M3-PN170045
Effective Date:
December 22, 2020
Issue Date:
January 27, 2021

ARCHITECTURE
ENGINEERING
CONSTRUCTION MANAGEMENT

STIBNITE GOLD PROJECT

FEASIBILITY STUDY TECHNICAL REPORT

Valley County, Idaho

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The effective date of this Report is December 22, 2020. The issue date of this Report is January 27, 2021. See Appendix I, Feasibility Study Contributors and Professional Qualifications, for certificates of Qualified Persons.

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1 SUMMARY

1.1 INTRODUCTION

Since inception, Midas Gold's vision for the Stibnite Mining District (the **District**) has been to use modern mining to redevelop an abandoned, brownfield mine site, provide long-term employment and business opportunities for a rural area in Idaho, funded by an economically viable project. The Project, as envisioned in this Feasibility Study (**FS**), would become one of the largest and highest-grade open pit gold mines in the United States and the country's only primary producer of antimony, a critical and strategic mineral. The FS builds upon Midas Gold's Plan of Restoration and Operations (**PRO**) (Midas Gold, 2016), identifying a suite of operational improvements and environmental refinements to achieve the Company's key objective for the financially viable restoration and brownfields development of the Stibnite mining district.

Restoration goals were established early on to address environmental impacts from over 100 years of historical mining activities and return the site to a fully functioning, self-sustaining ecosystem with improved water quality and habitat capable of supporting enhanced populations of fish, wildlife and flora. In addition to gold, the District also contains significant Mineral Reserves of antimony, a metal on the U.S. Department of Interior's final list of 35 critical minerals (Dept. of Interior, 2018) and referred to informally as a critical mineral herein.

This Technical Report (**Report**) provides a comprehensive overview of the Stibnite Gold Project (**Project**) and includes recommendations for future work programs required to advance the Project to a decision point. It provides information about the geology, mineralization, exploration potential, Mineral Resources, Mineral Reserves, mining method, process method, infrastructure, social and economic benefits, environmental protection, cleanup and repair of historical impacts, permitting, reclamation and closure concepts, capital and operating costs and an economic analysis for the Project. In summary, this Report defines an economically feasible, technically and environmentally sound Project that achieves redevelopment and restoration goals for the Stibnite Mining District.

For readers to fully understand the information in this Report, they should read this Report in its entirety, including all qualifications, assumptions and exclusions that relate to the information set out in this Report that qualifies the technical information contained in the Report. The Report intended to be read as a whole, and sections should not be read or relied upon out of context. The technical information in the Report is subject to the assumptions and qualifications contained in the Report. The economic and technical analyses included in this Report provide only a summary of the potential Project economics based on the assumptions set out herein. There is no guarantee that the Project economics described herein can be achieved.

1.2 BACKGROUND

After a number of years of collecting technical and environmental baseline data on the District and understanding the legacy impacts from past mining activity, engaging with stakeholders, and developing an environmentally, socially and economically feasible path forward, Midas Gold completed a preliminary feasibility study (**PFS**) in 2014 (M3, 2014) and submitted its PRO for the Project to regulators in September 2016. The PRO formed the basis for Alternative 1 in the Draft Environmental Impact Statement **(DEIS)** (USFS, 2020). Continued evolution of the Project following environmental modeling and analysis resulted in a modified PRO **(ModPRO)** (Brown and Caldwell, 2019) filed with regulators in May 2019, which formed the basis of Alternative 2 in the DEIS. The plan laid out in the PRO and ModPRO was founded on Midas Gold's core values of safety, environment, community involvement, transparency, accountability, integrity and performance. These core values led to the development of a number of key conservation guidance principles for the design of the Project:

- Meet society's present day needs for economic prosperity and mineral production while remaining protective of the environment and ensuring sustainability for future generations.

- Design with closure in mind, providing a long-term foundation for a naturally sustainable ecosystem.
- Conduct activities in an environmentally responsible manner.
- Reclaim, reprocess, or reuse legacy mining materials and restore legacy mining impacts during construction and early operations.
- Limit the Project footprint to previously disturbed areas, to the extent reasonably practicable and feasible.
- Improve on existing environmental conditions, especially with respect to water quality and fish and wildlife migration, populations and habitat throughout the Project life.
- Restore the impacts of development and replace the ecosystem function of affected features.
- Ensure local and regional financial and social benefits by prioritizing local hiring, training, purchasing, and contracting.

Since filing the PRO, Midas Gold has continued to advance the Project along two parallel paths: additional design and engineering studies in support of the FS; and further environmental modeling and analysis in support of Project permitting. In anticipation of the effects analysis in the DEIS, after considering comments received during stakeholder engagement discussions pre-release of the DEIS, and in response to the comments submitted during the official public comment period on the DEIS, Midas Gold has further refined the Project in this FS by incorporating a suite of operational improvements and Project modifications that reduce environmental, social and economic impacts identified in the PRO, ModPRO or DEIS. Key environmentally-focused modifications relative to the ModPRO incorporated in the Report include:

- Reducing the size of the Hangar Flats pit and associated water management risks and costs;
- Elimination of the Fiddle DRSF resulting in a reduction in Project footprint, and water management and reclamation requirements;
- Backfilling of the Hangar Flats pit to the pre-mining valley bottom elevation thereby preventing formation of a pit lake and mitigating impacts to water quality and stream temperature in Meadow Creek;
- Changes to the DRSF design and sequencing to allow for stockpiling and processing of low-grade ore, thereby eliminating the need to permanently place low-grade ore in DRSFs;
- Modifications to the stream and riparian restoration designs to further address stream temperature impacts;
- Optimization of limestone dosage into the pressure oxidation circuit to enhance the environmental stability of arsenic in mine tailings;
- Elimination of the countercurrent decantation circuit (**CCD**) reducing the process plant footprint and construction and operating costs; and,
- A comprehensive contact water management and water treatment plan.

These Project modifications are in addition to operational improvements and environmental protection measures adopted in the ModPRO and Alternative 2 of the DEIS (when compared to the PRO) that included:

- Elimination of the West End DRSF and partial backfilling of the Hangar Flats and West End pits;
- Installation of low permeability covers on DRSFs to reduce contact water seepage and infiltration;
- Onsite lime generation to reduce trucking requirements and operational expenses; and,
- Modifications to surface water management strategies to reduce the volume of water handling and improve site water quality.

The Project, as currently envisioned in this FS, integrates the results and findings of scientific investigations, engineering studies and stakeholder engagement activities conducted over the last decade into an environmentally, socially and economically feasible plan that redefines modern mining practices and principles to achieve environmental restoration of an abandoned mine site and create long-term economic benefits for the community and Project stakeholders.

1.3 KEY RESULTS

The Project consists of mining the Yellow Pine, Hangar Flats and West End deposits using conventional open pit methods, conventional processing methods to extract gold, silver and antimony, and on-site production of gold (**Au**) and silver (**Ag**) doré and an antimony (**Sb**) concentrate. The Project also entails an extensive reclamation and restoration program for historical impacts to the site including the recovery and reprocessing of Historical Tailings, restoration of fish passage during and after operations, relocation of historical mining wastes to engineered storage facilities, stream restoration, and reforestation of impacted areas. Midas Gold's plans for decommissioning the site include progressive and concurrent remediation, reclamation and restoration activities, beginning at the start of construction and continuing beyond the operations phase, through Project reclamation and closure.

The Stibnite Gold Project economics, as contemplated in the FS, are summarized in Table 1-1:

Table 1-1: Stibnite Gold Project Feasibility Study Highlights

Component	Early Production Years 1-4	Life-of-Mine Years 1-15
Recovered Gold (2) Total	1,853 koz	4,238 koz
Recovered Antimony Total	74 millon lbs	115 million lbs
Recovered Gold (2) Annual Average	463 koz/yr	297 koz/yr
Cash Costs(2) (Net of by-product credits)	$328/oz	$538/oz
All-in Sustaining Costs(2) (Net of by-product credits)	$438/oz	$636/oz
Initial Capital – including contingency	$1,263 million	
Case B at US$1,600/oz gold (Base Case) (1)		
After-Tax Net Present Value 5%	$1,320 million	
Annual Average EBITDA	$566 million	$292 million
Annual Average After Tax Free Cash Flow	$500 million	$242 million
Internal Rate of Return (After-tax)	22.3%	
Payback Period in Years (After-tax)	2.9 years	
Case C at US$1,850/oz gold (1)		
After-Tax Net Present Value 5%	$1,864 million	
Annual Average EBITDA	$678 million	$360 million
Annual Average After Tax Free Cash Flow	$584 million	$295 million
Internal Rate of Return (After-tax)	27.7%	
Payback Period in Years (After-tax)	2.5 years	

Notes:
(1) Base case prices US$1,600/oz gold, $20/oz silver and $3.50/lb antimony, Case C price based on metal selling prices of US$1,850/oz gold, $24/oz silver and $3.50/lb antimony, Post-Tax NPV at 5% discount rate.
(2) In this release, “M” = million, “k” = thousand, all amounts in US$, gold and silver reported in troy ounces (“oz”).
(3) See non-International Financial Reporting Standards (“IFRS”) measures below.
(4) All numbers have been rounded in above table and may not sum correctly.
(5) The FS assumes 100% equity financing of the Project.

The FS affirms that the Project can address legacy impacts left behind by previous mining operators including the recovery, reprocessing and safe storage of historical tailings, restoration of fish passage, stream restoration, and reforestation. The FS verifies a positive local economic benefit to Idaho communities bringing more than $1 billion in initial capital investment, approximately 550 direct jobs during operations, and hundreds of indirect and induced jobs, while generating significant taxes and other benefits to the local, state and national economies.

1.4 REGULATORY INFORMATION

This Report has been prepared based on the results of a FS completed for the Project, which is located in the Stibnite-Yellow Pine mining district (**District**), Idaho. The Project is wholly owned by direct or indirect subsidiaries of Midas Gold Corp. ("**MGC**"), a TSX-listed British Columbia company. Unless the context indicates otherwise, references to "**Midas Gold**" throughout this Report include one or more of the aforementioned subsidiaries of MGC.

The FS was compiled by M3 Engineering & Technology Corp. (**M3**) which was engaged by Midas Gold, through its subsidiary Midas Gold Idaho, Inc. (**MGII**), to evaluate the development of the Stibnite Gold Project based on information available up to the date of the FS. The FS was prepared under the direction of Independent Qualified Persons (**QP**s) and in compliance with National Instrument 43-101 the Canadian Securities Administrators (**NI 43-101**) standards for reporting mineral properties. Additional details of the qualifications and responsibilities of preparers are provided in Appendix I.

The FS supersedes and replaces the technical report entitled "Amended Preliminary Feasibility Study Technical Report for the Stibnite Gold Project, Idaho" prepared by M3 and dated March 28, 2019 and that report should no longer be relied upon. Mineral Resource Statements in the FS supersede and replace the Mineral Resources disclosed publicly on February 15, 2018, which should no longer be relied upon.

1.5 PROPERTY DESCRIPTION AND LOCATION

The Project is located in central Idaho, USA approximately 100 miles (**mi**) northeast of Boise, Idaho, 38 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho (see Figure 1.1). Mineral rights controlled by Midas Gold include patented lode claims, patented mill sites, unpatented federal lode claims, and unpatented federal mill sites and encompass approximately 27,104 acres or 42 square miles. The claims are 100% owned, except for 27 patented lode claims that are held under an option to purchase. The Project is subject to a 1.7% NSR Royalty on gold only; there is no royalty on silver or antimony.

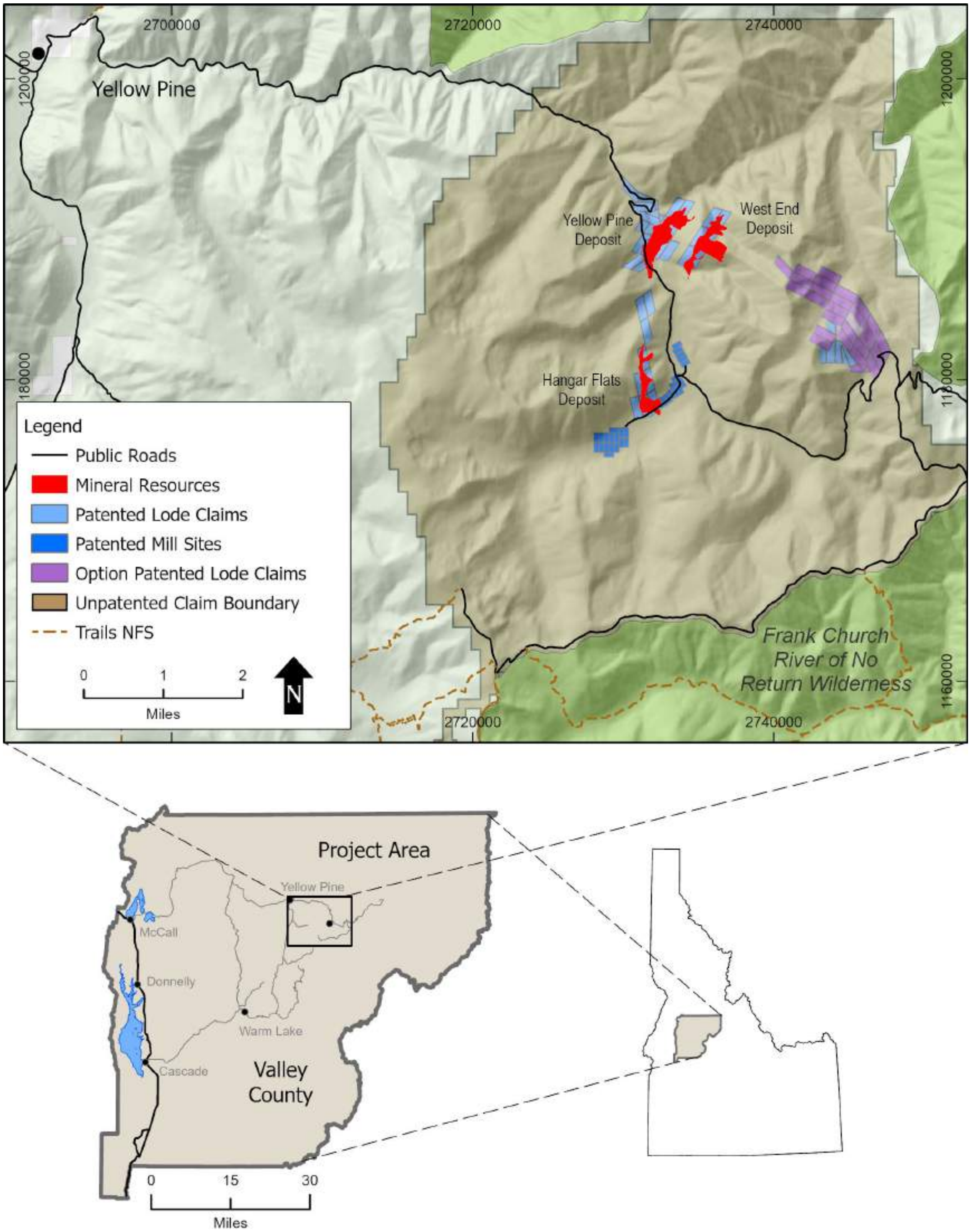
1.6 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The Project site is located approximately 152 road-miles northeast of Boise, Idaho within the East Fork of the South Fork of the Salmon River (**EFSFSR**) watershed at an elevation of ~ 6,500 feet (**ft**); nearby mountain peak elevations range from approximately 7,800 to 8,900 ft.

The climate is characterized by moderately cold winters and mild summers. Most precipitation occurs as snowfall in the winter and rain during the spring. The local climate allows for year-round operations, as evidenced by historical production over extended periods, and climate information.

Ground access to the Property is currently available by road from the nearby towns of Cascade, Idaho, an 84-mile drive and, during the snow free months, from McCall, Idaho, which is a 63-mi drive. Powerlines would need to be installed/upgraded from the main regional Idaho Power Corporation (**IPCo**) substation at Lake Fork to the Project site, a distance of 42 mi.

Figure 1.1: Location of the Stibnite Gold Project



1.7 SITE HISTORY

Two major periods of mineral exploration, development and operations have occurred in the District, leaving substantial environmental impacts that remain to this day. The first period of activity commenced in the mid-1920s and continued into the 1950s; it involved the mining of gold, silver, antimony, and tungsten mineralized materials by both underground and, later, open pit mining methods. During World War II and the Korean War, this District is estimated to have produced more than 90% of the U.S.' antimony and approximately 50% of the U.S.' tungsten; materials that were used in munitions, steelmaking, flame retardants and for other purposes. Mining of these strategic minerals was considered so critical that the U.S. federal government subsidized the mining activity, managed site operations and military time could be served at the mine site. Estimated production during this period totaled an estimated 0.53 Moz Au, 88 Mlbs of antimony and 13.6 Mlbs of contained tungsten.

The second period of major activity in the District started with exploration activities in the early 1970s and was followed by open pit mining and heap leaching from 1982 to 1997, with ore provided by multiple operators from a number of locations and processed in one-time and seasonal on-off heap leach facilities in Meadow Creek Valley. Gold production during this period totaled an estimated 0.45 Moz Au.

Both the East Fork of the South Fork of the Salmon River and its tributary Meadow Creek have been severely impacted by past mining activity. Additional impacts related to extensive forest fires and the failure of an earthen dam on "Blowout Creek", a tributary of Meadow Creek, have compounded the mining-related impacts and have increased soil erosion and impacted water quality.

1.8 GEOLOGICAL SETTING AND MINERALIZATION

Bedrock in the region can be subdivided into the pre-Cretaceous metasedimentary "basement," the Cretaceous Idaho Batholith, Tertiary intrusions and volcanics, and Quaternary unconsolidated sediments and glacial materials. The SGP is situated along the eastern edge of the Idaho Batholith, on the western edge of the Thunder Mountain caldera complex and within the Central Idaho Mineral Belt.

Large, north-south striking, steeply dipping structures exhibiting pronounced gouge and multiple stages of brecciation occur in the District and are often associated with east-west and northeast-southwest trending splays and dilatant structures. The Yellow Pine and Hangar Flats deposits are hosted primarily by intrusive phases of the Idaho Batholith along the Meadow Creek Fault Zone. The West End Deposit is hosted primarily by Neoproterozoic to Paleozoic metasedimentary rocks of the Stibnite roof pendant along the West End Fault Zone.

Mineralization and alteration in the District are associated with multiple hydrothermal alteration events occurring through the Paleocene and early Eocene epochs. Main-stage gold mineralization and associated potassic alteration typically occurs in structurally prepared zones in association with very fine-grained disseminated arsenical pyrite (FeS_2) and, to a lesser extent, arsenopyrite (FeAsS), with gold almost exclusively in solid solution in these minerals. Antimony mineralization occurs primarily associated with the mineral stibnite (Sb_2S_3). Additional gold mineralization effecting rocks of the Stibnite roof pendant is associated with epithermal quartz-adularia-carbonate veins.

Deposits of the District are not readily categorized based on a single genetic deposit model due to complexities associated with multiple overprinting mineralization events and uncertainties regarding sources of mineralizing hydrothermal fluids.

1.9 EXPLORATION

The District has been the subject of exploration and development activities for nearly 100 years, yet much of the area remains poorly explored due to its remote location, poor level of outcrop and extensive glacial cover. Midas Gold has completed extensive exploration work over the last decade that has included: geophysics; rock, soil and stream sampling and analysis; geologic mapping; mineralogical and metallurgical studies; and drilling.

This newer data has been integrated with datasets from previous operators and provides a comprehensive toolkit for future exploration. These efforts have led to the identification of over 75 prospects with varying levels of target support. These prospective areas include targets within, under and adjacent to existing deposits; bulk mineable prospects along known or newly identified mineralized trends; high grade underground targets and early-stage greenfield prospects and conceptual targets based on geophysics or geologic inference. Details of some of the more promising targets are summarized in Section 9 of this Report.

Exploration targets include conceptual geophysical targets, geochemical targets from soil, rock and trench samples, and results from widely spaced drill holes; as a result, the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

1.10 DRILLING

The Project area, including the three main deposits, has been drilled by numerous operators, totaling 793,769 ft in 2,723 drill holes, of which Midas Gold drilled 637 holes totaling over 344,465 ft since 2009. Pre-Midas Gold drilling was undertaken by a wide variety of methods and operators while Midas Gold employed a variety of drilling methods including core, Reverse Circulation, auger, and sonic throughout the District, but with the primary method being core.

1.11 DATA VERIFICATION

It is the opinion of the Independent QP responsible for the Mineral Resource estimates that the data used for estimating the Mineral Resources and Mineral Reserves for the Hanger Flats, West End, Yellow Pine and Historical Tailings deposits is adequate for this purpose and may be relied upon to report the Mineral Resources and Mineral Reserves contained in this Report.

1.12 MINERAL PROCESSING AND METALLURGICAL TESTING

1.12.1 Process Flowsheet Development

Process mineralogical studies supporting the 2012 PEA and 2014 PFS indicate that gold in all three deposits is hosted in pyrite and arsenopyrite and is predominantly refractory to direct cyanidation; however, discrete free gold is present in oxidized portions of the West End Deposit. Antimony in the Yellow Pine and Hangar Flats Deposits occurs almost entirely as stibnite and is typically coarse-grained when occurring at head grades above 0.1% antimony, and stibnite becomes sufficiently liberated for recovery via selective antimony flotation.

Considerable testing supporting the 2012 PEA and 2014 PFS studies were conducted on samples from the Yellow Pine, Hangar Flats and West End deposits that supported a process flowsheet entailing bulk sulfide flotation to maximize recovery of gold to a sulfide concentrate amenable to treatment by pressure oxidation for materials assaying less than 0.1% antimony. Based on this work, high antimony materials would be subject to a selective antimony flotation process, thereby producing a shippable antimony concentrate, with a gold-bearing bulk sulfide rougher concentrate to be floated from the antimony flotation tailings. Some of the oxidized West End ores are more transitional or free milling in nature, and an ore leaching process was developed to treat these materials. Testing was also conducted on samples of the historical (Bradley) tailings. This work showed the historical tailings could be processed using the same flowsheet most likely as a blend with fresh sulfide ores.

1.12.2 Comminution and Flotation Studies

Comminution testing, including 31 JK Drop Weight and SMC tests, 36 Bond Ball Mill Work Indices, 21 Bond Rod Mill Work Indices, 19 Crusher Work Indices and 14 Abrasion Indices have been conducted on samples from the project. These data show the ores to be amenable to SAG milling and the Bond Ball Mill work index to a closing size of 150 microns, averages 13.5 kWh/tonne.

The majority of flotation testwork conducted since the PFS focused on optimizing bulk sulfide rougher flotation and concentrate upgrading. Five master composites were subjected to different treatment schemes varying the selection and dosage of activators, depressants, collectors and frothers to economically optimize the dosage of each of the key flotation reagents. Concentrate upgrading was deemed necessary to reduce slurry viscosity and achieve autothermic conditions in the autoclave through reduction of potassium jarosite formation. Cleaner flotation testing of the rougher concentrate successfully upgrades sulfur concentration from 5% to 7.5% with gold losses of 1-2%. An extensive trade-off testing program identified the optimal grind size as 80% passing 85 microns based on replicate batch testing and locked cycle tests on a suite of master composites.

Flotation pilot plant runs on 3,600 kg of early production sulfide material from Yellow Pine were conducted to generate material for autoclave testwork and included rougher flotation and concentrate upgrading achieving the target 7.5% sulfur grade. Additional flotation pilot plant work was conducted to create a bulk antimony concentrate. Additional testwork focused on cyanide leaching of West End transitional flotation concentrates and flotation tailings, whole ore leaching of West End oxides, and the use of POX CCD overflow to liberate gold from cleaner tailings.

A variability study was conducted to assess performance of the mineral processing circuit on different ore sub-types and to support predictions of overall metallurgical recoveries. Forty-four variability composites were developed to represent the major lithological and alteration material blends to be processed from the three deposits during different project periods. Lithological controls were not found to impart significant variability on gold recoveries with the exception of clay rich fault gouge and transitional materials.

Projected gold flotation recoveries for low-antimony materials to a concentrate assaying 6.5% sulfur are estimated at 93.8% for Yellow Pine and 92.1% for Hangar Flats. Silver recoveries are estimated as 90.1% for Yellow Pine and 89.1% for Hangar Flats. Gold and silver flotation recoveries are independent of gold or sulfur grade. For high-antimony materials from the Yellow Pine deposit, gold misplacement to the antimony concentrate and overall gold recoveries to POX are functions of pyritic sulfur grade and are estimated to range from 83.6% to 95.5%. Constant gold and silver recoveries are projected for Hangar Flats high-antimony material at 89.7% for gold and 43.2% for silver.

West End sulfide material is highly refractory while transition material has a significant free milling gold content. Sulfide material will be processed by flotation, concentrate POX and cyanide leaching of the concentrate; transition material will be treated similarly, however the flotation tailings will also be leached; oxide materials will just be leached. Metallurgical predictions for West End are based on cyanide leachability and on a target concentrate carbonate to sulfur ratio of 1.3:1 CO_3/S, as the presence of excessive carbonate in the concentrate inhibits autothermic oxidation in the autoclave and associated gold recovery.

1.12.3 Hydrometallurgical Studies

Batch and pilot plant testwork for the POX and neutralization processes were completed at AuTec (Vancouver, Canada), CESL (Vancouver, Canada) and SGS (Malaga, Perth). These tests were performed on various concentrates derived from ore samples that represent parts of the deposits and mill feed over the life of mine.

Batch and pilot tests at AuTec showed that Project gold concentrates were amenable to acid pressure oxidation at 220°C, 462 kPa (67 psi) oxygen partial pressure, and a retention time of approximately 60 minutes. The optimized

POX feed density appeared to be in the range of 30-35% for all concentrates. After a hot acid cure and CIL, the gold recoveries were typically 95 to 98% of the gold in the concentrate feed. Mineralogical tests on POX residues generated at AuTec that were subjected to CIL confirmed the presence of potassium jarosite which in turn had some grains of occluded gold that had escaped extraction in CIL.

Oxidation tests were also undertaken at CESL and at SGS to investigate neutralization of acid inside the autoclave, or "*in-situ* acid neutralization" (**ISAN)**. Neutralization of acid inside the autoclave was accomplished by adding ground limestone in the POX feed to control free acid and sulfate concentrations and limit the formation of jarosites and basic iron sulfates. The objective would be higher ferric concentrations available for scorodite formation and lower sulfate concentrations that would inhibit pitticite (an unstable arsenic compound) formation. The SGS tests confirmed consistent gold recoveries in the 96.5-99.0% range.

The U.S. Environmental Protection Agency (**EPA**) Synthetic Precipitation Leaching Procedure (**SPLP**) results confirmed there were additional benefits from ISAN, with SPLP arsenic concentrations decreasing with increasing CO_3/S mass ratios to about 1.25 or higher. The CO_3/S ratio, which reflects the magnitude of limestone added, did not appear to affect the silver CIL recovery.

The continuous POX pilot plant was undertaken at SGS Malaga during the period of 20th to 26th November 2017. The test feed concentrate was generated from low-antimony samples from the Yellow Pine and Hangar flat deposits. The testing was conducted in a 22-liter autoclave with four compartments at feed rate of 4-6 kg/h and a nominal residence time of 75 minutes. The operating parameters were the same as those established in previous batch tests, but with varying levels of limestone additions to the feed to achieve a range of gross CO_3/S ratios. The autoclave residue was treated by hot acid cure (**HAC**) and neutralized prior to cyanide leaching.

The results show increasing gold extraction at higher CO_3/S ratios up to a value of 1.2; further increases in CO_3/S ratio appeared to have minimal effect. Increasing the CO_3/S ratio also appears to favor lower arsenic SPLP values and hence improved arsenic stability in the leach residues. Quantitative mineralogy on the pilot autoclave solids suggested that iron was precipitated as iron (III) hydroxide (or ferrihydrite), and arsenic was precipitated predominantly as scorodite, a stable arsenic product.

Overall projected life-of-mine metallurgical recoveries are provided in Section 1.16.2.

1.12.4 Arsenic Stability Studies

In the initial metallurgical pilot test work conducted at AuTec the arsenic in the pressure leach residues was unstable, possibly because of the preferential formation of pitticite over scorodite. In subsequent metallurgical testing at SGS the stability of arsenic improved with increases in the CO_3/S ratio to as high as 1.6. The alkalinity in the limestone was postulated to have reduced the propensity for hydroxy-sulfate compounds, such as basic ferric sulfate and potassium jarosite, to form and released iron to form ferrihydrite and to sequester arsenic as a more stable scorodite. However, subsequent environmental geochemical testing completed on commingled flotation and detoxified cyanide leach tailings from the SGS pilot plant indicated that arsenic destabilized at some point downstream of the POX process; consequently, a testing program was initiated at SGS commencing April 2020 to establish how and where the destabilization occurred. This program included ISAN POX tests with a terminal free acid of 8 to 13 mg/L of H_2SO_4, atmospheric arsenic precipitation (**AAP**), and a two-step neutralization procedure. The AAP process precipitates iron and arsenic slowly at an elevated temperature (92°C) by progressively adding limestone to achieve a pH of approximately 2 with a retention time of 4 to 5 hours. Test results suggest that under these conditions, a stable scorodite precipitate ($FeAsO_42H_2O$) formed.

Batch neutralization tests were conducted at two discrete pH regions: neutralization to pH 5 with limestone followed by neutralization to pH 10 with lime. The results show that the slurry temperature during the pH 5 neutralization step has no impact on arsenic stability; however, during the pH 10 neutralization step for slurry temperatures greater than 45°C arsenic destabilization occurred. The destabilization was postulated to be related to the reaction between free hydroxyl ions and the remaining pitticite. SPLP testing confirmed that reducing the neutralization temperature of the pH 5 slurry to 45°C prior to raising the pH to 10 minimize this reaction. Consequently, the FS flowsheet includes a two-step neutralization circuit, with a cooling circuit between the neutralization steps.

1.13 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimates for the Project were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (**CIM**) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" as adopted by CIM Council November 29, 2019 and are reported in accordance with NI 43-101 requirements. The Mineral Resource estimates for each of the Hangar Flats, West End and Yellow Pine deposits, and the Historical Tailings, were prepared using commercial mine-modeling and geostatistical software, take into account relevant modifying factors, and have been verified by an Independent QP. The consolidated Mineral Resource statement for the Project in metric tonnes (t) is shown in Table 1-2 based on a gold selling price of US$1,250/troy ounce limiting pit shell.

Table 1-2: Stibnite Gold Project Consolidated Mineral Resource Statement

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Measured ("M")							
Yellow Pine	4,902	2.42	382	3.75	590	0.24	25,831
Indicated ("I")							
Yellow Pine	45,350	1.72	2,509	2.07	3,020	0.09	85,774
Hangar Flats	25,861	1.44	1,194	3.24	2,697	0.15	84,463
West End	53,469	1.08	1,849	1.31	2,259	0.00	0
Historical Tailings	2,687	1.16	100	2.86	247	0.17	9,817
Total M & I	132,269	1.42	6,034	2.07	8,814	0.07	205,885
Inferred							
Yellow Pine	3,214	0.96	99	0.60	62	0.00	50
Hangar Flats	12,224	1.12	440	2.64	1,037	0.11	28,560
West End	20,540	1.06	700	1.11	733	0.00	0
Historical Tailings	191	1.13	7	2.64	16	0.16	662
Total Inferred	36,168	1.07	1,246	1.59	1,849	0.04	29,272

Notes:
(1) *All Mineral Resources have been estimated in accordance with CIM definitions, as required under NI 43-101.*
(2) *Mineral Resources are reported in relation to a conceptual pit shell to demonstrate potential for economic viability, as required under NI 43-101; mineralization lying outside of these pit shells is not reported as a Mineral Resource.* ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of inferred mineral resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.*
(3) *Open pit sulfide mineral resources are reported at an effective cut-off grade of 0.45 g/t Au and open pit oxide Mineral Resources are reported at an effective cut-off grade of 0.40 g/t Au.*
(4) *The Yellow Pine and Hangar Flats deposits contain zones with substantially elevated antimony-silver mineralization, defined as containing greater than 0.1% antimony. These higher-grade antimony zones comprise 18,477 kt grading 0.48% antimony of measured and indicated gold mineral resource estimates and 1,387 kt grading 0.93% antimony of inferred gold mineral resource estimates. Antimony mineralization is not classified separately from gold and is reported only if it lies within gold Mineral Resource estimates, and only if blocks meet gold cut-off grade criteria.*

The Yellow Pine and Hangar Flats deposits contain zones with substantially elevated antimony-silver mineralization, defined as containing greater than 0.1% antimony, relative to the overall Mineral Resource. The existing Historical Tailings Mineral Resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 1-3. Antimony Mineral Resources are reported only if they lie within gold Mineral Resource estimates.

Table 1-3: Antimony Sub-Domains within the Consolidated Mineral Resource Statement

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Measured							
Yellow Pine	2,142	2.76	190	5.79	399	0.52	24,429
Indicated							
Yellow Pine	7,086	2.17	495	5.28	1,204	0.52	80,606
Hangar Flats	6,562	2.10	443	7.89	1,664	0.55	79,179
Historical Tailings	2,687	1.16	100	2.86	247	0.17	9,817
Total M & I	18,477	2.07	1,228	5.91	3,513	0.48	194,031
Inferred							
Yellow Pine	10	1.21	0	2.78	1	0.18	41
Hangar Flats	1,185	2.40	92	15.27	582	1.07	27,829
Historical Tailings	191	1.13	7	2.64	16	0.16	662
Total Inferred	1,387	2.22	99	13.43	599	0.93	28,532

Notes:
(1) *Antimony mineral resources are reported as a subset of the total mineral resource within the conceptual pit shells used to constrain the total mineral resource in order to demonstrate potential for economic viability, as required under NI 43-101; mineralization outside of these pit shells is not reported as a mineral resource.* ***Mineral resources are not mineral reserves and do not have demonstrated economic viability. These Mineral Resource estimates include inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated.*** *All figures are rounded to reflect the relative accuracy of the estimate.*
(2) *Open pit antimony sulfide mineral resources are reported at a cutoff grade 0.1% antimony within the overall 0.45 g/t Au cutoff.*

1.14 MINERAL RESERVE ESTIMATES

The Mineral Reserve Estimates for the Project were estimated in conformity with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with NI 43-101. The Mineral Reserve estimates for each of the Yellow Pine, Hangar Flats, and West End deposits, and the Historical Tailings, were prepared to industry standards and best practices and take into consideration modifying factors including mining, processing, metallurgical, environmental, location and infrastructure, market factors, legal, economic, social, and governmental factors. The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery based on performance of the processing plant, and operating cost estimates.

The Mineral Reserve was developed by allowing only Measured and Indicated Mineral Resource blocks to contribute positive economic value and is a subset of the Mineral Resource comprised of the Probable Mineral Reserve that is planned for processing over the life-of-mine plan, with assumptions summarized in Sections 15 and 16. No economic credit has been applied to Inferred mineralization in the development of the Mineral Reserve, even if they lie within the Mineral Reserve pit.

The general mine planning sequence to produce the SGP Mineral Reserves estimate and associated mill feed schedule consisted of an ultimate pit limit analysis, pit shell selection, ultimate pit designs, internal pit phase design, mining sequence schedule, and mill feed optimization. A suite of nested pit shells for each deposit was generated using Geovia Whittle™ and a gold selling price ranging from $100 to $2,000 per troy ounce in $50 increments. The pit limit analysis was performed based on gold recovery only, to ensure the ultimate pit geometries would not be dependent on silver or antimony values. Mining costs used for the pit limit analysis are based on a first principal cost buildup for equipment requirements, labor estimates, and consumables price quotes. Selection of the optimal pit shells for each deposit was based on discounted cash flow analysis. For Yellow Pine and West End, the incremental change in discounted pit value **(NPV)** and strip ratio between potentially optimal pit shells is gradual, and pit shells representing gold selling prices of $1,250/oz and $1,300/oz respectively were selected. For Hangar Flats, the pit limit analysis suggested selecting the $1,150/oz pit shell but, due to additional technical considerations, the $750/oz pit shell was selected.

The ultimate pit designs were based on the selected pit shells, design parameters for 150-ton haul trucks, geotechnical design criteria, and additional mine sequencing and haulage considerations. Cut-off determination utilized a Net Smelter Return (**NSR**) methodology to account for varying ore types and separate process streams with unique process costs. The cut-off strategy applies elevated cut-off values to ensure the highest-grade ore available in the mine plan is processed preferentially and lower grade ore is stored in ore stockpiles for processing later in the Project life.

Cutoff grades for Mineral Reserves were developed assuming long term metal prices of $1,600/oz gold, $20.00/oz silver, and $3.50/lb antimony for material lying within the pit designs based on the pit shells selected above ($1,250, $750 and $1,300/oz Au for Yellow Pine, Hangar Flats and West End, respectively). This results in a Life-of-Mine (**LOM**) average gold cut-off grade of 0.48 g/t for open-pit mining. The Mineral Reserves are summarized in Table 1-4.

Table 1-4: Stibnite Gold Project Consolidated Mineral Reserve Summary

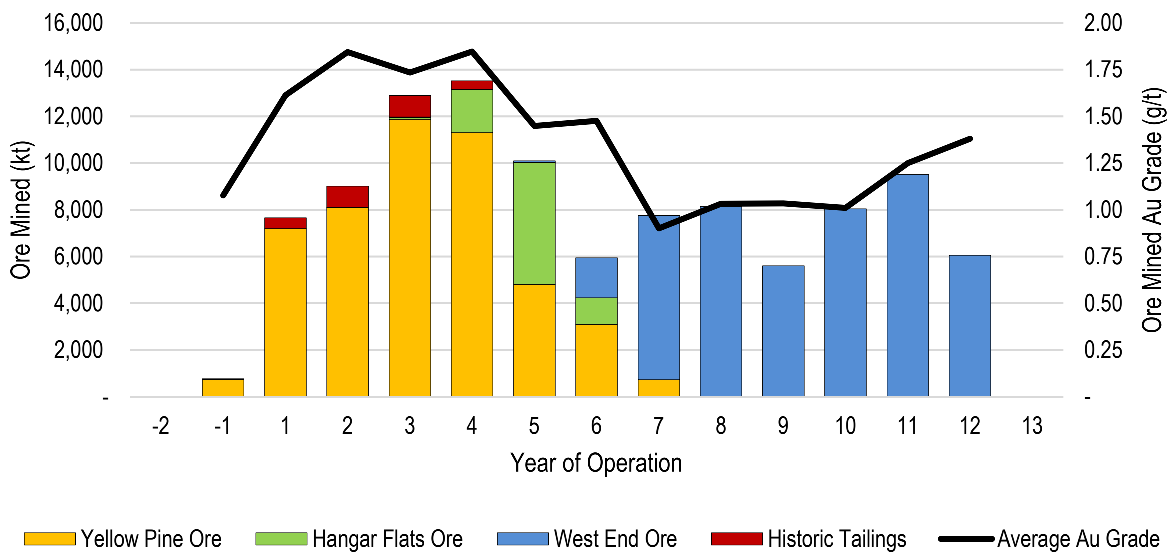
Deposit	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Yellow Pine							
Low Sb Sulfide – Proven & Probable	37,615	1.69	2,047	1.56	1,881	0.009	7,859
High Sb Sulfide – Proven & Probable	10,232	2.04	671	4.69	1,543	0.460	103,758
Yellow Pine Proven & Probable Mineral Reserves	**47,847**	**1.77**	**2,718**	**2.23**	**3,423**	**0.106**	**111,617**
Hangar Flats							
Low Sb Sulfide – Probable	5,167	1.34	223	1.65	273	0.018	2,104
High Sb Sulfide – Probable	3,095	1.92	191	4.85	483	0.369	25,148
Hangar Flats Probable Mineral Reserves	**8,262**	**1.56**	**414**	**2.85**	**756**	**0.150**	**27,252**
West End							
Oxide – Probable	4,749	0.54	83	0.87	133	-	-
Low Sb Sulfide – Probable	15,242	1.33	649	1.30	635	-	-
Transitional – Probable	25,839	1.03	855	1.49	1,236	-	-
West End Probable Mineral Reserves	**45,830**	**1.08**	**1,587**	**1.36**	**2,004**	**-**	**-**
Historical Tailings (1)							
Low Sb Sulfide – Probable	1,839	1.16	68	2.86	169	0.166	6,692
High Sb Sulfide – Probable	855	1.16	32	2.86	79	0.166	3,125
Historical Tailings Probable Mineral Reserves	**2,687**	**1.16**	**100**	**2.86**	**247**	**0.166**	**9,817**
Project Proven & Probable Mineral Reserves							
Oxide – Probable	4,749	0.54	83	0.87	133	-	-
Low Sb Sulfide – Proven & Probable	59,856	1.55	2,988	1.54	2,958	0.013	16,656
High Sb Sulfide – Proven & Probable	14,181	1.96	894	4.61	2,104	0.422	132,031
Transitional – Probable	25,839	1.03	855	1.49	1,236	-	-
Total Proven & Probable Mineral Reserves (2)(3)	**104,625**	**1.43**	**4,819**	**1.91**	**6,431**	**0.064**	**148,686**

<u>Notes:</u>
(1) *Historical Tailings ore type classification is proportional to the pit-sourced mill feed during Historical Tailings processing.*
(2) *Metal prices used for Mineral Reserves: $1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb.*
(3) *Antimony recovery is expected from High Sb Sulfide ore only, which contains 132,031 klbs of Sb.*

1.15 MINING METHODS

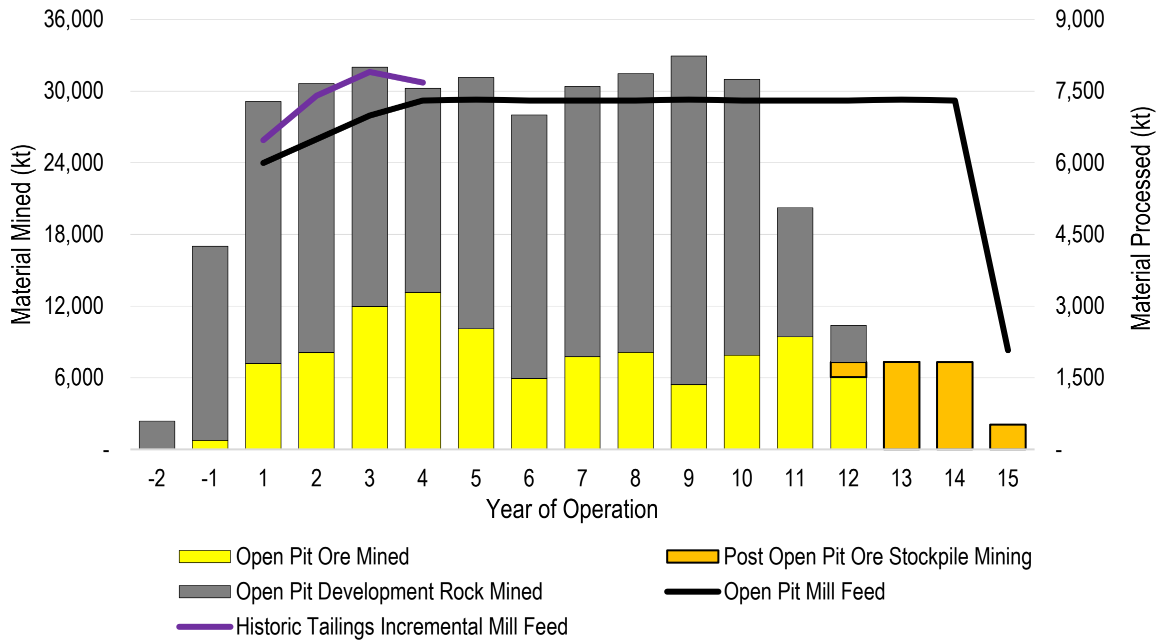
The mine plan developed for the Project incorporates the mining of the three *in situ* deposits: Yellow Pine, Hangar Flats, and West End and their related development rock; and the re-mining of Historical Tailings along with its cap of spent heap leach ore. The general sequence of open pit mining would be Yellow Pine deposit first, Hangar Flats deposit second, and West End deposit last, as shown on Figure 1.2. This sequence generally progresses from mining highest value ore to lowest value ore and accommodates the sequential backfilling the Yellow Pine and Hangar Flats open pits with material mined from West End open pit. Lower grade ore extracted during mining of the three pits is stockpiled and then processed during the operating life of the mill. The spent ore that overlies the Historical Tailings would be used as tailings storage facility ("TSF") construction material and is treated as stripping in the FS. Most development rock would be sent to one of five destinations: the TSF embankment, the TSF buttress, the Yellow Pine pit as backfill, the Hangar Flats pit as backfill, or the Midnight area within the West End pit as backfill. The Historical Tailings would be hydraulically transferred to the process plant during the first four years of operation, concurrent with mining ore from the Yellow Pine open pit.

Figure 1.2: Ore Mined by Deposit and Year



Mining at the SGP would be accomplished using conventional open pit hard rock mining methods with a production fleet consisting of two 28-yd^3 hydraulic shovels, one 28-yd^3 wheel loader, and a fleet of approximately eighteen 150-ton haul trucks. Mining is planned to deliver 7.30 Mt of ore to the crusher per year (nominally 20 kt per day) and approximately 22.1 Mt of development rock per year to DRSFs. Pre-stripping the open pits would begin two years prior to ore processing and open pit mining would continue until year 12 of operation. Once open pit mining is completed, the mining fleet will continue to provide ore to the mill from ore stockpiles until approximately the end of the first quarter in year 15 (Figure 1.3). A total of 102 Mt of ore would be mined from the three open pits and an additional 2.7 Mt of historic tailings would be mined. Approximately 254 Mt of development rock would be mined from the three open pits for a total of 356 Mt mined from the open pits and an average strip ratio (waste:ore) of 2.5.

Figure 1.3: Ore and Development Rock Mined by Year and Source



Long-term lower-grade ore stockpiles have been incorporated into the FS mine plan located for the most part within the footprint of the TSF buttress, thereby minimizing their incremental disturbance. The primary benefits to adding ore stockpile capacity is increased potential to optimize process ore feed value throughout the mine life, improved utilization of the Mineral Resource, reduced peak water treatment needs, reduced development rock tonnage and associated mining impacted water management. The stockpiling strategy is particularly significant during the first half of the mine life when Yellow Pine high value ore is mined at a rate greater than process plant throughput capacity. If stockpile capacity is not available, either the period-based cut-off value must increase resulting in ore converted to waste, or the mining rate reduced to align with process plant throughput capacity resulting in deferred access to high-value ore deeper in the open pit. The addition of long-term ore stockpiles allows for relatively high value ore mined from Yellow Pine open pit to be stockpiled and made available to process when lower value ore is being mined in West End open pit (Figure 1.4 and Figure 1.5).

Figure 1.4: Ore Stockpile Balance

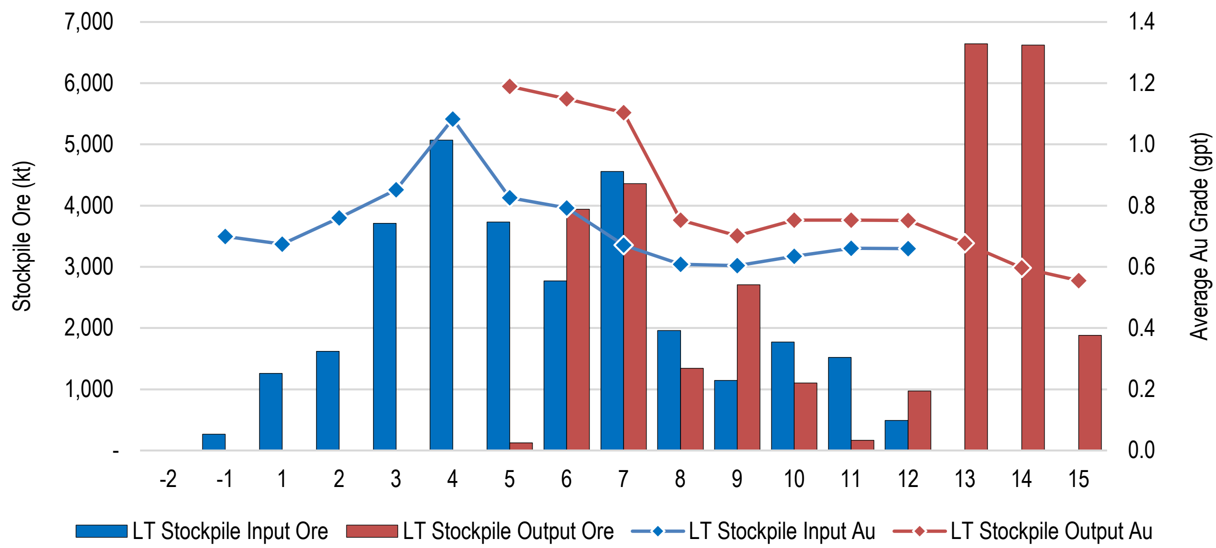
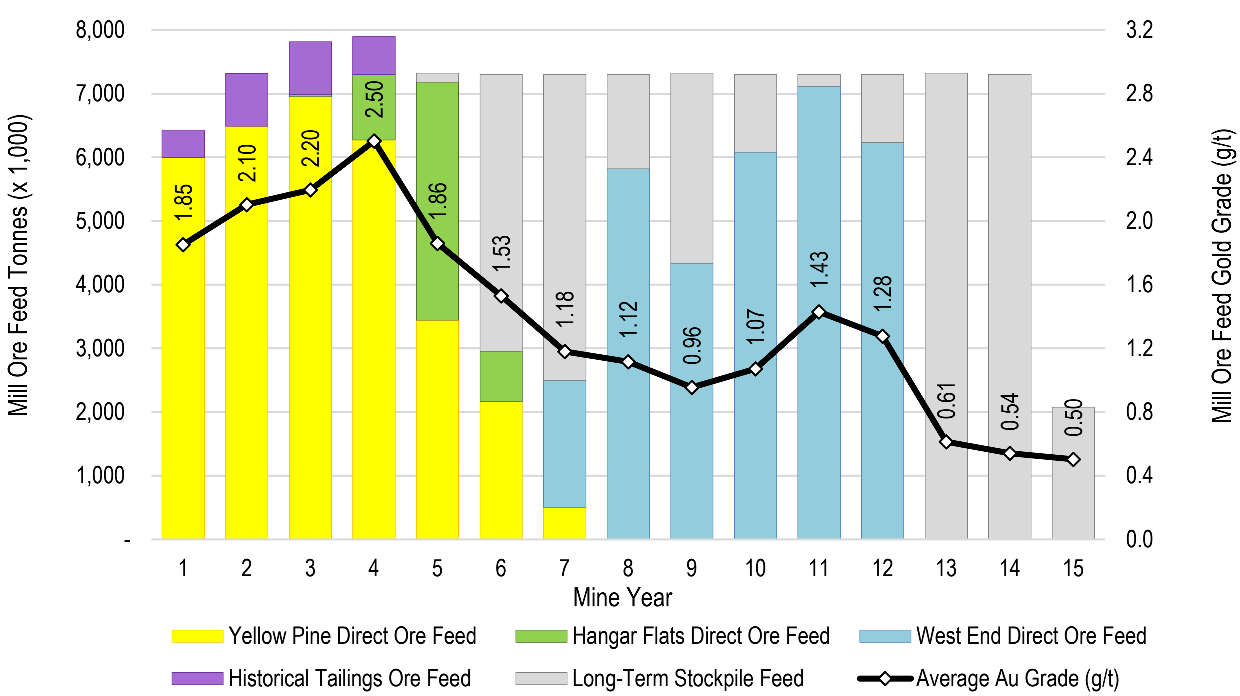


Figure 1.5: Mill Feed and Gold Head Grade by Deposit and Year



A summary of the mining statistics by ore type is provided in Table 1-5

Table 1-5: Life-of-Mine Mining Statistics

General Life-of-Mine Production	Unit	Value				
Open Pit Development Rock Mined	Mt	254				
Open Pit Ore Mined	Mt	102				
Open Pit Strip Ratio	waste:ore	2.5				
Historical Tailings Mined	Mt	2.7				
Mining Cost	$/t	2.47				
Daily Mill Throughput	kt/day	20.0				
Annual Mill Throughput	Mt/yr	7.30				
Mine Life	years	12				
Mill Life	years	14.3				
Life-of-Mine Average	**Unit**	**Total Ore**	**Oxide Ore**	**High Sb Ore**	**Low Sb Ore**	**Transition Ore**
Tonnage Milled	Mt	104.6	4.7	14.2	59.9	25.8
Contained Au Mined	koz	4,819	83	894	2,988	855
Contained Ag Mined	koz	6,431	133	2,104	2,958	1,236
Contained Sb Mined	klb	148,686	-	132,031	16,656	-
Contained Au Grade Mined	g/t	1.43	0.54	1.96	1.55	1.03
Contained Ag Grade Mined	g/t	1.91	0.87	4.61	1.54	1.49
Contained Sb Grade Mined	%	0.064	-	0.422	0.013	-

1.16 RECOVERY METHODS

1.16.1 Ore Processing

The Project's process plant has been designed to process sulfide, transition and oxide material from the Yellow Pine, Hangar Flats, and West End deposits. The processing facility is designed to treat an average of 20,000 t/d, or 7.3 Mt/y. Additionally, the Historical Tailings would be reprocessed early in the mine life to recover precious metals and antimony, and to provide space for the TSF embankment and buttress.

The process operations include the following components:

- **Crushing Circuit** – ROM material would be dumped onto a grizzly screen and into the crusher dump hopper feeding a jaw crusher operating at an average utilization of 75% yielding an instantaneous design-throughput of 1,111 tonnes per hour (**t/h**).
- **Grinding Circuit** – The grinding circuit incorporates a single semi-autogenous (**SAG**) mill, single ball mill design with an average utilization of 90%, yielding an instantaneous design-throughput of 926 t/h. When Historical Tailings are processed during early years of the operation, the slurry from the plant would also flow to the cyclone feed pump box. Cyclone underflow flows by gravity to the ball mill; cyclone overflow, at 33% solids with a target size of 80% passing (**P_{80}**) 85 microns, would be screened to remove tramp oversize and flow through a feed sample system and on to the antimony or gold rougher flotation circuit, depending on the antimony concentration of the material.
- **Flotation Circuit (Antimony and Gold)** – The flotation circuit consists of up to two sequential flotation stages to produce two different concentrates; the first stage of the circuit was designed to produce an antimony concentrate when the antimony grade is high enough, or bypassed if not, and the second stage was designed to produce a gold-rich sulfide concentrate. The antimony concentrate will be packaged and sold. The gold-rich sulfide concentrate will be stored in three surge tanks.
- **Pressure Oxidation Circuit** – Concentrate from the surge tanks would be pumped to the autoclave feed tank, which would feed the autoclave. The autoclave is designed to provide 75 minutes of retention time at

220 degrees Celsius (428 degrees Fahrenheit) to oxidize the sulfides and liberate the precious metals. Autoclave discharge would be processed through flash vessels and gas discharge would be condensed and the remaining gas cleaned through a scrubber.

- **Oxygen Plant** – An oxygen plant producing 607 t/d of gas at 95 percent oxygen and a gauge pressure of 40 bars is planned. The oxygen would be from a vendor-owned oxygen plant located near the autoclave building providing the autoclave with an "over the fence" supply.
- **Lime Plant** – Limestone quarried from the West End pit would be hauled to an area south of the primary crusher pad. The material would be crushed and screened to feed the limestone grinding mill and the lime kiln. Ground limestone slurry and milk of lime are used to control acid in the autoclave, neutralize solutions and slurries coming out of the POX process, and control pH for leaching.
- **Oxidized Sulfide Processing** – After pressure oxidation, slurry discharge from the flash vessels would be neutralized and cooled prior leaching. The slurry would then be leached in cyanide solution, followed by a seven-stage pump-cell carbon-in-pulp (**CIP**) circuit for precious metal recovery from this high-grade stream. The sulfide CIP tailings would be detoxified and discharged to the flotation tailings thickener. Alternatively, the sulfide leach tailings would be combined with flotation tailings when the latter undergoes cyanide leaching, as described in the next bullet point.
- **Oxide Carbon-in-Leach and Tailings Detoxification** – A future oxide leach circuit is included in the design of the process plant to be running in Year 7 of mill operations. This circuit would recover gold from non-refractory material in the flotation tailings when the mill is processing transition ore from the West End deposit. This circuit would also directly process oxide material from the West End deposit as a whole-ore leach process, that is, without undergoing flotation.
- **Carbon Handling** – Loaded carbon from the CIP circuit would be processed through a conventional carbon handling circuit, using the hot pressure-stripping of loaded carbon.
- **Gold Room** – Precious metals would be recovered from the strip solution by electrowinning.
- **Tailings** – Neutralized and thickened tailings would be pumped from the process plant to the TSF in a HDPE-lined carbon steel pipe.
- **Process Control Systems** – The process plant design includes an integrated process control system.

The two finished products from the Stibnite Gold Project ore processing facility will be: gold/silver bars, known as doré; and antimony-silver concentrate.

1.16.2 Projected Metallurgical Recoveries

Based on the metallurgical studies presented in Section 1.12, the mine plan provided in Section 1.15, and the process flowsheet included in Section 1.16, Figure 1.6 and Figure 1.7 summarize the projected LOM metallurgical recoveries to gold and siver-rich dore, and antimony concentrate, respectively.

Figure 1.6: Projected LOM Metallurgical Recoveries to Doré

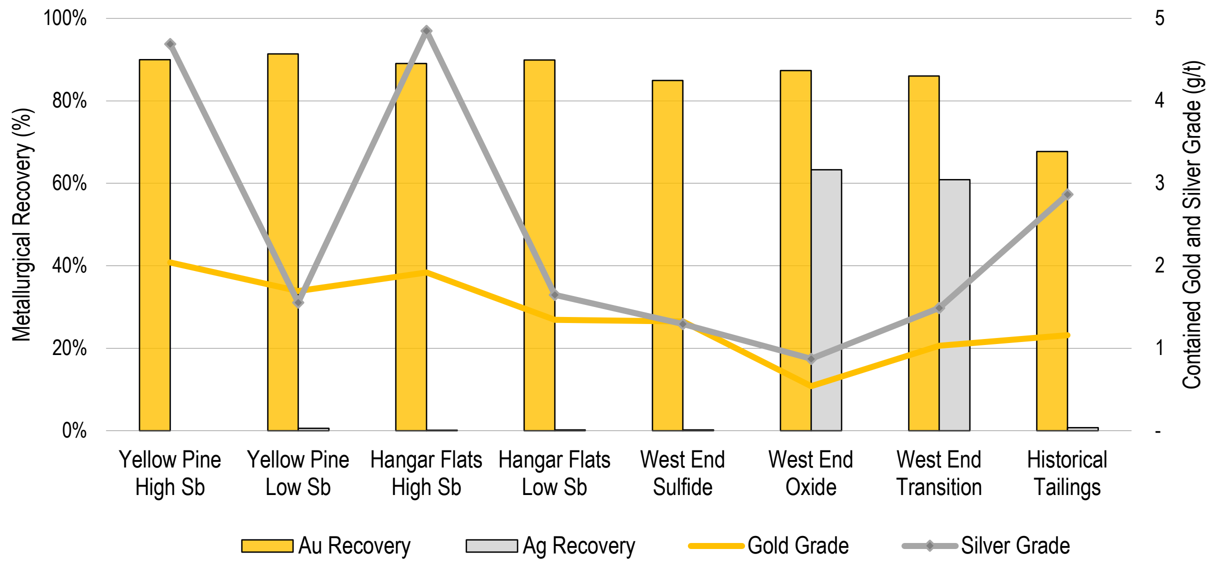
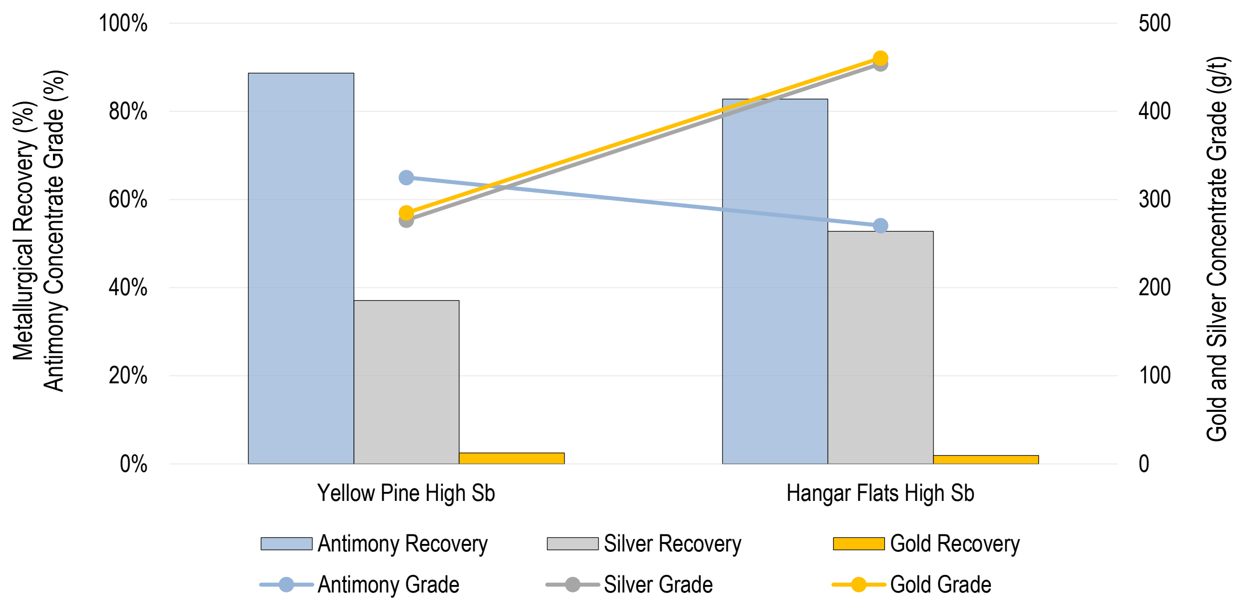


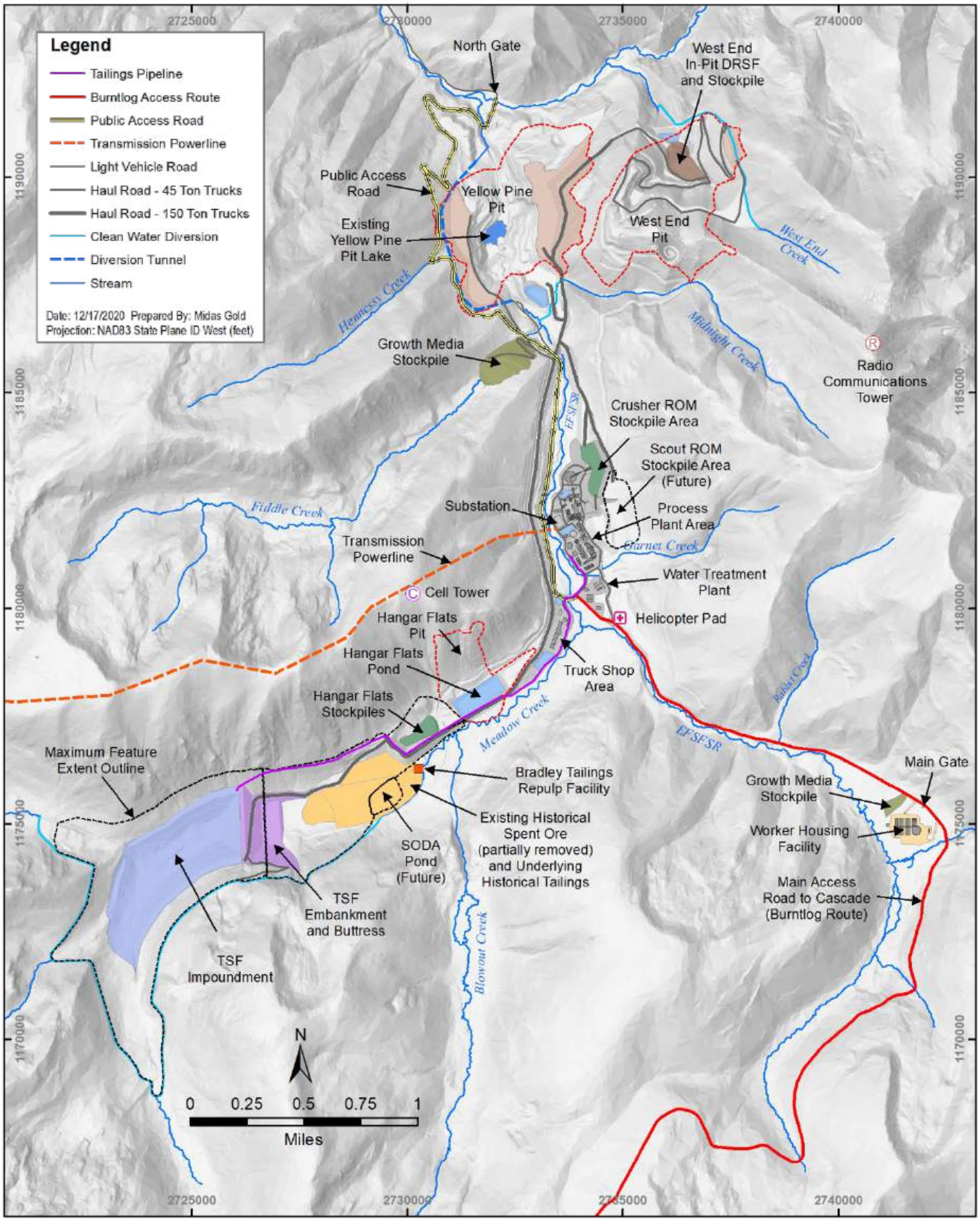
Figure 1.7: Projected LOM Metallurgical Recoveries to Antimony Concentrate



1.17 INFRASTRUCTURE

The Project will require upgrades to existing offsite infrastructure such as roads and power supply, as well as onsite and offsite infrastructure additions such as worker accommodations, water management systems, and tailings management systems. Section 18 provides a complete list and detailed descriptions of the infrastructure upgrades and additions required for the Project; provided below are summaries of some select key infrastructure. Figure 1.8 provides a general overview of the mine site at the beginning of the mine life.

Figure 1.8: Site Layout at the Beginning of Mine Life



1.17.1 Site Access

The site is currently accessed by the Stibnite Road, National Forest (NF-412), from the village of Yellow Pine, with three alternative routes up to that point. To address a number of shortcomings related to these routes, alternative access via the Burntlog Route was selected over several other possible alternatives because it provides safer year-round access for mining operations, reducing the proximity of roads to major fish-bearing streams, and this route respects the advice and privacy of community members close to the Project location. The route originates from the intersection of Highway 55 and Warm Lake Road and would be approximately 71 miles long. The route consists of 34 miles of existing highway (Warm Lake Road), 23 miles of upgraded road, and 14 miles of new road. The 37 miles of new and upgraded road would have a design speed of 20 mph, max 10% grade, a 21-foot width and intermediate-sized tractor trailer loading criteria. A maintenance facility would be constructed along the route. Additional details on the Burntlog Route and maintenance facility are provided in Section 18.

Midas Gold will provide buses and vans as the primary means of employee and contractor transportation to the site, reducing Project-related traffic along the access roads to site, thereby reducing risks to the safety of workers and the general public from traffic incidents, as well as minimizing the environmental impacts associated with vehicle traffic (particularly dust generation and sediment run-off, and also greenhouse gas and particulate emissions from vehicle use).

A through-site public access route will replace the current access through the SGP site during mine operations. During construction of the SGP, a new 12-foot-wide gravel road would be constructed to provide public access from Stibnite Road to Thunder Mountain Road through the mine site. A small segment of the road would be constructed on a widened bench within the Yellow Pine pit. South of the Yellow Pine pit, this road would parallel a mine haul road and use a partially revegetated historical mine road west of the EFSFSR.

1.17.2 Logistics Facility

Offsite administrative offices, transportation hub, warehousing and assay laboratory needed for the Project, referred to as Stibnite Gold Logistics Facility (**SGLF**), will be located on private land in Valley County, with easy access to State Highway 55. The SGLF will include offices for managers, safety and environmental services, human resources, purchasing and accounting personnel. Operating supplies for the mine will be staged and consolidated at the SGLF to reduce traffic to the site.

1.17.3 Power Supply and Transmission

Grid power was selected as the preferred primary power supply for the Project based on its low operating cost, low unit prices, and Idaho Power Company's existing clean energy portfolio. To provide the necessary power, the existing grid network would need to be upgraded to support the 50 to 60-megawatt (**MW**) load. This includes upgrading approximately 63 mi of existing powerlines to 138 kV, and approximately 9 miles of new 138 kV line. Additionally, new or upgraded 138 kV substations at Lake Fork, Cascade, Scott Valley, Warm Lake, Thunderbolt Drop, Johnson Creek, and Stibnite, as well as measures to strengthen the voltages on the IPCo system, are required. The 138-kV line would be routed to the Project's main electrical substation where transformers would step the voltage down to the distribution voltage of 34.5 kV.

1.17.4 Worker Accommodations

Midas Gold has an existing on-site worker housing facility with a capacity for approximately 60 workers. The existing facility would be expanded to provide accommodations during the initial year of construction and a new worker housing facility would be constructed approximately 2 miles south of the ore processing plant area to provide accommodations for the balance of the construction workforce and for the operations workforce. Since the peak construction accommodation requirements for approximately 1,000 workers is well in excess of the operations requirements of approximately 350 workers on site at any one time, leased accommodation units would be used during peak construction activity then demobilized following construction.

1.17.5 Water Management

Midas Gold will develop a water management system that protects or improves water quality in Project-area streams and provides water for ore processing, fire protection, exploration activities, surface mining (dust control), and potable water needs.

The key water management consideration for the Project site is the large amount of snowmelt runoff during the months of April through June, making spring melt the critical time for water management, storage, and treatment. In general, surface water that comes in contact with materials that have the potential to introduce mining- and process-related contaminants (contact water) is kept separate from surface water that originates from undisturbed, uncontaminated ground (non-contact water). This is accomplished by diverting clean water around mine facilities and collecting and reusing, evaporating, or treating and discharging contact water.

Meteoric and tailings consolidation water will be reclaimed from the TSF and would supply the majority of the water needed for ore processing. Additional water needs would be supplied from: pit dewatering, reuse of stored contact water, groundwater wells, and a surface intake near the upstream portal of the EFSFSR diversion tunnel.

Active dewatering will be required at the Yellow Pine and Hangar Flats pits, generally from alluvium and fractured bedrock wells, with total pumping ranging from zero to up to approximately 2,100 gpm over the life of mine. Excess dewatering water not used for ore processing would be treated, if required, and discharged to a surface outfall.

Major water diversions include construction of a tunnel and fishway to divert the EFSFSR and provide fish passage around the Yellow Pine pit, and surface diversions of Meadow Creek at the TSF, TSF Buttress, and Hangar Flats pit.

Contact water from the pits, stockpiles, TSF buttress, truck shop, ore processing facilities, and legacy materials exposed during construction would be collected in lined ponds or in-pit sumps for later use in ore processing, dust control, or treatment for discharge. Water management features would be phased in and out as mining progresses and the amount of surface area generating contact increases as pits and DRSFs expand and removed as backfilling and reclamation is completed. Aggregate contact water pond storage varies according to mine phase and is roughly 300 to 400 ac-ft over the mine life (excluding storage in pits), and approximately 200 ac-ft at the TSF in closure.

Three water types will require treatment over the life of the Project: contact water, including dewatering water, from mine facilities (construction through closure); process water from the TSF (closure); and sanitary wastewater (construction through early closure). Iron coprecipitation was selected for contact and process water treatment, as arsenic and antimony are the key constituents of concern in mine-impacted water at the site. During operations, treating and releasing contact water is generally limited to periods when a significant amount of dewatering water is being produced, or seasonally in wet years. During construction and at closure, absent a water demand for ore processing, less contact water can be consumed and proportionally more must be disposed of through evaporation or treatment and discharge. The variability in water excess is met with a phased water treatment approach, with approximately 300 gpm of treatment capacity during construction, 1,000 gpm early in operations, ramping up to 2,000 gpm during the peak of dewatering excess, and returning to 1,000 gpm through post-closure. Throughout the mine life, treatment would be augmented by forced evaporation when seasonal water storage and weather allows. Contact water volumes decline rapidly at closure as facilities are covered and reclaimed, but post-closure treatment is anticipated for the TSF until approximately 25 years after tailings deposition ceases, when tailing consolidation water is predicted to be minimal.

1.17.6 Tailings Management

The Project would produce approximately 120 million tons of tailings solids. The tailings would contain trace amounts of cyanide and metals (including arsenic and antimony), so a fully lined containment facility utilizing a composite liner is proposed to isolate the tailings and process water.

The TSF would consist of a rockfill embankment, a fully lined impoundment, and appurtenant water management features including a surface diversion of Meadow Creek and its tributaries around the facility. A rockfill buttress abutting the TSF embankment would substantially enhance embankment stability. Historical spent heap leach ore would be reused in TSF construction, in locations isolated from interaction with water, but the majority of the rockfill would be development rock sourced from the open pits. Design criteria were established based on the facility size and risk using applicable dam safety and water quality regulations and industry best practice for the TSF embankment on a standalone basis; the addition of the buttress substantially increases the safety factor for the design to approximately double the minimum requirements. The TSF impoundment, embankment, and associated water diversions would occupy approximately 420 acres at final buildout, with an approximately 465-foot ultimate height. The TSF location relative to other Project features is shown on Figure 1.8. Table 1-6 summarizes TSF design features.

Table 1-6: TSF Design Summary

Design Aspect	Description
Underdrains	Mains: perforated pipe and gravel in geotextile-wrapped trenches. Laterals: geo-composite drains.
Subgrade	Reworked and compacted in situ materials, or minimum 12 inches of liner bedding fill.
Liner Subbase	Geosynthetic clay liner.
Primary Liner	60-mil LLDPE, single-side textured.
Overliner drains	Geosynthetic strip drains.
Leak Detection	Sampling of underdrains and downgradient monitoring wells.
Deposition Strategy	Subaerial; depositing from perimeter of impoundment and embankment with pool on east side near, but not normally in contact with, embankment.
Reclaim	Pumped from barge (vertical turbine pumps).
Excess Water Disposal	Consumption in process (operations), mechanical evaporators (operations and closure), water treatment and discharge (closure).
Diversions	Surface channels, in rock cut or lined with geosynthetics, concrete cloth, or riprap and GCL. Parallel or embedded pipe for low flows (stream temperature mitigation measure).

1.18 METAL PRICES

The economic analysis completed for this FS assumed that gold and silver production in the form of doré with appropriate deductions for payabilities, refining and transport charges. The metal prices selected for the five economic cases in this Report are shown in Table 1-7.

Table 1-7: Assumed Metal Prices by Case

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(2) ($/lb)	
Case A	1,350	16.00	3.50	Lower bound case defined by the approximate 5-year trailing average gold price and consistent with the gold price used in the PFS (M3, 2014).
Case B (Base Case)	1,600	20.00	3.50	Base case derived from the weighted average of the 3-year trailing gold price (60%) and the 2-year gold futures price (40%).
Case C	1,850	24.00	3.50	Case corresponds to the approximate spot gold price at the effective date of this report.
Case D	2,100	28.00	3.50	Case corresponds with a gold price at approximately the peak 2020 spot price.
Case E	2,350	32.00	3.50	Upper bound case provides investors with insight into the revenues generated by the Project at a sustained elevated long term gold price.

Notes:

(1) The base case silver price was set at a gold:silver ratio ($/oz:$/oz) of 80:1 or $20/oz. The base case price was then varied similar to the way the gold price was varied (in this case by $4/oz Ag versus $250/oz Au) for the other cases.

(2) Antimony prices were assumed to be constant at $3.50/lb for all cases as antimony does not historically vary proportional to the gold and silver prices and is not expected to do so in the future. The $3.50/lb price was derived from a market study undertaken by an independent expert in antimony markets.

1.19 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL/COMMUNITY IMPACT

Midas Gold has a long-established environment, social and governance (**ESG**) approach, focused on a "net-benefit" goal, that is detailed in Chapters 2 and 6 of the PRO (Midas Gold, 2016), Section 20 of this Feasibility Study, and various corporate documents. In establishing the goal of net benefit to the environment and, as central principles to the proposed Project development, operations and closure, early in the design process, Midas Gold focused on a number of key restoration and mitigation principles. These principles included: conduct activities in an environmentally responsible manner; utilize previously disturbed areas; improve fish passage and habitat; remove, reprocess, or reuse legacy mine wastes to protect and improve water quality; revegetate disturbed or burned areas to improve wildlife habitat and reduce sediment loads; and restore or enhance wetlands and streams. By achieving this net benefit goal, Midas Gold will have provided Project restoration and mitigation projects that are both durable and additive; that is to say the mitigation outcomes will be above and beyond that which would have occurred in the absence of the Project (for additional details, see PRO Chapter 6, (Midas Gold, 2016)). The following provides a brief overview of each component of the goal as it intersects with the FS.

1.19.1 Environmental Legacies and Past Cleanup Efforts

The District has been mined extensively for tungsten, antimony, mercury, gold, and silver since the early 1900s, which left significant legacy environmental impacts that persist to this day, although multiple cleanup efforts undertaken by federal and state agencies and private entities have partially mitigated some of those historical impacts. Historical mining impacts have been compounded by extensive forest fires and subsequent damage from soil erosion, landslides and debris flows and resultant sediment transport.

Foremost remaining legacy issues include the presence of spent heap leach ore, tailings, abandoned surface and underground workings, and development rock dumps that interact with water, all leading to elevated arsenic and antimony in surface and groundwater at the site; and physical remnants of past mining disturbance such as the pit lake and fish passage barriers at the Yellow Pine pit and upstream, ongoing erosion of Blowout Creek, and deforestation and degraded stream habitat sitewide. Solutions for the most significant of these legacy issues are integrated with the SGP mine plan and associated restoration plans.

1.19.2 Environmental Studies

An extensive dataset demonstrating historical and existing conditions exists for the Project site, including data collected by contractors for the US Forest Service (**USFS**) and EPA, the US Geological Survey (**USGS**), prior mine operators, and Midas Gold and its contractors.

Assessments by several Midas Gold and Federal agency contractors determined that there were a number of pre-existing significant and moderate recognized environmental conditions and overall water quality in all drainages was impaired due to naturally occurring mineralization and impacts associated with historical mining.

Midas Gold's environmental resource baseline data collection program was initiated in 2011, and baseline monitoring reports were submitted in 2017 to regulators, but certain studies are ongoing to provide monitoring data, and additional supplementary studies have been prepared per agency requests. Baseline data from all sources informed environmental modeling and Project design.

1.19.3 Environmental Modeling

Midas Gold and its contractors developed predictive models for use in environmental evaluation and feasibility level engineering studies. Environmental models include air emissions modeling, a regional hydrogeologic/groundwater flow model and meteoric water balance, stream and pit lake network temperature model (**SPLNT**), geochemistry / site-wide water chemistry (**SWWC**) loading model, and site-wide water balance (**SWWB**). The modeling process involved development of conceptual models, work plan approval by the regulatory agencies, development and calibration of existing conditions models, and development of predictive models for the proposed action and alternatives to the proposed action. The suite of models facilitated environmental analysis, evaluation of alternate design scenarios, and design trade-offs. Environmental modeling has been a key tool for advanced engineering and identification of Project modifications (Section 1.2) and appropriate mitigation measures to reduce cost and environmental impact. Key Project changes and mitigation measures incorporated into the FS to address results of analyses in the DEIS, and comments received from stakeholders before and during the DEIS comment period, include: contact water treatment; a low-permeability cover on the TSF buttress; mine plan changes to eliminate some facilities, reduce facility size, backfill pits, and reduce the acreage of concurrent disturbance; and modifying water diversion designs to reduce summer stream temperatures.

1.19.4 Mine-Impacted Water Treatment

The seasonal water balance excess and predicted leaching of arsenic and antimony from mined materials lead to a need to dispose of water which would not meet discharge water quality standards absent treatment. Based on measured and predicted water quality and anticipated discharge water quality standards (typically either the acute cold-water biota or drinking water standards, depending on constituent), dewatering water, seepage, and contact stormwater would require treatment before discharge during operations. In closure, once other facilities are reclaimed, TSF water would require treatment. Mechanical evaporation would be used along with active, and potentially passive, water treatment to manage excess water at site. Due to the need to remove arsenic and antimony, iron coprecipitation was selected as the primary technology for active treatment. Required water treatment capacity varies from construction through closure, according to the site water balance changes and storage capacity, peaking in the middle of operations at approximately 2,000 gpm when both Hangar Flats and Yellow Pine pits are being mined, declining to approximately 1,000 gpm later in operations as facilities are concurrently reclaimed, and continuing until after the TSF is covered to manage tailings consolidation water. Post-closure water treatment will continue until approximately year 40 (approximately 25 years after the end of ore processing operations).

1.19.5 Permitting

Approval of the Project requires completion of the Environmental Impact Statement (**EIS**) in compliance with the National Environmental Policy Act (**NEPA**), which requires federal agencies to study and consider the probable environmental impacts of a proposed federal action before making a decision on that action. For the Project to proceed, there are multiple federal actions required as described in the Draft EIS (**DEIS**) for the Project which is available at https://www.fs.usda.gov/project/?project=50516. In addition to federal permits, the Project requires multiple state and local permits, which also are described in the DEIS. The DEIS was issued by the USFS for public review in August 2020, and the public comment period concluded in October 2020. State and local permitting processes are integrated through the Idaho Joint Review Process (**IJRP**) in progress concurrent with preparation of the EIS, and include water discharge (**IPDES**), air quality, cyanidation, groundwater, water rights, dam safety, mine and reclamation, building permits, sewer and water systems, among others. Once the USFS completes revisions to the DEIS, a Final EIS will be issued which will support the Records of Decision to be issued by the federal authorities.

Refinements to the Project reflected in the FS present opportunities to reduce the Project footprint and improve environmental outcomes. These refinements are responsive to comments received from stakeholders before the DEIS was published, comments received during the comment period and Midas Gold's own review of the environmental analysis. As such, the FS contemplates a Project that includes: contact water treatment; low-permeability cover on the TSF buttress; mine plan adjustments to reduce Project footprint; elimination of certain facilities; backfilling pits; and piping summer low flows to reduce stream temperatures.

Section 20 provides detailed descriptions and the status of each of the permits required prior to construction and operation of the Stibnite Gold Project.

1.19.6 Social and Community Impacts

Midas Gold's objective is to make the Project a fully integrated, sustainable, and socially and environmentally responsible operation through open communications and accessibility.

The Project would create approximately 550 direct jobs in Idaho during the almost 15 years of operations and would result in at least a similar number of indirect and induced jobs while generating significant taxes and other benefits to the local, state and national economies. The Project is also estimated to create substantial tax revenues from business, property, and individual taxes on Midas Gold, its employees, suppliers and contractors and their employees, and from induced economic activity. Midas Gold has committed to look to Idaho first, and particularly Valley County and neighboring Adams and Idaho counties, for its workforce and for the materials needed for the Stibnite Gold Project, encouraging local hiring, training, contracting, provision of supplies and services within the local communities and Valley County, and in expanding circles that include adjacent counties, the State and the balance of the U.S. (PRO Chapter 3 (Midas Gold 2016)).

Midas Gold has strived to develop a Project that respects and responds to the needs of all Project stakeholders, including local communities, tribes, and regional interests. In addition to board adoption of a formal Environmental Social and Governance commitment, Midas Gold has proactively implemented an iterative process of community engagement involving communicating with and listening to stakeholders through all aspects and phases of Project planning and design. These activities include interaction with potentially affected communities regarding potential Project economic impacts and opportunities, working with local communities to identify community needs and to plan for potential expansion of public services and infrastructure, engaging with tribal governments, and sponsoring and participating in community programs and educational events. Midas Gold's commitments also included entering into community agreements to ensure communication, coordination and transparency throughout the life of the Project and that financial benefits to local communities continue beyond the Project lifespan.

The public scoping and DEIS public comment phases of the NEPA process have also provided important feedback from communities and stakeholders that will be affected by the Project. It is notable that significant comment-driven Project changes, including modification of proposed public access through the Project site, backfilling of Hangar Flats pit, and additional fisheries and water quality mitigation measures were incorporated into Midas Gold's modifications of the Proposed Action, either previously incorporated as alternatives in the DEIS or proposed herein to further reduce Project environmental impacts, for adoption in the FEIS.

In order to better integrate the Project into the local communities and coordinate with them, in 2018 Midas Gold entered into a Community Agreement (**CA**) with the Village of Yellow Pine, the cities of Cascade, Donnelly, New Meadows, Riggins and Council, and Adams and Idaho counties (Midas Gold, 2018). As a regulator for the Project, Valley County determined it was not in a position to enter into the CA. The CA established the Stibnite Advisory Council, which brings communities together to discuss the challenges and opportunities presented by the Project; and the Stibnite Foundation, which distributes funds to projects from milestone and future share of profits contributions by Midas Gold.

Midas Gold respects the sovereign treaty rights of Native American tribes and has engaged them in good faith through all phases of Project exploration, development and planning. Through early engagement with the Nez Perce Tribe (**NPT**) commencing in 2012, Midas Gold has undertaken measures to mitigate potential impacts of its exploration activities identified by the NPT and has allowed the NPT full access to the Site and shared baseline environmental data. More recently, Midas Gold has been engaged with the Shoshone-Bannock Tribes (**SBT**) and has been undertaking efforts to educate Tribal representatives on its proposed plans to improve water quality, address legacy issues caused by prior mining companies and to collaborate on fisheries.

1.19.7 Avoidance and Minimization

Designing the site restoration for a net benefit was guided by a hierarchy of priorities: avoidance, minimization, then mitigation. Midas Gold sought to conserve existing natural resources and avoid and minimize environmental impacts in selection of Project facility locations, responsible operating plans, and facility design features. Avoidance and minimization measures reduced Project footprint, impacts to aquatic habitat, and the potential for water quality impacts.

1.19.8 Legacy Material Cleanup

Midas Gold will remove, reuse, reprocess, or isolate a variety of legacy materials from prior mining operations, in the course of re-mining this brownfield site. In addition to removals that will improve water quality, Midas Gold will repair a number of physical legacies that degrade fish habitat and limit fish migration.

1.19.9 Compensatory Mitigation

While Project facilities and infrastructure would be located in areas of previous disturbance wherever practicable, in some cases disturbance of wetlands and streams would be unavoidable. Under Section 404 of the Clean Water Act, unavoidable impacts to waters of the U.S. require compensatory mitigation – that is, replacement of their lost function – generally in advance of the disturbance taking place, either by the use of a mitigation bank or construction of replacement wetlands, generally in the same drainage basin.

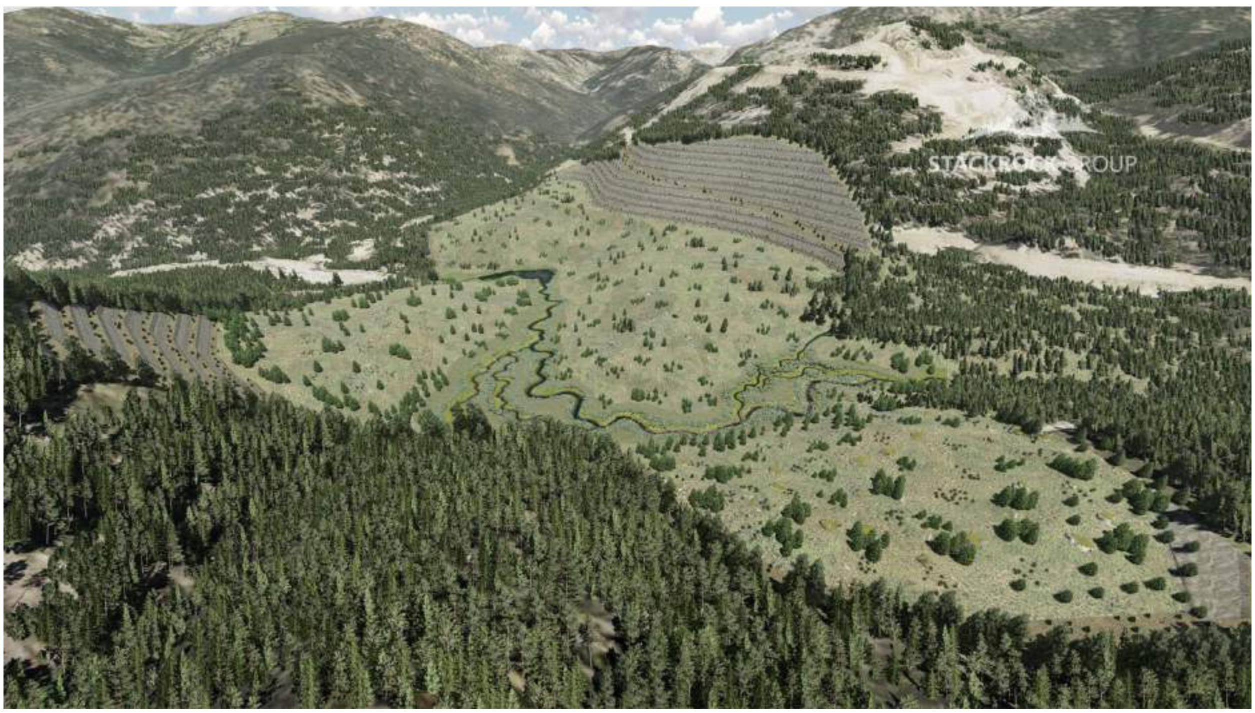
Owing to the combined effects of the Project sequence, limited valley-bottom land available, and lack of established mitigation banks in the basin, complete compensatory mitigation via a single means is impractical for the Project. Midas Gold is pursuing a comprehensive approach to wetland and stream compensatory mitigation that entails on-site enhancement and restoration of both streams and wetlands, banking, and off-site projects such as stream habitat enhancements and replacement of culverts that presently impede fish passage. Many of the compensatory mitigation measures are also closure and restoration projects. The U.S. Army Corps of Engineers (**USACE**) is evaluating the mitigation proposal concurrent to the NEPA process.

1.19.10 Closure and Restoration

Midas Gold developed closure and restoration plans with the objectives to establish a sustainable fishery with enhanced habitat to support natural populations of salmon, steelhead, and bull trout; improve water quality; establish vegetation; and enhance wildlife habitat, all contributing to a self-sustaining and productive ecosystem. Closure, reclamation and restoration activities would achieve post-mining land uses of wildlife and fisheries habitat and dispersed recreation at the mine site.

Significant components of reclamation and restoration occur concurrently with operations, including: removing and reprocessing and/or reusing historical tailings, development rock and spent ore; enhancing existing streams; improving water quality; backfilling and reclaiming the Hangar Flats and Yellow Pine (Figure 1.9) pits; stream restoration; and establishing permanent fish passage to the headwaters of the EFSFSR. The remaining closure activities occur in the first 10 years after operations cease: further improvements to water quality; restoring additional streams, wetlands, and riparian habitat throughout the site; decommissioning onsite infrastructure and facilities; replacing growth media; re-contouring artificial landforms to blend into the landscape; and replanting Project and historical disturbance areas. Closure maintenance, water treatment, and long-term monitoring are anticipated to continue longer to protect water quality gains and ensure that closure features are performing as intended.

Figure 1.9: Post Closure Isometric View of Yellow Pine Pit Area



1.19.11 Environmental Monitoring and Reporting

Midas Gold will employ environmental monitoring measures that will be part of permits and other approvals from the USFS, USACE, EPA, Idaho Department of Environmental Quality (**IDEQ**), Idaho Department of Lands (**IDL**), Valley County, and other appropriate agencies. The Project will operate under federal, state and local permit approvals that will mandate practices and procedures to mitigate environmental impacts, reclaim disturbed areas, and monitor restoration success and water quality. These agencies will conduct routine inspections to ensure compliance with applicable monitoring and reporting regulations.

1.20 CAPITAL & OPERATING COSTS

Capital expenditures or capital costs (**CAPEX**) and operating expenditures or operating costs (**OPEX**) estimates were developed based on Q3 2020, un-escalated U.S. dollars. Vendor quotes were obtained for all major equipment. Most costs were developed from first principles, although some were estimated based on factored references and experience with similar projects elsewhere. Vendor quotes were obtaining for all major equipment and operating consumables. Reclamation financial assurance costs are not included in the capital costs.

1.20.1 Capital Costs

The Project CAPEX estimate includes four components: (1) the initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, operations camp, and pre-production on and off site restoration and environmental mitigation; (2) the sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing concurrent restoration and environmental mitigation activities during the operating period; (3) working capital to cover delays in the receipts from sales and payments for accounts payable and financial resources tied up in inventory, and (4) closure CAPEX to cover post operations reclamation and restoration and water treatment costs. Initial and working capital are the two main categories that need to be available to construct the Project. Table 1-8 provides a CAPEX summary for the Project.

Table 1-8: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Closure CAPEX ($000s)[(1)]	Total CAPEX ($000s)
Direct Costs	Mine Costs	84,019	118,968	-	202,987
	Processing Plant	433,464	49,041	-	482,505
	On-Site Infrastructure	190,910	83,892	-	274,802
	Off-Site Infrastructure	115,940	-	-	115,940
Indirect Costs		232,684	-	-	232,684
Owner's Costs, First Fills, & Light Vehicles		38,351	-	-	38,351
Offsite Environmental Mitigation Costs		14,397	-	-	14,397
Onsite Mitigation, Monitoring, and Closure Costs		3,474	23,484	98,052	125,010
Total CAPEX without Contingency		**1,113,239**	**275,385**	**98,052**	**1,486,677**
Contingency		149,708	20,354	1,244	171,306
Total CAPEX with Contingency		**1,262,948**	**295,739**	**99,296**	**1,657,982**

Notes:
(1) Closure assumes self-performed closure costs, which will differ for those assumed for financial assurance calculations required by regulators.

1.20.2 Operating and All-In Costs

The Project OPEX estimate includes mine operating costs, process plant operating costs, and general and administrative (**G&A**) costs. Cash costs, expressed in dollars per short ton ($/st) milled or dollars per troy ounce of gold ($/oz Au) produced, are typically expressed before and after by-product credits (from antimony concentrate sales). Total cash costs include smelting and refining charges, transportation charges, and royalties. The All-In Sustaining Costs (**AISC**) and the All-In Costs (**AIC**) include non-sustaining CAPEX, and closure and reclamation CAPEX, respectively. A summary of these Project costs is presented in Table 1-9. The details that comprise the OPEX are provided Section 21.

Table 1-9: Operating Cost, AISC and AIC Summary

Total Production Cost Item	Years 1-4		LOM	
	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	9.71	156	8.22	205
Processing	13.13	211	12.76	318
G&A	3.54	57	3.43	85
Cash Costs Before By-Product Credits	**26.38**	**424**	**24.41**	**608**
By-Product Credits	(5.99)	(96)	(2.81)	(70)
Cash Costs After By-Product Credits	**20.40**	**328**	**21.60**	**538**
Royalties	1.69	27	1.09	27
Refining and Transportation	0.46	7	0.24	6
Total Cash Costs	**22.54**	**362**	**22.94**	**571**
Sustaining CAPEX	4.64	75	2.83	70
Salvage	-	-	(0.26)	(6)
Property Taxes	0.05	1	0.04	1
All-In Sustaining Costs	**27.23**	**438**	**25.54**	**636**
Reclamation and Closure(1)	-	-	0.95	24
Initial (non-sustaining) CAPEX(2)	-	-	11.65	290
All-In Costs	**-**	**-**	**38.14**	**950**

Notes:
(1) Defined as non-sustaining reclamation and closure costs in the post-operations period.
(2) Initial Capital includes capitalized preproduction.

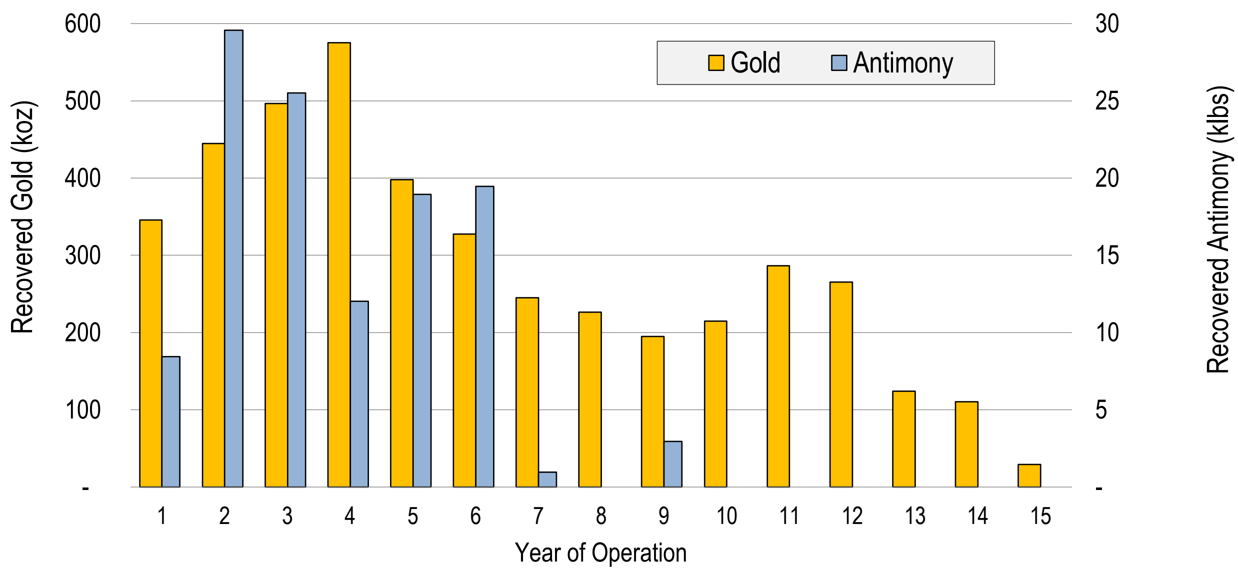
1.20.3 Metal Production

Recovered metal production by deposit is summarized in Table 1-10 and illustrated on an annual basis on Figure 1.10.

Table 1-10: Recovered Metal Production

Product by Deposit	Gold (koz)	Silver (koz)	Antimony (klbs)
Doré Bullion			
Yellow Pine	2,453	11	-
Hangar Flats	364	1	-
West End	1,333	839	-
Historical Tailings	68	0	-
Doré Bullion Recovered Metal Totals	**4,217**	**852**	**-**
Antimony Concentrate			
Yellow Pine	17	573	92,065
Hangar Flats	4	255	20,822
Historical Tailings	1	31	2,454
Antimony Concentrate Recovered Metal Totals	**21**	**858**	**115,342**
Total Recovered Metals	**4,238**	**1,710**	**115,342**

Figure 1.10: Annual Recovered Gold and Antimony



1.21 ECONOMIC ANALYSIS

The economic model described in this FS is not a true cash flow model as defined by financial accounting standards but rather a representation of Project economics at a level of detail appropriate for a FS level of engineering and design. The first year of analysis starts with the decision point of the Project, the completion of the EIS, and preliminary permit approval (Year -3 or three years before the start of commercial production). Taxation was taken into account using current federal, state, and county rates but the overall tax calculation is approximate and uses rudimentary depletion and depreciation estimates.

Four cases were run in the economic model to present a range of economic outcomes using varying metal prices. The metal prices used in the economic model are shown in Table 1-7. There is no guarantee that any of the metal prices used in the five cases are representative of future metals prices. The constant parameters for all cases are shown in Table 1-11.

Table 1-11: Financial Assumptions used in the Economic Analyses

Item	Unit	Value
Net Present Value Discount Rate	%	5
Federal Income Tax Rate	%	21
Idaho Income Tax Rate	%	6.9
Idaho Mine License Tax	%	1.0
Valley County Rural Property Tax Rate ($/$1,000 market value)	%	0.063
Percentage Depletion Rate for Gold and Silver	%	15
Percentage Depletion Rate for Antimony	%	22
Depreciation Term	Years	7
Equity Finance Assumption	%	100

The results of the pre- and after-tax economic analyses are provided in Table 1-12.

Table 1-12: Pre- and After-Tax Economic Results by Case

Parameter	Unit	Pre-tax Results	After-tax Results
Case A ($1,350/oz Au, $16.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	1,637	1,434
$NPV_{5\%}$	M$	896	771
Annual Average EBITDA	M$	223	-
Annual Average After-Tax Free Cash Flow	M$	-	189
IRR	%	17.3	16.2
Payback Period	Production Years	3.4	3.4
Case B ($1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	2,667	2,232
$NPV_{5\%}$	M$	1,599	1,320
Annual Average EBITDA	M$	292	-
Annual Average After-Tax Free Cash Flow	M$	-	242
IRR	%	24.3	22.3
Payback Period	Production Years	2.9	2.9
Case C ($1,850/oz Au, $24.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	3,697	3,026
$NPV_{5\%}$	M$	2,301	1,864
Annual Average EBITDA	M$	360	-
Annual Average After-Tax Free Cash Flow	M$	-	295
IRR	%	30.4	27.7
Payback Period	Production Years	2.4	2.5
Case D ($2,100/oz Au, $28.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	4,726	3,815
$NPV_{5\%}$	M$	3,002	2,404
Annual Average EBITDA	M$	429	-
Annual Average After-Tax Free Cash Flow	M$	-	348
IRR	%	35.9	32.4
Payback Period	Production Years	2.2	2.2
Case E ($2,350/oz Au, $32.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	5,755	4,603
$NPV_{5\%}$	M$	3,704	2,943
Annual Average EBITDA	M$	498	-
Annual Average After-Tax Free Cash Flow	M$	-	400
IRR	%	41.0	36.9
Payback Period	Production Years	1.9	1.9

The contribution to the Project economics, by metal, is approximately 96% from gold, 4% from antimony, and less than 1% from silver.

The undiscounted after-tax cash flow for Case B is presented on Figure 1.11. The payable metal value by year for Case B is summarized on Figure 1.12.

Figure 1.11: Undiscounted After-Tax Cash Flow for Base Case B

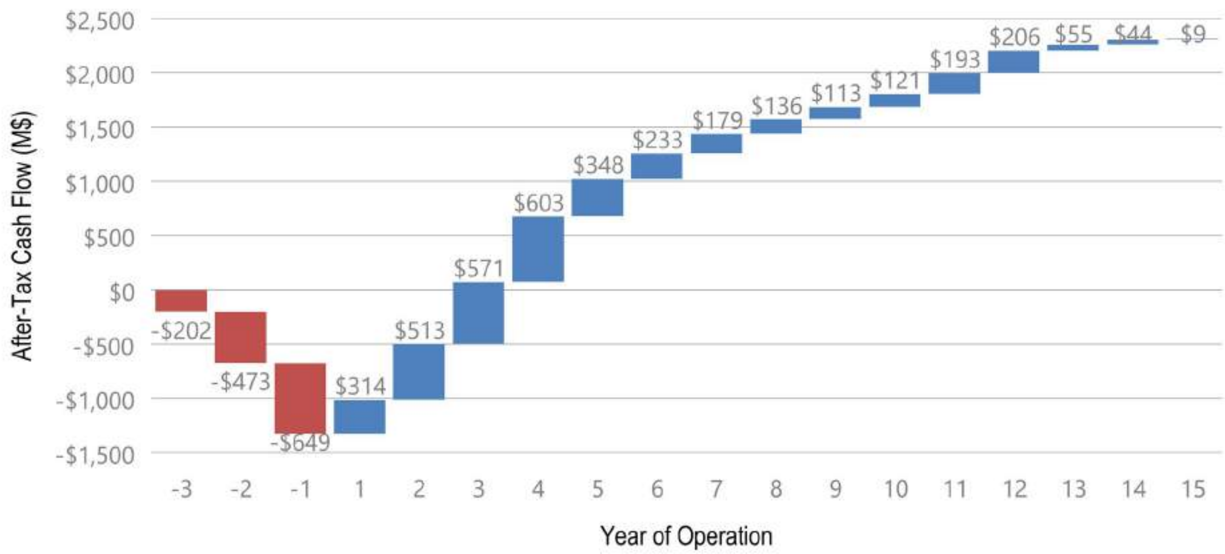
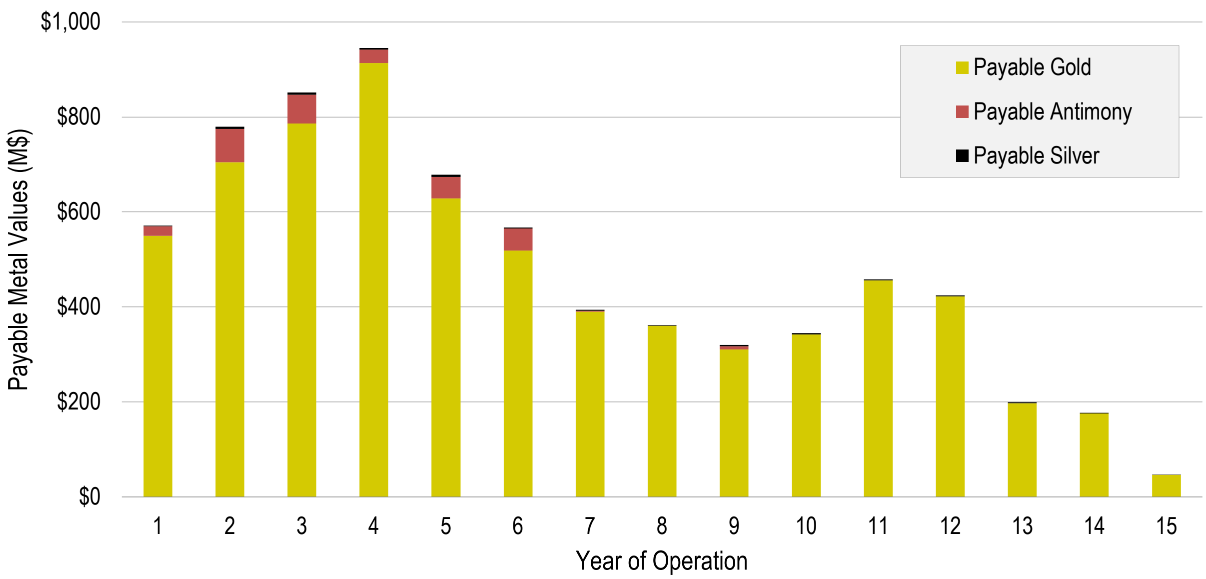


Figure 1.12: Payable Metal Value by Year for Case B



1.22 RISKS AND OPPORTUNITIES

A number of risks and opportunities have been identified in respect of the Project; aside from industry-wide risks and opportunities (such as changes in capital and operating costs related to inputs like steel and fuel, metal prices, permitting timelines, etc.), high impact Project specific risks and opportunities are summarized below.

Risks, which additional information could eliminate or mitigate include:

- Delay in permitting or necessary project changes resulting from permitting;
- Legal challenges to ROD or environmental complications associated with legacy mining impacts;

- Delays related to the Clean Water Act litigation initiated by NPT;
- Water management and chemistry that could affect diversion and closure designs and/or the duration of long-term water treatment;
- Geological uncertainties which may affect Mineral Resources and Mineral Reserves;
- Increases to estimated capital and operating costs; and
- Construction schedule.

Opportunities that could improve the economics, and/or permitting schedule of the Project, including a number with potential to increase the $NPV_{5\%}$ by more than $100 million include:

- In-pit conversion of approximately 9.8 Mt of Inferred Mineral Resources grading 1.02 g/t Au occurring within the Mineral Reserve Pits containing approximately 321 koz of gold, to Mineral Reserves, increasing Mineral Reserves and reducing the strip ratio;
- Out-of-pit conversion of approximately 26.2 Mt of Inferred Mineral Resources grading 1.09 g/t Au occurring outside the current Mineral Reserve Pits containing approximately 917 koz of gold, to Mineral Reserves;
- Out-of-pit conversion of approximately 27.1 Mt of Measured and Indicated Mineral Resources grading 1.26 g/t occurring outside the current Mineral Reserve Pits containing approximately 1,098 koz of gold, to Mineral Reserves;
- In-pit conversion of unclassified material currently treated as development rock to Mineral Reserves, increasing Mineral Reserves and reducing strip ratios;
- Definition of additional Mineral Reserves within the West End deposit through infill and resource definition drilling;
- Potential for the definition of higher grade, higher margin underground Mineral Reserves at Scout, Garnet or Hangar Flats; and,
- Discovery of other new deposits with attractive operating margins.

Mineral resources exclusive of mineral reserves are reported based on a fixed gold cut-off grade of 0.45 g/t for sulfide and 0.40 g/t for oxide, and in relation to conceptual Mineral Resource pit shells and Mineral Reserve pits to demonstrate potential economic viability as required under NI 43-101. Indicated mineral resources exclusive of mineral reserves are reported to demonstrate potential for future expansion should economic conditions warrant. Inferred mineral resources exclusive of mineral reserves are reported to demonstrate potential to increase in-pit production should inferred mineral resources be successfully converted to mineral reserves; mineralization lying outside of Mineral Resource pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated.

Opportunities with a medium impact ($10 to $100 million increase in Project $NPV_{5\%}$) include improved metallurgical recoveries, secondary processing of antimony concentrates, steeper pit slopes, and government funding of off-site infrastructure. A number of lesser impact opportunities also exist.

1.23 OTHER RELEVANT DATA AND INFORMATION

The Project would become the only domestic producer of antimony (stibnite) concentrate. Antimony was designated as a critical mineral in the U.S. Department of Interior's final list of 35 critical minerals published in 2018 (U.S. Dept. of Interior, 2018) as a result of zero domestic production in the U.S. and reliance on imports, directly or indirectly, from non-aligned countries such as China, Russia and Tajikistan which produce 92% of the world's antimony, according to the U.S. Geological Survey.

1.24 INTERPRETATION AND CONCLUSIONS

Industry standard mining, processing, construction methods, and economic evaluation practices were used to assess the Project. There was adequate geological and other pertinent data available to generate the FS.

The financial analysis presented in Section 22 of the FS demonstrates that the Project is financially viable and has the potential to generate positive economic returns based on the assumptions and conditions set out in this Report, while other sections of the FS demonstrate that the Project is technically and environmentally viable.

The FS has achieved its original objective of optimizing the PFS design, increasing the level of detail of the Project design and cost estimating resulting in decreased technical and financial risk, and strengthening the potential economic viability of the Project to standards appropriate for a FS.

The QPs of this Report are not aware of any unusual, significant risks or uncertainties that could be expected to affect the reliability or confidence in the Project based on the data and information available to date.

1.25 RECOMMENDATIONS

After many years of study, discussion, analysis, planning, and community and stakeholder input, Midas Gold prepared a comprehensive plan for the restoration and redevelopment of Stibnite, known as the PRO (Alternative 1 in the DEIS) and that plan was modified to form the ModPRO (Alternative 2 in the DEIS). This Feasibility Study lays out a safe, technically feasible, economically viable, environmentally sound and socially responsible path forward for the redevelopment and restoration of the Site. This path forward will comply with applicable laws and regulations and incorporates environmental improvements that were developed in response to comments received during the regulatory process, including the comment period for the DEIS, being undertaken under NEPA.

It is recommended that Midas Gold proceed with the NEPA process noted above in anticipation of a positive record of decision under NEPA. The estimated costs associated with this recommendation, and other ancillary recommendations included in Section 26, are approximately $14 million. Once a positive record of decision is in hand a construction decision would be the next logical step.

Restore the Site.

1.26 REFERENCES

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2 INTRODUCTION

2.1 BACKGROUND

The Stibnite Mining District's historical mining operations (Section 6 provides additional historical mining information) have resulted in significant environmental impacts, which have been compounded by forest fires that accelerated erosion and exacerbated the effects of past human activity. The large-scale equipment, power, transportation routes, process facilities, rock storage areas, water treatment, and tailings containment that would be needed to effectively accomplish restoration of the site would be readily available as part of a mining operation but would be prohibitively expensive on a standalone basis.

To develop a path forward for the site, since 2009, Midas Gold has been building relationships with individuals and organizations across Idaho, establishing existing environmental baseline conditions, and conducting extensive technical studies to determine the environmental, social, technical, and economic feasibility of redeveloping Stibnite. Midas Gold recognizes that there are sensitivities in developing a project in an historical mining area with an already impacted fishery and are committed to engaging in collaborative communication with all stakeholders to address their diverse interests throughout the life of the Stibnite Gold Project.

The Stibnite Gold Project is designed to incorporate the rehabilitation of an historically damaged site through the restoration of stream channels, wetlands, and fisheries through the cash flow generated by a profitable mining operation. The Project is also designed to generate financial returns for investors, boost the local economy, and supply extracted metals (including the critical metal antimony) while providing the means to comprehensive environmental restoration that is unlikely to occur by any other means.

A Plan of Restoration and Operations (**PRO**) was filed with the U.S. Forest Service and other agencies in September 2016. It presents the opportunity to use private investment for synergistic redevelopment of a brownfields[1] mining district and natural resource restoration. The PRO was evaluated as Alternative 1 in the Draft Environmental Impact Statement (**DEIS**) that was released by the U.S. Forest Service for public comment in August 2020. The PRO was updated with additional refinements, primarily for their environmental benefit, in a modified PRO (**ModPRO**) that was submitted to regulators in May 2019 and forms Alternative 2 of the DEIS. Since the date of the ModPRO, Midas Gold has continued to refine the Project based on its internal analysis, agency comments, and public input and that identified additional enhancements, again primarily focused on better environmental outcomes, that have been incorporated into this Feasibility Study.

An important aspect of the Project will be the recovery and sale of domestically sourced antimony concentrate. Antimony has been designated a critical mineral[2] by the U.S. due to its importance for national defense and economic security, the lack of domestic supply, and import reliance[3] (see Section 24 for additional information on antimony).

1 A "brownfields" site is one that has already been extensively disturbed by previous mining activity, as opposed to an area that has never been mined before and remains relatively wild or pristine.

2 In 2018, the U.S. Department of Interior issued its final list of 35 critical minerals, antimony among them (U.S. Dept. of Interior 83 Federal Register 23295, May 18, 2018) and, in 2019, the U.S. Department of Commerce issued a comprehensive "Federal Strategy to Secure Reliable Supplies of Critical Minerals" (June 4, 2019).

3 USGS Mineral Commodity Summary 2020, February 7, 2020.

2.2 MIDAS GOLD'S CORE VALUES

Midas Gold considers the health and safety of people, the protection of the environment and the sustainability of its activities to be the core values that drive all aspects of Project planning and development. This foundation of core values is reflected in the Company's policies as set out in the PRO and summarized below:

- Safety – The health and safety of employees, contractors, and the public is of the utmost importance.
- Environmental Responsibility – Go above and beyond what is required; find practical solutions to manage growth, while protecting and enhancing the natural environment.
- Community Involvement – As proud members of the community, actively strive to serve the community's needs, and to collectively enhance prosperity and well-being.
- Transparency – Fulfill our commitments in an open and transparent manner. Aim to be accurate, consistent, and straightforward in all information delivered to stakeholders.
- Accountability – As part of corporate governance, ensure that accountability guides actions, decisions, conduct, and reporting.
- Integrity & Performance – Set high moral standards and strive to fulfill commitments in an effective and sustainable manner.

In aligning the Stibnite Gold Project with these core values, Midas Gold also adopted the following conservation guidance principles for the Project, with the end goal that the Project bring a net environmental benefit:

- Conduct restoration, mining, ore processing, and reclamation activities in an environmentally responsible manner.
- Locate Project infrastructure on previously disturbed areas wherever practicable.
- Design, construct, operate, and close facilities to minimize impacts to aquatic and terrestrial wildlife, improve habitat across the Project site, protect anadromous and local aquatic populations, and remove impediments to fish passage.
- Protect and improve local surface water and groundwater quality.
- Preserve, restore, or enhance ecologically diverse stream channels and wetlands to mitigate those disturbed by legacy and new mine development.

Please see PRO Chapter 2 Core Values for further details.

2.3 PURPOSE OF REPORT

This feasibility study technical report (**FS** or **Report**) was commissioned by Midas Gold for its Stibnite gold-antimony-silver project (**Stibnite Gold Project** or **Project**) at Stibnite, Idaho. This Report has been prepared for Midas Gold Corp. (**MGC**), a British Columbia company exploring options for the redevelopment and restoration of the project area through its wholly-owned subsidiaries, Midas Gold Idaho, Inc. (**MGI**), MGI Acquisition Corp (**MGIAC**), Idaho Gold Holding Company (**IGHC**) and Idaho Gold Resources, LLC (**IGR**). Unless the context indicates otherwise, references throughout this Report to "**Midas Gold**" includes one or more of the aforementioned subsidiaries of MGC.

The Report has been prepared in compliance with the Canadian Securities Administrators (**CSA**) National Instrument 43-101 (**NI 43-101**) standards for reporting mineral properties, Companion Policy 43-101CP, and Form 43-101F1. The contents of this Report reflect the technical and economic conditions at the effective date of the Report. These conditions may change significantly over time; consequently, actual results may vary considerably from those depicted herein.

This Report provides a comprehensive overview of the Project and includes recommendations for future work programs required to advance the Project to a decision point. This Report defines an economically feasible, technically and environmentally sound Project that minimizes impacts and maximizes benefits. The key considerations that went into the design of the Stibnite Gold Project are as follows:

- The Project design began with the end in mind, contemplating the development, operation, and closure of the Project on a sustainable basis, meeting the needs of the present and enhancing the ability of future generations to meet their own needs. The Project design incorporates the key concepts of meeting the needs of society for a better life, providing economic prosperity, and remaining protective of the environment.
- The Project is designed to ensure ongoing positive local and regional fiscal and social benefits through tax payments, employment, and business opportunities, resulting in lower unemployment and higher annual wages.
- The Project has been designed for what will remain after closure. The closure plan is protective of the environment and incorporates inherently stable, secure features that will provide the foundation for evolution to a naturally sustainable ecosystem.
- The Project design incorporates the repair of extensive historical mining-related impacts much of which would occur during initial construction and early operations, and little or none of which would likely occur without the Project.
- The new facilities contemplated for the Project are tightly constrained and, to a large extent, placed in historically impacted areas to minimize the incremental Project footprint.
- Salmon, bull trout, steelhead, and other fishery enhancements are integral to the Project design. Removal of man-made barriers and restoration of natural habitat would allow fish migration into the upper reaches of the watershed for the first time since 1938.
- During development, operations, and closure, all aspects of the Project are designed to improve existing conditions where possible and remain protective of the environment, with the extensive costs related to remediation and reclamation of historical impacts accommodated by an economically feasible Project.

This Report provides information about the geology, mineralization, exploration, mineral resource potential, mining methods, ore process methods, infrastructure, social and economic benefits, environmental protection, repair of historical impacts, reclamation and closure concepts, capital and operating costs, and economic analysis for the Project. Economic and technical analyses included in this Report provide only a summary of the potential Project economics based on the many assumptions set out herein. There is no guarantee that the Project economics described herein can be achieved.

This Report and the information contained herein is current as of the effective date of the Report and supersedes earlier technical reports completed for Midas Gold including the "Stibnite Gold Project Prefeasibility Study Technical Report, Valley County, Idaho" effective December 8, 2014, amended March 28, 2019.

2.4 SOURCES OF INFORMATION AND QUALIFIED PERSONS

The sources of information include data and reports supplied by Midas Gold personnel, and documents referenced in Section 27. M3 Engineering & Technology Corp. (**M3**) used its experience to determine if the information from previous reports was suitable for inclusion in this Report and adjusted information that required amending. Revisions to previous data were based on research, recalculations, and information from other projects. The level of detail utilized was appropriate for this level of study.

This FS is based on information collected by the Qualified Persons (each a QP) during their site visits, and many meetings conducted between M3 and Midas Gold. This Feasibility Study Report is based on the following sources of information:

- Personal inspection of the Stibnite Gold Project site and surrounding area.
- Technical information provided to the QPs by Midas Gold through various reports.
- Budgetary quotes from vendors for engineered equipment.
- Technical and cost information provided by Idaho Power Co. and HDR, Inc. concerning power supply for the Project.
- Technical and economic information developed by M3 and associated consultants.
- Information provided by other experts with specific knowledge and expertise in their fields as described in Section 3 of this Report, Reliance on Other Experts.
- Additional information obtained from public domain sources.
- The information contained in this Report is based on documentation believed to be reliable. Information utilized in this Report will be either retained in Midas Gold's offices in Boise, Idaho or readily available from Midas Gold's consultants' Project files, subject to an appropriate agreement concerning confidentiality.

The individuals who have provided input to this FS have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. Table 2.1 provides a list of the QPs, their affiliation, sections for which they are responsible, date of the most recent site visit, and items reviewed on their site visits. The QP Certificates are provided as Appendix I.

2.5 ABBREVIATIONS, UNITS, AND TERMS OF REFERENCE

This FS is intended for the use of Midas Gold to further advance the Stibnite Gold Project toward a construction decision. It provides a mineral resource estimate, a classification of mineral resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (**CIM**) classification system, and an evaluation of the Project, which presents a current view of the potential economic outcome.

Imperial units (American System) of measurement are used in this Report. Other units of measurement used in this Report are defined when first used. Abbreviations are given in Section 2.5.4. All monetary values are in U.S. dollars ($) unless otherwise noted.

Table 2.1: List of Qualified Persons

Qualified Person	Company	Section Responsibility	Site Visit Date	Site Visit Review
Richard K. Zimmerman, R.G., SME-RM	M3 Engineering & Technology Corp.	1, 2, 3, 4, 5, 6, 7, 8, 9, 18 (excluding 18.8), 19, 20 (excluding 20.8), 21 (excluding 21.1.1, 21.1.6, 21.2.1), 22, 23, 24, 25, 26, 27	Mar 7, 2013	General site visit.
Art Ibrado, P.E.	M3 Engineering & Technology Corp.	17	See Note 1	
Grenvil Dunn, C.Eng.	Hydromet WA (Pty) Ltd.	13.9, 13.10	See Note 1	
Garth D. Kirkham, P.Geo.	Kirkham Geosystems Ltd.	10, 11, 12, 14	Apr 23-25, 2014, Jul 14-15, 2014, Jan 12–14, 2017 Jul 30-Aug 1, 2018	Site visit included inspection of the shops, offices, drill sites, the Yellow Pine, Hangar Flats, and West End mineral resource areas, miscellaneous outcrops, potential future mining operations infrastructure areas, and the core logging and storage facilities in Cascade.
Christopher J. Martin, C.Eng.	Blue Coast Metallurgy Ltd.	13.1, 13.2, 13.3, 13.4, 13.5, 13.6, 13.7, 13.8, 13.11, 13.12, 13.13	Aug 25, 2011	General site visit.
Chris J. Roos, P.E.	Value Consulting, Inc.	15, 21.1.1, 21.2.1	Oct 6, 2017	Reviewed Project geology, terrain, and operational constraints at site.
Scott Rosenthal, P.E.	Value Consulting, Inc.	16	Oct 6, 2017	Reviewed Project geology, terrain, and operational constraints at site.
Peter E. Kowalewski, P.E.	Tierra Group International Ltd.	18.8, 20.8, 21.1.6	Mar 7, 2013	General site visit.

Notes:

1) Art Ibrado and Grenvil Dunn have not been to the site. They have relied on Richard Zimmerman, who has visited the site, to inspect the proposed location of the ore processing plant facilities and infrastructure in relation to the existing topography.

2.5.1 Mineral Resources

As required by NI43-101, the Mineral Resources and Mineral Reserves in this Report have been classified according to the "*CIM Definition Standards for Mineral Resources and Mineral Reserves*" (May, 2014). Accordingly, the Mineral Resources have been classified as Measured, Indicated or Inferred; the Mineral Reserves have been classified as Proven, and Probable based on the Measured and Indicated Mineral Resources as defined below. By definition, a Mineral Resource must have reasonable prospects for eventual economic extraction[4].

"A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics, and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge."

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

"An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes."

"An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed."

2.5.2 Mineral Reserves

As required by NI43-101, Mineral Reserves have been defined according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (May 2014).

"A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified."

[4] CIM Estimation of Mineral Resources and Reserves-Best Practices Guidelines (CIM, Nov.29, 2019).

2.5.3 Glossary

Table 2.2 provides a glossary of certain terms that are used in this Report.

Table 2.2: Glossary

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All expenditures not classified as operating costs but excluding corporate sunken costs such as acquisition.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size by impact to render it more amenable for further processing.
Cut-off Grade	The grade of mineralized rock above which it becomes profitable to extract the mineralization.
Dilution	Waste, which is rock below an economic cutoff value mined with ore.
Dike	A sheet of igneous rock intruded along a crack in a rock mass and crystallized in place.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
District	A bounded division and organization of a mining region.
Fault	The surface of a fracture along which movement has occurred.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of a specific mineral within mineralized rock.
Historical Tailings	Approximately 3 Mt of uncontained tailings deposited in the Meadow Creek valley by previous operators.
Hydrocyclone	A process whereby particulate materials are segregated by size by exploiting the interaction between gravitational and centrifugal forces.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Lithological	Description of the physical characteristics of a rock.
Life of mine plans	Plans that are developed for the life of the mine.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Operating Expenditure	Operating expenditures/costs are costs required to operate the mine on a regular basis and includes mine operating costs, process plant operating costs, and general and administrative (G&A) costs
Oxide	Mineral that has undergone chemical reaction in which the substance has combined with oxygen.
Profile Sample	Profile samples are taken during pilot plant to provide a snapshot of the pilot plant conditions at a particular time
Project	A collaborative enterprise, involving research or design, that is carefully planned to achieve a particular aim.
Sedimentary	Pertaining to rocks formed by the lithification of accumulated of sediments, formed by the erosion of other rocks.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of the line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur-bearing mineral.
Sustaining Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Tailings	Finely ground waste rock from which valuable minerals or metals have already been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the spatial characteristics (usually grade).

2.5.4 Abbreviations

Table 2.3, Table 2.4, and Table 2.5 provide lists of abbreviations that are used in this Report.

Table 2.3: Abbreviations

Abbreviation	Unit or Term
A	amperes
AA	atomic absorption
AAP	atmospheric arsenic precipitation
AAS	atomic absorption spectroscopy
ABA	acid base accounting
ACI	American Concrete Institute
ADR	adsorption-desorption-recovery
AIC	American Institute of Constructors
AISC	American Institute of Steel Construction
Ag	silver
amsl	above mean sea level
ANFO	ammonium nitrate-fuel oil
AP	acid potential
APO	antimony pentoxide
~	approximately
aq	aqueous
ARD	acid rock drainage
As	arsenic
AT	after tax
$ATNPV_{5\%}$	after-tax net present value at a 5% discount rate
ATO	antimony trioxide
Au	gold
AuCN	assays that determine the cyanide soluble gold content
AuFA	assays that determine the total gold content using the fire assay technique
BDL	below detection limit
BDR	baseline Data Report
BIOX	biological oxidation of sulfides using bacteria in reactor tanks
BMP	best management practices established by the State of Idaho
°C	degrees Celsius
CAPEX	capital expenditures
CCD	counter-current decantation
cfm	cubic feet per minute
CIL	carbon in leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	carbon-in-pulp
CN	cyanide
CO_3	carbonate
CO_3/S	carbonate to total sulfur ratio
COC	chain of custody
CoG	cut-off grade
Con	concentrate
CN WAD	weak acid dissociable cyanide
CSAMT	controlled source audio magneto-tellurics geophysical survey method
CSIRO	Commonwealth Scientific and Industrial Research Organisation
CSMSC	Critical and Strategic Minerals Supply Chains committee
°	degree (degrees)
Detox	chemical destruction of cyanide in the gold barren liquors
dia.	diameter
EFMC	East Fork of Meadow Creek, commonly known as "Blowout Creek"

Abbreviation	Unit or Term
EFSFSR	East Fork of the South Fork of the Salmon River
EGL	effective grinding length
Eh	oxidation reduction potential, measured as mV with a Ag/AgCl 3.8M KCl probe unless otherwise specified
EM	electromagnetic geophysical survey technique
EMF	electromagnetic field
EMF	electromotive force
EPCM	engineering, procurement and construction management
EPH	early production high antimony mineralization from Yellow Pine
EPL	early production low antimony mineralization from Yellow Pine
EO	Executive Order
°F	degrees Fahrenheit
FA	fire assay
famsl	feet above mean sea level
Fe	iron (element)
ft	feet
ft^2	square feet
ft^3	cubic feet
ft^3/st	cubic feet per short ton
FOB	Free on Board
FS	feasibility study, as defined by NI 43-101
g	gram
gal	gallons
g/L	grams per liter
g-mol	gram-mole
gpm	gallons per minute
G&A	general & administration
GCL	geo-synthetic clay liner
GHG	greenhouse gasses
GPS	global positioning system
g/st	grams per short ton
g/t, gpt	grams per metric tonne
h	hour
HAC	hot acid cure
HC	hot arsenic cure
HCT	humidity cell test
HDPE	high-density polyethylene
HERCO	Hermitian Correction model, a statistical analytical tool
HF	Hangar Flats
HFH	Hangar Flats high antimony mineralization
HFL	Hangar Flats low antimony mineralization
HFZ	hidden fault zone at Yellow Pine
Hg	mercury
HMI	human-machine interface
hp	horsepower
HTH	Historic Tailings high-grade gold mineralization
HTL	Historic Tailings low-grade gold mineralization
HTM	Historic Tailings average grade gold mineralization
HWF	Hanging Wall fault at Yellow Pine
ICP	inductively coupled plasma
ICP AES	inductively coupled plasma atomic emission spectroscopy, an analytical method for assaying

Abbreviation	Unit or Term
ICP MS	inductively coupled plasma mass spectrometry, an analytical method for assaying
ICP OES	Inductively coupled plasma optical emission spectrometry
ID	Idaho, where context indicates
ID^2	inverse-distance squared
ID^3	inverse-distance cubed
IMPLAN	Impact analysis for planning
in	Inches
IP	induced polarization geophysical survey technique
IR	infrared
IRR	internal rate of return, a financial measure
I/S	insufficient Sample
kg	kilogram
kg/t	kilograms per metric tonne
koz	thousand troy ounces
kst	thousand short tons
kst/d	thousand short tons per day
kst/y	thousand short tons per year
kV	kilovolts
kW	kilowatts
kWh	kilowatt-hours
kWh/st	kilowatt-hours per short ton
L	liter
lb	pounds
LiDAR	Light Detection and Ranging distance measuring technology
LLDPE	linear low-density polyethylene
LOM	life-of-mine
m	meters
Ma	million years
MACRS	modified accelerated cost recovery system
MBR	membrane bioreactor
MCFZ	Meadow Creek fault zone
mg	milligram
mg/L	milligrams/liter
mi	miles
mi^2	square miles
MIBC	methyl isobutyl carbinol
min	minutes
mL	milliliter or 10^{-3} liters
MLA	mineral liberation analyzer
Mlbs	million pounds
Moz	million troy ounces
Mst	million short tons
Mst/y	million short tons per year
Mt	million tonnes
MFZ	Mule fault zone
MW	megawatts or million watts (where context indicates)
MWMP	Meteoric Water Mobility Procedure (Nevada)
mV	millivolt or 10^{-3} volts
MVA	megavolt amperes
N/A	not assessed

Abbreviation	Unit or Term
NAG	net acid generating
NEPA	National Environmental Policy Act of 1969 (as Amended)
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NNP	net neutralization potential
NP	neutralization potential
NPR	net of process revenue (NPR), defined as NSR less OPEX and G&A
NR	not reported
NSR	net smelter return
OHWM	ordinary high water mark
OPEX	operating expenditures
ORP	oxidation reduction potential, alternative to Eh
oz	troy ounces
oz/st	troy ounces per short ton
%	percent
P_{80}	80% passing a certain size
Pa,,(g)	pascal relative
PAX	Potassium amyl xanthate
PEA	Preliminary Economic Assessment as defined in NI 43-101
PFS	Preliminary Feasibility Study as defined in NI 43-101
PEP	Project Execution Plan
PFD	process flow diagram
pH	logarithmic molar concentration of hydrogen ions
PLC	programmable logic controller
PLS	pregnant leach solution
PMF	probable maximum flood
PoO	Plan of Operations
POX	pressure oxidative leach
ppb	parts per billion
ppm	parts per million (10-6)
ppmv	parts per million by volume
PSD	particle size distribution as measured by laser sizer
psi	pounds per square inch
$PTNPV_{5\%}$	pre-tax net present value at a 5% discount rate
QA/QC	quality assurance/quality control
QEMSCAN	Quantitative Evaluation of Minerals by Scanning electron microscopy
QP	NI 43-101 Qualified Person
RCA	riparian conservation area
RC	reverse circulation drilling
RMS CV	root mean squared coefficient of variation, a statistical tool
ROM	run-of-mine
RQD	rock quality designation
SAG mill	semi-autogenous grinding mill
SEC	U.S. Securities & Exchange Commission
sec	seconds
Sb	antimony
SG	specific gravity
SMC	Sag Mill Comminution
SVFZ	Scout Valley fault zone
SG	specific gravity

Abbreviation	Unit or Term
SIMS	secondary ion mass spectrometry
SMBS	sodium metabisulfite
SODA	spent ore disposal area
SOG	sale-of-gas
SOW	scope of work
SPLP	synthetic precipitation leaching procedure (West Mississippi) a pH 4.2 blended acid leach on synthetically produced metallurgical residues
SRCE	standardized reclamation cost estimator
st	short tons (2,000 pounds)
st/h	short tons per hour
st/d	short tons per day
st/y	short tons per year
TC-RC	treatment charges – refining charges, which are smelter charges
TDS	total dissolved solids
TIC	total inorganic carbon
ton	short ton of 2,000 lbs
tonne	metric tonne of 1,000 kg
tpa	tonnes per annum
TSF	tailings storage facility
TSS	total suspended solids
µ	microns, micrometers (one millionth of a meter)
UTM NAD83	Universal Transverse Mercator North American Datum of 1983 geodetic network
UV	ultra-violet light
V	volts
VFD	variable frequency drive
VHF	very high frequency
VLF-EM	very low frequency electromagnetic geophysical survey
W	watts, where context indicates
W	tungsten, where context indicates
WAD	Weak acid dissociable
WE	West End
WEFZ	West End fault zone
WEO	West End oxide mineralization
WES	West End sulfide mineralization
WRSF	waste rock storage facility
w/w	Weight by weight
XRD	x-ray diffraction
XRF	x-ray fluorescence
Y or y	year
yd	yards
yd^2	square yards
yd^3	cubic yards
YP	Yellow Pine
YPH	Yellow Pine high antimony mineralization
YPL	Yellow Pine low antimony mineralization

Table 2.4: Agency and Related Legal and Regulatory Abbreviations

Abbreviation	Agency Name and Related Act or Regulation or Term
ASTM	ASTM International, known until 2001 as the American Society for Testing and Materials
BEHS	Bureau of Environmental Health and Safety, Division of Health, Idaho Department of Health & Welfare

Abbreviation	Agency Name and Related Act or Regulation or Term
BFPP	bona fide prospective purchaser under CERCLA
BLM	Bureau of Land Management, U.S. Dept. of Interior
CERCLA	U.S. Comprehensive Environmental Response, Compensation, and Liability Act (1980, as amended)
CERCLIS	Comprehensive Environmental Response, Compensation, and Liability Information System
CFR	Code of Federal Regulations (US)
CIM	Canadian Institute of Mining, Metallurgy & Petroleum
CIM Standards	CIM definition standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014
CPO	contiguous property owner under CERCLA
DMEA	Defense Minerals Exploration Administration, Defense Minerals Administration, U.S. Dept. of Interior
DoD	U.S. Department of Defense
EA	Environmental Assessment
EHSP	Environmental Health and Safety Plan
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EPA	U.S. Environmental Protection Agency
ESA	Environmental Site Assessments under ASTM
FAA	U.S. Federal Aviation Administration, U.S. Dept. of Transportation
FCC	U.S. Federal Communications Commission
FLPMA	Federal Land Policy Management Act (1976, as amended)
HAZWOPER	Hazardous Waste Operations and Emergency Response
IDEQ	Idaho Department of Environmental Quality
IDL	Idaho Department of Lands
IDWR	Idaho Department of Water Resources
ID Team	USFS Interdisciplinary Team
IJRP	Idaho Joint Review Process
IPDES	Idaho Pollutant Discharge Elimination System
IRS	Internal Revenue Service
MOU	Memorandum of Understanding under IJRP
MSGP	Multi-Sector General Permit
MSHA	Mine Safety and Health Administration, U.S. Dept. of Labor
NEPA	U.S. National Environmental Policy Act (1969, as amended)
NOAA	U.S. National Oceanic and Atmospheric Administration, U.S. Dept. of Commerce
NOAA Fisheries	Formerly National Marine Fisheries Service, a division of NOAA
NPDES	National Pollutant Discharge Elimination System under the Clean Water Act (1972, as amended)
NPL	National Priorities List under CERCLA
OME	Office of Mineral Exploration, USGS, U.S. Dept. of Interior
RCRA	U.S. Resource Conservation and Recovery Act (1976, as amended)
REC	Recognized environmental condition under CERCLA
ROD	Record of Decision
SEC	U.S. Securities & Exchange Commission
SEDAR	System for Electronic Document Analysis and Retrieval
SOP	standard operating procedures designed by the State of Idaho
SPCC	Spill Prevention, Control and Countermeasures Plan
SRB	China's State Reserve Bureau
SWPPP	stormwater pollution prevention plan
TESCP	threatened, endangered, sensitive, candidate, and proposed species
TMDL	total maximum daily loads
USACE	U.S. Army Core of Engineers, U.S. Dept. of Defense
USBM	U.S. Bureau of Mines, U.S. Dept. of Interior

Abbreviation	Agency Name and Related Act or Regulation or Term
USFS	U.S. Forest Service, U.S. Dept. of Interior
USFWS	U.S. Fish and Wildlife Service, U.S. Dept. of Interior
USGS	U.S. Geological Survey, U.S. Dept. of Interior

Table 2.5: Corporate Abbreviations

Abbreviation	Company Name
AAS	American Analytical Services, an assay laboratory
AGP	AGP Mining Consultants Inc.
ALS	ALS Chemex Labs, Ltd., an assay laboratory
Barrick	Barrick Gold Corporation (formerly American Barrick Resources)
BCM	Blue Coast Metallurgy Ltd.
BioAnalysts	BioAnalysts, Inc.
Biomin	Biomin South Africa (Pty) Ltd, a biological oxidation metallurgical laboratory
Bradley	Bradley Mining Co.
BVRR	Boise Valley Railroad
CSA	Canadian Securities Administrators
Dakota	Dakota Mining Company
Dynamic Avalanche	Dynamic Avalanche Consulting Ltd.
Dynatec	Dynatec Metallurgical Technologies, a pressure metallurgy laboratory
El Paso	El Paso Mining and Milling
Franco Nevada	Franco Nevada Corporation
Gold Crest	Gold Crest Mines Inc.
GeoEngineers	GeoEngineers, Inc.
HDR	HDR, Inc.
Hecla	Hecla Mining Company
Homestake	Homestake Mining Company
IGHC	Idaho Gold Holding Company, a subsidiary of MGC
IGR	Idaho Gold Resources, LLC, a subsidiary of IGHC
IGS	Idaho Geologic Survey
IMC	Independent Mining Consultants, Inc.
INPR	Idaho Northern Pacific Railroad
IPCo	Idaho Power Company
MCSM	Meadow Creek Silver Mines Company
McMillen Jacobs	McMillen Jacobs Associates
MGC	Midas Gold Corp.
MGI	Midas Gold, Inc., a subsidiary of MGC
MGIAC	MGI Acquisition Corporation, a subsidiary of MGI
Midas Gold	Unless otherwise specified, one or more of the subsidiaries of MGC
MinVen	MinVen Corporation
MSE	Millennium Science & Engineering, Inc.
MWH	MWH Americas, Inc.
PAH	Pincock, Allen and Holt
Parametrix	Parametrix, Inc.
Pegasus	Pegasus Gold Corporation
Pioneer	Pioneer Metals Corporation
Ranchers	Rancher's Exploration Company
Rio ASE	Rio ASE, LLC
SGS	SGS Minerals Inc.
SMI	Stibnite Mines Inc., a subsidiary of MinVen and later Dakota
SRK	SRK Consulting (Canada), Inc.

Abbreviation	Company Name
Strata	Strata, a professional services corporation
Superior	Canadian Superior Mining (U.S.) Ltd.
Tierra Group	Tierra Group International, Ltd.
URS	URS Corporation
Vista	Vista Gold Corp.
Vista US	Vista Gold US Inc., a subsidiary of Vista

2.5.5 Standard Core Hole Diameters

Table 2.6 presents standard core hole and core size dimensions referred to in this Report. The conversions have been rounded to the nearest approximate whole fraction of an inch.

Table 2.6: Standard Core Hole Diameters

Size	Hole (outside) diameter	Core (inside) diameter
EX	37.7 mm (1-1/2 in)	21.4 mm (7/8 in)
AQ	48 mm (1-7/8 in)	27 mm (1-1/16 in)
AX	48 mm (1-7/8 in)	30 mm (1-3/16 in)
BQ	60 mm (2-3/8 in)	36.5 mm (1-7/16 in)
BX	60 mm (2-3/8 in)	42.1 mm (1-5/8 in)
NQ	75.7 mm (3 in)	47.6 mm (1-7/8 in)
NX	75.7 mm (3 in)	54.8 mm (2-5/32 in)
HQ	96 mm (3-3/4 in)	63.5 mm (2-1/2 in)
PQ	122.6 mm (4-13/16 in)	85 mm (3-3/8 in)

2.6 REFERENCES

Brown and Caldwell, 2019. SGP Environmental Impact Statement (DEIS) Modified Proposed Action – Chapter 2, Draft, Prepared for Midas Gold Idaho, Inc., May 2019.

U. S. Department of Defense strategic and non-fuel defense shortfalls (2015).

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3 RELIANCE ON OTHER EXPERTS

The Stibnite Gold Project Feasibility Study Technical Report (**Technical Report**) relies on reports and statements from legal and technical experts who are not Qualified Persons as defined by NI 43-101. The Qualified Persons responsible for preparation of this Technical Report have reviewed the information and conclusions provided and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report. This same information was also used to support permitting of the Project under National Environmental Protection Act (**NEPA**) to ensure alignment of the NEPA process and feasibility study assumptions.

3.1 PROPERTY OWNERSHIP AND TITLE

Legal review of the Stibnite Gold Project property ownership and title, presented in Section 4, was completed by multiple qualified, independent, title examiners. Independent legal opinions in respect of mineral title have been prepared on behalf of Midas Gold in support of its initial listing as a public company, subsequent financings, and sale of a royalty to a third party. The most recent opinion and current as of the date of this report was completed on April 25, 2019 by the law firm of Parsons, Behle & Latimer (**PB&L)** building on a comprehensive earlier review by Givens Pursley LLP (**Givens Pursley**). A series of Landman Reports by Almar Professional Land Services, Inc. (**Almar**) were completed in accordance with reasonable industry standards to provide data for the subsequent title opinions.

3.2 WATER RIGHTS

Mr. Terry Scanlan, P.E., P.G. of SPF Water Engineering, LLC (**SPF**) performed a comprehensive review of Midas Gold's water rights portfolio. The water rights held by Midas Gold are summarized in Section 5 of this report.

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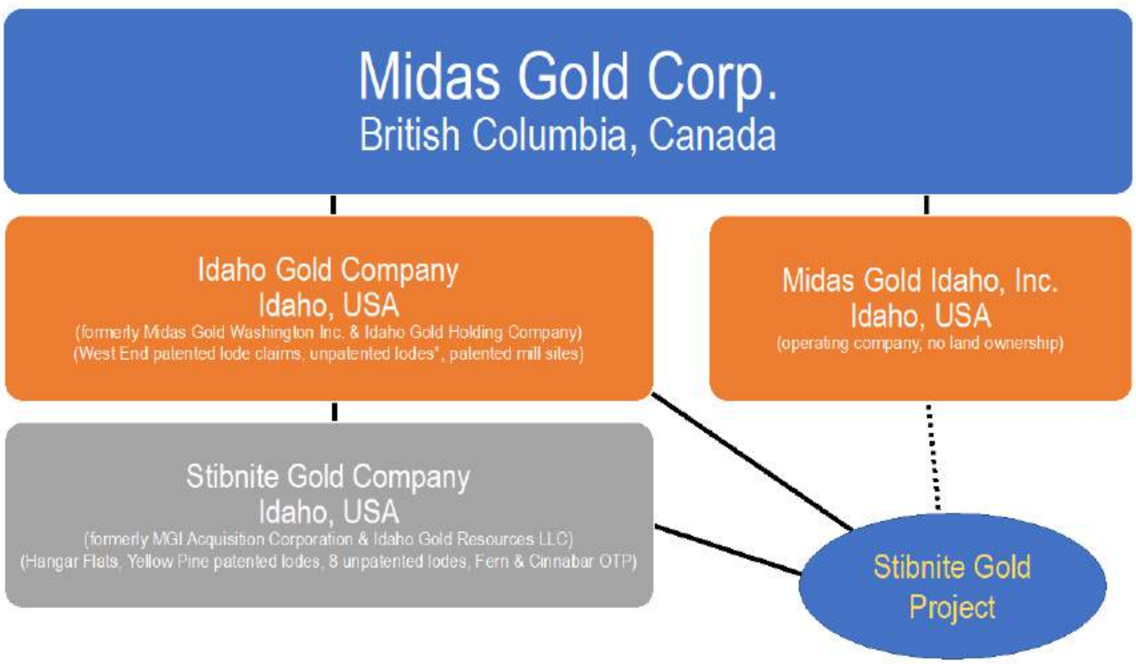
4 PROPERTY DESCRIPTION AND LOCATION

4.1 MINERAL TITLE

Midas Gold's property holdings consist of wholly owned patented lode mining claims, patented mill site claims, unpatented federal lode mining claims and unpatented federal mill site claims (collectively, "Claims") which cover approximately 29,340 acres (approximately 45 mi^2) as shown on Figure 4-2. Additional patented lode claims containing approximately 487 acres adjacent to the SGP area to the east are subject to an Option to Purchase agreement. Appendix II presents a land status map, concession summary and tables listing the unpatented lode claims and mill sites. In a legal opinion, dated April 25, 2019, by Jason Mau of the law firm of Parsons, Behle & Latimer, the patented and unpatented lode mining and mill site claims are owned or optioned by Midas Gold's U.S. subsidiaries; Idaho Gold Resources Company LLC (**IGRCLLC**) and its wholly owned subsidiary Stibnite Gold Company (**SGC**), both Idaho registered business entities. No significant flaws or title issues have been identified in multiple formal title reviews of the Claims performed by qualified, independent, title examiners. A number of independent legal opinions in respect of mineral title have been prepared on behalf of Midas Gold in support of its initial listing as a public company, subsequent financings, and sale of a royalty to a third party.

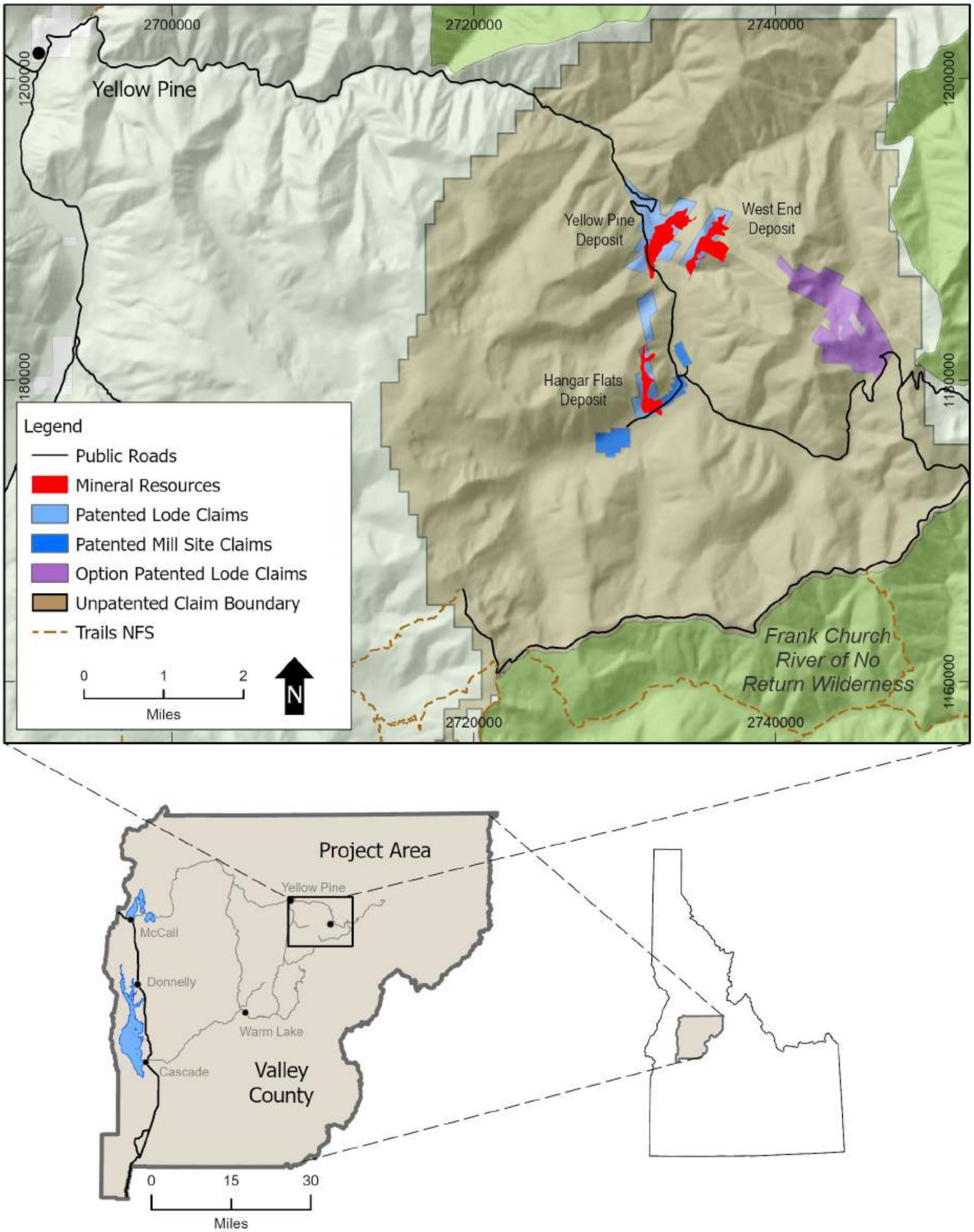
The organizational structure of the entities holding title to the Properties is provided on Figure 4-1.

Figure 4-1: Corporate Organizational Structure



Through a series of name changes and consolidations, the various subsidiaries identified in this Report have been consolidated into three entities: Idaho Gold Resources Company, LLC, an Idaho limited liability company; Stibnite Gold Company, an Idaho corporation and wholly owned subsidiary of Idaho Gold Resources Company LLC which is turn is a wholly owned subsidiary of Midas Gold Corp. Midas Gold Idaho, Inc., an Idaho corporation and wholly owned subsidiary of Midas Gold Corp. holds no real property ownership interests but is the operating company for the land-owning interests.

Figure 4-2: Project Location Map



4.2 LOCATION

The Project is located in Valley County, Idaho approximately 98 mi northeast of Boise, Idaho, 40 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho (Figure 4-2) in all or part of the following sections (Boise Meridian):

- Township 17 North, Range 8 East, Sections 12 to 13, 23 to 24, and 26;
- Township 17 North, Range 9 East, Sections 4 to 8 and 13 to 19;
- Township 18 North, Range 9 East, Sections 1 to 30 and 32 to 36;
- Township 18 North, Range 10 East, Sections 5 to 8, 17 to 20 and 29 to 30;
- Township 19 North, Range 9 East, Sections 21 to 28 and 32 to 36; and
- Township 19 North, Range 10 East, Sections 19, 30, and 31.

The Project area elevations range from approximately 6,500 ft to more than 8,900 ft above sea level and is centered at latitude 44°54'25" N and longitude 115°19'37" W and, in State Plane Idaho West coordinates, at 1103:1,181,270 ft US N and 1103:2,734,259 ft US W.

4.3 NATURE AND EXTENT OF TENURE

The following description was accurate as of the effective date of this Technical Report. Claim groups under Midas Gold's U.S. subsidiaries ownership are discussed in this section while those with encumbrances are detailed in Section 4.4.

4.3.1 Patented Lands

On June 11, 2009, a predecessor to Stibnite Gold Company acquired and exercised an option to purchase (**OTP**) the Meadow Creek group of nine patented lode claims totaling approximately 184 acres from Bradley Mining Co. (**Bradley**).

A predecessor to Idaho Gold Resources Company, LLC secured an OTP agreement from the J.J. Oberbillig Estate on June 2, 2009 to acquire 30 patented mill site claims totaling approximately 149 acres and six patented lode claims totaling approximately 124 acres. The Oberbillig OTP agreement was exercised and title to property rights were acquired on June 2, 2015. An associated transaction included the purchase and extinguishment of a 5% Net Smelter Return (**NSR**) royalty to the Oberbillig estate covering certain lands withing the SGP area. The majority of the mineralization constituting the West End Deposit is located within portions of these patented lode claims. Hecla Mining Company **(Hecla)** retains some surface rights on portions of six of the patented mill sites, but no mineral rights and IGRCLLC has a right to use the surface for the purposes of mining.

An OTP for patented lode mining claims covering of portions of the Yellow Pine Deposit was conveyed to Midas Gold in 2011 by way of a company merger between a predecessor to IGRCLLC and a subsidiary of Vista Gold Corp. (**Vista**) that was agreed to February 22, 2011. The OTP for the subject patented claims was exercised on November 28, 2012. As a result of the merger, the predecessor to IGRCLLC became a wholly owned subsidiary of Midas Gold Corp. The Yellow Pine claim group includes 17 patented lode mining claims totaling approximately 301 acres and eight unpatented lode mining claims (already included in the unpatented total above).

On April 28, 2011, a predecessor to Stibnite Gold Company purchased 6 patented lode claims east of the Project area. This group of claims is referred to as the Fern claim group, totaling approximately 100 acres.

Property taxes for the patented claim groups are paid in full as of the effective date of this Report and are included in Appendix II.

4.3.2 Unpatented Federal Lode Mining Claims and Unpatented Mill Site Claims

A subsidiary of a predecessor to IGRCLLC acquired 229 federal unpatented claims by purchase from previous owners in 2009 and 2011. These included 46 federal mill site claims and 183 federal unpatented lode-mining claims. In addition to the purchased claims, IGRCLLC predecessors or subsidiaries acquired by staking an additional 36 federal unpatented lode mining claims in 2009, 217 lode claims in 2010 and 901 federal unpatented lode-mining claims in 2011, and one federal unpatented lode-mining claim in 2012. An additional 126 unpatented lode claims were staked in 2015. Minor modifications and amended claim locations have occurred since original staking and/or acquisition. A complete list of active claims is included in Appendix II. Currently, 1,465 unpatented lode mining and 46 mill site claims totaling approximately 28,482 acres (11,526 hectares) constitute part of the overall land position as of the effective date of this report.

Maintenance of unpatented federal claims requires that IGRCLLC and SGC provide a list of claims and serial numbers to the Bureau of Land Management (**BLM**) along with annual maintenance fees, currently $165 for each lode mining claim or mill site on or before September 1st each year. This was completed for the most recent filing year on August 3, 2020, and an Affidavit of Satisfaction was subsequently recorded in Valley County in August 26, 2020. There is no underlying royalty on these federal lode mining claims and mill site other than the Franco-Nevada Corporation (**Franco-Nevada**) royalty detailed in Section 4.4. None of the Claims are subject to back-in rights.

4.3.3 Stibnite Gold Logistics Facility

On September 9, 2016, Idaho Gold Resources Company, LLC agreed to purchase a fee simple undeveloped 25-acre property in Section 7, Township 14N, Range 5E, Boise Meridian from private interests and closing of the property occurred on October 26, 2016. The property's metallic and non-metallic mineral rights, with the exception of aggregate materials needed for construction purposes on the property were retained by the previous owners. The property, in an area known locally as Scott Valley, has frontage on the Cascade-Warm Lake Highway and was purchased to serve as a project logistics center. The agreement provides for maintenance of certain pre-existing rights-of-way, easements and rights, none of which would be expected to inhibit use of the property for the intended purposes. Idaho Gold Resources Company, LLC has applied for a Conditional Use Permit from the Valley County Planning and Zoning Commission which was granted on October 5, 2020.

4.4 ROYALTIES, OPTION AGREEMENTS AND ENCUMBRANCES

4.4.1 Option Agreements

On May 3, 2011, a predecessor to SGC entered into an option to purchase 27 patented lode claims totaling approximately 485 acres from the J.J. Oberbillig Estate (the Cinnabar option claims). This agreement was modified in an Amended and Restated Real Property Purchase Agreement effective December 1, 2016. The amended agreement also includes an option on a Right of First Refusal to purchase the surface rights associated with portions of certain patented mill site claims that J.J. Oberbillig Estate sold to Hecla under a Real Estate Purchase and Sale Agreement dated effective as of December 30, 2002. The agreement also includes granting of a renewable easement for a communications tower. Midas Gold is obligated to make option payments to maintain the OTP to obtain title to these claims. As of June 30, 2020, the remaining option payments due on the Cinnabar property are US$80,000, which will be paid over the next two years. The agreement includes an option to extend up to 20 years. Property tax information for all claim groups is included in Appendix II.

On December 10, 2019, a Midas Gold subsidiary entered into an option agreement to purchase 3.74 acres from private interests for an electrical switching station site. The OTP has biannual payments of US$2,500 through 2033.

4.4.2 Royalty Agreement

Effective May 9, 2013, Midas Gold's US subsidiaries granted a 1.7% NSR royalty on future gold production from the Project properties to Franco-Nevada. The royalty does not apply to production of antimony and silver. The royalty agreement applies to all patented and unpatented mineral claims, with the exception of the Cinnabar claim group where Midas Gold holds an option to purchase but would be extend to the Cinnabar claim group were the OTP exercised.

4.4.3 Right of First Refusal

On May 16, 2018, Midas Gold entered into an investor rights agreement as part of a financing arrangement with Barrick Gold Corporation (**BGC**) which included a Right of First Refusal (**ROFR**) for BGC to purchase gold concentrates, if shipped off site, produced from the Project subject to certain terms and conditions as outlined in the agreement. The ROFR stipulates BGC has 45 days from the date of delivery to BGC of any offer to purchase concentrates or establish a streaming agreement pertaining to the concentrates from the Project to exercise its ROFR in respect thereof and to acquire such concentrate Interest on substantially the same terms and conditions as are set forth in the ROFR offer or on such other terms and conditions that provide substantially equivalent benefits to Midas Gold having regard to the financial, commercial and other relevant terms. The rights and obligations of the ROFR would terminate immediately at such time as Barrick Gold Corporation ceases to hold at least 10% of the issued and outstanding Common Shares of MGC.

4.4.4 Consent Decrees under CERCLA

Several of the patented lode and mill site claims held by IGRCLLC and SGC comprising part of the West End Deposit, and the Cinnabar claims held under an OTP from the Estate of J.J. Oberbillig are subject to a consent decree entered in the United States District Court for the District of Idaho (United States v. Estate of J.J. Oberbillig, No. CV 02-451-S-LMB (D. Idaho)) in 2003, involving or pertaining to environmental liability and remediation responsibilities with respect to the affected properties described therein. This consent decree provides property access to the regulatory agencies that were party to the agreement and the right to conduct remediation activities under their respective Comprehensive Environmental Response, Compensation, and Liability Act (**CERCLA**) and Resource Conservation and Recovery Act (**RCRA**) authorities as necessary and required to prevent the release or potential release of hazardous substances. In addition, the consent decree requires that heirs, successors and assignees refrain from activities that would interfere with or adversely affect the integrity of any remedial measures implemented by government agencies.

Certain mineral properties held by SGC and that portion of the mineral properties acquired from Bradley estate pursuant to the Bradley Mining Agreement (i.e. the Yellow Pine Deposit) are subject to a consent decree (United States v. Bradley Mining Co., No. 3:08-CV-03986 TEH (N.D. Cal.)). The consent decree was lodged on February 14, 2012 and approved on April 19, 2012. The consent decree states that if the U.S. Environmental Protection Agency (**EPA**) or the USDA Forest Service determines that "land/water use restrictions in the form of state or local laws, regulations, ordinances or other governmental controls are needed to implement response activities at the Stibnite Mine Site, ensure the integrity and protectiveness thereof, or ensure non-interference therewith" Bradley Mining or its heirs successors or assigns agree to cooperate with EPA's or the Forest Service's efforts to secure such governmental controls.

Midas Gold cannot ensure it has identified every consent decree or administrative order which may affect the Stibnite Gold Project.

Under CERCLA, a "bona fide prospective purchaser" defense is a legal defense available to an owner who, after conducting appropriate inquires, establishes that environmental liability occurred before the owner acquired the property. Midas Gold has taken and will continue to take all steps required to establish itself as a bona fide prospective purchaser.

4.5 ENVIRONMENTAL LIABILITIES

The Project is located in a historical mining district with extensive and widespread exploration and mining activity, and related environmental effects, spanning nearly 100 years from the early 1900s until today. For detailed ownership and mine development history in the District refer to Section 6 of this Report.

Actions by prior operators and government agencies have addressed some of the historical environmental issues at the site, but extensive disturbance and adverse environmental impacts remain. Potential environmental liabilities from legacy operations and activities that could have impacts on development of the Project are discussed in Section 20.

4.6 PERMITTING

4.6.1 Exploration Permits

The exploration programs completed by Midas Gold Idaho, Inc. (**MGII**) IGRCLLC and SGC (and predecessor companies) to date consisted of road and drill pad construction to support drilling on both public and private lands. There are different permitting requirements for activities on the respective public and private land holdings.

The USFS, Payette National Forest, Krassel Ranger District has jurisdictional authority over mitigating surface disturbance associated with exploration and mining-related activities on public lands within its administrative area. Although some of the claims are in the Boise National Forest, the Payette National Forest has been granted administrative authority for the entire Project area. Idaho Department of Lands **(IDL**), Payette Lakes Area District has jurisdictional authority over exploration and mining-related activities on private lands (as well as oversight on activity on public lands as well) within its administrative area.

MGII, on behalf of IGRCLLC and SGC, is currently conducting exploration in the District on patented property under an annual IDL Notice of Exploration. MGII currently has an exploration Plan of Operations (**PoO**) filed with the USFS under POO-2014-049059, which was issued in 2016 and valid for three years but was subsequently extended in 2020 and is in force at the time of this Report. This permit has an associated bond of US$169,000 held by the USFS to cover potential and any existing liabilities associated with the exploration permit.

MGII, IGRCLLC and SGC are in full compliance with applicable laws and regulations related to its exploration activities. The staff of IDL, USFS, U.S. Fish and Wildlife Service (**USFWS**), EPA, IDEQ, and National Marine Fisheries Service (**NMFS**) have toured the Project site several times during ongoing activities and have issued required permits and granted approval for Midas Gold's subsidiaries' activities on the site.

4.6.2 Mine Development Permits

The environmental permitting process for the development of a mine within the Project boundaries involves water quality permits, wetlands permits, surface and ground water use permits, authorizations to relocate stream channels, permits addressing design and construction of a tailings dam, air-quality permits, a cyanide use permit, and approval of a final operating and reclamation plan. In total, over 30 separate local, state, and federal environmental permits and licenses would be required to construct and operate a mine within the Project boundaries. See Section 20 for a discussion of permits required for development, operations and closure.

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5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

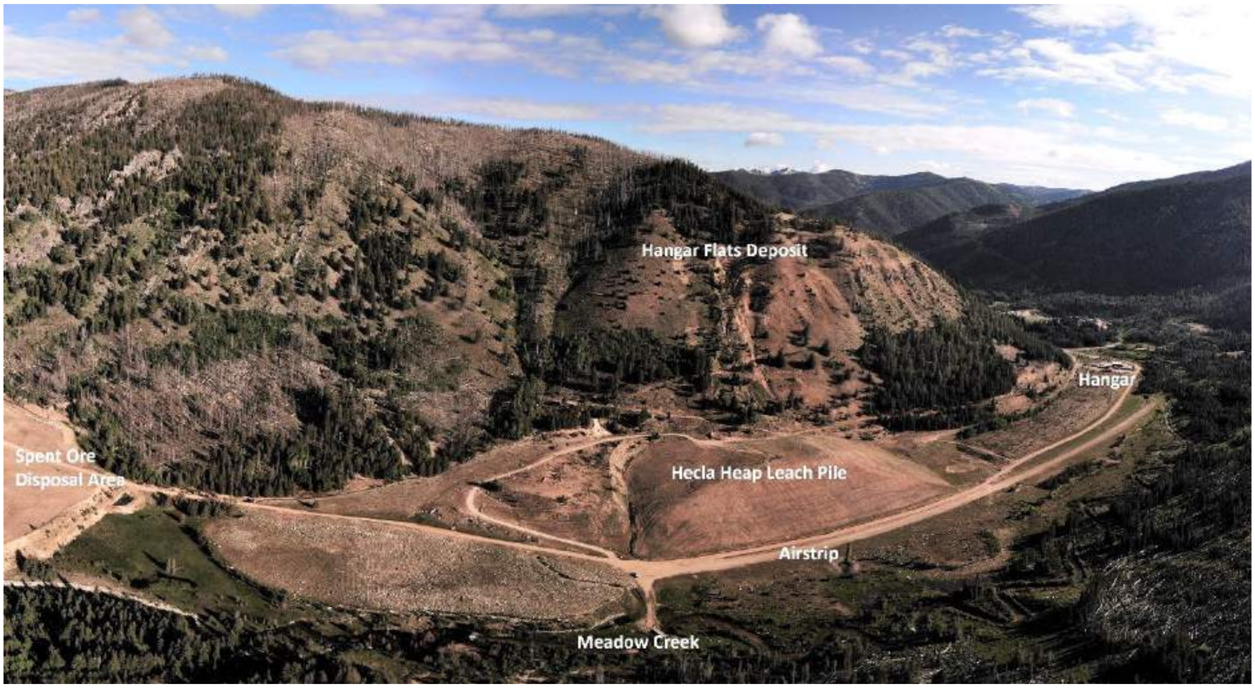
5.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The Project is located within the Salmon River Mountains of Central Idaho. The area consists of uplifted rocks of the Idaho Batholith deeply incised by the East Fork of the South Fork of the Salmon River (**EFSFSR**). The area is comprised of steep, rugged, and forested mountains at an elevation of approximately 7,800 to 8,900 ft with narrow, flat valleys at an elevation of approximately 6,500 ft. The land is heavily wooded with fir and pine trees and underbrush common. Large forest fires burned much of the area in 2002, 2006 and 2007. Photograph 5-1: and Photograph 5-2 depict local topography, vegetation, and surface features.

Photograph 5-1: View Looking South Along the EFSFSR

Source: Midas Gold, 2020

Photograph 5-2: View Looking North Toward Hangar Flats Deposit



Source: Midas Gold, 2020

5.2 CLIMATE AND LENGTH OF OPERATING SEASON

The climate is characterized by moderately cold winters and mild summers. Most precipitation occurs as snowfall in the winter and rain during the spring. The local climate allows for year-round operations as evidenced by historic production and climate information.

Weather records indicate that the average precipitation (equivalent rainfall) is approximately 32.2 inches per year. Average monthly temperatures and precipitation are shown in Table 5-1.

Table 5-1: Project Climate Data

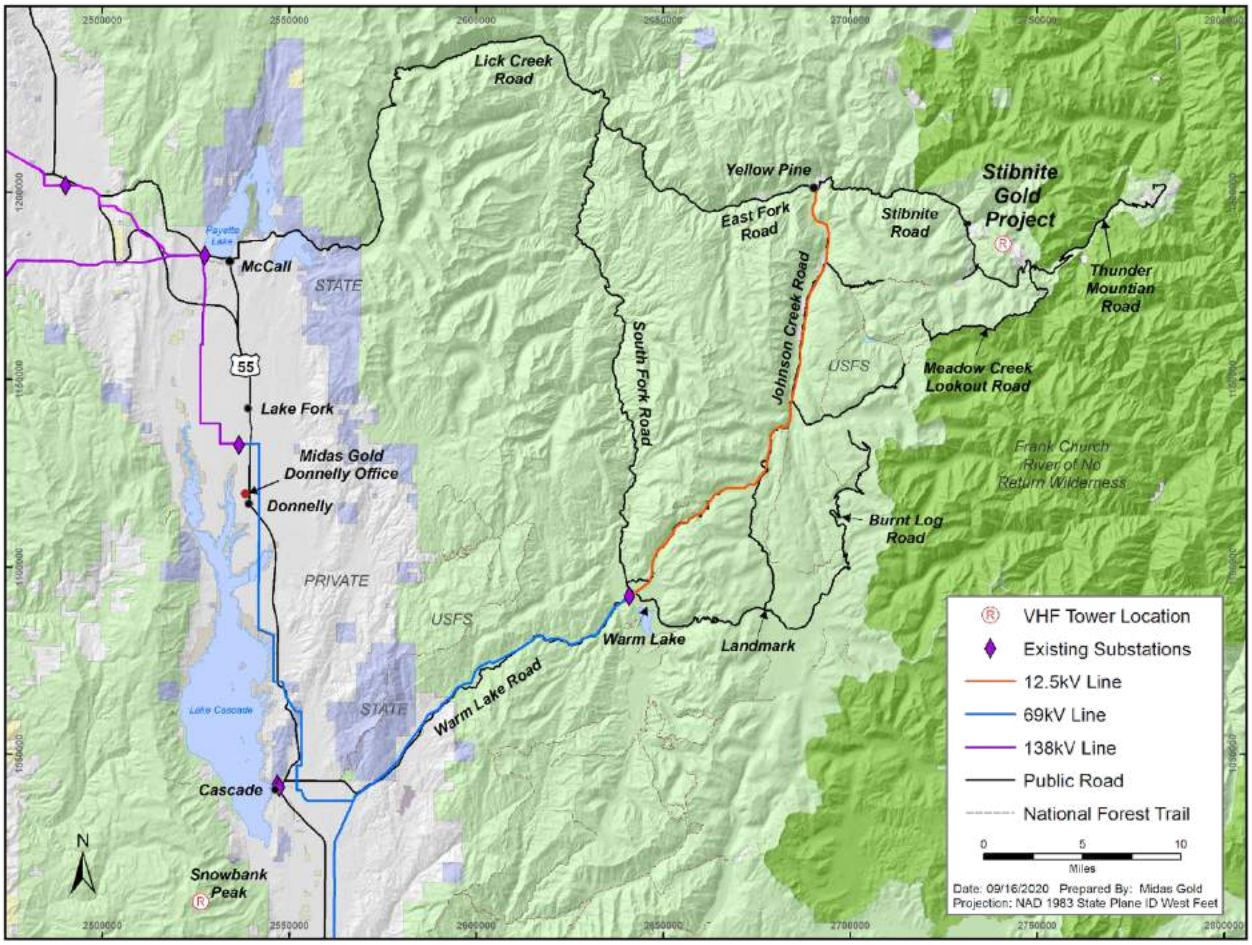
Month	Average Temperature (˚F)	Average Precipitation (in)
January	20.1	4.1
February	21.8	3.3
March	27.7	3.5
April	32.9	3.0
May	40.7	2.6
June	48.7	2.1
July	58.1	1.0
August	56.5	1.0
September	48.7	1.8
October	39.2	2.1
November	26.3	3.7
December	18.8	4.0
Average	**36.6**	**32.2**

5.3 ACCESS TO PROPERTY

The property is located approximately 152 road-miles northeast of Boise, Idaho. Figure 5-1 shows a map of current access routes. The primary access to the Project area is known as the Johnson Creek Route and includes:

- Boise to Cascade – Highway 55 (77.4 mi);
- Cascade to Landmark – two-lane, paved Warm Lake Road (35.6 mi);
- Landmark to Yellow Pine – single-lane, unpaved Johnson Creek Road (25.3 mi); and
- Yellow Pine to Stibnite – single-lane, unpaved Stibnite Road (14 mi).

Figure 5-1: Site Access and Pertinent Existing Regional Infrastructure



The **Johnson Creek Route** measures approximately 74 mi from Cascade to Stibnite and is not available at certain times of the year when Johnson Creek Road is impassable due to snow. Alternatively, the **South Fork Route** provides year-round access to Stibnite because it maintains a lower elevation profile. The route follows Warm Lake Road before turning north on the South Fork Road and then turning east onto the East Fork Road towards Yellow Pine and on to the Project site via Stibnite Road. The distance from Cascade to Stibnite is approximately 96 mi along this alternate South Fork Route.

Another route available in snow-free months starts by travelling east on Lick Creek Road near McCall, Idaho, towards Yellow Pine and onto Stibnite (the "**Lick Creek Route**"). The distance from McCall to Stibnite along the Lick Creek Route is approximately 67 mi and approximately 94 mi from Cascade to Stibnite via McCall.

A grass airstrip is located along Johnson Creek Road approximately 3 mi south of the town of Yellow Pine and a 2,300 ft long improved gravel airstrip is located at Stibnite.

5.4 SUFFICIENCY OF SURFACE RIGHTS

Midas Gold's US subsidiaries control approximately 28,482 acres of unpatented and 1,356.6 acres of patented lode and millsite claims (Appendix II provides a detailed listing of these claims). Surface facilities associated with development of the Stibnite Gold Project would be located on a combination of public and private property under rights established by the 1872 Mining Law, current USFS regulation, and IDL regulations for private property mining development. Authorization for such development will come in the form of a Record of Decision (**ROD**) by the USFS, approving (potentially with modifications) the Plan of Restoration and Operations (**PRO**), after completion of the Final Environmental Impact Statement (**EIS**). Ancillary permits from other agencies, required financial assurance for reclamation, and a mined land reclamation plan approved by IDL will also be required for development. Agency reviews of the PRO, reclamation plan, and EIS are in progress, with the Draft EIS public comment period having concluded just before this writing. Additional information on Midas Gold subsidiaries' patented and unpatented claims is provided in Section 4; additional information on permitting is included in Section 20.

5.5 LOCAL RESOURCES AND INFRASTRUCTURE

5.5.1 Power Supply

The nearest powerline is located along Johnson Creek Road, roughly 8 mi west of Stibnite (Figure 5-1). The powerline along Johnson Creek Road provides 12.5 kV distribution power to local residents along the route and the village of Yellow Pine but would be insufficient to support a mining operation. In order to support operations related to the Project, powerline infrastructure would need to be installed / upgraded from the main regional Idaho Power Company (**IPCo**) substation at Lake Fork to the Project site. A description of the proposed powerline upgrade is addressed in Section 18 of this Report.

5.5.2 Water Supply

Midas Gold's US subsidiaries have four permanent and three temporary water rights in the district (collectively, "**Water Rights**"). The permanent Water Rights were transferred from the estate of J.J. Oberbillig and Bradley (Table 5-2).

Midas Gold US subsidiaries' current water rights are insufficient to support the proposed Stibnite Gold Project development plan included herein, and additional rights will need to be secured through direct permit application and subsequent approval of such rights from the IDWR. Additional information regarding water rights and permitting is included in Section 20.

Table 5-2: Water Rights Summary

Water Right ID	Type	Source	Location of Point of Diversion	Beneficial Use	Maximum Diversion Rate (ft^3/s)	Maximum Annual Diversion (acre-feet)
77-7122	Surface Water	EFSFSR	NW ¼ of the NW ¼ , Section 14,T 18N, R9E	Storage and Mining	0.33	7.1
77-7141	Ground Water	Well	SW ¼ of the SW ¼, Section 11, T18N, R9E	Domestic	0.20	11.4
77-7285	Ground Water	Well	SE ¼ of the NE ¼, Section15, T18N, R9E	Storage and Mining	0.50	39.2
77-7293	Surface Water	Unnamed Stream (Hennessy Creek)	SW¼ of the NE¼, Section3, T18N, R9E	Mining	0.25	20.0
Source: IDWR, 2019						

5.5.3 Rail

The Idaho Northern Pacific Railroad (**INPR**) is a Class II railroad that owns railroad tracks that terminate in Cascade, Idaho. The INPR formerly operated the Cascade Branch rail line on approximately 100 mi of track between Payette, Idaho and Cascade with a switchyard in Emmett, Idaho. INPR presently operates between Payette and Emmett; however, if Project and other freight were sufficient, the line between Emmett and Cascade could be reactivated. Existing tracks from Cascade to Payette connects the line to the Union Pacific Railroad, which is capable of reaching ports in California, Oregon, Washington, and British Columbia.

The INPR previously operated a tourist train, the Thunder Mountain Line, on its Cascade Branch, which ran from Horseshoe Bend to Banks, Idaho. Active freight service to Cascade ceased in the mid-1990s, but INPR continues to perform maintenance and inspections required by the Federal Railroad Association of out of service trackage to Cascade. INPR owns land at the terminus of the rail line for switching and transloading facilities. Currently, facilities at the Cascade end of the track are limited.

Also serving the area and connecting to the Union Pacific Railroad is the Boise Valley Railroad (**BVRR**) at Nampa, Idaho located approximately 176 mi from the Project site. Currently, the BVRR is a short-line railroad connecting Nampa with the state capital Boise, Idaho. Of the two rail lines, the BVRR is much further from the Project site; however, in May 2010, the City of Boise signed a letter of intent with the BVRR to explore construction of a transloading and intermodal services facility in southeast Boise. Though construction of the proposed facility has not progressed beyond the initial letter of intent, if constructed, this facility would enable container freight to transfer directly from truck to train. Currently, the nearest facility for direct container handling of the type proposed is in Portland, Oregon.

5.5.4 Ports

The closest access for sea transportation is through the ports of Portland, Oregon; Tacoma, Washington; Seattle, Washington; and Vancouver, British Columbia. Each of these ports is located in the Pacific Northwest and can be accessed by truck or by rail with distances ranging from 573 to 727 mi from the Project. The Port of Portland is the closest of these four options; Terminal Six is the predominant container terminal at the port and is presently served by the SM Line on a weekly basis.

Additionally, the inland Port of Lewiston, Idaho, is located on the Clearwater River, just upstream from its confluence with the Snake River and is approximately 274 mi from the Project site. The port is served by truck and rail and loads barges for shipment down the Snake and Columbia Rivers. The port is used primarily for shipping agricultural products. Wheat is shipped in bulk, but many of the other commodities are shipped in containers. The port also hosts a transloading facility where items are containerized for shipment. Containers travel down the Columbia to Portland's Terminal Six a few days prior to being loaded onto a vessel for Asian ports of call.

5.5.5 Communications

In 2013, Midas Gold completed a microwave relay tower atop a 9,000-ft peak on the east side of the property (Figure 5.1). The tower is on leased patented land and provides a reliable long-term link to the regional communications hub on Snowbank Mountain 52 mi to the southwest. The relay operates at 5.8 GHz and uses a 6 ft diameter parabolic antenna (40 ft above land surface on the Stibnite end of the link) to provide a high bandwidth connection to a commercial leased tower facility, access to which is maintained year-round by the Federal Aviation Administration (**FAA**). A second smaller radio system relays the signal down to the valley floor via an intermediate tower near Midas Gold's Very High Frequency (**VHF**) repeater at West End. At the Stibnite tower sites, continuous and reliable power is provided by solar panels and battery systems designed to withstand the winter conditions at these locations.

Another 20 mi microwave link connects the Snowbank facility directly to Midas Gold's Donnelly office, providing an entirely private and Midas Gold-owned communication path. A virtual private network connects the Boise office directly into this system and creates an environment where all Idaho facilities are under one virtual roof with respect to electronic data. Local servers are backed up offsite on a nightly basis to a Midas Gold-owned co-located server.

5.5.6 Potential Processing Site

The majority of the Project area is characterized by steeply-sloping, mountainous terrain. Flat terrain with competent foundation conditions suitable for mine infrastructure is generally limited; these areas are typically in the valley bottoms, near the geologic contact between bedrock and colluvium or alluvium, which is consistent with infrastructure siting by previous mine operators.

The following methodology was used to arrive at the preferred process plant site:

1) Identify the primary physical constraints that limit the area that could be considered for process plant infrastructure such as: geotechnical constraints, avalanche constraints, regulatory constraints, project development constraints, etc.
2) Develop a scorecard that includes the key drivers/criteria that influence selection of the preferred process plant layout. The criteria could include environmental, permitting, and social considerations; safety considerations; capital expenditures (**CAPEX**); operating expenditures (**OPEX**); and operability considerations.
3) Develop conceptual project layouts that honor the preceding physical constraints with consideration to the key drivers.
4) Populate the scorecard in a workshop environment to identify the preferred process plant layout.

A large, gently sloping area immediately northeast of the confluence of Meadow Creek and the EFSFSR was selected as the preferred processing plant location. Section 18 provides a detailed discussion on the layout of the process plant and Section 20 presents a general site layout that includes the preferred process plant location relative to historically impacted areas of the Project site.

5.5.7 Potential Tailings Storage Area

Approximately 114 million tons of mineralized material are expected to be processed during the 14.25-year mine life of the Project. Ideally, from an environmental, technical, and financial perspective, all of the tailings generated from the operation would be stored in a single storage facility. To determine the preferred location for the tailings storage facility

(**TSF**), a siting assessment was completed that identified four locations that could provide sufficient storage capacity to contain the expected tailings quantities; this study is summarized in Appendix G of the PRO (Midas Gold, 2016).

The preferred tailings site was determined to be in the upper portion of the Meadow Creek valley, based on considerations such as: topography, hydrology, reuse of previously disturbed areas, environmental management and closure considerations, proximity to the processing plant, and expected cost. That area has sufficient capacity for both tailings and development rock, and a significant portion of the area has been previously disturbed by historical mining operations. This site keeps incremental disturbance to a minimum by overlapping historically disturbed areas used previously for tailings disposal; has superior long-term stability, reclamation, and closure characteristics (with 90% of the perimeter being mountains); and low impact on accessible fish habitat. A comprehensive description of the TSF is provided in Section 18.

Some of the land in the Meadow Creek valley is owned by Midas Gold and comprises patented mining claims; the balance of the land in the valley is Federal land managed by the USFS.

5.5.8 Potential Development Rock Storage Areas

There are several locations on the Stibnite Gold Project site where uneconomic mineralized or unmineralized material ("**development rock**") could be stored, which were evaluated in a similar manner and with similar considerations as the siting of the TSF. The preferred storage area for the Yellow Pine and Hangar Flats development rock is in the Meadow Creek valley downstream of the TSF, which would also provide a robust geotechnical stability buttress for the TSF. This site is preferred since it keeps incremental disturbance to a minimum by overlapping historically disturbed areas used previously for tailings disposal and spent heap leach ore disposal and keeps the development rock and tailings within the same area, which is preferable for water management and from a long-term reclamation and restoration perspective. The preferred storage location for the West End development rock is in the mined-out Yellow Pine open pit, which would enable the EFSFSR to be reestablished to its approximate pre-mining location and gradient, facilitating long-term fish passage to the headwaters of the EFSRSR and Meadow Creek. Additional West End development rock would be used to backfill the Midnight pit (a small satellite pit in West End) and Hangar Flats pit. Much of the proposed development rock storage land is owned by Midas Gold's US subsidiaries and comprises patented mining claims; the rest of the land in the valley is Federal land managed by the USFS. This layout keeps the maximum amount of disturbance within the existing footprint of historical disturbance. Sections 16 and 18 provide additional details on the development rock storage facilities.

5.5.9 Labor

Yellow Pine, which is the nearest town, is located approximately 14 road miles west of the Project. It has a population of approximately 60 people during the summer months, up to 40 in the winter, and limited services such as a general store (now closed), two restaurants, and a few lodging facilities. The nearby Valley County towns of McCall, Donnelly, and Cascade and surrounding areas have a combined population of several thousand people with many diverse services available.

Skilled miners, mining professionals, local laborers, and equipment operators would be identified from within Valley County and adjacent Adams and Idaho counties with additional workers sourced throughout Idaho and adjacent states if necessary.

MGI would likely become the largest employer in Valley County, Adams County, and potentially Idaho County, paying higher salaries than any other industry except the federal government. While unemployment in these counties is presently low, they have had some of the historically highest unemployment rates in Idaho. Further, studies indicate that there has been a significant out-migration of working-age people due to the lack of employment, which would likely be reversed were well-paid jobs available. Project-related jobs would strengthen the local manufacturing, services, and supply sectors and provide an important complement to the region's recreation industry. The property and sales taxes generated from the construction and mining operations would help support the region's schools and infrastructure. The infusion of new economic activity would likely help support every industry in the regional economy through the significant increase in direct, indirect, and induced employment in the region. Additional considerations on employment can be found in Chapter 4 of the DEIS.

Additional details on the Project labor requirements and approaches to meeting those needs are discussed in Section 20.

5.6 REFERENCES

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6 HISTORY

Although mineralization was first discovered by prospectors in the SGP area during the late 1800s, the earliest significant development activity didn't occur until after the turn of the century. Mining claims associated with the Meadow Creek Mine and Yellow Pine Mine (first staked in 1914 and 1923, respectively) were developed and many patented during this period by various interests. Over time ownership was consolidated primarily by two major landowners who controlled the majority, but not all of the land within the Stibnite Mining District (**District**). The eastern part was partially consolidated by United Mercury Mines with ties to the Oberbillig family; whereas the western part of the District was controlled by the Bradley interests, with ties to the Bradley family. The United Mercury Mines properties subsequently went through a series of ownership changes ultimately resulting in their control by the estate of JJ Oberbillig. The Bradley interests ultimately ended under the control of the Bradley Mining Co.

Bradley production was initially from the underground Meadow Creek mine (ca 1927 to 1937) and later from the larger Yellow Pine underground and subsequently open pit mine (1937 to 1952). Bradley's consolidation of the western portion of the District led to the Oberbillig family receiving royalties on some of the claims mined by Bradley. Bradley operated the Yellow Pine pit until 1952. Mining operations ceased after a worldwide collapse in antimony prices following the end of the Korean War, while milling and smelting continued periodically from stockpiled ores, as well as antimony-bearing materials from the Coeur d'Alene district. The former mill and smelter were subsequently dismantled, and the Stibnite town site abandoned completely in 1958, with many of the cabins and other buildings comprising the town site moved elsewhere. More detailed summaries of site history and ownership can be found in Mitchell (2000) and in the Midas Gold Plan of Restoration and Operations, Appendix D (Midas Gold, 2016).

The District lay dormant until the early 1970s aside from small-scale mining of antimony ores near the Murray Prospect (in the Garnet Creek drainage) and from the Bonanza Prospect (in the Sugar Creek drainage) by the Oberbillig interests in the 1960s, and mercury mining at the nearby Cinnabar mine by Holly Minerals Corp. in the 1950s. A sharp rise in gold prices in the 70's and the advent of heap-leach processing technology for oxide gold ores revitalized exploration in the District. Operators who conducted exploration and/or mineral extraction during this era included, in chronological order, Louisiana Land and Exploration Company, Canadian Superior Mining (U.S.) Ltd. (**Superior**), El Paso Mining and Milling (**El Paso**), Rancher's Exploration Company (**Ranchers**), Twin Rivers Exploration, MinVen Corporation (**MinVen**), Pioneer Metals Corporation (**Pioneer**), Hecla Mining Company (**Hecla**), Barrick Gold Corporation (**Barrick**, formerly American Barrick Resources), Stibnite Mines Inc. (**SMI**), and Dakota Mining Company (**Dakota**).

Hecla delineated a small zone containing oxide mineralization on the hill above the Hangar Flats Deposit but focused mainly on mining the nearby Homestake oxide gold deposit, which overlies the northeastern portion of the Yellow Pine Deposit. Superior delineated much of what is now the West End Deposit and brought that area into production in 1982. Superior was ultimately acquired by the Superior Oil Company of Houston, Texas, which was acquired by Mobil Oil. Mobil sold the West End Mine in 1986 to a 50/50 joint venture of Pioneer and MinVen, both Canadian-registered companies. Pioneer was the mine operator until it experienced financial problems in 1990, and ownership was conveyed to SMI, owned primarily by MinVen. MinVen later experienced financial problems and the mine was conveyed to Dakota. Operations in the District ceased after the 1997 season, when Dakota merged with USMX Inc. Rapidly falling gold prices in 1997, internal company financial problems, increasing environmental and regulatory issues, and delays in obtaining necessary operating permits led to the mine closure.

6.1 OWNERSHIP AND ROYALTIES

In 1990, during the course of these operations, six lode claims and 30 mill site claims (including mineral rights) were patented with ownership going to the estate of J.J. Oberbillig. These Oberbillig estate patented lands and the 5% Net Smelter Return (**NSR**) royalty interest on the Bradley Estate held by the Oberbillig estate were purchased by US subsidiaries of Midas Gold via exercise and satisfaction of an option to purchase and mortgage agreement in 2015.

Both the property and royalty mortgage notes were paid off and the subject lands and royalty became the property of Midas Gold's US subsidiaries.

On June 2, 2003, Vista's wholly owned subsidiary Vista Gold US Inc. (**Vista US**) entered into an Option to Purchase Agreement with Bradley regarding 17 patented lode mining claims owned by Bradley that covered the majority of the Yellow Pine Deposit. In addition, Vista, through its wholly owned affiliate, Idaho Gold Resources, LLC (**IGR**), acquired eight unpatented lode mining claims, also in the Yellow Pine Deposit area. On February 22, 2011, Midas Gold Inc. (**MGI**) entered into a combination agreement with Vista US and IGR whereby these entities became wholly owned subsidiaries of Midas Gold Corp. Midas Gold's US subsidiaries made final payment under the Option to Purchase on November 28, 2012 and now hold the title to these claims.

In 2006, much of the western portion of the District was staked by Niagara Mining and Development, a subsidiary of Gold Crest Mines Inc. (**Gold Crest**). These unpatented claims surround the patented lands of the former Bradley and Oberbillig estates. Additional, unpatented claims were staked by Gold Crest in 2007 covering the eastern portions of the District. All Gold Crest claims were purchased by MGI in 2009, and agreements were negotiated with the patented landowners.

On April 28, 2011, MGI's wholly owned subsidiary, MGI Acquisition Corp. (**MGIAC**), entered into an agreement with the owners of the six Fern patented mineral claims and now owns those rights.

On May 1, 2011, MGI's wholly owned subsidiary, MGIAC, entered into an option agreement with JJO, LLC, the owners of a number of patented and unpatented mineral claims comprising the former Cinnabar Mine property. JJO, LLC is a limited liability company and the personal representative of the estate of J.J. Oberbillig. The agreement granted MGIAC the right, but not the obligation, to acquire these claims over a period extending to May 1, 2017 in exchange for certain payments. MGIAC (now Stibnite Gold Company) made all payments required under the option agreement. The agreement was subsequently modified and renegotiated on December 1, 2016 and included granting of certain easements and a right of first refusal for purchase of certain surface rights within the District held by Hecla Mining Company. The option agreement remains in effect and is in good standing.

MGI subsequently completed staking of additional claims and modified its land position on several occasions between 2009 and 2015. Midas Gold Corp.'s U.S. subsidiaries were reorganized in 2015 and ownership is described in more detail in Section 4.

The entire property (excluding the Cinnabar group of claims) is subject to a 1.7% gold-only NSR royalty held by Franco-Nevada Corporation as of May 9, 2013. The older Oberbillig royalty was extinguished by purchase in 2015.

6.2 PAST EXPLORATION AND DEVELOPMENT

There have been two major periods of exploration, development and operations in the District prior to Midas Gold activity, one spanning from the early 1900s through the 1950s and another during the period from the early 1970s through the mid-1990s. These activities that occurred over the past century have left behind substantial environmental impacts that remain to this day. The history of development and mining in the District is summarized in numerous publications and additional references therein including: Larsen and Livingston (1920); Schrader and Ross (1926); White (1940); Cooper (1951); Hart (1979); Waite (1996); and Mitchell (1995; 2000) and various unpublished reports and documents. Much of the information contained in the text below is taken from these published sources and from unpublished company records.

The mining history of the region began in 1894 when the Caswell brothers began a sluice box operation in Monumental Creek in what is now known as the Thunder Mountain Mining District, located east of Stibnite. By 1902, a gold rush was underway at the Thunder Mountain District, along with associated development of roads and creation of the town of Roosevelt. By 1909, the gold rush was essentially over; that spring, a mudslide blocked Monument Creek creating present-day Roosevelt Lake and submerging the town of Roosevelt. During the Thunder Mountain gold rush, many prospectors passed through the area now known as the Stibnite-Yellow Pine District, discovering mercury, antimony, silver and gold. However, no work of any significance was completed until around 1917, when the World War I demand for mercury led to the development of several properties east of the main Project area, including the Hermes group of claims located by Pringle Smith in 1902, and the Fern group located by E. H. VanMeter in 1917 (Larsen and Livingston, 1920; Schrader and Ross, 1926).

The first period of large-scale development commenced in the mid-1920s and continued into the 1950s; it involved the mining of gold, silver, antimony, and tungsten mineralized materials by both underground and, later, open pit mining methods. During World War II, this District is estimated to have produced more than 90% of the Nation's antimony and approximately 50% of the Nation's tungsten; materials that were used in munitions, steelmaking, fire retardants and for other purposes. Mining of these strategic minerals was considered so critical that the federal government subsidized the mining activity, managed site operations, and allowed military time to be served at the mine site. Strategic metal mining operations at Stibnite continued through much of the Korean War. Antimony-gold-tungsten mining and milling ceased in 1952, near the end of the Korean War.

The second period of major activity in the District started with exploration activities in 1974 and was followed by open pit mining and seasonal on-off heap leaching and one-time heap leaching from 1982 to 1997, with ore provided by multiple operators from a number of locations and processed in adjacent heap leaching facilities.

Between these periods of development, numerous prospects were discovered and explored using soil sampling, rock sampling, trenching, drilling, geophysical methods and geology. Several of these prospects were developed into successful mining operations. Production records for these operations are discussed in Section 6.4. The history of exploration and development of the major deposits is discussed below and the major exploration activities by past operators and Midas Gold are summarized in Section 9.

The mining, milling and processing activities created numerous legacy impacts including underground mine workings, multiple open pits, development rock dumps, tailings deposits, heap leach pads, spent heap leach ore piles, a mill and smelter site, three town sites, camp sites, a ruptured water dam (with its associated erosion and downstream sedimentation), haul roads, an abandoned water diversion tunnel, an airstrip and other disturbances. Extensive forest fires have compounded the human-created impacts and have increased soil erosion and impacted water quality. Both the main stem of Meadow Creek and its East Fork tributary have been severely impacted by past mining activity. The East Fork of Meadow Creek, locally known as "Blowout Creek", is today one of the largest sources of sediment for this part of the Salmon River. "Blowout Creek" got its name from a water dam that failed in the 1960s with a washout that scarified an erosional channel and drained the meadow and the productive wetlands above. The erosional and dewatering effects continue today, with sediment being rushed downstream with every spring melt and every summer rainstorm, the finer sediments choking the spawning grounds of the EFSFSR. The EFSFSR, a branch of the Salmon River headwaters, currently runs though the old Yellow Pine pit (sometimes referred to locally as the "Glory Hole"). First mined in the late 1930s and abandoned in the late 1950s, the pit has since filled with river water and sediment and formed a lake. While recreationists currently camp on the old mine benches within the open pit and catch fish in the un-reclaimed pit lake, anadromous and local fish populations have not been able to migrate upstream from this point since 1938. Midas Gold (2016) provides more information (Chapter 4) on past mining and mining related legacy issues for further details.

6.2.1 Hangar Flats Deposit

Gold and antimony mineralization were discovered in what is now called the Hangar Flats area around 1900. Albert Hennessy staked the first claims here in 1914. Initial prospecting and development attempts focused on outcropping gold-silver-antimony mineralization, principally in the Meadow Creek area. By the mid-1920s, Albert Hennessy and his partners, who included J.J. Oberbillig, had established the Meadow Creek Silver Mines Company (**MCSM**) and had carried out intermittent, but considerable underground development work on what became known as the Meadow Creek Mine. Homestake Mining Company (**Homestake**) optioned the property and conducted sampling and metallurgical investigations during this period but decided not to complete a purchase of the property after initial metallurgical investigations indicated that they were unable to process the complex gold-antimony ores (Mitchell, 2000). In 1921, MCSM was superseded by United Mercury Mines and, by the mid-1920s, the Meadow Creek Mine area was consolidated under Bradley interests, and the mine was systematically explored and developed on six levels with numerous drifts, crosscuts, raises, winzes, and stopes. It subsequently produced gold, silver, and antimony from sulfide ores, which were milled on site from 1928 through 1938. Mine workings were systematically mapped and sampled, and exploration drilling (from both the surface and underground) was carried out to guide the mine development. About 25,426 ft of underground workings were developed in the Meadow Creek Mine, while substantial additional drilling was completed during this period (for details of drilling during this time period reference Section 10 of this Report). The Meadow Creek Mine produced gold, silver, and significant quantities of antimony between 1928 and 1937. Photograph 6-1 shows the processing facility and tailings pond for the Meadow Creek Mine during this time period. Most of the historical underground maps, tunnel assays, drill logs, and drill assay results can be found in Midas Gold's files or the Idaho Geological Survey archives.

Photograph 6-1: Bradley Mining Company Processing Plant and Tailings Pond

Source: Photograph circa 1942, courtesy of Robin McRae

In 1937, the Meadow Creek Mine was shut down and production shifted to development of the Yellow Pine Deposit in 1938. Beginning in 1943, a mostly unsuccessful attempt was made to re-open portions of the old Meadow Creek Mine workings to explore for antimony and tungsten in support of the war effort. From 1943 to 1945 additional core drilling was completed in the mine, after operations had ceased. A small amount of tungsten mineralized material was reportedly mined during this period from two levels of the mine that were not caved or flooded (Cooper, 1951).

From 1951 through 1954, the Defense Minerals Exploration Administration (**DMEA**) carried out an underground exploration program immediately north of the Meadow Creek Mine. The impetus for that work was provided by the Defense Production Act of 1950 (cf. 15 CFR §§700 to 700.93). It provided monetary assistance for companies to locate new reserves of strategic and critical minerals (Mitchell, 2000). If mineralized material was discovered, the companies that received assistance were required to reimburse the government from the proceeds of the operation. If no economic mineralization was discovered, the government loans were forgiven. Through the DMEA program, Bradley developed approximately 4,900 ft of underground workings on three levels (Mitchell, 2000) in the area immediately north of the Hangar Flats Deposit. Systematic mapping and sampling of the workings were carried out with the mining of bulk samples that were collected at roughly 5 to 10 ft intervals. Drilling of 27 core holes totalling 13,488 feet from underground stations was also carried out. Detailed drill logs and systematic assaying were well documented.

In the late 1970s, Ranchers leased property interests in the District from Bradley and completed a large soil grid over the trace of the Meadow Creek Fault system, including the area adjacent to the old Meadow Creek Mine. Ranchers' work outlined a number of large gold-in-soil anomalies over the old mine site, along the trace of the Meadow Creek Fault system, and north several kilometres to the Yellow Pine Deposit. Ranchers completed some trenching, but no drilling on the anomalies in this area; instead they focused their work on the Yellow Pine and Homestake deposits (Mitchell, 2000).

In the late 1980s, Hecla acquired Ranchers' interests and conducted trenching and ground geophysical surveys, as well as drilling 27 shallow reverse circulation (**RC**) holes in the area of the historical Meadow Creek Mine. Their trenching and RC drilling outlined a broad, but ill-defined zone of gold mineralization above the old workings and along strike to the north, as well as under the old Meadow Creek mill and smelter complex along the base of the hill (where the old Meadow Creek adits were located). Subsequently, Hecla constructed a heap-leach pad over a portion of the main mineralized area due to the need to find a location to leach the oxide ores from the Homestake area of the Yellow Pine Deposit. No further work, other than reclamation of the heap by Hecla and the mill and smelter by government agencies, occurred until Midas Gold's work was initiated in 2009.

6.2.2 Yellow Pine Deposit

The first claims were staked in the Yellow Pine Deposit by prospector Al Hennessy in 1923 who, with J. L. Niday, formed the Great Northern Mines Company. In 1929, the claims were optioned to F. W. Bradley's Yellow Pine Mining Company which drove the Monday and Cinnabar tunnels on opposing sides of the valley. In 1933, these claims were sold to J.J. Oberbillig. By 1938, when the Meadow Creek Mine was shut down, exploration, development, and production shifted to the Yellow Pine Deposit (Mitchell, 2000). A substantial amount of drilling in this area was completed by numerous operators from the late 1930s through the 1990s.

Between 1933 and 1952, Bradley and the United States Bureau of Mines (**USBM**) completed systematic exploration and development drilling in the Yellow Pine and Homestake areas in several drilling campaigns. These drilling programs were spurred on by both the demand for antimony, after the U.S. Government declared antimony a strategic metal (The Strategic Minerals Act of 1939), and the discovery of significant tungsten by U.S. Geological Survey (**USGS**) geologist Donald E. White who was studying USBM drill core from the district in 1941. Subsequent exploration and development included both underground and open pit exploration and development drilling, mapping, sampling and mining. Photograph 6-2 shows the Yellow Pine Open Pit in the early 1950s. During the World War II era, the Yellow Pine Mine was the major source of antimony and tungsten for the war effort and exploration during this period was focused on those commodities (Mitchell, 2000).

After operations shut down in 1952, little work was completed until the 1970s, when Ranchers and, later, its successor Hecla conducted extensive drilling campaigns on the deposit starting in the 1970s and continuing through the mid-1990s along with trenching, pit mapping, engineering, and environmental and metallurgical studies. Hecla completed a prefeasibility study focused on mining of the Yellow Pine deposit in 1987 (Brackebusch, 1987). Barrick optioned the property in the 1995 in a joint venture with Hecla and completed additional drilling and metallurgical test work before dropping the option. Hecla relinquished its control of the property back to the Bradley estate interests after closure and reclamation of the oxide operations at the Homestake pit in the late 1990s (Mitchell, 2000). Vista completed an independent mineral resource estimate prepared by Pincock, Allen and Holt (**PAH**) (2003) and a Preliminary Assessment (Pincock, Allen and Holt, 2006) but conducted no work on site in support of these reports. No additional exploration or development work was completed until MGI acquired their interests in a plan of arrangement between IGR and Midas Gold in 2011.

Photograph 6-2: Bradley Mining Company Open Pit Mine

Source: Photograph circa 1942, courtesy of J. Nock Family Collection

6.2.3 West End Deposit

Gold mineralization was first discovered along the West End Fault by Bradley interests in the late 1930s working with USBM staff conducting strategic minerals investigations; during this time Bradley's exploration focused on replacement of reserves at their Yellow Pine mining operation. Subsequent work by the USGS outlined a large multi-element soil anomaly (Leonard, 1973) that led to systematic follow-up by Superior and its successors. A modern era of exploration and development stretched from the mid-1970s to the late-1990s, prompted primarily by the rise in gold prices and the development heap-leach oxide gold recovery methods (Mitchell, 2000).

Superior conducted geological, geophysical, and geochemical investigations from 1974 to 1977 to evaluate the potential for heap-leach oxide gold in the West End and adjacent Stibnite deposit (now collectively known as West End). In 1979, Superior Oil Company, Superior's parent company, purchased Superior's outstanding shares and became sole owner of the West End Deposit. After completion of a favorable Feasibility Study and Environmental Impact Statement, five heap-leach pads were constructed, and a 2,000 to 3,000 st/d oxide mining operation began in 1982 (Photograph 6-3). Open pit mining at the West End Mine and heap-leach processing was conducted by Superior until 1984 when ownership of the deposit changed hands when Mobil Oil purchased Superior Oil. The West End mine did not operate in 1985, however heap leach processing of previously mined material continued throughout 1985 (Mitchell, 2000). Photograph 6-4 provides a 2011 oblique photo of the West End pit showing the partial backfill.

Photograph 6-3: Canadian Superior Mining (U.S.) Ltd. Heap Leach Processing Facility

Source: Photograph circa 1985, courtesy of the U.S. Forest Service

In 1986, Pioneer purchased the mine from Mobil with financing assistance from The Mining Finance Corporation and Twin Rivers Minerals who owned 25% of the West End Pit, and 18% of Pioneer's stock (Mitchell, 2000). At this time, Pioneer became the operator of the West End mine and continued to explore and produce until 1991. From 1991, ownership of the West End open pit mine and processing facilities changed hands from Pioneer to Pegasus Gold Corporation (**Pegasus**), and then to MinVen (later changed to Dakota). During this time the mining and exploration activities in the area continued under MinVen's (later Dakota's) subsidiary company, SMI. SMI continued to conduct sporadic drilling and development of the West End pit, including a small area on the east side of the West End Deposit known as the Stibnite pit, and a small pit approximately 1.5 miles to the south east known as the Garnet Pit, into the late-1990s. Between 1982 and 1997 crushed oxide material from the West End pits was placed in the Upper Meadow Creek Valley after being leached, neutralized, and rinsed (Mitchell, 2000) in an area now commonly referred to as the Spent Ore Disposal Area (**SODA**). For estimated production records during this time period see Table 6.3. Some spent ore was also used during reclamation as backfill in the Garnet pit and on some former access and haul roads during 1998 - 2000 reclamation by state and federal agencies.

Photograph 6-4: West End Pit showing Partial Backfill

Source: Photograph circa 2011, Midas Gold collection

6.3 HISTORICAL MINERAL RESOURCE AND RESERVE ESTIMATES

Through the years, various companies have completed mineral resource estimates of all or portions of the Meadow Creek Mine (now called Hangar Flats), West End, and Yellow Pine/Homestake deposits using different gold prices, cut-off grades, estimation methods, and datasets. These include multiple estimates by Ranchers, Hecla, Santa Fe Pacific Gold Corporation, Newmont Mining Corporation, and Barrick. Since these estimates were completed prior to 1998 and were not prepared in accordance with the requirements of Sections 1.2 and 1.3 of NI 43-101, there are no historical Mineral Resource or Mineral Reserve estimates that compare with the Mineral Resource and Reserve estimates in this Report. Historical data files contain various estimations of oxide and sulfide mineralized material consisting of individual mineralized lenses within the Hangar Flats, West End, and Yellow Pine deposit areas, but the Mineral Resource estimates and supporting backup data are incomplete or were for only small portions of larger deposits and are, therefore, not pertinent and are not reported here.

In 2003, Vista contracted with Pincock, Allen and Holt to complete an NI 43-101-compliant Mineral Resource estimate and Technical Report (Pincock, Allen, and Holt, 2003) on the Yellow Pine Deposit. This report was completed prior to any drilling by Vista or Midas Gold and has since been determined to be obsolete. The reader is referred to this report on the Canadian Securities Administrator's (**CSA**) System for Electronic Document Analysis and Retrieval (**SEDAR**) for details of the PAH resource estimation procedures and results.

Midas Gold has completed several Mineral Resource estimates for the Project. These include a maiden Mineral Resource estimate for the Hangar Flats, West End and Yellow Pine deposits (SRK, 2011), followed by updated Mineral

Resource estimates described in the PEA (SRK, 2012), the Stibnite Gold Project Prefeasibility Study (**PFS**) (M3, 2014), and an updated Mineral Resource estimate in 2018 (Midas Gold, 2018). The reader is referred to these reports by the issuer on SEDAR for details on procedures, assumptions, caveats and results from the previous Mineral Resource estimates. Information in this Report supersedes information reported in the PAH report, the SRK 2011 report, the PEA, the PFS and the 2018 Mineral Resource update.

6.4 HISTORICAL PRODUCTION

Historical production figures, because of limited surviving records, are estimates that have been pieced together from several sources. Victoria E. Mitchell of the Idaho Geological Survey (**IGS**) published a detailed report in 2000 titled "History of the Stibnite Mining Area, Valley County Idaho" and much of the history and production numbers used in this Report come from that document. Mitchell's report, however, does not detail all of the production from the three deposits for all of the years that their respective mines operated. As a result, other sources were utilized to fill in the gaps. Sources include public filing reports from the US Securities and Exchange Commission (**SEC**), unpublished company production records, Idaho State Mine Inspection records, and USBM reports. Occasionally, these sources contained conflicting data, in which case the company's production records were utilized. The production figures in many instances are only estimates and are not reported consistently for gold, silver, and antimony. Table 6-1 summarizes production for the Project by area, while additional details are provided in the following sub-sections.

Table 6-1: Stibnite District Estimated Historical Production

Area	Production Years	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (units)(1)
Hangar Flats	1928 - 38	303,853	51,610	181,863	3,758	67[2]
Yellow Pine	1938 - 92	6,493,838	479,517	1,756,928	40,257	856,189[3]
West End[4]	1978 - 97	8,156,942	454,475	149,760	-	-
Totals		**14,954,633**	**985,602**	**2,088,551**	**44,015**	**856,256**

Notes:
1. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.*
2. *The reported tungsten production in 1938 is from the 1943-1944 reopening of the Meadow Creek Mine during Strategic Minerals investigations by USBM.*
3. *Includes minor production from placer plant in late 1940s-early 1950s.*
4. *Includes 1995 production from Garnet pit.*

6.4.1 Hangar Flats Deposit

Gold, silver and antimony were produced from the Hangar Flats Deposit from 1928 to 1938. Based on available compiled records, the totals listed in Table 6.2 provide an approximation of the production from underground operations in the Meadow Creek Mine.

Table 6-2: Hangar Flats Deposit Estimated Production Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (units)[1]
Bradley	1928-31	19,767	Unknown	Unknown	Unknown	-
Bradley	1932	34,366	6,916	18,488	489	-
Bradley	1933	45,710	10,412	29,817	588	-
Bradley	1934	54,000	10,491	25,384	404	-
Bradley	1935	50,965	8,373	25,217	550	-
Bradley	1936	43,324	7,798	32,615	729	-
Bradley	1937	39,521	5,514	36,572	755	-
Bradley	1938	16,200	2,106	13,770	243	67
TOTALS		**303,853**	**51,610**	**181,863**	**3,758**	**67**

Notes:

1. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten. The reported tungsten production in 1938 is from the 1943-1944 reopening of the Meadow Creek Mine during Strategic Minerals investigations by USBM.*

6.4.2 Yellow Pine Deposit

Gold, silver and antimony were produced from the Yellow Pine Deposit starting in 1938, with the addition of tungsten in 1941 with continuous production from 1938 to 1952. Based on available compiled records, the totals listed in Table 6-3 provide an approximation of the production from underground and open pit operations during this time period. Additionally, gold was produced by Hecla from open pit operations (1989 to 1992) in the Homestake Mine, an oxide gold deposit which overlies the northeastern portion of the Yellow Pine Deposit.

Table 6-3: Yellow Pine Deposit Estimated Production

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)[1]
Bradley	1938	22,680	1,423	3,917	136	-
Bradley	1939	56,074	5,810	14,844	228	-
Bradley	1940	132,297	12,401	15,825	18	-
Bradley	1941	95,156	10,355	18,981	380	27,921
Bradley	1942	96,861	2,714	85,161	2,801	181,230
Bradley	1943	178,747	4,529	109,307	2,734	303,502
Bradley	1944	211,382	6,110	74,498	2,031	233,664
Bradley	1945	109,796	6,505	87,815	2,895	85,572
Bradley	1946	147,505	14,276	68,564	1,477	-
Bradley	1947	584,483	44,393	324,582	6,699	-
Bradley	1948	655,682	49,400	318,090	7,948	-
Bradley	1949	610,988	68,423	127,403	2,104	-
Bradley	1950	620,800	61,763	177,594	3,747	5,899
Bradley	1951	546,163	39,242	226,274	4,575	11,220
Bradley	1951[2]	26,355	-	-	-	4,990
Bradley	1952	310,201	24,747	104,073	2,484	2,191
Hecla	1988	278,193	20,701	-	-	-
Hecla	1989	910,475	29,436	-	-	-

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)[1]
Hecla	1990	900,000	57,747	-	-	-
Hecla	1991	Unknown	17,542	-	-	-
Hecla	1992	Unknown	2,000	-	-	-
TOTAL		**6,493,838**	**479,517**	**1,756,928**	**40,257**	**856,189**

Notes:
1. *A unit of WO_3 (tungsten trioxide) is 1% of a short ton (20 pounds), and WO_3 is 79.3% tungsten. A short ton unit of WO_3, therefore, equals 20 pounds of WO_3 and contains 15.86 pounds of tungsten.*
2. *Production from reprocessing of tailings and fluvial gravels through placer plant.*

6.4.3 West End Deposit

Gold and silver were produced from the West End Deposit from 1982 to 1993. Based on public filings, published reports, and unpublished company production records, the totals listed in Table 6-4 provide an approximation of the production from operations in the West End, Splay, Stibnite and Garnet pits, all of which, except Garnet, are located within the West End Deposit.

Table 6-4: West End Deposit Estimated Production Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)
Superior	1978	1,500	60	-	-	-
Superior	1982	200,000	7,832	3,287	-	-
Superior	1983	480,000	29,000	8,207	-	-
Superior	1984	487,295	28,645	8,107	-	-
Superior	1985	-	-	-	-	-
Superior	1986	630,865	45,508	28,719	-	-
Superior	1987	764,121	40,802	25,750	-	-
Pioneer	1988	278,193	32,347	17,418	-	-
Pioneer	1989	910,475	29,436	9,778	-	-
Pioneer	1990	982,240	63,357	9,942	-	-
Pioneer	1991	863,783	31,555	11,008	-	-
Pioneer-Pegasus	1992	950,000	31,549	12,818	-	-
MinVen-Dakota	1993	91,000	2,042	1,330	-	-
SMI	1994	-	-	-	-	-
SMI	1995	300,340	20,949	5,378	-	-
SMI (from Garnet Pit)	1995	300,130	59,190	-	-	-
SMI	1996	927,000	32,203	8,019	-	-
Total		**8,166,942**	**454,475**	**149,760**	**-**	**-**

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7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project area is located in the Salmon River Mountains, a high-relief mountainous physiographic province in central Idaho. Bedrock in the region can be subdivided into several groups based on age, lithology and stratigraphic relationships. In a broad sense, rock sequences in the region can be subdivided into those that are part of the pre-Cretaceous metasedimentary "basement," the Cretaceous Idaho Batholith, Tertiary intrusions and volcanics, and Quaternary unconsolidated sediments and glacial materials. The SGP is situated along the eastern edge of the Idaho Batholith, on the western edge of the Thunder Mountain caldera complex and within the Central Idaho Mineral Belt (Figure 7-1).

The pre-Cretaceous basement rocks were deposited during a protracted rifting event from Neoproterozoic through middle Paleozoic time and record the development and subsequent tectonic overprinting of the western Laurentian continental passive margin. Metamorphosed rift and passive margin sedimentary sequences are exposed in a broad northwesterly trending belt extending from southeast Idaho into northeast Washington and beyond (Lund et al., 2003; Lewis et al., 2012). The metasediments occur within stacked thrust sheets to the southeast, adjacent to the Idaho Batholith or as discontinuous roof pendants in central Idaho. The stratigraphic succession, although dismembered and isolated from adjacent areas having more continuous exposure, is similar to often mineralized Paleozoic miogeoclinal rocks of southeast Idaho and northern Nevada. In the Stibnite area, the sequence contains three unconformity-bound carbonate cycles. All the sedimentary rocks are strongly recrystallized and regionally metamorphosed. The metasediments and batholith rocks are cut by numerous regional-scale fault systems that trend north-south and northeast-southwest and vary in age (Figure 7-2).

These rocks likely correlate with the Mesoproterozoic Belt Supergroup, the Neoproterozoic Windermere Supergroup and the Neoproterozoic to lower to middle Paleozoic passive margin miogeoclinal successions (Lund et al., 2003; Lewis et al., 2012) found along strike regionally. Subsequent metamorphism, structural complexity and preservation of only small erosional remnants of these sequences make an accurate measurement of original thicknesses, stratigraphic associations and original facies relationships difficult. However, new district- and regional-scale mapping and isotopic dating work conducted by the Idaho Geological Survey (IGS), U.S. Geological Survey (USGS), and various academic partners suggests the youngest metasedimentary rocks within the Project area are correlative in part to passive margin slope and shelf rocks exposed in southeast Idaho in the Bayhorse region (Figure 7-1) and in the northern Panhandle of Idaho (Lewis et al., 2014; Stewart et al., 2016).

Pre-Cretaceous basement rocks in central Idaho underwent several periods of deformation, including regional folding and faulting in the early Paleozoic followed by extensive early Mesozoic folding and west to east thrust faulting associated with the Cretaceous-Tertiary Sevier and Laramide orogenies, (Lund et al., 2003). Each subsequent orogenic event resulted in eastward contractional deformation of the miogeoclinal sequence and underlying, older rift-related units. In Idaho, these orogenic events are associated with accretion of the Blue Mountains island arc complex to the western margin of North America along the Salmon River Suture Zone, situated west of the Project area (Figure 7-1; Figure 7-2). This suture marks the transition zone between Precambrian continental crust of North American affinity to the east and accreted Paleozoic to Mesozoic oceanic crust and island arc rocks to the west, as defined by various petrologic and geochemical studies, isotopic data and geophysical models (Piccoli and Hyndman, 1985; Kleinkopf, 1988; Lund and Snee, 1988; Strayer et al., 1989). The western margin of the Idaho Batholith is metamorphosed and foliated parallel to the Salmon River Suture Zone, which indicates that it was emplaced while the suture zone was still active (Manduca et al., 1993). The West Idaho Shear Zone is a steeply dipping younger fault system within the broader Salmon River suture and records right-lateral strike-slip displacement with a significant component of shortening perpendicular to the fault system under a transpressional kinematic environment during the Late-Cretaceous (Montz and Kruckenberg, 2017; Braudy et al., 2017).

Figure 7-1: Central Idaho Generalized Geologic Map and Gold Prospects

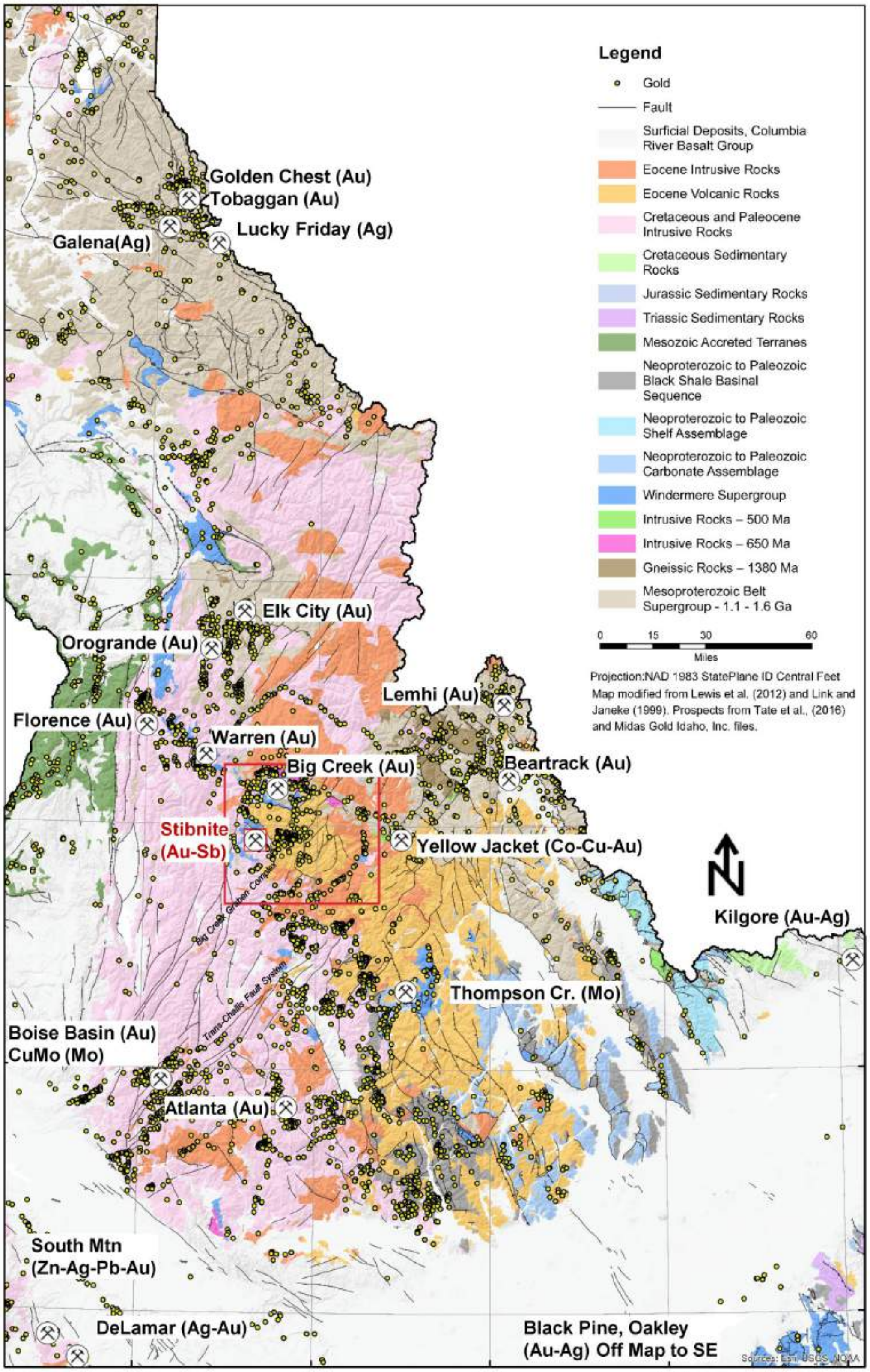
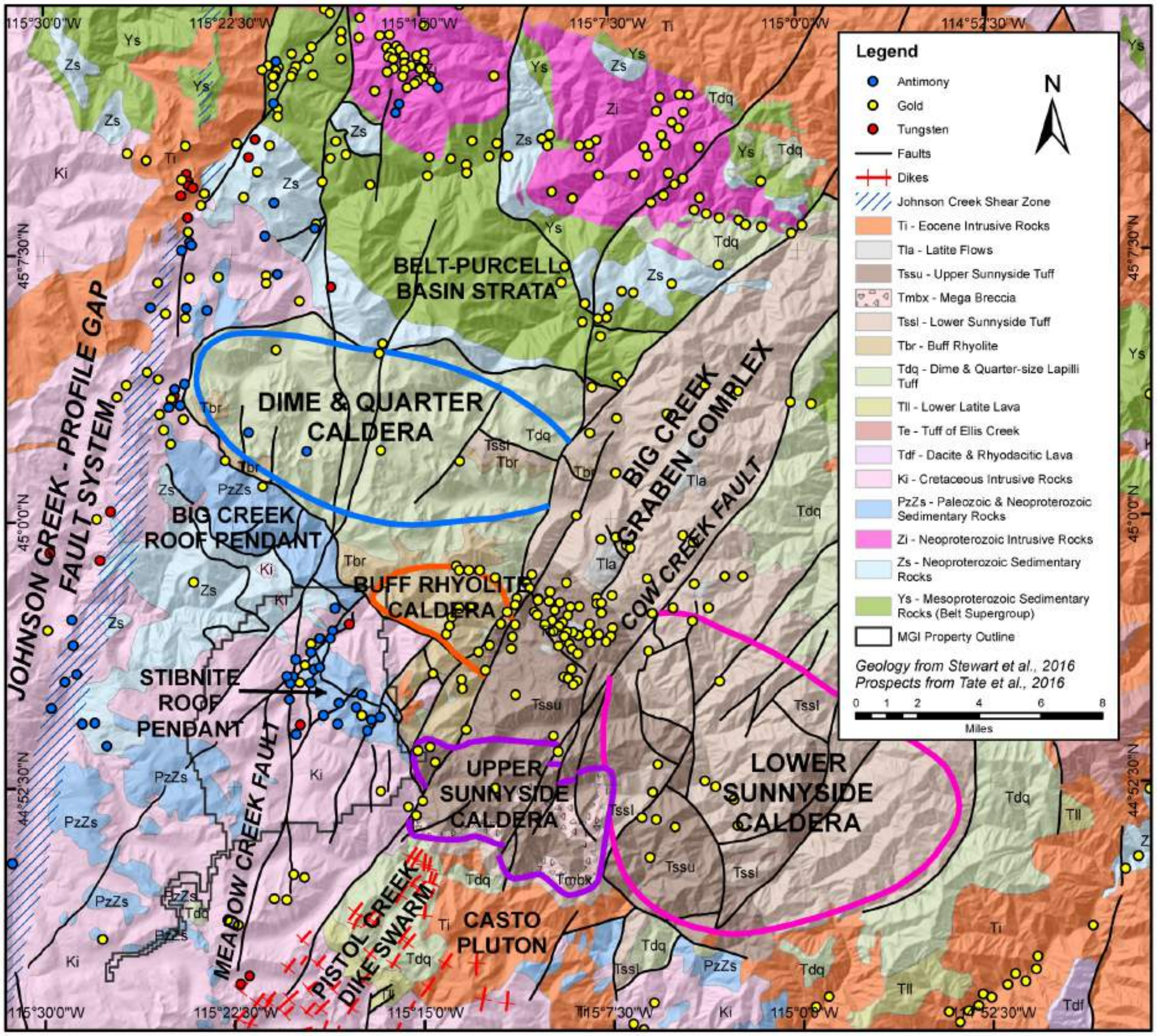
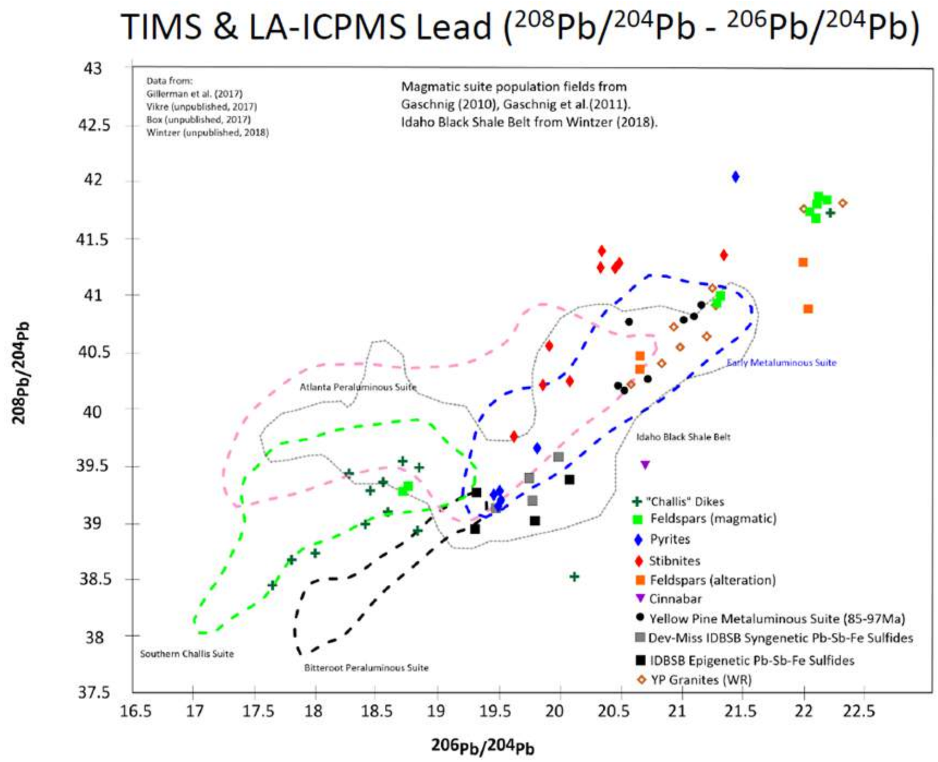


Figure 7-2: Regional Geologic Map and Prospects



The Idaho Batholith, a major feature of the Cordilleran orogen, intruded the sedimentary sequences in the mid--to--late Cretaceous. The batholith formed as a result of continuous magmatism lasting approximately 60 million years (Ma), marked by multiple pulses of igneous activity, each with distinctive compositions, tectonic setting, and geographic distribution. Regionally, the Atlanta Lobe of the Idaho Batholith, in which the SGP is located, shows a progression from early mantle-derived metaluminous magmatism from 98 Ma to 87 Ma, followed by more voluminous and evolved, crustal-contaminated peraluminous magmatism from 83 Ma to 67 Ma. This change in magmatic character is variably attributed to orogenic crustal thickening (Lund, 1999; Gaschnig et al. 2011), crustal contamination of more primitive magmas or magmatic differentiation. The majority of SGP area Cretaceous-age plutonic rocks are peraluminous but were emplaced coevally with the early metaluminous suite of Gaschnig et al., 2011; Wintzer, 2019). Based on common lead isotopes, regional correlations and geophysical data, it is likely that partial melting or assimilation of additional metasedimentary sources, including highly radiogenic Archean crust and basinal Paleozoic rocks, shifted compositions to the peraluminous fields (Gillerman et al., 2019; Wintzer, 2019). Wintzer (2019) compared common lead isotopes of ore minerals and whole-rock chemistry of intrusive rocks from the SGP area to those found along strike to the southwest in the Idaho Black Shale Belt, a basinal sequence of lower to middle Paleozoic metalliferous lithologies suggesting the batholith in this area may have assimilated some of these units and could provide a source of metals and reducing conditions (Figure 7-3).

Figure 7-3: Common Lead Isotope Signatures of SGP Ores and Rocks vs the Idaho Black Shale Belt



The regional tectonic regime was compressional during Cretaceous batholith and pluton emplacement through the early Tertiary during uplift as the batholith was unroofed. An extensional setting was present during Tertiary dike and epizonal pluton emplacement in the region (Yonkee and Weil, 2015). Eocene intrusions related to the Challis Volcanic Field are common near the eastern margin of the Atlanta Lobe of the Batholith and include dikes, dike swarms, and stocks (Bennett and Knowles, 1985). The intrusions generally are porphyritic in texture and intermediate to felsic in composition. Many of these plutons contain disseminated molybdenum, tungsten and tin mineralization (Bennett, 1980) and also are associated with distinctive isotopic signatures indicative of the mixing of meteoric water with magmatic fluids in large hydrothermal systems (Criss and Taylor, 1983). This is consistent with the presence of miarolitic cavities, normative mineralogy and experimental modeling (Bennett, 1980; Rehn and Lund, 1981). A large Tertiary intrusive complex known as the Casto Pluton is located east of the SGP and phases of this complex may extend beneath the SGP area (Anderson et al., in preparation). The isotopic signatures of this pluton are interpreted to be the result of assimilation of significant quantities of hydrothermally altered wall rocks in the magma under highly reducing conditions (Criss et al., 1984; Larson and Geist, 1995). These younger Challis intrusions and associated volcanics range in age from 51 Ma to 39 Ma and were derived from both crustal and mantle sources. The Thunder Mountain Caldera Complex of the Challis Volcanic Field lies immediately east of the SGP area and is described by Leonard and Marvin (1982) and Ekren (1985). It consists of predominantly felsic volcanic, pyroclastic, and epiclastic rocks that were erupted and deposited in subaerial and lacustrine environments. There were significant gold-silver mining operations in the Thunder Mountain area from the 1920s through the early 1980s within the volcanic complex.

Brittle faulting in west-central Idaho has occurred throughout the Cenozoic and includes Eocene extension and later Miocene Basin and Range normal faulting (Figure 7-1). This extension, to some extent, reversed the effects of the earlier compressional events leading to the development of reactivated thrust faults as low angle extensional faults, core complexes and in a general way placed the basinal rift and passive margin strata back in the original alignments with basinal rocks to the west and shallow water strata to the east. Approximately 10 miles to the west of the Project area, the mile-wide, 80-mile-long north-south trending Johnson Creek-Profile Gap Shear Zone is marked by dike swarms, heavy fracturing, multi-stage brecciation and pervasive alteration, and shows evidence of both Cretaceous and Tertiary intrusive and tectonic activity. The Meadow Creek Fault Zone (**MCFZ**), parallel to and approximately 10 miles east of the Johnson Creek Profile Gap structure, is situated along the west side of the Thunder Mountain Caldera, and can be traced for over 10 miles in a north-south-direction and has similar orientation and kinematic indicators to the Johnson Creek Profile Gap structure. Mineralization styles in prospects in the Johnson Creek – Profile Gap Fault System show similarities as well. A northeast-trending Tertiary graben complex named the Big Creek Graben (Stewart et al., 2016) is located 5 miles southeast of the Project area and cuts Eocene intrusive rocks. Numerous epithermal mines and prospects and shallowly emplaced Tertiary dike swarms are associated with this feature, including the Thunder Mountain District and Pistol Creek District mines. This structure is of a similar scale and is parallel to the well-studied Trans-Challis fault system located to the south. Much of the regional Mesozoic contraction may have been reversed during later younger extensional tectonic activity (Skipp, 1985; Janecke et al., 1993; Janecke et al., 1997). To the west of the Project area, evidence of widespread extensional deformation is concentrated in the Long Valley fault, which has resulted in the development of the Long Valley basin and West Mountain escarpment near the towns of New Meadows, McCall, Donnelly and Cascade. Within the project area, north-south, north-east and north-west striking brittle fault systems reflect regional structural trends.

Pleistocene-age valley glaciers created U-shaped valleys with over-steepened, talus-covered sides, and hanging valley tributaries with cirques and tarns in their upper reaches. U-shaped valleys also have lateral, terminal, and recessional moraines, remnants of moraine-dammed lakes, and glacial outwash deposits at their lower ends. Broadly glaciated areas have rounded hills with glacially scraped and scoured up-glacier slopes and ground -moraine covered down-glacier slopes. Modern Holocene-age stream drainage patterns indicate high rates of erosion and have deposited coarse-grained fluvial sedimentary deposits in floodplains often composed of a mixture of angular clasts from adjacent bedrock sources combined with more rounded reworked glacial deposits.

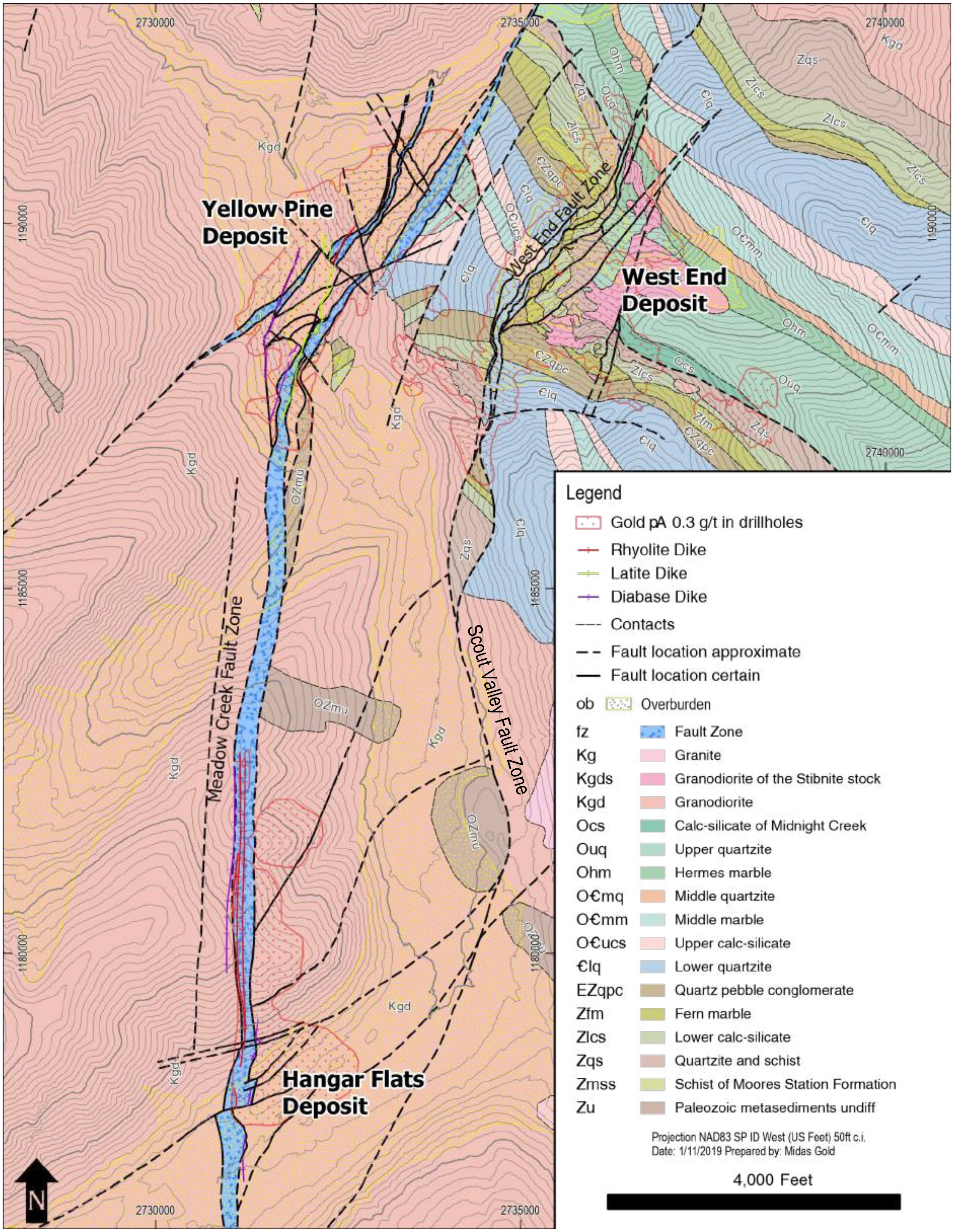
7.2 LOCAL GEOLOGY

7.2.1 Lithology

The Hangar Flats Deposit is hosted by Cretaceous intrusive phases of the Idaho Batholith. The West End Deposit is hosted primarily by metasedimentary rocks of the Stibnite roof pendant, but also by intrusive phases. The Yellow Pine Deposit is hosted primarily by intrusive phases of the Idaho Batholith but also by metasedimentary rocks. Other post-mineralization intrusive igneous rocks associated with the Challis Volcanics occur within the Yellow Pine, Hangar Flats, and West End deposits. Figure 7-4 illustrates the various lithologic units located within the Stibnite-Yellow Pine District (**the District**).

Numerous workers have described the stratigraphy and lithologic characteristics of the intrusive, metasedimentary, volcanic, and unconsolidated rocks exposed in the Project area including Larsen and Livingston (1920); Schrader and Ross (1926); Currier (1935); White (1940); Cooper (1951); and Smitherman (1985). The descriptions that follow are derived from these sources as well as from unpublished petrographic studies by past operators and Midas Gold.

Figure 7-4: Bedrock Geology of the West Side of the Stibnite Mining District



7.2.1.1 Igneous Rocks

Igneous rocks in the District can be subdivided into two groups, Cretaceous intrusive igneous rocks associated with the Atlanta lobe of the Idaho Batholith and Tertiary dikes associated with Eocene magmatism and the Thunder Mountain Caldera Complex. Cretaceous intrusive rocks have been broken down into four main phases (Gillerman et al., 2019; Wintzer, 2019; Lewis et al., in press 2020) with pulses at approximately 98-95 Ma, 88Ma, 84Ma, and 82 Ma, but additional dating would likely show even more complexity to the Cretaceous intrusive history.

<u>Quartz Monzonite</u>

The dominant type of intrusive rock exposed in the District and intersected in drilling consists of light to medium gray, equigranular, medium- to coarse-grained granodiorite with metaluminous to peraluminous compositions. For consistency with historical reports and drill logs, the granodiorites are generally referred to as quartz monzonites. When unweathered and unaltered the quartz monzonite typically consists of approximately 25-30% quartz, 50-60% feldspar (mostly calcic oligoclase and the remainder microcline and orthoclase), and 5-10% biotite. Hornblende and other mafic minerals are rare. Accessory minerals include muscovite, chlorite, apatite, sphene, and various carbonates and clay minerals. The unaltered quartz monzonite weathers to a white to light gray-colored, chalky textured grus with rusty orange discoloration due to weathering and oxidation of biotite. Locally the biotites may show a weak alignment and the rock may be coarsely porphyritic with large feldspar phenocrysts. Recent dating of zircons via LA-ICPM, SHRIMP, and ID-TIMS (Gillerman, 2017; Gillerman et al., 2019; Wintzer, 2019) gives an average age of approximately 89.2 Ma from granodiorites from the district with some dates as old as 95 Ma. Geochemical data from the granodiorites and quartz monzonites indicate that they are metaluminous in composition and the more felsic and younger intrusions have more peraluminous compositions (Wintzer, 2019).

<u>Alaskite-Leucogranite</u>

Leucogranites are widespread in the District and occur as dikes, sills, and segregations cross cutting the quartz-monzonite and range in width from less than 1 inch to over 30 ft. For consistency with historical reports and drill logs, the leucogranites are referred to as alaskites. The alaskites are siliceous, are typically fine-grained, sucrosic textured, and can be distinguished from the quartz monzonite by the lack of biotite or other mafic minerals. The alaskite dikes can be coarsely crystalline to pegmatitic locally. Core logging and field observations indicate multiple phases of aplite dikes cutting the quartz monzonites and granites and isotopic dating shows a spread of ages consistent with leucogranite phases associated with each of the major plutonic suites. The dikes may contain minor fine-grained disseminated euhedral magnetite and occasionally medium-grained euhedral arsenopyrite and often garnet. The alaskites typically occur as narrow 8- to 20-inch wide dikes in swarms that may range in overall width from a few feet to tens of feet across. Recent dating of zircons via LA-ICPM, SHRIMP and ID-TIMS (Stewart el al., 2016; Box et al., 2016; Gillerman, 2017; Gillerman et al., 2019; Wintzer, 2019) gives an average age of approximately 86.2 Ma from leucogranites from the district. This is consistent with field observations of the leucogranites occurrence in swarms or large stockworks in the cupolas of larger granite bodies. Petrochemistry data also suggest that the leucogranites represent late-stage differentiates of the granites.

<u>Pegmatite</u>

Pegmatite dikes are coarsely crystalline consisting of large euhedral grains of interlocking potassium feldspar and quartz. The pegmatite dikes range in width from 2 inches to more than 10 ft. Early pegmatite dikes cut through the quartz monzonite, but alaskite dikes have also been observed cutting through the early pegmatite dikes. Later pegmatite dikes cut through alaskite dikes.

Biotite Granite

Biotite granite is exposed in several areas in the District and a large northeast-trending body is exposed and intersected in drill holes between the West End Deposit and the Stibnite Pit and has been informally named the Stibnite Stock. The biotite granite is typically fine- to medium-grained, equigranular with large black to dark brown biotite, and contains traces of hornblende, zircon, and apatite as accessories. Muscovite is present but in smaller quantities than biotite. The biotite granite crosscuts both the main quartz monzonite body of the batholith and the metasedimentary sequence. Isotopic dating of zircons from outcrops of the Stibnite Stock in the Stibnite Pit produced a late Cretaceous age of 84.9 Ma ± 2.0 Ma; a more precise age was recently reported from drill core in hole MGI-10-37 at 50ft., producing a concordant age of 85.7 ± 0.1 Ma (Gillerman et al., 2014). Clasts of the biotite granite occur in mineralized potassium feldspar and quartz-cemented breccias in the West End Deposit suggesting mineralization at least locally postdates the stock and is consistent with $^{40}Ar/^{39}Ar$ isotopic dating of hydrothermal potassium feldspar (adularia) selvages on quartz veins cutting the Stibnite Stock; the feldspar was dated at 50 Ma ± 0.4 Ma (Gillerman et al., 2014).

Granite

Granites are common throughout the District and occur as small stock-like bodies and, based on cross-cutting relationships and isotopic dating, appear to be younger than the quartz monzonites and granodiorites that volumetrically make up the bulk of the Idaho Batholith in the Project area. Distinct granite bodies are known to be present along the Meadow Creek Fault Zone at the Hangar Flats and Yellow Pine deposits and in the northern DMEA, Rabbit, and Prometheus prospect areas. A large body of granite is exposed in the southwestern portion of the former Yellow Pine open pit and underlies the western portions of the Yellow Pine Deposit at depth. Recent dating of zircons via LA-ICPM, SHRIMP, and ID-TIMS (Stewart et al., 2016; Gillerman, 2017; Gillerman et al., 2019; Wintzer, 2019) gives an average age of approximately 86 Ma from granites from the district. Zircons have been dated in drill core (MGI-12-306, 926-956 ft) at 86 ± 5.4 Ma (Gillerman et al., 2014) and 84.1 ± 1.1 Ma (Box et al., 2016). The granites are phaneritic, fine- to medium-grained, equigranular, and typically light gray to white. Principal components include feldspar, quartz, and fine-grained mica.

Diorite

Diorite has been cut in several drill holes in the district at Yellow Pine, Hangar Flats, and near Scout and is exposed in the area around the Rabbit prospect. The diorites are fine- to medium-grained and are often weakly magnetic due to the presence of magnetite and/or pyrrhotite. Diorite clasts are observed as inclusions within quartz monzonite. Primary mineralogy is plagioclase with equal parts amphibole and biotite (approximately 20% each) and very rarely quartz. Much of the amphibole may be an alteration product of pyroxene. Calcite or dolomite as well as magnetite occur as accessories. Trace amounts of sphene have also been observed within this lithology, likely as an alteration product. No isotopic dates have yet been determined for the diorites.

Rhyolite

The general sequence of Tertiary dike ages based on limited field observations of cross-cutting relationships and dating, although potentially imprecise, suggests the rhyolites are likely the oldest dikes, the latite dike suite intermediate in age, and diabase dikes the youngest. Wintzer (2019) reported analyses from laser ablation uranium-lead (U-Pb) dates on zircons from dike samples from the Yellow Pine pit and drill core with ages ranging from 29-50 Ma consistent with other data showing Eocene to Oligocene ages. Several rhyolite dikes are found within the district and are associated with the MCFZ. On the eastern side of the district, they occur adjacent to the margin of the Thunder Mountain caldera. The rhyolites are aphyric to porphyritic and are light- to dark-gray to beige when fresh. The rhyolite contains sparse sub-inch sized, often resorbed, quartz and feldspar phenocrysts within an aphanitic, often partially devitrified, groundmass. Rhyolite dikes are up to 40 ft wide and are often sheared or strongly broken when they are located within fault zones. Xenoliths of mineralized quartz monzonite within the rhyolite have been observed in drill core and rhyolites likely were emplaced after the main pulses of mineralization. Both pyrite and stibnite have been observed in the rhyolites in small vugs and cavities suggesting remobilization of metals during emplacement. Based on similarities to dated rhyolites elsewhere in the area, these rhyolites are considered Tertiary in age.

Latite and Trachyte Porphyries

Porphyritic dikes are variable but typically latite and trachyte in composition, are common in faults throughout the district, and occur as small plugs and sills in the eastern part of the Project area. The Pistol Creek Dike Swarm, located just southeast of the District, and the Smith Creek Dike Swarm in Big Creek are both large regional-scale dike swarms of similar texture, mineralogy, and composition and likely are of similar age. The dikes are light greenish-gray when fresh, weather to an olive-green to orange-gray color, and often make a sticky, clay-rich soil likely due to alteration of devitrified glasses. Phenocrysts of sanidine, andesine, biotite, and rare quartz are set in a groundmass of fine-grained feldspar ± fine-grained biotite. These dikes cross-cut the quartz monzonite and the granites and have been observed cutting the rhyolite dikes. A latite dike sampled by the IGS from within the Yellow Pine Deposit produced an $^{40}Ar/^{39}Ar$ age of 45.9 ± 0.3 Ma (Gillerman et al., 2014). This dike is well exposed in the Yellow Pine Deposit and, although moderately altered, appears to be later than the main pulses of mineralization at Yellow Pine. Fragments of a similar lithology occur as clasts in mineralized breccias within the West End Deposit.

Diabase

Diabase dikes, up to 50 ft wide, often occur within or adjacent to fault zones within the district. Historic literature occasionally noted these as lamprophyres. Based on cross-cutting relationships, the dikes are likely Eocene or younger. They typically are brecciated and heavily fractured when they occur within structures, typically aphanitic to very finely porphyritic in texture, medium- to dark-green when fresh, and contain small partially resorbed grains of pyroxene and hornblende with phenocrysts making up less than 5% of the rock unit within an aphanitic groundmass primarily of plagioclase feldspar. Magnetite is a common accessory and is generally magnetic. Locally, they contain circular to ovoid, calcite-filled amygdules similar in appearance to outcropping Eocene basalt flows associated with the latest stages of Eocene volcanism within the adjacent Thunder Mountain Caldera and to the west in younger Miocene basalt flows in Long Valley. Rarely, xenoliths of rhyolite dike material have been found as fragments within the diabase dikes, indicating that diabase dikes are the youngest rock unit and were emplaced after the main phases of mineralization. However, stibnite has been observed in the diabases in small vugs and cavities along late fractures suggesting remobilization of metals during emplacement.

7.2.1.2 Metasedimentary Rocks

Early workers believed that the metasedimentary rocks in the district of the roof pendant were Proterozoic in age, partly because of their proximity to the Belt sedimentary basin. However, recent work has determined that at least some of the rocks are likely Paleozoic in age. Based on coral and bryozoan fossils, researchers in the early 1980s used biostratigraphy to place the Stibnite metasedimentary package in the Ordovician Period (Lewis and Lewis, 1982). Additional bryozoan fossils were discovered in 2012 by the IGS from the Hermes Marble near Sugar Creek. Detrital zircons recovered by the IGS from within the suite show ages in the Meso- and Neo-Proterozoic (Lewis, et al., 2014).

At least some of the metamorphosed Neoproterozoic through Ordovician stratigraphy found in central Idaho roof pendants near Stibnite and Edwardsburg likely correlate to the units exposed in southeastern Idaho in the Bayhorse and Clayton 7.5' quadrangles (Lund et al., 2003; Stewart et al., 2017; Isakson, 2017). The basal dolomite of Bayhorse Creek, Garden Creek Phyllite, Bayhorse Dolomite, and Ramshorn Slate in the Bayhorse quadrangle likely correlates with the Moores Station Formation and possibly partially with the underlying Edwardsburg Formation. These units likely correlate with the interbedded quartzite and siltite and overlying Clayton Mine Quartzite (Brennan et al., 2020; Krohe et al., 2020). Neoproterozoic schist of Moores Station Formation outcrops about a mile northwest of the District and likely underlies the stratigraphic section at Stibnite and is exposed as small scattered outcrops in drill intercepts in the valley bottom areas of the District (Stewart et al., 2016). The Moores Station Formation is overlain predominantly by quartzites of the Moores Lake and Umbrella Butte Formations, which likely are correlative to at least the lower stratigraphic section at Stibnite. Stewart et al. (2016) correlate the lower quartzite unit of Smitherman (1985) as Cambrian and the upper quartzite unit as Ordovician (Kinnikinic or Eureka equivalent). Stewart et al. (2016) interpreted the Middle Marble of Smitherman as Cambrian or Ordovician and below the Hermes marble, which they correlate with the Middle Ordovician Ella Dolomite exposed at Bayhorse, Idaho.

Early rudimentary stratigraphy was presented by Currier (1935), but Smitherman (1985) constructed a more detailed and comprehensive stratigraphic column of the Stibnite roof pendant (Figure 7-5). The metasedimentary rock units are divided into ten informal units. They are, in ascending stratigraphic order: Quartzite-schist, Lower Calc-silicate, Fern Marble, Quartz Pebble Conglomerate, Lower Quartzite, Upper Calc-silicate, Middle Marble, Middle Quartzite, Hermes Marble, and Upper Quartzite. The following descriptions are based mainly on Smitherman's work (1985) and include additional information from various unpublished studies completed by previous operators and by Midas Gold.

Quartzite Schist

The Quartzite-Schist unit in the SGP area is observed in exposures up to 460 ft thick and is apparently the oldest unit exposed in the immediate Project area. The unit consists of interbedded quartzite 4 inches to 4 ft thick and schist forming distinct compositional banding, likely reflecting original lithologic bedding, and local isoclinal folding. Intermediate lithologies between quartzite and schist are common and the unit is subdivided into quartz-mica schist, garnet-bearing quartz-biotite schist, and micaceous quartzite (Smitherman, 1985). Based on regional mapping in the Big Creek area and northeast of the District by the IGS, this unit is interpreted to be Neoproterozoic in age (Lewis et al., 2014).

Lower Calc-Silicate

The lower Calc-Silicate unit is 165 ft to 900 ft thick and consists of thin-bedded siltites and calc-silicate-bearing rocks. The contact between the quartzite schist and the calc-silicate sequence appears to be gradational. Minor folds are common and probably account for much of the variation in thickness. The unit contains grey quartz-feldspathic layers with alternating green calc-silicate beds in the lower portion and light grey calcitic marble with green calc-silicate interlayers in the upper portion. Locally the rocks have been altered to a coarse-grained skarn assemblage of garnet, epidote, diopside, calcite, pyrite, and iron oxide.

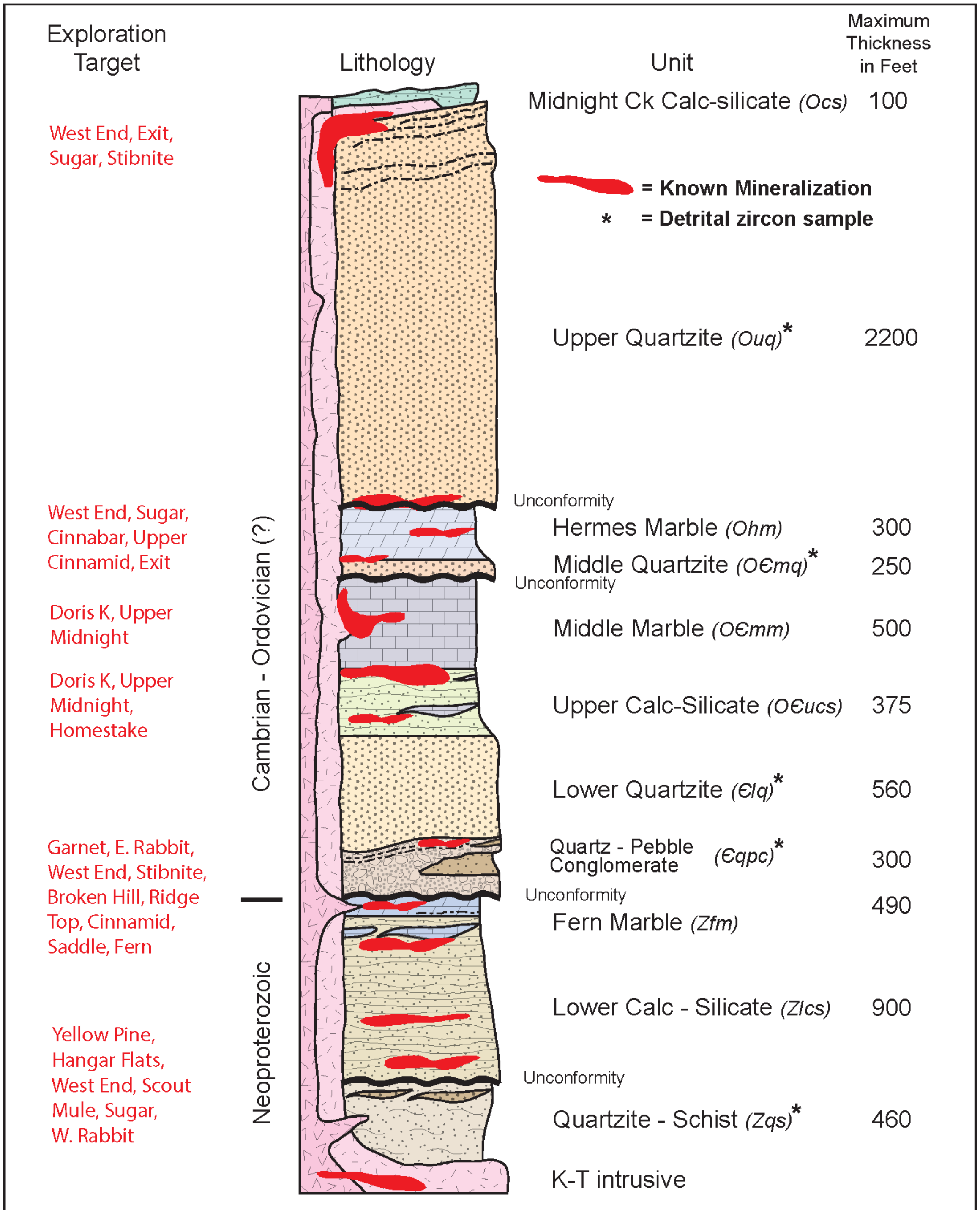
Fern Marble

The Fern marble overlies the lower calc-silicate and reaches a maximum thickness of about 500 ft. The marble is massive and consists primarily of coarse dolomite with rare quartz. Green-gray calc-silicate marble is locally common within 500 ft of the batholith contact.

Quartz-Pebble Conglomerate

The Quartz-Pebble conglomerate is a coarse-grained, pebbly quartzite unit, which contains lenses of metamorphosed pebble conglomerates and bodies of quartz-mica schist. The contact with the Fern marble is well exposed and likely represents an unconformity. Small schist lenses occur locally and consist of quartz, muscovite, biotite, sillimanite and/or andalusite. Detrital zircon dated LA-ICPMS methods and regional relationships suggest this unit is likely Neoproterozoic in age and possibly correlative in age to the Neoproterozoic Caddy Canyon quartzite exposed near Pocatello in southeast Idaho (Lewis et al., 2014).

Figure 7-5: Stibnite Roof Pendant Stratigraphy



Stibnite Roof Pendant Bedrock Stratigraphic Column after (Smitherman, 1985; Stewart et al., 2016)

Lower Quartzite

The quartz-pebble conglomerate unit grades upward into a muscovite-bearing quartzite that is 295 ft to 560 ft thick. The quartzite is typically light gray and commonly shows dark gray streaks, which appear to be relict bedding. Outcrops are large and bold, occurring along ridges and on slopes. The rock weathers into large blocks and vast talus fields. Thin sections show that the quartzite is 95% fine-grained to very coarse-grained quartz with up to 5% muscovite and 2% andalusite.

Upper Calc-Silicate

The upper Calc-Silicate consists of biotite, plagioclase, calc-silicate rock. The unit thickness varies from about 100 ft to about 375 ft, likely due to zones of isoclinal folding. The internal stratigraphy of the unit includes four subunits ranging from the lower dark gray, laminated plagioclase-calc-silicate rock to plagioclase-biotite rock to massive, calcareous, plagioclase-scapolite-diopside rock to centimeter-scale interbedded calc-silicate and calcitic-marble.

Middle Marble

The upper calc-silicate unit grades upward into a calcitic marble unit that is 260 ft to 490 ft thick. The unit is dominantly a massive, blocky, thick-bedded, blue-gray, finely crystalline limestone interbedded with thinner, light gray, thin-bedded, (1 inch) laminated marble. The rock is 80% to 99% calcite with minor biotite, diopside, and graphite.

Middle Quartzite

A quartzite unit 30 ft to 250 ft thick lies above the Middle Marble. It is a light gray, fine- to coarse-grained, vitreous quartzite. Accessory minerals are K-feldspar, sericite, graphite, leucoxene, zircon, and iron oxide. Carbonate cement is locally present, as well as rare biotite schist bodies near the lower contact.

Hermes Marble

The Middle Quartzite is overlain by 195 ft to 295 ft of dolomitic marble. The lower 195 ft consist of a light gray massive dolomite marble containing 80% dolomite and 20% altered tremolite porphyroblasts. Alteration of the tremolite is probably hydrothermal and resulted in clay replacing 90% of the tremolite. Minor pyrite and iron oxide are locally present. The upper portion is a gray, laminated marble that has essentially the same mineralogy but is generally unaltered. the Hermes Marble is often silicified and converted to maroon to gray-red jasperoids throughout its outcrop area, in underground workings, and drill holes within the Cinnabar Mine complex east of the District.

Upper Quartzite

A quartzite unit with minor siltite overlies the Hermes Marble and varies in thickness from 1,400 ft to 2,200 ft. The quartzite is nearly pure quartz with less than 3% muscovite. Locally, black quartzite contains intergranular graphite. Accessory minerals include zircon, magnetite, sericite, and secondary iron oxide after pyrite. Laminated gray siltite occurs in the upper portion of the unit. The siltite is composed of 70% to 90% fine quartz grains with the remaining 10% to 30% comprising biotite and minor muscovite. Preliminary detrital zircon dating as reported by the IGS suggests the unit is likely an age equivalent with the Ordovician Kinnikinic Quartzite of the Bayhorse area along strike to the southeast in southeast Idaho (Lewis et al., 2014).

7.2.2 Structure

7.2.2.1 District Structure

The District has a complex structural history including regional metamorphism associated with fold-thrust belt tectonism, transpressional to transtensional strike-slip faulting, and normal-oblique faulting associated with regional extension. Historic surface and underground mining records, field mapping, data from oriented drill core, and geophysical surveys indicate three dominant trends within the district.

Several major regional-scale structural features cut through the Project area in addition to numerous smaller subsidiary structures. Large, north-south striking, steeply dipping to vertical structures occur in the central and eastern portions of the property and include the MCFZ, the Scout Valley fault zone (**SVFZ**), the West End fault zone **(WEFZ) (**Figure 7-4**)**, and the Mule fault zone (**MFZ**). These features exhibit pronounced gouge and multiple stages of brecciation, suggesting multiple periods of movement. They are poorly exposed due to recessive weathering and often are found under or along the flanks of glacially carved valleys. The MCFZ can be traced from the main Yellow Pine Deposit south 1.85 mi through the Hangar Flats Deposit and continues for another 1.25 mi to the south, where it is cut by the Big Creek Graben Complex (Figure 7-2). The WEFZ is the northern continuation of the SVFZ and extends from the West End Deposit 1.5 mi to the northeast to its intersection with the Sugar Creek fault.

The MCFZ had early west-side up movement followed by right-lateral displacement based on kinematic indicators in underground and surface exposures and oriented drill core but variations in sense and amount of relative displacement are common. The WEFZ also has right lateral displacement based on offset of macroscale fold hinges but this is associated with east-side up displacement. Both the MCFZ and WEFZ are often associated with east-west and northeast-southwest trending splays and dilatant structures.

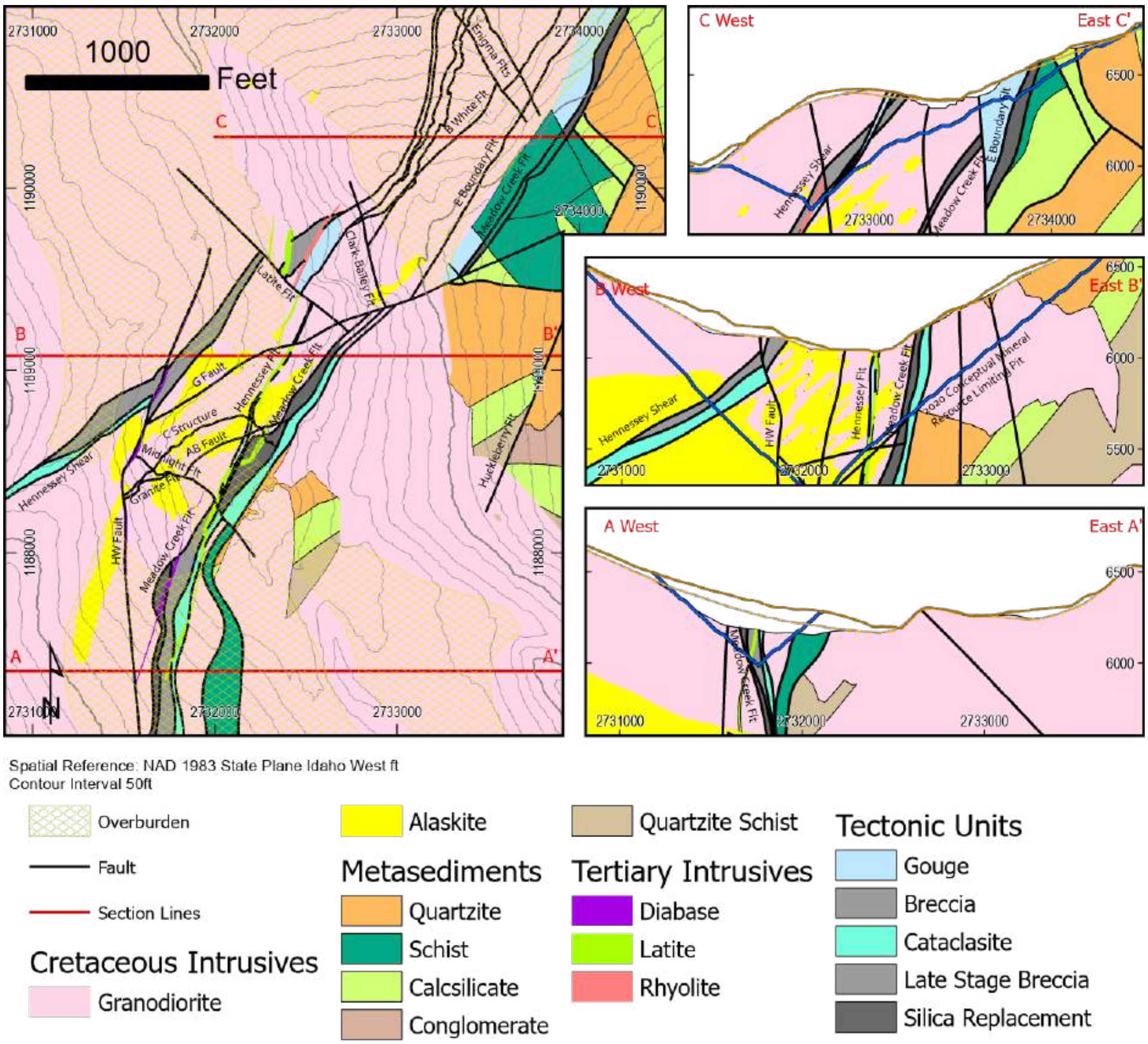
Large northeast striking structures in the district are coincident with major topographic lineaments and include the Salt Creek fault, the Sugar Creek fault, the Fern fault and other north-easterly structures. These are interpreted as either splay structures from the north-south faults or as younger structures that offset earlier faults. Large, northwest-southeast trending geophysical features cut through and within the metasedimentary rocks of the roof pendant and continue to the northwest across batholith rocks and through the younger caldera sequence to the southeast suggesting these features have at least some movement after development of the north-south and northeast elements.

Konyshev (2020) reported results of an apatite fission track study to evaluate the timing of uplift and evaluate potential block faulting and rotation of the Stibnite Roof Pendant. There are implications for future exploration at depth beneath and laterally around mineral occurrences in the eastern and central portions of the district if the roof pendant has been block faulted and tilted prior to or after the various periods of mineralization. For instance, if the low temperature epithermal mineralization found in the Fern and Cinnabar areas has been rotated after mineralization, higher-grade feeder structures may not lie vertically beneath areas of currently outcropping mineralization. Konyshev (2020) reported a weighted mean apatite fission-track age of 53.2±1.2 Ma for 12 samples spread throughout the area. He postulated the data point to rapid cooling of the apatites through their closure temperature due to passage of hydrothermal fluids potentially representing the waning stages of the main stage gold event, but prior to the lower temperature epithermal event. He also suggested the data did not support block rotation of the roof pendant after the ~53 Ma event since the samples do not show evidence of differential cooling through the apatite closure temperature. There is circumstantial evidence for tilting of the roof pendant in the west side of the district which, based on this data, would require that tilting, if it occurred, to have happened prior to ~53 Ma.

7.2.2.2 Yellow Pine Deposit Structure

In the Yellow Pine Deposit area (Figure 7-6), major structures include the northerly striking Meadow Creek Fault Zone and the northeasterly striking Hidden fault zone. These structures show evidence for multiple stages of motion, both pre- and post-main stage gold mineralization. Additional structures include subsidiary splay faults to the MCFZ, the northeasterly striking Letter faults and northwesterly striking scissor faults. The structural setting of the Yellow Pine Deposit is interpreted as a broad damage zone of the strike-slip Meadow Creek fault system accommodating progressive displacement and amalgamation of the Hidden Fault through development of sympathetic and antithetic shears and extensional and transpressional block rotation within the zone.

Figure 7-6: Geological Model for the Yellow Pine Deposit



The earliest recognized fabric in the intrusive rocks consists of aligned biotite in the Cretaceous granodiorite/quartz monzonite which defines a shallowly east-northeast dipping planar fabric interpreted as magmatic foliation formed during crystallization of the batholith (Figure 7-7A). Aplitic and granitic dikes are generally steeply dipping striking either northerly or east-northeasterly or are sub-horizontal. Locally, particularly above the Yellow Pine granite which underlies the main deposit, the dikes have a wide variety of orientations and have a stockwork-type configuration.

The earliest recognized faults in the intrusive rocks (D1a) consist of cohesive silicified breccias with ductile fabric elements (Figure 7-7, A; Figure 7-8, B). D1a breccias commonly contain stretched and elongated asymmetric clasts defining a shear fabric within a silica-pyrite matrix. D1a breccias generally occur as localized discontinuous bodies truncated by later faults and are not typically continuous between drill holes. However, two steeply dipping tabular bodies are recognized parallel to the MCFZ trend in the southern Yellow Pine Deposit, west of the Au-Sb-mineralized fault zone and along the north-south MCFZ at the boundary between Cretaceous intrusive rocks metasedimentary rocks of the Stibnite Roof Pendant. These structures are interpreted to have been active early based on presence of pre-mineralization sulfides and cross cutting relations. Absence of pervasive fabric development in adjacent rocks suggests that asymmetric stretched clasts are indicative of quartz-microplasticity under high fluid pressure conditions rather than true mylonitic processes within the ductile crustal regime.

Secondary D1b structures consist of breccias and vein arrays associated with the main stage gold mineralization event (Figure 7-7, C-E; Figure 7-8, B). D1b breccias are characterized by angular to slightly asymmetric clasts within a silica-pyrite or K-spar-silica-pyrite matrix. Clasts have undergone only minor displacement and rotation within breccia zones consistent with fluid-assisted brecciation processes rather than associated with tectonic compressive stress. D1b silica-pyrite veins and breccia veins predominantly strike northwesterly and are spatially associated with gold mineralization. Broad D1b breccia zones occur throughout the central Yellow Pine main-mineralized zone and also as tabular zones within the MCFZ and Hidden fault zone. D1b breccias are interpreted to have partially reactivated, overprinted, and obliterated earlier D1a structures during progressive deformation. Geochemically, D1b veins and breccias are have lower sulfur/arsenic ratios than D1a breccias, as is characteristic of main-stage, ore-grade gold mineralization.

D2 structures consist of brittle fault zones and epithermal veins. D2 fault zones contain fault gouge and cataclasite formed through wear abrasion processes during brittle tectonic comminution. D2 faults predominantly strike east-northeast and north-south (Figure 7-7, F-G; Figure 7-8, C-D). They include regions of the north-south striking MCFZ, east-northeasterly striking letter faults in central Yellow Pine, the Hidden fault along Hennessey Creek and other district scale northeast striking block faults which accommodate normal displacement of metasedimentary units in the Stibnite roof pendant (Smitherman, 1985). The Granite fault at Yellow Pine is a D2 structure interpreted to have significant post-gold-mineralization normal displacement, juxtaposing sericite-ankerite altered granite against the central Yellow Pine main-mineralized zone. The Hanging Wall fault and Hennessey fault are steeply dipping, northerly striking D2 brittle fault structures which juxtapose mineralized and unmineralized fault blocks. The East Boundary fault, a sub-unit of the MCFZ mapped by Hecla geologists in the eastern part of the Homestake pit, is a broad D2 gouge zone which marks the eastern limit of disseminated gold mineralization within the intrusives. The East Boundary fault gouge offsets the silicified corridor of the MCFZ with apparent east side up displacement, occurring in the hanging wall near surface and in the footwall at depth, reflecting oblique-reverse offset of D1 breccias during D2 brittle faulting.

Drill holes in the hanging wall of the Hidden fault and within the gap zone, between central Yellow Pine and Homestake, delineate a broad region of brittle deformation manifested as shattered rubble zones within rheologically competent felsic aplites and as weakly sheared or deformed argillically altered quartz monzonite. This zone may represent the extension of deformation associated with the Hennessey Creek structure across the Hidden fault and into the “gap” zone.

D2 structures host the tertiary latite and diabase dikes and also contain clasts of dike rocks in some drill hole intersects. Association with tertiary dikes and offset and entrainment of main-stage gold mineralized material are indicative of Eocene or later activation of D2 structures, based on current geochronology.

Figure 7-7: Stereonets of Oriented Core Data from Yellow Pine

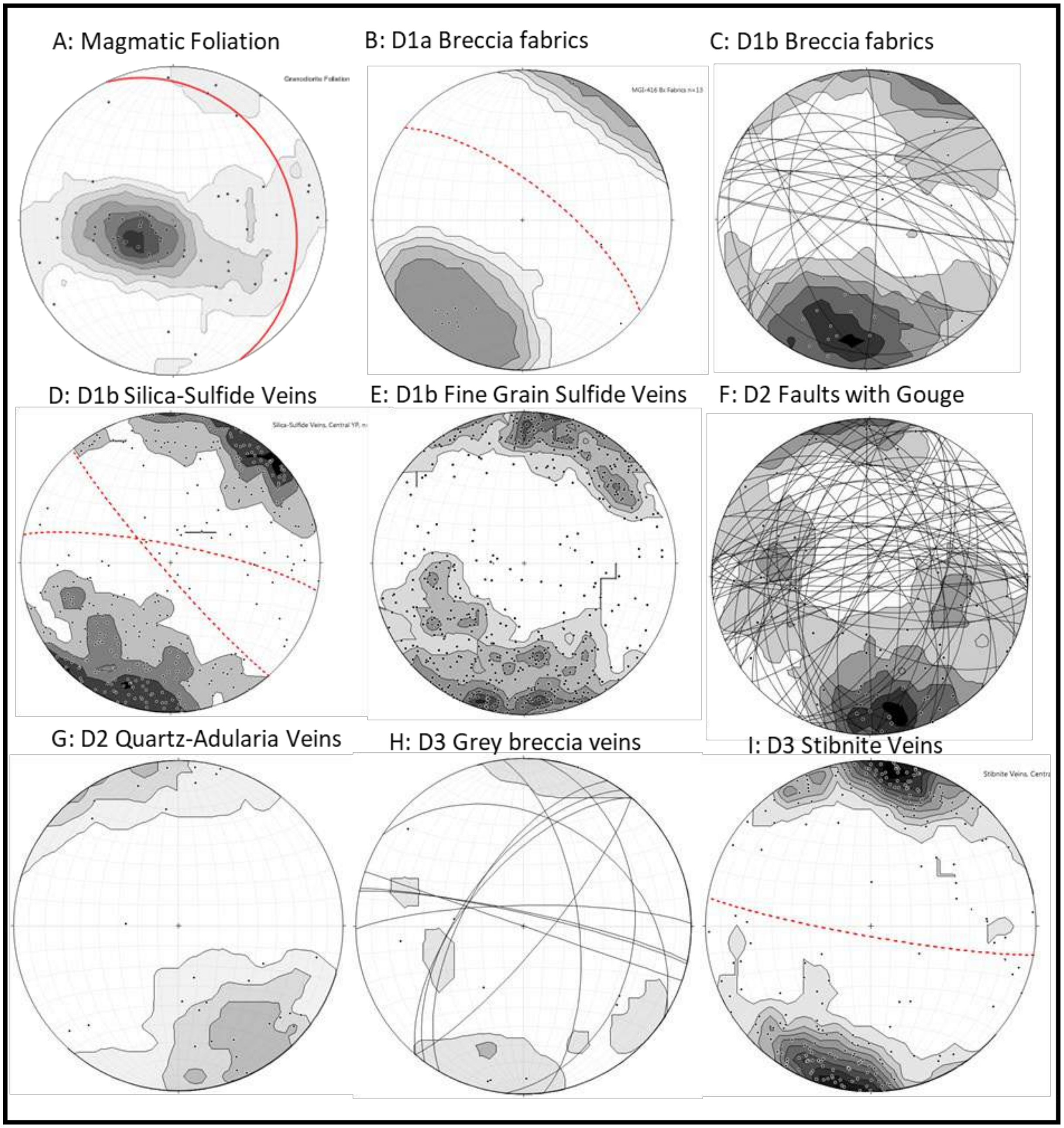
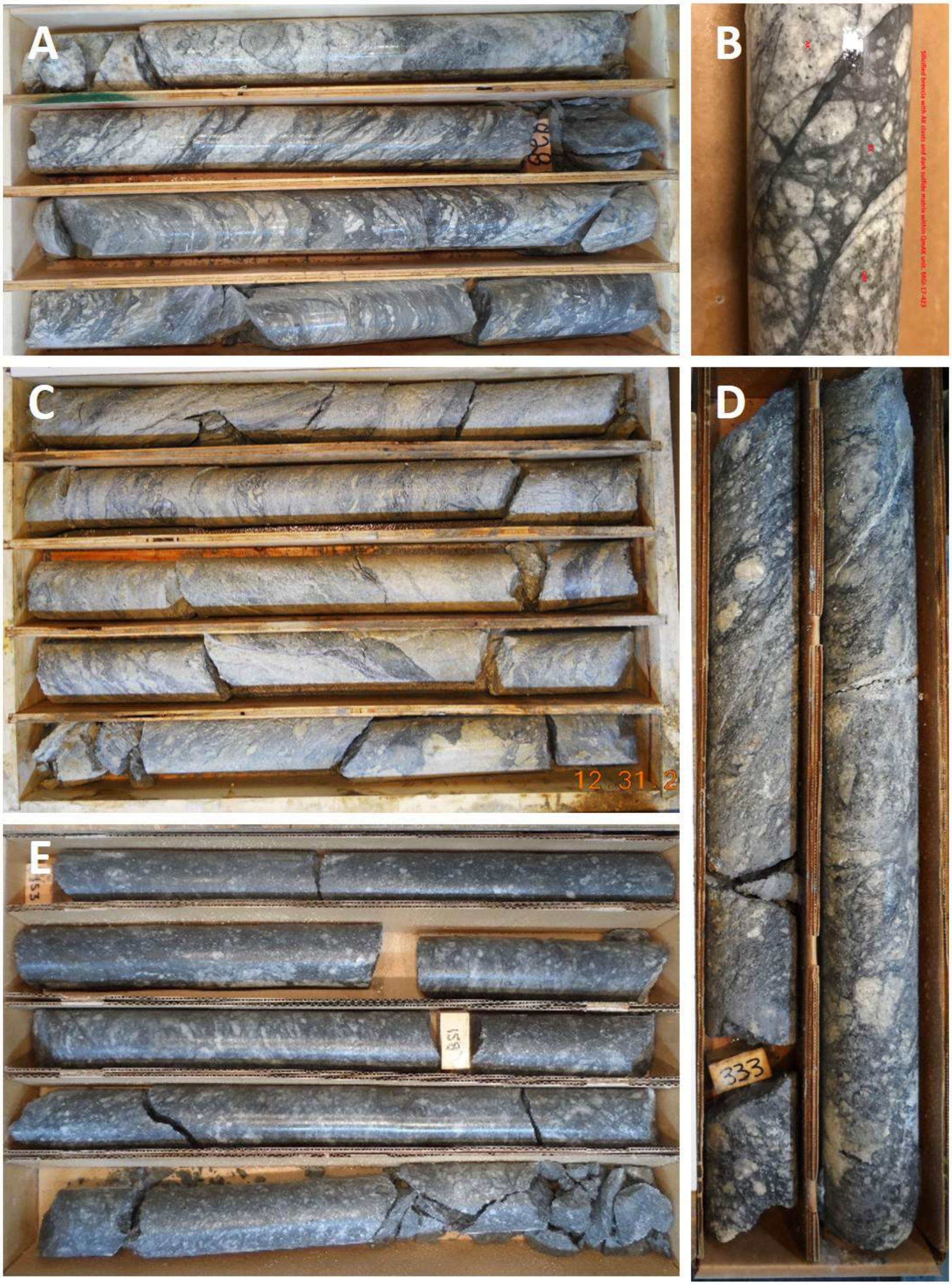


Figure 7-8: Structural Fabrics in the Yellow Pine Deposit



Northeasterly striking epithermal quartz-calcite-adularia veins occurring east of the MCFZ at Yellow Pine and West End are also attributed to D2 faulting (Figure 7-7, G). These veins show textural evidence for open-space filling such as coliform banding. Gold mineralization is only sometimes associated with these veins and is geochemically distinguished from the main-stage gold at Yellow Pine event by high gold/arsenic ratios, lower sulfur contents and narrower zones of potassic alteration in surrounding rocks.

D3 structures consist of northwesterly striking brittle fault zones sometimes associated with antimony mineralization and silicification (Figure 7-7, I; Figure 7-8, E). The northwesterly striking Midnight fault marks the southern boundary of antimony mineralization in the central Yellow Pine main mineralized zone. The fault consists of 1-2m-wide zone of silicified, locally stibnite-cemented matrix-supported breccia with distinct milled clasts. Stibnite mineralization in the footwall occurs as sheeted vein arrays sub-parallel to the Midnight fault. Northwesterly striking shear bands parallel to the Midnight fault and slicken-sided fault planes cut the silicified breccia exposed in the southwestern Yellow Pine pit. The Midnight fault is also interpreted to offset the D2 Granite fault and may extend across the Hanging Wall fault into a structurally complex zone of the Hidden fault zone.

Additional D3 structures are interpreted to offset latite dikes and the Hidden fault zone north of the central Yellow Pine mineralized zone and are also modeled in the Homestake area south of the Clark Tunnel and north of the historical Homestake pit, where they were termed the “enigma faults” by Hecla geologists. Northwesterly and north-northeasterly silica-cemented micro breccia veins cutting D2 epithermal veins in the roof pendant east of the MCFZ are attributed to D3. Limited kinematic indicators from D3 minor faults indicate strike-slip or dip-slip motion. Modeled offsets of dikes and earlier fault zones are inconsistent and the D3 structures are interpreted to have undergone variable, scissor type offset accommodating differential block rotation during continued strike slip faulting. The association of stibnite mineralization and rare scheelite within breccia matrix fill in D3 structures may place a late Eocene age constraint on this faulting event.

7.2.2.3 Hangar Flats Deposit Structure

The MCFZ, and northeasterly striking splay faults, are the principal structures controlling mineralization in the Hangar Flats Deposit. Like at the Yellow Pine Deposit, the MCFZ shows evidence for multiple stages of movement and fault strand reactivation both prior to- and post main-stage gold mineralization, antimony mineralization and Eocene dike emplacement. The MCFZ at the Hangar Flats Deposit is a complex structural zone of steeply dipping anastomosing fault strands and intact fault blocks of annealed breccias, cataclasite zones, clay gouge and rubble intruded by Tertiary rhyolite and diabase dikes.

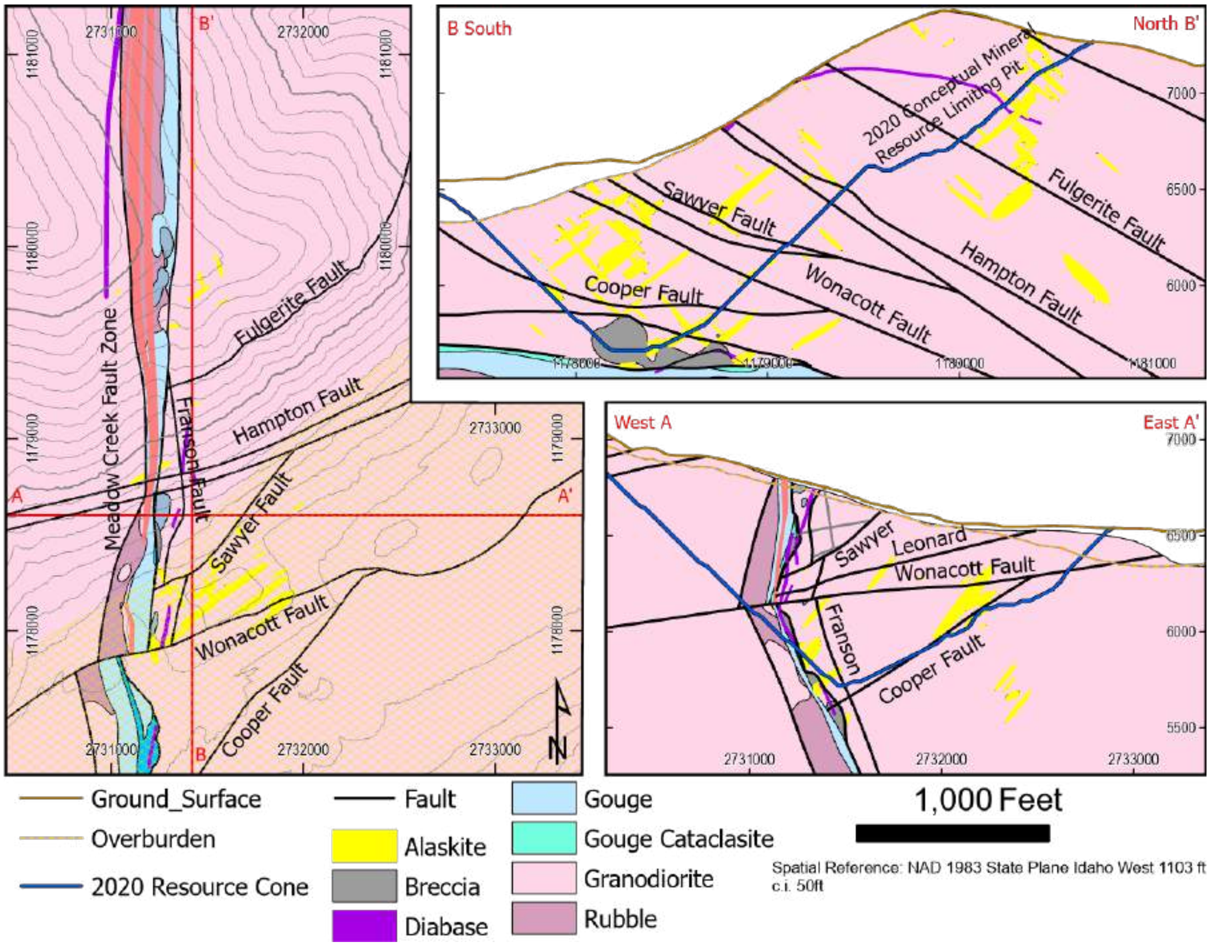
Subsidiary fault strands parallel to the MCFZ, as well as northeasterly striking, shallowly northwest dipping splay structures, and their intersections with the MCFZ, are important controls on mineralization in the Hangar Flats Deposit and have been mapped in legacy underground workings and intersected in drill holes. The north-striking Franson, Shake, and Frylock faults are parallel fault strands east of the MCFZ. The northwesterly dipping Cooper, Leonard, Sawyer, and No Name faults are splays associated with the MCFZ. The Wonacott and Hampton faults, also dipping northwesterly, are interpreted as splay structures with post-mineralization reactivation and offset of mineralization.

Multiple stages of movement are described in the historic literature from underground mapping, within unpublished company files, and are observed in Midas Gold drill core. Mineralized fragments have been rotated and then re-mineralized, indicating several periods of movement coincident with at least some stages of sulfide mineralization. Various kinematic indicators suggest the latest movement along the MCFZ involved right lateral and high angle reverse (i.e. west side up) movement and is marked by gouge and brecciation. Late movement is also indicated by broken Eocene rhyolite and diabase dikes observed in drilling.

At the Hangar Flats Deposit, the MCFZ displays a dip reversal across the Wonacott fault, a north-easterly splay structure interpreted to cut and offset the MCFZ with approximately 200 ft of oblique-right-lateral-normal displacement.

The MCFZ dips about 80° west, north of the Wonacott fault and dips 70° east in the footwall of the Wonacott fault, as shown on Figure 7-9. The Hampton fault Set is mapped in underground workings as offsetting and rotating diabase dikes by small amounts across multiple fault planes.

Figure 7-9: Geological Model for the Hangar Flats Deposit

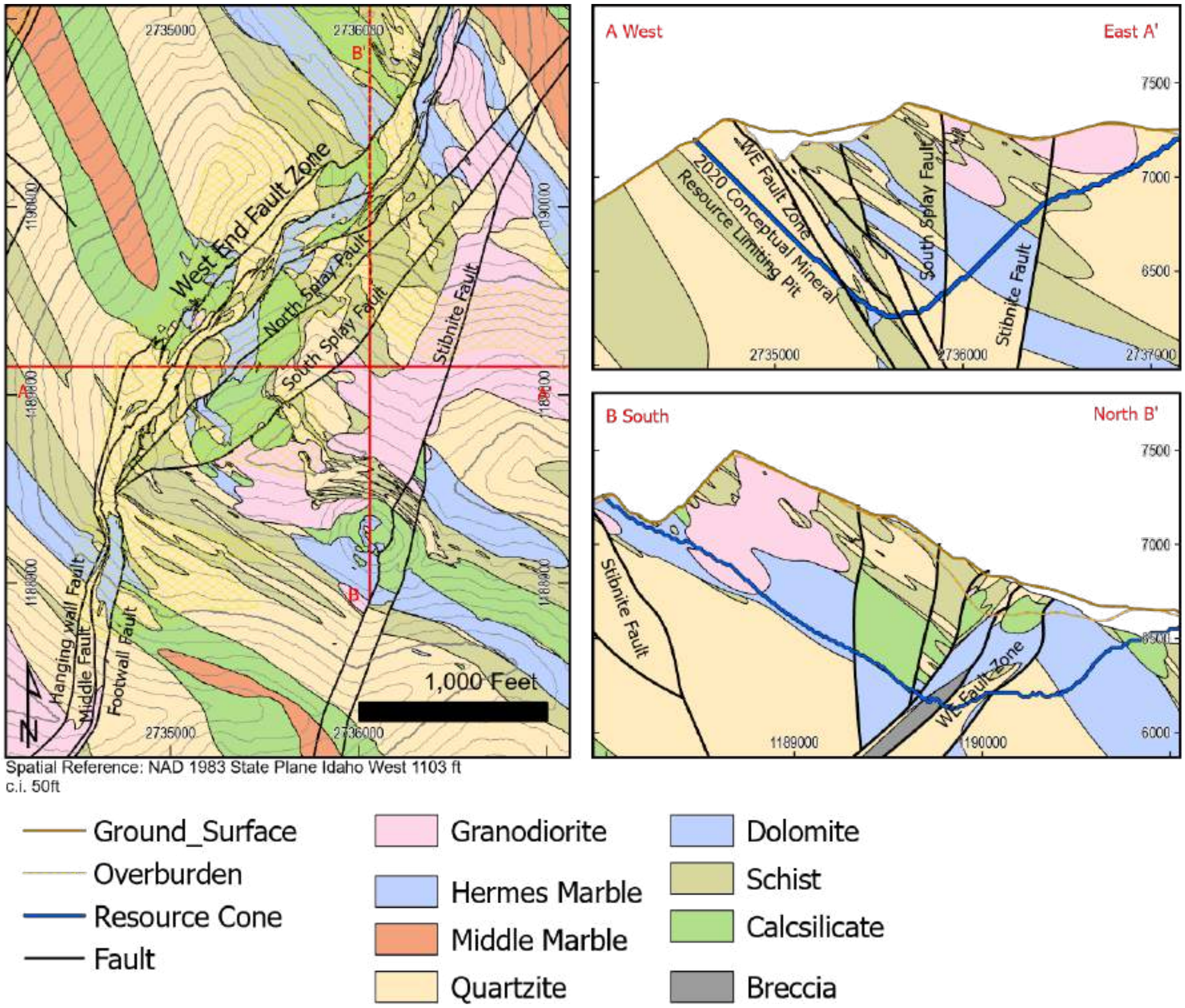


7.2.2.4 West End Deposit Structure

Metasedimentary rocks in the West End Deposit record both brittle and ductile deformation. Ductile deformation associated with Cretaceous regional metamorphism affected the roof pendant prior to District gold mineralization events. Brittle faulting provided important structural controls for mineralizing fluids during hydrothermal alteration and also continued following gold mineralization. Metasedimentary rocks in the deposit occur on the overturned limb of the Stibnite Syncline, striking northwest and dipping steeply to the northeast. Stratigraphic facing indicators as well as regional stratigraphic relations indicate beds are steeply overturned, younging to the southwest. Ductile deformation in the metasedimentary rocks resulted in development of minor folds, penetrative cleavage and bedding parallel thrusts or reverse faults. Shallowly northwest plunging minor fold axes and fold asymmetry are consistent with parasitic folding on the limb of the regional Stibnite Syncline.

The West End fault zone (**WEFZ**) is the predominant brittle structure associated with the West End Deposit. The main fault zone consists of three high angle faults, all striking north-northeast and dipping 50° to 75° to the southeast. The width of the WEFZ as measured between the footwall and the hanging wall faults varies from 100 ft to 295 ft. Based on offset of the shallowly plunging hinge of the Stibnite Syncline and other kinematic indicators, the WEFZ has experienced approximately 1,800 ft of right lateral and normal (down to east) offset (Figure 7-10).

Figure 7-10: Geological Model for the West End Deposit



The three high angle faults comprising the WEFZ occur as zones of gouge and silica-carbonate cemented polyphase breccias. The breccias often host some of the higher-grade gold mineralization in the deposit. Breccia clasts include both Latite and mineralized granodiorite suggesting at least some of the displacement along the WEFZ occurred following the main-stage gold mineralization event and emplacement of post-mineralization dikes. Some fault strands are locally pervasively oxidized to depths greater than 500 ft below the ground surface. Contiguous deformed blocks of metasedimentary rock occur between the main fault strands of the WEFZ and generally maintain the stratigraphic sequence of the roof pendant as it is displaced across the structure.

Several east-northeast-striking structures appear to splay off the primary structural zone and include the Splay faults, Stibnite fault, and Northeast Extension fault structures. The subsidiary splay structures have strikes ranging from azimuth 060° to 090° and dip steeply north and south. These faults generally offset the roof pendant with apparent right-lateral displacements of 100 ft to 300 ft. Other major faults recognized in the West End deposit include the Cinnabar Peak Thrust occurring at the base of the quartzite-schist formation and the Huckleberry fault, occurring 1,200 ft to the west of, and parallel to the main WEFZ. The Stibnite Stock biotite granite is emplaced into the roof pendant approximately 1,200 ft east of the WEFZ and forms a sub-vertical tabular body roughly parallel to the WEFZ with sills extending into the Lower Calc-silicate formation.

7.3 ALTERATION

Alteration and mineralization of intrusive rocks from the Yellow Pine and Hangar Flats deposit was described by White (1940), Cooper (1951), Lewis (1984), Cookro et al. (1987), and others. Recent petrographic, fluid inclusion, isotopic and dating studies completed through a collaborative partnership involving state, federal and academic research partners have significantly improved the understanding of deposit paragenesis and geochronology. Investigations by Gillerman et al. (2014); Gillerman et al. (2019); Marsh et al. (2016); Konyshev and Muntean (2016); Wintzer, 2019; and other ongoing unpublished research have documented progressive alteration and mineralization associated with multiple hydrothermal events occurring through the Paleocene and early Eocene epochs. A generalized paragenetic sequence is shown on Figure 7-11.

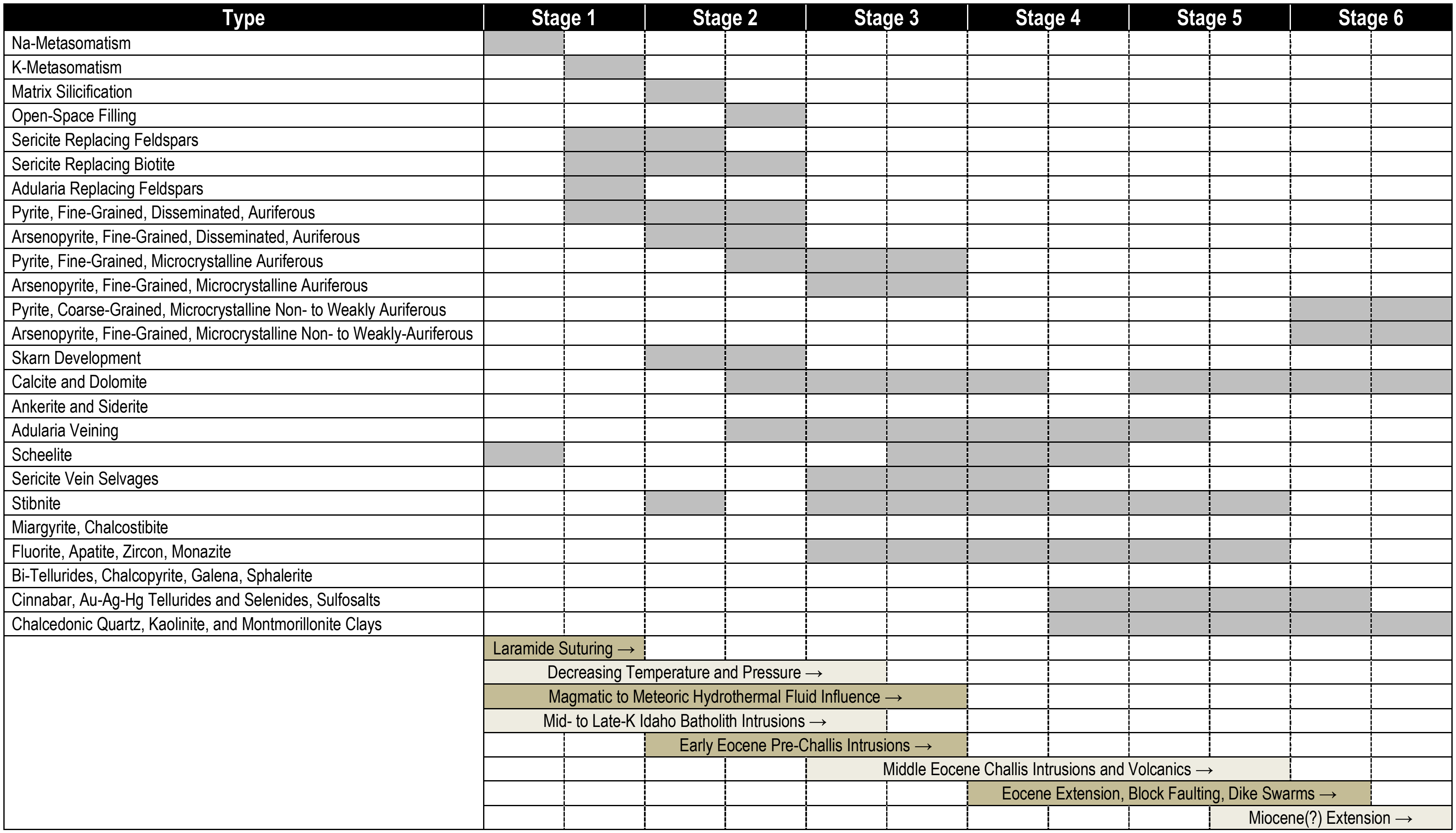
Multiple workers have estimated temperature-pressure conditions for hydrothermal events using a variety of methods. Konyshev (2020) examined vein relationships, textures, and chemistry from metasedimentary-hosted mineralization at West End and in several prospects and outlined four stages of mineralization. Marsh et al. (2016) utilized fluids inclusions to estimate pressure-temperature conditions of intrusive-hosted mineralization from Yellow Pine samples. Lewis (1984) applied the arsenopyrite-pyrite-pyrrhotite geothermometer and modeled stable isotopes to estimate pressure and temperature of mineralization throughout the district. Wintzer (2019) utilized lutetium–hafnium (Lu-Hf) geochronology and trace element contents along with garnet-aluminosilicate-silica-plagioclase (GASP) geothermobarometry to study conditions of metamorphic and igneous garnet formation.

There were multiple pre-mineralization Cretaceous age magmatic events in the district with at least four geochemically and temporally distinct intrusive pulses. At least four pulses of felsic to intermediate intrusions with peraluminous to metaluminous compositions are documented by extensive zircon geochronology at around 92-98 Ma, 88 Ma, 84-86Ma, 82-83 Ma (Gillerman et al., 2014; Stewart et al, 2016; Wintzer, 2019). Examination of zircon textures and dating work also suggests that there was cooling of the Cretaceous Idaho Batholith intrusive rocks between 78-71 Ma, which is temporally consistent with the onset of regional extension.

Systematic $^{39}Ar/^{40}Ar$ dating of gangue mineralogy includes dates on pre-mineralization magmatic feldspar and biotite, hydrothermal adularia, altered biotite, and illite associated with main stage arsenical pyrite mineralization (Gillerman et al., 2014; 2019; Hofstra et al., in preparation). This work indicates the main phase of gold mineralization occurred between 65-55 Ma (Gillerman et al., 2019; Hofstra et al., in preparation) which were overprinted by a later quartz-carbonate-adularia vein event around 50-51 Ma (Gillerman et al., 2019). Wintzer (2019) reported dates on scheelite that ranged from 45.8-61.3 Ma. No intrusions of the same age as the main stage gold mineralization are known to be present in the District or surrounding area.

The first alteration event is subdivided into two phases. Phase 1a is recognized at the West End and involves gold barren pyrite mineralization in calc-silicates and emplacement of milky-white, molybdenum-bearing quartz veins. Phase 1b, which effects the batholith and the roof pendant also involves milky-white quartz veins containing coarse, gold-barren pyrite with coarse muscovite and/or potassium feldspar (Gillerman et al, 2019). Phase 1 veins were interpreted based on texture to have formed under ductile conditions at temperatures >375° to 400°C (Konyshev, 2020). Re-Os dating of molybdenum from Phase 1a veins indicates 83.6 ± 0.4 and 86.3 ± 0.4 Ma ages (Konyshev and Munteen, 2016; 2019) which are similar to 86-82 Ma intrusive activity, including the 85.7 ± 0.1 Ma Stibnite Stock (Gillerman et al., 2019). The age of Phase 1b alteration is constrained by regional 79-75 Ma rutile and zircon rim cooling ages attributed to a regional late Cretaceous hydrothermal cooling event generally correlative with polymetallic vein events in the Big Creek district (GIllerman et al., 2019; Gammons, 1988).

Figure 7-11: Stibnite – Yellow Pine District Paragenesis



Source: Modified from Lewis, 1984; Cookro et al., 1987; Surface Science Western, 2012

The first alteration event is thought to have occurred following regional metamorphism and early magmatism. Wintzer (2019) used GASP geothermobarometry to estimate mid-crustal pressures that ranged from 7.5 to 7.0 kilobars (depths of 28.3 to 26.5 km) from samples collected west of the District near Salt Creek. These samples also provided an amphibolite-facies metamorphic temperature of approximately 775°C. A crosscutting leucogranite dike with an approximate age of 99 Ma, slightly older than most intrusions in the District was crystallized in a similar pressure range of 6 to 8.5 kilobars and a temperature range of 775° to 825°C, indicating the district remained at mid-crustal depths at this time.

The second alteration event is pervasive potassic alteration associated with main stage sulfidation and gold deposition in the Yellow Pine and Hangar Flats deposit. This event involves sericitization, replacement of biotite by pyrite, replacement of igneous plagioclase by potassium feldspar, and addition of quartz-pyrite-arsenopyrite and quartz-carbonate-pyrite-arsenopyrite veining (Lewis, 1984; Gillerman et al., 2019). Alteration is not typically texture-destructive at hand specimen scale and many of the primary textural relations of the typical quartz monzonite host rock are still evident. Main stage veins have more planar vein wall characteristics than the deep early veins, lack textures characteristic of epithermal veins, and were formed by brittle fracturing at temperatures probably less than 375˚C to 400˚C (Konyshev, 2020). Phase 1 veins are cross-cut by main stage veins composed mainly of quartz, iron carbonate, arsenian pyrite, and arsenopyrite. Main stage veins contain higher sulfide content and carry significant gold, which occurs in arsenian pyrite and arsenopyrite.

Petrography and microscopy by numerous workers document the early formation of very fine-grained disseminated euhedral pyrite associated with the replacement of plagioclase by hydrothermal potassium feldspar. This was followed by subsequent pyrite etching events with the growth of gold-bearing arsenical pyrite rims, and the replacement of adularia by coarse-grained sericite (Lewis, 1984; Palko and Martin, 2011a; Palko and Martin, 2011b; Palko, 2012; Gillerman et al. 2019, Konyshev and Muntean, 2016). SWIR spectroscopy indicates illite and ammonium illite are the dominant alteration products associated with Phase 2 (Zinsser, 2015). The timing of main stage gold mineralization in the district has been previously reported at 77.9 (Gammons, 1988), 57 Ma (Lewis, 1984), and 51 Ma (Gillerman, 2017), based on K-Ar and Ar-Ar dating of adularia. Recent work by the USGS and IGS, including step heating and single-crystal total fusion of sericite and adularia separates, indicates multiple ages of ~65 Ma for Phase 2 main stage mineralization based on samples from Yellow Pine and Hangar Flats. These new ages are believed to distinguish Phase 2 alteration from alteration associated with earlier and later Phase 1b and Phase 3 potassium feldspars, but uncertainty due to geological complexity is still present (Gillerman et al., 2019). The temperature of main stage gold-bearing arsenical pyrite mineralization was estimated by Lewis (1984) to be approximately 260°C, based on the pyrite-arsenopyrite-pyrrhotite geothermometer, but Lewis noted a range of 100° to 400°C, indicating a wide range of fluid temperatures and/or overprinting events. Marsh et al. (2016) estimated a range from 233° to 175°C from fluid inclusions in quartz associated with gold mineralization.

The third alteration event overprints main-stage gold mineralization and involves the deposition of tungsten as scheelite and antimony as stibnite. The W-Sb alteration event is associated with silicification and brecciation resulting in stibnite veining and distinctive black matrix breccias within discrete structural zones. Stibnite mineralization is texturally late relative to scheelite and stibnite vein arrays occur over a larger area relative to the tungsten mineralization mined historically at Yellow Pine. ID-TIMS and LA-ICPMS dates for scheelite at Hangar Flats yield circa 57 Ma ages similar to step-heating ages of adularia within a scheelite-stibnite breccia dated at 56 Ma (Gillerman et al, 2019). This event is coeval with scheelite mineralization at Quartz Creek near the town of Yellow Pine (Gammons, 1988). Fluid inclusion studies of hydrothermal quartz associated with stibnite mineralization indicate epizonal depth of formation at pressures of 11-6 bars, temperatures of 160° to 189°C, and fluid salinities of 4.2-10.9 wt % (Marsh et al., 2016).

The fourth alteration phase is an additional gold mineralization event effecting rocks of the Stibnite Roof pendant expressed as epithermal quartz-carbonate-pyrite veins with adularia selvages which occasionally contain very fine-grained free gold. Compound banded veins, open-space textures, chalcedonic or drusy quartz and local bladed calcite are typical of phase 4 and indicative of temperatures <220°C (Koneshev and Muntean, 2016). Adularia ages from Phase 4 veins in the West End deposit indicate 51.9-51.2 +/- 0.8 Ma for this event, slightly older than the initial onset of Eocene volcanic activity at Thunder Mountain (Gillerman et al, 2019). A microthermometry study of fluid inclusions quartz from epithermal veins in the Yellow Pine deposit by Marsh et al. (2016) produced homogenization temperatures that ranged from 175°C - 233°C.

There is some evidence for an additional metallic mineralization event consisting of dark, fine-grained quartz and pyrite microbreccia veins, which locally contains stibnite and scheelite crosscut the epithermal veins. This later scheelite and stibnite mineralization likely occurred at cooler temperatures estimated to be less than 180°C from δ18O values (Lewis (1984) and between a range of 189-160°C from fluid inclusions in stibnite (Marsh et al., 2016; Marsh et al., in preparation). The fluid inclusion data from stibnite (Marsh et al., 2016) indicates a depth of formation of ~ 2 km at ~45-50 Ma, thus documenting about 24 km of exhumation in approximately 50 million years between peak metamorphism and Tertiary intrusive activity.

The fifth alteration phase affecting the Stibnite district involved the deposition of post-mineralization carbonate veins and clay alteration of Eocene dikes, tentatively associated with cinnabar mineralization in the nearby mercury district as well as Au-Ag mineralization in the Thunder Mountain volcanics based on 45.8 +/- 0.3 Ma adularia ages. Late stage cinnabar mineralization in the eastern part of the district is associated with very low-temperature clays and may have occurred after oxidation of pyrite (Leonard, 1985). Remobilization of mercury under very low temperatures (<50°C) from original mineral matrix-bound mercury may have occurred, and some mercury mineralization may have occurred with higher temperature mineralization. This interpretation is consistent with observations from D-SIMS and petrographic analyses of West End samples for gold deportment studies (Surface Science Western, 2012) where mercury sulfides were found to be intergrown with iron oxides and texturally were later than, or at least coeval with, oxidation of pre-existing arsenical sulfides.

7.4 MINERALIZATION

Intrusive hosted precious metals mineralization typically occurs in structurally prepared zones in association with very fine-grained disseminated arsenical pyrite (FeS_2) and, to a lesser extent, arsenopyrite (FeAsS). Base metal sulfides are uncommon. Mineralogical studies of sulfide morphology and mineral chemistry were completed for metallurgical process flow sheet testing using x-ray diffraction (**XRD**), dynamic secondary ion mass spectrometry (**D-SIMS**), QEMSCAN®, mineral liberation analyzer (**MLA**), and petrographic studies (Palko and Martin, 2011a; Palko and Martin, 2011b; Palko, 2012). These studies, combined with past academic research (White, 1940; Cooper, 1951; Lewis, 1984; Cookro et al., 1987; Gillerman et al, 2019) indicate that there are multiple periods of pyrite development and associated precious metals mineralization. Arsenical pyrite is the primary host for gold mineralization and the vast majority of the gold occurs in solid solution within the sulfide crystal lattice. Arsenopyrite is the only other significant gold-bearing sulfide mineral in the intrusive hosted deposits. Gold rarely occurs as discrete sub-micron particles in pyrite and other sulfides. Base metals are rare and occur at very low concentrations, at or below typical crustal abundance levels. Various oxidized products of the weathering of the primary sulfides are found in the intrusives including goethite, hematite, jarosite, and scorodite, and host precious metal mineralization in the oxidized portions of the deposits.

Antimony mineralization occurs primarily in the form of the mineral stibnite (Sb_2S_3). Other antimony-bearing phases include miargyrite ($AgSbS_2$), gudmundite (FeSbS), chalcostibite ($CuSbS_2$), tetrahedrite [$(Cu, Fe)_{12}Sb_4S_{13}$], and owyheeite [$(Pb)_{10}(Ag)_{3-8}(Sb)_{11-16}(S)_{28}$]. There is a weak, but persistent association of volumetrically small base metal mineralization, typically <0.25%, associated with the antimony mineralization and includes rare occurrences of chalcopyrite ($CuFeS_2$), galena (PbS), sphalerite (ZnS) and molybdenite (MoS_2). Zones of high-grade, silver-rich mineralization locally occur with antimony and are related to the presence of pyrargyrite (Ag_3SbS_3), hessite (Ag_2Te), and acanthite (Ag_2S).

Tungsten mineralization is associated with the mineral scheelite ($CaWO_4$). Observations indicate that some of the tungsten is associated with the stibnite mineralization but may also precede it since stibnite has been found in numerous past studies cementing veins and brecciated scheelite fragments.

Although mercury mineralization is rare in the area of the three main deposits and in the west side of the District, studies of the mineral occurrences to the east in the Cinnabar district, where mercury was historically produced, indicate the primary mercury-bearing minerals are cinnabar (HgS), coloradoite (HgTe), and, to a lesser extent, tiemannite (HgSe) and amalgam (HgAg).

Metasediment-hosted mineralization has a similar sulfide suite and geochemistry, but with higher carbonate content in the gangue and a much more diverse suite of late stage minerals. As in the intrusive-hosted mineralization, gold is associated with very fine-grained arsenical pyrite and is tied up in the pyrite lattice. Rarely, submicron-sized native gold occurs as inclusions and along fractures and may be disseminated in highly fractured zones and may produce locally high grades and a minor nugget effect. Metallurgical test work completed by Midas Gold to date suggests around 20% of the gold in the West End metasediment-hosted mineralization may be particulate in nature, but extremely fine grained.

7.5 MINERALIZED ZONES

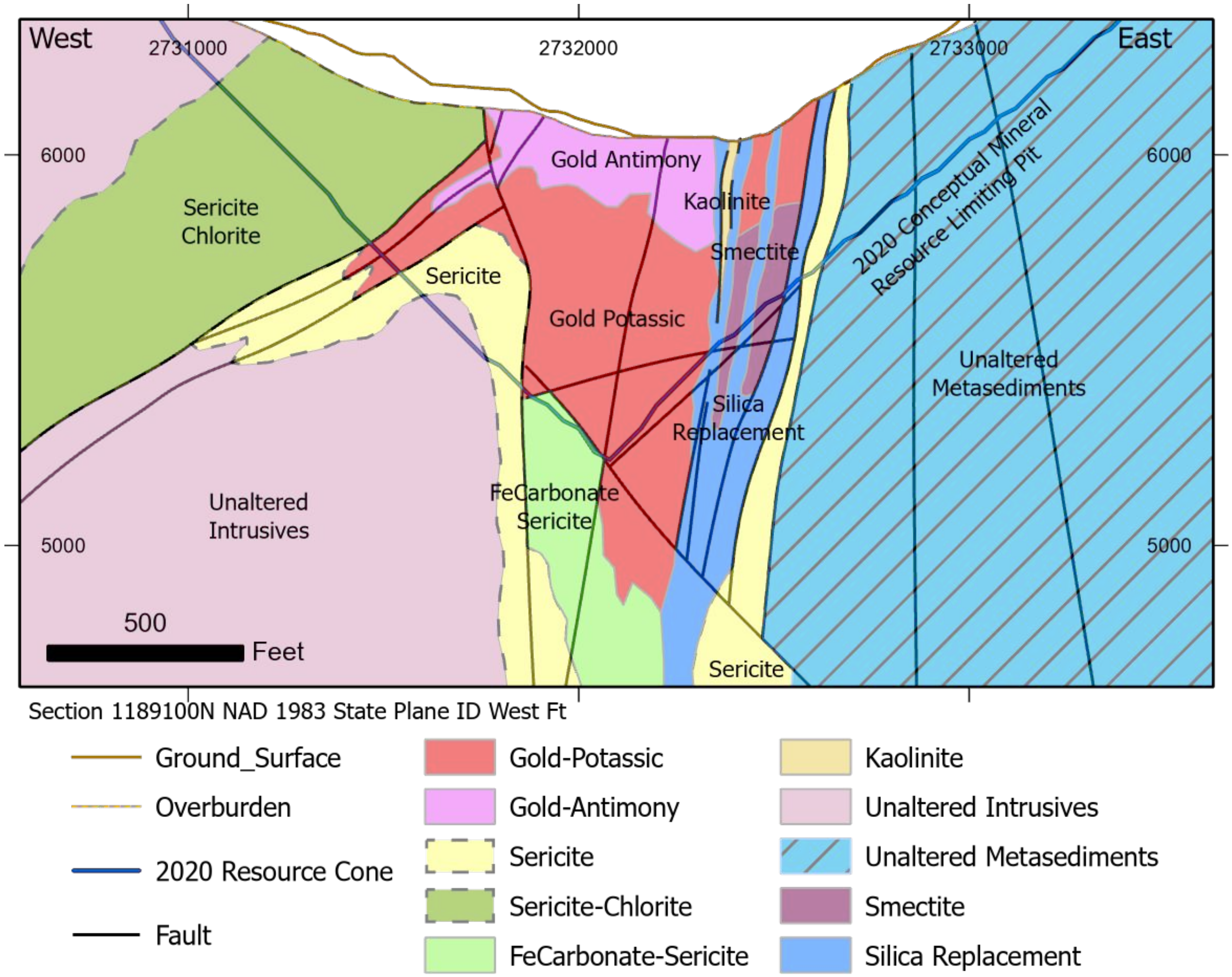
7.5.1 Yellow Pine Deposit

Mineralization in the Yellow Pine Deposit is structurally controlled and localized by the northerly striking MCFZ and by conjugate splay or cross structures associated with the MCFZ. The deposit shows zonation with gold mineralization occurring throughout the deposit footprint but with antimony and tungsten primarily in the central and southern portions of the deposit. The majority of mineralization in the deposit occurs west of the MCFZ and east of the Hidden fault zone. The geometry, width and continuity of precious metals mineralization changes along strike in the deposit in conjunction with a bend in the MCFZ and its intersection with the Hidden fault zone. To the south, gold and antimony mineralization occur within a D1b breccia zone of the MCFZ bounded to the east by post-mineralization gouge of the MCFZ and bounded to the west by the pre-gold mineralization D1a breccia zone. Here, the width of mineralization ranges from 80 ft to 165 ft, extends for over 800 feet along strike, and is open at depth.

In the central region of the deposit, between 1,188,200N and 1,189,600N, mineralization is broadly disseminated over a width of 500 feet east of the Hanging Wall fault and west of the post-mineralization Hennessey fault, except where Hennessey fault has offset the western part of the mineralization to the north (Figure 7-12). Gold and antimony mineralization in the central region of the deposit are bounded to the south by a complex fault network consisting of the C-structure, the Granite fault, and the northwesterly striking Midnight fault. The width of mineralization in Central Yellow Pine ranges from 165 ft to over 650 ft wide, over 1,400 feet of strike length and extends down dip over 1,200 ft.

Mineralization in the northern Homestake area of the Yellow Pine deposit ranges from 80 to 150 ft thick and extends for over 800 ft along strike and down dip. Here, mineralization occurs as a tabular body in the hanging wall of the Hidden fault/Clark Tunnel structure. The tabular zone steepens to the west, possibly due to down-dropping along post mineralization faults and is truncated to the west against the East Boundary fault, a gouge zone within the MCFZ. Directly east of the MCFZ gouge, is a silicified fault corridor which is moderately mineralized in the Homestake area. Gold mineralization also occurs within the metasediments at Homestake, where both disseminated and vein-hosted gold occurs within the upper Calc-Silicate and Middle Marble formations.

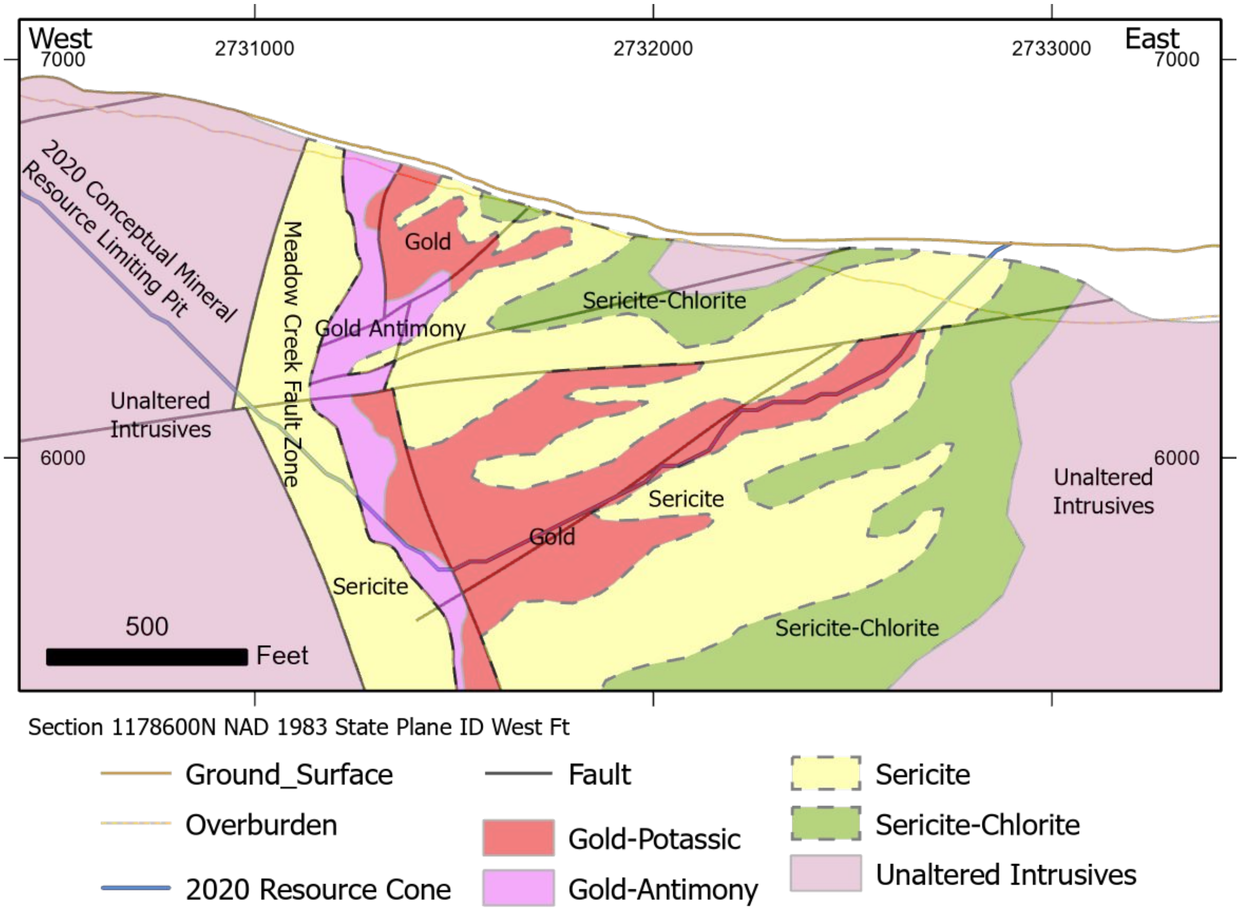
Figure 7-12: Yellow Pine Mineralized Zone and Generalized Alteration Zonation



7.5.2 Hangar Flats Deposit

Mineralization in the Hangar Flats Deposit is entirely intrusive hosted and is localized in and along the flanks of the MCFZ. The highest grades of gold, silver, and antimony, defined on the basis of drilling and legacy production, occur within sub-vertical, north-plunging, tabular to pipe-like breccia bodies formed at the intersection of the main north-south structural features and shallowly northwest-dipping dilatant splay structures. These mineralized breccia zones range from 16 ft to over 330 ft in true thickness and can be traced several hundred feet down dip. Disseminated replacement style gold mineralization occurs throughout the MCFZ and eastern footwall encompassing the higher-grade tabular breccia zones. Disseminated gold mineralization also occurs as shallowly dipping tabular bodies along the northwest dipping splay structures which pinch out to the east away from the main MCFZ (Figure 7-13).

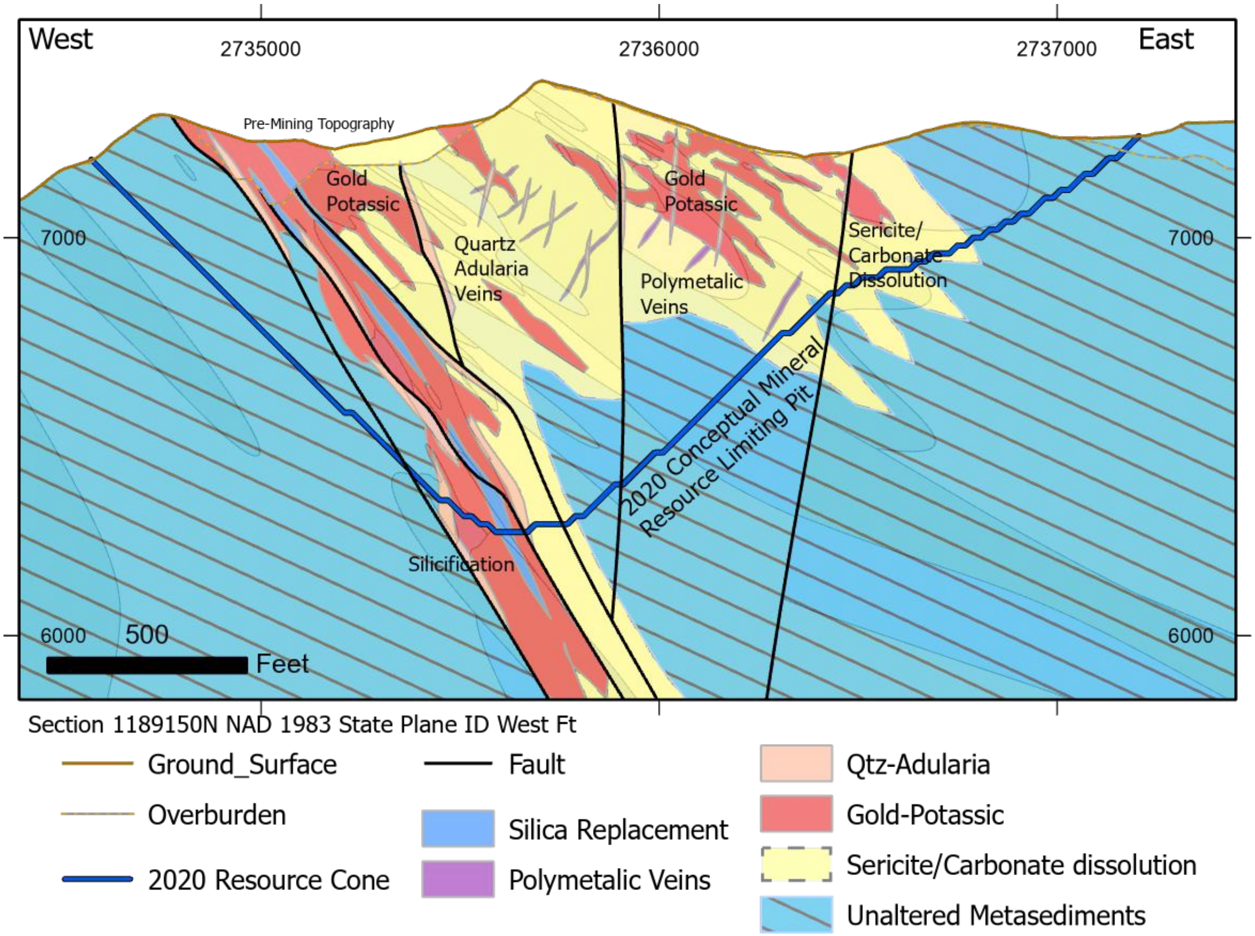
Figure 7-13: Hangar Flats Mineralized Zone and Generalized Alteration Zonation



7.5.3 West End Deposit

Mineralization in the West End Deposit is structurally and stratigraphically controlled. Within the WEFZ, gold mineralization occurs within silicified breccia zones and as replacement style mineralization where the northeast-dipping calc-silicate and schistose units are sheared and offset by the structure. Outside of the WEFZ, mineralized zones occur as stacked ellipsoidal bodies plunging along the intersection of favorable lithologic units and structural zones and as tabular bodies extending along bedding (Figure 7-14). Mineralization also occurs as fracture filling within siliciclastic sequences and other less favorable lithologic units. True widths of these bodies range from 50 ft to over 330 ft. Drilling by Midas Gold has intersected gold mineralization associated with the WEFZ well below the historic pit bottom – as deep as 1,300 ft below the original ground surface - where mineralization was exposed prior to mining. The hanging wall of the WEFZ tends to exhibit relatively more dilatant and dispersed structures relative to the footwall and, therefore, is more significantly mineralized. Open-space-fill quartz veins and silicified breccias are typical within higher grade zones of mineralization. Degree of oxidation in the West End Deposit is a function of both depth and proximity to faults and fractures. Both pervasive and fracture hosted oxidation is common throughout the deposit to depths of approximately 300 ft below the pre-mining topographic surface. Discrete zones of pervasive oxidation occur below this depth in the vicinity of the WEFZ and subsidiary structures. Oxidation is interpreted to have resulted from both infiltrating precipitation and from deep-seated circulation of meteoric fluids through structural zones.

Figure 7-14: West End Mineralized Zone and Generalized Alteration Zonation



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8 DEPOSIT TYPES

8.1 DEPOSIT MODELS

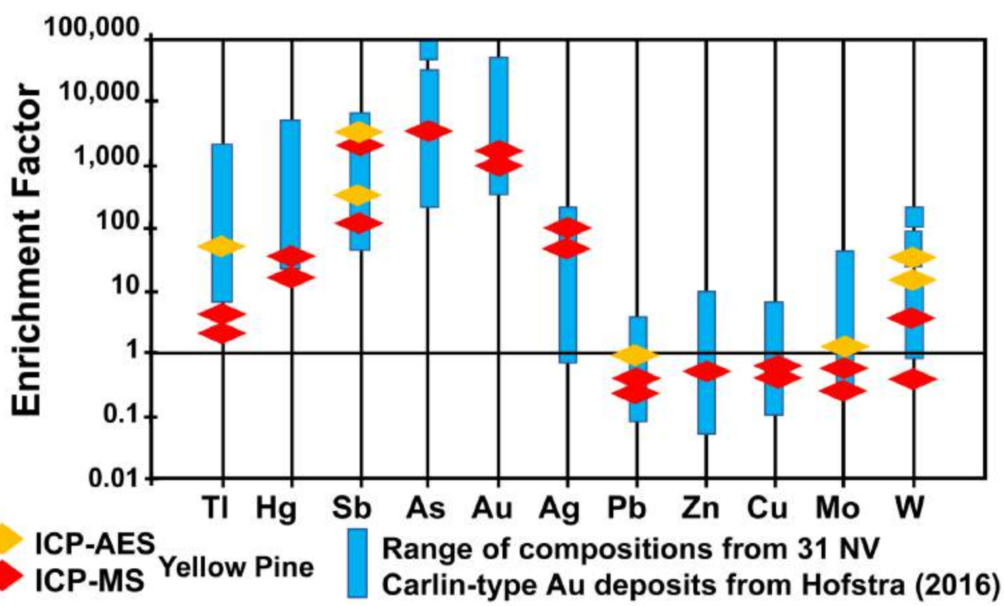
Gold-antimony-silver-tungsten deposits of the Stibnite Mining District (the **District**) are not readily categorized based on a single genetic deposit model due to complexities associated with multiple overprinting mineralization events and uncertainties regarding sources of mineralizing hydrothermal fluids. Early workers attributed the mineralization to the Idaho Batholith (Schrader and Ross, 1926); to hot springs associated with igneous intrusions (Thomson, 1919); to the Thunder Mountain caldera (Larsen and Livingston, 1920); to Tertiary dikes and small stocks (Bell, 1918; Thomson, 1919); or to both the batholith (gold and antimony) and the volcanics (mercury) (Currier 1935). Workers in the early 1970s considered some of the mineralization to be similar in style to deposits in the Yellow Jacket Co-Cu-Au belt farther east and attributed the precious metal mineralization to iron formations associated with what were interpreted as metavolcanics rocks (Jayne, 1977). Cookro (1985) attributed the tungsten to Cretaceous skarns. Cookro et al. (1987) noted isotopic signatures that suggested an igneous or metamorphic origin likely of Late Cretaceous age but also noted the potential for overprinting Tertiary mineralization. Criss et al. (1983; 1991) noted associations between Tertiary intrusions and meteoric dominated epithermal systems including Yellow Pine. Bookstrom et al. (1998) attributed the various metals in the District to a variety of deposit types including distal disseminated gold, Au-Ag and mixed metal veins, simple antimony veins, disseminated antimony, quartz-scheelite veins and breccia deposits, mixed metal skarns and hot springs mercury. Konyshev (2020) noted similarities to the reduced intrusive systems in the Tintina Belt, specially Donlin Creek. Others have noted similarities to Carlin-type systems and reduced intrusion gold deposits (Dail et al., 2015; Dail, 2016; Hofstra et al., 2016) and orogenic gold to antimony-gold bearing Carlin-like systems in China (Dail, 2014; Gillerman et al., 2019b). The complicated paragenesis and prolonged extent of mineralizing events in the area spanning tens of millions of years preclude application of a single genetic model.

Within the Project area, the focus of past exploration for, and development of, Au-Ag-Sb-W-Hg deposits has been from both disseminated deposits extracted using conventional open pit methods and higher grade structurally controlled Au-Sb, W-Sb, Hg and Au-only deposits extracted using various underground mining methods. Mineralization occurs in numerous locations throughout the District in medium- to coarse-grained, felsic to intermediate intrusive host rocks and typically occurs as disseminated replacement mineralization within structurally prepared dilatant zones or adjacent to district- and regional-scale fault zones. Mineralization also occurs in association with sheeted veins, stockworks, endoskarns, and complex polymictic breccias. In the metamorphosed sedimentary rocks, mineralization occurs in association with dense fracture zones in structurally prepared sites and as stratiform manto-style replacements in reactive carbonate and calcareous siltite and schist units, as well as in cross-cutting breccia veins and dikes and jasperoids (quartz-replaced carbonates).

Field observations, petrographic studies, geochronology, and metallurgical studies, indicate that there were multiple stages of mineralization separated by extended time periods, as discussed in Section 7. A generalized model for the earliest disseminated gold-arsenic replacement mineralization event could involve assimilation of reduced metals-enriched black shales in ascending magmas with subsequent differentiation of metal enriched volatile phases and passage of those fluids into the shallow crustal environment along regional, deep seated structures. In southeast Idaho, Hall et al. (1978) and McIntyre et al. (1976) noted scavenging of metals from Paleozoic rocks by magmatic and meteoric fluids associated with both Cretaceous and Tertiary granites in a 145-km long by 15-45-km wide belt of metalliferous Neoproterozoic and Paleozoic units known as the Idaho Black Shale Belt. These rocks are not present in the District but do occur directly along strike and may be present at depth beneath the District (Dail, 2015; Gillerman et al., 2019; Wintzer, 2019). The Bayhorse area stratigraphic succession, which includes the Idaho Black Shale Belt, is interpreted to be at least partly correlative to the Paleozoic sediment package at Stibnite (Yonkee et al., 2015; Lewis et al., 2014) and Neoproterozoic to Ordovician carbonate and siliciclastic sequences in north Idaho and eastern Washington also may be correlative. Geochemical and isotopic associations imply hydrothermal cells scavenged at least some metals from older strata not exposed in the District or immediate area including some with derivation from Archean crust or protoliths. Gillerman et al., (2014; 2019) reported Pb isotopic values from Stibnite ore minerals that included a component derived from Archean crustal sources. Wintzer (2019) compared the common lead signature of rocks and ores in the metalliferous Black Shale sequence in southeast Idaho to ores and minerals in the District and there is a close correlation providing evidence for magmatic assimilation and/or deep circulation of hydrothermal fluids to deep crustal levels where rocks with these lead isotopic signatures may be present. Taylor et al. (2007), using strontium and neodymium isotope data, reported similarities between southeast Idaho Neoproterozoic to Paleozoic sediment isotopic signatures to the Atlanta lobe of the batholith, inferring these sediments were assimilated into the batholith. In the Idaho Panhandle, Rosenberg and Wilkie (2016) reported an isotopic link between Late Cretaceous-Early Paleocene hydrothermal systems and buried Archean crust in in Snowbird-type fluorite deposits contemporaneous with extensional faulting and intrusion of the Bitterroot lobe of the Idaho Batholith, suggesting assimilation of the shale belt may have occurred along much of the length of the Cretaceous accretionary margin.

Isotopic and petrochemical characteristics that suggest hydrothermal fluids may have been sourced from reduced magmas that incorporated older metasedimentary rocks during crustal ascension (Gillerman et al, 2019). However, there are no known intrusive rocks of the same age in the District or area. Fluid chemistry, mineralogy, timing and tectonic setting of this mineralization event is consistent with the gold deposition mechanism proposed by Muntean et al. (2011) for formation of Carlin-type gold deposits (**CTGD**s), in which magmas formed due to asthenospheric upwelling during Tertiary slab delamination and underwent a mid-crustal fractionation process preferentially incorporating copper into a monosulfide solid solution and generating residual gold-rich magmas. Fluids resulting from volatile saturation during magma ascent underwent additional segregation in which gold, sulfur and arsenic are concentrated in the vapor phase and iron partitions into the brine phase allowing significant mass transport of gold in vapor while precluding pyrite precipitation. Subsequent mixing of the vapor with meteoric water, reaction of acidic gold-rich fluids with carbonate minerals and scavenging of iron from host rocks results in deposition of gold in rimmed arsenian pyrite and arsenopyrite over broad regions of disseminated mineralization characteristic of both Carlin-type and Stibnite deposits. Host rock lithologies differ from CTGDs, but tectonic setting on the passive Paleozoic margin, absence of causative intrusions, fluid chemistry, depth of formation, overall geochemical relationships (Figure 8-1), transtensional to extensional structural associations and timing of mineralization relative to Laramide slab delamination are compellingly similar.

Figure 8-1: Geochemistry of CTGD Deposits Compared to SGP-Area Deposits

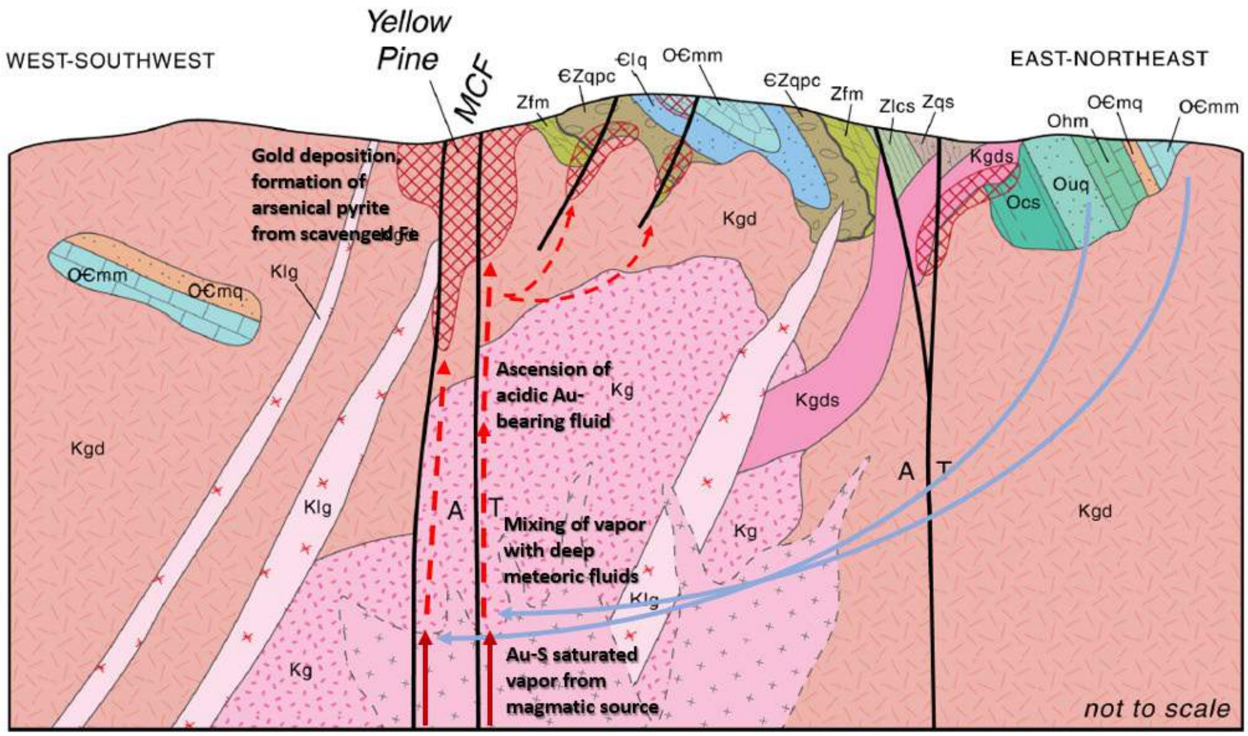


Source: Modified from Hofstra, A. 2016

The earlier gold-arsenic mineralization event was overprinted by younger, lower temperature Sb-W mineralization and, subsequently, epithermal gold mineralization associated with quartz-adularia-carbonate veins. The Sb-W deposits of the Stibnite Mining District share similarities with other Au-Sb-W deposits in Spain, Portugal, Bolivia and China, as described by Dail (2014; 2016) and Gillerman et al. (2019) to include associations with major shear zones, Paleozoic host rocks (especially carbonate sequences), quartz and carbonate gangue mineralogy, and low temperatures of formation. Based on similar ages, the epithermal vein mineralization, and possibly the Sb-W mineralization, resulted from circulation of meteoric fluids driven by shallow Eocene intrusions in an extensional environment.

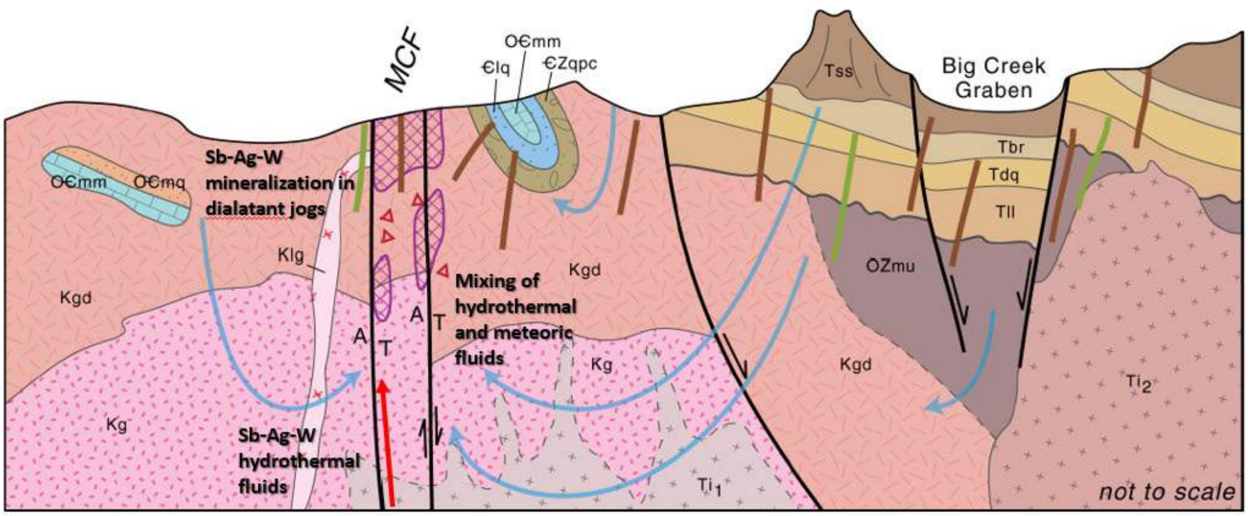
A schematic of the geologic setting for the various deposits and exploration prospects is shown on Figure 8-2 to Figure 8-4. These figures (modified from Gillerman et al. (2019b) illustrate the spatial relationships of each major deposit type, the intrusion(s), and the associated hydrothermal systems.

Figure 8-2: Main Stage Gold Mineralization (70-65 Ma)



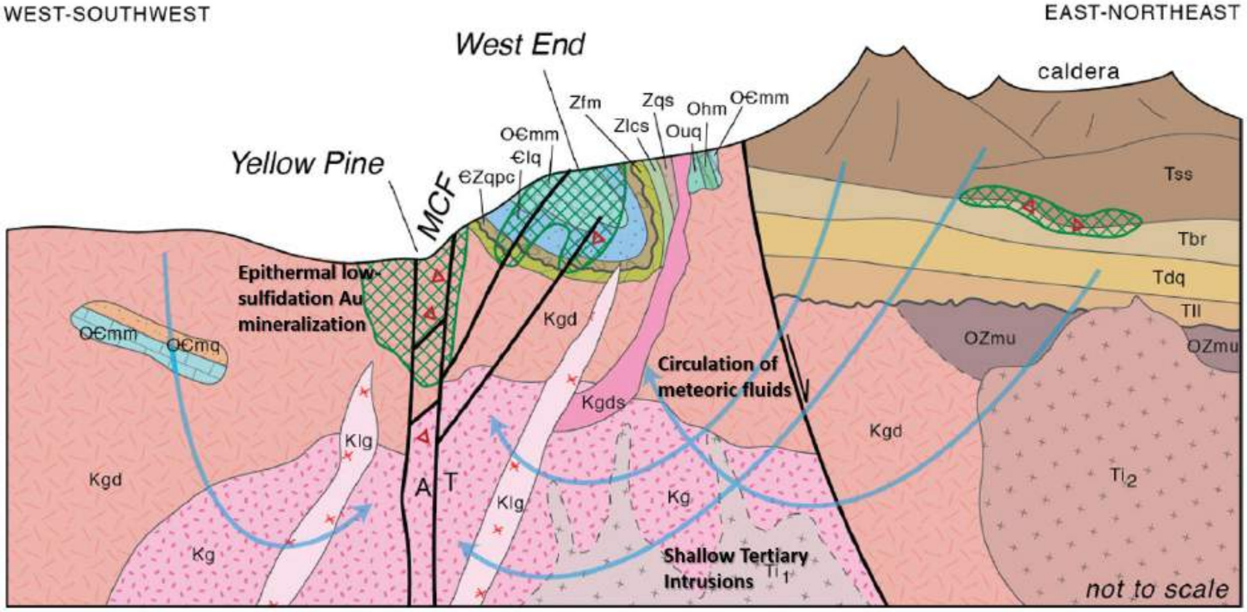
Source: Modified from Gillerman et al, 2019b

Figure 8-3: Antimony-Tungsten Mineralization



Source: Modified from Gillerman et al, 2019b

Figure 8-4: Epithermal Gold Mineralization Stage (~50-38 Ma)



Source: Modified from Gillerman et al, 2019b

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9 EXPLORATION

9.1 EXPLORATION POTENTIAL

Numerous prospects have been discovered during exploration and development activities in the Stibnite Mining District (District) over the past 100 years using a variety of methods; some of these prospects were developed into mines while others remain undeveloped. Besides pit expansion possibilities around and below the three main deposits (Yellow Pine, Hangar Flats, and West End), other exploration targets may one day warrant consideration for development if they can be proven viable after additional exploration, environmental, socio-economic, metallurgical, engineering, and other appropriate studies and following any required permitting. Midas Gold has developed an extensive pipeline of over 70 discrete high-potential exploration targets within the core of the District, but much of the District and land position is poorly explored even today after over 100 years of activity in the area.

The exploration targets discussed in this section include: pit expansion opportunities along strike and/or down-dip from known deposits; targets adjacent to proposed open pits; high-grade prospects with potential for discovery of deposits amenable to underground development; prospects along favorable structural or stratigraphic trends;untested or inadequately tested geochemical and geophysical targets; and conceptual targets with limited support. Several of the targets discussed are advanced prospects that have had past production and/or adequate drilling to infer good potential for discovery of new open pit or underground deposits but do not currently have developed mineral resources.

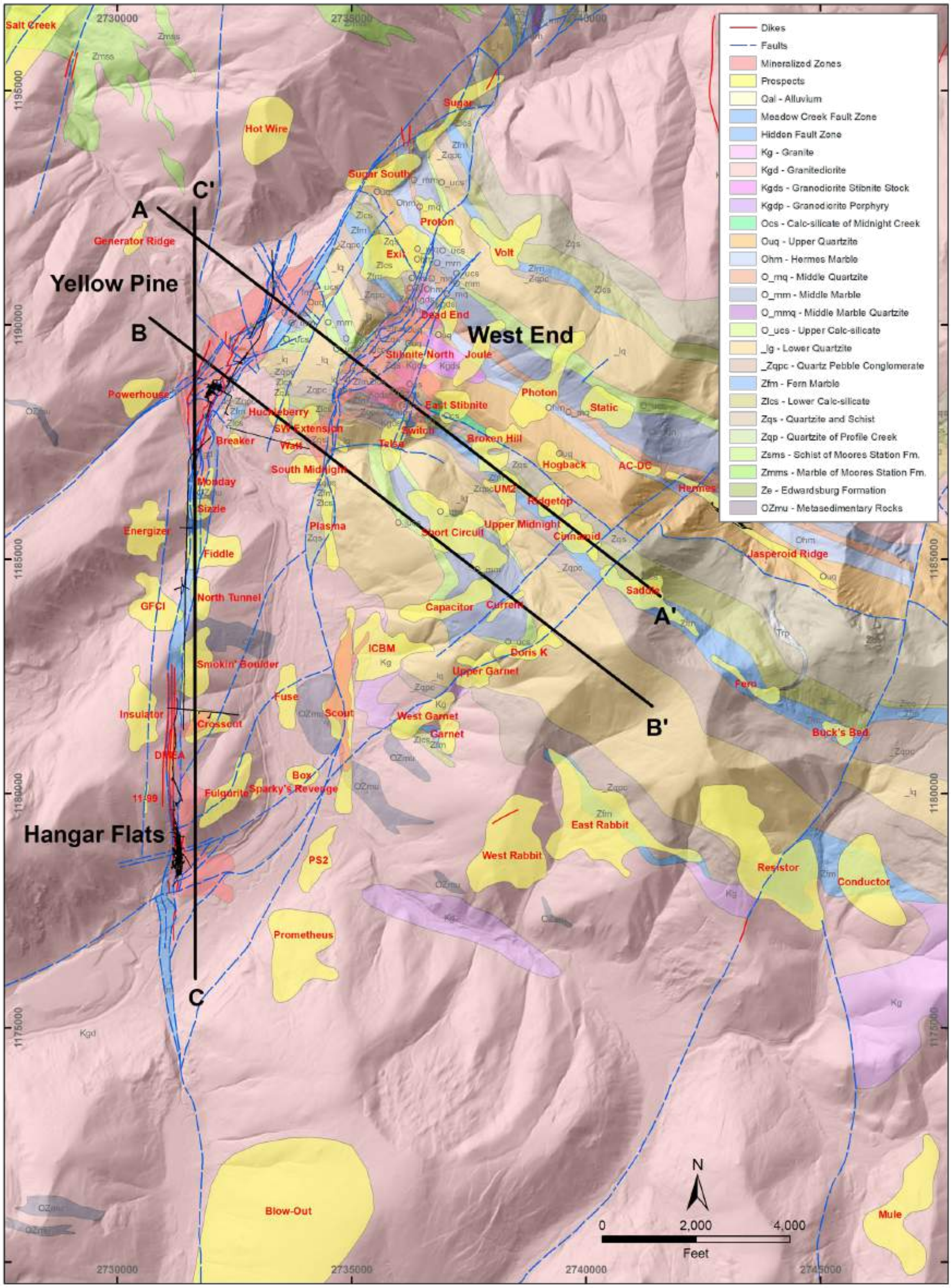
Some of the more significant prospects are summarized below and shown on a simplified geologic map (Figure 9-1) modified from Stewart et al. (2016). Conceptualized cross sections through the east side and west sides of the District are provided as Figure 9-2. See Section 7 for details on the District geological setting and Section 8 for additional details on deposit types and models.

Exploration data for the target areas discussed in this section include geophysical data; geochemistry from soil, rock, and trench samples; and results from widely spaced drill holes. As a result, the potential size and tenor of the targets are conceptual. There has been insufficient exploration to define mineral resources on these prospects and these data may not be indicative of the occurrence of a mineral deposit. Such results do not assure that further work will establish sufficient grade, continuity, metallurgical characteristics, and economic potential to be classed as a category of mineral resource. Some of the targets include areas with inferred mineral resources. Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

9.2 GEOLOGIC MAPPING

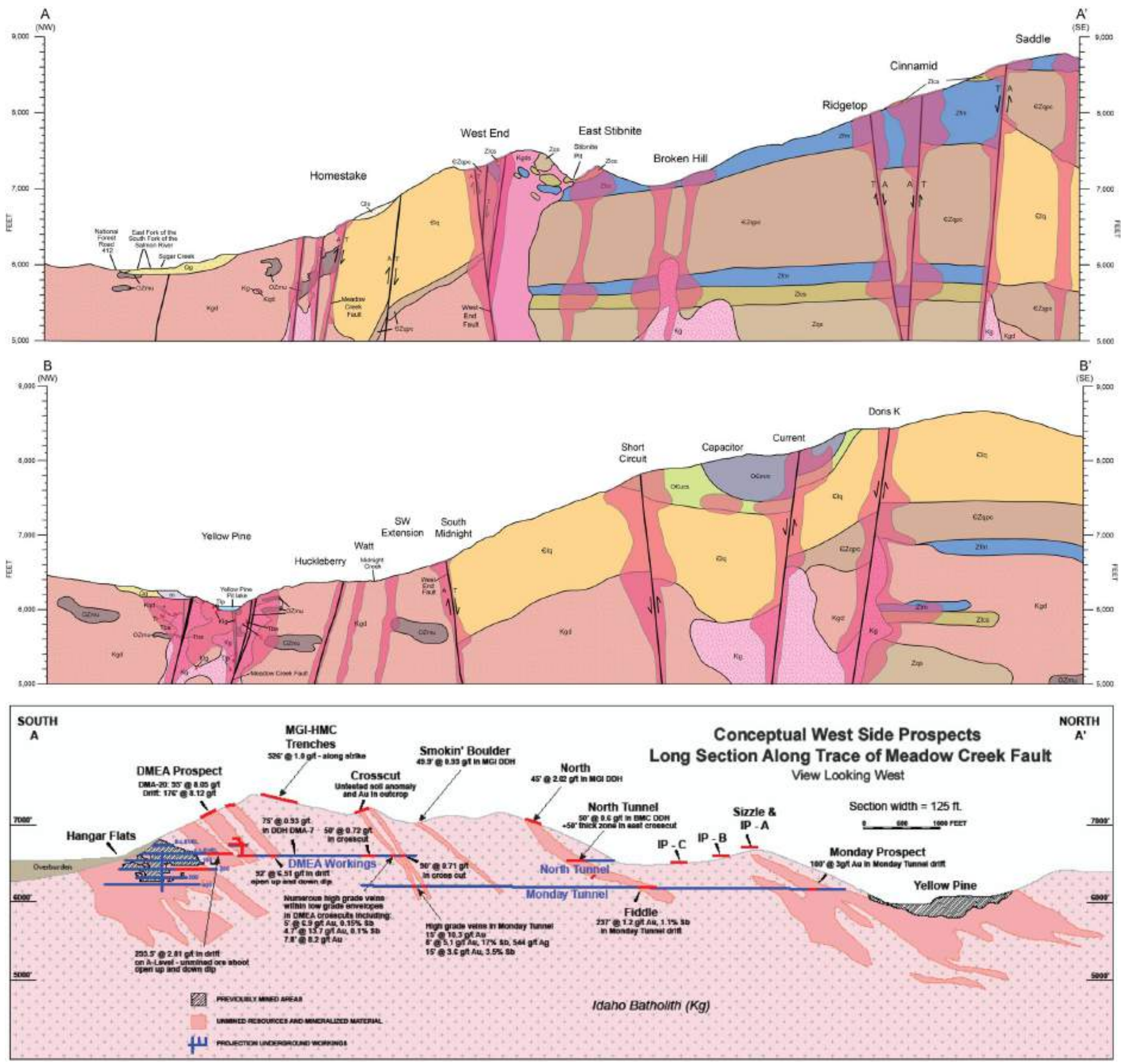
The SGP area has been mapped by numerous past workers. Midas Gold staff have remapped areas, where needed, to obtain additional information. A generalized 1:24,000 map of the district modified from Stewart et al. (2016) prepared under a contract with the Idaho Geological Survey showing prospects is provided as Figure 9-1. See Section 7 of this report for additional details of geology.

Figure 9-1: Prospects and Conceptual Long Sections Sections on Generalized Geology



Note: Geology modified from Stewart et al, 2016. Grid: 1983 Idaho State Plane West (feet)

Figure 9-2: East Side and West Side Long Sections through the District



Note: Cross sections modified from Wintzer, 2019 and Midas Gold. Unit name abbreviations and color scheme same as Figure 9-1. Red hatch areas represent conceptualized zones of alteration and mineralization.

9.3 GRIDS AND SURVEYS

Numerous local grids have been used on-site since the 1920s and historical survey control points have been re-established where possible and practical and tied to modern coordinate system datum and projections. See Section 10 for details on legacy and Midas Gold survey control and surveys.

9.4 GEOCHEMICAL SAMPLING

Numerous geochemical studies have been conducted in the District (Bannister, 1970; Leonard, 1973; and Curtin et al., 1974; Erdman et al., 1985). Past operators collected and analyzed tens of thousands of soil, rock chip, underground channel, surface chip, trench, and drill hole samples utilizing a variety of laboratories and methods. Not all sample information is fully documented with chain of custody, preparation and laboratory analytical methods, lower and upper detection limits, and/or QA/QC. However, the bulk of the geochemical data are considered reliable enough to utilize for basic exploration purposes and data thought to be unreliable is either updated with new information or excluded in the discussions that follow. Midas Gold collected additional stream sediment, soil, and rock samples using current industry-standard protocols, chain of custody procedures, and certified analytical laboratories.

9.4.1 Stream Sediment Surveys

9.4.1.1 Historical Sampling Programs

Summaries of stream sediment geochemistry at the regional- and district-scale include contributions by McDanal et al. (1984), McHugh et al. (1993), Hopkins et al. (1996), Watts and King (1998), and references therein. These data have been critical in the assessment of the District for exploration, development, and environmental purposes. Details of the public domain surveys can be obtained from the applicable documents. Within the SGP area, there are limited legacy industry stream sediment sampling data available, and results have been integrated with Midas Gold results were appropriate and the data deemed reliable.

9.4.1.2 Midas Gold Stream Sediment Surveys: Design and Methods

Midas Gold completed minus 80 mesh stream sediment surveys including collection of field parameters (eH, pH, temperature, dissolved oxygen, flow, etc.) in a high-density survey covering approximately 50 mi^2 to supplement existing surveys. The average catchment size from Midas Gold surveys resulted in a density of approximately one sample per 0.13 mi^2. The surveys were used: 1) to define the limits of potential mineralization for land acquisition; 2) to prioritize airborne geophysical anomalies for follow-up; 3) to establish natural background for permitting; 4) define lithogeochemical characteristics in areas with known geology to assist in determining the potential locations of favorable host rocks in unmapped or covered areas; and 5) to assess anthropogenic impacts from historical mining and processing.

Stream sediment samples were field sieved to -10 mesh (0.0787 inches) or sampled in bulk and transported to certified analytical laboratories with chain of custody procedures. Samples were air dried to minimize loss of volatiles and mercury and then sieved through an 80 mesh sieve. The material passing through the sieve was split, pulverized, and digested with aqua regia then analyzed by inductively coupled plasma atomic emission spectroscopy (ICP-AES) and inductively coupled plasma mass spectrometry (ICP-MS) using ALS Chemex Method ME-MS41L. Mercury was analyzed by the cold vapor method and gold by conventional fire assay (FA) and Atomic Absorption (AA) finish methods. The aqua regia digestion, while not complete, is appropriate for this type of survey and places most ore minerals and pathfinder elements into solution. Routine field and laboratory duplicates and laboratory standards and blanks were also analyzed for quality control purposes and found to be within acceptable limits. Results from the surveys were then analyzed using basic statistical and graphical methods to determine areas of anomalous geochemistry for further follow-up and prospecting. Lower and upper instrumental reporting limits for the various methods can be found on the ALS Chemex website at (https://www.alsglobal.com/en-us/services-and-products/geochemistry/geochemistry-downloads/ALS Geochemistry Fee Schedule USD.pdf accessed 8/2/2020).

9.4.1.3 Midas Gold Stream Sediment Surveys Results

The stream sediment surveys provided valuable data to meet early project assessment exploration and geophysical survey screening. A distinctive Au-Ag-As-Sb-Tl-W-Hg pathfinder suite "*bulls-eye*" marks the core of the District (Figure 9-3). Compared to average crustal abundances, base metals are depleted typical of reduced intrusive realted gold deposits. Incompatible elements, those that tend to evolve into late stage magmatic and hydrothermal fluids such as P, Y, Ce, La, and Nb show a marked increase in a ring around the roof pendant possibly reflecting the complex and highly evolved Cretaceous leucogranite intrusive phases common in the District. These incompatible elements tend to concentrate in evolved magma melt phases and associated hydrothermal fluids. The terranes drained by metasedimentary rocks have a distinctive Fe-Co-Ni-Cu-Mg-Ca stream sediment signature and surface water draining these areas generally have higher conductivity reflecting their differing chemical composition than the igneous terrane. The spatial distribution patterns and elemental associations in the stream sediment sampling data are similar to those found in bedrock (outcrop and drill hole data) and soil geochemical data from the district. Numerous stream sediment anomalies remain to be investigated.

9.4.2 Historical and Midas Gold Soil and Rock Chip Sampling Surveys

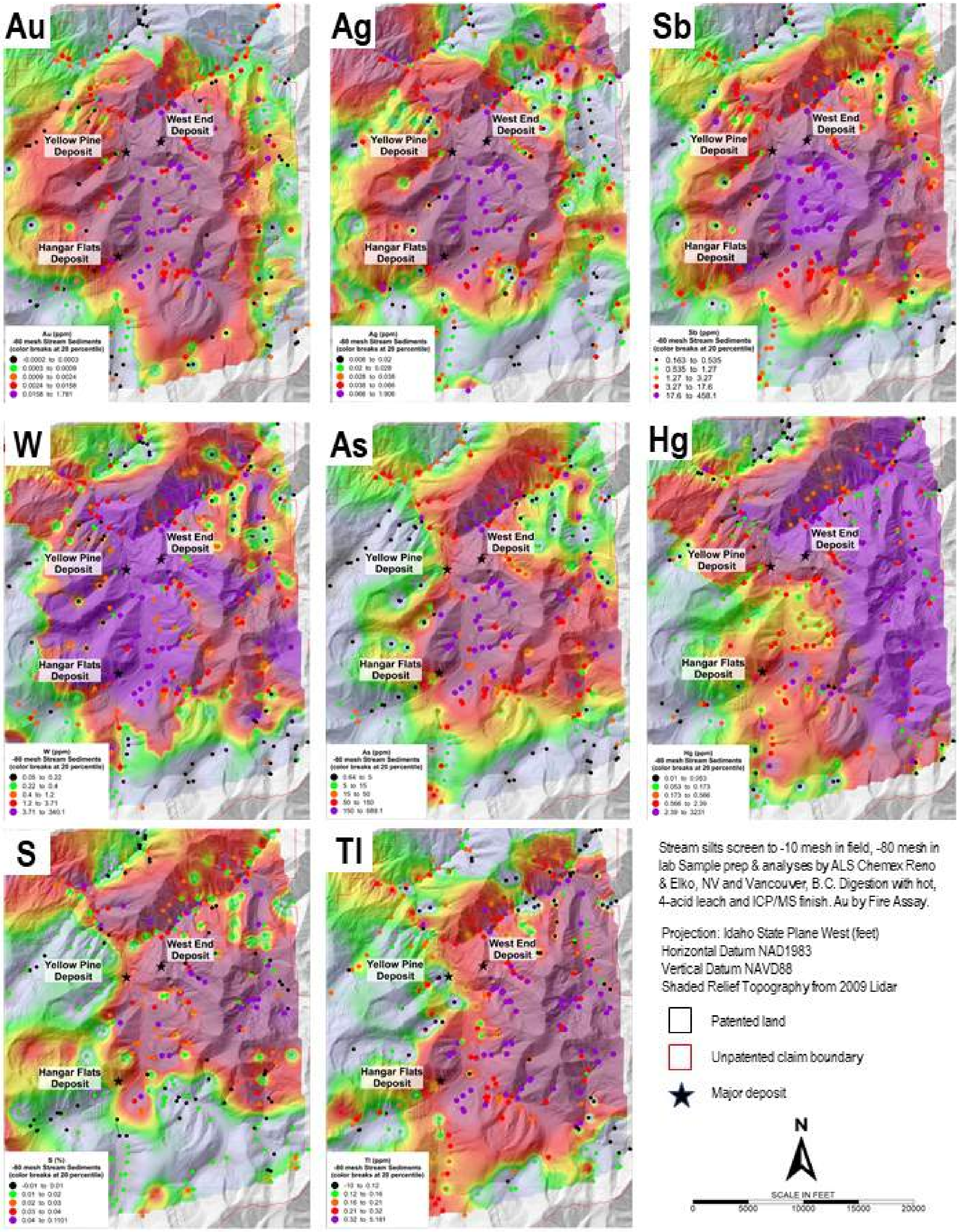
9.4.2.1 Survey Design and Methods

Over 5,000 historical soil samples were collected by previous operators were digitally captured from paper files. Past operators collected mull (forest floor litter), B-horizon and C-horizon samples on grids, contour and base of slope soil sampling lines, and rock chips from roadcuts. In addition, over 5,000 rock samples were taken from extensive 1970s to 1990s haul roads, trenches, and outcrops. Much of the detail for many of the historical soil and rock sampling surveys has been lost, and only summary reports and maps are available. Past soil and rock surveys used a variety of analytical methods and in some cases the data are of limited value due to high lower detection limits associated with the instrumentation of the era. Much of the 1970s to 1990s rock chip samples were analyzed for cyanide-soluble gold only, which can underestimate the total gold content due to lack of significant oxidation.

Midas Gold soil surveys included the collection of approximately 5,000 soil samples utilizing a minus-80-mesh fraction and an aqua regia digestion. Samples were analyzed by ICP-AES and ICP-MS using ALS Chemex Method ME-MS41L. In some areas, a sample split was also digested with a 4-acid prep and analyzed by ALS Chemex Method ME-MS61. The aqua regia digestion, while not complete, is appropriate for this type of survey and places most ore minerals and pathfinder elements into solution. The 4-acid digestion and lower detection limits in the MS-61 method were utilized in areas where substantial glacial cover was noted to pick up low concentrations of pathfinder elements associated with hydromorphic dispersion from springs and seeps through the unmineralized glacial cover. Where sites consisted of talus or sub-cropping regolith they were typically handled and treated as rock samples. Lower and upper instrumental reporting limits for the various elements can be found on the ALS Chemex website (https://www.alsglobal.com/en-us/services-and-products/geochemistry/geochemistry-downloads/ALS Geochemistry Fee Schedule USD.pdf accessed 8/2/2020).

Rock sampling included collection of approximately representative 2,500 samples of mineralized and unmineralized materials utilizing channel, chip, panel, and select samples depending on the site and outcrop characteristics. Typically, if veining or areas of concentrated mineralized were observed, the highly mineralized and/or veined material was sampled separately from the host rock to determine the host for mineralization and grade distribution. Rock sampling utilized the same analytical methods and protocols as the Midas Gold drilling program as described in Section 10. Routine field and laboratory duplicates, laboratory standards, and blanks were also analyzed for quality control purposes. Results from the surveys were analyzed using basic statistical and graphical methods to determine areas of anomalous geochemistry for further follow-up and prospecting.

Figure 9-3: Gridded -80 mesh Stream Sediment Pathfinder Element Geochemical Plot

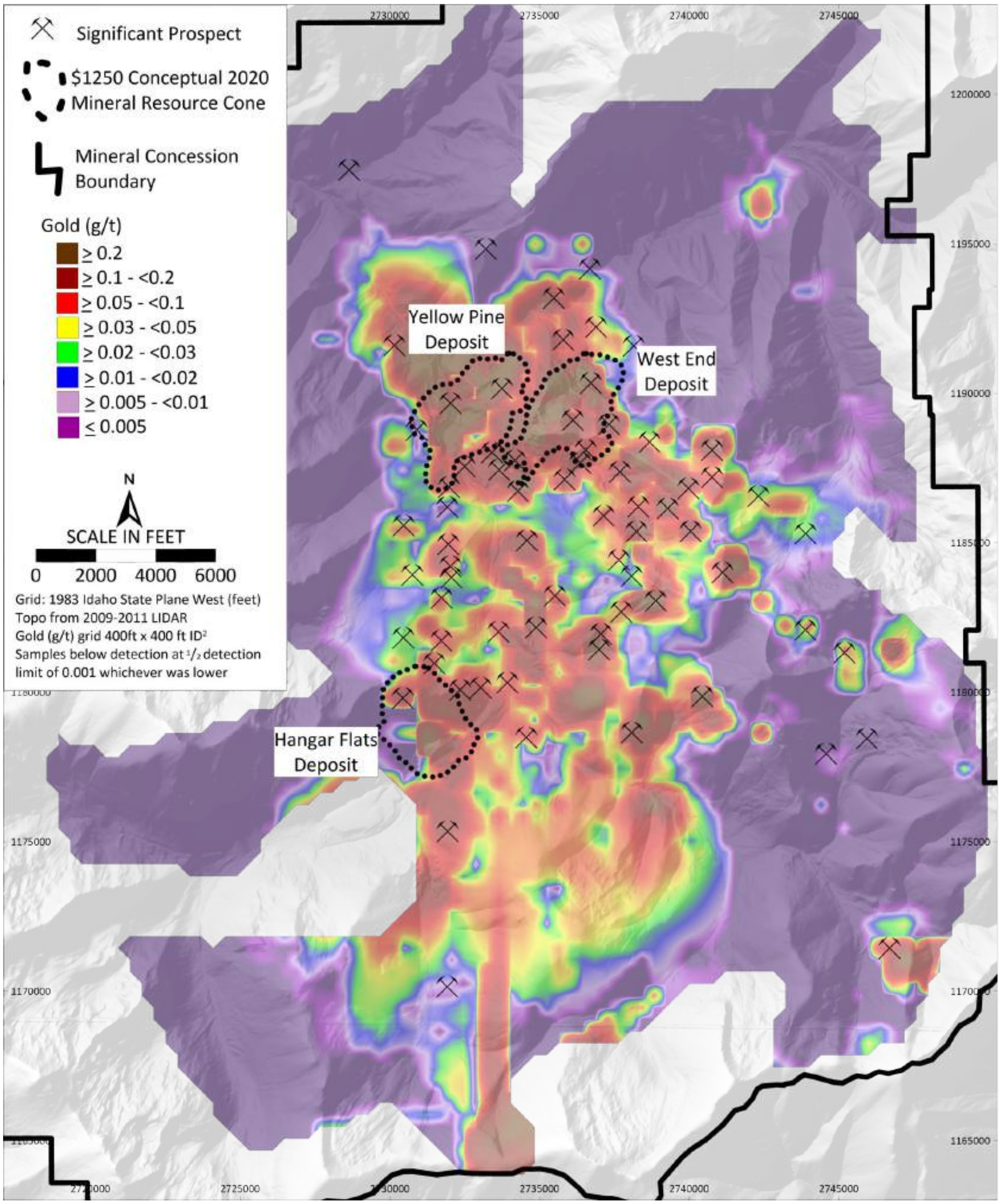


Note: Data from Midas Gold surveys 2009-2015. Samples results gridded and smoothed on 300 ft x 300 ft ID² grid.

9.4.2.2 Soil and Rock Sampling Survey Results

Soil survey and rock chip sampling results mimic and refine data identified in the coarser stream sediment surveys. Generally, besides gold itself, silver, arsenic, and to a lesser extent antimony, thallium, and mercury are the best pathfinders in soils for bedrock gold mineralization. A 400 ft x 400 ft gridded image of gold in soils, rocks, and drill hole samples (projected vertically to the surface) is provided as Figure 9-4.

Figure 9-4: Gridded Gold in Soils, Rocks and Drill Holes



Note: Data gridded on 400 ft x 400 ft grid cells with drill data composites projected to surface.

9.5 GEOPHYSICS

9.5.1 Historical Surveys

Regional geophysical compilations that cover the SGP area include work reported in Mabey and Webring (1985), McCafferty (1992), Rodriguez et al. (1996), and Kleinkopf (1998). A compilation of industry aeromagnetic and electromagnetic data, including reprocessing and reinterpretation of the regional datasets, is underway (Anderson et al., in preparation). A map of all available reliable legacy and Midas Gold ground geophysical survey data is provided as Figure 9-5. These data supplement airborne data coverage described in Section 9.5.2. Detailed geophysical surveys are limited to unpublished industry data (Bar, 1990; Nye, 1990). Ground geophysical survey methods proven valuable in targeting mineralization and structures include Very Low-Frequency Electromagnetics (**VLF**), Self-Potential (**SP**), time- and frequency-domain induced polarization (**IP**), and Controlled Source Audio-frequency Magnetotellurics (**CSAMT**). These systems have had variable success detecting faults, alteration, and mineralization and in several cases have led to drilling and discovery of blind mineralization beneath unmineralized cover materials.

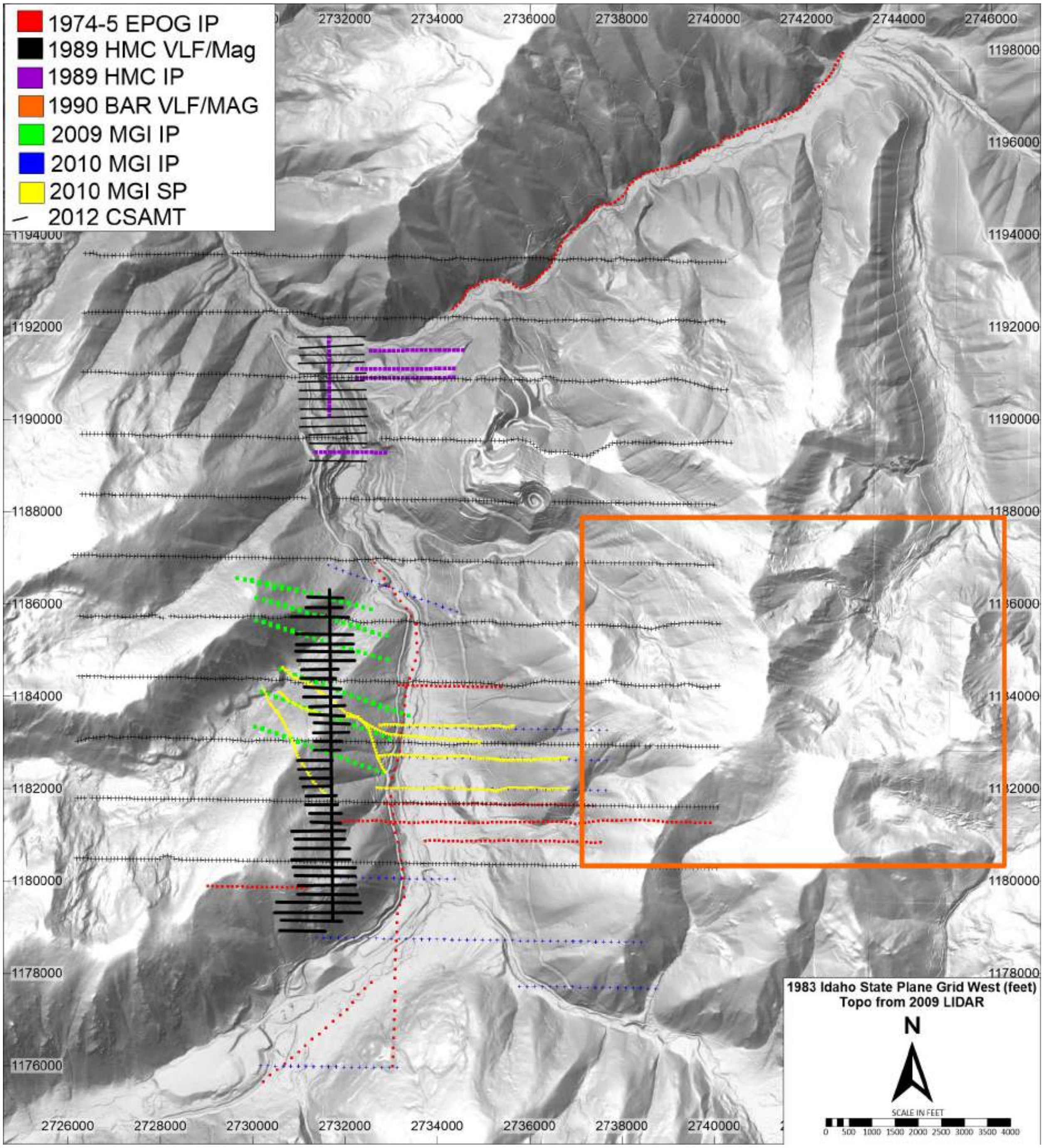
9.5.2 Midas Gold Surveys

Midas Gold contractor Fugro completed a helicopter-supported, 222 line-mile aeromagnetics survey in 2009, covering 33 mi^2, followed by a more detailed 595 line-mile airborne electromagnetic (**EM**) and aeromagnetics survey covering a larger area in 2011. The data were filtered, gridded, post-processed, and integrated with geologic and geochemical data to generate and evaluate target areas. Midas Gold's work included induced IP along 13 lines, totaling 13 line-miles; SP surveys along 6 lines totaling 4.23 line-miles; and CSAMT along 13 lines totaling 31 line-miles over the central part of the District (Figure 9-5). Numerous, high-quality anomalies were identified and indicate a large area of anomalous IP and CSAMT responses between the Yellow Pine and Hangar Flats deposits as well as in other areas. Additional surveys are recommended to provide fill-in or extend coverage over anomalous features or evaluate newly identified prospects since the last surveys were completed.

9.5.2.1 Airborne Geophysical Surveys

The airborne geophysical survey datasets were initially processed by Fugro and post-processed by Condor Consulting, Inc. to develop a series of layered earth inversions, stacked profile plots, and various derivative products (Condor, 2012). These data were integrated with other geologic, geochemical, and structural data to identify new previously unmapped faults, prospects, and potential buried stocks, which are easily interpreted and distinguished on both aeromagnetic and electromagnetic-derived resistivity maps when integrated (Figure 9-6 and Figure 9-7). For example, the prominent north-south feature associated with the Meadow Creek Fault Zone (A) and northeast-southwest feature associated with the Hennessy and Hidden fault zones (B) is readily discernable, as are potential splay faults, some confirmed by drilling. Prominent aeromagnetic and EM features in the roof pendant show strong stratigraphic, fault, dike, and in some cases alteration zone responses. An approximately 3,000-ft, roughly circular, high-amplitude feature on the aeromagnetics (C), centered around the Saddle, Fern, Buck's Bed, Resistor, and East Rabbit prospects, is interpreted to be the response associated with a meteoric water-dominated hydrothermal system above a potential buried Tertiary intrusion. Support for this interpretation includes the presence of widespread dikes (D), stock-like bodies that occur in scattered outcrops reports from historical underground mines, and widespread low-temperature epithermal alteration with meteoric water isotopic characteristics. Skarns are common along the base of the roof pendant where carbonate rocks contact the batholith (E), typically have strong magnetic responses, and often are conductive if they contain high magnetite or sulfide content. A draped Layered Earth Inversion (LEI) conductivity model (Figure 9-6) shows a broad, low-amplitude, high resistivity feature(C), coincident with the circular magnetic feature (Figure 9-7), potentially indicative of a response associated with the destruction of magnetite and high silica content.

Figure 9-5: Ground Geophysical Survey Data from 1974-2012



Note: EPOG is El Paso Oil and Gas, HMC is Hecla Mining Company, BAR is BAR Geophysics. Map excludes Pioneer IP & VLF at West End.

Figure 9-6: Layered Earth Inversion Conductivity at 10 meters below Ground Surface

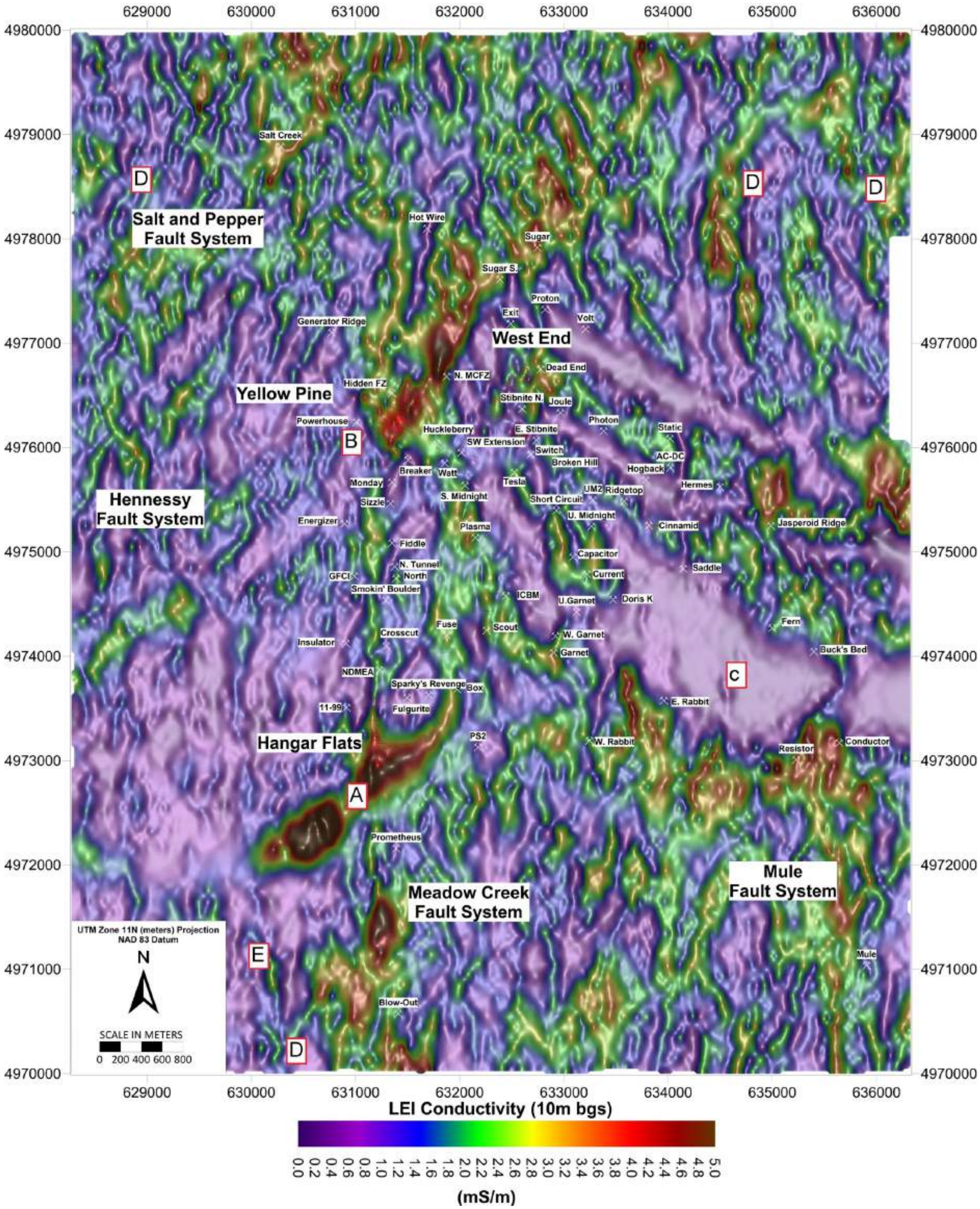
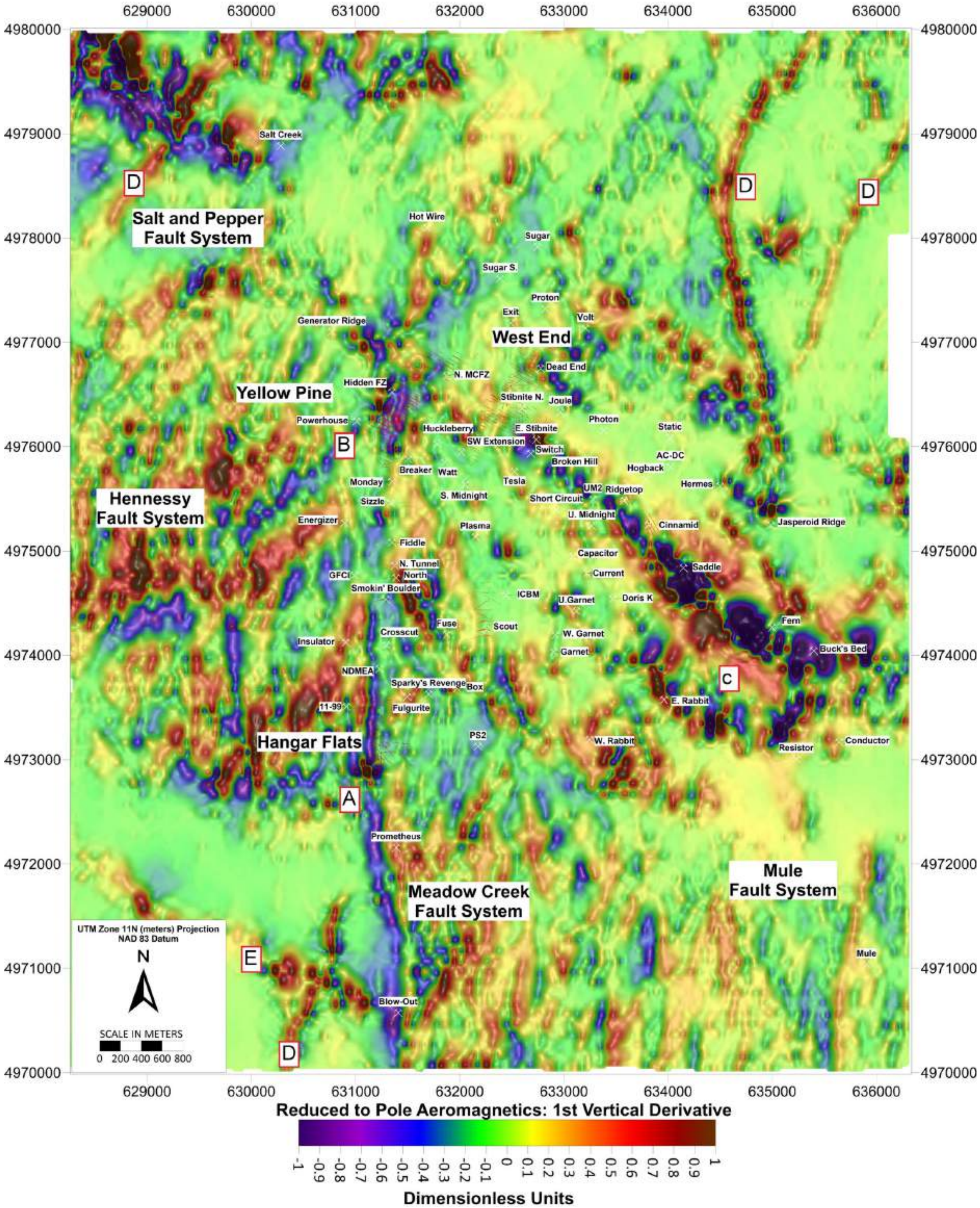


Figure 9-7: ZS[R] Filtered 1st Vertical Derivative Reduced to Pole Total Field Aeromagnetics



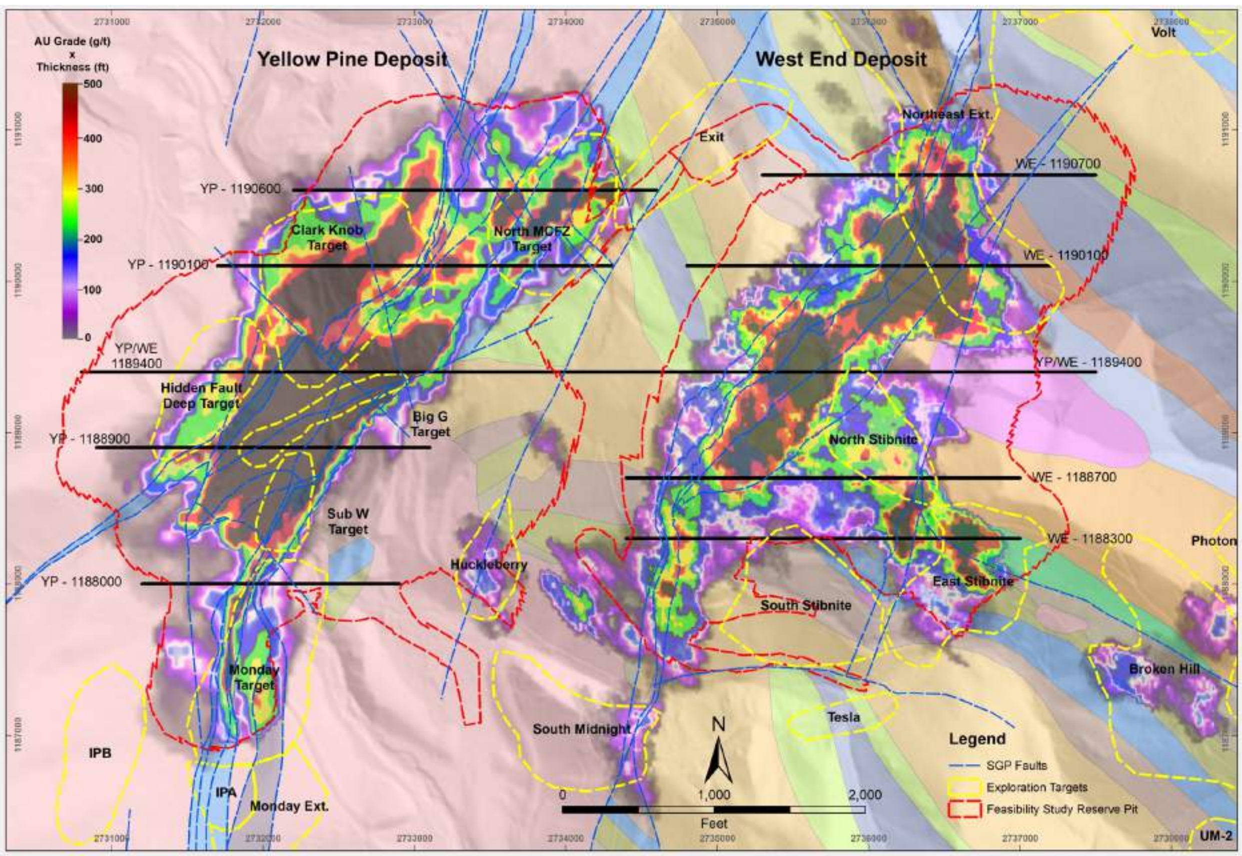
9.6 PETROLOGY, MINERALOGY AND RESEARCH

Extensive research on mineralogy, paragenesis, and timing of alteration and mineralization has been completed by Midas Gold, its contractors, and its research partners in academia and government. This work examined such items as ore and gangue mineral chemistry, morphology, whole-rock and trace-element chemistry, spectral characteristics, and mass balance calculations. A number of relevant vectoring tools for future exploration have been developed based on this research including the use of arsenic content in pyrite, pyrite grain morphology, and sodium depletion/potassium enrichment haloes around known mineralized zones.

9.7 POTENTIAL FOR EXPANSION OF THE YELLOW PINE, HANGAR FLATS AND WEST END DEPOSITS

All three major deposits with reported Mineral Resources (Section 14) remain open to expansion and this potential is described in the following sections. Mineralized material occurs between, beneath, and laterally around both the mineral reserve pits and conceptual mineral resource cones for all three deposits. A map showing the 2020 Feasibility study reserve pit limits and some of these opportunities at the Yellow Pine deposit and West End deposits is provided as Figure 9-8.

Figure 9-8: Yellow Pine and West End Block Modeled Gold Grade x Thickness

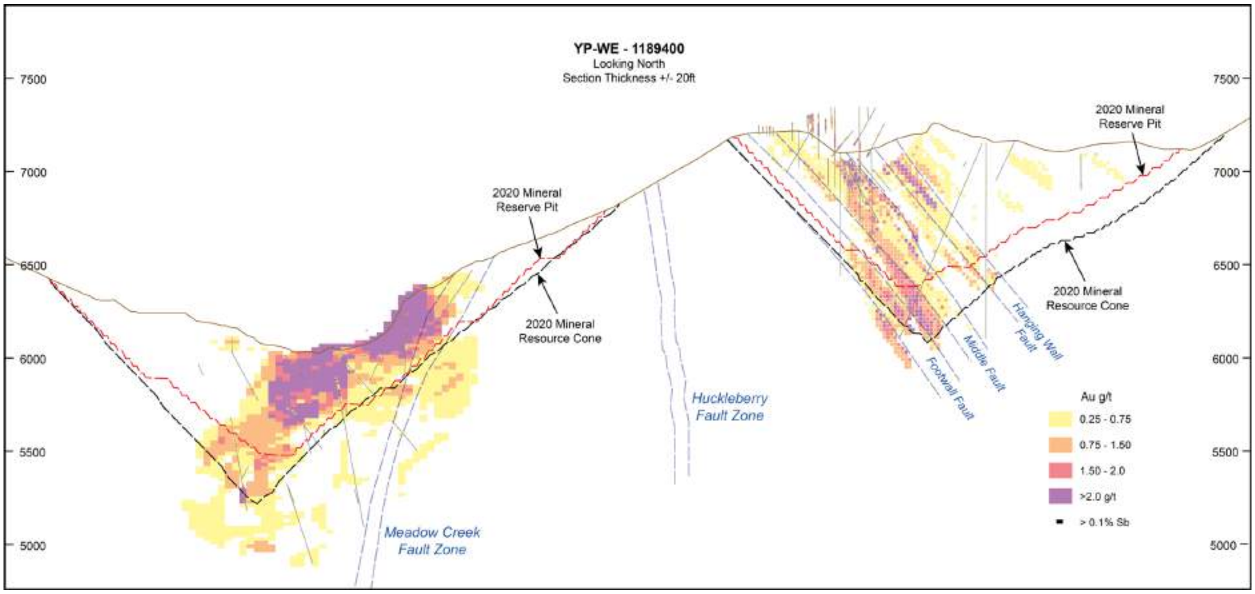


Note: Grade x thickness calculated by summing 2020 Conceptual Mineral Resource block grades to existing ground surface datum.

9.7.1 Yellow Pine

The Yellow Pine Deposit is open at depth and along strike in the north, northeast, and southwest directions (Figure 9-8) along the Meadow Creek Fault Zone (**MCFZ**) and subsidiary structures. Targets are defined by mineralized holes drilled by both Midas Gold and pre-Midas Gold operators. The area between the two deposits is also poorly tested and is mostly covered with talus, but mineralization is known to exist along the Huckleberry Fault Zone (Figure 9-9) and presents a promising but poorly evaluated target. Highlights of some of the other targets in and around the Yellow Pine deposit are discussed below.

Figure 9-9: E-W Cross Section 1,189,400N through the Yellow Pine and West End Deposits



9.7.1.1 Monday Tunnel Target

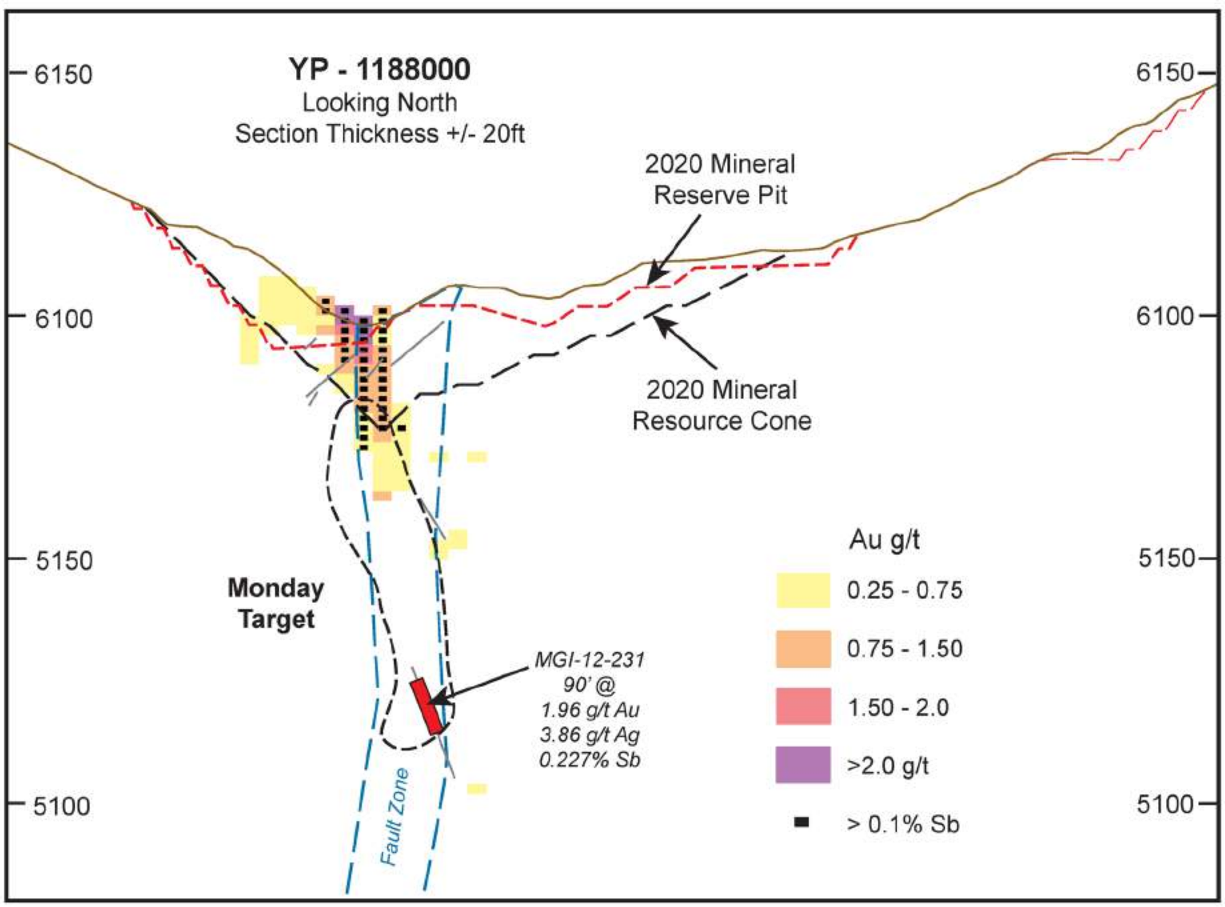
The Monday Tunnel Target on the continuation of the MCZF south of the main Yellow Pine Deposit has been the subject of underground exploration and limited drilling by the U.S. Bureau of Mines and Bradley Mining Company in the 1940s-1950s and more recently by Midas Gold. Mineralization is relatively narrow and steeply dipping and occurs in intrusive rocks along and within the MCFZ and metasedimentary rocks east of the fault but is covered beneath a relatively thick (100-200 ft) veneer of glacial materials. Midas Gold drilling intercepted mineralization over 500 feet below the feasibility study resource pit (Figure 9-8, Figure 9-10) and the zone remains open along strike to the south and down dip. Selected intercepts outside of the Mineral Reserve pit shell are provided in Table 9-1. The target is approximately 1,400 ft long in a north-south direction and approximately 650 ft wide in an east-west direction, ranges from 5,150 to6,320 ft in elevation, and is open at depth and along strike to the south. Several of the Midas Gold holes drilled here terminated in mineralization (MGI-11-138, MGI-11-140). Just south of the southernmost drilling area, IP line survey line A indicates the presence of a strong geophysical anomaly along the projection of mineralization. CSAMT survey data indicate the presence of a coincident low-resistivity feature that is beneath responses interpreted to be the glacial overburden. The CSAMT, IP, drill data, and legacy underground workings data suggest mineralization in the Monday target is likely continuous from the Yellow Pine pit area to the south for at least several hundreds of feet beyond the current conceptual Mineral Resource pit shell but lies beneath a southward thickening, 150-200 ft zone of unconsolidated glacial cover materials. While the grades and thicknesses here are promising, the steep slopes, thick unconsolidated glacial cover, and narrow, steep character of the mineralization make pit laybacks problematic without removal of significant volumes of unconsolidated glacial materials and development rock. Nonetheless, high grades of gold and antimony occur locally within the zone and could justify the removal development rock at high metal prices. Some zones potentially could support underground development if grades and continuity can be demonstrated with additional drilling. Examples of high-grade stibnite veins occurring within broader, lower grade intervals include 52 ft averaging 3.4 g/t Au, 26 g/t Ag, and 1.7% Sb in MGI-12-337 and 22 ft averaging 4.5 g/t Au, 31 g/t Ag, and 2.1% Sb in MGI-12-339.

Table 9-1: Select Drill Intecepts for the Monday Target

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Monday	Midas Gold	MGI-11-138	-70	120	480	765	285	1.31	2.35	0.006
					900	990	90	1.10	2.16	0.006
Monday	Midas Gold	MGI-11-140	-50	145	510	610	100	0.63	0.51	0.008
					850	865	15	2.33	0.37	0.025
Monday	Midas Gold	MGI-11-141	-70	145	435	485	50	1.02	1.23	0.006
Monday	Midas Gold	MGI-12-231	-70	120	845	935	90	1.95	3.86	0.227
Monday	Midas Gold	MGI-12-337	-60	120	488.5	514.5	26	0.93	1.15	0.011
Monday	Midas Gold	MGI-18-508	-20	256	375	415	40	0.81	1.23	0.355
					503	538.6	35.6	1.15	0.45	0.030
Monday	Bradley/USBM	MC-19	-14	265	0	40	40	1.42	-	0.249
Monday	Bradley/USBM	MC-20	-34	085	105	130	25	0.51	-	0.640
Monday	Bradley/USBM	MC-35	-37	079	195	235	40	0.54	-	0.975
Monday	Bradley/USBM	MC-54	-16	118	50	115	65	0.56	-	0.140
					180	250	70	0.75	-	0.511

*Note: Composites minimum 15 ft with lower cutoff grade (**COG**) of 0.25 g/t Au. May include up to 50% material below COG. MC series may include sludge and core values. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.*

Figure 9-10: E-W Cross Section 1,188,000N through the Monday Tunnel Target



9.7.1.2 Hidden Fault Deep Target

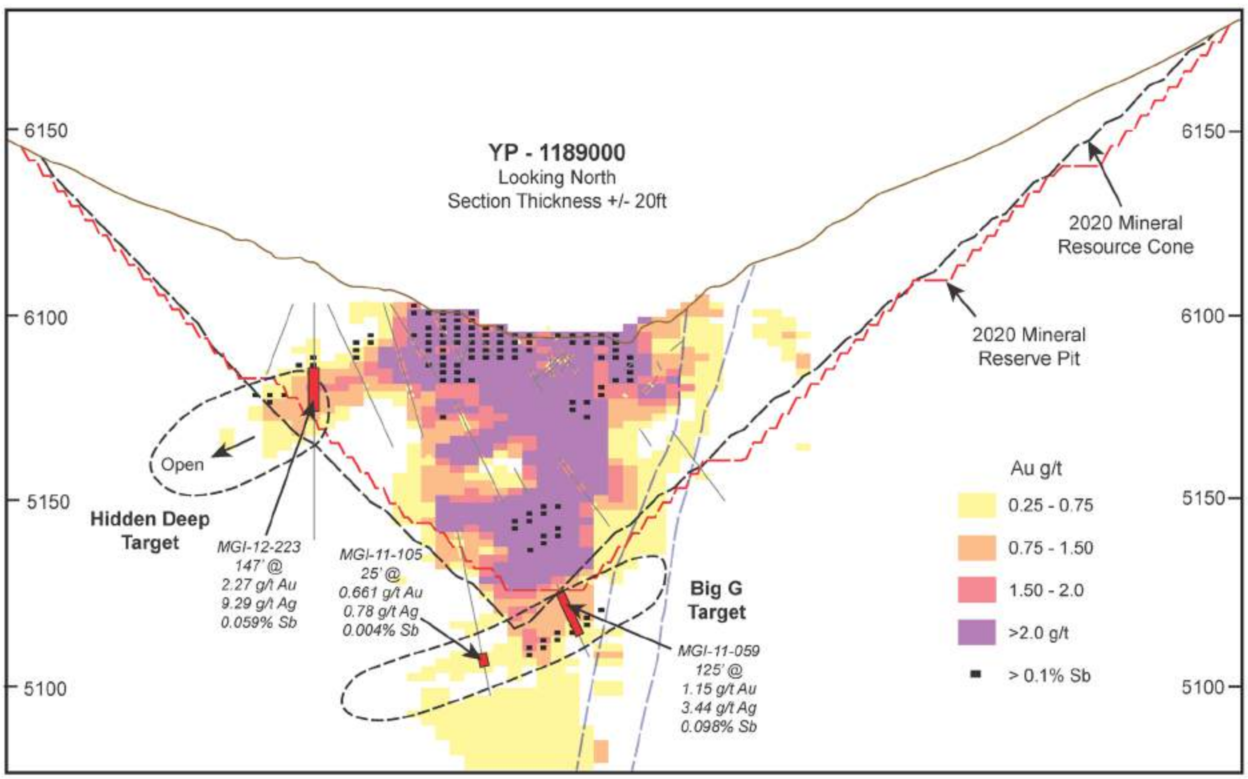
The Hidden Fault Deep Target is located at the northwest edge of the Yellow Pine Deposit (Figure 9-8, Figure 9-11) along the trace of the Hidden Fault Zone (**HFZ**). The target is supported by three Midas Gold holes covering an area approximately 1,100 ft (NE-SW) by 450 ft (NW-SE) over a range of 5,185 to 5,900 ft in elevation. The HFZ is poorly defined away from the main pit area, likely has had post-mineral movement, but remains open to the southwest and down dip. Selected intercepts are provided in Table 9-2.

Table 9-2: Select Drill Intercepts for the Hidden Fault Deep Target

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
HFZ	Midas Gold	MGI-11-131	-75	310	601	721	120	1.75	1.55	0.008
					776	806	30	0.74	0.59	0.002
HFZ	Midas Gold	MGI-12-224	-79	120	909	1017.5	108.5	1.97	1.49	0.005
HFZ	Midas Gold	MGI-13-397	-60	220	402.8	449	46.2	1.29	0.68	0.002

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. May include up to 50% material below COG. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Figure 9-11: E-W Cross Section 1,189,000N through the Hidden Fault Target



Note: Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

9.7.1.3 Big G Target

A Big G Target comprises a northeast-trending zone 1,050 ft long by 500 ft wide lying along the trace of the G-Fault at depth and contains some promising intercepts (Table 9-3, Figure 9-8). The G-Fault is a structure originally mapped underground and in the open pit during the Bradley era and is interpreted to be a mineralized structure that underwent post-mineralization movement. Eight holes are drilled along and around the trace of the fault at elevations between 5,075 and 5,875 ft. Although well drilled at higher elevations, additional drilling into the deeper sections below the structure during pit development may identify additional mineralization for pit expansion.

Table 9-3: Select Drill Intercepts for the Big G Target

Target	Operator	Drill Hole ID	Collar Inclination (o)	Collar Azimuth (o)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Big G	Midas Gold	MGI-11-105	-73	122	1035	1055	25	0.661	0.780	0.004
Big G	Midas Gold	MGI-12-199	-45	120	610	731.5	121.5	0.800	1.130	0.002
Big G	Midas Gold	MGI-12-241	-45	120	478	517	39	0.787	1.350	0.004
Big G	Midas Gold	MGI-12-245	-45	120	829	853.5	24.5	0.658	1.050	0.002
Big G	Midas Gold	MGI-13-365	-64	192	532	569.5	37.5	1.293	0.680	0.002

Note: Composites minimum 15 ft with lower COG value of 0.25 g/t Au. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

9.7.1.4 North Meadow Creek Fault Target

The North Meadow Creek Fault Target lies on the northeast side of the Yellow Pine deposit and is defined by fifteen Midas Gold and several pre-Midas Gold drill holes (Figure 9-8, Figure 9-12). The zone is bounded on the northwest by the East Boundary Fault and extends across an elongated ellipsoidal target area to the northeast and southwest. Mineralization is hosted in intrusive rocks west of the MCFZ and within metasedimentary rocks and intrusives east of the fault. The MCFZ exhibits post-mineralization displacement with the latest movement likely having a right-lateral sense of displacement. This post-mineralization movement has attenuated mineralization, forming lenses that vary in grade depending upon the amount of mineralized rock versus unmineralized rock caught up in the structural zone. The target extends at least 1,100 ft in a northeast-southwest direction and is roughly 500 ft wide in a northwest-southeast direction with a vertical range from 5,650 to 6,250 ft in elevation. High grades have been encountered in several holes over thick intervals in several historical and Midas Gold drill holes where they intersected favorable reactive metasedimentary host rocks within or adjacent to faults (Table 9-4). The zone is inadequately drilled at appropriate orientations to fully test the favorable metasedimentary rocks in the limbs and hinge of the Garnet Syncline and its intersection with the MCFZ. Additional drilling at appropriate orientations should be a priority here during pit development. Locally, grades are high enough that they could potentially support underground development if widths, continuity along strike and down dip, and other factors were found to be favorable after additional drilling and appropriate engineering, geotechnical, and other studies.

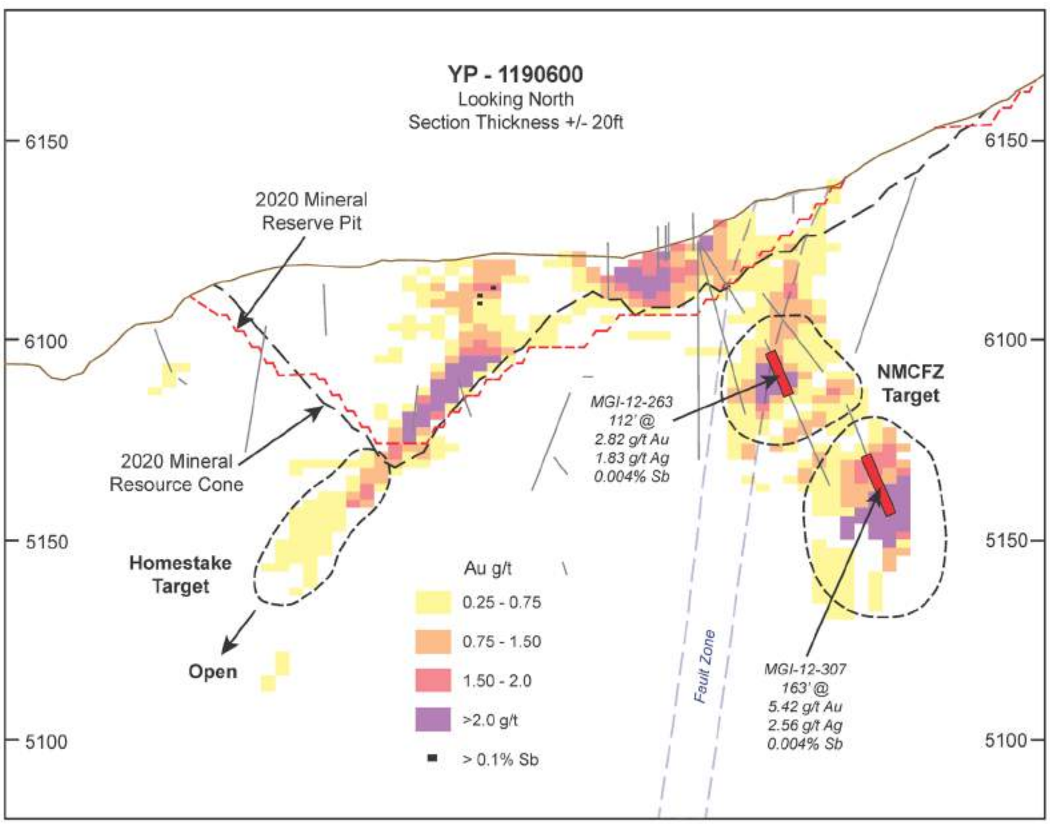
Table 9-4: Select Drill Intercepts for the North Meadow Creek Fault Zone Target

Target	Operator	Drill Hole ID	Collar Inclination (o)	Collar Azimuth (o)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
NMCFZ	Midas Gold	MGI-11-079-RC	-90	-	390	600	210	0.92	0.26	0.002
NMCFZ	Midas Gold	MGI-11-082	-50	117	393	642	249	1.07	0.61	0.002
NMCFZ	Midas Gold	MGI-11-084-RC	-50	120	290	450	160	0.71	0.61	0.003
NMCFZ	Midas Gold	MGI-11-108-RC	-50	120	230	500	270	2.03	1.97	0.004
NMCFZ	Midas Gold	MGI-11-111-RC	-70	120	380	425	45	1.52	0.98	0.002
					455	550	95	0.55	1.15	0.004
NMCFZ	Midas Gold	MGI-12-205-RC	-60	120	195	220	25	1.04	0.39	0.075
					300	505	205	1.03	1.33	0.003
					565	620	55	0.45	0.40	0.002
NMCFZ	Midas Gold	MGI-12-263	-16	118.9	334.5	446.5	112	2.82	1.83	0.004
					522	554.5	32.5	1.23	0.84	0.002

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
NMCFZ	Midas Gold	MGI-12-267	-45	120	273	350	77	1.26	2.20	0.002
					395	518.5	123.5	1.26	2.20	0.002
NMCFZ	Midas Gold	MGI-12-276	-45	120	175	227.5	52.5	1.62	4.88	0.006
NMCFZ	Midas Gold	MGI-12-307	-60	130	212	258	46	0.85	1.67	0.002
					331.5	399	76.5	0.63	0.87	0.001
					429	451.5	22.5	2.71	1.90	0.001
					510.5	540.5	30	0.55	1.95	0.03
					659.5	822	162.5	5.42	2.56	0.004
NMCFZ	Midas Gold	MGI-12-318	-67	120	198	237	39	1.06	1.76	0.002
					379	405	26	0.54	0.25	0.001
NMCFZ	Midas Gold	MGI-12-325	-79	120	233	253	20	2.98	2.86	0.004
					279	311.5	32.5	0.53	1.80	0.004
					324.5	377.5	53	0.76	2.87	0.004
					406.5	563	156.5	1.48	2.63	0.004
NMCFZ	Midas Gold	MGI-12-335	-65	120	378	565	187	1.31	0.81	0.002
NMCFZ	Midas Gold	MGI-13-358	-44	120	195	268	73	1.50	2.05	0.007
NMCFZ	Midas Gold	MGI-13-360	-65	120	290	361.5	71.5	1.19	2.81	0.005

Note: Composites minimum 15 ft with lower COGf value of 0.25 g/t Au. May include up to 50% material below COG. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Figure 9-12: E-W Cross Section 1,190,600N through the North Meadow Creek Fault Target



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

9.7.1.5 Clark Knob Target

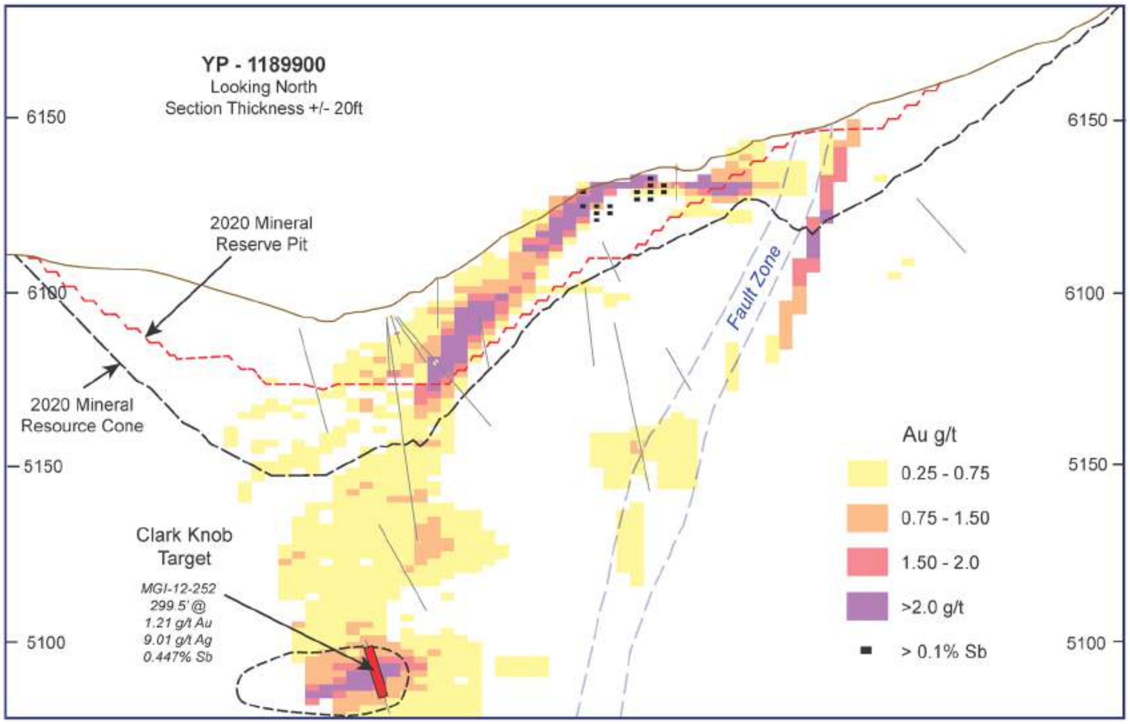
The Clark Knob Target consists of a large ovoid area located beneath and along the flanks of the northwestern end of the Yellow Pine Mineral Reserve pit and contains a large number of drill holes that encounter mineralization (Table 9-5). Mineralization has been intersected down-dip of the intervals within the mineral reserve pit both within the conceptual mineral resource cone and below it (Figure 9-8, Figure 9-13, Figure 9-14). This target, supported by 21 Midas Gold holes and additional holes by pre-Midas operators is roughly 1,000 ft by 1,200 ft and ranges in elvation from 4,860 ft to 6,110 ft. The area is roughly bounded by the Latite Fault to the southwest and the Clark-Bailey Fault to the northeast. Much of the mineralization is quite deep but may be developed by a future pit layback if metal prices, and other considerations are favorable. Several holes bottomed in mineralization and the target is open down-dip and along strike in both directions.

Table 9-5: Select Drill Intercepts for the Clark Knob Target

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Clark Knob	Midas Gold	MGI-11-124	-69	120	1027	1084	57	0.51	0.45	0.003
Clark Knob	Midas Gold	MGI-11-132	-81	116	602	681	79	0.93	3.12	0.005
					528	562	34	1.38	3.06	0.004
Clark Knob	Midas Gold	MGI-12-187	-72.5	120	769.5	964	194.5	1.09	0.89	0.002
Clark Knob	Midas Gold	MGI-12-243	-73	120	540.5	604	63.5	0.36	0.58	0.002
					705	871.5	166.5	0.96	1.67	0.004
					969.5	1008.5	39	1.33	0.62	0.003
					1146	1245	99	0.57	0.39	0.002
Clark Knob	Midas Gold	MGI-12-252	-70	120	883	1182.5	299.5	1.21	9.01	0.447
Clark Knob	Midas Gold	MGI-12-253	-70	176	823	903.5	80.5	0.60	0.81	0.003
					942	1031	89	0.68	0.98	0.002
Clark Knob	Midas Gold	MGI-12-261	-52	120	591	662	71	0.86	0.63	0.003
					690	784	94	0.52	2.64	0.003
Clark Knob	Midas Gold	MGI-12-272	-64	061	426.5	478.5	52	0.60	0.85	0.003
Clark Knob	Midas Gold	MGI-13-356	-80	162	605	695	90	1.64	1.03	0.002

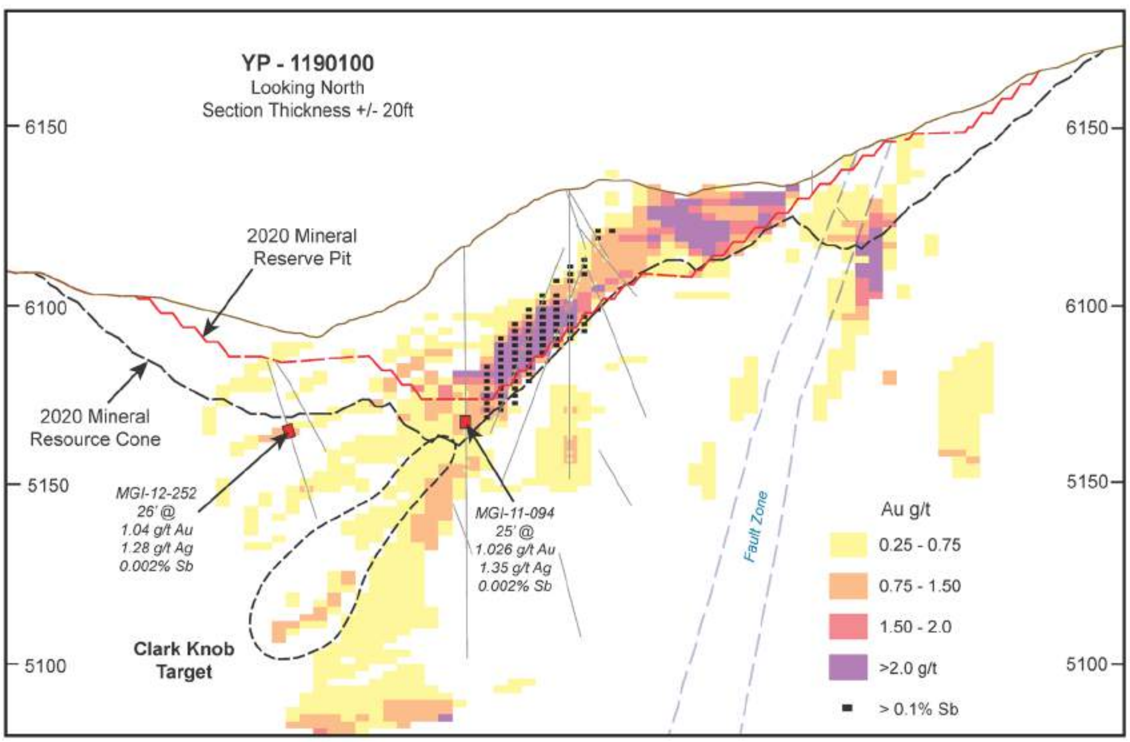
Note: Composites minimum 15 ft with a lower COG of 0.25 g/t Au. May include up to 50% material below COG. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Figure 9-13: E-W Cross Section 1,189,900N through the Clark Knob Target



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

Figure 9-14: E-W Cross Section 1,190,100N through the Clark Knob Target



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

9.7.1.6 Sub W Target

The Sub W Target consists of the area beneath the former Yellow Pine pit and Mineral Reserve pit at depth and contains several intercepts in an area approximately 650 ft (NE-SW) by 300 ft (NW-SE) over elevations ranging between 5,000 and 5,700 ft (Figure 9-8). The drill hole intercepts in this area are relatively low-grade (Table 9-6) but indicate that mineralization is open at depth. Structurally, the target area is cut by extensive faults and contains a wide variety of intrusive rock types and metasediments.

Table 9-6: Select Drill Intercepts for the Sub W Target

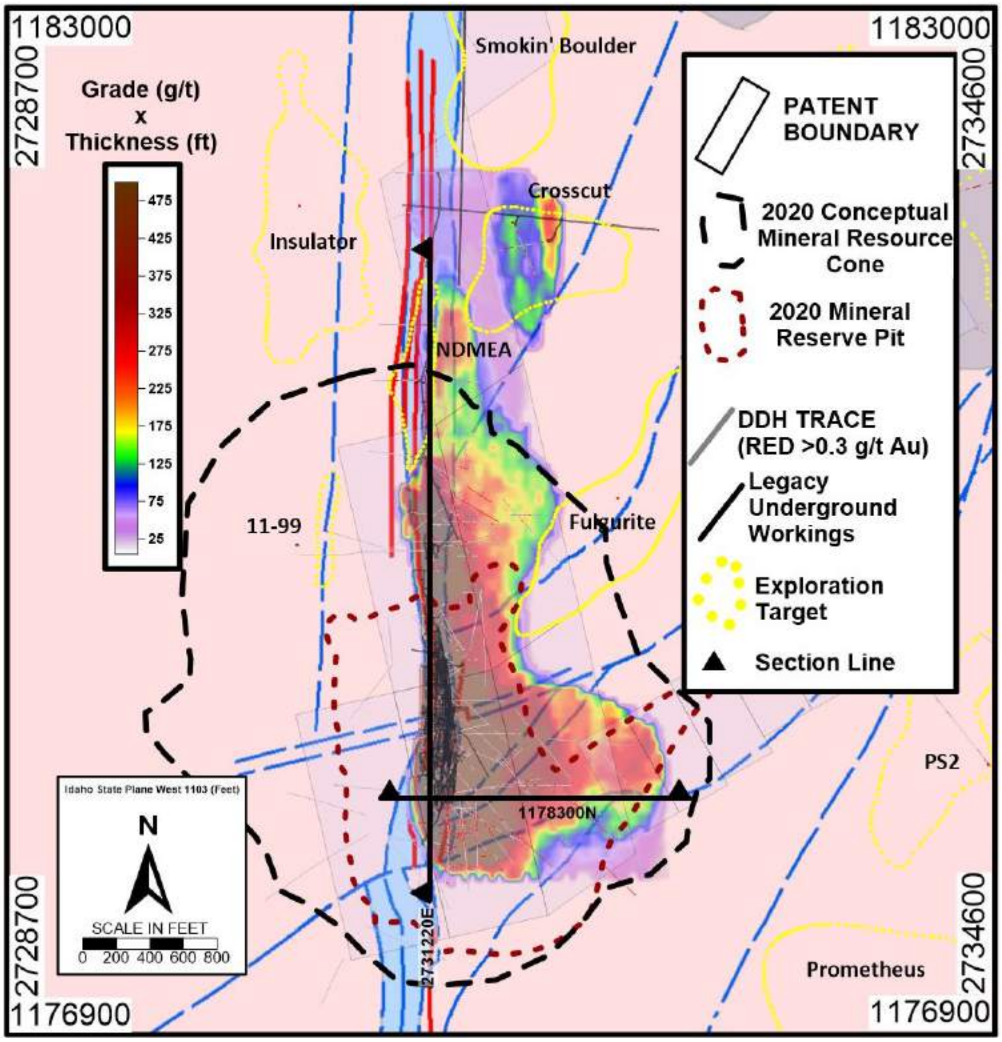
Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Sub W	Midas Gold	MGI-11-127	-54	131.5	963	1007	44	0.45	0.40	0.003
					1160	1184	24	0.57	2.56	0.004
					1209	1235	26	0.80	1.25	0.008
Sub W	Midas Gold	MGI-11-145	-45	112	1251	1324	73	0.78	0.87	0.007
Sub W	Midas Gold	MGI-12-235	-50	089	563	607	44	0.76	1.34	0.006
Sub W	Midas Gold	MGI-12-236	-70	300	981.5	1033.5	52	1.08	2.67	0.005
					1098.5	1288	189.5	0.75	2.52	0.011

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

9.7.2 Hangar Flats

The Hangar Flats Deposit is located along the MCFZ and the intersection of a series of subsidiary or splay faults that trend east-northeast and northeast and dip to the northwest. A corridor more than 3,000 ft long north, east, and west of the main deposit is inadequately drill-tested outside of the known deposit (Table 9-7, Figure 9-15, Figure 9-17).

Figure 9-15: Plan Map Showing the Hangar Flats Expansion Targets



9.7.2.1 Hangar Flats Deep Target

Historical sampling and production records from the former Meadow Creek Mine define the Hangar Flats Deep (HFD) Target, a zone of high-grade gold-antimony mineralization in a 30- to 330-ft-wide corridor along the western boundary of the MCFZ that remains open along strike and down dip. This was historically called the "*West Ore Body*" by Cooper (1951) and was never mined by previous underground mining operations. Figure 9-16 shows several drill holes, which intersected multiple high-grade intercepts, some containing several percent antimony and highly anomalous tungsten values within broad zones of gold mineralization that represent a portion of this body of mineralized material. Currently, much of this mineralization is drilled sufficiently to be classified as measured and indicated resources and some of this material falls within the conceptual Mineral Resource cone. Up-dip portions of this zone were mined underground in the 1920s-1930s. Stopes ranged from 3 to 40 ft in true thickness and show continuity over hundreds of feet of plunge, producing mill head grades averaging greater than 6 g/t gold and several percent antimony. One of the more significant intercepts in this target area, cut in drill hole MGI-12-192, included 294 ft grading 1.57 g/t gold and 2.76% Sb. Within this broad zone there were several higher-grade intervals including 25 ft averaging 6.09 g/t gold and another averaging 1.6 g/t Au, 108 g/t Ag, 6.4 % Sb, and 2.4% W over 75 ft. The grades and thicknesses encountered in this zone potentially could support underground development under appropriate metal prices.

Table 9-7: Select Drill Intercepts for the Targets beneath and peripheral to Proposed Hangar Flats Pit

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
HF Deep	Midas Gold	MGI-10-21	-90	-	964	1044	80	2.58	6.49	0.888
HF Deep	Midas Gold	MGI-11-67	-90	-	913	1000	87	1.59	12.90	1.024
HF Deep	Midas Gold	MGI-10-21	-90	-	964	1044	80	2.58	6.49	0.888
HF Deep	Midas Gold	MGI-12-165-RC	-78	320	920	965	50	1.21	13.94	0.911
HF Deep	Midas Gold	MGI-12-168-RC	-67	293	890	1010	120	1.39	36.55	2.319
HF Deep	Midas Gold	MGI-12-171-RC	-79	043	800	900	100	0.89	0.72	0.008
HF Deep	Midas Gold	MGI-12-173-RC	-67	140	620	685	65	1.13	1.70	0.004
					790	815	30	0.86	0.95	0.003
HF Deep	Midas Gold	MGI-12-179-RC	-90	-	615	710	95	0.93	1.34	0.004
HF Deep	Midas Gold	MGI-12-191	-72	320	897	932	35	1.00	1.90	0.047
HF Deep	Midas Gold	MGI-12-192	-83	280	902	927	25	0.71	0.47	0.006
					1006	1300	294	1.57	36.61	2.761
HF Deep	Midas Gold	MGI-12-193	-90	-	1029	1170	141	1.28	13.54	1.301
HF Deep	Midas Gold	MGI-12-203	-65	320	864	884	20	0.89	1.19	0.003
					912	938	30.5	1.71	1.18	0.004
					1006	1074.5	68.5	1.39	119.18	6.382
HF Deep	Midas Gold	MGI-12-195	-85	140	823	861.5	38.5	1.14	1.49	0.005
HF Deep	Midas Gold	MGI-12-220	-65	320	712.5	743	30.5	1.62	9.67	0.525
					816	865	40	0.80	112.43	0.007
					932	957.5	25.5	2.02	19.02	0.629
HF Deep	Midas Gold	MGI-12-225	-78	320	1328	1395	67.5	0.81	3.00	0.129
HF Deep	Midas Gold	MGI-12-229	-90	-	1283	1313	30	0.55	0.36	0.004
					1343	1365.5	22.5	0.70	0.25	0.003
					1431	1470	39	2.30	8.40	0.327

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. May include up to 50% material below COG. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Figure 9-16: E-W Cross Section 1,178,300N through the Hangar Flats Deep Target

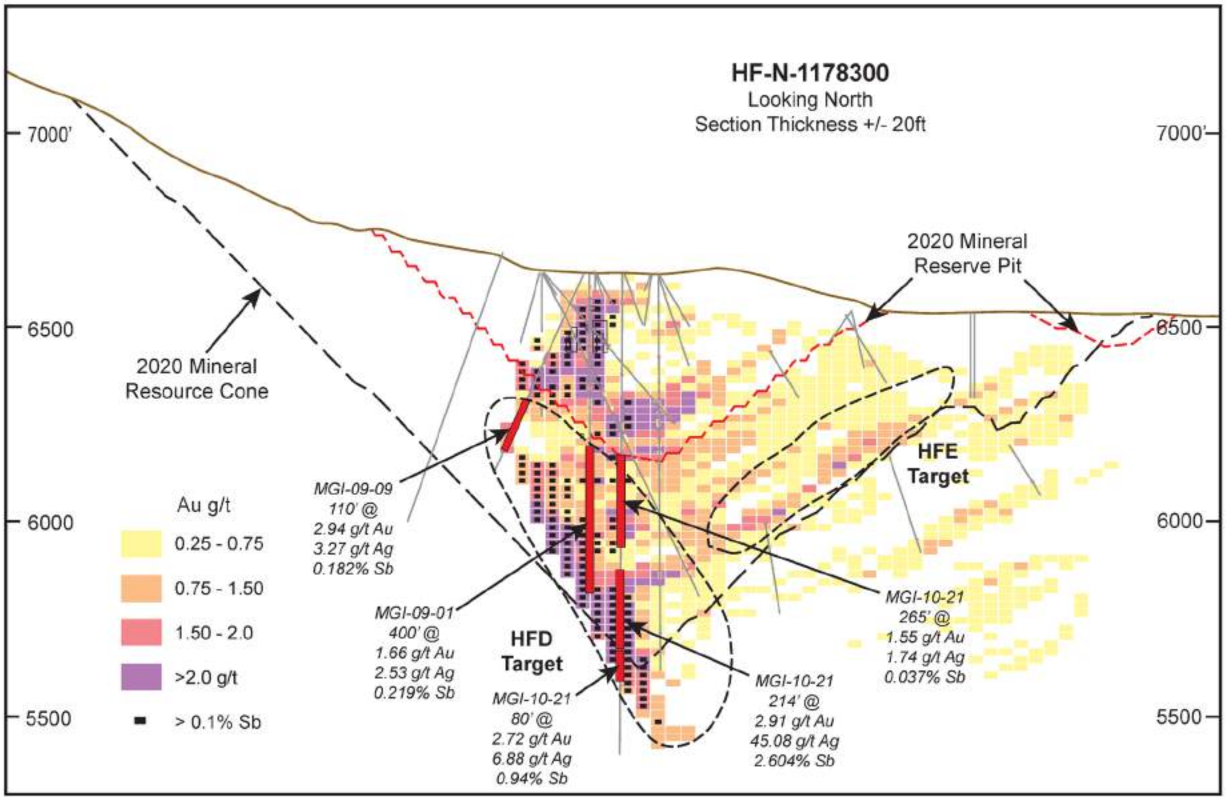
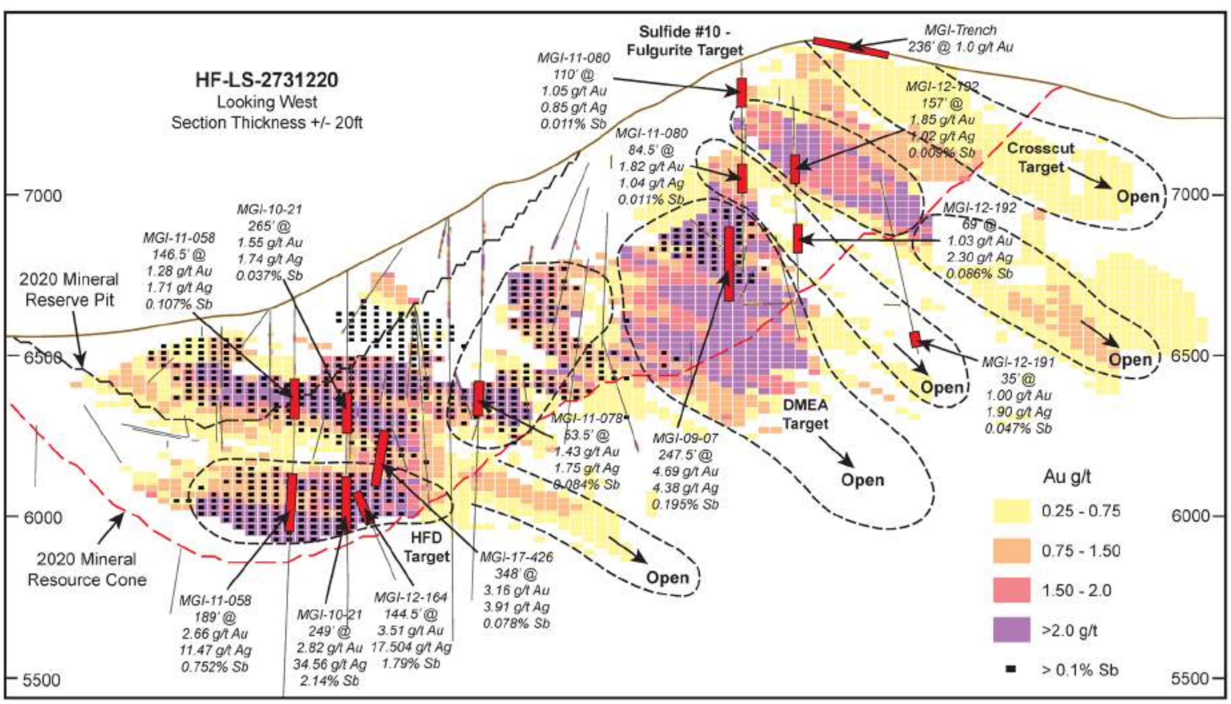


Figure 9-17: N-S Long Section 2,731,220E through the Hangar Flats Deposit



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

9.7.2.2 DMEA Target

The DMEA target lies beneath the northern part of the Hangar Flats Mineral Reserve Pit and was initially discovered in the early 1950s by USBM and Bradley during underground exploration under the federally sponsored Defense Mineral Exploration Administration program. The MCFZ is poorly tested over a distance of at least several thousand feet beyond the DMEA prospect, which has been explored by a single drift driven along the eastern side of the MCFZ fault trace in this target area. The underground workings were extensively mapped and sampled in the 1950s and indicate the presence of northeast-trending high-grade vein systems. A large zone of mineralization was sampled perpendicular to the MCFZ by pre-Midas Gold underground channel sampling, which produced a length-weighed average gold grade of 6.5 g/t over 92 ft (1.56 g/t over 300 ft). Underground drill holes intersected significant high-grade intercepts (Table 9-8). Mineralization cut in the tunnel and underground holes demonstrated continuity of mineralization over a vertical distance of at least 375 ft. This high-grade gold-antimony mineralization has been intersected over a strike length of 2,000 ft. Widely spaced holes show mineralization extends over 1,000 ft of vertical extent and remains open at depth. One of the more significant intercepts in this target area (MGI-09-07) verified the grades and thicknesses reported in historical DMEA underground workings, including several higher-grade intervals. This higher-grade mineralization might be developed in a conceptual underground development scenario, were continuity, scale, and other factors suitable. and included: 28.5 ft averaging 9.37 g/t Au, 7.32 g/t Ag, 1.136% Sb and another interval averaging 12.64 g/t Au, 7.87 g/t Ag, and 0.319% Sb over 22 ft (Table 9-8, Figure 9-17).

Table 9-8: Select Drill Intercepts for the DMEA Target

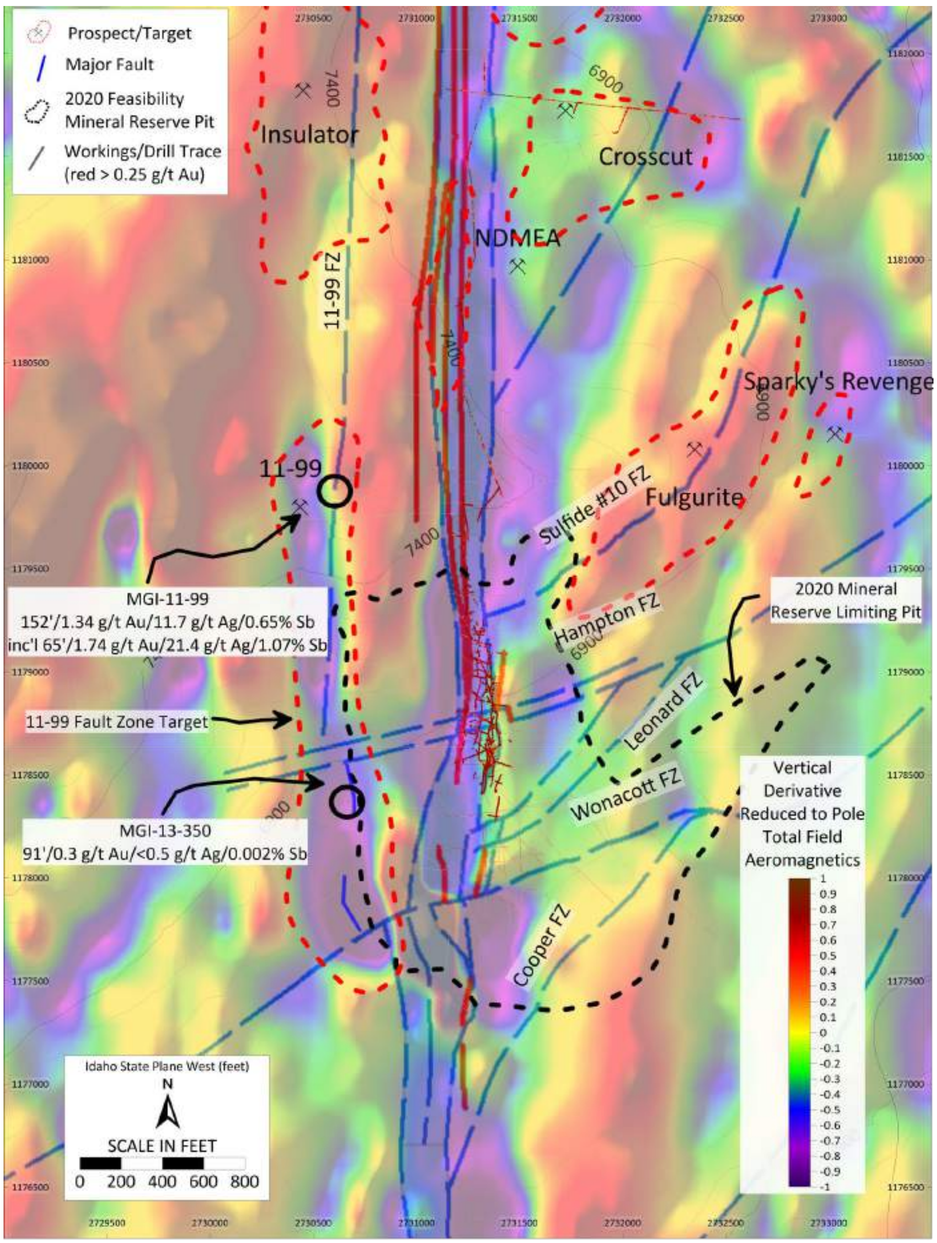
Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval ft)	Gold (g/t)	Ag (g/t)	Sb (%)
DMEA	Midas Gold	MGI-09-07	-70	090	678.5	926	247.5	4.69	4.38	0.195
				inc'l	689	704	15	6.28	7.30	0.247
				inc'l	720.5	749	28.5	9.37	7.32	1.136
				inc'l	772	792	22	12.64	7.87	0.319
DMEA	Midas Gold	MGI-09-10	-68	052	1012	1032	20	2.03	2.93	0.008
DMEA	Midas Gold	MGI-09-11	-65	155	666	700	34	0.70	3.16	0.006
DMEA	Midas Gold	MGI-12-197	-90	-	838	889	51	2.38	3.78	0.027
DMEA	Bradley/USBM	DMA-15	1	329	185	225	40	2.85	-	0.155
DMEA	Bradley/USB	DMA-17	-32	132	10	35	25	0.75	-	0.096
DMEA	Bradley/USBM	DMA-18	2	271	35	105	70	1.83	-	0.04

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. May include up to 50% material below COG. DMA series may include sludge and core values. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

9.7.2.3 11-99 Target

A geotechnical hole (MGI-11-099), drilled west of the Hangar Flats deposit in 2011 intercepted a previously unidentified zone of high-grade gold-antimony mineralization that cut 152 ft averaging 1.34 g/t Au, 12 g/t Ag, 0.65% Sb. The intercept was at considerable depth downhole and the hole terminated in mineralization. Surface examination above the intercept did not disclose any altered or mineralized rocks at the surface. Mineralization appears open and possibly extends along strike, down dip, and possibly up dip from the drill intercept, based on airborne geophysical surveys (magnetics and EM), CSAMT, and interpretation of oriented core data. However, given the intercept is in a single hole, the trend of the zone is uncertain. Another hole was drilled in the vicinity of the geophysical feature thought to represent the structure hosting mineralization (MGI-12-346), but was drilled to intersect the geophysical feature at a significantly higher elevation (~1100 ft) than the zone cut in hole MGI-11-099 (Figure 9-18). Another hole in the area was too far east (MGI-17-428) to adequately test the feature. A hole (MGI-13-350) along the same geophysical trend 1,500 ft to the south, intersected two broad zones of anomalous gold and arsenic, strong alteration, intense fracturing, and gouge at the same approximate location is was predicted to occur. The upper interval in MGI-13-350 was approximately 73 ft thick and averaged ~0.1 g/t gold and 1150 ppm arsenic and the second interval 100 feet farther downhole averaged ~0.3 g/t gold and ~1300 ppm arsenic over 91 ft. If the structure is steep, then the drill hole intercept approximates the true width (~150-165 ft). Both MGI-11-99 and hole MGI-13-350 show similar widths of alteration, large arsenic and antimony haloes, and a pronounced potassium enrichment and sodium + calcium depletion typical of mineralized structures within the SGP area. CSAMT data across the feature suggests it does not reach the surface and is essentially “blind”, but indicates that the feature continues to the north and south, potentially terminated against the Wonacott Fault to the south. The 11-99 intercept is approximately 1,300 ft below the ground surface and the intercept in MGI-13-350 is over 1,600 ft below the ground surface. Although too deep for any open pit consideration, the high gold-silver-antimony grades in MGI-11-99 warrant additional drill follow-up. Within the 152-ft interval, a higher-grade zone ~65 ft wide averaged over 1% Sb, ~21 g/t Ag, and 1.74 g/t Au. At appropriate metal prices, these grades might support underground operations if exploration is successful in defining a mineral resource.

Figure 9-18: Vertical Derivative of Reduced to Pole Aeromagnetics showing 11-99 Fault Zone & Target



9.7.2.4 HF East Target

A large area east of the Hangar Flats Deposit, the HF East Target, has only limited drill testing and there are several large structures (Wonacott, Leonard, and Hampton faults) that could potentially be mineralized along their traces northeast along strike and up dip from the deeper zones intersected in the main deposit area along the MCFZ. Fan drilling in 2009-2010 and historical DMEA-sponsored drilling under the airstrip confirms the northeast striking and northwest dipping low angle faults extend beneath the valley bottom to the northeast and are at least locally mineralized (Figure 9-15).

9.7.3 MCFZ Trend

The MCFZ trend consists of a ~2-mile long north-south string of prospects aligned along the MCFZ and associated cross structures. The Hangar Flats Deposit lies at the southern end with the Yellow Pine Deposit at the northern end. The major prospects along this trend are shown in Figure 9-2 and Figure 9-19, with selected drill results summarized in Table 9-9. Targets vary from conceptual open-pit to underground, with variable availability of supporting data.

The Monday Tunnel was driven in the 1920s-30s from the southern edge of the current Yellow Pine pit towards the Meadow Creek Mine but was abandoned before reaching the Meadow Creek Mine workings. The tunnel was partially reopened to explore for additional tungsten and antimony in the 1940s and minor production from the workings was reported but was not tabulated separately from Yellow Pine deposit production.

The North Tunnel was driven south through glacial materials in the Fiddle Creek drainage in the 1920s. It was a short exploration tunnel with minor production that was reopened in the 1940s to complete a small underground drilling program.

The DMEA tunnel was driven westward towards the MCFZ between the North Tunnel and the Meadow Creek Mine workings in the 1950s, discovered high-grade mineralization during underground sampling and drilling, but recorded no production. Other historical surface exploration was conducted along the MCFZ trend from Yellow Pine to Hangar Flats including ground-based geophysical surveys, soil grids, trenches, prospect pits and rock sampling.

Prospects along the MCFZ trend contain mineralization in high-grade Au-Sb-Ag±W veins and disseminated Au-Sb-Ag mineralization. One prospect contains molybdenite veining associated with minor greisen development in an undated leucogranite. The Idaho batholith is the predominant bedrock unit along the trend, but some metasedimentary rocks may be present, as suggested by drill intercepts and geophysical indicators. The majority of the trend is covered with glacial outwash deposits, landslides, and thick forested soil cover. There has been only limited drilling along the trend. Evidence of mineralization is mostly derived from previous underground exploration workings and limited widely spaced surface and underground drilling or inferred from geophysics and soil sampling data. The main MCFZ has been mapped underground as a north-south steeply dipping structural zone several hundreds of feet wide with a series of intersecting shallow to moderately-dipping cross structures striking northeast and east-west.

Pre-Midas Gold underground mapping at the DMEA, Monday, and North tunnels outlined extensive zones of Au-Sb-W mineralization and demonstrates the potential for high-grade mineralization along the trend. Beneath the Fiddle Creek drainage in the Monday Tunnel, an intercept of 240 ft grading 1.1% Sb and 0.75 g/t Au was reported just east of the main MCFZ trend (White, 1940). In the DMEA workings north of the Hangar Flats Deposit, intercepts of Au-Sb-W mineralization are common in northeast trending shear zones and disseminated within intrusive rocks. , Continuity of mineralization from these underground zones up to the surface is suggested by broad soil and ground geophysical anomalies covering the projected surface expression along the trend of the vein and shear systems in the North DMEA area. Several of the major prospects along the trend are described in the sections that follow and are outlined on Figure 9-19.

The trend is underlain predominantly by granodiorites and apparently younger leucogranite bodies with screens of primarily schistose metasedimentary rock units with contact metamorphic assemblages along the slopes to the east of the MCFZ. The eastern slopes and the northern part of the trend are covered with glacial outwash limiting geochemical prospecting. Midas Gold work here consisted of compilation of legacy data, geologic mapping, extensive IP, CSAMT, SP, soil grid sampling, backhoe trenching, hand-dug test pits, and some drilling. The airborne EM and magnetics data and their derivatives outline the main fault systems. The IP and CSAMT provide a means to screen areas under glacial cover and to assist in distinguishing barren structures from potentially mineralized structures.

Table 9-9: Select Drill Intercepts for the Exploration Targets along the MCFZ Trend

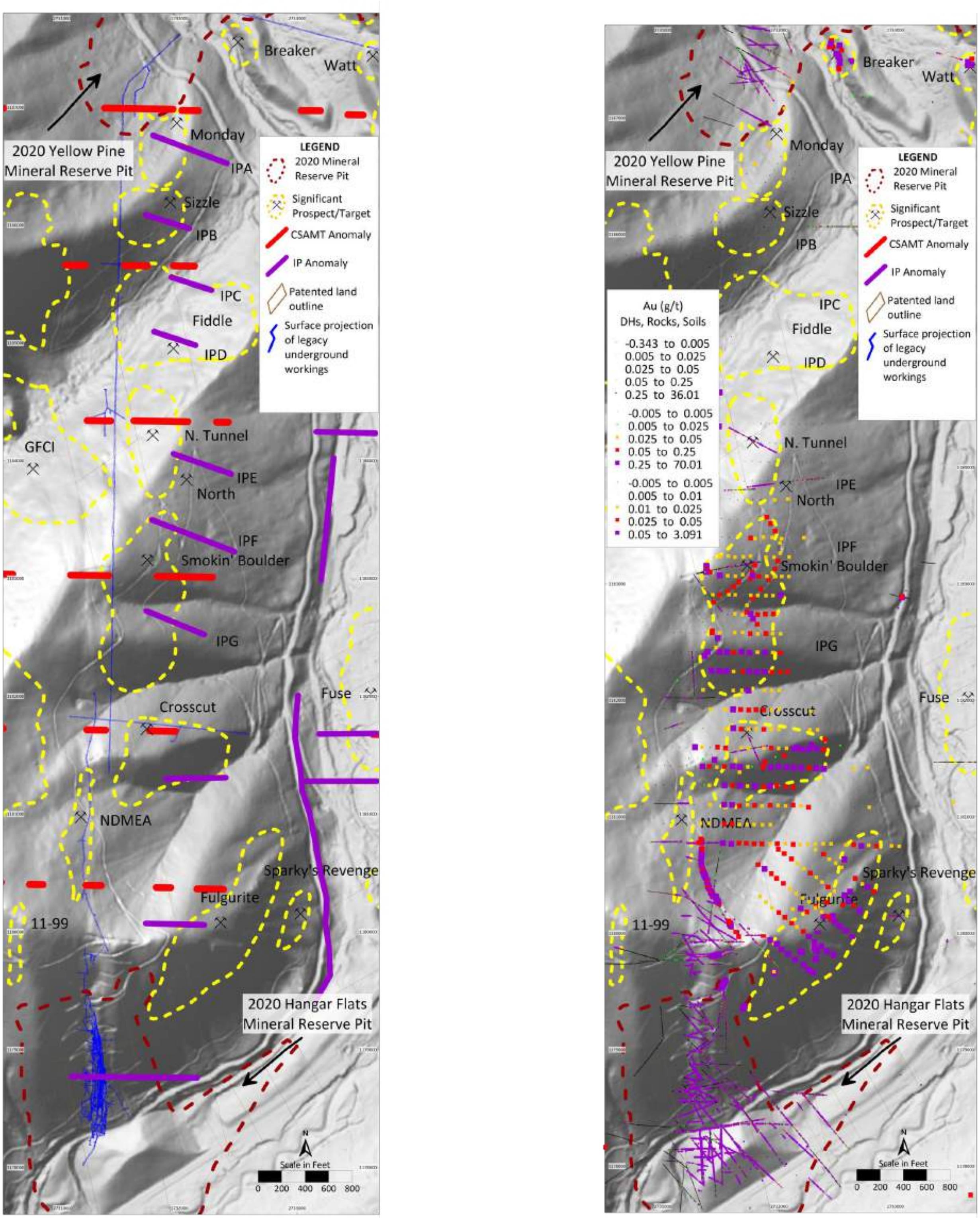
Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
North	Midas Gold	MGI-10-39	-45	083	110	153	43	0.58	0.44	0.007
					244.5	290	45.5	2.03	1.01	0.010
					310	335	25	1.03	0.91	0.004
North	Bradley/USBM	FC-1	1	118	15	110	95	0.55	-	0.113
Smokin' Boulder	Midas Gold	MGI-10-32	-60	081	1044	1071	27	0.57	1.31	0.005
Smokin' Boulder	Midas Gold	MGI-10-34	-45	081	570	595	25	1.48	4.06	0.185
Crosscut	Bradley/USBM	DMA-05	3	259	15	75	60	0.59	-	0.043
					120	140	20	0.49	-	0.035
Crosscut	Bradley/USBM	DMA-06	-1	157	35	70	35	0.71	-	0.040
					315	345	30	0.46	-	0.042
NDMEA	Bradley/USBM	DMA-07	0	089	5	65	60	1.07	-	0.112
					250	270	20	0.69	-	0.078
NDMEA	Bradley/USBM	DMA-09	0	269	130	170	40	0.62	-	0.069
NDMEA	Bradley/USBM	DMA-10	0	106	0	55	55	0.76	-	0.056
Fulgurite	Midas Gold	MGI-11-170-RC	-90	-	350	370	20	0.37	0.41	0.002
					530	560	30	1.29	0.54	0.040
Fulgurite	Midas Gold	MGI-12-180	-71	187	995	1035	40	0.72	0.62	0.002
MCFZ-West	Midas Gold	MGI-11-099	-75	310	1507	1659	152	1.34	11.70	0.653

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. May include up to 50% material below COG. DMA and FC series may include sludge and core values. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Conceptual targets north and northeast of the Hangar Flats deposit include at least four large, stacked mineralized zones known as the Sparky's Revenge, Fulgurite, NDMEA, and Crosscut prospects that are northeast striking, shallow to moderately northwest-dipping, and altered (Figure 9-19, Figure 9-20, and Figure 9-21). There are several other targets not discussed nor shown in Figure 9-19, mostly along the western side of the MCFZ and at depth, that are defined by geophysical interpretation and geologic inference.

The discovery of mineable mineralization in one or more of these targets could potentially affect or reduce strip ratios on the Hangar Flats pit due to deposit geometries and slope angles. There are several targets, all having similar dimensions, each potentially 100-200 ft thick by 600-800 ft along strike and 300-400 ft down dip. Collectively, they could amount to 5-20 million tons of mineralized material and could contain 150,000 oz - 875,000 oz in aggregate, assuming gold grades ranging from 1-1.5 g/t. However, the presence of steep slopes, floodplains, and other factors for some of the targets could make open pit operations problematic at some sites.

Figure 9-19: Plan Map of MCFZ Prospects, Geophysical Anomalies (l) and Geochemical Anomalies (r)

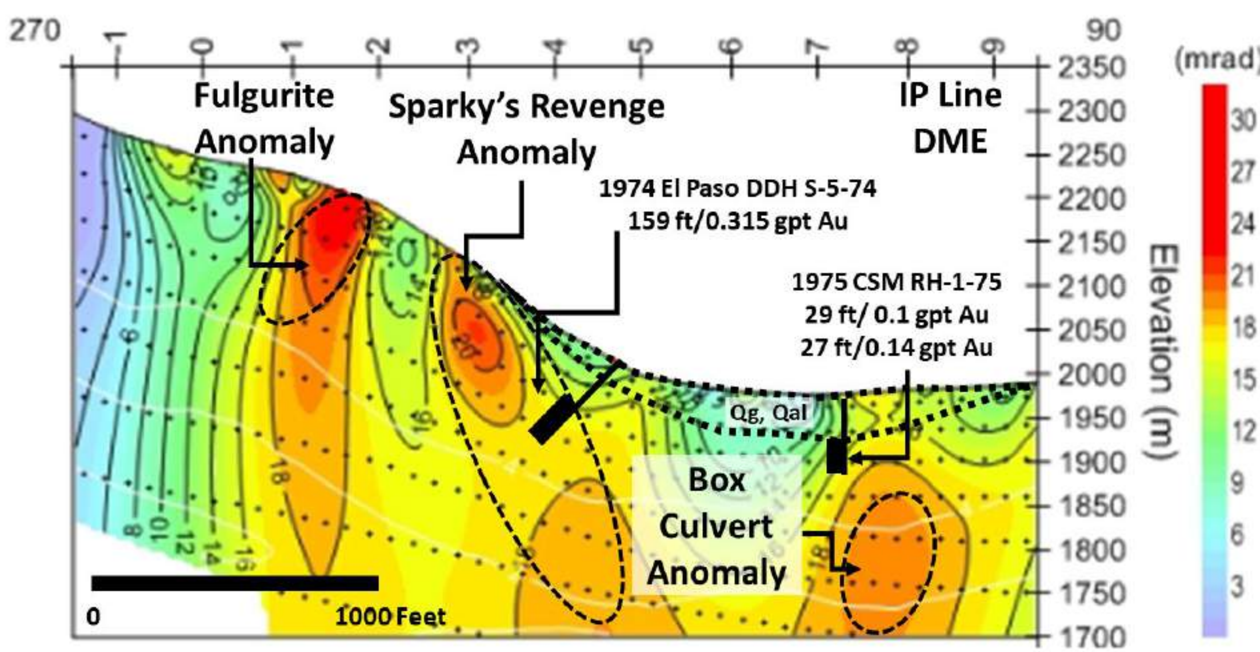


9.7.3.1 Sparky's Revenge

The Sparky's Revenge prospect does not outcrop and lies under 30-50 feet of glacial outwash and talus and is located at the site of a 1974 El Paso core hole S-5-74 that encountered strong pyrite-sericite alteration, veining, gouge, and multiple altered dikes. It cut a broad interval of low-grade gold mineralization (159 ft grading 0.315 g/t gold) from 259.5 ft downhole and bottomed in mineralization. A higher-grade interval was cut in the top of the interval averaging 0.57 g/t gold over 54 ft. The hole was a small-diameter (BX) hole and included core and sludge assays. The sludge samples showed locally higher grades and thus the actual grade of the material intersected in the hole is unknown. The mineralized intercept is coincident with a 1974 Canadian Superior IP-resistivity anomaly that was the hole's target. It is also directly along strike of northeast-striking, shallow, northwest-dipping mineralized structures that were intersected in Meadow Creek Mine workings over 2,000 ft to the southwest. If these zones connect, then it would suggest, as does the Fulgurite prospect, that mineralization may underlie much of the ridge east of the Hangar Flats deposit. A large gold-in-soils anomaly covers the ridge and its slopes, which have minimal outcrop.

This prospect is a conceptual target composed of the strong IP anomaly up-dip of the broad zone of low-grade gold and intensely altered core recovered from S-5-74. The reported angles of mineralized veins and structures logged in the hole by past operators are consistent with the interpreted northeast strike and shallow northwest dip of mineralization. A 2012 Midas Gold time-domain IP line (line DME, Figure 9-20) transected this feature to provide better data to evaluate the low-grade intercept. A series of several strong polarization and resistivity anomalies fall along the geophysical line coincident with either outcropping (Fulgurite) or gold mineralization revealed by drilling (Sparky's Revenge and Box Culvert). A very strong polarization response was encountered directly west of the drilled intercept in S-5-74 in an area covered with talus and closer to the MCFZ, which could represent increased gold-bearing sulfide concentrations and warrants drill testing. However, the dip indicated by the inverted IP line is the opposite direction (east) from the interpreted orientation of the structure. The feature may represent a dip reversal or another structural feature, but drilling results are insufficient to evaluate those possibilites. The strong IP response west and uphill from the S-5-74 intercept merits drilling to evaluate the target's potential.

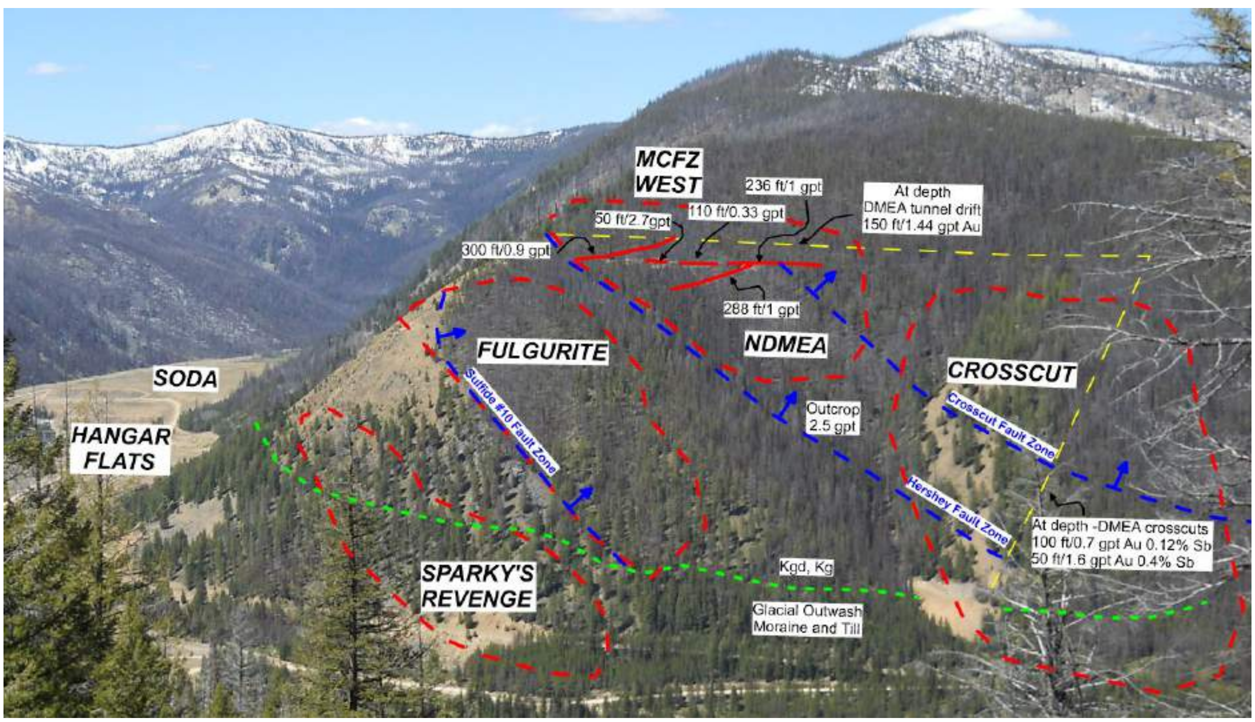
Figure 9-20: Inverted IP Profile, Line DME, Fulgurite and Sparky's Revenge Targets



9.7.3.2 Fulgurite Target

The Fulgurite Target extends northeast from the main Meadow Creek Mine area and was historically known as the Sulfide #10 prospect in Bradley Mining Company records and U.S. Bureau of Mines literature. A zone of mineralization is nearly continuously exposed for over 600 feet along strike in roadcuts, outcrops and historical trenches on a steep slope northeast of the former underground mine. In outcrop, the zone is relatively narrow, but consistently ranging from 15-25 feet thick (true) along strike until it is obscured by talus and glacial cover to the northeast. No significant drilling has been done on the zone along its northeastern extension away from the MCFZ. The zone was intersected its southwestern end in several holes including a Midas Gold drill hole approximately 425 feet down-dip. Another intercept was approximately twice the thickness of the up-dip outcrops and in the legacy DMEA drilling at similar grades. The alteration, mineralization, and fault structures in the Fulgurite/Sulfide #10 prospect area outcrops can be projected down-dip to a lens of mineralization adjacent to the high-grade core of the DMEA zone within the MCFZ. The zone shows consistent strikes and dips in outcrop well away from the MCFZ. The plunging high-grade shoot found underground at the DEMA Target could have several hundreds of feet of additional down-plunge extent along the intersection of the Meadow Creek and Sulfide #10 Fault. Grades averaging 8.55 g/t Au, 6.65 g/t Ag and 0.57% Sb over 71.5 ft of true width in MGI-09-07 support the possibility of underground development. The strike and dip are consistent with the majority of other known northeast-striking, shallow northwest-dipping faults within the Hangar Flats deposit itself. This a compelling area for early exploration drilling to investigate the potential for mineralization amenable to underground development.

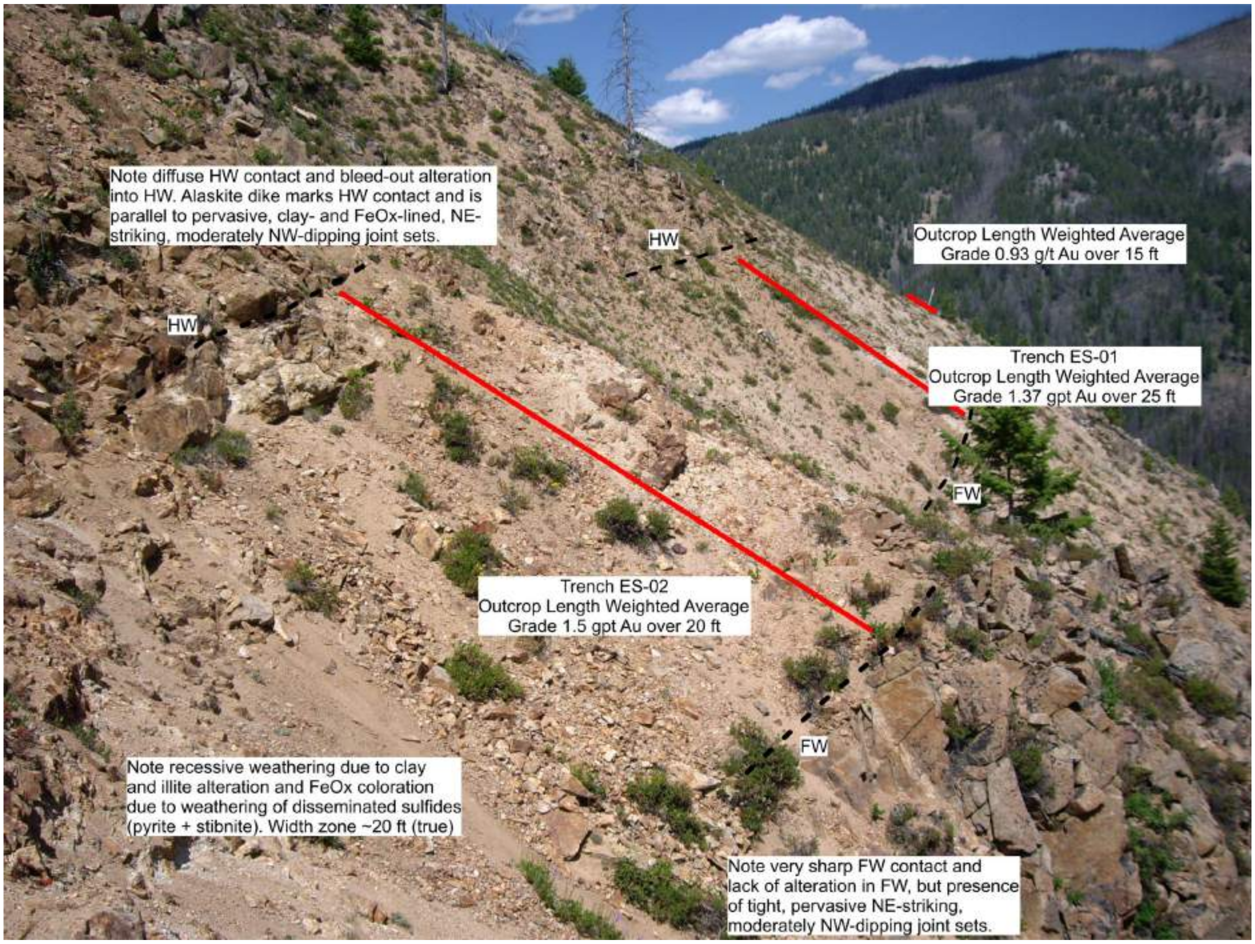
Figure 9-21: Photo looking west towards Sparky's Revenge, Fulgurite, NDMEA and Crosscut Prospects



Note: Blue lines are mapped, projected, or inferred faults; arrows indicate the dip direction. Red solid lines are 1988-99 Hecla and 2010-2011 Midas trenches yellow dashed line is vertical projection of DMEA exploration crosscut and drift vertically to surface; green dashed line is contact with bedrock derived soils upslope and glacial derived materials downslope; red dashed lines are target areas. Distances are along ground lengths and all locations approximate.

Data from outcrops and the limited drilling indicate a strike azimuth of approximately 030°-045° and dips of 35°-45° to the northwest similar to other lenses within the Hangar Flats area. Figure 9-22 is a photo showing the outcrop of a portion of the zone along the steep slopes northeast of the Hangar Flats deposit. A 2011 Midas Gold time-domain IP anomaly (line DME, Figure 9-20) coincides with the feature, suggesting the presence of increased sulfide concentrations and possibly higher grade and thicker gold mineralization at depth, closer to the MCFZ. A series of 42 hand-dug test pits covering the projection of the zone across the vegetated talus covered dip slope, where the zone projects between the Sulfide #10 Fault and surface projection of the NDMEA faults.Assays from the test pit samples ranged from 0.005 g/t gold to 3.76 g/t gold and averaged 0.71 g/t gold. All of the pits intersected intrusive rock that was intensely silicified, flooded with potassium feldspar, and impregnated with pyrite, similar in appearance to materials found on the DMEA dumps and in Midas Gold drilling into the DMEA target at depth.

Figure 9-22: Photo Looking Northeast Along Trace of Outcropping Fulgurite Target Mineralization



9.7.3.3 NDMEA Target

The NDMEA target represents outcropping and down-dip extensions of mineralization encountered in legacy Hecla Mining Company (**Hecla**) and Midas Gold roadcut and trench sampling, in shallow Hecla RC holes, and at depth in legacy underground workings (Table 9-10). The target is situated between the hanging wall of the Hershey Fault and footwall of the DMEA Crosscut structure. It extends from the ground surface to over 500 ft beneath it. Two parallel Hecla trenches in the 1980s intersected the zone northeast of the Meadow Creek Fault Zone. One Hecla trench above the road cut encountered approximately 300 ft of mineralization averaging 0.93 g/t Au and 300 feet along strike to the northeast. The zone was exposed below the road in another trench over 288 ft, averaging 1 g/t gold. The old trenches were widened and deepened between 2010-2012 during access road improvements. Resampling gave similar results and extended the zone to the north where a roadcut was excavated well into competent bedrock, averaging 1 g/t gold over 236 feet. Farther to the northeast and hundreds of feet downhill, a single outcrop exposed through glacial cover along the County Road cut slope averaged 2 g/t gold over approximately 25 feet, the limit of the exposure, suggesting the feature may have more strike length. The zone is expressed at depth in mineralization cut in the 1950s DMEA exploration drift 2DS, where legacy sampling intersected 210 ft of mineralized granitic rock averaging 4.02 g/t gold. A series of 12 hand-dug test pits completed in 2010 on the up-dip projection of the zone to the northeast encountered highly silicified, potassium feldspar-flooded, pyritic intrusive rock over a broad area with gold assay values ranging from 0.005 g/t to 0.375 g/t and averaging 0.131 g/t. Only limited legacy underground drilling has been conducted here, mostly to the south of the zone and at inappropriate orientations to adequately test the feature. The combination of broad zones of disseminated gold mineralization cut by higher grade Sb-W rich veins at the intersection of north-south and northeast structures is similar to the setting of the Hangar Flats and Yellow Pine deposits.

9.7.3.4 DMEA Crosscut

The DEMA Crosscut target consists of northeast-striking and shallow northwest-dipping disseminated pyrite-hosted gold mineralization and northeast- and north-south-striking steeper and higher grade crosscutting vein-related gold-antimony ± tungsten mineralization associated with several subparallel northeast-trending structures. These features were originally discovered in the Monday Tunnel in the 1930s and described in patent applications and mineral examiner field notes. The up-dip extensions of these zones were intersected in the 1950s DMEA exploration crosscut and along the main drift farther to the southwest (Table 9-10). The disseminated mineralization is reflected at shallow depths in samples from the main DMEA crosscuts 1XC, which cut 208 ft (approximately true width) averaging 0.59 g/t Au and 0.12% Sb, 3DS, which cut 57 ft averaging 1.44 g/t Au and 0.4% Sb, and the main 2DS drift, which hosts a broad low-grade interval along strike several hundred feet to the southwest. The veins while widespread and of good grades typically were widely spaced and narrow (0.3-1 ft in width), although in several areas maps show high vein densities suggesting swarms may be present in favorable structural settings as at Yellow Pine and Hangar Flats. Several short underground small diameter core holes were drilled through portions of the zone and cut mineralization similar in tenor to the drift and crosscut sampling (Table 9-11).

Table 9-10: Legacy Underground Sample Results for Crosscut and NDMEA Targets

Target	Operator	Location	Type Segment	Location ID	Type Sample	Composite Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Crosscut	Bradley	Monday Tunnel	Crosscut	5805	Rib channel	15	10.30	-	-
Crosscut	Bradley	Monday Tunnel	Crosscut	5665	Rib channel	8	5.10	-	16.97
Crosscut	Bradley	Monday Tunnel	Crosscut	5805	Rib channel	15	3.60	-	3.50
Crosscut	USBM	DMEA Tunnel	Crosscut	-	Rib channel	5	6.90	-	0.15
Crosscut	USBM	DMEA Tunnel	Crosscut	-	Rib channel	4.7	13.70	-	0.10
Crosscut	USBM	DMEA Tunnel	Crosscut	-	Rib channel	7.8	8.30	-	-
Crosscut	USBM	DMEA Tunnel	Crosscut	1XC	Rib channel	208	0.59	-	-
Crosscut	USBM	DMEA Tunnel	Drift	1D-S	Face advance muck	57	1.44	-	-
Crosscut	USBM	DMEA Tunnel	Drift	3D-S	Face advance muck	50	1.60	-	0.40
NDMEA	USBM	DMEA Tunnel	Drift	2D-S	Face advance muck	210	4.02	-	-
Note: Underground sample results digitized and captured from legacy maps and data.									

Patent survey documents from the 1930s indicate the Monday Tunnel, approximately 300 hundred feet below the DMEA workings cut intervals of disseminated mineralization and higher-grade veins and structures. Limited production (several hundreds of tons averaging ~3 g/t Au and ~3% Sb) from these zones is described in the patent documents but is not reported in the Bradley production records, possibly because the tonnages were comingled with Meadow Creek Mine materials. A detailed 2013 Midas Gold soil survey grid further defined the target with anomalies identified along the up-dip projections of the zones cut in the legacy Monday and DMEA underground workings and drill holes, outlining a Au-Sb-W soil anomaly 300 ft wide by 900 ft long several hundred feet up-dip of the zones cut in the workings.

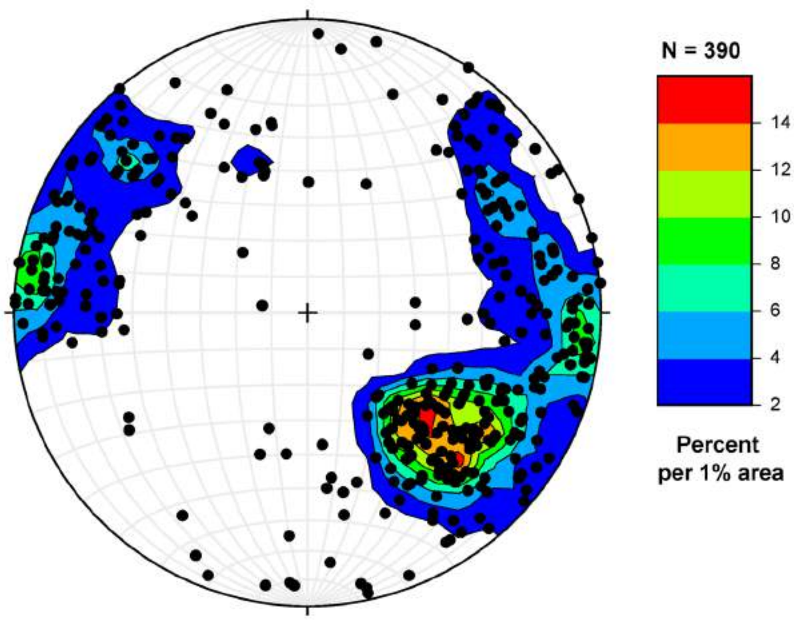
There is no modern drilling on this target. A stereonet plot of veins and mineralized faults compiled from historical maps of the collapsed DMEA underground workings is provided as Figure 9-23. It demonstrates the consistency of structural element trends in the prospects northeast of the Hangar Flats deposit. Limited Midas Gold oriented core fracture orientation data collected here is consistent with the legacy underground operation observations.

Table 9-11: Legacy Underground Drill Results for Crosscut and NDMEA Targets

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval ft)	Au (g/t)	Sb (%)
Crosscut	USBM	DMA-05	3	259.5	10	80	70	0.53	0.072
					115	140	20	0.49	0.042
					195	255	60	0.45	0.046
Crosscut	USBM	DMA-06	-1	157	10	70	60	0.54	0.041
					155	235	80	0.32	0.049
Crosscut	USBM	DMA-08	0	270	10	105	95	0.76	0.044
NMDEA	USBM	DMA-07	0	89	0	315	315	0.41	0.055
				incl	5	65	60	1.07	0.109
NMDEA	USBM	DMA-09	0	269	5	75	70	0.27	0.031
					130	195	65	0.49	0.060
NMDEA	USBM	DMA-10	0	106	0	55	55	0.76	0.061

Note: Composites minimum 15 ft with lower COG of 0.25 g/t Au. May include up to 50% material below COG. DMA series may include sludge and core values. Some intercepts may fall within 2020 Conceptual Mineral Resource Cone.

Figure 9-23: Stereonet Plot of Veins/Mineralized Faults from DMEA Underground Workings



Note: Data compiled from DMEA program maps and figures. Plotted using Stereonet, Richard W. Allmendinger © 2011-18.

9.7.3.5 Smokin' Boulder Target

This prospect is located near the site of several legacy trenches and prospect pits from the 1930s by Bradley and 1970s by Ranchers Exploration & Development Corp. (Ranchers). A spur trail off the main access trail crosses the trace of the MCFZ. Small prospect pits and cuts expose intensely altered granites and granodiorites with abundant disseminated pyrite and stibnite. Roadcuts below the old prospect pits were sampled in 2009 from outcrop or with hand augers and outline a broad area of anomalous gold (0.05-0.5 g/t) approximately 1,050 feet long in a north-south direction and over 300 ft wide in an east-west direction just east of the MCFZ and below and downslope of the historical trenches.

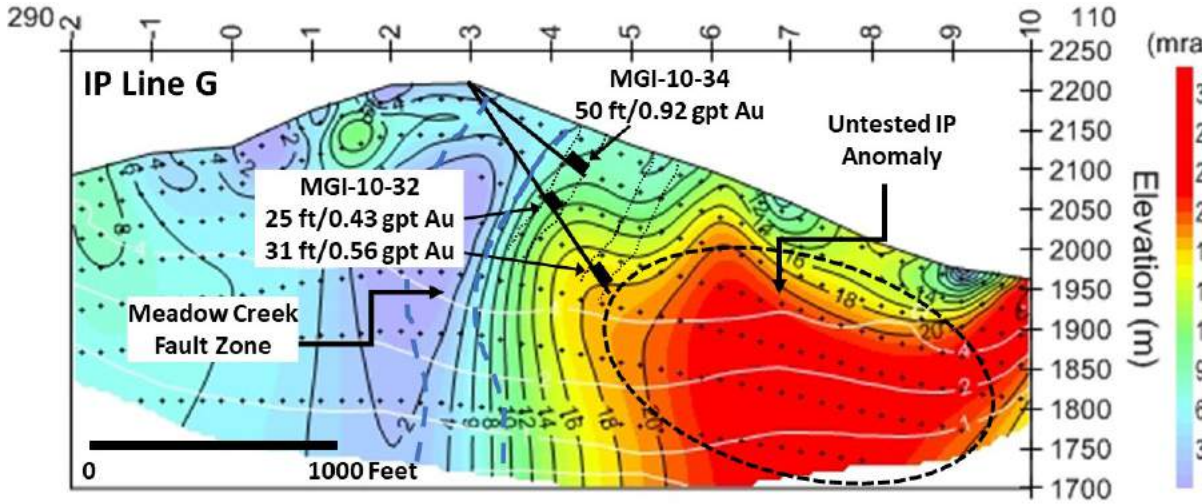
One of the old trenches above the roadcuts was reopened, sampled, and then reclaimed in 2010. The bulk of the western end of the trench was in heavily weathered, unmineralized gouge. The eastern end of the trench, which cut across an old prospect pit, contained anomalous gold (nine samples averaging 0.29 g/t Au, 2.67 g/t Ag, and 142 ppm Sb) over a 50 ft x 80 ft area. A follow-up soil grid covering this and a larger area in 2012 outlined a distinctive circular molybdenum anomaly as well as Au, As, Sb, and Hg over the target. Three shallow-angled holes were drilled in a fan pattern to the east and southeast from the trench where it crosses an old road (Table 9-12). All three holes intersected an upper narrow zone of mineralization 25-50 ft in true width, striking north-northeast, and dipping moderately steeply to the west. The deepest hole intersected another ~30 ft wide zone (estimated true thickness) deeper and farther east. The molybdenum soil anomaly is consistent with the presence of the younger suite of leucogranites (~83-85 Ma) in the district and an interval of leucogranite 60 ft wide with greisen containing abundant molybdenite with anomalous tungsten in one of the holes was encountered beneath the soil anomaly, suggesting the presence of a potential stock of leucogranite (similar to the Yellow Pine granite stock) at depth.

Table 9-12: Select Midas Gold Drill Intercepts for the Smokin' Boulder and North Tunnel Targets

Target	Operator	Drill Hole ID	Collar Inclination(°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Smokin' Boulder	MGII	MGI-10-032	-60	081	557	582	25	0.43	2.26	0.003
					1040	1071	31	0.56	1.24	0.004
Smokin' Boulder	MGII	MGI-10-034	-45	081	570	620	50	0.92	2.55	0.005
Smokin' Boulder	MGII	MGI-10-035	-60	254	750	770	20	0.31	0.57	0.004
North Tunnel	MGII	MGI-10-038	-50	087	200	210	10	0.14	0.83	0.002
					339	350	11	0.48	0.25	0.002
					402	427	25	0.25	2.54	0.008
North Tunnel	MGII	MGI-10-039	-45	070	41	89	48	0.37	0.30	0.011
					110	230	120	0.47	0.47	0.005
					244.5	360	115.5	1.12	0.69	0.006
					374	405	31	0.23	0.45	0.003
					455	482	27	0.48	0.84	0.006
					785	825	35	0.22	0.41	0.007
North Tunnel	USBM/BMC	FC-1*	1	118	15	65	50	0.60	0.09	-
					75	110	35	0.59	0.14	-
					225	255	30	0.51	0.07	-
					290	350	60	0.87	0.11	-
					410	475	65	0.30	0.07	-

*Note: Drill intercepts composited using 0.1 g/t COG over 10 ft for Au over minimum 10 ft. Reported composite may up to 25% material below COG. *Assays by core and sludge. FC series may include assays of core and sludge. Reported intercepts are estimated to be approximately 75-95% true width.*

Figure 9-24: Inverted IP Profile, Line G, Smokin' Boulder Target



Both holes MGI-10-032 and MGI-10-034 intersected five broad alteration zones ranging in width from 40-100 feet in estimated true thickness with highly anomalous pyrite content, Au, Sb, As, and other pathfinders. These zones all exhibit distinct potassium enrichment and sodium depletion similar in character to intrusive-hosted mineralization at Yellow Pine and Hangar Flats and suggest proximity to a larger mineralized system, perhaps representing a leakage halo along fracture systems to a deeper or adjacent zone. The presence of a molybdenum-bearing leucogranite could have provided additional fracturing in the cupola zone of the potentially younger granite stock in the surrounding granodiorites that could make the granodiorites more favorable hosts to disseminated mineralization. None of the holes tested the strong IP anomalies present on the adjacent 2010 IP Lines F and G between stations 500-700 farther to the east and down-slope (Figure 9-24) and these remain viable and high-quality drill targets given results from these scout drill holes and drill results along strike at the North Prospect approximately 900 ft to the north.

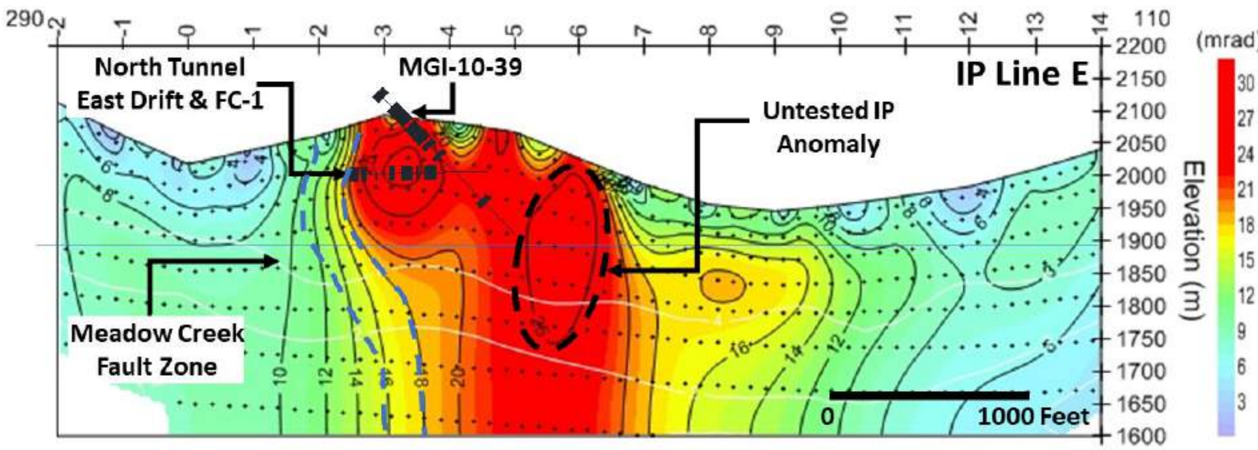
9.7.3.6 North Tunnel and IP Lines E and F Targets

The North Tunnel is a historical prospect from the 1930s Bradley era near an old, collapsed portal and development rock dump. Patent records and old newspaper reports indicate that an unknown and presumably minor amount of production occurred in the 1930s and 1940s that was probably reported with the Meadow Creek Mine and/or Yellow Pine production. The tunnel was driven south through glacial overburden into the MCFZ on a steep north-facing slope with no outcropping bedrock exposure along a distinctive recessive weathering topographic linear that follows the trace of the MCFZ from the Hangar Flats deposit north to the Fiddle drainage it is lost in glacial soils and overburden.

The workings consist of two drifts. Drill stations were established in the workings when they were reopened in the 1940s by the U.S. Bureau of Mines and Bradley Mining Company. Two holes were drilled underground. The western directed hole (FC-2) reportedly encountered gouge and unmineralized intrusive rock. The eastern drift reportedly had a 50 ft thick zone. Hole FC-1 was drilled easterly from a drill station in this drift and encountered several broad intervals approximately 240 ft thick in aggregate, containing anomalous gold mineralization (Figure 9-25, Table 9-12). The FC series holes were sampled for both sludge and core, but actual grades are uncertain due to small diameter core, use of sludge samples, and poor recoveries.

Two holes were drilled from the surface south of the North Tunnel in 2010 by Midas Gold (Figure 9-25, Table 9-12). The first hole, MGI-10-38, encountered a number of narrow low-grade intervals but was lost when the drill pad became unstable and the hole was abandoned.. Most of the hole intersected weakly altered, cataclastic rocks of the MCFZ. The second hole, MGI-10-39, was successful in testing part of the IP anomaly and penetrated several mineralized zones, likely correlative to the mineralized underground drill hole intervals to the north including an interval that averaged 1.12 g/t gold over 115.5 ft (approximate true width) coincident with the western and smaller portion of a much larger IP chargeability anomaly along IP Line E. This drill hole warrants step-out drilling. The large undrilled IP anomaly located just beyond the termination point of the drill hole is a compelling target.

Figure 9-25: Inverted IP Profile, Line E, North Tunnel Target



9.7.4 West End

There is potential to expand the West End Deposit at depth down-dip and along strike to the northeast and southwest, peripheral to the proposed Mineral Reserve pit. Most of the upper parts of the deposit that were previously mined were in oxidized or transitional materials and, in some cases, legacy operators only utilized cyanide-leach gold assay methods even when the holes intersected sulfide-bearing materials. This was the same for most of the prospects peripheral to the conceptual Mineral Resource pit, as described below. This could result in under-reporting of grades in these target areas and within zones of the West End Deposit itself. Some of the peripheral targets include Exit and Dead End targets on the northern end of the reserve pit; the Stibnite North, Tesla, and Switch targets to the southeast; and the South Midnight and Southwest Extension targets to the south and southwest. The Joule prospect is located east of the resource pit and is defined by soil, rock chip, and geophysical anomalies, but has never been drilled. Highlights of significant drill intercepts from these areas are listed in Table 9-13 and shown on Figure 9-26.

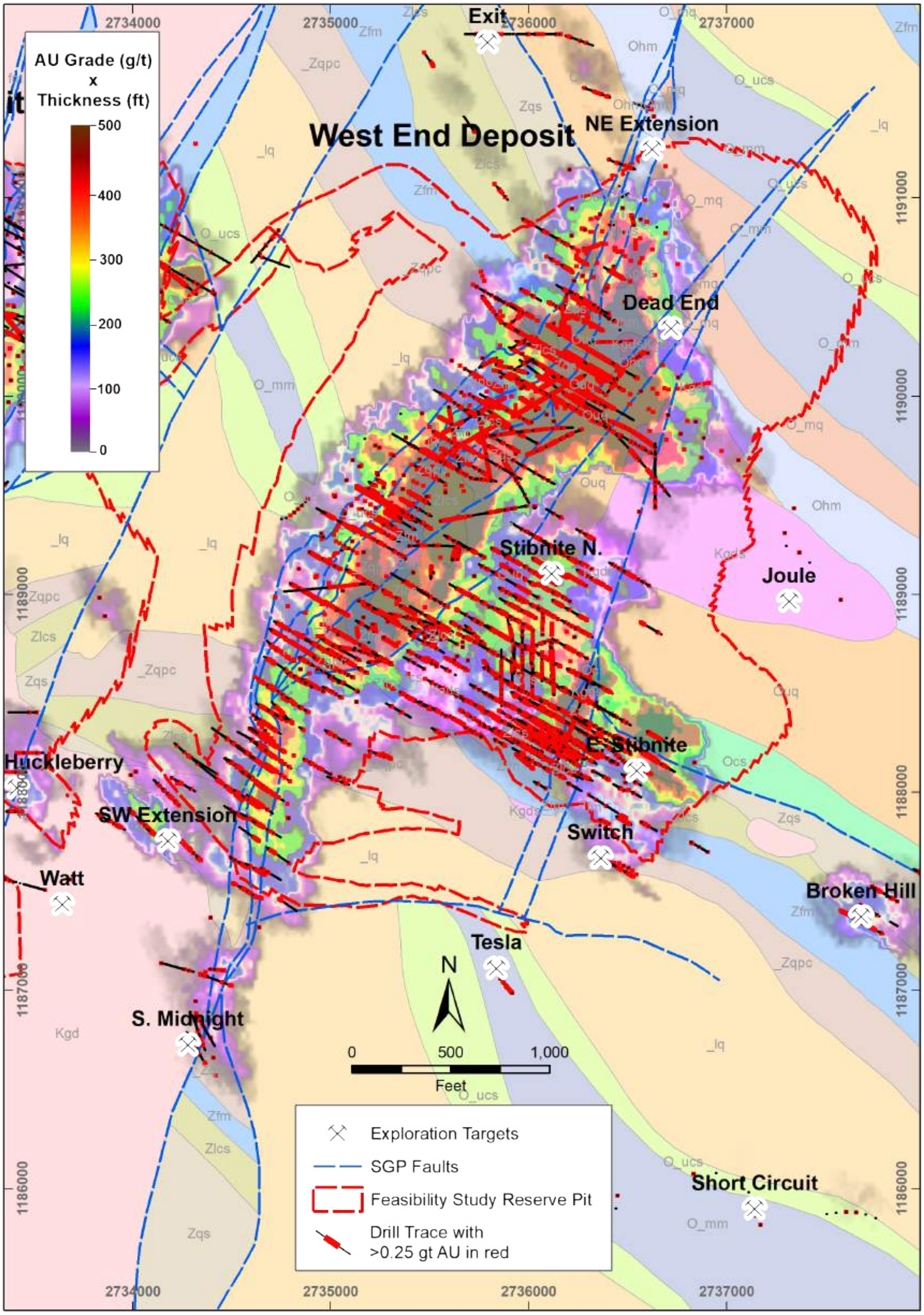
Table 9-13: Select Drill Intercepts for West End Expansion Targets

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Switch	Pioneer	PM-90-15	-70	294	115	135	20	0.85	-	-
Switch	Pioneer	PM-90-32	-70	300	135	170	35	0.83	-	-
Tesla	Superior	W-118	-55	140	75	105	30	1.10	-	-
						120	165	0.82	-	-
SW Ext	SMI	97-46LG	-50	235	240	270	30	2.19	0.69	-
SW Ext	SMI	97-47LG	-50	300	135	160	25	1.78	1.55	-
SW Ext	SMI	97-36SM	-55	300	140	160	20	1.00	12.33	-
					185	210	25	0.97	1.91	-
SW Ext	Superior	WER83-09	-55	304	170	190	20	0.96	0.90	-
SW Ext	MGII	MGI-12-316	-90	-	370	430	60	0.85	0.99	0.003
Huckleberry	Superior	WER83-34	-70	270	20	110	90	1.89	2.30	-
Huckleberry	Superior	WER84-05	-60	270	130	180	50	0.64	-	-

Target	Operator	Drill Hole ID	Collar Inclination (º)	Collar Azimuth (º)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
Huckleberry	Superior	WER84-06	-57	270	190	230	40	0.75	-	-
Huckleberry	Pioneer	PM91-12	-90	-	130	175	45	0.99	-	-
					210	235	25	0.76	-	-
South Midnight	Pioneer	PM90-04	-50	275	35	75	40	0.56	-	-
South Midnight	Pioneer	PM90-06	-50	150	120	155	35	0.71	-	-
South Midnight	Pioneer	PM90-08	-50	340	50	115	65	0.61	-	-
Exit	Superior	WER-83-26	-60	110	120	200	80	0.76	1.53	-
Exit	Pioneer	PM92-42	-55	270	0	40	40	0.253*	-	-
					155	255	100	0.211*	-	-
					540	590	30	0.92	-	-
					710	745	35	0.76	-	-
Exit	MGII	MGI-10-42	-90	-	552	572	20	2.67	3.20	0.004
Exit	MGII	MGI-10-50	-90	-	670	715.5	45.5	2.13	0.94	0.087
Exit	MGII	MGI-12-312	-90	-	492	511.5	19.5	1.25	1.67	0.008
Stibnite North	MGII	MGI-11-121	60	135	764	855	91	1.37	5.45	0.007
Stibnite North	MGII	MGI-12-273	-76	104	745	771	26	3.10	5.78	0.009
					810	1062	252	1.60	1.40	0.015
					1110	1185.5	75.5	1.60	1.40	0.015
Stibnite North	MGII	MGI-12-309	-69	249	893.5	923.5	30	1.19	0.34	0.002
Dead End	Superior	W-098	-90	-	285	340	55	0.72	1.99	-
Dead End	Superior	W-110*	-80	055	125	180	55	1.04	0.17	-
Dead End	Superior	WER83-23*	-90	-	110	340	230	1.10	3.00	-

*Note: Drill intercepts composited using 0.25 g/t COG over 20 ft for Au over minimum 20 ft with exception of those marked with a * which have a 0.1 g/t COG. Reported composite may up to 25% material below COG. Reported intercepts are estimated to be approximately 85-100% true width. Some intercepts may fall within 2020 Conceptual Mineral Resource cone.*

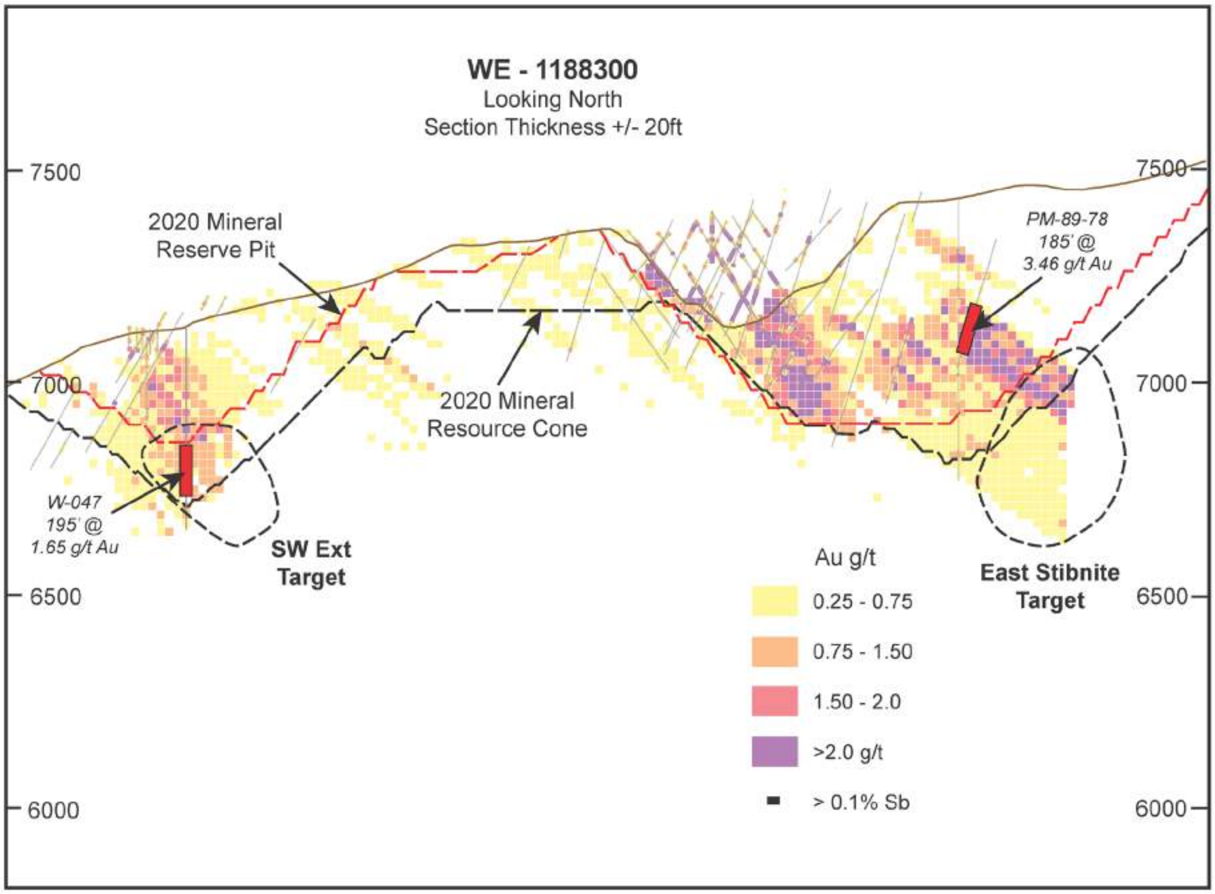
Figure 9-26: Significant West End Drill Intercepts and Expansion Targets



9.7.4.1 West End Down Dip Targets

The West End deposit is open down-dip along nearly its entire strike length (Figure 9-8, Table 9-14). The target consists of a poorly explored area 330 ft wide and extending approximately 2,100 ft along strike beneath the 2020 Mineral Reserve Limiting Pit.

Figure 9-27: E-W Cross Section 1188300N of the West End SW Extension and East Stibnite Targets



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

Table 9-14: Select Midas Gold Drill Intercepts Down Dip of West End Deposit

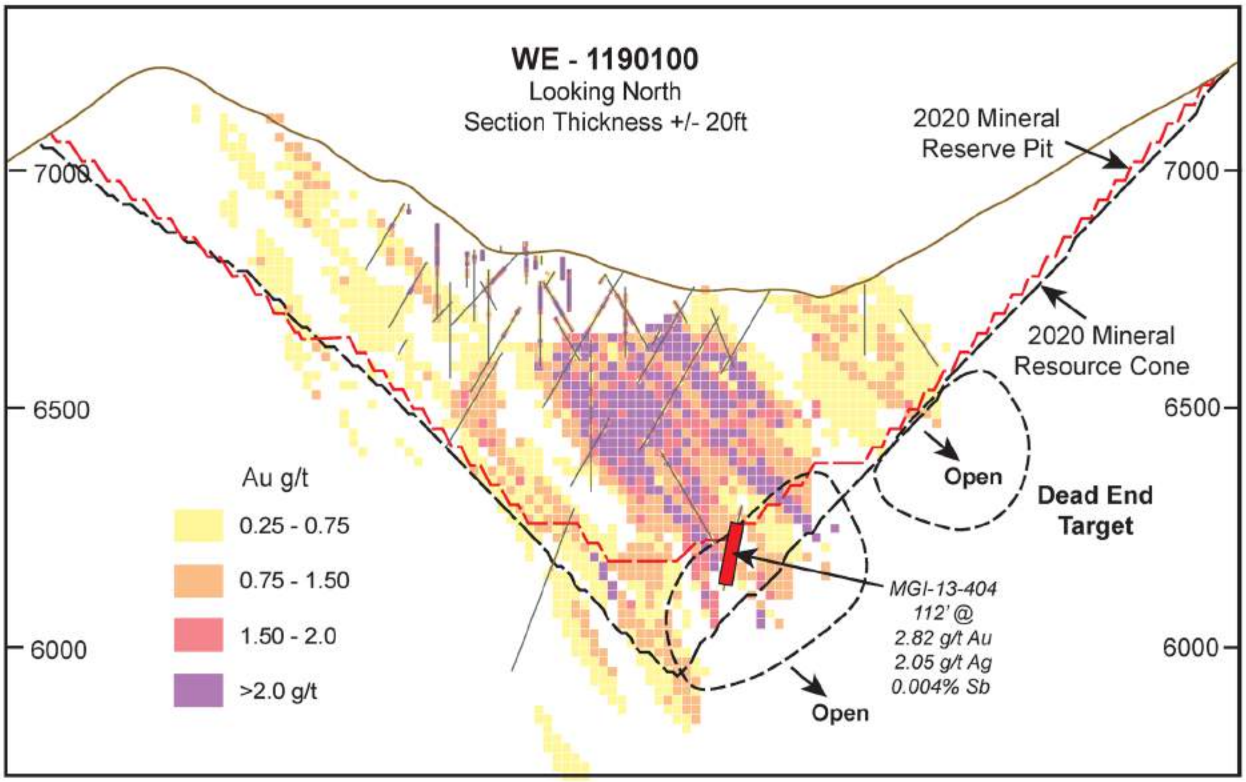
Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Ag (g/t)	Sb (%)
WE DownDip	MGII	MGI-11-139	-66	260	1120	1145	25	0.66	1.04	0.003
WE DownDip	MGII	MGI-11-142	-90	-	920	990	70	0.64	0.88	0.005
WE DownDip	MGII	MGI-12-254	-76	333	917	939.5	22.5	0.59	1.83	0.006
WE DownDip	MGII	MGI-12-282	-72	300	1030	1072	42	1.00	0.64	0.002
WE DownDip	MGII	MGI-12-286	-90	-	1095	1231.5	136.5	2.06	2.67	0.004
WE DownDip	MGII	MGI-12-290	-71	298	762	906	144	1.07	1.14	0.003
WE DownDip	MGII	MGI-12-294	-82	300	941	1054.5	113.5	0.81	1.47	0.003
WE DownDip	MGII	MGI-12-295	-90	-	953	1000.5	47.5	1.37	4.00	0.200

Note: Drill intercepts composited using 0.25 g/t COG over 20 ft for Au over minimum 20 ft Reported composite may up to 25% material below COG. Reported intercepts are estimated to be approximately 85-100% true width. Some intercepts may fall within 2020 Conceptual Mineral Resource cone.

9.7.4.2 Dead End Fault Target

The Dead End Fault Target lies below and along the northeast flank of the 2020 West End Mineral Reserve pit near the former NE Extension pit. Mineralization in the NE Extension pit is hosted within the Hermes Marble and the Stibnite Stock and is composed of dense quartz-adularia vein stockworks and sheeted vein arrays, biotite replacement by illite and sulfides, and polylithic breccias in a series of steeply dipping northeast striking faults.

Figure 9-28: E-W Cross Section 1,190,100N through the Dead End Target

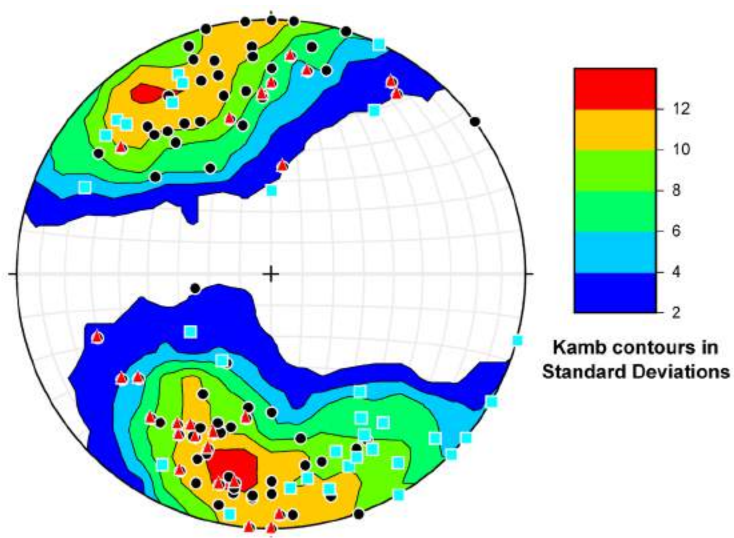


<u>Note:</u> Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

The target is defined by eight shallow holes drilled by pre-Midas Gold operators, with the most significant intercept being 230 ft with an average grade of 1.1 g/t gold along the east-northeast striking Dead End Fault that originates at a bend in the West End Fault system. Several of the holes bottomed in mineralization and/or are outside the limits of the current conceptual 2020 Mineral Resource cone boundary. Other favorable metasedimentary host lithologies, including a reactive iron-rich schist unit, may be present on the east side and/or beneath the stock intersecting the fault systems known to host mineralization. The stock is geometrically sill-like, but it and the adjacent and underlying lithologies remain poorly tested by drilling where they project into the fault system. The Exit target lies approximately 1,000 ft to the northwest and the former NE Extension Pit lies midway between the two prospects. Systematic rock chip sampling in the former Northeast Extension Pit highwall (Konyshev, 2000) outlined a continuous interval of 385 ft averaging 1.21 g/t Au and 5.18 g/t Ag, suggesting there are additional opportunities for drill targets between the Exit and Dead End prospects. The high Ag:Au ratio in the outcrop and drill samples in this prospect and adjacent prospects is atypical of disseminated sulfide-related mineralization elsewhere in the West End vicinity. It is more typical of the low to intermediate sulfidation quartz-adularia and low-temperature epithermal veins. This is consistent with field observations of dense arrays of west-northwest to east-west striking epithermal veins here (Figure 9-28). Konyshev (2020) reported relatively high Ag:Au ratios in probed pyrites from sulfides associated with veining at West End when compared to those associated with disseminated mineralization. Past drilling here has been poorly oriented to test all of these vein array features adequately.

The Dead End target is located beneath and beyond the boundaries of the former NE Extension Pit. The dimensions of mineralized material was determined by plan level and cross-sectional data derived from limited historical drilling. It outlines a conceptual potential open pit expansion target in the 1-5 million ton range. This conceptual target could contain 45-250 koz gold in a zone of approximately 50-100 ft thick by 500-800 ft wide and 800-1000 ft along strike at grades ranging from 1 g/t -1.5 g/t gold. The development of a new pit or pit expansion here could be inhibited by steep slopes, the legacy West End Creek stream diversion, historical development rock dumps, and the proposed adjacent West End pit if further exploration were to demonstrate a mineral resource for this target.

Figure 9-29: Stereonet Plot of Main Stage & Quartz-Adularia Veins from Northeast Extension Pit Mapping

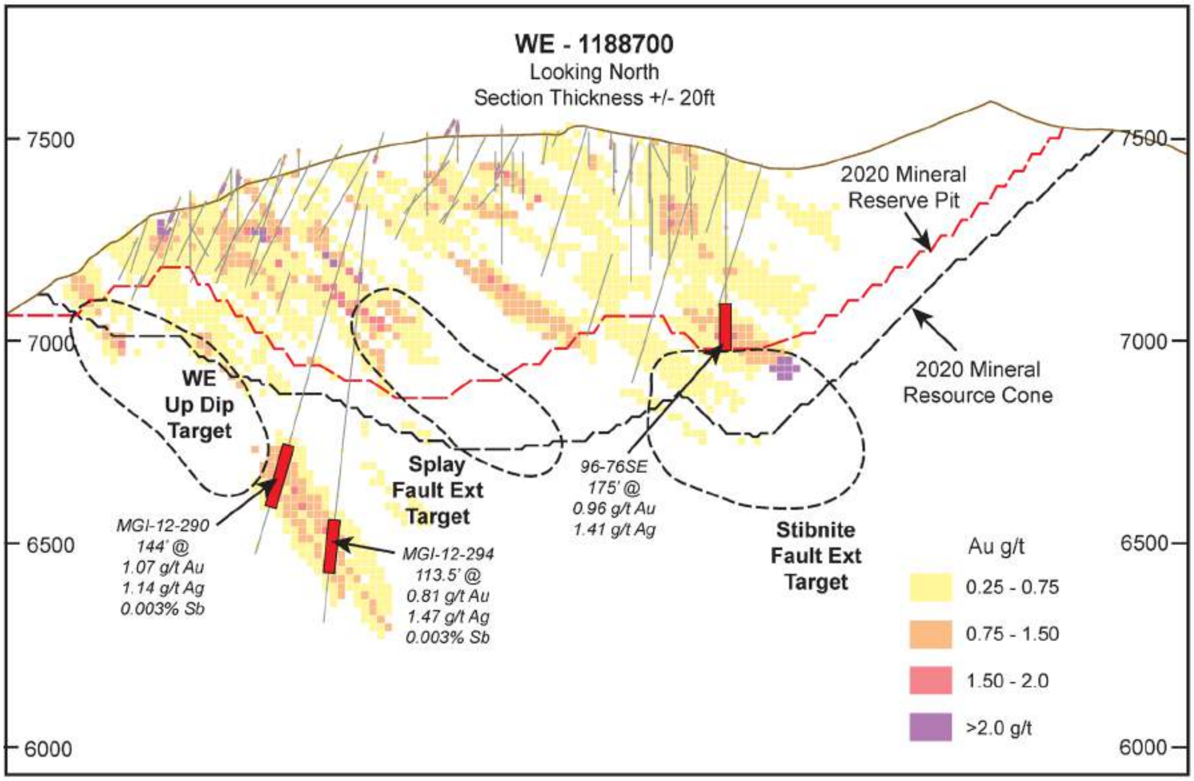


Note: Data from Konyshev (2020), N=122. Red triangles are deep early pre-gold event veins; blue squares are main stage and quart-adularia veins in marble; black circles are main stage and quartz-adularia veins in quartzite. Plotted using Stereonet, Richard W. Allmendinger © 2011-18.

9.7.4.3 Stibnite North Target

The Stibnite North target is defined by Midas Gold and Pioneer Metals drill holes (Table 9-13, Figure 9-30). Mineralization may continue down-dip and along strike within favorable faults and lithologies extending past the Mineral Reserve Limiting Pit. Limited outcrop exposures in old, partially backfilled roadcuts indicate the presence of abundant gold-bearing, quartz-adularia-sulfide veins. Structural analysis of the limited outcrop and drill data here suggests a northeast-striking vein swarm that is steeply dipping to vertical. That vein swarm intersects the lower calc-silicate sequence directly beneath the resource pit. The Ag:Au ratios in drill intercepts are consistent with the presence of these vein systems, which tend to have higher ratios than those in mineralization from the main West End Deposit.

Figure 9-30: E-W Cross Section 1,188,700N through the Splay and Stibnite North Targets



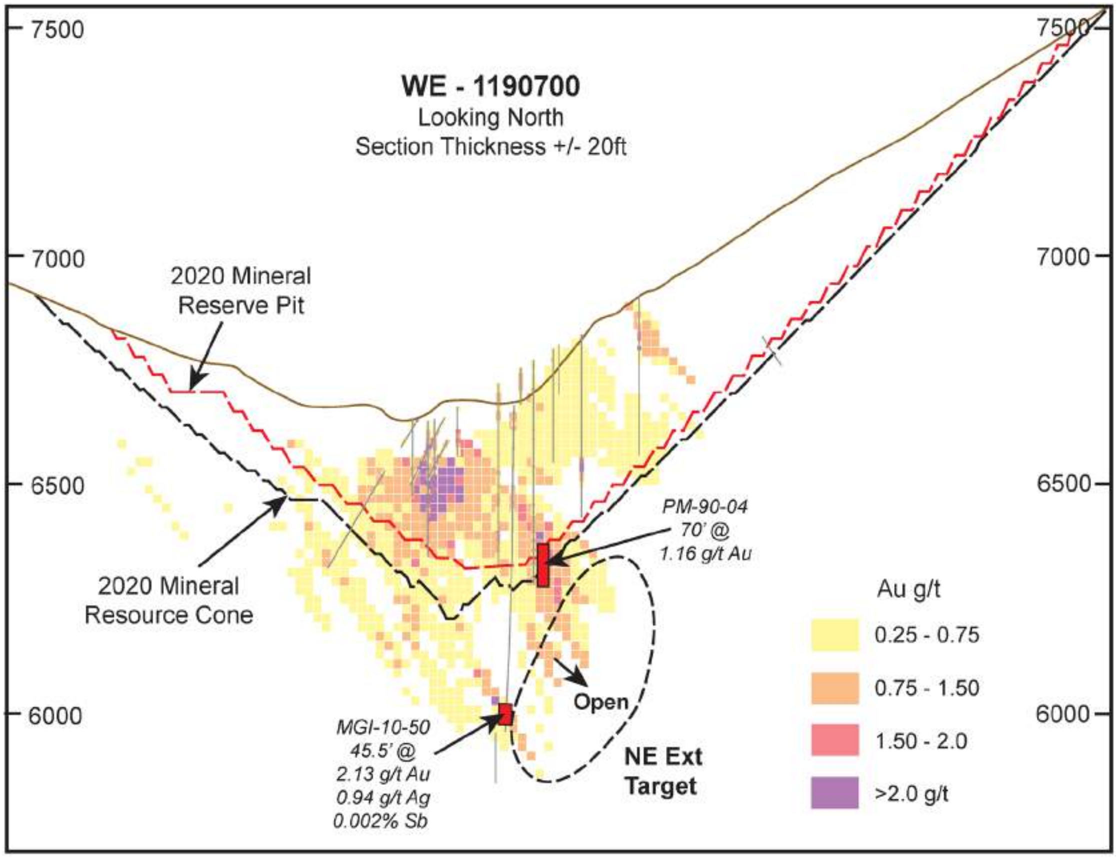
Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

9.7.4.4 Exit and NE Extension Targets

The Exit Target is located northwest of the main West End Fault Zone and includes an extension of the fault to the east-northeast. The area is identified by a strong surface soil and rock chip gold anomaly over an area of approximately 950 ft by 1,600 ft. Canadian Superior identified an apparently continuous zone of gold mineralization over 360 ft in outcrop within a distinctive sequence of magnetite schist and phyllite that averages 0.72 g/t gold in chip samples from road cuts (unpublished Canadian Superior maps and records). Midas Gold geologists confirmed portions of this anomaly by mapping and sampling exposures that were still accessible along the former road cuts. Adjacent outcrops show extensive quartz flooding and east-west trending, steeply dipping quartz-adularia vein arrays that have been inadequately tested by past drilling, warranting further exploration. There are several favorable structural features and stratigraphic intervals that define the target including the Cinnabar Ridge Fault Zone, the northeast extension of the Huckleberry Fault Zone, the Hermes Marble, Fern Marble, and the lower calc-silicate sequence, in addition to the schist sampled by Superior in the backfilled roadcut. Ten shallow legacy rotary and percussion holes with unfavorable orientations tested the near-surface portion of the anomaly. Several Midas Gold holes were drilled down-dip of the feature, cutting several intervals of anomalous gold and merit follow-up (Table 9-13).

A west-oriented 1992 Pioneer RC drill hole cut multiple intervals of gold mineralization down-dip of the variably mineralized outcrops that supports an open pit expansion exploration target. Many of the cuttings intervals were only assayed for cyanide leachable gold, despite the presence of sulfides as described in the drill logs. Thus, the actual total gold grades may have been higher than reported. The upper part of the hole cut anomalous gold associated with quartz-adularia veining within structurally prepared zones within the upper quartzite typical of the style of mineralization in that host rock unit. The Cinnabar Ridge Fault was intersected farther downhole and cut over 100 feet (approximate true width) of low-grade mineralization within the fault system itself averaging 0.21 g/t gold. The quartzite-schist, lower calc-silicate, and Fern Marble sequences were intersected across the northwest side of the Cinnabar Ridge Fault. The grade of gold mineralization encountered within both units was above the FS reserve cut-off grade.

Figure 9-31: E-W Cross Section 1,190,700N through the NE Extension Target



Note: Potential mineralization shown here may be partially included within the 2020 Conceptual Mineral Resource Cone as discussed in Section 14 of this Report.

The Exit target is adjacent to and northwest of NE Extension target (Figure 9-31) which is located within and along the extension of the fault northeast of the former West End Pit. A conceptual target composed of the Exit and NE Extension zones generally trends northwest-southeast within favorable stratigraphic units where they cut northeast and north-south structural features similar in style to mineralization in the adjacent West End Pit. The target is located beneath and beyond the boundaries of the former open pit. The dimensions of mineralized material is estimated from level plan and cross-sectional data from historical sources and outlines a potential open pit expansion in the range of 3-12 million tons.This target could contain 45-360 koz gold in a zone of stacked mineralized intervals approximately 100-250 ft thick (in aggregate true width) by 250-400 ft down plunge by 1,000-1,500 ft along strike at grades ranging from 0.5 g/t gold to 1 g/t gold. The development of a new pit or pit expansion could be impaired by steep slopes, the legacy West End Creek diversion, historical development rock dumps, and the proposed adjacent West End pit if further exploration were to demonstrate a mineral resource for this target.

9.8 POTENTIAL HIGH-GRADE UNDERGROUND MINING PROSPECTS

9.8.1 Scout

Scout is a potentially underground-mineable Au-Ag-Sb exploration prospect discovered in the 1930s by Bradley interests and further evaluated during Strategic Minerals investigations in the 1940s. Detailed exploration by other operators followed between 1947 and 1990 and included IP, VLF electromagnetic surveys, mapping, drilling, and resource estimation. Pre-Midas Gold drilling includes 18 holes totaling 6,912 ft. Midas Gold work includes IP and CSAMT surveys, mapping, rock and stream sediment sampling, and completion of 21 drill holes totaling 15,629 ft. Table 9-15 presents the results of the significant intercepts from that drilling.

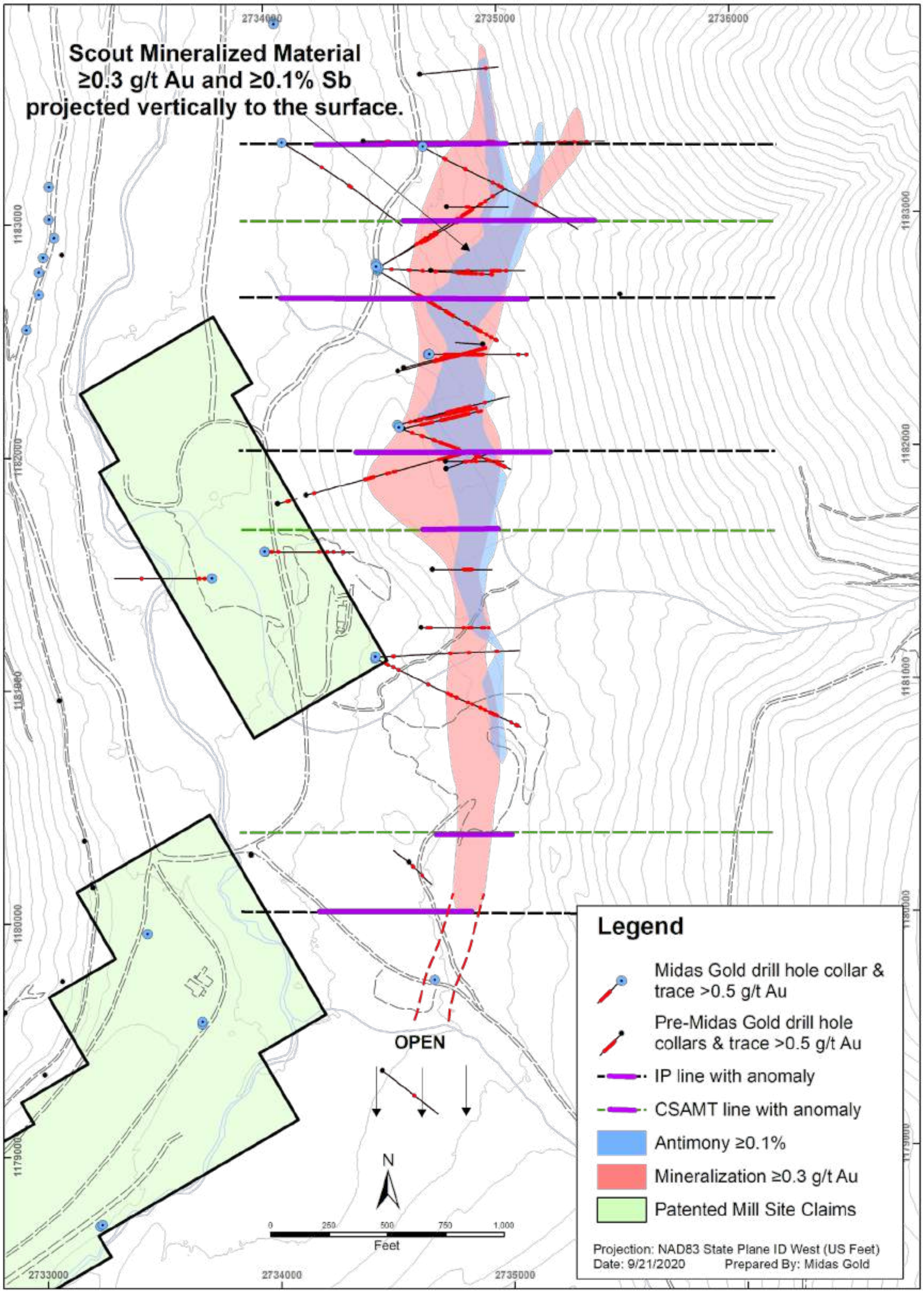
Mineralization at the Scout prospect is hosted by quartzite, schist, quartz diorite, and monzonite. Controls on mineralization are related to the Scout Valley Fault Zone, which trends north-south and dips steeply west, and includes east-west and southwest-northeast faults. Drilling results are insufficient as yet to define a mineral resource, but suggest a potential underground target. The dimensions of the potential target were estimated by simple polygonal and sectional methods from drillhole data and supplemented by trench and geophysical survey data. The target has dimensions of approximately 25-75 ft in true thickness, 2,000-3,000 ft along strike, and 250-300 ft down-dip at grades ranging from 1-2 g/t Au, 1-4% Sb, and 5-25 g/t Ag., The potential underground target is in the range of 2-5 million tons and contains between 50,000-300,000 oz Au; 40-150 million lbs Sb; and 300,000-1,500,000 oz Ag. Mineralization is open to the south, where monitoring well MWH-B08 cut 35 ft of 0.98 g/t Au and 40 ft of 0.97 g/t Au with 0.21% Sb coinciding with an IP and CSAMT anomaly.

Table 9-15: Select Drill Intercepts for the Scout Prospect

Drill Hole ID	Operator	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Au(1) (g/t)	Ag (g/t)	Sb(1) (%)
MGI-12-198-RC	Midas Gold	-90	-	294.9	335.0	40.0	0.83	-	1.9
				565.0	605.0	40.0	2.16	-	1.1
MGI-12-238	Midas Gold	-66	77	683.1	769.0	86.0	1.33	7.36	1.06
MGI-12-244	Midas Gold	-45	77	305.4	429.1	123.7	2.37	5.88	0.5
				including					
				331.4	363.8	32.5	5.70	15.30	1.46
MGI-12-249	Midas Gold	-53	115	311.7	862.5	550.9	0.78	14.80	2.02
				including					
				419.3	490.8	71.5	0.82	43.5	4.63
MGI-12-302	Midas Gold	-45	120	495.1	534.4	39.4	4.55	4.65	1.71
				651.6	677.5	25.9	1.68	8.42	2.86
MGI-12-345	Midas Gold	-44.6	116	764.1	816.9	52.8	1.68	48.0	5.42
MGI-12-347	Midas Gold	-50	90	771.3	784.4	13.1	5.96	114.6	12.3
MC-58	USBM	-20	75	625.0	730.0	105.0	1.77	-	0.3
MC-60	USBM	-45	77	109.9	419.9	310.0	1.0	-	0.33
				464.9	487.9	23.0	2.87	-	0.19
S-04-74	Superior	-45	90	276.6	325.8	49.2	-	-	1.44

Note: Selected intercepts composited with length weighted averages with COG of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb. Reported drill intercepts are approximate true widths.

Figure 9-32: Plan Map of the Scout Prospect



9.8.2 Garnet

The Garnet prospect is the site of past underground exploration in the 1920s-30s and a later open pit in the 1990s. The prospect was known in the 1920s as the Murray Prospect when at least two, short underground adits were excavated on antimony-tungsten-gold occurrences (Cooper, 1951). In the 1940s the prospect was briefly examined by the Bradley interests for antimony and tungsten, but there was minimal work and no reported development during that era. The former underground prospects located to the northwest of the Garnet Creek drainage were reported to have been reopened as dozer cuts in the 1960s and produced and shipped minor amounts of hand-cobbed antimony ore (<100 tons) by the Oberbillig interests from large stibnite veins exposed in the now collapsed and mined out Murray Prospect upper adit (Savage, 1963). The prospect is a potential underground exploration target, but there is a remote possibility of an open pit target as well. Steep slopes likely would impede economic development of an open pit of any significant size for mineralization identified to date due to the interpreted geometry which plunges into the hillside.

Bannister (1970) outlined a gold anomaly (> 600 ppb) in ashed humus samples 2,500 ft long by 1000 ft wide centered on the Garnet prospect during USBM Strategic Metals investigations. A joint venture comprising El Paso Oil and Gas and Superior expanded on the Bannister biogeochemical work with conventional soil sampling in the area leading to the discovery of a broad zone of outcropping high-grade gold mineralization (>30 g/t gold) here in the mid-1970s and, between 1974 and 1995, four other companies explored the prospect. Superior Mining Company reported a small reserve here in 1978 (Jayne, 1978) and subsequently completed a feasibility study (Superior, 1981). Pioneer updated the reserve and feasibility study in 1988 for an open pit but did not develop the prospect. These earlier reserve estimates were completed prior to NI 43-101 reporting requirements and are not considered relevant nor reliable by the Qualified Person. A small open pit was opened here by Stibnite Mine Inc. in 1995, which was abandoned after the northeast highwall of the pit began to fail. Production from the pit was leached on the on-off leach pads in the Meadow Creek valley. For approximate historical production records see Section 6 of this Report. The pit was partially backfilled with material reportedly from the pit development rock and spent ore from the West End pit (Jim Egnew, USFS Minerals and Geology Program Manager, personal communication to Chris Dail, 2010) and reclamation was completed in 1998. An active scarp above the former pit highwall remains unstable and could impact any future development should results dictate.

Pre-Midas Gold drilling includes 105 RC, core, rotary, and percussion holes totaling 16,261 ft. There are numerous unmined intervals of gold mineralization reported from beneath the limits of the former open pit ranging in apparent thickness from 15 feet to over 100 feet present in over 40 of the holes. Many of the holes bottomed in mineralization beneath the pit and some intersected significant mineralization adjacent to, but outside, the former pit limits. The length-weighted average grade of pre-Midas Gold down-hole drill composites that remain underneath the former open pit (using cutoff detailed in Table 9-16) is 2.4 g/t Au with an average interval width (90-100% of true width) of 52 feet. Using a higher cut-off grade for composites (as outlined in Table 9-16) a high-grade zone with length-weighted average grades and widths that could potentially represent a potential underground target. Data from 22 holes beneath the pit outline cut high-grade mineralization (using a 3 g/t gold cutoff over minimum intercept width of 5 feet) with a length-weighted average of 7.8 g/t gold over an average interval width of 12.3 ft (approximately 90-100% of true width). Highlights of some of the broad low-grade and narrower high-grade drill intercepts in the unmined portions of the prospect are tabulated in Table 9-16. Fire assay grades of the material mined were approximately twice the cyanide leachable grade reported in the blast hole data. Many of the remaining intervalswere not tested using fire assay methods despite the reported presence of sulfides and contain long low grade intervals of cyanide-leachable gold with highly anomalous values. Whether these intervals contain mineralization with higher grades is uncertain. Midas Gold work includes mapping and rock, soil, and stream sediment sampling, but no drilling.

Mineralization occurs in heavily dolomitized Fern carbonate, retrograde tactite skarn, laminated calc-silicate, quartzites, and locally in two granitic sills within and adjacent to a series of closely spaced faults in a small isolated roof pendant.It lies along steep valley wall slopes in an west-northwest-striking, north-dipping block of metasedimentary rocks that are assigned to the Fern and lower calc-silicate sequences of Stewart et al. (2016). Extremely iron-rich and retrograde tactite skarn occurs both in the Fern carbonate and lower calc-silicate adjacent to the sills and is composed of calcic garnet, calcium-iron-rich pyroxene, epidote, wollastonite, scapolite, and hematite after magnetite with minor pyrrhotite and trace amounts of chalcopyrite. Locally, endoskarn is developed in the sills themselves.

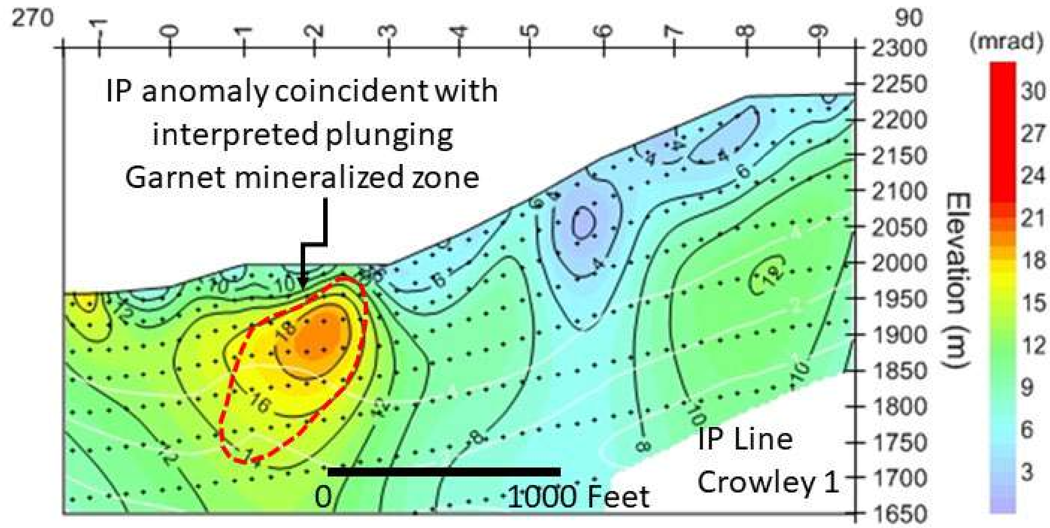
Wintzer (2019) dated zircon rims in garnet-clinopyroxene skarn from the Garnet pit yielding an age of 82.0 ± 2.2 Ma and the cores gave an age of 99.1 ± 1.1Ma. The skarn was adjacent to a granite that was dated at 98.1 ± 1.1 Ma and small leucrogranite dike cutting the granite was dated at 88.4 ± 1.4 Ma. The cores of zircons in the skarns gave ages similar to the adjacent granite and the ages are within the margin of error for the samples suggesting the skarn is locally endoskarn and not developed in the metasedimentary rocks. The leucogranite dike age would indicate intrusive active later than the intrusion of the granite at ~98 Ma and earlier than the ~82 Ma skarn-forming event. There are no dates on the later sulfide mineralization which appears epithermal in character, but based on other occurrences in the District, gold mineralization is likely late Cretaceous or more likely Tertiary.

Gold is associated with disseminated arsenical pyrite and antimony with massive stibnite veins in sulfide- and silica-impregnated zones. Mineralization occurs within a northerly plunging body developed at the intersection of north-south, northeast-southwest, and northwest-southeast fault zones steeply to moderately dipping to the west. The main body of mineralization appears to be associated with the northeast-southwest Garnet Greek Fault Zone (GCFZ) dipping to the northwest at its intersection with the Fern dolomite creating a north-northwest plunging ellipsoidal body of mineralization. Legacy blast hole data suggest a sharp footwall within the mined-out portion of the prospect that may represent the GCFZ, which is no longer exposed due to pit backfill and slumping on the pit walls. The carbonate unit is exposed in a west-northwest body approximately 1,300 ft long and approximately 250-500 ft wide. The unit has been intersected down-dip beneath granite sills and a quartzite sequence several hundred feet down-dip to the north and west indicating the favorable host rock body has much larger dimensions than the outcrop pattern would suggest. The lower calc-silicate unit is a highly favorable host rock elsewhere in the Stibnite roof pendant and is present sporadically in the deeper drill holes. The metasedimentary sequence is intruded by at least two altered and locally mineralized north-striking(?), east-dipping(?) granite sills.

The high iron content in the skarn made it a favorable host rock and the iron acted as a reductant for sulfide precipitation from hydrothermal fluids and likely is at least partly responsible for the high grades found in the prospect. The favorable Fern and lower calc-silicate stratigraphy extend beneath the upper sill northwest of the GCFZ for several hundred feet based on legacy drilling and may host mineralization to the northwest where other northeast-trending faults are known or suspected to be present. That interpretation is supported by strong soil, rock chip, and stream sediment anomalies and pronounced IP anomalies in the area immediately north of the pit.

There are several targets in and around the former Garnet open pit (Figure 9-33). Mineralization is open down dip of previously mined mineralization hosted within the Fern marble unit (A on Figure 9-33). Much of the past drilling did not penetrate the lower calc-silicate or was not assayed due to sulfide content since the 1970s-1980s drilling was targeting cyanide-leachable oxide ores. Specifically, the intersection of the lower calc-silicate unit, a favorable host sequence elsewhere in the District, and the GCFZ is mostly untested and a promising target (B in Figure 9-33). Holes drilled to the south and west of the open pit were too shallow to have tested that intersection, as were holes to the north. A low resistivity, high chargeability feature at the projected depth and location of the interpreted mineralized body was identified in both an east-west 1976 El Paso frequency-domain EM line approximately 150 ft north of the pit and 2011 Midas Gold time-domain IP line, Crowley 1, approximately 450 ft north of the pit (Figure 9-32).

Figure 9-33: Inverted IP profile, Line Crowley 1, Approximately 450 ft North of Former Garnet Pit



Approximately 600 ft west of the former open pit (C on Figure 9-33) a small soil grid was established to evaluate altered outcrops and the main body of the lower calc-silicate sequence's intersection with the northeast-trending Saddle Fault which extends to the Doris K and Saddle prospects to the northeast and projects towards the Hangar Flats Deposit to the southwest. The grid returned highly anomalous gold, silver, arsenic, and antimony results over its entire area, which is typical of mineralized areas elsewhere in the district. A 250 ft by 350 ft area was covered by three soil lines and eighteen samples. Gold values in minus 80 mesh soils from 0.35 g/t to 2.04 g/t and average 1.37 g/t at the intersection of the fault and the lower calc-silicate. Past drilling just east of this area and west of the open pit along old haul roads (in the late 1970s and early 1980s) only utilized CN leach assay methods yet reported long intervals (hundreds of feet) of detectable gold in the 0.X g/t gold range. The drill holes were typically too shallow to test the metasedimentary rocks or intersected mineralized metasedimentary rocks, suggesting the presence of a large area of stratiform-style mineralization within the lower calc-silicate. Actual gold grades in these old boreholes cannot be determined with available data. This area should be redrilled and samples fire assayed to determine actual gold grades.

The upper sill is poorly exposed in the high wall of the partially backfilled open pit. It was intruded along the unconformable contact with the overlying quartz pebble conglomerate unit and lower quartzite sequence. The lower sill appears to have intruded along the lower calc-silicate and underlying quartzite schist sequence, but locally has assimilated the lower calc-silicate unit. Both sills are locally mineralized. Disseminated mineralization similar to that found at Hangar Flats and Yellow Pine may be present in the sills where they intersect the GCFZ or the Saddle Fault structures.

The upper Murray adit and several prospect pits along the projected trace of the Saddle structure show evidence of this disseminated style of mineralization (biotite replacement by sulfides and sericite) and disseminated stibnite and large stibnite veins (D on Figure 9-33). Soil and rock chip sampling indicate gold and antimony are present over a broad area above where these sills would intersect favorable structures.

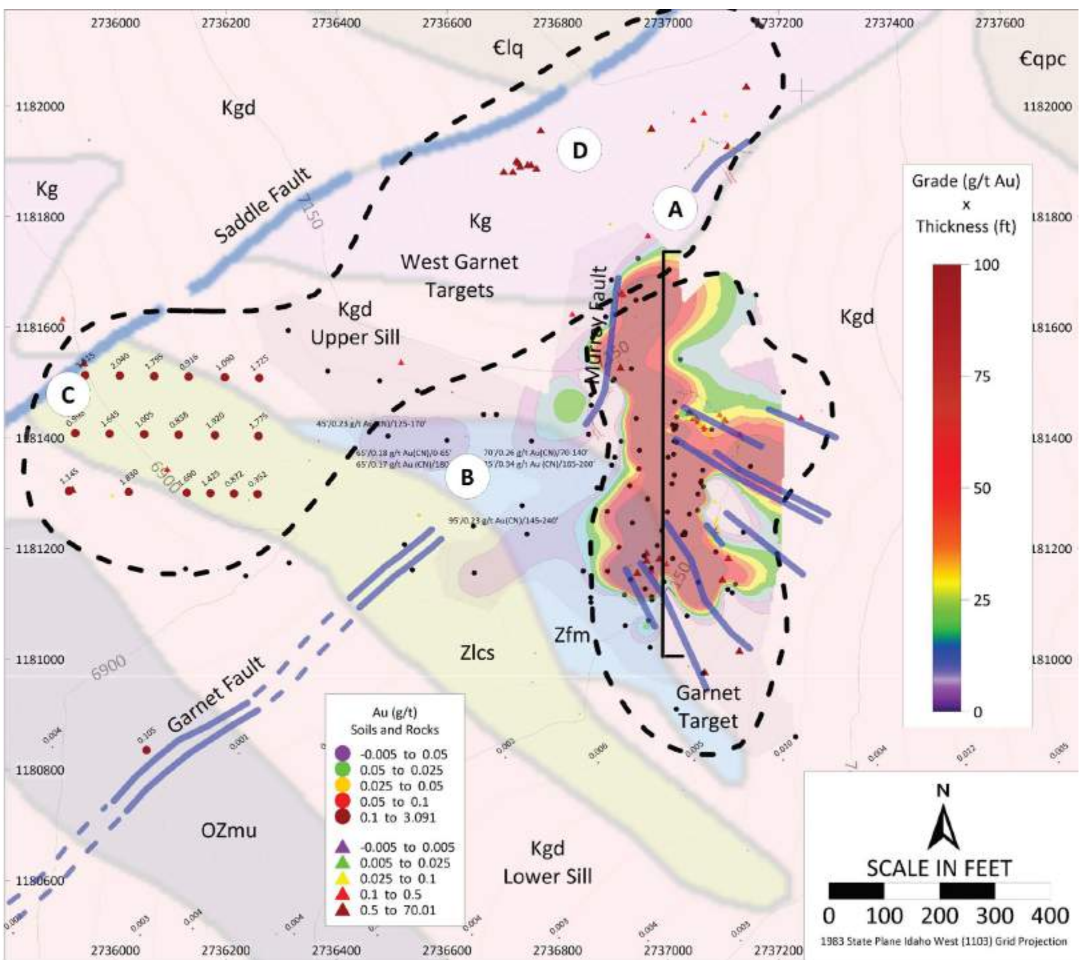
The conceptual underground target down the plunge of the mineralization exploited in the former open pit generally trends north-northwesterly from the Garnet Pit (Figure 9-33). The dimensions of mineralized material located beneath and beyond the boundaries of the former open pit are 30-60 ft thick (true) by 160-250 ft wide by 1,300-1,800 ft down plunge at grades ranging from 5 g/t to 8 g/t gold as estimated from geophysical data and historical drilling using level plans, cross sections, and polygonal methods (Table 9-16). The potential underground target ranges from 1 to 2 million tons and contains 250-500 koz gold. Other targets surrounding this provide further upside potential.

Table 9-16: Select Pre-Midas Gold Drill Intercepts below 1995 Garnet Pit

Drill Hole ID	Operator	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold[1] (g/t)
78-01GD	Superior	-90	0	175.0	190.0	15.0	18.52
G-07	Pioneer	-90	0	90.0	125.0	35.0	3.07
G-10	Pioneer	-90	0	100.0	185.0	85.0	3.06
RH76-25	El Paso	-90	0	143.0	158.0	15.0	9.13
S-23-76	El Paso	-55	337	186.0	196.5	10.5	7.18
S-29-76	El Paso	-62	189	215.0	287.0	72.0	3.35
				including			
				262.8	278.5	15.7	9.73
Xray-05-75	El Paso	-90	0	127.0	158.0	31.0	7.24
				including			
				135.0	148.5	13.5	15.56

Note: (1) Drill hole composites over 3 g/t Au reported, >30 ft composite length, and <10 ft of internal waste below 0.5 g/t Au. Higher-grade composites >6 g/t reported, >10 ft composite length, and <5 ft of internal waste below 3 g/t Au. These intercepts are located beneath the bottom of the former open pit and estimated true widths are 90-100% of the reported intercept lengths.

Figure 9-34: Plan Map of Grade x Thickness at the Garnet Prospect



Note: Grade x thickness calculated using all data and includes mined out material within the former open pit. Data gridded and summed using inverse distance squared and a 25 ft x 25 ft grid and smoothed. Grades below lower detection limit given zero value and data includes cyanide and fire assay analyzed samples. Gridding did not utilize pit blast hole data. Geologic unit symbology same as on Figure 9-1.

Figure 9-35: N-S Long Section of the Garnet Prospect with a 150 ft Corridor

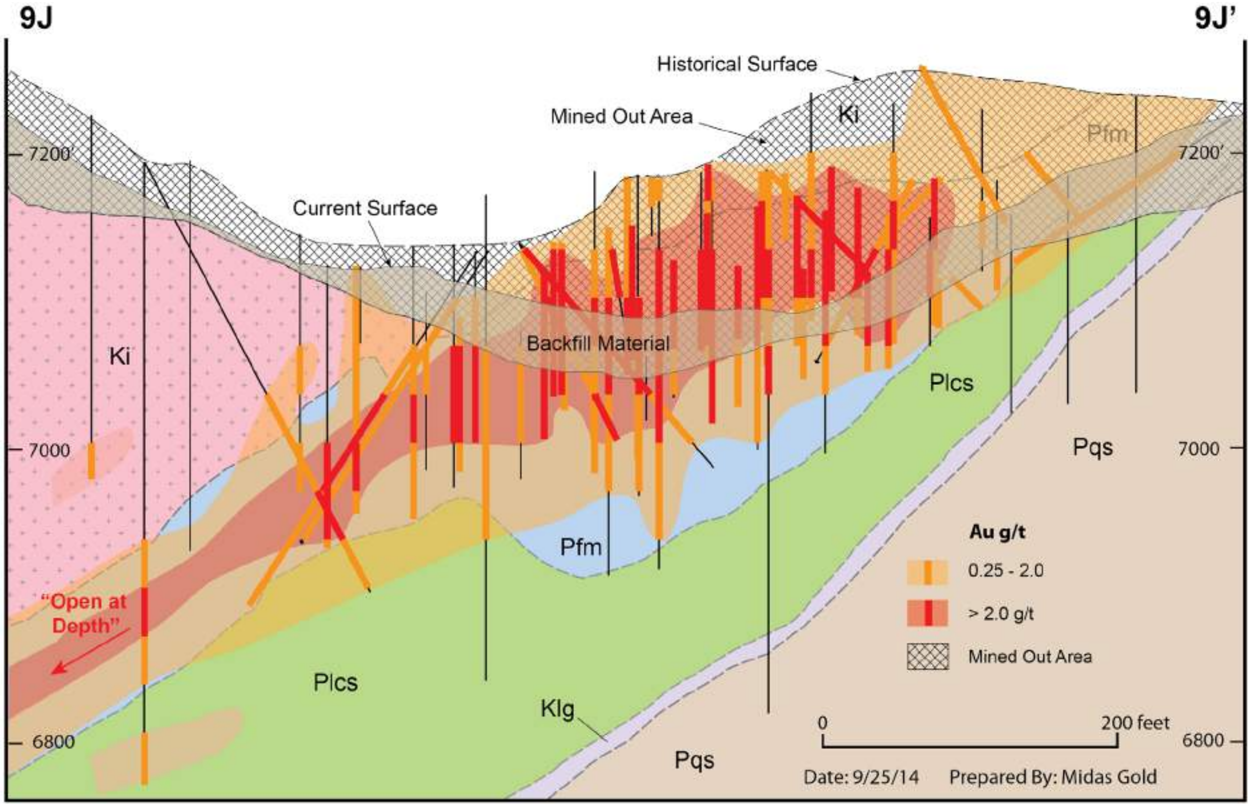
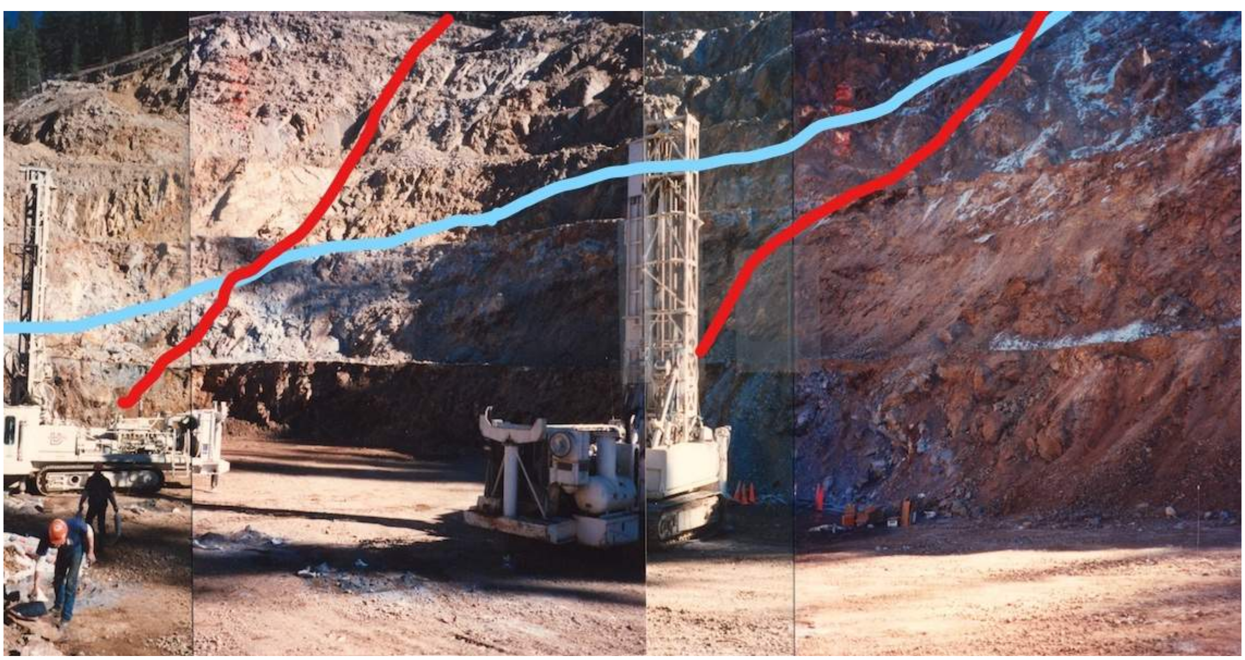


Figure 9-36: Photograph of Garnet Open Pit prior to backfill (October 27, 1995)

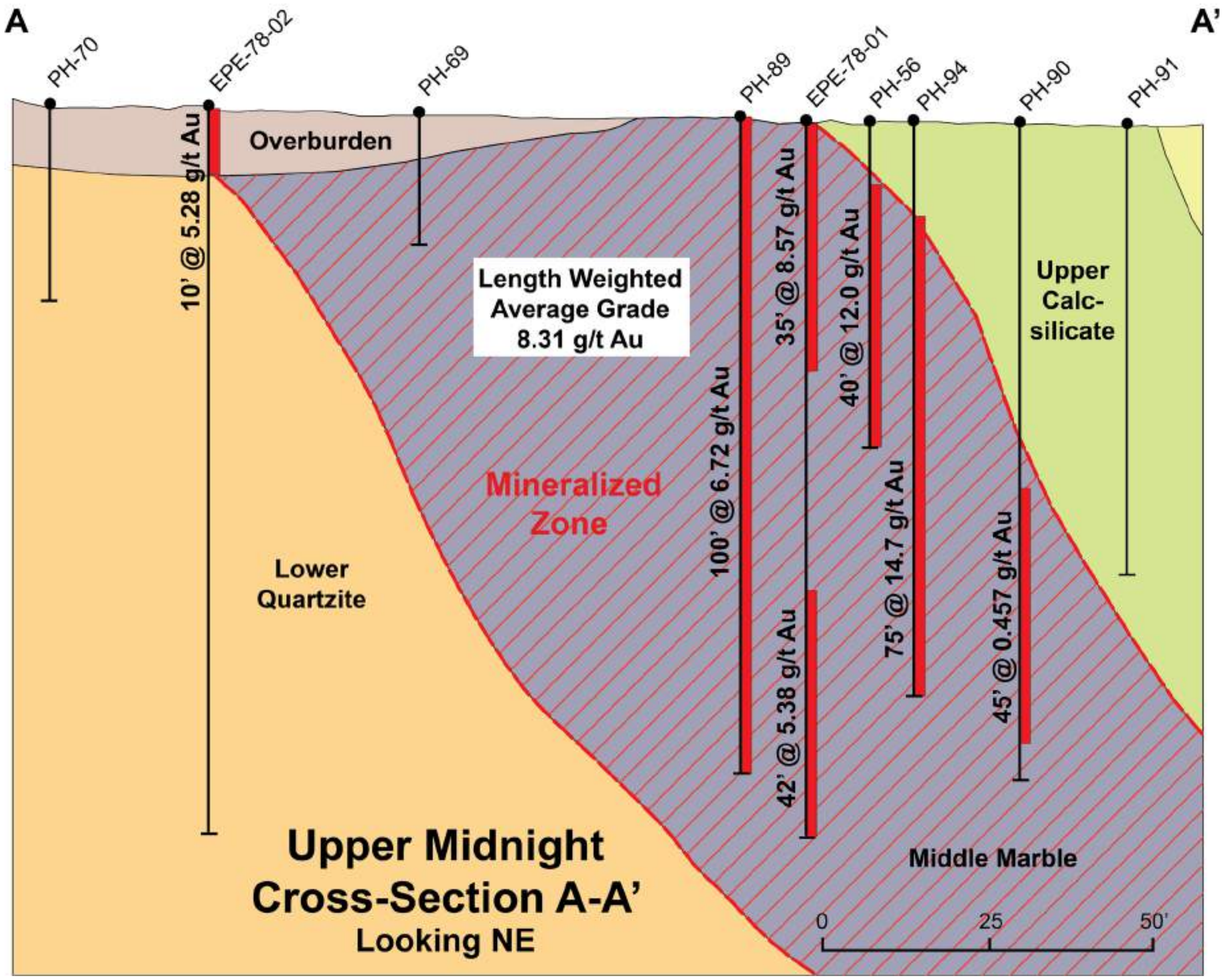


Note: Photo looking northeast along strike Garnet Fault System (between red lines) in last open pit bench in 1995. Upper benches (above blue line) in upper Kg sill and lower benches in the Zfm unit. Note blue unoxidized sulfide material right of the drill rig. Photo courtesy Virginia Gillerman, Idaho Geological Survey.

9.8.3 Upper Midnight

The Upper Midnight prospect is located north-northeast and northeast of the Garnet Prospect. It was originally identified prior to World War II and briefly examined for the prospect's antimony potential by the Bradley interests in the 1950s. The prospect was re-discovered in the early 1970s when El Paso and Superior sampled and defined numerous large gold-in-soil and rock chip anomalies in the immediate area of the WWII era prospects. In 1976, sampling of a black carbonate outcrop at Upper Midnight returned high-grade gold assays (>40 g/t), which were followed up by air track, core, and RC drilling that confirmed the presence of a small, poorly defined, but high-grade mineralized zone. Subsequent drilling campaigns included 2,349 ft in 28 shallow core, RC, and air track percussion holes but did not adequately test the down-dip extent or strike extensions of this zone, which appears to be approximately 60 ft thick (true width) with a length-weighted average gold grade within the mineralized zone of 8.33 g/t (Figure 9-36). In the early 1990s, large soil, ground magnetic, and Very Low Frequency Electro-Magnetic (VLF-EM) surveys were completed over the Upper Midnight prospect and outlined several coincident magnetic anomalies and strong conductive features spatially associated with the anomalous geochemical features. Petrographic studies from drill core and outcrop samples collected in the 1970s indicate the host rocks for gold mineralization are dark carbonaceous siltites, calc-silicates, silicified carbonates, and a tactite skarn similar to the nearby Garnet Deposit. The prospect has no recorded historical production. The prospect occurs along the southwestern strike projection of the structures that host mineralization at the Ridgetop Prospect along strike approximately 1,000 feet to the northeast.

Figure 9-37: Long Section through Upper Midnight Target looking Northeast



Between 2010 and 2013, Midas Gold collected stream sediment, soil, and outcrop samples covering a broader area, including Upper Midnight to confirm and expand on past exploration work. At Upper Midnight, 60 rock chip samples outlined anomalous gold values including 3 ft chip samples of 5.24 g/t within brecciated quartzites and 2.79 g/t in altered carbonates within a large 400 ft by 500 ft soil anomaly (> 0.1 g/t Au). The original prospect site along the former road has been completely backfilled during reclamation and the high-grade outcrops are no longer accessible.

The Upper Midnight conceptual target consists of a sediment-hosted, structurally and stratigraphically controlled, high-grade, gold deposit with underground mining potential. The target is within the upper calc-silicate unit near its contact with the middle quartzite sequence in conjunction with several faults. Due to steep slopes and small footprint, the open pit potential is very low. The mineralized zone, as defined by soil, rock, trench, drilling, and geophysical data, could extend along strike for a length of 500-600 ft and down-dip to the northeast for 150-200 ft. With the grades encountered to date, Upper Midnight represents an excellent high-grade target with underground mining potential.

Table 9-17: Select Pre-Midas Gold Drill Intercepts for the Upper Midnight Prospect

Drill Hole ID	Operator	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold[(1)] (g/t)
EPE-78-01	El Paso	-90	-	0	112.0	112.0	5.56
				including			
				0	35.0	35.0	11.35
				100.0	112.0	12.0	15.91
PH-056	Superior	-90	-	0	50.0	50.0	4.62
PH-089	Superior	-90	-	0	100.0	100.0	6.73
				including			
				5.0	30.0	25.0	15.61
PH-094	Superior	-90	-	15.0	90.00	75.0	14.75

Note:
(1) *Selected intercepts composited with length-weighted averages with COG of 0.75 g/t Au and/or 0.3% Sb reported, >10 ft composite length and <10 ft of internal waste below 0.5 g/t Au and/or 0.1% Sb.*

9.8.4 Doris K

The Doris K prospect has been known since at least the mid-1920s when it was known as the Doris K #3 prospect (Schrader and Ross, 1926). It was reported to be a high-grade gold-silver-antimony occurrence that was exposed in hand-dug cuts in the 1920s over a 15 ft by 100 ft area with much of the material reportedly averaging over 70% stibnite. Mineralization was described as trending parallel to stratigraphy (northwest striking) with an unmineralized quartzite hanging wall and a mineralized, vuggy, quartz-altered carbonate footwall. Savage (1963) reported that the Oberbillig interests processed some of the antimony-bearing material from this prospect at the United Mercury Mine Mill on Johnson Creek in 1963, but the amount and grade of material removed and processed is unknown. In the 1970s, both El Paso and Canadian Superior conducted soil and rock sampling in the area with one roadcut averaging 2.6 g/t gold over 25 feet within a 90-150-ft wide by 450-ft long roughly north-south breccia body. The prospect was proposed for open pit mining during the late 1990s by Stibnite Mines Inc. who reported a mineral resource here in public filings, but no backup information is available in company files to support the estimate. It was reported prior to NI 43-101 standard were enacted. The Qualified Person does not consider that historical resource valid and it should not be relied upon. Limited and widely spaced drilling has been completed here by past operators. Midas Gold has only recovered some of this drill data. The proposed development was primarily based on road cut samples and large bulk samples collected from trenches across the area (Tom Wonacott, personal communication to Chris Dail, 2010).

The prospect is northeast of the Garnet Prospect along the same structural feature in the same stratigraphic section and position as the Upper Midnight Prospect. The prospect is situated along the nose of the northwest plunging Garnet

Syncline where it intersects the down-to-the-west Garnet Creek normal fault. The fault is marked on the ground by a northeast-trending zone of intense siliceous stockwork veining and iron oxide-cemented tectonic breccias cutting a blocky schistose quartzite (lower quartzite) and a friable recrystallized limestone unit within the upper calc-silicate sequence. The breccia zone is approximately 50 ft-150 ft wide where exposed and is traceable for at least 500 ft in a northeast-southwest direction. It is exposed over 200 ft of vertical extent across the nose of the syncline. The mapped breccia may be the same zone roughly oriented north-south that was described in earlier El Paso and Superior reports.

Large angular, gossanous float blocks are common in the prospect area, but outcrop is relatively poor. Historical soil surveys and rock chip sampling by Superior and El Paso along reclaimed and backfilled roadcuts outline a broad area approximately 600 ft wide x 1,900 ft long of anomalous gold and antimony. Legacy rock samples along roadcuts include approximately 100 samples with over half reporting greater than 0.1 g/t gold. A total of 46 rock samples from outcrop exposures was collected around the Doris K prospect by Midas staff with 19 of the 46 samples resulting in values greater than 0.1 g/t gold. Gold values up to 15.7 g/t within a brecciated quartzite and 13.55 g/t within gossanous carbonate came from exposures near the anomalous legacy roadcut sample anomalies. These high-grade samples were taken within 150 ft of each other and also within a historical 300-ft by 250-ft gold-in-soils anomaly near the collapsed historical adit that likely was the original Doris K prospect reported by Schrader and Ross (1926). Large blocks of nearly pure stibnite and stibnite cementing oxidized carbonate matrix-supported breccias occur in several portions of the old 1990s trenches above the collapsed adit (Figure 9-38). A CSAMT survey line (Line 9) outlined a very low resistivity geophysical feature 500 ft long here, which is consistent with the presence of conductive rocks that could be related to mineralization, the fault trace, or conductive lithological units through the prospect or a combination of these sources. Airborne EM surveys produced strongly anomalous conductive responses directly over known or suspected mineralized zones that coincide with the CSAMT feature. Both underground and open-pit targets may exist here, Given the high grades encountered in surface sampling, widespread disseminated mineralization, and lack of drilling. Additional work including drilling is warranted to evaluate the potential of the prospect.

Figure 9-38: Sawn Hand Specimen Slab of Stibnite-Cemented Carbonate Breccia

Note: Slab approximately 10 in. across. Photo courtesy Tanya Nelson.

9.8.5 Fern

High-grade gold occurrences in the Fern area have been known since 1903 (Bell, 1918) when prospectors examined the area during the Thunder Mountain gold rush. Placer mining for gold was attempted around the turn of the century but apparently was unsuccessful. Mercury exploration, development, and minor production (approximately 40 flasks) occurred intermittently from 1917 to the 1920s over a vertical distance of approximately 1085 ft (Larsen and Livingston, 1920; Schrader and Ross, 1926). In the 1940s, the USBM conducted mercury exploration, including extensive trenching (USBM, 1943a, 1943b). In the 1950s, additional trenching was completed under the DMEA program. In the early 1960s, additional sampling, trenching, and drilling of 5 holes totaling 1,503 ft were completed under an Office of Mineral Exploration (**OME**) contract.

In 1983-84, Canadian Superior briefly explored the area and discovered high-grade gold mineralization in outcrops of jasperoid breccias developed in the Fern Marble. Two IP lines were completed followed by three RC holes totaling 1,780 ft in 1984. Pioneer did some follow up rock-chip sampling in 1987 and drilled five holes totaling 2,400 ft in 1990. Further soil sampling was completed by Barrier Reef Inc. in 1990. Several drill holes targeted stratiform-style jasperoid mineralization. They were oriented subparallel to faults and the outcropping crosscutting breccias, had very poor recoveries, and were only analyzed with cyanide methods. Nonetheless, significant alteration and mineralization were encountered in several of the holes (Table 9-18).

There is potential to discover a low-tonnage, high-grade, gold deposit with underground and/or open-pit mining potential where low-grade gold mineralization would be associated with replacement bodies in the Fern Marble or in the lower calc-silicate sequence. Twentynine systematic outcrop chip samples taken by Pioneer Metals in 1987 (near an adit driven adjacent to a northeast-trending jasperoid breccia) produced gold assays ranging from 0.87 g/t to 42.7 g/t, averaging 14.3 g/t. Holes drilled nearby at the time did not adequately test the mineralized structure but the holes did cut significant mineralization suggesting the possibility of a larger, open-pit target and underground potential from the main structure.

The area was mapped and sampled by Midas Gold in 2015. In addition to surface work, accessible old underground workings were mapped and sampled, and a structural analysis was completed (Dunbar and Dail, 2015). Over 350 epithermal calcite ± chalcedonic quartz veins and jasperoid replacement breccias were mapped and had remarkably consistent trends. Figure 9-38 is an example of a typical jasperoid vein cutting weathered and unaltered Fern Marble. Sampling of these veins indicates gold is associated with the quartz veins, which are both chalcedonic and opaline, but is rare in the carbonate veins. Quartz veins range in width from a few inches to over 30 feet, mostly 1-3 ft. In many cases, stratiform ledges of jasperoidal quartz replace bedding adjacent to thicker northeast-trending faults, which likely are feeders to the mineralization. These veins show a consistent geometry and thicken at intersections of two distinct fracture/vein sets. The geometry of the intersection lineation of the vein trends is identical to the long axis orientation of observed open-space vugs. These ellipsoidal vugs are thickest at the intersection bedding veins that strike northwest and dip northeast and veins that strike northeast and dip to the northwest suggesting larger mineralized zones may exhibit this geometry. Examination of the vugs indicates they often contain vuggy quartz after calcite, pyrite, cinnabar, and occasionally other unidentified sulfides.

Despite the high grades, no free gold has been observed. Assays vary widely across short distances, suggesting the presence of a nugget effect. Gregory (2017) conducted a QEMSCAN and petrographic examination of samples from the high-grade Fern Jasperoid Breccia Vein and noted that the typical mineralogy consists of massive chaldedonic quartz, bladed quartz after calcite in vugs, arsenosiderite, cinnabar, scheelite, arsenian pyrite, and hematite after pyrite. Konyshev (2020) conducted SEM and petrographic studies of samples from the same area and noted very little wall rock alteration on the opaline and chalcedonic veins except clay, no decalcification around the veins, and very low silver values even in samples with very high gold concentrations. Konyshev (2020) also noted similarities in textures from the veining and alteration at Fern to the Mule Canyon epithermal deposit in Nevada. Unlike samples on the west side of the district, fluid inclusions are common in veins in the Fern area, but there has been no work investigating the pressure-temperature conditions of mineralized veins found in this area.

Figure 9-39: Typical Northeast Striking, Northwest Dipping Jasperoid Breccia Vein

Note: Typical northeast striking, northwest dipping jasperoid breccia vein cutting unaltered Fern Marble. Vein by flag approximately 16 inches wide, north is towards top of photo.

Two veins were systematically sampled to evaluate structural controls on gold and obtain samples to examine mineralogy. A small vein exposed in outcrop just east of an old mercury prospect was sampled over a 2 ft width and averaged 4.5 g/t gold. It was sampled underground approximately 150 ft down dip from the outcrop over 80 ft of exposed strike length and had a length-weighted average gold grade of 4.8 g/t over 8 ft true width (Figure 9-39, Figure 9-40).

A second, larger jasperoid breccia vein first noted by Canadian Superior was systematically panel sampled in 2015 to confirm historically reported grades (Figure 9-40). A length-weighted average of the approximately 30 samples collected from the outcropping vein over approximately 168 ft of exposed strike length and 13 ft average width was 28 g/t. A short NX-diameter (2.16 in.) core hole drilled in 2017 across the middle of the jasperoid outcrop averaged 38 g/t gold over 10 ft true width. The outcrop exposure likely correlates with a drill intercept in former 1990 Pioneer hole P-90-36 (Table 9-18), approximately 125-175 ft down plunge which suggests continuity of mineralization to that level. All the previous holes drilled in this prospect utilized cyanide assay methods even in material logged as having sulfides and reported poor recoveries in the heavily silicified and vuggy textured carbonates. Cyanide leach methods may under-report total gold content where refractory sulfides are present.

Figure 9-40: Conceptual Cross Section Fern High Grade Jasperoid Breccia Vein (120 ft corridor)

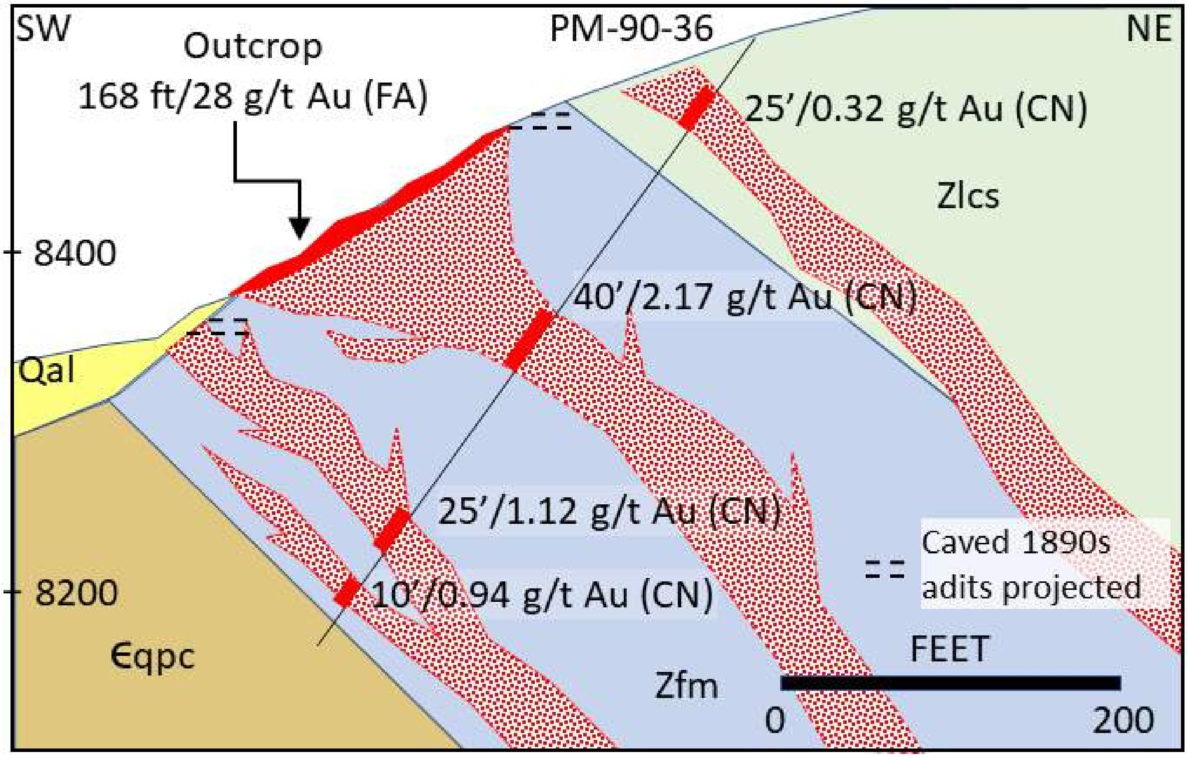
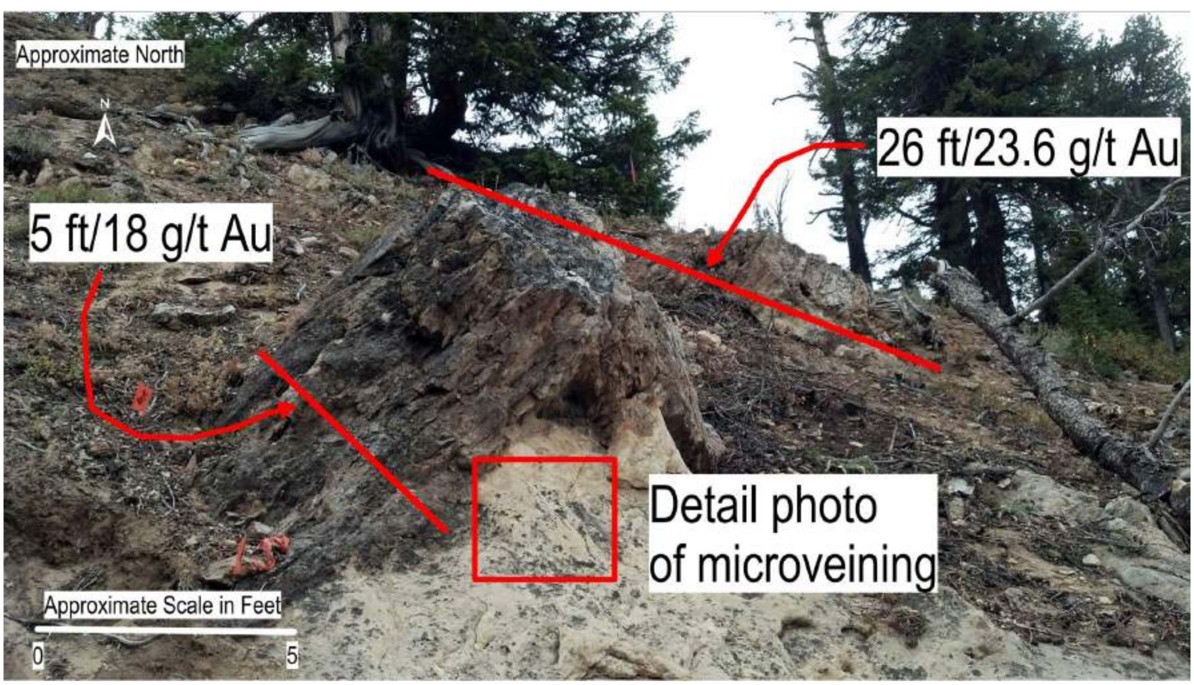


Figure 9-41: Photograph of Fern High Grade Jasperoid Breccia Vein



Although the primary targets at Fern are underground-style deposits where epithermal veins are well developed and have good grades, there are several areas where thick lower grade intercepts suggest the possibility of mineralization that may have bulk mining potential similar to the prospects at the Saddle, Cinnamid, and Ridgetop areas along strike to the northwest. In particular, the unaltered lower calc-silicate in the area typically contains from 1-4% disseminated pyrrhotite in the thinly laminated silty intervals. Promising targets occur where these pyrrhotite-rich stratigraphic sections are cut by northeast faults, such near the top of hole P-90-37 where the structure hosting the high-grade Fern vein crosses out of the Fern Marble into the lower calc-silicate. Additional veins and structures are present throughout the area and warrant additional sampling, prospecting, and drilling.

Table 9-18: Select Drill Intercepts in the Fern Prospect Area

Target	Operator	Drill Hole ID	Collar Inclination (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Au (g/t)
Bucks Bed	Canadian Superior	84-01	-65	215	120	340	220	0.18
			and		450	480	30	0.13
Fern HG	Pioneer	P-90-35	-60	251	125	150	25	0.22
Fern HG	Pioneer	P-90-36	-55	215	55	75	25	0.31
			and		205	245	40	2.17
			and		345	370	25	1.12
			and		390	400	10	0.94
Fern HG	Pioneer	P-90-37	-60	180	20	30	10	0.50
			and		70	120	50	0.78
			and		195	205	10	0.15
			and		300	320	20	0.19
Fern HG	Canadian Superior	84-3	-65	215	0	620	-	-

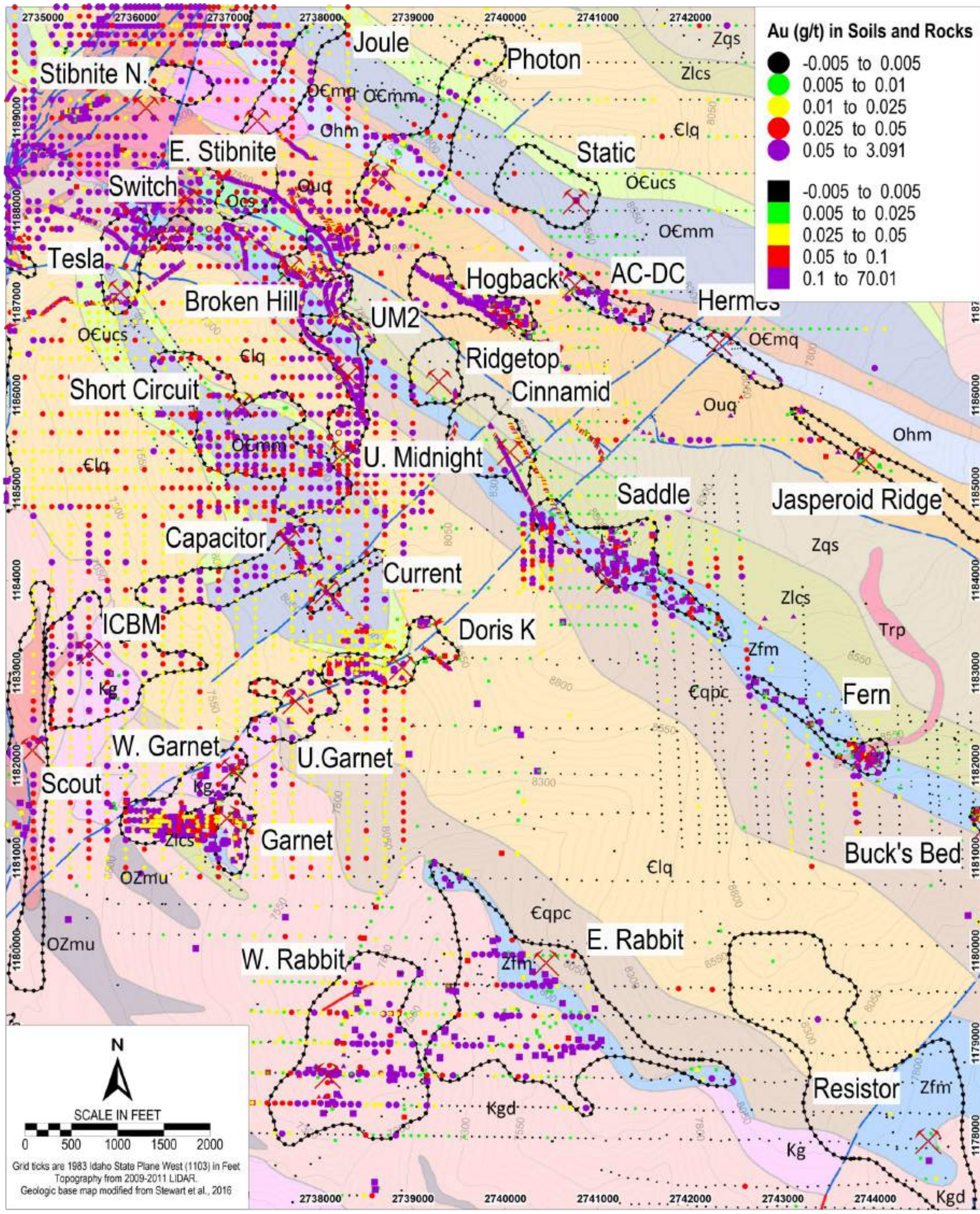
Note: Composites calculated using 0.1 g/t Au COG and can include up to 25% internal waste below COG. All assays by cyanide leach.

9.9 PROSPECTS FOR DISCOVERY OF NEW BULK MINING POTENTIAL

The discovery and development of the disseminated, metasedimentary rock-hosted West End Deposit in the 1970s, and later development in the 1980s-1990s, spurred exploration for similar oxidized bulk-mineable deposits amenable to heap leaching within the District area by a succession of entities. Exploration for similar deposits initially focused on the ridge trending southeast from the West End deposit where the calc-silicate sequence that hosts the West End oxide deposit is exposed. One deposit, the West End Extension (later known as the Stibnite Pit), was developed between 1995-1997. Large soil sampling grids and ground geophysical surveys followed by trenching and drilling led to the discovery of other prospects along the ridge and in adjacent areas. Several of these were proposed for mining and a draft Environmental Impact Statement was prepared in the mid- to late-1990s, but falling gold prices and other factors prevented their development.

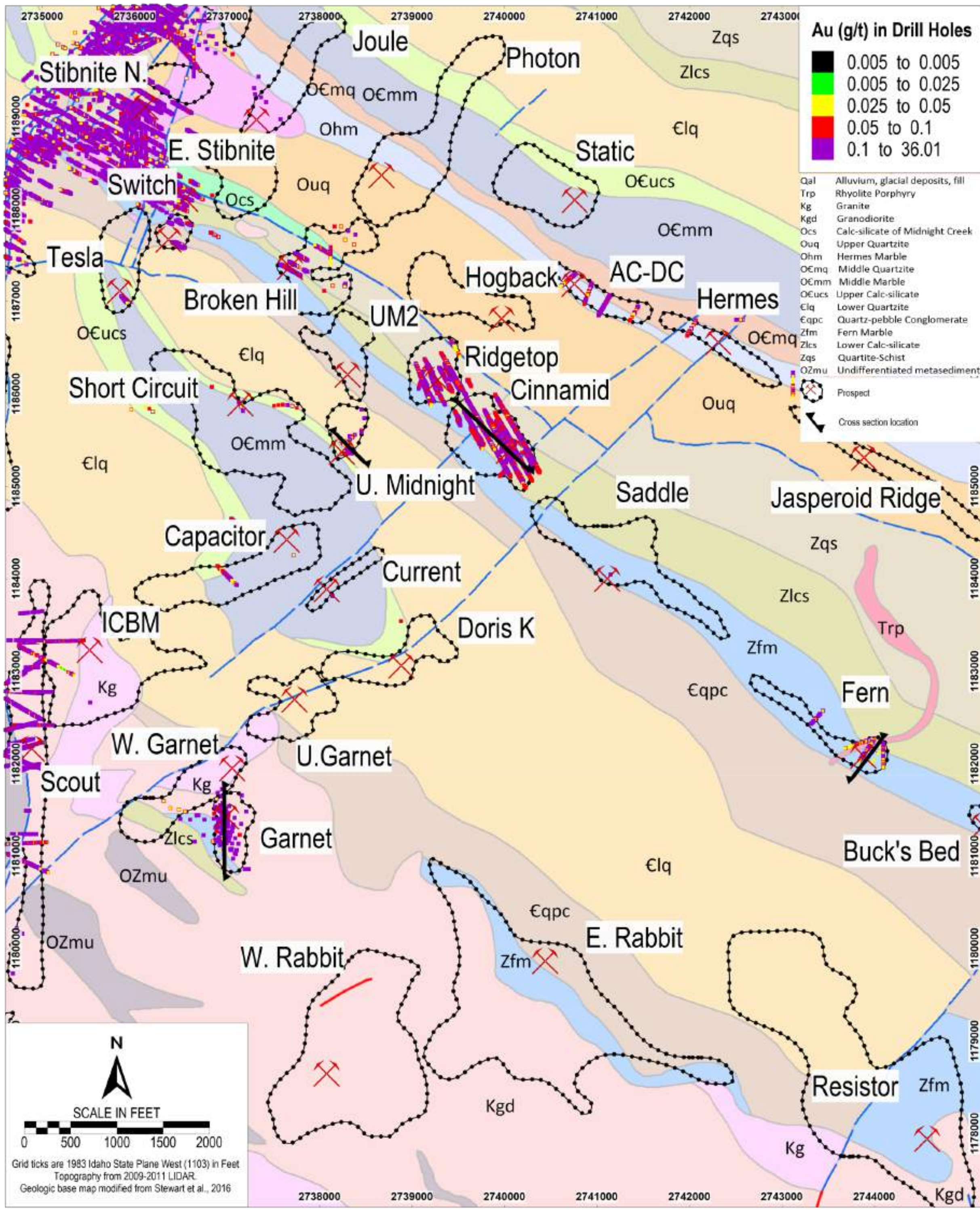
One group of prospects aligned along the outcrop trace of the contact between the Fern marble and lower calc-silicate in the overturned limb of the Garnet syncline, southeast of the former Stibnite pit. This alignment of old prospects, geochemical and geophysical anomalies has been named the Broken Hill - Ridgetop - Cinnamid - Saddle - Fern Trend. All of the prospects have some level of pre-Midas Gold drilling, and some exhibit size and grades that suggest potential for large-tonnage, bulk-mineable deposits. All of them are open to expansion based on available data. Gold in soils and rocks chips collected from these prospects is provided in Figure 9-41 and shown conceptually in long section (Figure 9-2). Figure 9-41 shows the widespread distribution of soil and rock chip anomalies in this area. Figure 9-42 shows how few of these prospects has been tested by drilling. This indicates a high potential for additional discoveries at the intersection of northeast-trending fault systems and favorable stratigraphy in the northwest-trending metasedimentary units, a setting to very similar to that of the West End deposit.

Figure 9-42: Gold in Soils and Rocks Stibnite Roof Pendant



Note: Stratigraphic unit symbology same as on Figure 9-1.

Figure 9-43: Gold in Drill Holes Stibnite Roof Pendant



In particular, note the northwest-southeast continuity of soil and rock chip anomalies between Cinnamid and Saddle parallel to favorable stratigraphy and the Cinnabar Peak Thrust Fault; the northeast-southwest alignment of soil and rock chip anomalies over 7,000 ft long along the Garnet Fault from the former Garnet open pit to Doris K; the alignment of anomalies and prospects between ICBM and Capacitor and other clusters without any systematic drilling. Many of these prospects are newly identified by Midas Gold staff and have multiple lines of support (soil, rock, ground and airborne geophysical anomalies, mapped alteration, etc.) and provide a pipeline of prospects for future exploration.

Surface rock-chip and soil sampling between the Cinnamid and Saddle prospects suggests continuity of gold mineralization at the surface along the 2,000 feet between the prospects. A groundwater monitoring well drilled in 1996 intersected 120 ft of cyanide-soluble gold averaging 0.92 g/t in oxidized Fern Marble, confirming the strong gold soil anomaly at Saddle (Figure 9-42). Since most analytical work from drill sample assays along the trend utilized CN-soluble assay methods, Gold grades may be significantly under-reported since most assays of drill hole samples were CN-soluble gold assays, especially considering that unoxidized sulfides are present at shallow depths. Midas Gold has mapped and rock-sampled all of the prospects along this trend but has not completed any additional drilling. In 2013, Midas Gold expanded the soil grid northeast of Broken Hill, which generated a large soil anomaly on strike with the northeast structures controlling mineralization at Broken Hill in the monoclinal stratigraphic succession northeast of the Cinnabar Ridge Fault. This suggests that these structures have additional potential along strike where limited or no past exploration work has been completed and favorable stratigraphic units are cut by northeast-trending structures.

Geologically, the Broken Hill - Saddle Trend is defined by the intersection of northwest-striking, northeast-dipping metasediments, and district-scale northeast-striking, moderate- to high-angle faults that provided the conduits for gold mineralization. Gold was preferentially deposited along the trend in reactive siltites, calc-silicates, Fern Marble, and in a sequence of interbedded quartzites, magnetite-bearing phyllites, and schists. Lithologic units rich in iron, which acted as a reductant for gold-bearing hydrothermal fluids, make more favorable host rocks, as at West End. Ridgetop and Cinnamid have been drilled extensively by previous entities, who focused on shallow oxide mineralization. A small number of holes have been drilled at Broken Hill, and only a single groundwater monitoring well was drilled at Saddle. However, the potential to discover additional sulfide mineralization along the entire trend is excellent. Drilling, trenching, and rock chip sampling has defined a body of mineralized material occurring as stacked lenses within the lower calc-silicate, Fern Marble, and quartzite-schist sequences (Figure 9-44) on the southeast end of the trend, between the Ridgetop and Saddle prospects with an aggregate true thickness ranging from between 75-125 ft (200-325 ft in plan view) that is from 3,000-3,500 ft along strike and extends approximately 200-325 ft vertically below the ground surface (the limits of current drilling). This represents conceptual open pit target ranging from 4-10 million tons at grades between 1-2 g/t gold and potentially containing between 120koz and 600koz. All previously drilled mineralization remains open to expansion along-strike and down-dip. There is considerable upside potential considering that the trend is over 3.5 miles long and is only drill-tested over a short section of the favorable stratigraphy. Strong conductivity responses in ground VLF and airborne EM surveys coincide with the trace of mapped, sampled, altered, and mineralized zones along this trend, suggesting that geophysical methods may useful for locating deeper zones of high sulfide content that are typically associated with gold mineralization.

9.9.1 Broken Hill

Soil geochemistry has been an important exploration tool for the identification of drill targets in the Stibnite roof pendant. Broken Hill was initially discovered in 1982 during soil sampling by Canadian Superior (Figure 9-41). However, follow up work outside of reconnaissance did not take place until 1990 after the discovery of the Stibnite deposit and the recognition of the potential economic importance of the numerous northeast trending faults cutting the roof pendant southeast along the ridge. Between 1990-1992, Pioneer Metals undertook an extensive program of road cut channel sampling and mapping to identify the source of the soil anomalies at Broken Hill.

In 1991, fourteen reverse circulation holes totaling 4,675 feet were drilled at Broken Hill looking for shallow oxide gold mineralization to feed the ongoing cyanide leach operations. Two additional holes were drilled in 1995 and 1996. One of the holes was a groundwater monitoring well (reverse circulation) while the other was a core hole drilled for geotechnical information. Significant intercepts are listed in Table 9-19.

Gold mineralization is found in all of the exposed units including marble, calc-silicate, quartzite-schist, and quartzite. Gold deposition is mostly controlled by two N50-70E fault zones exposed on drill roads were mapped and sampled by Pioneer Metals in 1991. The Broken Hill Fault is strongly brecciated over a width of 100 feet in the upper quartzite and quartzite-schist metasedimentary units. Holes were designed to test for oxide mineralization, leaving much of the favorable reactive lithologic units at depth within the lower calc-silicate sequence untested.

Table 9-19: Select Drill Intercepts within the Broken Hill – Saddle Trend

Prospect	Operator	Drill Hole ID	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold(1) (g/t)
Broken Hill	Pioneer	91-31	-90	N/A	30	110	80	1.06
Ridgetop	Pioneer	92-47	-90	N/A	135	205	70	2.23
Ridgetop	SMI	95-63	-90	N/A	155	190	35	2.06
Ridgetop	SMI	96-67	-90	N/A	105	180	75	2.41
Cinnamid	Pioneer	92-49	-90	N/A	235	285	50	4.54
Cinnamid	SMI	95-69	-90	N/A	15	90	75	2.56
Cinnamid	SMI	95-70	-90	N/A	110	185	75	3.06
Cinnamid	SMI	96-62	-90	N/A	330	460	130	1.23
Saddle	SMI	MW-96-01	-90	N/A	25	145	120	0.92(2)
Fern	Pioneer	90-36	-55	215	205	235	30	2.73
					345	365	20	1.36
Fern	Pioneer	90-34	-50	222	285	300	15	1.01
Fern	Pioneer	90-37	-60	180	85	120	35	0.77

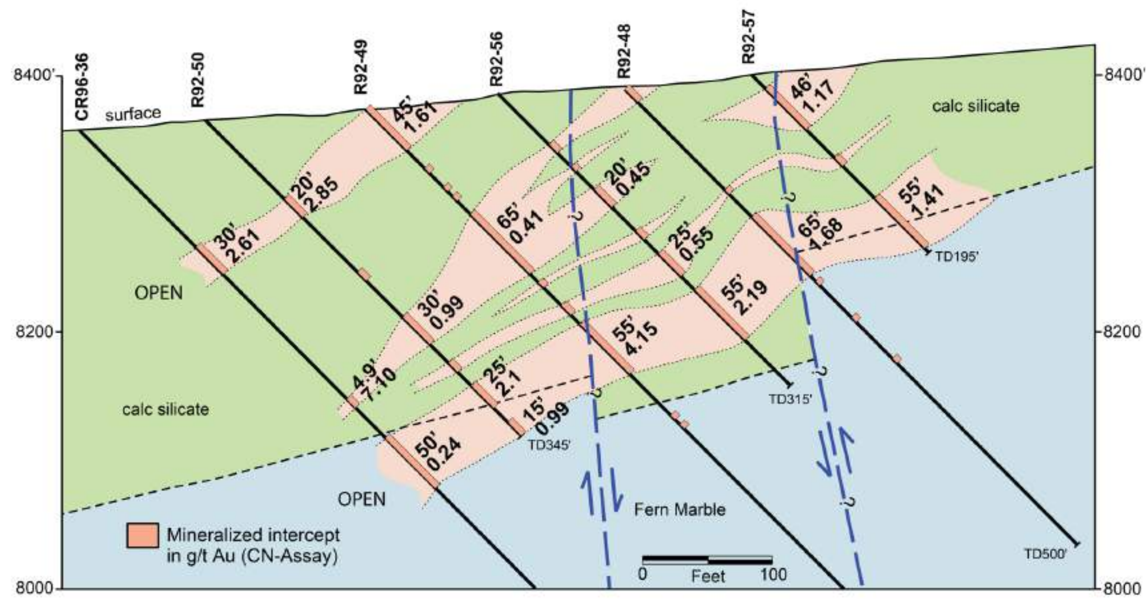
Notes:
(1) Selected intercepts composited with length-weighted averages of continuous mineralization with over 0.75 g/t Au reported, >10 ft composite length, and <10 ft of internal waste below 0.5 g/t Au.
(2) Cyanide assay method only.

The Photon Prospect was identified by a Midas Gold soil sample survey in 2013 as the mineralized northeast extension of the Broken Hill Fault zone across the trace of West End Creek to the northeast. The fault zone can be traced for about 2,600 feet to the northeast. The structure changes strike from about N50°E to N30°E as it crosses the drainage. The fault zone is represented in the soil survey as a moderately strong gold-antimony-arsenic-mercury anomaly that widens near the lower contact of the Hermes Marble. Two of the 2012 CSAMT lines cross the feature, The southwestern line identified a two-pronged feature 400 feet wide directly along strike with the two faults observed at the Broken Hill prospect to the southwest. The lower Hermes contact occurs along the drainage edge and exposures exhibit extensive dolomitization, fracturing, and development of silica-lined vugs in zones parallel to bedding that may account for the low resistivity features noted in the CSAMT. the current interpretation is that the Broken Hill Fault Zone continues to the northeast, based on soil sampling and geologic mapping. The Hermes mercury mine lies uphill and southwest of the Photon Prospect in the same drainage and at the same stratigraphic contact. The contact is highly sheared and brecciated and appears to be a bedding/contact-parallel fault at the head of West End Creek where there are good continuous exposures,. Cinnabar was deposited at the contact between the upper Quartzite and the Hermes Marble typically within the marble at the Hermes Mine and suggests this sequence has appropriate favorable stratigraphic and mineralogic characteristics and structural preparation to make a good gold target. The close proximity of the proposed West End pit to the Broken Hill prospect makes Broken Hill a high priority target for drill testing.

9.9.2 Ridgetop-Cinnamid

In 1990-1992, SMI and Pioneer Metals undertook an extensive program of road cut channel sampling and mapping to identify the source of the soil anomalies at Broken Hill, Cinnamid-Ridgetop and Saddle. Rock sampling included 775 rock chip channel samples at Cinnamid and Ridgetop (Figure 9-41), 110 roadcut and outcrop samples at Saddle, and over 1,000 rock-chip channel samples between the Stibnite pit and Ridgetop on access and drill roads. At Cinnamid, nearly the entire length of a 1,200-ft roadcut produced anomalous rock-chip channel sample assays (90 of 122 samples assayed greater than 0.1 ppm). Dakota Mining (Dakota) mapped and sampled three trenches at Cinnamid in 1996, confirming the previously sampled gold mineralization. Bar Geophysics conducted an extensive VLF and ground magnetics survey that coincided with a large tightly spaced soil grid covering the area between Cinnamid-Ridgetop and the Fern prospect.

Figure 9-44: Long Section through the Cinnamid Prospect Looking Northeast

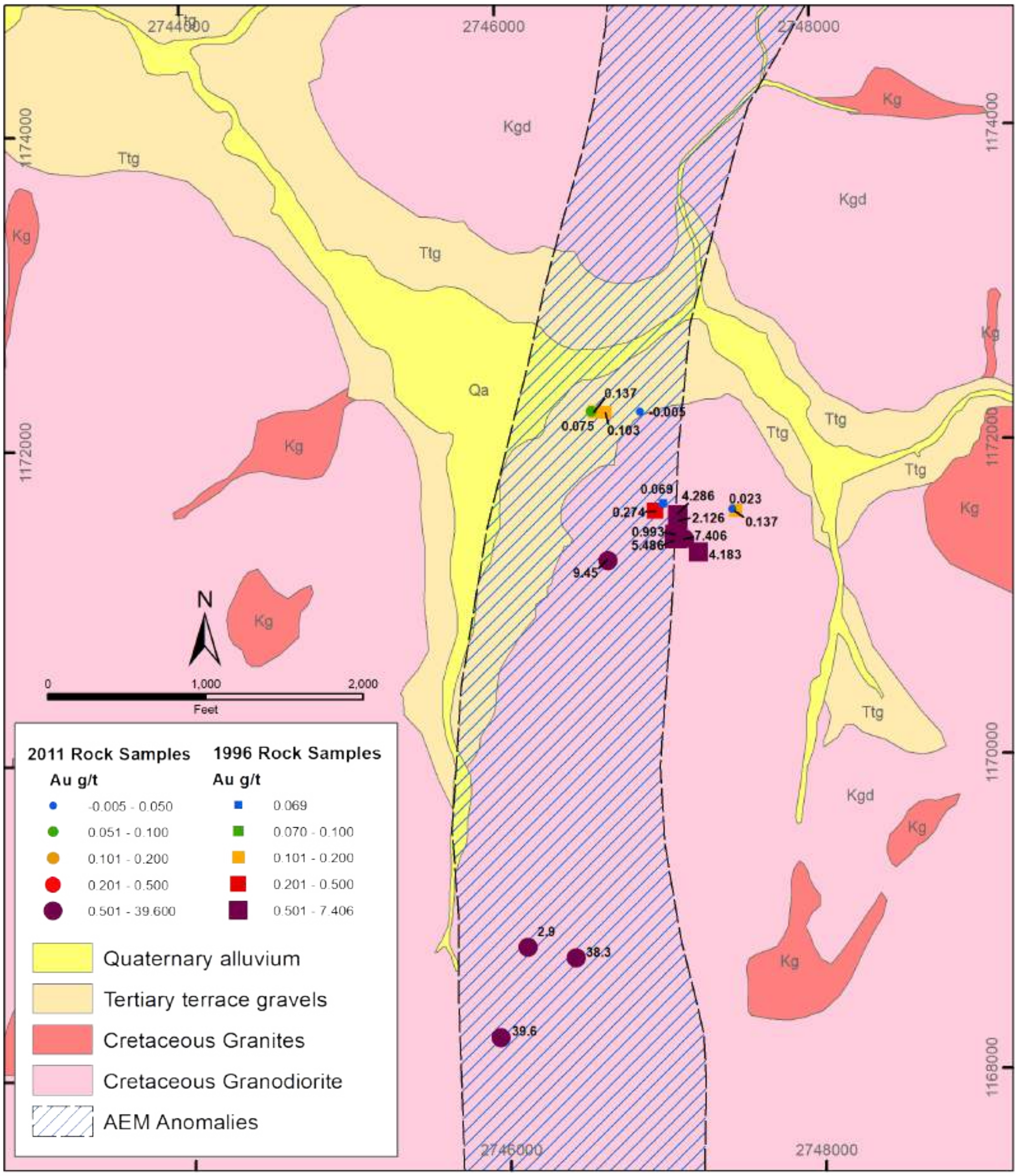


9.9.3 Mule

The Mule Prospect (Figure 9-44)) is a potential open pit and/or underground gold prospect associated with high-grade sulfidic quartz veins and lower grade disseminated mineralization hosted in intrusive rocks. The prospect is located less than a mile west of the volcanic terrane associated with the Tertiary Thunder Mountain Caldera and a thin veneer of volcanic ash and pyroclastic deposits are present just south and east of the prospect. Pioneer excavated three trenches that cut a N30°E vein system that dips 30°W. The first trench included a 1-2 ft wide vein that averaged 51 g/t CN-leachable gold within an altered zone 40 ft wide averaging 0.58 g/t CN-leachable gold (excluding the vein intercept). The second trench, located approximately 175 ft north of the first, included a 2 ft wide vein that assayed 6.03 g/t CN-leachable gold within a zone 158 ft wide (~119 ft true width) that averaged 0.4 g/t CN-leachable gold (excluding the vein). A third trench situated too far to the west to intercept the vein based on the projections from the northern trenches 100 ft south of the first trench did not intersect the vein nor cut altered rocks and no assays were reported.

Midas Gold's 2011 airborne magnetic and EM surveys outlined a large N-S geophysical feature several miles long through this area continuing to the north through the Fern and Cinnabar mines. This survey resulted in geophysical characteristics similar to the Meadow Creek Fault Zone farther west that hosts both the Yellow Pine and Hangar Flats deposits. Follow-up work included mapping and rock, soil, and stream sediment sampling. In early 2012, a reconnaissance soil grid of a 550 samples was established over the area and outlined two large soil anomalies near the old trenches and another anomaly farther to the south. Eighteen rock samples were collected from the limited bedrock exposures within and around these soil anomalies and consistently indicated the presence of narrow high-grade gold veins within broader zones of silicified intrusive rocks.

Figure 9-45: Mule Target Soil and Rock Geochemistry

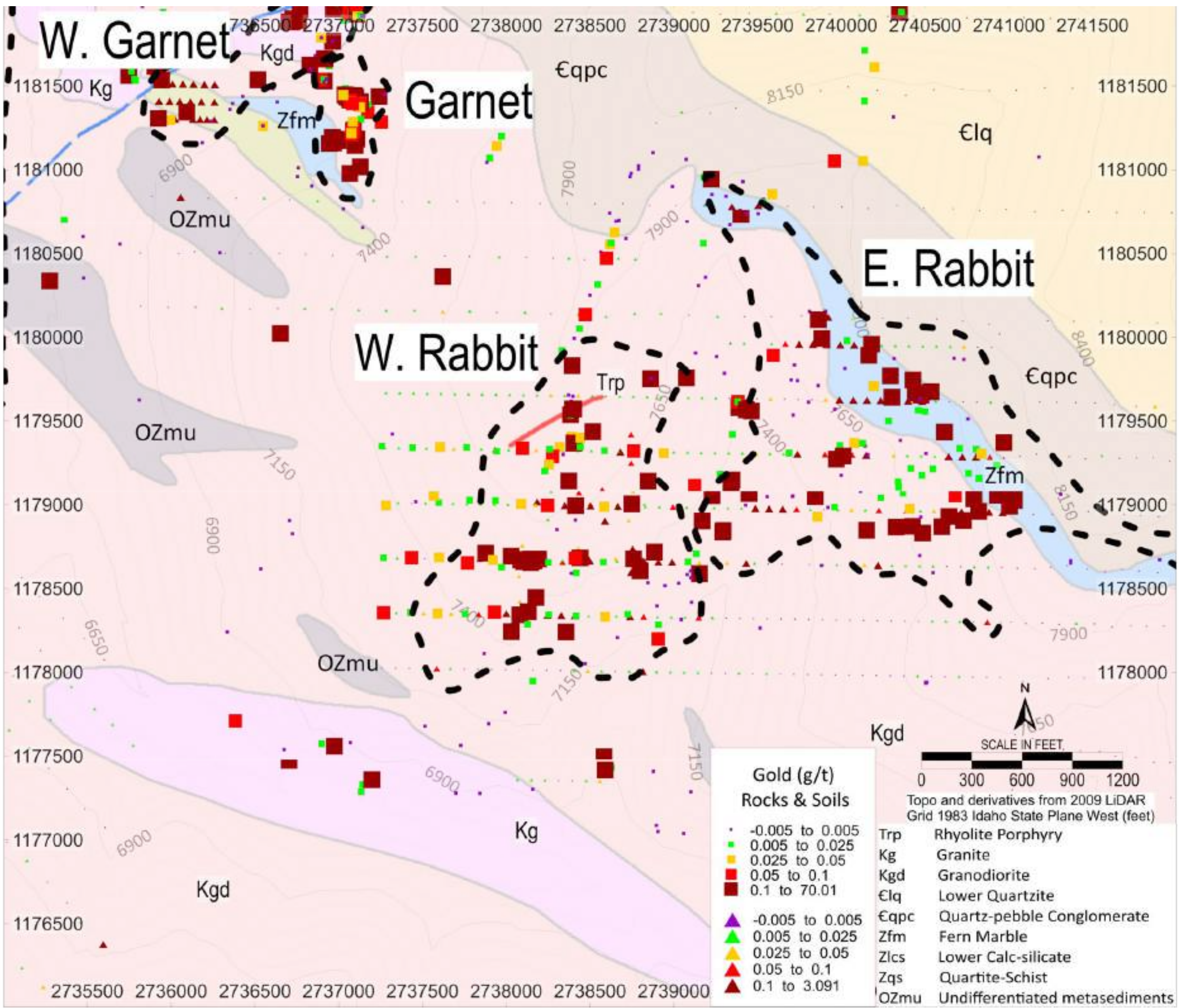


The Mule Prospect is not associated with arsenic or antimony anomalies associated with gold typical of mineralization elsewhere in the District. Sampling by Midas Gold outlined two large soil and rock chip anomalies associated with sericitized and silicified granite and high-grade gold veins and silicified gold-bearing intrusives. The largest of the anomalies in the south is at least 1,500 ft north-south by 750 ft wide and is open to the south. The northern anomaly covers an area of past historical trenching approximately 350 ft by 750 ft and is truncated by cover. The narrow but high-grades gold veins and silicified zones in the intrusive rocks could represent possible underground exploration targets. The historical trench results and large soil anomalies suggest two broad areas that might host potential bulk tonnage mineralization amenable to open pit mining.

9.9.4 Rabbit

The Rabbit Prospects are situated southeast of the Garnet Prospect along strike of the metasedimentary bedding and consist of large, coincident, multi-lobed soil, rock chip, and geophysical anomalies. The area was targeted for further exploration after compilations suggested a similar structural setting and presence of legacy prospects. Figure 9-46 shows soil and rock chip values and the two Rabbit targets. Mineralization occurs in both areas and in the intervening area, and is associated with silica, clay and sulfide impregnations and with extensive quartz-sulfide veining. Textural features suggest later epithermal alteration overprinting higher temperature alteration similar to Garnet. Midas Gold conducted mapping, stream sediment, rock chip, soil, and test pit sampling and two lines of IP-resistivity.

Figure 9-46: Plan Map of Soil and Rock Geochemistry at the Rabbit Prospect



The West Rabbitt Prospect was originally discovered in the 1920s or earlier (Schrader and Ross, 1926) and is located on the south slope of Cinnabar Ridge, on the northwest side of Rabbit Creek, about 900 ft above the East Fork South Fork Salmon River flood plain. Schrader and Ross (1926) reported that it consists of a mineralized zone with several siliceous ore shoots 100 ft wide and noted it was similar to reports by Bell (1918) of "*several large antimony-bearing quartz-breccia veins associated with large porphyry dikes*." The mineralized outcrops extend through a vertical range of about 300 feet. An adit was driven approximately 250 ft into altered quartz monzonite. Minor placer workings were excavated in the creek below the adit. Altered and mineralized diorite is found on the dumps from the former excavations and unaltered diorites outcrop to the south of the prospects. Minor prospecting was completed by modern explorers, but the prospect has reportedly never been drilled. Midas Gold geologists mapped, completed a soil grid, and a C-horizon auger sampling program, expanding the area of anomalous geochemistry considerably.

The West Rabbit area is underlain by soil and rock sample anomalies (Figure 9-45) that outline a conceptual intrusive-hosted target zone roughly 825 ft wide by 1,475 ft long with over 500 ft of vertical relief. Narrower high-grade antimony-bearing veins cross-cut the disseminated gold mineralization and may be potential underground targets.

The East Rabbit Prospect was discovered by Midas Gold geologists in 2010 (Figure 9-45) based on anomalous soils and rocks that outline a conceptual target zone hosted by metasedimentary rocks roughly 650 ft wide by 1,975 ft long with over 600 ft of vertical relief.

9.9.5 Short Circuit

In the 1970s and 1980s, large soil sampling grids were established by multiple entities that outlined large gold, arsenic, and antimony anomalies throughout the footprint of the Stibnite roof pendant. The Short Circuit Prospect is one such gold soil anomaly northeast of the Scout prospect on the southwestern limb of the Garnet Syncline. The soil anomaly is subparallel to the northwest-southeast axis of the syncline for over 2,500 ft and ranges from 600-1,100 ft wide, as defined by soil samples with gold greater than 50 ppb. The anomaly extends into the outcrop area of the upper calc-silicate unit (Ocs), the same host unit for the Upper Midnight mineralization. Twenty-two air track holes were drilled in traverses along roadcuts in three clusters to investigate a small portion of the large soil anomaly primarily in areas underlain by the Ocs unit. Several of the holes cut mineralization that warrants follow-up including more drilling. PH-164 intersected 85 ft of 1.82 g/t gold, PH-159 intersected 45 ft of 0.42 g/t gold, and PH-174 intersected 20 ft of 0.53 g/t gold. A single, short vertical RC hole was drilled approximately 300 ft northwest of PH-164 in 1987 by Pioneer that cut two intervals of gold mineralization including 15 ft of 0.48 g/t gold and 10 ft of 0.84 g/t gold, but did not test the air track hole with the thickest intercept of highly anomalous gold.

9.10 SUMMARY

The prospects identified in this section have attributes that make them promising exploration targets. However, extensive additional investigations would be required before elevating any of the prospects into resource or reserve status. That work would be extensive and likely include additional drilling, resource and reserve delineation, engineering, metallurgical, geotechnical and geochemical characterization, and other studies. In addition, federal, state, and local permitting would be required, including additional review and analyses under the National Environmental Policy Act, for any prospect to be developed other than those currently being permitted for mining under the USFS Environmental Impact Study process for the SGP.

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10 DRILLING

10.1 INTRODUCTION

The District has been drilled by numerous operators over the past 90 years. Table 10-1 shows the number of holes and footage catalogued within the Midas Gold database consisting of a variety of drilling types including percussion, auger, churn, core, reverse circulation (**RC**), rotary, and sonic drilled from both underground and surface drill stations.

This section summarizes the drilling methods, protocols and results employed for by each of the operators by year. The independent Qualified Person (**QP**) responsible for Section 10 of this report, Garth Kirkham, P. Geo., believes that the the methods and procedures for all Midas Gold drilling are consistent with industry standards and best practices supporting their use in mineral resource and mineral reserve estimation as detailed in this study.

Table 10-1: Pre-Midas Gold and Midas Gold Drilling by Mineralized Area

Mineralized Area	Pre-Midas Gold Drilling		Midas Gold Drilling		Total Drilling	
	# Holes	Feet	# Holes	Feet	# Holes	Feet
Yellow Pine	770	148,545	253	160,585	1,023	309,130
Hangar Flats	117	30,631	143	109,265	260	139,896
West End	889	208,039	53	39,680	942	247,720
Historical Tailings	26	1,554	63	5,725	89	7,279
Scout	18	6,912	28	15,859	46	22,771
Other	266	53,624	97	13,352	363	66,976
Totals	**2,086**	**449,304**	**637**	**344,465**	**2,723**	**793,769**

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

Pre-Midas Gold drilling was completed in conjunction with several surface and underground mining operations. Midas Gold drilling has been conducted for the purposes of exploration, mineral resource definition, metallurgy, and geotechnical engineering. The location of each mineralized area, along with their associated drill hole collars for both Midas Gold and Pre-Midas Gold drilling, can be found on Figure 10-1.

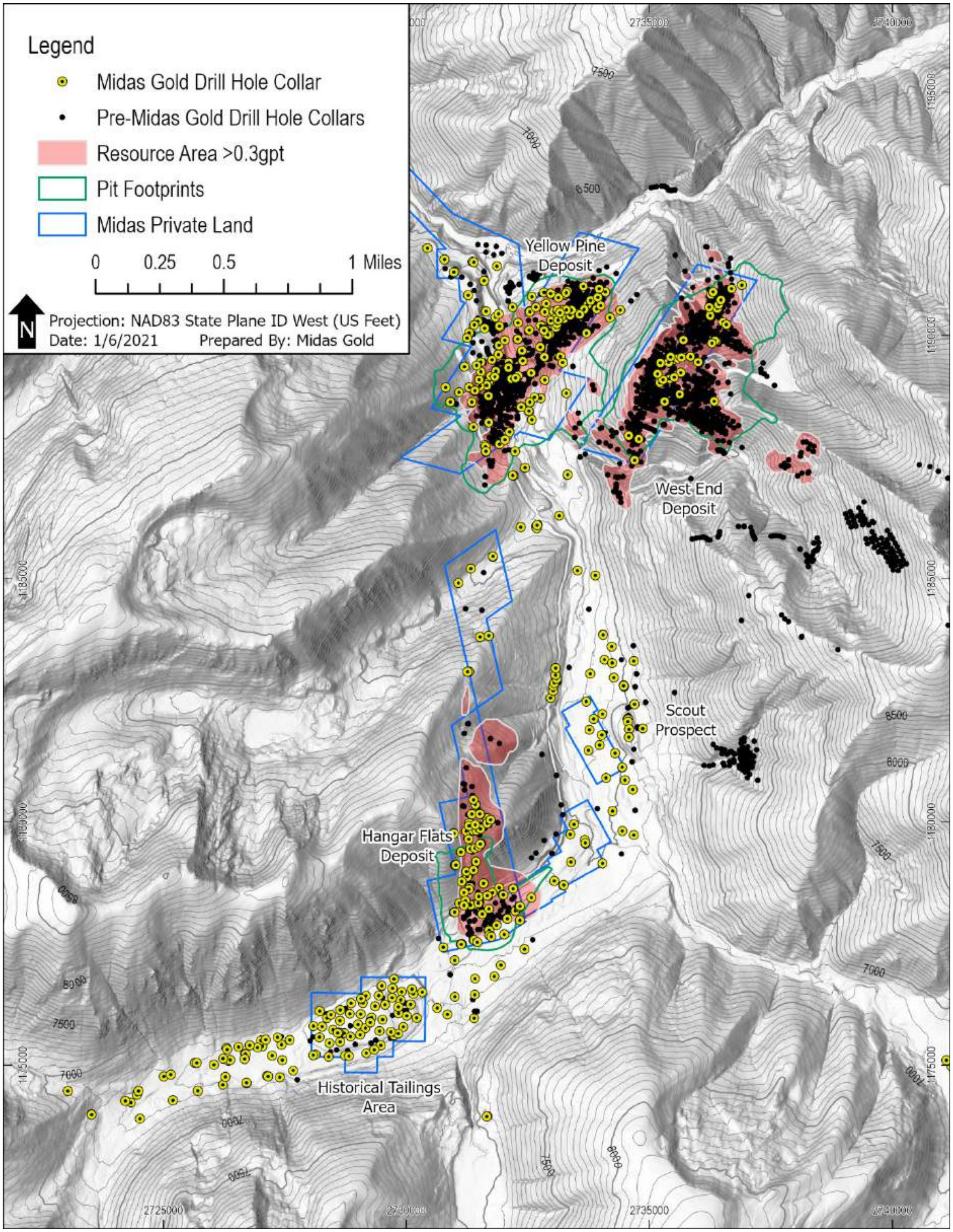
The Yellow Pine mineralized area has been drilled by 10 operators over the past 80 years and the total Yellow Pine database comprises approximately 309,130 ft of drilling in 1,023 holes. Drilling employed a variety of methods including core, RC, rotary, and air track. The pre-Midas Gold drilling was primarily performed in conjunction with surface and underground mining operations.

The Hangar Flats mineralized area has been drilled by six operators over the past 90 years totaling approximately 139,896 ft of drilling in 260 holes. Drilling employed a variety of methods including surface and underground core, RC, rotary, and sonic. Much of the pre-Midas Gold drilling was performed in conjunction with underground mining operations.

The West End mineralized area has been drilled by six operators over the past 80 years and the total West End database comprises approximately 247,720 ft of drilling in 942 holes. Drilling employed a variety of methods including core, RC, rotary, and air track. The pre-Midas Gold drilling was primarily performed in conjunction with surface mining operations.

The Historical Tailings area has been drilled by 2 operators over the past 25 years and the total Historical Tailings database comprises approximately 7,279 ft of drilling in 89 holes. Drilling employed a variety of methods including RC, sonic, and auger. Pre-Midas Gold drilling was conducted for well construction.

Figure 10-1: Drill Hole Collar Locations



The Scout prospect has been drilled by 5 operators over the past 65 years and the total Scout database comprises approximately 22,771 ft of drilling in 46 holes. Drilling employed a variety of methods including core, RC, and air track. All drilling at Scout has been conducted as exploration drilling or geotechnical investigations.

Project wide drill holes in the mineralized areas were drilled on a variety of orientations to intersect north-, northeast-, and northwest- striking structural features which control mineralization. Less than one-third of exploration and mineral resource development drillholes were drilled vertically.

10.2 DRILLING METHODS

Many drilling methods have been used by previous operators and by Midas Gold. Methods have varied by operator, time period, and deposit across the District. Methods have included air track, auger, churn, both surface and underground core, RC, rotary, sonic, percussion holes, and cone penetration tests. Cone penetration tests are included in this section as they are included in the drilling database as drill holes. This section presents a discussion on pre-Midas Gold drilling followed by a discussion of Midas Gold drilling.

10.3 PRE-MIDAS GOLD DRILLING

The extent and data quality of pre-Midas Gold drilling varies significantly by drilling campaign and operator. Table 10-2 shows the pre-Midas Gold drilling by year and type.

Table 10-2: Pre-Midas Gold Drill Holes

Year	Operator	Type	Holes	Feet
1929	Bradley	Core	10	5,586
1939	USBM	Core	6	1,331
1940	Bradley	Core	286	60,887
	USBM	Core	46	14,758
1945	Bradley	Churn	1	101
1946	Bradley	Core	18	3,661
1947	Bradley	Core	6	1,621
1948	Bradley	Core	8	3,169
1949	Bradley	Core	2	870
1950	Bradley	Core	3	825
		Churn	9	1,386
1951	Bradley	Core	15	4,761
		Churn	6	272
1952	Bradley	Core	1	371
	USBM	Core	4	1,141
1953	Bradley	Core	8	3,874
	USBM	Core	8	2,528
1954	Bradley	Core	5	2,235
		Churn	10	894
	USBM	Core	11	1,752
1955	Bradley	Core	4	1,448
	USBM	Core	4	357
1973	Ranchers	Core	6	820
	Twin River	Core	5	1,396
1974	El Paso	Core	10	2,509
		Rotary	1	200
1975	El Paso	Core	20	4,803
	Superior	Core	2	607

Year	Operator	Type	Holes	Feet
1976	El Paso	Core	11	2,526
		RC	24	2,198
	Superior	Core	17	6,661
		RC	12	1,080
1977	Superior	Air Track	62	5,140
		Core	24	6,618
1978	El Paso	RC	7	741
	Superior	Air Track	127	11,129
		RC	19	2,548
		Rotary	66	11,635
1981	El Paso	Air Track	35	1,660
		RC	8	2,000
	Superior	Rotary	9	1,750
1982	Ranchers	Core	63	12,194
	Superior	Air Track	34	1,543
1983	Ranchers	Rotary	26	5,580
	Superior	RC	44	10,921
		Rotary	29	3,422
1984	Ranchers	Core	9	1,193
		RC	55	7,845
	Superior	RC	15	4,433
1986	Pioneer	Air Track	4	275
		Percussion	5	845
		RC	40	7,865
		Rotary	7	1,808
1987	Hecla	RC	29	1,080
	Pioneer	Air Track	8	470
		RC	73	16,110
		Rotary	7	1,100
1988	Hecla	Auger	5	134
		RC	68	14,519
		Test Pit	15	158
	Pioneer	RC	49	20,560
1989	Hecla	Core	2	593
		RC	38	5,050
	Pioneer	RC	79	32,930
1990	Pioneer	RC	46	15,135
1991	Pioneer	RC	32	11,610
	SMI	RC	71	2,167
1992	Barrick	Core	14	11,427
		RC	3	1,655
	Pioneer	RC	57	17,175
1994	SMI	Auger	12	769
1995	SMI	Core	4	668
		RC	24	8,160
1996	SMI	Core	3	1,136
		RC	112	32,448
1997	SMI	RC	68	16,480
		Totals	2,086	449,304

The availability of pre-Midas Gold drilling data has varied by operator, time period, and deposit. Midas Gold has reviewed and incorporated all pertinent and available data into its databases. Incorporated data include geologic logs, drilling recovery, assay values, surface and down-hole surveys, and relevant Quality Assurance/Quality Control (QA/QC) measures.

Geologic logging associated with pre-Midas Gold drilling varied in format between past operators. General logging procedures utilized paper logs including both visual logs and written observations. Characteristics recorded included core, cuttings and sludge recovery, lithology, alteration, pertinent mineralogy, sulfide percentage, oxide percentage/intensity, structures, and assay values such as gold, silver, antimony, and tungsten.

Drilling recovery varied by era of drilling. Early drilling by Bradley and USBM had poor recovery due to the drilling technology of the time. Core recovery from later operators, however, was much better with Pioneer, Hecla, and Superior showing moderate recovery (averages in the 60-70% range), El Paso and Ranchers showing better recovery (averages in the 70-80% range), and Barrick exceeding 90% recovery.

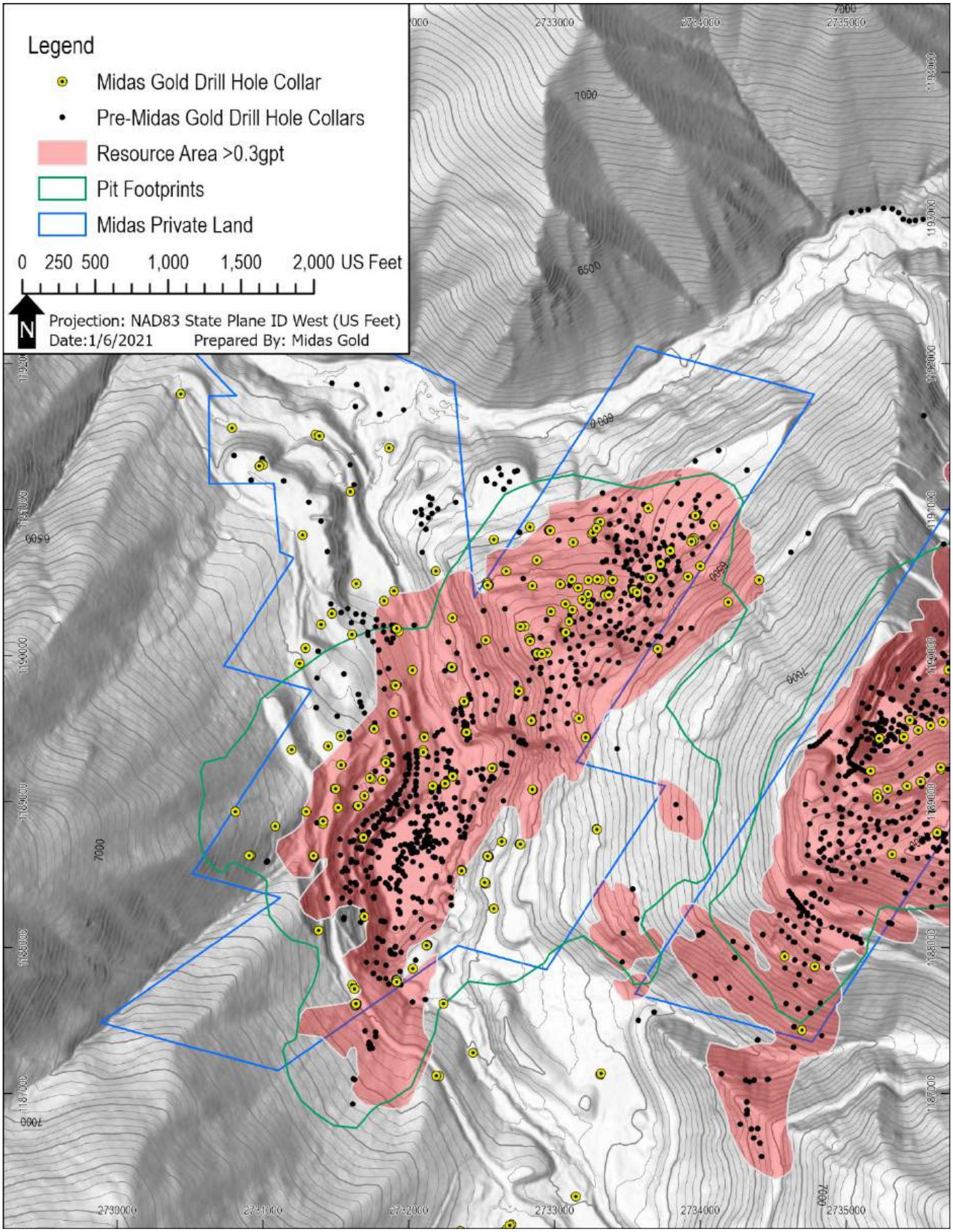
Data for QA/QC programs were available from some pre-Midas Gold operators and are discussed in further detail within Section 11 and data applicability to gold resources is discussed in Section 14.

10.3.1 Yellow Pine

Pre-Midas Gold drilling within the Yellow Pine mineralized area was conducted with multiple methods by a number of different companies (see Figure 10-2). The historical Bradley and USBM drilling (conducted prior to 1955) used conventional core drills of the time to drill AX, EX and BX sized core. The Hecla, Superior, Ranchers and Barrick drilling used wire line core drills with core sizes similar to Midas Gold, including PQ, HQ, and NQ. The RC drilling was conducted with buggy, track, and truck-mounted drills under dry and wet drilling conditions. The RC drill typically used a down-hole hammer with a 5.5-inch bit. Samples were collected by both a center return bit and an above-hammer interchange, and then traveled up the center of the drill string so that minimal contamination could occur. Typically, only a short section of casing was required. According to existing drill logs, operators began plugging their drill holes in the mid-1980s, prior to that time there was no hole-abandonment remediation required for previous drilling.

The operation was an active mine during parts of the drilling and the drill logs, plan maps, and sections illustrate the surveying standards that existed at the time of exploration, development, and mining activity. Historical files do not always describe in detail the methods used for locating holes, however, many survey records from pre-Midas Gold drilling do exist, are well preserved, and were utilized to construct the drill hole database. In addition, a considerable number of survey control points, old adits and shafts, and pre-Midas Gold drill hole collars were located by Midas Gold and included in its surveys, providing increased confidence in the location of pre-Midas Gold data including drill holes.

Figure 10-2: Yellow Pine Drill Hole Collar Locations

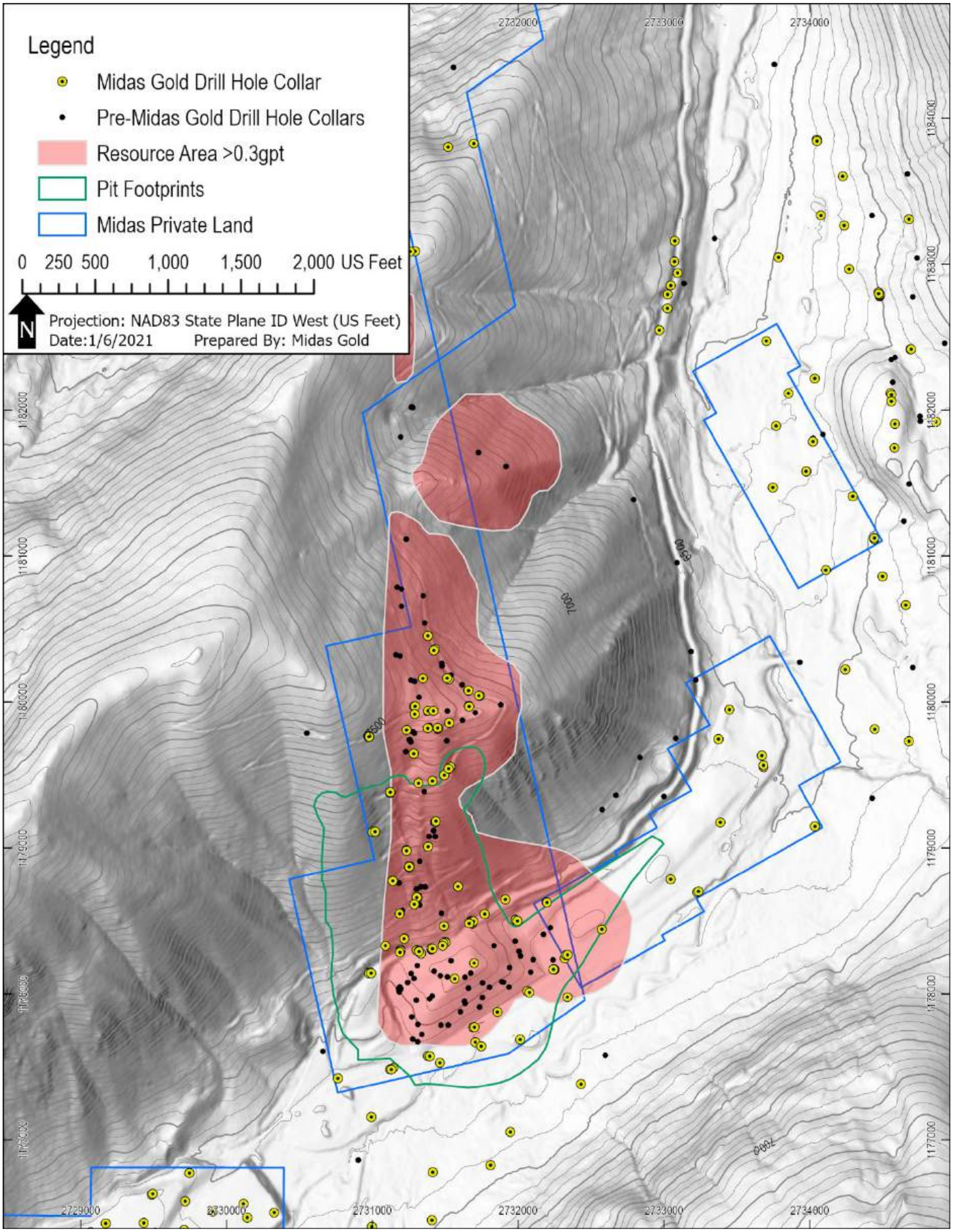


10.3.2 Hangar Flats

Pre-Midas Gold drilling within the Hangar Flats mineralized area was conducted with multiple methods by a number of different companies (see Figure 10-3). Most pre-Midas Gold drilling was conducted prior to 1960. Known drill core sizes utilized by pre-Midas Gold operators included AX, EX, BX and NX and were reduced as drilling conditions required. Typically, only a short section of casing was required. According to existing drill logs, operators began plugging their drill holes in the mid 1980's, prior to that time there was no hole-abandonment remediation required.

The drill logs, plan maps, and sections illustrate the surveying standards that existed at the time of exploration, development, and mining activity. Many survey records from previous drilling and underground development work by Bradley, as well as later campaigns under contract to the Defense Minerals Exploration Administration (DMEA) do exist, are well preserved, and were utilized to digitize the historical underground development workings and catalog drill data. Several of the older 1940's drill hole collars are still preserved and were surveyed and found to be within 3 - 6 ft of their expected locations, however, most collars were typically not preserved. Most of the later generation of drill holes, completed by Hecla in the area during the late 1980's, were located and surveyed in 2009 and 2010 and were found to be accurate to within 10 - 20 in.

Figure 10-3: Hangar Flats Drill Hole Collar Locations



10.3.3 West End

Pre-Midas Gold drilling within the West End mineralized area was conducted with multiple methods by many different companies, all of which were reputable industry operators or contractors. Most of the drilling was conducted in the 1970's and 1980's. Core drilling was much less common than RC and Air Track drilling, consisting of about 10% of drillholes mostly completed in the 1970's. The RC drilling was conducted with buggy, track, or truck-mounted drills under dry and wet drilling conditions. The RC drills typically used a down-hole hammer with a 5.5-inch bit. Samples were collected by both a center return bit and an above-hammer interchange, and then traveled up the center of the drill string so that minimal contamination could occur. Typically, the overburden in the mineralized area was very thin, and only a short section of casing was required. According to existing drill logs, operators began plugging their drill holes in the mid 1980's, prior to that time there was no hole-abandonment remediation required for previous drilling.

Historically, a drill location was first laid out by the mine surveyors with a specified easting and northing, and then a drill pad was constructed. After the pad was completed, the collar point was re-established. Original surveyor's records for most of the pre-Midas Gold drill holes are well preserved, and surveyed coordinates were verified against logs, as well as the dataset used in the mineral resource models. Pre-Midas Gold drill hole collars were typically not preserved due to post-drilling mining operations in the area, but some collars have been located by Midas Gold in its surveys and found to be accurate to within 3-15 ft. with some exceptions.

10.3.4 Historical Tailings

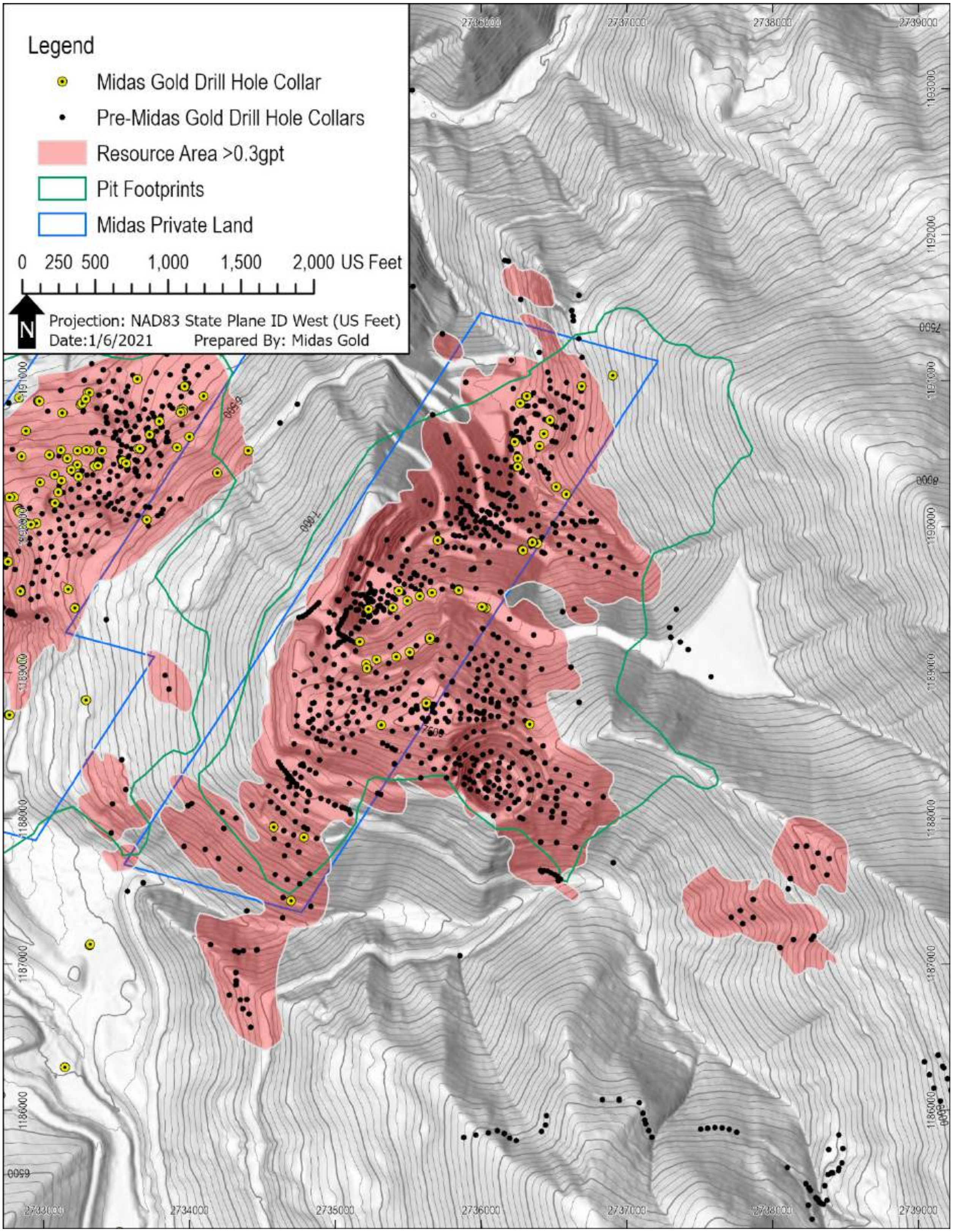
Pre-Midas Gold drilling within the Historical Tailings area was conducted primarily for water quality monitoring purposes. Stibnite Mines Inc. is the only known pre-Midas Gold operator to have drilled in this area and they used both RC (in 1996) and auger (in 1994) drilling techniques.

10.3.5 Scout

Pre-Midas Gold drilling in the Scout area was conducted with multiple methods by many different companies. Bradley generally drilled AX, EX and BX core in the 1940's and 50's while Pioneer and El Paso drilled BQ, BX, NX, and HQ in the 1990's and 1970's respectively. According to existing drill logs the overburden thickness in this area is significant and, in some instances, operators were forced to abandon drill holes due to collapsing conditions. There was no hole-abandonment remediation required at the time of the previous drilling.

Historical files do not always describe in detail the methods used for locating holes, but conventional survey methods tied to existing ground control were typically utilized. However, the drill logs, plan maps, and sections illustrate the standards that existed at the time of exploration. Some of the pre-Midas Gold hole collars are still preserved and were surveyed and found to be within 3 - 6 ft of their expected locations. Most collars were not preserved.

Figure 10-4: West End Drill Hole Collar Locations



10.3.6 Pre-Midas Gold Coordinates and Grid Conversions

Three common local mine grids were used for surveying hole locations by pre-Midas Gold operators: the Bradley, Ranchers, and Hecla grids. Some other grids were occasionally used, but they were able to be converted into one of the main three grid systems. Each of the three grid systems had a known conversion into Idaho State Plane West with NAD27 Datum.

Midas Gold has used two separate methods for grid conversion from historical coordinate systems. From the Project inception until 2013, coordinates were converted by first converting historical coordinates into the Hecla grid, then into Idaho State Plane (NAD27). Standard reprojection techniques with GIS software were used. In 2013, Midas Gold contracted Russell Surveying, Inc., a licensed and registered professional surveyor in Idaho to create conversions from various grid systems directly into NAD83 UTM coordinates. These conversions provided the basis for GIS coordinate systems in the historic grids that can be projected into any modern coordinate system and vice-versa accurately. These GIS coordinate systems provide the current conversion method for pre-Midas Gold grid coordinates to 1983 Idaho State Plane (feet).

10.4 MIDAS GOLD DRILLING

Midas Gold drilling is detailed in Table 10-3. Core and RC drilling was primarily conducted by Midas Gold for mineral resource definition and geotechnical data collection., Air lift and sonic drilling was conducted for monitoring wells and bedrock depth determination. Auger drilling was conducted for geotechnical investigation of unconsolidated material and resource definition of historical tailings. Cone penetrometer tests were performed for geotechnical investigation of unconsolidated materials.

Table 10-3: Drilling by Area Completed by Midas Gold

Hole Type	Year	# Holes	Feet
Yellow Pine			
Air Lift	2012	3	414
Auger	2015-2018	10	923
Core	2011-2018	181	129,911
RC	2011-2012	49	28,187
Sonic	2011-2012	10	1,150
	Totals	253	160,585
West End			
Air Lift	2012-2013	3	962
Core	2010-2017	35	29,408
RC	2011-2012	15	9,310
	Totals	53	39,680
Hangar Flats			
Air Lift	2012	6	948
Cone-Penetrometer Test (CPT)	2017	5	5
Core	2009-2017	108	91,967
RC	2012	18	14,955
Sonic	2011-2012	6	1,390
	Totals	143	109,265

Hole Type	Year	# Holes	Feet
Historical Tailings			
Air Lift	2012	1	60
Auger	2013-2017	52	4,596
CPT	2017	2	2
Sonic	2011-2017	8	1,067
	Totals	63	5,725
Scout			
Core	2012-2013	16	11,319
RC	2011-2012	5	4,310
Sonic	2011	7	230
	Totals	28	15,859
Non-Resource Areas (e.g. Planned Infrastructure Sites)			
Air Lift	2012	2	600
Auger	2013-2018	60	3,150
Core	2010-2017	11	7,603
CPT	2017	7	7
RC	2012	1	1,000
Sonic	2012	16	992
	Totals	97	13,352

Notes:
(1) For clarity the numbers in the table have been rounded to the nearest whole number.

At Yellow Pine, drilling was conducted in a wide range of orientations with approximately 80 – 160 ft spacing within the deposit. Drillholes are typically oriented to the southeast, south or northwest and inclined steep to moderately. This orientation provides an oblique angle of intersection between the predominant orientation of mineralization and the drill hole.

At Hangar Flats, drilling was conducted in a wide range of orientations with approximately 100 - 210 ft spacing. The holes typically bear to the south through west and are moderately inclined on average. The drilling that is oriented to the south and southeast intercepts the northeast trending mineralization at a preferable orientation near true thickness. The drilling oriented approximately easterly that is targeting the subvertical north-south trending mineralization commonly intercepts the mineralization at an oblique angle.

At West End, most drill holes are arranged in parallel at 65 - 100 ft spacing on section lines and inclined steeply to the northwest along parallel sections 100 ft apart. The mineralization is interpreted to follow two main orientations controlled by both the fault planes and stratigraphy, of which the drill holes intercept at a variety of angles.

In the Historical Tailings, drilling has defined a flat-lying zone of fine-grained mine tailings of potentially economic grade. Drilling was completed with an auger rig using vertical holes with approximately 230 ft spacing which crosscut the tailings perpendicular to the body. Intercepts are considered nearly true thickness.

At Scout, drilling is widely spaced (approximately 275 – 400 ft) and is oriented to the east to drill across the main mineralized zone to obtain true thickness.

10.5 SITE CHARACTERIZATION DRILLING

Numerous drilling campaigns have been conducted on the site for purposes other than resource exploration and definition. These programs included monitoring well installation, geotechnical investigations such as infrastructure site evaluation, and environmental monitoring. Several of the previous operators conducted geotechnical and hydrological drilling for various purposes and many of their records still exist. The existing geotechnical data has been used by Midas Gold for initial planning purposes and several of the previous wells are still being utilized for water supply and monitoring purposes.

Seventy-two core drillholes were drilled with tooling to collect oriented structural data. Core in split tubes was logged for geotechnical purposes by a geologist at the rig or in the core shack. These drillholes were also utilized for resource estimation and geologic modeling. Numerous non-resource holes were drilled for geotechnical analysis in soils for environmental or infrastructure site planning purposes. These drillholes included auger, sonic, core, and cone penetration test methods. Some of these holes were usable in resource estimation and geologic modeling but most were not drilled within resource areas. For example, holes around the Historical Tailings area generate data for site condition evaluation beneath the potential tailings storage facility. Other areas with drilling for site condition investigation include the potential mine camp site, the potential mill site, the potential development rock stockpile sites, and the potential diversion tunnel site.

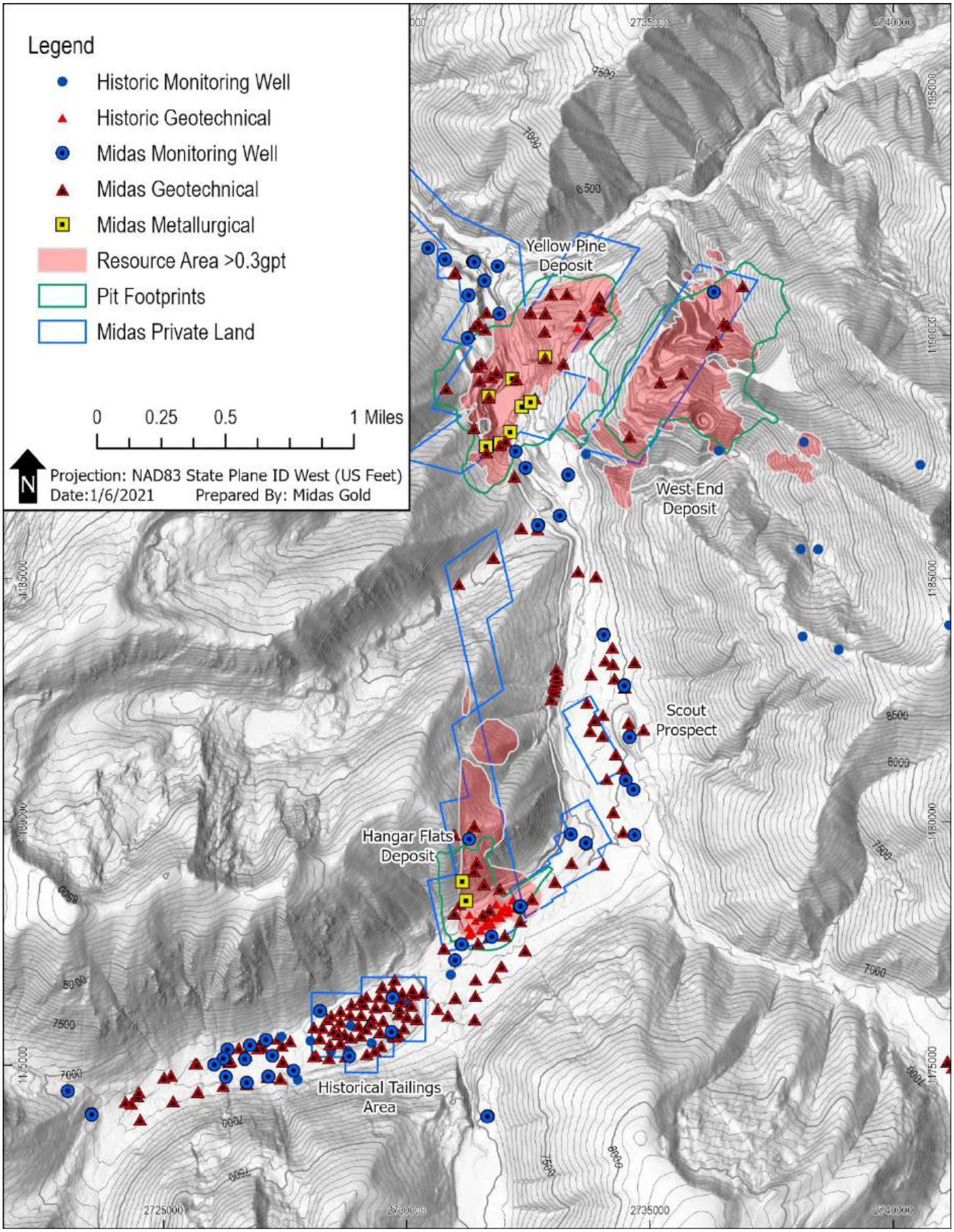
Some historic drillholes for purposes such as geotechnical investigation and water monitoring have surviving records. The current drilling database contains 25 pre-Midas Gold water monitoring wells which were drilled by SMI in the mid-90's, generally in the area currently known as Historic Tailings. Sixteen auger drillholes were commissioned in 1988 by Hecla for geotechnical investigation purposes, but not water monitoring, on the area currently known as the Hecla Heap. Hecla also drilled 2 geotechnical core holes at Yellow Pine in 1989 which have surviving geotechnical records.

10.6 METALLURGICAL DRILLING

Midas Gold drilled 15 core holes in PQ core size to provide metallurgical sampling material. Quartered core from these holes was assayed for use in mineral resource estimation, typically half-core was submitted or retained for metallurgical work, and the remaining quarter-core archived in the Midas Gold core storage facilities.

Additionally, core samples were taken from 302 other drill holes to be used for metallurgical testing. These holes were generally drilled with HQ core (excepting the Historical Tailings which were drilled via hammer-sampler auger) and were selected to generate representative samples for metallurgical programs such as variability testing, flotation cell testing, and pilot plant testing.

Figure 10-5: Site Characterization Drilling



10.7 GEOLOGIC LOGGING

Geologic logging performed by Midas Gold utilized paper log sheets in 2009 - 2010 and digital logging methods from 2011 - present. In 2009 and 2010, geologic logging on paper was completed onsite after core was received from the drillers. Logs included both visual and written observations recording lithology, alteration, pertinent mineralogy, sulfide percentage, oxide intensity, and structures. These paper logs were digitally captured after the 2009 and 2010 field seasons.

In 2011 - 2017, preliminary core logging was completed on site and detailed logging was completed at the core logging facilities in Valley County. Preliminary geological logging performed at Stibnite after core was received from drillers identified general geology and alteration for hole-tracking and daily reporting purposes. Subsequent detailed geologic logging was conducted using Microsoft Access digital logging forms. Pertinent geologic observations were digitally recorded including recovery, rock quality, lithology, alteration, mineralization, and structures. The Microsoft Access form was also used to record sample intervals and basic header information including azimuth, inclination, survey coordinates, logging geologist, drilling contractor, etc. Once logging was completed for a hole, the completed log was added to Midas Gold's Microsoft Access database after data verification. All logging was completed on-site beginning in 2017 and is located onsite at present.

Reverse circulation chip logging in 2011 and 2012 was completed using paper logs either at the drill rig or at the Stibnite core facility. These paper logs were later entered digitally using Microsoft Access® logging forms and the logs were added to the database.

10.8 DRILLING RECOVERY

In general, both RC and core recovery were good for all drilling completed by Midas Gold. Core recovery averaged 90.5%, and RC recovery was good to excellent. Whenever the RC drilling encountered voids, recovery suffered significantly, and if it could not be regained, the hole was terminated.

Numerous studies and statistical evaluations have been performed by Midas Gold staff testing the relationship between recovery and grade across the Project for both Pre-Midas Gold drilling and recent drilling conducted by Midas Gold. No meaningful relationship could be found.

Cyclicity issues were identified within a small number of the RC holes drilled by Midas Gold. Individual intervals were analyzed and those showing cyclicity were flagged for omission in mineral resource modeling. Problematic intervals were only identified and flagged in a small number of RC holes which were all drilled in 2011 and, as a result, these holes were excluded from mineral resource estimation.

10.9 ROCK QUALITY DESIGNATION

Rock Quality Designation (**RQD**) is a measure of naturally occurring fractures in a rock and was calculated when possible as part of the standard core logging procedures. RQD was measured as the sum of all complete core fragments with lengths greater than 3.9 in (10 cm) in a given core run with > R1 hardness value (will not crumble under a firm blow with the point of a geologic hammer) over the length of the core run. Lengths were measured along the centerline of the core, ignoring fault gouge or other low competency material and paying close attention to mechanical breaks from drillers boxing the core, as these are not naturally occurring fractures.

10.10 DRILL HOLE COLLAR SURVEYS

During the Midas Gold drilling programs, drill sites were located using handheld Global Positioning System (GPS) receivers. Drill hole orientations were calculated based on actual drill collar locations to ensure that holes were properly oriented. Alignment stakes were set and drill alignments surveyed using conventional survey tools or in some cases a Brunton-style compass.

Once holes were completed, the collar was marked with a cement cap containing a steel pin attached to a steel chain extending above ground surface with a tag identifying the drill hole number. Over the course of Midas Gold's drilling programs, these collars were either surveyed by a professional surveyor or an onsite geologist using a backpack GPS unit. Approximately 75% of drillholes collars were surveyed by a professional surveyor.

10.11 DOWN HOLE SURVEYS

Down hole surveys were performed on core holes using various survey instruments including an acid etch clinometer, tropari or, for Midas Gold drilling programs, a Reflex EZ-Shot tool to measure deviation from the collared orientations. Surveys were generally taken every 200 ft down hole with some exceptions due to lost or collapsed holes.

Survey values were received from drill contractors on paper logs and were captured in a master spreadsheet for entry into the drilling database. Magnetic declination corrections were applied by the drilling database manager prior to database import. Declination corrections were modified at least annually based on changing magnetic declination, sourced from the NOAA.

10.12 SAMPLE LENGTH AND TRUE THICKNESS

Sample length was a set value for the RC (5 ft) and auger drilling (5 - 10 ft within spent ore material, 2 ft within tailings). For core drilling, sample length was determined by the geological relationships observed in the core and was generally 5 - 7.5 ft. Changes in lithology and mineralization were used as sample breaks, and regular sample intervals were used within lithologic units and intervals of similar mineralization intensity.

Based on the wide range of drill hole orientations, many of the intercept lengths do not represent true thickness of mineralization. In general, at Hangar Flats and West End the drill hole intercept length is greater than the true thickness of mineralization. In the southern and northern areas of Yellow Pine, where mineralization occurs as discrete zones, the drill hole intercept length is generally greater than the true thickness. In the central region of the Yellow Pine deposit where mineralization is broadly disseminated, intercept lengths are equal to, or greater than true thickness.

10.13 CORE, CUTTINGS, REJECT AND PULP STORAGE

Core and cuttings were received by Midas Gold personnel from the drilling contractors and remained under supervision until shipped to Midas Gold's core logging facility in Valley County, ID. Once at the facility, core and cuttings were stored within the building and supervised during the workday and locked when vacant (nights and weekends). After core was logged and sampled, the remaining halved core was stored within Midas Gold's warehouses, or behind a secured chain-link fenced compound at the Cascade warehouse. Rejects were stored in the same locations. Once pulps were received back from the assay labs, they were stored by Midas Gold. Rejects are stored inside of the chain-link fence at the warehouse in Cascade. All storage locations remain locked when no Midas Gold personnel are present. In Cascade, both the fence and the warehouse remain locked.

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11 SAMPLE PREPARATION ANALYSES AND SECURITY

This section provides an overview of the sample preparation, analyses, and security procedures used by Midas Gold; where available similar information is also provided for pre-Midas Gold activities.

Sample preparation and analyses programs have been undertaken by the operators and vintages of drill campaigns. This section summarizes the verification work and practices employed for by each of the operators by year. The independent Qualified Person (**QP**) responsible for Section 11 of this report, Garth Kirkham, P. Geo., believes that the the sample collection, preparation, analysis and security for all Midas Gold drilling are consistent with industry standards and best practices supporting their use in mineral resource and mineral reserve estimation as detailed in this study.

11.1 SAMPLING METHODS

Throughout the last 90 years, multiple drilling and sampling methods have been used across the district by pre-Midas Gold operators as well as Midas Gold. Sampling methods and quality control measures have varied based on the era and the type of drilling.

11.1.1 Pre-Midas Gold Sampling

Drilling on site has utilized industry standard methods for sampling. Early operators utilized methods with small core diameters that required tripping out to recover the core samples. To achieve enough mass to assay, dehydrated drill cuttings and muds (sludge) were combined with the recovered core as was appropriate during that era. Modern era core drilling shifted to larger core size and the use of wireline methods allowing sample recovery without tripping out the drill stem between runs. Reverse Circulation (**RC**) drill holes were drilled under both wet and dry conditions and samples collected from a cyclone or similar splitter. Sample lengths were generally 5 ft in length, although many sample intervals were selected based on changes in lithology or changes in intensity of alteration and mineralization. Few documents have survived to describe sample preparation methods and little to no chain of custody records for previous operators are available.

11.1.2 Reverse Circulation Drill Sampling

Midas Gold RC holes were cased into competent bedrock and drilled wet. Samples were collected every five feet and holes flushed and cleaned between samples with water and drilling products. Sampled material was collected from a cyclone splitter into plastic totes. A flocculent was added if necessary and, after settling, the excess clear water was decanted off and the remaining sample was poured into labeled sample bags. QA/QC samples were inserted at the drilling rig by the attending geologist and typically included 1 certified standard, 1 blank and 1 cyclone splitter reject every 20th sample. (i.e. every 100 ft). Sample bags were placed into larger rice bags which were placed into bulk storage sacks and transported to Valley County facilities for shipping to the laboratory. Pre-numbered bar codes were utilized for sample tracking by both Midas Gold and the recipient lab.

11.1.3 Core Drill Sampling

From the beginning of the core drilling program in 2009, core was generally sampled on 5 ft intervals with sample breaks made at significant changes in lithology or intensity of alteration and/or mineralization. An exception is a period in 2012 when sample intervals for core were varied based on the logging geologist's interpretation of the intensity of mineralization such that if core was mineralized, samples were selected in 6.5 ft lengths; if core was not mineralized samples were selected in 7.5 ft lengths. The core logging geologist marked the core with a lumber crayon to provide a line for the core sawyer to split veins and joints into representative halves. Half of the cut core was placed into canvas sample bags, which were placed into labeled rice bags, and then placed into bulk storage sacks for shipment to the laboratory. Typically, sampling was conducted in batches of 50 samples including 2 certified standards, 2 blanks, and 2 quarter-core duplicates. Pre-numbered bar codes were utilized for sample numbering.

11.1.4 Sonic and Auger Drill Sampling

Sonic drilling samples were collected by the drilling contractor and placed into plastic sleeves which were set into cardboard boxes. This material was sampled in a manner similar to drill core samples.

Mineral resource definition in the unconsolidated Historic Tailings within the Spent Ore Disposal Area (**SODA**) was conducted with a hollow stem auger drilling method. Auger drilling utilized a split tube and samples were divided in half by the geologist. Material was composited into 10 ft samples within the SODA material and 2 ft samples within the tailings material and then placed into canvas sample bags. The other half of the tailings samples were retained and placed in wooden core boxes. In the Historic Tailings, at least one sample from 35 of the 42 drill holes was taken as a Shelby sample for specific gravity and particle size analysis. The geologist inserted one standard and one blank into the sample set for each hole within the tailings. The split tube was washed thoroughly between samples to prevent cross-contamination. Sampling of auger material in non-tailings drillholes was conducted in a similar fashion except samples were collected based on split tube recovery rather than composited depending on material type.

11.2 SECURITY AND CHAIN OF CUSTODY

All samples were kept under direct supervision of Midas Gold staff and its contractors or within locked facilities. Changes in custody were documented with signed and dated Chain of Custody (**COC**) forms.

RC and auger samples were bagged at the drill rig and prepped for shipment to the assay lab under supervision of the rig geologist. RC and auger samples were shipped to the Valley County logging facility in bulk storage bags accompanied by a signed COC form detailing drill hole numbers, footages, sample numbers, and the shipment date.

Drill core was picked up at the drill rig by the site geologist while performing the daily rig inspections. After inspecting the core boxes for errors, a COC form was completed documenting the transfer of core from the rig to the Stibnite core shack. Often the initial COC would be documented on the driller's daily log and included the box numbers, footages, date, and geologist's name and signature. At the core shack, a summary log was completed to verify and record box numbers, footages, lithology, mineralization and other rock characteristics. Upon completion of the summary log, the core was prepared for shipping to the Valley County logging facility by Midas Gold staff or contractors. When shipped, core was accompanied by a signed COC form detailing the hole numbers, footages, box numbers, and shipment date.

Once the core or samples were received at the Valley County facility, the receiver checked the COC for errors and stored the core for future logging/sampling in a secured site which was locked when no personnel were present. Once detailed logging and sampling of core was complete, the samples were prepped for shipping, bagged in rice bags, and sealed with tamper-proof security tape. From 2015 to present, most of these steps were conducted at on-site facilities and samples were transported to Valley County facilities ready for shipment to the assay lab. Each shipment was accompanied by another COC form to the assay lab. Upon receipt, the lab then verified that the security tape was undisturbed and completed the COC form.

11.3 DENSITY

In 2010, Midas Gold sent 61 samples from the 2009 and 2010 drilling campaign to ALS Chemex Labs, Ltd. (**ALS**) for density determination using a paraffin wax coating. Beginning in 2011, density measurements for core material were determined using in-house hydrostatic weighing. Measurements were collected by Midas Gold geologists on approximately 0.5 ft core intervals every 50-200 ft downhole, or within different lithologic units, totaling 3,318 intervals. Four hundred seventy-eight (14% of the 3,318) of these density samples were also submitted to ALS for density determination with paraffin wax coating. ALS results compared to measurements by Midas Gold showed a root mean squared coefficient of variation (**RMS CV**; a statistical tool routinely used to determine precision through using the quadratic mean of the relative standard deviation for each pair) of 0.988%, indicating there was no significant difference (assuming a value of zero means perfect measurement duplication) between the in-house measurements and third-party, independent certified lab results for density.

For the unconsolidated material within the Historic Tailings, 35 samples were sent to Strata Geotechnical Testing Laboratories in Boise, ID for density determination using the ASTM D2937 method. This method involves collecting an in-situ sample using a drive-cylinder with a known volume, weighing the sample, and calculating the density of the collected material.

11.4 ANALYTICAL LABS AND METHODS

There is little documentation of the sample preparation, analysis, and security for most samples from pre-Midas Gold operators. United States Bureau of Mines (**USBM**) utilized a government laboratory and analyzed drill core and sludge using a conventional 30 g fire assay pre-concentration method followed by gravimetric analysis. Other operators used several assay laboratories (both for primary and check assays) with CN-leach assays followed by atomic absorption (**AA**) for oxide mineralization and conventional fire assay techniques for sulfide mineralization. Bradley drilling sludge samples were analyzed using conventional fire assay techniques in company owned Yellow Pine and Boise laboratories. Table 11-1 shows the various analytical labs used by different operators. The various analytical methods utilized at various laboratories by pre-Midas Gold operators had different lower detection limits, upper reporting limits and sensitivities which are documented in the company's database and archives.

Table 11-1: Off-Site Assay Laboratories Used by Pre-Midas Gold Operators

Laboratory	Location	Operator	Year
T.S.L. Laboratories Limited	Spokane, WA, USA	El Paso	1973, 1978
		Superior	1975-1978, 1981
Union Assay	Salt Lake City, UT, USA	Ranchers	1973, 1975-1978 1982, 1984
Bondar Clegg	BC, Canada	Superior	1976
	North Vancouver, BC, Canada	SMI	1995-1996
Rocky Mountain Geochemical Corp.	Midvale, UT, USA	Superior	1976-1977
	Reno, NV, USA	Ranchers	1983-1984
Monitor Geochemical Laboratory	Elko, NV, USA	Superior	1978
Hazen Research	Golden, CO, USA	Ranchers	1982
Peter Mack	Wallace, ID, USA	Ranchers	1982
South Western Assayers and Chemists	Tucson, AZ, USA	Ranchers	1982
Mountain States Research and Development	AZ, USA	Ranchers	1982-1984
Silver Valley	Osburn, ID, USA	Superior	1983
Hunter	Sparks, NV, USA	Pioneer	1986-1988
ALS Chemex Labs Inc.	N. Vancouver, BC, Canada	Hecla	1989
		Barrick	1992
SVL Analytical Inc.	Kellogg, ID, USA	SMI	1997

11.4.1 Assay Laboratories

Midas Gold utilized multiple laboratories for assay, check assay, and metallurgical work in both the US and Canada. All labs were ISO 17025 or 9001 certified. Table 11-2 summarizes the assay laboratories used by Midas Gold for sample analysis from 2009 to present. A total of four labs have been used in the United States and Canada for primary and check assays. Midas Gold has utilized the same primary lab, currently known as ALS Global, for the entirety of the Stibnite Gold Project.

Table 11-2: Analytical Laboratories Used by Midas Gold

Laboratory	Location	Certification/ Accreditation	Use
ALS Global (ALS)	Elko, Reno, and Winnemucca, NV, USA; Vancouver, BC, Canada	ISO 17025:2005 ISO 9001:2008	Primary Lab 2009-Present
American Analytical Services (AAS)	Osburn, ID, USA	ISO 17025	Check Assays
Inspectorate	Reno, NV, USA	ISO 9001:2008	Check Assays Cyanide Gold Assays
SGS Canada, Inc.	Vancouver, BC, Canada	CAN-P-1579 17025:2005	Check Assays Cyanide Gold Assays

11.4.2 Metallurgical and Geochemical Laboratories

Table 11-3 summarizes the laboratories used by Midas Gold for feasibility study analysis. A total of thirteen labs have been used in the United States and Canada for metallurgical and geochemical testing in preparation for feasibility.

Table 11-3: Metallurgical and Geochemical Testing Laboratories Used by Midas Gold

Laboratory	Location	Certification/Accreditation	Use
SGS Canada, Inc.	Burnaby, BC, Canada	CAN-P-1579, CAN-P-1587, CAN-P-4E (ISO/IEC 17025:2005)	Metallurgical Testing
SGS Australia	Malaga, WA, Australia	ISO 9001:2015	Metallurgical Testing
Pocock Industrial, Inc.	Salt Lake City, UT, USA	Not Certified	Metallurgical Testing
McClelland Laboratories	Sparks, NV, USA	EPA ID #: NV00933	Geochemical Testing
Western Environmental Testing Laboratory	Sparks, NV, USA	EPA ID #: NV000925	Geochemical Testing
AuTec Innovative Extractive Solutions Ltd. (AuTec)	Vancouver, BC, Canada	Not Certified	Metallurgical Testing
CESL Limited	Richmond, BC, Canada	Not Certified	Metallurgical Testing
Blue Coast Research	Parksville, BC, Canada	Not Certified	Metallurgical Testing
CSIRO	Waterford, WA, Australia	Not Certified	Metallurgical Testing
FLSmidth USA Inc.	Midvale, UT, USA	Not Certified	Metallurgical Testing
Surface Science Western	London, ON, Canada	ISO 9001:2015	Metallurgical Testing

11.5 SAMPLE PREPARATION AND ANALYSIS

Midas Gold samples were received and weighed by the primary assay lab. Core samples were prepared based on laboratory specifications which involved crushed to 70% passing a ¼ inch mesh (6 mm) and drying at a maximum of 140 degrees Fahrenheit (60 degrees Celsius). Dried material was split and pulverized to 70% passing No. 10 mesh, split again, and pulverized to 85% passing No. 200 mesh. Material passing through the No. 200 mesh was then run with four primary analytical techniques.

Multi-element analysis entailed a 4-acid digestion followed by inductively coupled plasma atomic emission spectroscopy (**ICP-AES**) for 33 elements. Every 20th sample was digested in aqua regia followed by an inductively coupled plasma mass spectrometry (**ICP-MS**) finish for 51 elements with a fluorine add-on. Arsenic had a 5 parts per million (**ppm**) lower detection limit and a 10,000 ppm upper reporting limit. Samples reporting > 10,000 ppm As were re-analyzed by using a digestion in 75% aqua regia followed by an ICP-AES finish with a lower detection limit of 0.01% and an upper reporting limit of 60%. Antimony had a 5.0 ppm lower detection limit and a 10,000 ppm upper reporting limit. Samples reporting values > 500 ppm Sb were re-analyzed using 0.9 g sample added to 9.0 g Lithium Borate flux and fused in an auto fluxer. A disc was prepared from the melt and analyzed using X-ray fluorescence **(XRF)** spectroscopy with a lower detection limit of 0.01% (100 ppm) and an upper reporting limit of 50%. SGS check assays submitted in 2017 tested an alternate antimony assay method of sodium peroxide fusion with an ICP finish. Statistical comparison of XRF and this new ICP method did not show an appreciable difference in results. Sulfur had a 0.01% lower detection limit and a 10% upper reporting limit. Samples reporting values > 2% S were re-analyzed by using a 0.01 – 0.1 g sample in a Leco sulfur analyzer using an Infrared (**IR**) detection system with a 0.01% lower detection limit and a 50% upper reporting limit. Mercury analysis changed in 2015 from an aqua regia digestion and cold vapor AAS finish to an aqua regia digestion with mass-spec finish. Mercury values in excess of 100 ppm require an aqua regia digestion with ICP finish.

All gold assays were performed using a 30 g fire assay charge followed by an atomic absorption spectroscopy finish with a 0.005 ppm lower reporting limit and a 10 ppm upper reporting limit. Samples reporting values > 6 ppm were re-analyzed using a 30 g fire assay charge followed by a gravimetric finish with a 0.05 ppm lower reporting limit and a 1,000 ppm upper reporting limit. Samples reporting values >10 ppm were analyzed by metallic screen method with a 0.05 ppm lower reporting limit and a 1,000 ppm upper reporting limit.

Silver was analyzed via the initial multi-element ICP-AES analysis with a 0.5 ppm lower detection limit and a 100 ppm upper reporting limit. Samples reporting values > 10 ppm Ag were reanalyzed using an ICP-AES or AA finish with a 1.0 ppm lower detection limit and a 1,500 ppm upper reporting limit. Samples reporting values > 750 ppm Ag were reanalyzed using a 50 g fire assay charge followed by a gravimetric finish with a 5 ppm lower detection limit and a 10,000 ppm upper reporting limit.

11.6 DATABASE VERIFICATION

Midas Gold employs multiple electronic verification measures to regularly validate the database for accuracy in addition to the periodic manual verifications discussed in Section 12. Interval verification tools are run to check for intervals that are overlapping or out of sequence. Digital assay data received from the primary assay laboratory are imported directly into the database and then manually verified against pdf lab certificates. Assay data in the database are periodically verified against a master assay spreadsheet and original laboratory analytical reports to prevent assay value errors. Furthermore, sample number ranges are examined for unreasonable differences that may indicate sample switches or typing errors.

11.7 QUALITY ASSURANCE AND QUALITY CONTROL

Midas Gold exercised strict and rigorous QA/QC protocols throughout the different drilling campaigns from 2009 to 2018. Periodically these protocols were assessed for adequacy and improved accordingly. Pre-Midas Gold operators conducted various QA/QC programs for both their drilling and mine assay operations but not all records of QA/QC measures have survived to be reviewed by Midas Gold. However, Section 11.7.1 details the records that Midas Gold has collected and catalogued.

11.7.1 QA/QC Pre-Midas Gold

Pre-Midas Gold operators had varying QA/QC programs, but not all records have survived. Historical reports indicate that Bradley used duplicates and standards as QA/QC measures at Hangar Flats, but exact insertion rates are unknown. QA/QC data which are available from existing records are detailed in Table 11-4 for each operator by deposit.

Table 11-4: Pre-Midas Gold QA/QC Measures and Insertion Rates

Company	Deposit	Check(2)	Reject(3)	Rerun(4)	Standard	Blank	Totals(1)
Pioneer	West End	1.74%	5.54%	0.07%	8.67%	-	16.02%
SMI	West End	2.00%	-	2.56%	1.27%	0.35%	6.18%
Superior	West End	10.57%	-	0.56%	1.25%	-	12.38%
Pioneer	Yellow Pine	-	-	-	18.35%	-	18.35%
Ranchers	Yellow Pine	4.42%	6.44%	-	-	-	10.86%
Superior	Yellow Pine	1.19%	-	-	-	-	1.19%
Barrick	Yellow Pine	3.88%	-	-	-	-	3.88%

Notes:
(1) Percentage insertion rates stated are based on QA/QC analyses recovered from historical files and are likely not comprehensive.
(2) Check assays were performed at third party laboratories.
(3) Rejects consisted of a combination of sample rejects and sludge samples run at internal and third-party laboratories.
(4) Rerun assays were performed at internal laboratories.

11.7.2 QA/QC by Midas Gold (2009-2018)

Midas Gold exercised strict and rigorous QA/QC protocols throughout the different drilling campaigns and retained independent qualified persons to review and help improve QAQC procedures. Current procedures include insertion of standards (both certified and in-house customized), blanks, and duplicate samples into the sample stream to ensure confidence in external lab results. In addition, coarse rejects were re-labeled and sent to the primary lab for assay to test splitting and comminution practices. Pulp material was also sent to other laboratories for cross comparison. Finally, the primary lab analyzes pulp duplicates internally which are reviewed by Midas Gold and included in the QAQC analysis. Table 11.5 shows the insertion rates of various QA/QC measures used in Midas Gold drilling since project commencement. The various QA/QC measures are described in detail in the following sections.

Table 11-5: Midas Gold QA/QC Measures and Insertion Rates

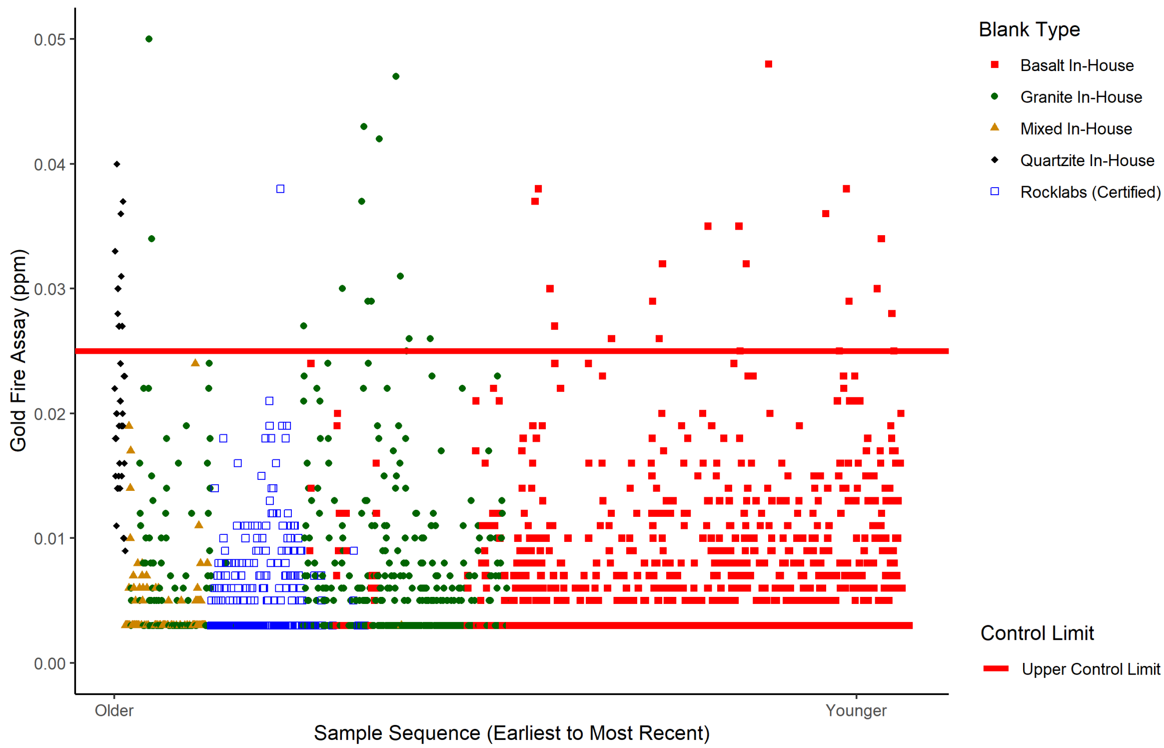
Deposit	Assays	Blank	Standard	Field Duplicates	Pulp Duplicates	Check	Reject	Totals
Yellow Pine	25,347	4.6%	5.2%	4.5%	5.0%	5.5%	1.5%	26.3%
Hangar Flats	19,246	4.5%	5.0%	4.3%	6.3%	2.4%	1.7%	24.2%
West End	6,251	4.5%	4.2%	4.5%	6.5%	3.5%	2.0%	25.2%
Historical Tailings	990	2.3%	5.8%	0.0%	4.7%	4.7%	0.0%	17.5%
Scout	2,341	4.8%	3.9%	4.8%	5.1%	0.9%	1.6%	21.1%

11.7.3 Blanks QA/QC

Midas Gold used a total of 2,493 blanks in the sample stream, 318 of which were certified (Figure 11-1). Non-certified in-house blanks were composed of locally sourced, unmineralized quartzite, basalt, or granite.

Gold grades of 0.025 ppm Au were selected as a control limit for blanks based on background cross contamination observed following spike samples. Upon evaluation, blanks reporting values below 0.025 ppm Au, a limit consistent with assay lab protocols, were considered satisfactory. Treatment of non-satisfactory samples is discussed in Section 11.7.8. Certified blanks reported all but 1 value under this limit and non-certified blanks reported 97.5% of values under this limit.

Figure 11-1: Blank Performance – Gold



11.7.4 Standard Reference Materials QA/QC

Insertion rate of standards typically exceeded 5% for drilling within all deposits. Midas Gold used a total of 1,705 certified gold standards, 1,044 non-certified gold standards, and 565 certified antimony standards (Figure 11-2, Figure 11-3). Some antimony standards were not certified at the time of use, but subsequently received certification.

Upon evaluation, standards reporting within two standard deviations of the expected value were considered satisfactory. Standards were flagged for evaluation when reporting between two and three standard deviations from the expected value and flagged as failed when reporting over three standard deviations. Standards flagged for evaluation were re-run on a case-by-case basis while the procedures for standards flagged as failed are described in Section 11.7.9. Certified gold standards reported 91.5% of values within satisfactory limits, non-certified gold standards reported 90% of values within satisfactory limits, and certified antimony standards reported 94.5% of values within satisfactory limits.

Figure 11-2: Certified Gold Standards

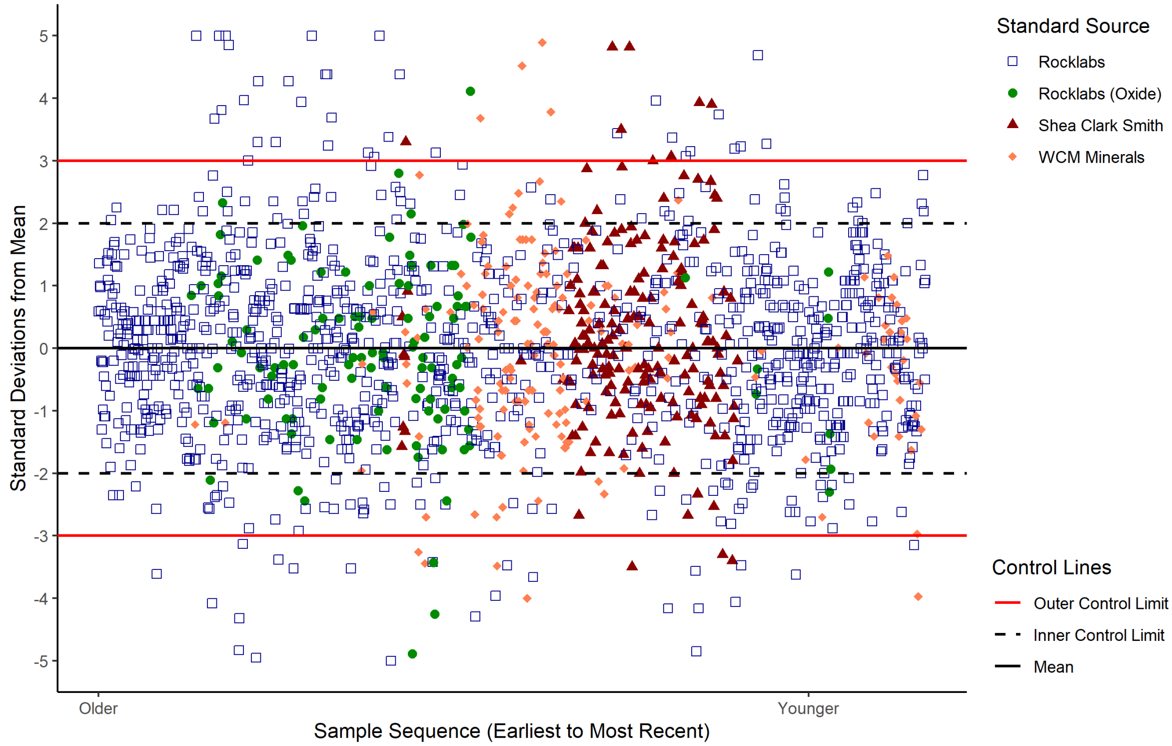
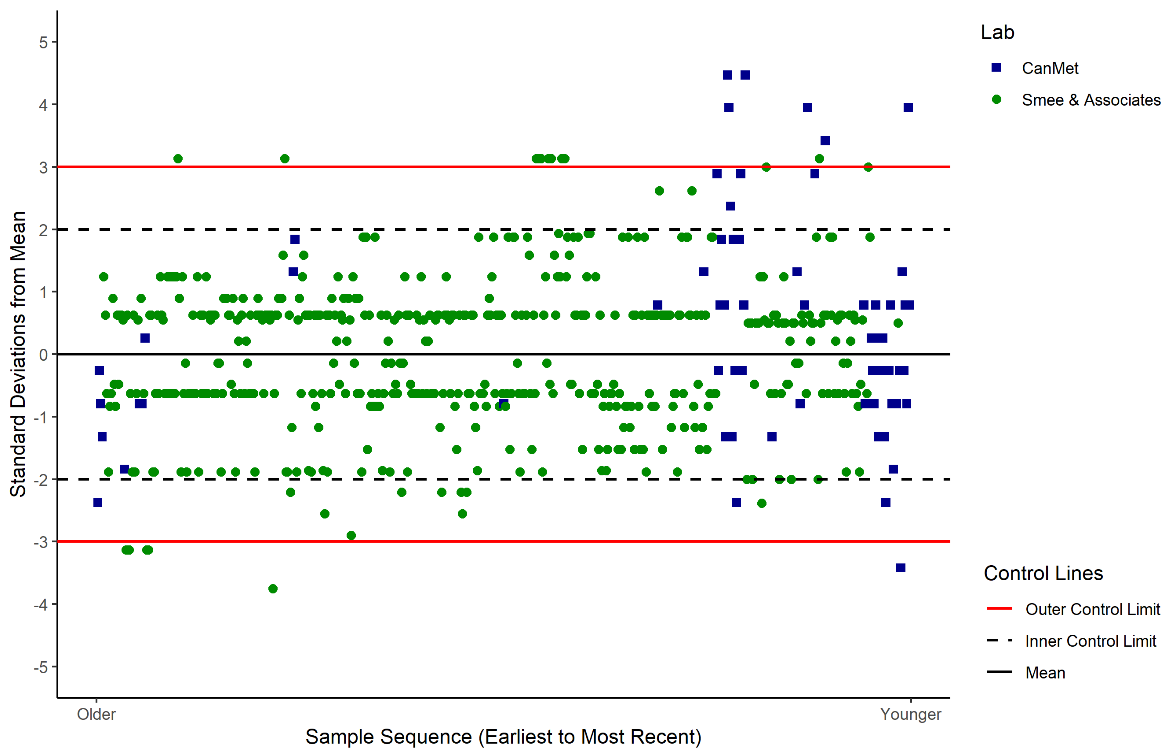


Figure 11-3: Certified Antimony Standards

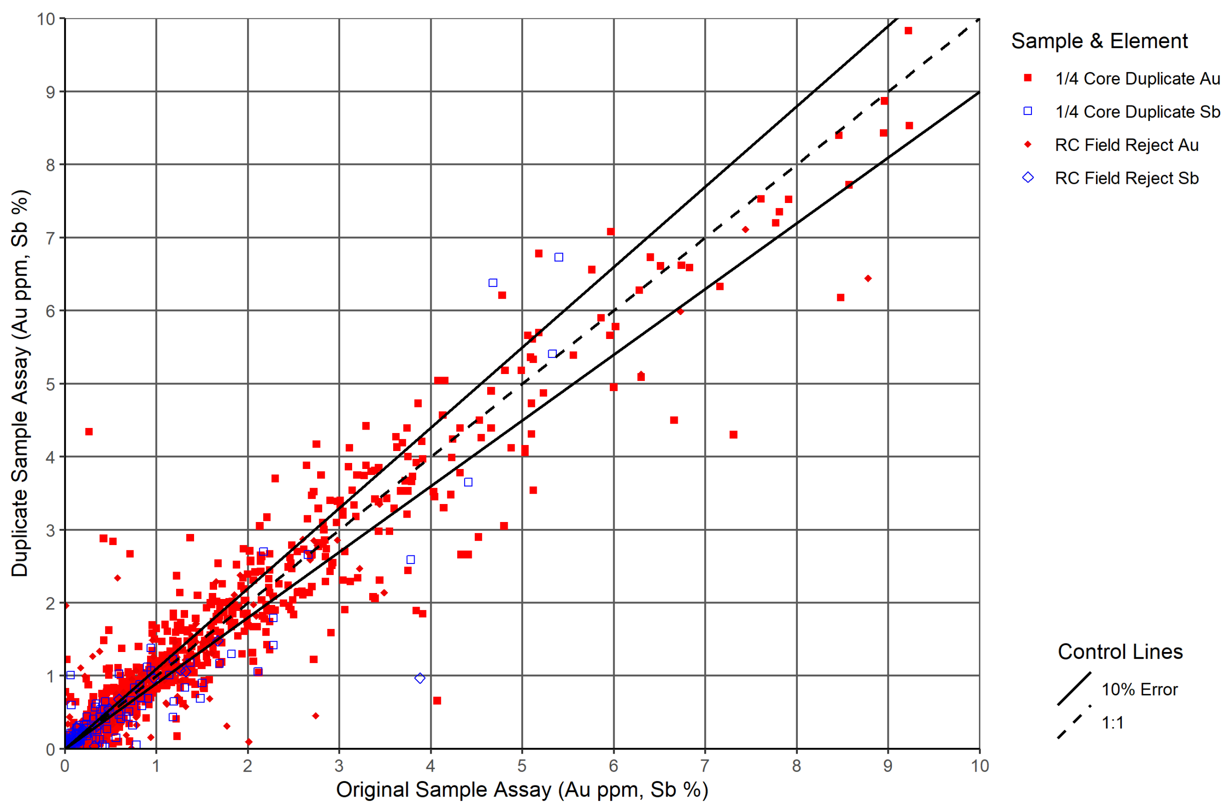


11.7.5 Field Duplicates QA/QC

Midas Gold generated 1,880 quarter core duplicates from core holes of which 1,115 were above 0.025 ppm by gold fire assay and 130 were above 0.05% antimony. Reproducibility for quarter core duplicates was fair for both gold and antimony with a RMS CV of 26% for gold and 37% for antimony however the correlation coefficients for both are excellent at 0.97 (i.e. 1 is perfect). In addition, removal of outliers significantly improves the RMS CV.

Midas Gold generated a total of 536 RC field rejects of which 365 were above 0.025 ppm by gold fire assay, and 19 were above 0.05% antimony. Reproducibility for RC field rejects was poor to fair for both gold and antimony with an RMS CV of 23.5% for gold and 18.8% for antimony, respectfully. Figure 11.4 shows a scatter plot of both field duplicate types. The correlation coefficient for the gold trendline is 0.88 and 0.33 for antimony, the latter being impacted by a limited number of analyses and by outliers.

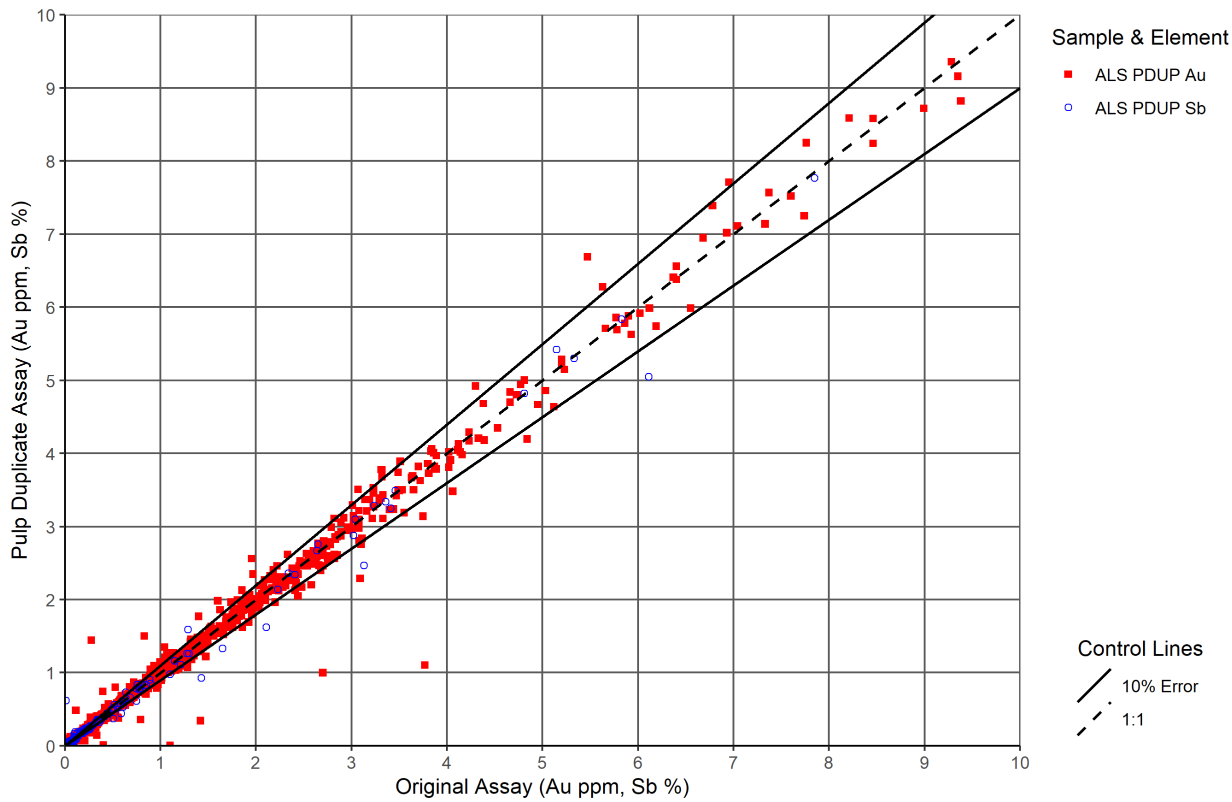
Figure 11-4: Field Duplicates



11.7.6 Pulp Duplicates QA/QC

ALS prepared one pulp duplicate for every twenty samples submitted. A total of 3,414 pulp duplicates were produced and assayed of which 1,788 were above 0.025 ppm for gold and 165 were above 0.05% antimony. Reproducibility for pulp duplicates was excellent for gold with an RMS CV of 8.7% and reproducibility was good to moderate for antimony with an RMS CV of 11.9%. Figure 11-5 shows scatter plots of the original assay values versus the pulp duplicate values.

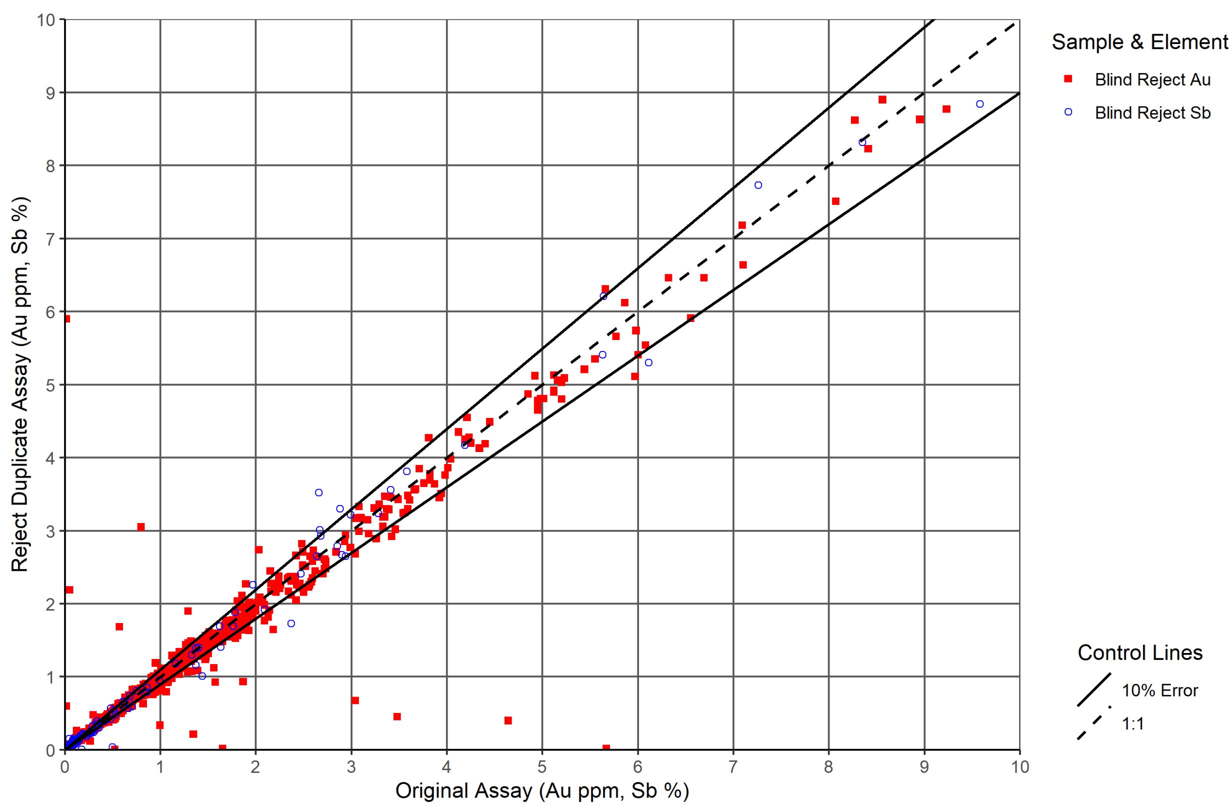
Figure 11-5: ALS Pulp Duplicates



11.7.7 Check Assays QA/QC

Midas Gold re-submitted 853 rejects with new sample numbers to ALS for assay to test for reproducibility and consistency (blind rejects). Out of the submitted rejects, 786 were above 0.025 ppm by gold fire assay and 118 were above 0.05% antimony by XRF. Within these parameters, the RMS CV for gold was 12.4% and the RMS CV for antimony was 10.4%, both values showing acceptable reproducibility. A scatterplot of these values is shown on Figure 11-6.

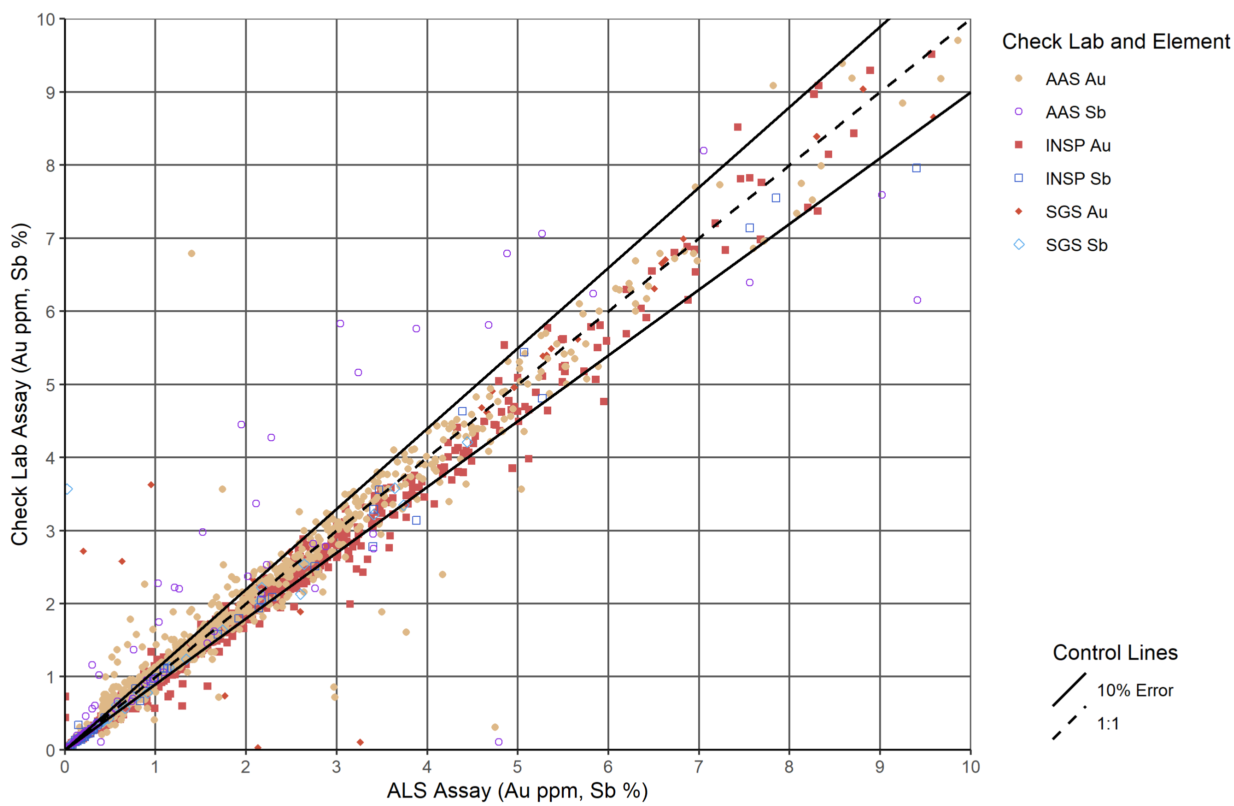
Figure 11-6: Blind Rejects Assays



Pulps were submitted to three different ISO certified laboratories for umpire assays as a cross check of ALS performance including: American Assay Labs, Inspectorate, and SGS. A total of 1016 pulps were submitted to Inspectorate for gold fire assay of which 988 were above 0.025 ppm. The average percent difference between the Inspectorate assay and the reported ALS assay was -4.57%. Of these samples, 125 were also assayed for antimony of which 63 exceed 0.05% antimony. The average percent difference between ALS and Inspectorate antimony assays for these samples was -4.41%. A total of 1,031 pulps were submitted to AAS for gold fire assay of which 908 were above 0.025 ppm. Eighty-five samples were assayed for antimony that exceeded 0.05%. The average percent difference between the AAS assay and the reported ALS assay was 4.49% for gold and 21.84% for antimony. Removal of sample outliers (absolute percent difference more than 75%) reduces the average antimony difference to 6.88%. Discrepancies are attributed to sample numbering issues at the check lab.

SGS analyzed 177 samples of which 62 were assayed for gold only and 115 were assayed for gold and antimony. One hundred sixty-two samples were above 0.025 ppm gold and 43 samples were above 0.05% antimony. The average percent difference between the SGS assay and the reported ALS assay for gold was 1.08% and for antimony was -3.95%. Figure 11.7 shows the QQ plot of umpire laboratory check assays of pulps.

Figure 11-7: Pulp Check Assays



11.7.8 Work Order Evaluation and Corrective Actions

Assay shipments containing drill samples, duplicates, standards and blanks are grouped as work orders, typically containing 50 samples total. Beginning in 2012 and retroactively, each standard and blank within ALS work orders was systematically evaluated using the criteria discussed in Sections 11.7.3 and 11.7.4. If a work order was flagged as questionable, the failed standards or blanks were re-assayed along with the 5 samples sequentially above and below the failure. Some work orders required assay revisions and others contained results that were confirmed by re-assay. When necessary, ALS would re-issue revised certificates and the Midas Gold database was updated accordingly. Table 11-6 summarizes the total and revised work orders over the Stibnite Gold Project to date.

Table 11-6: Work Orders and Revisions by Year

Year	Work Orders	Flagged Work Orders	Flagged Work Order Proportion	Work Orders with Original Results Confirmed	Revised Work Orders
2009-2014 (PEA & PFS)	678	104	15%	75	29
2014-2018 (FS)	32	2	6%	0	2

11.8 CONCLUSIONS

It is the opinion of the Independent Qualified Person that the sample collection, preparation, analysis and security for all Midas Gold drilling are consistent with appropriate methods for disseminated gold–antimony–silver deposits:

- Midas Gold drill programs included insertion of blank, duplicate and standard reference material samples;
- Midas Gold QA/QC program results do not indicate any problems with the analytical programs or procedures;
- Midas Gold data are subject to validation, which includes checks on lithology data, mineralization/alteration data, sample numbers, and assay data. The checks are appropriate and consistent with industry standards;
- independent data audits have been conducted, and indicate that the sample collection and database entry procedures are acceptable; and
- all core has been catalogued and stored in secure designated areas and is appropriately safeguarded against weather.

Where historical data are available, sample collection, preparation, analysis, and security for pre-Midas Gold drill programs, are generally considered to have used accurate methods for disseminated gold–antimony–silver deposits but can only be partially verified with appropriate supporting QA/QC results. The QP is of the opinion that the quality and reliability of the sample collection methods, sample security protocols, sample preparation and gold, antimony, and silver analytical data from the pre-Midas Gold drilling programs is sufficient to support their use in mineral resource and mineral reserve estimation with the exception of certain holes flagged and determined to be unreliable due to lack of supporting data, poor sample quality, lack of survey control, inappropriate analytical methods or reporting limits or obvious bias. Furthermore, the QP is of the opinion that the quality of the gold, antimony, and silver analytical data from Midas Gold drill programs is sufficiently reliable to support their use in mineral resource and mineral reserve estimation with the exception of certain reverse circulation holes that are flagged for exclusion due to cyclicity issues. These assumptions of validity are based on various reviews including analysis and inspection of original drill logs, assay certificates, statistical validations, assessment of geological continuity between pre-Midas Gold and Midas Gold drill holes, density of drilling, available pre-Midas Gold operator laboratory check assays and standards and inter-hole continuity.

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12 DATA VERIFICATION

12.1 INTRODUCTION

Data verification programs have been undertaken by numerous independent consultants as well as Midas Gold personnel, as discussed in previous NI 43-101 technical reports (SRK, 2011; SRK, 2012; M3, 2014) and performed subsequently. This section summarizes the verification work and practices employed for both historical and current data. The independent Qualified Person (**QP**) responsible for section 12 of this report, Garth Kirkham, P. Geo., believes that the datasets are validated and verified sufficiently to support their use in mineral resource and mineral reserve estimation for each of the respective deposits.

The QP has made multiple site visits to Midas Gold facilities in Valley and Ada Counties, Idaho. The QP visited the Lake Fork, Idaho offices and facilities April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas July 13 - 16, 2014, as well as January 12 – 14, 2017 and visited the Boise offices July 30 – August 1, 2018.

The tour of the offices, core logging, and storage facilities showed a clean, well-organized, professional environment. Onsite staff led Kirkham through the chain of custody and methods used at each stage of the logging and sampling process. All methods and processes are to industry standards and best practices and no issues were identified.

Four complete drill holes were selected by Kirkham and laid out at the core storage area. Site staff supplied the logs and assay sheets for verification against the core and the logged intervals. The data correlated with the physical core and no issues were identified. In addition, Kirkham toured the complete core storage facilities. No issues were identified, and core recoveries appeared to be very good.

The 2014 site visit entailed inspection of the workshops, offices, reclaimed drill sites, the Yellow Pine, Hanger Flats and West End mineral resource areas along with the outcrops, historical drill collars, and areas of potential disturbance for potential future mining operations. In addition, the site visit included a tour of the village of Yellow Pine, ID, which is the most likely populated area to be affected by any potential mining operation along with surrounding environs. The 2017 site visit entailed inspection of active core drilling operations in the Hangar Flats deposit as well as the onsite core logging and core cutting facilities, logging and change of custody proceedings. The drilling, logging and sample handling operations were conducted in a professional manner to industry standards and the onsite facilities were clean, well organized and of professional norms.

Kirkham is confident that the data and results are valid based on the site visits and inspection of all aspects of the Project, including methods and procedures used. It is the opinion of Kirkham that all work, procedures, and results have adhered to best practices and industry standards as required by NI 43-101. No duplicate samples were taken to verify assay results, but Kirkham is of the opinion that the work is being performed by a well-respected company and management that employs competent professionals that adhere to industry best practices and standards. Kirkham also notes that authors of prior technical reports (SRK, 2011; SRK, 2012) collected duplicate samples and had no issues.

12.2 MIDAS GOLD DATA REVIEWS

Kirkham reviewed the data storage and practices employed by Midas Gold, summarized as follows. Midas Gold professional personnel have constructed and maintained the drill hole and geologic solids databases in-house since project inception. A designated database geologist is supervised by an on-site resource geologist who is responsible and accountable for all data stored in the drill hole database and MineSight project directories. Midas Gold has updated and revised the drillhole database on numerous occasions, as outlined in the PFS (M3, 2014). In preparation for this study, Midas Gold has updated the database in the following manner:

1. Migration of the drillhole database from MS Access/GEMCOM to SQL/MineSight Torque;

2. Introduction of new drilling information from ongoing campaigns;
3. Addition of QA/QC from ongoing Midas Gold drilling;
4. Conversion of drillhole collars, downhole depths and other spatial data to the NAD 1983 Idaho State Plane West (feet) geographical coordinate system;
5. Revision of below-detection assay value assignments;
6. Addition of pre-Midas Gold blast hole assay information; and
7. Numerous minor changes and additions to database tables and structures.

Midas Gold and its contractors have conducted numerous audits of manual inputs of pre-Midas Gold drill hole information from original paper log copies. In-house audits completed by Midas Gold geologists include a 100% audit of drill hole collar locations (March, 2013), a 5% audit of pre-Midas Gold assay records (January, 2013), a 100% audit of gold assays and lithology records for the West End Deposit (April, 2013) and a 100% audit of USBM assay records for the Yellow Pine Deposit (May, 2013). In addition, Midas Gold routinely electronically verifies assay records in the drill hole database against original electronic laboratory certificates for Midas Gold drilling. Independent contractors completed a 1% audit of pre-Midas Gold assay records against the original paper log copies and a 5% audit of Midas Gold assay records against PDF lab certificates (February, 2014) and a 100% electronic audit of Midas Gold Yellow Pine assay records against original electronic lab certificates as well as a 100% audit of post-PFS drillhole data in 2018.

12.3 HISTORICAL DRILL HOLE DATA

Midas Gold and its contractors have completed numerous validations to assess the accuracy of the historical drill hole data and evaluate what data sets are appropriate for estimation of mineral resources. Kirkham has directed and reviewed these validations throughout his involvement with the Project, allowing for confidence in the quality of legacy data. Midas Gold and previous operators on the property have conducted extensive confirmation drilling programs that provide the basis for statistical and graphical inter-campaign drill hole data validations. Statistical validations completed in 2014 (M3, 2014) included paired sample analysis, comparison of de-clustered population statistics, panel comparisons and block kriging using different data sets. Prior to statistical validation, data from some drillhole campaigns were deemed unreliable and were removed from the database used for mineral resource estimation, as discussed in the PFS (M3, 2014) and in Section 14.

The review indicates that post-1973 drilling in the Yellow Pine, Hangar Flats and West End deposits generally show overall good agreement with Midas Gold drilling and between pre-Midas Gold campaigns, with certain exceptions. Pre-1953 USBM drilling and Bradley Mining Company surface drilling also compare well to Midas Gold and other post-1973 drilling campaigns for gold. Underground drilling generally shows a moderate high bias as compared to Midas Gold drilling, as do antimony assays in the Hangar Flats and Yellow Pine deposits. Observed bias in legacy underground drilling campaigns was attributed to orientation bias and structural controls on mineralization rather than analytical or sampling bias, as is discussed at length in the PFS (M3, 2014).

Midas Gold completed mineral resource sensitivity studies to further quantify the potential impact of use or exclusion of various drillhole information. Sensitivities for Yellow Pine in 2014, as previously discussed in the PFS, found only a 4% increase in contained gold using all drillhole data when compared to using only post-1973 data. Similar magnitude changes were observed when excluding Hecla drillhole data for estimation of mineral resources in the Homestake area of the Yellow Pine deposit. Mineral resource sensitivities in 2018, using updated geological models, indicated <4% change for Yellow Pine and <3% change for Hangar Flats by excluding pre-Midas Gold data. These sensitivity results are well within acceptable limits for validation of legacy drillhole information and the use of legacy drillhole information in estimation of mineral resources.

12.4 CONCLUSIONS

Kirkham visited the Valley County, ID offices and facilities April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas July 13 - 16, 2014, and again January 12 – 14, 2017. During these visits, no issues were identified, and all procedures and protocols were to industry standards.

The datasets employed for use in the mineral resource estimates are a mix of historical data and current, modern data. There is always a concern with respect to validity of the historical data. Extensive validation and verification must be performed in order to ensure that the historical data may be relied upon.

Kirkham directed and reviewed extensive validation and verification studies along with procedures performed by external consultants and by Midas Gold in order to ensure validity of the mineral resource estimates. The methods and procedures entailed detailed analysis and resulted in sub-sets of data being excluded.

It is the opinion of the author that the data used for estimating the current mineral resources for the Yellow Pine, Hanger Flats, West End and Historical Tailings deposits is adequate for this feasibility stage project and may be relied upon to report the mineral resources and mineral reserves contained in this report.

12.5 REFERENCES

M3 Engineering & Technology (2014). Stibnite Gold Project Prefeasibility Study Technical Report, prepared for Midas Gold, December 8, 2014, amended March 28, 2019.

SRK (2011). Technical Report on Mineral Resources for the Golden Meadows Project, Valley County, Idaho, prepared for Midas Gold, June 6, 2011.

SRK (2012). Preliminary Economic Assessment Technical Report for the Golden Meadows Project Idaho, prepared for Midas Gold, September 21, 2012.

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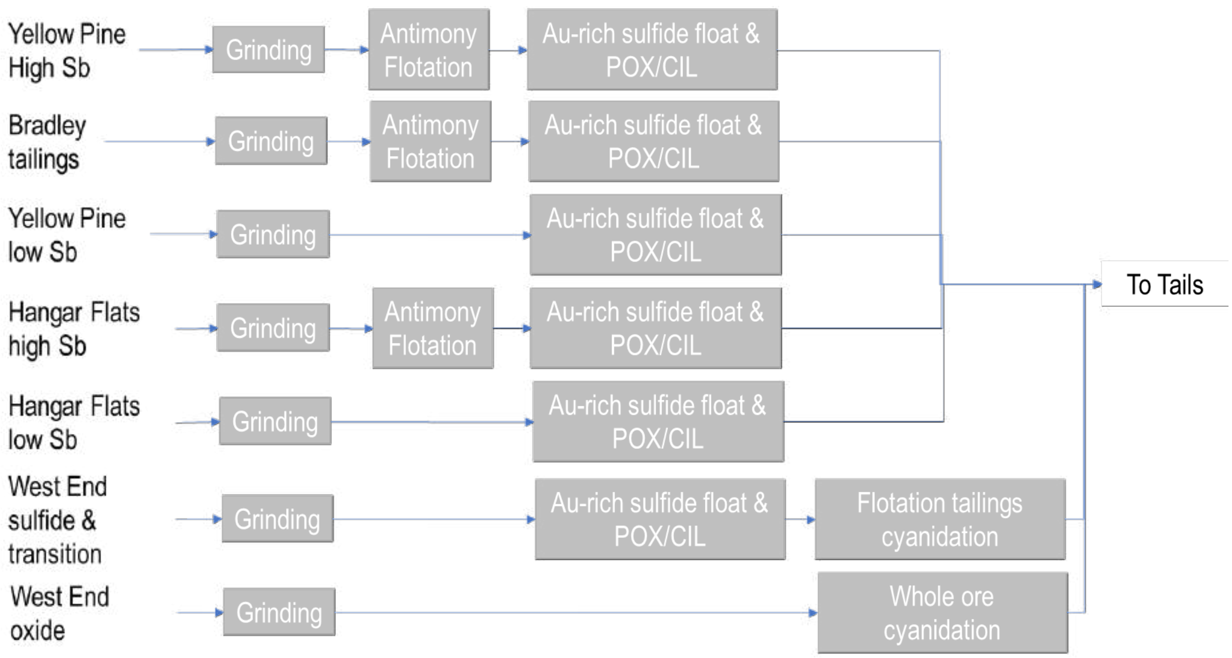
13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Testing has been conducted to support the feasibility study, on samples from the Yellow Pine, Hangar Flats and West End deposits. This work has included extensive mineralogical studies and developmental metallurgical test work on various ore types from each of the deposits. From the project outset in 2009, the primary objective of the metallurgical testwork program was to identify the most economic route to process the different feed types through the same mineral processing and hydrometallurgical facility. From a mineral processing standpoint, this required the need to develop grinding, antimony flotation, bulk sulfide flotation and tailings or whole ore cyanidation processes that could be utilized on an as-needed basis, as driven by the material type to be processed at the time. This allows for the seven overall feed sources to be processed using different permutations of the same equipment, as on Figure 13-1.

Figure 13-1: Different Process Routes for Different Feed Materials



This makes it possible to design a single plant that can process all ores from the Project as they are mined, using stockpiles to smooth ore characteristics and support batch processing by ore type.

Work for the feasibility study comprised four major phases, namely:

- Fine-tuning of the mineral processing circuit using master composites representing the major feed types described above (except Bradley Tailings);
- Development and fine-tuning of the hydrometallurgical circuit mostly in a single (Yellow Pine dominated) feed type;
- Application of the mineral processing circuit to different ore sub-types leading to design and execution of a variability program for the project; and
- Hydrometallurgical variability testing of a series of concentrates from most of the key feed types listed above material.

13.2 SAMPLE SELECTION AND COMPOSITE PREPARATION

Some 2,543 drill hole intervals from the Yellow Pine, Hangar Flats and West End deposits were delivered from site to the metallurgical laboratories for the purposes of building master composites for flowsheet development (Austin Zinsser (a), 2016) (Austin Zinsser (b), 2016).

These samples were used to create 19 master composites: 6 for flotation flowsheet optimization and the remainder for the production of concentrates for hydro-metallurgical and downstream cyanidation testing. They were also used to create 62 samples for diagnostic testing, 42 samples for variability flotation and tailings leach testing, and 57 West End variability samples for leaching only. The latter were used to re-test and validate the link established in the PFS (M3, 2014) between the AuCN assay and actual leach extraction. Finally, some 15 composites were prepared for grindability testing, to supplement data gathered in previous phases of the project (Table 13-1).

Table 13-1: Primary Metallurgical Composites for Testing

Comp. ID	Description	No. of Holes	Purpose	Head Grades					
				Au (g/t)	Ag (g/t)	As (%)	Sb (%)	$S_{(t)}$ (%)	CO_3
POL	Payback Period Comp low Sb	28	Flowsheet optimization	2.06	4.1	0.42	0.02	1.12	n/a
POH	Payback Period Comp high Sb	9	Flowsheet optimization	2.50	5.4	0.32	0.40	1.40	n/a
YPOL	Yellow Pine Composite	22	Flowsheet optimization	2.21	2.3	0.42	<0.01	1.19	n/a
HFOL	Hangar Flats Low Sb	18	Flowsheet optimization	1.57	1.7	0.49	0.02	1.08	n/a
HFOH	Hangar Flats High Sb	8	Flowsheet optimization	2.48	8.0	0.42	0.41	1.27	n/a
WESO	West End Sulfide	6	Flowsheet optimization	1.38	1.6	0.33	n/a	0.68	9.89
5208 Low Sb	Yellow Pine/Hangar Flats Bulk	73	Pilot float & Hydromet	2.20	n/a	0.49	0.02	1.16	n/a
5208 High Sb	Yellow Pine/Hangar Flats Bulk	26	Pilot float	1.78	n/a	0.31	0.34	1.25	n/a
5231 Low Sb	Yellow Pine/Hangar Flats Bulk	613*	Pilot float & Hydromet	1.78	n/a	0.36	n/a		n/a
YP 0-3 Low	Yellow Pine Years 0-3 Low Sb	5	Bulk float-Hydromet	2.21	n/a	0.42	n/a	1.05	n/a
YP 0-3 High	Yellow Pine Years 0-3 High Sb	3	Bulk float-Hydromet	2.11	n/a	0.37	0.47	1.37	n/a
YP 4+ Low	Yellow Pine Years 4+ High Sb	4	Bulk float-Hydromet	2.33	n/a	0.37	n/a	1.14	n/a
HFFZ	Hangar Flats MC Fault Zone	10	Bulk float-Hydromet	1.45	n/a	0.46	0.15	1.16	n/a
HFO	Hangar Flats non-Fault	7	Bulk float-Hydromet	1.39	n/a	0.39	n/a	1.03	n/a
WEAAP	West End High Carbonate	15	Bulk float-Hydromet	1.31	n/a	n/a	n/a	0.63	15.2
WELC	West End Low Carbonate	9	Bulk float-Hydromet	1.24	n/a	n/a	n/a	0.68	6.67
WEBL	West End Low Carbonate Blend	24	Bulk float-Hydromet	1.25	n/a	n/a	n/a	0.76	10.3
WEBM	West End Med Carbonate Blend	24	Bulk float-Hydromet	1.30	n/a	n/a	n/a	0.67	10.4
WEBH	West End High Carbonate Blend	24	Bulk float-Hydromet	1.33	n/a	n/a	n/a	0.69	12.4
HFWE-B1	Hangar Flats-West End Blend #1	31	Flowsheet optimization	1.38	n/a	n/a	n/a	0.85	7.72
HFWE-B2	Hangar Flats-West End Blend #2	41	Bulk float-Hydromet	1.38	n/a	n/a	n/a	0.92	6.80
*No of intervals									

13.3 GRINDING CHARACTERIZATION

A total of three JK Drop Weight Tests, twenty-eight SAG Mill Comminution (**SMC**), thirty-six Bond Ball Mill Work Index, twenty-one Bond Rod Mill Work Index, fourteen abrasion index and nineteen crusher work index tests have been conducted on the project. All the work was conducted by SGS Lakefield and SGS Vancouver; the results from these tests are provided in Table 13-2 ((Sun, 2017). When benchmarked against the SGS databases, the three deposits have average grindability characteristics of which West End is the least amenable to SAG milling but also the softest in ball milling.

A number of the samples studied in the earlier phases of work were selected from locations that, at a later stage of project development, dropped outside of the pit. However, as they represent the same geological materials as the material to be mined, the data has been included in the overall datasets summarized in Table 13-2.

Table 13-2: Grinding Characterization Samples

Test	Units	Yellow Pine			Hangar Flats			West End		
		No. of Tests	Avg.	75th Percentile	No. of Tests	Avg.	75th Percentile	No. of Tests	Avg.	75th percentile
JK Drop Weight SAG Testing										
A x b	N/A	1	103.5	n/a	1	123.2	n/a	1	63.4	n/a
Ta	N/A	1	0.68	n/a	1	1.5	n/a	1	0.37	n/a
SMC Testing										
A x b	N/A	10	93.6	17.5	10	159.0	105.2	8	50.0	37.6
Ta	N/A	10	0.93	0.84	10	1.61	1.00	8	0.49	0.37
Crusher and Mill Index Testing										
Crusher WI	kWh/Mt	7	5.7	6.1	7	6.0	7.0	5	9.6	12.5
Abrasion Index	N/A	6	0.21	0.25	5	0.19	0.22	3	0.24	0.31
Bond Rod Mill WI	kWh/Mt	9	11.2	11.3	7	10.5	10.8	5	13.9	15.0
Bond Ball Mill WI @ 150µm	kWh/Mt	7	13.7	14.1	7	13.3	13.6	7	13.0	13.5
Bond Ball Mill WI @ 100µm	kWh/Mt	5	16.2	16.4	5	16.0	17.1	5	16.2	16.4

13.4 MINERALOGY

Process mineralogical studies were conducted by SGS Vancouver, Process Mineralogy Consultants, Surface Science Western and Actlabs under the guidance of Blue Coast Metallurgy (Palko, 2011a) (Palko, 2012a) (Palko, 2012b).

Full gold deportment studies were conducted on twelve samples (four from each deposit), while 212 samples were subjected to bulk mineralogical analysis using QEMSCAN (101, 58, 50 and 3 from Yellow Pine, Hangar Flats, West End and the Bradley Tailings, respectively).

The gold is predominantly refractory to direct cyanidation, being present in solid solution or colloidal form in the host pyrite and arsenopyrite minerals. Discrete gold is particularly rare in the Yellow Pine and Hangar Flats deposits, but somewhat more abundant in the West End Deposit where some of the sulfides have been oxidized by weathering and other geological processes. Any discrete gold occurrences are very fine, typically ranging up to 10 microns (µm) in size. The mean grades of the gold hosting sulfides, as identified using laser-ablation ICP-MS are provided in Table 13-3.

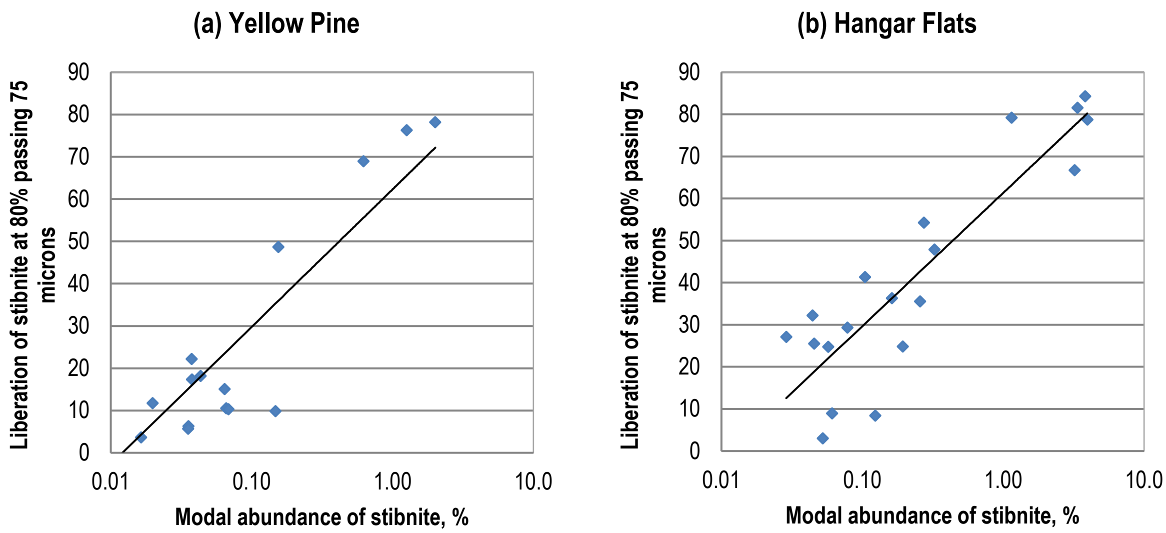
Both pyrite and arsenopyrite are non-stoichiometric. The pyrite is often strongly arsenian and the arsenopyrite commonly arsenic-deficient. Accordingly, whereas in many deposits of this type the gold is enriched in arsenopyrite, at this project it occurs in all iron sulfides. Gold is, however, primarily enriched within more porous or finely disseminated pyrite and arsenopyrite. The coarse crystalline sulfides contain relatively little gold, especially in Hangar Flats where coarse pyrite and arsenopyrite are virtually barren of gold.

Antimony occurs almost entirely as stibnite, which is typically coarse-grained when occurring in higher-grade samples. At head grades above 0.1% (0.14% stibnite) antimony, stibnite becomes sufficiently liberated to expect reasonable recoveries from the selective antimony float. Other antimony hosts such as myargyrite and tetrahedrite exist but at extremely low abundances.

Table 13-3: Discrete and Solid Solution Gold Mineralogy

Gold Mineralogy		Yellow Pine	Hangar Flats	West End
Free-Milling Gold		1-5%	1-17%	5-86%
Refractory Gold Host		Grade of Gold in Host Mineral (ppm)		
Pyrite	Coarse	23	5	19
	Porous	42	168	216
	Disseminated	108	212	104
Arsenopyrite	Coarse	54	3	17
	Porous	62	77	152
	Disseminated	88	n/a	n/a
Stibnite		1	n/a	n/a

Figure 13-2: Liberation vs Modal Abundance of Stibnite



The host rock bulk mineralogy is shown in Table 13-4, which describes the mean, 20th percentile and 80th percentile of each of the major components in a total of 212 samples analyzed by QEMSCAN using a standard Specimen Identification Protocol (**SIP**) tailored for the project. Some key features of these data include:

- Pyrite is, on average, the most abundant sulfide mineral in all deposits, although the arsenopyrite content is also significant. The samples of Bradley tailings studied were particularly rich in stibnite.
- West End tends to be poorer in sulfides, so mass pull to the pre-oxidation circuit would be lower. The effects of this low sulfide sulphur content on mass pull to pressure oxidation would be compounded by the need to control the carbonate content in West End concentrates subjected to autoclaving, so increasing the required upgrading in cleaner flotation.
- Quartz and k-feldspar are the dominant non-sulfide minerals. The mean quartz content is quite consistent between the deposits but varies more widely within each deposit. Yellow Pine is especially enriched in k-feldspar. West End contains relatively little k-feldspar. Instead, West End has a moderately high carbonate content making carbonate rejection a key objective in West End flotation making the need for cleaning greater in the processing of this material.

- Clays are best represented in these data by the sericite/muscovite category and tend to be richest in the Hangar Flats Deposit and West End Deposit.

Table 13-4: Distribution of Modal Mineral Abundances (%) by QEMSCAN Analysis

Mineral	Yellow Pine			Hangar Flats			West End			Bradley Tailings		
	mean	20th percentile	80th percentile	mean	20th percentile	80th percentile	mean	20th percentile	80th percentile	mean	20th percentile	80th percentile
Sulfides												
Pyrite	2.1	1.2	2.9	1.7	1.1	2.6	1.0	0.2	1.6	0.5	0.4	0.6
Arsenopyrite	1.0	0.3	1.6	1.0	0.4	2.1	0.7	0.1	1.0	0.2	0.2	0.2
Stibnite	0.1	0.0	0.3	0.1	0.0	0.4	0.0	0.0	0.0	0.2	0.2	0.3
Other sulfides	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Siliceous gangue												
Quartz	37.6	31.2	42.5	36.5	31.3	41.8	38.9	22.5	54.1	37.1	33.5	41.2
K-Feldspar	40.7	33.1	50.5	33.3	27.2	38.0	19.1	5.9	29.6	46.9	46.1	47.5
Sericite/Muscovite	9.8	6.5	13.3	11.7	6.3	18.4	13.8	6.7	18.7	7.7	5.6	9.6
Plagioclase	0.7	0.0	0.7	2.0	0.2	10.8	2.4	0.1	2.6	1.5	1.1	1.8
Other silicates	1.9	0.6	2.7	2.4	1.2	4.6	7.2	1.7	12.0	3.0	2.0	3.8
Carbonates												
Dolomite	1.7	0.4	2.3	0.4	0.1	1.1	7.6	0.9	11.7	1.1	0.9	1.3
Ankerite	1.7	0.5	2.0	0.5	0.0	1.2	5.8	0.8	8.7	0.3	0.2	0.4
Calcite	1.1	0.3	1.8	0.3	0.0	0.7	1.5	0.2	1.6	0.3	0.2	0.4
Other carbonates	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.1
Other gangue												
Oxides	1.1	0.4	1.6	0.8	0.3	1.6	1.7	0.5	2.7	0.8	0.6	0.9
Sulfates	0.1	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.1
Others	0.4	0.1	0.7	0.5	0.3	0.8	0.3	0.2	0.4	0.4	0.4	0.5

Mercury occurs primarily as discrete, microscopic grains of cinnabar (HgS) and coloradoite (HgTe).

13.5 FLOTATION TESTING

13.5.1 Past Testing

Considerable testing has been conducted on samples from the Yellow Pine, Hangar Flats and West End deposits in the recent past, supporting the PEA (SRK, 2012) and PFS (M3, 2014) studies.

For materials assaying more than 0.1% antimony, this work led to the development of a selective antimony flotation process, with a gold-bearing bulk sulfide rougher concentrate to be floated from the antimony flotation tailings. For materials assaying less than 0.1% antimony, selective antimony recovery is no longer feasible, so a bulk sulfide flotation process was developed, designed to maximize recovery of gold to a sulfide concentrate amenable to treatment by pressure oxidation.

The flowsheets developed from this work, reported in the PEA and PFS reports, formed the baseline processes on which the flotation testwork was conducted for the feasibility study. In brief, these flowsheets were:

- High Antimony Materials: The feed is milled under mildly alkaline conditions (with lime) using inert grinding media in the presence of sodium cyanide to depress the gold-rich, iron-bearing sulfides. Lead nitrate, an effective and selective activator for stibnite, was added to and conditioned with the ground product. The activated stibnite was floated using small doses of the Cytec dithiophosphinate collector Aerophone® 3418-A. The antimony concentrate was cleaned twice. The antimony circuit tailings were conditioned with copper sulphate and then a xanthate collector before flotation of a rougher concentrate. MIBC was used as the frother throughout.

- Low Antimony Materials: The feed is milled at natural pH with copper sulphate, then conditioned with isobutyl xanthate before rougher flotation with MIBC frother.

13.5.2 Rougher Flotation Development

The majority of the flotation testwork focused on bulk sulfide rougher flotation. Initially, a 5% sulfur grade was assumed as the baseline requirement for the feed to the autoclave. As this could be attained through rougher flotation alone, initially very little cleaner testwork was conducted – however as hydrometallurgical testing exposed the need for higher concentrate grades, options to upgrade the rougher concentrate were explored. This concentrate upgrading work is described in Section 13.5.3.

Rougher flotation development focused on, and was measured by, optimisation of the net present value of the project. Different treatment schemes varying the selection and dosage of activators, depressants, collectors and frothers were tested and their performance measured using the financial model created during the PFS. Once the flotation chemistry was considered optimal, a separate test program was run to optimise the primary grind size.

Five master composites were studied, namely:

- POH and POL – Yellow Pine–dominant, high and low antimony composites representing material to be mined during the anticipated pay-back period of the mine (Gajo, 2018a) (Gajo, 2018b);
- YPOL – low antimony material to be milled after year 4 from Yellow Pine); and,
- HFOH and HFOL – high and low antimony material to be milled after Year 4 from Hangar Flats).

Samples from each composite were subjected to a program of tests covering a matrix of conditions, designed to economically optimise the dosage of each of the key flotation reagents. Using the PFS financial model, by changing the input reagent cost data (per kg of reagent), this testwork led to an economic rationalisation of reagent requirements and a drop in the consumption of all flotation reagents, although the dithiophosphate, Cytec Aero® 3477, was added to the suite for some materials as the associated enhanced recoveries outweighed the cost of the reagent. The life of mine reduction in reagent tonnages and associated cost savings relative to the PFS are summarised in Table 13-5.

Table 13-5: Economic Impact of Process Optimization

Reagent	Tonnes of Reagent			Unit Cost ($/kg)	Life-Of-Mine Saving ($)
	PFS	FS	Reduction		
Lime	2,635	2,314	321	0.28	89,964
Sodium cyanide	828	607	221	2.50	551,700
Lead nitrate	3,917	2,421	1,496	2.77	4,144,114
Collector 3418A	187	161	26	11.75	302,022
Copper sulphate	16,445	8,770	7,674	2.90	22,254,890
Xanthate	15,938	12,649	3,288	2.40	7,891,848
Collector 3477	-	2,633	-2,633	2.50	-6,583,125
TOTAL			**10,393**		**28,651,413**

These reagent cost savings were achieved with an attendant increase in gold recovery. This improvement in recovery varied by composite but was typically 2-3 percent over that achieved using the PFS flowsheet on the same composite.

13.5.3 Yellow Pine and Hangar Flats Bulk Sulfide Concentrate Upgrading

In the PFS, only the West End feed material was cleaned, owing to the higher abundance of carbonates in the deposit (Table 13-4). However, as hydrometallurgical testing progressed, the need to upgrade concentrates from Yellow Pine and Hangar Flats became increasingly apparent. Early BTAC autoclave tests using batch Parr autoclaves revealed that excessive amounts of gangue could have a negative effect on autoclaving performance at the high percent solids needed to make the process autothermic. Further, the high levels of k-feldspar in the dilute concentrates led to the production of potassium jarosites. This had the dual effects of robbing iron from scorodite production (so destabilising the arsenic-bearing POX products) and entrapping gold in the jarosite matrix, so slightly reducing gold recovery.

While fine-tuning the autoclaving procedure, described later in this section, served to limit the production of jarosites, concentrate upgrading was deemed necessary to address the rheological challenges being faced in hydrometallurgy.

The initial aim was to enhance the sulfur grade of the concentrate from 5% to approximately 7.5%, so requiring a ~33% drop in mass recovery. Various approaches to concentrate upgrading were explored including concentrate desliming, flotation cleaning of just the slower-floating part of the concentrate and of the entire concentrate. Flotation cleaning of the entire rougher concentrate, though more costly than some other options, was selected as the preferred approach as it yielded the best metallurgy. This easily achieved the target mass reduction, while gold losses were kept to 1-2%.

The reagent scheme for concentrate cleaning was fine-tuned in a program of ten tests on the POL composite. Only light doses of collectors were found to be needed. This program explored using copper sulphate, Aero® 3477 and amyl xanthate collector at various doses. While there was little difference in response between all the conditions tested, the test using 25 g/t amyl xanthate and excluding all other reagents proved to be marginally better, so this was selected as the standard cleaner scheme.

13.5.4 West End Flotation Optimization

Fourteen batch tests were run on a West End Sulfide Optimization (**WESO**) composite, to optimise the West End flotation flowsheet through fine-tuning of the copper sulphate and collector doses. In the process of doing this, the copper sulphate dose to the roughers was halved from the PFS to 100 g/t and PAX dose was established at 125 g/t. A single stage of cleaning was found to be enough to achieve the target 1.3:1 mass ratio of carbonate to sulfur, indicated from hydrometallurgical work to be the maximum viable for autoclaving without acid addition. Unlike Yellow Pine and Hangar Flats feeds, which are essentially sulfide feed materials, West End feed almost always contains a partially oxidised transition fraction. This makes it slower to float so requiring more reagents in cleaning. Both copper sulphate (50 g/t) and PAX (60 g/t stage-added to the cleaners) were found to be necessary to maximise cleaner recovery, while regrinding was found to be detrimental to cleaner performance.

13.5.5 Reagents and Conditions Adopted for the Feasibility Study

The final reagent suites used for confirmatory tests and for this feasibility study are as shown in Table 13-6.

Table 13-6: Reagents used on Flotation of each Feed Material (all g/t)

Circuit	Reagent	High Antimony		Low Antimony		
		Yellow Pine	Hangar Flats	Yellow Pine	Hangar Flats	West End
Grinding	Sodium cyanide	35	35	-	-	-
	Lime	200	225	-	-	-
	Copper Sulphate	-	-	100	100	100
Sb Conditioning	Lead nitrate	200	250	-	-	-
	Cytec 3418A	15	10	-	-	-
Antimony rougher flotation	Cytec 3418A	-	-	-	-	-
	MIBC	20	25	-	-	-
Antimony cleaner flotation	Sodium cyanide	20	20	-	-	-
	Cytec 3418A	-	4			
	Lead nitrate	-	20			
	MIBC	-	-	-	-	-
Bulk sulfide Conditioning	Copper Sulphate	120	100	-	-	-
	PAX	65	60	35	35	35
	Aero 3477	-	-	10	10	-
Bulk sulfide rougher flotation	PAX	135	90	90	90	90
	Copper Sulphate	30	-	-	-	-
	Aero 3477	-	-	40	40	-
	MIBC	35	15	45	25	
Bulk sulfide cleaner conditioning	Copper Sulphate	-	-	-	-	50
Bulk sulfide cleaner flotation	PAX	25	25	25	25	60
	Aero 3477	-	-	-	-	-
	MIBC	4	4	4	4	-

Although the reagent dosage for each composite was optimised independently, the final low Sb flowsheet proved to be quite similar for all low Sb composites, incorporating 100 g/t copper sulphate added to the mill at natural pH, and 125 g/t PAX and 50 g/t Aero 3477 (Yellow Pine and Hangar Flats) stage added through the rougher float, with varying doses of MIBC frother. The float was conducted at natural pH. This similarity in optimal flowsheets bodes well for the blending of material from the different resources in any ratio required.

The high Sb flowsheet was also similar for the antimony-bearing composites from Hangar Flats and Yellow Pine. It included 200-225 g/t lime and 35 g/t sodium cyanide added to the mill, then 200-250 g/t lead nitrate added for stibnite activation in conditioning ahead of flotation with 15 g/t of the dithiophosphinate collector 3418A and 20-25 g/t MIBC added ahead of a very short stibnite rougher flotation stage. Copper sulphate, at 100-120 g/t, was added to the antimony flotation tailings, mostly in conditioning ahead of bulk sulfide rougher flotation (in Yellow Pine treatment, an additional 30 g/t is dosed midway through the float) and 200 g/t amyl xanthate was stage added through the float. MIBC (~35 g/t) was used as the frother. A small amount of sodium cyanide (10 g/t per cleaner stage) was the only reagent added to the two stages of antimony cleaning, which, in the laboratory, were each 2 minutes in duration.

The test conditions (grind sizes and residence times) adopted as standard for the FS are shown in Table 13-7.

Table 13-7: Residence Times and Grind Sizes Adopted for Feasibility Study

Circuit	Reagent	High Antimony		Low Antimony		
		Yellow Pine	Hangar Flats	Yellow Pine	Hangar Flats	West End
Grinding, 80% passing size (microns)		85	85	85	85	85
Residence times (minutes)						
Sb conditioning	Lead nitrate	1	1	-	-	-
	Cytec Aerophine® 3418A	1	1	-	-	-
Sb rougher flotation		2	2	-	-	-
Sb cleaner conditioning	Sodium cyanide	1	1	-	-	-
	Lead nitrate	1	1	-	-	-
	Cytec Aerophine® 3418A	1	1	-	-	-
Sb Cleaner 1		2	2	-	-	-
Sb cleaner conditioning	Sodium cyanide	1	1	-	-	-
	Lead nitrate	1	1	-	-	-
	Cytec Aerophine® 3418A	1	1	-	-	-
Sb Cleaner 2		2	2	-	-	-
Bulk sulfide conditioning	Copper Sulphate	3	3	-	-	-
	PAX/Aero® 3477	1	1	1	1	1
Bulk sulfide float	Rougher flotation	31	31	31	31	31
Bulk sulfide float	Cleaner flotation	30	30	30	30	30

All flotation was configured in open circuit except for the antimony second cleaner tailings, which was recycled to the antimony first cleaner feed.

Following completion of the test program, it was concluded that the optimal concentrate grade for autoclaving of Yellow Pine and Hangar Flats material, was 6.5% sulfur. This is lower than had been the objective at the time of process development and may lead to a further review of process selection in due course. Desliming of the concentrates would be much cheaper and could easily achieve this level of upgrade without much loss of gold. In fact it may also be possible to demonstrate that rougher flotation alone can achieve this sulfur grade at target recoveries. These should be studied further in the value engineering phase. However, the process as described in this section was left unchanged for the sake of the FS.

13.5.6 Primary Grind Optimization

The primary grind/metallurgy trade-off was established for each of the deposits, using their respective master composites. Populations of data on the effect of primary grind size on flotation performance were established for a variety of master composites. This was done either by suites of replicate batch tests where the selected flowsheet is entirely open circuit – i.e. the low antimony floats on Yellow Pine, Hangar Flats and West End), or locked cycle tests where the flowsheets included circulating streams, as in high antimony materials. The majority of the work was done on the low Sb samples which represent the bulk of the ore feed for the Yellow Pine and Hangar Flats deposits.

Recognising changing the primary grind size can be expected to have an impact on the reagent need, the reagent dose was slightly reduced for coarser primary grind sizes and slightly increased for finer primary grind sizes. In addition, applicable data from earlier programs were included to further boost the statistical robustness to the analysis.

Recovery of gold as a function of grind size is shown for selected composites on Figure 13-3. The POL and POH composites were designed to represent early production low- and high- antimony material respectively, predominantly comprised of Yellow Pine material. YPOL and HFOL composites are comprised of mid-life low Sb feed from the Yellow Pine and Hangar Flats pits. As the amount of higher Sb material to be mined is quite limited in these later years, a specific high Sb composite for the later years was not tested. Instead, the data from the POH composite was used to represent this material. West End was excluded from the analysis at the time, mainly as the West End flowsheet development was not complete at the time, but also because West End is only fed into the plant at any major tonnage towards the end of the mine life so the impact on project economics of non-optimal grind size selection for West End was discounted and any modifications to the primary grind can be addressed during the operating life. Previous work in the PFS had indicated that the grind needs for West End were quite similar to the other deposits. Best fit regressions were obtained on each dataset, and using the mix of feed materials as developed in the PFS mine plan, the year-by-year life of mine recoveries were obtained as a function of different grind sizes, as shown in Table 13-8.

Figure 13-3: Effect of Primary Grind size on Gold Recovery to a 5% Sulfur Concentrate

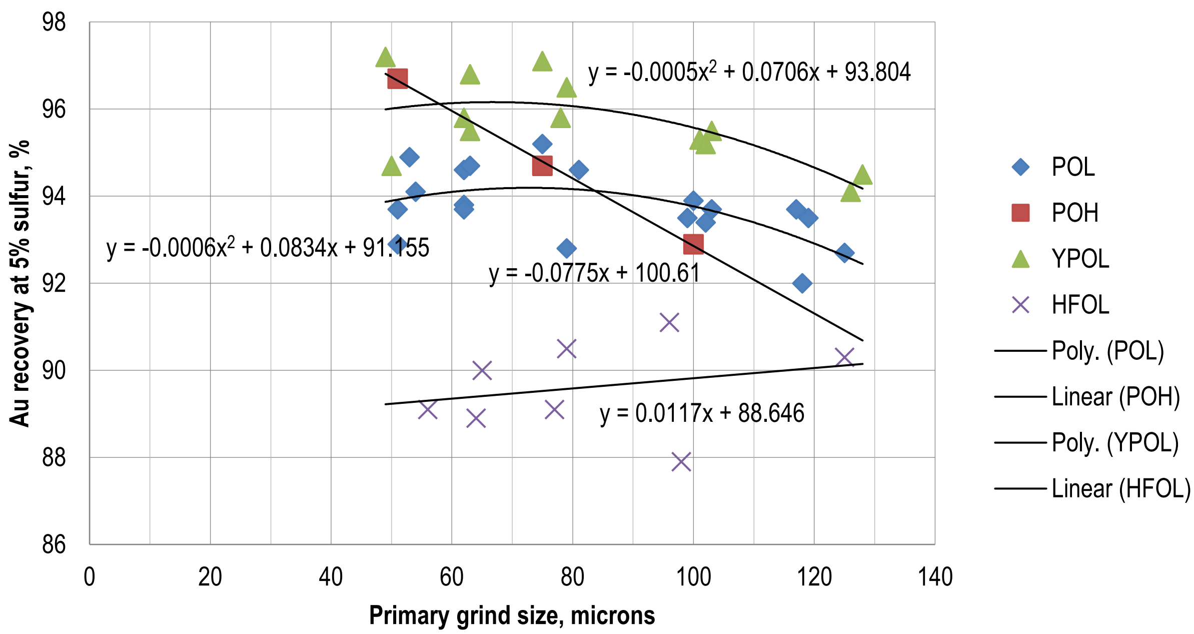


Table 13-8: Effect of Grind Size on Gold Flotation Recoveries

Grind Size p80, microns	Project Wide Gold Flotation Recoveries										
	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Ave
50	94.8%	94.4%	94.1%	96.1%	95.9%	95.4%	94.7%	93.0%	90.6%	89.9%	93.9%
65	94.5%	94.3%	94.2%	96.2%	96.0%	95.5%	94.7%	92.6%	90.6%	90.0%	93.9%
75	94.3%	94.2%	94.1%	96.1%	95.9%	95.4%	94.6%	92.2%	90.5%	90.0%	93.7%
85	93.9%	93.9%	93.9%	96.0%	95.8%	95.2%	94.4%	91.9%	90.4%	90.0%	93.5%
100	93.3%	93.4%	93.4%	95.7%	95.4%	94.9%	94.1%	91.4%	90.4%	90.1%	93.2%
125	91.8%	91.9%	92.1%	94.6%	94.2%	93.7%	93.0%	90.3%	89.8%	89.7%	92.1%

This size-by-size metallurgical forecast, together with estimated grinding costs using the grindability data described earlier, was used in an economic trade-off exercise to identify the optimum primary grind size (80% passing 85 microns).

13.5.7 Production of Concentrates for Hydrometallurgical Testing

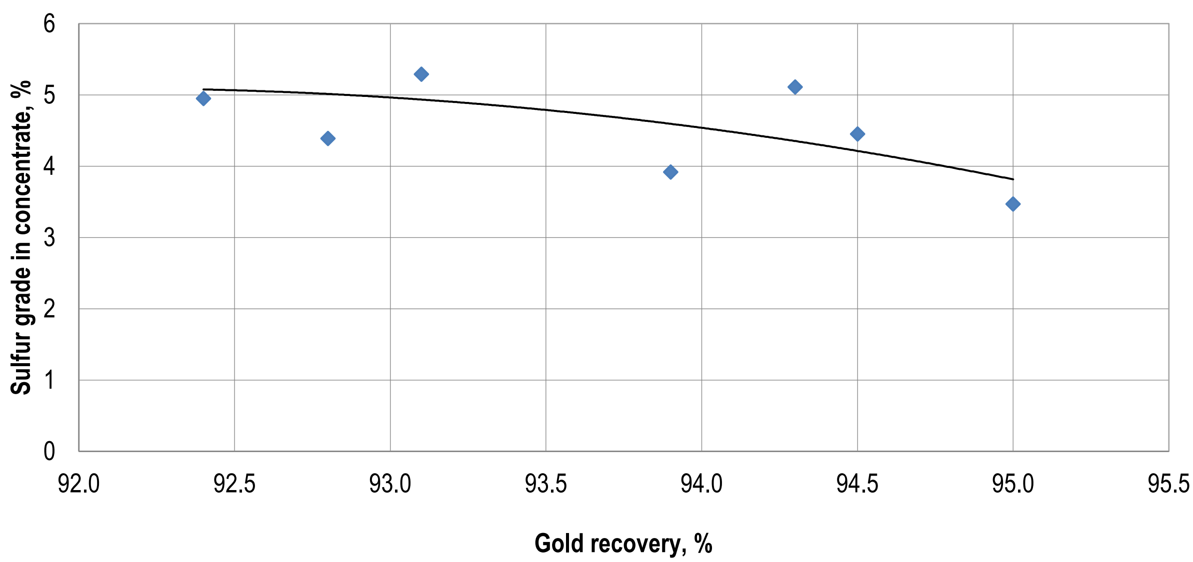
13.5.7.1 Pilot Plant Testing

Three flotation pilot plant runs were conducted through the feasibility study. The first two runs were executed purely to make concentrate for pilot-level pre-oxidation work. The first run milled and floated 3,600 kg of low antimony material mostly from Yellow Pine, in eleven dayshift campaigns using a pilot plant consisting of a single stage ball mill operating in closed circuit with a screen and targeting 80% passing 80 microns. Two banks each consisting of 4x28L Hazen-Quinn agitated flotation cells, performed the rougher float. Throughput was at 56 kg/hr.

As the purpose of the pilot work was to produce concentrate for autoclave testing, no process optimisation was conducted and relatively little tuning-in of the pilot plant was possible in the short run times available. Accordingly, care should be taken not to over-interpret the piloting metallurgy from a metallurgical forecasting perspective.

Further, this run was completed before the decision was made to produce higher grade concentrates, so it employed just crushing, grinding and rougher flotation. On average it produced a concentrate assaying 4.2% sulfur at 97% sulfur recovery and 94% gold recovery, while 5% sulfur concentrates were floated at about 93% gold recovery (Figure 13-4).

Figure 13-4: Pilot Plant Run #1 Rougher Flotation, Sulfur Concentrate Grade vs Gold Recovery



In addition, a small amount (600kg) of high-Sb feed was processed in a single campaign. No optimization was possible during this run, which yielded Sb concentrates assaying of 41-56% antimony, at antimony recoveries up to 86%.

A second pilot plant program was run on the rougher concentrates several months later, once hydrometallurgical testing had advanced to a point where a target concentrate grade could be assigned. After being stored under ambient conditions for this time, the rougher concentrate had become tarnished and proved difficult to re-float, however a concentrate assaying 7.5% sulfur was produced (Bond, 2018).

The third pilot plant project employed a pilot Stage Flotation Reactor (**SFR**) from Woodgrove Technologies (Leon, 2017). This work was initiated based on the success of the technology at Dundee Precious Metals' Chelopech operation in the flotation of pyrite concentrates. It was run in parallel with batch laboratory testing to evaluate the relative SFR performance. SFR, incorporating froth washing, proved to produce higher grade concentrates through very effective rejection of fine gangue, but recoveries were poor, and the technology was only competitive metallurgically when operated with conventional scavenger flotation. A subsequent trade-off study, executed by M3, could not make a case for incorporation of the SFR technology into the feasibility study – however this may warrant further investigation in the value engineering phase of the project.

13.5.7.2 Bulk Concentrate Production

A bulk flotation program was executed at SGS in Burnaby to produce concentrate for pressure oxidation and downstream processing variability testing at SGS in Malaga (Gajo, 2018). In this program, cleaner flotation was used to make concentrates from a total of 14 different feed composites (see Table 13-1). The background to the shipped concentrate composites and their assays is shown in Table 13-9. Estimated recoveries are included but should be used with caution as cleaner flotation was conducted several months after rougher flotation and some cleaner recoveries were poor on the likely tarnished rougher concentrates.

Table 13-9: Bulk Flotation Concentrate Sample Assays for POX Variability Testing

Description	SGS #	Mass	Au (g/t)	Sulfur (%)	CO_3 (%)	Au Recovery To Conc
Yellow Pine Years 0-3 Low Sb	Conc 3	5.1	16.9	8.8	3.9	89
Yellow Pine Years 0-3 High Sb	Conc 4	5.8	12.8	8.1	3.2	74
Yellow Pine Years 4+ Low Sb	Conc 5	9.0	20.8	9.7	4.9	93
Hangar Flats Remote from Meadow Creek Fault (HFO)	Conc 6	10.0	11.9	8.9	2.4	91
Hangar Flats Meadow Creek Fault Zone (HFFZ)	Conc 7	6.3	9.5	7.6	2.6	83
West End High Carbonate, sighter composite (WEBH)	Conc 8	3.7	21.4	16.4	8.9	78
West End High Carbonate, sighter composite (WEBH)	Conc 9	5.9	16.6	11.3	10.8	79
Yellow Pine/Hangar Flats Pilot Sample (5208)	Conc 10	552	11.1	7.6	3.5	92
West End High Carbonate, bulk composite	Conc 11	50.9	17.1	14.7	9.6	79
Hangar Flats/West End blend (HF/WE)	Conc 12	57.7	11.3	8.1	6.7	86
West End Medium Carbonate (WEBM)	Conc 13	1.4	27.3	18	8.0	82
West End Low Carbonate (WEBL)	Conc 14	1.1	19.7	12.6	9.3	80

13.5.8 Flotation Concentrate Characterisation

Several multi-element analyses have been conducted through the various phases of the project. The list of assays provided in Table 13-10 represents the average analyses from locked cycle tests on Yellow Pine and Hangar Flats materials conducted to date on the project.

Table 13-10: Average Multi-element Scans of Yellow Pine and Hangar Flats Sb Concentrates

Element	Yellow Pine	Hangar Flats	Units	Element	Yellow Pine	Hangar Flats	Units	Element	Yellow Pine	Hangar Flats	Units
Au	8.9	4.8	g/t	La	42	<10	ppm	Zn	1816.7	1560.0	ppm
Ag	307	698	g/t	Li	<10	<10	ppm	SiO_2	7.6	4.8	%
As	0.3	0.2	%	Mo	19	<10	ppm	Al_2O_3	1.5	0.8	%
Sb	60.8	54.8	%	Ni	215	130	ppm	CaO	0.4	0.7	%
Fe	2.3	1.1	%	Pb	1580	1390	ppm	Cr_2O_3	0.01	0.01	%
S	26.9	25.5	%	Sc	<5	<5	ppm	Fe_2O_3	3.4	1.2	%
Hg	168	357	ppm	Se	297	62	ppm	K_2O	1.1	0.7	%

Element	Yellow Pine	Hangar Flats	Units	Element	Yellow Pine	Hangar Flats	Units	Element	Yellow Pine	Hangar Flats	Units
Ba	80.0	76.7	ppm	Sn	<50	<50	ppm	MgO	0.1	0.3	%
Be	<5	<5	ppm	Sr	20	43	ppm	MnO	0.01	0.03	%
Bi	< 20	n/a	ppm	Tl	< 30	< 30	ppm	P_2O_5	0.05	0.05	%
Cd	<10	<10	ppm	U	< 40	< 40	ppm	TiO_2	0.08	0.04	%
Co	<10	<10	ppm	W	<50	<50	ppm	V_2O_5	0.002	0.002	%
Cu	280	423	ppm	Y	<5	<5	ppm				

Multi element assays of the bulk sulfide concentrates used for hydrometallurgical testing at SGS Malaga are shown in Table 13-11.

Table 13-11: Analyses of Gold Bearing Sulfide Concentrates for Hydromet Testing

Sample	Deposit	Description	Au, g/t	Fe, %	As, %	Sb, %	S^t, %	Al, %	K, %	Ag, %	Hg, %	CO_3
Con 10	Project Master Composite		11.1	8.6	2.8	0.2	7.0	7.6	5.7	15	4	2.5
Con 3	Yellow Pine	Yr 0-3, low Sb	17.5	8.9	3.4	0.1	7.6	7.7	4.5	18	8	4.8
Con 4	Yellow Pine	Yr 0-3, high Sb	11.9	7.4	2.1	0.4	6.7	7.3	6	17	5	4.2
Con 5	Yellow Pine	Yr 4+, low Sb	20.7	14.3	4.7	0.0	12.4	5.9	4.8	12	3	5.4
Con 6	Hangar Flats	Non-fault zone	9.1	8.4	2.8	0.5	6.2	8.3	5	16	10	3.2
Con 7	Hangar Flats	Fault zone	6.4	10.5	2.1	0.3	9.2	7.8	4.4	15	28	3.6
Con 8	West End	Low CO_3/S	22.0	18.3	4	0.2	17.9	4.3	2.5	25	11	12.5
Con 9	West End	Medium CO_3/S	23.3	15.2	3.3	0.1	12.8	4.6	2.5	18	14	13.5
Con 11	West End	High CO_3/S	11.9	11.5	2.4	0.1	8.7	5.3	3	17	4	13.7
Con 12	HF/WE blend	HF/WE blend	8.0	9.1	2.5	0.5	6.8	6.8	4	16	6	9.0

13.5.9 Flotation Behaviour of Mercury

Mercury assays, on average, 0.7 ppm in Yellow Pine, 1.9 ppm in Hangar Flats and 1.0 ppm in the West End mineable feed. Its content tends to be elevated in the Sb-rich feed.

Where present with antimony, much of the mercury tends to float with the antimony. A balance derived during the PFS indicated that, life-of-mine, 9% of the mercury will report to the Sb concentrate, 55% to the bulk sulfide concentrate reporting to the autoclave and the remaining 36% reporting to the flotation tailings. In the case of West End, some of the mercury in the flotation tailings would be leached. This would mostly report to the activated carbon to ultimately be retorted in the refinery.

13.6 CYANIDATION PROCESS DEVELOPMENT

13.6.1 Overview

Extensive testing has been conducted on process products from Yellow Pine and Hangar Flats to investigate the potential for supplemental gold recovery from both rougher and cleaner flotation tailings. Most of this work has focused on leaching of the tailings stream “as-is” and is described in Section 13.6.2.

The cyanide testing of West End sulfide and transition materials focused solely on flotation tailings and is described in Section 13.6.3.

A variety of approaches to leaching of West End oxide samples was tested, including direct leaching, as well as the leaching both flotation concentrates and tailings from West End oxides (Section 13.6.4).

13.6.2 Hangar Flats and Yellow Pine Process Tailings

A brief program of process optimisation was conducted on the cyanide leaching of flotation tailings from Yellow Pine and Hangar Flats master composites (POL and HFOL). The results are summarised in Table 13-12.

Table 13-12: Effect of Cyanide Conditions on Leaching of Gold from HF and YP Tailings

	Parameters	Yellow Pine	Hangar Flats	Yellow Pine	Hangar Flats
Leach Conditions	Cyanide concentration	0.25 g/L	0.25 g/L	0.1 g/L	0.1 g/L
	Tailings leach time, hours	8	8	16	16
	Cyanide consumption (including initial dose),	0.31	0.31	0.18	0.18
	Lime consumption (including initial dose), kg/t	0.74	0.65	0.74	0.65
Gold Extraction	Gold distribution to rougher tailings (based on mill feed)	7.2%	4.5%	7.2%	4.5%
	Leach extraction of gold from rougher tailings	9.5%	9.2%	9.5%	9.2%
	Gold distribution to cleaner tailings (based on mill feed)	1.8%	1.9%	1.8%	1.9%
	Leach extraction of gold from cleaner tailings	12.1%	13.5%	12.1%	13.5%

Extractions were quite low and independent of cyanide concentration down to 0.1 g/L.

13.6.3 Leaching of West End Sulfide and Transition Flotation Tailings

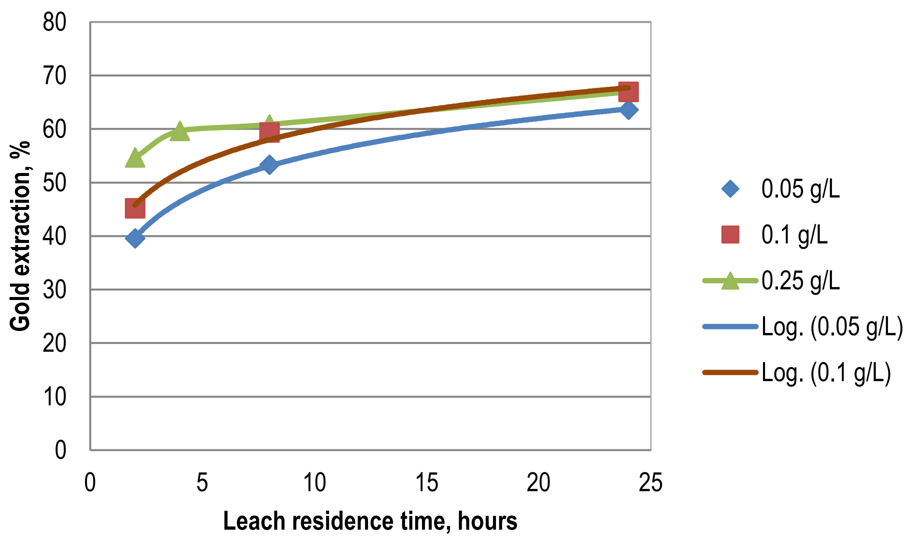
Results from a factorial design study on the effect of cyanide concentration and pulp density on the leaching of West End sulfide flotation tailings, are shown in Table 13-13. Gold extraction was largely independent of both cyanide dose and pulp density within the range tested.

Table 13-13: Optimization of Cyanide Leaching of West End Sulfide Tailings

Test ID	Pulp Density (%)	Leach Time (hrs)	NaCN Conc. (g/L)	NaCN Dosage (kg/t)	pH	NaCN		CaO		Au Grade			Au Extraction (%)
						add'n (kg/t)	Cons. (kg/t)	add'n (kg/t)	Cons. (kg/t)	Residue (g/t)	Calc. Head (g/t)	Direct Head (g/t)	
WES-LCT1-L1	35%	48	0.05	0.09	10.5 - 11	0.18	0.08	0.77	0.71	0.17	0.35	0.39	51.1
WES-LCT1-L2	40%	48	0.07	0.11	10.5 - 11	0.16	0.07	0.85	0.80	0.17	0.35	0.39	51.1
WES-LCT1-L3	45%	48	0.08	0.10	10.5 - 11	0.18	0.08	0.76	0.72	0.17	0.36	0.39	53.3
WES-LCT1-L4	35%	48	0.14	0.25	10.5 - 11	0.30	0.06	0.83	0.75	0.18	0.41	0.39	56.0
WES-LCT1-L5	40%	48	0.17	0.26	10.5 - 11	0.32	0.08	0.80	0.74	0.17	0.40	0.39	57.8
WES-LCT1-L6	45%	48	0.21	0.25	10.5 - 11	0.34	0.08	0.78	0.75	0.16	0.36	0.39	55.1
WES-LCT1-L7	35%	48	0.27	0.50	10.5 - 11	0.56	0.08	0.79	0.73	0.16	0.41	0.39	61.2
WES-LCT1-L8	40%	48	0.33	0.50	10.5 - 11	0.59	0.08	0.79	0.74	0.17	0.39	0.39	56.1
WES-LCT1-L9	45%	48	0.41	0.50	10.5 - 11	0.62	0.12	0.80	0.75	0.17	0.36	0.39	53.3

The effect of cyanide concentration on the kinetics of gold leaching from West End flotation tailings was tested, as is shown on Figure 13-5. A cyanide concentration as low as 0.1 g/L appears to be adequate for leaching of the West End transition material, the kinetics of which appear to be quite slow with a “tail” to the leach curve extending to at least 24 hours.

Figure 13-5: Effect of Cyanide Concentration on Gold Extraction from West End Comp Flotation Tailings



13.6.4 Whole Ore Leaching of West End Oxides

An initial program of tests was conducted on West End oxide material ground to 80% passing 75 microns (Table 13-14). This round of tests evaluated pulp density (at 35%, 40%, and 45% solids) and cyanide dose effects (at 0.26, 0.5 and 0.75 kg/t). All tests were run for 48 hours with no kinetic samples. There was no consistent effect of pulp density or statistically significant effect of cyanide dose on final recovery.

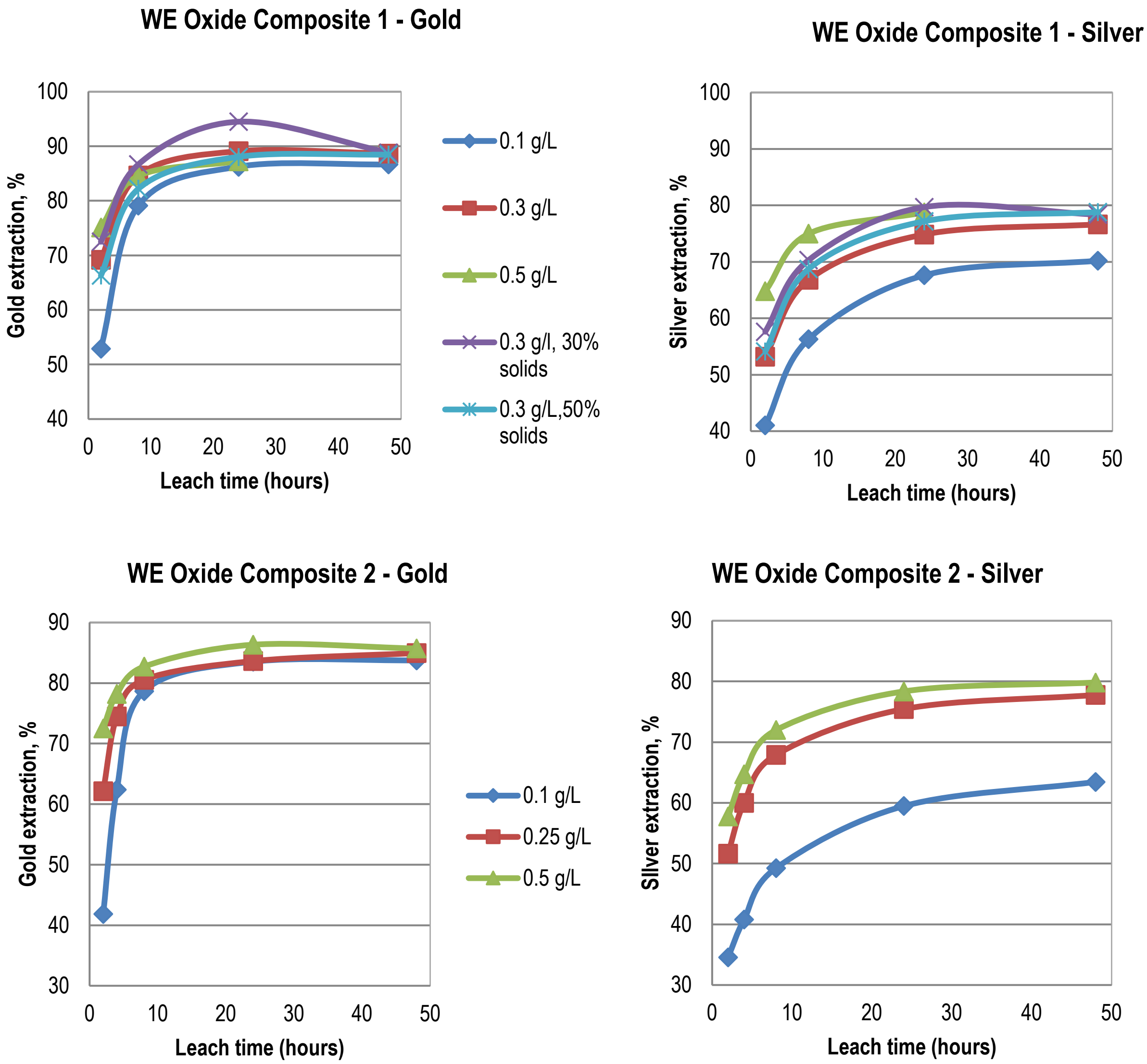
Table 13-14: Effect of Pulp Density and CN dose on Whole Ore Leaching of West End Oxides

Test ID	Pulp Density (%)	Leach Time (hrs)	NaCN Conc. (g/L)	NaCN Dosage (kg/t)	pH	NaCN		CaO		Au Grade			Au Extraction (%)
						add'n (kg/t)	Cons. (kg/t)	add'n (kg/t)	Cons. (kg/t)	Residue (g/t)	Calc. Head (g/t)	Direct Head (g/t)	
WEO-L1	35%	48	0.14	0.26	10.5 - 11	0.26	0.00	0.78	0.68	0.23	1.02	0.99	77.4
WEO-L2	40%	48	0.17	0.26	10.5 - 11	0.30	0.00	0.77	0.69	0.26	1.02	0.99	74.5
WEO-L3	45%	48	0.21	0.26	10.5 - 11	0.26	0.00	0.85	0.76	0.24	0.93	0.99	74.2
WEO-L4	35%	48	0.27	0.50	10.5 - 11	0.50	0.02	0.81	0.70	0.22	1.01	0.99	78.2
WEO-L5	40%	48	0.33	0.50	10.5 - 11	0.53	0.02	0.83	0.73	0.22	1.00	0.99	77.9
WEO-L6	45%	48	0.41	0.50	10.5 - 11	0.50	0.00	0.79	0.72	0.22	1.00	0.99	78.0
WEO-L7	35%	48	0.40	0.74	10.5 - 11	0.75	0.00	0.78	0.67	0.23	0.99	0.99	76.8
WEO-L8	40%	48	0.50	0.75	10.5 - 11	0.81	0.00	0.85	0.73	0.22	1.00	0.99	78.0
WEO-L9	45%	48	0.61	0.75	10.5 - 11	0.83	0.06	0.84	0.75	0.23	1.00	0.99	76.9

However, the above program provided no insight into the kinetics of the leach, nor the effect of cyanide concentration on kinetics. Two more composites of West End oxide material were leached in subsequent kinetic tests, following grinding to 80% passing about 80 microns, at 40% solids using cyanide doses ranging from 0.1 to 0.5 g/L. One composite was also leached using 0.3 g/L cyanide at 30% and 50% solids. All tests were run at pH 10.5-11.0.

Invariably, the gold leach kinetics were fast, being complete within 24 hours, and about 96% complete after 8 hours. Leaching with 0.1 g/L cyanide slowed the leach somewhat and led to slightly poorer extractions after 24 hours. Silver leached more slowly than gold and was more sensitive to cyanide concentration and pulp density.

Figure 13-6: Gold and Silver Metallurgy from Whole Ore Leaching of Two West End Composites



Occasionally, with free milling ores, it pays to float the floatable gold and leach the concentrate and tailings separately. Further, as even the West End oxide ores have a small refractory component, if this was floated to a specific concentrate, it may pay to oxidise or regrind it before leaching. Accordingly, kinetic products from oxide flotation were leached to evaluate recoveries as a function of floatability. All products leached quite well. None of the leach residues were rich enough to warrant pre-oxidation suggesting that none of the products contain sufficient refractory gold to be oxidised, and the high recoveries from cleaner concentrates 1, 2 and 3 suggest regrinding would offer little benefit.

The weighted average leach recovery was the same as from whole ore leaching, so no case could be made of running the float on West End oxide material.

Table 13-15: Leaching of Flotation Products from West End Oxides

Flotation Product	Gold Assay, g/t		Extraction %
	Feed	Residue	
Cleaner 1	4.01	0.61	84.7
Cleaner 2	4.44	0.60	86.4
Cleaner 3	3.95	0.40	89.9
Cleaner tailings	1.57	0.17	89.1
Rougher Tailings	0.83	0.17	79.3

13.7 GEOMETALLURGY AND VARIABILITY TESTING

While basic geometallurgical principals were observed in this study, the study did not aim to "geometallurgically enable" the resource model. This was deemed too complex for a project with three different deposits, plus the Bradley Tailings resource, and at least two significantly different feed types per deposit. Once the lithotypes were included, the potential existed for 15-20 geometallurgical units for just gold recovery, each requiring recovery algorithms would have been needed, requiring the testing of hundreds of variability samples.

However, a geometallurgy-based diagnostic program was conducted to evaluate the metallurgical response from key lithotypes to identify if there were sizeable differences in response that would require this more complex approach to variability testing (Hall, 2018).

So the geometallurgy and variability program comprised a sequence of steps including:

- Identification of geometallurgical samples by project geologists, comprising discrete, loggable lithologies in each deposit, and the collection of multiple samples for testing, to represent each lithotype in each deposit.
- Geometallurgical (Diagnostic) testing of each sample so building populations of data describing the flotation recovery related to each lithology in each deposit.
- Concurrent mining sequence and geological evaluation of the lithologies to (a) identify if they would likely be mined as discrete blocks and (b) if so, how the mill feed mix of each lithotype would vary through the life of the mine. This was based on the PFS mine model) at the time, but reasonably resembles the FS mine model.
- Based on this analysis, design of samples for variability testing.

This program started with the geometallurgical (diagnostic) studies on the following discrete lithology samples (Table 13-16):

Table 13-16: Lithologies tested in the Diagnostic Geometallurgical Program

Lithology	Yellow Pine	Hangar Flats	West End
Alaskite	✓	✓	
Gouge	✓	✓	✓
Quartz-Monzonite	✓	✓	
Granite	✓		
Breccia	✓	✓	✓
Mix	✓		
Quartz-schist			✓
Granodiorite			✓
Calc-silicate			✓
Oxide			✓

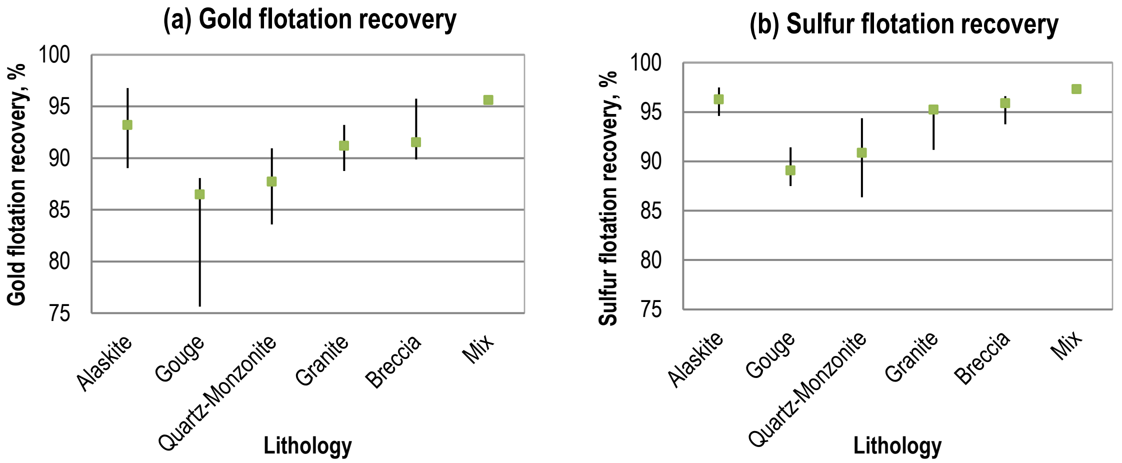
13.7.1 Geometallurgical Studies

A generic "mini-flotation" procedure was applied to all diagnostic samples to assess flotation recovery characteristics. Note the "mini-flotation" procedure was designed as a standard diagnostic mini-test. It was not optimal, so recoveries tended to be lower than expected project wide.

For Yellow Pine, this was applied to 32 samples, representing six different lithotypes, or an average of about 5 samples per lithotype. Recoveries are shown on Figure 13-7. Granite, breccia and alaskite yielded the highest sulfur and gold recoveries with gouge and quartz-monzonite yielding the worst. However, the differences were not substantial and were not deemed statistically significant given the internal variability within each lithotype.

Figure 13-7: Effect of Lithotype on Gold and Sulfur Recovery from Yellow Pine Materials

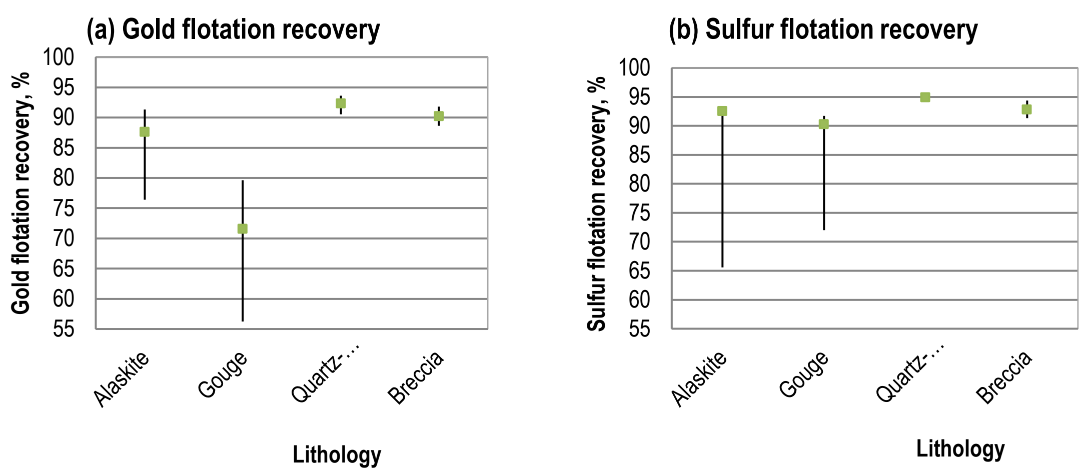
Top and bottom whiskers represent the 20th and 80th percentiles



For Hangar Flats four lithotypes were tested (Figure 13-8). Gouge material was again the poorest floating, however unlike with Yellow Pine, quartz-monzonite was now the best. Again, there was significant overlap in recovery data.

Figure 13-8: Effect of Lithotype on Flotation Metallurgy of Hangar Flats Materials

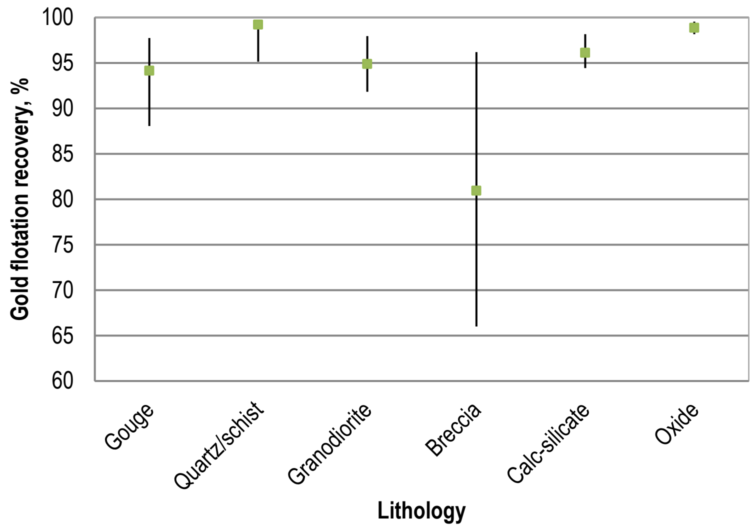
Top and bottom whiskers represent the 20th and 80th percentiles



West End has also been analyzed in a similar fashion, although it is the discrimination of oxide/transition/sulfide mineralization (determined through the cyanide soluble gold measurements) that truly drives variability in West End metallurgy.

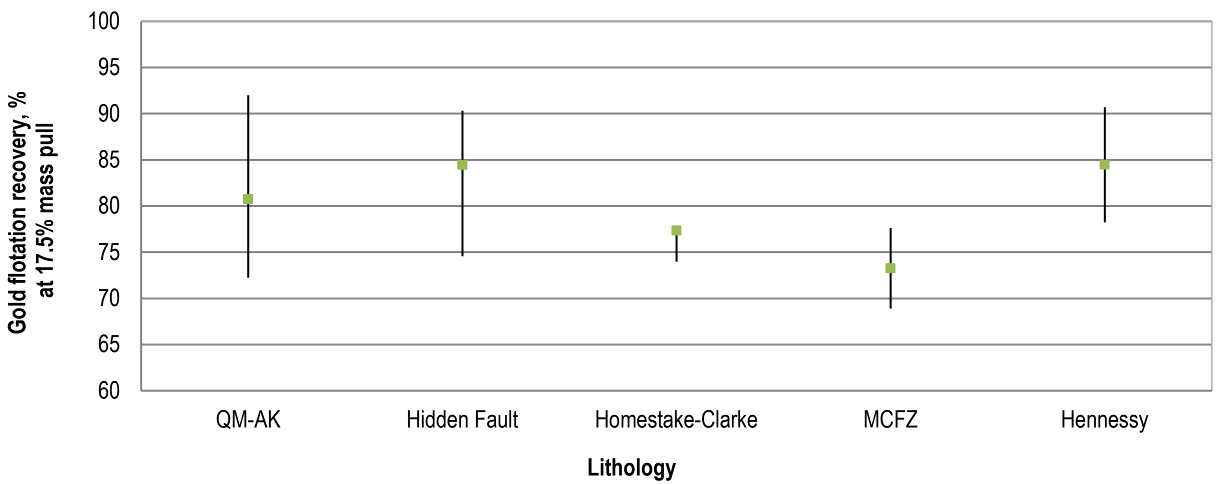
Except for breccia, sulfide recovery from all lithotypes was high with no statistically significant differences between the lithotypes (Figure 13-9). The over-riding driver behind gold flotation recovery is the refractoriness of the gold so this has not been included in this analysis.

Figure 13-9: Effect of Lithotype on West End Sulfide Flotation Recovery



Using project-wide data, the impact of proximity to key geological structures including the Meadow Creek, Hidden and Hennessy Faults, as well as the Clarke Tunnel was also evaluated, compared with quartz monzonite/alaskite material distal from these main structures. Material from the Meadow Creek Fault Zone (**MCFZ**) tended to yield poorer metallurgy, while that from the Hidden and Hennessy Faults yielded quite good metallurgy (Figure 13-10).

Figure 13-10: Effect of Location of Sample within Geological Structure on Gold Recovery



In summary, lithology does not appear to be a major driver behind flotation performance in any of the deposits, with the possible exception of gouge material. Accordingly, while recognizing the potential for recovery differences by lithotype in the design of the variability samples was important, there was no need to build individual discrete algorithms for each lithotype.

13.7.2 Variability Studies

The design of the variability composites reflected (1) the lithological drivers behind metallurgy as described in Section 13.7.1 and (2) the range in lithotype blends expected to be mined at any one time through the life of the mine. In total, the variability program studied 44 major composites, each being subjected to flotation and tailings leaching tests.

13.7.2.1 Yellow Pine

Alaskite **(AK)** tends to occur as dykes and sills, often as swarms in the quartz monzonite **(QM)**, while granites also tend not to occur as distinct mineable lithotypes. They would likely be mined as a blend (QM-AK) so were treated this way in variability composite design. This lithotype will be a major component in the mill feed mix, ranging from 25-75% of the tonnage from Yellow Pine (Figure 13-11), the range being reflected in their content in the Yellow Pine variability samples (Figure 13-12).

The fault materials, however, were seen as more discrete. The MCFZ would be the primary source of gouge material, likely the worst actor in flotation. In mining, the content of MCFZ material in the mill feed for a given monthly production period is expected to range from 0-20%, so most variability samples were designed reflecting this. Hidden Fault material, suspected to a different acting fault material, will form at least a minor component in the ore mix throughout most of the Yellow Pine pit life so is a component in most of the variability samples (Figure 13-12). One sample, YP-316 contained 100% MCFZ material and was used as an end member sample. This was designed to push variability to the limit, though in practice such a blend is extremely unlikely for a prolonged period of time. The data should therefore be taken as a predictor of plant metallurgy with caution.

The Homestake-Clark Tunnel zone is a major source of tonnage in Years 2 and 3, likely comprising up to 50% of the tonnage, so comprising a major part of 3 samples. Breccia from the Hennessy fault area is a minor component of the feed mix and some variability samples.

These composites, therefore, mostly spanned the likely normal compositions of low Sb materials to be seen in the mill, plus some “worst case scenario” end members. Sample YP-309 contained a high proportion of partially oxidised transition material from the Homestake–Clarke Tunnel area, where 49% of the gold is cyanide soluble, and is a transition end-member sample not representing significant tonnage in the deposit.

In addition, five higher Sb samples were included, from the MCFZ, Hidden Fault and Homestake-Clarke Tunnel structures, as well as one end member very high Sb sample (Figure 13-12).

Figure 13-11: Composition of Mill Feed Material Mined from Yellow Pine Based on PFS Mine Model

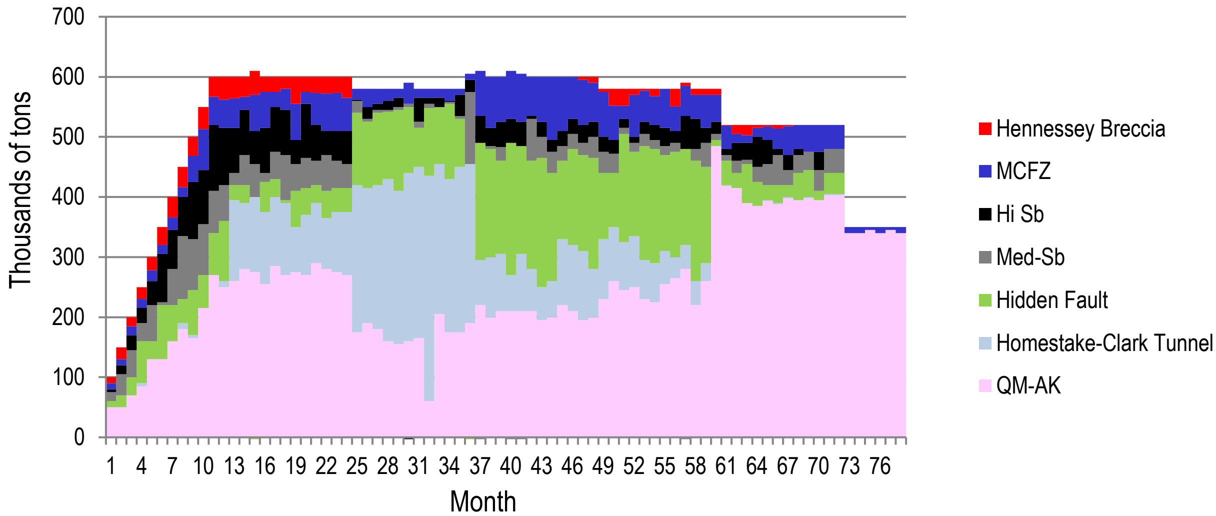
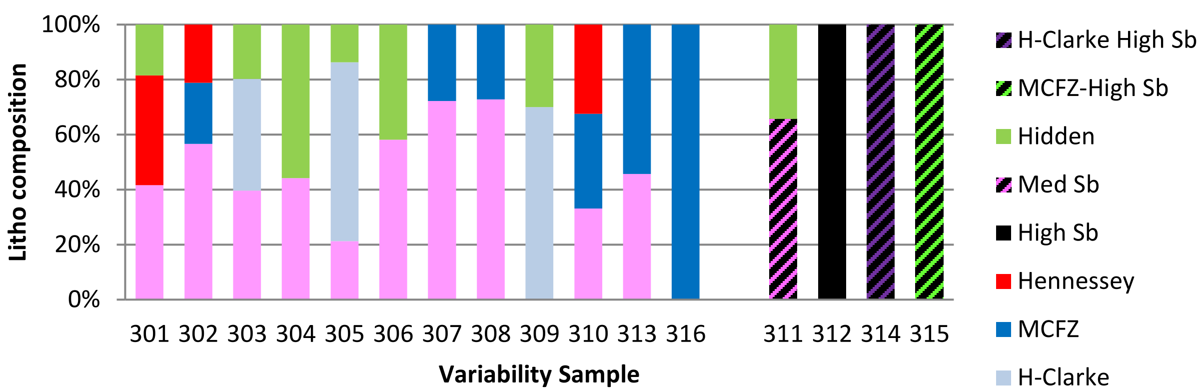


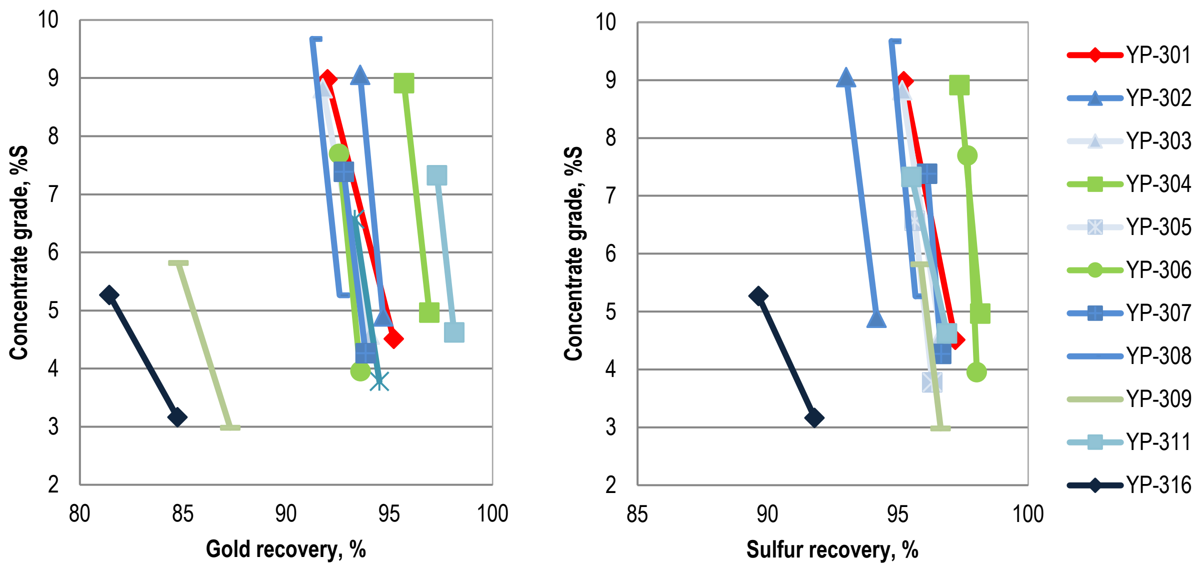
Figure 13-12: Composition of Yellow Pine Variability Samples



Gold flotation recoveries as a function of sulfur grade are shown in Figure 13-13 below, with the rougher and first cleaner points shown for each composite. The samples are colour-coded for the dominant non-QM-AK component, being dark blue for the Meadow Creek Fault Zone (MCFZ), green for Hidden Fault, the Hennessy-rich sample is in red, Homestake-Clark (H-Clark) is shown in pale blue.

The data are clustered together, in a range of 92.3-96.5% to a concentrate assaying 6.5% sulfur, for all samples except the end members YP-309 and YP-316. Recovery from the YP-309 transition composite was poorer because of the presence of ultra-fine discrete gold (as demonstrated by a 46% leach extraction of gold from the Homestake-Clark Tunnel component in this sample), while YP-316 was designed to be a “worst case” scenario sample containing 100% Meadow Creek Fault material, recovered just 81% of the gold to a 5% sulfur concentrate. Excluding these two samples, the average first cleaner gold recovery was 93%, and the sulfur recovery 96.5% to a concentrate averaging 8% sulfur.

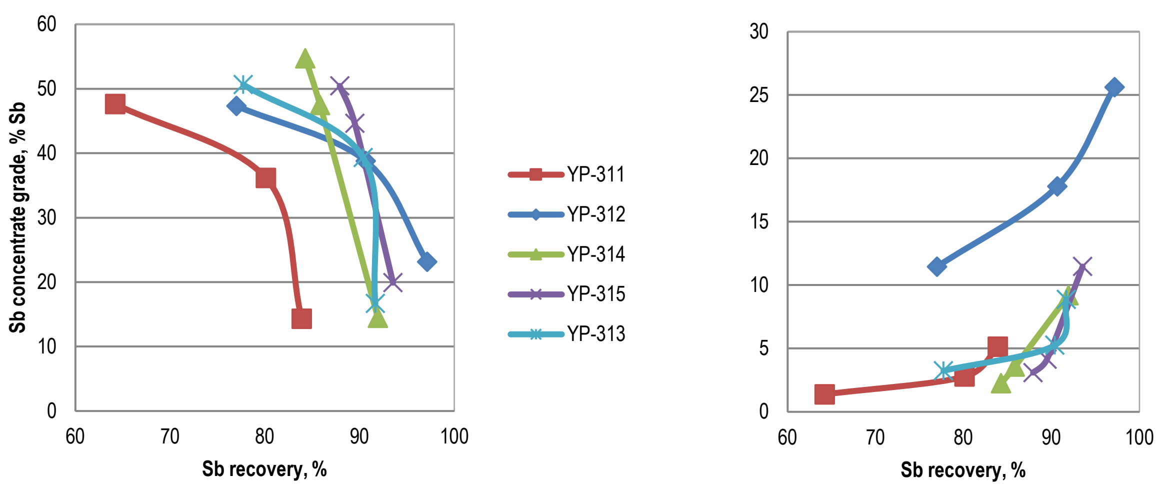
Figure 13-13: Gold Metallurgy from Yellow Pine Low Sb Variability Composites



Head grades in the four Sb-rich samples ranged from 0.18% to 1.15% Sb. Antimony flotation metallurgy was good for all samples except the lowest grade YP-311 (head grade 0.18% Sb).

With such a wide variation in head grades, reagent doses were probably not optimal, which may have led to the lower Sb concentrate grades seen in the tests. However, batch recoveries to 40% Sb tended to be around 90%, validating the metallurgical data on the master composites. It is the opinion of the QP, that laboratory tests often fail to match the plant for concentrate grade, where the latter is using column technology which is very difficult to simulate in the laboratory. Gold loss to the antimony concentrate averaged 2.5% except for the high Sb outlier sample YP-312 (Figure 13-14).

Figure 13-14: Antimony Flotation from Yellow Pine Variability Composites



Gold recoveries to the gold flotation concentrate were calculated from the Sb-Au flotation tests assuming all Sb middlings would report to the gold concentrate. They are shown in Table 13-17 below, averaging 92% (excluding the high Sb end member):

Table 13-17: Gold Recovery from High Sb Variability Samples

Sample	Gold Recovery, %
YP-311	95.1
YP-312	82.7
YP-313	91.7
YP-314	92.8
YP-315	88.6

The rougher and cleaner flotation tailings from all variability samples were leached to allow for a more robust assessment of the potential for leaching of both the rougher and cleaner flotation tailings in the plant. On average, only 0.8% of the gold in the mill feed was recoverable by leaching the tailings. Excluding the transition sample this drops to 0.4% of the gold, equivalent to 0.01 g/t gold in the leach feed. Similarly, leaching the cleaner tailings added 0.2% gold recovery, dropping to 0.1% gold recovery when the transition sample is excluded.

More detailed gold metallurgical variability data for Yellow Pine are tabulated in Table 13-18.

Table 13-18: Yellow Pine Variability Test Results Summary

Sample	Flotation Test	Conc Sulfur Grade (%)	Gold Flotation Recovery (%)			Gold Leach Extraction (%) Based on Leach Feed		Gold Leach Extraction (%) Based on Float Feed		Gold in Feed and Recovered by Flotation and Leaching (g/t)			
			Cleaner Conc	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Head	Floated	Leached Rougher Tails	Leached Cleaner Tails
YP-301	VF-10RR	9.0	92.0	4.8	3.2	8.1	8.8	0.4	0.3	2.59	2.38	0.01	0.01
YP-302	VF-11	9.1	93.6	5.3	1.1	5.7	8.3	0.3	0.1	3.01	2.82	0.01	0.00
YP-303	VF-12R	8.8	91.8	6	2.3	8.9	11.8	0.5	0.3	3.33	3.06	0.02	0.01
YP-304	VF-13RR	8.9	95.7	3.1	1.2	3.1	8.7	0.1	0.1	2.20	2.11	0.00	0.00
YP-305	VF-14	6.6	93.3	5.5	1.2	2.2	7.3	0.1	0.1	1.33	1.24	0.00	0.00
YP-306	VF-15	7.7	92.6	6.4	1.0	7.3	7.2	0.5	0.1	0.87	0.81	0.00	0.00
YP-307	VF-16	7.4	92.8	6.2	1.1	1.6	6.2	0.1	0.1	0.99	0.92	0.00	0.00
YP-308	VF-17	9.7	91.3	7.4	1.3	7.5	7.7	0.6	0.1	1.18	1.08	0.01	0.00
YP-309	VF-18	5.8	84.8	12.7	2.5	62.0	64.4	7.9	1.6	1.94	1.65	0.15	0.03
YP-310	VF-19	8.0	92.4	6.1	1.5	5.1	8.5	0.3	0.1	2.84	2.62	0.01	0.00
YP-311	VF-20R*	8.1	95.1	2.5	1.0	8.6	4.8	0.2	0.0	2.78	2.64	0.01	0.00
YP-311	VF-38RR	7.3	95.5	3.1	1.4	13.7	4.4	0.4	0.1	2.78	2.65	0.01	0.00
YP-312	VF-21RR*	6.4	82.7	4.7	1.2	6.8	13.0	0.3	0.2	1.84	1.52	0.01	0.00
YP-313	VF-22R*	7.6	91.7	3.7	1.4	12.5	8.0	0.5	0.1	2.72	2.49	0.01	0.00
YP-314	VF-23R*	5.5	94.2	3.6	1.4	3.8	6.1	0.1	0.1	2.08	1.96	0.00	0.00
YP-315	VF-24R*	6.8	88.6	6.3	2.0	7.6	5.1	0.5	0.1	2.15	1.90	0.01	0.00
YP-316	VF-25R*	5.3	81.4	15.3	3.3	10.2	16.7	1.6	0.6	0.73	0.59	0.01	0.00
	Averages*	**7.5**	**91.8**	**5.4**	**1.7**	**10.3**	**11.6**	**0.8**	**0.2**	**2.08**	**1.91**	**0.02**	**0.00**

**Flotation numbers do not add up to 100% due to gold losses to Sb concentrate*

13.7.2.2 Hangar Flats

The Hangar Flats variability composites were prepared to reflect different expected blends of Hangar Flats materials in the mill feed, as depicted on Figure 13-15. Consistent with the range of monthly mill feed mixes in the mine plan, the blends were mostly comprised of Quartz-Monzonite and Alaskite-Granite with smaller amounts of Breccia and gouge material. The latter may be somewhat over-represented in the variability samples, which in effect may add a degree of conservatism to the metallurgical data.

Figure 13-15: Proportion of Low Sb Hangar Flats Material by Lithotype in PFS Mine Plan

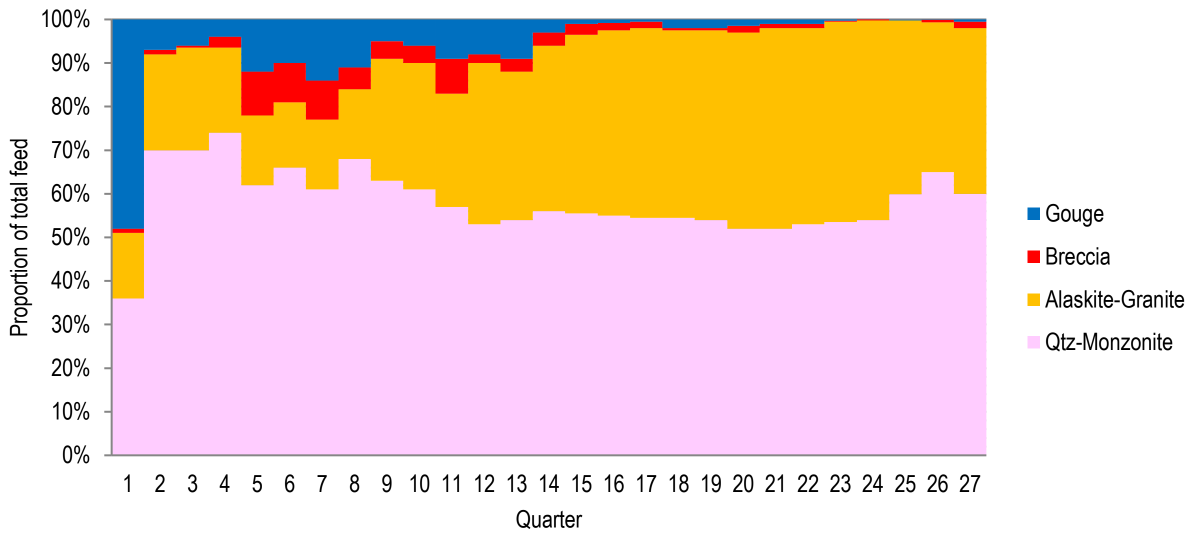
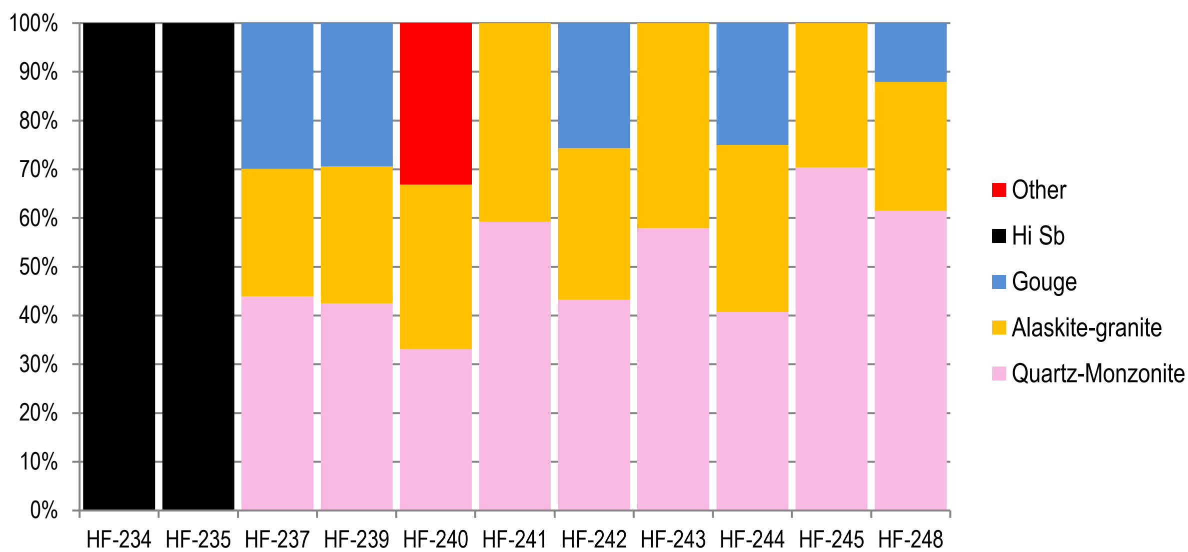
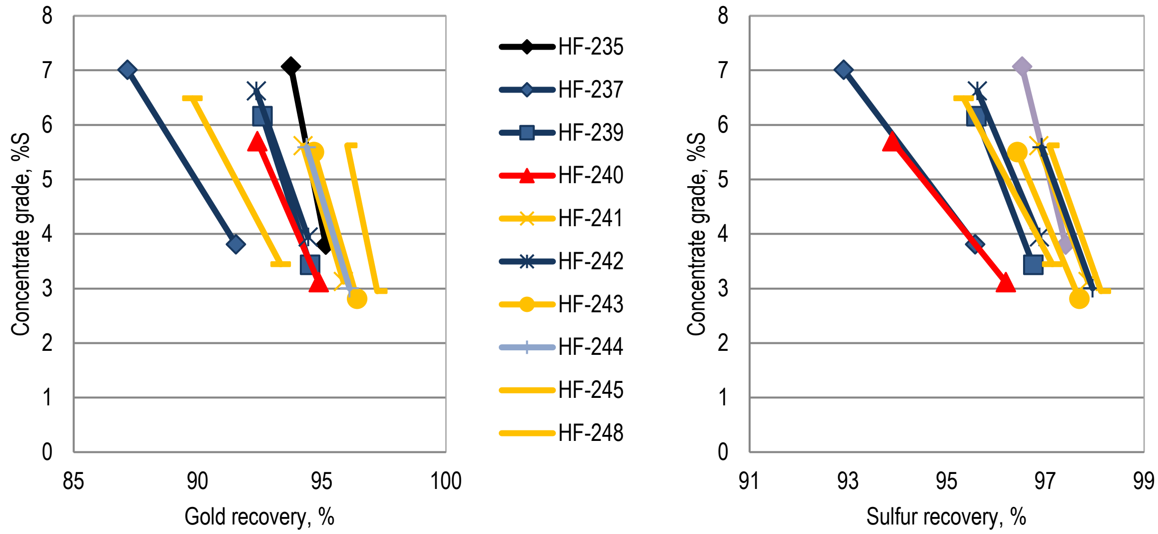


Figure 13-16: Approximate Mix of Lithotypes in Low Sb Variability Samples



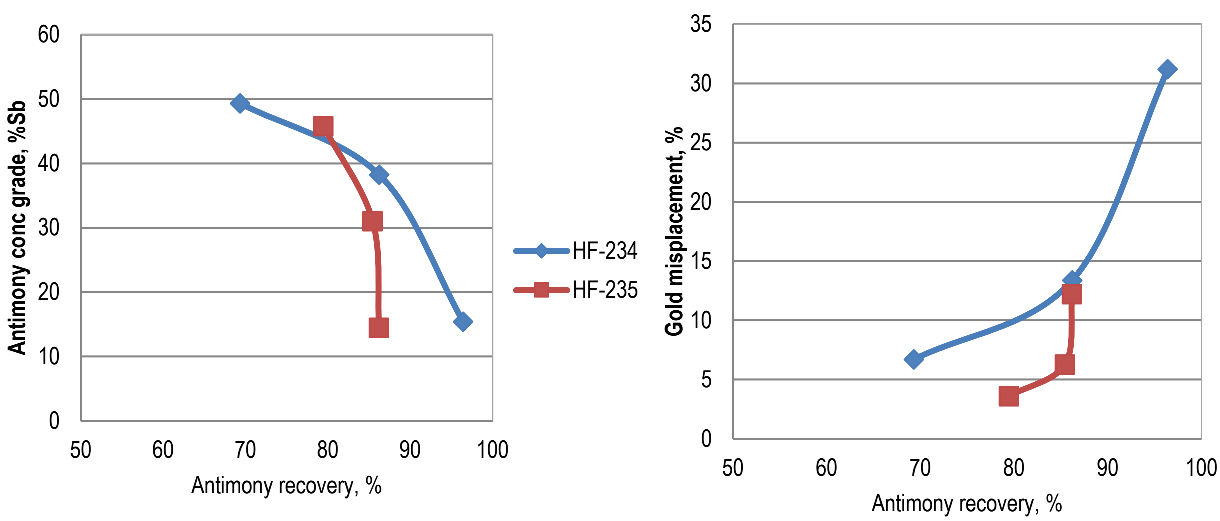
The flotation variability data is shown in Figure 13-17. For the low Sb samples, gold recovery to the first cleaner concentrate, assaying 6.1% sulfur, averaged 92.7%. Silver recovery averaged 89.6%. HF-235 was floated as a high-and low-Sb sample in this study. There was no clear trend in recovery between the alaskite-granite rich material and the gouge-rich blends.

Figure 13-17: Flotation Metallurgy of Hangar Flats Low Sb Variability Samples



Two antimony-rich samples were tested. As with the Yellow Pine variability samples, antimony concentrate grades missed expectation. However antimony recoveries were again quite good given the cursory nature of variability testing. Recoveries to concentrates assaying 30-40% antimony averaged 86%. Gold misplacement was quite high at close to 5 percent (Figure 13-18).

Figure 13-18: Antimony Flotation Metallurgy from Hangar Flats High Sb Samples



Gold recovery to the first cleaner concentrate from the two antimony-rich samples averaged 90.0%.

The rougher and cleaner tailings from all the Hangar Flats variability samples were leached to investigate the potential for subsequent gold recovery by cyanidation. As with the Yellow Pine samples, gold recoveries were poor, at 0.4% for the rougher tailings and 0.2% for the cleaner tailings.

More detailed gold metallurgical variability data for Yellow Pine are tabulated in Table 13-19.

Table 13-19: Gold Recovery from Hangar Flats Variability Samples by Flotation and Tailings Leaching

Sample	Flotation Test	Conc Sulfur Grade (%)	Gold Flotation Recovery (%)			Gold Leach Extraction (%)				Gold in Feed and Recovered by Flotation and Leaching (g/t)			
						Based on Leach Feed		Based on Float Feed					
			Cleaner Conc	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Head	Floated	Leached Rougher Tails	Leached Cleaner Tails
HF-234	VF-26RR*	7.4	88.7	3.4	1.2	7.5	5.7	0.3	0.1	2.32	2.06	0.01	0.00
HF-235	VF-27R*	7.6	91.3	3.4	1.7	17.8	5.2	0.6	0.1	1.41	1.29	0.01	0.00
HF-237	VF-28R	7.0	87.2	8.5	4.4	5.8	6.8	0.5	0.3	1.32	1.15	0.01	0.00
HF-239	VF-29	6.2	92.6	5.5	1.9	8.3	7.7	0.5	0.1	2.24	2.07	0.01	0.00
HF-240	VF-30	5.7	92.4	5.1	2.5	1.4	6.4	0.1	0.2	1.62	1.50	0.00	0.00
HF-241	VF-31	5.6	94.2	4.1	1.6	7.1	11.1	0.3	0.2	1.62	1.53	0.00	0.00
HF-242	VF-32	6.6	92.4	5.6	2.1	10.7	10.7	0.6	0.2	1.49	1.38	0.01	0.00
HF-243	VF-33	5.5	94.7	3.6	1.7	9.6	11.0	0.3	0.2	2.05	1.94	0.01	0.00
HF-244	VF-34	5.6	94.4	3.9	1.7	17.2	14.0	0.7	0.2	1.48	1.40	0.01	0.00
HF-245	VF-35	5.6	96.0	2.8	1.2	18.9	13.5	0.5	0.2	1.30	1.25	0.01	0.00
HF-248	VF-36	6.5	89.8	6.7	3.6	8.2	4.2	0.5	0.2	1.30	1.17	0.01	0.00
HF-235	VF-37	7.1	93.8	4.8	1.4	3.6	64.5	0.2	0.9	1.45	1.36	0.00	0.01
	Averages*	**6.4**	**92.3**	**4.8**	**2.1**	**9.7**	**13.4**	**0.4**	**0.2**	**1.63**	**1.51**	**0.01**	**0.00**
**Flotation numbers do not add up to 100% due to gold losses to Sb concentrate*													

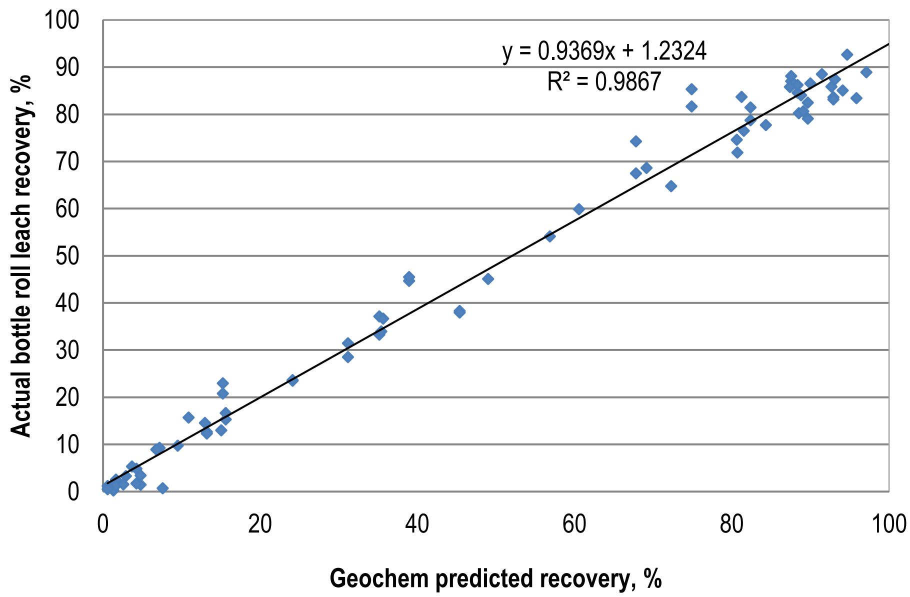
13.7.2.3 West End

Variability in West End metallurgy is primarily driven by the refractoriness of the contained gold. Lithological factors are less influential although samples have been split into the carbonate, calc-silicate, schist and fault zone lithotypes.

13.7.2.3.1 Leaching

The leachability of gold contained in a sample is best predicted using a geochemical "cyanide soluble" gold assay, and in the PFS, the cyanide soluble assay procedure that best predicted bottle roll performance was identified. This was checked again during the FS, to ensure the link between the geochemical test and actual leach performance continued to stand scrutiny using a different dataset. The resulting correlation is shown on Figure 13-19.

Figure 13-19: Correlation of Geochem Inferred Recovery and Actual Leach Recovery



The regression was very similar to that determined in the PFS.

13.7.2.4 Flotation

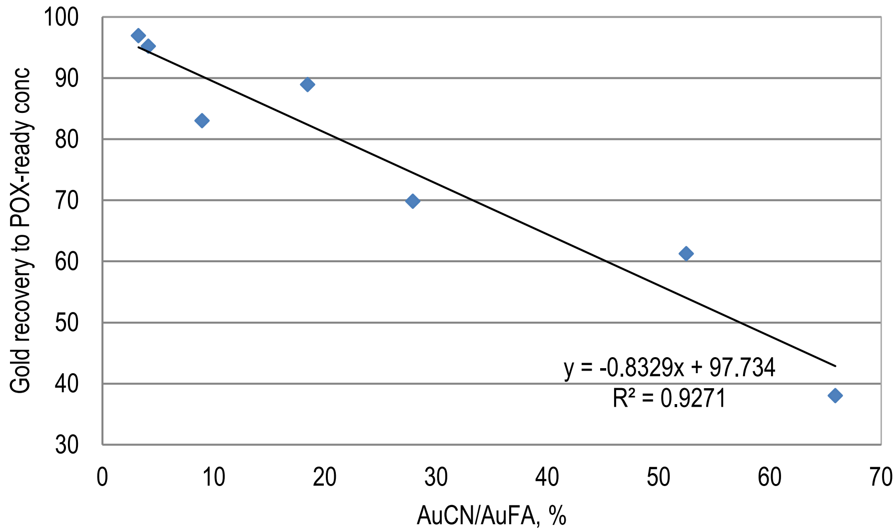
The FS variability study also tested the response of a much smaller suite of variability composites to the overall mineral processing flowsheet (Table 13-20):

Table 13-20: West End Variability Samples

Comp #	Oxidation	Lithotype
WE-401	Sulfide	Schist
WE-402	Sulfide	Calc-silicate
WE-403	Sulfide	Carbonate
WE-404	Transition	Schist
WE-405	Transition	Calc-silicate
WE-406	Transition	Carbonate
WE-407	Transition	West End Fault Zone
WE-408	Oxide	Schist
WE-409	Oxide	Calc-silicate
WE-410	Oxide	West End Fault Zone

The oxide samples all assayed less than 0.1% sulfur, and all failed to make a POX-ready concentrate as defined for this project, using the flowsheet as developed. For the sulfide and transition samples, the recovery of gold to a POX-ready concentrate was linked closely to the cyanide soluble gold content in the sample:

Figure 13-20: Flotation Recovery of Gold vs CN Soluble Gold Assay

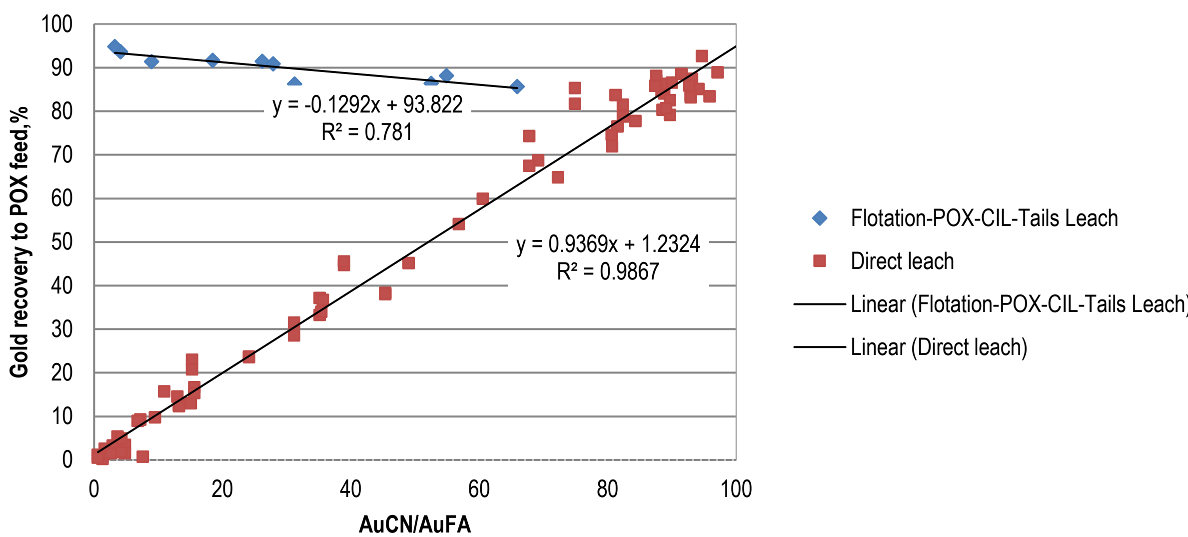


Although the data population is limited, the link between AuCN/AuFA and gold flotation recovery tracks that from the PFS well, so helping to add confidence to the relationship.

Leaching of the flotation tailings added significant gold recovery to flotation alone from the transition samples, bringing gold recoveries to over 85%. Superimposing the float-leach recoveries on the whole ore leach data from Figure 13-19 provides some insights into what materials are best floated, pre-oxidized and leached, and what are best leached directly although the much higher operating costs associated with the full integrated circuits means to a point lower recoveries by direct leaching alone can be more economic.

The point at which flotation and pre-oxidation becomes economic is probably in the range of up and reasonably close to 75% AuCN/AuFA.

Figure 13-21: West End Float-POX-CIL-Tailings Leach and Direct Leach vs CN Soluble Gold Content



The flotation and leach metallurgy from the West End and West End/Hangar Flats variability samples are shown in Table 13-21.

Table 13-21: Hangar Flats and West End Variability Test Results Summary

Sample	Flotation Test	Conc Sulfur Grade (%)	Gold Flotation Recovery (%)			Gold Leach Extraction (%)				Gold in Feed and Recovered by Flotation and Leaching (g/t)			
						Based on Leach Feed		Based on Float Feed					
			Cleaner Conc	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Rougher Tailings	Cleaner Tailings	Head	Floated	Leached Rougher Tails	Leached Cleaner Tails
WE-401	VF-39	7.4	95.4	3.7	0.9	24.6	24.6	0.9	0.2	1.25	1.19	0.01	0.00
WE-402	VF-40	6.6	91.4	5.6	3.0	31.8	31.8	1.8	1.0	3.45	3.16	0.06	0.03
WE-403	VF-41	8.0	97.1	1.9	1.0	21.8	21.8	0.4	0.2	1.28	1.24	0.01	0.00
WE-404	VF-42	7.0	92.1	6.8	1.2	45.6	45.6	3.1	0.5	0.80	0.74	0.02	0.00
WE-405	VF-43	6.4	77.2	18.1	5.0	70.9	70.9	12.8	3.5	1.28	0.99	0.16	0.05
WE-406	VF-44	5.6	48.3	45.3	6.4	75.9	75.9	34.4	4.9	0.92	0.44	0.31	0.04
WE-407	VF-45	6.6	63.1	30.3	6.6	68.8	68.8	20.9	4.5	1.12	0.71	0.23	0.05
WE-408	-	-	-	-	-	84.5	-	84.5	-	1.47	-	1.24	-
WE-409	-	-	-	-	-	85.3	-	85.3	-	1.07	-	0.91	-
WE-410	-	-	-	-	-	86.1	-	86.1	-	2.38	-	2.05	-
	Averages	**6.8**	**83.8**	**12.9**	**3.3**			**8.1**	**1.8**				
West End-Hangar Flats Blend													
Yr 6	VF-49	6.9	87.6	8.7	3.7	41.7	41.7	3.6	1.6	1.40	1.23	0.05	0.02
Yr 7	VF-50	6.6	91.6	5.8	2.6	36.4	36.4	2.1	0.9	1.65	1.51	0.04	0.02
Yr 8	VF-51	5.9	91.0	6.5	2.5	42.2	42.2	2.8	1.1	1.50	1.36	0.04	0.02
Yr 9	VF-52	7.1	89.7	7.2	3.1	54.3	54.3	3.9	1.7	1.96	1.76	0.08	0.03
	Averages	**6.6**	**90.0**	**7.0**	**3.0**			**3.1**	**1.3**				

13.7.2.5 Historical Tailings Reprocessing

Almost three million tonnes of historical tailings, produced by the Bradley Mining Company, are located in the Meadow Creek Valley. They assay approximately 1.19 g/t gold, 2.92 g/t silver and 0.17% antimony. During the PFS a test program was completed to assess the metallurgical response of these materials to the flowsheet. This work was not repeated during the FS given the consistency of the results and the minor contribution of these materials to the overall project, and the results reported in the PFS have been adopted for the FS. For the sake of completion, the results are summarized in this section.

Particle size analyses of six composites tested show an average P_{80} of 184 µm with a range from 109 µm to 323 µm. The average gold head grade was 1.15 g/t, ranging from 0.78 g/t to 1.51 g/t (Table 13-22).

Table 13-22: Head Grade and Particle Size Analyses of Historical Tailings Composites

Head Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	0.98	1.12	1.51	0.78	1.17	1.31
Ag, g/t	4.00	3.00	3.80	-	-	-
As, %	0.09	0.15	0.18	0.09	0.15	0.17
Sb, %	0.14	0.16	0.22	0.07	0.23	0.17
S, %	0.43	0.36	0.29	0.18	0.37	0.28
PSA P_{80}, µm	323	142	109	139	276	116

Grinding testwork using historical tailings material blended with Yellow Pine material at a ratio of 15% of tailings and 85% indicated that blending the historical tailings material with fresh ore may reduce the operating work index of the total feed to the grinding circuit by 10 - 14% (Gajo, 2014b).

Two programs of flotation testwork were undertaken at SGS on the Historical Tailings (McCarley, 2014). The results are summarized in Table 13-23.

Table 13-23: Summarized Metallurgy from Re-processing of Historical Tailings Composites

Head Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	0.98	1.12	1.51	0.78	1.17	1.31
Sb, %	0.14	0.16	0.22	0.07	0.23	0.17
S, %	0.43	0.36	0.29	0.18	0.37	0.28
PSA P_{80}, µm	323	142	109	139	276	116
Antimony Concentrate Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	n/a	n/a	n/a	n/a	7.62	6.06
Sb, %	n/a	n/a	n/a	n/a	50.4	6.79
S, %	n/a	n/a	n/a	n/a	20.7	3.67
Antimony Concentrate Recovery	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	n/a	n/a	n/a	n/a	0.94	0.53
Sb, %	n/a	n/a	n/a	n/a	27.5	3.44
S, %	n/a	n/a	n/a	n/a	7.16	1.16
Sulfide Concentrate Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	16.0	13.2	4.64	11.5	25.7	12.4
Sb, %				n/a	1.44	1.48
S, %	6.45	4.65	0.93	2.85	8.94	4.47
Sulfide Concentrate Recovery	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	74.4	62.0	33.2	55.4	68.7	36.6
Sb, %				n/a	17	24.9
S, %	75.7	68.6	34.5	61.1	67	47.3
Tailings Leach Recovery	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH
Au, g/t	0.06	0.1	0.6	0.15	0.05	0.17
Au, %	26	37.2	57	35.6	16.5	31.7

This culminated in locked cycle testing to evaluate the effect on flotation of blending 15% historical tailings with 85% fresh Yellow Pine material (Gajo, 2014b). The results, shown below, indicate no adverse effects of blending in the tailings on overall metallurgy, with both antimony and gold recoveries very similar to those from testing Yellow Pine alone, and antimony concentrate grades remaining close to the sulfur grades required for POX.

Table 13-24: Flotation of Blended Yellow Pine Early Production Feed and Historical Tailings

Material	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
Blend of Early Production High Sb (85%) & Historical Tailings (15%)											
LCT Sb Final Concentrate	17.2	0.86	11.4	0.39	58.7	25.8	n/a	3.3	0.8	89.1	15.9
LCT Au Rougher Concentrate	354.7	17.7	15.1	2.09	0.28	6.3	1.7	91.3	92.2	8.9	80.9
LCT Au Rougher Tail	1635.4	81.5	0.19	0.03	0.01	0.05	0.85	5.4	7.0	1.9	3.1
Blend of Early Production Low Sb (85%) & Historical Tailings (15%)											
BT Au Rougher Concentrate	633.4	15.8	12.4	2.17	n/a	5.98	1.45	94.3	93.4	n/a	95.7
BT Au Rougher Tail	3367.7	84.2	0.14	0.03	n/a	0.05	n/a	5.7	6.6	n/a	4.3
Note: LCT - Locked Cycle Test, BT - Batch Test											

The above evidence suggests that the Yellow Pine and Historical Tailings materials can be successfully co-processed.

13.8 METALLURGICAL PERFORMANCE FORECAST

13.8.1 Yellow Pine

13.8.1.1 Yellow Pine Low Antimony

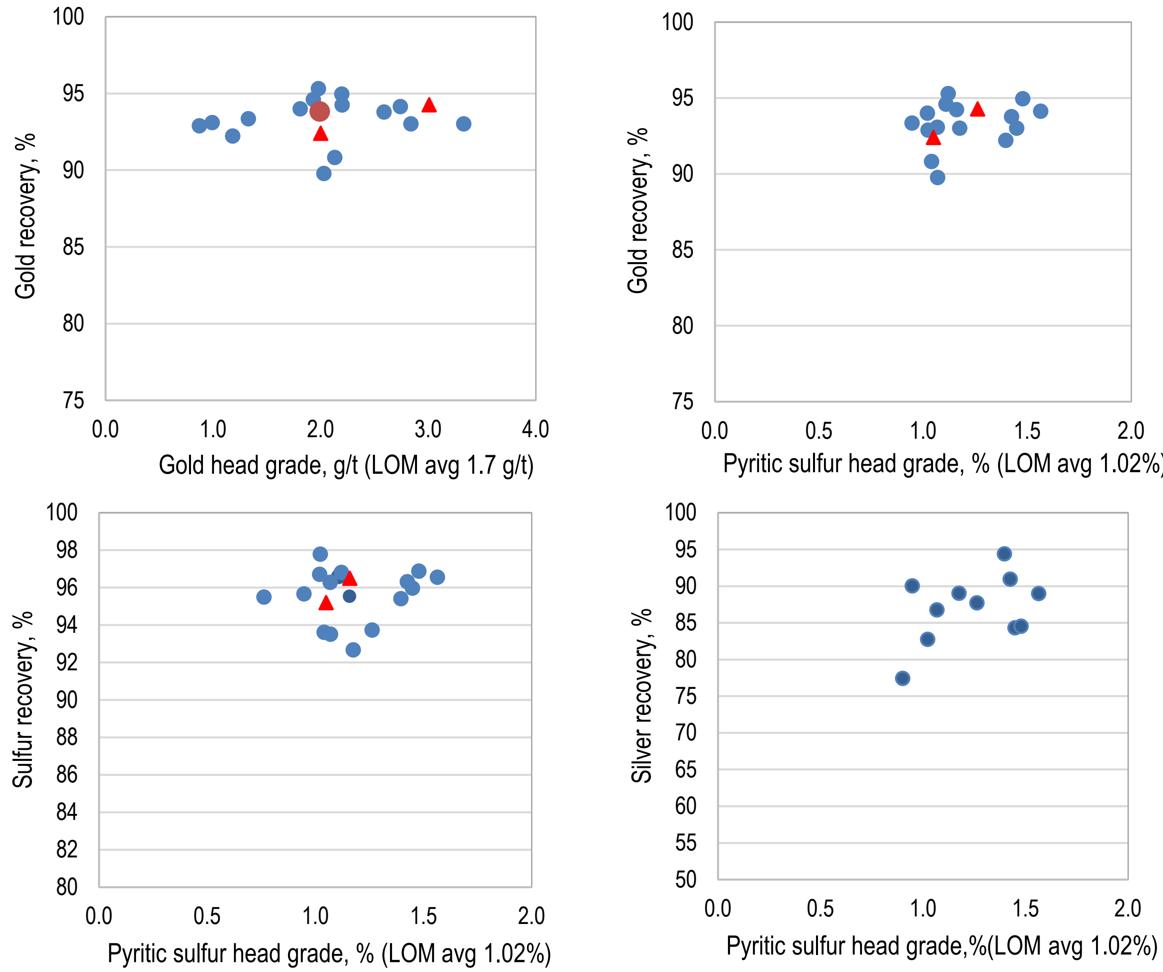
Metallurgical data used to estimate the recovery of gold, silver and sulfur from Yellow Pine (low antimony) feed material were derived from testing on three master composites, 12 variability composites, two bulk flotation composites and two pilot plant composites (both for bulk concentrate production for POX testing). Recoveries, plotted in Figure 13-22 are to a concentrate assaying 6.5% sulfur, and estimated by interpolation or if necessary, extrapolation of stage recovery data from the different programs, except for the pilot plant where actual recoveries to final concentrates were used. These averaged 6.6% sulfur.

Test data from two outlying samples have been excluded as these were oxide transition samples. Minimal oxide material is present in the Yellow Pine mineral resource (see Section 14).

Pilot plant data are shown as red triangles. Gold recovery is independent of gold or sulfur head grade across the range expected from the mine plan. Excluding data from the transition samples, the average gold recovery to the sulfide flotation concentrate is 93.8%. This has been assumed as a fixed number for the purpose of the FS metallurgical forecast.

Sulfur recovery to the concentrate is also plotted as a function of pyritic sulfur head grade on Figure 13-22. There is no trend with sulfur head grade, so a fixed recovery has been assumed as the average from all the data on sulfide (non-transition) samples. This is 96.1% sulfur recovery. Silver recovery is also assumed to be fixed at 90.1%.

Figure 13-22: Yellow Pine Low Sb Bulk Sulfide Flotation



13.8.1.2 Yellow Pine High Antimony

Head grades, concentrate grades and metal recoveries used for metallurgical forecasting are listed in Table 13-25. Sample YP-312, assaying 1.12% Sb, is a grade outlier. Gold losses from this sample, noted in the table in italics were abnormally high due to the high Sb grade. Locked cycle testing yielded a weak relationship between antimony head grade and recovery as shown on Figure 13-23.

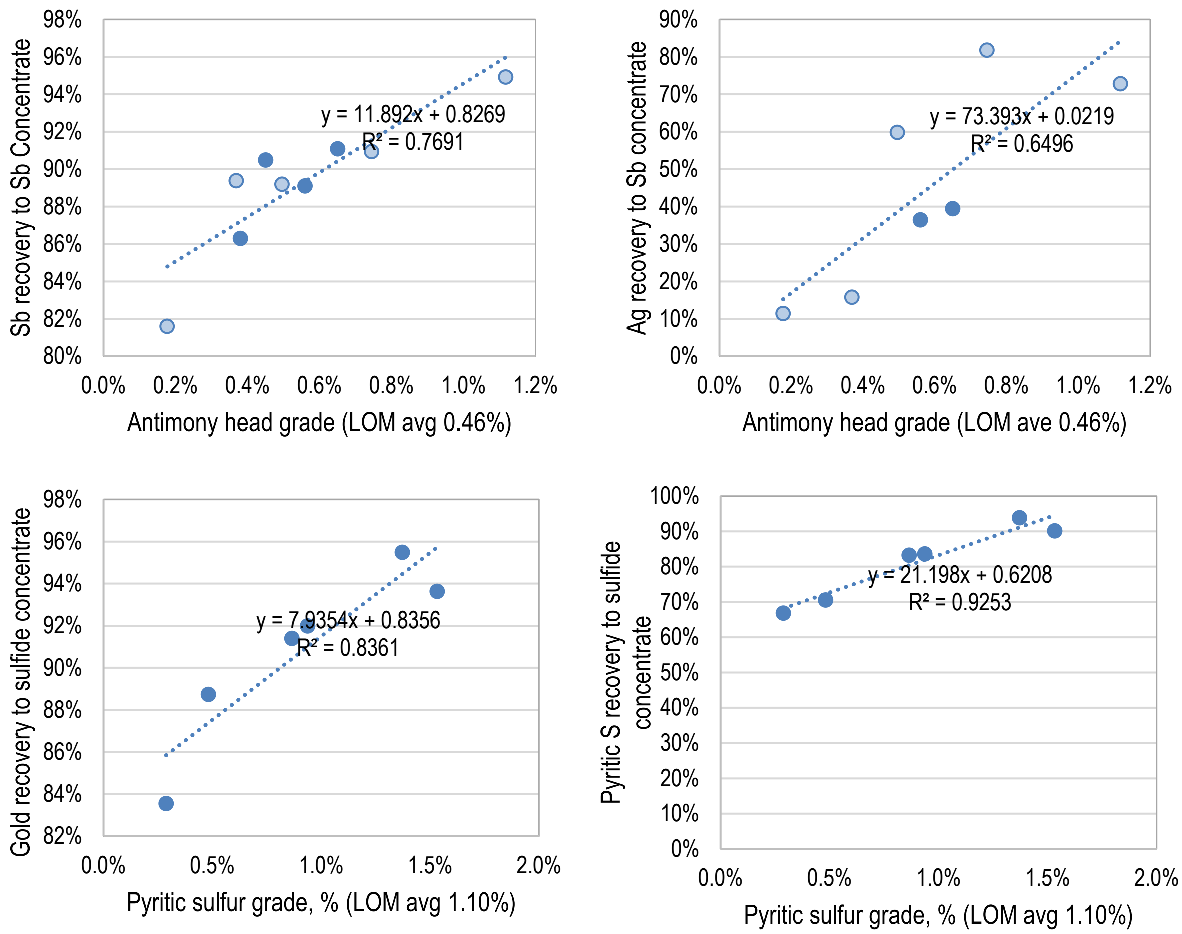
Table 13-25: Antimony-Rich Yellow Pine Composites and Tests used for Metallurgical Forecast

Sample	Test	Feed Grade			Sb Concentrate Grade			Recovery to Sb Concentrate			Recovery to Au Concentrate		
		Au (g/t)	Ag (g/t)	Sb (%)	Au (g/t)	Ag (g/t)	Sb (%)	Au (%)	Ag (%)	S (%)	Au (%)	Ag (%)	S (%)
YP-311	VF-20R	2.67	2.35	0.18	15.4	112	47.6	1.40	11.40	64.2	95.5	85.1	93.9
YP-312	VF-21R	1.97	17.67	1.12	11.3	695	50.0	*10.60*	72.70	82.8	83.6	21.8	66.9
YP-313	VF-22R	2.73	13.68	0.50	11.6	1020	50.6	3.40	59.80	81.8	92.0	35.2	83.7
YP-314	VF-23R	1.93	4.57	0.37	7.6	126	54.7	2.30	15.70	84.4	93.6	75.6	90.2
YP-315	VF-24R	1.97	16.37	0.75	4.7	1029	50.4	3.10	81.80	88.1	88.7	16.7	70.6
POH	LCT3	2.45	n/a	0.45	4.5	n/a	65.1	1.10	n/a	90.5	91.4	n/a	83.3
EPHB	LCT1	1.92	4.95	0.56	11.4	211	58.7	3.30	36.50	89.1	91.3	45.4	80.9

Sample	Test	Feed Grade			Sb Concentrate Grade			Recovery to Sb Concentrate			Recovery to Au Concentrate		
		Au (g/t)	Ag (g/t)	Sb (%)	Au (g/t)	Ag (g/t)	Sb (%)	Au (%)	Ag (%)	S (%)	Au (%)	Ag (%)	S (%)
EPH	LCT1	3.24	8.39	0.65	10.5	349	62.1	3.10	39.50	91.1	92.0	29.3	81.0
YPH	LCT1	2.11	0.25	0.38	9.3	n/a	57.1	2.50	n/a	86.3	92.1	n/a	80.2
YP0-3H	Bulk	1.98	n/a	n/a	n/a	n/a	n/a	2.50	n/a	89.3	85.9	n/a	81.3

Sufficient parallel batch and locked cycle tests have been conducted on the same samples to determine a factor to adjust batch test recoveries to project locked cycle recoveries. Using this, projected antimony and silver recoveries in closed circuit, as a function of head grade, have been plotted on Figure 13-23 using the adjusted batch test recoveries (in pale blue) on variability samples, as well as the locked cycle test recoveries from the master composites (dark blue).

Figure 13-23: Yellow Pine High Sb Testwork: Sb Head Grade vs Sb and Ag Recovery to Final Concentrate



Gold misplacement to the antimony concentrate tends to be linked to the amount of pyrite and arsenopyrite misplaced to this concentrate. The more stibnite floated to the antimony concentrate, the more pyrite and arsenopyrite is caught in the float so gold misplacement rises. A relationship describing this has been developed to predict gold losses to the antimony concentrate.

The concentrate grade has been assumed to be 65% antimony. This was achieved in the only locked cycle test run on a high Sb Yellow Pine sample in the FS. Given the inclusion of column flotation in the plant flowsheet, the QP believes it is reasonable to assume this can be achieved commercially.

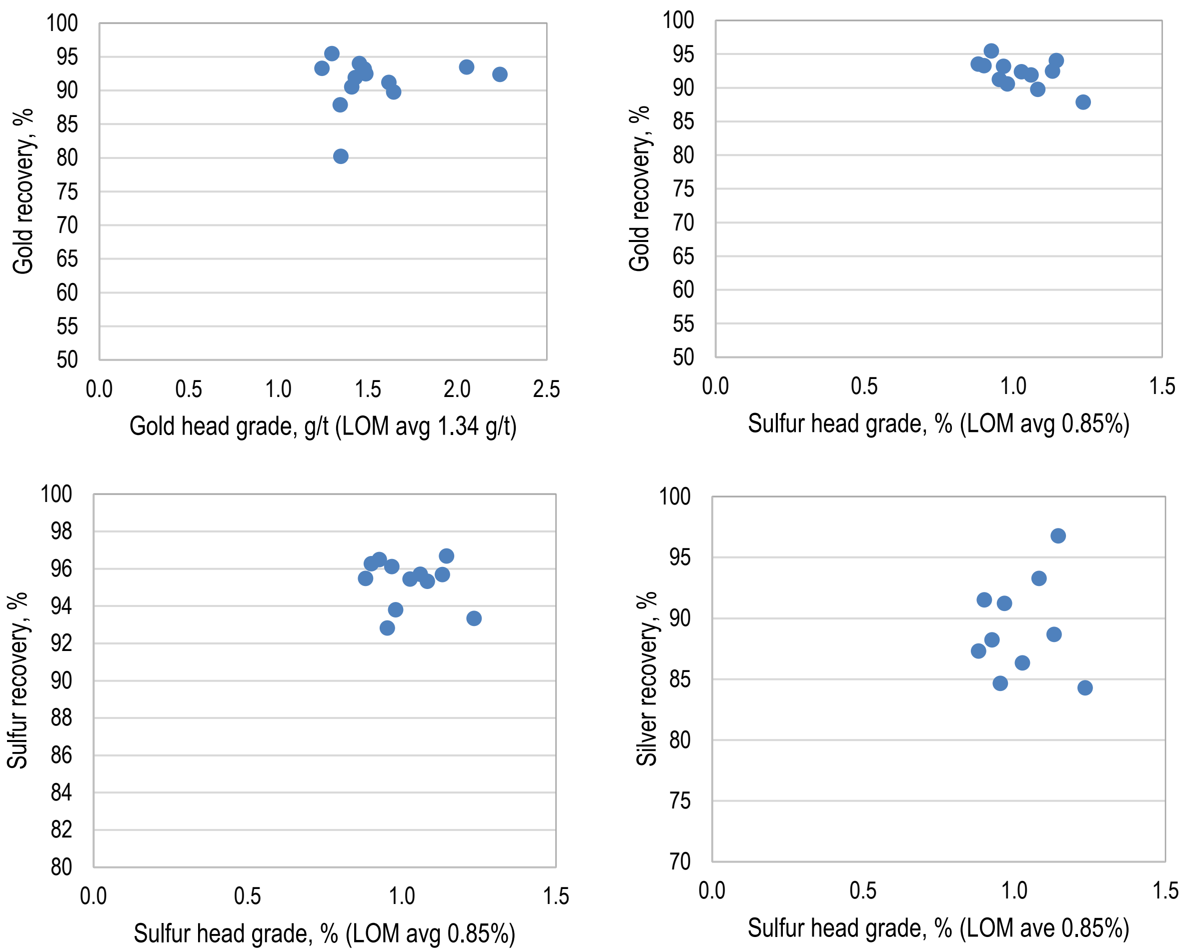
The test data in Table 13-25 were also used to build the metallurgical forecast for bulk sulfide flotation from Yellow Pine high antimony material. The recovery of gold and sulfur to the POX feed are linked to the pyritic sulfur grade in the feed (Figure 13-23). Silver recovery to the sulfide concentrate shows no trend against any head assay, so has been assumed to be an average of all the data - 46.9%.

13.8.2 Hangar Flats

13.8.2.1 Hangar Flats Low Antimony

Data from one process development composite (HFOL), ten variability composites and two bulk flotation composites have been used to determine recovery from low antimony Hangar Flats material. As with Yellow Pine, the recovery of gold to a POX-ready sample containing 6.5% sulfur from Hangar Flats materials, is independent of gold and sulfur head grades within the ranges defined by the mine plan. However, the mine plan for Hangar Flats contains considerable material with sulfur grades that lie outside of the range of tested samples, so the level of confidence in the metallurgical forecast is somewhat poorer. Overall, gold recovery has been assumed to be fixed at 92.1% (Figure 13-24). Within the range of sulfur head grades tested, sulfur recovery is also independent of sulfur head grade. Accordingly, sulfur recovery has been assumed to be fixed at 95.3%. Silver recovery was found to be independent of silver, sulfur or gold head grades and has been assumed to be fixed at 89.1%, the average of the data plotted on Figure 13-24.

Figure 13-24: Hangar Flats Low Antimony Metallurgy



13.8.2.2 Hangar Flats High Antimony

The samples and test data that formed the basis of the metallurgical forecast for the processing of high antimony material from Hangar Flats are provided in Table 13-26.

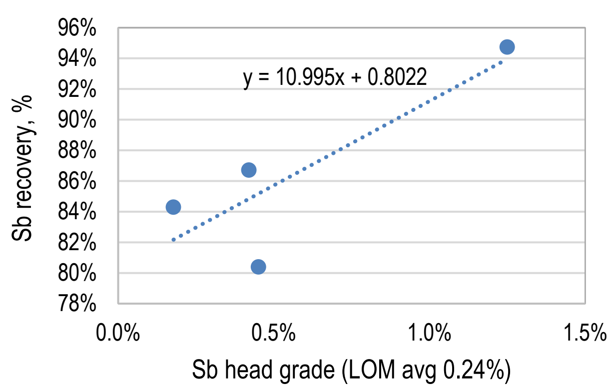
Table 13-26: Test Data Forming Basis of Met Forecast for High Sb Hangar Flats Material

Sample	Test	Head Grade			Sb Concentrate Grade				Recovery to Sb Concentrate			Recovery to Nominal 6.5% Sulfur	
		Au (g/t)	Sb (%)	S (%)	Au (g/t)	As (%)	Sb(%)	S (%)	Au (%)	Sb(%)	S (%)	Au (%)	S (%)
HFOH	LCT1	2.22	0.42	1.27	5.4	0.3	50.1	23.3	1.80	86.7	13.3	91.8	80.2
HFH	LCT1	1.85	0.45	1.06	4.9	0.2	58.1	25.8	1.60	80.4	15.3	89.8	80.3
HF-234	VF-26R	2.31	1.25	1.69	10.7	1.0	48.0	24.8	9.40	78.4	29.9	85.4	67.4
HF-235	VF-27R	1.43	0.18	1.16	16.7	5.0	45.8	27.6	3.60	80.2	7.4	91.7	89.7

Throughout the project, it has been observed that antimony recovery has been linked to head grade. This was explained mineralogically in the PFS as the stibnite grain sizes tend to coarsen with higher head grades.

For the two open circuit tests, closed circuit recoveries to final concentrate have been estimated by multiplying the rougher recoveries in each test by the average LCT/rougher recovery factor for the two samples where locked cycle test data are available (Figure 13-25).

Figure 13-25: Sb Head Grade vs Sb Recovery and Au Misplacement to Sb Conc, Hangar Flats High Sb



With higher Sb head grades, the propensity to lose gold to the Sb concentrate tends to rise, so gold misplacement is assumed to be linked to the Sb head grade. Gold misplacement is projected from the relevant data from the four tests (Figure 13-26).

Silver recovery to the antimony concentrate has been assumed to be the average of the tests where Ag metallurgical data are available (52.8%). The antimony concentrate grade has been assumed to be the average of the two locked cycle test concentrate grades (54.1% Sb).

The forecasted recoveries of gold, silver and pyritic sulfur have been assumed to be fixed at the average from the four tests (Au, 89.7%; Ag, 43.2% and sulfur 79.4%).

13.8.3 West End

The modelling of gold recovery from West End is more complex than from Yellow Pine and Hangar Flats. The broad range of oxide, transition and sulfide ore types coupled with the different flowsheet options leads to the need for a multivariate model.

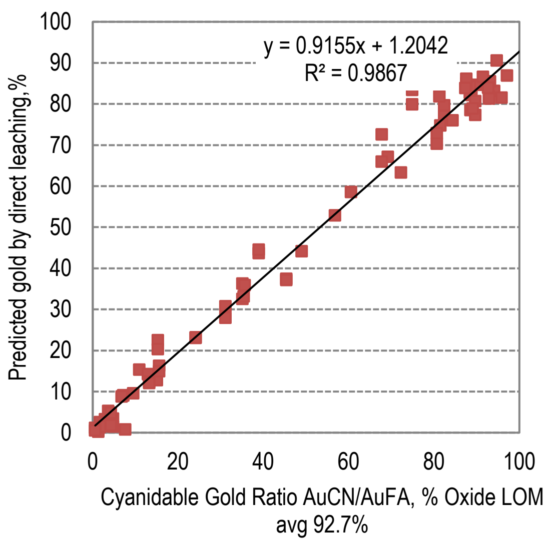
13.8.3.1 West End Oxide

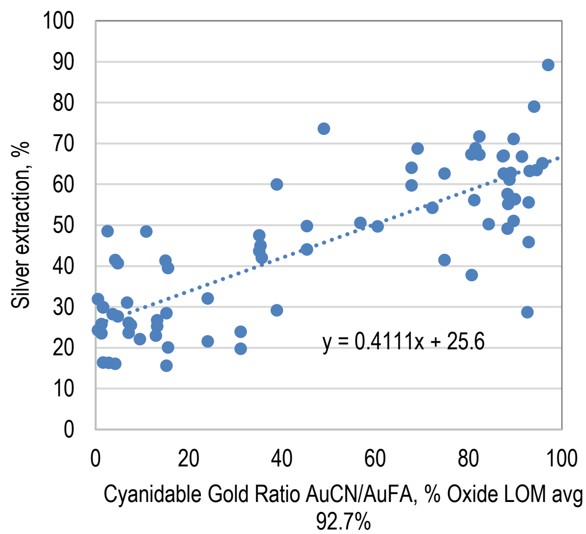
In past work the proportion of "cyanide leachable" gold ($^{CN}/_{FA}$) has been shown to be closely linked to (a) actual gold leach performance and (b) inversely linked to gold flotation recovery.

Some 78 samples were tested for $^{CN}/_{FA}$ and actual leach kinetic recovery. From the latter, the 12-hour leach recovery has been estimated, and 0.8% gold losses from leach solution have been applied to cover carbon losses and incomplete carbon absorption.

These data, for gold and silver plotted against AuCN/AuFA, are shown on the scatterplot on Figure 13-26:

Figure 13-26: Gold and Silver Recovery by Direct Cyanidation vs Cyanidable Gold Ratio





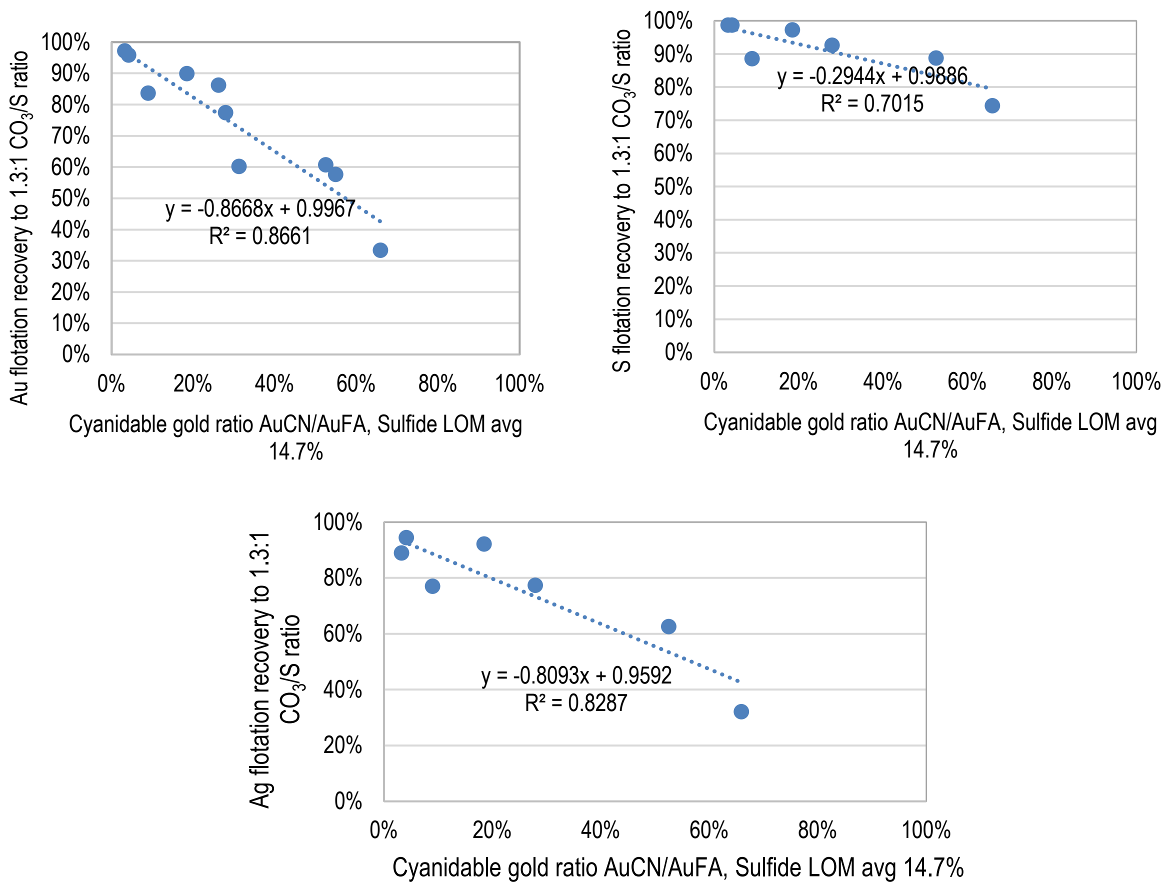
13.8.3.2 West End Sulfide

West End sulfide material is highly refractory while transition material has a significant free milling gold content. Sulfide material will be processed by flotation, concentrate POX and oxidation residue leaching, transition material will be treated by flotation, concentrate POX and oxidation residue leaching, plus leaching of the flotation tailings.

Unlike Yellow Pine and Hangar Flats, where the sole criterion for flotation concentrate production is to make an autothermic concentrate at 6.5% sulfur, the presence of carbonates in West End adds another tier of complexity to the float. Excessive carbonates in the concentrate neutralize the acid leach and may lead to excessive oxygen losses in the CO_2 off-gas from the autoclave. Testing has shown that the concentrate should not contain a mass ratio of carbonate to sulfur more than 1.3:1. Usually, this is the primary criterion in determining the nature of the flotation concentrate from West End samples and with it, gold recovery.

The recovery of gold and sulfur to a flotation concentrate assaying below the 1.3:1 CO_3/S limit, as a function of $^{CN}/_{FA}$ is shown on Figure 13-27.

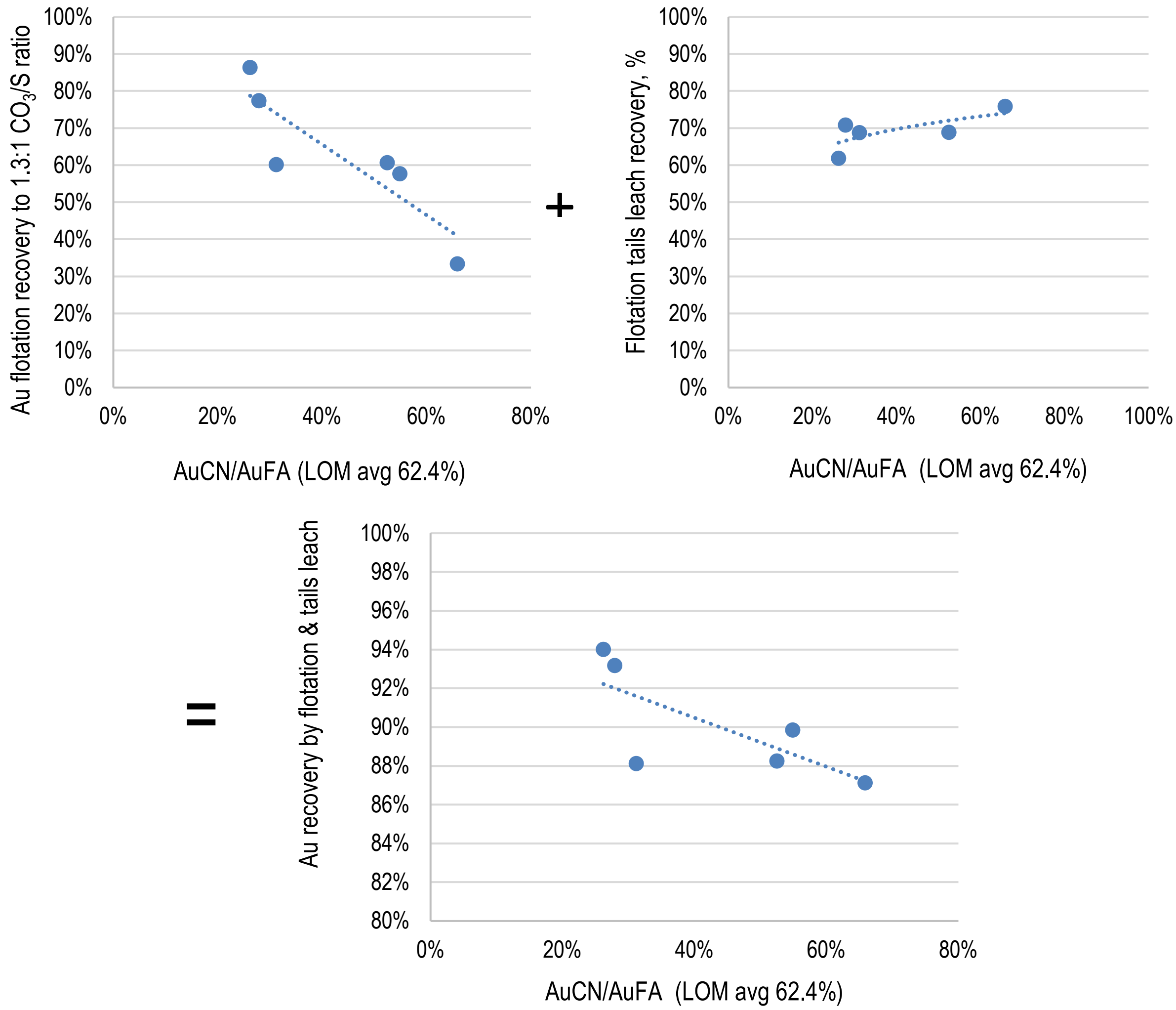
Figure 13-27: Correlation of Cyanidable Gold Ratio with Flotation Recoverable Gold, Sulfur and Silver



13.8.3.3 West End Transition

As described above, transition material will be processed by flotation with concentrate pre-oxidation, with the flotation tailings and the POX product both leached for gold recovery. As the flotation-POX-CIL process and the direct leach process are most effective on low and high $^{CN}/_{FA}$ gold respectively, running both processes tends to lead to consistently high recoveries. In fact, gold recoveries from West End transition material will be as high as any other material in the project (Figure 13-28):

Figure 13-28: West End Gold Recovery by Flotation and Tails Leaching vs Cyanidable Gold Ratio



The leach extraction of silver from West End flotation tailings shows no clear relationship with $^{CN}/_{FA}$. Given that it is a relatively minor contributor to the project, and the lack of clear connection with $^{CN}/_{FA}$, Ag or S head grade, the average tested recovery of 60.9% (including carbon and solution losses) has been assumed.

13.8.4 Bradley Tailings

13.8.4.1 Antimony Flotation

Antimony rougher flotation was conducted on three composites, namely S06, HTM and HTH, all using lead nitrate as activator, cyanide as a pyrite depressant and 3418A as a collector. Two of the three achieved antimony recoveries ranging from 40-55% (Table 13-27). The HTH composite, which hosted the most liberated stibnite, yielded the poorest flotation recoveries, suggesting the conditions tested were not appropriate for this particular sample.

Table 13-27: Antimony Rougher Flotation from Bradley Tailings

Composite	Test	Mass Pull, %	Sb Recovery
S06	14129-002 F-10	2.3	55.2
HTM	14129-002 F-11	1.62	40.3
HTH	14129-002 F-12	3.2	11.6

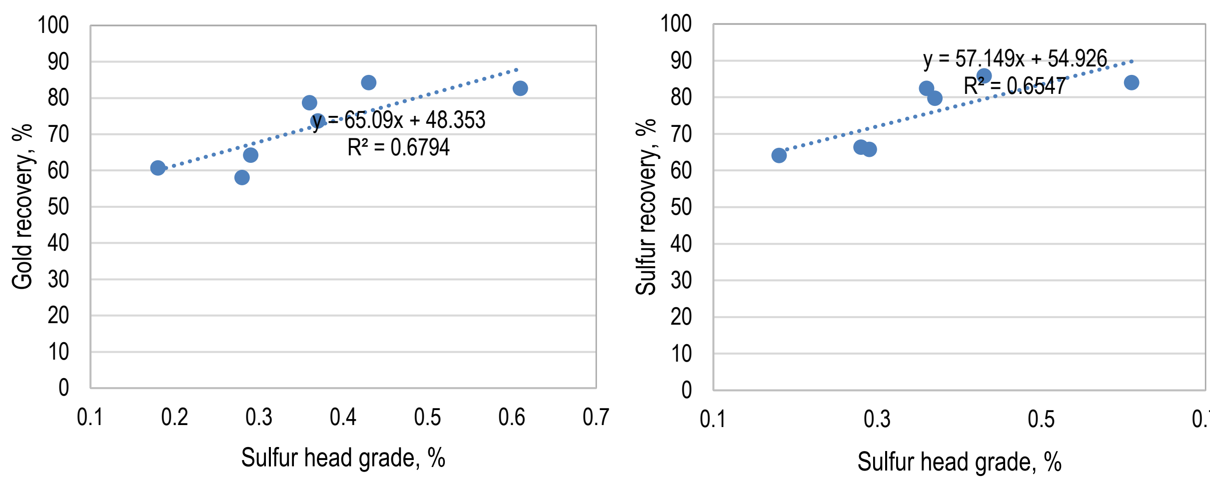
There is a paucity of reliable cleaner data, as so few tests were conducted on the samples at the time and cleaner flotation requires a greater degree of optimization testing than roughing. The best grades achieved were 50% Sb from the HTM composite and 60% Sb from the S06 composite, both suggesting that, if floated, there is a reasonable prospect that most of the Bradley tailings will yield saleable Sb concentrate grades. Gold grades in these concentrates were 8 and 3 grams per tonne respectively. Silver was not assayed.

Given the above information, it is the opinion of the QP that, for the sake of the feasibility study, 25% of the antimony in the Bradley Tailings can be assumed to report to the final antimony concentrate, at a grade equal to that produced from the primary ore sources. Gold losses to the concentrate can be assumed to be 0.5%.

13.8.4.2 Gold Flotation

Historic Bradley tailings will be co-processed with Yellow Pine and Hangar Flats material. The floatability of these tailings is highly dependent on the degree of oxidation of the material, and testing of this material yielded gold and sulfur recoveries linked to sulfur grade in the Bradley tailings (Figure 13-29). In the mine plan, the sulfur grade in the Bradley Tailings has been assumed to be fixed at 0.33%, so this points to a gold recovery to the flotation concentrate (POX feed) of 70%. Sulfur recovery to the autoclave feed is forecasted at 74%.

Figure 13-29: Re-flotation of Bradley Tailings: Feed S Grade vs Au Recovery



13.9 HYDROMETALLURGICAL TESTWORK

13.9.1 Introduction

Batch and pilot plant testwork for the pressure oxidative leach (**POX**) and neutralization processes were completed for the Stibnite Gold Project on behalf of Midas Gold Idaho Inc. The testwork was carried out in 3 laboratories, namely AuTec (Vancouver, Canada), CESL (Vancouver, Canada) and SGS (Malaga, Perth).

The test programs, in chronological order, are as follows:

- Batch Pressure Leach Testwork at AuTec (Le, 2017a).
- Pre-Autoclave Pilot Batch Testwork at AuTec (Le, 2017b).
- Continuous Pressure Oxidation and Cyanidation on Two Midas Gold Project Concentrate at AuTec, April 2017 (Ahern, et al., 2017).
- Solid Liquid Separation Testwork on Pilot Plant Feed and Discharge at AuTec, May 2017 (Pocock Industrial, 2017).
- Stibnite Gold Project Total Oxidative Leach (TOL) Bench Program at CESL, April – May 2017 (Teck Resources Limited, 2017).
- POX Discharge Diagnostic Leach Program at AuTec, June 2017 (Le & Erwin, 2017).
- Stibnite and West End POX Testwork at AuTec, August 20174 – January 2018, (Erwin, 2018).
- POX Batch Test at SGS Australia, July 2017 to March 2018, (Lima, 2018c).
- Pilot POX Test Program at SGS Australia, November 2017 (Lima, 2018c).
- Neutralisation Batch Test at SGS Australia, January – February 2018 (Lima, 2018a).
- Pilot Neutralisation Test at SGS Australia, March 2018 (Lima, 2018a).
- Geochemical Batch Test Program at SGS Australia, May 2018, (Lima, 2018d).
- Batch Test Program – arsenic destabilization identification at SGS Australia, April – June 2020 (SGS Minerals Metallurgy, 2020).

A majority of the gold in the Stibnite Gold Project ore was found to be refractory from the early testwork. Less than 5% of the gold was found to be amenable to a cyanide leach. Mineralogy on the Yellow Pine ore confirmed that the non-discrete gold predominantly occurred in the pyrite and to a lesser extent in the arsenopyrite.

To overcome the refractory nature of the ore, the ore was first concentrated via flotation and the flotation concentrate was then subjected to pressure oxidative leach to liberate the gold for downstream cyanide leach. The pressure oxidation process produced an acidic waste liquor stream which was required to be neutralized. The pressure oxidative leach and neutralization batch and pilot tests investigated the optimum process conditions for these processes.

Early pressure oxidative leach tests undertaken at AuTec and CESL were made on the following concentrate feed:

- Process optimization Low Sb – Yellow Pine Early Production Composite (used in early AuTec Batch test R2017-026).
- BCR Rougher Flotation Pilot Plant 1 Concentrate - ~88% Yellow Pine and 12% Hangar Flats Low Antimony ore (Concentrate AC 1, 2, 3 and 4 – used in AuTec pre pilot, pilot and CESL batch test).

In total there were 12 concentrate blends that were tested at SGS and these were:

- Con 1- advanced subsample of Con 10,
- Con 2- advanced subsample of Con 10,
- Con 3 – Yellow Pine – Low Sb (Yr. 0-3),
- Con 4 – Yellow Pine – High Sb (Yr. 0-3),
- Con 5 – Yellow Pine – Low Sb (Yr. 4+),

- Con 6 – Hangar Flats – Low Sb (Outside Fault Zone),
- Con 7 – Hangar Flats – High Sb (Fault Zone Influenced),
- Con 8 – West End low CO_3/S ratio,
- Con 9 – West End high CO_3/S ratio,
- Con 10 - Feasibility Concentrate – Low Sb Flotation (Pilot Plant Concentrate),
- Con 11 – West End – Low Sb (High Carbonate Composite), and
- Con 12 – West End – Low Sb (West End/ Hangar Flats Blend Year 7+).

The SGS batch and pilot continuous test campaigns in 2017/2018 demonstrated:

- That increasing the CO_3/S ratio to (1.3-1.5) in the concentrates reduces jarosite formation, increased gold recovery and reduced soluble arsenic in the residue and removed the requirement for a hot acid cure.
- Gold could be recovered at greater than 95% when the pressure leach residue was subjected to standard carbon in leach.
- The EPA Synthetic Precipitation Leaching Procedure (**SPLP**) arsenic values were typically less than 0.5mg/L in the SPLP leachate.
- That the variability concentrates from Yellow Pine, Hangar Flats and West End could all be pressure leached at CO_3/S ratio of between 1.3-1.5 and delivered gold extraction in CIL of greater than 95%.
- Even though the 2018 Geochem testwork at SGS undertaken on the co-mingled neutralized POX residue (pH of ~9) with flotation tailings had shown acceptably low SPLP concentrations of As (<2 mg/L), Hg (<0.02 mg/L) and Cr (<0.1 mg/L), the blended cyanide detox residue/flotation tailings were found to have unacceptably high SPLP arsenic.

As a result, an additional batch testwork program was carried out at SGS Laboratory (Malaga, Perth) for the Stibnite Gold Project on behalf of Midas Gold Idaho Inc. in the period of April to June 2020.

The objective of this 2020 testwork program was to (a) identify under what conditions the arsenic was destabilized in the downstream processing of the concentrate, and, (b) establish the impact on the downstream processes after pressure oxidation leach on the solute values especially mercury, arsenic and antimony.

The following process steps were examined:

- Pressure oxidative leach (**POX**);
- Atmospheric Arsenic Precipitation (**AAP**);
- Slurry neutralization;
- Cyanide leach / Carbon-in-Leach (**CIL**);
- Continuous cyanide detox; and
- Blending of cyanide detox residue and flotation tailings.

13.9.2 Review of Testwork

13.9.2.1 Batch Testwork at AuTec

The early batch tests were undertaken at AuTec on a blended concentrate (20% mass pull to yield a typical 5% sulfide concentrate) in March 2017 (Lee, 2017a). The purpose of the testwork program was to investigate the amenability of the gold concentrate to pressure oxidation and to optimize the autoclave operating parameters, hot acid cure (**HAC**), carbon-in-leach (**CIL**) process, and determine the rheological properties of the autoclave feed slurry. In this program, an acid pre-treatment was employed on the concentrate to neutralize the carbonate in the concentrate prior to pressure oxidation.

The main outcomes of this batch testwork campaign were:

- The Stibnite Gold concentrate was amenable to acid pressure oxidation at 220°C and a retention time of approximately 60 minutes. After a hot acid cure and CIL, the gold recoveries were 95 to 98%. The recovery of silver was poor at between 1 to 12%.
- Optimised leach feed densities appeared to be in the range of 30-35% for all concentrates.
- The concentrate P80 was 46µm and 50% of the gold were present in the fractions finer than 25 µm.
- Arsenic in the pressure leach residues was not stable and leached in the hot acid cure step.
- In CIL, the average cyanide consumption was 1.14 kg/t, and limestone addition was 7.2 kg/t.

13.9.2.2 Pre-Pilot Plant Batch Autoclave Testwork at AuTec

Batch Pre-Autoclave pilot testwork was carried out on two new concentrate samples (AC1 and AC2) prior to an autoclave pilot campaign at AuTec (refer to Appendix 5 for the full laboratory report). These concentrates had similar sulfide concentrations to the concentrates that were used in the pilot campaign. Concentrate AC1 contained 4.92% S and AC2 contained 6.63% S. The purpose of this batch testwork program was to understand their response in a 6-hour continuous pilot program. As per previous batch testwork at AuTec, this batch test campaign examined:

- autoclave operating parameters,
- the acid hot acid cure process,
- CIL process, and
- rheology properties of the autoclave feed slurry.

The batch pressure leach feed was also subjected acid pre-treatment by addition of sulfuric acid to pH 2 at ambient temperature.

The batch testwork on AC1 and AC2 concentrates demonstrated:

- Gold recoveries of 70% to 97%.
- Silver recoveries of 1% to 17%.
- These outcomes were achieved at 220°C and 35% solid density.
- Mercury extraction were 0 – 85%; some of which reported to the vent.

13.9.2.3 Pilot Testwork at AuTec

Pilot testwork were carried out on a 5% sulfide concentrate (AC3) in Run 1 and a 7% sulfide concentrate (AC4) in Run 2 at AuTec laboratory in April 2017 (Refer to Appendix 6 for the laboratory testwork report). The pilot campaign employed a five compartment (6 cell) autoclave using the following conditions:

- Acid pre-treatment – Concentrated sulfuric acid was added to the concentrate slurry to adjust the pH to 1.8 at ambient temperature.
- Autoclave temperature of 220°C.
- Autoclave retention time of 1 hour.
- Oxygen partial pressure of 462 kPa (67 psi).
- A 6-hour campaign for both AC3 and AC4.
- A feed slurry density of 37% solids.
- Hot acid cure was undertaken on AC 3 and AC4 at 95°C with 2 hours of retention time.
- 6-hour hot acid cure was also undertaken on AC4 autoclave discharge slurry.
- Both the AC3 and AC4 (post hot acid cure) residues were committed to the standard 24h CIL bottle roll cyanide leach test.

The outcomes from the pilot testwork were:

- Gold recovery of 95.6 and 97.6% for AC3 and AC4, respectively.
- Silver recoveries were 0.5% and 0.6% in AC3 and AC4 respectively.
- Soluble arsenic in the pressure leach was typically 2 to 3 g/L As and this increased further to 6-14 g/L As after hot acid cure in AC3. All soluble arsenic external to the autoclave was pentavalent.
- Trivalent arsenic was only detected in the autoclave first compartment at 0.45 g/L and 0.36 g/L for AC3 and AC 4 respectively.
- There was a significant amount of dissolved aluminum (4 to 5 g/L).
- Aqueous fluoride exiting the autoclave varied between 148 to 190 ppm F (total fluoride).
- The mercury deportment to the gas phase during the pilot plant was minimal (ORP during the pilot plant was kept below 550 mV (Ag/AgCl 3.8M KCl).
- Free acid concentrations in the autoclave discharge were 43 to 44 g/L H_2SO_4 for both AC3 and AC4.
- The dominant arsenic containing residue in the autoclave discharge was pitticite ($Fe.(AsO_4).(SO_4).H_2O$). Little to no scorodite ($FeAsO_4.2H_2O$) was detected.
- The instability of the pitticite was thought to contribute to the release of arsenic in hot acid cure.

The concern surrounding the instability of pitticite and its dominance over scorodite led to a review of the leach conditions. Additional to these arsenic containing products was a concern that excessive quantities of potassium jarosite were being made. The outcomes of this hydrometallurgical introspection were that, if sulfate in the aqueous phase could be sequestered to the extent that if it could be made to report to the solid phase without associating with ferric iron, then it may be possible to encourage $Fe\text{-}AsO_4$ association over that of $Fe\text{-}SO_4\text{-}AsO_4$. Additionally, $Fe\text{-}OH\text{-}SO_4$ as in jarosite was thought to lock low levels of gold in a non-cyanide leachable association. Overall, the hydrometallurgical solution suggested that in-situ acid neutralization (ISAN) may resolve arsenic stability. The AuTec autoclaves were not engineered to accommodate ISAN. Consequently, CESL was approached to assist with testwork that could accommodate the introduction of limestone during the leach.

13.9.2.4 Batch Testwork at CESL

Five batch pressure oxidation tests were undertaken at CESL Laboratory on the 5% Sulfur (AC3) and 7% Sulfur (AC4) concentrates in April to May 2017 (Appendix 7, Teck Resources, 2017). The batch tests investigated conditions with both acid pre-treatment and *in-situ* acid neutralization (**ISAN**).

The outcomes from the batch testwork were:

- High gold recoveries of >97% for AC3 and AC4.
- A retention time of 40 minutes was identified a possible minimum.
- High gold extraction of 96% was achieved on residues with as low as 94% sulfur oxidation.
- ISAN resulted in higher gold extraction compared to acid pre-treatment.
- It was found that there was an increase of as much as 7% in gold extraction with a reduction of 10 g/L in the final free acid in pressure oxidation.
- The active fluoride concentrations were typically at less than 1 mg/L.
- Arsenic in the batch leach aqueous phase was typically 0.8 to 2.6 g/L As at 60 minutes pressure oxidation discharge. However, during hot acid cure, additional arsenic was leached resulting in aqueous arsenic concentrations of 0.8 to 4.2 g/L in the hot acid cure discharge.

The introduction of limestone to the autoclave generated an additional non-condensable gas. Neither AuTec nor CESL were able to control the gas composition in the vapor phase. The decision was made to engage SGS Malaga, Perth to grow an understanding of the *in-situ* acid neutralization process and to be able to control not only the limestone addition but also the oxygen partial pressure.

13.9.2.5 Batch Testwork at SGS (2017/2018)

13.9.2.5.1 Introduction

The main purpose of ISAN was to improve gold extraction and produce a more stable arsenic precipitate that contains more scorodite and less pitticite. The batch testwork results confirms the benefits from the addition of limestone, with gold recoveries consistently between 96.5 and 99%. The SPLP arsenic concentrations decreased with increasing carbonate to sulfur ratio (refer to Figure 13-30).

Prior to the pilot testwork, batch pressure oxidative leach tests with ISAN were undertaken on 12 concentrates. Nine of these were variability concentrates (Concentrate 3 through to 9 and Concentrate 11 and 12). Concentrates 1, 2 and 10 represent the dominant concentrates. A summary of the result of the batch testwork are shown in Table 13-34.

Figure 13-30: SPLP Arsenic Concentration versus Carbonate to Sulfur Ratio

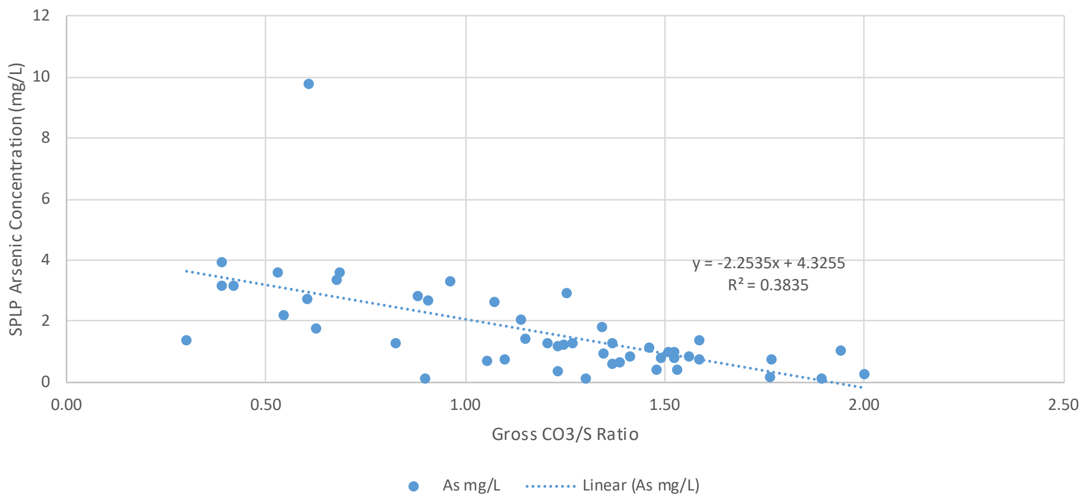
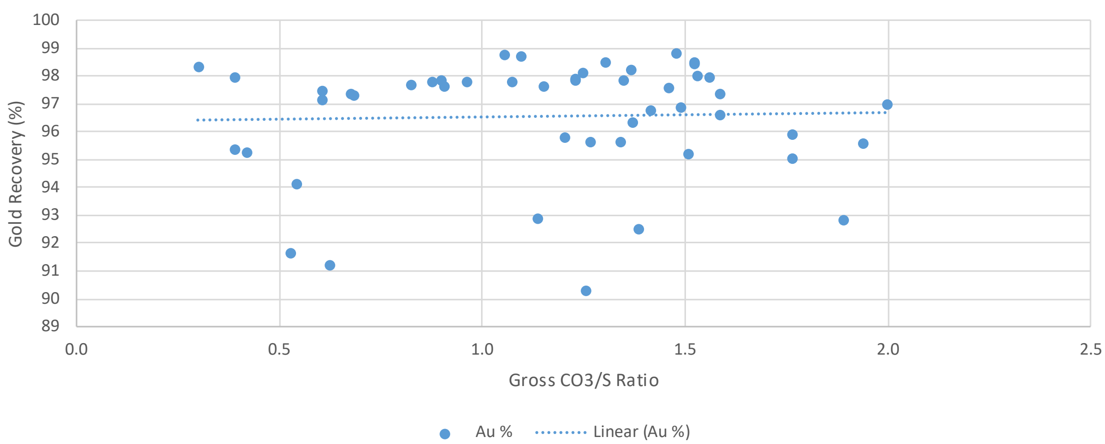


Figure 13-31: Gold Recovery versus Carbonate to Sulfur Ratio



In the batch tests, the 12 concentrates were initially subjected to a standard test procedure i.e. without the addition of any external carbonate. However, some of the concentrates had high in-situ CO_3/S ratio and therefore required the standard procedure to be altered to either:

- a "simulated" first compartment pressure leach procedure where some acid and ferric were present, or
- a ferric-initiated leach by adding a small quantity of sulfuric acid at the strike temperature to release ferric ions and thereby catalysing the sulfide oxidation process.

In the standard test procedure, the concentrate was mixed with water and heated to 220°C under a nitrogen blanket. When the slurry was at the target temperature, oxygen was added into the autoclave and the oxidation process was commenced.

Concentrates that naturally have a low CO_3/S mass ratio (<0.5) generated sufficient soluble ferric ions to support the ongoing oxidation of sulfide and the exothermic process. However, in some of the batch tests where no exothermic reaction was observed after more than 5 minutes, a very small quantity of sulfuric acid was added to attempt to generate ferric and catalyze the oxidation process. This latter process thus simulated the first compartment of a continuous autoclave.

This simulated first compartment procedure involved progressive addition of the concentrate slurry into the batch autoclave filled with water with sulfuric acid. Unfortunately, it is impossible to get the kinetic information from this type of test as the concentrate was progressively leached as it was added to the autoclave. Table 13-28 identifies the optimum conditions for each concentrate type based on the batch test results and also whether the concentrates were able to support auto-thermal conditions.

The result showed the following:

- For optimised results (gold extraction>95% and SPLP (As) <3 mg/L), gross CO_3/S ratio was generally higher than 0.9 for all concentrate types (average of approximately 1.3).
- The retention time of the concentrate that requires progressive leaching to simulate the first compartment could not be determined. This type of concentrates may be slower acting and may require longer retention time in the autoclave. This can only be confirmed by conducting a pilot plant campaign.
- Concentrate 8 was very slow to oxidise under autoclave conditions.
- CIL gold extraction on the pressure oxidative leach residue of the optimised batch tests was generally high (greater than 95% Au extraction) except for concentrate 8 that yielded 90% gold extraction.

Concentrate 10 (Con 10) was the Feasibility Study concentrate composite from a low antimony ore flotation campaign. This concentrate was generated from the flotation Pilot Plant.

Table 13-29 addresses the response of Concentrate 10 in batch tests employing three (3) limestone grades. These limestones were:

- Analytical reagent grade CaCO3,
- By-product from a local industry - AS limestone, and
- Middle Marble – an “on-site” limestone.

13.9.2.5.2 Summary of SPLP Results

The SPLP values for select batch tests undertaken on the Feasibility Study Concentrate (i.e. Concentrate 10) are given in Table 13-30.

Prior to the pressure oxidative leach pilot plant campaign in November 2017, several batch tests were undertaken employing AR Grade limestone. The same limestone quality was employed in the pilot plant. The results from this work provided guidance on what CO_3/S mass ratio was relevant for consideration in the pilot plant.

Table 13-28: Optimized Batch Tests Condition for the 12 Concentrates

Conc. Type	Test No.	Sulfide (S^{2-}) in Conc. %	Sulfur as Sulfate in Conc.*** %	CO_3/S Ratio *In situ* within Conc.	Effective CO_3/S Ratio Applied to Conc.	Calculated Gross CO_3/S Ratio (*In situ* +Applied)	Temp °C	Test Retention Time min	Feed Density % solids	CIL Au Ext. %	CIL Ag Ext. %	SPLP As Conc. mg/L	Able to Support Auto Thermal Conditions?
Con 1	361-11	6.78	0.45	0.15	0.75	0.90	220	90	30	97.9	3	0.2	Yes
Con 2	361-30	6.3	1.09	0.34	1.06	1.41	220	90	34	98.2	2.9	NA**	Yes
Con 3	361-55	5.19	1.44	0.62	0.90	1.51	220	NA*	32.3	95.2	0	1.0	No
Con 4	361-73	4.87	1.29	0.63	0.80	1.41	220	NA*	34	96.8	0	0.8	No
Con 5	361-63	9.75	0.3	0.56	0.65	1.20	220	NA*	32	95.8	0	1.32	No
Con 6	361-103	5.12	0.61	0.42	1.07	1.49	220	90	34.8	96.9	0.2	0.8	Yes
Con 7	361-113	7.33	1.14	0.39	0.76	1.15	220	90	35.9	97.6	0.1	1.4	Yes
Con 8	361-123	16.80	0.21	0.7	0.56	1.25	220	240	35	90.3	0.8	2.9	Yes – but slow acting
Con 9	361-45	12.40	0.27	1.05	0.47	1.52	220	NA*	35	98.4	0	0.8	No
Con 10	361-84	6.26	0.19	0.39	0.86	1.25	220	90	35.4	98.1	0.7	1.3	Yes
Con 11	361-142R	7.27	0.31	1.48	0	1.48	220	NA*	35	98.8	1.1	0.4	No
Con 12	361-133	6.98	0.07	1.14	0.39	1.53	220	NA*	35	98.0	1.6	0.4	No
Average				0.66	0.69	1.34	220	115	34	96.8	0.9	1.0	
Max				1.48	1.07	1.53	220	240	36	98.8	3.0	2.9	
Min				0.15	0.00	0.9	220	90	30	90.3	0.0	0.2	

* Retention time is not available as the tests were undertaken using the simulated first compartment
** SPLP As data not available
*** Sulfur as sulfate in concentrate was calculated by difference (i.e. Sulfur (Sulfate) = Sulfur (Total) – Sulfide – Elemental Sulfur)

Table 13-29: Concentrate 10 Summary of Pressure Oxidative Leach Batch Testwork

Test No	Test Objective	Temp	Feed Density	Calculated Gross CO_3/S Mass Ratio (*In situ* + Applied)	CIL		SPLP in Final Sample			Pressure Oxidative Leach at 60 min							
										Residue		Aqueous					
					Au Ext.	Ag Ext.	Hg	Cr	As	Fe	S^{2-}	S	Fe	Fe^{2+}	K	As	Free Acid
		°C	% Solids		%	%	mg/L	mg/L	mg/L	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	g/L
361-81	Base Line Test - No $CaCO_3$ Addition	220	34.67	0.39	97.5	1.1	<0.02	<0.1	3.9	7.1	0.050	24000	3730	168	115	1330	43
361-82	Varying $CaCO_3$	220	34.86	0.68	97.3	0.6	<0.02	<0.1	3.4	7.6	BDL	21400	3790	279	88	979	37
361-83	Varying $CaCO_3$	220	34.87	0.96	97.9	0.5	<0.02	<0.1	3.3	7.4	0.450	17200	2070	223	60	423	33
361-84	Varying $CaCO_3$	220	35.39	1.25	97.1	0.7	<0.02	<0.1	1.3	7.6	1.300	12000	1200	279	23	136	22
361-85	Varying $CaCO_3$	220	35.17	1.46	98.1	2.2	<0.02	<0.1	1.2	6.9	BDL	6260	297	0	21	100	10
361-86	Using AS limestone	220	38.4	1.19	36.2	52.1	<0.02	<0.1	14.5	8.7	3.720	694	1	0	227	559	0
361-87	Using AS limestone	220	35	1.59	97.3	1.1	<0.02	<0.1	1.4	6.5	0.080	10000	928	167	8	45	17
361-88	Using AS limestone	220	35.4	1.59	96.6	2.1	<0.02	<0.1	0.8	6.6	BDL	6070	120	19	16	10	8
361-89R	Using AS limestone - Varying $CaCO_3$	220	34	1.39	92.5	0.0	<0.02	<0.1	0.7	6.7	0.010	10400	1220	110	18	61	17
361-90	Varying Middle Marble $CaCO_3$	220	35.3	0.68	97.3	0.0	<0.02	<0.1	3.6	7.5	BDL	24400	4580	240	109	1320	40
361-91	Varying Middle Marble $CaCO_3$	220	35.1	0.88	97.8	0.0	<0.02	<0.1	2.9	7.4	BDL	19000	2070	200	78	539	34
361-92	Varying Middle Marble $CaCO_3$	220	35.1	1.07	97.8	0.0	<0.02	<0.1	2.6	7.4	BDL	18200	1770	74	85	287	32
361-93	Varying Middle Marble $CaCO_3$	220	35.2	1.27	95.6	2.1	<0.02	<0.1	1.3	7.1	BDL	13400	1060	56	35	146	21
361-94	Varying Middle Marble $CaCO_3$	220	34.9	1.65	95.2	8.2	<0.02	<0.1	0.4	7.0	0.030	9212	376	0	55	29	11
361-95	Geochemical Test with Middle Marble $CaCO_3$	220	36.3	1.26	89.1	29.4	<0.02	<0.1	0.9	6.9		6350	13		100	47	2
361-95 t=60 CIL	Geochemical Test with Middle Marble $CaCO_3$	220	36.3	1.26	99.1	11.6	<0.02	<0.1	0.9	6.9	0.020	6350	13		100	47	2

Table 13-30: Concentrate 10 Batch Testwork SPLP Results

Test No.	Feed Density % w/w	Limestone Addition kg/t	Gross CO_3/S Mass Ratio= (*In situ* + Applied)	CIL Extraction, %		SPLP in Final Sample, mg/L		
				Au	Ag	Hg	Cr	As
361-81	34.67	0	0.39	97.5	1.1	<0.02	<0.1	3.9
361-82	34.86	33	0.68	97.3	0.6	<0.02	<0.1	3.4
361-83	34.87	67	0.96	97.9	0.5	<0.02	<0.1	3.3
361-84	35.39	100	1.25	97.1	0.7	<0.02	<0.1	1.3
361-85	35.17	125	1.46	98.1	2.2	<0.02	<0.1	1.2

The results showed the following:

- At a gross CO_3/S ratio lower than 1.25, the arsenic concentration in the SPLP leach was significantly higher compared to the tests at CO_3/S ratio higher than 1.25.
- The CO_3/S ratio did not appear to affect the gold or silver CIL recovery.

13.9.3 SGS Pilot

The SB100 pressure oxidative leach pilot plant was undertaken at SGS Laboratory in Malaga, Western Australia, during the period of 20th to 26th November 2017. The testing was conducted in a 22-liter autoclave with four compartments at feed rate of 4-6 kg/h and a nominal residence time of 75 minutes. The autoclave residue was treated by hot acid cure (**HAC**) and neutralized prior to cyanide leaching

Concentrate 10 was employed as the pilot plant concentrate. Feed composition of every 5th bag of feed concentrate were subsampled and assayed, and the assays are shown in Table 13-31.

Table 13-31: Pilot Plant Concentrate 10 Composition

Bag No.	Solids %	Al %	As %	Fe %	K %	P %	Sb %	Ag ppm	Hg ppm	C %	S %	S° %	S^2 &-
1	73.1	7.72	2.57	7.98	5.58	0.07	0.19	16	10.8	0.55	6.57	0.14	6.06
5	72.3	7.74	2.58	7.71	5.66	0.07	0.19	16	12.7	0.54	6.51	0.56	5.88
10	70.7	7.35	2.51	7.7	5.34	0.07	0.19	13	12.5	0.55	6.66	0.26	6.36
15	71.7	7.8	2.56	7.7	5.7	0.07	0.2	15	9.6	0.58	6.71	0.68	5.07
20	73.1	7.72	2.65	8.25	5.57	0.07	0.21	15	11	0.55	6.26	0.25	5.37
25	71.2	7.68	2.54	7.83	5.47	0.08	0.19	13	10.5	0.54	6.66	0.45	5.48
30	71.9	7.52	2.55	7.58	5.61	0.07	0.2	15	10.9	0.55	6.26	0.44	5.76
35	71.6	7.53	2.59	7.79	5.62	0.07	0.2	13	11.1	0.57	6.75	0.88	5.56
40	71.5	7.72	2.58	7.59	5.56	0.07	0.19	14	11	0.56	6.44	0.26	5.85
45	71.9	7	2.42	7.18	5.23	0.07	0.19	13	10.5	0.55	6.54	0.42	6.05
50	72.3	7.61	2.55	7.96	5.6	0.07	0.2	14	14	0.57	6.51	0.29	5.72
55	72.3	7.37	2.77	8.46	5.43	0.07	0.21	13	11	0.57	7.46	0.07	7.31
60	73.9	7.89	2.65	7.77	5.65	0.07	0.2	18	11.2	0.61	6.9	0.77	4.28
Min	**70.7**	**7.0**	**2.4**	**7.2**	**5.2**	**0.1**	**0.2**	**13.0**	**9.6**	**0.5**	**6.3**	**0.1**	**4.3**
Mean	**72.1**	**7.6**	**2.6**	**7.8**	**5.5**	**0.1**	**0.2**	**14.5**	**11.3**	**0.6**	**6.6**	**0.4**	**5.8**
Max	**73.9**	**7.9**	**2.8**	**8.5**	**5.7**	**0.1**	**0.2**	**18.0**	**14.0**	**0.6**	**7.5**	**0.9**	**7.3**

During the pilot testing, samples were taken to provide a snapshot of the pilot plant conditions at a particular time. These samples are referred to as "profile samples." The process conditions for the pilot plant when the profiles samples were taken are shown in Table 13-32.

Table 13-32: Process Conditions for Pilot Plant Profile Samples

Profile No.	CO_3/S Ratio Gross (*In situ* + Added)	Autoclave Feed Rate (Solids) kg/h	Autoclave Feed % Solids %w/w	Total Oxygen Flow in Autoclave NL/min	Autoclave Temperature Profile (Average) °C	Autoclave Operating Pressure kPa(g)	Oxygen Partial Pressure kPa	Autoclave Retention Time min	Hot Acid Cure?
SOW Target	-	5	~30-38	-	220	3000-3100		90	Y
No Profile	0.422	3.7	33.0	20.4	217	3000	598	118	N
Profile 1	0.834	5.9	34.4	28.5	210	3000	915	78	Y
Profile 2	0.826	6.1	34.0	24.0	215	3100	640	74	Y
Profile 3	1.164	4.2	28.2	20.8	211	3100	1058	85	Y
Profile 4	1.150	4.6	26.4	18.5	216	3100	871	72	Y
Profile 5	1.505	4.9	32.1	20.9	216	3133	888	86	N
Profile 6	1.560	5.7	32.1	30.0	218	3300	929	74	N

Quantitative mineralogy was undertaken on the pilot autoclave solids. The main purpose of this mineralogical testing was to determine the quantitative contribution of the mineral phases of the residues for the project mass and energy balance.

The SGS mineralogy results are shown in Table 13-33 (QEMSCAN). Table 13-34 confirms that SGS was not able to account for the iron in the sample (refer to the differences between the QEMSCAN and ICP Chemical Assay highlighted in yellow in Table 13-34). Consequently, the Commonwealth Scientific and Industrial Research Organisation (CSIRO) were engaged to do a mineralogical inference (QXRD and direct extraction) of the iron phase and the results of the CSIRO work are shown in Table 13-35.

Table 13-33: SGS Pressure Oxidative Leach Pilot Plant Mineralogy (QEMSCAN)

Mineral	Mineral Mass (%) at Various Gross CO_3/S Ratios (*In situ* + Added)							
	0.422			0.826*		1.150**		1.560***
	Feed Concentrate	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge
Quartz	21.44	20.39	19.51	23.43	22.89	21.28	22.76	20.26
Albite	1.31	1.36	1.40	1.69	1.68	1.54	1.41	1.18
Orthoclase	37.73	38.43	30.43	37.34	37.14	37.12	36.37	35.80
Muscovite	13.03	11.64	17.50	14.08	11.03	11.21	12.75	12.04
Chlorite	0.82	0.08	0.38	0.35	0.35	0.64	0.72	0.96
Other Silicates	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Biotite	0.32	2.48	7.54	3.34	0.69	2.46	2.16	2.58
Arsenopyrite	4.80	0.01	0.00	0.00	0.00	0.00	0.00	0.00
Arsenian Pyrite	0.14	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Pyrite	9.77	0.00	0.04	0.00	0.01	0.02	0.00	0.07
Stibnite	0.69	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Sulphides	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Fe-Oxide	0.19	0.08	0.03	0.05	0.05	0.08	0.13	0.13

Mineral	Mineral Mass (%) at Various Gross CO_3/S Ratios (*In situ* + Added)							
	0.422			0.826*		1.150**		1.560***
	Feed Concentrate	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge
Rutile	0.27	0.34	0.27	0.76	0.29	0.49	0.46	0.35
Zircon	0.07	0.04	0.07	0.09	0.02	0.10	0.11	0.05
Other Oxides	0.00	0.00	0.00	0.00	0.00	0.06	0.00	0.03
Pitticite	0.17	12.88	10.53	9.90	12.28	11.06	9.88	8.94
Scorodite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Jarosite	0.02	11.03	5.65	4.10	5.99	3.90	3.60	1.00
Anhydrite/ Gypsum	0.11	0.79	6.04	4.32	6.92	9.45	9.12	15.99
Alunite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Other Sulphates	0.08	0.32	0.48	0.38	0.54	0.52	0.42	0.47
Carbonates	8.62	0.04	0.02	0.01	0.01	0.01	0.00	0.01
Phosphates	0.43	0.12	0.10	0.17	0.10	0.09	0.10	0.12
Totals	**100.01**	**100.03**	**99.99**	**100.01**	**99.99**	**100.03**	**99.99**	**99.98**
* from Pilot Plant Profile 2. ** from Pilot Plant Profile 4. *** from Pilot Plant Profile 6.								

Table 13-34: Comparison of SGS QEMSCAN Assay versus Chemical Assay

Element		Mineral Mass (%) at Various Gross CO_3/S Ratios (*In situ* + Added)							
		0.422			0.826*		1.150**		1.560***
		Feed Concentrate	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge	Hot Acre Cure Discharge	Pressure Leach Discharge	Hot Acid Cure Discharge	Pressure Leach Discharge
Al (%)	QEMSCAN	7.29	7.3	8.23	6.97	7.47	7.01	6.95	6.98
	Chem	7.55	7.2	7.08	7.08	7.43	7.02	6.84	7.23
As (%)	QEMSCAN	2.26	2.66	2.16	2.51	2.02	2.01	2.24	1.81
	Chem	2.13	2.64	1.96	1.96	1.78	2.08	2.1	1.77
Ca (%)	QEMSCAN	3.21	0.27	1.79	2.05	1.29	2.7	2.81	4.73
	Chem	2.9	I/S	I/S	2.62	2.29	3.59	3.6	4.98
Fe (%)	QEMSCAN	6.75	7.87	5.49	5.99	4.77	4.66	5.1	3.56
	Chem	7.37	7.14	5.72	7.23	6.6	6.56	6.75	6.77
K (%)	QEMSCAN	5.03	5.83	5.72	5.23	5.59	5.21	5.22	4.97
	Chem	5.37	5.37	5.09	5.09	5.33	5.11	5.09	4.96
Na (%)	QEMSCAN	0.23	0.24	0.22	0.23	0.26	0.23	0.25	0.22
	Chem	0.22	I/S	I/S	0.26	0.21	0.21	0.21	0.2
S (%)	QEMSCAN	6.22	2.6	3.04	3.45	2.32	3.34	3.57	4.59
	Chem	6.06	2.29	2.9	3.91	2.93	3.4	3.42	4.05
Si (%)	QEMSCAN	24.1	23.7	23.3	23.7	25.6	24.4	23.71	23.04
	Chem	22.6	I/S[Note 4]	I/S [Note 4]	24.1	24.3	22.2	19.5	22
* from Pilot Plant Profile 2. ** from Pilot Plant Profile 4. *** from Pilot Plant Profile 6. I/S = Insufficient Sample									

Table 13-35: CSIRO QXRD Pilot Plant Mineralogy on Pressure Oxidation Pilot Plant Profile 6

Mineral	Mineral Mass, % w/w				
	POX Autoclave Compartment 1	POX Autoclave Compartment 2	POX Autoclave Compartment 3	POX Autoclave Discharge	Concentrate 10 Geochem
Quartz	19.9	20.6	20.9	17.4	17.8
Albite	2.2	2.3	2.3	2.4	2.4
K-feldspars	30.1	30.9	29.7	27.5	26.5
Muscovite/Illite	26.5	26.7	28.2	30.4	26.7
Pyrite	1.5	1.2	0.5	0.4	0.3
K-Jarosite	0.0	0.0	0.0	0.0	0.0
Anhydrite/Bassanite/Gypsum	0.7	0.3	1.8	4.1	6.1
Philipsbornite	0.4	0.4	0.6	0.6	0.7
Kaolinite	0.0	0.0	0.0	0.4	0.1
Clinochlore	0.0	0.0	0.0	0.0	0.0
Ca silicate/amphibole	0.0	0.0	0.0	0.0	0.7
$FeAsO_4$ (max.)	7.12	7.41	6.94	7.28	6.03
$Fe(OH)_3$ (max)	7.62	7.97	9.68	11.3	9.76
$FeAsO_4 + Fe(OH)_3$ (max)	14.7	15.4	16.6	18.6	15.8
$FeAsO_4 + Fe(OH)_3$ (min)	10.2	9.1	8.4	10.4	15.4

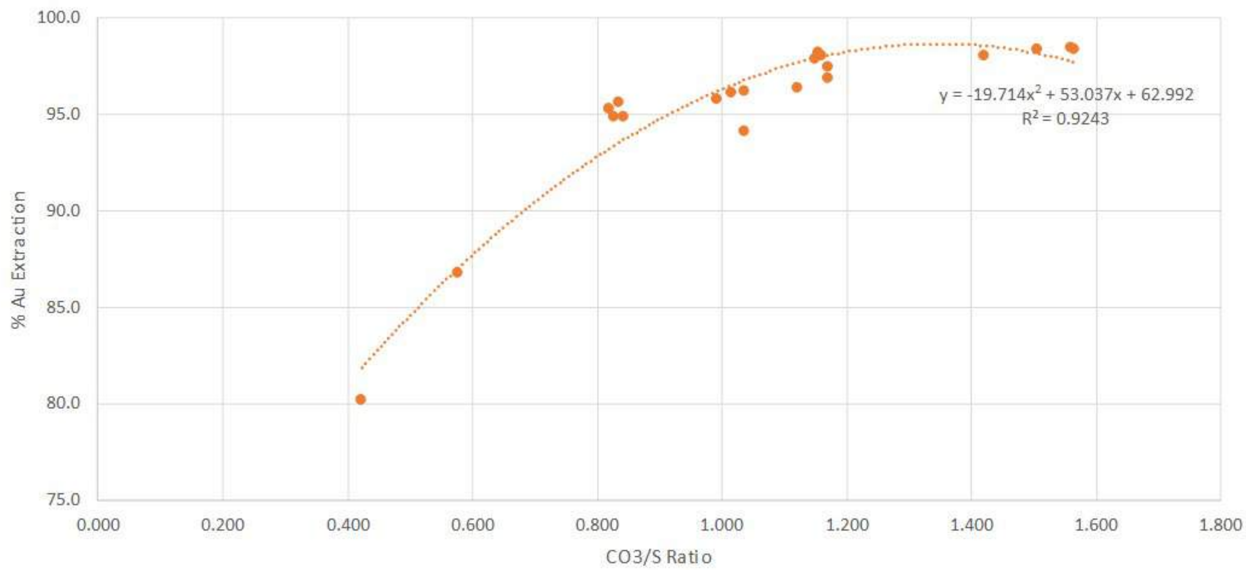
The CSIRO QXRD mineralogy results suggested that:

- iron was precipitated as iron (III) hydroxide (or ferrihydrite), and
- arsenic was precipitated predominantly as scorodite.

For non-iron species, the QEMSCAN values should be considered in Table 13-33. All the QXRD and QEMSCAN values should be considered with an understanding of the limitations of the mineralogical equipment employed.

The hot acid cure (**HAC**) or Pressure Oxidative Leach washed filter cakes collected during the pilot plant campaign (including the profile samples) were leached using cyanide to determine the gold extraction. Figure 13-32 shows the effect of varying gross CO_3/S ratio (in situ + applied) on CIL gold extraction. There appeared to be an increasing gold extraction at higher CO_3/S ratio up to a value of 1.2 where after any further increase in CO_3/S ratio appeared to be minimal.

Figure 13-32: Effect of Varying Gross CO_3/S Ratio on CIL Gold Extraction



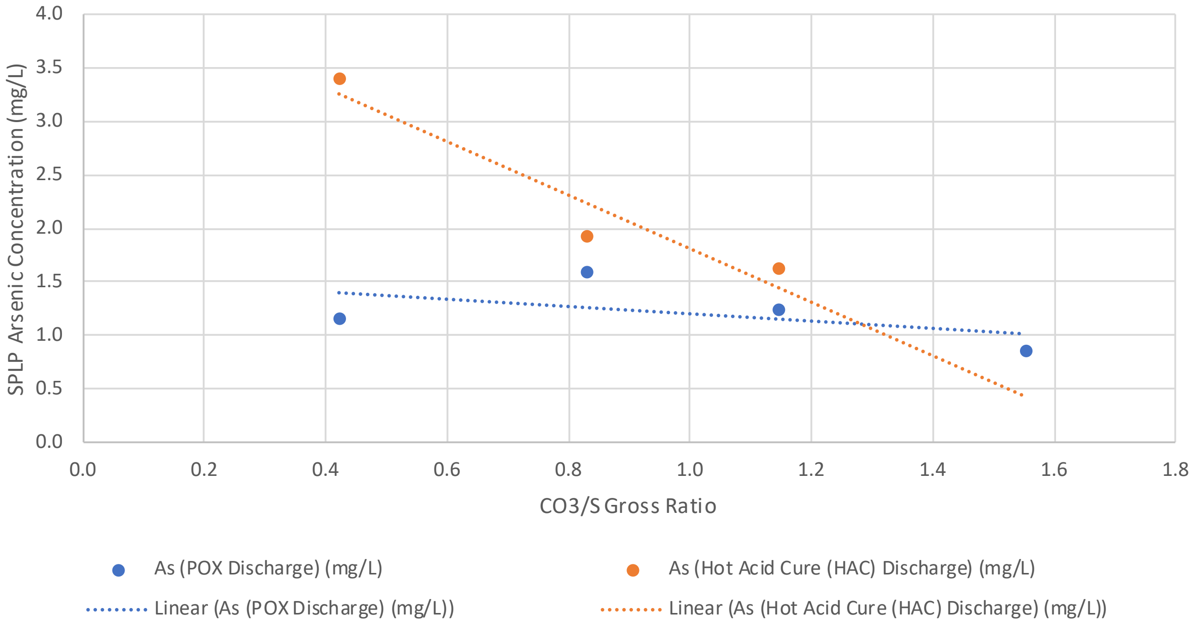
The effect of varying CO_3/S ratio in the pressure oxidative leach autoclave on the SPLP of the pressure oxidative or hot cure residue was examined. A characterization leaching test using EPA Synthetic Precipitation Leaching Procedure (SPLP 1312) was conducted on the profile samples for the four CO_3/S ratios campaigns in the Pressure Oxidative Leach Pilot Plant. A summary of the SPLP test results is shown in Table 13-36.

Table 13-36: Summary of Pilot Plant SPLP Results

Sample	CO_3/S (*In situ* + Added)	SPLP Concentration (mg/L)						
		SO_4	As	Al	Cd	Pb	Cu	Hg
No Profile - POX C4	0.42	1176	1.15	1	0.002	<0.02	0.10	<0.02
No Profile – Hot Acid Cure Discharge	0.42	1608	3.40	2	<0.002	<0.02	0.26	<0.02
Profile 2 - POX Discharge	0.83	1983	1.59	7	0.004	<0.02	0.45	<0.02
Profile 2 - Hot Acid Cure Discharge	0.83	1983	1.92	2	0.004	<0.02	0.10	<0.02
Profile 4 - POX Discharge	1.15	1935	1.24	<1	0.003	<0.02	0.05	<0.02
Profile 4 - Hot Acid Cure Discharge	1.15	1938	1.63	1	0.003	<0.02	0.07	<0.02
Profile 6 - POX Discharge	1.6	1965	0.85	<1	0.004	<0.02	0.14	<0.02

All samples tested produced low-level metal concentrations. However, in the case of arsenic, increasing the CO_3/S ratio appears to favor lower SPLP values and hence improved arsenic stability in the leach residues (refer to Figure 13-33).

Figure 13-33: Effect of Varying Gross CO_3/S Ratio on SPLP Arsenic Concentration



The solubility of arsenic 5+ in the aqueous phase was similar to that of iron. Arsenic precipitates produced at low pH and at elevated temperatures within the autoclave were stable but not when precipitated at high pH and external to the autoclave (refer to Figure 13-34 and Figure 13-35).

Figure 13-34: Arsenic Concentration in Hot Acid Cure and Pressure Oxidative Leach Discharge Residue and Liquor at Varying Gross CO_3/S Ratio

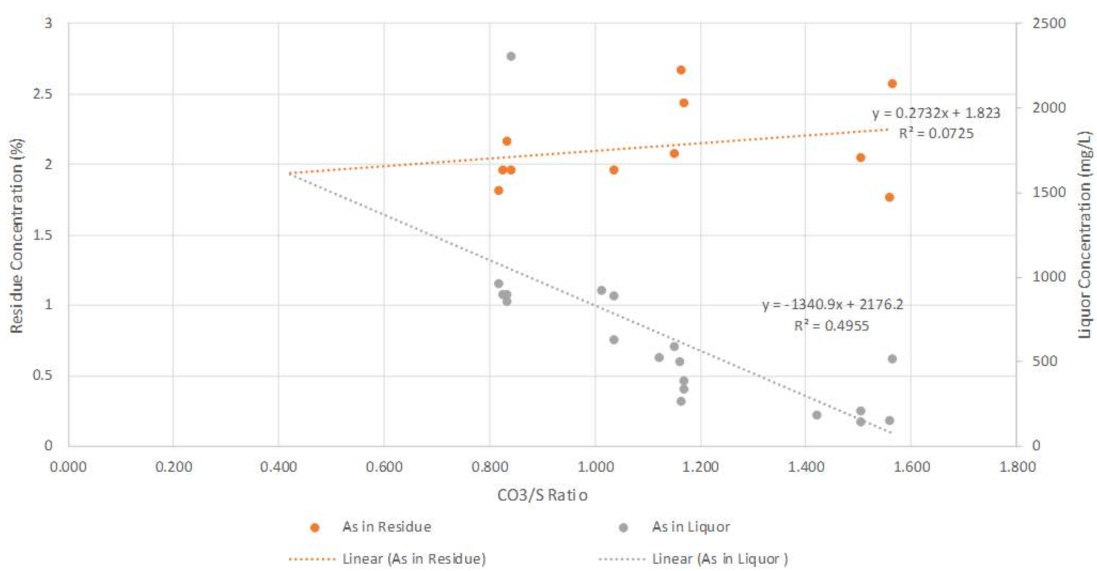
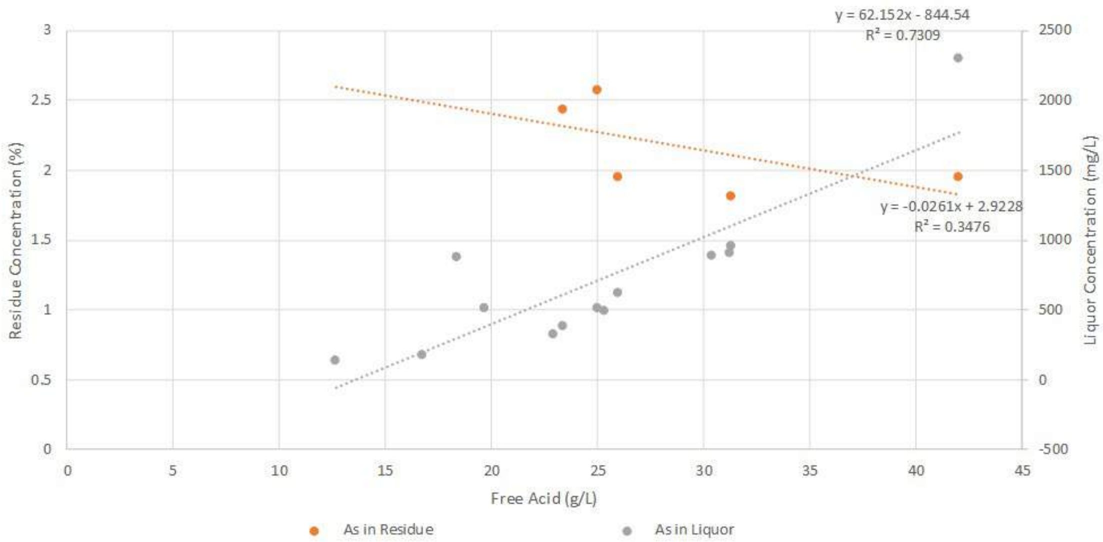


Figure 13-35: Arsenic Concentration in Hot Acid Cure and Pressure Oxidative Leach Discharge Residue and Liquor at Varying Free Acid Concentration



13.9.3.1 Summary of POX Testwork

The 2016 pilot test work at AuTec was undertaken with pre-acidulation of the autoclave feed slurry. The residue from the autoclave was cream-yellow in color. The free acidity in this discharge liquor was 30+g/L H_2SO_4. The dominant arsenic mineral in the residue was found to be pitticite. In the discharge, the arsenic concentrations were found to be 3-6 g/L and more appeared in the hot acid cure step. In the 2017 autoclave pilot plant at SGS, a partial acid neutralization (*in-situ* acid neutralization) was conducted in the autoclave and the free acid was reduced to approximately 8 to 15 g/L. With this partial acid neutralization, the arsenic levels were reduced to 0.2-0.3 g/L. Mineralogy conducted at CSIRO (Perth) confirmed a reduction in pitticite and an increase in scorodite (a more stable mineral containing arsenic). The partially neutralized autoclave discharge slurry was distinctly light brown in color. The arsenic concentration in the POX autoclave leachate with and without *in-situ* acid neutralization is shown in Table 13-37.

Table 13-37: Arsenic in Leachate

Parameters	Units	Autoclave – *In-situ* Acid Neutralization	
		None	Partial
Free Acid (g/L)	g/L	30+	10 - 13
Soluble Arsenic (g/L)	g/L	3 - 6	0.2 – 0.3
SPLP Arsenic (mg/L)	mg/L	4	0.68

The *in-situ* acid neutralization in the POX autoclave was modelled and a summary of the SysCAD output is shown in Table 13-38.

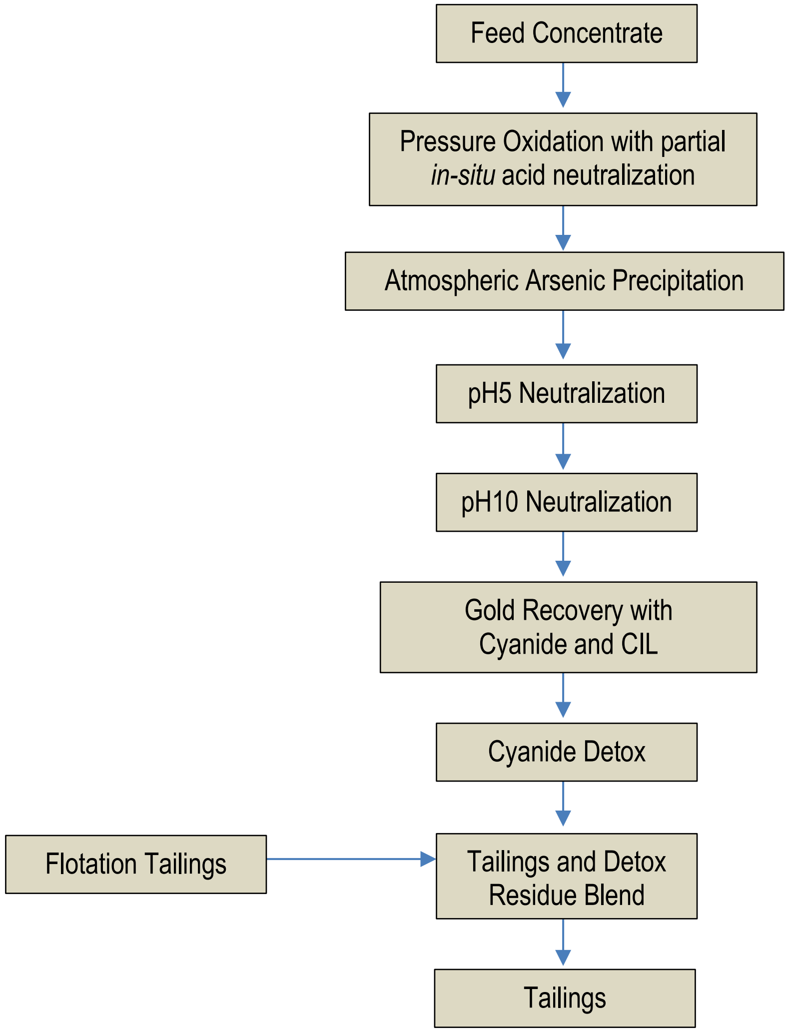
Table 13-38: Summary of Autoclave Conditions

Parameters	Units	Value
Feed Flow Concentrate	dry t/h	154.0
Feed Flow Slurry	t/h	450.7
Temperature Comp#1/Comp#3/Comp#5	°C	215/ 222/ 222
Limestone to Autoclave	kg/t concentrate	135.2
Partial Oxygen Pressure	kPa	700
Vessel Total Pressure	kPa(g)	4100
Vapor Phase Oxygen Content	% of Total	20
Quench Addition	m^3/h	22.0
H_2S in Vent (STP)	mg/m^3	25
Vent Gas Flow	t/h	25.5
Composition of Autoclave Discharge		
H_2SO_4	g/L	10.9
As	g/L	0.207
Al	g/L	0.211
Cu	g/L	0.105
Fe	g/L	1.26
K	g/L	0.140
Ni	g/L	0.016
P	g/L	0.018
S	g/L	8.9
Sb	g/L	0.0002
Si	g/L	0.574
Zn	g/L	0.071

13.9.4 Arsenic Stability Investigation (2020)

The stability of arsenic was a concern flowing out of the 2018 metallurgical product environmental geochemical results. A testwork program was initiated at SGS commencing April 2020 to examine where arsenic destabilization occurred. The testwork flowsheet is given on Figure 13-36.

Figure 13-36: Arsenic Stability Investigation (2020) Testwork Flowsheet



13.9.4.1 Pressure Oxidation

The feed to the atmospheric arsenic precipitation step was derived from three batch pressure oxidation leach tests that were undertaken in the 2020 testwork program employing the conditions shown in Table 13-39. Concentrate 10 that remained after the 2017/18 testwork program was employed as the feed for the tests and the work was carried out at conditions similar to that in the 2017/2018 pilot campaign; namely: 35% solids, 220°C and 3015 kPa (g). The gross CO_3/S mass ratio was varied to examine its impact on gold extraction and the leach and downstream solute compositions. Pressure oxidation leach tests (393.02-POX 1 to 3) embraced a partial *in-situ* acid neutralization with terminal free acid of 8 – 13 g/L of H_2SO_4.

Table 13-39: Summary of 2020 Batch Pressure Oxidative Leach

Parameters			Units	393.02. POX1	393.02. POX2	393.02. POX3
Conc. Type				Con 10	Con 10	Con 10
Test Objective				Initial Test	Free Acid 8-13 g/L	Free Acid 8-13 g/L
Temperature				220	220	220
Feed Density				35	30	30
Sulfur Conc. in Feed				6.895	6.895	6.895
CO_3/S Mass Ratio *In situ* within Conc.				0.28	0.28	0.28
Effective CO_3/S Mass Ratio Applied to Conc.				1.21	0.93	0.93
Calculated Gross CO_3/S Mass Ratio (Applied + *In situ*)				1.49	1.2	1.2
SPLP in Final Sample Extraction Solution		As	mg/L	1.3	0.37	0.37
		Hg	mg/L	<0.02	<0.02	<0.02
		Sb	mg/L	0.082	0.014	0.017
		Se	mg/L	<0.1	<0.1	<0.1
Pressure Oxidative Leach 120 min	Residue	Fe	%	6.52	7.35	7.53
		S^{2-}	%	0.12	0.02	0.02
	Aqueous	As	mg/L	129.5	36.7	64.1
		Fe	mg/L	255	172	158
		Hg	mg/L	1.86	2.19	2.88
		Sb	mg/L	0.06	0.09	0.11
		Se	mg/L	BDL	BDL	BDL
		Free Acid	g/L	8.21	11.5	7.7

The POX autoclave discharge slurry was employed as the feed to the atmospheric arsenic precipitation.

13.9.4.2 Atmospheric Arsenic Precipitation (**AAP**)

This process step precipitated iron and arsenic slowly at an elevated temperature (92°C) by progressively adding limestone to achieve a pH of approximately 2. It is thought that under these conditions, a stable scorodite precipitate ($FeAsO_4.2H_2O$) would form instead of the less stable pitticite precipitate ($Fe.(AsO_4).(SO_4).H_2O$) and calcium arsenate. Possible precipitation reactions are as follows:

Scorodite precipitation: $4H_3AsO_4(aq) + 2Fe_2(SO_4)_3(aq) + 8H_2O(l) \rightarrow 4FeAsO_4.2H_2O(s) + 6H_2SO_4(aq)$

Pitticite precipitation (unbalanced equation): $H_3AsO_4(aq) + Fe_2(SO_4)_3(aq) + H_2O \rightarrow Fe.(AsO_4).(SO_4).H_2O(s) + H_2SO_4(aq)$

Calcium arsenate precipitation: $2H_3AsO_4(aq) + 3CaCO_3(s) \rightarrow Ca_3(AsO_4)_2(s) + 3H_2O(l) + 3CO_2(g)$

The atmospheric arsenic precipitation tests were undertaken in a temperature controlled 5L closed top reactor with an overhead agitator and vent condenser. Some oxygen was thought to be required but was discontinued when all the soluble iron was found to be in the ferric form. The test conditions employed a slurry density of 30 to 35% solids, 92°C with limestone powder being progressively added to the slurry to achieve a ramped pH of approximately 1.5 to 2.0.

The test conditions and results of the batch atmospheric arsenic precipitation are shown in Table 13-40.

Table 13-40: Atmospheric Arsenic Precipitation Test Summary

Parameters		Units	393.02.POX1. AAP1	393.02.POX2. AAP1	393.02.POX2. AAP2	393.02.POX2/3. AAP3
Feed Source			393.02.POX1 Final Slurry	393.02.POX2 Final Slurry	393.02.POX2 Final Slurry	393.02.POX02/3 Final Slurry
Test Objective			Sighter Test - AAP	AAP at pH 1.5 - 2 and 92°C	AAP at pH 2 and shorter residence time	Same conditions as 393.02.POX2.AAP2
Temperature		°C	85	92	92	92
pH Target			1.5-2.5	1.5–2.0	2	2
Reagent			$CaCO_3$	$CaCO_3$	$CaCO_3$	$CaCO_3$
Retention Time		minutes	345	330	300	300
Feed Solution	As	mg/L	130	37	37	59
	Hg	mg/L	1.86	2.2	2.2	2.77
	Sb	mg/L	0.068	0.097	0.097	0.112
	Se	mg/L	BDL	BDL	BDL	BDL
Final Solution	As	mg/L	1.85	3.4	3.96	6.22
	Hg	mg/L	0.18	BDL	0.04	BDL
	Sb	mg/L	0.066	0.029	0.039	0.054
	Se	mg/L	0.1	BDL	BDL	0.1
Final Solid	As	%	2.2	2.52	2.42	2.53
	Hg	ppm	3	8.8	8.8	8.3
	Sb	ppm	1183	1944	1720	1868
	Se	Ppm	BDL	BDL	BDL	13

The assay of key elements across the atmospheric arsenic precipitation is provided in Table 13-41.

- Arsenic was reduced from 66 to approximately 4 mg/L.
- Antimony was reduced from 0.09 to 0.05 mg/L.
- Mercury was reduced from 2.3 to 0.11 mg/L.
- Aluminum was reduced from 129 to 38 mg/L.
- Potassium was reduced from 27 to 18 mg/L.
- Total sulfur was reduced from 4,993 to 2,385 mg/L in line with the removal of matching cations.
- Iron was reduced from 190 to 7 mg/L.
- Free sulfuric acid was reduced from approximately 10 g/L in the feed to approximately 0.6 g/L in the discharge.

Table 13-41: Elemental Assay Across the Atmospheric Arsenic Precipitation Step

Parameters		Atmospheric Arsenic Precipitation Feed					Atmospheric Arsenic Precipitation Discharge				
		393.02. POX1. AAP1-N1	393.02. POX2. AAP1	393.02. POX2. AAP2	393.02. POX2/3. AAP3	Average	393.02. POX1. AAP1-N1	393.02. POX2. AAP1	393.02. POX2. AAP2	393.02. POX2/3. AAP3	Average
pH							2.05	2.02	2.59	1.95	
Free Acid H_2SO_4) g/L		9.45	6.7	6.7	0		0.58	0.62	0.62	0.21	
Aqueous Assay (mg/L)	Ag	0.1	0.16	0.16	BDL	**0.14**	0	0.06	BDL	BDL	**0.08**
	As	130	37	37	59	**66**	2	3.4	4	6.2	**4**
	Cu	84	954	954	732	**681**	26	720	647	604	**499**
	Fe	255	172	172	160	**190**	4	7	6	11	**7**
	Ni	7	1880	1880	1140	**1227**	4	1700	1690	1100	**1123**
	Zn	54	65	65	62	**62**	23	56	52	57	**47**
	Al	181	99	99	135	**129**	22	33	BDL	60	**38**
	Mg	1150	1090	1090	1012	**1086**	535	974	911	953	**843**
	Na	28	7	7	2	**11**	39	8	2	3	**13**
	Ca	590	497	497	475	**515**	414	511	496	494	**479**
	Si	329	335	335	393	**348**	156	214	200	224	**199**
	K	60	21	21	6	**27**	32	13	8	19	**18**
	P Note 1	N/A	N/A	N/A	N/A	**N/A**	1.1	3.8	7.2	7.8	**5**
	Cd	0	0	0	0.1	**0**	BDL	BDL	0	0	**0**
	S(t)	5120	5020	5020	4813	**4993**	1190	2880	2800	2670	**2385**
	Hg	1.9	2.2	2.2	2.8	**2.3**	0.18	BDL	0	BDL	**0.11**
	Cr	1.1	0.6	0.6	0.8	**0.8**	0.2	BDL	BDL	BDL	**0.2**
	Mn	106	124	124	109	**116**	52	131	119	118	**105**
	Pb	BDL	BDL	BDL	0.9	**0.9**	0.7	BDL	BDL	BDL	**0.7**
	Sb	0.07	0.01	0.01	0.01	**0.09**	0.07	0.03	0.04	0.05	**0.05**
	Se	BDL	BDL	BDL	BDL	**BDL**	0.1	BDL	BDL	BDL	**0.1**

Notes: N/A = Not Assessed; BDL = Below Detection Limit

13.9.4.2.1 SPLP Results for the Atmospheric Arsenic Precipitation

Table 13-42 groups all the atmospheric arsenic precipitation SPLP data. The AAP appears to improve the stability of the arsenic and antimony in the residue emanating from the pressure oxidation leach autoclave confirming, it seems, that a stable arsenic compound (possibly scorodite) is produced.

Table 13-42: SPLP of the Final Unwashed Atmospheric Arsenic Precipitation Residue

Parameters	Units	393.02. POX2.AAP1	393.02. POX2.AAP2	393.02. POX2/3.AAP3	393.02. POX1.AAP1-N1	Average
Sulfate, SO_4	mg/L	2100	2013	1851	2043	2002
Arsenic, As	**mg/L**	**0.22**	**0.27**	**0.33**	**0.31**	**0.28**
Aluminum, Al	mg/L	3	2	4	3	3
Selenium, Se	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	0.004	0.004	0.005	<0.002	0.004
Lead, Pb	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02
Chromium, Cr	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1
Manganese, Mn	mg/L	6.7	5.6	4.8	3.9	5.2
Iron, Fe	mg/L	<1	<1	<1	<1	<1
Nickel, Ni	mg/L	68.2	66.6	39.8	0.19	43.7
Copper, Cu	mg/L	30.7	31.9	22.4	1.63	21.7
Zinc, Zn	mg/L	2.1	2.16	2.02	1.09	1.8
Magnesium, Mg	mg/L	43.1	40.8	39.6	31.2	38.7
Sodium, Na	mg/L	18.4	3.2	3.6	14.9	10.0
Potassium, K	mg/L	8	<5	<5	8	8.0
Phosphorus, P	mg/L	1.8	<0.3	<0.3	0.6	1.2
Mercury, Hg	mg/L	<0.02	<0.02	0.03	<0.02	<0.02
Calcium, Ca	mg/L	751	728	669	833	745
Antimony, Sb	mg/L	0.01	0.011	0.016	0.033	0.018

13.9.4.2.2 Summary of Atmospheric Arsenic Precipitation Testwork

The atmospheric arsenic precipitation step was considered as a ''follow-on'' from the partial acid neutralization step in the autoclave in that it provided a further means of attenuating arsenic as scorodite employing residual iron in the autoclave discharge liquor.

Arsenic removal to levels of approximately 5mg/L in the autoclave discharge slurries were achieved with the atmospheric arsenic precipitation and required the following conditions:

- Temperatures above 90 °C,
- Aqueous iron-to-arsenic ratios in excess of 2:1,
- Graded pH profile of between 1.2 and 2 over approximately four agitated tanks,
- Retention time of approximately 4-5 hours, and
- The stability of the atmospheric arsenic precipitation residue from the SPLP arsenic result was very acceptable at 0.28 mg/L.

The key process conditions for the atmospheric arsenic precipitation are provided in Table 13-43.

Table 13-43: Process Conditions in Atmospheric Arsenic Precipitation

Parameters	Units	Value
Temperature	°C	90 – 92
Retention Time	h	4 – 5
pH Range	-	1.2 – 2.0
Iron – to - Arsenic Ratio	-	≥ 2 : 1
Number of Reactors	#	4
Middle Marble Limestone Dose	kg/t concentrate	52.5
Steam Required	kg/t concentrate	65
Slurry Density	% solids	41
Discharge Aqueous Composition		
Fe	g/L	0.13
As	g/L	0.004
Sb	g/L	0.0003
Hg	g/L	0.0001
K	g/L	0.013

13.9.4.3 Batch Neutralization

Batch neutralization was split into two discrete pH regions:

- pH ~5 – with little to no free hydroxide present in the aqueous, and
- pH 9.5-10 in preparation for CIL where there is free hydroxide present.

13.9.4.3.1 Batch Neutralization pH 5 Test Conditions

The conditions and results of batch neutralization testwork employing a termination pH of 5 are shown in Table 13-44. The tests were undertaken in a temperature controlled 2 L closed top reactor fitted with a Rushton impeller. In each test, analytical grade limestone powder was gradually added until the final pH of typically 5 was achieved.

13.9.4.3.2 Batch Neutralization pH 5 Effects of Temperature

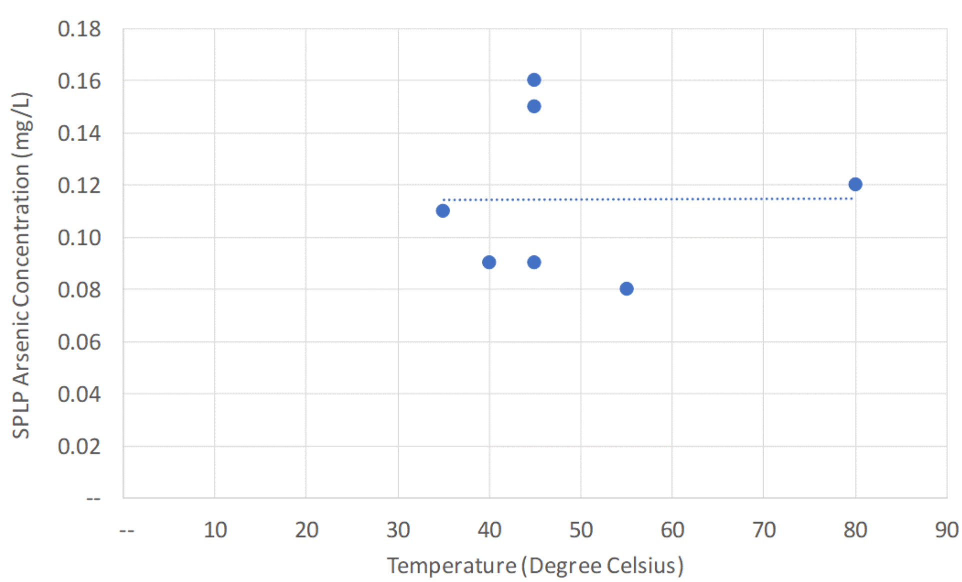
The impact of the neutralization temperature on the SPLP arsenic concentration was investigated. Scatter plot Figure 13-37 suggests that the temperature at which the pH 5 neutralization was conducted has little to no impact on the stability of the arsenic precipitated from both the autoclave and the atmospheric arsenic precipitation as a feed source. Figure 13-37 also confirms that the SPLP arsenic values in the pH 5 neutralization step are acceptably low at <0.2 mg/L.

Table 13-44: Batch Neutralization (pH 5) Test Summary

Parameters		Units	393.02. POX2.AAP1. N1+ CN	393.02. POX2.AAP1. N2+ CIL	393.02. POX2.AAP1. N3	393.02. POX2.AAP1. N4	393.02. POX2.AAP2. N5+CIL	393.02. POX2.AAP2. N6	393.02. POX2/3.AAP3. N1+ CN
Feed Source			POX 2 AAP 1 Slurry	POX 2 AAP 1 Slurry	POX 2 AAP 1 Slurry	POX 2 AAP 1 Slurry	POX 2 AAP 2 Slurry	POX 2 AAP 2 Slurry	POX 2/3 Blend AAP 3 Slurry
Test Objective			Varying Temp	Varying Temp	Varying Temp	Varying Temp	Varying Temp	Varying Temp	Varying Temp
Temperature		°C	45	40	35	55	45	80	45
Terminal pH			5.07	5.87	4.89	5.16	5.37	6.4	5.45
Retention Time		min	~60[Note 1]	~60[Note 1]	~60[Note 1]	~60[Note 1]	~60[Note 1]	60	60
Neutralizing Reagent			$CaCO_3$	$CaCO_3$	$CaCO_3$	$CaCO_3$	$CaCO_3$	$CaCO_3$	$CaCO_3$
Feed Solution	As	mg/L	3	3	3	3	4	4	6
	Hg	mg/L	BDL	BDL	BDL	BDL	0.04	0.04	BDL
	Sb	mg/L	0.029	0.029	0.029	0.029	0.039	0.039	0.054
	Se	mg/L	BDL	BDL	BDL	BDL	BDL	BDL	0
Solution at pH 5	As	mg/L	0.45	0.32	0.35	0.38	0.61	0.56	1.1
	Hg	mg/L	BDL	BDL	BDL	BDL	0.04	0.04	BDL
	Sb	mg/L	0.03	0.02	0.03	0.02	0.02	0.02	0.03
	Se	mg/L	BDL	BDL	BDL	BDL	BDL	BDL	BDL
Final Solid	As	%	2.52	2.49	2.47	2.42	2.39	2.51	2.46
	Hg	ppm	8.1	8.3	8.4	8.4	7.4	8.3	8.3
	Sb	ppm	1850	1956	1892	1804	1653	1746	1732
	Se	ppm	BDL	BDL	BDL	BDL	BDL	BDL	BDL

Note 1: Approximate time when pH 5 samples were taken.

Figure 13-37: SPLP Arsenic Concentration of Neutralized Residue (pH 5) at Varying Neutralization Temperatures



13.9.4.3.3 Impurity Deportment in pH 5 Neutralization

Table 13-45 provides the assay data for the important solute components in the pH 5 neutralization step.

Table 13-45: Elemental Assays in the pH 5 Neutralization Step

Parameters	Test No	pH	Temp	Aqueous Assay (mg/L)								
				As	Cu[Note 1]	Fe	Al	Mg	Si	P	Mn	Sb
Feed Liquor	393.02.POX2.AAP1.N1+CN	2.02		3.4	720	7	33	974	214	3.8	131	0.03
	393.02.POX2.AAP1.N2+CIL	2.02		3.4	720	7	33	974	214	3.8	131	0.03
	393.02.POX2. AAP1.N3	2.02		3.4	720	7	33	974	214	3.8	131	0.03
	393.02. POX2. AAP1.N4	2.02		3.4	720	7	33	974	214	3.8	131	0.03
	393.02.POX2.AAP2.N5+CIL	BDL		4	647	6	BDL	911	200	7.2	119	0.04
	393.02.POX2.AAP2.N6	BDL		4	647	6	BDL	911	200	7.2	119	0
	393.02.POX2/3.AAP3. N1+CN	BDL		6.2	604	11	60	953	224	7.8	118	0.05
Discharge Liquor at pH 5	393.02.POX2.AAP1.N1+CN	5.07	45	0.5	9	1	BDL	932	96	BDL	113	0.03
	393.02.POX2.AAP1.N2+CIL	5.87	40	0.3	3	1	BDL	892	75	BDL	105	0.02
	393.02.POX2.AAP1.N3	4.89	35	0.4	41	2	1	901	94	0.7	115	0.03
	393.02.POX2.AAP1.N4	5.16	55	0.4	7	1	3	890	89	BDL	106	0.02
	393.02.POX2.AAP2. N5+CIL	5.37	45	0.6	34	BDL	BDL	824	106	BDL	100	0.02
	393.02.POX2.AAP2. N6	6.4	80	0.6	1.2	BDL	BDL	979	72	BDL	81	0.02
	393.02.POX2/3.AAP3.N1+CN	5.45	45	1.1	1.88	1	4	908	90	4.7	83	0.03

Note 1: The high copper values in the feed liquor are an artifact of the copper contamination by SGS in poorly cleaned test autoclaves.

- Potassium, copper and zinc were precipitated consistently to less than 10 mg/L
- Iron, aluminum and phosphorus were precipitated consistently to less than 5 mg/L
- Arsenic, antimony, mercury, cadmium, chromium and selenium precipitated consistently to less than 1 mg/L

13.9.4.3.4 SPLP Results for the pH 5 Neutralization

The SPLP result for the final unwashed pH 5 neutralized residue is shown in Table 13-46. The results show that acceptable levels of arsenic (0.18 mg/L), chromium (<0.1 mg/L), mercury (<0.02 mg/L) and antimony (0.0097 mg/L) were achieved.

13.9.4.3.5 Batch Neutralization pH 10 Test Conditions

The test conditions and results of the batch neutralization testwork at pH 9.5 - 10 are shown in Table 13-47. All the tests were undertaken in a temperature controlled 2L closed top reactor fitted with a Rushton impeller. The feed slurries were derived from a prior pH 5 Neutralization in which limestone had been employed to adjust the pH. The pH adjustment employed dry slaked lime ($Ca(OH)_2$) powder with additions taking place slowly allowing sufficient time for the kinetically impaired hydrolysis reactions to take place.

The objective of these tests, besides providing a slurry feed for CIL, was to understand how the pH adjustment conditions impacted the stability of arsenic.

The results show that arsenic is solubilized with increasing pH and retention time. Temperature did not seem to affect arsenic stability within the range tested (35 to 55°C). With a constant retention time of 180 minutes, dissolved arsenic levels increased with pH, from 0.2 mg/L As to 0.81 mg/L As over a pH range of 9.5 to 10.4. Dissolved arsenic also increased with retention time, from 0.68 to 1.91 when the retention time increased from 180 minutes to 340 minutes.

Table 13-46: SPLP of pH5 Batch Neutralization Residues

Parameters	Units	393.02. POX 2. AAP1.N1 (45°C)	393.02. POX2. AAP1.N2 (40°C)	393.02. POX2. AAP1.N3 (35°C)	393.02. POX2. AAP1.N4 (55°C)	393.02. POX2. AAP2.N5 (45°C)	393.02. POX2. AAP2.N6 (80°C)	393.02. POX2/Pox 3.AAP3.N1	393.02 POX1. N1 (80°C)	Average
Sulfate, SO_4	mg/L	2073	2061	2046	2025	1306	1908	1785	1827	1879
Arsenic, As	mg/L	0.09	0.09	0.11	0.08	0.16	0.12	0.15	0.65	0.18
Aluminum, Al	mg/L	1	<1	<1	<1	<1	<1	<1	<1	<1
Selenium, Se	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	<0.002	<0.002	0.002	<0.002	0.003	<0.002	<0.002	<0.002	0.003
Lead, Pb	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Chromium, Cr	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Manganese, Mn	mg/L	6.77	6.28	6.65	6.79	5.23	5.7	4.97	0.25	5.3
Iron, Fe	mg/L	<1	<1	<1	<1	<1	<1	<1	<1	<1
Nickel, Ni Note 1	mg/L	64.219	58.097	63.989	64.879	57.14	40.43	31.03	0.04	47
Copper, Cu Note 1	mg/L	2.062	0.967	7.362	1.35	7.93	0.14	0.4	<.05	3
Zinc, Zn	mg/L	1.42	1.03	1.88	1.24	1.79	0.3	0.7	<.05	1
Magnesium, Mg	mg/L	43.2	40.6	41.2	42	38.9	45.4	41.9	10.4	38
Sodium, Na	mg/L	18.8	18.9	19.5	18.4	2.7	2.9	3.6	4.2	11
Potassium, K	mg/L	7	7	8	6	<5	<5	<5	<5	7
Phosphorus, P	mg/L	0.5	0.5	0.5	<0.3	<0.3	<0.3	<0.3	<0.3	1
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	0.03	<0.02	<0.02
Calcium, Ca	mg/L	772	785	760	766	731	723	653	770	745
Antimony, Sb	mg/L	0.008	0.007	0.01	0.006	0.01	0.013	0.011	0.013	0.0097
Note 1: The high nickel and copper values in the feed liquor are an artefact of the nickel and copper contamination by SGS in poorly cleaned test autoclaves.										

Table 13-47: Batch Neutralization (pH 9.5-10) Test Summary

Parameters	Units	393.02. POX2.AAP1 .N1+CN	393.02. POX2.AAP1 .N2+CIL	393.02. POX2.AAP1 .N3	393.02. POX2.AAP1 .N4	393.02. POX2.AAP2 .N5+CIL	393.02. POX2/3. AAP3.N1A +CN	393.02. POX2/3. AAP3.N1B +CN	393.02. POX2/3. AAP3.N1C +CN	393.02. POX2,3. N2A (Sighter)
Feed Source		POX 2 AAP1 Slurry	POX 2 AAP1 Slurry	POX 2 AAP1 Slurry	POX 2 AAP1 Slurry	POX 2 AAP2 Slurry	POX 2/3 Blend AAP3 Slurry	POX 2/3 Blend AAP3 Slurry	POX 2/3 Blend AAP3 Slurry	POX 2/3 Blend Slurry
Test Objective Variable		Impact of Temperature	Impact of Temperature	Impact of Temperature	Impact of Temperature	Impact of Temperature	Retention Time and Target pH 9.5	Retention Time and Target pH 9.6	Retention Time and Target pH 9.8	Sighter Test
Temp	°C	45	40	35	55	45	45	45	45	45
Terminal pH		9.51	9.9	9.82	10.4	10.4	9.6	9.8	9.88	9.6
Retention Time	Minutes	180	180	180	180	120	340	240	180	120
Neutralizing Reagent		$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$	$Ca(OH)_2$
Feed Solution at pH 5 (mg/L)	As (mg/)	0.45	0.32	0.35	0.38	0.61	1	1	1	
	Hg (mg/)	BDL	BDL	BDL	BDL	0.04	BDL	BDL	BDL	
	Sb (mg/)	0.03	0.02	0.03	0.02	0.02	0.025	0.025	0.025	
	Se (mg/)	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	
Solution at pH 9.5-10 (mg/L)	As (mg/)	0.2	0.31	0.42	0.81	0.61	1.91	1.54	0.68	5.5
	Hg (mg/)	BDL	BDL	BDL	BDL	0.04	BDL	BDL	BDL	BDL
	Sb (mg/)	0.041	0.037	0.049	0.054	0.033	0.06	0.06	0.069	0.059
	Se (mg/)	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL

Parameters	Units	393.02. POX2.AAP1 .N1+CN	393.02. POX2.AAP1 .N2+CIL	393.02. POX2.AAP1 .N3	393.02. POX2.AAP1 .N4	393.02. POX2.AAP2 .N5+CIL	393.02. POX2/3. AAP3.N1A +CN	393.02. POX2/3. AAP3.N1B +CN	393.02. POX2/3. AAP3.N1C +CN	393.02. POX2,3. N2A (Sighter)
Final Solid	As (%)	2.52	2.49	2.47	2.42	2.39	2.35	2.54	2.41	2.6
	Hg (ppm)	8.1	8.3	8.4	8.4	7.4	7.7	7.6	7.7	2.6
	Sb (ppm)	1850	1956	1892	1804	1653	1718	1824	1747	1811
	Se (ppm)	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL

13.9.4.3.6 Effects of Temperature

Raising the pH above 5, where free alkalinity was present, destabilized the arsenic in the residue and resulted in the release of soluble arsenic in the 2018 geochemical tests. In gold CIL, free alkalinity is required to stabilize the cyanide. Because the activity of free hydroxyl ions increases with temperature, and in view of the limited test time available to explore the relationship between temperature and arsenic stability at elevated pH, all ensuing testwork was made at temperatures in the 35 to 45°C range.

Several tests (POX 2 Atmospheric Arsenic Precipitation 1 Neutralization Tests 1, 2, 3 and 4) were made to explore the impact of temperature on the stability of arsenic in residue. All of these tests employed the POX 2 Atmospheric Arsenic Precipitation 1 Final Slurry. The retention time was 180 minutes.

13.9.4.3.7 Impurity Deportment in pH 10 Neutralization

The neutralization pH of 9.5 to 10.0 resulted in lower concentrations of most solute elements (Table 13-48).

- Arsenic and antimony were precipitated to concentrations below 2 and 0.06 mg/L respectively.
- Phosphorus was precipitated to concentrations below 4 mg/L.
- All the base metals were reduced to below 14 mg/L in total.
- Magnesium was reduced to below 96 mg/L.

Table 13-48: Solute Concentration at Terminal pH of 9.5-10

Test No.	Terminal pH	Concentration of Discharge Liquor (mg/L)								
		As	Cu	Fe	Al	Mg	Si	P	Mn	Sb
393.02.POX2 .AAP1.N1+CN	9.51	0.2	0.13	1	BDL	95.5	6	BDL	0.5	0.04
393.02.POX2. AAP1.N2+CIL	9.9	0.31	0.18	1	BDL	42.4	7	BDL	0.4	0.04
393.02.POX2 .AAP1.N3	9.82	0.42	0.69	4	2	32.9	8	0.5	2.3	0.05
393.02.POX2. AAP1.N4	10.36	0.81	BDL	1	2	4.9	22	BDL	0.3	0.05
393.02.POX2. AAP2.N5+CIL	10.38	0.61	0.14	BDL	4	3.1	11	2.9	0.1	0.03
393.02.POX2/3.AAP3.N1A+CN	9.6	1.91	BDL	1	4	5.3	33	3.5	0.2	0.06
393.02.POX2/3.AAP3.N1B+CN	9.8	1.54	0.3	BDL	2	4.4	26	3.7	BDL	0.06
393.02.POX2/3.AAP3.N1C+CN	9.88	0.68	0.05	BDL	BDL	13.7	14	0.6	BDL	0.07
393.02.POX2,3.N2A (Sighter)	9.6	5.5	BDL	1	BDL	3.8	20	BDL	0.2	0.06

13.9.4.3.8 Destabilization of Arsenic

Figure 13-38 provides a plot of SPLP against neutralization temperature for the pH 5, pH 8.5 and pH 9.5-10 tests. Elevated pHs required to support the CIL testwork appear to destabilize arsenic in the gold containing residues even at temperatures as low as 45°C.

13.9.4.3.9 SPLP Results for pH 10 Neutralization

The SPLP results for the final unwashed pH 9.5-10 neutralized residues are provided in Table 13-49. The results clearly demonstrate elevated SPLP arsenic values for most of the pH 9.5-10.0 neutralization tests.

Figure 13-38: Comparison of SPLP Arsenic Concentration at Varying Neutralization pH

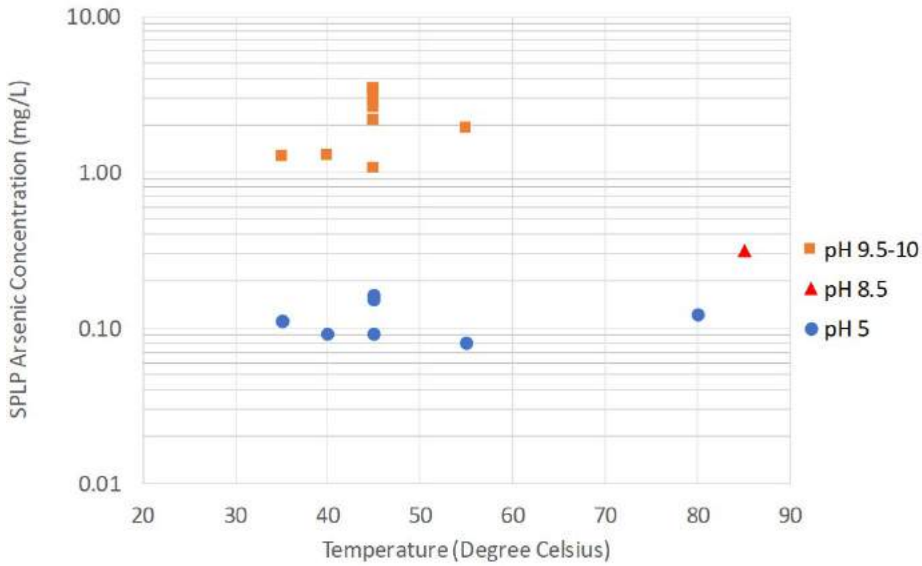


Table 13-49: SPLP of pH9.5 - 10 Batch Neutralization Residues

Parameters	Units	393.02. POX2. AAP1. N1 (45°C)	393.02. POX2. AAP1. N2 (40°C)	393.02. POX2. AAP1.N3 (35°C)	393.02. POX2. AAP1.N4 (55°C)	393.02. POX2. AAP2. N5 (45°C)	393.02. POX2/3. AAP3. N1a (45°C)	393.02. POX2/3. AAP.3. N1b (45°C)	393.02. POX2/3. AAP3. N1c (45°C)	393.02 POX1.N1 (80°C)	Average
Final Neut pH		9.76	9.70	9.76	10.07	10.23	9.23	9.30	9.37	8.0	
Sulfate, SO_4	mg/L	1590	1533	1563	1521	1758	1329	1335	1386	1740	1528
Arsenic, As	mg/L	1.05	1.28	1.24	1.88	2.57	2.11	2.93	3.35	3.32	2.19
Aluminum, Al	mg/L	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Selenium, Se	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002
Lead, Pb	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Chromium, Cr	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Manganese, Mn	mg/L	0.02	0.02	0.02	0.01	0.02	0.02	0.01	0.02	0.02	0.018
Iron, Fe	mg/L	<1	<1	<1	<1	<1	<1	<1	<1	<1	<1
Nickel, Ni	mg/L	0.133	0.102	0.08	0.051	<.05	0.08	<.05	<.05	<.05	0.072
Copper, Cu	mg/L	0.013	0.019	0.011	0.015	<.05	<.05	<.05	<.05	<.05	0.034
Zinc, Zn	mg/L	<0.05	<0.05	<0.05	<0.05	<.05	<0.05	<0.05	<0.05	<.05	<0.05
Magnesium, Mg	mg/L	23.2	18.1	17.9	9.7	7.5	9	6..3	5.6	21.9	14.1
Sodium, Na	mg/L	18.8	19	18.6	18.4	2.9	5	4.5	4.5	14.5	11.8
Potassium, K	mg/L	7	7	7	7	<5	<5	<5	<5	6	6.8
Phosphorus, P	mg/L	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	<0.3	0.4	0.4
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	0.02	<0.02	<0.02	<0.02	<0.02
Calcium, Ca	mg/L	651	646	642	643	778	562	569	589	727	645
Antimony, Sb	mg/L	0.024	0.028	0.025	0.032	0.028	0.031	0.034	0.036	0.032	0.03

13.9.4.3.10 Summary of Neutralization Testwork

Neutralization to pH of 5 with limestone resulted in no arsenic destabilization with SPLP arsenic values consistently below 0.2 mg/L. Temperatures in the range of 35 to 80°C had no impact on the SPLP with pHs up to 5.0. The retention time in pH 5.0 neutralization was typically 0.5 hours.

Neutralization employing lime to raise the pH to approximately 9.5 -10 in preparation for gold recovery was where arsenic destabilization was noticed. It was postulated to be related to the reaction between free hydroxyl ions and remaining pitticite and so consequently, the neutralization temperature was reduced to approximately 45°C. The retention time at pH 10 was typically 0.5 to 1h. Arsenic in SPLP increased significantly and was measured in the 1.05 to 3.35 mg/L range.

Slurry Cooling Towers were considered for cooling the slurry prior to the pH 10 neutralization step. No slurry cooling testwork has been done, however testwork for this circuit should be considered prior to project implementation.

The process conditions in neutralization are given in Table 13-50.

Table 13-50: Process Conditions in Neutralization

Parameters	Units	Neutralization Stage pH	Value
Temperature	°C	5	≤ 80
	°C	10	≤ 45
Reagent	-	5	Middle Marble Limestone
	-	10	Lime
Retention Time	h	5	≤ 0.5
	h	10	~ 1.0 Note 1
Slurry SG	% solids	Feed to Neutralization	~41.2
	% solids	5	~ 45.1
	% solids	10	~ 45.5
Slurry Cooling Drift Loss	%	5	~ 0.002
Solids	kg/h	5	~14
Evaporation	t/h	5	~26
Cooling Range	°C	5	~15 (estimate)
Terminal Temperature	°C	5	~60
Note 1: Controlled Adjustment required.			

13.9.4.4 Cyanidation Tests

13.9.4.4.1 Test Conditions

Activated carbon (**CIL**) was employed in the batch cyanide leach tests where the gold recovery was required.

The CIL test conditions and results are shown in Table 13-51. The CIL test conditions were:

- An initial sodium cyanide concentration of 1.5 g/L was employed in all tests except for Test 393.02.POX2.AAP2.N5 where the initial sodium cyanide concentration was lower at 1.0 g/L. In all tests, the cyanide concentration drifted lower during the leach.
- The CIL pH was maintained at approximately 9.8-10 with the addition of lime powder.
- The dissolved oxygen was maintained greater than 20 ppm.

- A 20 g/L activated carbon charge was added in all CIL tests.
- The temperature was maintained at 40°C.
- The initial slurry density was variable between 28 to 35%w/w solids.
- The leach time in all cases was 24 hours.

Table 13-51: Summary of CIL Testwork

Parameters		Units	393.02. POX4.CIL	393.02. POX4.CIL2	393.02. POX4.CIL	393.02. POX2. AAP1.N2	393.02. POX2. AAP2.N5
Feed Type			POX 4 Final Slurry	POX 4 Final Slurry	POX 5 Final Slurry	POX 2 AAP 1 Neutralization 2 Slurry	POX 2 AAP 1 Neutralization 5 Slurry
Initial NaCN		g/L	1.5	1.5	1.5	1.5	1
NaCN at 24h		g/L	0.4	0.6	0.6	0.3	0.03
pH at 24h			9.5	9.6	9.6	9.8	9.6
Temperature		°C	40	40	40	40	40
Dissolved Oxygen		ppm	>20	>20	>20	>20	>20
Carbon Conc.		g/L	20	20	20	20	20
Feed Slurry Density		% Solids	34	30	35	28	30
Reagent Consum	Cyanide	kg/t	2.4	2.6	1.7	5.2	4.1
	Lime	kg/t	17.2	33.2	22	8.2	--
Au Extraction		g/t	12.3	13	9	10	10
		%	98.2	98.4	97.6	96.7	93.5 Note 1
Ag Extraction		g/t	2.1	3	1	0	NR
		%	14.2	27	10	3.4	NR
Calc. Head	Au	g/t	12.5	13.3	9.6	10.8	10.8
	Ag	g/t	14.8	12.3	11.1	10.4	NR

Note 1: The gold extraction was low as a result of low sodium cyanide concentration during leach. The sodium cyanide concentration fell to 0.1g/L after 4

13.9.4.4.2 SPLP Results for CIL Tests

The SPLP results for the final washed CIL residue are shown in Table 13-52. Arsenic values generally exceed the "cut off" value of 2.0mg/L thus confirming that the destabilization of arsenic that commenced in the pH10 neutralization persistent in CIL.

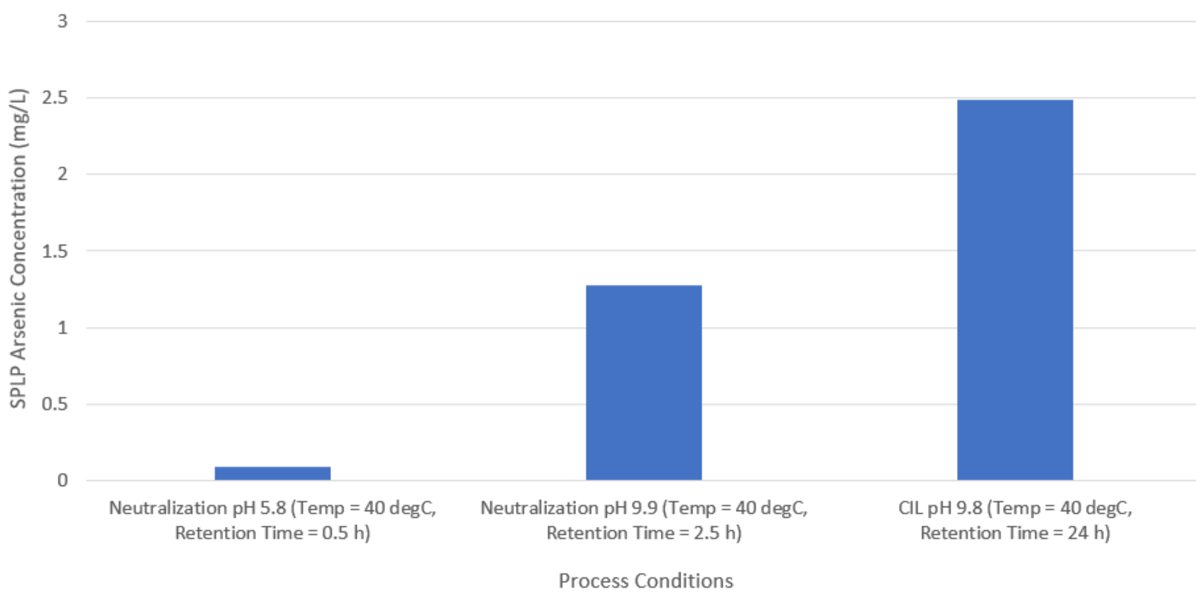
Table 13-52: SPLP of Final Washed CIL Residue (after Carbon Removal)

Parameters	Units	393.02. POX2. AAP2.N5. CIL	393.02. POX2. AAP1.N1. CN	393.02. POX2/3. AAP3.N1a. CN	393.02. POX2/3. AAP3.N1b. CN
Sulfate, SO_4	mg/L	1389	1407	1326	1341
Arsenic, As	**mg/L**	**2.98**	**4.07**	**4.14**	**4.69**
Aluminum, Al	mg/L	<1	<1	1	<1
Selenium, Se	mg/L	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	<0.002	<0.002	<0.002	<0.002
Lead, Pb	mg/L	<0.02	<0.02	<0.02	<0.02
Chromium, Cr	mg/L	<0.1	<0.1	<0.1	<0.1
Manganese, Mn	mg/L	0.02	0.02	<0.01	<0.01
Iron, Fe	mg/L	<1	<1	<1	<1

Parameters	Units	393.02. POX2. AAP2.N5. CIL	393.02. POX2. AAP1.N1. CN	393.02. POX2/3. AAP3.N1a. CN	393.02. POX2/3. AAP3.N1b. CN
Nickel, Ni	mg/L	0.91	4.27	3.5	4.67
Copper, Cu	mg/L	0.32	1.22	0.9	1.38
Zinc, Zn	mg/L	<.05	<0.05	<0.05	<0.05
Magnesium, Mg	mg/L	6.7	6.5	5.7	4.4
Sodium, Na	mg/L	9	15.9	13.3	15.8
Potassium, K	mg/L	6	<5	<5	<5
Phosphorus, P	mg/L	<0.3	<0.3	<0.3	<0.3
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02
Calcium, Ca	mg/L	619	608	579	586
Antimony, Sb	mg/L	0.033	0.033	0.039	0.039

Figure 13-39 provides a typical trend for the destabilization of arsenic across neutralization and CIL. The results are for test 393-02-POX 2-AAP1 residue. The time duration that the residues were exposed to elevated levels of alkalinity could be significant in explaining the SPLP arsenic trend on Figure 13-39.

Figure 13-39: Comparison of SPLP arsenic concentration at Neutralization and CIL



13.9.4.4.3 Summary of CIL Testwork

Table 13-53 summarizes the CIL testwork data and results.

Table 13-53: Summary of CIL Testwork

Parameters	Units	Value

Sodium Cyanide	kg/t dry feed	1.19
Oxygen	kg/t dry feed	0.96
Lime ($Ca(OH)_2$)	kg/t dry feed	21.2
Water Dilution to Achieve 40% solids	t/h	~ 64
Gold Recovery	%	96.7 - 98.4
Silver Recovery	%	3 - 27
Temperature	°C	~ 40
pH at 24hr	-	9.5 - 9.8
Feed SG	% Solids	40%

13.9.4.5 Cyanide Detox Tests

13.9.4.5.1 Test Conditions

Batch continuous cyanide detox tests were employed. Samples were taken after 4 turnovers had been achieved. The cyanide detox test conditions and results are shown in Table 13-54.

All tests were conducted in a temperature controlled 1L continuously fed agitated reactor. The slurry was heated to approximately 40°C and, once at temperature, oxygen (99.5% pure) and sodium metabisulfite (at 20% strength) in an aqueous solution were continuously added. Lime slurry (at 20% solids) was added intermittently to maintain the pH above 8.5. Only the discharge slurry after the first 4 turn overs of the continuous detox test was retained and analyzed.

The objective in this testwork was twofold:

- The first and primary reason for testing the cyanide detox step was to understand its impact on the stability of arsenic, and
- The second objective was to present a post-detox slurry to the flotation tailings blend step.

No attempt was made to optimize the sodium metabisulfite, oxygen concentration and lime dosage. In most tests the feed slurries into detox were diluted to satisfy the ''continuous process'' requirement.

Table 13-54: Cyanide Detox Summary

Parameters	Units	393.02.POX2. Detox.01	393.02.POX2/3. Detox.02	393.02.POX4. CIL2.Detox.1	393.02.POX.5. CIL.Detox.1
Feed Source		393.02.POX2.AAP2. N5.CIL Slurry	393.02.POX2/3.AAP3. N1.CN Slurry	393.02.POX4. CIL2 Slurry	393.02.POX5. CIL Slurry
Solids Feed Rate	dry kg/h	0.17	0.47	0.17	0.18
Feed Slurry Density Note 1	% w/w solids	10	23	10	10
Detox Retention Time	Minutes	25.5	18.1	26	23.4
Sodium Metabisulfite Feed	g / kg dry solids	24.6	39.9	41.4	24.8
Hydrated Lime Feed	g/kg dry solids	21	27	24.6	41.8
Average Slurry pH		8.85	8.69	8.81	8.64
Average Dissolved Oxygen Note 2	ppm	16.7	18.1	27.8	28.3
Average Temperature	°C	40.4	39.4	37	36
Initial WAD CN Concentration	ppm	100	20	60	50
Final WAD CN Concentration	ppm	0	0	0	0
Note 1: The feed slurry was diluted to permit a continuous process with limited feed slurry availability. Note 2: Oxygen concentration was not controlled.					

13.9.4.5.2 SPLP Results for Cyanide Detox

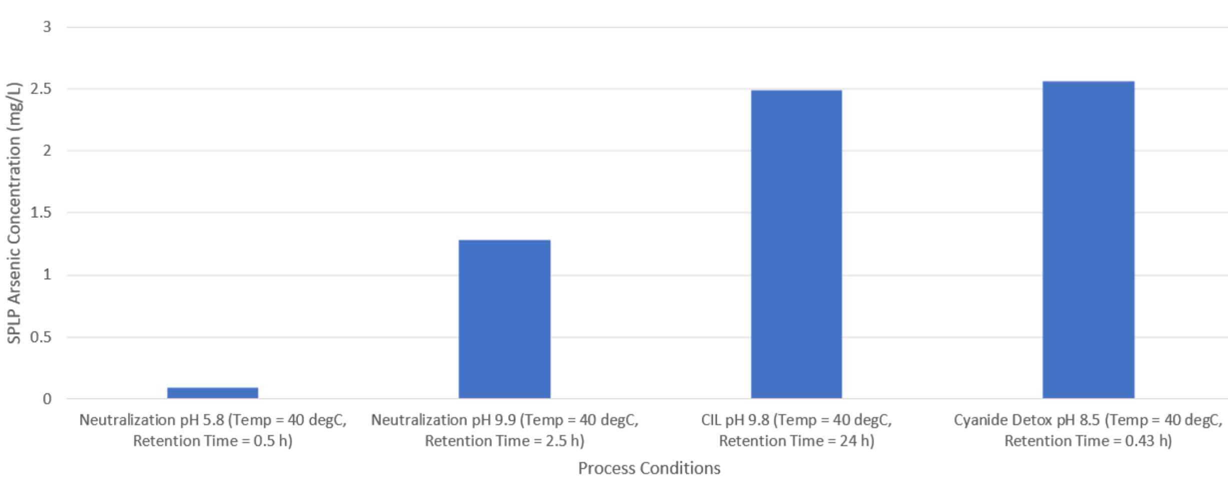
The SPLP values for the final unwashed cyanide detox residue are given in Table 13-55. Arsenic continues to leach from the detox residues possibly because of the pH 10 destabilization that occurred in the gold recovery steps. There was no significant change to the antimony, mercury and chromium leachate values.

Table 13-55: SPLP of the Final Unwashed Cyanide Detox Residue

Parameters	Units	393.02.POX2. Detox.1	393.02.POX2/3. Detox.2	393.02.POX4. Detox1	393.02.POX5. Detox1	Average
Sulfate, SO_4	mg/L	1449	1614	1824	1746	1658
Arsenic, As	mg/L	2.56	1.56	4.26	3.24	2.91
Aluminum, Al	mg/L	<1	<1	<1	<1	<1
Selenium, Se	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	<0.002	<0.002	<0.002	<0.002	<0.002
Lead, Pb	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02
Chromium, Cr	mg/L	<0.1	<0.1	<0.1	<0.1	<0.1
Manganese, Mn	mg/L	0.02	<0.01	0.04	<0.01	<0.01
Iron, Fe	mg/L	<1	<1	<1	<1	<1
Nickel, Ni	mg/L	0.1	6.21	<0.05	<0.05	3.16
Copper, Cu	mg/L	<.05	1.8	<0.05	<0.05	0.49
Zinc, Zn	mg/L	<.05	<0.05	<0.05	<0.05	<0.05
Magnesium, Mg	mg/L	7.1	10.4	9.2	8.7	8.9
Sodium, Na	mg/L	64.8	157	72	46.9	85
Potassium, K	mg/L	6	7	14	18	11
Phosphorus, P	mg/L	<0.3	1	<0.3	<0.3	1
Mercury, Hg	mg/L	<0.02	0.07	<0.02	<0.02	0.033
Calcium, Ca	mg/L	588	580	599	631	600
Antimony, Sb	mg/L	0.032	0.023	0.017	0.008	0.02

Figure 13-40 provides a simple SPLP arsenic trend for the overall metallurgical process after the pressure leach and atmospheric arsenic precipitation. The trend confirms that the primary arsenic destabilization commenced in the pH10 Neutralization and continued into the gold recovery cyanide CIL.

Figure 13-40: Comparison of SPLP As concentration at Neutralization (pH 5 & 10), CIL and Cyanide Detox



13.9.4.5.3 Summary of Cyanide Detox Testwork

The cyanide detox testwork results is summarized in Table 13-56.

Table 13-56: Summary of Cyanide Detox Testwork

Parameters	Units	Value
Sodium Metabisulfite (Over-Stoichiometric)	%	10
Dissolved Oxygen Concentration	ppm	16 - 29
Lime Addition ($Ca(OH)_2$)	kg/t dry solid feed	0.021 – 0.042
Temperature	°C	36-40
pH	-	8.6 - 8.9
Retention Time	min	18 - 26

13.9.4.6 Cyanide Detox Discharge Slurry and Flotation Tailings Blend

13.9.4.6.1 Test Method

The cyanide detox slurry was blended with concentrator tailings thickener underflow and the blend was examined for arsenic stability. The blend ratio for Concentrate 10 was:

- 75.2% rougher tailings,
- 12.0% cleaner tailings, and
- 12.8% cyanide detox residue.

The blend was sampled for assay.

13.9.4.6.2 SPLP Results for Cyanide Detox

Table 13-57 provides the SPLP data for the cyanide detox and flotation tailings blend.

Table 13-57: SPLP Values for the Blended Detox Residue and Flotation Tailings

Parameters	Units	393.02.POX2/3. AAP3.N1a.CIL. Tailings	393.02.POX4. CIL. Tailings	393.02.POX2/3. Detox2. Tailings Note1	393.02.POX4. Detox1. Tailings	393.02.POX5. Detox1. Tailings	Average
Arsenic, As	**mg/L**	**0.73**	**0.53**	**0.23**	**0.59**	**0.49**	**0.514**
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Antimony, Sb	mg/L	0.055	0.028	0.069	0.081	0.069	0.060
Note 1: This is the SPLP results for the blended partial *in-situ* acid neutralization detox residue and flotation tailings							

The cyanide detox residue from a single pressure oxidation test (POX 5 CIL Detox residue) was submitted to a "kinetic" SPLP program to identify whether time had any impact on the stability of the residue. The residue was stored below its supernatant at 20°C. The results of this tests are provided in Table 13-58 and Table 13-59.

Table 13-58: Kinetic SPLP of POX 5 CIL Detox Residue

Parameters	Units	Week 0	Week 1	Week 2	Week 3	Week 4	Week 5	Week 6
Arsenic, As	mg/L	3.28	3.60	3.75	3.88	3.44	3.62	3.44
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Antimony, Sb	mg/L	<0.1	<0.1	0.016	0.035	0.013	0.018	0.018

Table 13-59: Kinetic SPLP of POX 5 CIL Detox Residue Blended with Tailings

Parameters	Units	Week 0	Week 1	Week 2	Week 3	Week 4	Week 5	Week 6
Arsenic, As	mg/L	0.41	0.46	0.46	0.59	0.46	0.46	0.47
Mercury, Hg	mg/L	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Antimony, Sb	mg/L	<0.1	<0.1	0.173	0.276	0.208	0.32	0.33

The assay values in Table 13-58 and Table 13-59 suggests that there may be some destabilization of antimony. However, the SPLP arsenic in the Detox Residue blended with tailings does not indicate cause for concern.

13.9.4.7 Arsenic Deportment across Metallurgical Circuit

The blending of flotation tailings and detox residue slurries reduced the arsenic SPLP to approximately 0.5mg/L. Table 13-60.

Table 13-60 summarizes the SPLP arsenic trends across the SGS metallurgical testwork. It also provides the expected SysCAD modelled concentrations of arsenic across the entire metallurgical circuit incorporating the concentrator and the Tailings Storage Facility.

Table 13-60: Arsenic Trends in the Metallurgical Process

Process Steps	pH Range	As SPLP (mg/L)	SysCAD Soluble Arsenic (mg/L)
Pressure Oxidative Leach	~ 0.8	0.68	207
Atmospheric As Precipitation	1.5 – 2.0	0.28	6.5
pH 5 Neutralization	5	0.11	7.6
pH 10 Neutralization	9.5 - 10	2.2	20.1
Carbon in Leach	9.5 - 10	3.1	17.4
Cyanide Detox	8.5 – 9.0	2.06	17.3
Tailings – Cyanide Detox Blend	7.8 - 8.5	0.23	7.4

Arsenic destabilization appears to be an inevitable outcome of raising the pH of the POX leach residues for the recovery of gold employing the cyanide carbon-in-leach step. The destabilization of arsenic does not seem to be reversible at pHs above neutral and only appears to be arrested when the pH is reduced to approximately 8.5 in Cyanide Detox. Arsenic is expected to leach from POX residues and report to the process liquors. The only sink for aqueous arsenic is in the pore water within the tailings facility and in the autoclave and neutralization circuits where arsenic containing process water is employed in the feed repulp, reagent make up and quench water. The SysCAD mass balance models both these arsenic sinks and the values in Table 13-60 reflect the expected arsenic values across the circuit.

13.10 HYDROMETALLURGICAL RECOVERY

Test data using the FS flowsheet has been generated on gold (pyrite/arsenopyrite) concentrates from composite samples of the following ore types and blends:

- Yellow Pine High Antimony (Con 4)

- Yellow Pine Low Antimony (Con 2, 3, 5)
- Hangar Flats High and Low Antimony (Con 6, Con 7)
- West End Sulfide/Transition (Con 8, Con 9, Con 11)
- Blended Composite representing years 1-4 production consisting of 85% Yellow Pine and 15% Hangar Flats (Con 1, Con 10)
- Blended composite representing periods of blended Hangar Flats/West End production (Con 12)

The data, shown in Table 13-61 include batch POX test data on all samples except Con 10, and continuous Pilot Plant data on Con 10.

Con 8 was an outlying West End composite, with very high sulfur grade (17%). It was created in case POX testwork indicated that for the carbonate-rich West End material a greater degree of carbonate rejection was needed. This proved not to be the case so, while this sample was tested, optimization of the treatment scheme for such high sulfur grades was not pursued, hence the low recovery.

A series of options for the prediction of hydrometallurgical recoveries have been considered:

- Option 1: Average all the data, weighted evenly
- Option 2: Average of all data, weighted evenly, excluding Con 8. All subsequent options excluded Con 8.
- Option 3: Projected recoveries for each ore type/source reflected average result from the relevant tests.
- Option 4: Projected recoveries for each ore type/source reflected average result from the relevant tests except Yellow Pine Low Sb, where the pilot plant result alone was used.
- Option 5: Only the pilot plant result was used.

M3 chose to adopt Option 4 for forecasting the extraction of gold to solution for the project metallurgical forecast. The pertinent input data and chosen recoveries are shown in Table 13-61.

Downstream processing steps (carbon absorption, desorption and refining) all incur small gold losses. Using a standard M3 parameter, these have been assumed to add up to 0.8% for both gold and silver. Accordingly, recovery to doré from leach solution has been assumed to be 99.2%.

No testing was done on concentrate from re-flotation of Bradley Tailings material, so it is assumed that the POX recovery will be the same as for Yellow Pine low Sb material.

Table 13-61: Input Data and Chosen POX-CIL Recoveries

Metal	Ore Source/Type	Composite	Individual Recoveries, %	Selected Recovery, %	Notes
Gold	YP High Sb	Con 4	96.7	96.0	Con 4 recovery, carbon and solution losses assumed to be 0.8%
	85% YP/15% HF	Con 1	97.9	96.7	Average of pilot plant (98.1%) and average of batch tests on different composites (96.8%). Carbon and solutions losses assumed to be 0.8%
	YP Low Sb	Con 2	98.2		
	YP Low Sb	Con 3	95.2		
	YP Low Sb	Con 5	95.8		
	85% YP/15% HF	Con 10	98.1		
	HF	Con 6	96.9	96.5	Average recovery (97.3%), carbon losses of 0.8%
	HF	Con 7	97.6		

Metal	Ore Source/Type	Composite	Individual Recoveries, %	Selected Recovery, %	Notes
	50:50 HF:WE	Con 12	98.0	97.6	Average recovery, excluding Con 8, carbon losses 0.8%
	WE	Con 8	90.3		
	WE	Con 9	98.4		
	WE	Con 11	98.8		
Silver	YP High Sb	Con 4	0.0	0.0	
	YP Low Sb	Con 1	1.1	2.3	Average recovery
	YP Low Sb	Con 2	3.7		
	YP Low Sb	Con 3	2.7		
	YP Low Sb	Con 5	2.8		
	85% YP/15% HF	Con 10	1.2		
	HF	Con 6	0.2	0.4	Average recovery
	HF	Con 7	0.6		
	50:50 HF:WE	Con 12	1.6	5.9	Average recovery
	WE	Con 8	1.7		
	WE	Con 9	7.1		
	WE	Con 11	13		

13.11 FEASIBILITY METALLURGICAL PROJECTIONS

Based on the above-described rationale, Table 13-62 provides the metallurgical projections that have been adopted for the Stibnite Gold Project Feasibility Study.

Table 13-62: Summarized Metallurgical Forecast Algorithms

Ore Body	Ore Type	Product	Parameter	Metallurgy Forecast Algorithms
Yellow Pine	High Antimony	Antimony Con	Au Recovery into Sb Concentrate	3.40 x Sb grade + 0.0089
			Ag Recovery into Sb Concentrate	73.39 x Sb grade + 0.022
			Sb Recovery in Sb Concentrate	11.89 x Sb grade +0.83
			Sb Concentrate Grade	65.0%
		Doré	Au Flotation/POX/CIL Recovery (for 6.5% S con)	(7.94 x pyritic S grade + 0.836) x 0.960
			Ag Flotation/POX/CIL Recovery (for 6.5% S con)	0.0%
		Sulfide Con	Sulfide Sulfur Flotation Recovery (for 6.5% S con)	21.20 x pyritic S grade + 0.621
	Low Antimony	Doré	Au Flotation/POX/CIL Recovery (for 6.5% S con)	90.7%
			Ag Flotation/POX/CIL Recovery (for 6.5% S con)	0.6%
		Sulfide Con	Sulfide Sulfur Flotation Recovery (for 6.5% S con)	96.1%
Hangar Flats	High Antimony	Antimony Con	Au Recovery into Sb Concentrate	6.82 x Sb head grade + 0.002
			Ag Recovery into Sb Concentrate	52.8%
			Sb Recovery in Sb Concentrate	11.00 x Sb head grade + 0.80
			Sb Concentrate Grade	54.1%
		Doré	Au Flotation/POX/CIL Recovery (for 6.5% S con)	86.6%
			Ag Flotation/POX/CIL Recovery (for 6.5% S con)	0.1%
		Sulfide Con	Sulfide Sulfur Flotation Recovery (for 6.5% S con)	79.4%
	Low Antimony	Doré	Au Flotation/POX/CIL Recovery (for 6.5% S con)	88.9%
			Ag Flotation/POX/CIL Recovery (for 6.5% S con)	0.2%
		Sulfide Con	Sulfide Sulfur Flotation Recovery (for 6.5% S con)	95.3%

Ore Body	Ore Type	Product	Parameter	Metallurgy Forecast Algorithms
West End	Oxide	Doré	Au Direct CIL Recovery	(0.916 x $^{CN}/_{FA}$ + 0.0120) x 0.992
		Doré	Ag Direct CIL Recovery	(0.411 x $^{CN}/_{FA}$ + 0.256) x 0.992
	Sulfide and Transition	Doré	Au Flotation/POX/CIL Recovery (to 1.3 CO_3/S Con)	(-0.867 x $^{CN}/_{FA}$ + 0.997) x 0.976
		Doré	Ag Flotation/POX/CIL Recovery (to 1.3 CO_3/S Con)	(-0.809 x $^{CN}/_{FA}$ + 0.959) x 0.009
		Sulfide Con	Sulfide Sulfur Flotation Recovery (to 1.3 CO_3/S Con)	-0.294 x $^{CN}/_{FA}$ + 0.989
		Doré	Au Flotation Tailings CIL Recovery, Low $^{CN}/_{FA}$	(1.767 x $^{CN}/_{FA}$ + 0.162) x 0.992 for $^{CN}/_{FA}$ < 0.31
		Doré	Au Flotation Tailings CIL Recovery, High $^{CN}/_{FA}$	(0.451 x $^{CN}/_{FA}$ + 0.549) x 0.992 for $^{CN}/_{FA}$ > 0.31
		Doré	Ag Flotation Tailings CIL Recovery	60.9%
Bradley Tailings	Low Antimony	Doré	Au Flotation/POX/CIL Recovery	67.7%
		Doré	Ag Flotation/POX/CIL Recovery	0.2%
		Sulfide Con	Sulfide Sulfur Flotation Recovery	74.0%

13.12 METALLURGICAL OPPORTUNITIES

13.12.1 Acid Treatment and Leaching of Flotation Cleaner Tailings

The cleaner tailings from processing both Yellow Pine and Hangar Flats materials contain refractory gold encapsulated in ultra-fine sulfides. As these sulfides are so fine grained it has been proposed that atmospheric acid treatment may be successful in partially or wholly oxidising them (essentially equating to an Albion type process). A cursory examination of the potential for use of the highly acidic un-neutralised POX-CCD overflow to leach the cleaner flotation tailings was made and indicated that some of the gold did indeed become leachable.

This was not added to the flowsheet as (a) it was deemed too complex for the feasibility study and (b) the supporting data was too limited to allow for a reliable trade-off exercise to be conducted.

13.13 ALTERNATIVE PROCESSES

13.13.1 Alternatives to Upgrading of Rougher Concentrates

While cleaner flotation was selected as the process of choice when hydrometallurgical data pointed to the need for a 7.5% sulfur flotation concentrate, this has subsequently been dropped to 6.5% sulfur.

At such a target grade, cyclone desliming may well be adequate for the production of autoclave-ready concentrates, so saving substantial capital and operating costs compared with cleaner flotation.

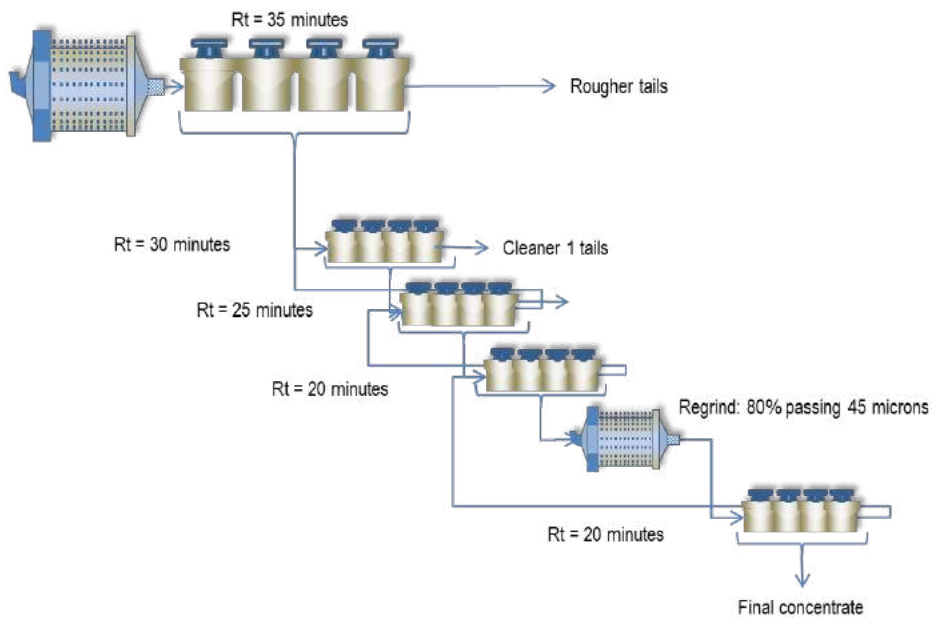
In practice, the deep, well-drained froths produced by a commercial flotation plant may allow for concentrates assaying 6.5% sulfur in rougher flotation only. This may especially be the case if a hybrid Woodgrove SFR/conventional tank cell circuit is employed in conjunction with conventional tank cell flotation (prior testing using SFR alone failed to match recoveries with a laboratory cell, but grades were far better and laboratory flotation of the SFR tails showed promise at increasing overall grades without affecting recoveries. Prior to detailed design of the plant, a pilot plant study perhaps incorporating Woodgrove and/or similar technologies should be considered as this may lead to the decision to eliminate cleaner flotation entirely, prior to the treatment of West End material.

13.13.2 Production of Shippable Concentrates

The potential for cleaner flotation to produce a concentrate suitable for shipment off-site, as an alternative to on-site sulfide oxidation and gold leaching, was investigated during the FS. Flowsheet development by batch testing and confirmation by locked cycle testing was conducted on life-of-mine low Sb composites from Yellow Pine, Hangar Flats and West End, while a selection of variability samples from the feasibility study have been used to evaluate metallurgical consistency in cleaning (Martin, 2018), (Hall, 2018).

The adopted flowsheet employed either four or five stages of cleaning, with regrinding of the third cleaner concentrate. Three stages of cleaning were found to be important ahead of regrinding as it allowed for removal of barren slimes. This made the final 1-2 stages of cleaning of the reground concentrate more effective. Regrinding was to a product size of 80% passing 40-45 microns. The example of the Yellow Pine flowsheet is shown on Figure 13-41 below:

Figure 13-41: Typical Flowsheet Employed in Locked Cycle and Variability Cleaner Testing



Recognizing the complexity and cost of such a cleaner circuit, laboratory-scale batch column flotation was explored as a means of both simplifying the flowsheet and potentially raising concentrate grades. High concentrate grades, close to 34% sulfur, were achieved in several tests but recoveries were lower than when using a conventional agitated laboratory cell. For the sake of simplicity at this stage in the project, further work on column flotation was curtailed and confirmation and variability work was conducted using conventional agitated flotation. If shippable grade concentrate production was to be explored in more detail in the future, pilot plant column testing should be considered.

Two locked cycle tests on the Yellow Pine master composite targeted the production of low and high sulfur concentrates, to provide information on the grade/recovery relationship in cleaning. The tests produced 23.4% and 26.5% sulfur concentrates at 91.5% and 90.6% gold recovery respectively – suggesting there could be some supplemental recovery loss associated with cleaning to the higher-grade concentrate. One locked cycle test was run on each of the Hangar Flats and West End master composites. The Hangar Flats composite yielded 89.8% of the gold into a concentrate assaying 24.9% sulfur, while, with leaching of the rougher and first cleaner tailings, the West End composite yielded a total of 88% gold recovery to a concentrate assaying 25.1% sulfur. 77% of the gold reported to the concentrate while 11% was leached and would be recovered as doré. Gold grades in concentrates from the four locked cycle tests were 40-50 g/t.

Compared with flotation to produce a concentrate for on-site autoclaving (typically assaying 6.5% sulfur), cleaning to shippable grades incurred a supplemental gold loss of 3.6% for Yellow Pine, 2.3% for Hangar Flats and 3.9% for West End.

Variability testing to produce a shippable concentrate was conducted on 13 different samples, representing some of the best and worst acting samples from the feasibility study. Excluding one outlying Hangar Flats sample, the average estimated supplemental loss in gold recovery was 3.3%, compared with the flotation of an on-site POX-ready concentrate.

13.13.3 Antimony Concentrate Processing

Two scoping studies were undertaken to evaluate the options for antimony concentrate processing by Midas Gold, as opposed to direct sales of concentrate to a third party. The two evaluated options were roasting (at Kingston Process Metallurgy, Kingston, Ontario) and leach-electrowinning (at SGS Lakefield, Lakefield, Ontario).

The concentrates sent for the two studies were produced from a high-grade antimony mixture of material from the Hangar Flats and Scout Ridge prospect areas of the Project. These were produced from 26 x 10 kg batch tests with two stages of cleaning of the antimony concentrates which produced approximately 11 kg of concentrate at an average grade of 50.4% antimony.

13.13.3.1 Stibnite Roasting

Roasting scoping studies were conducted at Kingston Process Metallurgy in a two-phase program involving static kiln tests at three temperatures (700, 800, and 900 ºC) followed by two rotary kiln runs at temperatures near the optimum identified in the static tests. Product from the tests were forwarded to SGS for cyanidation of the calcines to evaluate amenability to gold extraction. Based on the results of the rotary kiln tests, a preliminary heat and mass balance was also evaluated (Pettingill, Davis, & Roy, 2013).

Results of the static kiln tests showed the best antimony removal at temperatures of 800 ºC and higher, with greater than 99% removal of antimony from the concentrates. Precipitates from the condensation zone ranged from 79.3 - 83.6% Sb, 0.77 - 0.81% As and 0.08 - 0.24% Fe. The final rotary kiln results showed that at 950 ºC, 99.9% of the antimony and 95% of the sulfur off-gassed (as SO_2) in the first 2 hours. Cyanidation of the calcines was able to extract 95% of the gold remaining in them.

13.13.3.2 Stibnite Leach – Antimony Electrowinning

The second study conducted on the concentrates was done at SGS Lakefield and involved scoping testwork into a stibnite leach – antimony electrowinning process. A significant potential upside to the leach-electrowinning program is that the leach residues from the process would be available for reprocessing in the autoclave, rendering that gold recoverable.

Scoping testwork involved investigating four leach methods: ferric chloride, caustic, caustic sulfur and caustic sulfide, followed by Hull cell electrowinning. Leach parameters investigated included reagent concentration and leach temperature; leach tests were conducted with kinetic samples pulled to assess the extraction vs. time curve for each. The final solutions were placed into a Hull electrowinning cell to test deposition of antimony on the cathode, configuration of the Hull cell tested current densities from 0 - 500 ampere per square meter (A/m^2). Parameters investigated in electrowinning included: temperature, degree of mixing, current intensity, current density and cathode type: stainless steel, copper or brass (Lupu & Gladkovas, 2014).

Caustic leach: Results of the caustic leach showed that antimony extraction of 99.5% could be achieved with a 10% NaOH solution at 25ºC in 3 hours, when conducted at 2% solids in the leach. All tests exceeded 90% extraction of antimony, achieved within one hour. Gold was not leached, and silver dissolution was less than 10%. In the single test where it was measured, 94% of the arsenic was extracted.

Caustic sulfide leach: Antimony extraction of 99.9% was achieved in the first few hours of the leach with both sulfur sources (sodium sulfide or elemental sulfur) under all test conditions. Use of elemental sulfur as the sulfur source appeared to leach about 24% of the gold from the concentrate while silver extraction was about 10%. Use of sodium sulfide as the sulfur source leached less than 10% of the gold and leached 26 - 30% of the silver. Greater than 75% of the arsenic also appeared to be leached in the tests.

Ferric Chloride: Results of the ferric chloride leach showed that at 90 ºC and 150 g/L sodium chloride, greater than 93% extraction of antimony could be achieved. The parameters tested resulted in extractions ranging from 55 to 93%. There was no indication of gold leaching in any of the tests, while silver extraction ranged from about 30 to 67%. Arsenic extraction was varied as well, with tests leaching from 25 to 85% of the arsenic.

Electrowinning of antimony from all solutions was successful, though the degree of metal adhesion varied with each leach solution and cathode material.

The caustic sulfide leach was tested in a brief locked cycle test employing leaching, electrowinning and re-leaching to provide preliminary insight into the suitability of the spent solutions from electrowinning for re-leaching a new batch (Gladkovas, 2014).

In both leach cycles, antimony extraction was close to 99%. Current efficiency dropped quickly in electrowinning due to depletion of the antimony in solution, but by the end of the second leach, antimony loading had risen to the point where significantly more efficient electrowinning could be expected. Initial indications are, therefore, that the process will prove to be workable in commercial operation – however no analyses of the electrowon product were obtained to explore its potential marketability. Mineralogical analyses of the leach residues from the study indicated that the gold-bearing pyrites and arsenopyrites were intact and likely to be available for processing in the autoclave with other gold concentrates. The state of any remaining silver was not investigated and should be evaluated in the future.

13.13.3.3 Neutral pH Pressure Cyanidation of Antimony Concentrates

Conventional cyanidation of otherwise free-milling gold is not possible in antimony-rich materials as the antimony consumes large amounts of the cyanide at high pH levels. Accordingly, neutral pH cyanidation is practiced under pressure using a pipe reactor at Consolidated Murchison in South Africa. Such a process may allow for extraction of silver and some of the gold from the antimony concentrates and could be tested in due course. Mild pre-oxidation of the stibnite has also been proposed as an alternative, whereby the stibnite surface is sufficiently oxidized to be passivated from reaction with the cyanide.

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14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The Mineral Resource Statement presented herein represents the fourth mineral resource evaluation prepared for Midas Gold in accordance with the Canadian Securities Administrators' National Instrument 43-101 (**NI 43-101**). This evaluation includes updated Mineral Resource estimates for the Project's three lode gold deposits; Yellow Pine, Hangar Flats and West End, and also reports the Mineral Resource Estimate for the historical tailings deposit, which is unchanged since the PFS (M3, 2014).

This section describes the mineral resource estimation methodology and summarizes the key assumptions. In the opinion of Garth Kirkham, P.Geo., Qualified Person, the mineral resource estimates reported herein are a reasonable representation of the mineral resources found within the Project at the current level of sampling. The mineral resources were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (**CIM**) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (CIM, Nov. 2019) and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Updated Mineral Resources reported herein supersede and replace the Mineral Resources disclosed publicly (Midas Gold, 2018), which should no longer be relied upon. **It is important to note that mineral resources that are not mineral reserves do not have demonstrated economic viability. Mineral resource estimates do not account for mine-ability, selectivity, mining loss and dilution. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated.**

The mineral resource evaluation reported herein is for Yellow Pine, Hangar Flats and West End is current and supersedes earlier mineral resource estimates completed for Midas Gold including:

- Technical Report on Mineral Resources for the Golden Meadows Project (SRK, 2011).
- Preliminary Economic Assessment Technical Report for the Golden Meadows Project Idaho (SRK, 2012).
- Preliminary Feasibility Study Technical Report for the Stibnite Gold Project (M3, 2014).

The mineral resource estimates were reviewed and verified by Garth Kirkham, P.Geo., the Independent Qualified Person for the mineral resource estimates for the Project and included in this Report. Midas Gold's field work on the Project from 2009-2015, including drilling, was carried out under the supervision of Chris Dail, CPG and Richard Moses, CPG, who were Midas Gold's senior geologists responsible for certain aspects of the programs during the periods they were employed by Midas Gold. Field work, including drilling, completed in 2015-2017 was carried out under supervision of Kent Turner, independent senior geology consultant and SME-Registered Member, and Austin Zinsser, Midas Gold's Senior Resource Geologist and SME-Registered Member.

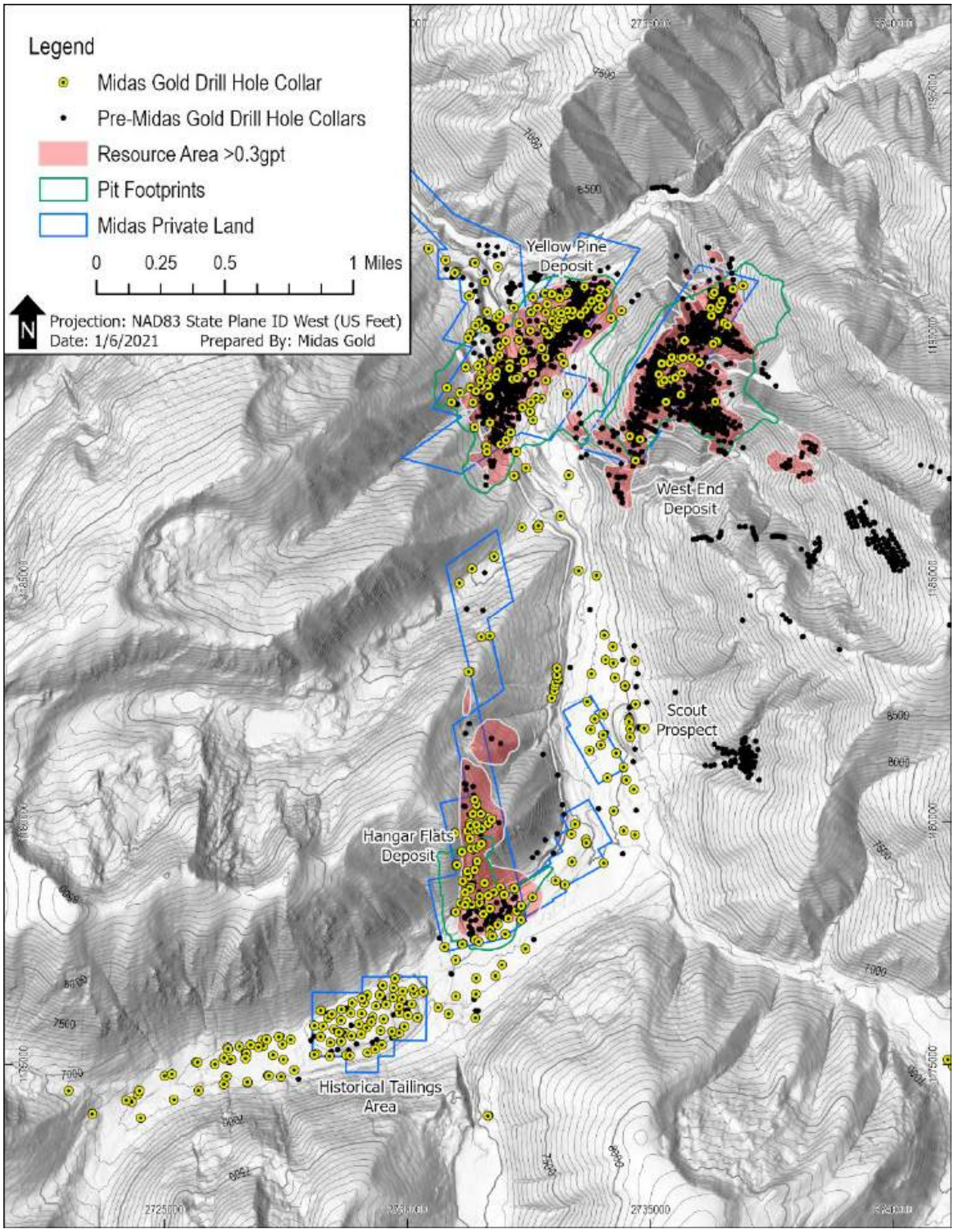
The general mineral resource estimation methodology for all deposits involved the following procedures:

- generation of updated geological models and review of structural controls on mineralization;
- database verification and validation;
- data exploration, compositing and evaluation of outliers;
- construction of estimation domains for gold, antimony and silver;
- spatial statistics;
- block modeling and grade interpolation;

- mineral resource classification and validation;
- assessment of "reasonable prospects for eventual economic extraction;" and
- preparation of the mineral resource statement.

The drillhole database and data utilized in the Mineral Resource Estimate is discussed in Section 14.2 with detailed mineral resource evaluation methodologies discussed in subsequent sections for Yellow Pine (Section 14.2.1), Hangar Flats (Section 14.2.2), and West End (Section 14.2.3). An assessment of reasonable prospects for eventual economic extraction and mineral resource statements, including that for the Historical Tailings are presented in Sections 14.8 and 14.9. Figure 14-1 shows a plan view of the Stibnite Project area along with drillhole locations and deposits that are the subject of the Resource Estimation reported herein.

Figure 14-1: Plan Map of the Stibnite Gold Project Area Showing Drillhole Locations and Deposits



14.2 DRILL HOLE DATABASE

Midas Gold's drill hole database used for Mineral Resource Estimation, is stored as an SQL database in Hexagon Minesight Torque™ and contains collar locations stored as NAD83 State Plane feet grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold and silver analyses by fire assay and/or cyanide soluble assay, other geochemical assays including antimony and sulfur, geologic intervals with rock types, core recovery information, and core density measurements. The most common assay lengths are approximately 5 ft long, with the majority of assays between 3 ft and 7 ft in length. The drill hole database contains 1,843 specific gravity measurements, collected on core samples using a water immersion method and verified with independent, third party laboratory measurements.

14.2.1 Yellow Pine Drill Hole Database

The Yellow Pine area was explored for gold and antimony by numerous operators, up to and including Midas Gold between 2011 and 2017. The Yellow Pine deposit was previously in production in the 1930s - 1950s from the Bradley Pit area, while the Homestake area was in production in the late 1980s. The drill hole database contains data for 1,016 separate drill holes representing a mixture of pre-1953 and modern drilling programs. Historical data (i.e. pre-Midas Gold) accounts for approximately 48% of the drill hole database by footage, as previously described (Section 10). Multiple statistical validations were completed to assess the quality of the historical drill hole data, as discussed in the PFS. A significant number of historical holes were removed from the dataset used for resource estimation including holes missing critical supporting information, holes with long downhole composited assays, air-track drill holes, R.C. holes showing evidence for cyclicity, and all historical pre-1953 drill holes in the northeast Homestake area of the deposit. In addition, certain historical holes were removed from the estimate which appeared to be mis-located or otherwise erroneous based on improved understanding of controls on mineralization.

For the Yellow Pine deposit, gold, antimony and silver mineral resources were estimated in addition to oxidation intensity and a suite of geochemical concentrations. Table 14-1 shows the number of drill holes and estimation composites utilized in the estimate for the primary commodities, which illustrates that the metal values for gold, antimony, and silver were not consistently analyzed for all sample intervals throughout the various historical drilling campaigns nor were all drillholes deemed to have reliable information for all elements.

Table 14-1: Drill Hole Data Used in the Yellow Pine Mineral Resource Estimate

Company	Gold			Antimony			Silver		
	# Holes	# Samples	Feet	# Holes	# Samples	Feet	# Holes	# Samples	Feet
Barrick	17	2,538	12,817	14	2,164	10,932	-	-	-
Bradley	107	4,056	20,650	70	2,380	12,087	7	212	1,078
El Paso	1	52	258	1	52	258	1	52	258
Hecla	68	2,282	11,582	-	-	-	58	1,954	9,929
Midas Gold	223	28,510	143,748	223	28,454	143,465	223	28,686	144,651
Pioneer	2	86	435	-	-	-	-	-	-
Ranchers	145	4,660	23,649	54	2,150	10,900	-	-	-
Superior	16	384	1,951	-	-	-	-	-	-
USBM	50	2,714	13,758	50	2,602	13,195	-	-	-
All	629	45,282	228,848	412	37,802	190,836	289	30,904	155,915

14.2.2 Hangar Flats Drill Hole Database

The database for the Hangar Flats deposit contains data for 260 separate drill holes representing both historical and modern drilling programs, as previously described in Section 10. The drill holes were reviewed, and certain drill holes were not considered for use in mineral resource estimation, including air-track, rotary, and pre-collar drill holes, as well as historical drilling where the methods were questionable or documentation lacking.

Gold and antimony were mined from the Hangar Flats deposit by the Bradley Mining Company from 1928 to 1938 and the deposit was later explored by Bradley in the 1940s, the United States Bureau of Mines from 1951-1954, the Hecla Mining Company from 1988 to 1989, and by Midas Gold beginning in 2009, as discussed in Section 10. The majority of sampling used in the mineral resource estimate for Hangar Flats is from Midas Gold drilling completed primarily from 2009 to 2012. Data from pre-1940s Bradley operations includes exploration drill holes and underground drift samples and was used solely for construction of the geologic model due to uncertainty regarding sampling and analytical methods. Post 1940s Bradley drillholes, United States Bureau of Mines exploration drillholes and drillholes by Hecla were used for mineral resource estimation as this drillhole data is well documented, has been validated by Midas Gold drilling and is deemed reliable.

For the Hangar Flats deposit, gold, antimony and silver mineral resources were estimated in addition to oxidation intensity and a suite of geochemical concentrations. Table 14-2 shows the number of drill holes and sample intervals utilized in the estimate for the primary commodities, and illustrates that the metal values for gold, antimony, and silver were not consistently analyzed for all sample intervals throughout the various historical drilling campaigns nor were all drillholes deemed to have reliable information for all elements. Note that samples outside of the domains are not tabulated here as they were not used in estimation of gold, silver, or antimony.

Table 14-2: Drill Hole Data Used in the Hangar Flats Mineral Resource Estimate

Company	Gold			Silver			Antimony		
	# Holes	# Samples	Feet	# Holes	# Samples	Feet	# Holes	# Samples	Feet
Bradley	28	856	4,491	0	0	0	19	407	2,176
Hecla	22	701	3,505	22	684	3,420	0	0	0
Midas Gold	114	14,703	74,872	114	14,717	74,949	60	3,817	19,247
USBM	22	632	3,149	0	0	0	0	0	0
All	186	16,892	86,017	136	15,401	78,369	79	4,224	21,423
Note: Drill hole information includes un-sampled intervals. Data outside of estimation domains is excluded from tabulation.									

14.2.3 West End Drill Hole Database

The West End deposit was in explored from 1978-1996 by multiple operators and was previously in production as a heap leach operation during the 1980s and 1990s. The West End drill hole database consists of 943 holes drilled using various methods, as previously described in Section 10. The database consists of collar locations in State Plane grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types, core recovery information and specific gravity measurements. Certain drill holes were not considered reliable for use in mineral resource estimation, including rotary and air-track drill holes, and other unreliable holes flagged by Midas Gold. After removal of selected drill holes and non-bedrock intervals, the final database used for estimation of total gold mineral resources contained 674 drill holes. Approximately 78% of the assay records have gold fire assays **(AuFA)** and 75% have cyanide soluble gold assays **(AuCN)**. Only Midas Gold, Canadian Superior Mining Ltd. (**Superior**) and Stibnite Mines Inc. (**SMI**) drill holes were assayed for silver, with the latter exclusively assayed for cyanide soluble silver.

Table 14-3: Drill Hole Data Used in the West End Mineral Resource Estimate

Company	Gold Fire Assay			Gold Cyanide Assay			Silver		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
El Paso	1	18	30	0	0	0	0	0	0
Midas Gold	53	6,020	11,499	52	5,148	9,872	53	6,020	11,499
Pioneer	336	21,313	32,498	336	21,281	32,449	136	6,947	10,586
SMI	118	6,851	10,431	118	6,851	10,431	118	6,851	10,431
Superior	163	6,573	11,626	132	2,850	6,196	71	2,642	5,448
Twin River	3	160	256	0	0	0	0	0	0
All	674	40,935	66,340	638	36,130	58,948	378	22,460	37,964
Note: Drill hole information excludes samples within overburden and includes un-sampled intervals.									

Drill holes in the West End deposit form an irregular grid and are primarily vertical or oriented on 120-degree azimuths. Mean drill hole spacing is approximately 40 m above 2,100 m elevation increasing to 70 m near the base of the drill pattern at 1,900 m elevation.

14.3 YELLOW PINE

14.3.1 Mineral Resource Estimation Procedures

The Yellow Pine Mineral Resource estimate is based on the validated drill hole database, interpreted digital geologic model, digitized as-built data of historical workings, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed primarily using commercial three-dimensional block modelling and mine planning software Hexagon Minesight™ MS3D Version 15.10.

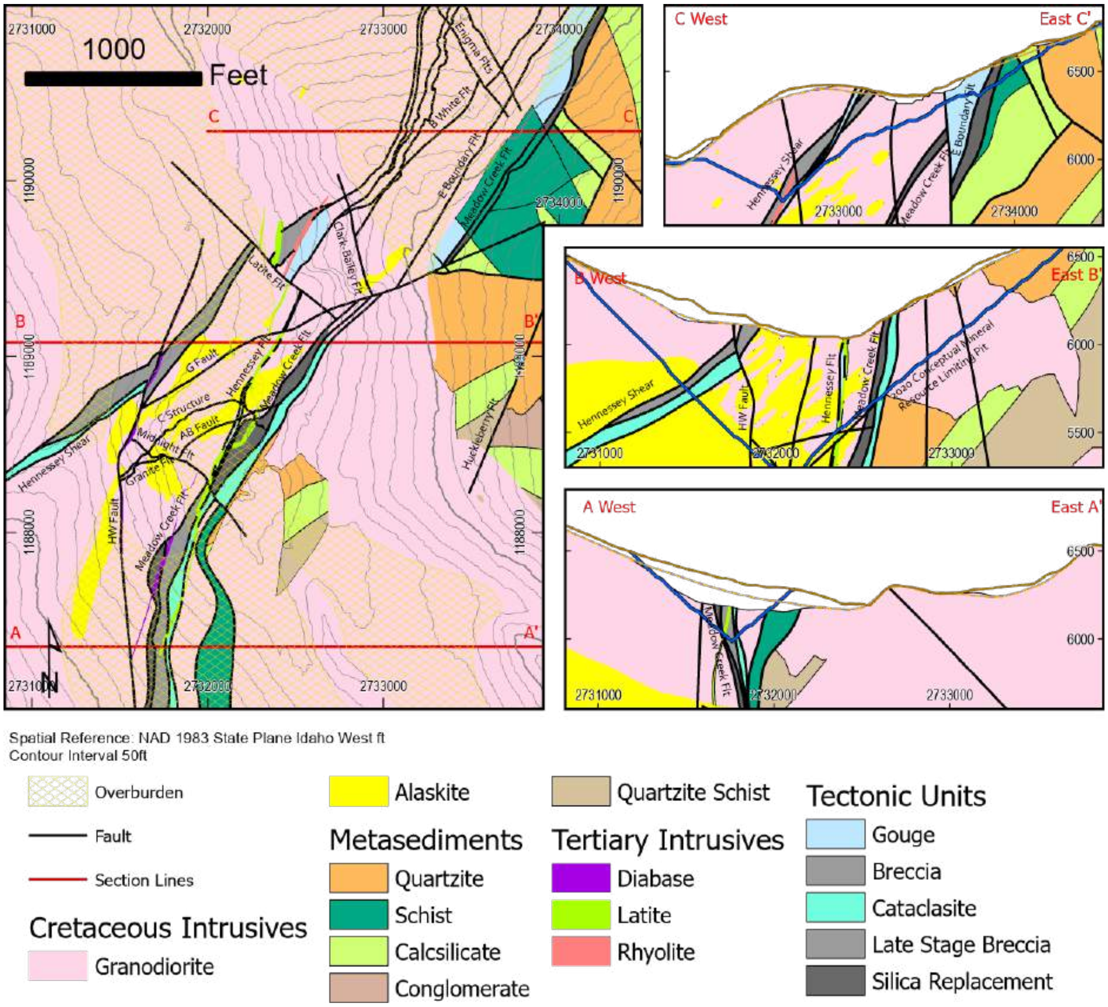
14.3.2 Geologic Modeling

The Yellow Pine Mineral Resource estimate is based on a generalized geologic model consisting of major rock types, major structures, surfaces, and historical underground workings and pit bottom surfaces (as shown on Figure 14-2).

The Yellow Pine Geological model was significantly updated from that used in the PFS. Additional oriented core drilling completed in 2016-2017, re-logging of key fault zones from core photos and integration of structural data, legacy data sets and drillhole geochemistry have allowed for a more detailed 3D structural interpretation of the Yellow Pine deposit. These data sets were integrated into the detailed geological model first using GIS software to capture and geo-reference historical spatial data and then using Hexagon MineSight MS3D to construct geological boundaries through sectional and implicit modeling methods to incorporate logging information, geochemistry and oriented core data. Geological surface TINs were generated from digitized polylines using MineSight's surface interpolation tools and subsequently trimmed manually against fault surfaces based on the deformation sequence for the deposit. Principal changes to the Yellow Pine Geological model since the PFS include:

- subdivision of the Meadow Creek Fault Zone (**MCFZ**) and Hidden Fault Zone (**HFZ**) into syn- and post-mineralization structural corridors consisting of silicified breccias and gouge zones respectively;
- recognition of a pre-gold mineralization silicified breccia zone at southern Yellow Pine adjacent to the primary ore body;
- recognition of northwesterly faults in central Yellow Pine which control antimony mineralization and postdate gold mineralization;
- improved models for post-mineralization Tertiary dikes and their offset by faults;
- digitization of historically mapped northeasterly and northwesterly post-mineralization faults at Homestake which serve as important controls on gold mineralization and oxidation;
- a detailed model for the metasedimentary lithostratigraphic units of the roof pendant east of the MCFZ, some of which are preferential hosts to mineralization; and
- implicit and geostatistical models representing felsic dike swarms cutting the granodiorite.

Figure 14-2: Yellow Pine Geological Model



14.3.3 Controls on Mineralization

As discussed in Section 7, mineralization in the Yellow Pine deposit is structurally controlled and localized by the northerly striking MCFZ and by north striking gently west dipping conjugate splay or cross structures associated with the MCFZ. The majority of mineralization in the deposit occurs west of the MCFZ and east of the Hidden Fault Zone (**HFZ**), a wide, moderately northwest dipping fault and fracture zone. To the south, gold mineralization occurs within a breccia zone of the MCFZ bounded to the east by post-mineralization gouge of the MCFZ and bounded to the west by the pre-gold mineralization ductile breccia zone. In the central region of the deposit, between 1188200N and 1189600N, mineralization is primarily disseminated and occurs east of the Hanging Wall Fault (**HWF**) and west of the post-mineralization Hennessey Fault, except where Hennessey Fault has offset the western part of the orebody to the north. Gold and antimony mineralization in the central region of the deposit are bounded to the south against the C-structure/granite fault, a normal fault which is locally offset by the northwesterly striking Midnight Fault. In the northern Homestake area of the deposit, mineralization occurs in the hanging wall of the Hidden Fault/Clark tunnel structure and is truncated against the East Boundary Fault, a historically mapped gouge zone within the MCFZ occurring directly east of a silicified fault corridor which is moderately mineralized in the Homestake area. Gold mineralization also occurs within the metasediments at Homestake, where both disseminated and vein hosted gold occurs within the upper-calc silicate and Middle Marble formations. These complex relationships between faults and mineralization were applied towards construction of estimation domains in the Yellow Pine Mineral Resource Estimate.

The geologic model also includes solids representing minor late-stage dikes; numerous adits, drifts and underground development workings; and surfaces representing current and pre-mining topography; and the current top-of bedrock surface. The surface representing the top of bedrock was digitized from drill hole data and from 1950s and 1990s engineering drawings depicting the historical Bradley pit and Homestake pit bottoms prior to backfilling. Midas Gold drilling has confirmed the pit-bottom in the Homestake area and the location of legacy underground workings. Drillholes drilled from barges through the pit lake by the Rancher's Exploration Company (**Ranchers**) have confirmed the Yellow Pine pit-bottom as captured from engineering drawings.

14.3.4 Exploratory Data Analysis and Data Preparation

Exploratory data analysis and graphical data review was performed on raw assays within 39 geological solids to aid in construction of appropriate geostatistical estimation domains. Quantitative data analysis included generation of descriptive statistics, box plots, histograms, log-probability plots, and analysis of multivariate relations. The data was also reviewed relative to surfaces representing historical underground and surface mining. Data preparation included assignment of numeric values to samples assaying below detection limits (generally 1/2 detection limit or lower for legacy data) and to intervals which were selectively un-assayed. In addition, samples sourced from non-bedrock materials, including those from backfilled pits and waste rock dumps, were removed from the dataset.

14.3.5 Estimation Domain Modeling

The principal change in the Yellow Pine Mineral Resource estimate relative to the PFS estimate is due to definition of improved geostatistical estimation domains based on the updated geological model. The gold estimate utilized sixteen estimation domains; six primary mineralized domains and ten secondary domains. Gold mineralization occurs in all domains, but 77% of assays greater 0.3 g/t Au occur within the primary domains. The estimation domains consist of 3D geological solids representing discrete fault zones, fault blocks, and lithologic units including metasedimentary formations and intrusive dikes (Table 14-4). The large number of domains was deemed appropriate due to the structural complexity of the deposit and distribution of gold within the updated geological model, especially in order to represent the truncation of mineralization across post-mineralization fault boundaries. The principal gold domains include the mineralized silicified breccia corridor of the southern MCFZ (D3), the Hennessey Shear/Hidden Fault Zone (D5) consisting of silicified breccia and post mineralization gouge, broadly disseminated mineralization occurring in fault bounded blocks of the Central Yellow Pine (D6) and Hennessey fault block (D7), the Homestake deposit area including the hanging wall of the Clark Tunnel structure/northern extension of the Hidden Fault west of the "East Boundary Fault" (D11), and the silicified breccia zone of the northern MCFZ (D12) at the contact with the metasediments. The secondary domains generally have lower gold grades and include the post-mineralization gouge zones of the MCFZ, rhyolite and latite dike solids, three groups of contiguous metasedimentary formations, strongly altered but lower-gold-grade fault blocks occurring below primary gold domains and hanging wall zones occurring west of the ore-body (as shown in Table 14-4 and Figure 14-3).

Table 14-4: Yellow Pine Gold Estimation Domains and Descriptions

Domain Number	Name	Category	Lithology	Description
1	W Intrusives	Secondary Domain	Mixed intrusives	Primarily chloritic altered intrusives with diorite at depth bounded to the east by the MCFZ and north by the HCSZ.
2	S_YP_SiO2_Bx	Secondary Domain	Silicified Breccia	Silicified breccia zone with high sulfide content but low gold and arsenic.
3	S_YP_Au-Sb-Bx	Primary Domain	Silicified Breccia	Silicified breccia corridor of the MCFZ in southern YP bounded by gouge to the east and gold-barren breccia to the west. Midnight fault is northern boundary.

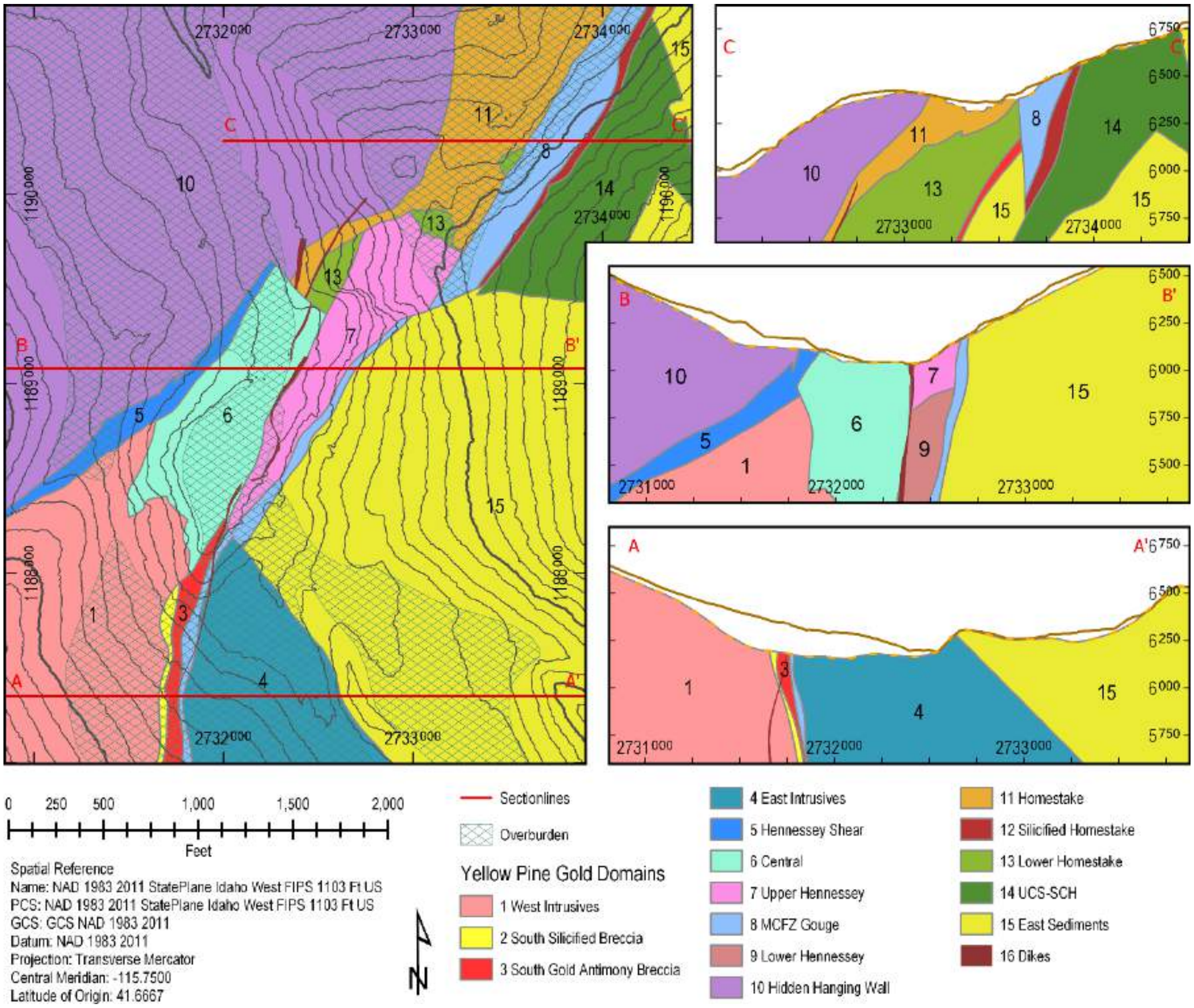
Domain Number	Name	Category	Lithology	Description
4	E Intrusives	Secondary Domain	Intrusives, schist, diorite	Mixed lithologies cut by steeply dipping anastomosing gouge fault strands of the MCFZ.
5	Hennessey Shear	Primary Domain	Breccia, Gouge, Rubble	Hennessey shear zone in which significant gold occurs within post-mineralization gouge zones and rubble zones in the footwall bounded to the east by Domain 6.
6	Central YP	Primary Domain	Mixed intrusives	Disseminated mineralization within the central YP area bounded to south by Midnight and Granite faults, bounded to east by Hennessey Fault and to north by latite fault NW; includes the diabase dike.
7	Hennessey	Primary Domain	Mixed intrusives	Bounded to the west by the Hennessey fault and to the east by gouge of the MCFZ. Includes silicified breccias of the MCFZ in central YP. Lower contact is a geochemical boundary marked by abrupt drop in gold grades but no appreciable change in sulfur or arsenic.
8	MCFZ Gouge	Secondary Domain	Gouge and cataclasite	Gouge and foliated cataclasite of the MCFZ, local mineralized materials entrained.
9	Lower Hennessey	Secondary Domain	Mixed intrusives	Weakly mineralized block of rock beneath Hennessey domain, east of Hennessey Fault and west of MCFZ gouge
10	Hidden Hanging Wall	Secondary Domain	Mixed intrusives	The hanging wall of the Hennessey Creek and Hidden Fault zones characterized by weak chloritic to sericitic alteration
11	Homestake	Primary Domain	Mixed intrusives	Homestake domain includes the northern Hennessey fault zone gouge and the northern Hidden fault breccia corridor, as well as the hanging wall of the Clark tunnel fault zone. Domain is bounded to the east by the East-boundary fault zone, part of the MCFZ gouge corridor.
12	Hmstk SiO2 Bx	Primary Domain	Silicified Breccia	Narrow zone of breccia in the MCFZ between sediments and gouge of the east boundary fault zone. Contains elevated calcium and has low arsenic/gold ratio.
13	Lower Homestake	Secondary Domain	Mixed intrusives	Material beneath Homestake. Sericite-pyrite-arsenopyrite alteration
14	UCS-SCH-QZ	Secondary Domain	Schist, calc-schist, quartzite and breccia	Significant gold mineralization occurs within upper-calc-silicate and schist packages east of MCFZ within hinge zone of stibnite syncline.
15	E Sediments	Secondary Domain	Metaseds and granodiorite sill	Sediments outside of domain 14
16	Dikes	Secondary Domain	Latite and Rhyolite	Dikes within the deposit but excluding the Diabase

Antimony mineralization is controlled by many of the same structures as gold mineralization but is more spatially restricted, occurring primarily south of 1,189,100N with some additional mineralization associated with the Clark Tunnel fault. The northern boundary of the antimony domain was defined using indicator kriging and the southern boundaries are defined by the same structures that control gold mineralization. Bradley Mining Company data was excluded from the 0.01% indicator kriging shell definition due to low precision of antimony assays in this data set.

Silver estimation domains were based on a combination of the antimony domains and gold domains discussed above as high-grade silver occurs preferentially in regions of stibnite mineralization. The deposit was divided into four silver domains: silver domains 2 and 3 correspond to the Southern- and Clark Tunnel antimony domains respectively, silver domain 1 comprises other regions of the primary gold ore domains, and silver domain 4 makes up the rest of the deposit. Use of a similar estimation plan for both antimony and silver was selected to help maintain the multivariate relationship between the primary economic metals in the deposit.

An oxide shell was constructed to encompass the oxidized region of the deposit and contains the majority of samples with cyanide recoverable gold, primarily located in the Homestake area.

Figure 14-3: Yellow Pine Estimation Domains



14.3.6 Compositing

Gold, antimony and silver were composited downhole on 10 ft intervals with composite lengths adjusted to break at gold estimation domain boundaries and to eliminate residual short composites. The 10 ft composite length is an even multiple of the 5 ft average sample length and is also appropriate for estimation into 20 ft bench height blocks. The majority of samples in the deposit average 5 ft but some campaigns used longer samples outside of mineralized zones. Composites were assigned to estimation domains by tagging within the 3D domain solids in MS3D.

14.3.7 Composite Statistics and Capping

Descriptive statistics, histograms and probability plots were generated for ten-foot composites within each estimation domain for both clustered and declustered composites. Outliers were identified using log probability plots and were also reviewed spatially in MS3D. For gold, capping grades of 12 g/t Au within Domains 6,7 and 11; and 7 g/t Au within other domains were selected. Capping grades of 8% antimony and 100 g/t silver were selected within the main antimony shell with 10 g/t silver applied elsewhere. Capped grade statistics are presented for comparative purposes in Table 14-5 through Table 14-8 but outliers in the estimation plan are handled employing a 40 ft range restriction on high-grade composites rather than through explicit capping.

Table 14-5: Descriptive Statistics for Primary Gold Domain Composites (g/t Au)

Domain	Data Set	Number	Mean	Std Dev	Coeff Var	Max	Upper uartile	Median	Lower Quartile	Capping Grade	Metal Removed
Au_Dom3	raw composites	407	1.42	1.41	0.99	6.77	2.26	1.04	0.24	n/a	0.0%
	capped + declus		1.22	1.31	1.07	6.77	1.93	0.75	0.16		
Au_Dom5	raw composites	602	1.29	1.38	1.07	11.68	2.05	0.93	0.13	7	0.9%
	capped + declus		1.13	1.24	1.09	7	1.89	0.69	0.08		
Au_Dom6	raw composites	4774	2.39	1.79	0.75	20.31	3.2	2.15	1.21	12	0.5%
	capped + declus		2.11	1.89	0.89	12	2.95	1.81	0.69		
Au_Dom7	raw composites	1602	2.09	2.22	1.06	18.24	3.23	1.28	0.42	12	0.6%
	capped + declus		1.64	1.91	1.16	12	2.48	0.87	0.25		
Au_Dom11	raw composites	3058	1.57	2.07	1.32	21.66	2.18	0.78	0.19	12	1.4%
	capped + declus		1.4	1.9	1.35	12	1.85	0.63	0.17		
Au_Dom12	raw composites	195	0.85	1.55	1.83	14.4	1.08	0.36	0.07	7	4.9%
	capped + declus		0.78	1.16	1.49	7	1.11	0.41	0.07		

Table 14-6: Descriptive Statistics for Low Grade Secondary Gold Domain Composites (g/t Au)

Domain	Data Set	Number	Mean	Std Dev	Coeff Var	Max	Upper Quartile	Median	Lower Quartile	Capping Grade	Metal Removed
Au_Dom1	raw composites	2182	0.21	0.59	2.82	7.94	0.13	0.02	0	7	0.0%
	capped + declus		0.24	0.63	2.97	7	0.17	0.03	0		
Au_Dom2	raw composites	88	0.28	0.73	2.58	3.94	0.21	0.01	0	n/a	0.0%
	capped + declus		0.3	0.68	2.28	3.94	0.29	0.02	0		
Au_Dom4	raw composites	366	0.22	0.31	1.38	2.82	0.32	0.12	0.04	n/a	0.0%
	capped + declus		0.24	0.33	1.37	2.82	0.34	0.12	0.04		
Au_Dom8	raw composites	1032	0.43	0.76	1.79	9.47	0.49	0.2	0.04	7	2.3%
	capped + declus		0.43	0.76	1.75	7	0.49	0.2	0.04		
Au_Dom9	raw composites	239	0.46	0.83	1.81	6.34	0.49	0.22	0.08	n/a	0.0%
	capped + declus		0.54	1.07	1.97	6.34	0.5	0.22	0.08		
Au_Dom10	raw composites	2944	0.19	0.42	2.21	6.08	0.17	0.05	0.01	n/a	0.0%
	capped + declus		0.2	0.42	2.09	6.08	0.19	0.06	0.01		
Au_Dom13	raw composites	3996	0.21	0.51	2.47	8.87	0.18	0.06	0.01	7	0.0%
	capped + declus		0.21	0.46	2.16	7	0.21	0.07	0.01		
Au_Dom14	raw composites	831	0.47	1.21	2.6	17.02	0.4	0.15	0.04	7	10.4%
	capped + declus		0.43	0.88	2.06	7	0.4	0.14	0.04		
Au_Dom15	raw composites	1028	0.14	0.36	2.64	4.54	0.1	0.02	0	n/a	0.0%
	capped + declus		0.14	0.35	2.46	4.54	0.12	0.03	0.01		
Au_Dom16	raw composites	215	0.42	0.77	1.82	4.68	0.53	0.08	0	n/a	0.0%
	capped + declus		0.44	0.87	1.98	4.68	0.43	0.06	0		

Table 14-7: Descriptive Statistics for Antimony Composites (% Sb)

Domain	Data Set	Number	Mean	Std Dev	Coeff Var	Max	Upper Quartile	Median	Lower Quartile	Capping Grade	Metal Removed
Sb_Dom0	raw composites	13455	0.014	0.094	6.742	3.89	0.03	0.02	0.001	n/a	0.0%
	capped + declus		0.013	0.095	7.108	3.89	0.003	0.002	0.001		
Sb_Dom2	raw composites	5240	0.359	0.96	2.677	14.9	0.27	0.02	0.003	8	1.7%
	capped + declus		0.281	0.796	2.736	8	0.17	0.006	0.002		
Sb_Dom3	raw composites	206	0.534	1.492	2.796	15.12	0.48	0.12	0.02	8	6.7%
	capped + declus		0.362	0.774	2.139	8	0.353	0.093	0.02		

Table 14-8: Descriptive Statistics for Silver Composites (g/t Ag)

Domain	Data Set	Number	Mean	Std Dev	Coeff Var	Max	Upper Quartile	Median	Lower Quartile	Capping Grade	Metal Removed
Ag_Dom1	raw composites	2671	1.43	3	2.1	85.71	1.77	0.7	0.25	10	15.2%
	capped + declus		1.34	1.69	1.26	10	1.73	0.7	0.25		
Ag_Dom2	raw composites	2408	4.66	12.02	2.58	152.34	3.17	1.72	0.75	100	1.7%
	capped + declus		4.05	10.16	2.51	100	2.79	1.38	0.51		
Ag_Dom3	raw composites	199	9.27	37.78	4.07	457.5	6.05	2.98	1.79	20	32.3%
	capped + declus		4.1	4.76	1.16	20	4.71	2.52	1.42		
Ag_Dom4	raw composites	10174	0.5	1.63	3.29	99.92	0.31	0.25	0.25	7	7.8%
	capped + declus		0.47	0.67	1.42	7	0.35	0.25	0.25		

14.3.8 Spatial Statistics

Semi-variogram models were generated for gold, antimony, silver and cyanide gold recovery ratio (oxidation) to determine spatial continuity and to guide search ellipse orientations and anisotropies. Experimental variograms were generated in GSLIB software for the primary gold, antimony and silver estimation domains. Variograms were not modeled for secondary domains or for the Clark tunnel antimony shell. Gold mineralization typically displays greatest continuity parallel to northeasterly striking fault zones while antimony and silver show maximum continuity along northwesterly striking antimony vein arrays. Oxidation follows the historical topographic surface. Gold variogram models typically have a nugget of 10-18%, a short-range structure achieving 60% of the sill at a distance of approximately 40 to 50 ft and a maximum range of 130-295 ft.

14.3.9 Block Model Parameters and Grade Estimation

The block model mineral resource estimate for Yellow Pine was developed with block dimensions of 40 x 40 x 20 ft with coordinates defined in Table 14-9. Blocks were discretized into a 4 x 4 x 2 array of points during estimation.

Table 14-9: Block Model Definition for Yellow Pine

Deposit	Dimension (ft)			Origin (ft)[1]			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Yellow Pine	40	40	20	2,729,740	1,185,700	4,500	155	170	152	0

Notes: [1]*Lower left hand block model corner, NAD83 ID State Plane West feet*

The Yellow Pine drillhole database contains 1,843 core density measurements within an average density of 2.63 g/cc. Average density values for were calculated for each gold estimation domain after removal of outliers and assigned to the block model.

A multiple percent model was used for the Yellow Pine deposit to accurately capture discrete regions of mineralization occurring within some narrow geological zones and to also allow for accurate forecasting of mining dilution under different extraction scenarios. The volume of each block occurring within each of the 16 gold domains was calculated and stored in the model as a percentage. For the blocks occurring within multiple domains, blocks were assigned a domain code and percentage for both a primary gold domain and a secondary gold domain based on majority by volume. Gold grade estimates were then stored in two fields, primary and secondary, to allow accurate reporting of partial block in-situ resources as well as full block diluted grades. Blocks were assigned to silver and antimony domains by majority.

Gold, antimony and silver were estimated using ordinary kriging or inverse distance squared interpolation. Generally, blocks were estimated using a two-pass search strategy with approximately 2/3 estimated in the first pass and the remaining estimated in the second pass within the ore domains. The estimates used hard boundary conditions with only samples in an estimation domain used to inform blocks in that domain. Ordinary kriging was used to estimate grades in the five primary gold domains, the primary antimony shell and the four silver domains. Inverse distance squared was used for the remaining gold domains. The first gold estimation pass range was generally based on the ranges of the variogram model, approximately 100-200' for the primary direction with the second pass expanded to twice the range of the first pass. A minimum of three octants and five composites was required in the first pass with octant requirements relaxed in additional passes. Densely drilled domains had a maximum of three composites per hole with more composites allowed in other domains. The inverse distance searches either increased the search ellipse range or decreased the octant requirements in subsequent passes to control grade extrapolation away from data and estimate an appropriate number of blocks. Capping was applied in the software using a range limiting method with un-capped samples allowed up to a maximum distance of 40' (one block) and capping grades of 8 or 12 g/t Au applied after that. This method was selected to adequately capture local high grade in the deposit, which was often clustered, while limiting the extrapolation of higher grade beyond reasonable distances.

Figure 14-4: Yellow Pine Gold Block Model

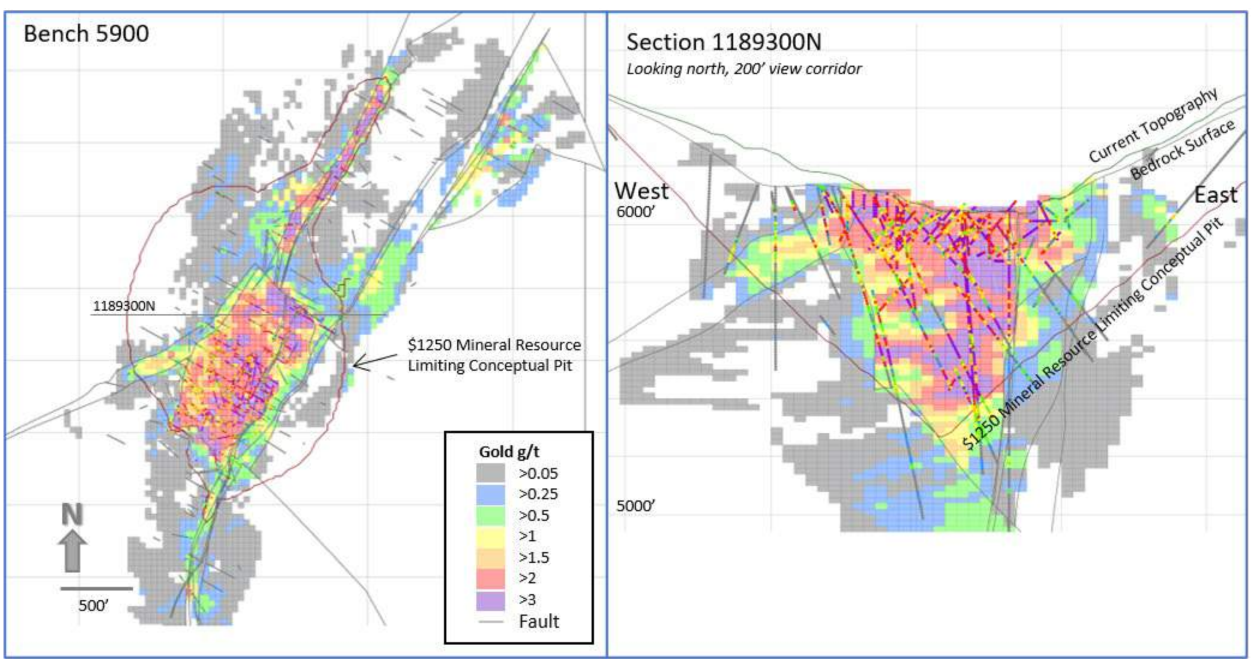
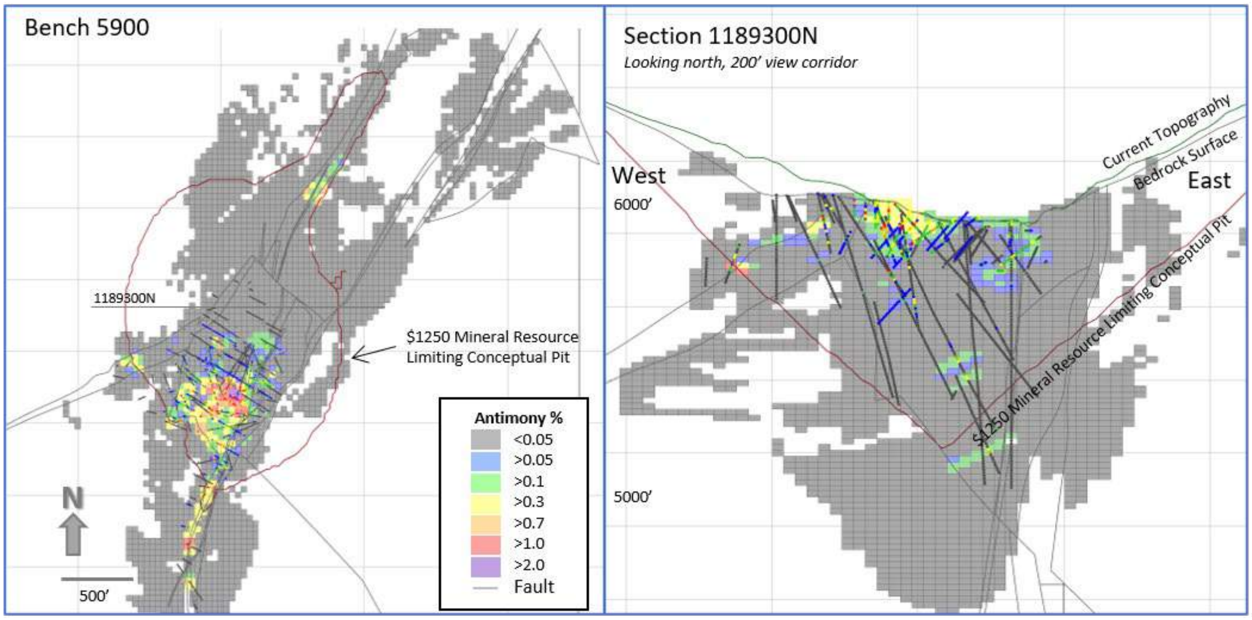


Figure 14-5: Yellow Pine Antimony Block Model



14.3.10 Block Model Validation

The block model for Yellow Pine was validated by completing a series of graphical inspections, bias checks, sensitivity studies, comparison to prior estimates and reconciliation against historical production records. Graphically, the model was validated by visually comparing the composites to estimated block grades on plan and section views. Global bias was assessed through comparison of average declustered composite grades and block grades for each estimation domain. Multiple model sensitivities were run to assess the impact of historical data on the estimate, selection of capping grades, kriging search neighborhood and choice of interpolation method. Exclusion of the pre-1953 drill hole data results in a 2.2% reduction in mineralized tonnage with no appreciable reduction in gold grade at a 0.75 g/t Au cutoff grade, reported within a conceptual pit shell. Other sensitivities showed similar magnitude changes to the Yellow Pine Mineral Resource.

14.3.11 Geochemical Estimates

In addition to gold, antimony and, silver, a suite of estimates of geochemical element concentrations were prepared to support geo-metallurgical and geo-environmental engineering. Additional elements estimated include sulfur, arsenic, mercury, iron, calcium, magnesium and potassium which were all analyzed for Midas Gold drillholes. The estimation methodology generally followed that used for the commodities consisting of data exploration, domain definition, block estimation and model validation. Elements were composited into the same 10' intervals as used for gold and were estimated using either ordinary kriging or inverse distance interpolation. Capping was not warranted as geochemical elements are typically more normally distributed than the precious metals and underestimation of deleterious elements poses a risk to the project. A summary of the estimates is provided below:

- Arsenic and sulfur were estimated within six estimation domains broadly similar to those used for gold, segregating regions of hydrothermal alteration from less altered rock units. Arsenic and sulfur were estimated using ordinary kriging. The sulfur estimate was limited to pyritic sulfur with stibnite sulfur calculated from the antimony block estimate. Intrusive host rock lithology was also used to correct for variations in sulfur grade observed between granodiorite and more felsic intrusives.

- Mercury was estimated within four domains based on a modified antimony shell, southern MCFZ, Homestake area and elsewhere. Mercury was estimated using inverse distance cubed interpolation to capture observed short range variability of the late stage overprinting mercury mineralization event.
- Calcium, magnesium and iron were estimated within nine estimation domains generally constructed to honor lithologic units including clastic vs carbonate metasediments, fault zones, and intrusive rocks. These elements were estimated using inverse distance cubed interpolation.
- Potassium shows only minor variability throughout the deposit and was estimated using ordinary kriging within a single estimation domain. The resultant model adequately captures the potassic alteration zonation associated with the main stage gold mineralization event as well as variations within the metasediments.

14.4 HANGAR FLATS

14.4.1 Mineral Resource Estimation Procedures

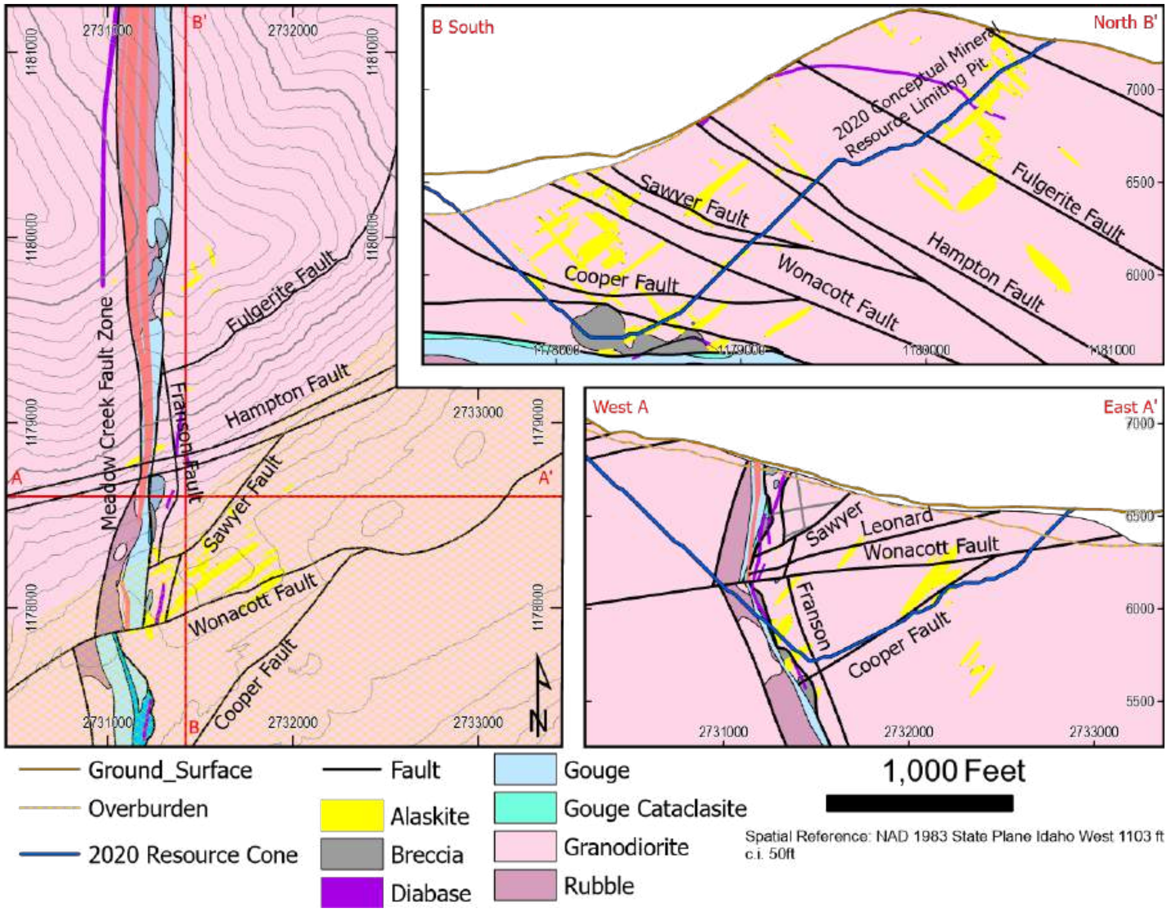
The Hangar Flats Mineral Resource estimate is based on the validated drill hole database, interpreted three-dimensional geological model, digitized as-built data of historical workings, and LiDAR topographic data. The geologic modeling was completed using the commercially available software Seequent Leapfrog Geo 4.3. The estimation of mineral resources was completed using commercial three-dimensional block modelling and mine planning software Hexagon Minesight™ MS3D Version 15.10.

14.4.2 Geologic Modeling

The Hangar Flats Mineral Resource estimate is based on a generalized geologic model consisting of major rock types, pre- and post-mineralization structures and post-mineralization tertiary dikes. Modeling was conducted using both sectional and implicit modeling methods to guide surface construction and incorporate legacy underground mapping information captured in GIS. Tertiary dike rocks; rhyolite and diabase, cut gold mineralization and were modeled as sets of dikes striking north-south oriented sub-vertically to allow for accurate estimates of mining dilution. Unconsolidated overburden consisting of till, alluvium, and backfilled ground, were modeled using data from drilling and field observations.

The most important control on mineralization in the Hangar Flats deposit is the Meadow Creek Fault Zone (MCFZ) which is a wide, northerly striking, right lateral shear zone with zones of clay gouge and silicified breccias which forms the western boundary mineralization within the deposit. Modeling of the shear zone focused on defining discontinuous blocks of highly mineralized breccia and quartz monzonite adjacent to and entrained in the anastomosing clay gouge zones. The gouge itself was subdivided into three units, post-mineralization light colored gouge, foliated cataclasite, and sulfidic dark colored gouge. The plutonic rocks are divided into a felsic alaskite and a slightly more intermediate quartz monzonite. These rocks were distinguished from one another through geologic logging and geochemical classification and modeled using implicit modeling techniques. Mineralization in the Hangar Flats deposit is controlled primarily by the north-south trending MCFZ which has been mapped in underground mining of the Meadow Creek Mine (MCM) and the DMEA Tunnel. The secondary control of mineralization is a series of northeast trending structures that splay from or cut the MCFZ and dip moderately to the northwest. These structures have provided ground preparation and served as conduits for mineralized fluids. Three series of faults were modeled, north-south faults parallel to the MCFZ, northeast striking shallowly dipping splay structures, and northeast striking post-mineralization faulting. Figure 14-6 illustrates the Hanger Flats Geological model.

Figure 14-6: Hangar Flats Geological Model



14.4.3 Controls on Mineralization

The MCFZ is the principal structure controlling mineralization. The eastern mineralized corridor of the MCFZ varies in width from about 100 to 250 feet. Gold mineralization and antimony mineralization form elongate ore shoots adjacent to the eastern boundary of the MCFZ at the intersections of the MCFZ and numerous low angle structures. Mineralization occurs as north-plunging breccias and shoots of massive stibnite antimony mineralization, sulfide biotite replacements, and stockworks of quartz-sulfide veining. Mineralization to the east is in northeast striking, moderately northwest dipping structures that are interpreted as splays of the MCFZ with gold and silver mineralization in quartz-sulfide veins and sulfide biotite replacements. Late stage faulting locally offsets the MCFZ. The MCFZ changes from dipping nearly vertical in the north to dipping 45 degrees east to the south across the Wonacott fault, a major northeasterly striking structure. The geometry and spatial extents of mineralization on the west side of the MCFZ are uncertain due to low density of drilling.

14.4.4 Exploratory Data Analysis and Data Preparation

Exploratory data analysis and graphical data review were performed on raw assays within ten geological solids to aid in construction of appropriate geostatistical estimation domains. Quantitative data analysis included generation of descriptive statistics, box plots, histograms, log-probability plots, and analysis of multivariate relations. The data was also reviewed relative to surfaces representing historical underground and surface mining. Data preparation included assignment of numeric values to samples assaying below detection limits (generally 1/2 detection limit or lower for legacy data) and to intervals which were selectively un-assayed. In addition, samples sourced from non-bedrock materials, including those from backfilled pits and waste rock dumps, were removed from the dataset.

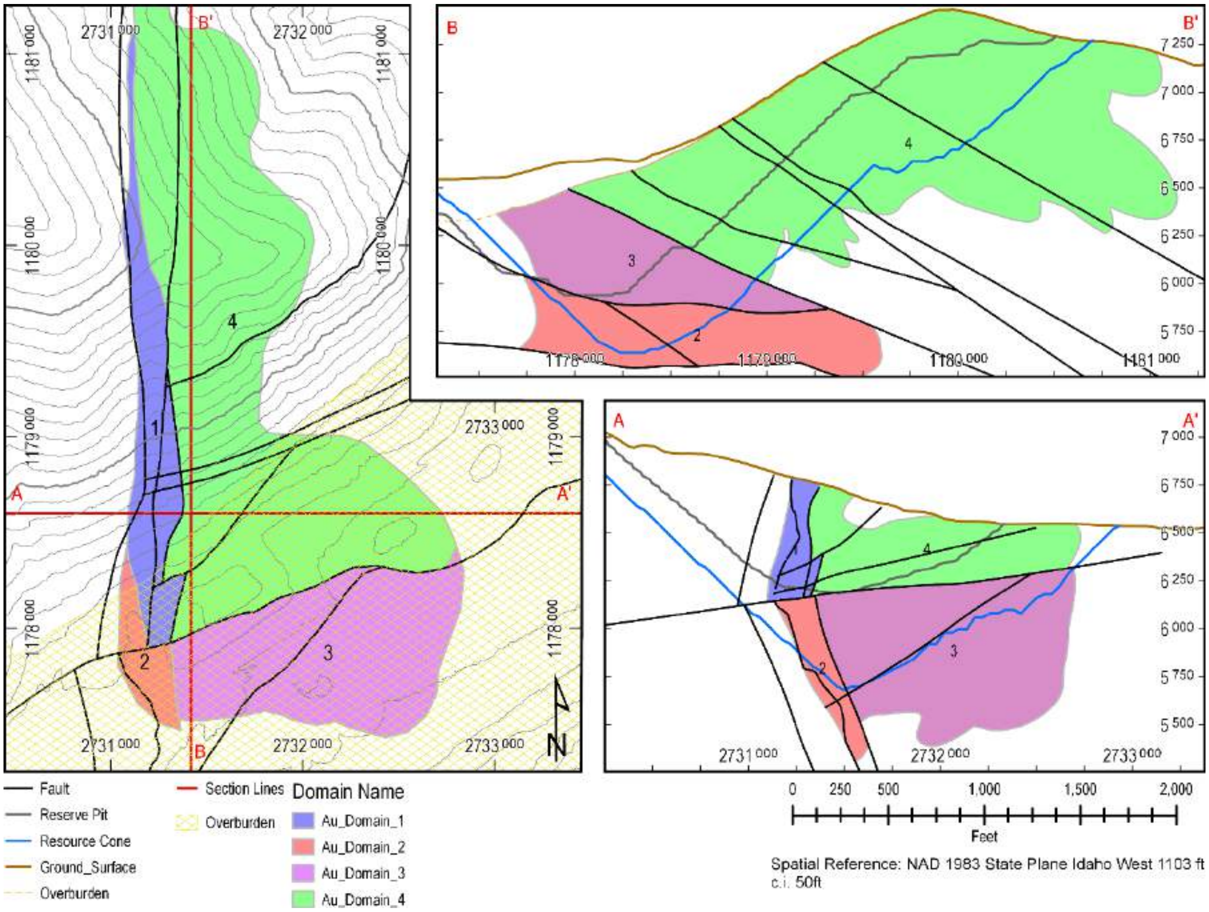
14.4.5 Estimation Domain Modeling

The Hangar Flats estimation domains are based on the major fault zones and fault units in the geological model as well as grade shells constructed using indicator kriging methods. Four estimation domains were defined for gold within the 0.1 g/t grade shell; D1 is the MCFZ structural corridor between the north striking Franson Fault a MCFZ gouge in the hanging wall of the Wonacott Fault; D2 is the MCFZ structural corridor in the footwall of the Wonacott Fault; D3 is the footwall of the Wonacott Fault, east of D2 and D4 is the hanging wall of the Wonacott Fault east of D1. The antimony estimation domains use the same structural boundaries as gold but are constrained within a 0.05% antimony grade shell that is less extensive than the gold and silver mineralization. The silver estimation domains are the same as those used for the gold estimate. Oxidation in the deposit is primarily controlled by depth below the ground surface and two domains were constructed by defining two areas with different topographic slopes; northeast versus south. See Table 14-10 and Figure 14-7.

Table 14-10: Gold & Antimony Estimation Domain Codes

Domain Number	Name	Category	Lithology	Description
1	Au_Domain_1 Sb_Domain_1	NS Trending	Intrusives & faulted rocks	Domain 1 is located between the Franson Fault and the approximate eastern margin of the MCF gouge. The southern boundary is the Wonacott Fault that cuts and displaces the mineralization. This zone nearly encompasses the historical Meadow Creek Mine. Mineralization is oriented NS with moderately plunging shoots with a minor axis in the EW direction. This area is limited with a 0.1 gpt grade shell for Au or 0.05% for Sb.
2	Au_Domain_2 Sb_Domain_1	NS Trending	Intrusives & faulted rocks	Domain 2 is located between the Frylock Fault and the approximate eastern margin of the MCF gouge. The northern boundary is the Wonacott Fault and the south boundary is a 0.1 gpt grade shell Au or 0.05% for Sb. Mineralization in this domain strikes north-south and plunges north.
3	Au_Domain_3 Sb_Domain_1	NE Trending	Quartz Monzonite & Alaskite	Domain 3 is bounded to the west by the Frylock Fault and the north by the Wonacott Fault. The rest of the boundary is a 0.1 gpt shell Au or 0.05% for Sb. The mineralization strikes northeast and dips northwest.
4	Au_Domain_4 Sb_Domain_4	NE Trending	Quartz Monzonite & Alaskite	Domain 4 is bounded on the west by both the MCF gouge zone and the Franson Fault. The southern boundary is the Wonacott Fault. The remainder is defined by a 0.1 gpt shell Au or 0.05% for Sb. The mineralization strikes northeast and dips northwest.
0	Au_Domain_0	Unmineralized	Intrusives & faulted rocks	This domain is primarily unmineralized and encompasses all of the area of the model outside of the other gold domains and below the ground surface.

Figure 14-7: Hangar Flats Estimation Domains



14.4.6 Compositing

Gold, antimony and silver were composited downhole on 10 ft intervals with composite lengths adjusted to break at gold estimation domain boundaries and to eliminate residual short composites. The 10 ft composite length is an even multiple of the 5 ft average sample length and is also appropriate for estimation of 20 ft bench height blocks. The majority of samples in the deposit average 5 ft but some campaigns used longer samples outside of mineralized zones. Composites were assigned to estimation domains by tagging within the 3D domain solids in MS3D.

14.4.7 Composite Statistics and Capping

To mitigate risk associated with use of high-grade statistical outliers, capping grades were selected for each estimation domain after declustering and weighting raw composite data. Capping grade was evaluated through log probability plots and analysis of contained metal within deciles and centiles, following the Parrish Method (Parrish, 1997). Both methods yielded similar results and final composite capping levels are shown in Table 14-11 through Table 14-13.

Table 14-11: Descriptive Statistics for Gold Domain Composites (g/t Au)

Gold	Data set	Number	Mean	Std Dev	Coeff Var	Max	Lower Quartile	Median	Upper Quartile	Capping Grade	Metal Removed
Au_Domain_1	raw composites	1,344	1.74	2.08	1.19	15.53	0.19	0.98	2.62	10	0.46%
	declustered		1.52	1.93	1.27	15.53	0.19	0.98	2.62		
	capped + declus		1.51	1.90	1.26	10.00	0.19	0.98	2.62		
Au_Domain_2	raw composites	780	1.18	1.45	1.23	8.16	0.10	0.66	1.65	7.5	0.14%
	declustered		1.13	1.42	1.25	8.16	0.10	0.66	1.65		
	capped + declus		1.13	1.41	1.25	7.50	0.10	0.66	1.65		
Au_Domain_3	raw composites	2,216	0.79	1.22	1.53	14.09	0.05	0.28	1.04	7.5	0.81%
	declustered		0.61	1.07	1.75	14.09	0.05	0.28	1.04		
	capped + declus		0.61	1.02	1.69	7.50	0.05	0.28	1.04		
AU_Domain_4	raw composites	4,390	0.35	0.73	2.11	8.71	0.02	0.09	0.32	7.5	0.26%
	declustered		0.37	0.78	2.12	8.71	0.02	0.09	0.32		
	capped + declus		0.37	0.77	2.10	7.50	0.02	0.09	0.32		

Table 14-12: Descriptive Statistics for Silver Domain Composites (g/t Ag)

Silver	Data set	Number	Mean	Std Dev	Coeff Var	Max	Lower Quartile	Median	Upper Quartile	Capping grade	Metal Removed
Au_Domain_1	raw composites	1172	11.21	108.89	9.71	3160.00	0.48	1.80	4.13	150	45%
	declustered		12.54	120.54	9.61	3160.00	0.48	1.80	4.13		
	capped + declus		6.15	19.13	3.11	150.00	0.48	1.80	4.13		
Au_Domain_2	raw composites	668	8.52	30.54	3.58	381.95	0.43	1.35	3.70	150	10%
	declustered		8.78	32.54	3.71	381.95	0.43	1.35	3.70		
	capped + declus		7.63	23.38	3.06	150.00	0.43	1.35	3.70		
Au_Domain_3	raw composites	2112	1.11	2.45	2.21	65.96	0.25	0.46	1.25	7	5.4%
	declustered		0.91	1.98	2.17	65.96	0.25	0.46	1.25		
	capped + declus		0.86	1.08	1.26	7.00	0.25	0.46	1.25		
Au_Domain_4	raw composites	3990	0.82	5.90	7.20	238.18	0.25	0.25	0.53	7	25%
	declustered		0.87	5.27	6.09	238.18	0.25	0.25	0.53		
	capped + declus		0.61	0.94	1.54	7.00	0.25	0.25	0.53		

Table 14-13: Descriptive Statistics for Antimony Domain Composites (% Sb)

Antimony	Data Set	Number	Mean	Std Dev	Coeff Var	Max	Lower Quartile	Median	Upper Quartile	Capping Grade	Metal Removed
Sb_Domain_1	raw composites	1114	0.34	0.91	2.63	9.13	0.01	0.02	0.25	4	10.76%
	declustered		0.31	0.90	2.94	9.13	0.01	0.02	0.25		
	capped + declus		0.27	0.69	2.50	4.00	0.01	0.02	0.25		
Sb_Domain_2	raw composites	618	0.54	1.98	3.65	25.54	0.00	0.01	0.14	7	19.32%
	declustered		0.54	2.00	3.72	25.54	0.00	0.01	0.14		
	capped + declus		0.43	1.22	2.81	7.00	0.00	0.01	0.14		
Sb_Domain_3	raw composites	442	0.16	0.37	2.22	3.50	0.01	0.03	0.17	2	8.44%
	declustered		0.19	0.44	2.32	3.50	0.01	0.03	0.17		
	capped + declus		0.17	0.34	2.00	2.00	0.01	0.03	0.17		
Sb_Domain_4	raw composites	75	0.17	0.48	2.79	2.62	0.00	0.00	0.07	0.7	47.49%
	declustered		0.20	0.53	2.62	2.62	0.00	0.00	0.07		
	capped + declus		0.11	0.19	1.80	0.70	0.00	0.00	0.07		

14.4.8 Spatial Statistics

Semi-variogram models were generated for gold, antimony, and silver in GSLIB software for the primary gold, antimony and silver estimation domains. Continuity of gold, silver, and antimony mineralization in domains 1 and 2 is typically greatest parallel to the north-south, steeply dipping orientation of the MCFZ. Other domains show greatest continuity along NE to EW striking, shallowly to moderately NW dipping trends, parallel to north-easterly faults. Variogram models typically reach the sill at a range of 140-250 ft and obtain 60% of the sill at distances of approximately 40 ft.

14.4.9 Block Model Parameters and Grade Estimation

The Mineral Resource Estimate for Hangar Flats was developed with block dimensions of 40 x 40 x 20 ft with coordinates defined in Table 14-14. The selected block size is approximately 30% of the median spacing of Midas Gold drill holes and is consistent with conceptual mining bench heights. Blocks were discretized into a 4 x 4 x 2 array of points during estimation.

Table 14-14: Block Model Definition for Hangar Flats

Deposit	Dimension (ft)			Origin (ft)[1]			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Hangar Flats	40	40	20	2,729,000	1,176,700	5,140	112	152	138	0

[1]*Lower left hand block model corner, NAD83 Datum Idaho State Plane West (feet)*

The Hangar Flats drillhole database contains 917 bulk density measurements from MGI drill core. Most measurements were made by MGI on core samples using a hydrostatic weighting method with approximately 10% verified by an outside laboratory using a wax coating water immersion method. Density variations were observed within rock types and associated with mineralization. Density was estimated using inverse distance squared interpolation within 500 ft of samples or assigned the mean density for the rock type, found in Table 14-15.

Table 14-15: Density Assignment Values for Hangar Flats Rock Types

Rock Type	Sample Count	Mean ρ (g/cc)	Std Dev
Alaskite	44	2.61	0.033
Breccia	34	2.66	0.123
Cataclasite	18	2.63	0.057
Dark Gouge	12	2.56	0.058
Diabase	17	2.63	0.078
Fault Material	18	2.65	0.028
Light Gouge	42	2.53	0.053
Quartz Monzonite	691	2.63	0.043
Overburden	-	1.75*	-
Rhyolite	16	2.54	0.029
Rubble	25	2.62	0.028

A multiple percent model was used for the Hangar Flats block model to account for percentage of unmineralized materials (dikes & overburden) contained in each block. Blocks were assigned to domains based on majority by volume.

The Hangar Flats Mineral Resource Estimate was completed for gold, antimony and silver using the estimation domains and shells discussed previously. Gold was estimated within the four estimation domains discussed above. Blocks were estimated using a three-pass search strategy to achieve an appropriate degree of smoothing. The gold estimate used a mixture of hard and soft boundaries based upon contact plot analysis to limit grade extrapolation into unmineralized areas. Ordinary kriging was used to estimate gold and required a minimum of five composites in the first pass with a maximum of two composites for each octant. Subsequent passes relaxed the sample requirements and increased the search ranges. The first estimation pass major axis range of 200 ft was based on the variogram range and appropriate for the average drillhole spacing. The second pass used a maximum search range to 350 ft with a final estimation pass range of 500 ft. The orientation and anisotropy of the search ellipses was based on observed continuity of mineralization and variography. The estimation for silver used the same search parameters as those for gold. Antimony was estimated similarly but with reduced search ranges of 100, 200, and 300 ft, consistent with lower continuity of antimony mineralization. The ratio of cyanide recoverable gold to total gold was estimated to model degree of oxidation using a single pass inverse distance interpolation in each of the two domains discussed above. The search ellipses in each domain were aligned parallel to the general topographic surface and had a maximum range of 500 ft. Figure 14-8 and Figure 14-9 show a section and plan view of the Hanger Flats block model for gold and antimony, respectively.

Figure 14-8: Hangar Flats Gold Block Model

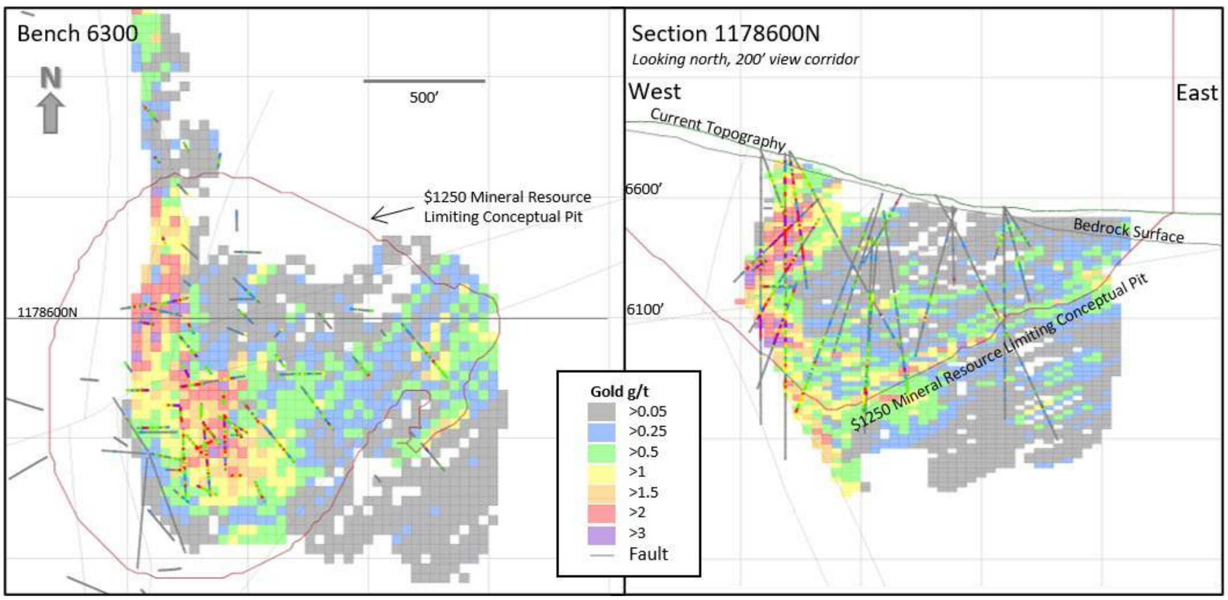
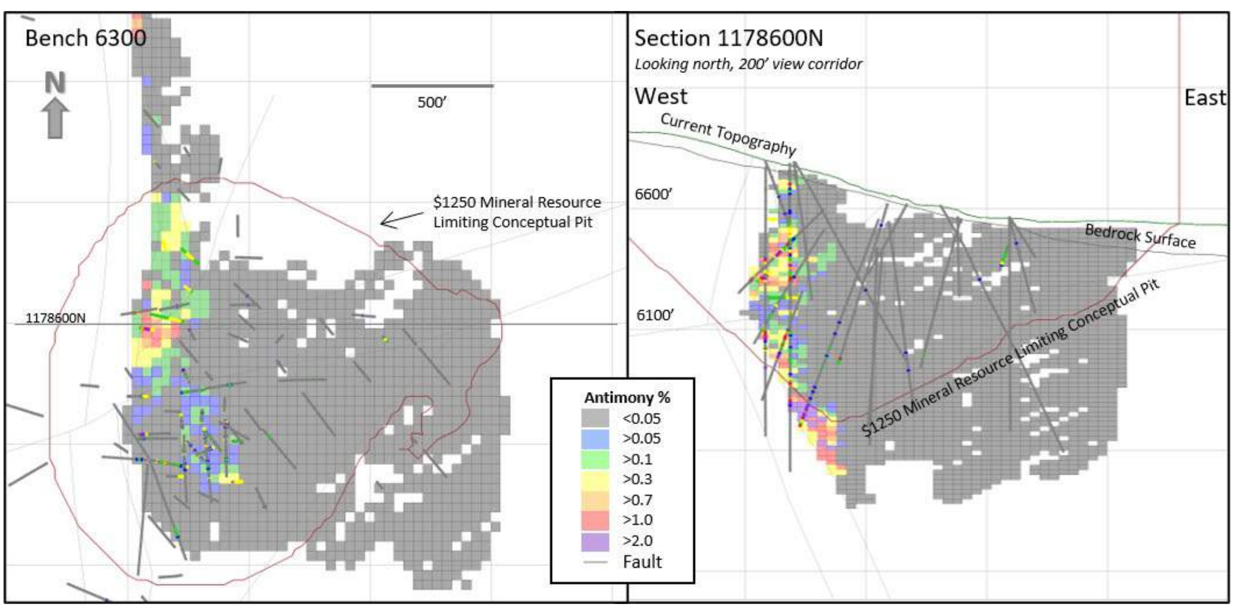


Figure 14-9: Hangar Flats Antimony Block Model



14.4.10 Block Model Validation

The block model for Hangar Flats was validating using graphical inspections, statistical comparisons, sensitivity studies, and bias checks. Graphically, the model was compared to sample composites displayed in 3D and in various sectional orientations. Descriptive statistics and plots for gold and antimony were compared with declustered statistics for each domain to assess global bias. Swath Plots were produced and inspected for local bias between composites, kriged blocks and nearest neighbor declustered block grades. Various sensitivities were run to assess the impact of estimation methods, capping grades, and sample requirements. Model sensitivities using un-capped gold composites produced a 0.4% increase in gold ounces and inverse distance cubed interpolation produced a 3.4% increase in gold ounces as reported within a conceptual pit shell.

14.4.11 Geochemical Estimates

In addition to gold, antimony and, silver, a suite of estimates of geochemical element concentrations were prepared to support geo-metallurgical and geo-environmental engineering. Additional elements estimated include sulfur, arsenic, mercury, iron, calcium, magnesium and potassium which were all analyzed for Midas Gold drillholes. The estimation methodology generally followed that used for the commodities consisting of data exploration, domain definition, block estimation and model validation. Elements were composited into the same 10 ft intervals as used for gold and were estimated using either ordinary kriging or inverse distance interpolation. For all estimates, sample selection was restricted to composites occurring within the same geological solid as the block estimated. Capping was not warranted as geochemical elements are typically more normally distributed than the precious metals and underestimation of deleterious elements poses a risk to the project. A summary of the estimates is provided below:

- Pyritic sulfur grade was estimated into blocks using ordinary kriging within the five gold domains. Pyritic sulfur was calculated for composites by subtracting out sulfur associated with stibnite. Stibnite sulfur was calculated from the estimated antimony block estimate and total sulfur grade was calculated as the sum of pyrite sulfur and stibnite sulfur. This methodology mitigates risk for metallurgical forecasting associated with disparate search strategies for sulfur and antimony.
- The elements arsenic, calcium, mercury, potassium, and sodium were estimate in five gold domains described above using either ordinary kriging or inverse distance squared interpolation using a four-pass strategy. The gold domains appropriately segregate hydrothermally altered rocks from the rest of the country rock which is the primary control on the distribution of mobile cations and deleterious metals in the deposit. Search orientations were derived from the gold estimate to best maintain the multivariate relationships observed in the samples.
- Aluminium, iron, and magnesium had their grades estimated using ordinary kriging in a single domain across the deposit in two estimation passes.
- Estimates were constrained to 1,000 feet from their nearest composite. Un-estimated blocks for all elements were assigned a mean average value for the rock for the geologic solid “rock type”.

14.5 WEST END

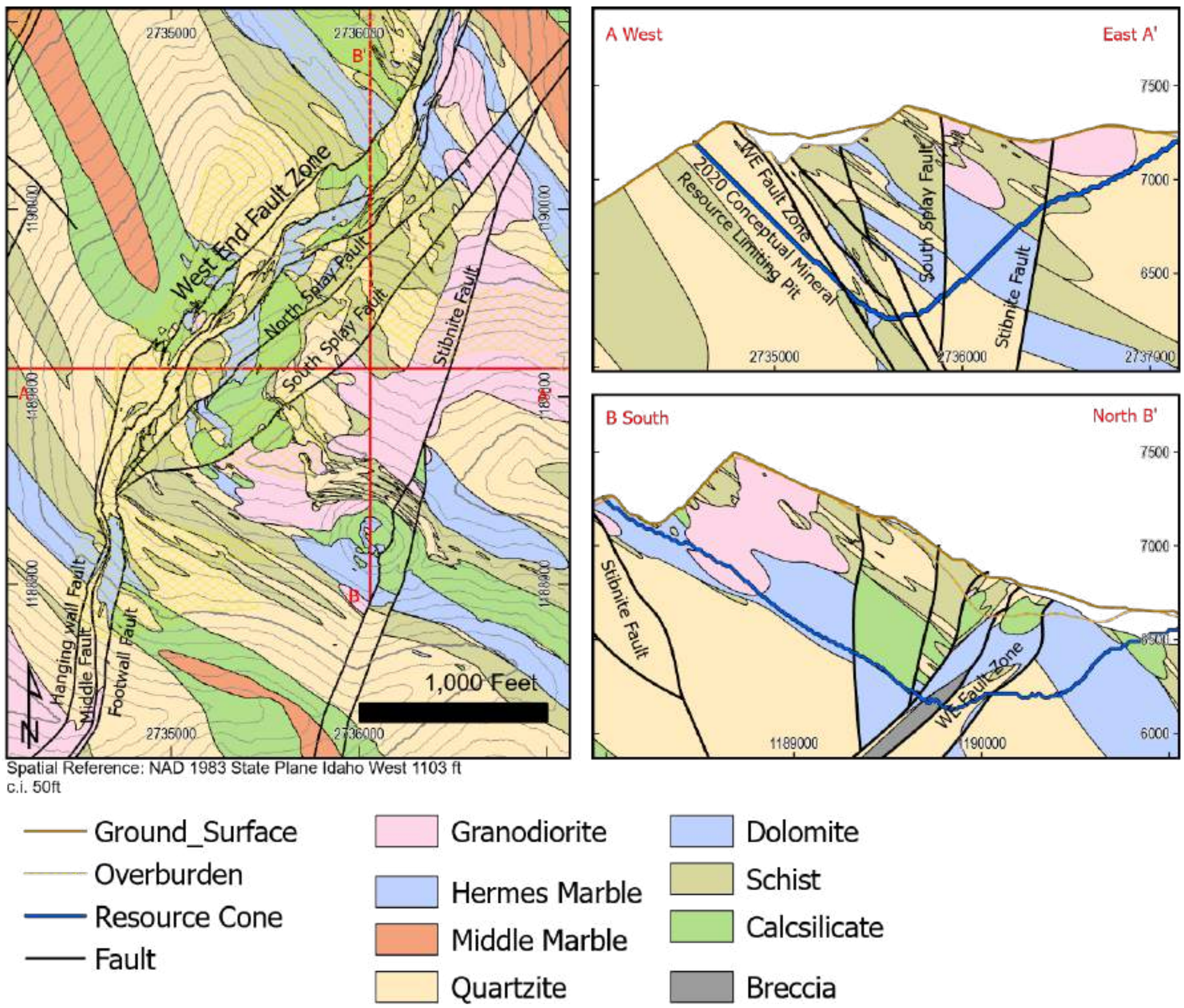
14.5.1 Mineral Resource Estimation Procedures

The West End Mineral Resource estimation is based on the validated and verified drill hole database, interpreted lithologic units, interpreted fault structures, and LiDAR topographic data. The geologic model was constructed using ARANZ Leapfrog® Geo software (Leapfrog®). The estimation of mineral resources was completed utilizing Vulcan™ resource modeling software.

14.5.2 Geologic Modeling

The West End Mineral Resource Estimate is based on a generalized geologic model consisting of major rock types, major structures, LiDAR topography, historical topography and historical pit bottom surfaces. The deposit occurs in an overturned sequence of steeply dipping Proterozoic to Paleozoic metasediments comprising the Stibnite Roof Pendant. The meta-sedimentary rocks are intruded by quartz-monzonite and granitic stocks. Mineralization occurs within fault zones, principally the southeast dipping WEFZ; as well as disseminated within preferential lithologic hosts. As discussed in Section 7, lithologic formations consist of quartzite, quartz-pebble conglomerate, interbedded quartzite and schist, limestones, dolomitic marble, and calc-silicate rocks and range in thickness from 230 – 590 feet. See Figure 14-10.

Figure 14-10: West End Geologic Model



The West End Geological Model was significantly updated from that used in the PFS. Compilation of various historical data sets from 1980s and 1990s operators including bench mapping, CAD cross sections, blast hole assays, pit-bottom as-built surfaces, as well as incorporation of additional mapping and sampling completed by Midas Gold in 2015-2017, allowed for construction of a more detailed 3D structural interpretation of the West End deposit. These data sets were integrated using Leapfrog software to geo-rectify, code and merge historical maps and sections with exploration drilling to generate the 3D geological model solids. The resulting geological model reasonably captures the geological complexity of the deposit, which has undergone numerous ductile and brittle deformation events. The geological model consists of eight lithologic units and seven fault surfaces, as well as pre- and post-mining topographic and bedrock surfaces. Principal changes to the West End Geological model since the PFS include:

- modeling of individual rock types rather than metasedimentary formations; specifically, subdivision of the quartzite-schist and quartz-pebble conglomerate formations into discrete siliciclastic and schistose geological solids;
- projection of historical surficial geological mapping data into the sub-surface;
- modeling of major splay faults as offsetting stratigraphic units in the roof pendant, as based on geological mapping;
- modeling the "Middle Fault" of the West End Fault Zone as juxtaposing various metasediment fault blocks between the hanging wall and footwall faults; and
- an improved 3D surface representing the historical West End pit bottom which accurately models individual benches and better defines pit geometry in areas with previously limited data.

14.5.3 Controls on Mineralization

Gold mineralization in the West End deposit occurs within all lithostratigraphic units with higher-grade mineralization preferentially occurring in the schist and calc-silicate lithologies as well as within silicified fault breccias of the WEFZ. Gold mineralization is associated with both disseminated sulfide replacement mineralization and with silica alteration occurring as quartz-veinlets, stockworks and zones of silica flooding. Gold also occurs along oxidized fractures and broadly disseminated within fracture zones and within intrusive units where gold is associated with sulfide-sericite alteration. Gold is concentrated along and adjacent to the WEFZ and its subsidiary structures; with mineralized drill holes observed crossing the modeled hanging wall and footwall with no apparent disruptions in gold grade. Silver mineralization within the deposit is generally low-grade and erratic. Silver mineralization is locally elevated within the WEFZ. Significant antimony mineralization is not recognized in the West End deposit.

The oxidation level in the deposit is of moderate and variable depth, with pervasive oxidation occurring at shallow levels, preferentially within certain lithologic units, and locally at deeper elevations between strands of the WEFZ and along splay structures. Significant zones of transition material are not recognized.

14.5.4 Exploratory Data Analysis and Data Preparation

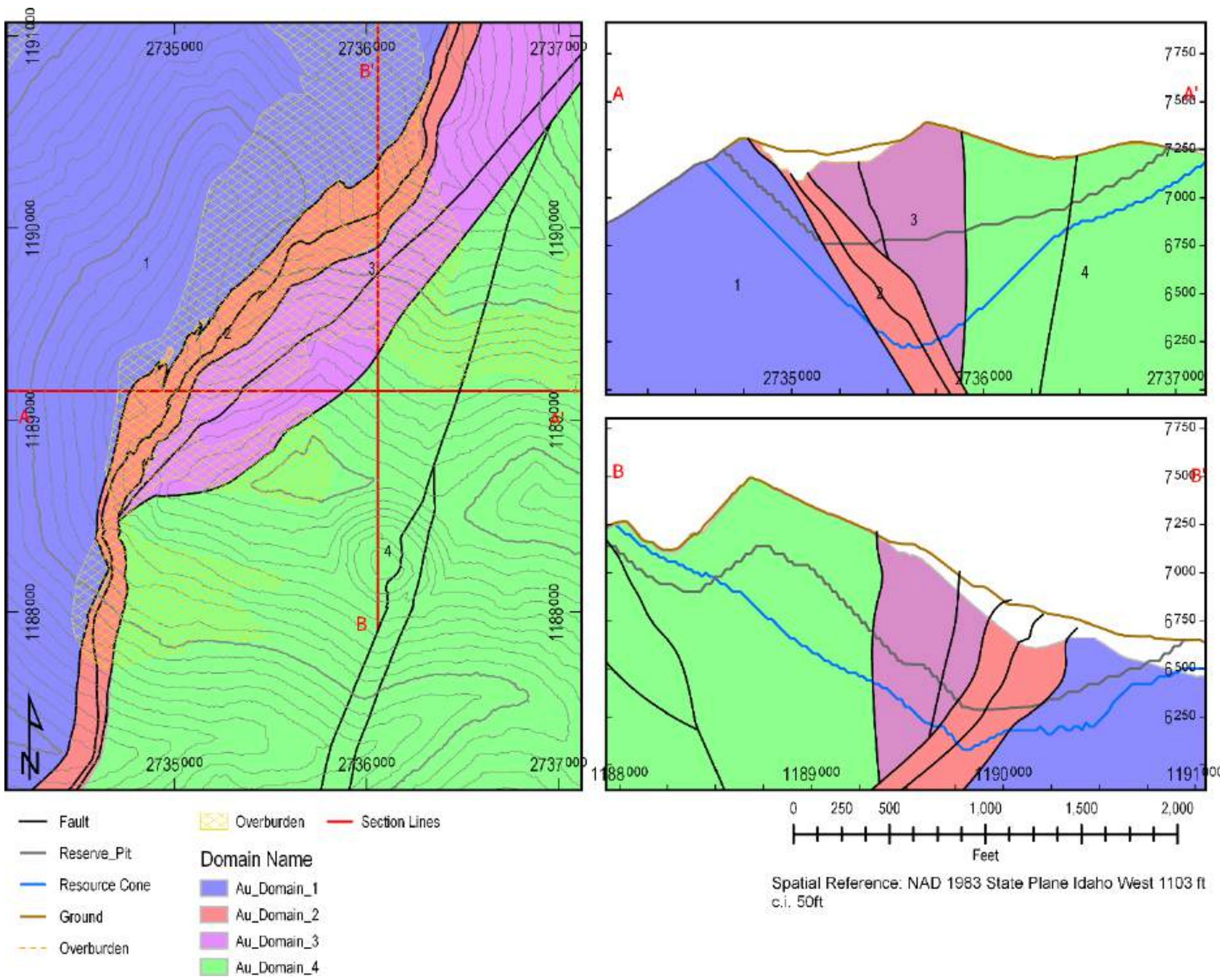
Exploration drilling in the West End deposit was conducted by multiple operators using multiple drilling and assaying methods. Detection limits for gold are quite variable, depending on the drilling campaign and assay lab used. Detection limits were adjusted to values equal to half the detection limit; levels well below those of economic interest. Some historical operators selectively used fire assays within the sulfide zones where sulfide mineralization was observed, resulting in an apparent high bias because higher-grade intervals were preferentially assayed. To address this, a new variable was created (Au_Final) combining AuFA if available, and AuCN if not, ensuring that an assay is available for every interval in holes containing partial fire assay data. While this treatment is somewhat conservative, it affects a relatively small subset of drill holes in a restricted area of the deposit and as such will not result in over-estimation of *in situ* mineral resources based on selective spot assaying of higher-grade intervals. Similar to the treatment of partial gold assays, a new variable Ag_Final was created combining fire assay and cyanide soluble silver assays for use in silver estimation.

Lithology imparts a significant control on the distribution of gold mineralization within the West End deposit. For statistical evaluation and Mineral Resource Estimation, the data was assigned to three lithologic groups with similar grade distributions. The calc-silicates, breccia and schistose lithologies are assigned to lithology group 1; the quartzites, including that of the quartz-pebble conglomerate formation to lithology group 2; and the Fern Dolomite and Granite to lithology group 3. Very little gold and silver mineralization is recognized outside of these lithologies within the Middle Marble and Hermes carbonates.

14.5.5 Estimation Domain Modeling

In addition to the lithology groups discussed above, four structural domains were defined based on the preferred orientation of mineralization being either parallel to lithology units or to fault structures. The structural domains are based on the footwall and hanging wall of the WEFZ, as well as the eastern splay fault, which shows up to 200 ft of apparent displacement of stratigraphy. Mineralization in the WEFZ (domain 2) occurs parallel to the main structure. Mineralization within the other structural domains occurs parallel to bedding within favorable lithologic units. The resultant grade estimate was therefore conducted within 12 separate estimation domains based on three lithology groups and four structural domains (see Figure 14-11).

Figure 14-11: West End Structural Domains



14.5.6 Capping and Compositing

The original drillhole sample assay values were assessed for statistical outliers using log probability plots. Gold capping levels were chosen independently for each of the lithology groups. The silver capping levels did not vary between the lithologic groups. Capping grades for samples within each lithology group are provided in Table 14-16.

Table 14-16: Capping Grades for Samples

Metal	Lith Group	Assay Type	Capping Grade	# Samples Capped	Minimum Capped Grade (g/t)	Maximum Capped Grade (g/t)	% of Metal Lost to Capping
Au	1	Total Fire & CN	23	6	23.1	26.4	0.06
		Fire	23	6	23.1	26.4	0.06
		CN Soluble	15	6	15.6	17.6	0.10
	2	Total Fire & CN	13	12	13.7	18.9	0.45
		Fire	13	12	13.7	18.9	0.43
		CN Soluble	7	12	7.1	14.1	0.66
	3	Total Fire & CN	15	7	16.1	28.2	0.70
		Fire	15	7	16.1	28.2	0.70
		CN Soluble	13	6	13.3	27.9	0.77
Ag	1	Total Fire & CN	17	7	18.6	154.3	3.20
	2	Total Fire & CN	17	12	17.1	70.3	1.60
	3	Total Fire & CN	17	12	17.7	54.5	2.50

Gold, silver, and cyanide soluble gold and silver were composited downhole on 10 ft intervals with no breaks at lithologic contacts. The 10 ft composite length is an even multiple of the average (mode) 5 ft sample length and is also appropriate for estimation of 20 ft bench height blocks. Descriptive statistics for capped composites are provided in Table 14-17 through Table 14-20.

Table 14-17: Descriptive Statistics for West End Capped Total Gold Composites

Lith Group	Count	Mean	Std	Median	Upper Quartite	Max	CV
1	9864	0.91	1.56	0.31	1.08	22.28	1.71
2	6208	0.68	1.16	0.26	0.72	15.43	1.70
3	6228	0.49	0.90	0.21	0.54	21.94	1.85

Table 14-18: Descriptive Statistics for West End Cyanide Capped Gold Composites

Lith Group	Count	Mean	Std	Median	Upper Quartite	Max	CV
1	8329	0.53	1.06	0.15	0.52	15.00	1.99
2	5224	0.41	0.75	0.17	0.40	8.33	1.84
3	5417	0.34	0.68	0.15	0.36	14.17	2.01

Table 14-19: Descriptive Statistics for West End Capped Total Silver Composites

Lith Group	Count	Mean	Std	Median	Upper Quartite	Max	CV
1	4920	1.18	1.76	0.43	1.43	17.00	1.50
2	3645	1.01	1.68	0.38	1.10	17.00	1.67
3	3237	1.20	1.93	0.41	1.41	17.00	1.60

Table 14-20: Descriptive Statistics for West End Cyanide Capped Silver Composites

Lith Group	Count	Mean	Std	Median	Upper Quartite	Max	CV
1	1551	0.69	1.49	0.22	0.66	16.46	2.17
2	729	0.69	1.64	0.26	0.62	17.00	2.35
3	1354	0.55	1.29	0.19	0.46	17.00	2.36

14.5.7 Spatial Statistics

Semi-variogram models were generated for gold and silver for each lithology group to determine spatial continuity of mineralization for use in block estimation. Gold variogram models typically have a nugget of 25-35% and a maximum range of approximately 60 ft, reaching 60% of the sill at a range of 15 to 20 feet. Silver variogram models typically have a nugget of 15-30% and a maximum range of 135-195 ft reaching 60% of the sill at a range of 15-25 feet.

14.5.8 Block Model Parameters and Grade Estimation

The West End block model used for mineral resource estimation was developed with 20 x 20 x 20 ft blocks (Table 14-21). This block size is smaller than the 40 x 40 x 20 ft blocks used for the Yellow Pine and Hangar Flats deposits and was selected to allow for accurate estimation of mineralized tonnage within narrow geological units. This method was selected in lieu of the multiple percent model approach used for Yellow Pine and Hangar Flats block models.

Table 14-21: Block Model Definition for West End

Deposit	Dimension (m)			Origin (ft)[1]			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
West End	20	20	20	2732700	1185400	5680	290	370	116	0

[1] Lower left hand block model corner, NAD83 Idaho State Plane West feet

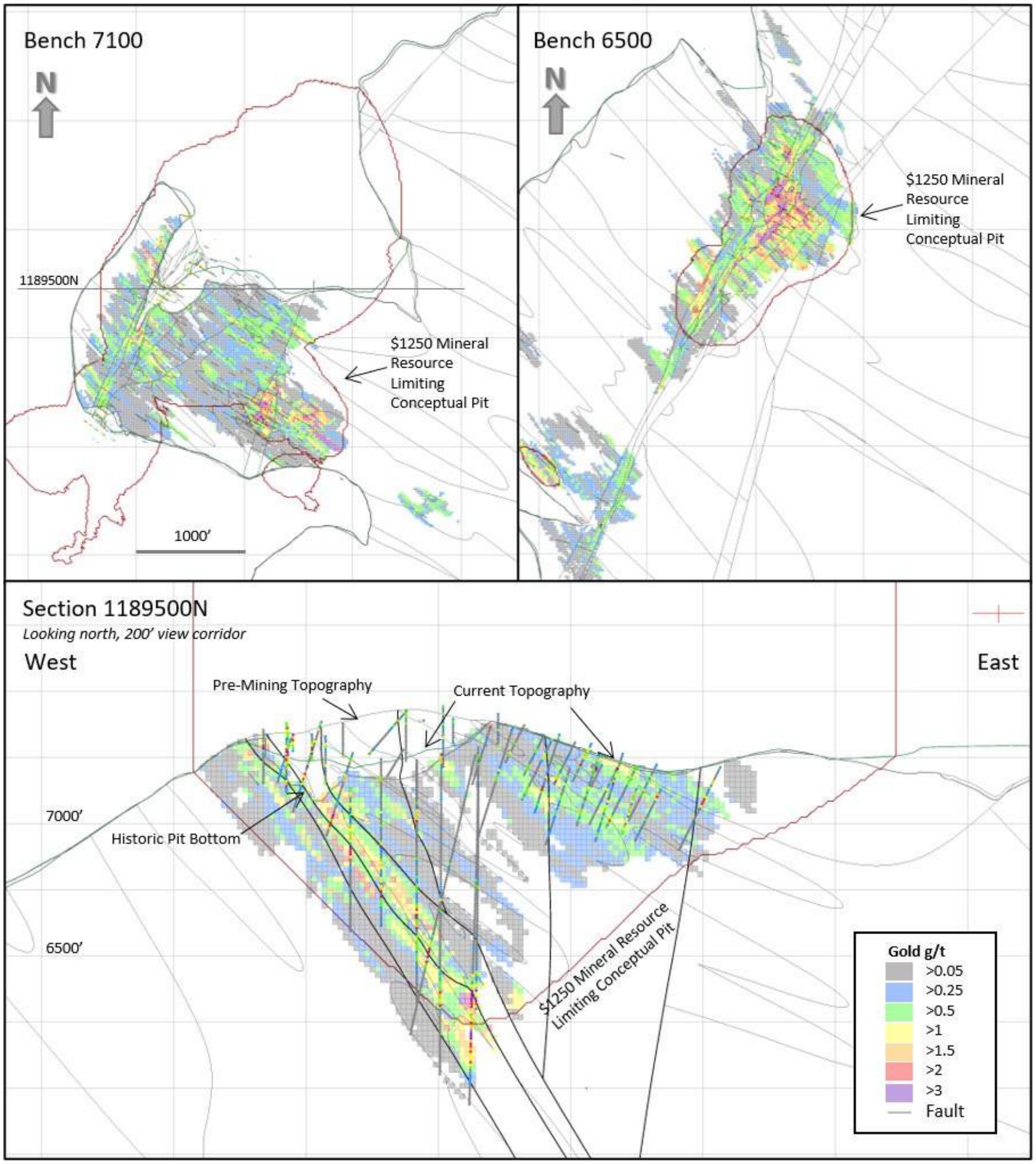
The drill hole database contains 166 density measurements from the primary lithologic units, the majority of which were determined onsite using the water immersion method, with a number of independent third-party measurements completed offsite using the same methodology. Because of the relatively small number of density measurements, density values were averaged for each lithologic unit and assigned to the geologic model after removal of outliers, as summarized in Table 14-22.

Table 14-22: Density Assignment Values for West End Lithologic Units

Rock Model Unit	Bulk Density (g/cm^3)
Breccia	2.50
Quartzite	2.61
Schist	2.70
Upper & Lower Calc-Silicate	2.75
Fern Marble	2.78
Middle Marble	2.80
Hermes Marble	2.78
Stibnite Stock	2.61
Overburden	1.75

Total gold, cyanide soluble gold and silver were estimated using ordinary kriging with estimation domains based on the lithology groups and structural domains discussed above. Lithology groups served as hard boundaries for sample selection. The grade estimations for all metals in all domains, utilize a three-pass sample search strategy with each pass searching longer distances than the previous. The first estimation pass used an anisotropic search ellipse with a maximum range of 150 feet which was expanded to 250 and 300 feet in subsequent passes. Estimation was limited to those blocks within 225 feet of the closest composite. As discussed previously, the model is subdivided into four search domains. Domains 1, 3 and 4 all use static search orientations which are aligned parallel to the average strike and dip of the lithologic layering. Search domain 2 represents the West End Fault Zone where a dynamic search orientation was used based on the average strike and dip of the overlying, Hanging Wall Fault and the underlying, Foot Wall Fault. All estimations require a Min/Max of 3/16 samples respectively, utilize a minimum of two drill holes and a maximum of 2 samples per octant. A high-grade composite restriction was also applied in certain parts of the model to prevent excessive grade extrapolation into sparsely drilled areas of the deposit. Figure 14-12 shows plan and section views of the West End block model for gold.

Figure 14-12: West End Gold Block Model



14.5.9 Block Model Validation

The block model for West End was validated by completing a series of graphical inspections, bias checks, comparison to prior estimates and reconciliation against historical production records. Graphically, the interpolated block grades were visually checked on sections, plan views and in 3-D for comparison to the composite assay grades. The general model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data including the number of composites used, number of drillholes used, average distance to samples used, and the number of blocks estimated in each pass. Global and local bias was assessed through comparison of estimated block grades to the composite sample data and by construction of swath plots at 50 m spacing across the deposit. The final validation compared the grade estimate within the material which was historically mined to the accumulated production data from that mining period.

14.5.10 Geochemical Estimates

In addition to gold, cyanide gold and silver, a suite of estimates of geochemical element concentrations were prepared to support geo-metallurgical and geo-environmental engineering. Additional elements estimated included sulfur, arsenic, mercury, iron, calcium, sodium, magnesium and potassium. These elements were analyzed for Midas Gold drillholes but are only rarely analyzed in legacy holes, which comprise the majority of drillholes in the deposit.

- Sulfur and arsenic, for which data is limited, generally correlate with gold and were estimated within three domains using collocated co-kriging incorporating the gold block model as the secondary variable used to guide the estimate in areas with sparse Midas Gold drilling. This method reproduced the multivariate gold-arsenic-sulfur relationships observed in the composite data with total gold and oxide gold respectively.
- The major cations (Fe, Ca, Na, Mg and K) are primarily controlled by metasedimentary lithology and were estimated within domains based on lithology solids using inverse distance squared interpolation.
- Elevated mercury occurs along north-easterly striking splay structures and was estimated using inverse distance cubed interpolation within a single domain.

14.6 HISTORICAL TAILINGS

The Historical Tailings Mineral Resource estimate was not updated for the SGP Feasibility Study. The 2014 PFS details development of the Historical Tailings Mineral Resource Estimate.

14.7 MINERAL RESOURCE CLASSIFICATION

Mineral Resources are classified under the categories of Measured, Indicated and Inferred according to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) guidelines. Mineral resource classification for gold was based primarily on drillhole spacing and on continuity of mineralization. Antimony and silver are not classified separately and are reported based on gold classification. Measured resources were defined at Yellow Pine as blocks with an average distance to three drillholes of less than 50 feet and occurring within the Central Yellow Pine or Homestake estimation domains where historical production occurred. Indicated resources were defined as those with an average distance to three drillholes of less than 120 feet at Yellow Pine and 100 feet at Hangar Flats. Indicated resources at West End were defined as those with an average drillhole spacing of less than 100 feet and meeting additional requirements. Final resource classification shells were manually constructed on sections to smooth the classification categories. The drillhole spacing used to define indicated resources in Yellow Pine and Hangar Flats is generally consistent with classification strategy in the 2014 PFS and was independently validated by a drillhole spacing study assessing theoretical grade uncertainty under different drillhole patterns. This study indicates that a drillhole spacing of 120 feet reduces annual uncertainty to ±15-20% and that a drillhole spacing of 50 feet reduces quarterly uncertainty to ±15-20% with 90% confidence. See Figure 14-13 through Figure 14-15.

Figure 14-13: Mineral Resource Classification for Yellow Pine

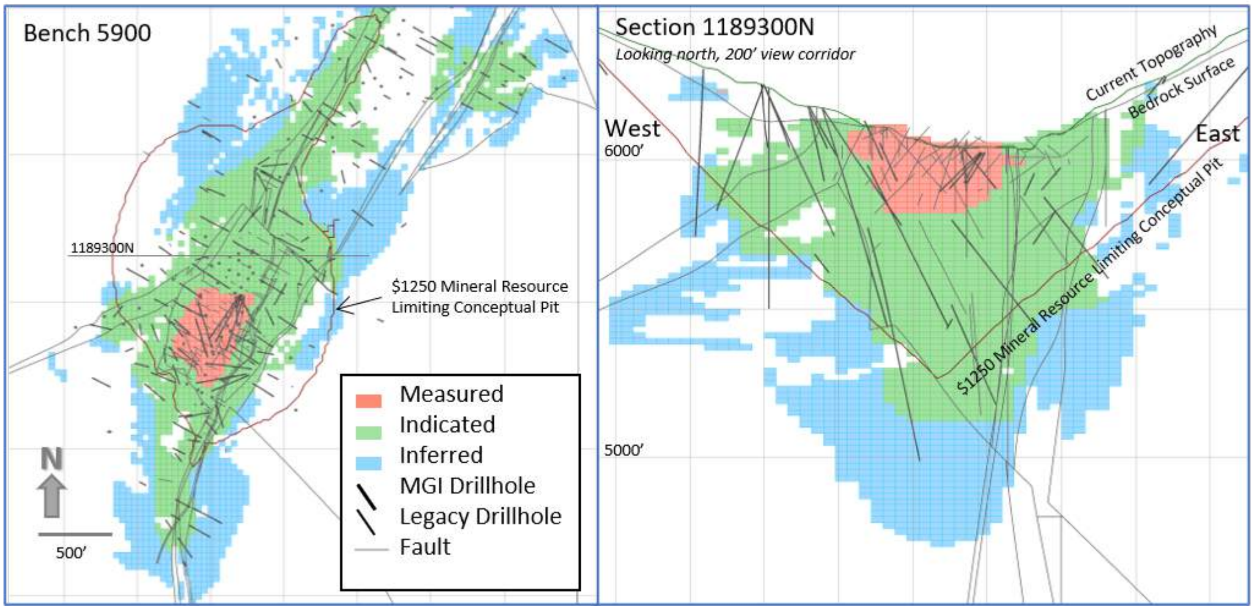


Figure 14-14: Mineral Resource Classification for Hangar Flats

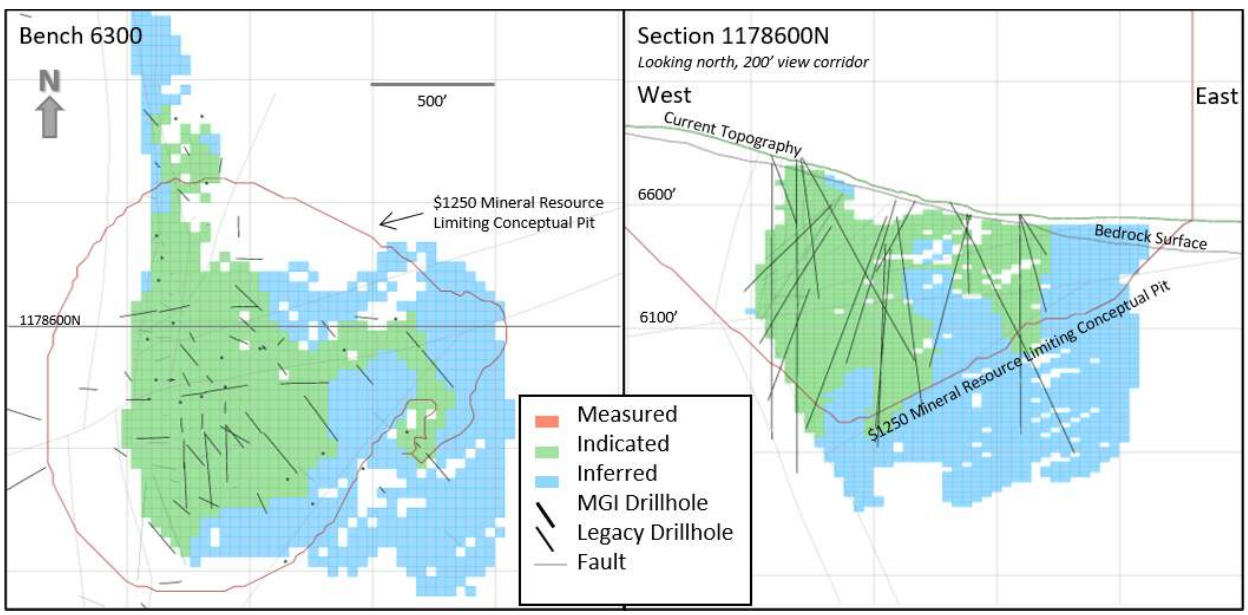
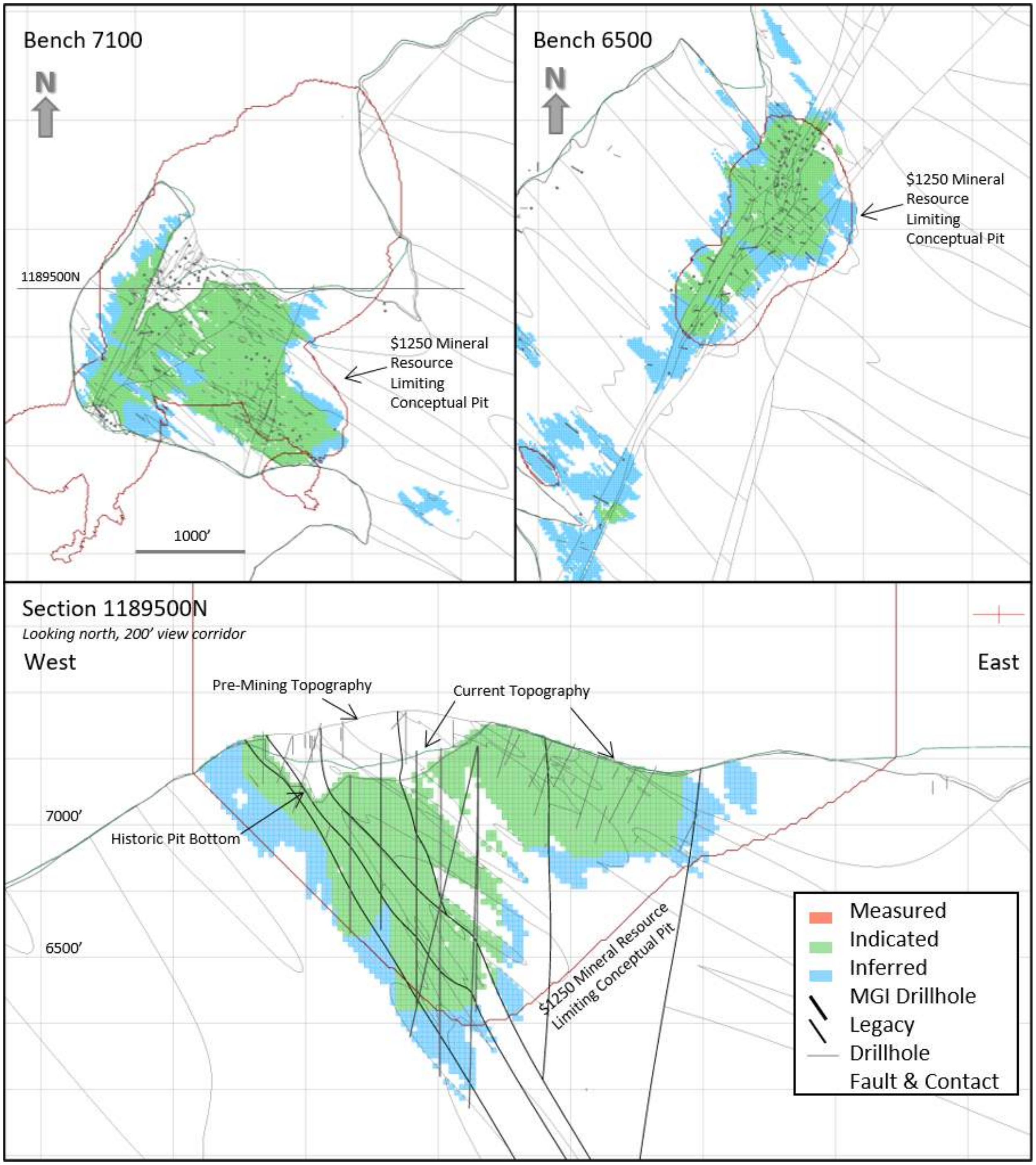


Figure 14-15: Mineral Resource Classification for West End



14.8 ECONOMIC CRITERIA AND PIT OPTIMIZATIONS

CIM Best Practices for Mineral Resources and Mineral Reserves requires that Mineral Resources have "reasonable prospects for eventual economic extraction" requiring that mineralization meet certain grade and material volume thresholds under reasonable production and recovery scenarios at reasonable cutoff grades. The potential for eventual economic extraction was assessed using an open-pit optimization Pseudoflow algorithm in MineSight® Version 15.10

software. Input parameters were developed on the basis of advanced cost estimates, metallurgical recoveries indicated by bench and pilot scale testwork and from feasibility level design engineering studies, as shown in Table 14-23. Relative to the 2014 PFS, sulfide processing costs have decreased, and pit slopes have been flattened, as discussed in Sections 21 and 15.

Table 14-23: Pit Optimization Parameters by Deposit

Economic Parameters	Units	Yellow Pine & Hangar Flats	West End
Mining Cost - Waste	$/tonne mined	2.00	2.00
Mining Cost - Ore	$/tonne mined	2.00	2.00
Ore Type Classification	-	-	Value Based
Oxide Processing Cost	$/tonne mined	-	7.20
Oxide Au Recovery	%	-	R*92.75%+1.22%
Transition Processing Cost	$/tonne mined	-	12.28
Transition Au Recovery	%	-	92.37%-R*8.93%
Sulfide Processing Cost	$/tonne milled	10.69	10.69
Sulfide Au Recovery	%	93%	96.42%-R*84.72%
Dore Transport Cost	$/oz Au	1.15	1.15
Dore Refining Cost	$/oz Au	1.00	1.00
G&A and Rehabilitation Cost	$/tonne milled	4.00	4.00
Pit Slopes	degrees	36-46	36-46
Au Payability	%	99.5	99.5
Au Selling Price - Base Case	$/oz	1250	1250
Mining dilution	%	0	0
Mining recovery	%	100	100
NSR Royalty on Au	%	1.7	1.7

Assumptions used to derive the cutoff grades and define the resource-limiting pits were estimated in order to meet the NI43-101 requirement for mineral resource estimates to demonstrate “reasonable prospects for eventual economic extraction” and vary from those used to limit the mineral reserves reported herein.

Because of the flat and shallow geometry of the Historical Tailings deposit, and due to potential use of the overlying material in conceptual construction scenarios, economic criteria were not assessed using a pit optimization. Instead, cost estimates for removing the overlying SODA material were compared to potential revenue from processing the tailings material and were shown to be positive.

14.9 MINERAL RESOURCE STATEMENTS

Mineral resources presented herein comply with guidelines of the Canadian Securities Administrators' National Instrument 43-101 and conform to CIM Definitions and Standards for Mineral Resources and Mineral Reserves (CIM, 2018). The mineral resources reported in Table 14-24 to Table 14-29, inclusively, are contained entirely within conceptual pit shells developed from the parameters discussed above. Based on these parameters, cutoff grades for Hangar Flats, West End and Yellow Pine were calculated based on a $1,250/oz gold selling price, which resulted in an open pit sulfide cutoff grade of approximately 0.45 g/t Au and an open pit oxide cutoff grade of approximately 0.40 g/t Au. Only mineral resources above these cutoffs and within the mineral resource-limiting pits are reported and, as such, mineralization falling below this cutoff grade or outside the mineral resource-limiting pit is not reported, irrespective of the grade. To demonstrate mineral

resource sensitivity to gold price and cut-off grade, mineralized tonnage and grade is reported in Table 14-30 within multiple conceptual pit shells optimized at different gold selling prices.

Table 14-24: Consolidated Mineral Resource Statement for the Stibnite Gold Project

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Measured							
Yellow Pine	4,902	2.42	382	3.75	590	0.24	25,831
Indicated							
Yellow Pine	45,350	1.72	2,509	2.07	3,020	0.09	85,774
Hangar Flats	25,861	1.44	1,194	3.24	2,697	0.15	84,463
West End	53,469	1.08	1,849	1.31	2,259	0.00	0
Historic Tailings	2,687	1.16	100	2.86	247	0.17	9,817
Total M & I	***132,269***	***1.42***	***6,034***	***2.07***	***8,814***	***0.07***	***205,885***
Inferred							
Yellow Pine	3,214	0.96	99	0.60	62	0.00	50
Hangar Flats	12,224	1.12	440	2.64	1,037	0.11	28,560
West End	20,540	1.06	700	1.11	733	0.00	0
Historic Tailings	191	1.13	7	2.64	16	0.16	662
Total Inferred	***36,168***	***1.07***	***1,246***	***1.59***	***1,849***	***0.04***	***29,272***

Notes:

(1) All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI43-101").

(2) Mineral Resources are reported in relation to a conceptual pit shell to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a Mineral Resource. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. These Mineral Resource estimates include Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is also no certainty that these inferred Mineral Resources will be converted to the Measured and Indicated categories through further drilling, or into Mineral Reserves, once economic considerations are applied. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.

(3) Open pit sulfide Mineral Resources are reported at a cutoff grade of 0.75 g/t Au and open pit oxide Mineral Resources are reported at a cutoff grade of 0.45 g/t Au.

The Yellow Pine and Hangar Flats deposits contain zones with substantially elevated antimony-silver mineralization, defined as containing greater than 0.1% antimony, relative to the overall mineral resource. The existing Historical Tailings Mineral Resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 14-25 to illustrate the potential for antimony production from the Project and are contained within the overall mineral resource estimates reported herein. Antimony zones are reported only if they lie within gold mineral resource estimates.

Table 14-25: Antimony Sub-Domains Consolidated Mineral Resource Statement

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Measured							
Yellow Pine	2,142	2.76	190	5.79	399	0.52	24,429
Indicated							
Yellow Pine	7,086	2.17	495	5.28	1,204	0.52	80,606
Hangar Flats	6,562	2.10	443	7.89	1,664	0.55	79,179

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Historic Tailings	2,687	1.16	100	2.86	247	0.17	9,817
Total M & I	***18,477***	***2.07***	***1,228***	***5.91***	***3,513***	***0.48***	***194,031***
Inferred							
Yellow Pine	10	1.21	0	2.78	1	0.18	41
Hangar Flats	1,185	2.40	92	15.27	582	1.07	27,829
Historic Tailings	191	1.13	7	2.64	16	0.16	662
Total Inferred	***1,387***	***2.22***	***99***	***13.43***	***599***	***0.93***	***28,532***

Notes:

(1) Antimony mineral resources are reported as a subset of the total mineral resource within the conceptual pit shells used to constrain the total mineral resource in order to demonstrate potential for economic viability, as required under NI43-101; mineralization outside of these pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate.

(2) Open pit antimony sulfide mineral resources are reported at a cutoff grade 0.1% antimony within the overall 0.45 g/t Au cutoff.

Table 14-26: Yellow Pine Mineral Resource Statement Open Pit Oxide + Sulfide

Classification	Tonnage	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Oxide (1,3)							
Measured	145	0.99	5	1.30	6	0.01	25
Indicated	1,241	0.99	39	1.05	42	0.00	108
Total M & I (2)	1,386	0.99	44	1.08	48	0.00	133
Inferred	15	0.79	0	0.80	0	0.00	0
Sulfide (1,3)							
Measured	4,758	2.47	377	3.82	584	0.25	25,806
Indicated	44,109	1.74	2,469	2.10	2,978	0.09	85,666
Total M & I (2)	48,866	1.81	2,847	2.27	3,563	0.10	111,472
Inferred	3,198	0.96	98	0.60	62	0.00	50

Notes:

(1) Mineral resources are reported in relation to a conceptual pit shell to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.

(2) Total Measured and Indicated Mineral Resources are inclusive of resources stated above.

(3) The mineral resources were tabulated based on the open pit optimization parameters discussed previously and a gold selling price of US$1,250/oz. These economic parameters equate to a cutoff grade of 0.45 g/t Au for open pit sulfide mineral resources and 0.40 g/t Au for open pit oxide mineral resources.

Table 14-27: Hangar Flats Mineral Resource Statement Open Oxide + Sulfide

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Oxide (1,2)							
Indicated	444	0.85	12	1.20	17	0.00	0
Inferred	128	0.68	3	1.08	4	0.00	0
Sulfide (1,2)							

Indicated	25,417	1.45	1,182	3.28	2,680	0.15	84,463
Inferred	12,096	1.12	437	2.66	1,033	0.11	28,560

Notes:

(1) Mineral resources are reported in relation to a conceptual pit shell to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.

(2) The mineral resources were tabulated based on the open pit optimization parameters discussed previously and a gold selling price of US$1,250/oz. These economic parameters equate to a cutoff grade of 0.45 g/t Au for open pit sulfide mineral resources and 0.40 g/t Au for open pit oxide mineral resources.

Table 14-28: West End Mineral Resource Statement Open Pit Oxide + Sulfide

Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
Oxide [1,2]							
Indicated	22,290	0.86	614	1.30	931	0.00	0
Inferred	6,317	0.84	171	1.10	223	0.00	0
Sulfide [1,2]							
Indicated	31,179	1.23	1,235	1.32	1,328	0.00	0
Inferred	14,223	1.16	529	1.12	510	0.00	0

Notes:

(1) Mineral resources are reported in relation to a conceptual pit shell to demonstrate potential for economic viability, as required under NI43-101; mineralization lying outside of these pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.

(2) The mineral resources were tabulated based on the open pit optimization parameters discussed previously and a gold selling price of US$1,250/oz. These economic parameters equate to a cutoff grade of 0.45 g/t Au for open pit sulfide mineral resources and 0.40 g/t Au for open pit oxide mineral resources.

Table 14-29: Historical Tailings Mineral Resource Statement Open Pit Sulfide

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Sulfide [1]							
Indicated	2,687	1.16	100	2.86	247	0.17	9,817
Inferred	191	1.13	7	2.64	16	0.16	662

Notes:

(1) Mineral resources are reported in total above cutoff since all the spent heap leach ore stacked on top of the tailings would be removed for construction purposes and the tailings fully exposed. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of Inferred mineral resources could be upgraded to Indicated. All figures are rounded to reflect the relative accuracy of the estimate.

14.10 GRADE SENSITIVITY ANALYSIS

The mineral resources and associated conceptual pit shell geometries are sensitive to the gold selling price used for reporting. To demonstrate sensitivity of the mineral resource to different gold prices and associated cut-off grades, multiple conceptual pit shells were developed across a range of gold selling prices and the mineralized material tonnage reported at cut-off grades appropriate for each selling price based on the economic parameters in Table 14-23. These results are shown in Table 14-30. It should be noted that this information does not constitute a Mineral Resource Statement and is presented only to demonstrate sensitivity of the deposits to cutoff grade selection.

Table 14-30: Combined Mineral Resource Sensitivity to Cutoff Grade

Gold Price ($US/oz)	Sulfide Cutoff Grade (g/t)	Oxide Cutoff Grade (g/t)	Classification	Tonnage (000s)	Gold Grade (g/t)	Contained Gold (000s oz)	Silver Grade (g/t)	Contained Silver (000s oz)	Antimony Grade (%)	Contained Antimony (000s lbs)
1,000	0.60	0.55	Measured	4,731	2.49	379	3.84	585	0.25	25,827
			Indicated	98,248	1.60	5,056	2.24	7,063	0.07	160,822
			Total M & I	102,979	1.64	5,435	2.31	7,648	0.08	186,649
			Total Inferred	17,526	1.30	730	2.08	1,173	0.06	23,694
1,250	0.45	0.4	Measured	4,902	2.42	382	3.75	590	0.24	25,831
			Indicated	127,367	1.38	5,652	2.01	8,223	0.06	180,054
			Total M & I	132,269	1.42	6,034	2.07	8,814	0.07	205,885
			Total Inferred	36,168	1.07	1,246	1.59	1,849	0.04	29,272
1,500	0.40	0.35	Measured	4,949	2.41	383	3.72	592	0.24	25,834
			Indicated	143,106	1.29	5,936	1.91	8,805	0.06	189,761
			Total M & I	148,055	1.33	6,319	1.97	9,397	0.07	215,595
			Total Inferred	52,077	0.96	1,610	1.40	2,342	0.03	32,771
1,750	0.35	0.3	Measured	5,000	2.39	383	3.69	593	0.23	25,834
			Indicated	160,402	1.21	6,219	1.83	9,431	0.06	199,671
			Total M & I	165,402	1.24	6,602	1.89	10,024	0.06	225,504
			Total Inferred	69,527	0.87	1,941	1.32	2,946	0.03	39,816

Resource sensitivity information presented above is reported from conceptual pit shells optimized using cost parameters discussed previously and gold selling prices denoted in the left-hand column. This information does not constitute a Mineral Resource Statement and is presented only to demonstrate sensitivity of deposits to cutoff grade and gold selling price.

14.11 DISCUSSION ON MATERIAL AFFECTS TO THE MINERAL RESOURCE ESTIMATE

To the extent known, the Qualified Person is not aware of any environmental, permitting, legal, title, taxation, marketing, political or other factors that would affect the resource estimates specifically.

14.12 CONCLUSIONS

It is the opinion of the Qualified Person that the Mineral Resource Estimates for the Yellow Pine, Hanger Flats, West End and Historical Tailings deposits were prepared using industry standards and best practices by qualified professionals and may be relied upon for public reporting and for estimating Mineral Reserves contained in this Report.

14.13 REFERENCES

CIM (2019). CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines, CIM Mineral Resources and Mineral Reserve Committee, November 29, 2019.

M3 Engineering & Technology (2014). Stibnite Gold Project Prefeasibility Study Technical Report, prepared for Midas Gold, December 8, 2014, amended March 28, 2019.

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15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

This section describes the Mineral Reserve estimation methodology, summarizes the key assumptions used, and presents the Mineral Reserve estimates for the Project.

Mineral Reserves are defined in the Canadian Institute of Mining and Metallurgy (**CIM**) Definition Standards for Mineral Resources & Reserves (May 19, 2014) as "*those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals*."

The Qualified Person (**QP**) for the estimation of the Mineral Reserve was Chris Roos, P.E. of Value Consulting, Inc. The Mineral Reserve estimates reported herein are a reasonable representation of the Mineral Reserves within the Project at the current level of analysis. Mr. Roos has reviewed the risks, opportunities, conclusions, and recommendations summarized in Sections 25 and 26, and he is not aware of any unique conditions that would put the Stibnite Gold Project Mineral Reserve at a higher level of risk than any other North American developing project.

The Mineral Reserves were estimated in conformity with CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" (Nov 19, 2019) and are reported in accordance with the Canadian Securities Administrators' NI 43-101. CIM states "*to be considered a Mineral Reserve, modifying factors must be applied to the Mineral Resource estimate . . . including mining, processing, metallurgical, environmental, location and infrastructure, market factors, legal, economic, social, and governmental. The Mineral Reserve estimates are based on a mine plan and pit design developed using modifying parameters including metal price, metal recovery based on performance of the processing plant, and operating cost estimates*."

15.1.1 Estimation Methodology

The SGP Mineral Reserves estimate equates to the mill feed schedule as presented in Section 16. The general mine planning sequence to produce the mill feed schedule consisted of an ultimate pit limit analysis, pit shell selection, ultimate pit designs, internal pit phase design, mining sequence schedule, and mill feed optimization. Section 15 includes a description of the reserve estimation process through ultimate pit design. Section 16 includes the remaining processes requisite to schedule mill feed and estimate Mineral Reserves. The mine planning process followed to estimate Mineral Reserves is summarized in Table 15-1.

Table 15-1: Mineral Reserve Estimation Process

Mineral Reserve Estimation Process	Process Inputs	Process Outputs	Section
Ultimate Pit Limit Analysis (UPLA)	Geologic resource block model Pit slope geotechnical limits Mining cost estimates Process cost estimates Metallurgical forecast algorithms Metal sell price estimate Metal sell costs (incl. royalties) Discount rate	Nested pit shells	15.2

Mineral Reserve Estimation Process	Process Inputs	Process Outputs	Section
Ultimate Pit Shell Selection	Nested pit shells	Guidance pit shells for ultimate pit design	15.3
Ultimate Pit Design	Guidance pit shells for ultimate pit design Pit design parameters (i.e. road width & grade, bench height & face angle)	Ultimate pit designs (defining extent of mined material included in Reserve Estimate)	15.4
Pit Shell-to-Design Reconciliation Analysis	Selected guidance pit shells Ultimate pit designs	Pit shell-to-design reconciliation	15.5
Dilution & Mining Losses	Geologic resource block model	Diluted resource block model	15.2.2
Cut-off Grade Analysis	Diluted resource block model	Cut-off grade methodology	15.6
Reserve Estimation	Diluted resource block model Cut-off grade methodology	Preliminary Reserve Estimate	15.7
Internal Pit Phase Analysis	Ultimate pit deigns Nested pit shells	Ultimate pit phase designs	16.2
Mine Sequence Analysis	Ultimate pit phase designs Production fleet equipment alternatives Mine production rates by fleet and activity Mill feed quantity and quality requirements	Fleet alternative analysis Strategic mine plan	16.3
Mine Development Plan	Strategic mine plan (incl. bench access schedule) Construction material requirements	Mine development and pre-stripping schedule Development fleet schedule	16.4
Stockpile Strategy Analysis	Strategic mine plan Process costs and metallurgical forecast algorithms Stockpile rehandle cost estimate Site layout incl. stockpile location options	Strategic stockpile schedule including capacity by ore type and grade class	16.5
DRSF Strategy Analysis	Strategic mine plan Strategic stockpile schedule	DRSF and stockpile design and schedule	16.6
Mill Feed Optimization	Strategic mine plan DRSF and stockpile schedule	Mill feed schedule Final Reserve Estimate	16.7
Mine Production Schedule Analysis	Strategic mine plan Mine development schedule Fleet alternative analysis	Mine production schedule Load & haul equipment schedule Drill & blast schedule Production support fleet schedule	16.8
Mine Consumables Estimate	Mine production schedule Drill & blast schedule Equipment consumables rates (i.e. fuel, tires, GET)	Mine consumables schedule	16.9
Maintenance Estimation	Mine production schedule Equipment rebuild and replacement schedule Preventive maintenance schedule Equipment parts life estimates	Equipment maintenance schedule Mine maintenance equipment schedule	16.10
Staffing Estimation	Mine production schedule Equipment maintenance schedule	Mine operations staff schedule Mine maintenance staff schedule Mine management staff schedule	16.11
Capital and Operating Cost Estimation	Equipment schedules Equipment cost vendor quotes Equipment maintenance schedule Mine consumables schedule Staffing schedules	Capital and operating cost schedule	16.12
Ultimate Pit Limit Analysis Validation	Capital and operating cost schedule	UPLA Validation	16.12.3

15.1.2 Mineral Reserves Summary

A summary of the Mineral Reserves for the Project is shown in Table 15-2. Detailed Mineral Reserves are presented in Section 15.7.

Table 15-2: Summary of Mineral Reserves

Deposit	Gold Cut-off (3)	Tonnage	Average Grade			Total Contained Metal		
			Gold	Antimony	Silver	Gold	Antimony(5)	Silver
Imperial Units	(oz/st)	(kst)	(oz/st)	(%)	(oz/st)	(koz)	(klbs)	(koz)
Yellow Pine – Proven		5,507	0.069	0.232	0.106	378	24,594	584
Yellow Pine – Probable		47,235	0.050	0.091	0.060	2,340	86,024	2,840
Yellow Pine – Proven & Probable	**0.013**	***52,742***	***0.052***	***0.106***	***0.065***	***2,718***	***111,617***	***3,423***
Hangar Flats – Probable (1)	0.014	9,107	0.046	0.150	0.083	414	27,252	756
West End – Probable (1)	0.014	50,519	0.031	-	0.040	1,587	-	2,004
Historical Tailings – Probable (1)	0.011 (4)	2,962	0.034	0.166	0.084	100	9,817	247
Proven & Probable Mineral Reserves (2)		**115,330**	**0.042**	**0.064**	**0.056**	**4,819**	**148,686**	**6,431**
Metric Units	(g/t)	(kt)	(g/t)	(%)	(g/t)	(t)	(t)	(t)
Yellow Pine – Proven		4,996	2.35	0.232	3.63	11.8	11,609	18.2
Yellow Pine – Probable		42,851	1.70	0.091	2.06	72.8	39,020	88.3
Yellow Pine – Proven & Probable	***0.46***	***47,847***	***1.77***	***0.106***	***2.23***	***84.5***	***50,629***	***106.5***
Hangar Flats – Probable (1)	0.49	8,262	1.56	0.105	2.85	12.9	12,361	23.5
West End – Probable (1)	0.49	45,830	1.08	-	1.36	49.3	-	62.3
Historical Tailings – Probable (1)	0.39 (4)	2,687	1.16	0.166	2.86	3.1	4,453	7.7
Proven & Probable Mineral Reserves (2)		**104,625**	**1.43**	**0.064**	**1.91**	**149.9**	**67,443**	**200.0**

Notes:
(1) *Deposit does not have a measured mineral resource. Reporting uses only an indicated mineral resource.*
(2) *Metal prices used for Mineral Reserves: $1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb.*
(3) Gold cut-off values are approximated due to application of the Net Smelter Return cut-off methodology as explained in Section 15.2.9.
(4) The Historic Tailings mineral resource was estimated using a composite of drill hole data to establish average mineral grades for the entire deposit. Therefore, the cut-off value provided is an approximate break-even cut-off grade.
(5) Antimony recovery is expected from the High Sb Sulfide ore only and contains 132,031 klbs (59,888 t) of Sb.

15.2 ULTIMATE PIT LIMIT ANALYSIS

Ultimate pit limit optimization and phase analysis (**UPLA**) was performed by the QP with Geovia Whittle™ version 4.7 using the Pseudoflow algorithm option. This section describes the optimization inputs. Pit limit optimization analysis results and pit shell selection is presented in Section 15.3.

The Pseudoflow algorithm performs the same function as the traditional Lerchs-Grossman (**LG**), however by structuring the UPLA as a maximum flow problem, the Pseudoflow algorithm can arrive at exactly the same solution in a fraction of the time. In either approach, Whittle™ applies approximate costs and recoveries along with approximate open pit slope criteria to establish theoretical economic breakeven pit geometries (pit shells). The resulting pit geometries should be considered as approximate as they do not assure pit bench access or bench working space requirements. The primary result of the incremental pit geometries (nested pit shells) is the relative change in pit size and estimated increase in total pit value. This provides guidance for designing detailed ultimate pit designs and identifying potential mining phases to bring forward value in the mining sequence.

15.2.1 Geologic Resource Block Model

Garth Kirkham is the QP responsible for the mineral resource block models used in this mineral reserve estimate. The models comprise parameters that describe lithology, in-situ density, resource classification, ore and waste percentage, oxidation, and metal grades, as explained in detail in Section 14.

For mine planning purposes, the block model dimensions for individual blocks should correspond to an increment of proposed mining bench height. Bench height has a potentially significant impact on project value due to the relationship between bench height, grade dilution, mine operating cost, mine production rates, and processing cost. A bench height trade-off analysis was conducted to evaluate bench heights ranging from 10 to 50 feet in 10-foot increments. Based on the analysis, a bench height of 20 feet for ore zones and 40 feet for waste zones was selected as the most economical way to mine the deposits. These bench heights will allow optimizing productivity in waste zones while maintaining ore selectivity in ore zones to reduce potential grade dilution.

Based on the bench height trade-off analysis, a block model with uniform block dimensions of 40 x 40 x 20 feet representing the selective mining unit (**SMU**) was created for each deposit using the resource block model as detailed in Section 14. Only blocks classified as measured or indicated were used in the mineral reserve estimate. Blocks classified as inferred were reclassified as waste with zero payable metal content. The modified mineral resource block model is hereafter referred to as the reserve block model.

15.2.2 Ore Dilution and Mining Losses

CIM defines dilution as "*material that is below the cut-off grade or value but is intentionally or inadvertently mined and must be considered in Mineral Reserve estimates because it dilutes the average grade estimate and increases the volume mined*". Dilution can be classified as either internal or external. Internal dilution occurs within a mining block in which pockets of material below cut-off grade cannot be removed selectively during the digging operation. External dilution typically occurs because of blasting which causes material movement and mixing of ore and waste along mining block boundaries.

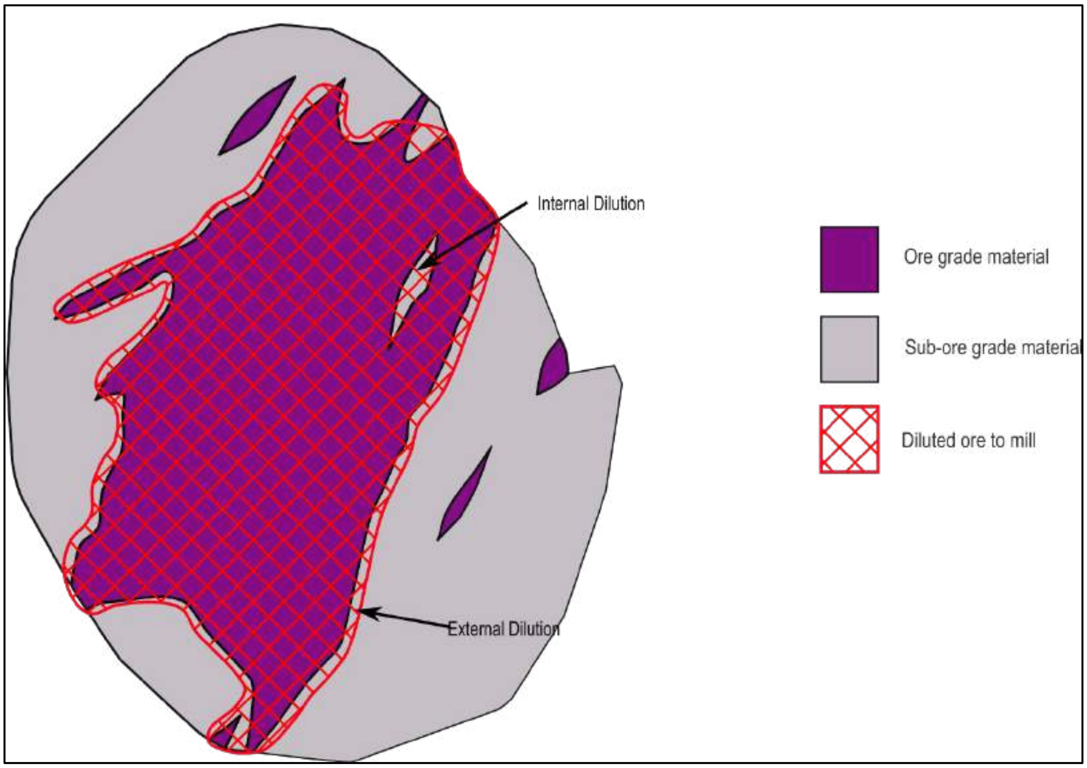
Internal dilution was estimated in the reserve block model by averaging metal content within each 40 x 40 x 20-foot block provided in the resource block model. Both the Yellow Pine and Hangar Flats resource block models were modeled using an ore percent approach to estimate the amount of waste within a single block with dimensions 40 x 40 x 20 feet. The West End resource block model was estimated on a whole block basis using 20 x 20 x 20-foot blocks to account for narrow geological controls as discussed in Section 14.5. Internal dilution at West End was estimated by consolidating the blocks, i.e., re-blocking the model into 40 x 40 x 20-foot blocks.

Additionally, ore type designation dilution was estimated by applying an algorithm to identify blocks with an ore-type classification that did not match at least 30% of the adjacent 8 horizontal blocks. These blocks were reclassified to match the predominant adjacent ore classification. This resulted in some blocks being reclassified from ore to waste, waste to ore, and one ore-type classification to another, e.g., oxide ore reclassified as low antimony ore.

An external dilution study was conducted to estimate dilution occurring along ore block boundaries between adjacent blocks. A 10% dilution boundary for each block was applied to estimate external dilution resulting from blasting. This equates to an 8-foot mixing zone which is approximately half the distance between blasthole spacing. This degree of dilution would result in an approximately 3% increase in ore mined with a loss of approximately 2% gold mass for an effective grade dilution of 5%. To account for this, a mining dilution factor of 5% was input to the Whittle™ pit limit analysis. Figure 15-1 illustrates internal and external dilution estimation.

CIM defines mining losses as "*the percentage of ore grade material within the mine designs that will not be extracted for various reasons*". Mining losses are typically more significant in underground operations where material may need to be left in place for safety and geotechnical considerations. Mining losses are not expected for the SGP due to the geologic characteristics and pit designs with ramps primarily in waste.

Figure 15-1: Internal and External Dilution



15.2.3 Overall Pit Slope Angles

Overall pit slope angles and sectors were provided by the Project geotechnical consultant STRATA, A Professional Services Corporation (**STRATA**) for all three open pits as shown on Figure 15-2, Figure 15-3, and Figure 15-4. Slope sectors were coded into the block model prior to importing into Whittle™.

Figure 15-2: Yellow Pine Overall Pit Slope Angles

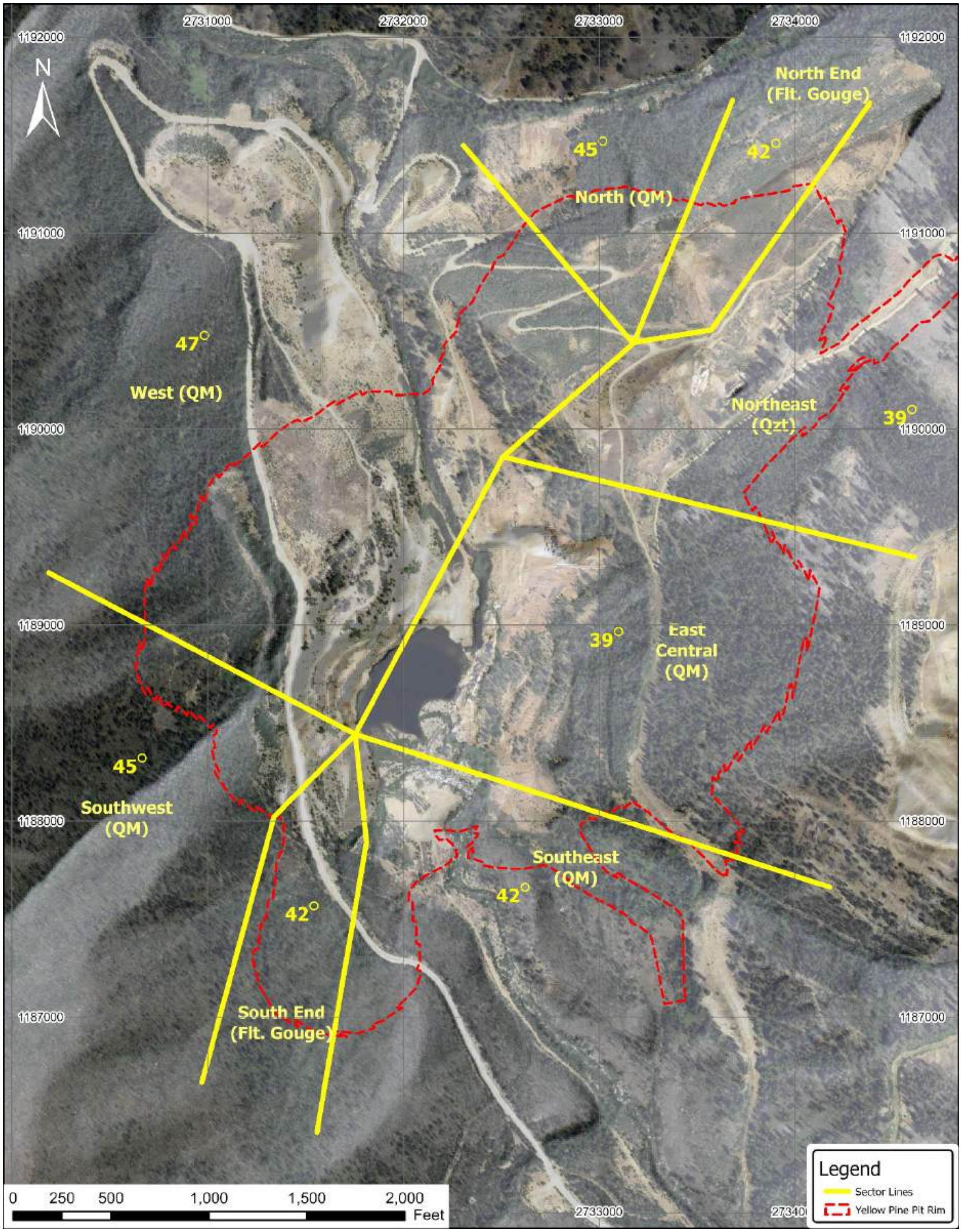


Figure 15-3: Hangar Flats Overall Pit Slope Angles

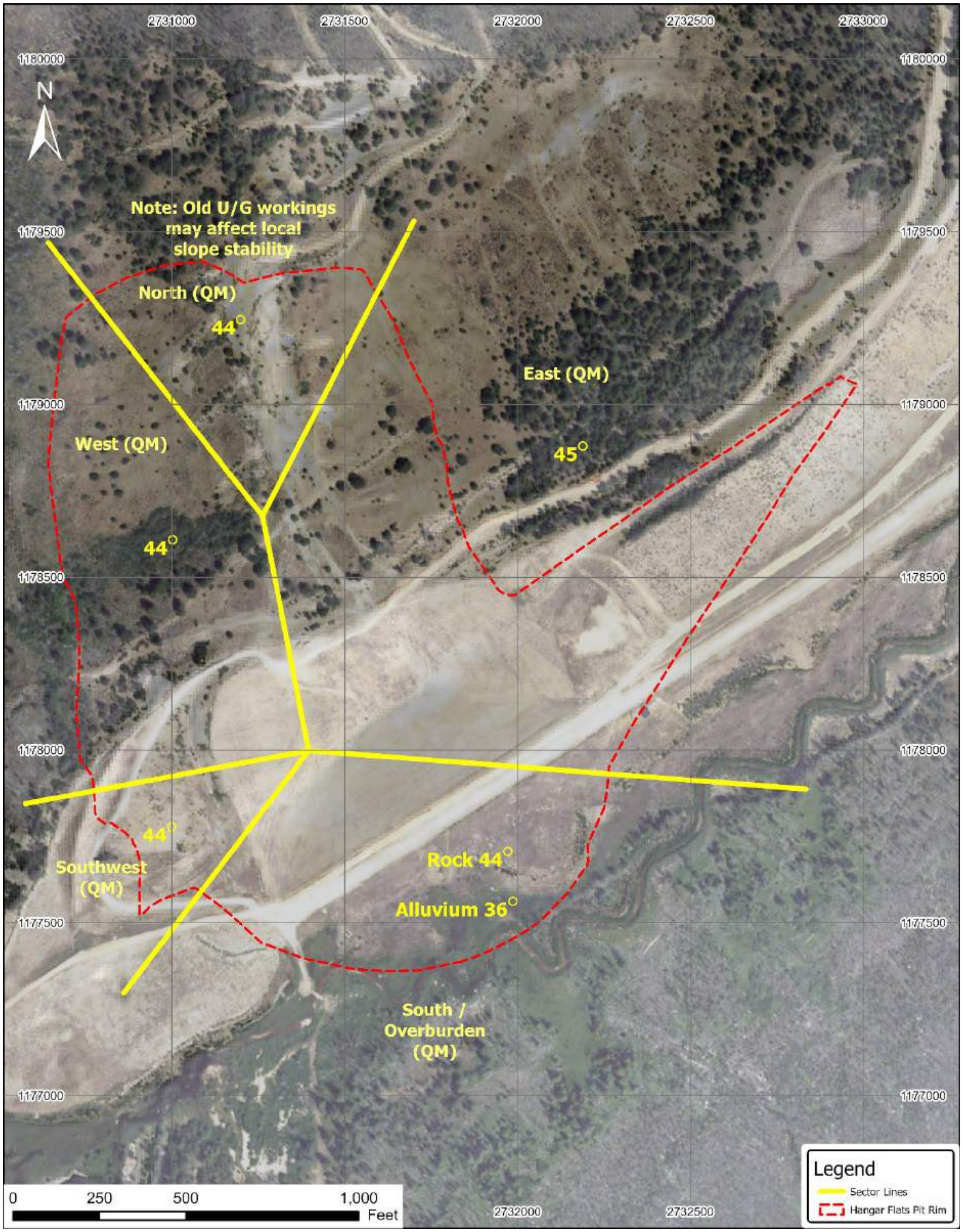
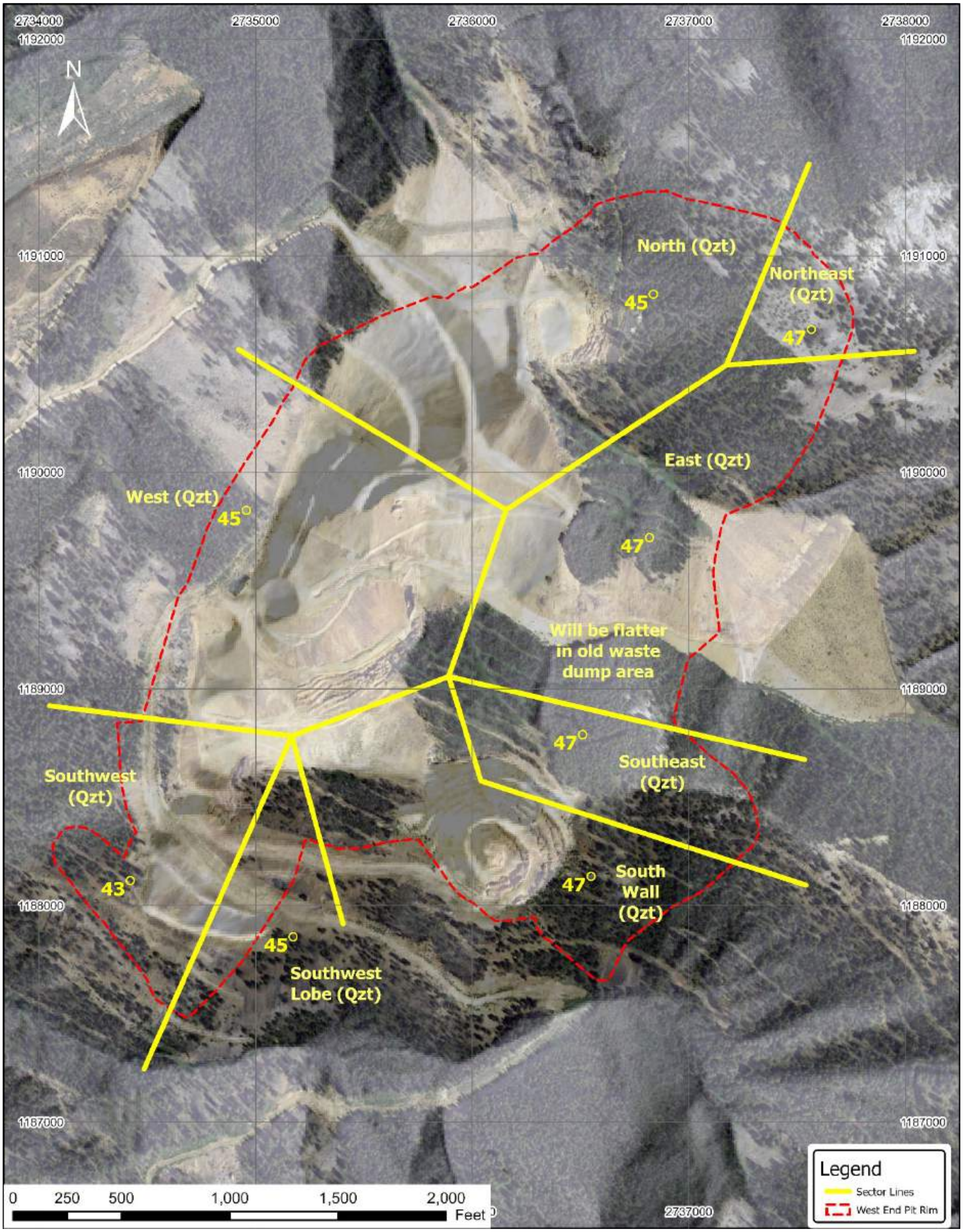


Figure 15-4: West End Overall Pit Slope Angles



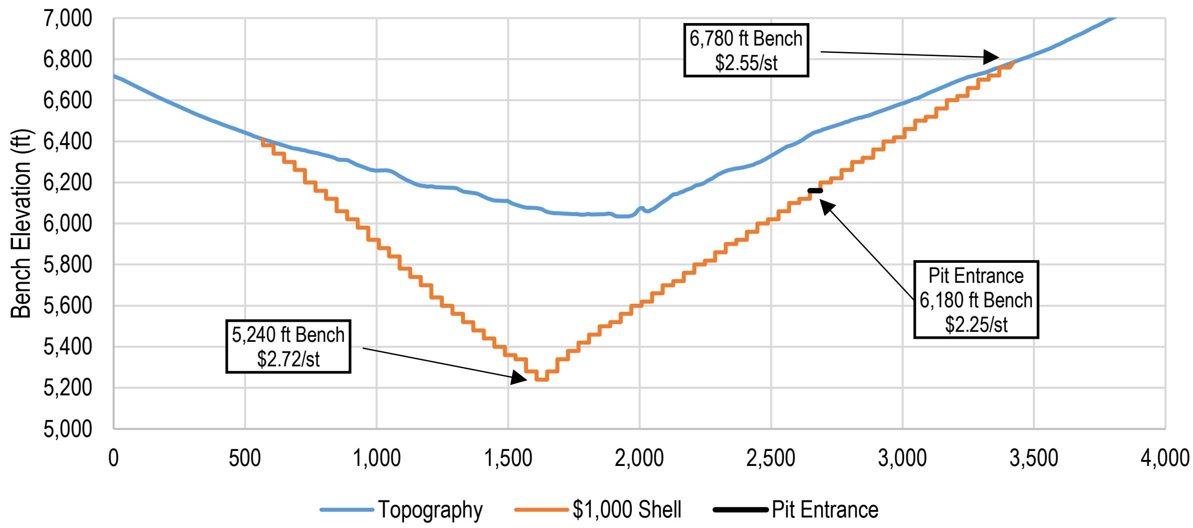
15.2.4 Mining Method and Mining Costs

Conventional owner-operated truck and shovel open pit mining methods were selected as the most viable mining method for the deposits at this time. Mining costs used for the pit limit analysis are based on the calculations presented in the Prefeasibility Study and first principle cost buildup based on equipment requirements, labor estimates, and updated consumables price quotes. The mining costs comprise pit and dump operations, delivery of the ore to the crusher or stockpiles and waste to the DRSFs, road maintenance, mine supervision, and mining-related technical services. The mining cost estimate increased as compared to the PFS primarily due to updated equipment operating cost estimates, labor estimates, and additional mine development costs added to account for in-pit production access in steep terrain. Note that, while one mining cost is presented, the QP evaluated a range of mining costs to test the sensitivity of the UPLA to mining cost parameters and concluded that the selected ultimate pit limits were not highly sensitive to costs within the expected accuracy (+/- 15%).

A reference mining cost of $2.25/st plus an incremental cost of $0.01 per 20-foot bench both below and above the pit rim was applied to each pit individually. This incremental cost was added to benches below the pit rim to account for additional haulage cost when hauling from the pit loaded. Due to the site topography, the incremental bench cost was added for benches above the pit rim to account for access road development to upper pit benches and decreased mining efficiency on smaller benches within the pit upper reaches. Validation of cost assumptions applied to the UPLA are presented in Section 16.12.3.

As an example, Yellow Pine mining cost per ton for the $1,000 shell ranges from $2.72 at the bottom bench (elevation 5,240 feet) to $2.25 at the pit rim entrance (elevation 6,180 feet) to $2.55 at the highest bench (elevation 6,780 feet) as shown on Figure 15-5.

Figure 15-5: Yellow Pine Mining Cost by Bench Elevation



15.2.5 Metallurgical Recoveries Forecast Algorithms

Metallurgical recovery functions and costs were applied to gold, silver, and antimony as presented in Section 13. The pit limit analysis was performed on gold recovery only, to ensure the ultimate pit geometries would not be dependent on silver or antimony value. Silver and antimony recoveries were incorporated into the mine schedule once the ultimate pit designs were completed as discussed in Section 15.4.

15.2.6 Process Costs, Selling Costs, Payability, and Royalties

Each unit of mined material from the three pits and historical tailings was classified into one of six ore type designations as shown in Table 15-3. The designation corresponds to the highest Net Smelter Return (**NSR**) value as further discussed in Section 15.2.9. Process and selling costs applied in the UPLA are shown in Table 15-4. The QP also performed a sensitivity analysis of the UPLA relative to process costs and concluded once again that the selection of an ultimate shell is not highly sensitive to cost.

Table 15-3: Ore Type Designation

Ore Type	Description	Deposit Occurrence
Low Sb Sulfide Ore	Only a gold-bearing sulfide concentrate would be produced and processed onsite through POX and cyanide leaching.	All Deposits
High Sb Sulfide Ore	An antimony concentrate would be produced followed by a gold-bearing sulfide concentrate. The sulfide concentrate would be processed onsite through pressure oxidation (**POX**) and cyanide leaching.	Yellow Pine, Hangar Flats
Oxide Ore	Gold would be recovered through whole ore cyanide leaching.	West End
Low Sb Transitional Ore	A gold-bearing sulfide concentrate would be produced and processed onsite through POX and cyanide leaching. Additional gold would be recovered through cyanide leaching of the tailings.	West End
Historical Tailings Sulfide Ore	Processed concurrent with both High Sb Sulfide Ore and Low Sb Sulfide Ore sourced from the open pits	Historical Tailings
Waste	Material not meeting the NSR cut-off value.	All Deposits

Table 15-4: Ore Process Costs, Selling Costs, Payabilities, and Royalties

Cost Item	Unit	Cost
Ore Processing		
Oxide	$/st ore	9.58
Low Sb Sulfide	$/st ore	12.17
High Sb Sulfide	$/st ore	13.96
Low Sb Transitional	$/st ore	13.04
Historical Tailings Sulfide	$/st ore	8.91
G&A	$/st ore	3.47
Reclaim Cost	$/st ore	0.57
Payability*		
Au in Sb Concentrate	%	20.0
Sb in Sb Concentrate	%	68.0
Ag in Sb Concentrate	%	45.0
Au in Doré	%	99.0
Ag in Doré	%	95.0
Transportation, Refinement & Royalty		
Sb Concentrate	$/wet st	175
Au in Doré	$/paid oz	8.00
Ag in Doré	$/paid oz	0.50
Royalties	% net of smelter Au	1.7

15.2.7 Metal Selling Prices

A suite of nested pit shells for each deposit was generated using revenue factors that reflected a gold selling price ranging from \$100 to \$2,000 per troy ounce in \$50 increments. The nested pit shells generated using the Pseudoflow algorithm in Geovia Whittle™ represent the optimal pit shell geometry based on undiscounted cash flow. Each nested pit shell is then evaluated using the estimated metal sell price expected during operations. The gold price used in the nested shell evaluation was \$1,600 per troy ounce. Antimony and silver value were not included in the pit limit analysis to prevent their value from influencing the pit design and provides additional conservatism that de-risks the dependence of the project on revenues from those metals.

Sensitivity analyses were performed on both the Yellow Pine pit and Hangar Flats pit to assess the potential impact silver and antimony could have on pit geometry. The pit shell size increase resulting from either addition of silver, antimony, or the combined value was insignificant as compared to pit shells calculated using only gold value.

15.2.8 Discount Rate

CIM states "*As a minimum, the NPV must be positive using a reasonable discount rate appropriate for all project risks, in order for the grade and tonnage to qualify as a Mineral Reserve*". For the ultimate pit limit analysis, an annual discount rate of 10% was applied using a high-level scheduling algorithm in the Whittle™ "Pit by Pit Graph" and choosing the ultimate pit limit based on an incremental analysis of the discounted NPV generated by that schedule.

15.2.9 Block Value Calculation

A Net Smelter Return (**NSR**) cut-off methodology was adopted to calculate block value and ore type due to the polymetallic nature of the ore deposits and separate process streams with unique process costs. CIM states "*For the NSR method, the dollar value that each metal contributes towards the total value is calculated and is expressed as one value referred to as the NSR value. The calculation of an NSR value considers revenues, metallurgical recoveries, smelter deductions, treatment charges, penalties, and transportation costs for all metals of potential economic interest. This NSR value can then be used to derive a cut-off value, where the NSR cut-off value is then the dollar value of a given sample or block that equals the total operating costs, as appropriate*."

The Net of Process Revenue (**NPR**), defined as NSR less process plant operating expenditures (**OPEX**) and general and administrative costs (**G&A**), was calculated on a block-by-block basis in dollars per ton of ore to estimate the value of a block for each available process stream. Mining costs are not included in the calculation of NPR because it will be approximately the same for an ore block regardless of process stream designation. The potential process stream designations used to define each block ore type are explained in Table 15-3.

For the pit limit analysis, antimony and silver are assumed to have no value therefore the high antimony sulfide ore process stream is effectively unavailable due to the process cost associated with producing an Sb concentrate with no Sb value. In effect, the pit limit analysis evaluated the project based on on-site gold processing only. Once the pit is designed, silver and antimony NPR are calculated on a block-by-block basis and included in the reserve estimate. An example of NPR calculation and block ore type classification determination is shown in Table 15-5.

Table 15-5: Sample Block Value Calculation

Resource Block Model – Sample Block Values				
Block Mass	2,617 st			
	Grade	Contained Metal	Transport Cost	Sell Price
Au	2.35 gpt	179 oz	$8.00 /oz	$1,600 /oz
Sb	0.20%	10,416 lb	$175 /st conc.	$3.50 /lb
Ag	2.45 gpt	187 oz	$0.50 /oz	$20 /oz

	High Sb Sulfide Ore Doré Revenue			Low Sb Sulfide Ore Doré Revenue		
	Au	Sb	Ag	Au	Sb	Ag
Doré Recovery	88.36%	0.0%	0.0%	90.70%	0.0%	0.6%
Doré Recovered Metal	158.3 oz	0 lb	0.0 oz	162.5 oz	0 lb	1.1 oz
Doré Payability	99.0%	0.0%	0.0%	99.0%	0.0%	95.0 %
Doré Payable Metal	156.7 oz	0 lb	0.0 oz	160.9 oz	0 lb	1.1 oz
Doré Metal Value	$250,729	$0	$0	$257,373	$0	$21
	$250,729			$257,395		

	High Sb Sulfide Ore Sb Concentrate Revenue			Low Sb Sulfide Ore Sb Concentrate Revenue		
	Au	Sb	Ag	Au	Sb	Ag
Sb Con Recovery	1.57%	85.4%	16.8%			
Sb Con Contained Metal	2.8 oz	8,891 lb	31.4 oz			
Sb Con Metal Payability	20.0%	68.0%	45.0%	N/A		
Sb Con Payable Metal	0.6 oz	6,046 lb	14.1 oz			
Sb Con Metal Value	$898	$21,162	$282			
Total Sb Con Metal Value	$22,342			0$		

	High Sb Sulfide Ore Net Smelter Return (NSR)			Low Sb Sulfide Ore Net Smelter Return (NSR)		
	Au	Sb	Ag	Au	Sb	Ag
Net Smelter Payable Metal	157.3 oz	6,046 lb	14.1 oz	160.9 oz	0 lb	1.1 oz
Net Smelter Metal Sell Value	$251,628	$21,162	$282	$257,373	$0	$21
Total Net Smelter Value	$273,071			$256,107		
Sb Con Mass		6.84 st			n/a	
Transport & Refinement Cost	$1,254	$1,197	%0	$1,287	n/a	$1
Net Smelter Return	$270,621			$256,107		

	High Sb Sulfide Ore Net of Process Revenue (NPR)	Low Sb Sulfide Ore Net of Process Revenue (NPR)
	Total	Total
Ore Processing Unit Cost	$13.96 /st	$12.17 /st
Ore Processing Cost	$36,533	$31,849
G&A Cost	$9,081	$9,081
Royalties (1.7% Au NSR)	$4,278	$4,375
Net of Process Revenue	$220,729	$210,802
Net of Process Unit Rev	$84.34 /st	$80.55 /st

Block Ore Designation: High Sb Sulfide since the unit NPR is greater than Low Sb Sulfide

15.3 ULTIMATE PIT LIMIT SHELL SELECTION

Regarding ultimate pit limit shell selection, CIM states "*To select the ultimate pit limit, an analysis of incremental pit shells can be carried out to evaluate the contribution of each consecutive pit shell to NPV at a constant processing plant capacity. Optimization results from each of the shells are analyzed independently to select a final pit shell to use for preparation of the final pit design, along with any starter or phase pit selections. The objective of the final pit shell is often to maximize grade and project NPV. To determine the optimum pit shell, cash flow analyses are performed considering the sequence of mining for all the nested pit shells.*"

The cash flow analyses for nested pit shells were performed by the QP using Whittle™ software. The analyses produce two discounted values for each nested shell often referred to as "Best Case" and "Worst Case". The "Worst Case" values are calculated for each pit shell as if the shell is mined in its entirety bench-by-bench without internal phasing. This delays access to higher-grade ore and reduces NPV as compared to a phased mining approach. The "Best Case" values are calculated sequentially from the smallest to largest pit shell, where each shell represents an internal pit phase. Each pit shell increment is scheduled as if the prior shell has already been mined and processed allowing for the pit to advance downward quickly and access higher-value ore and increased NPV. The actual mining sequence is likely to be in-between these two scenarios, including internal phases while maintaining large enough benches for consistent mine productivity.

Discussion of nested pit shells in this section is limited to selecting shells for ultimate pit designs. There is further discussion in Section 16 regarding internal pit phase design as it relates to nested pit shells.

Nested pit shell cash flow analysis for all three pits was performed on a suite of shells ranging in gold sell price between \$100/oz to \$2,000/oz in increments of \$50/oz.

15.3.1 Yellow Pine Pit Shell Selection

The Yellow Pine maximum discounted value shells for the "worst case" and "best case" are \$950 and \$1,550; respectively (see Figure 15-6). The incremental change in discounted pit value (NPV) and strip ratio between these two shells is gradual which implies the value of Yellow Pine is not highly sensitive to the selection of a specific shell. Whittle™ allows for a third, Specified Case, however, due to the nature of the deposit, the "nested" shells did not accurately represent the likely mining sequence so "directional" shells were ultimately chosen to guide internal phases as discussed in Section 16.2.1.

To properly analyze and select an ultimate pit within the range specified above, the QP opted to perform an incremental analysis of each subsequent pit to determine the point where the additional mining no longer adds significant value (see Figure 15-7). This analysis utilizes an "incremental return" which is approximated as the incremental change in discounted value divided by the incremental change in discounted total costs. The resulting incremental return can be compared to the project minimum acceptable rate of return (MARR, 10%) to determine when incremental additions no longer generate significant value. As the actual value is recognized to be between the best- and worst-case scenarios, the QP chose to use a weighted average return to reflect the likely results of a realistic schedule. Due to the topography at the site, the worst-case is highly unlikely as it begins mining at the top of the mountain, neglecting the accessible ore in the bottom of the valley. With this in mind, the average was weighted at 75% of the best-case and only 25% of the worst-case and \$1,250 was chosen as its average return (12.5%) was the last incremental return above the MARR.

Figure 15-6: Yellow Pine Nested Pit Shell Discounted Value

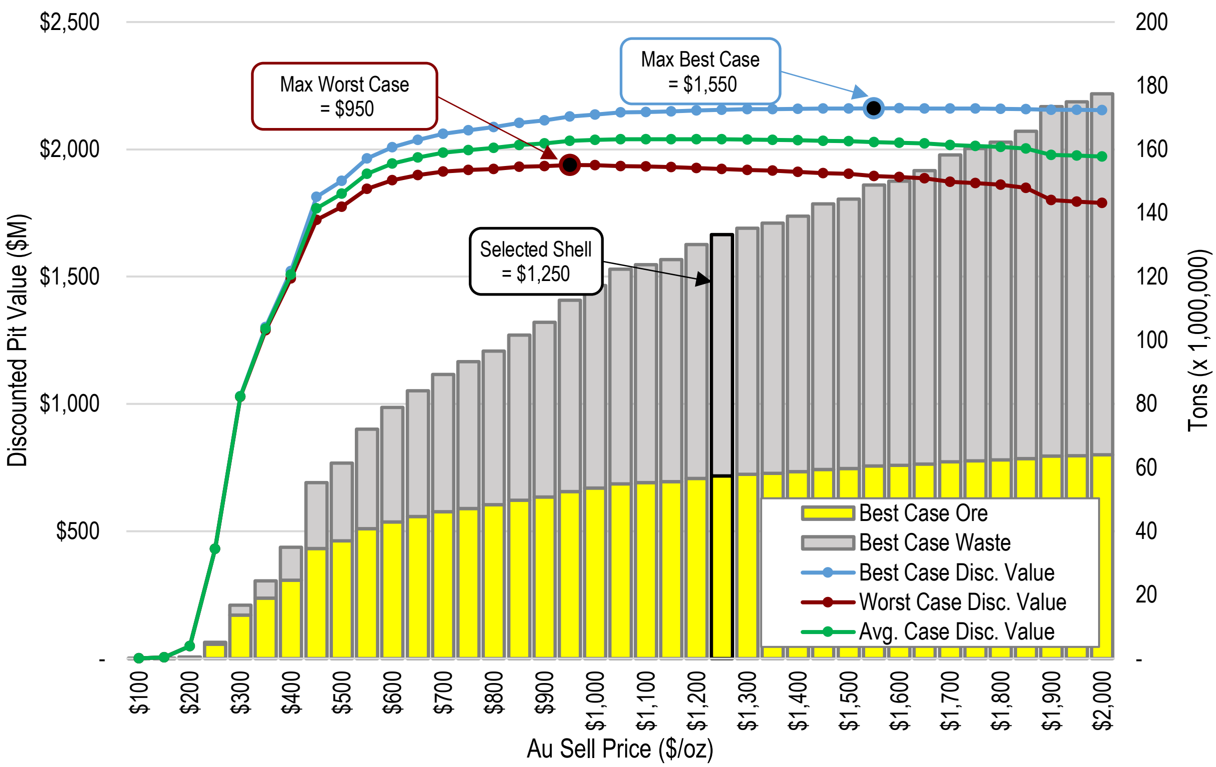
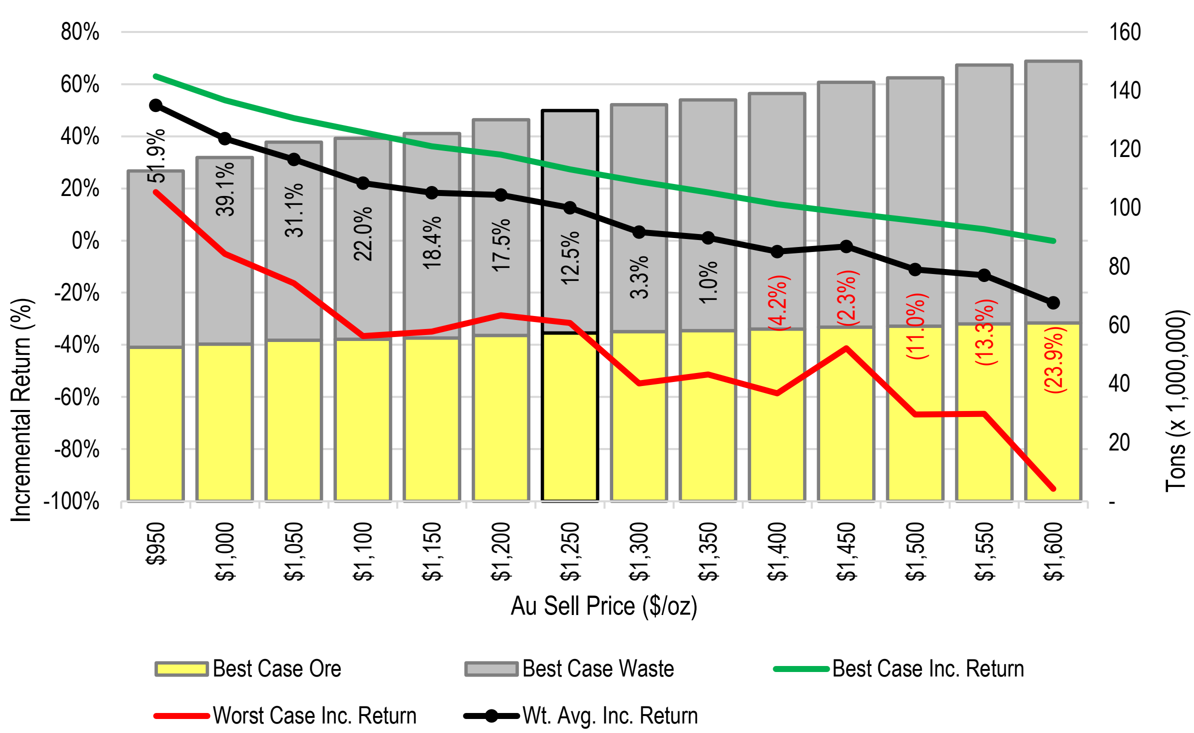


Figure 15-7: Yellow Pine Nested Pit Shell Incremental Return



15.3.2 Hangar Flats Pit Shell Selection

For the Hangar Flats deposit, a similar analysis to Yellow Pine resulted in an ultimate pit between $1,100 and $1,600 with an incremental analysis suggesting $1,150 be chosen as the ultimate pit (see Figure 15-8). However, upon review, this "large" Hanger Flats pit presented a number of technical challenges, risks, and costs associated with mining through the extensive historical underground workings and development of a haul road from the Fiddle Creek basin to access its upper benches. Based upon a mine sequence analysis, the project team selected a much smaller footprint for the initial Hanger Flats pit ($750 shell). As this shell (see Figure 15-9) may be an internal phase of a larger Hanger Flats pit it allows for additional study of the true costs associated with a potential layback and a better understanding of the operational requirements of mining through the historical workings.

Figure 15-8: Hangar Flats Nested Pit Shell Discounted Value

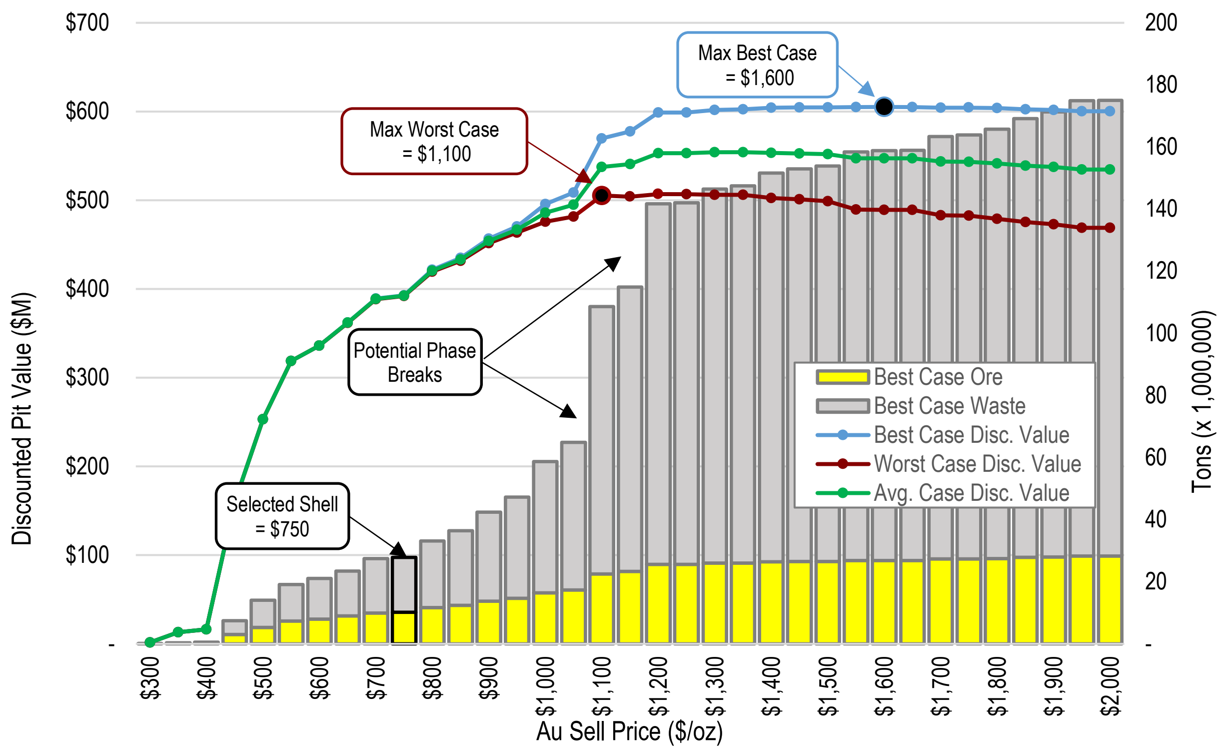
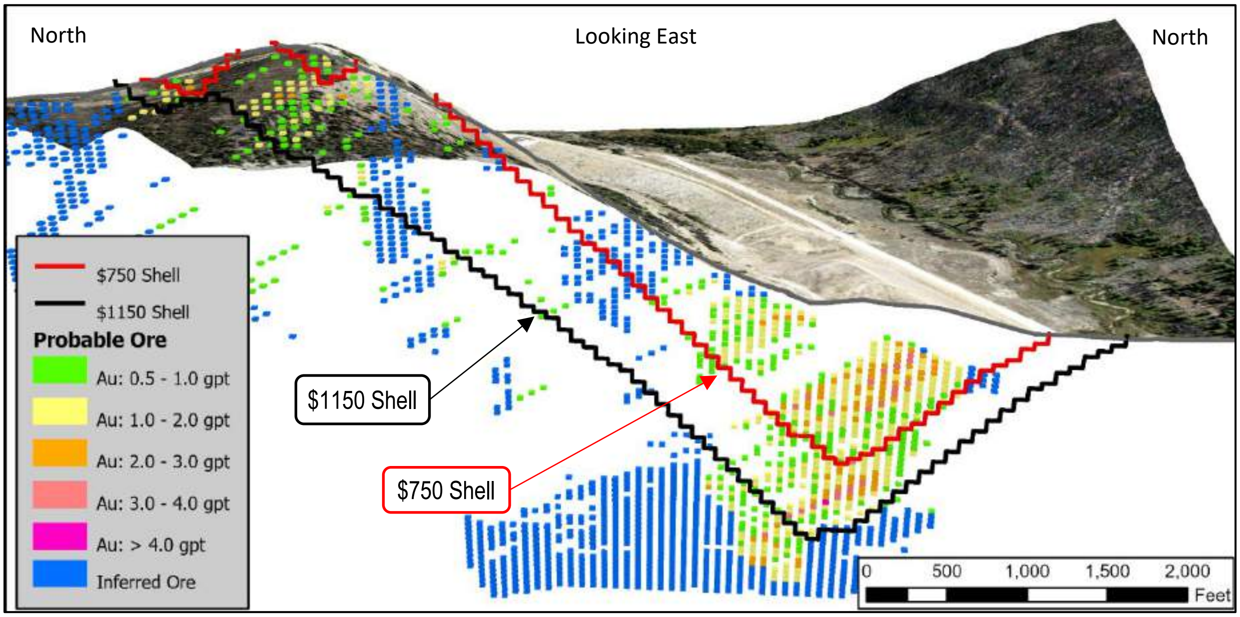


Figure 15-9: Hangar Flats Nested Pit Shell Cross-Section



15.3.3 West End Pit Shell Selection

Similar to Yellow Pine, the incremental pit shell changes in discounted value and strip ratio are relatively gradual without any substantial incremental change between the maximum values for worst-case and best-case, as shown in Figure 15-10.

Reviewing the incremental return, as discussed in the Yellow Pine Pit Shell Selection, results in an ultimate pit selection of $1,300 for West End that has an incremental return of 10.9%.

Figure 15-10: West End Nested Pit Shell Discounted Value

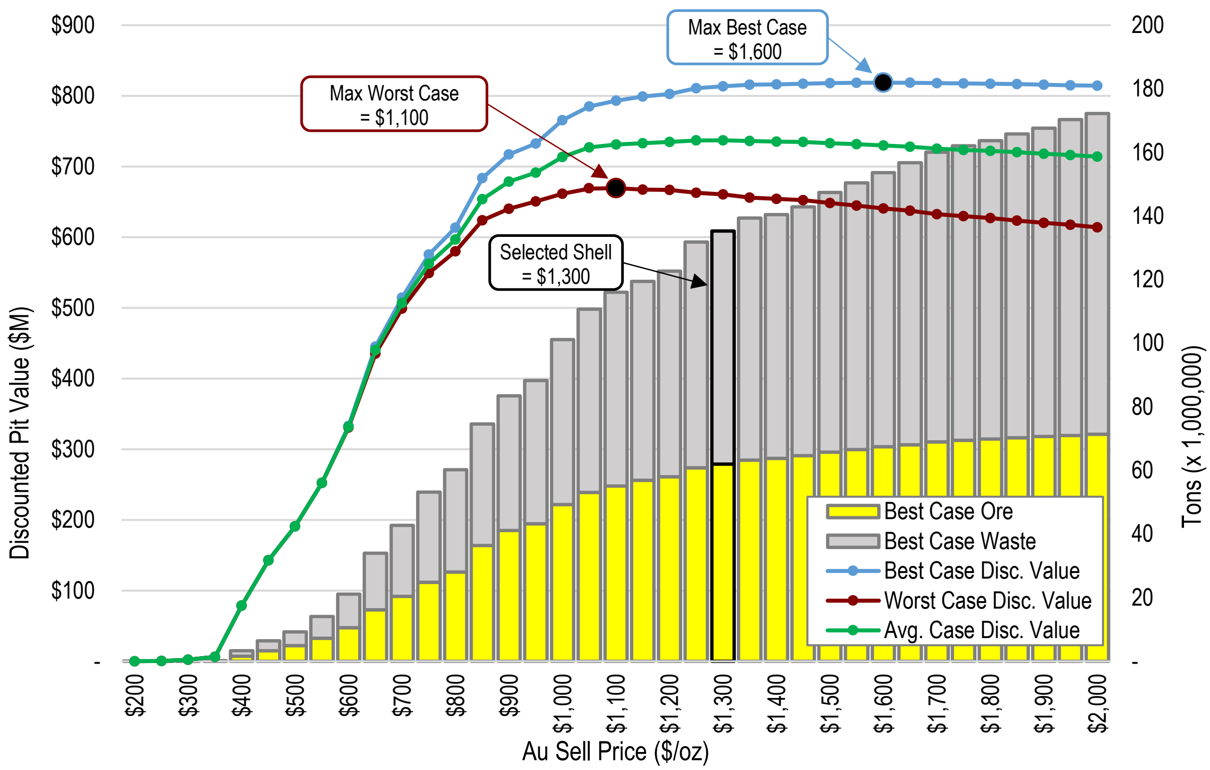
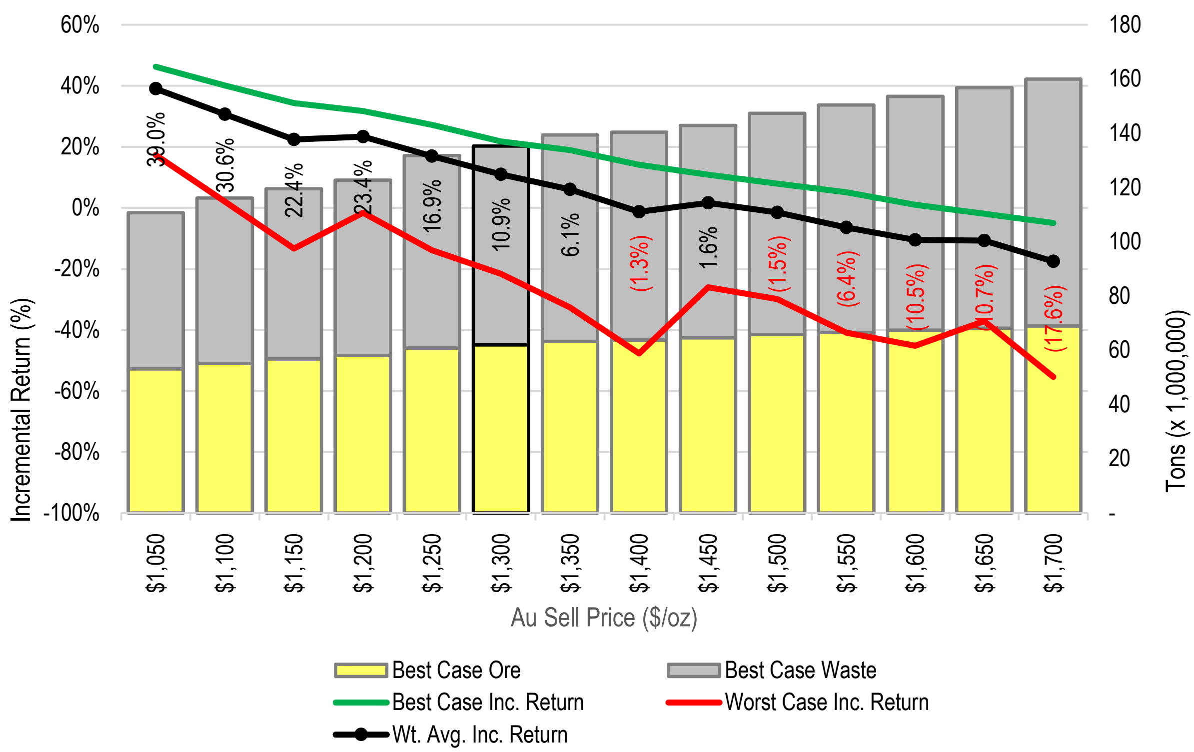


Figure 15-11: West End Nested Pit Shell Incremental Return



15.4 ULTIMATE PIT DESIGNS

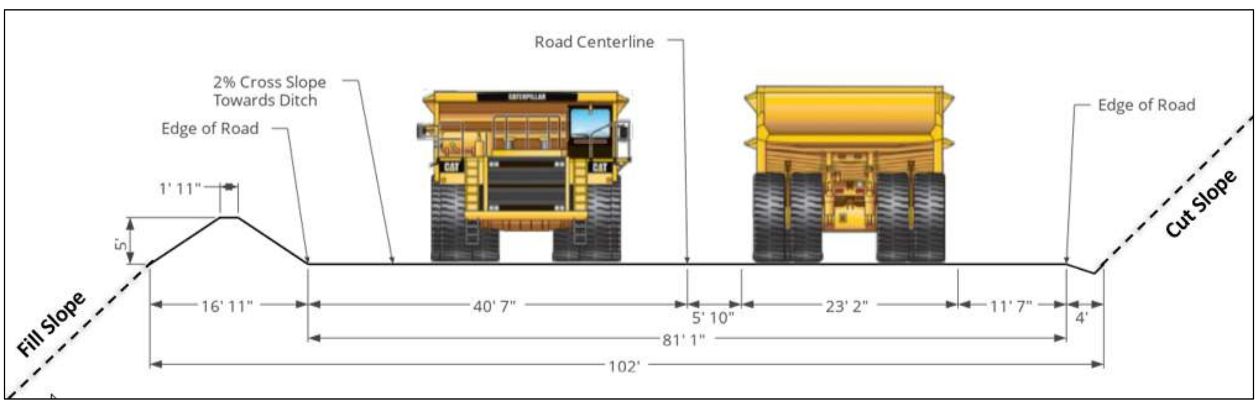
15.4.1 Pit Design Parameters

The ultimate pit design for each pit was based on the selected pit shells and the pit design parameters summarized in Table 15-6. Figure 15-12 presents a typical haul road cross-section that illustrates the 150-t class haul truck running surface design parameter.

Table 15-6: Pit Design Parameters

Design Parameter	Value	Comment
Bench Height	20 ft 40 ft	Single bench ore mining Double bench waste mining; final pit configuration
Bench Face Angle	63º 45º	Bedrock Alluvium
Catch Bench Width	20 ft	
Inter-ramp Angle	36º to 47º	
150t Truck Ramp Width (2-Lane)	102 ft	Including berm and ditch (Figure 15-12)
45t Truck Ramp Width (2-Lane)	50 ft	Including berm and ditch
150t Truck Running Surface	81.1 ft	3.5 x truck operating width
Safety Berm Height	5 ft	½ truck tire height
Safety Berm Width	16.9 ft 1.9 ft	Width at base Berm top
Road Ditch Width	4 ft	
Maximum Ramp Gradient	10% 12%	150t Haul Trucks 45t Articulated Trucks
Minimum Road Bend Radii	64 ft	
Minimum Production Fleet Bench Width	250 ft	Benches less than 250 ft wide are mined with the development (45t haul truck) fleet

Figure 15-12: Typical Haul Road Cross-Section

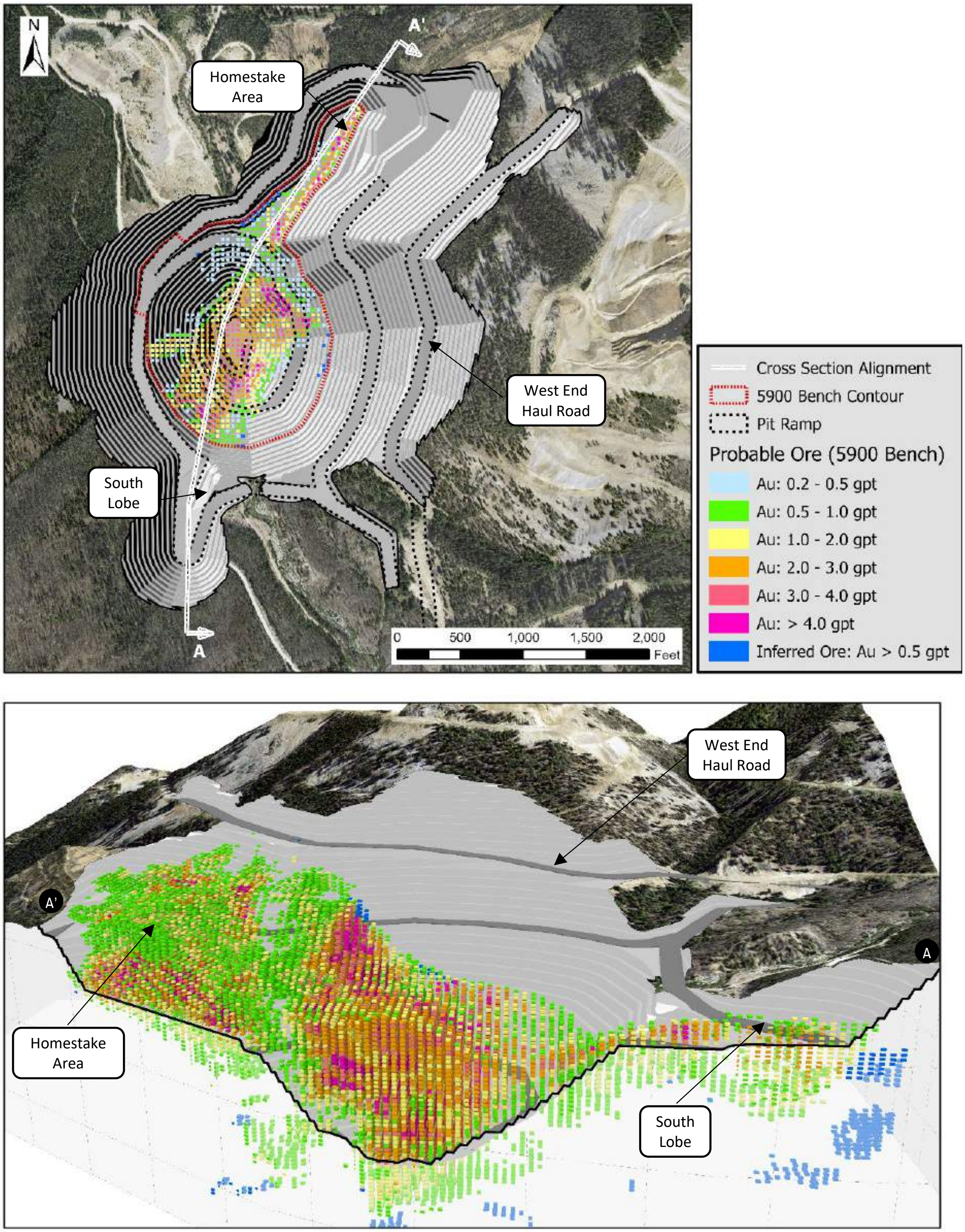


15.4.2 Yellow Pine Ultimate Pit Design

The $1,250 shell was used as a guide for the Yellow Pine ultimate pit design. The pit design deviates from the shell in the following locations as shown in Figure 15-13 and Figure 15-17:

- upper west wall to accommodate the West End Haul Road used to access West End Pit resulting in additional waste;
- south lobe to accommodate the access ramp switchback resulting in reduced access to ore under the ramp; and
- the north lobe (Homestake area) due to limited mine equipment working width to reach the narrow shell bottom following steeply dipping ore resulting in reduced access to ore.

Figure 15-13: Yellow Pine Mineral Reserves and Mineralized Material

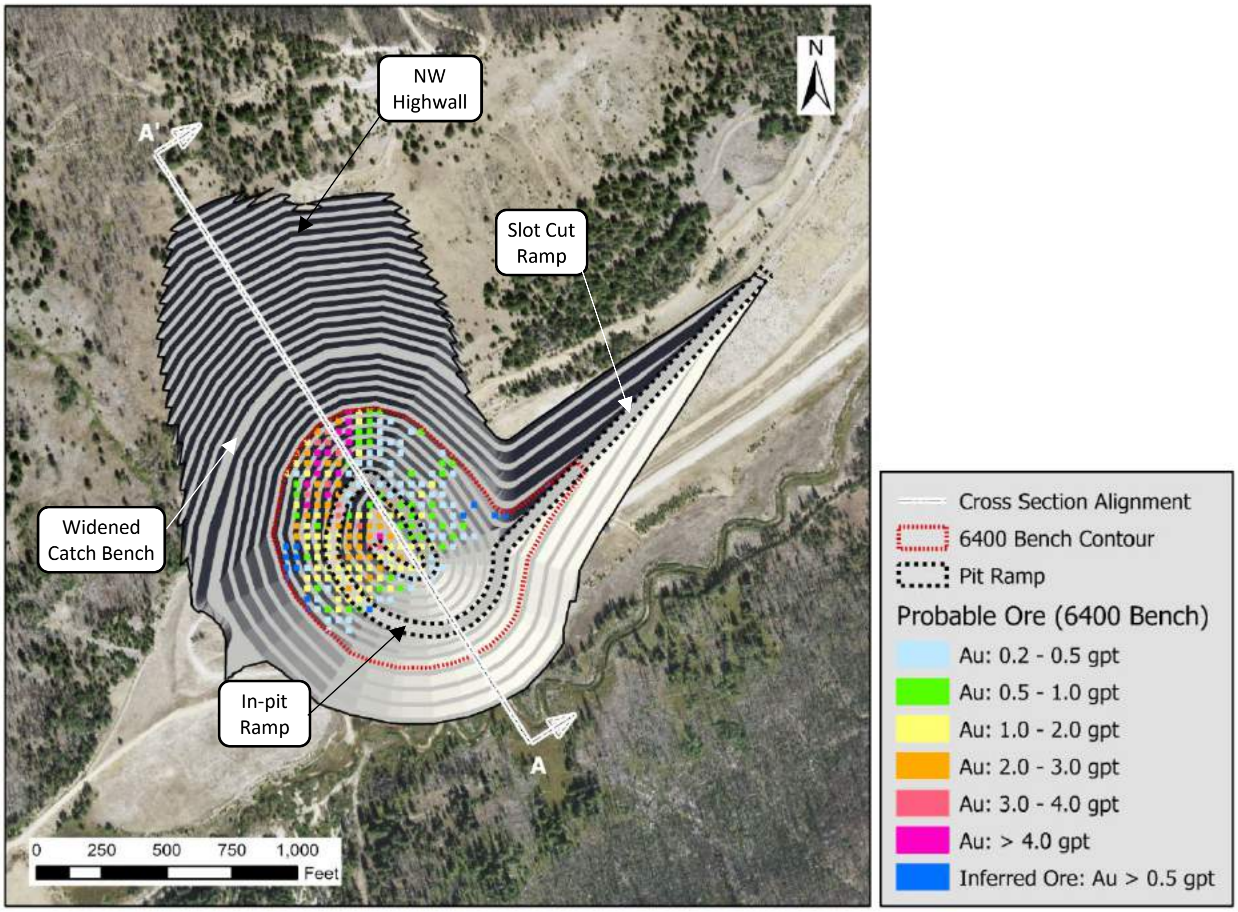


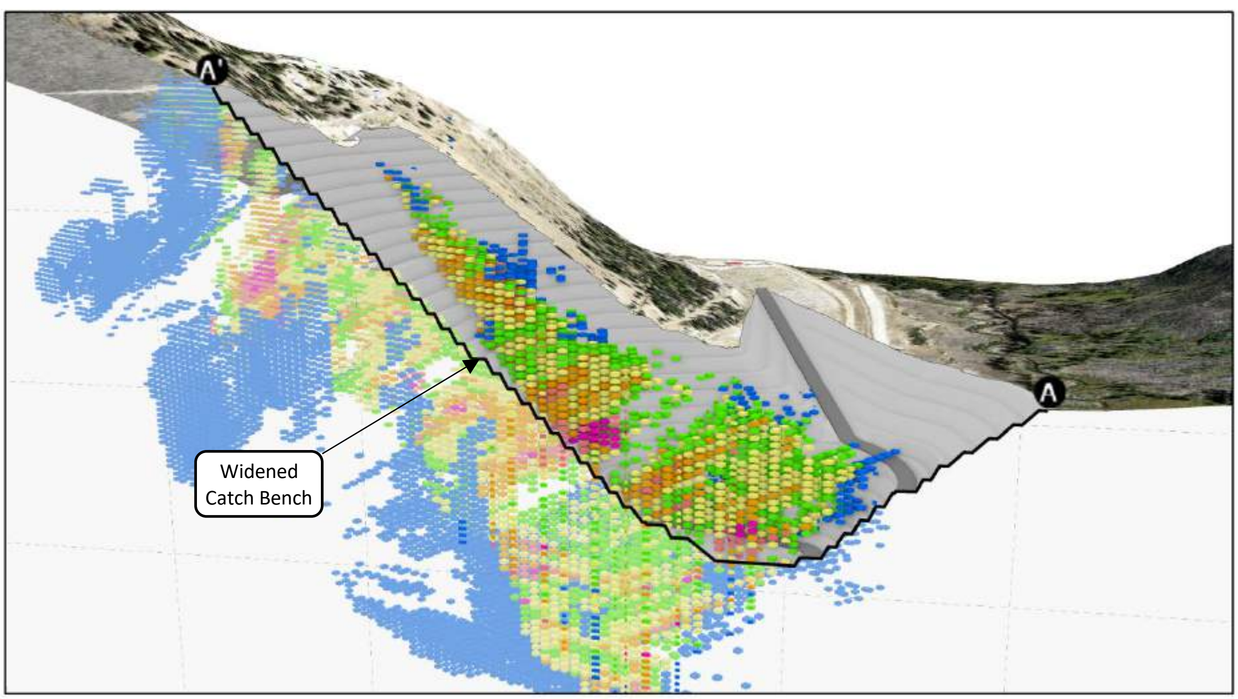
15.4.3 Hangar Flats Ultimate Pit Design

The $750 shell was used as a guide for the Hangar Flats ultimate pit design. The pit design deviates from the shell in the following locations as shown in Figure 15-14 and Figure 15-18:

- slot cut ramp access resulting in additional waste primarily in the alluvium;
- in-pit ramp forcing the ultimate pit limit to extend beyond the shell resulting in additional waste and access to high-value ore at the bottom of the shell;
- limited haul ramp access from the valley floor to upper NW reaches of the shell due to steep topography resulting in the NW portion of the pit highwall designed inside of the shell; and
- a single highwall catch bench widened approximately halfway up the NW highwall to accommodate potential local geotechnical instability resulting from historical underground workings.

Figure 15-14: Hangar Flats Mineral Reserves and Mineralized Material



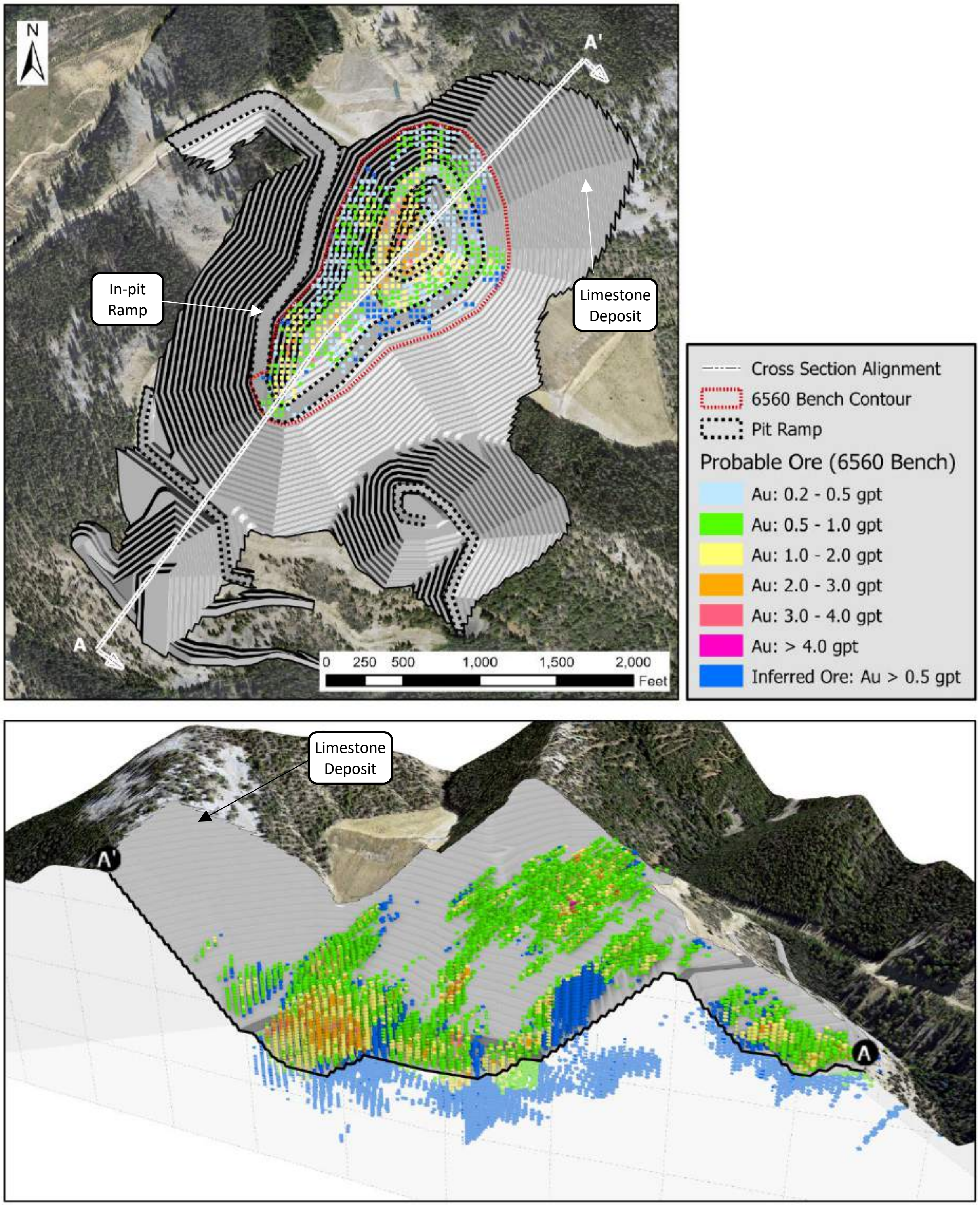


15.4.4 West End Ultimate Pit Design

The $1,300 shell was used as a guide for the West End ultimate pit design. The pit design deviates from the shell in the following locations as shown in Figure 15-15 and Figure 15-19:

- in-pit ramp forcing the ultimate pit limit to extend beyond the shell resulting in additional waste and access to high-value ore at the bottom of the shell; and
- mining equipment access and working width required in the NE portion of the pit to allow access to the limestone deposit for on-site lime generation.

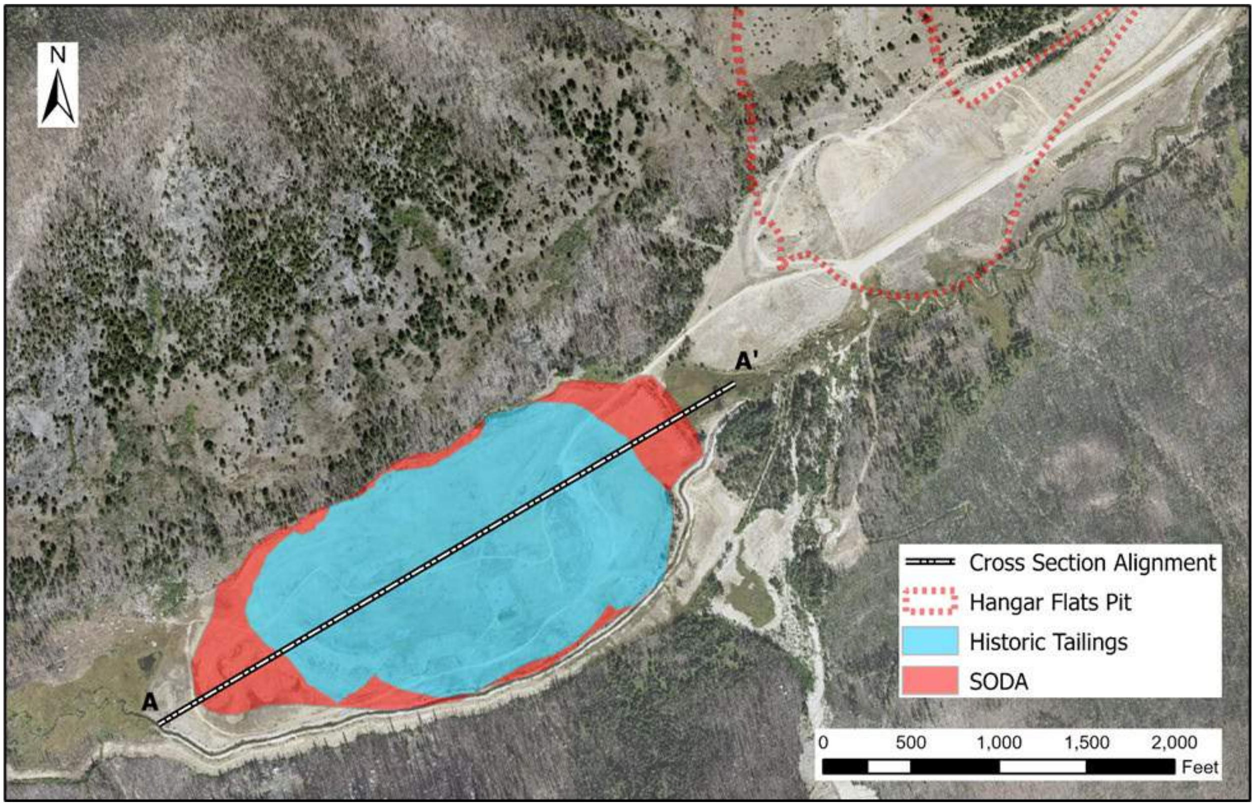
Figure 15-15: West End Mineral Reserves and Mineralized Material

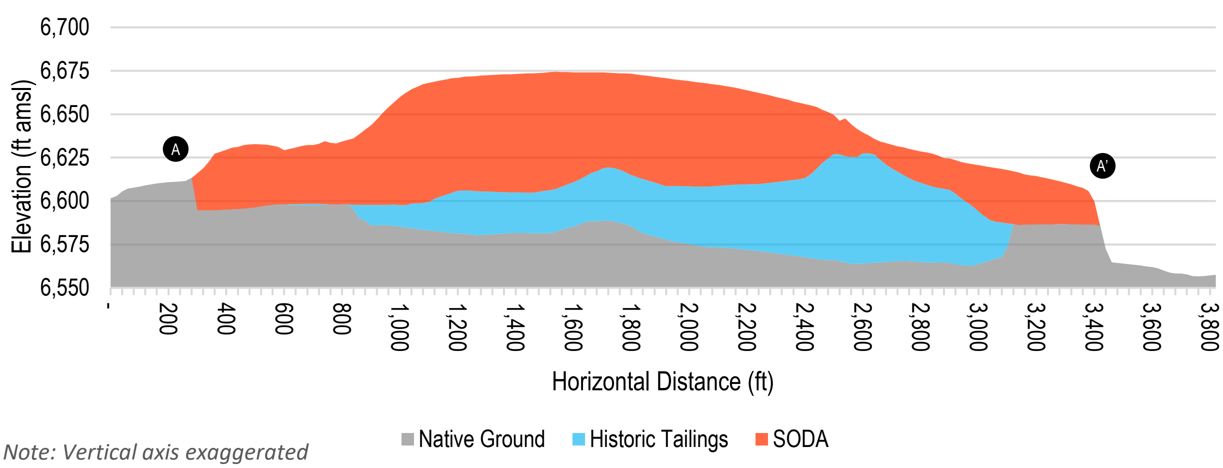


15.4.5 Historical Tailings

The Historical (Bradley) Tailings are located below the Spent Ore Disposal Area (SODA) southwest of the Hangar Flats open pit and partially within the planned development rock storage facility footprint (Figure 15-16). Metallurgical test results show that the contained gold in the Bradley Tailings produces an economic benefit when fed to the process plant concurrent to primary ores. Therefore, the Bradley Tailings are planned to be mined and processed through the mill and are included in the Mineral Reserve.

Figure 15-16: Historical Tailings Mineral Reserves and Mineralized Material





Note: Vertical axis exaggerated

15.5 PIT SHELL TO ULTIMATE DESIGN RECONCILIATION

The CIM recommends that "*a shell-to-design reconciliation be prepared to examine the effectiveness of the design in maintaining the optimized shell configuration. In many cases, the Life-of-Mine design will not follow the outline of the selected optimized pit, as Practitioners will often have to include additional waste materials or exclude mineralized material if this yields a better outcome for a final pit design.*"

For all three pit designs, development rock beyond the selected shell extent is included in the ultimate pit design to accommodate pit haul ramps. Summary reconciliation results are shown in Table 15-7 and cross-section comparisons are shown in Figure 15-17, Figure 15-18, and Figure 15-19.

Table 15-7: Pit Shell to Pit Design Comparison

Yellow Pine	Total (kt)	Ore (kt)	Waste (kt)	Au (koz)	Sb (klb)	Ag (koz)	Au (gpt)	Sb (%)	Ag (gpt)
$1,250 Shell	133,211	51,009	82,202	2,868	118,514	2,868	1.75	0.105	2.24
Pit Design	146,275	47,836	98,439	2,733	106,413	3,420	1.78	0.101	2.22
Pit to Shell Variance (%)	***9.8***	***(6.2)***	***19.8***	***(4.7)***	***(10.2)***	***(7.0)***	***1.6***	***(4.3)***	***(0.8)***
Hangar Flats	Total (kt)	Ore (kt)	Waste (kt)	Au (koz)	Sb (klb)	Ag (koz)	Au (gpt)	Sb (%)	Ag (gpt)
$750 Shell	27,825	9,068	18,757	471	32,674	904	1.62	0.163	3.10
Pit Design	28,783	8,261	20,523	418	27,238	759	1.57	0.150	2.86
Pit to Shell Variance (%)	***3.4***	***(8.9)***	***9.4***	***(11.4)***	***(16.6)***	***(16.1)***	***(2.7)***	***(8.5)***	***(7.9)***
West End	Total (kt)	Ore (kt)	Waste (kt)	Au (koz)	Sb (klb)	Ag (koz)	Au (gpt)	Sb (%)	Ag (gpt)
$1,300 Shell	135,210	45,068	90,142	1,604	-	2,004	1.11	-	1.38
Pit Design	177,761	48,859	131,902	1,612	-	2,011	1.09	-	1.36
Pit to Shell Variance (%)	***31.5***	***(1.8)***	***46.3***	***(0.5)***	***-***	***0.4***	***(1.3)***	***-***	***(1.4)***
All Open Pits	Total (kt)	Ore (kt)	Waste (kt)	Au (koz)	Sb (klb)	Ag (koz)	Au (gpt)	Sb (%)	Ag (gpt)
Shells	296,246	105,145	191,101	4,943	151,188	6,584	1.95	0.065	1.95
Pit Designs	352,819	101,956	250,863	4,762	133,651	6,190	1.89	0.059	1.89
Pit to Shell Variance (%)	***19.1***	***(3.0)***	***31.3***	***(3.7)***	***(11.6)***	***(6.0)***	***(3.0)***	***(8.8)***	***(3.0)***

Figure 15-17: Yellow Pine Pit Shell to Ultimate Design Reconciliation

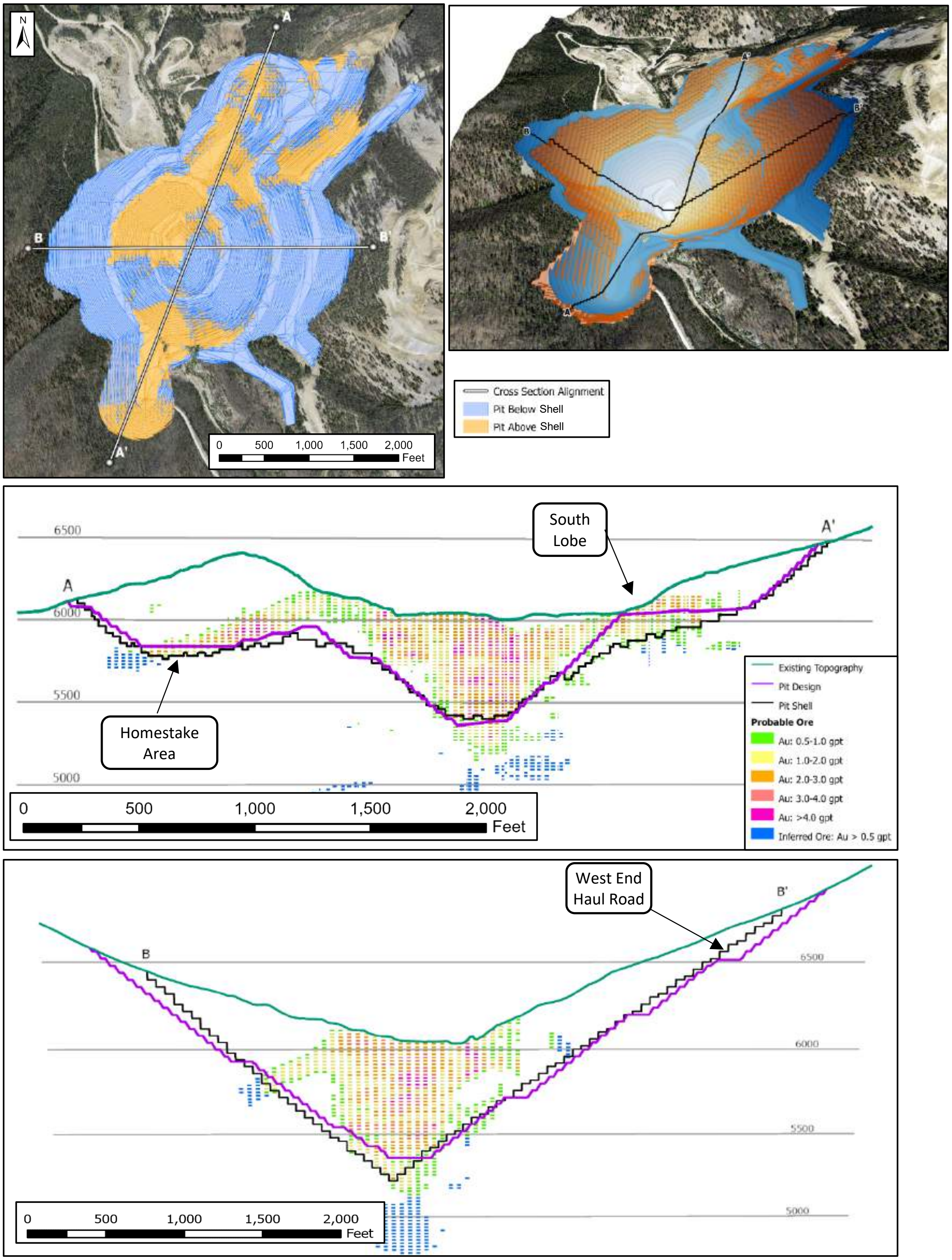


Figure 15-18: Hangar Flats Pit Shell to Ultimate Design Reconciliation

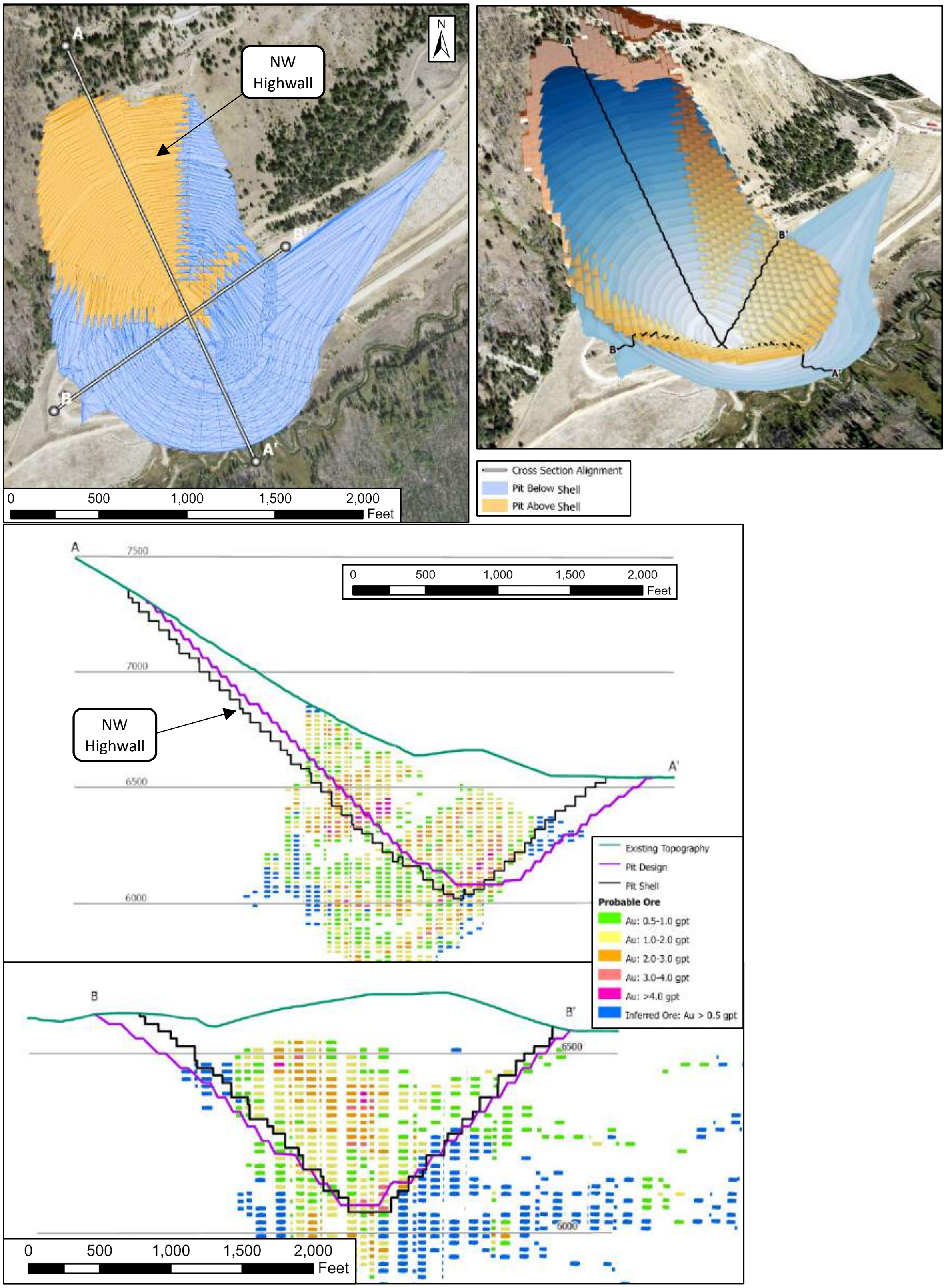
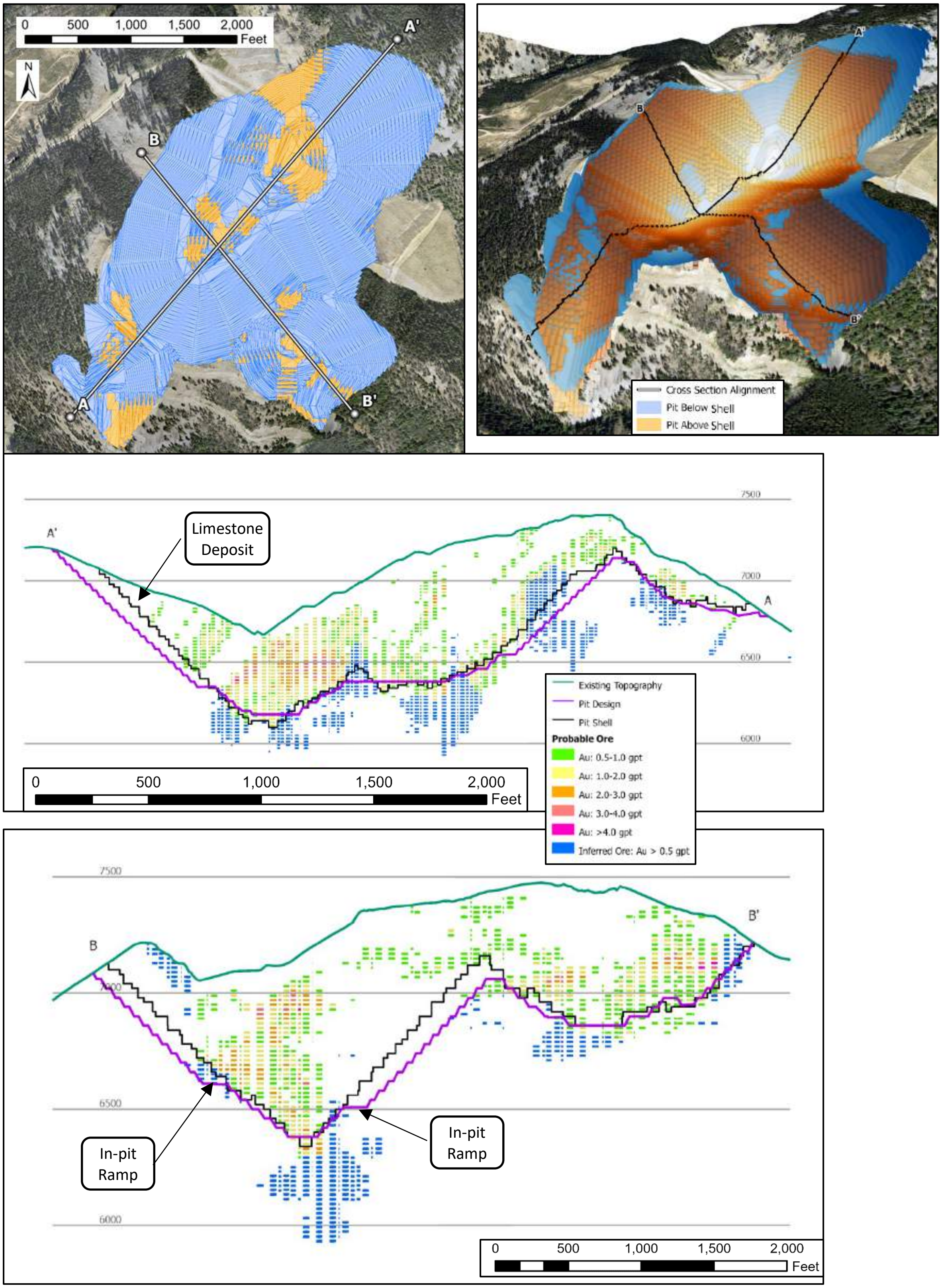


Figure 15-19: West End Pit Shell to Ultimate Design Reconciliation



15.6 CUT-OFF GRADE AND RESOURCE ORE TYPE CLASSIFICATION

The CIM states "*The concept of a cut-off grade or value is a fundamental component in the preparation of Mineral Reserve estimates, mine designs, and mine production schedules. A cut-off grade or value is defined as the grade or value that is used to differentiate between ore and waste for a given set of conditions, parameters and time frame.*"

Initial mine planning was performed using the ultimate pit designs and break-even cut-off values on a block-by-block basis. This resulted in the ore mining rate exceeding the mill throughput rate unless either the mining rate was significantly reduced, or substantial stockpiles could be established to accept the lower value ore. Reducing the mining rate would defer access to higher-value ore and subsequently reduce the project NPV. Stockpile capacity is limited by steep terrain and the intent to restrict site disturbance. Therefore, an optimal mineral reserve cut-off strategy was developed using elevated cut-off values in the mine schedule to maximize recoverable metal and efficiently utilize the available stockpile capacity. This cut-off strategy enabled a practical mining rate that improved project value by processing higher value ore earlier in the mill feed schedule. The life-of-mine cut-off values are shown in Table 15-8. The approximate variable cut-off values over time identified in the mineral reserve cut-off strategy analyses are shown on Figure 15-20 and Figure 15-21.

Cut-off values in the FS are lower than in the PFS primarily due to incorporating long-term stockpiles into the mine plan and lower processing costs. In the PFS, there was no provision for long-term stockpiles resulting in elevated cut-off values to ensure the highest grade ore available in the mine plan was processed. The addition of long-term stockpiles in the FS allows for lowering the elevated cut-off value as compared to the PFS while maintaining the highest grade available ore is processed throughout the mine life and extends the mill life by approximately 2 years.

Ore type classification for the three open pits was determined on a block-by-block basis by calculating the block NPR value for each potential process stream designation (i.e. high Sb sulfide, low Sb sulfide, oxide, low Sb transitional) and classifying the block ore type by whichever process stream designation had the highest potential value. The Historical Tailings will be processed concurrently with ore sourced from the open pits during the first four years of operations. Therefore, the Historical Tailings ore type classification is proportional to the open pit ore type classification during the first four years of operations since the Historical Tailings will accompany the open pit ore process stream designation.

Table 15-8: Life-of-Mine Cut-off Values

Deposit	Net of Process Revenue Cut-off ($/st)	Approximate Equivalent Gold Cut-off (gpt)
Yellow Pine	5.18	0.46
Hangar Flats	5.31	0.49
West End	3.68	0.49
Open Pit Average	**4.52**	**0.48**
Historical Tailings	4.52	0.39

Figure 15-20: Approximate Gold Cut-off by Schedule Year

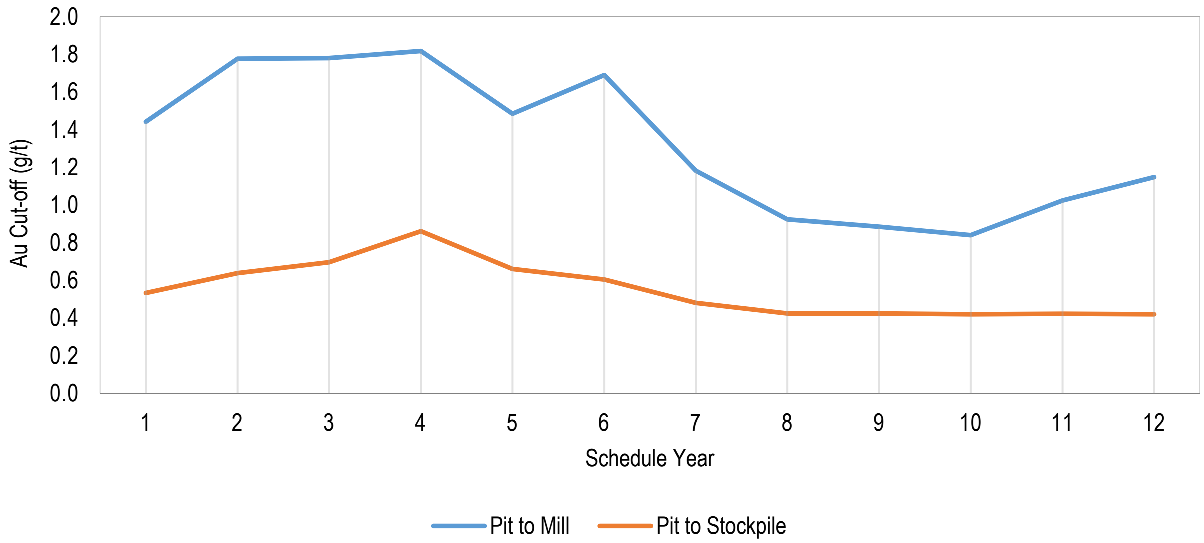
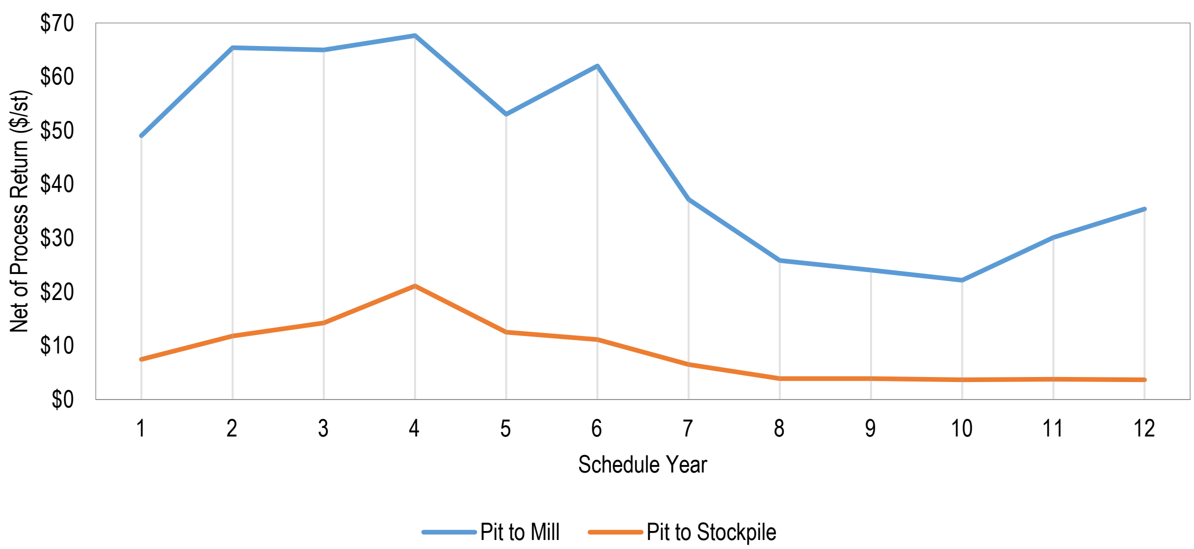


Figure 15-21: Approximate NPR Cut-off by Schedule Year



15.7 MINERAL RESERVE ESTIMATE

The Stibnite Gold Project Mineral Reserves are presented in Table 15-9 and Table 15-10.

Table 15-9: Probable Mineral Reserves Summary (Imperial Units)

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony(4)	Silver
Imperial Units	(kst)	(oz/st)	(%)	(oz/st)	(oz)	(lb)	(oz)
Yellow Pine							
Low Sb Sulfide – Proven	2,797	0.062	0.022	0.059	173	1,213	166
High Sb Sulfide – Proven	2,710	0.076	0.450	0.154	205	24,380	417
Yellow Pine Total Proven	**5,507**	**0.069**	**0.232**	**0.106**	**378**	**25,594**	**584**
Low Sb Sulfide – Probable	38,667	0.048	0.009	0.044	1,874	6,646	1,715
High Sb Sulfide – Probable	8,568	0.054	0.463	0.131	466	79,378	1,125
Yellow Pine Total Probable	**47,235**	**0.050**	**0.091**	**0.060**	**2,340**	**86,024**	**2,840**
Low Sb Sulfide – Proven & Probable	41,463	0.049	0.009	0.045	2,047	7,859	1,881
High Sb Sulfide – Proven & Probable	11,279	0.060	0.460	0.137	671	103,758	1,543
Yellow Pine Proven & Probable Mineral Reserves	**52,742**	**0.052**	**0.106**	**0.065**	**2,718**	**111,617**	**3,423**
Hangar Flats (1)							
Low Sb Sulfide – Probable	5,696	0.039	0.018	0.048	223	2,104	273
High Sb Sulfide – Probable	3,411	0.056	0.369	0.141	191	25,148	483
Hangar Flats Probable Mineral Reserves	**9,107**	**0.046**	**0.150**	**0.083**	**414**	**27,252**	**756**
West End (1)							
Oxide – Probable	5,235	0.016	-	0.025	83	-	133
Low Sb Sulfide – Probable	16,801	0.039	-	0.038	649	-	635
Transitional – Probable	28,483	0.030	-	0.043	855	-	1,236
West End Probable Mineral Reserves	**50,519**	**0.031**	**-**	**0.040**	**1,587**	**-**	**2,004**
Historical Tailings (1)(2)							
Low Sb Sulfide – Probable	2,019	0.034	0.166	0.084	68	6,692	169
High Sb Sulfide – Probable	943	0.034	0.166	0.084	32	3,125	79
Historical Tailings Probable Mineral Reserves	**2,962**	**0.034**	**0.166**	**0.084**	**100**	**9,817**	**247**
Proven & Probable Mineral Reserves							
Oxide – Probable	5,235	0.016	-	0.025	83	-	133
Low Sb Sulfide – Proven & Probable	65,980	0.045	0.013	0.045	2,988	16,656	2,958
High Sb Sulfide – Proven & Probable	15,632	0.057	0.422	0.135	894	132,031	2,104
Transition – Probable	28,483	0.030	-	0.043	855	-	1,236
Total Proven & Probable Mineral Reserves (3)	**115,330**	**0.042**	**0.422**	**0.056**	**4,819**	**148,686**	**6,431**

Notes:
(1) Deposit does not have a Measured Resource. Only Indicated Resource reported.
(2) Historical Tailings ore type classification is proportional to the pit-sourced mill feed during Historical Tailings processing.
(3) Metal prices used for Mineral Reserves: $1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb.
(4) Antimony recovery is expected from High Sb Sulfide ore only and contains 132,031 klbs of Sb.

Table 15-10: Probable Mineral Reserves Summary (Metric Units)

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony	Silver
Metric Units	(kt)	(g/t)	(%)	(g/t)	(t)	(t)	(t)
Yellow Pine							
Low Sb Sulfide – Proven	2,537	2.12	0.022	2.04	5.4	550	5.2
High Sb Sulfide – Proven	2,459	2.60	0.450	5.28	6.4	11,059	13.0
Yellow Pine Total Proven	**4,996**	**2.35**	**0.232**	**3.63**	**11.8**	**11,609**	**18.1**
Low Sb Sulfide – Probable	35,078	1.66	0.009	1.52	58.3	3,014	53.3
High Sb Sulfide – Probable	7,773	1.86	0.463	4.50	14.5	36,005	35.0
Yellow Pine Total Probable	**42,851**	**1.70**	**0.091**	**2.06**	**72.8**	**39,020**	**88.3**
Low Sb Sulfide – Proven & Probable	37,615	1.69	0.009	1.56	63.7	3,565	58.5
High Sb Sulfide – Proven & Probable	10,232	2.04	0.460	4.69	20.9	47,064	48.0
Yellow Pine Proven & Probable Mineral Reserves	**47,847**	**1.77**	**0.106**	**2.23**	**84.5**	**50,629**	**106.5**
Hangar Flats (1)							
Low Sb Sulfide – Probable	5,167	1.34	0.018	1.65	6.9	954	8.5
High Sb Sulfide – Probable	3,095	1.92	0.369	4.85	5.9	11,407	15.0
Hangar Flats Probable Mineral Reserves	8,262	1.56	0.150	2.85	12.9	12,361	23.5
West End (1)							
Oxide – Probable	4,749	0.54	-	0.87	2.6	-	4.1
Low Sb Sulfide – Probable	15,242	1.33	-	1.30	20.2	-	19.7
Transitional – Probable	25,839	1.03	-	1.49	26.6	-	38.5
West End Probable Mineral Reserves	**45,830**	**1.08**	**-**	**1.36**	**49.3**	**-**	**62.3**
Historical Tailings (1)(2)							
Low Sb Sulfide – Probable	1,832	1.16	0.166	2.86	2.1	3,036	5.2
High Sb Sulfide – Probable	855	1.16	0.166	2.86	1.0	1,417	2.4
Historical Tailings Probable Mineral Reserves	**2,687**	**1.16**	**0.166**	**2.86**	**3.1**	**4,453**	**7.7**
Proven & Probable Mineral Reserves							
Oxide – Probable	4,749	0.54	-	0.87	2.6	-	4.1
Low Sb Sulfide – Proven & Probable	59,856	1.55	0.013	1.54	92.9	7,555	92.0
High Sb Sulfide – Proven & Probable	14,181	1.96	0.422	4.61	27.8	59,888	65.4
Transitional – Probable	25,839	1.03	-	1.49	26.6	-	38.5
Total Proven & Probable Mineral Reserves (3)	**104,625**	**1.43**	**0.064**	**1.91**	**149.9**	**67,443**	**200.0**

Notes:
(1) Deposit does not have a Measured Resource. Only Indicated Resource reported.
(2) Historical Tailings ore type classification is proportional to the pit-sourced mill feed during Historical Tailings processing.
(3) Metal prices used for Mineral Reserves: $1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb.
(4) Antimony values are reported only for ore scheduled in the mine plan that is classified as High Sb Sulfide.

15.8 REFERENCES

Strata, A Professional Services Corporation (2014). Preliminary Feasibility Study Slope Designs for Three Proposed Open Pits at the Golden Meadows Project in the Stibnite Mining District, Valley County, Idaho, February 14, 2014.

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16 MINING METHODS

16.1 INTRODUCTION

The Stibnite Gold Project FS mine plan consists of mining three primary mineral deposits and re-mining the Historical Tailings using conventional open pit shovel and truck mining methods. The mining operation will deliver 8.05 million short tons (st) of oxide and sulfide mineralized ore to the crusher per year (nominally 22,050 st per day).

Ore from the three open pits, Yellow Pine, Hangar Flats, and West End, will be sent to either the crusher located near the processing plant or one of several ore stockpiles located throughout the Project site. The Historical Tailings will be trucked to a re-pulping facility adjacent to the tailings deposit and hydraulically transferred to the process plant grinding circuit via a re-pulping facility. Most of the development rock from the three open pits will be sent to one of five destinations: the TSF embankment, the TSF Buttress, the Yellow Pine open pit backfill, the Hangar Flats open pit backfill, and the West End open pit backfill as shown on Figure 16.1. A small portion of the development rock will be used in various development projects especially during pre-production as further discussed in Section 16.4. A summary of the ore tonnage by process route and waste tonnage from each of the primary deposits and the Historical Tailings is provided in Table 16.1.

The general sequence of open pit mining is Yellow Pine first, Hangar Flats second, and West End last. This sequence generally progresses from mining highest value ore to lowest value ore and accommodates backfilling the Yellow Pine and Hangar Flats open pits with material mined from West End open pit thereby accelerating concurrent reclamation and restoration of the EFSFSR. The Historical Tailings will be mined and processed during the first four years of operation concurrent with mining ore from the Yellow Pine open pit.

The mine planning methodology applied in the SGP FS consisted of the following general procedures:

- designing ultimate pits designs (Section 15.4);
- designing internal pit phases for each open pit (Section 16.2);
- developing the strategic mine plan (Section 16.3);
- scheduling mine development work and incorporating it into the strategic mine plan (Section 16.4);
- designing and scheduling stockpiles and development rock storage facilities (Section 16.6);
- optimizing the process ore feed schedule (Section 16.7);
- scheduling a detailed mine plan (16.8);
- developing equipment maintenance and consumables schedules (Section 16.9);
- developing staffing schedules (16.11);
- estimating the mine capital cost and operating cost schedule (Section 16.12); and,
- performing an ultimate pit limit analysis validation (Section 16.12.3).

Figure 16.1: Sitewide Mining Related Features

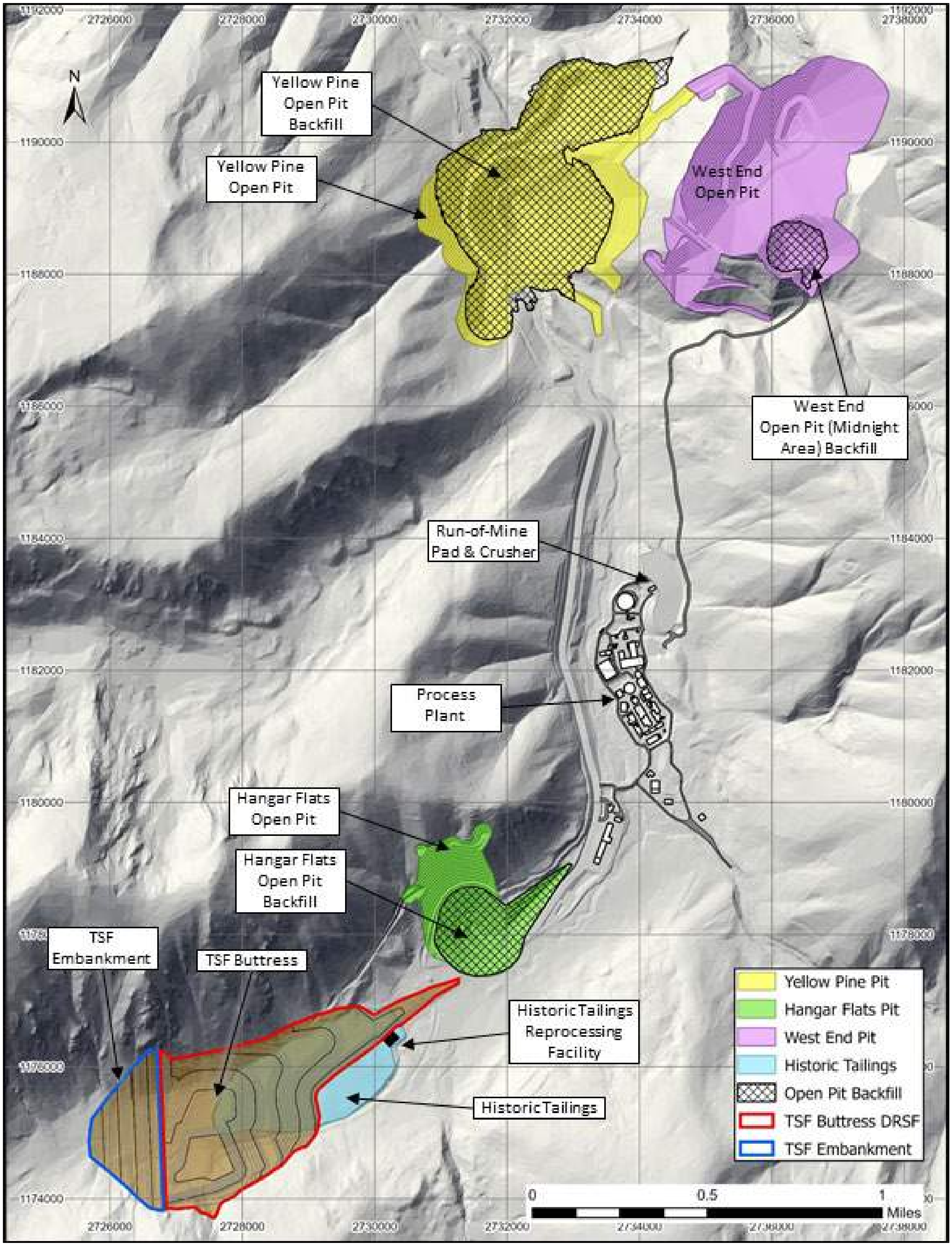


Table 16.1: Summary of Mine Plan Ore Type and Tonnage by Deposit

Deposit & Ore Type	Tonnage (000s)	Average Grade Gold (g/t)	Average Grade Antimony (%)	Average Grade Silver (g/t)	Total Contained Metal Gold (000s oz)	Total Contained Metal Antimony (klbs)	Total Contained Metal Silver (000s oz)
Yellow Pine							
Low Sb Sulfide	37,615	1.69	0.009	1.56	2,047	7,859	1,881
High Sb Sulfide	10,232	2.04	0.460	4.69	671	103,758	1,543
Total Ore	**47,847**	**1.77**	**0.106**	**2.23**	**2,718**	**111,617**	**3,423**
Development Rock	99,666						
Total Tonnage	147,512						
Strip Ratio	2.08						
Hangar Flats							
Low Sb Sulfide	5,167	1.34	0.018	1.65	223	2,104	273
High Sb Sulfide	3,095	1.92	0.369	4.85	191	25,148	483
Total Ore	**8,262**	**1.56**	**0.150**	**2.85**	**414**	**27,252**	**756**
Development Rock	20,066						
Total Tonnage	38,328						
Strip Ratio	2.43						
West End							
Oxide	4,749	0.54	-	0.87	83	-	133
Low Sb Sulfide	15,242	1.33	-	1.30	649	-	635
Transitional	25,839	1.03	-	1.49	855	-	1,236
Total Ore	**45,830**	**1.08**	**-**	**1.36**	**1,587**	**-**	**2,004**
Development Rock	134,031						
Total Tonnage	179,861						
Strip Ratio	2.92						
Historical Tailings							
Low Sb Sulfide	1,832	1.16	0.166	2.86	68	6,692	169
High Sb Sulfide	855	1.16	0.166	2.86	32	3,125	79
Total Ore	**2,687**	**1.16**	**0.166**	**2.86**	**100**	**9,817**	**247**
Development Rock[1]	5,218						
Total Tonnage	7,905						
Strip Ratio	1.94						
All Deposits							
Oxide	4,749	0.54	-	0.87	83	-	133
Low Sb Sulfide	59,856	1.55	0.013	1.54	2,988	16,656	2,958
High Sb Sulfide	14,181	1.96	0.422	4.61	894	132,031	2,104
Transitional	25,839	1.03	-	1.49	855	-	1,236
Total Ore	**104,625**	**1.43**	**0.064**	**1.91**	**4,819**	**148,686**	**6,431**
Development Rock	258,980						
Total Tonnage	363,605						
Strip Ratio	2.49						

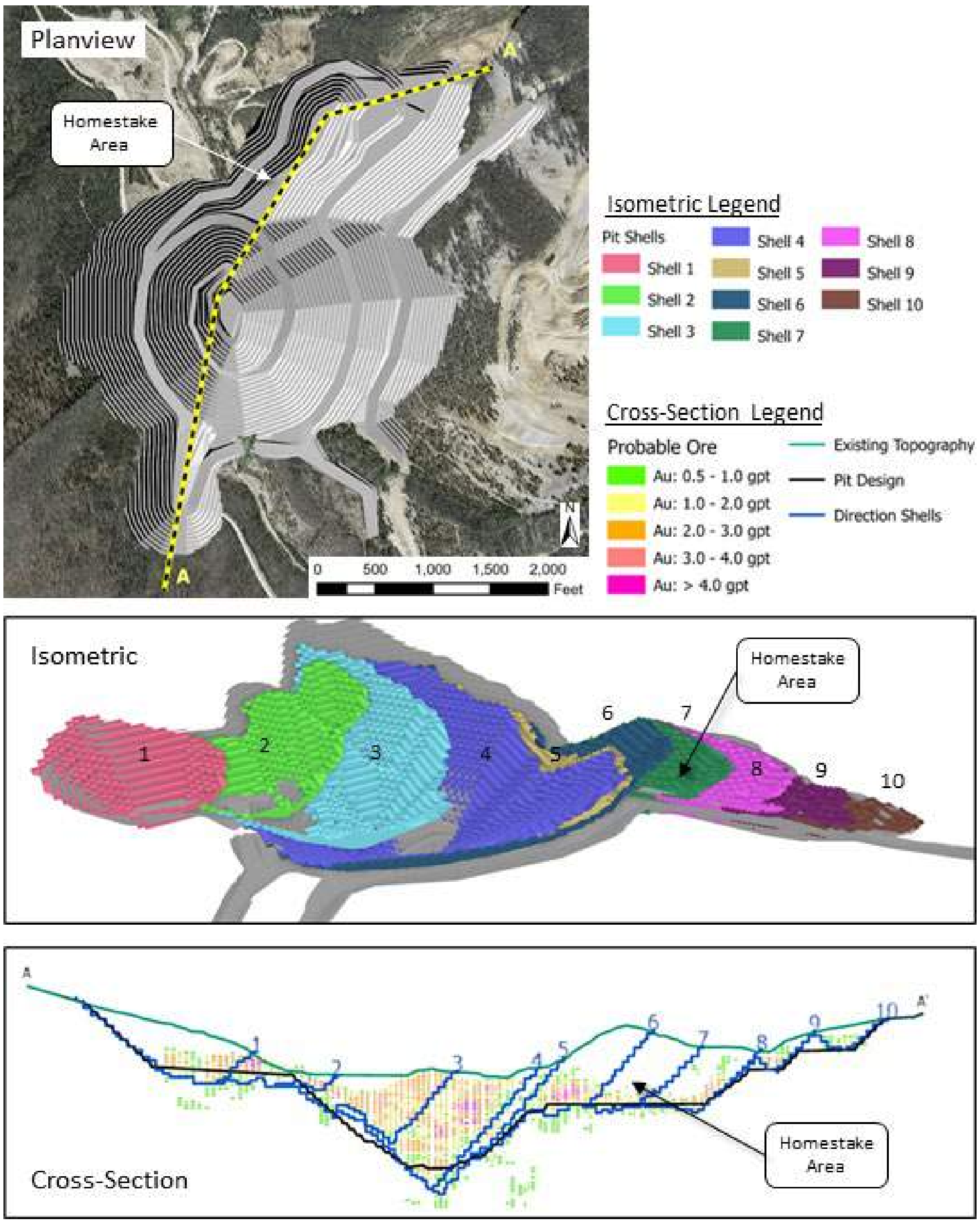
16.2 OPEN PIT PHASE DESIGN

The purpose of designing phases within the ultimate pit designs is to balance development rock stripping and ore access, bring higher-value ore forward in the mine schedule, guide detailed mine scheduling, allow for concurrent backfilling of pits and to facilitate concurrent reclamation and restoration. The open pit phase designs were based on the nested pit shells generated in the Ultimate Pit Limit Analysis described in Section 15.4. Phase designs include all interim in-pit access roads to develop each phase and allowance for adequate equipment operating requirements.

16.2.1 Yellow Pine Pit Phase Design

In addition to the nested pit shells produced in the Ultimate Pit Limit Analysis, a suite of directional pit shells was generated for the Yellow Pine deposit to identify potential for mining the main portion of Yellow Pine first and the northern Homestake area last (Figure 16.2). This phasing sequence allows for accelerated access to high-value ore deep in the central Yellow Pine deposit and provides for a short development rock haul from the Homestake area to the Yellow Pine pit backfill to reduce haulage cost.

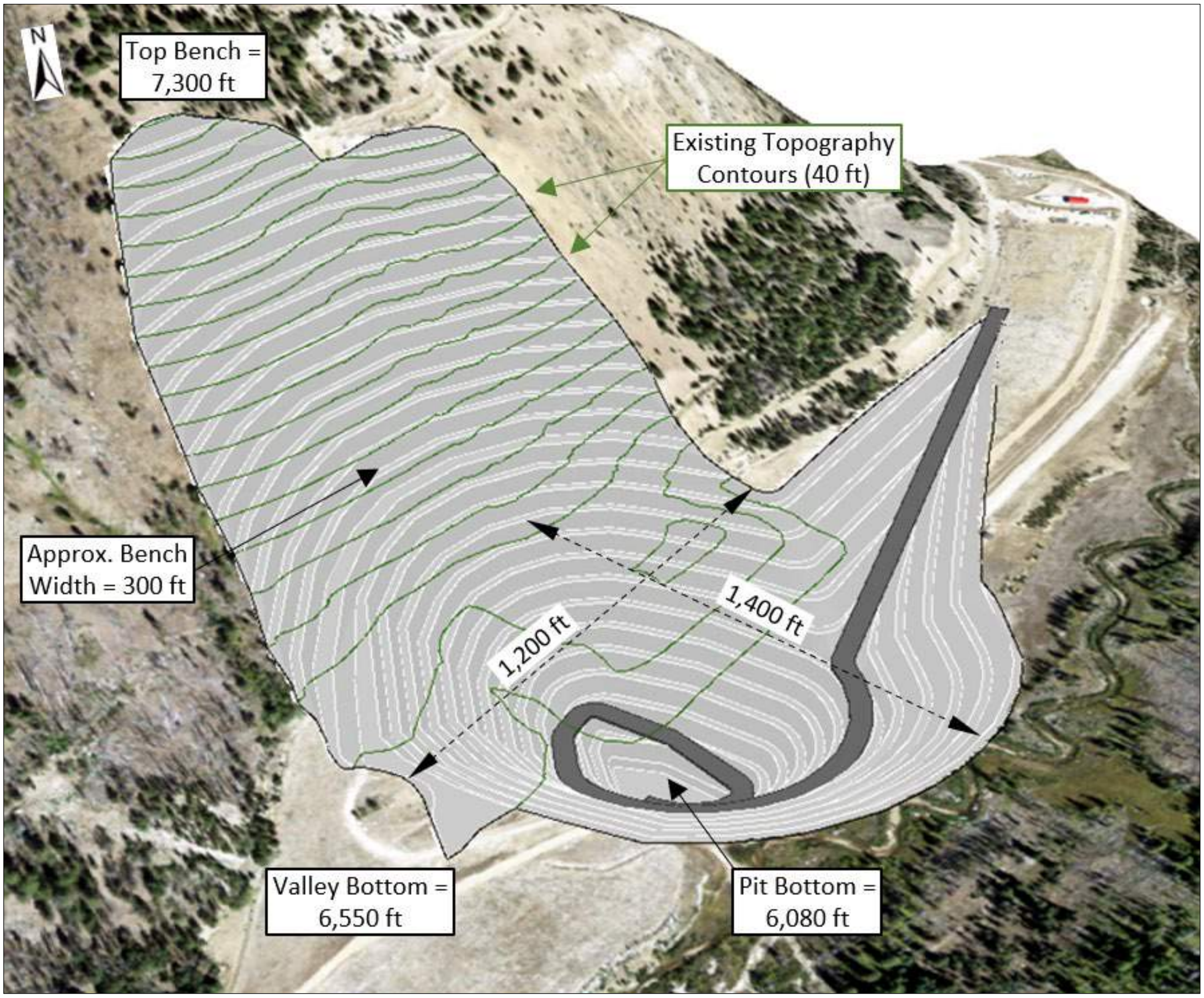
Figure 16.2: Yellow Pine Directional Pit Shells



16.2.2 Hangar Flats Pit Phase Design

The Hangar Flats pit design consists of a single phase due to its small size and steep topography which requires a top-down mining approach. An internal phase within Hangar Flats would likely result in very narrow bench widths in the northwest highwall causing significantly reduced mining production rates (Figure 16.3). Additional discussion regarding the Hangar Flats open pit geometry alternatives is provided in Section 16.3.2.

Figure 16.3: Hangar Flats Pit Design

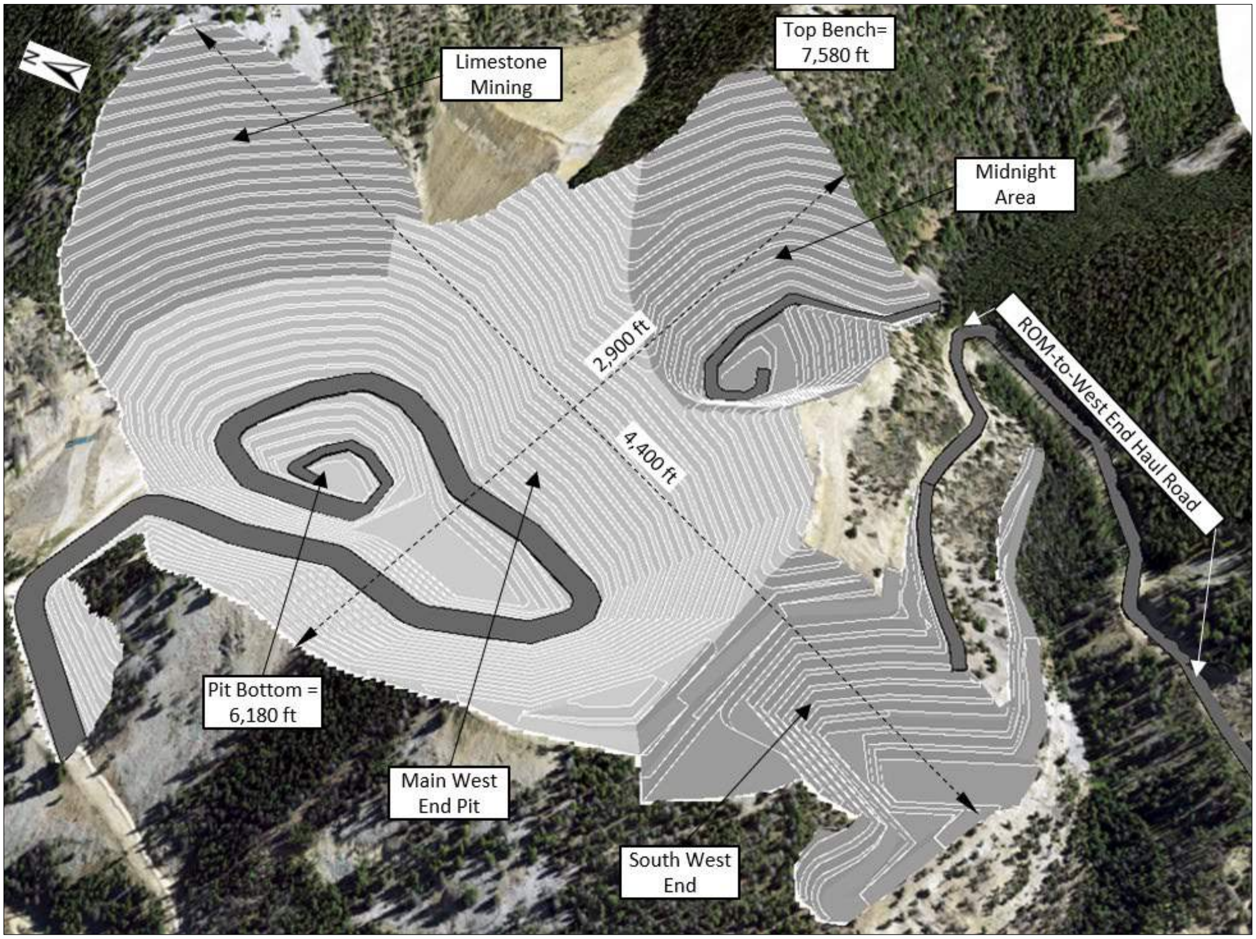


16.2.3 West End Pit Phase Design

Four pit phases were designed for the West End pit: (1) Middle Marble limestone mining, (2) Midnight area pit production, (3) South West End pit production, and (4) Main West End pit production as shown on Figure 16.4. Mining limestone from the Middle Marble geologic unit located in the northeast portion of the West End open pit is required for the lime kiln to produce lime used in ore processing. The Midnight Area phase sequence is primarily driven by when access is available for backfilling this area using development rock produced in the Main West End phase. The South West End phase is accessible via the ROM-to-West End Haul Road and can be mined independent of the Main West

End phase. The Main West End phase does not benefit significantly from additional phasing due to the homogeneous nature of the ore body.

Figure 16.4: West End Pit Phases

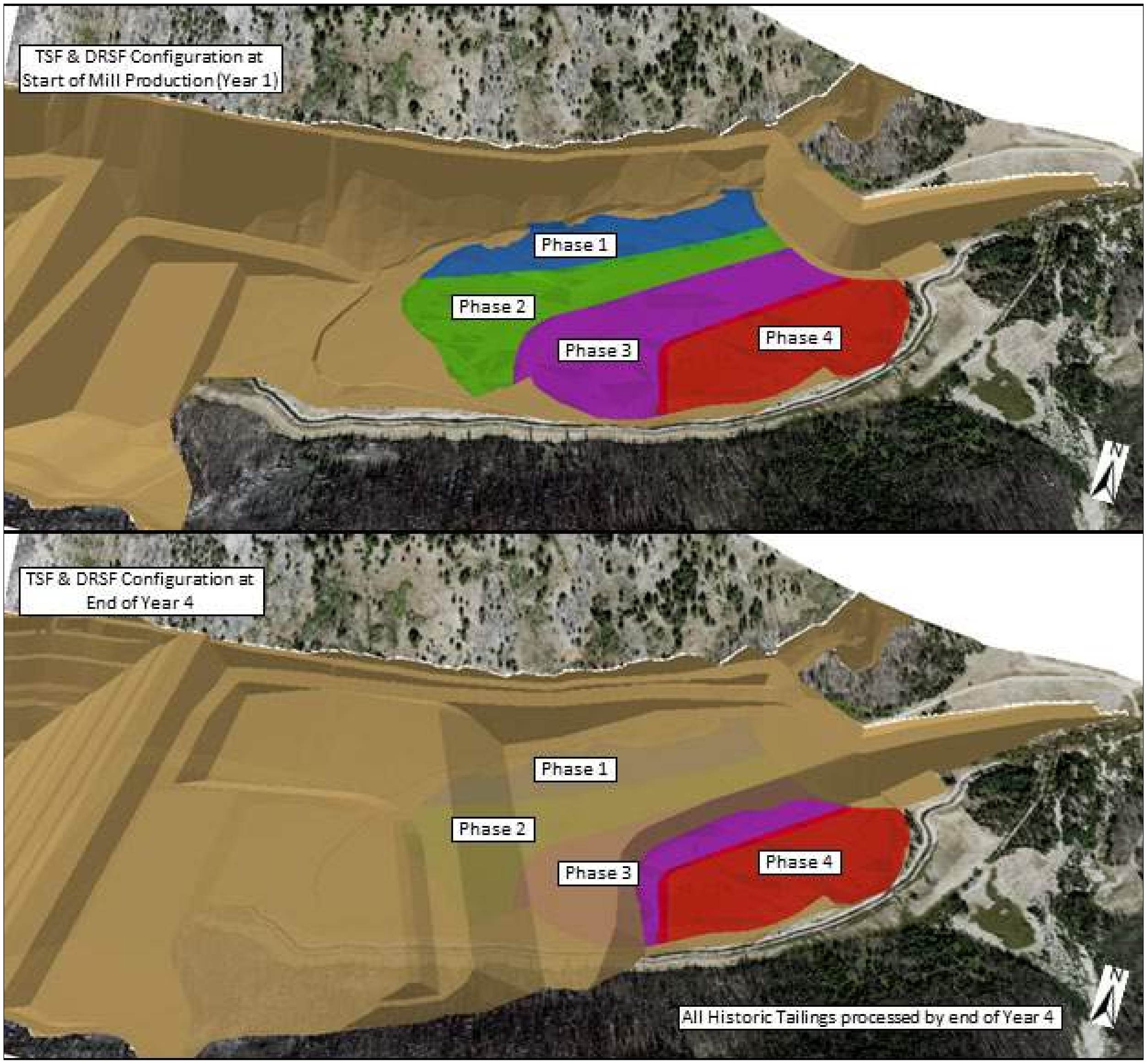


16.2.4 Historical Tailings Phase Design

Approximately 3 million tons of Historical Tailings from processing ore in the World War II era underlie spent heap leach material. The spent material will be removed and used as construction material for the TSF exposing the Historical Tailings. The 2,687 kt of Historical Tailings will be excavated and hauled by truck to a nearby handling facility where it would be screened, re-pulped, and pumped to the grinding circuit.

For mine planning purposes, the Historical Tailings resource is modeled with constant grade and value throughout the deposit. Therefore, phasing the Historical Tailings is not influenced by advancing access to higher value ore but instead by the need to accommodate construction of adjacent facilities and avoid costs associated with double handling of the material. The Historical Tailings are planned to be excavated and processed during the first 4 years of mill operation as shown on Figure 16.5.

Figure 16.5: Historical Tailings Phases



16.3 MINE SEQUENCE ANALYSIS

The mine sequence analysis consisted of evaluating various combinations of mining sequence, pit design alternatives, fleet alternatives, and mining production rates to optimize project value and produce a strategic mine plan. The strategic mine plan was then used as a blueprint for detailed mine planning including stockpile optimization, equipment scheduling, equipment cost estimating, development rock storage facility scheduling, mill feed optimization, and the life-of-mine production schedule. The primary objectives for the mine sequence analysis included:

- identify most favorable Hangar Flats open pit geometry;
- evaluate mine production ramp-up and peak production rate alternatives;
- maximize access to high value ore early in the mine life for increased project value;
- identify optimal mine production fleet criterion;

- maximize mine production equipment productivity and utilization;
- balance development rock stripping and access to ore;
- ensure consistent ore feed to the process plant throughout the mine life;
- provide pit sourced material to construction projects as needed particularly during construction;
- ensure project objectives and constraints are achieved such as backfilling the Yellow Pine and Hangar Flats Pits;
- support concurrent reclamation and restoration; and,
- generate a period-based (monthly prior to Year 3 and quarterly after) mine production schedule.

16.3.1 Process Facility Mined Material Requirements

There are four general types of mined material that affect the mining sequence and mining production rate:

- Run-of-mine (**ROM**) sulfide ore - Since all material to be processed during the first few years of operation is sulfide ore from the Yellow Pine open pit, the process plant throughput ramp-up schedule is based on ROM sulfide ore.
- ROM oxide ore - Substantial quantities of oxide ore are not encountered until the West End open pit is in full production. Therefore, a direct cyanide leaching circuit is planned to be operational starting in Year 7. High-value oxide ore mined prior to Year 7 will be stockpiled and rehandled to the crusher once the circuit is operational.
- Historical Tailings – Historical tailings are scheduled to be processed during the first four years of mill operations to allow for the advancing construction of the TSF Buttress and because the tailings add Project value without displacing ROM ore.
- Limestone – Limestone from the Middle Marble geologic formation will be mined and used directly as crushed limestone or processed in a lime kiln to provide the lime necessary to increase the pH of solutions and slurries as needed for processing sulfide ore.

The process plant, at full production capacity, is designed to process 8.05 million tons per year of ROM ore via the crusher and an additional 0.916 million tons per year of historical tailings. Process plant ROM ramp-up to full production is scheduled to occur during the first 3 years of operation and Historical Tailings ramp-up occurs during the first year of operation as shown on Figure 16.6. The ore processing schedule for mineralized material by ore type is shown on Figure 16.7.

Figure 16.6: Process Plant Throughput Ramp-Up Schedule

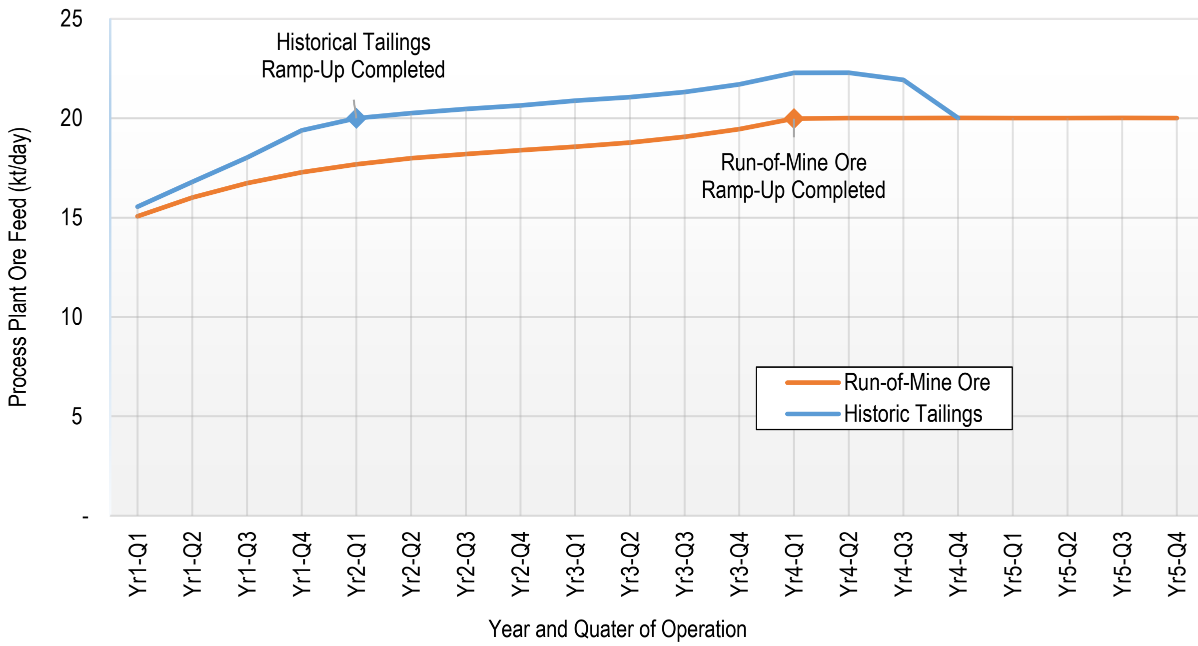
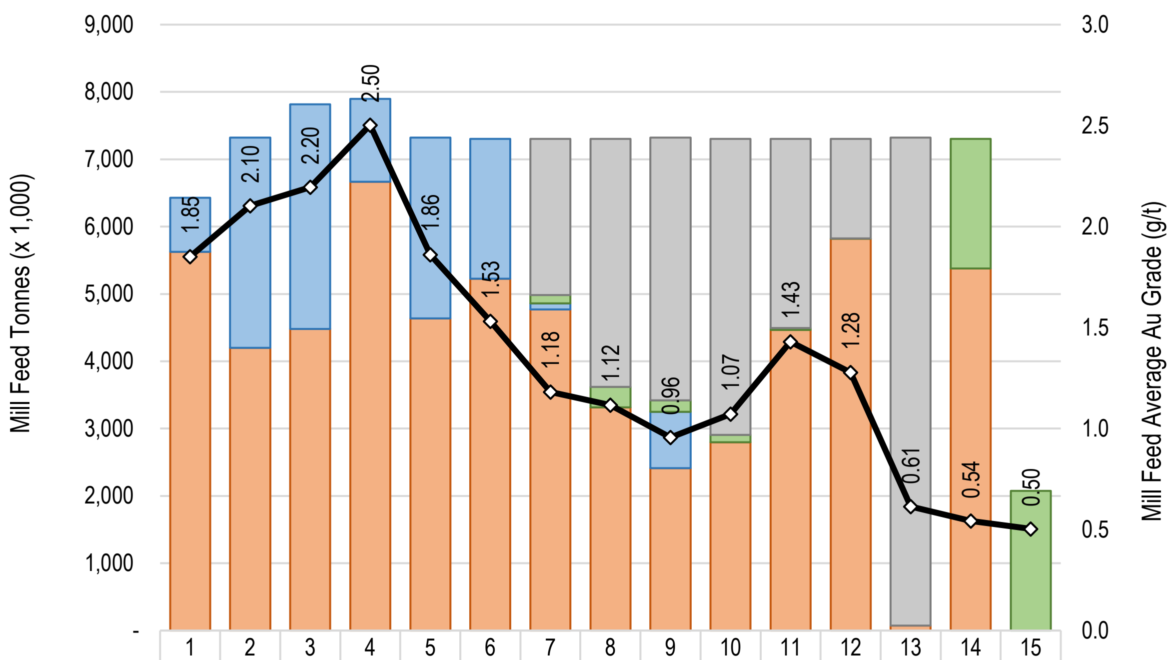


Figure 16.7: Process Plant Throughput Schedule by Ore Type, Year, and Average Grade



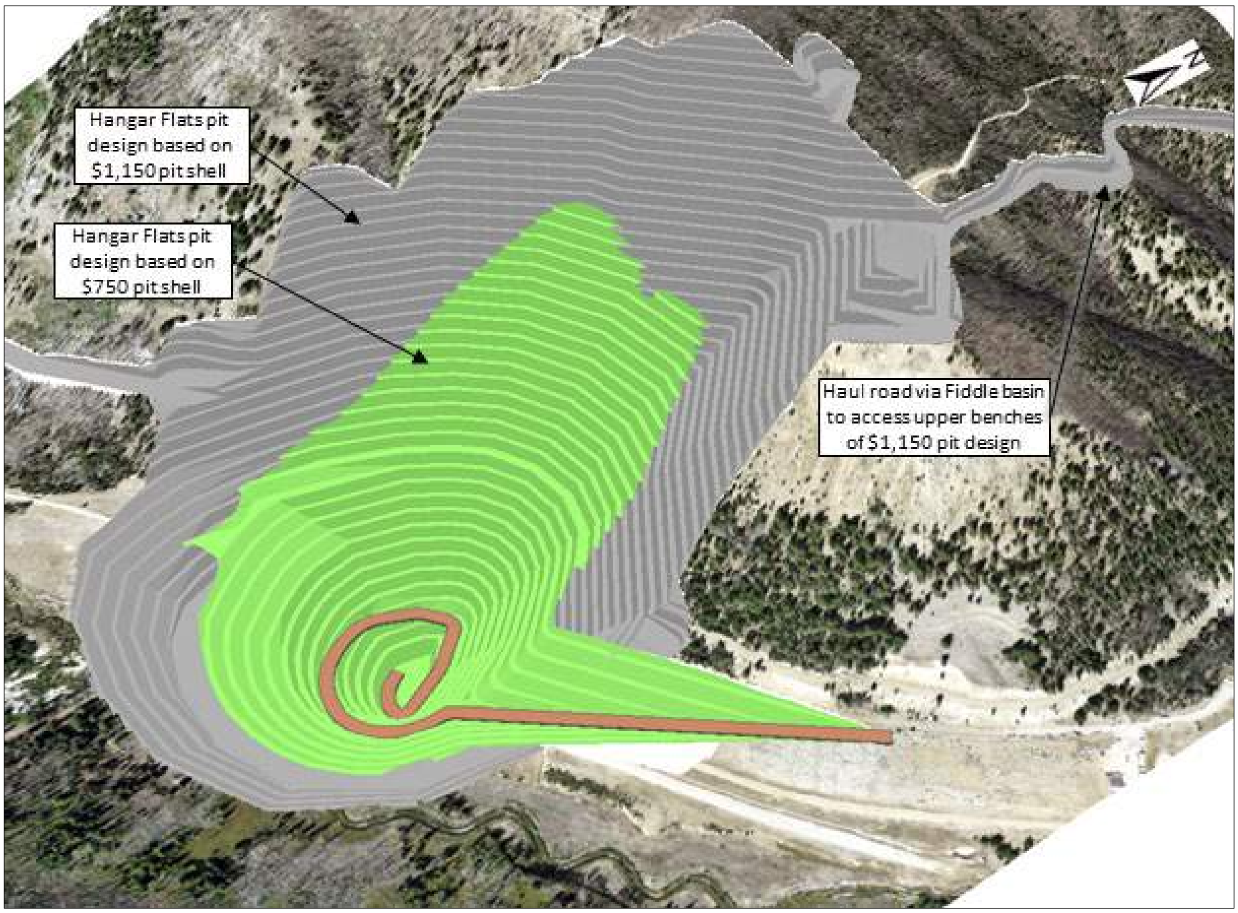
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Transition	-	-	-	-	-	-	2,319	3,682	3,905	4,394	2,810	1,479	7,251	-	-
Oxide	-	-	-	-	-	-	125	307	167	113	28	6	-	1,926	2,076
High Sb	803	3,120	3,337	1,230	2,685	2,079	88	-	840	-	-	-	-	-	-
Low Sb	5,626	4,201	4,479	6,665	4,638	5,224	4,770	3,314	2,411	2,796	4,464	5,818	72	5,377	-
Avg. Au (g/t)	1.85	2.10	2.20	2.50	1.86	1.53	1.18	1.12	0.96	1.07	1.43	1.28	0.61	0.54	0.50

16.3.2 Alternative Pit Geometry Evaluation

Alternative pit geometries based on pit shells may warrant evaluation in the mine sequence analysis if the nested pit shells for a deposit do not clearly identify the most suitable shell to use as guidance for the ultimate pit design. This can provide additional information beneficial to selecting the appropriate pit shell to be used in the ultimate pit design. Hangar Flats was the only deposit identified as having potential for higher value and less risk by evaluating pit designs outside the range of pit shells identified as optimal from the Ultimate Pit Limit Analysis.

Several Hangar Flats pit designs were evaluated, including a single-phase pit based on the $1,150/oz Au pit shell, a small single-phase pit based on the $750/oz Au pit shell, and a phased design incorporating both pit shells as shown on Figure 16.8. The single-phase design based on the $750/oz Au pit shell was selected to reduce costly access to upper benches, lower strip ratio, reduce project footprint, reduce the quantity of development rock generated and therefore the size of the DRSFs, allow elimination of the Fiddle DRSF, reduce closure cost, and reduce potentially detrimental effects on sitewide water management.

Figure 16.8: Hangar Flats Pit Geometry Alternatives ($750/oz Au Pit Selected)



16.3.3 Mine Production Rates

Evaluating mine production rates is essential to determine the duration of the mine life, duration of the process plant life (dependent on stockpile capacity), ore access schedule, and mining equipment fleet requirements. Mine production rate determination objectives included:

- balancing ore and development mining to maintain optimal process plant ore feed;
- accessing higher value ore earlier in the schedule while minimizing stockpile development and/or excessively elevating cut-off values;
- deferring development mining cost;
- minimizing stockpile rehandle cost;
- supporting concurrent pit backfilling and thereby accelerating concurrent reclamation and restoration;
- deferring equipment purchase capital cost;
- minimizing equipment capital and operating cost;
- scheduling a gradual fleet size ramp-up at start of operations;
- avoiding production fluctuations to maintain consistent staffing levels; and,
- providing adaptability in mine plan execution.

A suite of scenarios combining incremental production rates ranging from 28 to 48 million tons per year, Hangar Flats pit design alternatives, and variable production ramp-up schedules were evaluated to meet the objectives listed above. An approximate mine production rate of 34 million tons per year was selected based on stated objectives and Project value estimates. This production rate is substantially lower than the 42 million tons per year used in the PFS primarily due to reduced waste stripping requirements in the Hangar Flats pit, incorporating long-term stockpiles, and lower overall cut-off values.

16.3.4 Mine Production Fleet Equipment Selection

The SGP mine production fleet is typical for an open pit hard rock mine consisting of loading equipment (i.e. hydraulic shovels and wheel loaders), haul trucks, blast hole drills, and large dozers. The selected production fleet is the basis for mine production rates, detailed mine production schedules, and subsequent cost schedules.

Haul truck selection considerations included mine production rate, haul distance and profile, maneuverability, and fleet versatility to service multiple concurrent loading areas. Four haul truck size classes were considered in the production equipment fleet alternative analysis: 100-ton, 150-ton, 200-ton, and 250-ton. Based on a mine production rate of 34 million tons per year and an average round-trip haul distance of 6 miles, the number of trucks required for haul fleets consisting of 100-ton, 150-ton, 200-ton, and 250-ton trucks would be 24, 16, 12, and 10; respectively.

The 100-ton class haul truck was considered due to the maneuverability and versatility well suited for developing haul roads and operating productively on the narrow benches expected during open pit development. Although the 100-ton class haul truck could be effective for mine development work, they would be inefficient for production mining in the open pits once roads are established and initial benches developed. Therefore, the 100-ton class haul truck was eliminated from further evaluation and a separate development fleet was chosen to perform mine development, concurrent reclamation, and construction projects as described in Section 16.3.5.

The 250-ton class haul truck fleet size was also rejected for further analysis due to the estimated production inefficiency resulting from allocating a fleet of only 10 trucks to three concurrent loading areas (e.g. Yellow Pine open pit, Hangar Flats open pit, and a stockpile). A haulage simulation comparing 150-ton and 200-ton class haul trucks identified 150-ton class haul trucks as the best alternative due to the greater flexibility to serve multiple loading units and increased productivity offsetting added labor cost.

Loading equipment selection considerations included production rate, bench height, hydraulic shovel versus wheel loader, mobility, material selectivity, haul truck compatibility, and operational workspace requirements.

Hydraulic shovels were selected as the primary pit production loading equipment instead of wheel loaders because:

- the three open pits are mined sequentially allowing for loading equipment to remain in each pit for long durations reducing the need for mobility;
- narrow benches in some portions of the open pits favor hydraulic shovels which require less operational workspace;
- hydraulic shovels typically have a shorter truck loading cycle time than wheel loaders which contributes to increased fleet productivity;
- hydraulic shovels have greater material selectivity which reduces potential ore dilution;
- equipment longevity and mechanical availability; and,
- optional configuration (i.e. backhoe or shovel) for safe and productive operation on varied bench heights.

Hydraulic shovels with either 22-yd^3 or 28-yd^3 buckets are well-suited to load 150-ton class haul trucks. The approximate number of bucket-passes calculated to load a 150-ton class haul truck by 22-yd^3 and 28-yd^3 bucket hydraulic shovels is 5 and 4; respectively. A loading simulation was performed to compare productivity between 22-yd^3 and 28-yd^3 bucket hydraulic shovels including different material types and loading conditions anticipated throughout the mine life. The simulation projected a reduction in loading time of approximately 18,000 hours over the LOM for the 28-yd^3 bucket hydraulic shovel as compared to the 22-yd^3 bucket. Although the capital cost of the larger 28-yd^3 bucket hydraulic shovel is more than the 22-yd^3, the improved loading productivity and potential reduction in truck wait-time contributes to better Project economics. Two 28-yd^3 bucket hydraulic shovels were selected as the primary loading equipment matched to a fleet of 150-ton class haul trucks. One of the hydraulic shovels would be configured as a face shovel and the other as a backhoe to increase loading flexibility depending on bench height and workspace conditions. In addition to the two hydraulic shovels, a 28-yd^3 wheel loader is included in the production fleet to support loading during hydraulic shovel maintenance and loading stockpiled ore from various locations throughout the mine site as needed.

Rotary blasthole drills will be used for pit production drilling. Drills were selected primarily based on the ability to single-pass drill to a depth required for a 40-foot bench and drill hole diameter ranging from $6^1/_2$ inches to $10^5/_8$ inches. An average of five production drills with approximately 70,000-pound pulldown force are included in the production fleet as further detailed in Section 16.8.3.

Large dozers will be required to support hydraulic shovels and maintain development rock storage facilities. An average of five concurrently operating 600 horse-power dozers are included in the production fleet as further detailed in Section 16.8.4.

16.3.5 Mine Development Fleet Equipment Selection

The development fleet for the SGP is defined as the primary mining equipment used to construct haul roads, develop initial benches for production fleet mining, mine in-pit locations too confined for the production fleet, support various projects (e.g. TSF rind fills, water management ponds), and support concurrent reclamation. The development fleet is effectively a smaller version of the production fleet consisting of articulated haul trucks, excavators, loaders, surface drills, and medium size dozers.

16.3.6 Auxiliary, Maintenance, and Administrative Equipment Fleets

The additional equipment required to support the mine production fleet and mine development fleet are split into the following three fleets:

- Auxiliary Fleet – equipment primarily used to support production fleet;
- Maintenance Fleet – equipment used by maintenance department; and,
- Administrative Fleet – equipment used primarily by mine management departments.

A summary of mining equipment is listed by fleet in Table 16.2.

Table 16.2: Summary of Mining Equipment by Fleet

Equipment Type	Equipment Class	Approximate Number of Operating Units
Mine Production Fleet		
Shovel	28 yd^3	2
Large Wheel Loader	28 yd^3	1
Haul Truck	150 ton	16
Production Blasthole Drill	50 ft single pass, 70k lb pulldown	5
Large Dozer	600 Hp	5
Mine Development Fleet		
Excavator	5 yd^3	2-3
Wheel Loader	8 yd^3	2-3
Articulated Truck	45-ton ADT	8
Track Mounted Drill	3.5 – 5.0-inch diameter hole	2
Medium Dozer	215 Hp	2
Auxiliary Fleet		
Motor Grader	18 ft blade, 300 Hp	2
Motor Grader	14 ft blade, 240 Hp	1
Water Truck	9k gallon, 45 ton ADT	2
ANFO Truck	8 ton ANFO capacity	1
Stemming Truck	15 yd^3	1
Rock Spreader	100-ton capacity	1
Lowboy Trailer	100-ton capacity	1
Light Tower	20 kW, 29 ft extension	6
Maintenance Fleet		
Fuel & Lube Truck	45-ton ADT chassis	2
Mechanics Truck	35k lb chassis	2
Tire Service Truck	58/85-57 tire capacity	1
Flatbed Truck	Class 6 chassis	1
Forklift	6,000 lb lift capacity	1
Telehandler	11,000 lb lift capacity	1
Administrative Fleet		
Pickup Truck (4x4)	4x4 diesel crew cab	18
Man Van (4x4)	12-person capacity	4
Mine Radio	n/a	130
Dispatch System	High precision GPS on production fleet	n/a
Survey Equipment	Various	n/a
Mining Training Simulator	n/a	1

16.3.7 Strategic Mine Plan

The product of the mine sequence analysis is a strategic mine plan that defines the sequence of mining best suited to meet the objectives listed in the beginning of Section 16.3 and project-specific criteria including:

- backfill Yellow Pine open pit to support concurrent restoration of the original gradient of the EFSFSR;
- concurrent backfill Hangar Flats open pit to approximate the original valley elevation and gradient;
- concurrent backfill the Midnight area within the West End open pit;
- avoid concurrent mining of Yellow Pine and Hangar Flats open pit below valley elevation to reduce overlapping water management requirements;
- access the Middle Marble formation in West End early and stockpiling limestone prior to processing ore;
- construct growth medium stockpile bases from suitable in-pit glacial till; and,
- deliver material required for TSF construction and other construction related projects.

The strategic mine plan is used to evaluate stockpile strategy, DRSF construction sequencing, mill feed optimization, and guide the development of a mine production schedule.

To develop the strategic mine plan, each pit phase was split into cuts and assigned a mining fleet and production rate based on the fleet and type of mining activity. An example set of cuts for the Yellow Open pit is shown in Table 16.3. This methodology facilitated evaluating multiple mining sequences, pit geometries, equipment alternatives, and production rates with appreciable detail to determine the most favorable strategic mine plan. Each scenario included expected production delays due to road construction, bench operating limitations, drilling and blasting for bench access, periods of excessive average haul distance, and common factors such as equipment mechanical availability. The most favorable mine plan consisted of a Hangar Flats pit design based on the $750/oz Au pit shell, a production fleet based on 28-yd^3 hydraulic shovels matched to 150-ton class haul trucks, a development fleet based on 45-ton class articulated trucks, and a general mining sequence as shown on Figure 16.9. Material mined by deposit and year is shown on Figure 16.10. Ore mined by deposit and ore type is shown on Figure 16.11.

Table 16.3: Yellow Pine Open Pit Cut List

Phase	Cut ID	Cut Name	Fleet	Activity	Production Rate (st/hr)	Cut (kst)
YP Main	1	East ADT Road	Development	Road Construction	272	390
YP Main	2	East Cut ADT Starter	Development	Starter Bench	354	295
YP Main	3	East Cut ADT Production	Development	Production Mining	459	3,068
YP Main	4	East Cut Production Starter	Production	Starter Bench	1,179	3,965
YP Main	5	West ADT Road	Development	Road Construction	272	598
YP Main	6	West Cut ADT Starter	Development	Starter Bench	354	236
YP Main	7	West Cut ADT Production	Development	Production Mining	459	952
YP Main	8	West Cut Production Starter	Production	Starter Bench	1,179	1,015
YP Main	9	West Cut Production	Production	Production Mining	1,905	2,822
YP Main	10	West Ramp	Production	Truck Limited	1,542	1,117
YP Main	11	Stage 1 Ore Starter	Production	Starter Bench	1,179	1,360
YP Main	12	Stage 1 Ore Production	Production	Production Mining	1,905	12,145
YP Main	13	Stage 1 Waste Production	Production	Truck Limited	1,542	10,823
YP Main	14	Stage 2 Ore Production	Production	Production Mining	1,905	4,835
YP Main	15	Stage 2 Waste Production	Production	Production Mining	1,905	8,136
YP Main	16	Stage 3 Production	Production	Production Mining	1,905	5,903
YP Main	17	Stage 4 Waste Production	Production	Production Mining	1,905	10,487
YP Main	18	Stage 4 Ore Production	Production	Production Mining	1,905	8,349
YP Main	19	Stage 5 Ore Production	Production	Production Mining	1,905	14,666
YP Main	20	Stage 5 Waste Production	Production	Production Mining	1,905	33,764

Phase	Cut ID	Cut Name	Fleet	Activity	Production Rate (st/hr)	Cut (kst)
YP Main	21	Stage 6 Ore Production	Production	Production Mining	1,905	8,823
YP Main	22	Stage 6 Ore ADT Production	Development	Production Mining	459	1,339
Homestake	23	Homestake Waste Production	Production	Production Mining	1,905	15,033
Homestake	24	Homestake Ore Production	Production	Production Mining	1,905	9,683
Homestake	25	Homestake Ore ADT Production	Development	Production Mining	459	2,798

Figure 16.9: General Mining Sequence

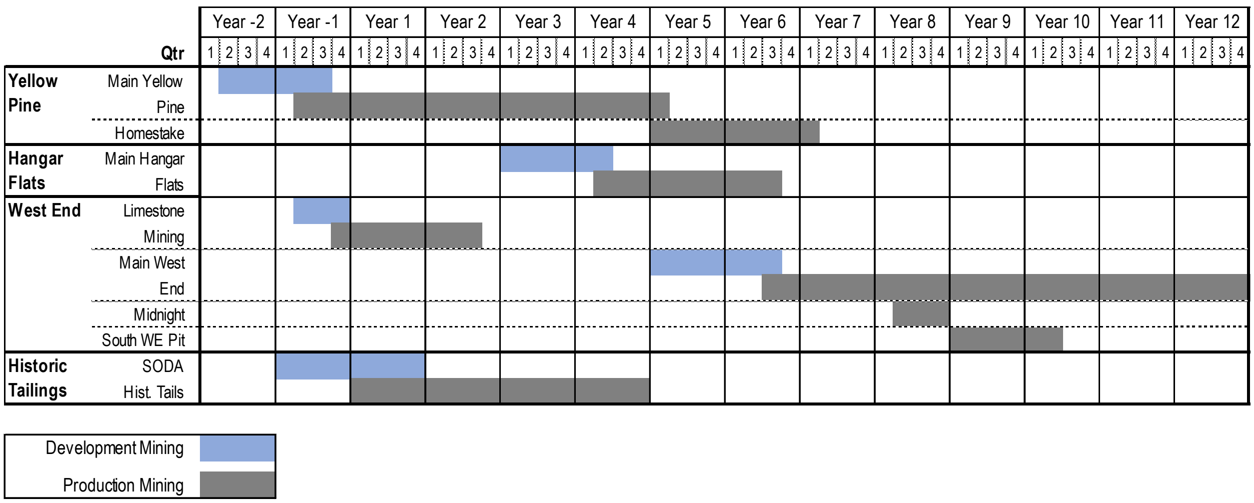
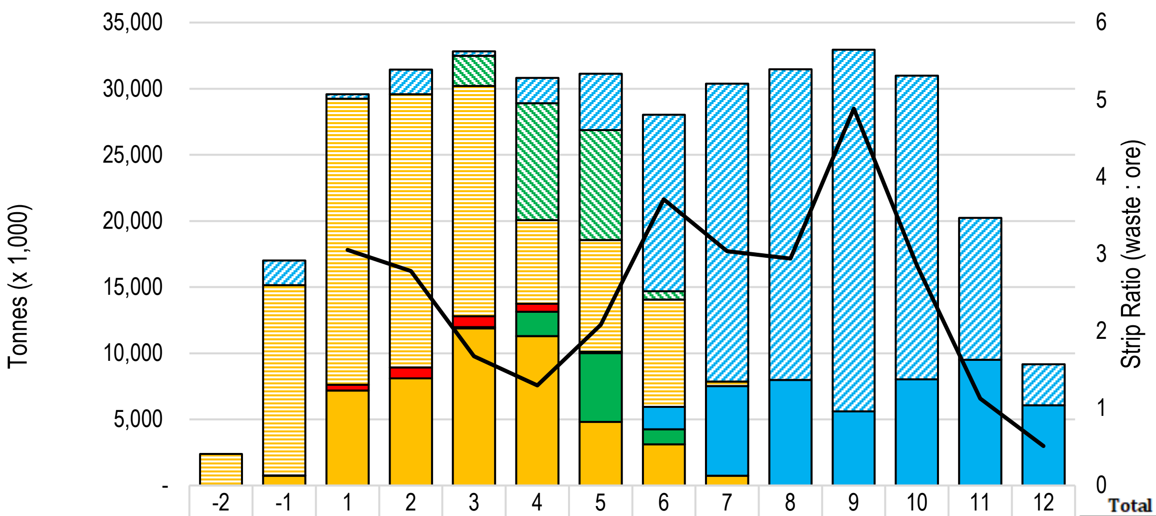


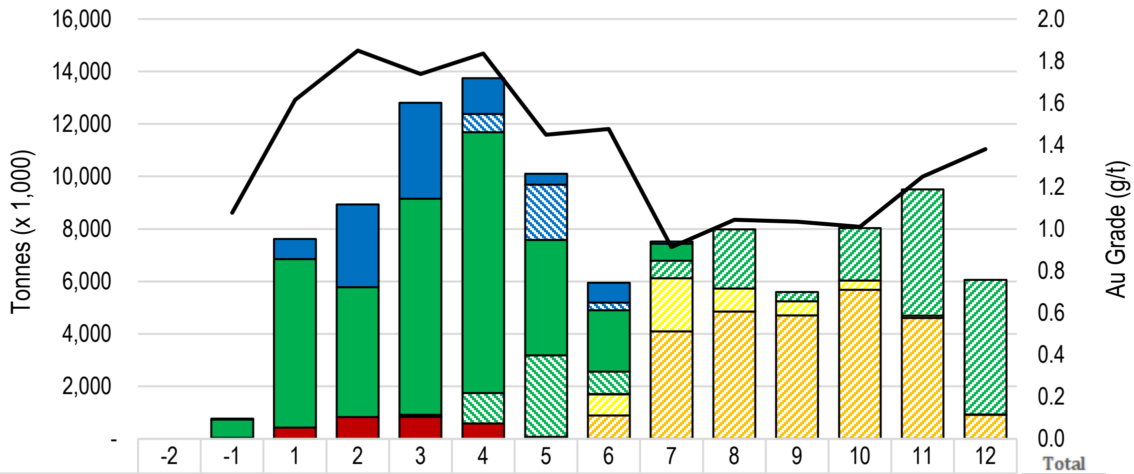
Figure 16.10: Ore and Development Rock Mined by Deposit and Year (000s tonnes)



	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
WE Dev. Rock	-	1,861	345	1,870	338	1,905	4,262	13,339	22,522	23,484	27,345	22,942	10,721	3,098	134,031
HF Dev. Rock	-	-	-	-	2,297	8,845	8,309	615	-	-	-	-	-	-	20,066
YP Dev. Rock	2,387	14,375	21,613	20,651	17,396	6,330	8,461	8,114	340	-	-	-	-	-	99,666
Hist. Tails.	-	-	432	831	831	592	-	-	-	-	-	-	-	-	2,687
WE Ore	-	36	-	-	31	-	79	1,713	6,790	7,979	5,599	8,035	9,509	6,058	45,830
HF Ore	-	-	-	-	57	1,849	5,219	1,138	-	-	-	-	-	-	8,262
YP Ore	-	737	7,188	8,102	11,883	11,299	4,804	3,100	734	-	-	-	-	-	47,847
Strip Ratio			3.05	2.78	1.67	1.30	2.08	3.71	3.04	2.94	4.88	2.86	1.13	0.51	2.49

Note: Values shown on Figure 16.10 are the result of the mine production schedule as presented in Section 16.8.

Figure 16.11: Ore Mined by Deposit, Ore Type, and Year (000s tonnes)



	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	Total
YP High Sb	-	33	769	3,154	3,649	1,358	416	761	91	-	-	-	-	-	10,232
HF High Sb	-	-	-	-	1	699	2,109	285	-	-	-	-	-	-	3,095
YP Low Sb	-	704	6,419	4,947	8,233	9,941	4,388	2,339	643	-	-	-	-	-	37,615
HF Low Sb	-	-	-	-	55	1,149	3,110	853	-	-	-	-	-	-	5,167
WE Low Sb	-	-	-	-	-	-	-	11	674	2,248	358	2,003	4,817	5,131	15,242
WE Oxide	-	22	-	-	16	-	27	804	2,015	879	538	355	88	6	4,749
WE Trans.	-	14	-	-	15	-	52	898	4,101	4,852	4,703	5,678	4,605	921	25,839
Hist. Tails.	-	-	432	831	831	592	-	-	-	-	-	-	-	-	2,687
Ore Au (gpt)		1.08	1.62	1.85	1.74	1.84	1.45	1.48	0.91	1.04	1.03	1.01	1.25	1.38	1.43

Note: Values shown on Figure 16.11 are the result of the mine production schedule as presented in Section 16.8.

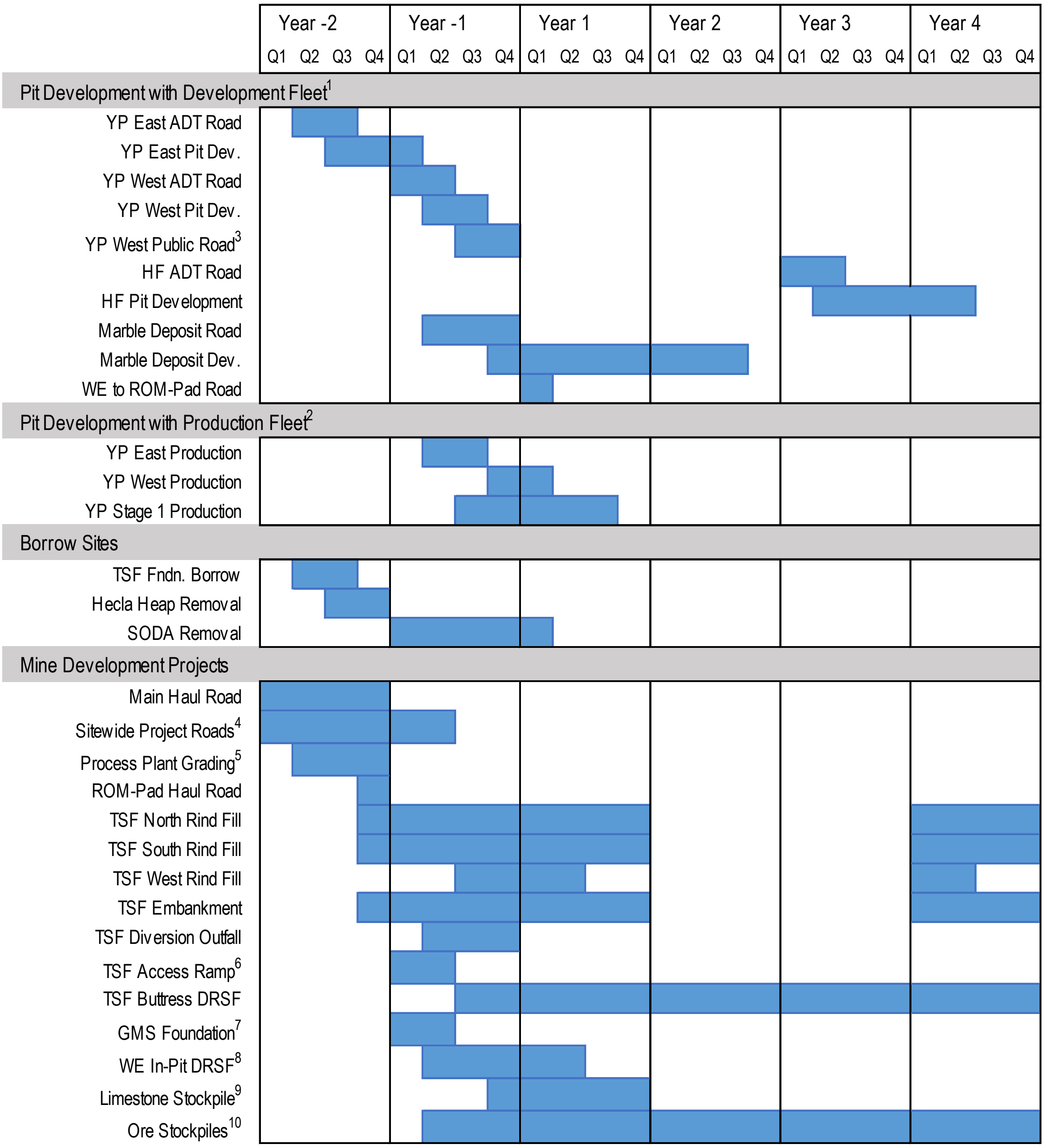
16.4 MINE DEVELOPMENT PLAN

The mine development plan consists of scheduling open pit development and sitewide construction activities that will be performed by the mining fleet equipment and staff. These activities include:

- constructing initial sitewide haul roads;
- constructing in-pit roads to access initial mine production working benches;
- pre-stripping and developing pit benches for the mine production fleet;
- mining upper benches within the Yellow Pine open pit as needed for the public access road;
- accessing and mining the Middle Marble formation to stockpile sufficient limestone prior to processing ore;
- mining, hauling, and placing fill material for TSF construction;
- supporting various sitewide construction activities; and,
- constructing growth media stockpile foundations.

The mine development plan was created using first principal calculations for drilling, blasting, loading, and hauling equipment requirements and activity scheduling. Example calculations are provided in Table 16.7, Section 16.8.2. This schedule was then incorporated into the equipment maintenance estimate, staffing estimate, and cost estimate. A summary of activities captured in the mine development plan are shown on Figure 16.12 and Figure 16.13.

Figure 16.12: Mine Development Plan Activity Schedule

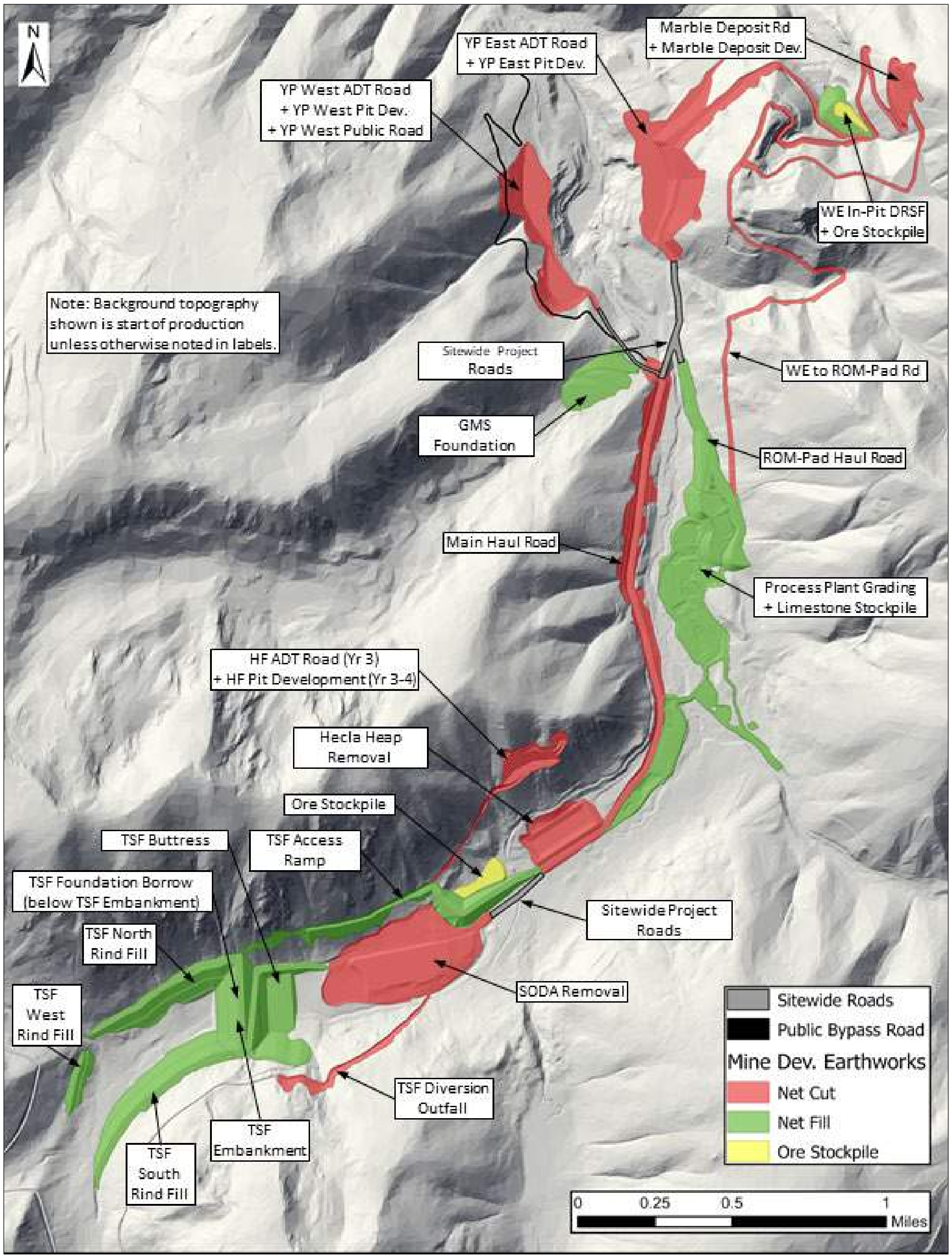


Notes:

(1) Pit Development with Development Fleet: Activities performed by the development fleet as described in Section 16.3.5.
(2) Pit Development with Production Fleet: Activities performed by the production fleet that produce material required for development projects.
(3) YP West Public Road: Mining at rim of Yellow Pine open pit to prepare for public bypass road construction.
(4) Sitewide Project Roads: Includes roads not explicitly captured in other line items that do not have substantial cut and fill imbalance.
(5) Process Plant Grading: Only fill material sourced from open pits is included, not cut to fill within the Process Plant area footprint.
(6) TSF Access Ramp: Includes the base of the main haul ramp to the TSF Embankment and access road for the tailings pipeline corridor.
(7) GMS Foundation: Growth Media Stockpile (GMS) foundational material sourced from the alluvial material in west side of the Yellow Pine open pit.

(8) *WE In-Pit DRSF: A temporary DRSF located entirely within the West End open pit footprint to reduce waste haulage requirements during development activities required to access the marble deposit and develop West End for production mining. It also serves as a foundation for stockpiling ore sporadically mined during development mining.*
(9) *Limestone Stockpile: Represents the required initial stockpile of limestone sourced from the marble deposit required at ore processing commencement. The stockpile is continually maintained during the LOM as needed for generating lime required for ore processing.*
(10) *Ore Stockpiles: Ore stockpiles are required at ore processing commencement and throughout the LOM.*

Figure 16.13: Mine Development Plan Activity Location Map



16.5 ORE STOCKPILE STRATEGY ANALYSIS

An improvement to the SGP FS mine plan as compared to the PFS is the addition of long-term ore stockpiles. The primary benefit to adding ore stockpile capacity is increased potential to optimize process ore feed value throughout the mine life, improve long term closure by processing lower grade ore that could otherwise become a source of metal leaching in the DRSFs, and support pit phasing and therefore concurrent backfilling and restoration activities. This is particularly significant during the first half of the mine life when Yellow Pine high value ore is mined at a rate greater than process plant throughput capacity. If stockpile capacity is not available, either the period-based cut-off value must increase resulting in ore converted to waste, or the mining rate reduced to align with process plant throughput capacity resulting in deferred access to high-value ore deeper in the open pit. The addition of long-term ore stockpiles allows for relatively high value ore mined from Yellow Pine open pit to be stockpiled and made available to process when lower value ore is being mined in West End open pit.

The principal objective of the ore stockpile strategy was to increase Project value by stockpiling ore with higher value than is available later in the mine plan. Additional objectives include:

- reducing peak mining rates particularly when pre-stripping West End and concurrently mining Hanging Flats and Yellow Pine open pits;
- stabilize mining rates by providing additional options to source ore for processing;
- provide operational ore blending and campaigning flexibility including deferral of oxide ore processing;
- support optimal utilization of the mineral resource while reducing low grade ore being sent to the DRSFs where it is more likely to be a source of metal leaching than once it is converted to tailings, metals extracted and neutralized and stored in a lined facility;
- reduce Project risk related to open pit ore production disruptions;
- extend process plant life while increasing Project value;
- increase Project value opportunity if metal sell prices increase; and,
- incorporate stockpile designs into DRSF layout to facilitate reclamation and minimize additional ground disturbance resulting from ore stockpiles.

The ore stockpile strategy analysis consisted of using the strategic mine plan and assigning each unit of material mined a value-based grade bin designation as shown for Yellow Pine in Table 16.4. An optimized mill feed schedule including stockpile rehandle cost was then created assuming unlimited stockpile capacity and segregation by grade bin and ore type (i.e. ten grade bins for each of the four open pit ore types). This mill feed schedule represents a best-case scenario but is unachievable due to geographical constraints and being operationally impracticable. Using this schedule as a guide, multiple iterations of DRSF design, DRSF sequencing, and stockpile design were evaluated to proximate the best-case scenario as described in the Section 16.6.

Table 16.4: Yellow Pine Ore Grade Bins

Ore Grade Bin	NPR Cutoff ($/st)	Low Sb (kst)	High Sb (kst)	Average Low Sb Au (gpt)	Average High Sb Au (gpt)
1	-	1,703	95	0.28	0.25
2	$2.50	2,973	119	0.34	0.29
3	$5.00	2,664	132	0.40	0.27
4	$10.00	4,570	270	0.49	0.38
5	$20.00	6,953	564	0.66	0.51
6	$30.00	5,486	675	0.90	0.70

7	$40.00	4,046	652	1.14	0.91
8	$60.00	6,105	1,256	1.49	1.26
9	$100.00	9,285	3,691	2.17	1.97
10	> $100	7,225	4,291	3.54	2.97
Note: Table calculations include Proven and Probable ore only.					

16.6 DRSF AND STOCKPILE ANALYSIS

The DRSF and stockpile analysis was an iterative process of designing and sequencing both DRSFs and ore stockpiles in combination to augment project value by advancing higher value ore feed to the mill and abate operating costs associated with haulage and stockpile rehandle. The outcome of this analysis is DRSF designs, DRSF construction sequence, ore stockpile designs and calculated ore type and grade for use in the mill feed optimization. Significant changes to DRSFs and ore stockpiling in the FS as compared to the PFS include:

- eliminating the West End DRSF to reduce Project disturbance area and potential impacts on water quality;
- adding a small interim DRSF and ore stockpile within the West End open pit footprint to receive waste and ore during pit development to reduce haulage requirements;
- eliminating the Fiddle DRSF to reduce Project disturbance area and potential for water quality degradation;
- backfilling the Hangar Flats open pit to restore the area to pre-existing conditions, create wetlands and create short term ore stockpile capacity;
- adding the Scout ROM Stockpile near the plant site to increase stockpile capacity with a short haul distance to the crusher; and,
- adding several long-term ore stockpiles on the TSF Buttress and within the Hangar Flats pit footprint to handle ore that would otherwise be sent to DRSFs.

Development rock from the three open pits is planned to be sent to five different permanent destinations over the mine life consisting of: the TSF embankment and rind fills; the TSF Buttress; the mined-out Yellow Pine open pit; the mined-out Hangar Flats open pit; and the Midnight area within the mined-out West End open pit. In addition to these five areas, other destinations will receive development rock from the three open pits including a temporary ore stockpile base within the West End open pit, a foundation for stockpiling growth medium and recovered seed bank material, a reclamation materials stockpile located on the TSF Buttress, and miscellaneous projects such as road fills and ore stockpile foundations.

Ore from the three open pits is planned to be delivered to either the crusher as direct feed for processing, short-term stockpiles located on the ROM pad, or long-term stockpiles located primarily on the TSF Buttress and Hangar Flats open pit backfill. The locations of waste and ore destinations are shown on Figure 16.14. The waste destination schedule is shown on Figure 16.15.

Figure 16.14: DRSF and Stockpile Locations

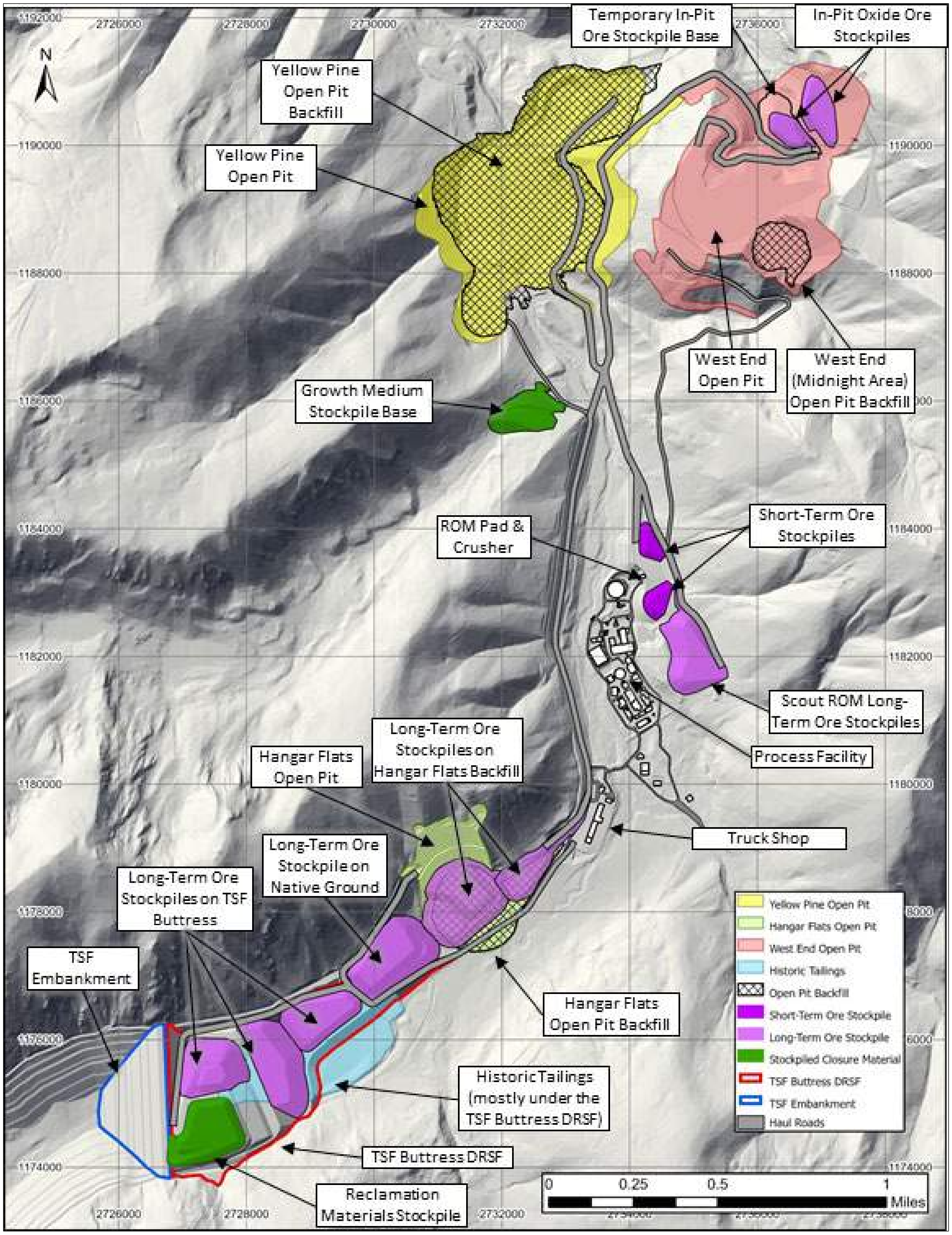
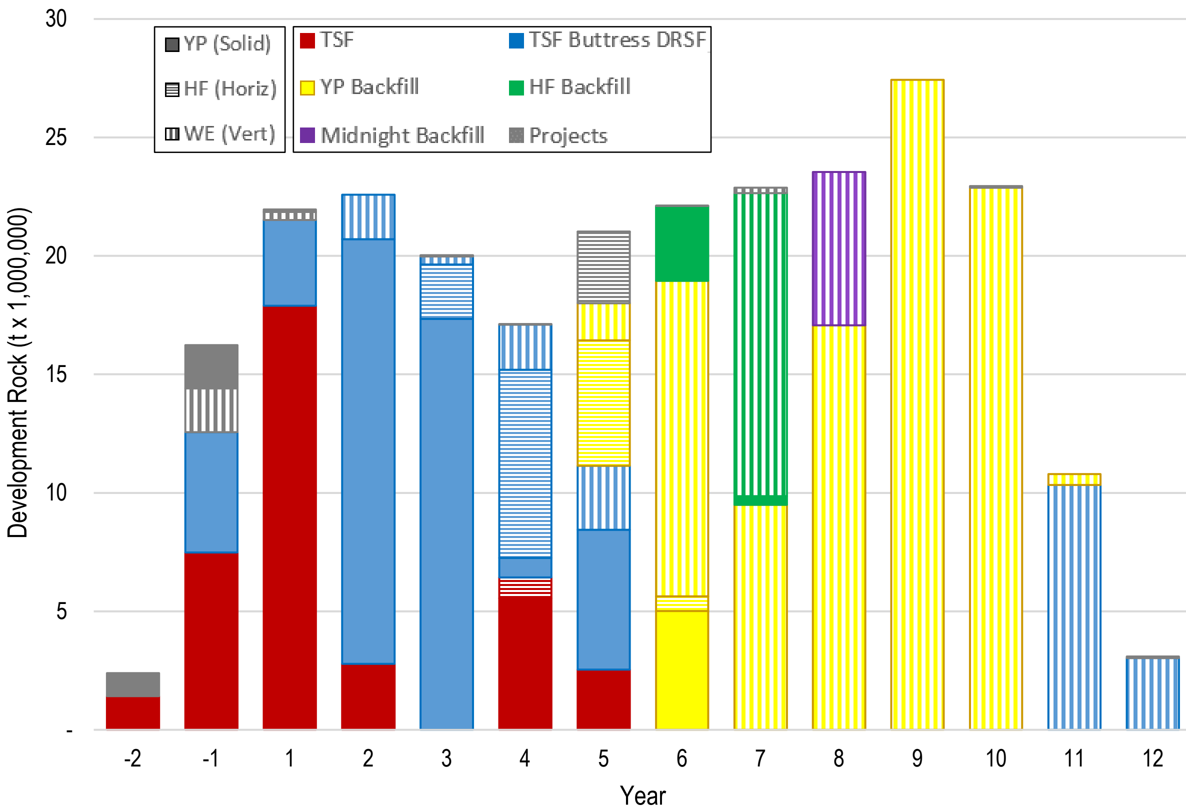


Figure 16.15: Development Rock Destination by Pit and Year



16.7 MILL FEED OPTIMIZATION

A mill feed optimization was conducted using the strategic mine plan and stockpile schedule to ensure the highest value ore available is processed and to create the final mill feed schedule. The optimization consisted of scheduling ore routing from pit-to-mill, pit-to-stockpile, and stockpile-to-mill on a monthly period until the end of Year 2 and then on a quarterly period for the remainder of the process plant life. This methodology was applied to identify suitable timing for constructing the oxide ore processing circuit and calculating ore load and haul requirements for input into the mine production schedule analysis. The final mill feed schedule is the basis for reporting Mineral Reserve Estimates as provided in Section 15.

Opportunity to increase Project value during the mill feed optimization was primarily driven by maximizing stockpile ore value available for process during periods when in-pit ore is lower in value than stockpiled ore. It was an iterative process of scheduling variable stockpile cut-off values by ore type while considering incremental cost between ore directly fed to process from an open pit versus rehandling ore from stockpiles to process. The outcome of this process defined each stockpile ore quantity, ore type, cut-off value, average value and grade, and stockpile duration. Ore processed by year and source is shown on Figure 16.16. Long-term ore stockpile inventory and progression is shown on Figure 16.17 and Figure 16.18; respectively.

Figure 16.16: Ore Processed by Year and Source

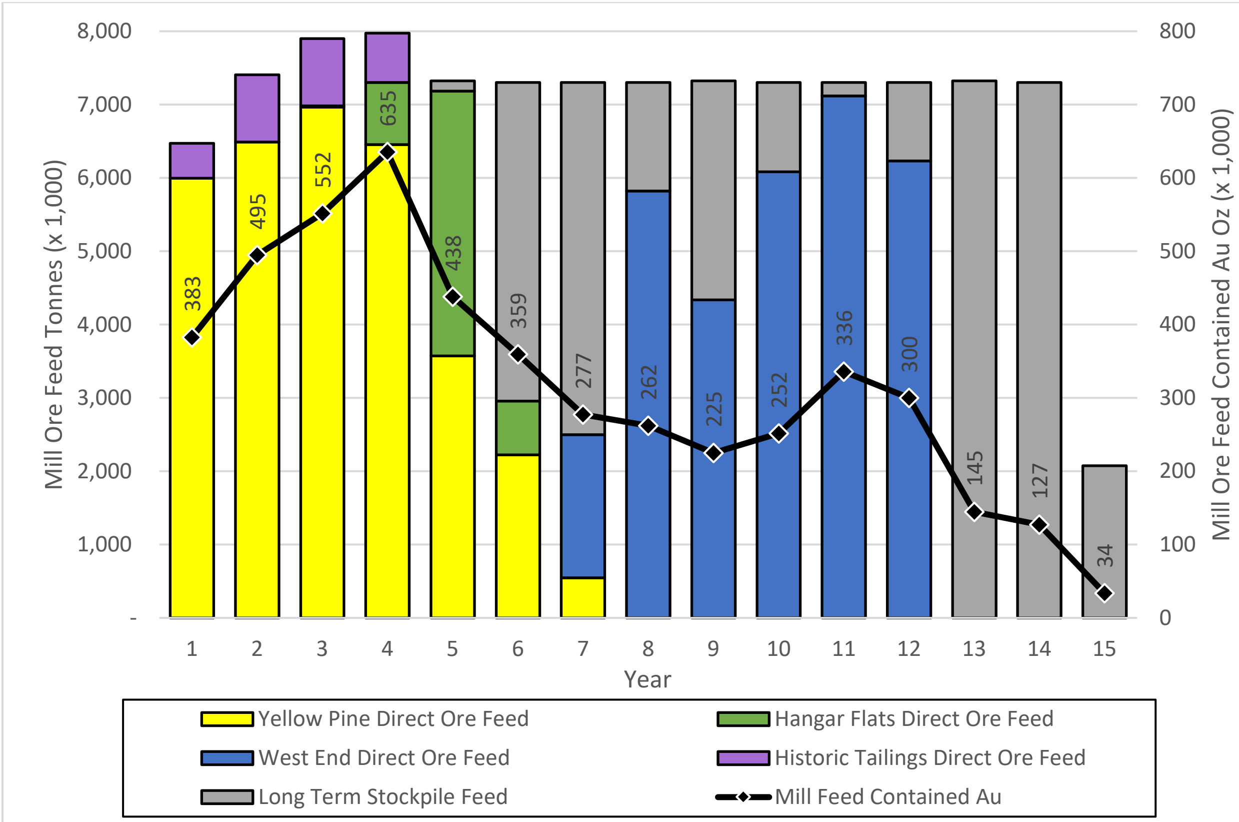
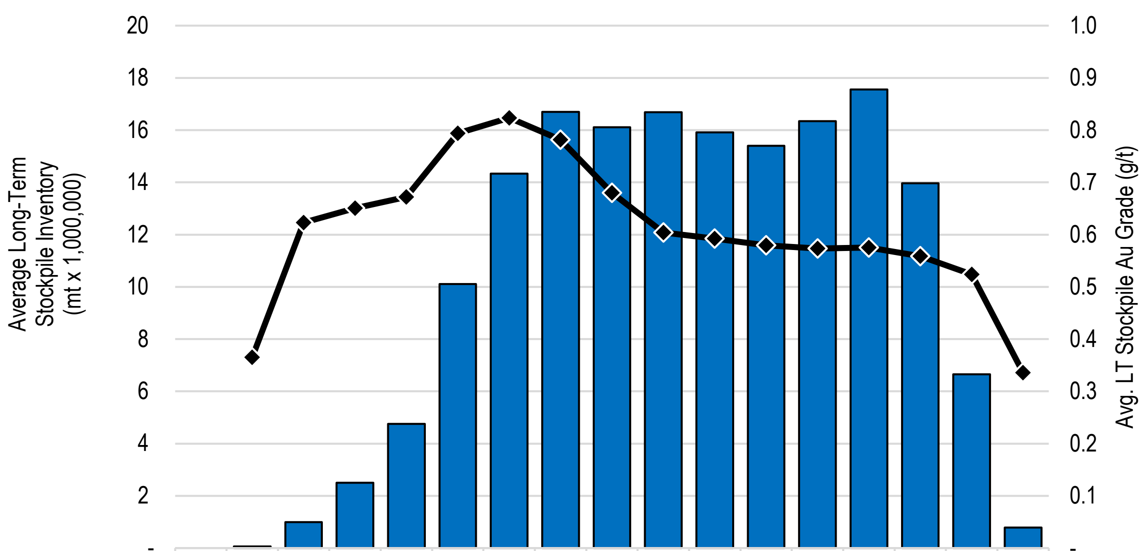
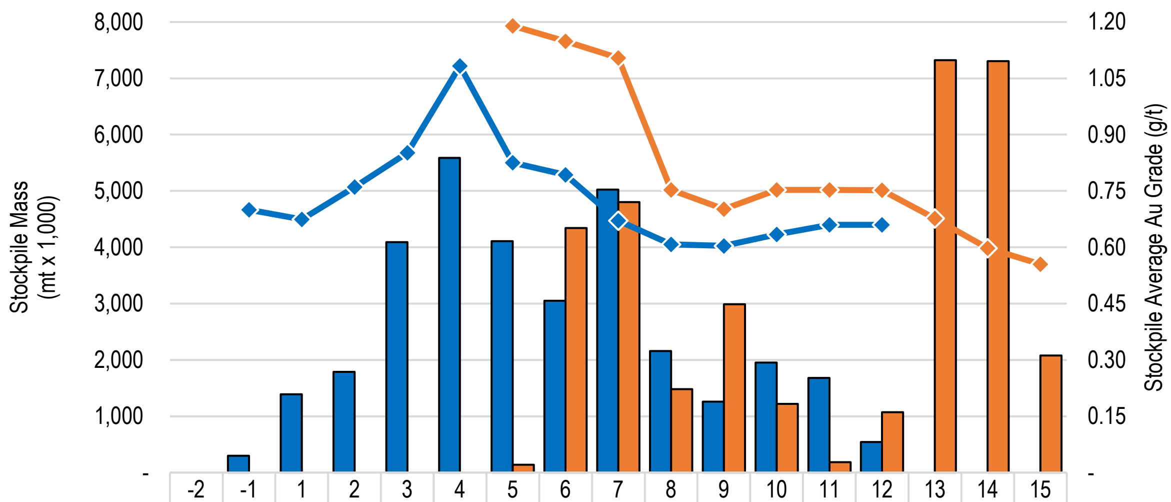


Figure 16.17: Long-Term Stockpiles Total Inventory



	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Inventory (st)	-	0.1	1.0	2.5	4.8	10.1	14.3	16.7	16.1	16.7	15.9	15.4	16.3	17.6	14.0	6.7	0.8
Au (gpt)		0.36	0.62	0.65	0.67	0.79	0.82	0.78	0.68	0.60	0.59	0.58	0.57	0.58	0.56	0.52	0.34

Figure 16.18: Long-Term Stockpiles Progression



	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
In Tons	-	296	1,38	1,78	4,09	5,59	4,11	3,05	5,02	2,15	1,26	1,95	1,67	540	-	-	-
Out Tons	-	-	-	-	-	-	139	4,34	4,80	1,48	2,98	1,21	186	1,07	7,32	7,30	2,07
In Au		0.70	0.67	0.76	0.85	1.08	0.83	0.79	0.67	0.61	0.60	0.63	0.66	0.66			
Out Au							1.19	1.15	1.10	0.75	0.70	0.75	0.75	0.75	0.68	0.60	0.56

Notes:
(1) Higher stockpile grade output during years 5-7 as compared to years -1-4 is achieved by grade segregation within the stockpile facility during early years.

16.8 MINE PRODUCTION SCHEDULE ANALYSIS

The mine production schedule analysis consisted of creating a detailed period based mine schedule derived from the strategic mine plan, mine development schedule, mill feed schedule, and DRSF and stockpile schedule. It is an aggregation of these schedules into a single schedule with the addition of equipment requirement calculations to generate the final mine production schedule used to estimate equipment requirements, equipment purchase schedule, and the mining operating expenditure schedule.

16.8.1 Work Schedule

Mining is scheduled for 365 days per year and 2 shifts per day of 12 hours duration each. A summary of equipment operator working time and delays are provided in Table 16.5.

Table 16.5: Summary of Equipment Operator Working Time

Shift and Rotation Duration	Stated Period
Calendar Hours (Hrs per Year)	8,760
Shift Count (Shifts per Day)	2
Shift Count (Shift per Year)	730
Shift Duration (Hrs per Day)	12
Rotation Duration (Days per Rotation)	14
Rotation Count (Rotations per Year)	26

Working Time Delays	Stated Period	Per Year (hours)
Weather Delay (Hrs per Year)	240	240
Lunch Break (Hrs per Shift)	1.00	730
Morning Break (Hrs per Shift)	0.25	183
Afternoon Break (Hrs per Shift)	0.25	183
Safety Meeting (Hrs per Rotation)	2.00	52
Shift Change Delay (Hrs per Rotation)	2.00	52
Total Delay (Hrs per Shift)	1.97	1,440

16.8.2 Load and Haul

Mine production loading is planned predominantly with 28-yd^3 hydraulic shovels supported by a 28-yd^3 wheel loader. Mine development loading is planned with 5-yd^3 excavators supported by 8-yd^3 wheel loaders. A summary of mining activity, loading equipment, hauling equipment, and drilling equipment is provided in Table 16.6.

Table 16.6: Mining Equipment by Mining Activity

Mining Activity	Loading Equipment	Hauling Equipment	Drilling Equipment	Dozer Support Equipment
Mine Access Road Construction	5-yd^3 Excavator	45-ton Articulated Truck	track mounted drill[4]	215 Hp Dozer
Mine Production Bench Development	8-yd^3 Wheel Loader	45-ton Articulated Truck	track mounted drill	215 Hp Dozer
Mine Production	28-yd^3 Hydraulic Shovel	150-ton Haul Truck	blasthole drill[5]	600 Hp Dozer
Pit Bottom Production[1]	8-yd^3 Wheel Loader	45-ton Articulated Truck	track mounted drill	215 Hp Dozer
South West End Pit Phase	8-yd^3 Wheel Loader	45-ton Articulated Truck	track mounted drill	215 Hp Dozer
Limestone Mining	8-yd^3 Wheel Loader	45-ton Articulated Truck	track mounted drill	215 Hp Dozer
Stockpile Rehandle	28-yd^3 Wheel Loader	150-ton Haul Truck	n/a	600 Hp Dozer
Construction Borrow	28-yd^3 Wheel Loader	150-ton Haul Truck	n/a	600 Hp Dozer
General Project Work	8-yd^3 Wheel Loader	45-ton Articulated Truck	track mounted drill	215 Hp Dozer
Concurrent Reclamation[2]	5-yd^3 Excavator	45-ton Articulated Truck	n/a	600 Hp Dozer
Closure Reclamation[3]	28-yd^3 Wheel Loader 5-yd^3 Excavator	150-ton Haul Truck 45-ton Articulated Truck	n/a	600 Hp Dozer 215 Hp Dozer

Notes:
1) Pit Bottom Production: The bottom benches of all three open pits are planned to be mined with the development fleet to access high-value ore inaccessible to the larger production fleet.
2) Concurrent Reclamation: Some concurrent reclamation will be performed using the 28-yd^3 wheel loader and 150-ton haul trucks dependent on project suitability and equipment availability.
3) Closure Reclamation: Large-scale reclamation projects are planned for production fleet concurrent with development fleet.
4) 3.5 - 5.0-inch diameter hole track mounted drill.
5) $6^1/_2$ - $10^5/_8$-inch diameter, 50 ft single-pass 70k lb pulldown blasthole drill.

Loading and hauling calculations for the mine production schedule consisted of pairing loading equipment to hauling equipment by fleet and mining task to estimate production rates. Production rates were calculated on first principal assumptions including: bucket capacity; truck bed capacity; material densities; fill factor; cycle time; truck spot time; face cleanup delay; mechanical availability over the machine life; usage, tramming; haul profiles; expected haul delays; and sitewide speed limits. Prior to estimating production rates all haulage routes for each source-to-destination were delineated and a suite of approximately 600 haulage routes were simulated to estimate travel load time, return time, truck bunching delay, and truck wait-to-load based on various loading equipment, hauling equipment, and fleet size. Sample calculations for the mine production fleet are shown in Table 16.7.

Table 16.7: Sample Load and Haul Productivity Calculations

Production Metric	Value	Comment
Period, Source, and Destination Details		
Period	Year -1 Month 7	Used to match delineated haul routes
Source Pit	Yellow Pine	
Source Cut	4 - East Starter	
Source Bench	6,460	
Bench Count	1	Used to estimate bench turnover delay
Destination	TSF.P1.Emb	TSF Phase 1 Embankment
Load and Haul Tons and Density		
Schedule Tons	236,420	
Bank Density (SG)	2.57	
Blasted Density (SG)	2.06	
Load and Haul Fleet Selection		
Fleet	Production 01	Mine plan consists of two production fleets, each with a single hydraulic shovel
Loading Unit Class	28 yd^3 FS	Hydraulic front shovel (FS) production rate is different than backhoe configuration
Loading Unit Count	1	
Hauling Unit Class	150 ton Truck	
Hauling Unit Count	10	
Haul Route Details and Haul Time Calculations		
Road ID	Yr-1.YP04.P1EM	Approximately 270 LOM haul routes digitized from bench exit point to destination bench entrance point
Off Bench Distance (ft)	202	Based on weighted average distance from bench exit to bench perimeter (not centroid)
Off Bench Slope (%)	10%	
Total 1-Way Haul Dist. (ft)	19,325	
Round-Trip Haul Dist. (mi.)	7.32	
Average Haul Time (min)	12.80	Calculated based on haulage simulations
Average Return Time (min)	7.97	Calculated based on haulage simulations
Round-Trip Speed (mph)	21.1	
Load and Haul Delay Entry		
2-Sided Truck Loading?	No	If mining on large open bench, double-sided loading assumed and loader truck spot reduced
Add Haul Truck Exchange?	Yes	Due to a confined "starter cut" truck exchange delay added to position truck for loading
Add Shovel Cleanup?	Yes	Starter cuts are likely to require a delay for shovel to cleanup working face and haul truck access
Bench Turnover Delay (hrs)	24	Delay per bench added to account for drill, blast, and bench preparation prior to mining new bench
Route Add Time (min)	2.00	Est. haul delay due to crossing other active haul roads and/or routing through active construction
Loader Productivity		
Effective Bucket Cap (yd^3)	25.92	Effective bucket capacity based on heaped bucket capacity and bucket fill factor
Bucket Tons per Pass	44.91	Calculated from effective bucket capacity and blasted density
Loader Full Passes	3	
Last Bucket Partial Fill (st)	18.56	
Last Bucket Fill Min. (%)	25%	Min % of last bucket fill to approximate whether a partially filled bucket pass is applied to top-off truck
Loader Cycle Time (min)	0.55	Based on published loader cycle times and field measurements
Spot Plus 1st Bucket (min)	0.80	Value based on whether double-sided truck loading and type of loading unit.
Loader Cleanup (min)	0.50	This value is zero for large open benches with dozer support

Production Metric	Value	Comment
Truck Productivity		
Truck Rated Payload (st)	146.0	Truck rated payload based on truck specifications
Truck Max Payload (st)	153.3	105% of rated payload. Used to estimate last loader bucket fill percent.
Truck Payload (st)	153.3	If last loader bucket is partial fill, assume truck is loaded to maximum payload. Note average payload for LOM is approximately 149 st.
Load With Exchange (min)	2.95	Includes loader full passes, partial pass if applicable, loader first bucket, and loader cleanup.
Dump & Maneuver (min)	1.00	Estimated maneuver plus dump time. Variable depending on truck class
Truck Bunching (min)	2.59	Estimated bunching delay based on truck fleet size and approximately 600 haulage simulations
Truck Exchange (min)	0.75	Included if 'Add Haul Truck Exchange?' is 'Yes'.
Truck Wait to Load (min)	-	Estimated wait time from same 600 haulage simulations used to estimate bunching and wait to load time
Total Truck Cycle Time (min)	30.07	Sum of Wait to Load, Load with Exchange, Avg Haul, Dump & Maneuver, Avg Return, Bunching, Truck Exchange, and Route Add Time
Truck Load Count	1,543	Count of total truck loads required to haul scheduled tons
Equipment Operating Hours		
Loader Operating Hours	88.7	(Truck Load Count) x [(Load with Exchange) + (Loader Cleanup)]
Operating Hours per Unit	88.7	Calculated on 1 loading unit in the fleet
Truck Operating Hours	773	(Truck Load Count) x (Cycle Time)
Operating Hours Per Truck	77.3	Calculated on 10 trucks in the fleet allocated to this loading unit.
Fleet Limiting Equipment	Loader	Calculated. Used to adjust number of truck or loader units to better match req. equipment op hours
Loader Hang Time (hrs)	-	Equals truck operating hours per unit less loader operating hours per truck (minimum of zero)
Hang Time per Load (min)	-	Total loader hang-time divided by load count
Truck Standby (hrs)	11.40	Equals loader operating hours per unit less truck operating hours per truck (minimum of zero)
Truck Standby (min / load)	0.44	Total truck standby divided by load count
Mining Activity Calendar Hours and Average Fleet Productivity		
Loader Utilization	68.2%	Estimated based on preventive maintenance, unscheduled down, operator breaks, shift change, rotation change, tramming, fuel & lube, safety observations, and weather delays
Truck Utilization	70.4%	Note: Utilization = equipment working time / calendar hours
Loader Total Hrs	130.0	Function of operating hours per unit and loader utilization
Truck Total Hrs	109.9	Function of operating hours per unit and truck utilization
Total Fleet Total Hrs	130.0	Total fleet calendar hours equals calendar hours associated with fleet limiting equipment
Mining Activity Total Hrs	154.0	Equals Total Fleet Calendar Hours plus bench turnover delay
Avg. Fleet Prod. (st/hr)	***1,535***	***Total activity tons divided by activity calendar hours***

Mining cut shapes were generated manually for each period to meet the scheduling objectives identified in the strategic mine plan, mill feed schedule, DRSF sequence, and stockpile designs. The load and haul calculations were performed for each ore type within a cut for each period in the mine schedule (i.e. monthly through the end of year 2 and quarterly after). The load and haul schedule includes equipment operating hours and required units by period and equipment type as summarized on Figure 16.19, Figure 16.20, and Figure 16.21.

Figure 16.19: Haul Truck and Articulated Truck (ADT) Unit Count

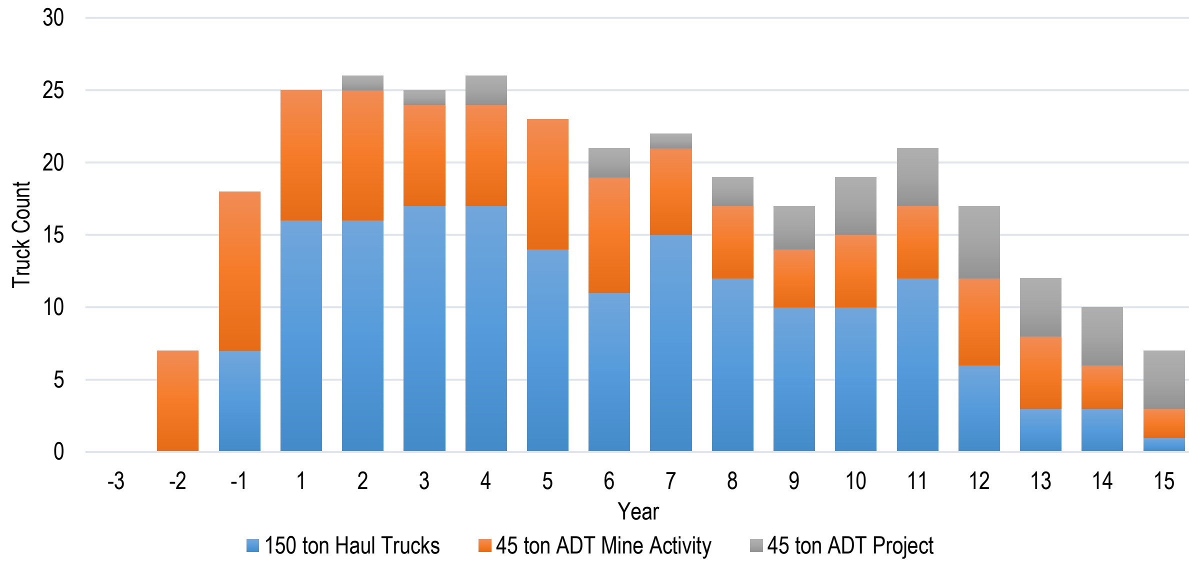


Figure 16.20: Mine Production and Development Loading Unit Count

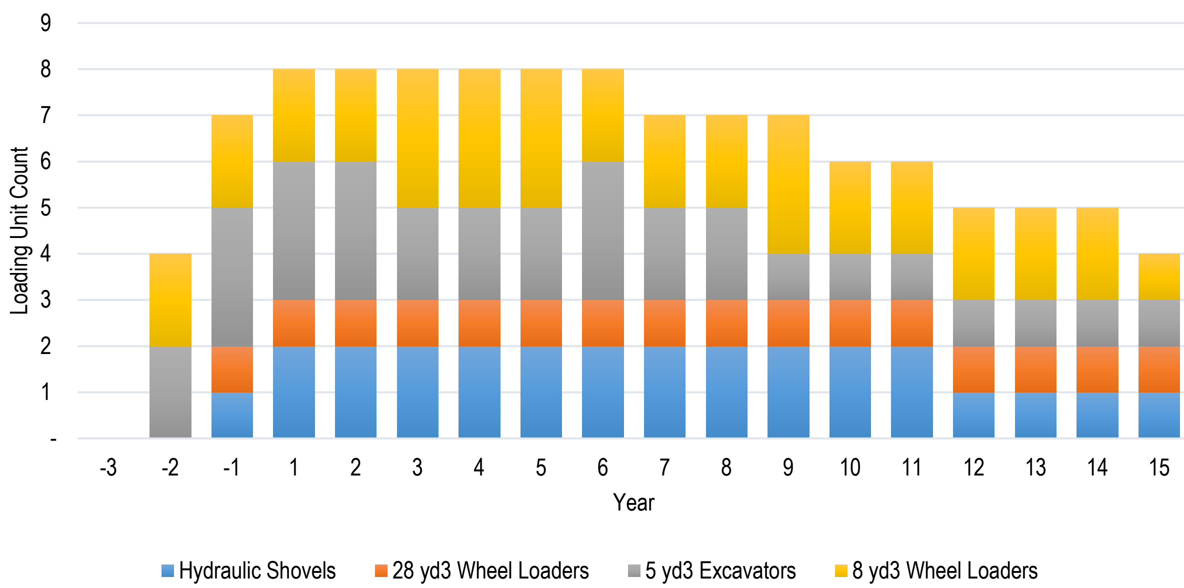
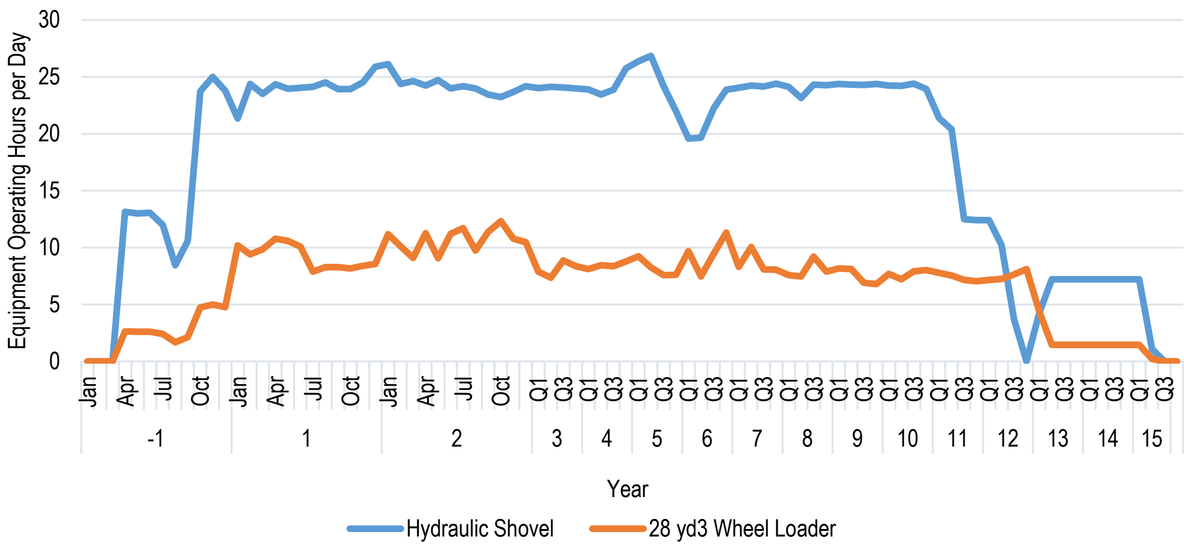


Figure 16.21: Mine Production Fleet Loading Equipment Operating Hours



16.8.3 Drill and Blast

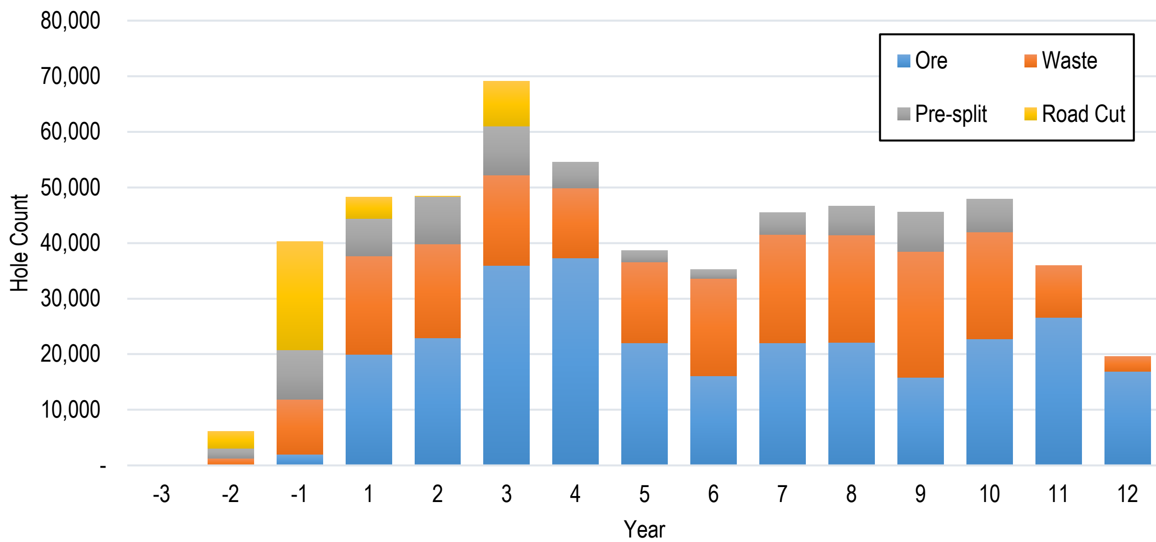
Drilling and blasting requirements were estimated based on the following four general types of blasting: ore blasting, waste blasting, highwall pre-splitting, and road development as shown in Table 16.8. Pre-splitting is a controlled blasting technique to create shear planes along the pit highwall to promote pit highwall stability and maintain pit design compliance during production mining. A commercial explosives and blasting systems provider is planned to be contracted to provide and manage ammonium nitrate-fuel oil mixtures (ANFO), emulsion, and blasting accessories. The explosives contractor will also provide and manage the explosive plant and mixing equipment. Scheduled blasthole count by year is shown on Figure 16.22.

Table 16.8: Drill and Blast Pattern by Blast Type

Item	Ore[1]	Waste	Road Cut[2]	PreSplit[3]
Life of Mine Qty	101,327 kst	237,103 kst	5,162 kst	1,161,766 yd²
Average ANFO Blend[4]	30/70	30/70	50/50	30/70
Redrills	2%	2%	8%	2%
Secondary Holes	2%	5%	15%	0%
Secondary Depth (ft)	15.0	30.0	6.0	n/a
ANFO Spillage	2%	2%	5%	5%
Stemming Spillage	5%	5%	8%	5%
Average Hole Count per Blast	350	350	150	100
Bench Height (ft)	20	40	8	40
Blast Hole Diameter (in)	6.75	8.00	3.50	3.50
Burden x Spacing (ft)	14 x 16	18 x 21	8 x 12	4
Sub-drill (ft)	3.0	4.0	2.0	0
Stemming (ft)	10.0	13.0	4.0	2.0
Base Charge (ft)	11.0	19.0	6.0	20.8
Deck Stemming (ft)	2.0	12.0	-	19.8
Powder Factor (lb/st)	0.52	0.39	0.68	5.16 lb/yd²

Notes:
1) Ore and waste tons include bedrock material only. Overburden is assumed to be dozer ripped and loaded without the need for blasting.
2) Road cuts include road construction and initial pit bench development to create benches with sufficient room to operate production blasthole drills effectively.
3) Pit and road highwall will only be pre-split for highwall above planned backfill elevation. Pre-split calculations based on 70 degree angled holes.
4) Based on average ANFO / emulsion blend estimated from degree of moisture expected in blastholes prior to loading explosives.

Figure 16.22: Blasthole Count by Blast Type and Year



16.8.4 Maintenance and Auxiliary Equipment

The maintenance and auxiliary equipment were selected based on production fleet size, development fleet size, open pit geometries, and the number of concurrent projects, pits, and DRSFs in operation. A list of maintenance and auxiliary equipment is provided in Table 16.2.

16.8.5 Mine Sequence Drawings

The SGP terrain comprises steep-walled valleys and, as a result, initial haul road access to the upper benches of the open pits will require significant effort to pioneer roads and develop initial mining benches. Construction of these roads is planned prior to production mining. Designs of the initial access roads and other necessary external haul roads are shown on the time sequence plans presented on Figure 16.23 to Figure 16.35, inclusively. Key details for each year of mining are provided with each figure.

Figure 16.23: Annual Mine Progression – End of Year -1 (Pre-Production)

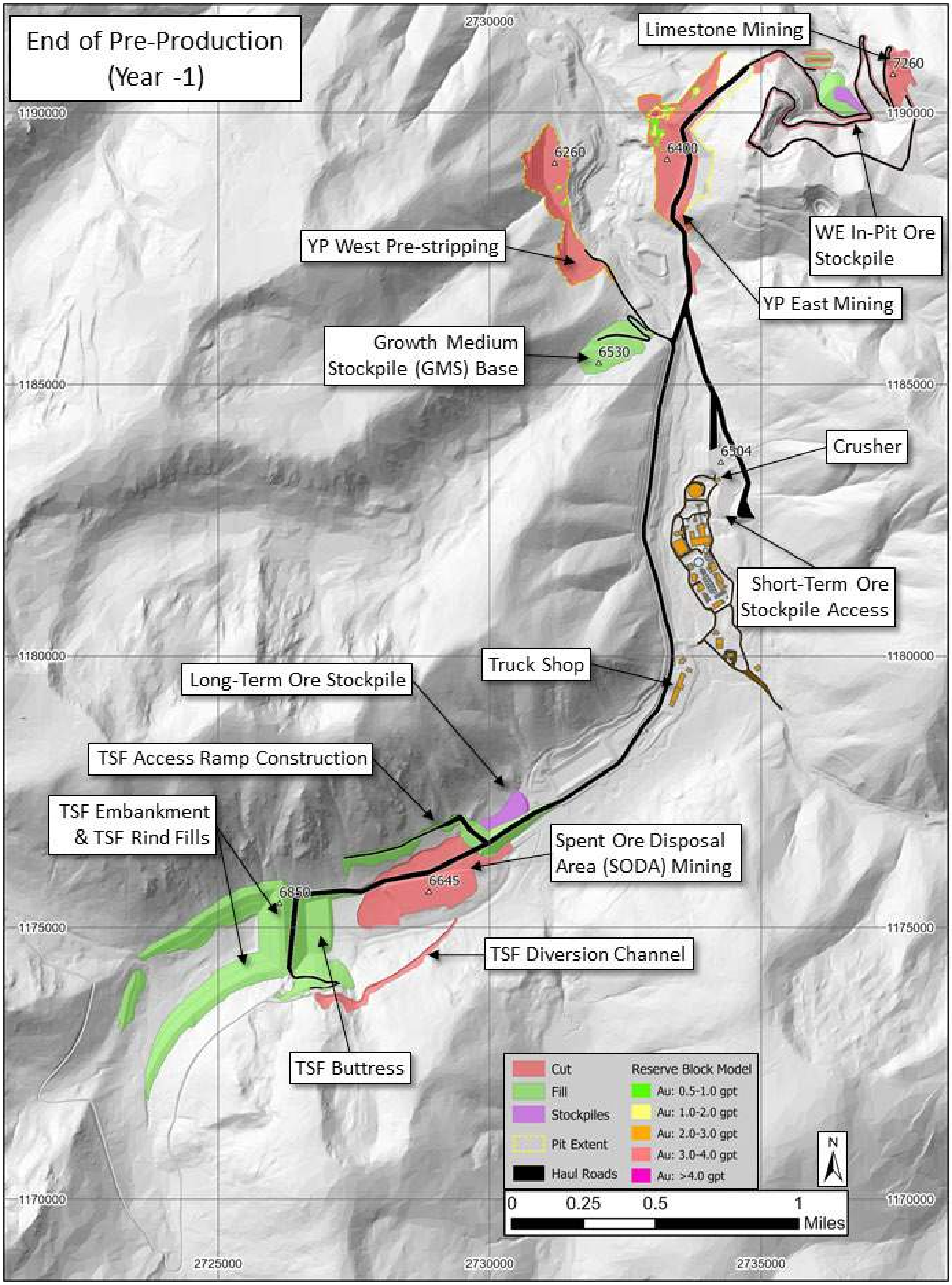


Figure 16.24: Annual Mine Progression – End of Year 1

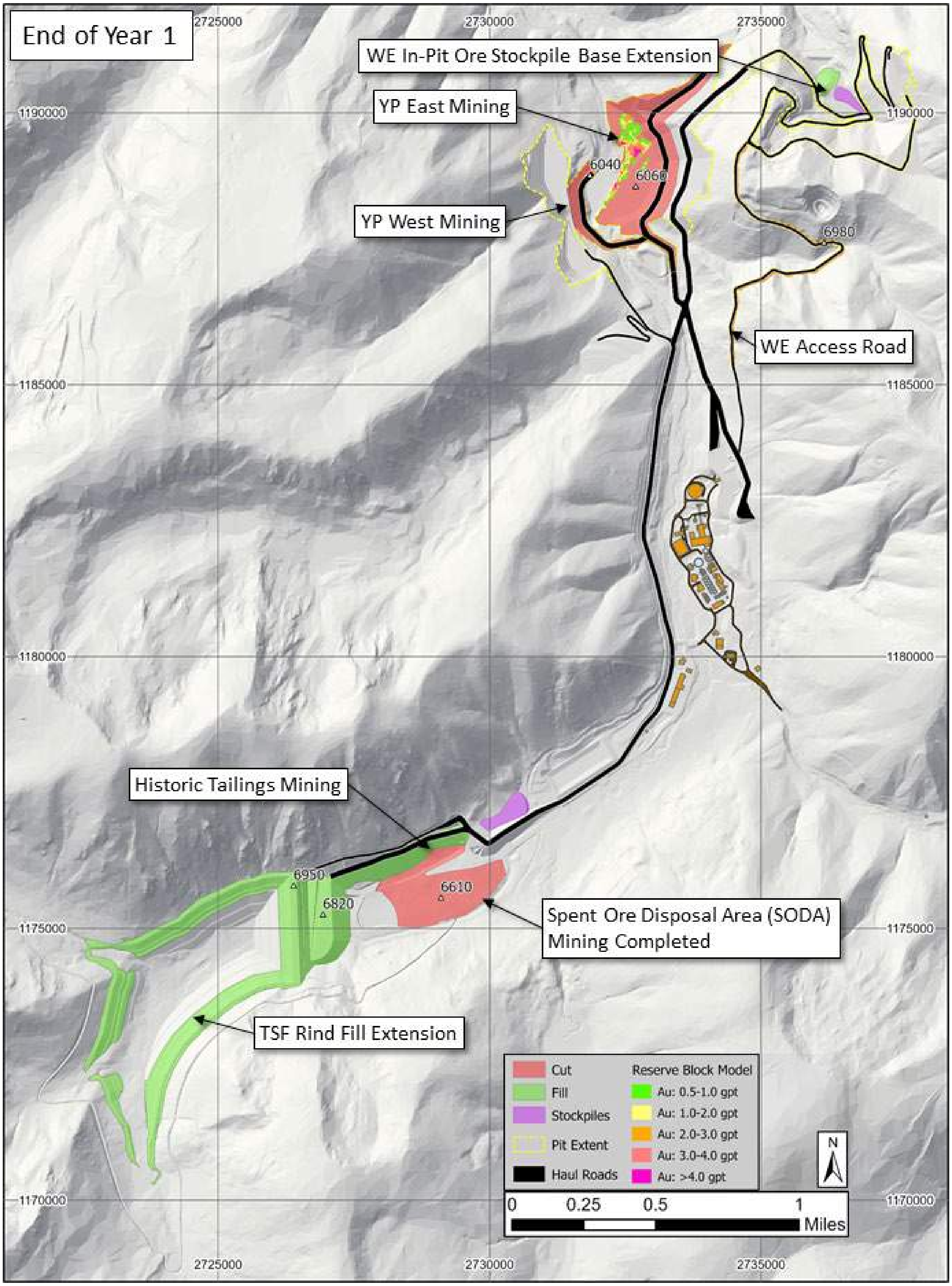


Figure 16.25: Annual Mine Progression – End of Year 2

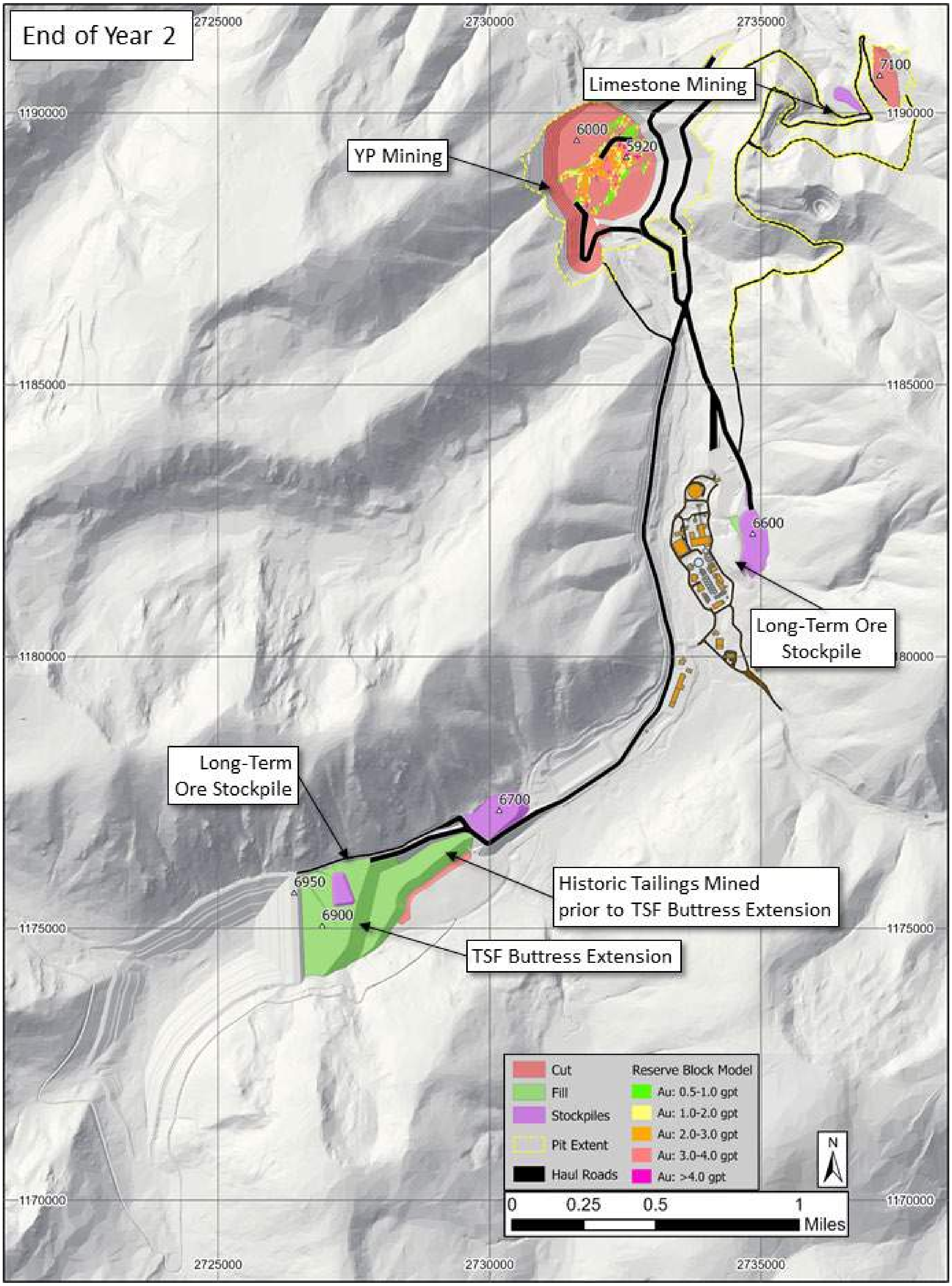


Figure 16.26: Annual Mine Progression – End of Year 3

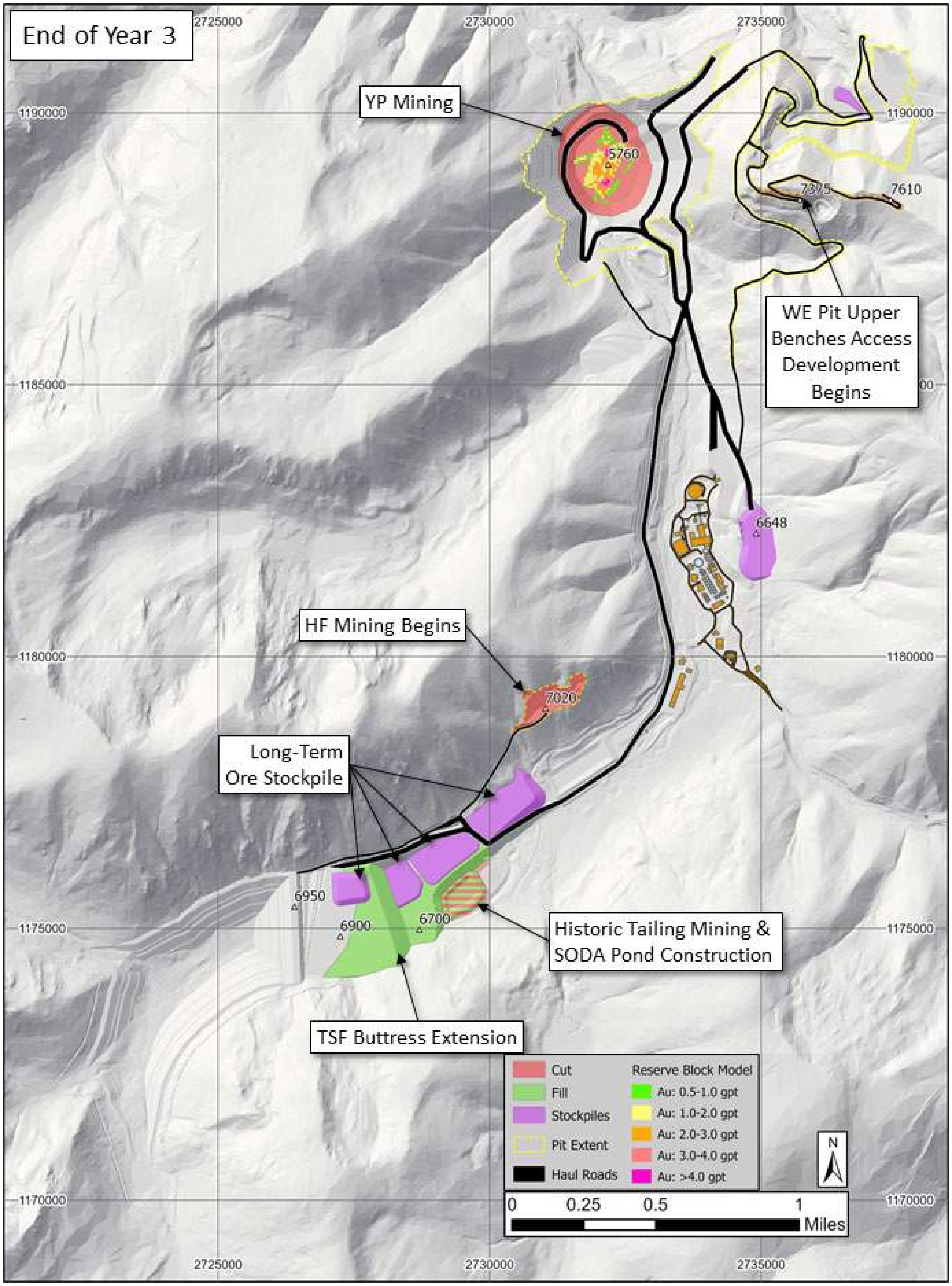


Figure 16.27: Annual Mine Progression – End of Year 4

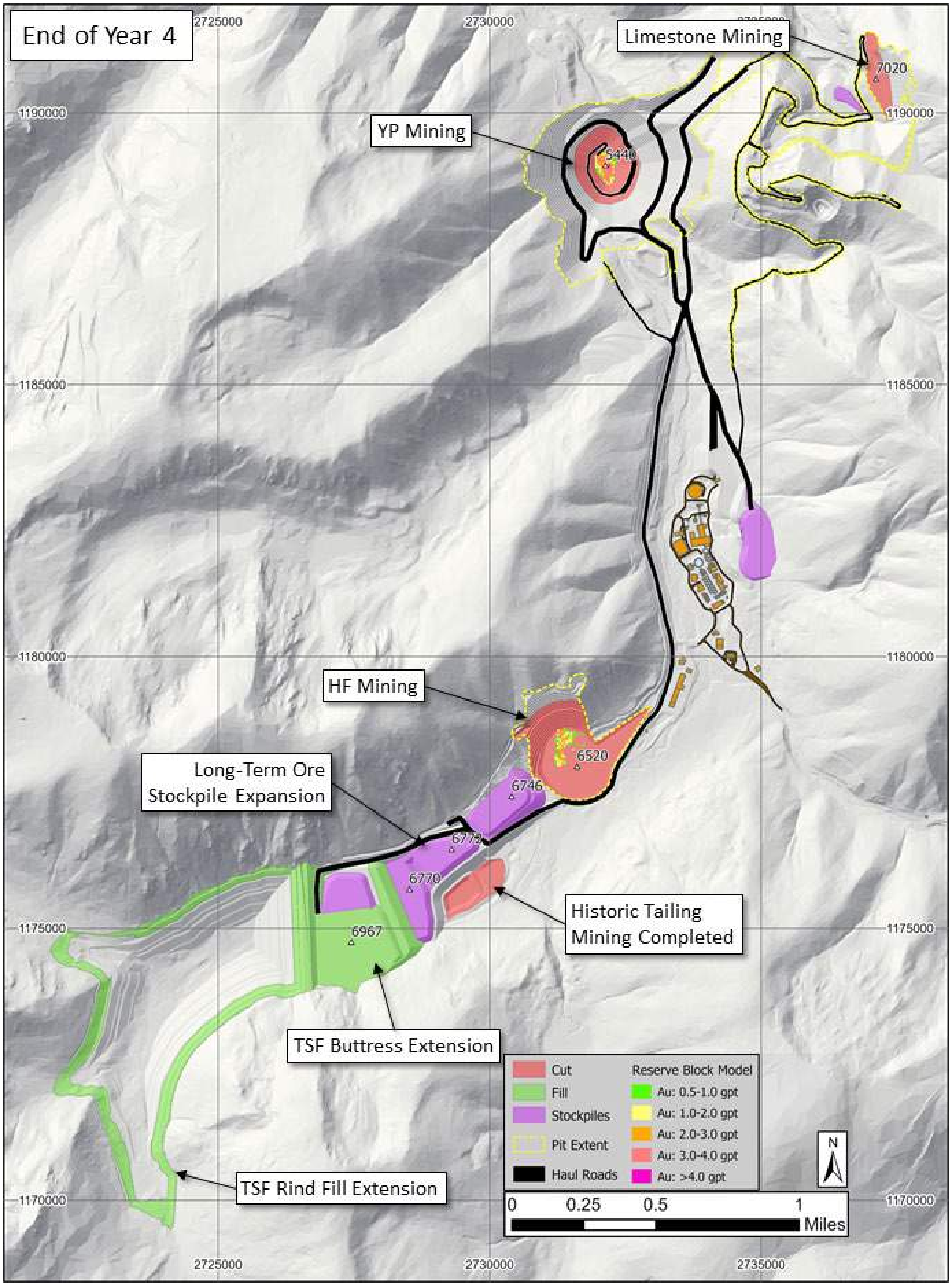


Figure 16.28: Annual Mine Progression – End of Year 5

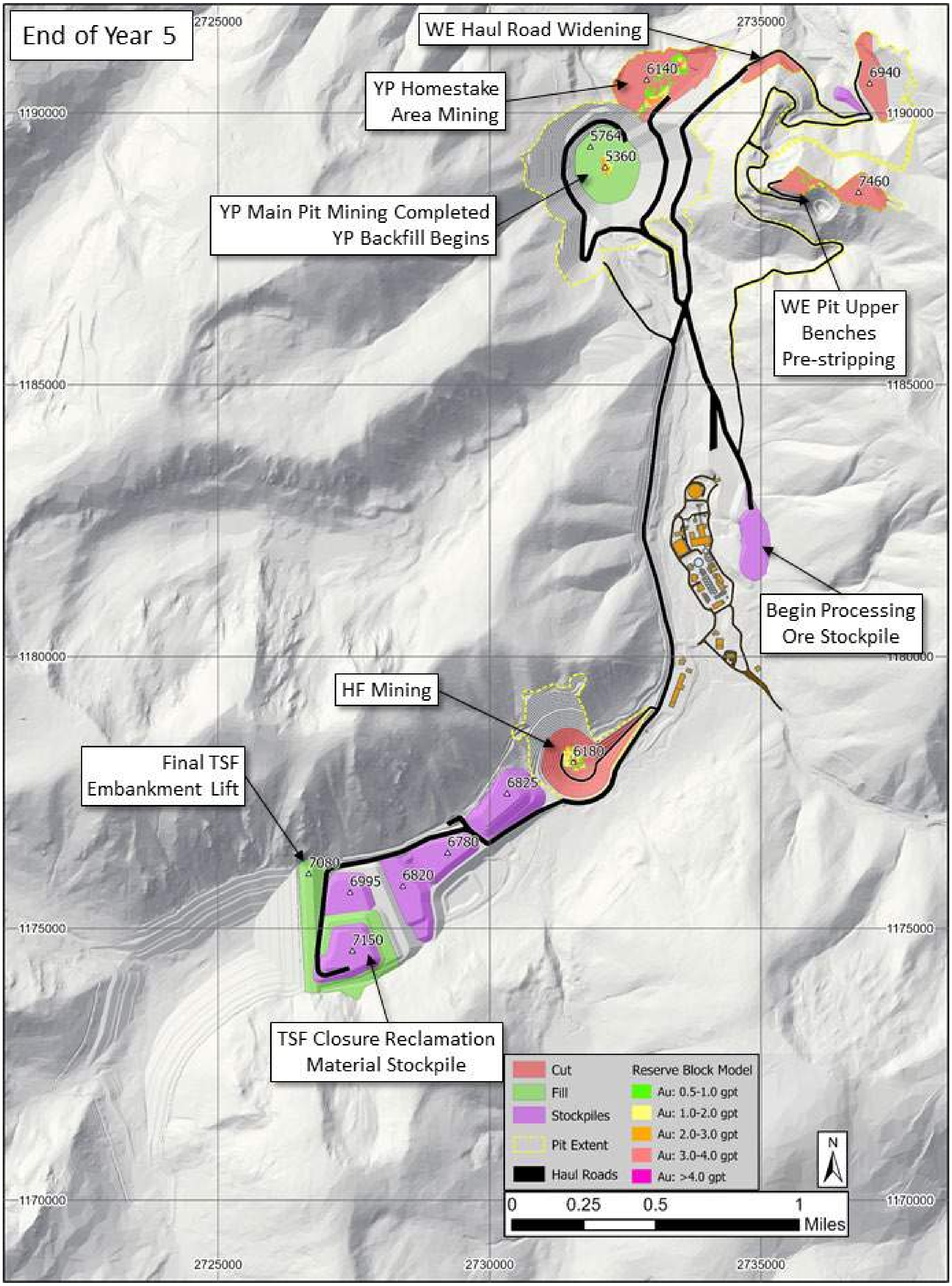


Figure 16.29: Annual Mine Progression – End of Year 6

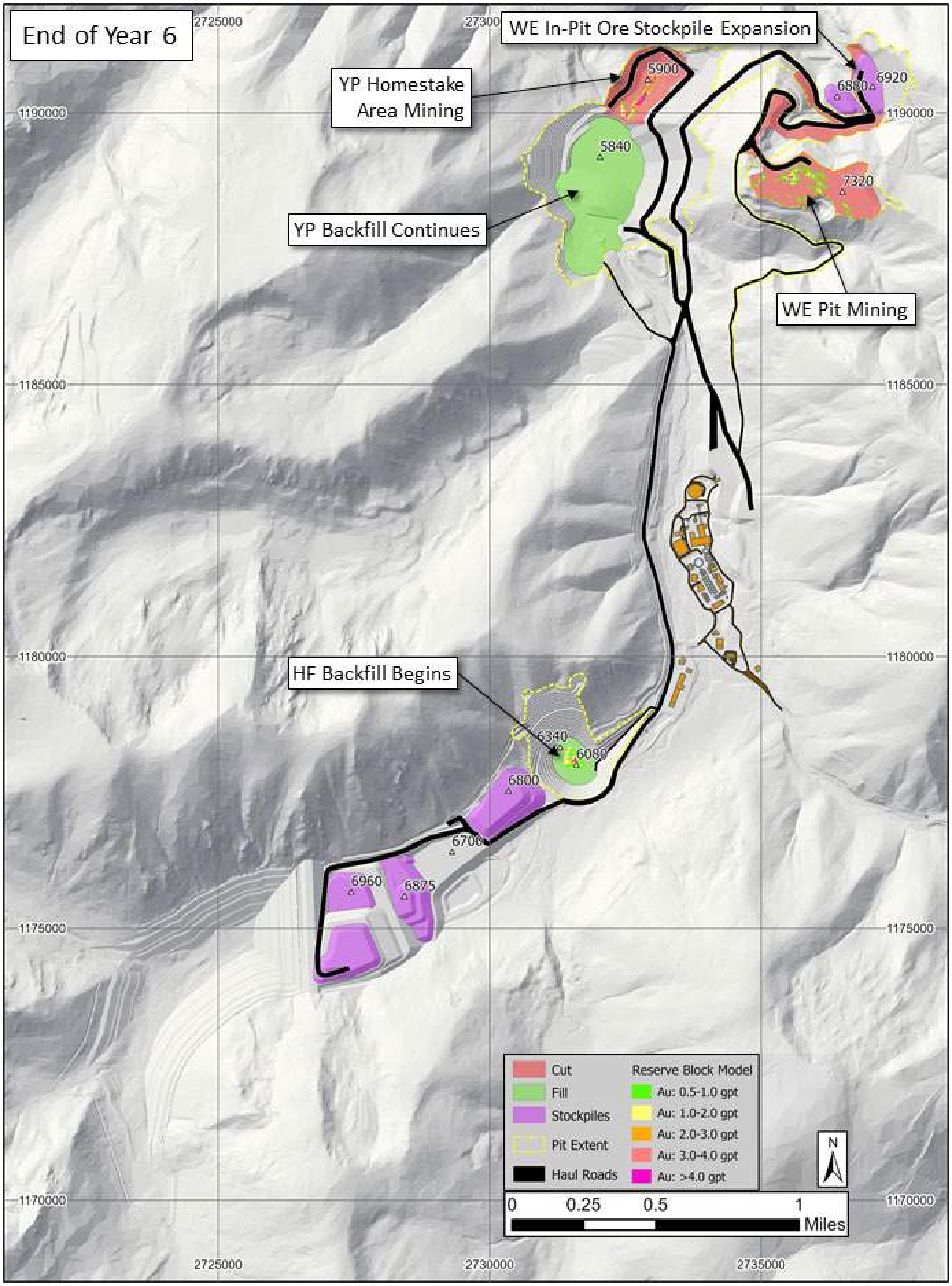


Figure 16.30: Annual Mine Progression – End of Year 7

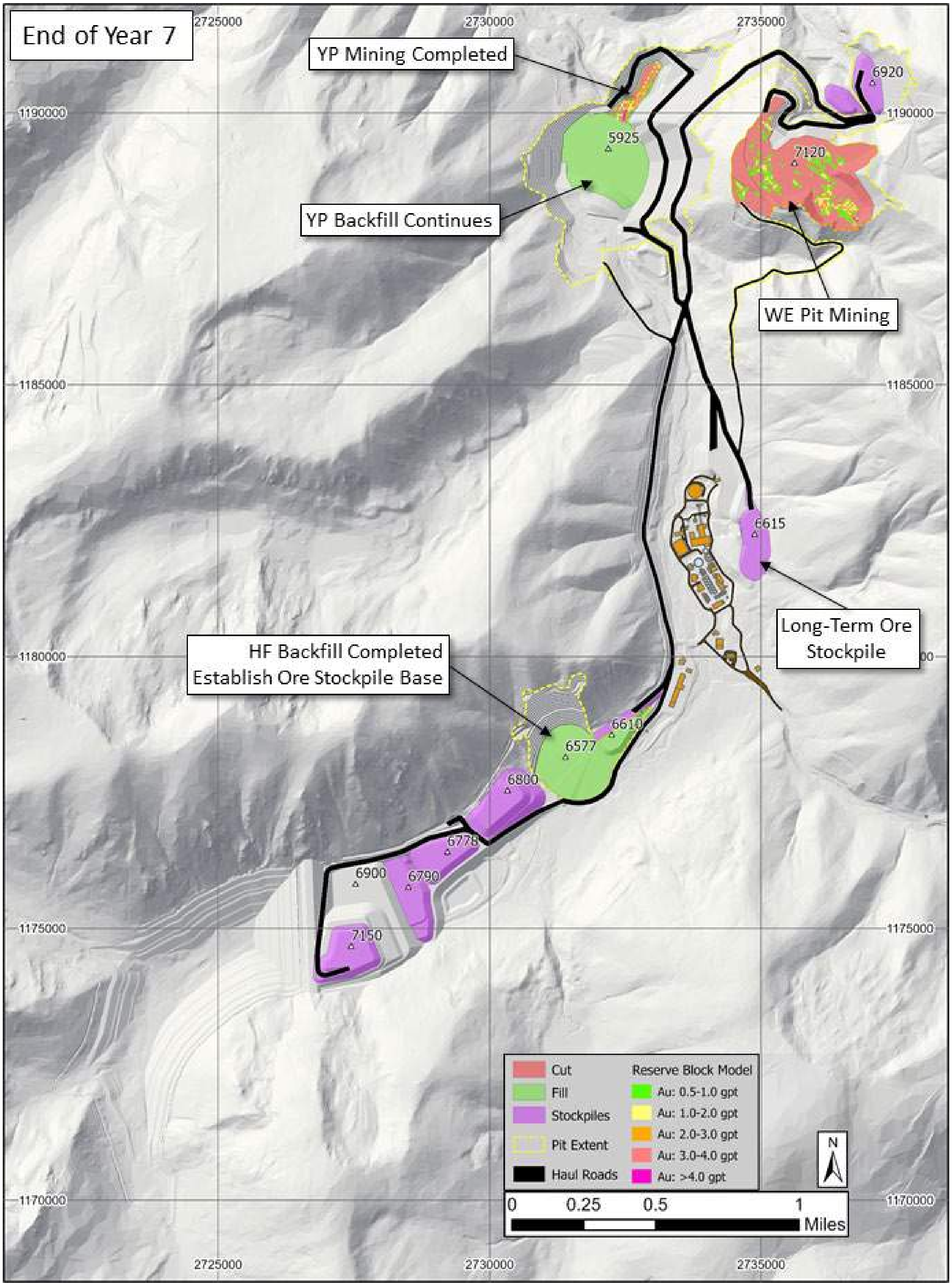


Figure 16.31: Annual Mine Progression – End of Year 8

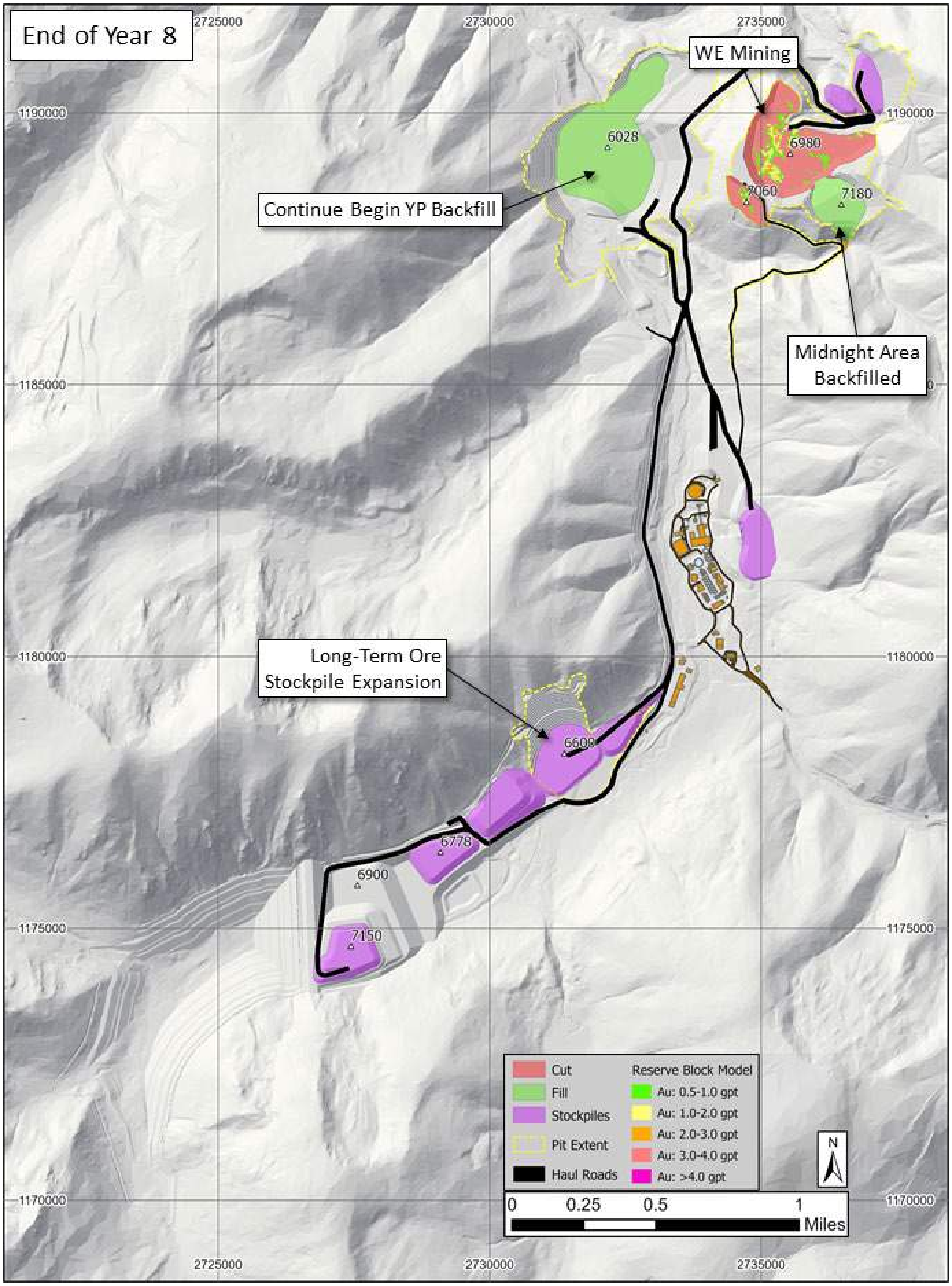


Figure 16.32: Annual Mine Progression – End of Year 9

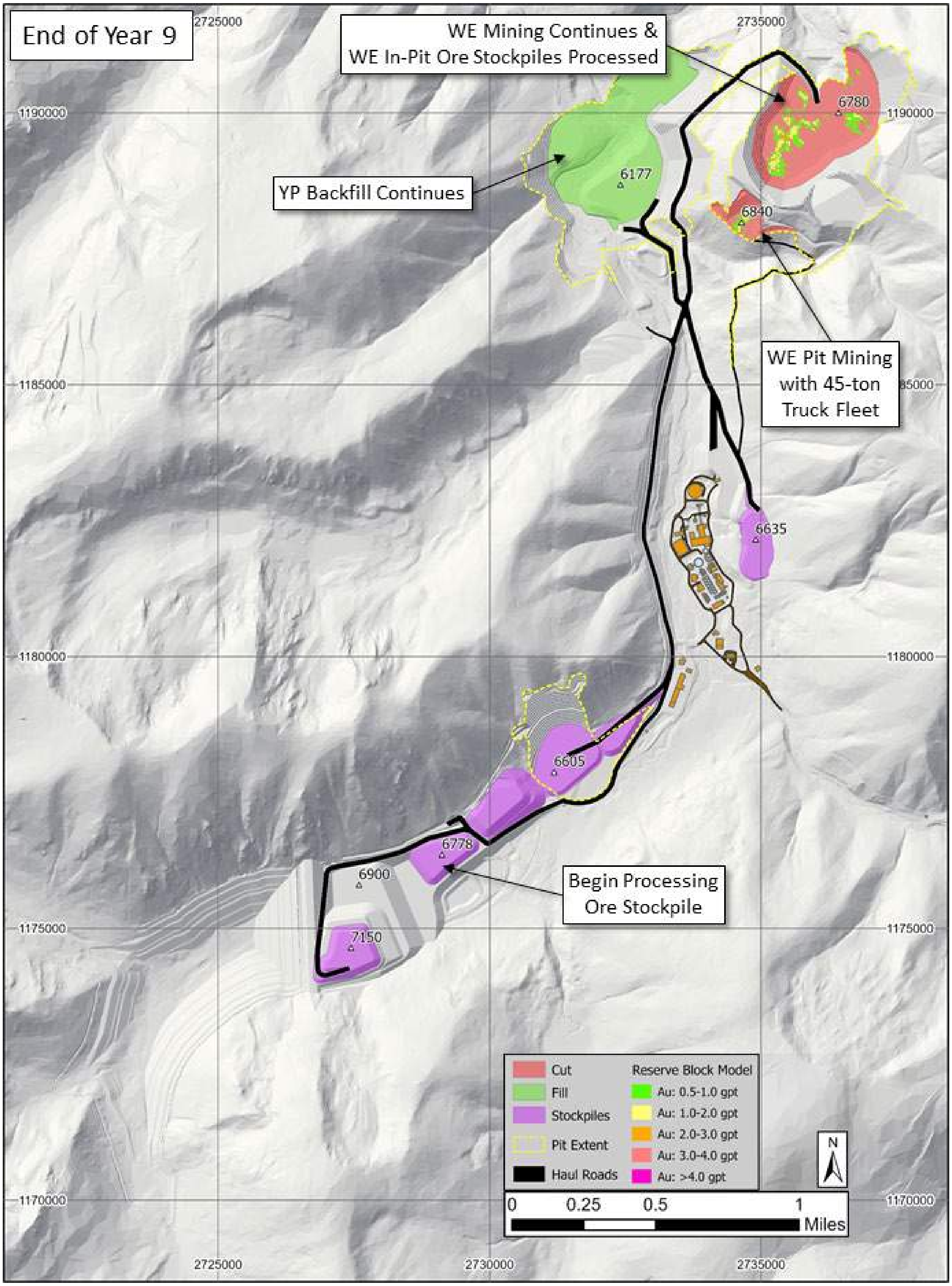


Figure 16.33: Annual Mine Progression – End of Year 10

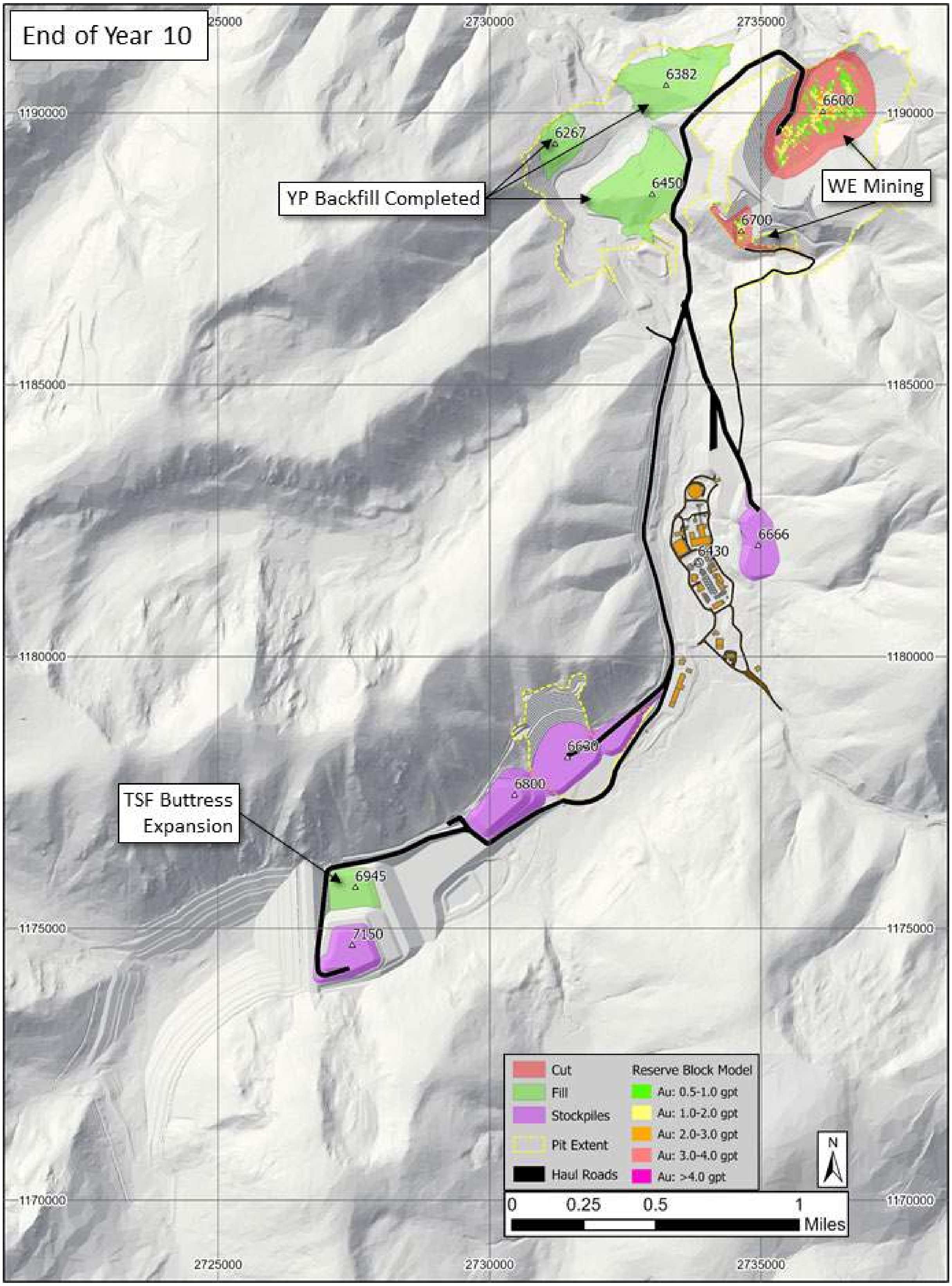


Figure 16.34: Annual Mine Progression – End of Year 11

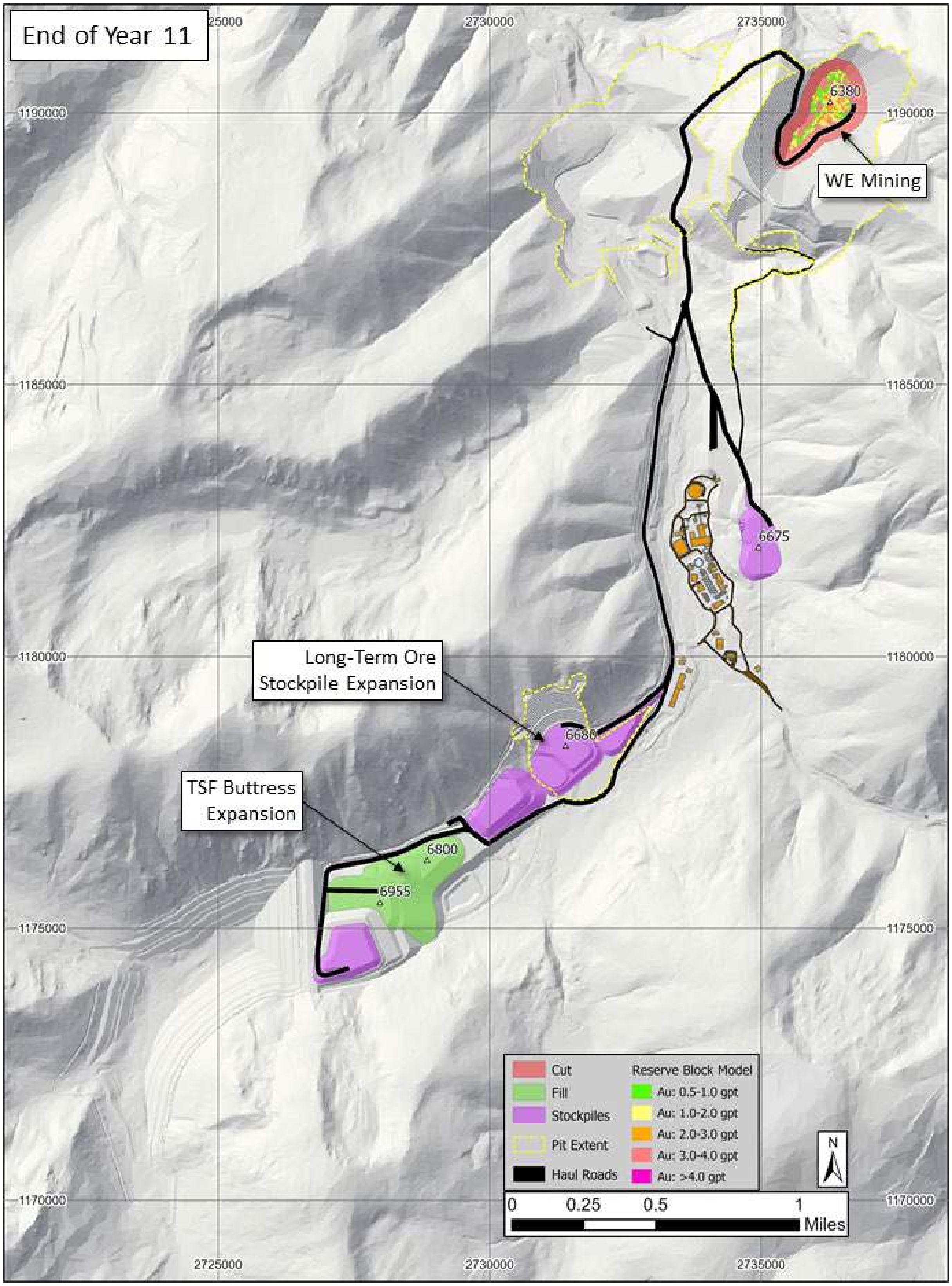
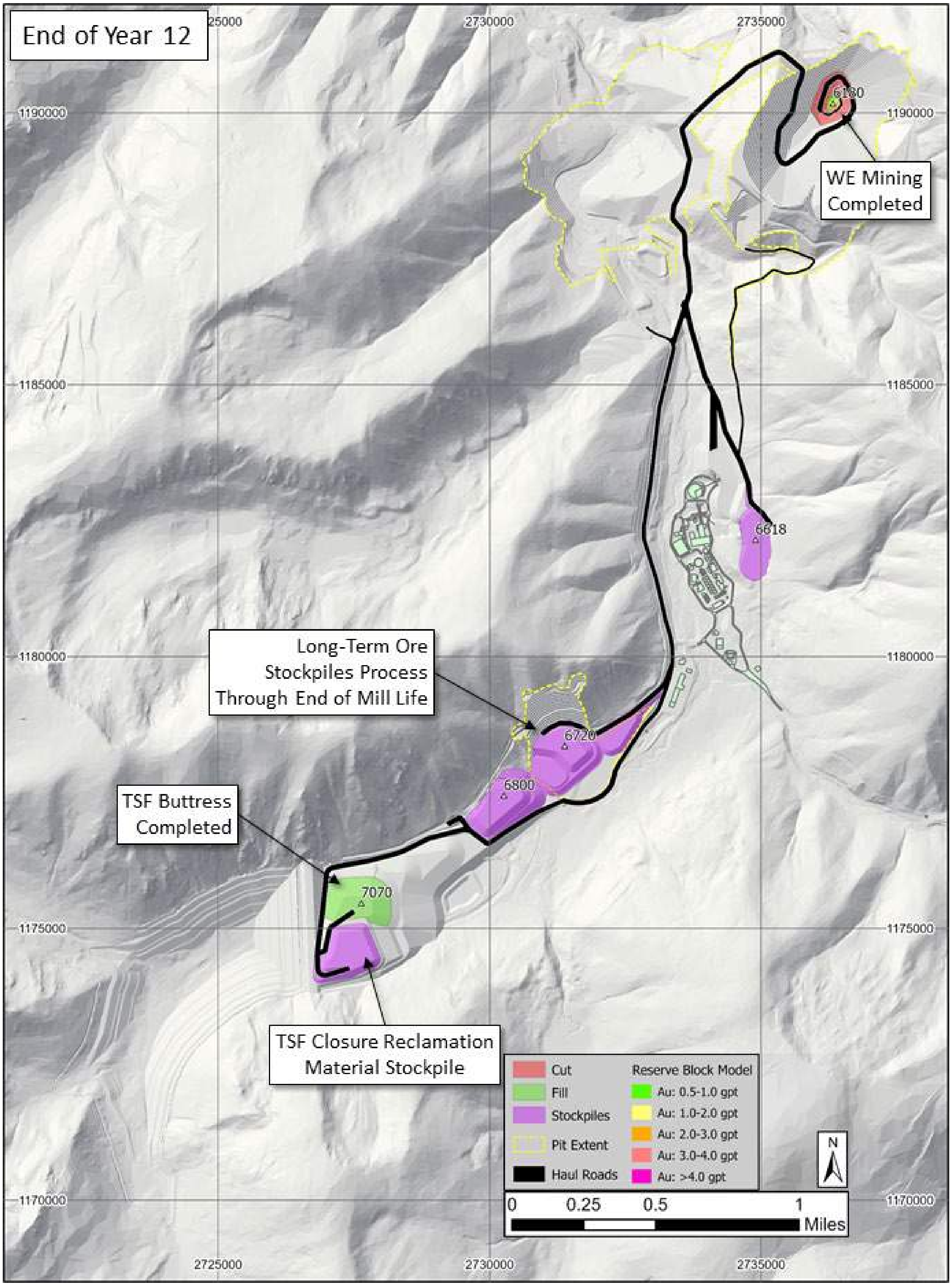


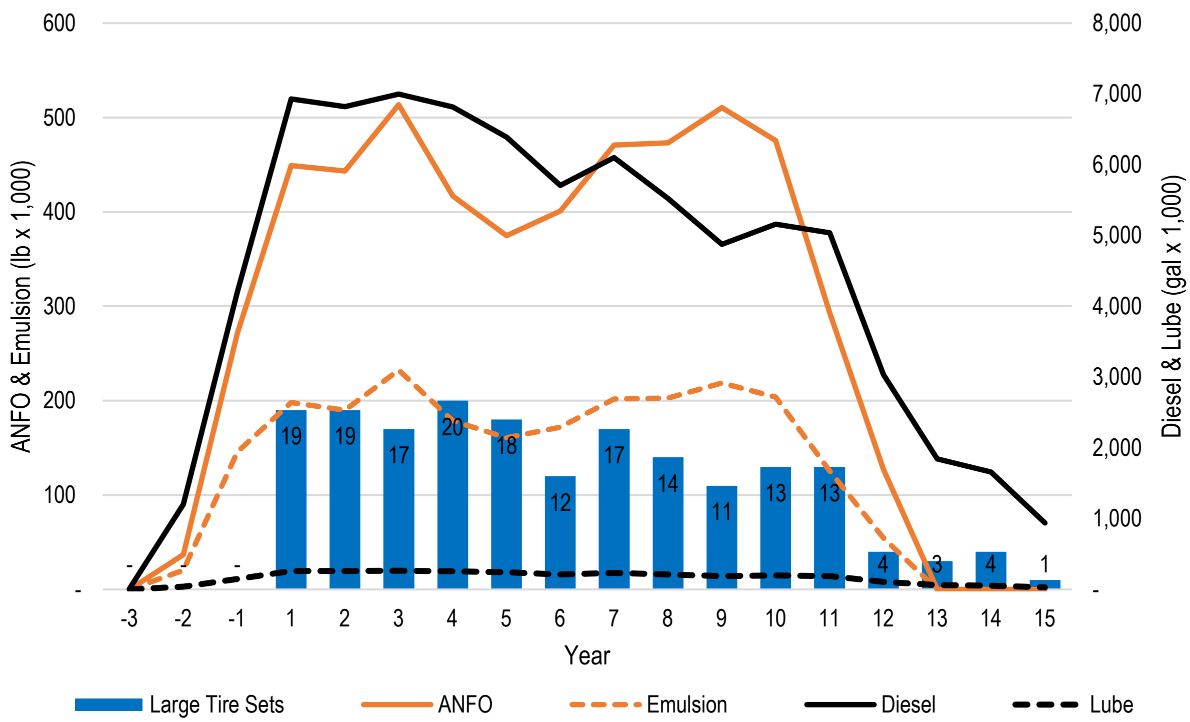
Figure 16.35: Annual Mine Progression – End of Year 12



16.9 MINE CONSUMABLES ESTIMATE

The mine consumables estimate incorporates the mine production schedule, drill and blast schedule, and equipment consumable rates assumptions to generate the mine consumables schedule used in the operating cost estimate. All mine consumable rates are based on equipment manufacturer values and actual mining data. Consumables for the loading, hauling, auxiliary, and support equipment primarily consist of diesel fuel, lube, tires, maintenance parts, and ground engaging tools. Additional consumables for drilling include drill steel, drill bits, hammers, bushings, and chucks. Blasting consumables were estimated separately based on pattern type and include ANFO, emulsion, stemming, detonation chord, boosters, detonators, and air deck plugs. The mine consumables schedule was developed using first principal calculations based on equipment engine hours and blast pattern designs for each period throughout the mine life. A summary of principle mine equipment consumables is shown on Figure 16.36.

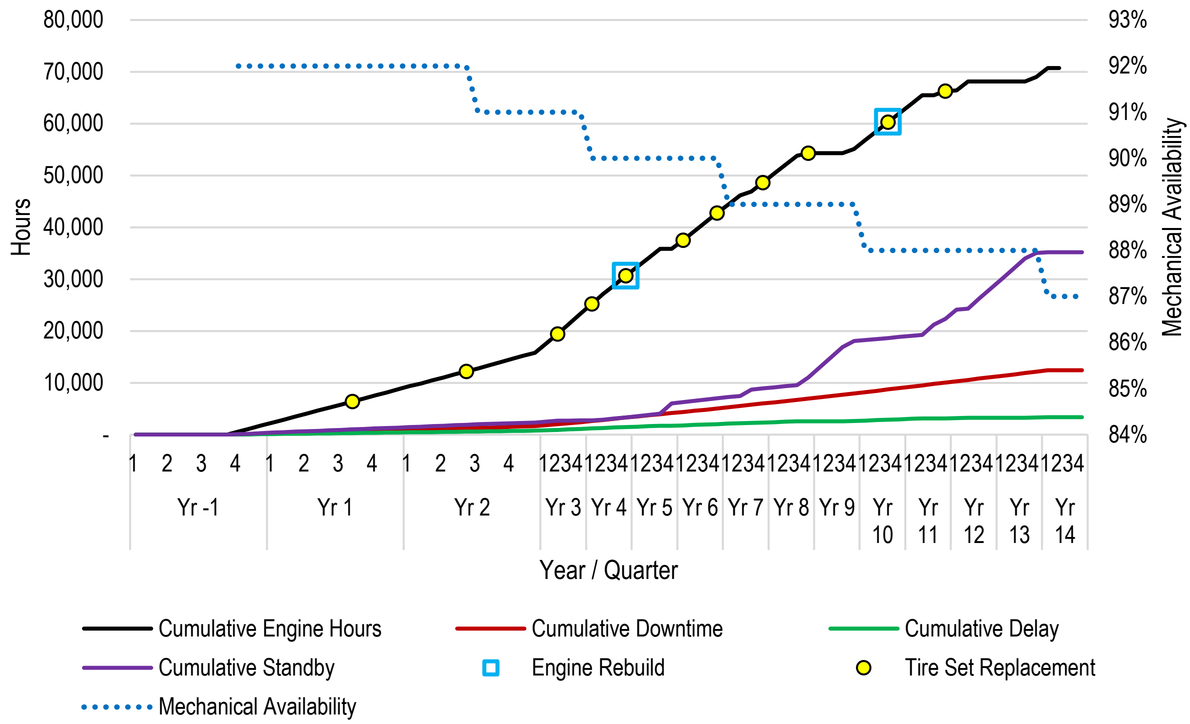
Figure 16.36: Principal Mine Equipment Consumables by Year



16.10 MINE MAINTENANCE ESTIMATE

The mine maintenance estimate consists of estimating equipment preventive maintenance schedules, major rebuild schedules, equipment parts life and cost estimates, and equipment mechanical availability to generate mine maintenance staffing requirements, equipment mechanical availability estimates, and operating costs. The basis for estimating mine equipment maintenance requirements was manufacturer estimates, actual mine data, and maintenance cost surveys. The maintenance schedule was generated for each individual equipment unit for the life of the equipment based on each unit's cumulative engine hours and unscheduled downtime assumptions as a function of each unit's progressive time in-service. An example of engine hours estimation, tire set replacement, and engine rebuild schedule for a haul truck in the production fleet is illustrated on Figure 16.37.

Figure 16.37: Haul Truck Engine Hours and Engine Rebuild Example



16.11 STAFFING ESTIMATION AND ORGANIZATIONAL STRUCTURE

The mine is scheduled to operate continuously 365 days per year with personnel working 12 hour shifts on a 2-week-on / 2-week-off rotation. All mine operations staff will rotate between day and night shift except for the blasting crew, technical staff, and management which will work day shift only. The staffing estimate is based on the mine equipment schedule, equipment maintenance schedule, and estimated technical workload during construction, mine operation, and closure. All mining staff are managed by the mine manager, who reports to the general manager as shown on Figure 16.38. Staffing headcount is summarized on Figure 16.39 and shown by position and year in Table 16.9 and Table 16.10.

Figure 16.38: Mining Organizational Structure

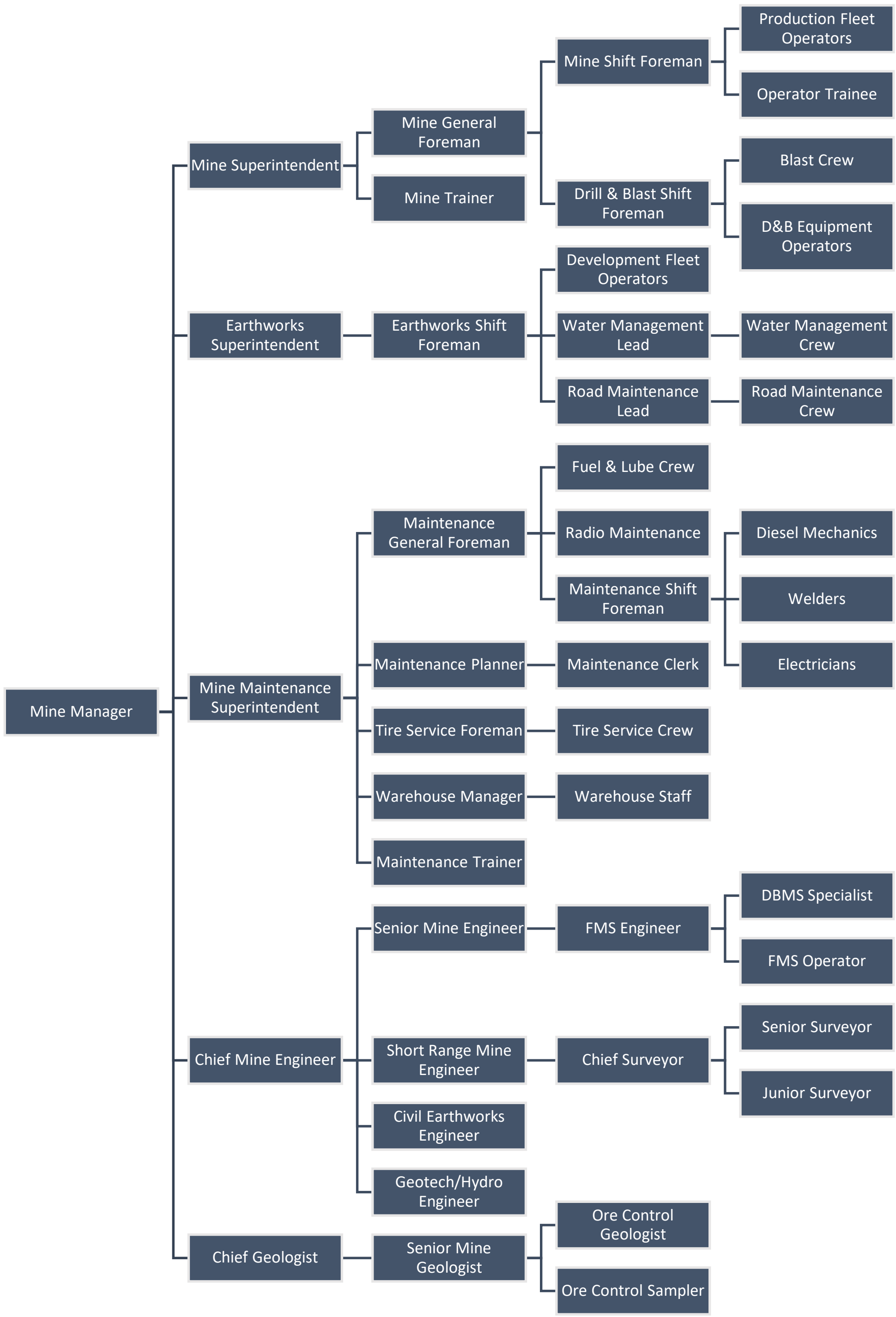
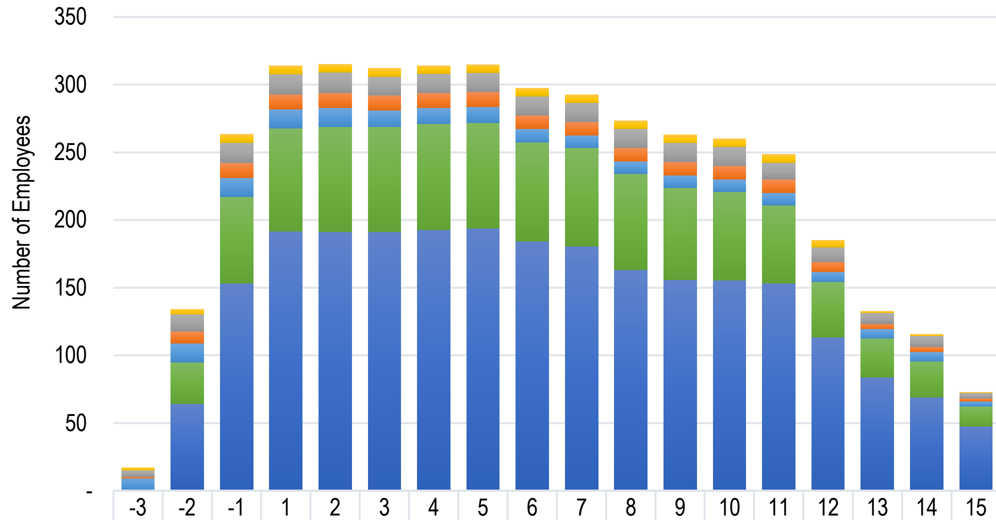


Figure 16.39: Salaried and Hourly Mining Personnel by Department and Year



	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mine Geology Salaried	2	4	6	6	6	6	6	6	6	6	6	6	6	6	5	1	1	1
Mine Engineering Salaried	5	12	15	15	15	14	14	14	14	14	14	14	14	13	11	8	8	4
Mine Maintenance Salaried	1	9	11	11	11	11	11	11	10	10	10	10	10	10	8	4	4	2
Mine Operations Salaried	9	14	14	14	14	12	12	12	10	9	9	9	9	9	8	7	7	4
Mine Maintenance Hourly	0	31	64	76	78	78	78	78	73	73	71	68	66	58	41	29	26	15
Mine Operations Hourly	-	64	153	192	192	192	193	194	184	181	163	156	156	153	113	84	69	48

Table 16.9: Salary Staff Requirements

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mine Manager	0.8	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations																		
Mine Superintendent	0.8	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Earthworks Superintendent	0.8	1	1	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine General Foreman	0.8	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Mine Shift Foreman	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2
Drill & Blast Shift Foreman	-	2	2	2	2	2	2	2	2	2	2	2	2	2	0.5	-	-	-
Earthworks Shift Foreman	1.5	2	2	2	2	1	1	1	1	-	-	-	-	-	-	-	-	-
Mine Trainer	1.5	2	2	2	2	2	2	2	-	-	-	-	-	-	-	-	-	-
Mine Operations Total	**8.3**	**13**	**13**	**13**	**13**	**11**	**11**	**11**	**9**	**8**	**8**	**8**	**8**	**8**	**6.5**	**6**	**6**	**3**
Mine Maintenance																		
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Maintenance General Foreman	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Maintenance Shift Foreman	-	3	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	1
Maintenance Planner	-	1.5	2	2	2	2	2	2	2	2	2	2	2	2	1.5	1	1	0.5
Maintenance Trainer	-	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-	-	-
Maintenance Clerk	-	1.3	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-	-
Mine Maintenance Total	**1**	**8.8**	**11**	**11**	**11**	**11**	**11**	**11**	**10**	**10**	**10**	**10**	**10**	**10**	**7.5**	**4**	**4**	**2**
Mine Engineering																		
Chief Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Senior Mine Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Short Range Mine Engineer	-	1.4	2	2	2	2	2	2	2	2	2	2	2	2	1	-	-	-
FMS Engineer	0.4	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
DBMS Specialists	0.4	1	2	2	2	2	2	2	2	2	2	2	2	1.5	0.8	-	-	-
Civil Earthworks Engineer	1	2	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Geotech / Hydro Engineer	-	-	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Chief Surveyor	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Senior Surveyor	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Junior Surveyor	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Mine Engineering Total	**4.8**	**12.4**	**15**	**15**	**15**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**14**	**12.5**	**10.8**	**8**	**8**	**4**
Mine Geology																		
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-
Senior Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0.5
Ore Control Geologist	-	1	2	2	2	2	2	2	2	2	2	2	2	2	1	-	-	-
Sampler	-	0.8	2	2	2	2	2	2	2	2	2	2	2	2	2	-	-	-
Mine Geology Total	**2**	**3.8**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**6**	**5**	**1**	**1**	**0.5**
Salaried Staff Total	**17**	**39**	**46**	**46**	**46**	**43**	**43**	**43**	**40**	**39**	**39**	**39**	**39**	**38**	**31**	**20**	**20**	**11**

Table 16.10: Hourly Staff Requirements

Year	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mining Equipment Operators																		
Mine Production Fleet	-	4	45	98	98	100	100	91	82	97	85	85	80	78	50	28	26	15
Mine Development Fleet	-	45	74	56	55	47	50	58	57	40	35	27	35	41	41	39	26	18
Mine Auxiliary Fleet	-	5	11	14	14	15	15	15	15	15	15	15	15	13	10	7	7	5
Mine Indirect Hourly	-	11	24	24	24	30	27	28	30	28	28	28	26	22	13	10	10	10
Mine Equipment Operator Total	**-**	**64**	**153**	**192**	**191**	**191**	**192**	**192**	**184**	**180**	**163**	**156**	**155**	**153**	**113**	**84**	**69**	**48**
Mine Maintenance Staff																		
Diesel Mechanics	1	8	20	27	27	28	28	28	24	24	22	20	20	20	14	9	8	5
Welder	-	3	8	10	10	10	10	10	10	10	10	9	8	6	5	2	2	1
Fuel & Lube Crew	-	5	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4	2
Tire Crew	-	5	8	8	8	8	8	8	8	8	8	8	8	8	4	4	4	2
Maintenance Laborer	-	6	14	17	18	18	18	18	17	17	17	17	15	12	8	7	7	4
Radio Maintenance Staff	-	1	2	2	2	2	2	2	2	2	2	2	2	2	1	-	-	-
Warehouse Staff	-	1	4	4	4	4	4	4	4	4	4	4	4	2	2	2	2	1
Mine Maintenance Staff Total	**1**	**31**	**64**	**76**	**78**	**78**	**78**	**78**	**73**	**73**	**71**	**66**	**66**	**58**	**41**	**28**	**25**	**15**
Hourly Staff Total	**1**	**95**	**217**	**268**	**269**	**269**	**270**	**270**	**258**	**253**	**234**	**224**	**221**	**211**	**154**	**112**	**95**	**62**

16.12 CAPITAL AND OPERATING COST ESTIMATE

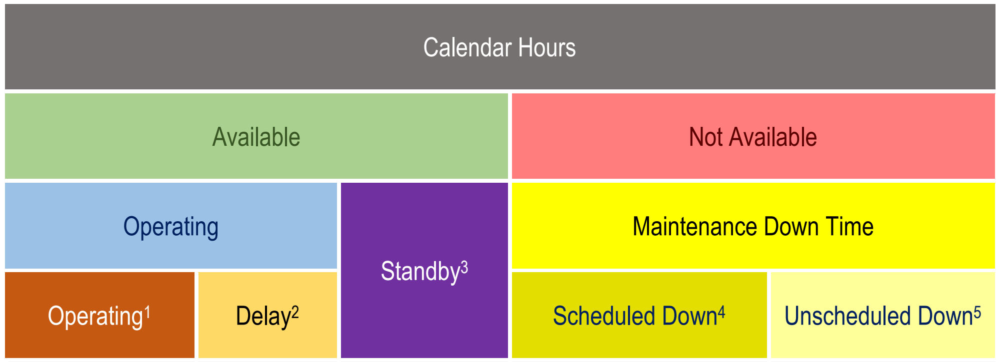
16.12.1 Mine Equipment Capital Cost Estimate

All capital costs for each equipment type were estimated using vendor budgetary quotes or recent mining industry surveys. Equipment capital costs include estimates for freight, assembly, spare parts, initial tire purchase, fire suppression, equipment advance payments, and potential equipment modifications. For equipment that is planned to be leased, pay schedules are based on quotes provided by equipment manufacturers. Capital and operating cost details are provided in Section 21. The equipment purchase schedule is shown in Table 21.3

16.12.2 Mining Operating Cost Estimate

Mine equipment operating costs were developed using first principals based on vendor provided hourly operating cost estimates and recent operating mine equipment survey data. Each equipment unit was scheduled on a monthly period through the end of year 2 and quarterly after using a time model as shown on Figure 16.40.

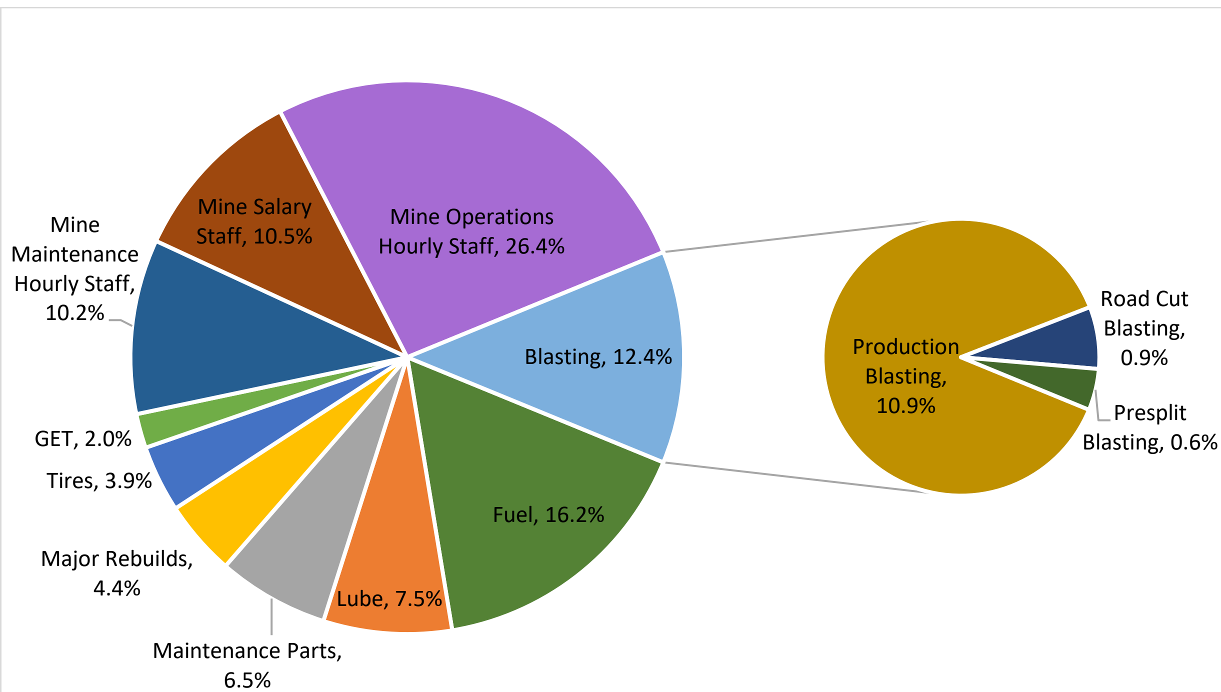
Figure 16.40: Equipment Time Category Model



Notes:
1) Operating Time: The time the equipment is running and doing productive work.
2) Delay Time: The time the equipment is running but not doing productive work.
3) Standby Time: The time the equipment is available for operation but not running.
4) Scheduled Down Time: The time the equipment is not operating because of scheduled maintenance.
5) Unscheduled Down Time: The time the equipment is not operating because of any failure, unscheduled maintenance, or scheduled maintenance overrun.

Once all time categories were estimated for each equipment unit, operating costs were calculated for each schedule period including fuel, maintenance parts, lube, tire replacement, ground engaging tool replacement, operator labor, and maintenance labor. If operating time for a fleet was not sufficient to accomplish the work required in the mine production schedule, additional units were added. A summary of major operating costs calculated by category is shown on Figure 16.41. Additional mine operating cost details are provided in Section 21.

Figure 16.41: Operating Costs by Category

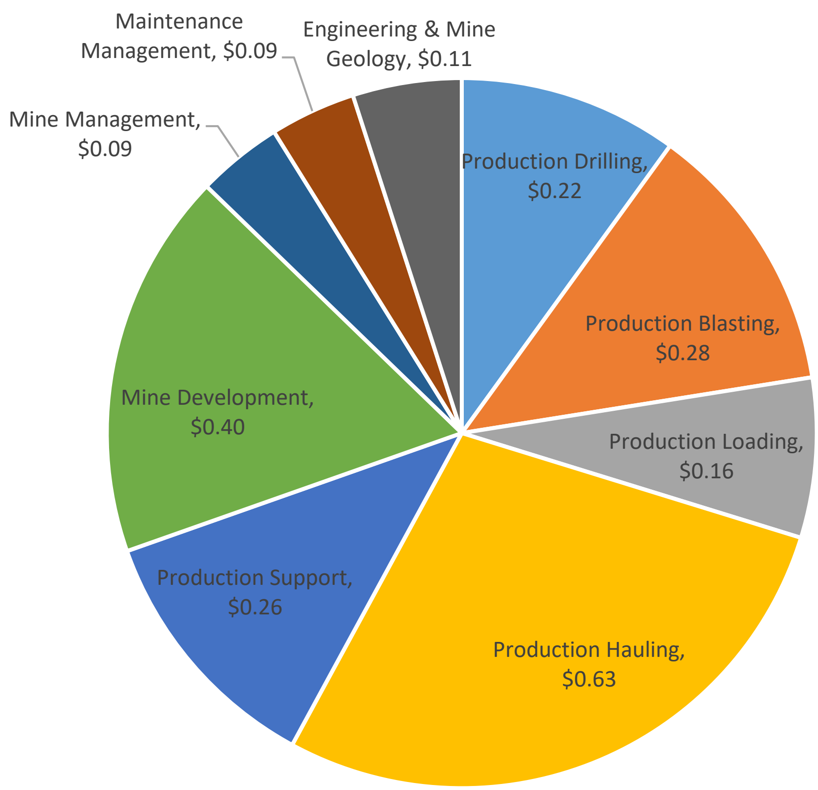


16.12.3 Ultimate Pit Limit Analysis Validation

Mining costs used for the UPLA in Section 15 were estimated based on first principal cost buildups and calculations presented in the Prefeasibility Study. Since the UPLA was a prerequisite for detailed FS mine planning, cost estimating, and the Mineral Reserve Estimate it is prudent to validate the mining cost assumptions input into the UPLA once the final FS cost estimate is completed. As stated in Section 15.2.4, the reference mining cost assumed was $2.25/st plus an incremental cost of $0.01 per 20-foot bench both below and above the pit rim for all open pits.

The average mining cost for the three open pits as calculated in the Feasibility Study is $2.24 per ton mined (Figure 16.42). This is slightly lower than initially estimated for the UPLA but similar enough to regard the selected pit shells as acceptable for guiding ultimate pit designs. The predominant factor driving a lower mining cost estimate was reduced fuel cost quotes between the time when the UPLA was conducted to when the final FS cost estimate was produced.

Figure 16.42: Mine Operating Unit Cost by Category



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17 RECOVERY METHODS

17.1 OVERVIEW

The Stibnite Gold Project process plant has been designed to process both sulfide and oxide mineralized material from three deposits (Hangar Flats, Yellow Pine, and West End) as well as Historical Tailings from former milling operations. The design of the processing facility was developed based on the laboratory testing, summarized in Section 13, to treat 8.05 million tons per year or 1,021 short tons per hour (**stph**) with a design availability of 90%.

Run-of-mine (**ROM**) material is crushed and ground and then subjected to flotation to recover an antimony concentrate of stibnite (with some silver and minor gold) and a gold-bearing sulfide concentrate of pyrite and arsenopyrite. The sulfide concentrate is processed using pressure oxidation (**POX**) to enable gold and silver to be leached and recovered to doré bars containing gold and silver. Small quantities of elemental mercury are collected in flasks to prevent its potential release into the environment. The design introduces Historical Tailings into the ball mill during the first 3-4 years of operation. Tailings from the operation are deposited in a geomembrane-lined tailings storage facility (**TSF**). A simplified process flow diagram is shown on Figure 17-1 and a list of major equipment, including the estimated connected power requirements, is shown in Table 17-1. The process facilities would be housed in several buildings and these are listed in Table 17-2.

Figure 17-1: Overall Process Flow Diagram

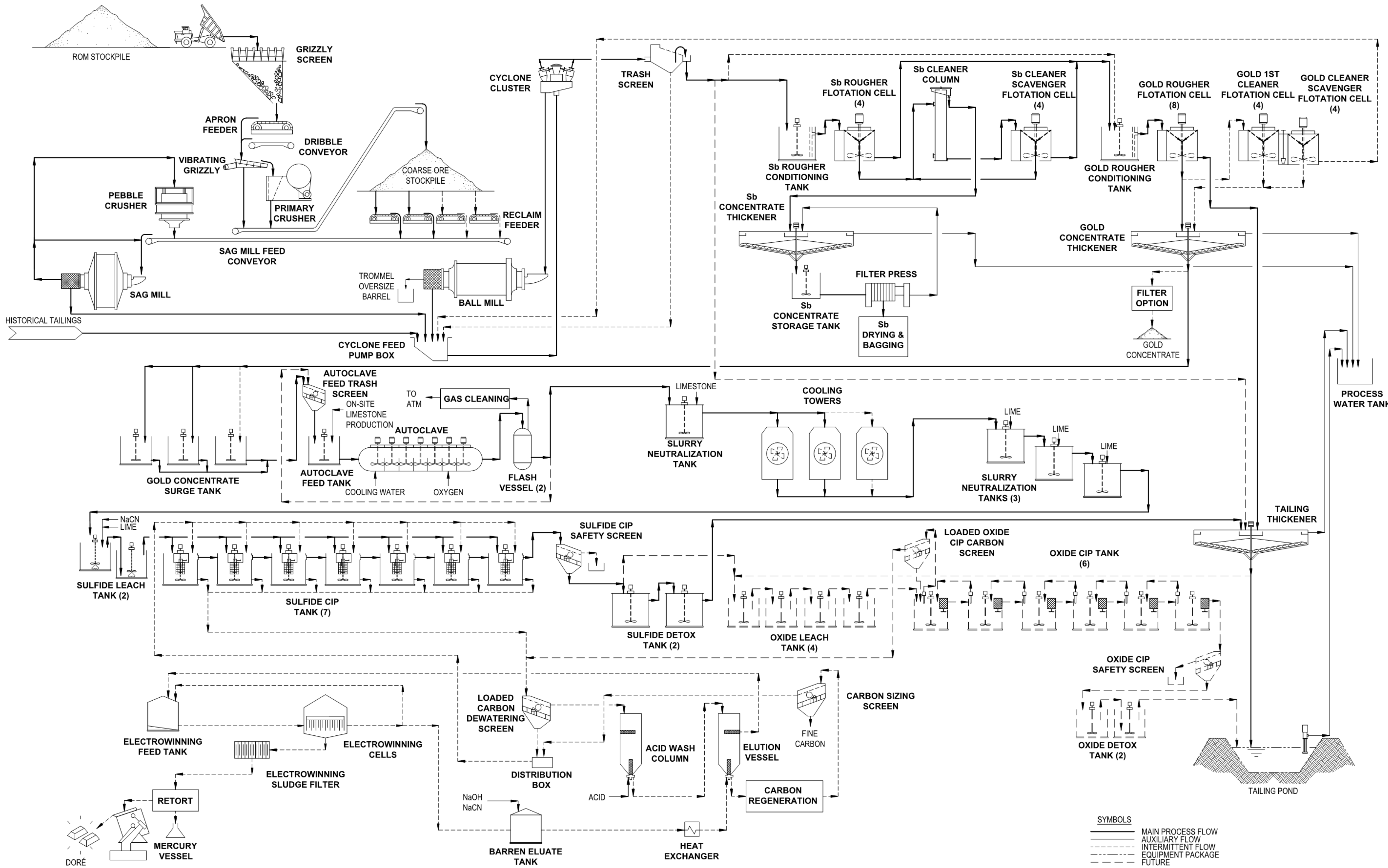

Table 17-1: Major Process Equipment List and Estimated Connected Power Requirements

Item	№	Description	Estimated Connected Power (hp)	
			Each	Total
Primary Crusher	1	Jaw Crusher; feed opening 75 x 63”	500	500
Semi Autogenous Grinding (SAG) Mill	1	30 ft diameter x 16 ft EGL, low-speed induction motor on VFD	11,390	11, 390
Pebble Crusher		Pebble cone crusher	670	670
Ball Mill	1	26 ft diameter x 44 ft EGL low speed dual-drive with synchronous motors, & clutch	11,390 per drive	22,780
Cyclone Cluster	1	16-place cyclone cluster; gMax26 type cyclones		
Sb Rougher Flotation	4	1,766 ft³ Tank Cell	75	300
Sb 1st Cleaner Flotation	1	9400 ft³, 8' diameter, 26' high Flotation Column		
Sb Cleaner Scavenger Flotation	4	706 ft³, 10.5’ diam x 12’ high Tank Cell	50	200
Gold Rougher Flotation Cell	7	22,248 ft³ Tank Cell	700	4,900
Gold Cleaner & Cleaner Scavenger Flotation Cells	8	4,591 ft³ Tank Cell	200	1,600
Gold Concentrate Thickener	1	83 ft diameter high-rate thickener	15	15
Autoclave Feed Pumps	2	Positive displacement pumps, 1768 gpm, 42 bars discharge pressure, 1 operating 1 standby	920	1,840
Autoclave	1	16.4 ft inside brick x 124.7 ft T/T inside brick, 8 cells, agitated, 5 compartments		
Autoclave Agitators	8	4 in Compartment 1, 1 each in Compartments 2 - 5	4x350, 4x150	2,000
Preheat Vessel	1	12 ft diam x 41.7 ft high T/T; 3 - 4.4 psig		
Flash Vessels	2	30 ft diameter I/S x 32.8 ft high T/T, brick lined		
Atmospheric Arsenic Precipitation Tanks (Future)	5	26 ft dia. x 28 ft high, UNS S32750 shell and bottom, UNS S31803 top cover, insulated, agitated	25	125
Slurry Neutralization Tanks	4	23 ft dia. x 25 ft high, LDX 2101 SS; Covered, Agitated	20	80
Slurry Cooling Towers	3	23 ft dia. x 39.4 ft high atmospheric cooling tower, with demister and fan; 2 operating 1 standby	73.7	221
Sulfide Leach Tanks	2	40 ft diameter x 42 ft tank height; CS, agitated	50	100
Sulfide CIP Tanks	7	27 ft diameter x 38.3 ft tank height; CS, agitated; with pump cell mechanism	50	350
Oxide Leach Tanks (Future)	4	50’ diam x 52’ height, CS, agitated	150	600
Oxide CIP Tanks (Future)	6	50’ diam x 52’ height, CS, agit, pumping screens	150	900
Carbon Regeneration Kiln	1	6 tpd carbon throughput; diesel fired;1,290 F (design temp)	11.2	11.2
Elution Vessel	1	6 ton, 4 to 1 height to diameter ratio; CS; 300° F (design temp); 100 psig		
Electrowinning Cells	2	2,500 L, 33 cathodes, 34 anodes, 2500 Amp 9v Rectifier		
Limestone Primary Crusher	1	Jaw Crusher, feed opening 42” x 28”	150	150
Limestone Secondary Crusher	1	HP 200 Standard Cone or equiv	177	177
Limestone Slurry Ball Mill	1	9.8’ x 15’ EGL overflow, SCIM on VFD	737	737
Lime Kiln	1	Maerz Vertical Lime Kiln	135	135
Lime Silo	1	37,000 ft³ bolted tank; 30 ft diameter x 52 ft cylinder height 60° cone bottom		
Lime Slaker Plant	1	Ball mill lime slaker system, 7.5’ diam x 12’ EGL	250	250
Oxygen Plant (Onsite supply contract)	1	31 (max 37) stph @ 95% purity; 82.4° F; 664 psia	14,000	14,000

Table 17-2: List of Process Buildings

Building	Construction	Dimensions, LxWxH, in feet
Primary crusher building	Steel and concrete	118 x 52 x 64 (eave)
Coarse ore stockpile	Metal dome structure on concrete ring	268 ft diameter x 99 ft high overall; metal dome only: 76.6 ft high
Grinding	Steel and concrete	238 x 101.5 x 114.25 (ridge)
Flotation	Steel and concrete	352 x 138 x 118.6 (ridge)
Autoclave wing of L-shaped building	L-shaped steel and concrete	177 x 63 x 72.7 (ridge)
Other wing (lab, steam gen and electrical room)		50 x 38.5 x 24.5 (peak)
Carbon Handling	Steel and concrete	84 x 70 x 68
Refinery (melting furnace and vault)	Masonry lower and steel superstructure	126 x 48 x 46.6 (peak)
Historical tailings repulping	Steel structure on concrete piers	77 x 48 x 57 (low ridge) 85.8 (high ridge)
Reagents Building 1	Steel and concrete	123 x 81 x 66.7 (ridge)
Reagents Building 2	Steel and concrete	183 x 83 x 71.2 (ridge)
Tailings Pumping System Building	Steel-framed supported on concrete piers	139 x 92 x 45 (eave)

17.2 MINE PRODUCTION SCHEDULE SUMMARY

A preliminary mine schedule, listing elemental concentrations of interest, is shown in Table 17-3. The mine schedule indicates that the gold, sulfur, and calcium concentrations within the Project deposits are highly variable. The material to be processed early in the life of the operation is relatively high in gold and sulfur concentration; however, after year six, the blend trends toward lower gold and sulfur but higher calcium concentrations. These variations have important implications for the process plant design.

Table 17-3: Mill Feed Schedule with Elements of Interest

Time Period	Ore (kst)	Au (oz/st)	Sb (%)	Sulfur (%)	Calcium (%)
1	7,087	0.054	0.074	1.17	1.34
2	8,070	0.061	0.221	1.01	0.99
3	8,616	0.064	0.187	0.99	0.89
4	8,905	0.072	0.101	0.95	0.75
5	8,072	0.054	0.147	1.05	1.08
6	8,050	0.045	0.140	0.99	1.27
7	8,050	0.035	0.012	0.69	2.17
8	8,050	0.034	0.001	0.56	3.88
9	8,072	0.029	0.024	0.58	3.59
10	8,050	0.032	0.001	0.49	3.44
11	8,050	0.042	0.000	0.66	4.78
12	8,050	0.037	0.001	0.77	5.73
13	8,072	0.018	0.000	0.24	3.84
14	8,050	0.021	0.002	0.67	2.95
Total / Average	113,243	0.043	0.066	0.77	2.61

During the first four years of the operation, material from Historical Tailings would be added to the higher grade, freshly mined material from Yellow Pine at a rate of about 10 - 15% of the total throughput. The Historical Tailings material is expected to average 0.03 oz/st gold and 0.3% sulfur, with a typical particle size of 80% passing 180 microns.

On average, 16% of the material shown in Table 17-3 would be processed through the antimony recovery circuit, with annual values ranging from approximately 49% in Year 2 to less than 1.2% in Year 7 and zero thereafter.

Approximately 2.6% of the material listed in Table 17-3 is oxide and responds well to conventional cyanidation, but poorly to flotation. An additional 25.2% of the material is characterized as transition material and yields variable gold recoveries by both flotation and conventional cyanidation. The remaining 72.2%, including both low and high antimony ores, is considered refractory to direct leaching but responds well to concentration by flotation followed by pressure oxidation.

17.3 PROCESS DESCRIPTION

The flowsheets developed for the Stibnite Gold Project FS are based on metallurgical test programs directed and supervised by Blue Coast Metallurgy (**BCM**) and Hydromet (Pty) Ltd. (**Hydromet**). The metallurgical testing was primarily conducted by SGS Minerals Inc. (**SGS**) and AuTec Innovative Extractive Solutions Ltd. (**AuTec**). Previous testing to support the PEA (SRK, 2012) and PFS (M3, 2014) was also supervised by BCM and conducted by SGS.

The process plant is designed to process 22,046 st/d through crushing, grinding, flotation, concentrate oxidation, leaching by cyanidation, and tailings processing operations.

Zones in both Yellow Pine (**YP**) and Hangar Flats (**HF**) contain sufficient amounts of antimony to warrant production of an antimony concentrate. Since most of the antimony occurs as stibnite, a sulfide that is amenable to flotation, a stibnite concentrate would be produced by flotation and shipped off-site for further processing.

Metallurgical testing indicates that refractory sulfides can be concentrated by flotation to recover gold and silver in the sulfide concentrate. The refractory gold in the concentrates can then be liberated by pressure oxidation, making it amenable to leaching by cyanidation.

Oxide mill feed contains gold and silver that are soluble in cyanide solutions but recovery by flotation is low. A direct cyanide leaching circuit is planned to process these ores starting in Year 7. The same circuit would be used to recover gold and silver left behind by flotation of mixed sulfide-oxide ores (transition ores). As with oxide ores, flotation of gold and silver associated with the oxide portion of transition ore is poor. This leaves tailings that may contain economic amounts of gold and silver that are amenable to cyanidation and justifying leaching of the transition ore tailings after flotation.

Once in solution, gold is separated from the leach slurry by adsorption on granular activated carbon, which, when sufficiently loaded, is eventually removed from the slurry by screening. Adsorbed gold and silver are then stripped off the carbon to a new solution, from which the precious metals are precipitated by electrolysis – a process commonly known in the industry as electrowinning (**EW**). Metallic gold and silver collect at the bottom of the EW cells as a sludge, which is filtered and melted in a furnace and finally cast as doré bars. Doré bars would be the final mine product that is sold to third party refiners.

Minor amounts of mercury are also present in the processed mill feed. The process is designed to remove the mercury that accompanies gold and transport it to a permitted off-site facility to prevent its discharge into the environment and maintain a safe working environment for employees.

Process design criteria were developed for each process area. Data used in the process design criteria are from various sources including:

- PEA (SRK, 2012);
- PFS (M3, 2014);
- client-provided historical data conducted by and for prior owners and operators of the Project;
- metallurgical testing;
- calculations;
- vendor data or recommendations;
- M3 database information;
- industry practice;
- handbooks;
- assumptions based on experience; and
- other reports and consultants.

The following sections provide a comprehensive summary of the FS process flowsheet based on the metallurgical testing and interpretation presented in Section 13; the major process equipment selected for the Project and a discussion of the alternatives considered; a description of the primary buildings required to support the major process equipment; and descriptions of the primary process support infrastructure including the water systems, process air systems and the tailings handling system. The layout of the facilities discussed in this section is discussed in Section 18.

17.3.5 Crushing and Stockpile

Haul trucks with 150-ton capacity are planned to transport mined material to the primary crusher pad for processing. The mill feed is either dumped directly into the primary crusher feed hopper or onto one of four 100,000-ton ROM stockpiles. The stockpiles allow blending of materials to control sulfide or carbonate concentrations. They also provide storage for mined materials that are segregated and accumulated for future processing as feed campaigns through the process plant.

The crushing circuit design is based on a 24-hour per day, 365-day year operation at an average utilization of 75% yielding an instantaneous design throughput of 1,225 stph. ROM material dumped onto a grizzly screen passes into the crusher dump hopper. Stockpiled material is fed to the crusher as needed for blending or campaigning using a front-end loader. The dump hopper is designed with live capacity for one haul truck. A rock breaker at the dump pocket breaks oversize materials, allowing them to pass the grizzly. An apron feeder draws material from the dump hopper to feed a vibrating screen feeder. The oversize feeds the jaw crusher and the undersize passes through to the stockpile feed conveyor. The crusher product joins the undersize on the conveyor for delivery to the coarse ore stockpile.

Belt scales are included in the design to monitor production rate – one on the coarse-ore stockpile feed conveyor and another on the SAG mill feed conveyor.

A concrete and steel structure is designed to support the primary crusher and associated equipment. Concrete piers are designed to support the concrete dump pocket, insulated metal roof and wall panels, and a 20-ton bridge crane. Water sprays would be installed at the crusher dump pocket and at material transfer points to reduce dust emissions.

The stockpile is designed with 12 hours of live capacity (11,023 st) and a total capacity of approximately 40 hours' worth of production (44,500 st). Four feeders would be provided for material reclaim to the milling circuit. A dome-shaped cover supported by a concrete ring structure is designed to reduce dust emissions and for protection from the weather. Twenty-four concrete piers 22 ft 6 inches tall at 15° spacing are designed to support the concrete ring and dome. The dome consists of coated a metal tube framing with metal roof/siding attached to the metal framing.

A concrete reclaim tunnel underneath the stockpile is designed to reclaim ROM material for conveying to the grinding circuit (Section 17.3.6). The reclaim tunnel is designed to be 20 x 20 x 160 ft with two perpendicular 38-ft tunnel segments at the center of the stockpile. Four 12 ft by 6 ft draw holes arrayed around the center of the stockpile provide feed to the SAG mill feed conveyor. The reclaim design contains two of the draw holes along the axis of the tunnel centered 46 feet from the center of the dome and two draw holes perpendicular to the axis of the tunnel centered 36 feet from the center, one on each of the perpendicular tunnel segments. The design permits stockpile material to be drawn from two of the four draw holes at a time to produce 1,021 stph of ore to grinding. Rotating among the draw holes is intended to provide even drawdown and mitigate "rat holing." Stockpile material flow from the draw holes is controlled by belt feeders at each draw hole that transfer the material to the SAG mill feed conveyor. A dust collector is designed to control dust in the reclaim tunnel.

17.3.6 Grinding

The SAG mill feed conveyor is designed to deliver reclaimed material from the stockpile to the SAG mill at an instantaneous design throughput of 1,021 stph, based on 90% availability. The grinding circuit design includes one SAG mill with a discharge trommel, one pebble crusher, one ball mill and a cyclone cluster.

The trommel screen on the SAG mill discharge returns oversize "pebbles" to the SAG mill feed conveyor after crushing. The trommel screen undersize falls into the cyclone feed pump box where it combines with the ball mill discharge. The combined slurry is then pumped to a hydrocyclone cluster for particle size classification. When historical tailings tons are processed during the early years of operation, the slurry from the tailings repulping plant would also flow into the cyclone feed pump box. Cyclone underflow flows by gravity to the ball mill for additional size reduction by grinding. The cyclones parameters will be set to achieve a target size of 80% passing 75 microns at 33% solids. Cyclone overflow is designed to flow by gravity to the flotation area after passing through a screen to remove trash.

The grinding area is to be enclosed in a steel structure supported on concrete piers with preformed insulated metal roof and wall panels. The grinding area floor is designed to be concrete on grade with containment walls to contain spills. The floor is sloped to a trench that directs spills to a sump that enables pumping back to the cyclone feed pump box.

17.3.7 Flotation

Two flotation circuits are included in the project design. One circuit produces an antimony concentrate, and the other produces a gold-rich sulfide concentrate, which is the main flotation product. Ore deliveries that are high in antimony would be processed by the antimony circuit to produce an antimony concentrate, and then proceeds to the sulfide flotation circuit to recover gold and silver in a sulfide concentrate. Low-antimony mill feed is processed in the gold flotation circuit only, bypassing the antimony circuit.

A single steel and concrete building designed to contain both flotation circuits and thickening, filtration, drying, packaging, and shipping of antimony concentrates. The steel structure is supported on concrete piers and includes insulated metal roof and wall panels. Each side of the building is equipped with a 20-ton overhead bridge cranes.

17.3.7.1 Antimony Flotation

Antimony flotation is designed to remove stibnite (Sb_2S_3) from the feed stream prior to gold flotation. Reagents are added to activate stibnite and depress the gold-bearing sulfides (pyrite and arsenopyrite) during antimony flotation. The antimony rougher flotation circuit recovers the stibnite as rougher concentrate, which is cleaned and scavenged. Antimony rougher tailings combined with antimony cleaner-scavenger tailings advance to the gold flotation circuit.

The antimony rougher operation includes one bank of four flotation tank cells with a total retention time of 4 minutes, which is 2 times the lab retention time and assumes 15% of the volume is occupied by air bubbles. The plant cell selection is a balance between the number of flotation cells in series to reduce the impact of short-circuiting, the maximum flow recommended for the flotation cells, and minimization of gold reporting to the antimony concentrate.

Antimony rougher concentrate is pumped to the antimony cleaner conditioning tank, where reagents are added for cleaner column cell flotation. Concentrate from the column is the final antimony concentrate and is pumped to the antimony concentrate dewatering. The cleaner column cell tailings are pumped to the cleaner-scavenger cells for flotation of additional stibnite. Cleaner-scavenger concentrate is returned to the cleaner column cell. The cleaner-scavenger tailings are combined with rougher tailings as feed to the gold flotation circuit.

The antimony concentrate is sampled, thickened in a 25-ft diameter thickener, filtered, dried, stored, and bagged for shipment. The antimony concentrate filter and dryer were sized based on general vendor guidelines for similar material. Dried concentrate would be stored in a bin prior to bagging and shipment.

17.3.7.2 Gold Flotation

Feed for gold flotation comes from combined tailings from the antimony rougher and antimony cleaner-scavenger flotation cells when treating high antimony ore, or directly from the ball mill cyclone overflow when treating low-antimony ore. Copper sulfate solution is added to the gold rougher conditioning tank to activate the sulfides for flotation. The conditioned pulp discharges to the gold rougher flotation bank. Gold-bearing sulfides are recovered in the rougher concentrate, which is pumped to the gold concentrate thickener. The gold concentrate thickener increases the pulp density prior to the oxidation step for efficient pulp storage and to facilitate autoclave temperature control. Thickener overflow is returned to the process water system. Gold rougher flotation tailings are pumped to the flotation tailings thickener. The gold rougher operation includes one bank of seven flotation tank cells with a total retention time of 90 minutes, which is 2.5 times the lab retention time and assumes 15% of the volume is occupied by air bubbles.

Metallurgical testing indicates that approximately 6.5% sulfide sulfur is adequate for autothermic pressure oxidation of gold concentrates (Section 13). A gold cleaning-scavenging circuit will be used when the sulfur grade of the rougher concentrate is less than 6.5%. The gold cleaner and cleaner-scavenger operation comprises eight flotation tank cells, four for cleaning and four for cleaner-scavenging, with a total retention time of 75 minutes, which is approximately 2.5 times the laboratory retention time.

Gold rougher concentrate is pumped to the gold cleaner conditioning tank where flotation reagents are added. The conditioned gold rougher concentrate then flows through the gold cleaner and gold cleaner-scavenger cells. Concentrates from both cleaner and cleaner-scavenger tanks flow to the gold cleaner concentrate pump box and pumped to the gold concentrate thickener. The gold cleaner scavenger tailings would be returned to the cyclone feed pump box for additional grinding to liberate gold-bearing sulfides or to the flotation tailings thickener.

17.3.8 Pressure Oxidation of Gold Concentrate

The pressure oxidation circuit has been designed to implement the *in-situ* acid neutralization (**ISAN**) process as discussed in Section 13. The objective of this process is to control free acid concentration in the autoclave by adding limestone slurry in the feed and, if required, by direct injection into the autoclave. Controlling the free acid concentration inhibits the formation of basic ferric sulfate and promotes ferric sulfate (scorodite) formation. The lower free acid in the autoclave discharge also allows direct neutralization without an acid-wash counter-current decantation (**CCD**) circuit, thus simplifying the flowsheet.

The product of the sulfide flotation process is a concentrate that comprises gold-containing pyrite, arsenian pyrite, arsenopyrite, other sulfides, and gangue materials. Gold and silver in this concentrate exhibit poor recovery by cyanide leaching alone because of encapsulation within the sulfide minerals. Oxidation of the sulfides breaks down their crystalline structure making the precious metals available for cyanide leaching and recovery. Several alternatives for oxidizing the sulfide concentrate were considered in the PFS (M3, 2014). POX using an autoclave with high pressure oxygen was selected as the preferred alternative for its industrially proven reliability and environmental performance.

Underflow from the gold concentrate thickener is pumped through a trash screen to the concentrate surge tanks. Three concentrate surge tanks provide approximately 36 hours of live storage for surge capacity and for blending to buffer variabilities in the sulfide sulfur and carbonate contents of the concentrate. Under normal operating conditions, each tank is filled or emptied (to feed the autoclave) independently. At any time, one tank would be feeding the autoclave, a second tank would be full and awaiting sulfide sulfur and carbonate assays, and a third tank would be receiving concentrate slurry. Once assays become available, limestone slurry is added to the second tank to attain a carbonate-to-sulfide ratio of approximately 1.25:1.

The concentrate slurry is pumped through the concentrate preheater where steam from one of the autoclave flash tanks heats it up in preparation for injection into the autoclave. A density adjust tank is used to attain the design pulp density of 35-40% solids before being pumped to the autoclave feed tank.

Gold concentrate slurry at a target sulfide sulfur grade of 6.5% is pumped into the autoclave at high pressure to overcome the pressure in the autoclave. Two independent trains of two pumps in series are needed to accomplish this. The first stage of each train is a centrifugal booster pump to supply the second-stage positive displacement pump, which provides the necessary injection pressure. During normal operation, both pump lines would be in operation. When one pump line is down for maintenance, the second pump line would continue to operate with a maximum capacity of 75-80% of required volume.

The autoclave is designed to operate at 428 ºF (220 ºC) and 622 psig (4,289 kPag). Steam generators are provided for the initial heat up of the autoclave to operating temperature with direct injection of steam. The preheated concentrate slurry would be pumped into the first and largest compartment of the autoclave containing four agitators at 35-40% solids. Oxygen is injected into the autoclave at an overpressure of 100 psi. Discharge from the first compartment flows through the remaining four compartments in series, each with one agitator. The oxidized slurry exits the autoclave to two flash vessels that are operating in parallel. Slurry discharge from each flash vessel flows by gravity to a cooled, agitated discharge tank with a live retention time of 3 minutes.

The autoclave vapor phase is vented at the first compartment to prevent carbon dioxide buildup that would displace oxygen in the vapor phase. The vent gasses are directed to the autoclave vent scrubber for treatment.

Slurry discharged to the flash vessels rapidly depressurizes, producing a vapor phase mostly composed of steam. Vapor-phase discharge from flash vessel No. 1 flows to the autoclave flash scrubber. Vapor-phase discharge from flash vessel No. 2 flows to the concentrate preheater.

The autoclave vent scrubber treats gasses from the autoclave vent with hydrogen peroxide and sodium hydroxide to condense the steam and convert any entrained hydrogen sulfide gas to sulfate. From the vent scrubber, the gas goes to the autoclave vent condenser where it passes through a vent gas cleaning tower, steam condensation tower, and vent gas mercury cleaning column. The mercury cooling column is packed with sulfur-impregnated carbon, which collects any remaining mercury in the off gas before discharge to the atmosphere. Condensed steam is recycled to the process water tank.

The autoclave flash scrubber receives the vapor-phase discharge from the No. 1 flash scrubber and off-gas from the concentrate preheater. The off gasses would pass through a steam cleaning tower and flash steam mercury cleaning columns before being released to the atmosphere through the flash steam condenser stack.

The designed nominal retention time in the autoclave is 75 minutes. The oxygen utilization used in the design was 85%, although the point of injection and mixing parameters warrant a higher value. Quench (cooling) water is pumped to the autoclave at flow rates required to control the autoclave temperature. A vendor-operated oxygen plant is designed to supply oxygen gas at 95% purity.

Sizing of the autoclave is based primarily on the rate that sulfide sulfur is fed to the autoclave. The autoclave design is based on a sulfide sulfur feed rate of 11.54 stph. At the stoichiometric requirement of 1.87 tons of oxygen per ton of sulfide sulfur, 21.6 stph of pure oxygen is required, which requires 26.7 stph of gas supply at 95% purity and an estimated utilization rate of 85%.

Design conditions of 75 minutes of retention time and cooling water addition for process control result in a live reactor volume of 22,425 ft^3. The total reactor volume allowing for head space and an estimated operating level of 81% results in a total volume of 27,722 ft^3. The resulting internal dimensions of the autoclave are shown in Table 17-1.

The autoclave building is an L-shaped steel and concrete structure supported on concrete piers and supports preformed insulated metal roof and wall panels. The building is equipped with a 10-ton overhead bridge crane. The larger wing of the L-shaped building houses the autoclave and supporting tanks and vessels. The other wing houses the site assay lab, the steam plant, and an electrical room.

17.3.9 Atmospheric Arsenic Precipitation (Future)

Arsenic levels in the mill feed are expected to increase during the life of mine. The atmospheric arsenic precipitation (**AAP**) circuit would be installed in Year 6 to be operational in Year 7. This circuit would treat the autoclave discharge to slowly precipitate iron and arsenic at 92 ^{o}C by progressively adding limestone to achieve a pH of approximately 2. This process would be carried out in 5 agitated tanks in series, with a total retention time of 5 hours. Each tank would be covered and insulated to maintain temperature. An intermediate slurry heater would be installed to compensate for heat loss, using POX flash steam as the heat source.

17.3.10 Slurry Neutralization and Cooling

POX discharge slurry needs to be neutralized to pH 10.5 before being subjected to cyanide leaching. This is achieved in four agitated tanks in series, each designed with a 30-minute retention time. Air is sparged through the tanks to facilitate removal of carbon dioxide evolved during the reaction. The discharge gas is routed to the neutralization circuit vent scrubber.

Metallurgical tests showed that neutralization must be carried out in two stages to preserve the stability of the ferric arsenate precipitate. Neutralization to pH 4.5 can safely be done at high temperatures. This is achieved in Neutralization Tank 1 with limestone added as a slurry. Neutralization from pH 4.5 to pH 10.5 requires prior cooling to 113 ºF (45 ºC) or lower. The slurry is therefore pumped from the first tank to the slurry coolers.

Slurry cooling is accomplished in two forced draught cooling towers with a third unit on standby. A cooling tower comprises a vertical cylindrical body with spray nozzles, a drift eliminator, and a horizontal cylindrical duct fan at the bottom. The slurry is fed under pressure to a spray manifold on top of the cooling tower. Several nozzles spray the slurry down into the tower against a counter-flowing air stream that cools the slurry down. The cooled slurry is collected in a basin at the tower's base.

The cooled slurry is pumped back to the second neutralization tank to continue pH adjustment to pH 10.5 with milk of lime added to Neutralization Tanks 2, 3, and 4. Once fully neutralized, the slurry is then pumped to the cyanide leach circuit.

17.3.11 Sulfide Leaching, Recovery, and Detoxification

Neutralized POX discharge is leached with cyanide solution in leach tanks causing the gold and silver to go into solution. Two leach tanks in series are designed to provide 3 hours of residence time each. The tanks are positioned to permit gravity flow from one tank to the other.

After leaching, the pulp flows by gravity to the CIP tanks containing activated carbon that adsorbs the precious metals in the leach solution. Seven Kemix pump-cell CIP tanks in series are used to process the leached concentrate. These tanks have built-in mechanisms that pump slurry from tank to tank such that the tanks can be all be built at the same level.

The CIP tanks are in the carousel arrangement where, instead of carbon moving upstream from tank to tank, the positions of the lead tank (first tank) and lag tank (last tank) move around the carousel. Slurry from the last leach tank flows to the CIP feed distribution box, where it is directed to the lead tank by a dart valve system. Carbon remains in each tank, while the pulp is moved to the next tank by the pumping action of the Kemix pump cell until it reaches the lag tank.

When the carbon in the lead tank is fully loaded, the pulp flow is redirected to the next tank downstream, which becomes the new lead tank. The former lead tank is drained, and the loaded carbon pumped out to a screen to separate it from the pulp, which is returned to the CIP distribution box. The loaded carbon is then sent to the carbon handling circuit. The empty tank then receives slurry from the former lag tank and barren activated carbon, to become the new lag tank.

Pulp from the lag vessel is pumped to the detoxification tanks and treated to reduce the cyanide concentration prior to discharge to the TSF. Weak acid-dissociable (**WAD**) cyanide is oxidized to cyanate using sodium metabisulfite and air, the efficacy of which was demonstrated by the laboratory results presented in Section 13. Copper may be added to catalyze the detoxification process, but adequate copper is expected to be present in the slurry. The slurry pH is maintained in the range of 8-9 by adding milk of lime. Air is sparged into the detoxification tanks below the agitators to maximize dispersion and dissolution of oxygen in the slurry. Two tanks arranged in series, provide a retention time of approximately 2 hours for the detoxification operation. Either tank can be fed directly or bypassed for maintenance without significantly affecting the performance of the process.

The detoxification circuit is designed to reduce cyanide concentrations in the tailings slurry to less than 50 ppm WAD cyanide before being pumped to the TSF. This concentration maximum is based on guidance from the International Cyanide Management Institute (**ICMI**) as a concentration protective of animals from the adverse effects of cyanide-bearing solutions. A lower WAD cyanide concentration in the reclaim water is desirable to maintain efficiency of the flotation process. The detoxification process as designed is likely to routinely achieve considerably lower concentrations than the ICMI target. Natural oxidation by UV radiation from sunlight will provide additional detoxification in the TSF to further lower WAD concentrations in the process water reclaimed from the TSF.

17.3.12 Carbon Handling and Refining

The carbon handling system processes loaded carbon to produce doré bullions. This involves recovering gold and silver from loaded carbon by desorption, electrowinning, and refining.

The loaded carbon is screened and washed using a single deck vibrating screen when it arrives at the carbon handling facility. The screen oversize (carbon) flows to the acid wash column and screen undersize (slurry) is returned to the CIP circuit for recovery of soluble metals. The acid wash and elution vessels are each designed with a carbon capacity of 6 tons to accumulate a daily gold loading of 100-200 oz/st.

Wash solution containing nitric acid is circulated from the acid wash circulation tank through the acid wash column to dissolved scale that has accumulated on the carbon. At the end of acid wash cycle, carbon is rinsed with fresh water, to remove excess acid. The rinse solution flows to the neutralization tank where it is neutralized with sodium hydroxide (caustic) solution before sending it to the cyanide detoxification tank or to the process water tank. An exhaust fan removes any hydrogen cyanide gas that may be generated from the acid wash circulation tank and acid wash columns.

Acid-washed carbon is pumped to the elution vessel for stripping precious metals using the modified pressure Zadra method with an eluant solution of sodium cyanide and sodium hydroxide. The eluant is pumped from the strip solution tank through heat exchangers to the bottom of the elution vessel at 65 psig and 290 ºF and a flowrate of 2 to 4 bed volumes per hour. Solution exits at the top of the elution vessel to a pregnant eluate tank.

The eluant solution is heated in two steps using plate-and-frame heat exchangers. The first step is heat exchange between the hot pregnant eluate exiting the elution vessel and the eluant. The second step is between the eluant and hot water generated by a propane-fired water heater. The second-step heat exchanger is the main or primary heat exchanger that takes the solution to operating temperature.

Pregnant eluate is pumped from the pregnant eluate tank to the electrowinning feed tank, from where the solutions flows by gravity to two electrowinning cells in series, each served by a 2,500-amp rectifier. Gold and silver are electrolytically reduced to their metallic state at the EW cathodes, with some of the precipitates falling off to the bottom of the EW cells to form a sludge. At the end of the process, the barren eluate flows to the barren eluate tank and pumped back to the strip solution tank to complete the circuit.

To protect the operators, the electrowinning cells are covered and exhausted to an electrowinning demister followed by a bed of sulfur-impregnated carbon to remove mercury in the exhaust gas. The cleaned exhaust is then blown out to the atmosphere.

Precious metals that collect in the electrowinning cells are periodically harvested as a sludge by pressure washing the stainless-steel cathodes. The precious metal sludge is pumped to a filter press, then dried in a retort furnace system. The retort also volatilizes mercury, which is condensed in a chiller and stored onsite in metal flasks prior to being sold or shipped to a secure disposal site. Off gas from the mercury recovery chiller then goes through a particulate filter and finally trough a bed of sulfur-impregnated carbon before being discharged to the atmosphere.

Dried sludge from the retort is mixed with flux and melted in the gold furnace. The molten precious metals are poured into doré bars, which are stored in a vault in the refinery building. Off gasses from the induction furnace are drawn by the furnace exhaust fan through a baghouse, HEPA filter, and carbon adsorption vessel.

Stripped carbon from the elution vessel is pumped to the kiln feed screen to remove undersized carbon. Screen oversize is accumulated in the kiln feed bin and fed to the carbon regeneration kiln. The kiln heats the carbon to 1,290 ºF for 10 minutes. The heat-treated carbon exits the kiln into a water quench tank. The carbon then undergoes another screening to remove fines before it is returned to the lag CIP tank in the Sulfide CIP circuit or the last tank in the Oxide CIP circuit.

The carbon handling and refinery building was designed as a composite building divided into two areas. The carbon-handling area houses the acid wash column, strip vessel, carbon regeneration kiln, and all the tanks and vessels. The refinery area contains the electrowinning cells, mercury retort, and gold furnace. The refinery area building is designed with a masonry lower floor with a steel superstructure with a shed roof conjoined with carbon-handling area. The masonry section provides additional security for the precious metal handling and refining area, including the doré vault.

17.3.13 Leaching of Oxides or Flotation Tailings, Recovery, and Detoxification

Evaluation of cyanide leaching of flotation tailings from the Yellow Pine and Hangar Flats pits indicate that it is uneconomic. However, the West End orebody contains oxidized and partially oxidized (transition) materials that could enhance the economics of the Project by leaching flotation tailings in the case of transition ore, or leaching the ore without flotation (whole-ore leach) in the case of oxide ore, as described in Section 13. Thus, a tailings/oxide leaching and CIP recovery circuit is planned for construction in Year 6 to be operational in Year 7 to coincide with the initiation of mining from the West End pit.

When processing transition mill feed, the pulp from the sulfide CIP circuit is to be combined with conditioned flotation tailings (from the flotation tailings thickener) into the tailing/oxide CIP circuit. Mixing the two streams extends leaching of the POX residue and utilizes its residual cyanide in the flotation tailings leach. Underflow from the tailings thickener is conditioned with milk of lime in a conditioning tank. The conditioned slurry then flows by gravity to a series of four tailing/oxide leach tanks. After leaching, the slurry flows to six CIP tanks in series. The aggregate retention time is approximately 12 hours between the leach tanks and CIP tanks. Additional cyanide solution and compressed air are added to the tailing/oxide CIP tanks to facilitate gold recovery.

Each CIP tank would be equipped with a Kemix-type pump cell with integrated carbon screen to advance the pulp to the next tank in series. Barren carbon from the elution circuit is added to the last CIP tank and advanced from tank to tank by carbon advance pumps installed in each of the tanks. In the case of processing transition material, the loaded carbon from tailing/oxide CIP stage is advanced to CIP carbon holding tank and on to the sulfide CIP tanks.

When processing oxide material, the entire slurry from the grinding circuit is directed to the tailings thickener where it is dewatered. Thickened oxide slurry is pumped to the oxide/tailings conditioning tank and combined with lime and cyanide. The conditioned oxide slurry flows through the leach and CIP tanks in the same manner as the combined POX residue and flotation tailings described above. When processing oxide material, the loaded carbon is pumped directly to the loaded carbon screen in the carbon handling area in advance of acid washing and elution.

Fugitive carbon is scavenged from tailing/oxide CIP pulp discharge with single-deck vibrating safety screen and collected in a safety screen bucket. The pulp passing through the screen flows by gravity to the tailing/oxide detoxification circuit to reduce the cyanide concentration prior to discharge. The tailing/oxide detoxification circuit is larger and distinct from the sulfide detoxification circuit to handle the larger (4.5 times) flow volume.

Two tanks in series each are required to provide the design retention time of approximately one hour for the detoxification operation. The process uses the sodium metabisulfite-air method process described in Section 17.3.11 to destroy unused cyanide in solution. The detoxification circuit is designed to reduce cyanide concentrations in the tailings slurry to less than 50 ppm WAD cyanide before being transported to the TSF.

17.3.14 Historical Tailings Reprocessing

Historical tailings from processing of Yellow Pine ores in the World War II era underlie leached material in the spent ore disposal area (**SODA**). Metallurgical testing indicates that the Historical Tailings contain gold and antimony, as detailed in Section 13, which can be recovered economically. The waste leached material from SODA will be removed and used as construction material for the TSF, exposing the Historical Tailing. Approximately 3 million tons of Historical Tailings are planned for reprocessing within the first 4 years of operation. The repulping facility is designed with an instantaneous throughput of 124 stph with an estimated availability of 75%.

A study conducted for the PFS (M3, 2014) indicated that the economically and environmentally best method to re-pulp the Historical Tailings involves excavation of the material, trucking to a screening plant for re-pulping, and pumping the slurry to grinding circuit. In accordance with this method, the material is excavated and hauled a short distance by truck to the Historical Tailings handling facility. The material is dumped onto a grizzly screen and passes through into a feed hopper. An apron feeder feeds the tailings to a vibrating screen. Screen oversize drops to a conveyor that stacks the trash for periodic removal. Water sprays onto the vibrating screen facilitate re-pulping of the tailings into a slurry, which flows into a sump as is pumped in a dedicated pipeline to the grinding area of the plant. The re-pulped tailings slurry is discharged to the cyclone feed box upstream from the ball mill. Water for the Historical Tailings re-pulping system is designed to come from the TSF reclaim water pipeline.

The Historical Tailings re-pulping process is enclosed in a building comprising a steel structure is supported on concrete piers and includes insulated metal roof and wall panels. The building is backed by a masonry stabilized earth (**MSE**) wall and backfilled slope to provide ramp access to the dump pocket at the top of the building. The dump pocket is protected by a roof and walls on three sides. The building is equipped with a 1-ton overhead bridge crane.

17.3.15 Reagents

Reagents required for various aspects of the SGP process are housed in two primary areas. Reagent Building 1 is located next to the flotation building and contains reagents primarily used in flotation. Reagent Building 2 is located on the south side of the plant area and contains reagents associated with gold recovery. A third reagent area has been added to the FS to produce limestone and lime for neutralization for the process.

17.3.15.1 Limestone and Lime

A significant change in reagents from the PFS (M3, 2014) is the use of ground limestone slurry to moderate acid generation in the autoclave. Limestone from the Middle Marble formation would be mined from the West End pit, crushed, screened, and ground to make limestone slurry. A coarse fraction of the crushed limestone would be designated as feed for a vertical lime kiln to provide the lime necessary to increase the pH of solutions and slurries as needed in the process.

Mined limestone would be trucked to a stockpile south of the primary crusher stockpile in 40-ton articulated haul trucks. Stockpile material would be fed to a dedicated jaw crusher. Crusher discharge would be conveyed to a sizing screen from which oversize and undersize material would be produced.

17.3.15.2 Process Reagent Mixing and Storage

Reagents requiring handling, mixing, and distribution systems are summarized in Table 17-4. The table also includes estimated reagent consumption rates for full-scale plant operation, which have been estimated based on metallurgical testing results.

The dry reagents would be stored under cover, then mixed in reagent tanks and transferred to distribution tanks for process use. The reagent building would be a steel-framed structure with metal roofing; metal siding would be installed to keep reagents dry and protected from the sun. The floors would be slab-on-grade concrete with concrete containment walls to capture spills.

Table 17-4: Estimated Primary Reagent Consumption Rates

Reagent	Use in Process Plant	Yellow Pine		Hangar Flats		West End			Historical Tailings
		High Sb	Low Sb	High Sb	Low Sb	Sulfide	Oxide	Transition	Low Sb
		lb/ton ore	lb/ton ore	lb/ton ore	lb/ton ore	lb/ton ore	lb/ton ore	lb/ton ore	lb/ton ore
Ground Limestone ($CaCO_3$)	Acid neutralizer during POX	47.4	47.4	47.4	47.4	10.3			47.4
	Acid neutralizer in POX discharge	17	17	17	17	10			17
Pebble Lime (CaO)	Pyrite depressant	0.60		0.68					
	POX leach and leach tails detox	8.7	8.7	8.7	8.7	5.2		0.26	8.7
	Oxide/tails leach and combined tails detox						4.2	4.2	
Lead Nitrate ($Pb(NO_3)_2$)	Antimony activator	0.40		0.5					
Aerophine 3418A	Antimony collector	0.030		0.02					
Copper Sulfate ($CuSO_4$)	Sulfide activator	0.30	0.20	0.20	0.20	0.30		0.20	0.30
Potassium Amyl Xanthate (PAX)	Sulfide collector	0.41	0.26	0.31	0.26	0.27		0.27	0.41
Methyl Isobutyl Carbinol (MIBC)	Frother	0.11	0.09	0.08	0.05	0.08	0.00	0.09	0.06
Sodium Cyanide (NaCN)	Gold and silver complexing agent, pyrite depressant, strip solution makeup	1.0	0.80	1.0	0.80	0.80	0.80	0.76	0.80
Flocculant, tailings	Promote settling	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06
Flocculant, conc.	Promote setting (lb/ton conc)	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05
Activated Carbon	Recover soluble gold and silver	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10
Sodium Metabisulfite ($Na_2S_2O_5$)	Cyanide detoxification of POX tailings	0.031	0.031	0.031	0.031	0.031		0.031	0.031
	Cyanide detoxification of float tailings/oxide leach					0.43	0.43	0.43	
Nitric Acid (HNO_3)	Descale activated carbon	0.08	0.08	0.06	0.06	0.05	0.05	0.05	0.04
Caustic (NaOH) (sodium hydroxide)	Strip solution makeup and neutralization of spent acid from carbon acid wash	0.07	0.07	0.06	0.06	0.05	0.05	0.05	0.05

17.4 WATER SYSTEMS

Two types of water systems are required for the Stibnite Gold Project process plant: fresh water and process water. Fresh water for the Project would be supplied from multiple sources including wells and contact water ponds. Groundwater wells located within the Meadow Creek valley alluvial deposits may contain elevated concentrations of metals and are considered to be the equivalent of contact water. Contact water includes seepage from storage piles and runoff from mine-impacted areas. Contact water from various sources would be pumped to the freshwater tank, which also serves as the firewater tank. Fresh water in the tank would be distributed to and used for:

- the freshwater distribution system;
- process water makeup;
- the firewater pipeline loop;
- the gland seal water tank and pumped by horizontal centrifugal pumps to be used as seal water for mechanical equipment;
- the mine water trucks to be used in road dust control; and
- the process uses points (e.g. crusher dust suppression, reagent mixing, etc.).

Process water would be reclaimed from several locations and returned to the process water tank. Overflow from the neutralization thickener, gold concentrate thickener, and the antimony concentrate thickener would be pumped to the process water tank. Water reclaimed from the TSF, contact and stormwater ponds, and condensate from autoclave flash steam and vent gas would also be pumped to the process water tank. Recirculating autoclave condensate would important to controlling the potential mercury content by recycling it through the autoclave and CIL circuits where contained mercury would contact carbon. The mercury adsorbed on carbons would be recovered in the mercury retort gas cleaning system.

Water obtained from condensation of steam from the autoclave vent and flash tanks would also be recovered. Because of its potential mercury content, the condensate use will be maximized within the autoclave and leach/CIL areas where all solutions eventually contact activated carbon. Mercury would be recovered in the mercury retort gas cleaning system.

17.5 PROCESS AIR SYSTEMS

Several of the agitated process tanks require injected air provided by blowers, including flotation cells, neutralization tanks, conditioning, leach tanks and CIP tanks. The flotation column air spargers run at a higher pressure (around 70 pisg), which would require a compressor and air receiving tank. Each of these systems has a dedicated blower or compressor and an installed spare to provide the necessary volume and pressure of air for the process.

Gaseous oxygen is provided to the autoclave at pressure of 664 psig to facilitate oxidation of the sulfides to liberate the precious metals. The oxygen would be supplied from a vendor-supplied oxygen plant located near the autoclave building.

17.6 TAILINGS HANDLING SYSTEM

M3 conducted a study to evaluate the methods to pump the tailings from the process plant to the TSF. The design basis involved pumping approximately 6,000 gallons per minute of tailing, with 55% solids and a specific gravity of 1.53, a vertical distance of 400 feet (starter dam) to 630 feet (final dam) and a horizontal distance of approximately 11,520 feet (starter dam) to 10,830 feet (final dam). Capital and operating costs for horizontal centrifugal and positive displacement pumps were compared and the centrifugal pumps were selected on the basis of lower life-of-mine cost, primarily due to lower initial capital cost. Various pipe types and configurations were evaluated in terms of calculated pressure and friction losses. HDPE-lined carbon steel pipe was selected for the tailings pipe from the process plant to the TSF because it was the lowest cost and best alternative that could handle the pressure and reduce friction losses.

The tailings would be pumped using six horizontal centrifugal pumps connected in series to lift the tailings to the starter dam crest elevation of approximately 6,850 feet AMSL. Six spare pumps would be installed in series to enable continued pumping if one of the pumps in the initial series should fail. The tailings would be transported in HDPE-lined carbon steel piping 24 inches in diameter in a lined trench or, when buried, in a containment sleeve. The pipeline is routed west from the thickener and crosses EFSFSR after approximately 1,500 feet. The pipeline routing then parallels the waste haulage road and then climbs up the slope on the northern side of the Meadow Creek valley, parallel to the surface water diversion around the development rock storage facility. Additional information on the configuration and management of the TSF is provided in Section 18.

Supernatant from the TSF would be reclaimed and pumped via three barge-mounted vertical turbine pumps and pipeline to the process water tank located in the process plant area; the reclaim water pipeline would share the same secondary containment as the tailings pipeline. The TSF impoundment must be raised periodically to provide additional tailings capacity; the tailings pipeline would be relocated and extended to accommodate these raises. One additional pump and one spare would need to be added to the tailings pumping system as the TSF dam rises to its ultimate height of approximately 7,080 ft AMSL.

The initial routing of the pipeline (and waste haulage road) transects the ultimate Hangar Flats open pit and must be moved to circumvent the pit when mining begins to encroach; Meadow Creek also has to be realigned since it transects the ultimate Hangar Flats pit. The pipeline, road, and Meadow Creek diversion would all be moved concurrently to be outside of the ultimate Hangar Flats pit.

The tailings pumping system would be housed in a steel-framed building supported on concrete piers with preformed insulated metal roof and wall panels. There is an overhead bridge crane for pump maintenance.

17.7 PROCESS CONTROL SYSTEMS

The Stibnite Gold Project process plant design includes an integrated process control system consisting of three tiers of control and monitoring systems. A conceptual description of the control architecture is provided below, followed by a conceptual control philosophy that depicts the level of automation and the principles that would guide decisions concerning instrumentation and control design in the next phase of this Project.

17.7.5 Process Control Architecture

Process control for the process plant would be accomplished by a multi-tiered monitoring, control, and recording system using an Ethernet backbone. The fiber optic network would be arranged in dual self-healing ring configuration for redundant peer-to-peer communications and control. The redundant fiber optic communication modules protect the integrity of the Ethernet network by maintaining network communications, even with a failure of a fiber path. The functions of the network include data collection and control on a single high-speed network, with tie-in to the plant management system. The devices on the network include servers, workstations, switches, Programmable Logic Controllers (**PLC**s), and Human-Machine Interfaces (**HMI**s).

The control system consists of three levels of control: local control, PLC control, and Process Control System (**PCS**). Local control of each piece of driven machinery is from a local hand control station, typically a station with Start and Stop pushbuttons. Field Stop pushbuttons are hard-wired directly to the motor control centers (**MCC**) to operate independent of the control system or selector switch position. Likewise, personnel safety features, such as conveyor pull cords, are directly connected to the motor controls. Each piece of driven machinery is equipped with a Local/Off/Remote selector switch located in the MCC. The selector switch is arranged to provide bump-less control between the local Start/Stop pushbuttons when in the Local position, and the PLC control system when in the Remote position.

PLCs control the process equipment when the local control switch is in the “Remote Mode” and provide monitoring and control of the equipment. PLCs are accessible to both field operators and operators in the control rooms. The PLC system would monitor the status of all local controls to supervise operations and alarm the operator of any anomalies in the system’s configuration.

The PCS integrates the system components, from the device-level communications and control, to the Ethernet networks and higher-level business systems. It incorporates redundant virtual servers and operator workstations into the network to enable operators in the mill control room, crusher control room, and in other designated control stations throughout the site to monitor and control the various component processes. These workstations would be configured to access the process screens and data associated with their specific process area. Two large screen monitors installed in the mill control room provide a process overview. Access to historical process records is provided by the historian server. An engineering workstation is installed and configured with access to all process interface screens, as well as the software required to provide system configuration and maintenance.

17.7.6 Process Control Philosophy

The process plant would incorporate modern, dependable and proven instrumentation and control systems. The monitoring and control systems would support the operation of the plant under the following parameters. The plant would operate on a two 12-hour shift per day basis. Planned maintenance shutdowns would take place on a regular basis. The plant would have an overall operating availability 90%, with lower availabilities for the crusher (75%). There are no holiday and/or other planned work stoppages during the calendar year. The maintenance of the monitoring and control systems would be performed in accordance and support of this operating and maintenance schedule.

The mill building control room would serve as the center for communications, fire systems monitoring and emergencies in general. The control room would be manned on a 24 hour-a-day basis. A base station radio would be assigned to the control room as well as an outside telephone line. The control room would also have the ability to communicate on all other site group frequencies. The control room operator would also have access to the company computer network and e-mail system.

Real time observation of strategic points along the operation would be by a TV camera system with monitors in the control room. PLC systems would be used for controlling the plant equipment. Proper graphic displays would be developed for the PLC systems. The control room would serve as the center of all control and recording of key process variables, outputs, functions and plant stoppages.

Safety systems would include, but are not limited to the following:

- The use of start-up warnings – horns, sirens or some other means – would be used throughout the property.
- Applicable interlocks would be used to protect people and equipment.
- All fire protection systems and fire detection systems would be monitored from the mill control room.
- Interlocks and/or other safety related protection would either be hard wired or in control logic depending upon which offers the greatest level of assured safety.

Real-time process control and monitoring systems that provide data to the operators would include but are not limited to the following.

- Instrumentation on the primary crusher would provide data on power draw, weigh scale on stockpile feed conveyor, crusher discharge hopper level indicators, etc. The primary crusher would also have a tramp iron magnet and an appropriate metal detector.
- Coarse ore stockpile would have a height measuring device and the reclaim conveyor would have vendor supplied variable speed controls for each feeder.
- Each reagent system would have the ability to be batched to the necessary strength and stored until used in the plant. The delivery systems would have the ability to be measured and controlled from the plant control room.
- The grinding area instrumentation would include the SAG mill feed conveyor weight scale, water and reagent control to the SAG mill, tramp steel magnet, cyclone feed sump levels and auto water addition to the sump, pulp densities for the cyclone feed pump discharge as well as cyclone pressure, and the ball mill power draw and automatic water addition. Both grinding mills would have the vendor supplied controls, interlocks and monitors to protect the equipment.
- The flotation circuits have on-stream X-ray analyzers. The following streams would be automatically sampled and analyzed: rougher flotation concentrate, rougher scavenger flotation concentrate, rougher tailing, first cleaner scavenger flotation tailing, and 2nd cleaner flotation concentrate. Flotation sumps would have level indicators and automatic valves for water and/or reagents where applicable. Flotation cells would have the vendor supplied packages to allow level control and other needed instrumentation normally associated with their product. Thickeners would have torque indicators with adjustable height rakes and automatic valves on the thickener underflow pumps.
- The antimony filter would have all typical vendor-supplied instrumentation. A truck scale would be necessary in order to weigh antimony concentrate prior to leaving the site. An automatic wheel wash system would be needed to ensure environmental requirements are met.
- The pressure oxidation process would be controlled by a PLC housed in the mill control room. The PLC would log the sulfur content, carbonate content and monitor the density of the slurry in the autoclave feed tanks, and monitor the pressure and temperature in the autoclave. Based on those measurements, the PLC would adjust the water and oxygen additions to the autoclave and venting of CO_2 to the flash vessels.
- The oxygen plant would be vendor supplied and vendor operated. Appropriate operating characteristics and alarms would be transmitted to the mill control room through the Ethernet.
- Slurry density, temperature and pH are monitored in the CCD process to enable the PLC to control addition of wash water and lime in the neutralization and leach pre-conditioning tanks.
- Cyanide concentration would be manually monitored and adjusted.
- Reagent addition in the detoxification tanks would be automatically metered by the PLC using monitoring information from the CIP/CIL tailing.
- The ADR plant would have vendor-supplied instrumentation and controls operated by plant personnel. Key operating parameters would be monitored by the PCS in the mill control room.
- The neutralization thickener would have a torque indicator and adjustable lift rakes. All typical vendor-supplied indicators and systems are anticipated. Thickener underflow and recycle systems would have automatic valves and a flow and density meter.
- The tailings system would have horizontal centrifugal pumps and would have remote start and stop control capability from the mill control room.
- The TSF reclaim water barge would have vertical turbine pumps with remote stop and start capabilities from the mill control room. Each pump would receive a control signal from the reclaim water storage tank. The

reclaim water storage tank would have a level indicator and an automatic control on the anti-scalant addition line.

Process control and monitoring systems that measure, weigh, monitor, and collect samples for assaying would include the following:

- a weigh scale on the coarse ore stockpile conveyor to enable reconciliation of mine-delivered tonnage with tons crushed;
- a weigh scale on the coarse ore reclaim conveyor for the metallurgical balance;
- automatic sample cutters would be utilized to ensure samples are taken on a regular basis and the shift composite samples would serve as a basis for the plant metallurgical balance;
- appropriate flow meters, scales and control valves would be installed where deemed necessary;
- before leaving the site, antimony concentrate would be weighed and sampled for moisture and antimony content as well as gold and silver content; and
- gold doré would be weighed and sampled for precious metal and impurity contents before being shipped offsite.

17.8 REFERENCES

M3 Engineering & Technology (2014). Stibnite Gold Project Prefeasibility Study Technical Report, prepared for Midas Gold, December 8, 2014, amended March 28, 2019.

SRK (2012). Preliminary Economic Assessment Technical Report for the Golden Meadows Project Idaho, prepared for Midas Gold, September 21, 2012.

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18 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

Existing infrastructure relevant to the development and operation of the Stibnite Gold Project was presented in Section 5. This section summarizes the infrastructure upgrades and infrastructure additions that would be required to support the mining and mineral processing activities that were discussed in Sections 16 and 17, respectively. The Project infrastructure needs that are discussed in this section include:

- Site Access – new road construction and upgrades to existing roads to support safe and reliable all-season vehicle access to the site.
- Power Transmission and Communication Systems – upgrade power supply system on and off-site, install reliable high-speed communications, and expand radio communications across the mine site and access road.
- Other Offsite Infrastructure – road maintenance facility, offsite logistics, warehousing, metallurgical laboratory, and administration facilities near Cascade.
- Site Preparation and Support Infrastructure – clearing, grubbing, growth media stockpiling, borrow sources, upgrades to the existing worker housing facility and construction of a new facility to support construction and operations.
- Ore Processing Plant – equipment, buildings, facilities, and infrastructure to process mineralized material and extract saleable concentrates and metals.
- Onsite Infrastructure – systems, facilities, and structures contributing to the entire operation including truck shop, oxygen plant, limestone crushing, lime calcining, freshwater system, reclaim and process water system, and water treatment plant for treating excess water to discharge standards.
- Tailings Management – tailings storage facility (**TSF**), buttress, and associated pumps and pipelines to safely manage ore processing by-products during operations and in the long term.
- Water Management – surface water diversions and contact water management infrastructure; freshwater, reclaim water, and potable water supply systems; mining-impacted water treatment and management infrastructure; and sanitary waste management infrastructure.

Initial capital, sustaining capital, and closure costs associated with the infrastructure discussed herein are provided in Section 21.

18.2 SITE ACCESS

18.2.1 Burntlog Route

Vehicle access to the Project site is currently via secondary roads that intersect Highway 55 near the communities of Cascade and McCall, as previously discussed in Section 5, and as shown on Figure 18-1. A new site access road alignment was developed that uses the existing US Forest Service road (NF-447) to facilitate safe year-round access for mining operations; reduce proximity of roads to streams, creeks and rivers; and respect advice of community members. NF-447 is known locally as the Burntlog Road and referred to as the "Burntlog Route" by Midas Gold. Figure 18-1 illustrates the alignment of the Burntlog Route, which would total 36.4 miles from Landmark to the Site. From Landmark, the route follows the Burntlog Road (NF-447) for 17.3 miles before transitioning to a new road alignment for 11.8 miles that traverses through the Trapper Creek and Riordan Creek drainage basins to connect to the existing Meadow Creek Lookout Road (NF-640). The route then follows the Meadow Creek Lookout Road for approximately

1.3 miles until the route deviates and begins a steady decline in elevation for 4.0 miles to Thunder Mountain Road (NF-375) and the worker housing facility. Approximately 2.0 miles of Thunder Mountain Road will be upgraded between the worker housing facility and the Project site.

Design criteria were based on jurisdictional policies of Valley County (Valley County, 2008) and the USFS (U.S Forest Service, 2011 and 2014) applicable to a Rural Resource Recovery Road. Key road design criteria include design speed of 20 mph (designed to 25 mph where possible or reduced to 15 mph where needed); maximum 10% vertical grade; 3% to 5% maximum cross slope; 21--foot width; and WB-50 (intermediate-sized tractor-trailer) design vehicle with American Association of State Highway and Transportation Officials (**AASHTO**) HL-93 loading. The critical design vehicle (only occurring in special situations with appropriate traffic control) is a lowboy trailer with mining equipment (similar to WB-67). Additional loading may be placed on structures for the delivery of the autoclave and other equipment. Typical sections of the access road are shown on Figure 18-2.

Improvements to existing route segments range from gravel surfacing, widening, and bridge/culvert replacement (where the Burntlog Route follows the existing route) to full reconstruction along generally parallel segments in order to meet grade or curve criteria. Retaining walls and rock blasting will be required for portions of the route, particularly the new segment connecting the existing Burntlog Road with Meadow Creek Lookout Road and Meadow Creek Lookout Road with Thunder Mountain Road.

Construction of the Burntlog Route would occur concurrently from both ends of the route on a seasonal basis (May to November), but construction could occur outside of those months if conditions allow. The southern portion workforce would be housed in three temporary trailer camps located near construction borrow sources or staging areas (Figure 18-1). The northern portion workforce would be housed at the construction housing facility at the mine site (see Section 18.5.5). Some construction workers could also be housed in the town of Cascade.

Up to eight construction aggregate borrow sites would be established along the Burntlog Route to meet construction and ongoing maintenance needs throughout the life of the operation and during closure and reclamation. Additionally, eight staging areas would be located along the route for the staging of construction equipment and supplies. Three construction camps would be located within disturbance areas for borrow sources or staging areas.

Figure 18-1: Offsite Infrastructure and Utility Upgrades

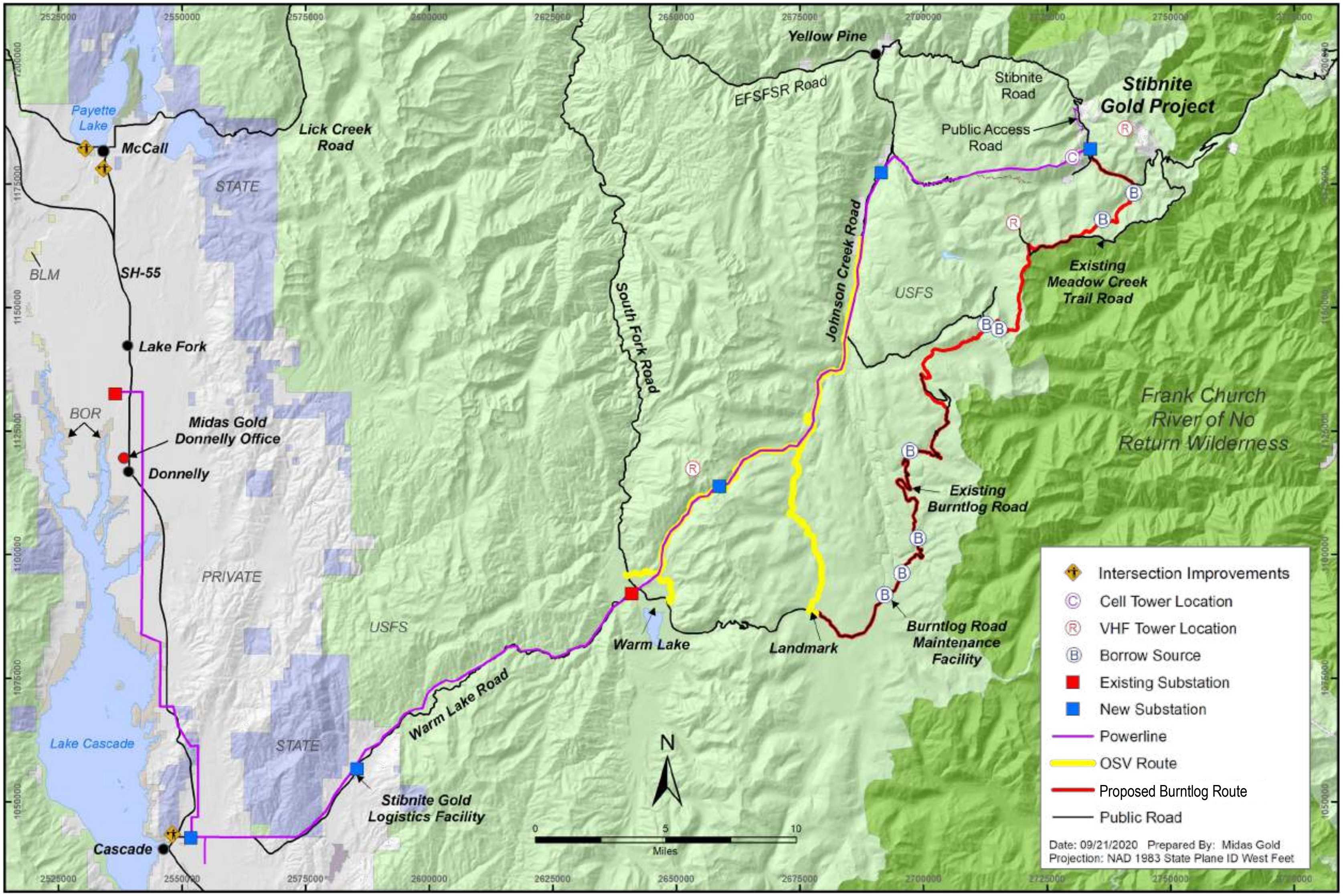
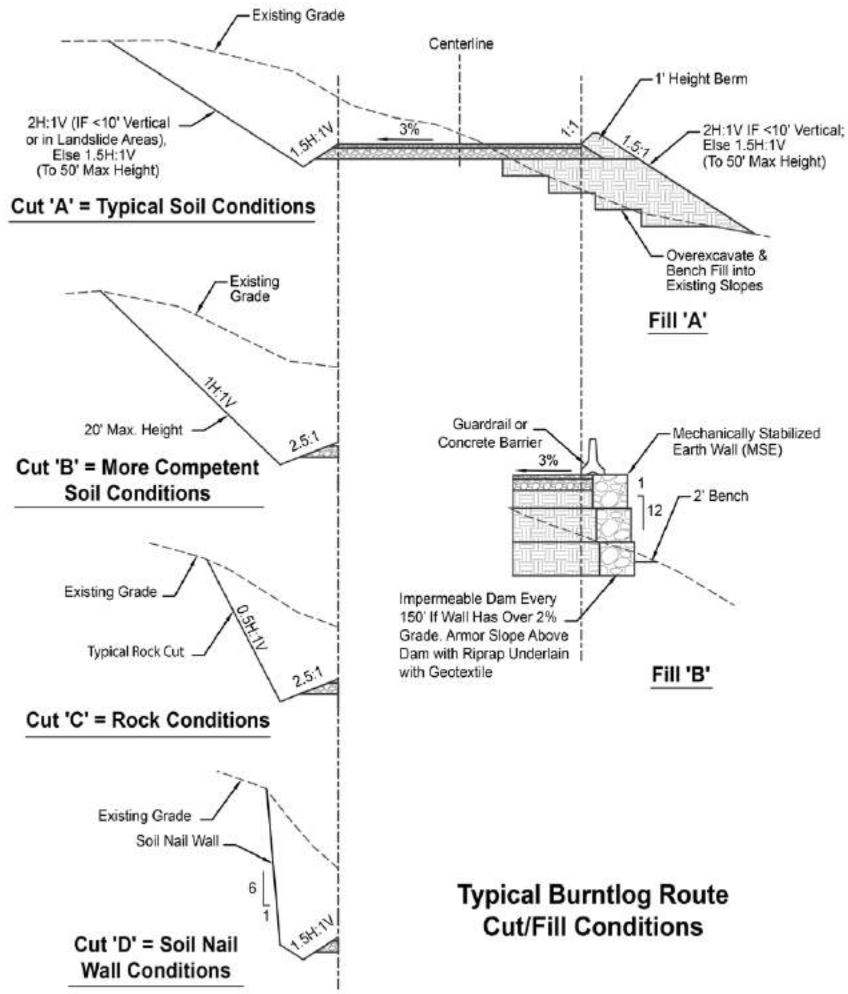


Figure 18-2: Burntlog Route Typical Sections



18.2.2 Public Access

The Burntlog Route would serve as a public access route from Landmark to Thunder Mountain Road and points beyond from the time of its completion in the mine construction phase to the time of its decommissioning and reclamation during mine closure. A public access route will replace the current access through the SGP site on Stibnite Road during mine operations. A new 12-foot-wide gravel road would be constructed during the construction of the SGP to provide public access from Stibnite Road to Thunder Mountain Road through the mine site (Figure 18-6). The road would be constructed on a widened bench within the Yellow Pine pit. South of the Yellow Pine pit, this road would parallel a mine haul road and then follow the route of a former mine haul road. The public access road would be constructed concurrently with the removal of development rock from the Yellow Pine pit. Berms, security fencing, and an underpass to allow the public road to pass beneath the mine haul road would separate the public access road from other mine site roads. The underpass would be located north of Fiddle Creek. The public access road would be temporarily closed during construction and maintenance of the public access road and mining activities that would be considered public safety hazards (e.g., highwall scaling, blasting).

During mine reclamation and closure, the portion of Stibnite Road passing through the site will be relocated to an alignment similar to its current configuration and will provide public access through the reclaimed site permanently. The permanently relocated Stibnite Road will also serve as mine site access for all post-closure monitoring and maintenance activities.

18.2.3 Over-Snow Vehicle Route Improvements

Valley County currently grooms for over-snow vehicle (**OSV**) use between Warm Lake and Wapiti Meadows (approximately 17 miles) along Warm Lake and Johnson Creek Roads. During construction and operations, Midas Gold would plow Warm Lake Road between Warm Lake and Landmark which requires an alternative route for OSV users. Midas Gold identified a route along Cabin Creek Road that will require minor upgrades and maintenance to facilitate grooming of the trail. Upgrades would include the installation of culverts and bridges at stream crossings, road stabilization, and road base addition for construction and maintenance activities. During construction, Johnson Creek Road will be plowed during the winter and an OSV route will be established parallel to the road to provide access to the Landmark area.

18.2.4 State Highway 55 Intersection Improvements

Primary access to and from the SGP (via Warm Lake Road) originates along State Highway 55 (**SH-55**), a major north-south transportation corridor connecting southern Idaho to northern Idaho. A traffic impact study (HDR, 2017) commissioned by Midas Gold evaluated five intersections along this corridor and recommended improvements to three of the intersections to maintain an adequate level of service on the state transportation network. Two of the intersections are located in McCall, Idaho and the third is located at the intersection of Warm Lake Road and SH-55 near Cascade, Idaho. Proposed traffic improvements would improve WB-67 semi-truck turning movements and include the addition of dedicated turn lanes, acceleration/deceleration lanes, and striping modifications.

18.3 POWER AND COMMUNICATIONS SYSTEMS

18.3.1 Power Transmission and Distribution

The proposed on-site mining and mineral processing facilities are estimated to require a total instantaneous power demand of approximately 60 megawatts (**MW**). In order to identify the preferred power supply and distribution option, Midas Gold completed a comprehensive trade-off study in which twelve power sources were considered and evaluated against environmental and social impacts, permitability, reliability and technical feasibility, and capital and operating costs. The results of this study were presented in the PFS (M3, 2014).

Clean energy options were considered for providing electrical power for the SGP. Renewable power generation options would not be reliable sources of power for the Project's requirements of 24 hours per day, 365 days per year and would require significant, redundant alternative self-generation methods. Grid power from IPCo's existing clean energy portfolio is deemed to be the most economical alternative and to have fewer environmental and social impacts than the self-generation alternatives.

A 69-kV power-line corridor was historically permitted and constructed to the town of Stibnite, which indicates the feasibility of permitting a modern powerline to the Project site. Future detailed Project execution studies will consider the expanded utilization of renewable energy, such as solar power which is currently being used for field operations, for areas like worker housing facility and offices, increasing the proportion of renewable energy utilized by the site.

The closest grid powerline to the Project site is a 12.5 kV distribution line along Johnson Creek Road supplying power to the nearby town of Yellow Pine, and the closest transmission line is a 69-kV line that provides power to Cascade and Warm Lake, Idaho. Since both powerlines are inadequate to carry the expected Project loads, the existing system would need to be upgraded to provide the additional service capability required.

The upgrades required to integrate the large load into the IPCo network include an increased 230/138 kV transformer capacity; approximately 41.3 miles of 69 kV lines upgraded to 138 kV; approximately 21.0 miles of 12.5 kV line upgraded to 138 kV line; and approximately 9.2 miles of new 138 kV line. Measures to increase the voltages on the IPCo system are required as shown on Figure 18-1 including new or upgraded 138 kV substations at Lake Fork, Cascade, Scott Valley, Warm Lake, Thunderbolt Drop, Johnson Creek, and Stibnite. IPCo would need to resupply small consumers between the Johnson Creek substation and users to the south via an underground 12.5 kV replacement line. Two route modifications were identified during public outreach and were incorporated into the design. The key reasons for the modifications were to avoid wetland disturbance and impacts to private property.

The 138-kV line would be routed to the Project site's main electrical substation where transformers would step the voltage down to the distribution voltage of 34.5 kV. The main substations would have redundant dual 138 to 34.5 kV transformers to prevent loss of power due to failure. The current Project design entails oxygen being supplied by a third party through a Sale-of-Gas (**SOG**) contract; therefore, a metered 34.5 kV line would be provided for the operator of the oxygen plant.

Power distribution from the Main Substation to various Project facilities would be at 34.5 kV. Power distribution to the primary crusher, truck shop, mine pits, TSF, and worker housing facility is designed to be overhead. Primary power distribution within the process plant area will be underground in duct banks.

During construction, power supply by three 1,000 kW propane or diesel generators operating at 4,160 volts is planned. The generators are planned for use as backup/emergency power during the operations phase of the Project. After primary power is provided via the 138-kV powerline, two of the generators would be relocated to the main substation for emergency power and the third would be relocated to the on-site worker housing facility as discussed below.

18.3.2 Communications Systems

Midas Gold's existing microwave relay detailed in Section 5 (Figure 18-1) was designed and constructed to be scalable to accommodate potential future increases in communication requirements. However, since the microwave relay was constructed, the regional hub on Snowbank Mountain reached capacity and will no longer provide the required increase in bandwidth (1,000 Mbps) to Stibnite. Midas Gold consulted with IPCo about adding fiber optic cable to the transmission line between Cascade and Stibnite. Approval was granted and Midas Gold would partner with local communication providers to add fiber to the transmission line. Other technologies will be considered prior to construction but fiber is the expected option at this time.

The communication facilities would also need to be expanded at the mine site and along the Burntlog Route to facilitate two-way rapid communication between equipment operators and ground personnel and to allow broadcasting of emergency messages. The two-way radio system would be supported by a series of repeaters placed on public and private land. A series of very high frequency (**VHF**) radio repeaters would be placed along the Burntlog Route as needed. The repeaters would be placed near the existing Meadow Creek Lookout and Thunderbolt Lookout communication sites, the new Burntlog Road Maintenance Facility, and on private parcels at the mine site as needed. The 10-foot towers on 3-foot by 3-foot concrete pads would be supported by solar panels, support hardware, and a backup battery case.

A cell tower also would be installed to facilitate area communications. The proposed cell tower would be approximately 60-feet tall and located near the proposed transmission line west of the main substation.

18.4 OTHER OFFSITE INFRASTRUCTURE

Midas Gold would require offsite facilities to support mine-related activities including administrative offices, a transportation hub, warehousing, and an assay laboratory. These facilities would be located at a facility Midas Gold refers to as the Stibnite Gold Logistics Facility (**SGLF**). In addition to the support infrastructure located at the SGLF, year-round road maintenance and snow removal activities would be supported from a facility Midas Gold refers to as the Burntlog Road Maintenance Facility (**BRMF**).

18.4.1 Stibnite Gold Logistics Facility

The administrative offices, training facility, assay laboratory, warehouse, storage, and transportation hub for the operation will be located at the SGLF near the town of Cascade to reduce traffic to and from the Project site and to reduce housing requirements at the site. MGII has acquired property along Warm Lake Road for the Logistics Facility; a plan view of the facility is presented on Figure 18-3. The southern portion of the facility includes parking for vehicles of construction and operations workers who will be bussed to the site. The northern area of the site is available as a laydown yard for equipment and material staging. An area on the east side of the Facility is reserved for potential future construction of a core storage facility.

The administration building and assay laboratory are designed to share a modular building consisting of sixteen 12 ft by 60 ft units, nine dedicated to administration and seven for the laboratory. The Administrative Building includes offices for managers, safety and environmental services, human resources, purchasing, and accounting personnel and includes conference rooms, a break room, and restrooms. Network servers and the communications link for the mine would also be located at this complex as well as the offsite repository for physical and electronic records for mine operations. Administration personnel in the SGLF would coordinate procurement of and payment for the goods and services required at the mine site. The assay laboratory includes offices and laboratory spaces for analytical testing. A sample receiving and handling section is attached to the rear of the laboratory to receive and prepare samples for analysis. Production samples are planned for daily delivery to the laboratory for processing and analysis, and the results would be transmitted electronically to mine operations and exploration personnel.

The SGLF design also includes a warehouse to accumulate parts and supplies and a parking area for trucks to check-in and assemble loads prior to traveling to the Project site. Drivers would check-in at this complex and either proceed to the site, typically in a convoy, or unload at the warehouse for temporary storage and assembly of a load. A truck scale is planned to verify loads going into and out of the warehouse area, as well as a laydown area for temporary outdoor storage.

18.4.2 Burntlog Road Maintenance Facility

The Burntlog Maintenance Facility would be located on NFS land 4.4 miles east of the intersection of Warm Lake and Johnson Creek Roads and would be accessed via the Burntlog Road. The maintenance facility would be located within the footprint of a borrow source established for construction of the Burntlog Route. The facility would include three buildings a 7,500-square foot maintenance building, a 7,100-square foot aggregates storage building, a 4,300-square foot equipment shelter, and an 825-square foot sleeping quarters (Figure 18-4). The maintenance building would house sanding/snowplowing trucks, snow blowers, road graders, and support equipment. Additional features of this facility may include covered stockpiles of coarse sand and gravel for winter sanding activities, and communications equipment.

This facility would include a double-contained fuel storage area housing three 2,500-gallon fuel tanks for on-road diesel, off-road diesel, and unleaded gasoline. Additionally, a 1,000-gallon used oil tank would be located inside the maintenance facility and a 1,000-gallon propane tank would be located at the facility for heating.

Figure 18-3: Stibnite Gold Logistics Facility

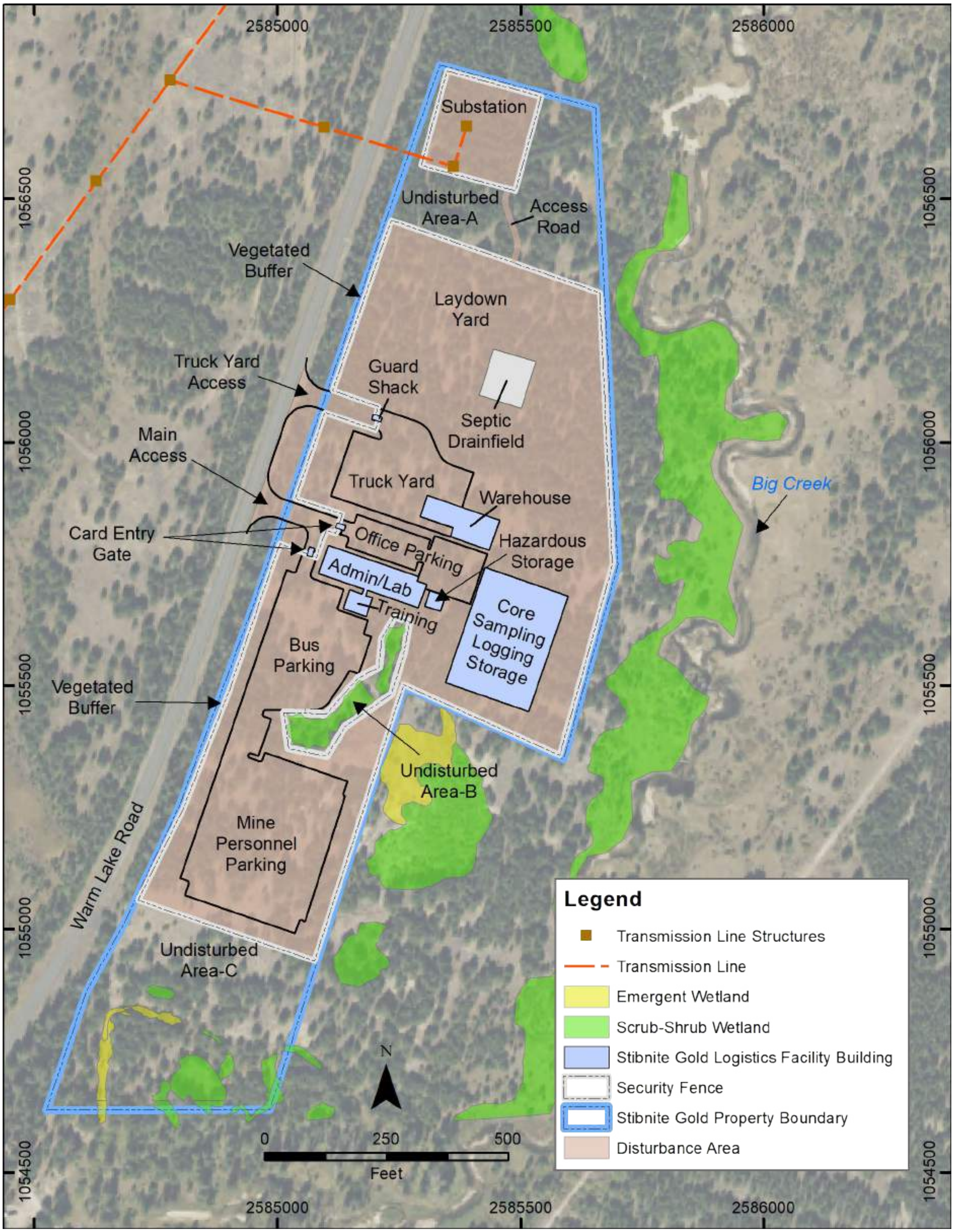
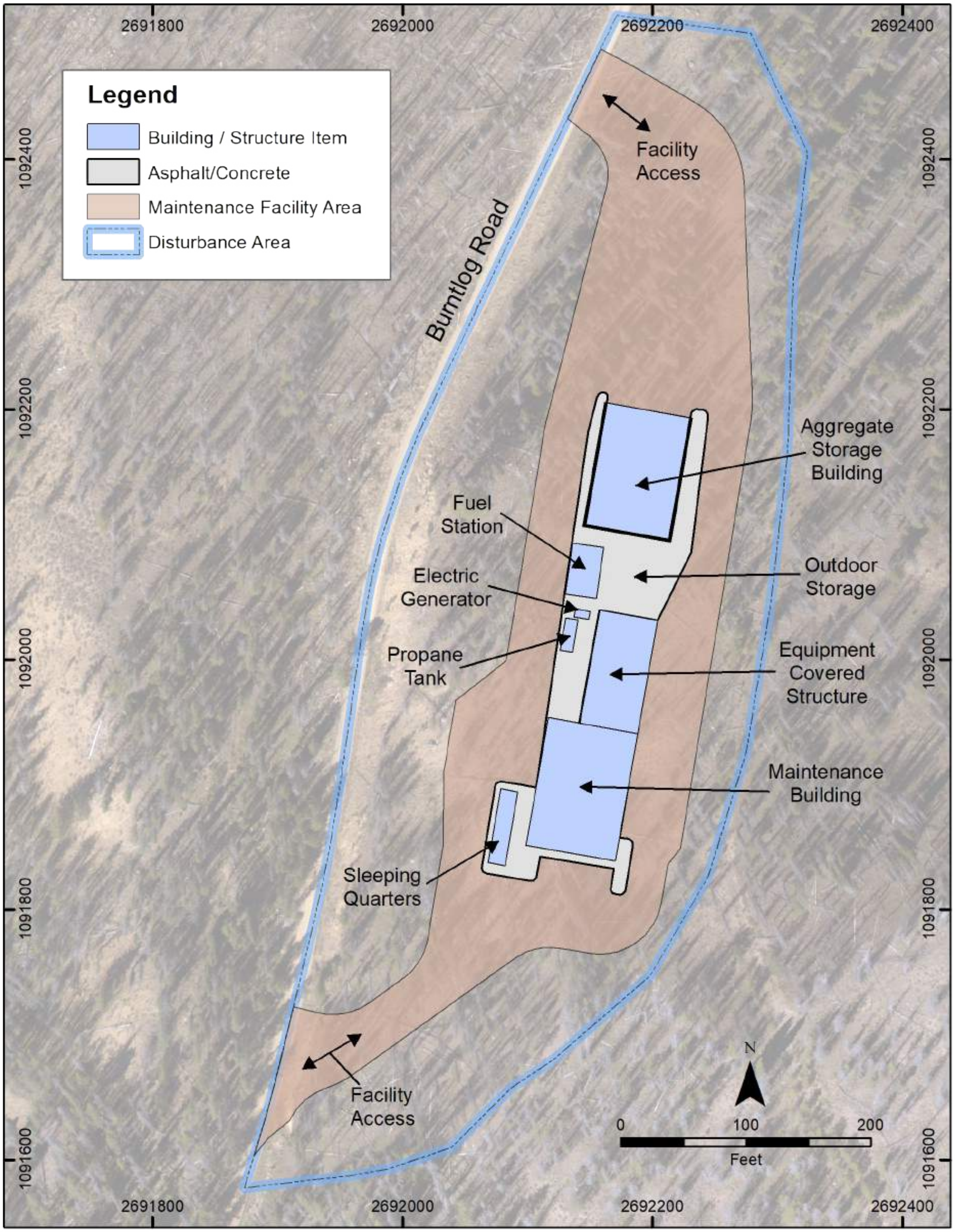


Figure 18-4: Burntlog Road Maintenance Facility



18.5 SITE PREPARATION AND SUPPORT INFRASTRUCTURE

The SGP would require construction of surface facilities, mine site haul roads, and water management features. Removal of some legacy mining features would be initiated during the construction phase. Midas Gold would install 15 to 20 temporary trailers on private lands adjacent to the existing exploration camp to accommodate construction crews. Prior to construction of each facility, vegetation would be cleared, and growth media (**GM**) salvaged and stockpiled. The existing exploration camp water supply and sanitary waste management systems would be used and/or expanded for early construction, while new, larger facilities are built.

Pre-construction water management activities would include the installation of surface water management features and implementation of best management practices to reduce erosion and sediment delivery to streams. These water management features and best management practices could include sedimentation ponds run-on water diversion ditches, trenches, and/or berms; runoff water collection ditches; silt fences; water bars; culverts; energy dissipation structures; terraces; and other features specified in construction permits.

18.5.1 Growth Media Stockpiles and Composting

Owing to prior site disturbance, topography, and geology, a reclamation GM deficit is anticipated at the Project site. Suitable GM material and wetland seed bank material (**SBM**) within the area proposed for operations would be salvaged for future reclamation following vegetation clearing, and stockpiled either within the Fiddle valley, at the Worker Housing Facility, or in short-term GM stockpiles (**GMSs**) within the footprint of the TSF.

Vegetation would be removed from upland operating areas before site preparation and construction of surface facilities. Merchantable timber on NFS surface lands could be purchased from the USFS. Non-merchantable trees, deadwood, shrubs, and slash would be removed, and any remaining vegetation would be grubbed using a bulldozer. The resulting material would be chipped and either stockpiled for use as mulch or temporarily left in place for blending with salvaged growth media. After vegetation removal, growth media and chipped vegetation would be salvaged and stockpiled. For wetlands, vegetation and recoverable organic soils (together, SBM) would be removed, allowed to drain, and stored at the Fiddle stockpile.

GMSs would be stabilized, seeded, and mulched to protect the stockpiles from wind and water erosion. Unconsolidated overburden (chiefly alluvial and glacial materials from Hangar Flats and Yellow Pine pits) would be stored in the upper lift of the TSF Buttress and under the Fiddle GMS to allow future access for use as cover material for reclamation of the TSF, TSF Buttress, and Hangar Flats backfill.

While the cost to produce compost on site is anticipated to be less than the cost to procure and transport commercially available compost to the Project, space limitations and timing of clearing operations preclude a large-scale composting operation on site. Importing and blending compost for generating growth media to cover the anticipated on-site deficit are included in the FS financial model. GM/SBM stockpiles are shown on Figure 18-6. Reclamation and closure plans are summarized in Section 20.

18.5.2 Mine Site Borrow Sources

Various types of earth and rock material would be used from borrow sources for construction, maintenance, closure, and reclamation activities. Most of these materials can be sourced at the mine site from existing development rock dumps, legacy spent heap leach ore, and from development rock removed as part of proposed surface mining and underground exploration activities. However, native materials would be required for some applications. Specific areas within the mine site that have large quantities of high-quality native alluvial and glacial granular borrow materials for use include:

- The alluvial and glacial soils in the Meadow Creek valley floor within the footprint of the TSF, TSF Buttress, and Hangar Flats pit;
- The outwash soils in the lower Blowout Creek alluvial fan; and,
- Glacial soils in the Fiddle Creek valley walls, within the footprint of the Fiddle GMS, and in the EFSFSR valley walls within the footprint of the Yellow Pine pit.

18.5.3 Landfills and Solid Waste Management

Solid waste from the worker housing facility, shops, and other work areas that cannot be composted or recycled would be collected in wildlife-resistant receptacles and hauled offsite for disposal in a municipal solid waste landfill. Early in mine life, inert construction and demolition (**C&D**) waste would be placed in an approximately 4-acre onsite landfill located on private land in the Fiddle Creek basin. Once pit backfilling is in progress and during active closure, additional C&D landfills would be created within pit backfills and at the ore processing facility site. No materials meeting the definition of municipal or hazardous waste nor any waste that could produce pollutants or contaminants that could travel off site would be placed in these facilities. The onsite landfills would be designed to meet non-municipal solid waste landfill regulations.

18.5.4 Mine Support Infrastructure

Onsite infrastructure to support the SGP mining and ore processing operations would include the following.

- A modular one-story mine administration building that would include offices for site management, environmental staff, and other administrative and technical staff.
- A maintenance workshop that would store materials and supplies.
- A truck wash facility that would include an oil/water separation system and water treatment facilities to enable reuse of the wash water.
- A worker housing facility that would be constructed on NFS lands adjacent to Thunder Mountain Road (NF-375) and would accommodate approximately 500 people. The worker housing facility would include indoor multiuse areas and outdoor recreation facilities that would include a sports field and cross-country ski trails across federally administered land.
- Haul roads which would be required within the mine site to transport ore, development rock, and reclamation materials from mining or storage areas, and to transport vehicles to the maintenance workshop. A typical haul road would be approximately 102 feet wide. The haul roads would be built and maintained for year-round access and would be surfaced with gravel aggregate. Road maintenance activities would be conducted to manage fugitive dust emissions and maintain stormwater management features.
- Culverts would be installed where haul roads cross drainages or to direct stormwater to collection and retention structures. Culvert inlets and outlets would be lined with rock riprap, or equivalent to prevent erosion and protect water quality. Crossings of known fish-bearing streams would be constructed to support fish passage with either appropriately designed and constructed culverts or bridges.
- Service roads and trails that would provide an internal access system for employees and visitors to the site. The service roads would typically be 12 to 15 feet wide. Some would be covered with gravel aggregate, while others would be dirt, two-track roads. There would be no planned public use of the mine site service roads or trails. The trail system would enable pedestrian traffic to move safely throughout the mine site operating area. Service roads and trails would be located within the overall disturbance area defined for the mine site and existing roads would be used to the extent possible.

- Employee and visitor parking would be maintained at multiple locations during construction and operations. During construction, the gravel parking areas would be located at the new worker housing facility, near the contractor/construction laydown areas. As operations are initiated, gravel parking areas would be maintained for buses, vans, and other miscellaneous vehicles for employees, contractors, vendors, and visitors at the new worker housing facility, at the shop area, and near the mine administration office.

18.5.5 Worker Accommodations

Since the Project is located in a relatively remote area of Idaho, accommodations will be required for construction and operations personnel. The location selected for the worker accomodations is approximately 1½ miles southeast of the confluence of the EFSFSR, just off the existing Thunder Mountain Road (Figure 18-5). The location is quiet yet located close enough to the site to yield minimal commute times, which should assist in attracting skilled operators to this remote location. For convenience, the construction camp will also be located near the operations worker accomodations area. The following sections describe how the construction camp and operations worker accomodations would be developed.

18.5.5.1 Construction Worker Accommodations

Midas Gold has been conducting exploration activities at the proposed Project location since 2009 and, as a result, has facilities on-site capable of housing workers.

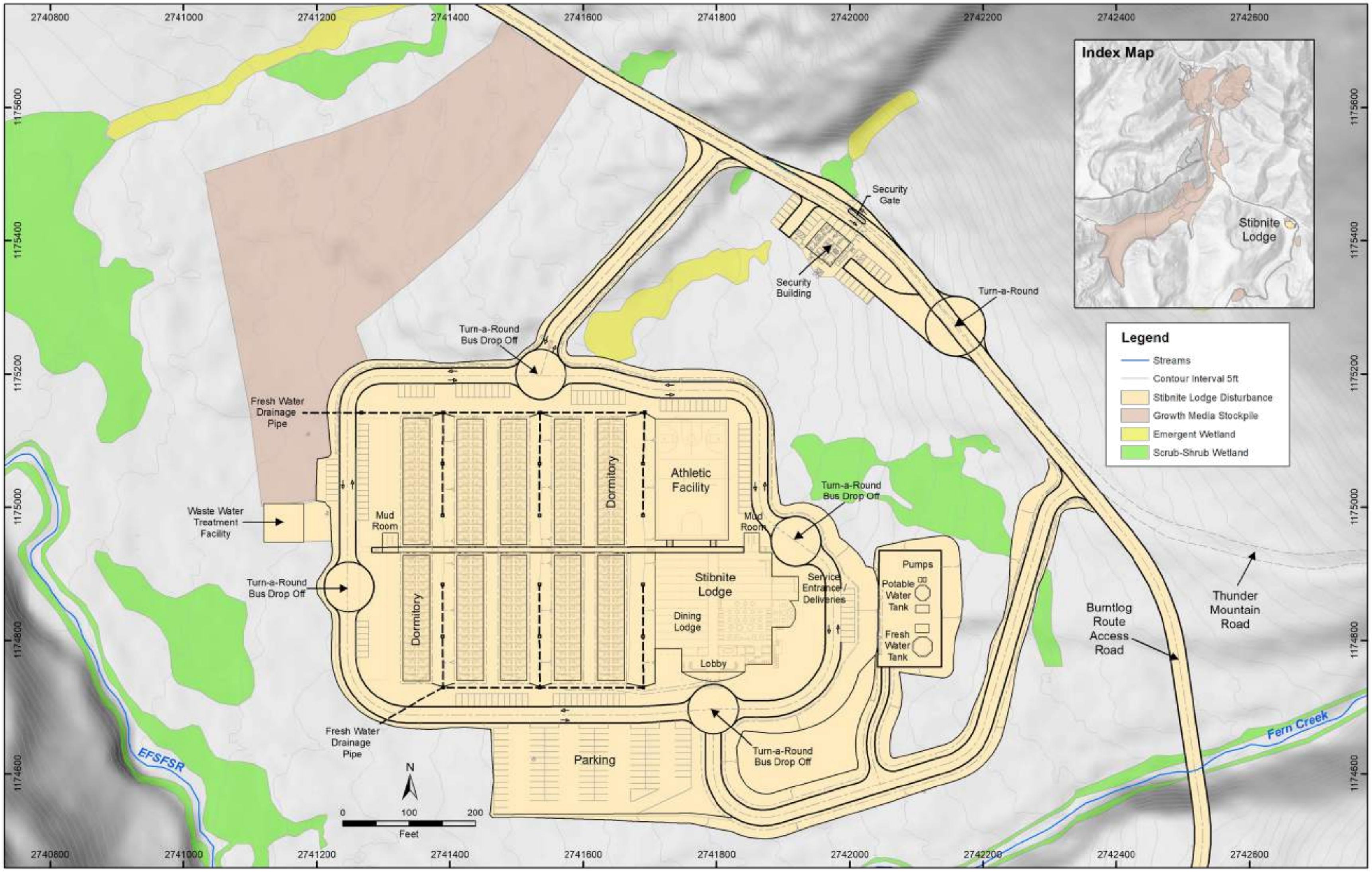
The current on-site facilities are located near the proposed future plant site location and include:

- a 60-person (maximum) housing facility;
- a kitchen/dining building capable of serving 125 workers per 12-hour shift;
- a public drinking water system capable of treating 6,250 gal/day average and a peak of 12,500 gal/hour. The existing potable water supply system would be used and expanded for the initial construction housing facility. The existing system would be supplemented with deliveries of potable water if needed. Supplemental water sources (i.e., water deliveries) would be used by personnel in remote construction areas.
- a Membrane Bioreactor (**MBR**) sewage treatment facility with a nominal treatment rate of approximately 9,000 gal/day average, and a peak treatment rate of 18,000 gal/day. Sanitation during construction would be provided through this sewage treatment system adjacent to the exploration camp. Portable sanitary facilities would be located throughout the mine site and at remote construction areas; and
- power provided by a 455 kW C-15 Caterpillar diesel generator.

The existing exploration housing facility would be relocated and expanded appropriately to manage the estimated 1,000-person construction workforce. The worker housing facility would be developed based on the following assumptions:

- each room will have two beds and locking storage facilities for 4 workers who “share” the room on alternating shifts (day and night) and work cycles;
- there will be one bathroom for every 2 bedrooms;
- supervisors will have a dedicated room that is not shared with rotating shift personnel;

Figure 18-5: Stibnite Operations Housing Facility



18.5.5.2 Operations Worker Accommodations

The operations worker housing facility would be developed by upgrading the construction housing facility. Approximately 517 employees are needed for the operation based on the overtime scheme associated with a modified "14 on, 14 off" work cycle. The bed count associated with this position assessment is approximately 250. As a result, the camp is designed to be a 300-person site residential facility, leaving approximately 50 beds for visitors and/or temporary workers of various types.

The distances from Cascade and McCall are too far for regular commuting from town to the Project site. Charter buses will be used to transport employees to and from the Cascade administration office/staging area and the Project site at the beginning and end of their work cycles, taking approximately two hours under good weather conditions. A charter bus company will operate a small fleet of 50- to 60-person buses, working on a schedule of staggered work cycles that will minimize the number of buses needed to handle the work cycle rotations.

Onsite transport of employees from the operations camp to the mine and plant work facilities would be accomplished by a small fleet of converted school buses and 14-person vans; the distance to transport employees from the operations camp to the various work facilities ranges from one to two miles. Operations personnel would double as bus and van operators. The onsite fleet would be winterized to handle snow conditions between the operation housing facility and the work areas.

18.6 ORE PROCESSING PLANT

The plant site location was selected during the PFS after careful consideration of four potential sites. The layout has been refined for the FS, as depicted on Figure 18-6, showing the initial overall layout of the site at the beginning of mining. The primary feature of the refined layout is the relocation of plant facilities to the valley floor and off Scout Ridge. Geotechnical evaluations of the area currently planned for the grinding circuit enabled this relocation and cost increases to the mill foundations were more than offset by reduction in civil work (particularly drilling and blasting) to create a construction pad on bedrock at the top of Scout Ridge. The FS layout also has the following benefits:

- The SAG mill feed conveyor can be shorter and less steeply inclined.
- The layout is more compact, saving costs for internal utility infrastructure, especially piping.
- Avoiding Scout Ridge reduces exposure to potential avalanches and enables the use of this area for ore stockpiling.
- Relocating facilities from Scout Ridge reduces the length and steepness of in-plant roads.

The configuration of the mine site has minor planned changes as the mining sequence progresses to the Hangar Flats and West End pits. The final configuration of the overall site is shown on Figure 18-7. Haulage roads, water diversions, contact water ponds, and stockpiles would be modified to accommodate the changing configuration of mining, but the processing plant remains the same except for the addition of the oxide leaching circuit in the sixth year of operation.

The Mine Access Road enters the plant area from the southeast and permits delivery and service traffic to come and go from the plant without interacting with mine traffic. The haulage to the Primary Crusher is isolated at the north end of the plant site and the haul road past the Truck Shop is west of the plant. The Admin and Warehouse facilities are located near the entrance to provide access by personnel and supplies without passing through the process area.

Figure 18-6: Initial Overall Site Layout

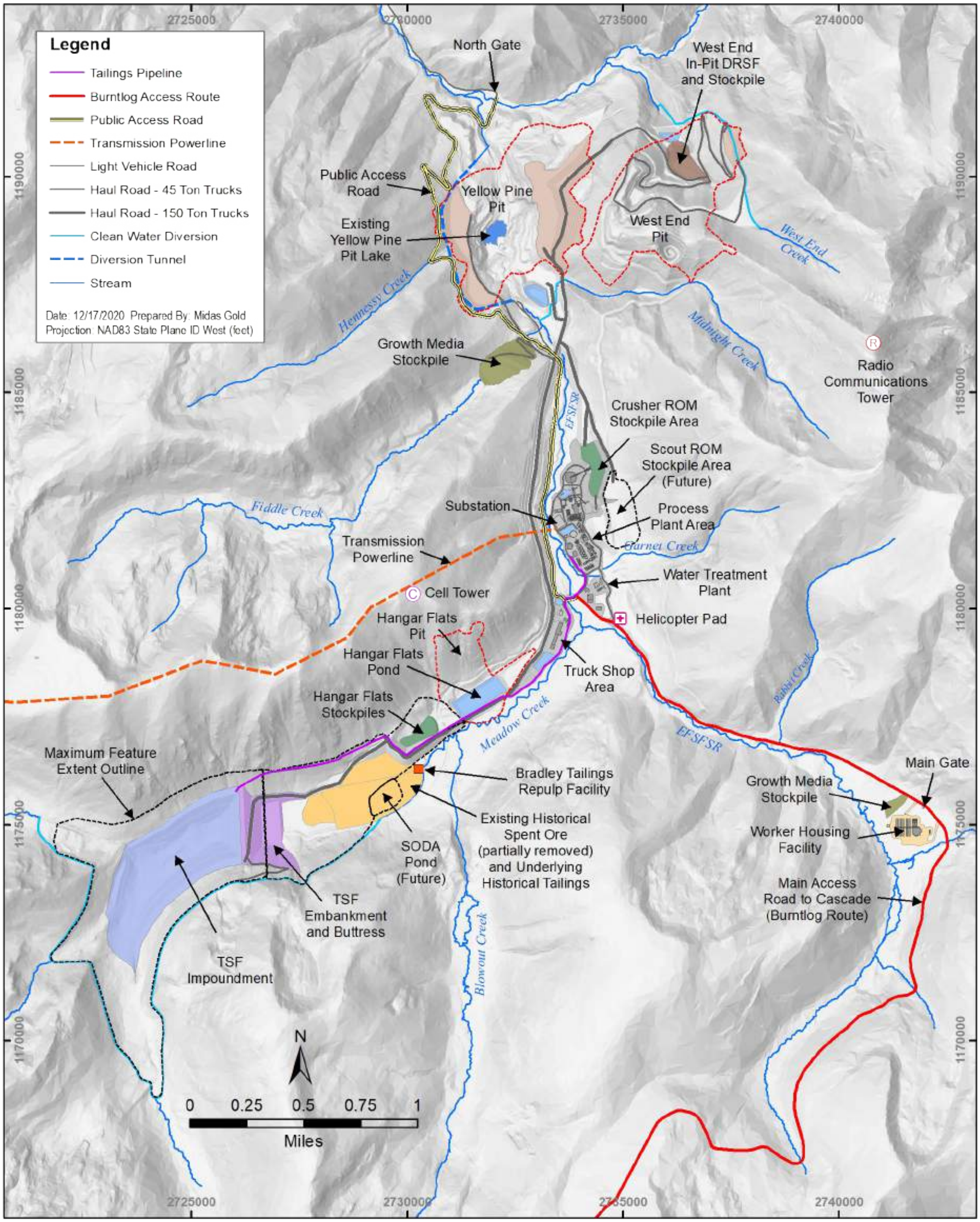


Figure 18-7: Final Overall Site Layout

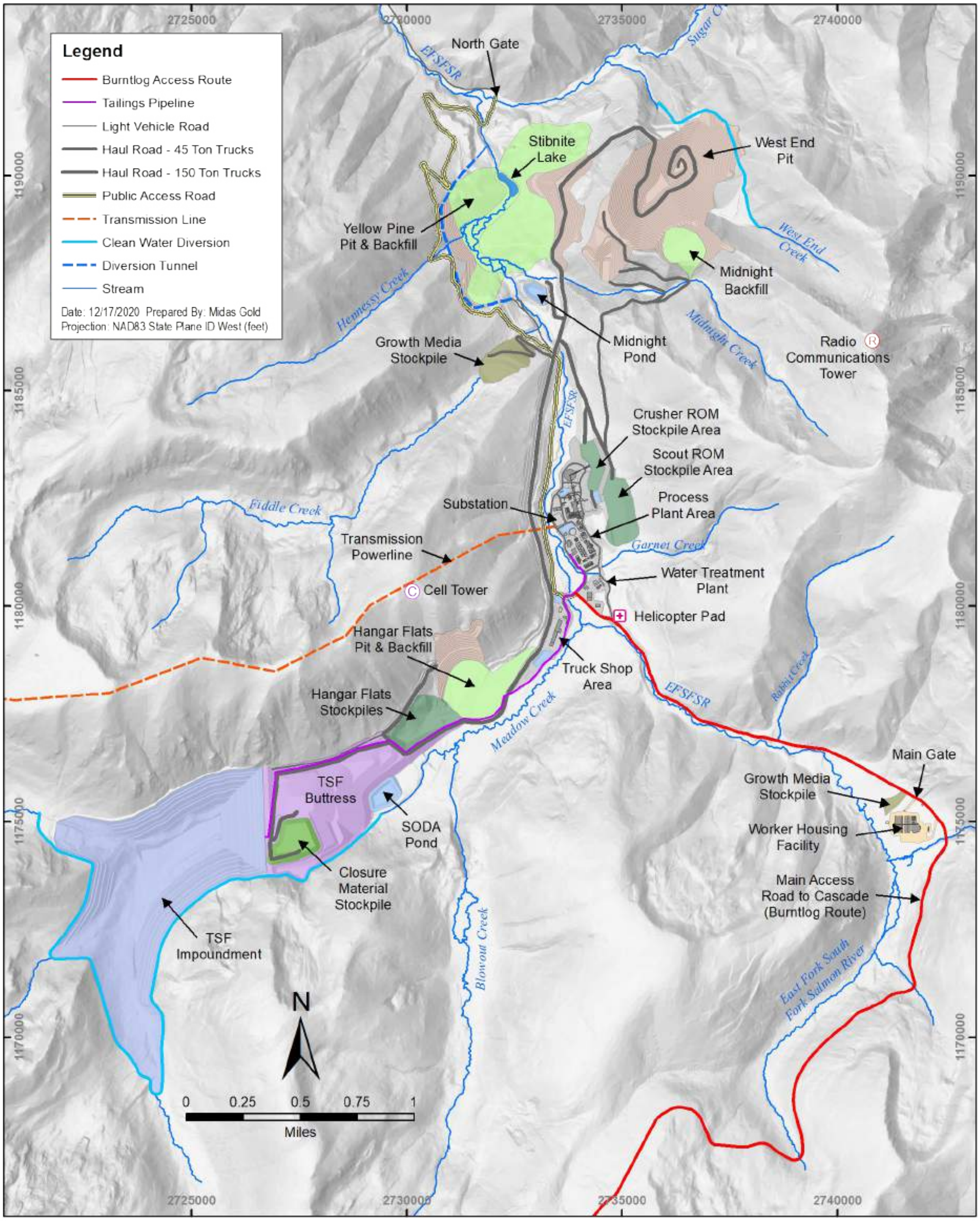


Figure 18-8: Process Area Detail

The layout of the plant site is designed to provide a direct linear flow of processed material through the plant area to minimize piping and other utilities (Figure 18-8). Mineralized material enters the process from the primary crusher on the north, passes through the Coarse Ore Stockpile, through grinding, flotation, and pressure oxidation in the autoclave before looping back through neutralization, cyanidation, detoxification, and tailings disposition. The linear arrangement of the plant on the valley floor enables underground piping placement to minimize overhead obstruction and alleviate the need for heat tracing in most cases.

18.7 ONSITE INFRASTRUCTURE

Infrastructure in the plant area includes a network of roads, power distribution, surface water diversions, and water pipelines. The contributing processes of oxygen supply, limestone crushing, lime calcining, truck servicing, and water treatment for discharge are also included as infrastructure. The roads that provide access to plant buildings and facilities connect to the access road before it reaches the haul road, facilitating deliveries of equipment, materials, and supplies without conflict with mine traffic. The main roads parallel the EFSFSR and have gentle grades, contributing to safety, even in winter months. Power distribution through most of the site is underground in duct banks or above ground in cable trays, contributing to safety and reduction of conflict with mobile cranes used for maintenance. Powerlines enter the site from the west side into the Main Substation and distributed underground to the Oxygen Plant substation and throughout the process area. Overhead power lines distribute power to the north and south of the plant area for water management, truck maintenance, and water reclamation from the TSF. Water from supply wells in the Meadow Creek valley is directed to a collection tank and pumped to the fresh/fire water tank, which is located along the access road at an elevation of approximately 6,800 feet amsl to provide make-up water and water for fire suppression by gravity. The pipelines to and from the fresh/fire water tank, as well as yard piping in the plant area, are buried to protect the lines from freezing.

Stormwater and snowmelt are diverted by berms and channels that pass through the process area to natural drainages (Figure 18-6). Contact water from the plant site is collected by berms and ditches that direct it to lined contact water ponds. Collected contact water is used as process makeup water after settling to reduce suspended solids. Some of the contact water ponds also act as secondary containment ponds for plant facilities.

18.7.1 Oxygen Supply

A cryogenic air separation unit (**ASU**) is planned to provide the supply of oxygen required in the pressure oxidation process (Figure 18-8). The plant would be supplied and managed by an oxygen supply vendor in an “over-the-fence” agreement. Site grading, concrete, and construction support would be provided by the EPCM contractors. Oxygen would be piped directly from the oxygen plant to the autoclave building. The oxygen plant would have its own electrical power substation adjacent to the plant.

18.7.2 Limestone and Lime

A limestone and lime area were added to the layout defined in the PFS because of changes in the neutralization strategy described in Section 17. Limestone quarried from the north end of the West End pit, as described in Section 16, would be hauled to a pad south of the primary crusher pad. Limestone would be crushed and screened to feed the lime kiln and the limestone grinding mill. Ground limestone slurry and milk of lime are used to control acid in the autoclave, neutralize solutions and slurries coming out of the POX process, and control pH for leaching.

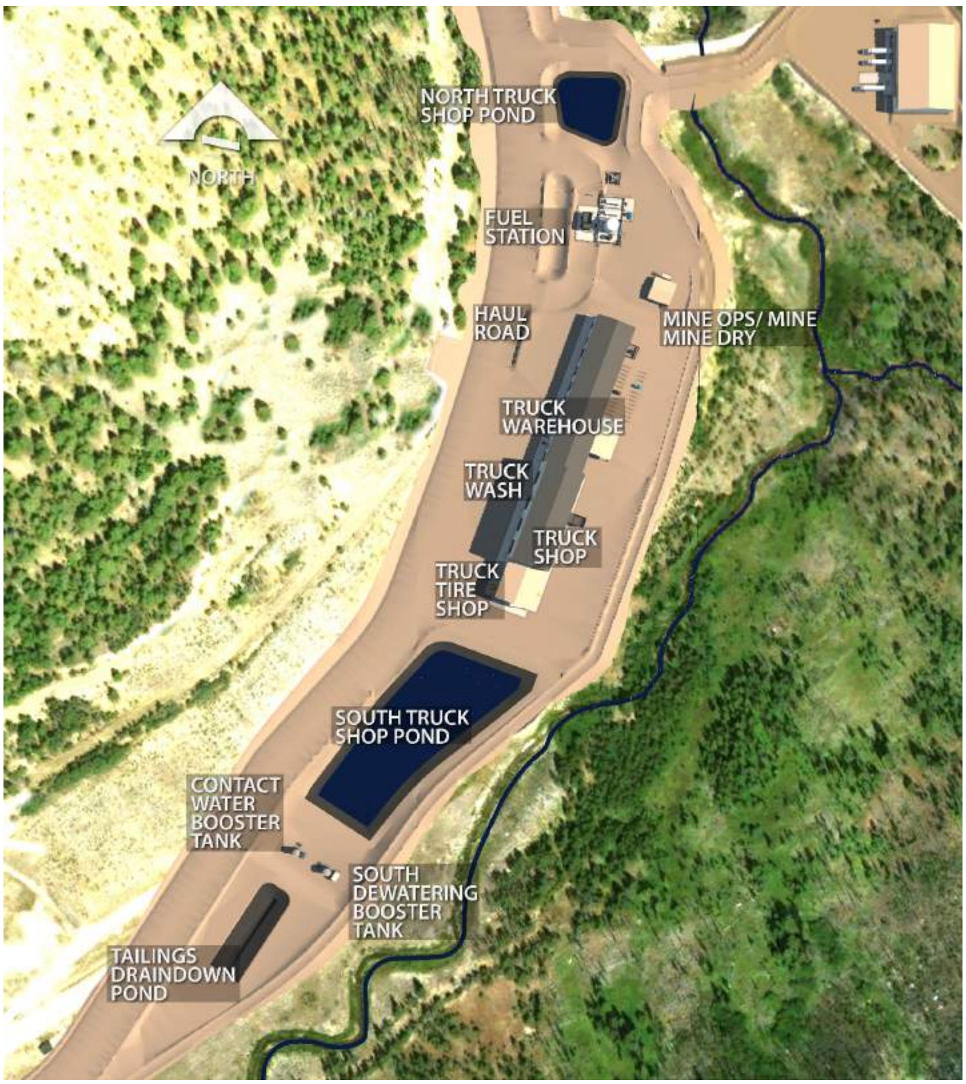
Limestone quarried from the site in the West End pit area is brought to the Limestone Crushing area for crushing and classification (Figure 18-8). The large-sized fraction of the crushed limestone is conveyed to the Lime Kiln to make lime for pH conditioning. The smaller fraction is conveyed to a limestone grinding area of the mill building to make a limestone slurry for acid neutralization, both within and after the autoclave.

18.7.3 Truck Shop Area

A truck servicing area is located along the main haul road near the Hangar Flats pit (Figure 18-9). The main truck shop complex includes a parts warehouse, repair shop, truck wash, and tire shop. The mine operations and change house (mine dry) is in an adjacent building. Contact water ponds are present to collect stormwater runoff at the north and south ends of the area. A containment pond for draining the tailings line is located at the far south end of the area adjacent to booster tanks for contact and dewatering water (Figure 18-9).

Fuel for the operation consists of diesel, gasoline, and propane. Truck and light vehicle fuel is stored and dispensed from tanks at the north end of the truck shop area (Figure 18-9). Propane is stored in a tank north of the lime kiln, which is its primary consumer.

Figure 18-9: Truck Shop Area Detail



18.7.4 Water Treatment Plant

Contact water and groundwater pumped from dewatering wells will be used to augment the operation's water supply demands. Periodically during mine life especially in Years 4 through 7, these sources are projected to produce more water than is required to satisfy operational demands. A water treatment plant (**WTP**) using iron co-precipitation treatment technology is planned to treat up to 2,000 gpm of excess water and discharge it to a permitted outfall on the perennial streams flowing through the site (Figure 18-6). The WTP would remain available for treating excess contact water generated during the post-closure period. The WTP will be relocated to private land on the TSF buttress after the buttress has been covered and reclaimed.

18.8 TAILINGS MANAGEMENT

The Project plans to produce approximately 120 million tons of tailings solids (approximately 115 million tons of ground ore plus approximately 5 million tons of lime, ground limestone, and gypsum resulting from the neutralization of oxidized sulfides) over a 14.25-year mill life. As the tailings would contain trace amounts of cyanide and metals (particularly arsenic and antimony), a fully lined containment facility utilizing a composite liner is proposed to contain the tailings and process water within the impoundment. This option is optimal to reduce Project footprint, provide for a single containment facility for monitoring and closure, and allow for the utilization of development rock and legacy material to construct and buttress the TSF. Section 16 discusses management of Project-generated development rock.

18.8.1 Overview and Design Criteria

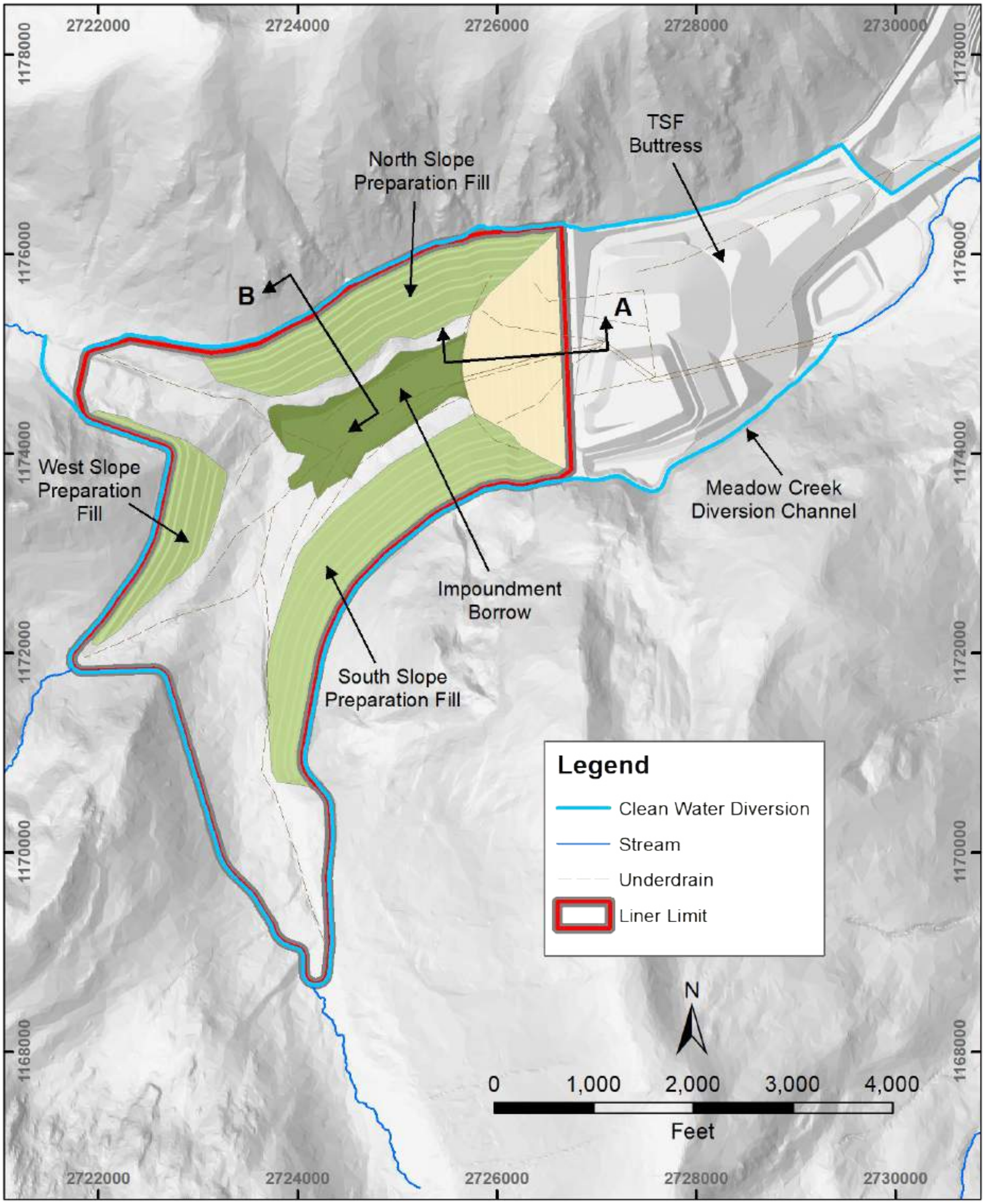
Based on previous siting optimization and tradeoff studies, a single TSF would be constructed to retain all tailings from the processing of the various ore types including legacy tailings and would be located on NFS lands within the Meadow Creek valley. The TSF impoundment, embankment, and associated water diversions would occupy approximately 423 acres at final buildout. The TSF location relative to other project features is shown on Figure 18-6.

The TSF would consist of a rockfill embankment, a fully-lined impoundment, and appurtenant water management features. The TSF Buttress located immediately downstream of, and abutting against, the TSF embankment would substantially enhance embankment stability. Design criteria were established based on the facility size and risk using applicable dam safety and water quality regulations and industry best practice for the TSF embankment on a standalone basis; the addition of the buttress substantially increases the safety factor for the design to approximately double the minimum requirements. Table 18.1 lists the design criteria for the TSF. Figure 18-10 shows a plan view of the TSF impoundment, embankment, buttress, and water diversions. Additional details are discussed in the following sections.

Table 18.1: Tailings Storage Facility Design Criteria

Parameter		Minimum Value	Comments
Solution and Water Management	Inflow Design Flood (IDF) – Impoundment	24-hour Probable Maximum Flood (PMF)	Facility will provide reserve storage capacity above the normal operating pool to store the IDF, assuming diversions fail at the onset of the storm. No operational spillway is included.
	IDF - Diversions	1% annual exceedance probability (AEP) (1-in-100-year event)	Diversions will convey peak flow from IDF without damage.
	Freeboard – Impoundment	4 feet	2' wave height + 2' dry freeboard above stored IDF and operational pool combined
	Freeboard – Diversions	1 foot	
Geotechnical Stability	Static Factor of Safety (FOS)	1.5	
	Pseudo-static (Earthquake) FOS	1.0	
	Design Earthquake	2,475-year – operations (OBE); Maximum Credible Earthquake (MCE) – embankment and post-closure	2,475-year OBE applies to temporary slopes (TSF interior, excluding the upstream embankment face) that are overtaken and buttressed by tailings as the facility fills. MCE applies to embankment during both operations and closure.

Figure 18-10: Tailings Storage Facility Layout



18.8.2 TSF Earthworks

The TSF embankment would be constructed of compacted mine development rock and overburden, repurposed spent heap leach ore, and native borrow sources within the impoundment footprint with a geosynthetic liner on the upstream embankment face (identical to and continuous with the impoundment liner discussed below). Rockfill would be placed in zones of successively more stringent lift height and compaction criteria approaching the liner (Figure 18-11), with the final liner bedding (directly under the liner system) consisting of well-graded silt, sand, and gravel. The development rock TSF Buttress would be placed on the east side of the TSF embankment, providing additional short- and long-term geotechnical stability. Engineered slope preparation fill (Figure 18-12) would be placed against steep slopes within the impoundment to flatten and smooth slopes to facilitate liner placement. Slope preparation fill would consist of spent ore, alluvium, colluvium, previously-mined rock, till, or rock borrowed from within the limits of the TSF or open pits, depending on material availability as the fills are expanded.

Spent heap leach ore would only be used in zones of the starter embankment and impoundment slope preparation fill that would be lined in the same phase of facility construction as the spent ore was placed, and a buffer of clean fill provided between the spent ore and groundwater. Placement of the spent ore below a synthetic liner but above the water table would minimize interaction with water and minimize the potential for further oxidation and mobilization of constituents from this material. Reuse of this already mined material would reduce the quantities of new materials required to construct these facilities.

18.8.3 TSF and Buttress Staging

The TSF would be expanded at intervals throughout the mine life to align with tailings storage and freeboard requirements, beginning with a starter embankment constructed to a crest elevation of approximately 6,850 feet (or approximately 245 feet above the existing ground surface). The final embankment height would be approximately 475 feet at a crest elevation of 7,080 feet. Predicted fill rates and staging are based on tailings consolidation testing and modeling, the mine plan (Section 16), and the site-wide water balance (Section 20). Buttress staging is driven by availability of development rock from the open pits (Section 16); development rock will only be placed in the buttress when not needed for embankment construction. The impoundment and starter embankment would be constructed and fully lined to the elevation of the first stage during preproduction. Due to mine sequencing, the bulk of the embankment and buttress rockfill would be placed well in advance of the need for lined storage, with the embankment crest reaching its maximum elevation by end of the fifth year of production. Subsequent facility expansions would thus consist of placement of the finer, thinner lift-height material on the upstream embankment face; clearing and fill within the impoundment; liner bedding placement; and liner installation and drain extensions throughout the facility. Five total stages are envisioned, with a facility expansion planned every 3 years on average during operations. Figure 18-11 and Figure 18-12 show the proposed TSF embankment and impoundment fill stages and material zones. Figure 18-13 shows the filling curve. Buttress and embankment phasing for select years is shown on Figure 18-14 and Figure 18-15. Additional construction and removal of stockpiles (Section 16) occurs at the TSF Buttress throughout operations but is not shown for clarity.

Figure 18-11: TSF Section A

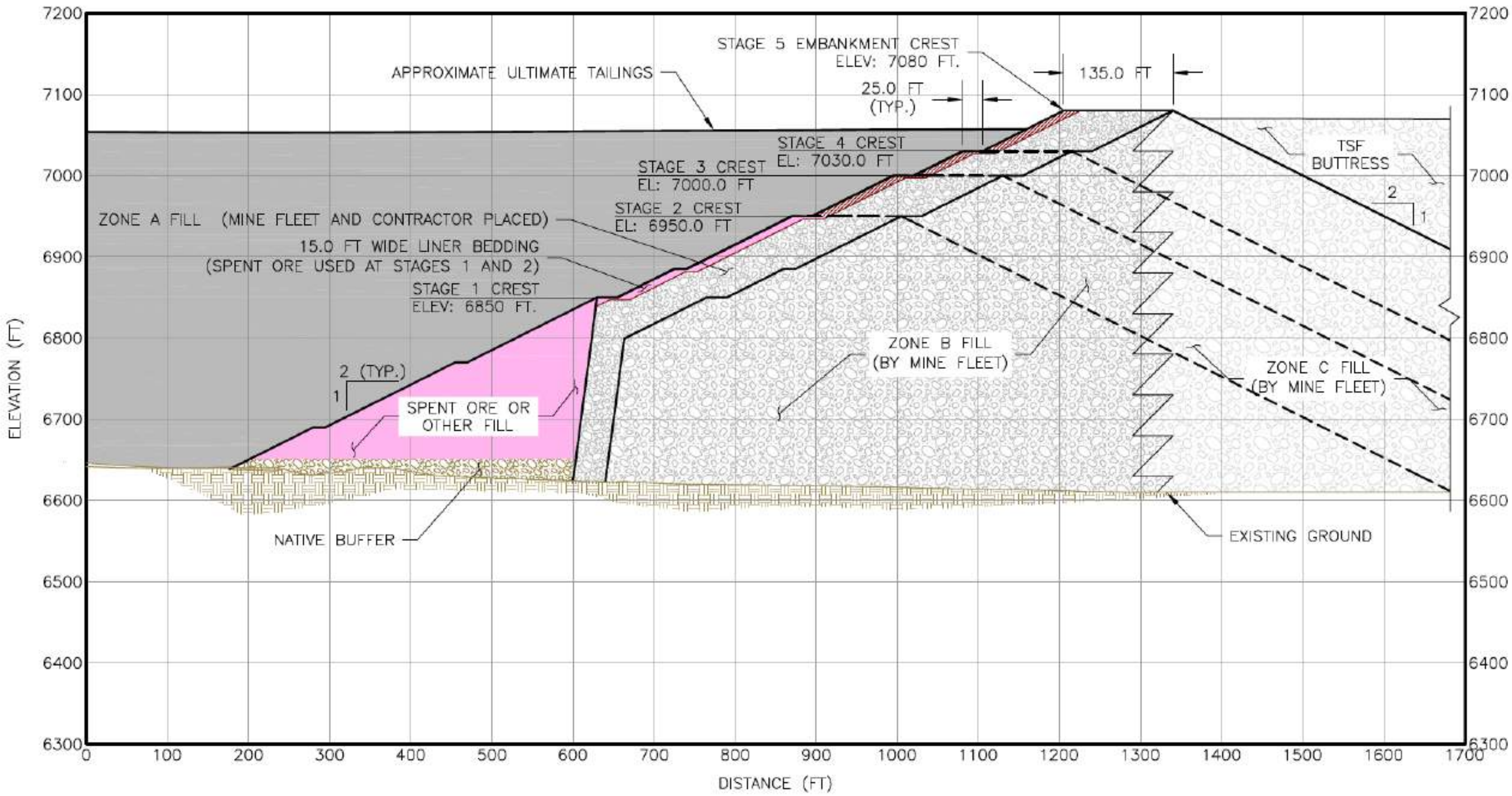
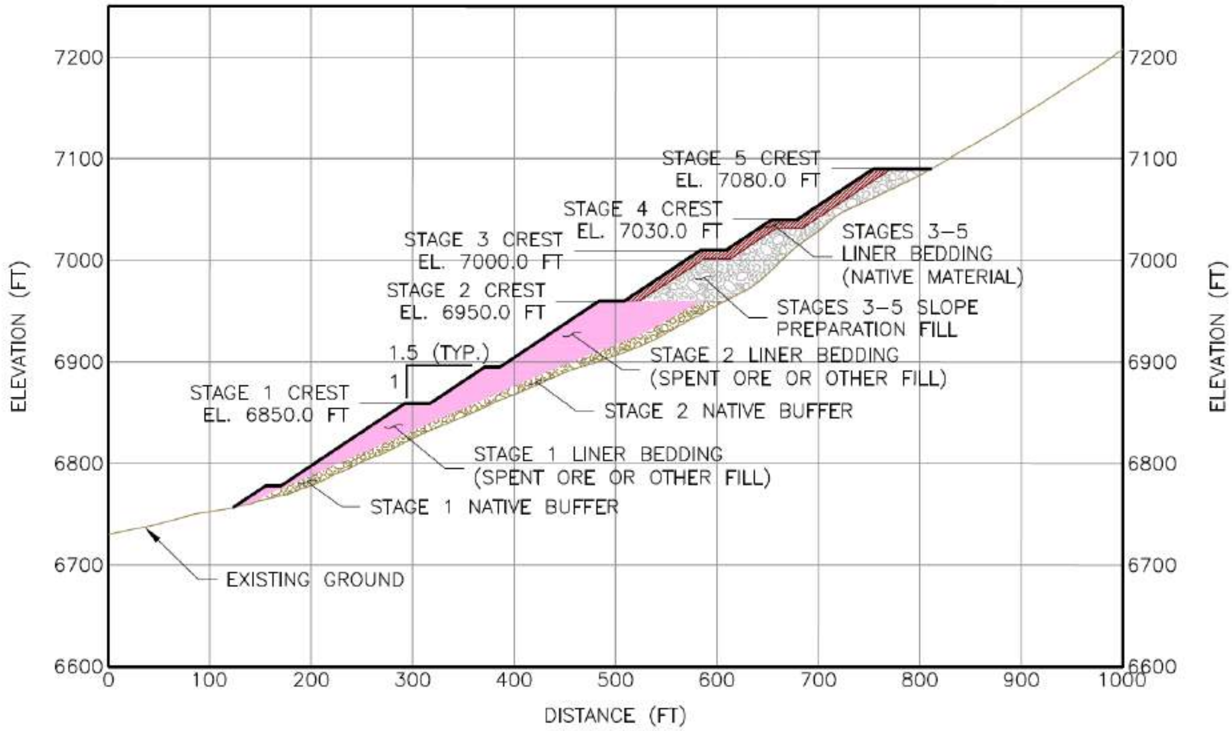


Figure 18-12: TSF Section B



Note:

1. Slope preparation fill crest is sloped. Referenced crest elevation is the slope preparation fill crest at the TSF embankment.

Figure 18-13: Tailings Storage Facility Fill Curve

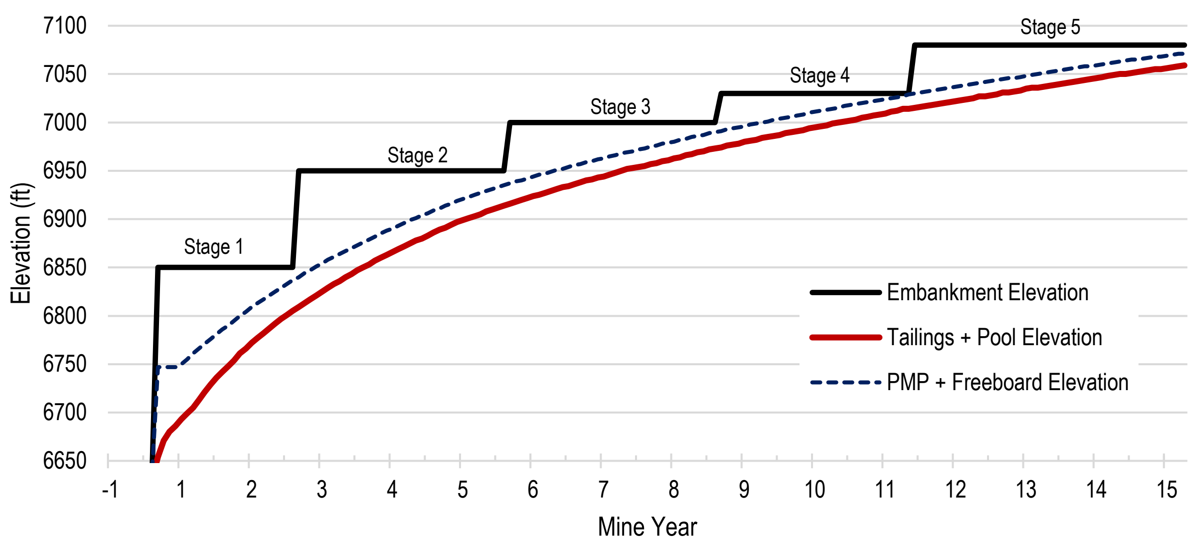


Figure 18-14: TSF and Buttress Development, Years -1 through 3

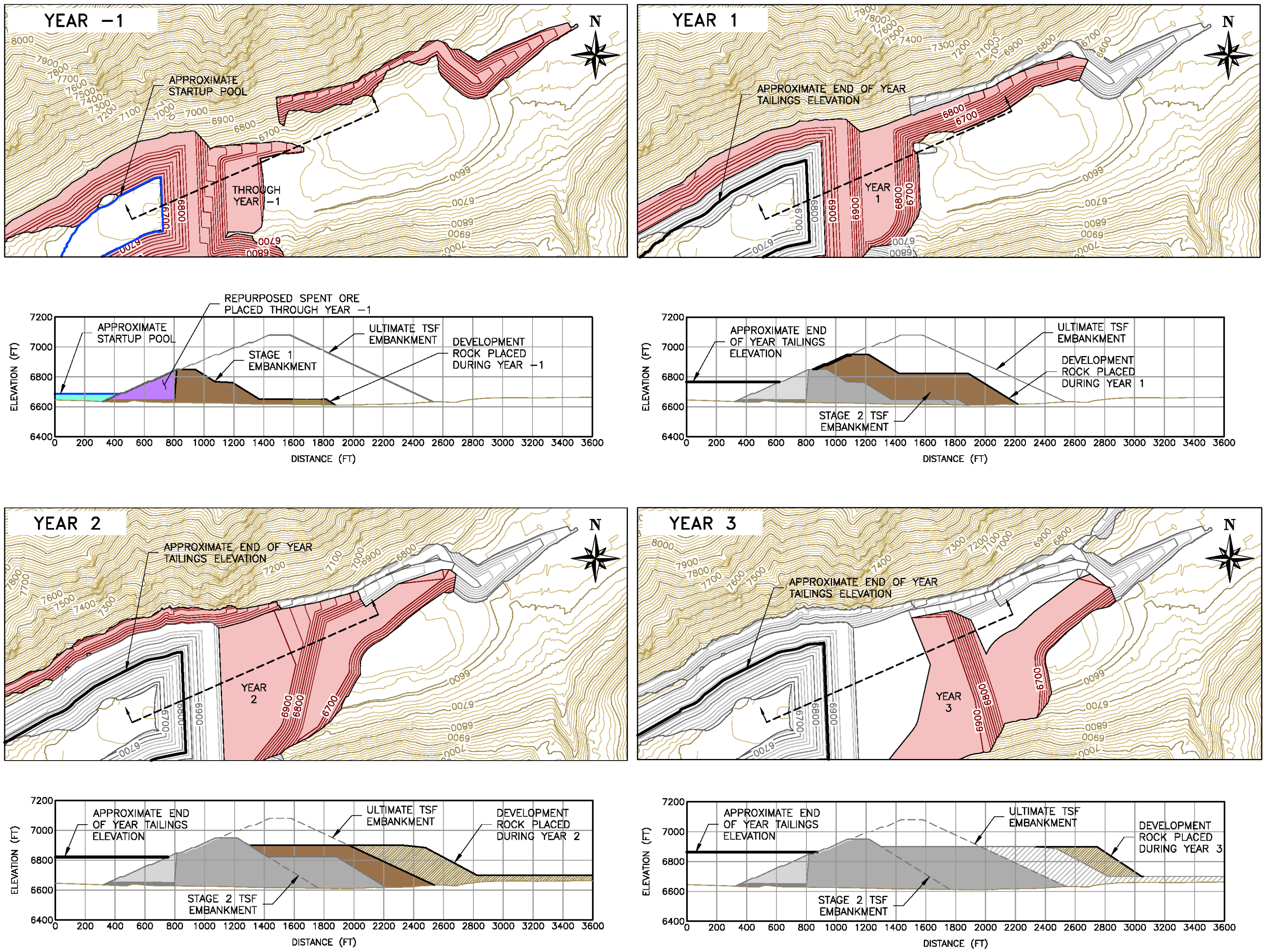
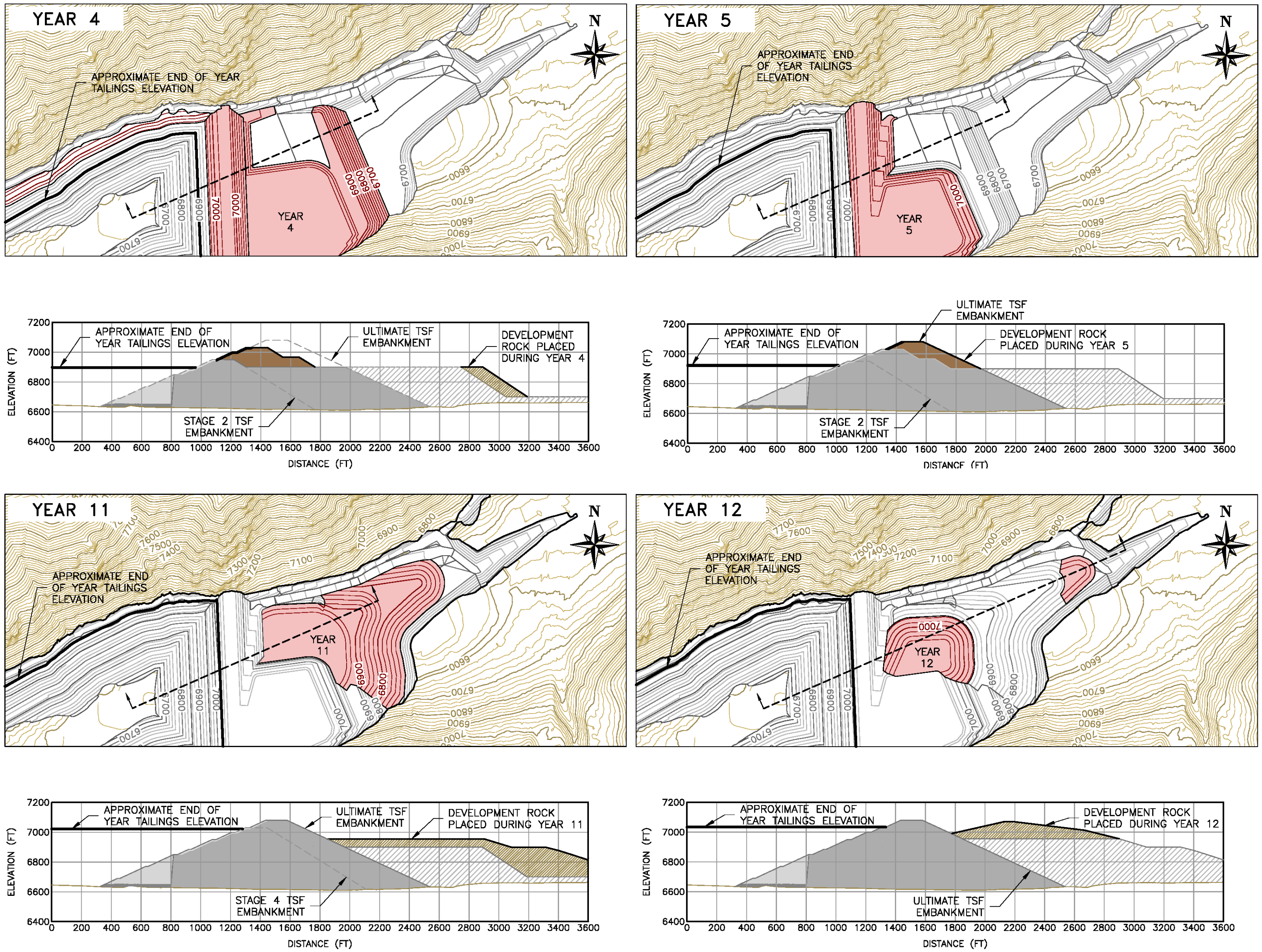


Figure 18-15: TSF and Buttress Development, Years 4 through 12



18.8.4 Impoundment and Liner System

Due to water quality regulations and the presence of dissolved metals (chiefly arsenic and antimony, with trace mercury; see Section 20) and residual cyanide in the tailings pore water and supernatant pool, the TSF impoundment (including the upstream embankment face) would be composite-lined with geosynthetic materials to prevent seepage of process water or transport of tailings out of the facility. The upper layer of the composite liner will consist of 60-mil (1.5 mm) linear low-density polyethylene (**LLDPE**) geomembrane, textured on one side for stability on slopes. A geosynthetic clay liner (**GCL**) will be placed underneath the geomembrane layer, providing a self-sealing leakage barrier should the geomembrane liner be torn or punctured, and improving contact between the liner system and the subgrade, both of which reduce leakage. A network of geosynthetic drains would be placed above portions of the geomembrane liner to reduce hydraulic head on the liner and excess pore pressure in the overlying tailings. The drains would report to a sump near the upstream embankment toe, and the water would be pumped out to the pool or reclaim system for reuse.

Where suitable soil exists (typically in valley bottoms), it would be scarified and re-compacted to prepare the liner subgrade, or a minimum of 12 inches of liner bedding fill would be placed. Steep, rocky hillsides (approximately 1/3 of the TSF footprint) would be covered with slope preparation fill (Section 18.7.2 and Figure 18-12) to cover rock outcrops and flatten slopes sufficiently to allow liner placement. Slope preparation fills will be progressively buttressed by the tailings as the facility fills and therefore are exposed without buttressing for months, not years or decades. Geotechnical design criteria for the slope preparation fills differ from the TSF embankment in that the 2,475-year earthquake was used in slope stability modeling rather than the MCE to reflect the slopes' shorter unbuttressed exposure window.

Underdrains installed during site preparation would collect spring and seep flows beneath the TSF impoundment liner and embankment, reducing hydrostatic uplift on the liner system, and convey the collected water beneath the TSF embankment and buttress. The underdrains would be a series of parallel drains with branching laterals. Underdrain flows would be collected in a sump upstream of the discharge point, monitored for water quality, then discharged to surface water or pumped to the ore processing facility for use as makeup water. The underdrain layout is included on Figure 18-10.

Cyanide would be reduced in the process plant to levels protective of wildlife. An 8-foot high, chain-link fence surrounding the TSF is designed to keep wildlife such as deer and elk from entering the impoundment area to prevent either liner damage or wildlife drowning.

18.8.5 Tailings Transport and Distribution

Thickened tailings slurry would be pumped from the tailings thickener at the process plant to the crest of the embankment and then around the perimeter of the TSF in a distribution header. The tailings pipeline and pumping system would require sufficient head and operating flexibility to deliver tailings to the back of the TSF as the embankment increases in height over the 14.25-year operational life of the facility. Horizontal centrifugal pumps that increase in number as the embankment height increases would be used to pump the tailings from the thickener to the TSF. The initial requirement includes five operating pumps and five standby pumps. A tailings booster station located at the south end of the truck shop area will be added to coincide with the Stage 3 TSF expansion. One additional pump and one standby will be added to provide the necessary lift to deliver tailings for the remainder of mine life. The ultimate configuration would include six operating pumps and six standbys.

Thickened tailings would be deposited in the TSF from a series of drop-pipes (spigots) originating from a 20" HDPE tailings distribution header along the facility perimeter bench. Subaerial tailings deposition would promote drying and consolidation of the tailings. Rotating the active deposition points would allow additional drying, and sequencing of deposition would allow gradual development of a tailings beach that slopes generally from west to east within the facility, mimicking the pre-Project valley drainage and simplifying facility closure. Development of a tailings beach around the perimeter would provide a measure of protection against floating ice damaging the liner system.

The tailings pipeline from the mill to the TSF would be HDPE-lined, 18-inch carbon steel pipe. Light vehicle roads and haul roads would connect the ore processing facility and the TSF. The tailings delivery and reclaim water return pipelines would parallel the roads with secondary containment provided throughout the pipeline length. Secondary containment for pipelines would consist of a backfilled geomembrane-wrapped trench, pipe-in-pipe, or open geosynthetic-lined trench, depending on location. The pipeline corridor would drain to one of two pipeline maintenance ponds – one at the truck shop and one at the ore processing facility. A 12-inch to 16-inch (size variable according to elevation) HDPE reclaim water line would be co-located in the trench to provide secondary containment of process water being reclaimed from the TSF. The slurry line from the Bradley Tailings recovery operation would also share this trench until it is no longer required.

The proposed routing of the tailings pipeline is designed to follow the haul road on the north side of the Meadow Creek valley (Figure 18-6). The pumping station would be on the west side of the plant area. The tailings line would be routed across the EFSFSR on a bridge in a double-contained pipe, then generally follow the haul road toward the embankment. After passing the vicinity of the future Hangar Flats pit, the pipeline corridor would be installed in a trench that climbs the slope on the north side of the valley. The pipeline corridor would be accompanied by a roadway to enable monitoring and servicing the pipeline and trench. The pipeline would be installed sufficiently high on the valley slope so that it is above the ultimate height of the TSF Buttress so that construction of the latter would not interfere with the tailings operation. In approximately Year 4, a portion of the tailings pipeline would be rerouted to the southeast to accommodate the growth of the Hangar Flats open pit and associated reconfiguration of haul roads.

18.8.6 TSF Water Management

TSF water management facilities include diversions, underliner, and overliner drainage systems, the reclaim system, and evaporators. The TSF would be operated as a zero-discharge facility meaning no water would be discharged to the surface water or groundwater except under unusual circumstances and in compliance with applicable laws, until closure when water treatment would be implemented (Sections 18.8.5.2 and 20). During operations, water collected in or falling on the surface of the TSF would drain to the supernatant pond on top of the tailings and be recycled along with tailings consolidation water for use in ore processing via barge-mounted pumps discharging to the reclaim pipeline (Section 18.7.5). Clean water would be diverted around and under the facility in surface diversions and underdrains. Surface water diversion channels would serve to temporarily divert Meadow Creek and its tributaries around the TSF and TSF Buttress, while underdrains (Section 18.7.4) constructed in valley bottoms would collect springs and natural seeps and prevent accumulation of water under the liner system. Snowmaker-type evaporators installed at the TSF would be used to dispose of excess water as needed. The geo-composite overliner drain system would report to a sump near the upstream embankment toe, from where it would be pumped out and the water returned to the TSF water pool.

18.8.7 Summary

Table 18.2 summarizes the TSF design.

Table 18.2: Summary of TSF Design

Design Aspect	Description
Underdrains	Mains: perforated pipe and gravel in geotextile-wrapped trenches. Laterals: geo-composite drains.
Subgrade	Reworked and compacted in situ materials, or minimum 12 inches of liner bedding fill.
Liner Subbase	Geosynthetic clay liner
Primary Liner	60-mil LLDPE, single-side textured
Overliner drains	Geosynthetic strip drains.
Leak Detection	Sampling of underdrains and downgradient monitoring wells.
Deposition Strategy	Subaerial; depositing from perimeter of impoundment and embankment with pool on east side near, but not normally in contact with, embankment.
Reclaim	Pumped from barge (vertical turbine pumps).
Excess Water Disposal	Consumption in process (operations), mechanical evaporators (operations and closure), water treatment and discharge (closure)
Diversions	Surface channels, in rock cut or lined with geosynthetics, concrete cloth, or riprap and GCL. Parallel or embedded pipe for low flows (stream temperature mitigation measure).

18.9 WATER MANAGEMENT

Water management infrastructure is needed at the site to supply water for ore processing, camp, offices, fire protection, exploration, and dust control; divert surface water around mine features and infrastructure; dewater pits and to control water that comes in contact with mine features. Key considerations for water management on the Project site are centered around a large amount of snowmelt runoff and run-on during the months of April through June. This spring melt is the critical time for water storage and treatment. Operational water management actions would be informed by climate predictions and the stored water in snowpack in the months preceding the spring melt.

In general, surface water that comes in contact with materials that have the potential to introduce mining- and process-related contaminants (contact water) is kept separate from surface water that originates from undisturbed, uncontaminated ground (non-contact water). This is accomplished by diverting clean water around mine facilities and collecting and reusing, evaporating, or treating and discharging contact water. Water management designs were guided by water balance and environmental modeling, described in Section 20. Section 20 also addresses water-related permitting and water rights. Site water management features are shown on Figure 18-16 (north) and Figure 18-17 (south).

Figure 18-16: Water Management Plan - North

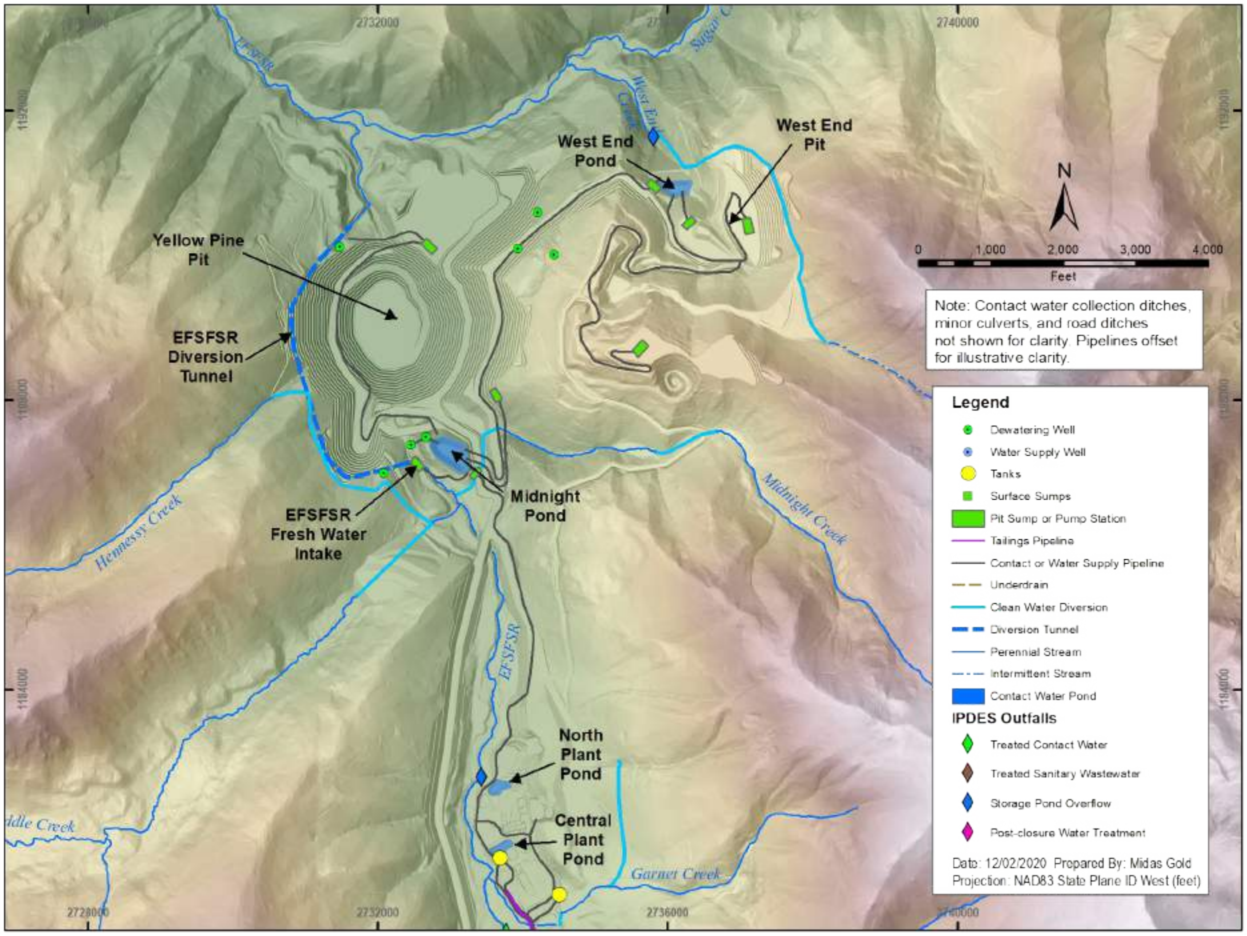
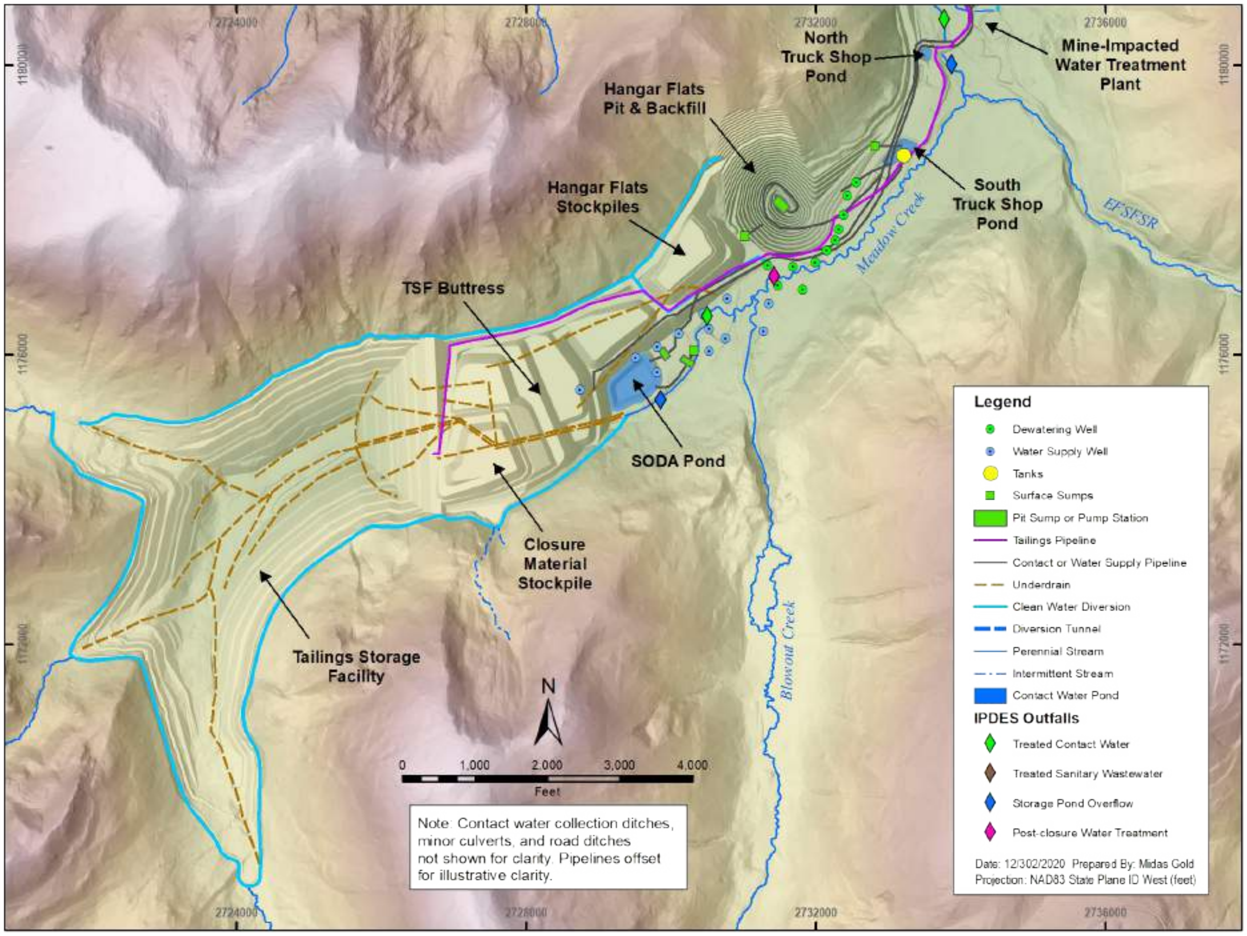


Figure 18-17: Water Management Plan - South



18.9.1 Water Supply

Water supply for the mine, process plant, worker housing facility, and site dust control would be provided by four types of water systems: freshwater (including fire water), potable water, reclaim water, and contact water re-use. Freshwater for the process would be supplied from groundwater resources by a surface intake on the EFSFSR near the south tunnel portal, a water supply well field, and dewatering wells associated with Hangar Flats and Yellow Pine pits. Reclaim water would be pumped back to the process facility from the supernatant water pond in the TSF. A water supply wellfield would be developed for potable water supply to the worker housing facility. Potable water would be filtered and chlorinated before use. Potable water for the office and other mine facilities would be supplied from the potable water tank at the worker housing facility via a holding (head) tank at the same location as the freshwater/fire water tank for the process. Contact water, when available, would be reused for process makeup and for dust control on haul roads, plant roads, backfills, stockpiles, and the TSF Buttress when of suitable quality for direct use.

18.9.1.1 Freshwater / Fire Water Supply

Freshwater for process needs would be supplied by an intake on the EFSFSR, a water-supply well field located in the Meadow Creek valley upstream from its confluence with the EFSFSR, and from dewatering of the Yellow Pine and Hangar Flats pits. Surface water from the EFSFSR intake would be pumped to a booster tank adjacent to the Midnight contact water pond, and then to the process plant. The mill water supply wellfield would consist of approximately twelve

14-inch diameter alluvial wells, ranging from 200 to 270 feet in depth. Groundwater pumped from dewatering and supply wells would be collected in equalization tanks (with destination tank depending on well location) and pumped either to the process plant or to the freshwater/firewater head tank located at approximately 6,800 ft amsl along the access road. Freshwater for process makeup would be drawn by gravity from the freshwater tank from an elevated nozzle to allow the water in the bottom of the tank to remain available for fire suppression use, thereby ensuring an adequate water supply and pressure from gravity for fire suppression at all times, even when there is no power. Intake and water supply well locations are shown on Figure 18-16 and Figure 18-17.

18.9.1.2 Potable Water Supply

Water for the worker housing facility would be obtained from a separate water supply wellfield located in the EFSFSR valley to its southwest. This water will be filtered and chlorinated for cleaning, cooking, showering, and consumptive use in the worker housing facility. Water from the worker housing facility potable water tank is designed to flow by gravity to a potable water tank at approximately 6,800 ft amsl on the access road to provide gravity flow of potable water to eye wash-safety showers and sinks, showers, and restrooms in the process plant and associated areas.

18.9.1.3 Reclaim Water System

Reclaim water pumped from the TSF supernatant water pond would be reused as process water. Water reclaimed from the TSF would be pumped to the reclaim water tank at the plant site. From the TSF to the plant site, the reclaim water pipeline would share a lined secondary containment trench with the tailings pipeline. At the plant site, the reclaim line would diverge and be located in its own secondary containment trench. Water from the reclaim water tank would be distributed to various points of use in the process. During reprocessing of the Bradley Tailings, an additional pipeline would shunt a portion of reclaim water from the main reclaim line to the repulping plant where it would be used to slurry the tailings for pumping to the processing plant.

18.9.1.4 Contact Water Reuse

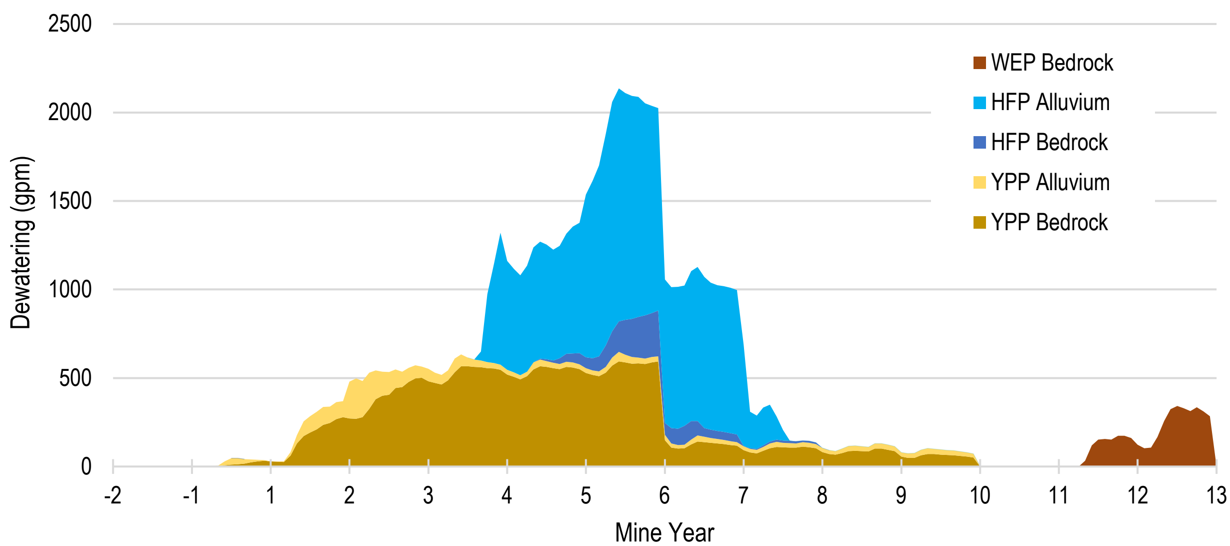
Contact water collected in ponds and in-pit sumps would be pumped via pipeline or directly into trucks to points of reuse or treatment/evaporation. Contact water for reuse in the process plant would report to the process water tank. Contact water for road/mine dust control would be pumped directly to water trucks. Stormwater retained in haul road sediment traps may also be pumped out for use in dust control on those roads or other mine features.

18.9.2 Pit Dewatering

Groundwater modeling and pump tests indicate that active dewatering will be required for the Hangar Flats open pit. Active dewatering will not be required for the West End pit and will be limited at the Yellow Pine pit. Dewatering of alluvium/overburden and possibly shallow (up to ~20 feet below bedrock surface) fractured and oxidized bedrock would be necessary for the Hangar Flats and Yellow Pine pits and would be accomplished with approximately eleven 10-inch to 14-inch diameter wells drilled to depths ranging from 240 to 290 feet at Hangar Flats pit, and four 12- to 14-inch wells, approximately 80 to 170 feet deep at Yellow Pine pit. An additional three 10-inch wells in bedrock at 500 to 800-foot depth may be needed at Yellow Pine pit and have been included in the FS cost estimate. Actual well layout and need will be refined based on additional long-term pump tests during construction and operations and site experience.

Water quality monitoring, geochemical testing, and geochemical modeling suggest that dewatering water quality may not be suitable for direct potable use or discharge without treatment for arsenic and antimony. Excess dewatering water (not used for process makeup) would be treated, if required, and discharged to a surface outfall to the EFSFSR near the process plant or a surface outfall located along the lined portion of Meadow Creek to augment stream baseflow and offset depletions resulting from dewatering Hangar Flats pit. Total dewatering would range from near zero to a peak of 2,100 gpm over the life of the mine, depending on the location of active mining among the three pits and the depth and area of each pit. Predicted dewatering rates are shown on Figure 18-18. Wellfield layouts and piping are shown on Figure 18-16 and Figure 18-17.

Figure 18-18: Predicted Pit Dewatering Rates Including Passive Groundwater Inflows



18.9.3 Non-Contact Water Management

Surface water management activities include diversion of non-contact runoff water originating offsite around mining operations, diversion of streams around mine facilities, and management of sediment from erosion occurring in the East Fork of Meadow Creek (Blowout Creek). Construction of stream crossings (culverts and bridges) is also required, incidental to road construction and not further discussed here.

18.9.3.1 Surface Water Diversions

Surface diversions are required to prevent offsite clean water from commingling with contact water and to prevent the accumulation of excess water in the TSF. The principal surface water diversions would route Meadow Creek around the TSF, the TSF Buttress, and the Hangar Flats pit. The tunnel routing the EFSFSR around Yellow Pine pit is discussed in Section 18.8.3.2. Other smaller-scale diversions are provided to intercept hillslope runoff and minor tributaries at the TSF, TSF Buttress, Fiddle GMS, Bradley Tailings reprocessing operation, open pits, and process plant area. Lower Meadow Creek would be diverted around the Hangar Flats pit prior to mining proceeding below the creek level. That diversion would feature a natural channel and a restored floodplain corridor southeast of the present channel and be left in place after closure. Surface diversions are sized to convey the runoff from a flood event required by applicable regulations and appropriate to the risk level of each facility – at least the 100-year flood for the Meadow Creek diversion at the TSF/Hangar Flats DRSF, Meadow Creek channel/floodplain at Hangar Flats pit, and diversions at the process plant. Other diversions would be designed for at least a 25-year event. The main stream diversion channels are either constructed in rock cut (on steep hillsides), or lined with rock riprap and GCL or geosynthetics (HydroTurf, concrete cloth, etc.) to prevent erosion and minimize seepage if the substrate is alluvium, colluvium, or fill. Portions of the diversions are piped in areas with steep slopes, notably West End Creek and Hennessy Creek. Perennial stream diversions (Meadow Creek at TSF/Buttress and West End Creek at West End pit) will feature low-flow pipes sized to convey late-summer baseflow as a stream temperature mitigation measure. TSF and DRSF diversion plans are shown on Figure 18-10 and Figure 18-17. The Meadow Creek diversion at Hangar Flats pit is shown on Figure 18-17.

18.9.3.2 EFSFSR Tunnel

Currently, the EFSFSR flows over a steep cascade, which is a fish migration barrier, into the existing Yellow Pine pit forming a pit lake. The lake outflow discharges northward toward the EFSFSR's confluence with Sugar Creek. Mining the Yellow Pine deposit requires the pit lake to be dewatered and the EFSFSR temporarily diverted around the pit during planned mining operations and subsequent stream restoration activities. The orientation of the Yellow Pine open pit relative to the surrounding steep terrain makes a surface diversion impractical; hence, the EFSFSR will be diverted around the open pit in a tunnel driven in rock.

The 0.9-mile long EFSFSR diversion tunnel would be 15 x 15 feet in cross section and require support ranging from rock bolts and shotcrete to steel sets, depending on ground conditions. The tunnel interior would feature a 5-foot wide weir/pool fishway and a 9-foot wide maintenance accessway separated by a 5-foot tall by 1-foot thick partition wall. A control weir at the upstream end of the tunnel would divert all low flows into the fishway, and at high flows would allow flow in both the fishway and the accessway – providing for maintenance access during low-flow periods, and limiting the flow range in the fishway to control water velocity within the range against which the target fish species (Chinook salmon, bull trout, and steelhead) can swim. 3D computational fluid dynamic modeling confirms that the design provides acceptable velocity and depth for fish passage during each respective species' migration period, and that the tunnel has a flood flow capacity in excess of the 500-year event. Transition channels to/from the tunnel and the EFSFSR would be armored against erosion and include concrete and rock weirs to maintain depth, sediment and debris control structures, and fish resting pools. A freshwater intake with fish screens (Section 18.8.1.1) would be located in the forebay upstream of the control weir near the south portal. Figure 18-19 shows the overall layout of the tunnel and key features. Figure 18-20 and Figure 18-21 present the tunnel design, including general layout, profile, support types, and fish passage and access features.

Figure 18-19: Isometric Cutaway View of Fish Passage Tunnel

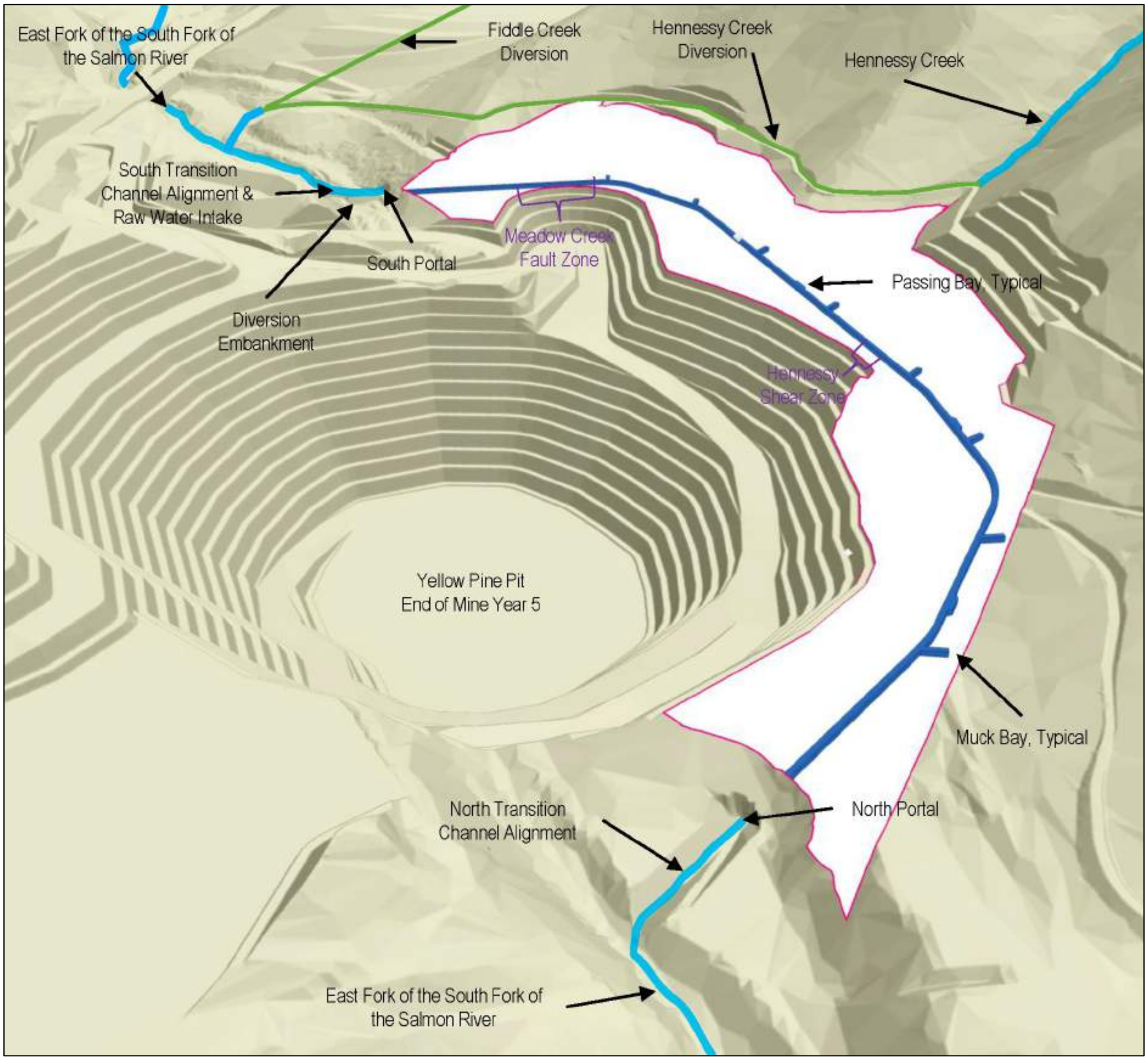


Figure 18-20: EFSFSR Tunnel Plan and Profile

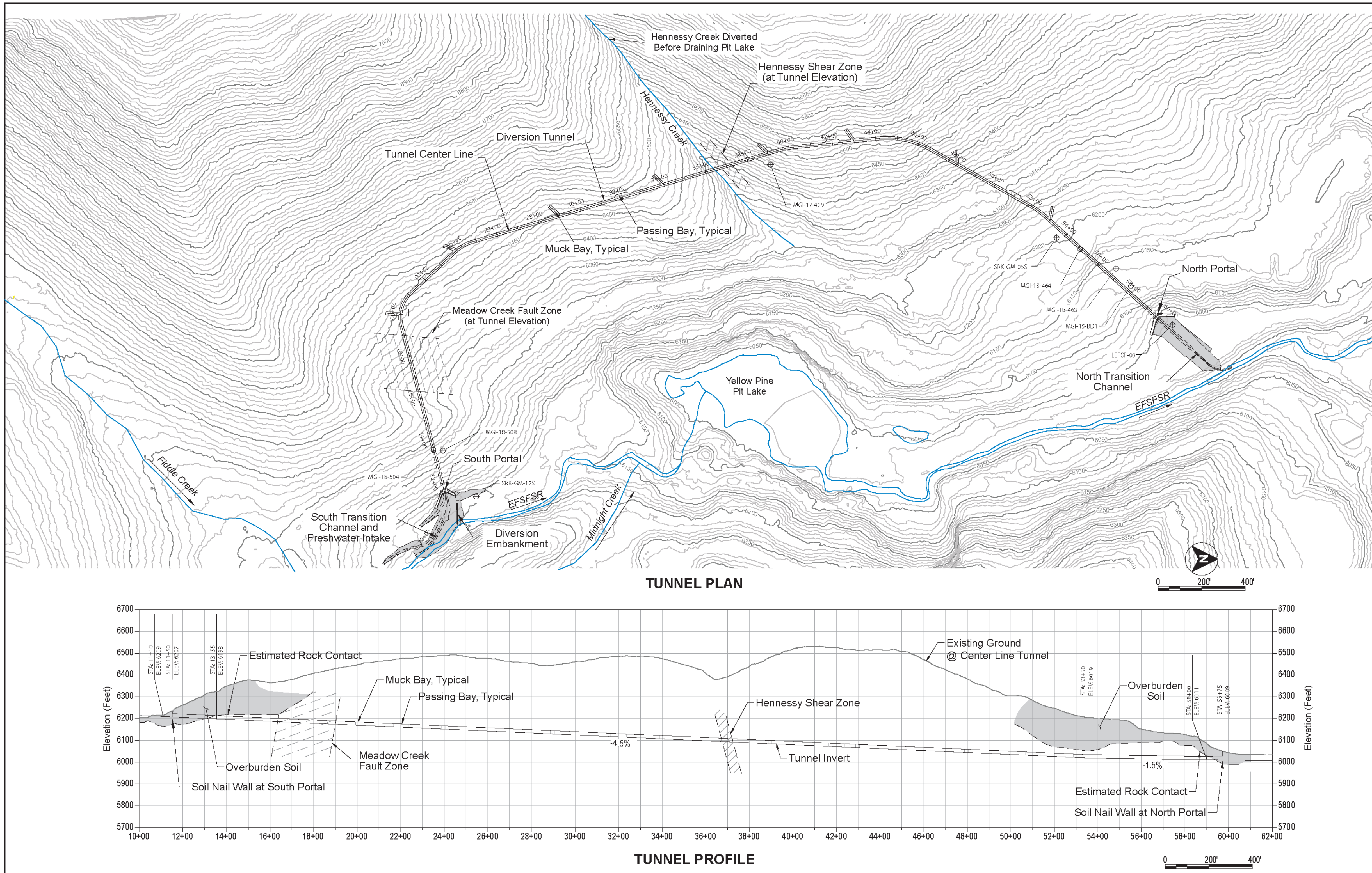
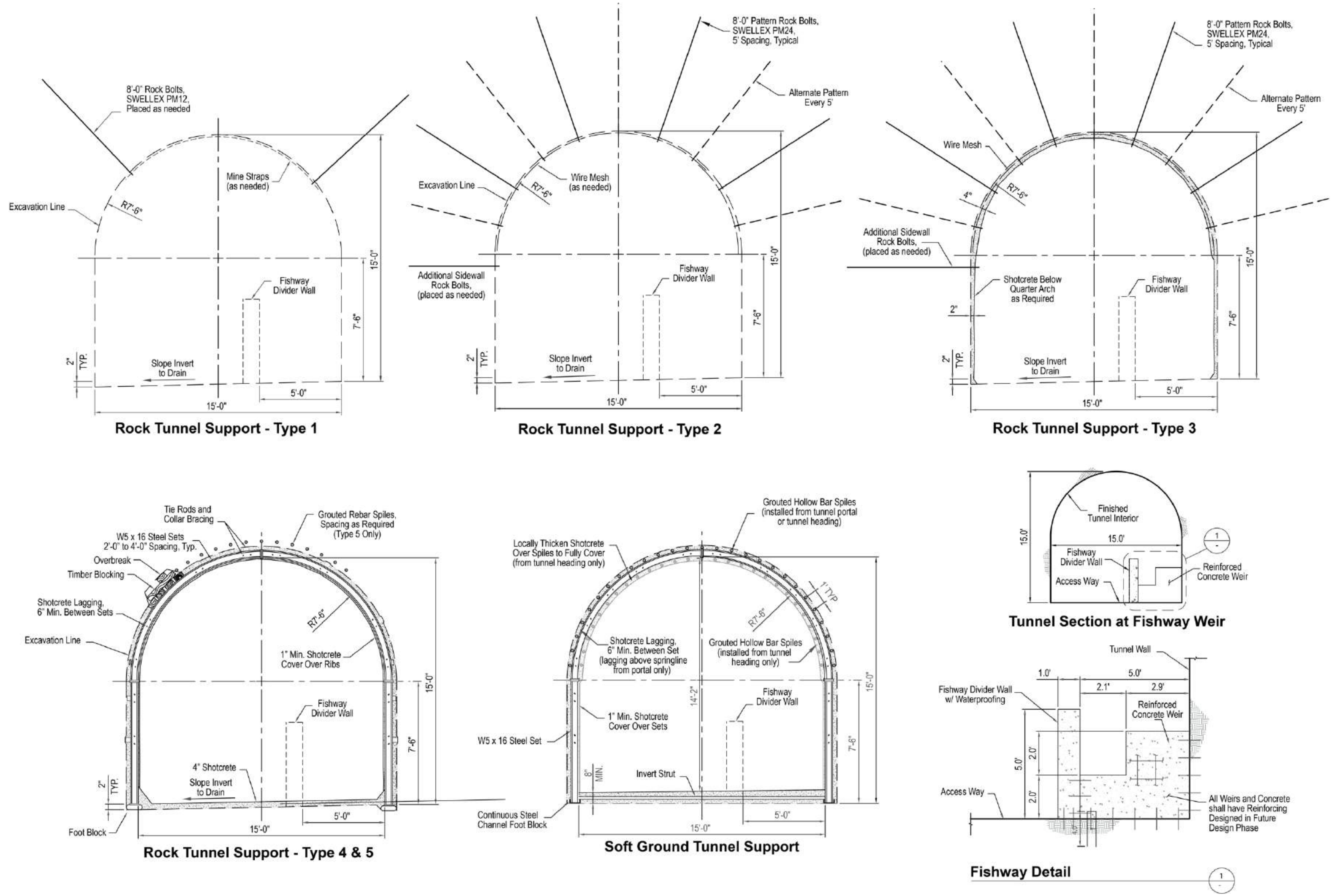


Figure 18-21: EFSFSR Tunnel Support Details



18.9.3.3 East Fork Meadow Creek Sediment Control

The East Fork of Meadow Creek (**EFMC**), which is commonly known as "Blowout Creek", introduces a significant sediment load to the EFSFSR and Meadow Creek due to a 1965 water dam failure in the upper EFMC watershed. The sediment is attributed to ongoing erosion of the gully and alluvial fan created by the "blowout", and degrades the quality of the salmon spawning gravels in Meadow Creek and beyond. A rock drain would be constructed in the eroded gully to separate the streamflow from sediment sources, stabilize the gully invert from further erosion, and develop a work area in the gully for fill placement, erosion control, revegetation, and ultimate restoration of a stable surface channel. At the top of the rock drain, a grade control structure would be installed to raise groundwater levels and thereby restore wetlands in the meadow upstream of the gully, where the former impoundment was located. Closure and mitigation features are discussed further on Section 20.

18.9.4 Contact Water Management

Contact water is surface water that has come into contact with the mine pits, ore stockpiles, spent leached ore (e.g., SODA and Hecla heap), historical tailings, development rock, or any other mining-related surface. Contact water may require active or passive treatment during construction, operations, and/or closure prior to discharge to the environment. For water management, as opposed to permitting purposes, contact water is not differentiated from mine drainage, which, under water quality regulations, includes runoff from pits but not necessarily development rock. Water management planning is predicated on the assumption that the same water quality standards apply to both and would drive the need to treat or manage mine-impacted water in advance of or in lieu of discharge, rather than classification by source facility. However, influent water quality may differ from various sources and seasonally (e.g., pits vs. the TSF Buttress), and treatment would be targeted accordingly.

Process water (including reclaim water pumped from the TSF) is addressed separately from contact water; see Section 18.8.5.2. Precipitation and snowmelt runoff or excess dust control water from the mine facilities will be collected, contained, and segregated from surface waters that are not in contact with mine facilities. The disposition of collected contact water will vary by facility, by water quality, seasonally, and from year-to-year depending on a number of factors including stored water inventory, anticipated dewatering and process makeup demand, and capacity of water treatment, evaporation, and storage facilities.

Contact water from the plant site, ore stockpiles, TSF Buttress, SODA/Bradley Tailings reprocessing operation, and Hecla heap would be collected and contained in ponds or sumps sized appropriately for their respective catchment area. Water would be retained in these ponds to settle sediments, then pumped to one of several locations – the ore processing facility for direct re-use, the tailings impoundment, evaporators located at the tailings impoundment or open pits, other contact water ponds for equalization storage, or to the water treatment plant for treatment and discharge.

Contact water originating in the pits (including surface runoff, snowmelt, and groundwater seepage not captured by dewatering wells) would be collected in sumps within the pits and pumped out to contact water storage ponds for reuse or disposal (after treatment). Small amounts of water (approximating summer/fall dust control needs) would be retained in the sumps and pumped out as needed for use in dust suppression in the pits. Surplus contact water collected in the pits and transferred initially to ponds would be reused, evaporated, treated for discharge, or pumped to the TSF for future use as reclaim/process makeup. Pits provide additional reserve storage for contact water generated from other facilities during unusually wet conditions, and spillways from contact water ponds route to pits where feasible rather than directly discharging to waterways in the event that water flows over the spillway.

Runoff from roads with the potential to be in contact with process reagents or hydrocarbons during vehicle maintenance or loading and unloading would be collected. Stormwater from other roads outside of the plant site, stockpiles, pits and

DRSFs would be treated locally with small-scale sediment control best management practices to remove sediment prior to discharge.

Water collected from the growth media stockpile in Fiddle valley would be used for dust suppression, used for irrigation of reclamation plantings, or discharged following settling of sediment.

Contact water management features would be phased in and out as mining progresses and the amount of surface area generating contact water increases as pits and DRSFs expand, and later shrink as backfilling and reclamation is completed. Figure 18-16 and Figure 18-17 show the planned contact water storage ponds, transfer pipelines, forced evaporation sites, and water treatment plant for a representative mine plan year. Table 18.3 provides additional detail on the contact water storage ponds.

Table 18.3: Water Storage Ponds Summary Information

ID	Location	Storage Volume (ac-ft)[2]	Operational Years (inclusive)	Facilities Served[1]
North Plant Pond	Processing Plant	6	-2 to 17	Process Plant, Crusher ROM Stockpile, Crusher
Central Plant Pond	Processing Plant	6	-2 to 17	Process Plant
Scout Pond	Processing Plant	5	2 to 15	Scout ROM Stockpile, Crusher ROM Stockpile
North Truck Shop	Truck Shop	3	-2 to 17	Truck Shop
South Truck Shop	Truck Shop	26	-2 to 17	Truck Shop, HF Stockpiles
HF Pond	Hecla heap	236	-2 to 4	TSF embankment, TSF Buttress, HFP
SODA Pond	TSF Buttress	166	3 to 17	TSF Buttress, HF Stockpiles
MN Pond	Midnight Townsite	95	-2 to 15	YPP, WEP
WE Pond	Lower West End Dump	34	-1 to 9	WEP, WE In-Pit Stockpile
TSF Water Storage Basins	TSF	200	23 to 40	TSF Cover[3]
Fiddle Pond	Lower Fiddle Creek valley	3	-2 to 24	Fiddle GMS

[1] Facility names abbreviated as follows: YPP = Yellow Pine pit, WEP = West End pit, HFP = Hangar Flats pit, TSF = Tailings Storage Facility.
[2] Volumes reported at pond rim. Active storage volumes are slightly lower due to freeboard and spillway depth.
[3] Cover runoff commingles with consolidation water, requiring collection and treatment through approximately year 40.

18.9.5 Water Treatment and Disposal

Three water types will require treatment over the life of the Project: contact water from mine facilities, which includes dewatering water (construction through closure); process water from the TSF (closure); and sanitary wastewater (construction through early closure). During operations, treating and releasing contact water is generally limited to periods when a significant amount of dewatering water is being produced, or seasonally in wet years. Outside of that time, much of the collected contact water can be put to beneficial use by storing that water into the summer and fall. During construction and at closure, absent a water demand for ore processing, less contact water can be consumed and proportionally more must be disposed of through evaporation or treatment and discharge. From construction through early closure, the camp and offices will produce sanitary wastewater needing treatment. Water quality standards, treatment technology selection, and water balance are further discussed in Section 20. Table 18-4 summarizes the phased water treatment capacity and treatment rates throughout the life of the Project. Due to contact water runoff seasonality, reuse, and equalization storage, average treatment rates are often significantly less than treatment capacity, except during Hangar Flats dewatering when a substantial proporotion of treated water is from relatively constant dewatering flows. Treatment plant and outfall locations are shown on Figure 18-16 and Figure 18-17.

Site-specific discharge standards may be negotiated with regulators as part of discharge permitting (Idaho Pollutant Discharge Elimination System, or IPDES). Should site-specific standards, more in line with baseline or background water quality, be established, water treatment costs and duration may be reduced.

Table 18-4: Mine-Impacted Water Treatment Summary

Mine Phase [years]	Installed Capacity (gpm)	Peak Treatment Rate (gpm)[1]	Mean Treatment Rate (gpm)[2]	Constituents of Concern	Treatment Location
Construction [-3 to -1]	300	60 to 300	20 to 130	As, Sb	Distributed
Early Operations [1 to 3]	1,000	0 to 100	0 to 12	As, Sb	Process facility
Middle Operations [4 to 6]	2,000	850 to 1,900	330 to 1,300	As, Sb	Process facility
Late Operations [7 to 15]	1,000	0 to 300	0 to 50	As, Sb	Process facility
Early Closure [16 to 23]	1,000	1,000	150 to 240	As, Sb, Hg, CN	Process facility
Late Closure [24 to 40]	1,000	750 to 900	120 to 270	As, Sb, Hg, CN	TSF buttress

[1] Peak treatment rate range over mine phase, for monthly 50th percentile water balance simulation result.
[2] Mean treatment rate range over mine phase, for monthly 50th percentile water balance simulation result.

18.9.5.1 Contact Water

Water quality permitting discussions are ongoing, but it is likely that the Project will need to adhere to stringent surface water quality standards for arsenic and antimony. Thus, coupled with the timing of water treatment needs with respect to the mining sequence and dewatering excess, treatment methods and capacity will be phased. During construction and early in operations, a modular, mobile, rented iron coprecipitation system is planned. Early in operations, this system would be replaced by a two-train iron coprecipitation system located at the ore processing facility. Sludge from the clarifiers during construction would be stored in a small impoundment in the TSF footprint or on previously disturbed land at SODA. During operations, the sludge would be stored on-site in the TSF.

The total area of the Project that would generate contact water varies though the life of the Project as various facilities come online, expand, and are closed. This is met with a staged water treatment strategy. The construction time period is paired with 300 gpm of peak capacity from package iron coprecipitation plants. The first three years of operations would require 1,000 gpm of total treatment capacity, using an iron coprecipitation plant that would remain until closure. During peak simultaneous dewatering of the Yellow Pine pit and the Hangar Flats pit, an additional 1,000 gpm of modular water treatment capacity will be brought online for approximately three years, then treatment capacity would be scaled back to 1,000 gpm for the remainder of operations and early closure. At closure, the plant would be modified to accommodate treatment of water from the TSF (Section 18.8.5.2). Later in closure, the plant would be relocated to the TSF Buttress as the TSF would be the only remaining water source requiring treatment.

Enhanced evaporation, using snowmaker-style misters located over the TSF, ponds, and/or pits, will supplement the treatment system, in particular to prevent surplus process water accumulation in the TSF and eliminate contact water inventory, if necessary, in the Hangar Flats or SODA ponds. Treatment and enhanced evaporation differ in their relative effectiveness, efficiency, usefulness in cold/wet conditions, and applicability to variable inflow water quality. Approximately 3,600 gpm of nominal evaporator capacity (i.e., throughput, which exceeds actual volume evaporated according to unit efficiency) will be available during operations and early closure (through year 17), then scaled back to approximately 1,200 gpm until the TSF is covered in approximately year 23.

After mine closure and final reclamation of the TSF Buttress and pit backfill surfaces, contact water treatment will no longer be required; process water treatment for the TSF (Section 18.8.5.2) will continue longer, through approximately year 40.

18.9.5.2 Process Water

There are no plans to treat process water for discharge during normal mine operations. Ore processing is a significant consumer of water due to evaporation inside of the process plant, and at the TSF from the tailings and supernatant pool surfaces and, most significantly, burial of entrained water with the tailings.

The TSF will be operated as a zero discharge facility, and accumulated water on the TSF, originating from incoming slurry, tailings consolidation, or precipitation, will be returned to the processing facility for reuse. In the event of excess water accumulation in the TSF, excess water would be disposed of using enhanced (mechanical) evaporation or prioritized for reclaim and reuse in ore processing (treating to reuse standards at the process plant if necessary). In the latter scenario, no contact water would be introduced to the process circuit, and contact water would be evaporated or treated for discharge instead of process water. As an emergency measure, package reverse osmosis units could be brought online to treat excess water. As maximum runoff volumes at site are driven by snowmelt, the risk of such a situation developing could be readily identified in advance based on snowpack measurements and equipment staged accordingly.

At closure, remaining water inventory on the TSF would be eliminated by a combination of mechanical evaporation and active water treatment. Under EPA regulations, the maximum annual process water treatment volume is limited to the net of annual precipitation and evaporation. Cover would be placed on the facility as surface conditions allow use of equipment.

The post-closure period begins after the placement of the cover material and the restoration of Meadow Creek to a lined floodplain corridor in the center of the TSF (Section 20). In post-closure, active water treatment would continue until water quality standards can be met either without treatment or with passive treatment methods, but the treatment plant will be relocated to private land on the TSF Buttress to minimize pipeline length and head, and flow equalization would be provided by shallow water storage basins on the TSF on either side of the Meadow Creek corridor. Treatment is predicted to be necessary until approximately year 40 (approximately 25 years after closure) when consolidation water inflow to the cover is predicted to be minimal. Once this threshold has been achieved the remaining diversions on the perimeter of the facility will be removed, and hillside runoff would be routed over the cover. As pilot studies for passive treatment have not been completed, closure costs are estimated assuming active treatment would be continued until no treatment was indicated. Passive treatment would be adopted as flows, water quality, and effectiveness permit.

18.9.5.3 Sanitary Wastewater

Early in construction, the currently permitted membrane bioreactor (**MBR**) plant at the existing exploration camp would be used, and treated effluent reused for flushing toilets and urinals or discharged to the existing permitted drain field, while the worker housing facility and its associated treatment plant is under construction. During operations and closure, sanitary wastewater from the worker housing facility, ore processing facility, and administration buildings would be treated at a new MBR or similar plant located at the worker housing facility and discharged to the EFSFSR via a permitted IDPES outfall. Vaults or portable toilets would be utilized at offsite facilities and remote locations onsite (TSF, pits, maintenance facility etc.), and serviced as needed using vacuum trucks. Treatment residuals would be hauled offsite to a permitted sanitary landfill. Vault/portable toilet wastewater would be hauled to a public / municipal wastewater treatment plant.

18.10 REFERENCES

HDR Inc. (2017). Traffic Impact Study, prepared for Midas Gold Idaho, Inc.

M3 Engineering & Technology (2014). Stibnite Gold Project Prefeasibility Study Technical Report, prepared for Midas Gold, December 8, 2014, amended March 28, 2019.

United States Forest Service, United States Department of Agriculture (2011, with partial amendments in 2014). "FSH 7709.56 – Road Preconstruction Handbook." Washington, DC.

United States Forest Service, United States Department of Agriculture (2014). "FSH 7709.56b – Transportation Structures Handbook." Washington, DC. Published November 24, 2014.

Valley County (2008). "Valley County, Minimum Standards for Public Road Design and Construction". Cascade, Idaho. Published April 2008.

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19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

19.1.1 Doré

The economic analysis completed for this FS assumed that gold and silver production in the form of doré could be readily sold without deleterious element penalties. Assumed gold and silver doré payabilities, refining and transport charges are provided in Table 19.1; these values are considered typical.

Table 19.1: Dore Payables, Refining and Transportation Assumptions

Parameter	Gold in Doré	Silver in Doré
Metal Payability in Doré	99.5%	98.0%
Refining Charges	$1.00/oz Au	$0.50/oz Ag
Transportation Charges	$1.15/oz Au	$1.15/oz Ag

19.1.2 Antimony Concentrate

A market study for the sale of antimony concentrate was completed by a confidential independent leading industry participant. The marketing study was based on preliminary antimony concentrate production estimates, and ranges for projected antimony, gold, silver, and deleterious element grades in the concentrate. The following information was derived from the antimony market study:

- Approximately 175,000 tonnes of antimony is presently produced annually around the world. One fifth of the production is from recycling while the remaining four-fifths result from primary production.
- The antimony concentrate production profile of this Project, based on the mine plan provided in Section 16, would make it one of the largest antimony producers outside of Asia.
- Antimony concentrate payables would potentially be:
  - 65 to 72% payable for an antimony concentrate with a grade of 55 to 60% antimony, respectively, with no treatment or refining charges and no minimum deductions.
  - deleterious element charges may apply, particularly for selenium.
  - gold would not be subject to refining or other deductions and would yield payables of 20 to 25% for the first five years of production where the gold content was greater than 8.5 g/t Au. In the later years of production it is likely that gold content would receive payability of 15 to 25%.
  - silver would not be subject to refining or other deductions and would yield payables of:
    - 40 to 50% for concentrate silver grades of 300 to 700 g/t, respectively; and
    - 50% for concentrate silver grades greater than 700 g/t.
- Currently only a small number of smelters, all of them located in Asia, have the capacity to treat the tonnage of antimony concentrate planned for production by the Project. However, other domestic and international smelting possibilities outside of Asia may be viable alternatives when the Project is operational.

Based on the payability information provided by an independent leading industry participant, and on the concentrate transportation costs discussed in Section 18, Table 19.2 summarizes the antimony concentrate payables and transportation charge assumptions for this study.

Table 19.2: Antimony Concentrate Payables and Transportation Assumptions

Parameter	Concentrate Payables and Transportation Charges
Antimony Payability	Constant at 68% (based on a constant life-of-mine concentrate grade of 59%)
Gold Payability	<5.0 g/t Au no payability ≥5.0 g/t ≤8.5 g/t Au payability of approximately 15 - 20% ≥8.5 g/t ≤10.0 g/t Au payability of approximately 20 - 25% ≥10.0 g/t Au payability of approximately 25%
Silver Payability	<300 g/t Ag no payability ≥300 g/t ≤700 g/t Ag payability of approximately 40 - 50% ≥700 g/t Ag payability of approximately 50%
Transportation Charges	$151/wet tonne from site to Asia

19.2 METAL PRICES

The metal prices selected for the four economic cases in this report are shown in Table 19.3; the basis for selection of these metal prices is also provided in the table.

Table 19.3: Assumed Metal Prices by Case

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(2) ($/lb)	
Case A	1,350	16.00	3.50	Lower bound case defined by the approximate 5-year trailing average gold price and consistent with the gold price used in the PFS (M3, 2014).
Case B (Base Case)	1,600	20.00	3.50	Base case derived from the weighted average of the 3-year trailing gold price (60%) and the 2-year gold futures price (40%).
Case C	1,850	24.00	3.50	Case corresponds to the approximate spot gold price at the effective date of this report.
Case D	2,100	28.00	3.50	Case corresponds with a gold price at approximately the peak 2020 spot price.
Case E	2,350	32.00	3.50	Upper bound case provides investors with insight into the revenues generated by the Project at a sustained elevated long-term gold price.

Note:
(1) *The base case silver price was set at a gold-silver ratio ($/oz:$/oz) of 80:1 or $20/oz. The base case price was then varied similar to the way the gold price was varied (in this case by $4/oz Ag versus $250/oz Au) for the other cases.*
(2) *Antimony prices were assumed to be constant at $3.50/lb for all cases as antimony does not historically vary proportional to the gold and silver prices and is not expected to do so in the future. The $3.50/lb price was derived from a market study undertaken by an independent expert in antimony markets.*

There is no guarantee that the gold, silver, and antimony prices used in the study cases would be realized at the time of production. Prices could vary significantly higher or lower with a corresponding impact on Project economics.

19.3 CONTRACTS

There are no mining, concentrating, smelting, refining, transportation, handling, sales and hedging, forward sales contracts, or arrangements for the Project. This situation is typical of a project that is still several years away from production.

19.4 REFERENCES

M3 Engineering & Technology (2014). Stibnite Gold Project Prefeasibility Study Technical Report, prepared for Midas Gold, December 8, 2014, amended March 28, 2019.

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20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACTS

As discussed in Section 6 of this Report and summarized in Appendix D of the PRO (Midas Gold, 2016), the Stibnite area has been mined extensively over the past century. Historical mining activities have altered the topography, hydrology and ecology of the District and left significant mining wastes that continue to impact soil, surface water and groundwater quality. Environmental studies and investigations in the Stibnite area conducted over the past several decades and summarized herein have identified and/or characterized these impacts which represent Recognized Environmental Conditions (**REC**s).

Cleanup efforts undertaken at the site by Federal and State agencies and private companies have been conducted pursuant to multiple cooperative agreements and have included stream improvements, tailings reclamation efforts, facility removal and cleanup, surface disturbance reclamation, and specific cleanup projects under the Comprehensive Environmental Response, Compensation, and Liability Act (**CERCLA**). These projects have provided incremental improvements to water quality and overall site conditions; however, numerous legacy materials and disturbances remain and continue to degrade aquatic and terrestrial wildlife habitat and impact surface water and groundwater quality. These conditions are compounded by extensive forest fire impacts and subsequent damage from soil erosion, landslides and debris flow, and resultant sediment transport.

Midas Gold submitted the PRO for mining on National Forest System (**NFS**) lands to the United States Forest Service (**USFS**) in September 2016, in accordance with USFS regulations for locatable minerals set forth in 36 Code of Federal Regulations (CFR) 228 Subpart A. Alongside mining, the PRO proposed cleanup, mitigation, and reclamation of legacy mining impacts before, during, and after the proposed mining activities. The Project was designed to align with Midas Gold's core values, conservation principles and sustainability goals. The USFS accepted the PRO as administratively complete in December 2016 and began to process the application per their responsibility under the National Environmental Policy Act (**NEPA**). During the NEPA analysis, Midas Gold identified further improvements to Project environmental performance, and submitted a modified PRO (**ModPRO**) in May 2019 (Brown and Caldwell, 2019) as an additional alternative to be analyzed in the Draft Environmental Impact Statement (**DEIS**). The DEIS (U.S. Forest Service, 2020) was released for public review on August 14, 2020 and analyzed environmental effects for the Proposed Action, three action alternatives including the ModPRO, and the No Action Alternative. The USFS did not identify a preferred alternative in the DEIS.

Concurrent to and following the preparation and public review of the DEIS, Midas Gold has continued to study alternatives that reduce the overall Project footprint, reduce associated wetland impacts, improve surface water and groundwater quality, reduce air emissions, improve fisheries and wildlife habitat, and improve upon the reclamation and restoration designs. Further, comments received from agencies during the past several years, and from the public during the USFS comment period on the DEIS, which closed on October 28, 2020, identified additional opportunities for improvements. These adjustments are incorporated in this Report and represent responses to comments and suggestions that reduce or mitigate impacts for an improved environmental outcome. It is intended that they be incorporated into the preferred alternative in the Final EIS (**FEIS**) currently being prepared by the USFS as further mitigations to impacts vs. the action alternatives.

The following sections provide information on historical and recent site characterization efforts, existing environmental conditions, status of project approval and permitting efforts, social and community considerations, proposed mitigation of stream and wetland disturbance, and reclamation and closure activities.

20.1 ENVIRONMENTAL STUDIES

20.1.1 Historical Environmental Studies

Mining operations that occurred at the site prior to 1970 were not subject to modern, rigorous environmental regulations. Thus few, if any, environmental studies were prepared for the site prior to 1970 and much of our understanding of pre-mining conditions is speculative and relies upon data from more recent investigations.

Historical environmental studies and effects analysis conducted for the site supported the preparation of Environmental Impact Statements for Superior's and Hecla's legacy mining and heap-leaching operations for the West End and Homestake mines, respectively. These operations were permitted in the 1980s and included subsequent expansion of West End mining activities in the 1990s, as discussed in Section 6 of this Report. More recent investigations have assessed the impacts of these and other legacy operations. An environmental site characterization was conducted at the Project site from 1998 through 2000 by URS for the USFS and the EPA (the "URS Report"; URS, 2000). This study included chemical, biological and habitat characterization and determined existing site conditions posed no unacceptable risks to the environment or human health; however, it did document continued metal release to surface water and groundwater downgradient of legacy mining impacts. Subsequent to the URS Report, Millennium Science and Engineering Inc. (**MSE**) conducted additional investigations and published an Engineering Evaluation and Cost Analysis (**EE/CA**) in 2003.

Several of the patented lode and mill site claims acquired, or under purchase option by Midas Gold, are subject to consent decrees under CERCLA that provide regulatory agencies the right to conduct remediation activities, and place limitations on activities that would adversely affect the integrity of any remedial measures implemented by government agencies. Pursuant to CERCLA and the Resource Conservation and Recovery Act (**RCRA**), the EPA, the Forest Service, and the State of Idaho have jointly conducted removal and remediation actions on site. The United States Geologic Survey (USGS) monitored water quality on the site between from 1983 to 1996 and re-initiated monitoring in 2011; USGS scientists have prepared multiple reports on water quality and aquatic biota.

Although some portions of the Project site were placed on the Federal Facilities Docket on September 25, 1991, and are currently listed on the Comprehensive Environmental Response, Compensation, and Liability Information System (**CERCLIS**) List (No. ID9122307607), in 2001 both the EPA and the Bureau of Environmental Health and Safety (**BEHS**), Division of Health, Idaho Department of Health and Welfare determined the risk to be too low for listing the site on the National Priorities List (**NPL**).

20.1.2 Midas Gold Environmental Studies

In 2009 and 2010, Midas Gold and Vista US contracted MSE to conduct Phase I and Phase II Environmental Site Assessments (**ESA**s) to identify RECs in connection with the Property. These studies are necessary to fulfill obligations for undertaking "all appropriate inquiry" as to site conditions as a requirement of satisfying the bona fide prospective purchaser, contiguous property owner, and the "innocent landowner" affirmative defenses under CERCLA. The ESAs identified a number of RECs, but none were categorized as imminent threats to human health or the environment; however, the ESAs indicated that overall water quality in all drainages was impaired due to naturally occurring mineralization and impacts associated with historical mining.

In 2011, Midas Gold initiated an environmental resource baseline data collection program to establish the existing environmental conditions, identify and quantify environmental risks and liabilities, monitor for potential impacts from onsite activities, and generate baseline reports for project approval and permitting efforts. The environmental baseline work plans were approved by USFS subject matter experts for each of the resource categories, with input from representatives from additional state and federal agencies. Table 20-1 summarizes the nature, timeframe, and contractors responsible for Midas Gold's environmental baseline studies. While baseline monitoring reports were initially submitted in 2017 in support of NEPA analysis, certain of the studies continue to provide monitoring data, and additional supplementary studies have also been prepared per adequacy review from the agency interdisciplinary teams convened for the NEPA analysis.

Table 20-1: Midas Gold Environmental Baseline Studies

Baseline Resource	Baseline Study Document(s)	Preparers	Date
Air Quality	Air Quality Baseline Study	Stantec Consulting	Apr 14, 2017
Aquatics	Aquatic Resources Baseline Study	MWH Americas	Apr 28, 2017
	Aquatic Resources 2016 Baseline Study - Addendum Report	GeoEngineers	Jul 19, 2017
Cultural	Cultural Resources Baseline Studies 2011-2017 Summary Report	HDR	Apr 14, 2017
Environmental Justice	Environmental Justice Baseline Study	HDR	Apr 14, 2017
Geochemistry	Phase 1 Baseline Geochemical Characterization Report	SRK	May 2, 2017
	Phase 2 Baseline Geochemical Characterization Report		May 5, 2017
Geology	Geological Resource Baseline Study	MGI	May 19, 2017
Geotechnical	Geotechnical Summary Report	STRATA	May 19, 2017
	Geotechnical Investigations Summary Report	Tierra Group	Dec 12, 2018
Groundwater Hydrology	Groundwater Hydrology Baseline Study	Brown and Caldwell	Jun 30, 2017
Groundwater Quality	Groundwater Quality Baseline Report	HDR	Jun 30, 2017
Hazardous Materials	Hazardous Materials Baseline Study	HDR	Apr 28, 2017
Land Use	Land Use Baseline Study	HDR	Apr 14, 2017
Noise	Noise Baseline Study	HDR	Apr 28, 2017
Public Health/ Safety	Public Health and Safety Baseline Study	HDR	Apr 28, 2017
Recreation	Recreation Baseline Study	HDR	Apr 14, 2017
Socioeconomics	Socioeconomic Baseline Study	Univ. of Idaho	Apr 28, 2017
Soils	Soil Resources Baseline Study	MGI	Apr 28, 2017
	Soil Salvage Report	Tetra Tech	Dec 20, 2017
Surface Water Hydrology	Surface Water Hydrology Baseline Study	HydroGeo	Jun 30, 2017
Surface Water Quality	Surface Water Quality Baseline Study	HDR	Jun 30, 2017
Transportation	Transportation Baseline Study	HDR	Apr 26, 2018
Vegetation	Vegetation Baseline Study Vegetations Baseline Study Addendum	HDR	Apr 14, 2017 Apr 26, 2018
Visual	Scenic Resources Baseline Study	HDR	Apr 14, 2017
	Key Observation Points and Viewshed Simulations	Tetra Tech	Jan 15, 2019
Water Resources	Water Resources Baseline Summary Report	Brown and Caldwell	Jun 30, 2017
Water Rights	Water Rights Baseline Study	HDR	Apr 19, 2017
Wetlands	Wetland Resources Baseline Study, Addendums, and Jurisdictional Determination	HDR	Apr 19, 2017
Wildlife	Terrestrial Wildlife Baseline Study	Strobilus Environmental	Dec 1, 2013
	Terrestrial Wildlife Baseline Study Updates	Garcia & Associates	Apr 14, 2017 Apr 26, 2018

20.1.3 Environmental Modeling

In 2017, Midas Gold contracted Brown and Caldwell, Air Sciences, and SRK Consulting to develop predictive models for use in environmental evaluation of the Stibnite Gold Project and feasibility level engineering studies. Environmental models include air emissions modeling, the Hydrologic Model and meteoric water balance, Stream and Pit Lake Network Temperature Model **(SPLNT),** Site-Wide Water Chemistry **(SWWC)**, and Site-Wide Water Balance **(SWWB)**. The modeling process involved development of conceptual models, work plan approval by the regulatory agencies, development and calibration of existing conditions models, and development of predictive models for the proposed action and alternatives to the proposed action. In addition, Midas Gold developed an additional, more detailed, site-wide water balance model for the Feasibility Study, to facilitate rapid evaluation of alternate design scenarios and perform trade-offs. Environmental modeling has been a key tool for advanced engineering and identification of appropriate mitigation measures.

20.1.3.1 Air Quality

Midas Gold contracted Air Sciences Inc. to complete detailed life-of-mine air emission predictive modeling for the Stibnite Gold Project (Air Sciences, 2019). The modeling was completed to fulfill the applicable requirements of Idaho Administrative Procedures Act (**IDAPA**) 58.01.01 – Rules of Control of Air Pollution in Idaho, and to obtain a minor source permit to construct. The modeling encompasses mining operations and ore processing activities and uses the AERMOD modeling system to model air dispersion based on planetary boundary layer turbulence structure and scaling concepts, including treatment of both surface and elevated sources, and both simple and complex terrain. Air emissions were estimated for the following:

- Criteria air pollutants: CO, NO_X, $PM_{2.5}$, PM_{10}, SO_2, Pb, and O_3 precursor VOCs;
- Applicable HAP from Section 112(b) of the Clean Air Act;
- Applicable TAP listed in IDAPA 58.01.01 Sections 585 and 586; and,
- Greenhouse gases: Carbon dioxide, methane, nitrous oxide, and carbon dioxide equivalent.

Given the SGP's proximity to Federal Class I areas, CALPUFF, a non-steady-state meteorological model, was also used to assess long range transport of pollutants and VISCREEN was used to estimate the potential impact of a plume of specified emissions for specific transport and dispersion conditions. The modeling results indicate that the total concentrations from the SGP do not exceed the applicable National Ambient Air Quality Standards (40 CFR part 50).

20.1.3.2 Hydrology and Hydrogeology

The Hydrologic Model predicts surface and groundwater flows for the operational and post-closure period and consists of a meteoric water balance that incorporates precipitation, infiltration, snow accumulation and melt and a MODFLOW-NWT numerical groundwater model. The hydrologic model was calibrated to baseline surface and groundwater monitoring data and is used to predict effects of proposed mining activities on groundwater levels and stream flows as well as operational water management requirements. The hydrologic model provides input data to SPLNT and SWWC models and is coupled with the SWWB model, providing input and receiving output from it.

The hydrologic model predicts streamflow depletions, particularly in Meadow Creek, during active dewatering and post-mining recovery of groundwater around the Hangar Flats pit and assisted in identifying mitigating measures to address these issues. Groundwater and thus streamflow depletions will be mitigated by a combination of lining channels with low-permeability geosynthetics, discharging treated excess dewatering water directly to streams for streamflow augmentation, withdrawing a portion of makeup water from a surface water intake at the EFSFSR tunnel farther downstream instead of wells in Meadow Creek valley, and backfilling pits (reducing the volume and time required to saturate backfill as opposed to filling an empty pit to form a lake).

20.1.3.3 Water Temperature

The SPLNT model is used evaluate the effects of proposed mining activities on stream temperature. It combines QUAL2K simulations for stream reaches with GLM (General Lake Model) simulations for pit lakes. The model is based on streamflow inputs from the hydrologic model, topographic and vegetative shading factors, meteorological inputs and simulations of heat transfer within pit lakes and stream channels.

SPLNT modeling indicated increases in stream temperatures due to loss of shade during both mine operations and for significant duration into closure as reclamation plantings grow to full height. Predicted temperature impacts will be mitigated by piping diverted summer low flows during operations and early closure, by changes to the width and vegetation species composition of riparian plantings along both restored and enhanced stream reaches, by a rock drain at Blowout Creek, and by inclusion of a small in-line lake on the EFSFSR within the lined floodplain corridor over the Yellow Pine pit backfill that mimics the temperature-moderating function of the present pit lake.

20.1.3.4 Geochemistry

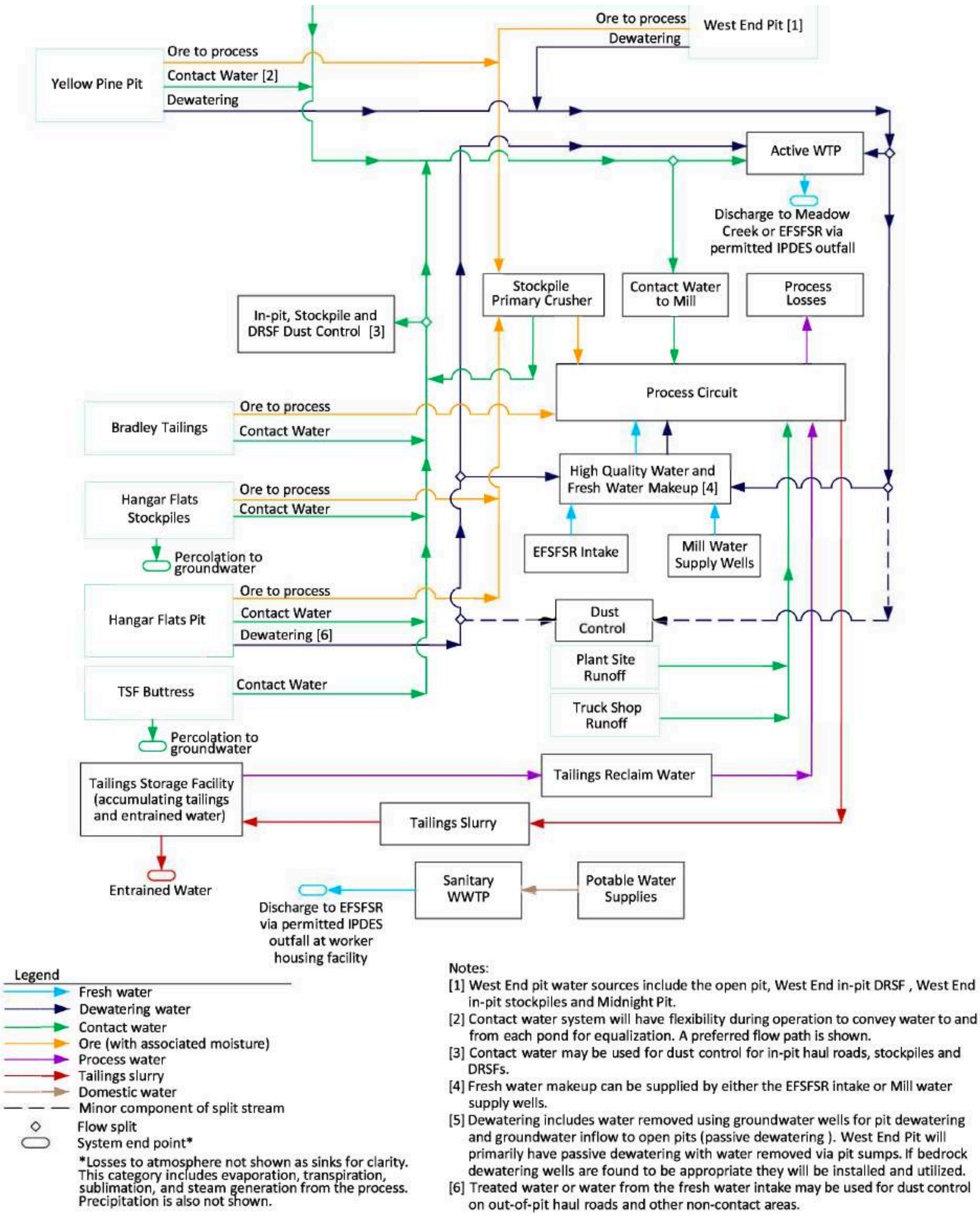
The SWWC model is used to assess ground and surface water quality changes resulting from proposed Project activities. The model predicts and aggregates constituent concentrations in ground and surface water at multiple prediction nodes downgradient of mining facilities for both operational and post-closure periods. Water quality predictions are based on estimated inflows to ground and surface water from each facility derived from the SWWB and hydrologic model; source term estimates of water quality for each facility based on geochemical characterization testing and PHREEQC geochemical equilibration modeling; estimated improvement due to removal of legacy mining facilities; and estimates of constituent dilution in surface water based on inputs from the hydrologic model. Source terms for certain legacy materials, tailings, development rock, and pit-wall runoff are based on scaled humidity cell release rates and geologic material types defined in the mineral resource block models.

SWWC analysis indicates that while ARD potential is negligible, neutral metal leaching of arsenic and antimony from legacy materials and natural mineralization occurs today (consistent with the conclusions of the water quality studies summarized in 20.1.1 and 20.1.2), has the potential to occur in the future from Project development rock and tailings, and could impact both surface water and groundwater quality. Collecting and treating contact water before discharge (during all major Project phases) and installing a low-permeability closure cap on the TSF buttress mitigate potential water quality impacts associated with development rock produced by the Project. TSF consolidation water will require treatment at closure to meet discharge standards for arsenic, antimony, mercury, and/or cyanide.

20.1.3.5 Water Balance

The SWWB model accounts for production, usage, reuse, consumption, handling and storage of process water, contact water and dewatering water over the course of the project. For use in conjunction with the operations-phase hydrologic model, the SWWB simulates variability in water volumes for a climatic scenario representative of a typical 14-year period within the available 122-year record, but containing both wet and dry years, using a probabilistic approach wherein every mine year is analyzed for every climate year within a given scenario. Additional output is generated using the full record, with water years sampled randomly, enabling assessment of the full range of climate variability. The SWWB uses monthly groundwater dewatering and runoff yield computed with the hydrologic model and meteoric water balance calculations, and annually varying facility (pit, TSF, buttress, backfill, and process plant) configurations based on the Feasibility Study mine plan. Tailings consolidation was modeled in CONDES based on the results of seepage-induced consolidation tests performed on representative full-flowsheet pilot plant tailings samples, and the resultant tailings density time series imported to the SWWB. Figure 20-1 shows the water balance flow diagram identifying inputs, outputs, and transfers.

Figure 20-1: General Water Management Flow Diagram



Notes:

[1] West End pit water sources include the open pit, West End in-pit DRSF , West End in-pit stockpiles and Midnight Pit.
[2] Contact water system will have flexibility during operation to convey water to and from each pond for equalization. A preferred flow path is shown.
[3] Contact water may be used for dust control for in-pit haul roads, stockpiles and DRSFs.
[4] Fresh water makeup can be supplied by either the EFSFSR intake or Mill water supply wells.
[5] Dewatering includes water removed using groundwater wells for pit dewatering and groundwater inflow to open pits (passive dewatering). West End Pit will primarily have passive dewatering with water removed via pit sumps. If bedrock dewatering wells are found to be appropriate they will be installed and utilized.
[6] Treated water or water from the fresh water intake may be used for dust control on out-of-pit haul roads and other non-contact areas.

The TSF has a negative water balance in isolation, due to reclaim to the ore processing facility, burial of pore water with the tailings, and evaporation; however, the Project site has a seasonally positive water balance in certain years, due to limited water storage capacity and the production of contact water from other mine facilities – particularly leading up to the midpoint of mine life, when both Hangar Flats and Yellow Pine pits are being mined, and contact stormwater and dewatering water are maximized. This leads to the need to dispose of excess water seasonally for the life of mine and into closure, using a combination of mechanical evaporation and water treatment and discharge. The SWWB facilitates optimization of contact water reuse, storage, TSF reclaim, and dewatering water disposal to maximize reuse and minimize water treatment cost.

One result of this is that the processing plant will prioritize use of contact water during and after spring melt (when ponds are full) instead of TSF reclaim, surface water or groundwater, returning to use of TSF reclaim water to draw down the supernatant pool through the summer to a minimal state, preventing carryover storage into the following melt season. In unusual situations, water can be transferred to pits, allowed to remain in pits, or transferred directly to the TSF to prevent any release of untreated water. The planned water treatment plant capacity and storage volume available in ponds prevent the need for water transfers or extended in-pit water storage up to the 95th percentile runoff conditions, and no untreated water would be discharged up to at least the 99^{th} percentile condition. During extended periods of in-pit water storage that prevents in-pit ore mining, ore would be processed from the long-term stockpiles.

20.1.4 Water Treatment

The seasonal water balance excess and predicted leaching of arsenic and antimony from mined materials lead to a need to dispose of water which would not meet the most stringent potentially applicable water quality standards absent treatment. Based on measured and predicted water quality and anticipated discharge water quality standards (typically either the acute cold-water biota or drinking water standards, depending on constituent), dewatering water, seepage, and contact stormwater would require treatment before discharge during operations. Early in closure, seepage and contact stormwater along with TSF water would require treatment before discharge; later, TSF water treatment would extend until approximately 25 years after the end of mill operations. Mechanical evaporation would be used along with active, and potentially passive, water treatment to manage excess water at site.

Midas Gold's consultants developed a water treatment plan, including technology selection, based on the geochemical (**SWWC**) and water balance (**SWWB**) model predictions and application of the most stringent potentially applicable discharge standards. Due to the need to remove arsenic and antimony, while avoiding introduction of chloride into the ore processing circuit, iron coprecipitation (with iron sulfate) was selected for active treatment. Vertical-flow wetlands appear viable as a passive treatment system, for diminishing TSF flows later in closure, subject to additional technology confirmation steps and pilot studies that would be accomplished during operations. Required water treatment capacity varies from construction through closure, according to the site water balance changes and equalization storage capacity, peaking in the middle of operations at approximately 2,000 gpm when both Hangar Flats and Yellow Pine pits are being mined, declining to approximately 1,000 gpm later in operations as facilities are concurrently reclaimed, and continuing until after the TSF is covered to manage commingled tailings runoff and consolidation water. Post-closure water treatment will continue until approximately year 40 (approximately 25 years after the end of ore processing operations). Details of water treatment capacity, phasing, and contact water equalization storage are discussed in Section 18.

Site-specific discharge standards may be negotiated with regulators as part of discharge permitting (Idaho Pollutant Discharge Elimination System, or IPDES). Establishing natural background concentrations, a necessary step in site-specific standards, is challenging as the site is highly mineralized and has elevated metal levels in surface and groundwater due to both natural conditions and legacy features from over 100 years of mining. Should site-specific standards, more in line with baseline or background water quality, be established, water treatment costs and duration may be reduced.

20.2 LITIGATION AND ENVIRONMENTAL RISKS

In August 2019, the Nez Perce Tribe (**NPT**) filed a lawsuit against Midas Gold and its related affiliates under the Clean Water Act (**CWA**) alleging unpermitted water pollution discharges associated with the RECs from historical mining operations. The NPT lawsuit is ongoing as of the effective date of this Feasibility Study.

20.3 PERMITTING

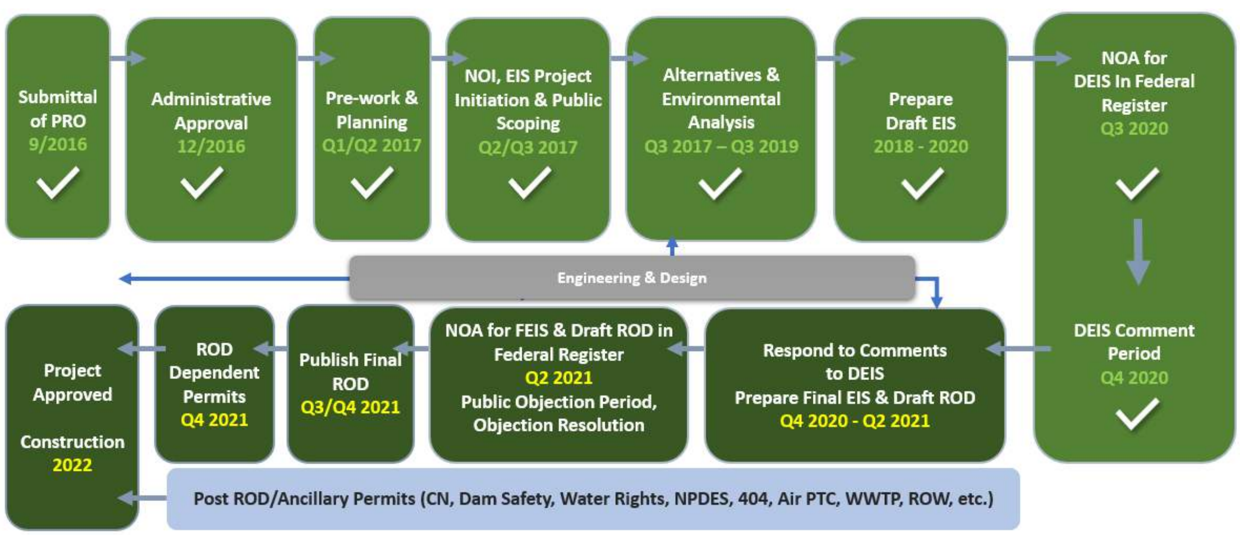
20.3.1 Environmental Impact Statement

USFS approval of the Final Plan of Restoration and Operations (**PRO**) / Reclamation Plan for the Project requires an appropriate level of environmental analysis under NEPA. As with most proposed mining operations that impact federal lands and will generate significant environmental effects, the USFS has determined that the Project requires preparation of an environmental impact statement (**EIS**). Preparation of an EIS under NEPA requires federal agencies to study and consider the likely environmental impacts of the proposed action, compare them to a range of reasonable alternatives to the proposed action, and then move forward with a decision-making process that identifies a preferred alternative, thus dictating discretionary federal action that is determined necessary for the Project to proceed. The final determination of the lead agency is memorialized in a Record of Decision (**ROD**) document. The administrative obligation of the USFS to conduct Project NEPA analysis is provided in the Draft EIS (**DEIS**) document which identifies their statement of purpose and need:

> *The (USFS's) purpose and need is to administratively process Midas Gold's application and reach a decision within the scope of its authority regarding Midas Gold's Plan in a timely manner…The role of the PNF under its primary authorities in the Organic Administration Act, Locatable Minerals Regulations (36 CFR 228 Subpart A), and the Multiple-Use Mining Act (1955, PL 167) is to ensure that mining activities minimize adverse environmental effects on NFS lands and comply with all applicable environmental laws. The PNF may impose reasonable conditions to protect national forest surface resources but cannot materially interfere with reasonably necessary activities under the General Mining Law that are otherwise lawful.*

Figure 20-2 presents the typical sequence of the NEPA process and the status of the Stibnite Gold Project EIS.

Figure 20-2: USFS NEPA Process – Timeline for Stibnite Gold Project



*Note: The timelines presented on Figure 20-2 are based on the current "Schedule of Proposed Activities" (**SOPA**) published by the USFS on Oct.1, 2020 but are subject to adjustment and change as the NEPA process continues to advance.*

The EIS and the related ROD for PRO approval serves as an overarching procedural permitting requirement, as well as that of at least three other primary federal and state authorizations or determinations:

- Idaho Pollutant Discharge Elimination System (**IPDES**) Permit for water discharge [formerly National Pollutant Discharge Elimination System (**NPDES**) – Idaho gained primacy from EPA for enforcement of this section of the CWA in 2018;
- United States Army Corps of Engineers (**USACE**) CWA Section 404 Dredge and Fill Permit and determination of the Least Environmentally Damaging Practicable Alternative (**LEDPA**); and
- Endangered Species Act (**ESA**) Biological Opinion.

The EIS and ROD for the PRO effectively drive the entire permitting process, since a completed final EIS and favorable ROD are generally required before these important clearances can be obtained or utilized.

Other primary federal and state authorizations and/or permits are described in the sections which follow. The discussion ties the EIS and other permitting requirements together in terms of an estimated schedule and costs for completing the program. Table 20-2 provides a summary of the status of the other federal, state and local permitting processes.

Table 20-2: Federal, State and County Permit Applications and Status

Federal Government	Permits and Approvals	Status	Submittal
Forest Service	• Road Use Permit • Mineral Material Permit • Timber Sale Permit and Contract • Powerline SUP	• Planning • Planning • Planning • Planning	• Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS
Army Corps of Engineers	• Clean Water Act Section 404 Permit • 401 Certification	• In Preparation • In Preparation	• Post-FEIS • Post-FEIS
US Bureau of Reclamation	• Transmission Line upgrade permit	• In Preparation	• Post-FEIS
Environmental Protection Agency	• Construction General Permit • Multi-Sector General Permit (2020) • Stormwater Pollution Prevention Plan • Spill Prevention Plan (SPCC) • Clean Water Act Section 401 Certification • EPA Waste Generator ID • SARA Title III – EPCRA • TSCA – TRI	• In Preparation • In Preparation • Issued, Update in Preparation • Planning • In preparation • Planning • Planning • Planning	• Post-FEIS • 4Q 2020 • Post-DEIS • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS
Federal Communications Commission	• Radio Authorizations	• Planning	• Post-FEIS
Bureau of Alcohol, Tobacco, Firearms and Explosives	• Permit for Transporting, Storage and Use of Explosives	• Planning	• Post-ROD
Mine Safety and Health Administration	• Mine Identification Number • Legal Identity Report • Ground Control Plan • Part 48 Training Plan • Commencement of Operations	• Planning • Planning • Planning • Planning • Planning	• Post-ROD • Post-ROD • Post-ROD • Post-ROD • Post-ROD
State of Idaho	**Permits and Approvals**	**Status**	**Submittal[1]**
Department of Environmental Quality	• Air Quality Permit to Construct • Cyanidation Permit (coordinate with IDL) • Idaho Pollutant Discharge Elimination System • Point of Compliance • Wastewater Treatment Permit • Drinking Water Permit • Solid Waste permits • Water Reuse Permit[1]	• Draft Permit in review • In Preparation • In Preparation • In Preparation • Planning • Planning • Planning • Planning	• 4Q 2020 • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS • Post-FEIS

Department of Health and Welfare	• Septic System Approval • Food Establishment License	• Planning • Planning	• Post-FEIS • Post-FEIS
Department of Water Resources	• Water Rights • Mine Tailings Impoundment Structure / Dam Safety approval • Dam Safety approval for contact water ponds	• In Preparation • In Preparation • Planning	• Post-FEIS • Post-FEIS • Post-FEIS
State Historic Preservation Office (SHPO)	• Cultural (SHPO) Clearance	• Planning	• Post-FEIS
Department of Lands	• Mine Operating Plan (PRO) • Mine Reclamation and Closure Plan (RCP) • Mine RCP for Preferred Alternative • Reclamation Financial Assurance	• Completed • In Preparation • In Preparation • In Preparation	• 3Q 2016 • Post-DEIS • Post-FEIS • Post-FEIS
Valley County	**Permits and Approvals**	**Status**	**Submittal[1]**
Planning and Zoning Department	• Conditional Use Permit (numerous)	• Various	• Variable
Building Department	• Building Permits	• Planning	• Post-FEIS
Road Department	• Annual road use permits	• Planning	• Post-FEIS

Note:
1. Permit requirement is under evaluation.

20.3.2 Idaho Pollutant Discharge Elimination System Permit

An IPDES Permit is required for point source discharges from the mining operation to "waters of the United States". In addition, since the Project is subject to performance standards for “new sources” for its respective industrial source category, the Project must demonstrate that it is applying the best available control technology (i.e., technology-based effluent limits, or TBELs) and that the discharge water quality will meet applicable water quality standards (water quality-based effluent limits; WQBELs). WQBELs are almost always more stringent than TBELs, and therefore are expected to control. The IPDES permit application must be submitted at least 180 days prior to the approved discharge.

Stormwater discharges associated with industrial activity that meet certain criteria can be authorized under a related permit, the Multi-Sector General Permit (**MSGP**). Stormwater is defined as "storm water runoff, snowmelt runoff, and surface runoff and drainage". MSGP stormwater would be managed with Best Management Practices according to an approved stormwater pollution prevention plan (**SWPPP**). This document must be submitted at least 60 days before commencing the discharge.

Where flows are from conveyances that are not impacted by operational activities, or do not come in contact with overburden or other mine waste, a discharge permit is not required, though a Stream Alteration Permit (State) and/or Department of the Army (“404 dredge and fill”) permit may be if the conveyance modifies or diverts a natural watercourse. To minimize the volume of stormwater runoff that is subject to applicable discharge permits, the water management scheme developed for the Project endeavours to collect and convey clean water around the mining operation and discharge downstream, wherever feasible and practicable.

20.3.3 U.S. Army Corps of Engineers Section 404 Dredge and Fill Permit

A Department of the Army (“Section 404” or “Dredge and Fill”) Permit is required under Section 404 of the Clean Water Act for the discharge of dredged or fill material placed into waters of the United States. A 2009 U.S. Supreme Court decision found mine tailings to be "fill", and can, therefore, be placed into waters of the United States with an approved Section 404 USACE Dredge and Fill Permit. Dredged or fill material includes tailings and waste/development rock. Other activities, in addition to the tailings and development rock storage that may require a 404 Permit are:

- road construction;
- bridges;

- construction of dams for water storage;
- stream diversions;
- other infrastructure (Power Transmission Line, Worker Housing Facility); and
- certain reclamation activities.

As a cooperating agency, the USACE works with the USFS to establish the range of reasonable alternatives in the Draft EIS. The next step is for the USACE to evaluate the practicability of alternatives to determine whether a practicable alternative to the proposed action exists that "would have less adverse impact on the aquatic ecosystem, so long as the alternative does not have other significant adverse environmental consequences (40 CFR 230.10[a]) also known as the least environmentally-damaging practicable alternative, or LEDPA. For the USACE to use the EIS as a supporting evaluation for its permit decision, there must be an alternative that is the LEDPA in accordance with the USACE Guidelines at 40 CFR 230.10(a) as it pertains to Section 404 of the CWA.

20.3.4 ESA Consultation

The purpose of the Endangered Species Act (**ESA**) is to protect and recover imperiled species and the ecosystems upon which they depend. Under ESA Section 7, Federal agencies must consult with the U.S. Fish and Wildlife Service (**USFWS**), or the National Oceanic and Atmospheric Administration (**NOAA**) Fisheries (together, Services), depending on the species, when any action the agency carries out, funds, or authorizes (such as through a permit) may affect a listed endangered or threatened species. The ESA prohibits the "take" (harm, harass, kill) of fish and wildlife species classified as endangered or threatened, and prohibits the destruction or adverse modification of their designated critical habitat, unless otherwise authorized. Federal agencies are required to "conserve endangered or threatened species, and to ensure that their actions are not likely to jeopardize the continued existence of any of these species or adversely modify their designated habitat" (ESA, 16 U.S.C. Section 1538(a). Some adverse effect is allowable, with the issuance of an incidental take permit made pursuant to a Biological Opinion (**BO**) by the USFWS of NOAA. The BO must first determine that the "federal action" (issuance of a federal permit in this case) would not jeopardize the continued existence of the species.

The following listed species are, or may be, in the vicinity of the Project site (orcas excepted), and consultation under ESA Section 7 is required on any Federal action that may affect these species or their designated critical habitats (*):

- Snake River spring/summer-run Chinook salmon (threatened)*;
- Snake River steelhead (threatened)*;
- Bull trout (threatened)*;
- Southern resident killer whale (endangered)*;
- Canada lynx (threatened*); and
- Northern Idaho ground squirrel (threatened).

ESA Section 7 consultation will be with NOAA Fisheries for Chinook salmon, steelhead trout, and Southern resident killer whale, and with the USFWS for the remaining species. The Section 7 informal consultation process has been ongoing, concurrent with the development of the EIS, and a draft biological assessment (**BA**) is in preparation. Once a BA that meets requirements of the Forest Service and the Services in completed, Section 7 formal consultation follows, and will culminate in two BOs issued individually by the Services.

A BA is a precursor to the Services' BOs, and draft BAs are often prepared by the third-party contractor preparing the EIS, the proponent as a designated non-federal representative (**NFR**), or by the lead federal agency. The Services' ESA Consultation Handbook promotes project applicants applying for the status, stating *"There is a clear need for* *early, regular and fully informed coordination among federal agencies and applicants, in order to as completely as possible inform the consultation, resolve conflicts and design the project to minimize adverse effects.*" There are similar statements in the handbook recommending the involvement of other agencies and tribes. Midas Gold was granted NFR status for the project and its first-party contractor is preparing the draft BA in collaboration with five federal agencies, three state agencies and three Tribes. The USFS has the authority to accept, modify or reject any of the content generated from this process. Ultimately, the USFS will prepare the final biological assessment prior to submitting it to NOAA and USFWS for review and acceptance. The USFWS and NOAA will use the BA to prepare the BOs which involves:

- A summary of the information upon which the USFWS' or NOAA's opinion is based;
- A detailed discussion of the effects of the actions on listed species and their critical habitat; and
- The USFWS' or NOAA's opinion as to whether the agency action would jeopardize "the continued existence of the species, or adversely modify their critical habitat"; and,
- Issuance of an Incidental Take Permit, as appropriate.

The formal BOs must generally be issued within 135 days from the date that the formal consultation is initiated (i.e. 45 days after the conclusion of the 90-day formal consultation period). The BA will be finalized and submitted to the Services after a Preferred Alternative is identified as the Federal Action which Midas Gold anticipates will be declared in the FEIS. The BO may require additional mitigation measures or design features to protect ESA-listed species or their habitat (i.e. reasonable and prudent measures), beyond those already included in Project operating plans.

Effects on other sensitive species, such as westslope cutthroat trout, North American wolverine, and bentflower milkvetch, are considered in NEPA but those species do not require consultation under ESA Section 7. However, one additional species – the whitebark pine (currently listed as proposed threatened) – occurs in the project area and may be affected. Whitebark pine will follow a different path during ESA consultation – referred to as Conferencing. Under current law, an agency must "Conference" with the USFWS or NOAA Fisheries on any agency action that is likely to jeopardize the continued existence of any species proposed to be listed or to destroy or adversely modify critical habitat proposed to be designated for such species (ESA §7(a)(4), 16 U.S.C. §1536(a)(4)). Because this species could warrant future protection under the ESA during the federal action timeframe, the USFS, USFWS, and Midas Gold as the NFR have agreed to assess the potential impacts on whitebark pine in this BA following the Conferencing process (USFWS and NMFS 1998).

Formal conferences follow the same procedures as formal consultation. The opinion issued at the end of a formal conference is called a conference opinion. It follows the contents and format of a biological opinion. However, the incidental take statement provided with a conference opinion does not take effect until the Services adopt the conference opinion as a biological opinion on the proposed action - after the species is listed. The conference process is beneficial to the species by providing the opportunity to actively manage the species prior to listing and is beneficial to applicants in that they would not have to re-initiate ESA Section 7 consultation if the species is listed.

20.3.5 Other Federal Programs

There is no comprehensive federal groundwater quality statute, in contrast to surface water and the Clean Water Act. Ground water protection is found in several programs which include: Safe Drinking Water Act, sections of CERCLA, and RCRA. The Safe Drinking Water Act was implemented by the State of Idaho to enforce drinking water regulations for municipalities, public water systems, and related facilities. Based on the anticipated number of personnel working on site and lodged at the worker housing facility, this operation would be classified as a public water system.

The federal Clean Air Act regulates air quality, and the Project would be subject to National Ambient Air Quality Standards; definitive air quality criteria would apply. The operation would be required to meet Prevention of Significant

Deterioration requirements, visibility regulations, and National Emission Standards for Hazardous Air Pollutants. This would involve pre-construction and operating permits issued and managed by the State of Idaho.

20.3.6 Major State Authorizations, Licenses, and Permits

The federal and state application processes would be integrated and processed concurrent with the EIS. The key authorizations, licenses, and permits required by the State of Idaho are as follows:

- IPDES (formerly **NPDES**) permit is discussed above. The State of Idaho gained primacy on this program from EPA in 2018, and Midas Gold has made application to IDEQ for this and related discharge permits. Effluent water quality standards set in the IPDES permit would influence water treatment designs.
- Air Quality Application for Permit to Construct and Operate – This permit is required by IDEQ prior to construction. The IDEQ Air Permit to Construction (**PTC**) assesses the air pollutant emissions from stationary sources, determines the allowable impacts to air quality and prescribes measures and controls to reduce and/or mitigate impacts.
- Cyanidation Permit – This permit is required by IDEQ and is applicable for cyanidation facilities, defined as; "That portion of a new ore processing facility, or a material modification or a material expansion of that portion of an existing ore processing facility, that utilizes cyanidation and is intended to contain, treat, or dispose of cyanide containing materials including spent ore, tailings and process water". Midas Gold intends to produce gold doré onsite and uses cyanide in its production. The regulations apply to both operations and closure and reclamation of any cyanide facility, which at the SGP includes the TSF and elements of the processing plant and associated pipelines.
- Ground Water Rule – This rule establishes minimum requirements for ground water protection through standards and a set of aquifer protection categories. Midas Gold has requested the establishment of points of compliance outside and downgradient from the mine area(s). Midas Gold is working with IDEQ to establish reasonable upper-tolerance limits for all compliance wells. These upper-tolerance limits would take into account the high baseline (due to off-site legacy mining) and naturally occurring background levels for several parameters.
- Total Maximum Daily Loads (**TMDL**) – In Idaho, TMDLs are generally assessed on a sub-basin level, which means water bodies and pollutants within a hydrologic sub-basin are generally addressed within a sub-basin report. An earlier TMDL for the main-stem South Fork Salmon River was approved by EPA in 1991. That TMDL set surrogate sediment targets for percent fines and cobble embeddedness. The Salmon River, South Fork Sub-basin report was updated in 2012 with an EPA approved addendum in February 2012 that proposed to remove the EFSFSR from the 303(d) list for sediments and metals. No TMDL has been established for the EFSFSR, and none is presently in progress.
- Water Rights – As described in Section 5 of this technical report Midas Gold currently holds four permanent water rights associated with the mining activity area. Additional water rights will need to be secured through direct permit application and subsequent approval of such rights from the Idaho Department of Water Resources (**IDWR**) to have sufficient water rights to support Project development. Preparation of an application for these water rights is in progress at the time of this writing. New water right appropriations from the Main Salmon River and tributaries are subject to Federal Wild and Scenic River water rights and State minimum streamflow rights on the Main Salmon River, South Fork Salmon River and East Fork South Fork Salmon River. The subordinations in the Federal and State water rights are sufficient to allow diversions under new water right permits proposed for industrial use. Diversions to storage, however, are not subordinated. To allow diversion to storage at the SGP under new appropriations, mitigation will need to be provided to offset the rate of flow diverted to storage, with mitigation water provided in a timely manner. Midas proposes to secure natural flow surface water irrigation rights for mitigation purposes. Identifying and acquiring appropriate irrigation rights is in progress by Midas Gold.

- Stream Channel Alteration Permit – This permit is required by the IDWR for a modification, alteration, or relocation of any stream channel within or below the mean high-water mark. The FS contemplates relocating portions of Meadow Creek, EFSFSR, and their tributaries, both initially temporarily and later permanently, as part of the overall mine plan. This permit would be obtained in conjunction with any USACE 404 permit obtained for the same purpose.
- Dam Safety – The IDWR must first approve construction of dams greater than 10 ft high impounding a reservoir exceeding 50 acre-feet in volume. The Application to Construct a Dam includes design plans and specifications for construction of the dam. Mine tailings impoundments greater than or equal to 30 ft high are regulated by IDWR in the same manner. Design and construction requirements for mine tailings impoundment structures are described in IDAPA 37.03.05; water dams are described in IDAPA 37.05.06. IDWR has indicated that the size and anticipated water pool volume for the SGP TSF will require application under the more stringent water dam criteria, and Midas Gold has prepared designs and application materials accordingly, with application submittal anticipated in 2021. Three of the proposed lined contact water storage ponds would also be jurisdictional, and applications for those ponds will be submitted to IDWR in 2021 or in advance of pond construction for ponds built later in mine life.
- Water and Wastewater Systems – The drinking water system(s) design for the contemplated work camp (construction and operations) must be approved prior to use. This would assure compliance with the Safe Drinking Water Act. IDEQ would also require approval of plans and specifications for any new sewage treatment and disposal for the work camp.
- Fuel Storage Facilities – Any proposed fuel storage must also comply with IDEQ design and operating standards, as well as Idaho State Fire Marshall and Valley County requirements. Spill reporting requirements for federal and state agencies are necessary components of spill prevention containment and countermeasures (SPCC) plans prepared under the authority of EPA.
- Reclamation Plan – All surface mines must submit and obtain approval of a comprehensive reclamation plan (Title 47) for mining activities on patented land as administered by the Idaho Department of Lands (**IDL**). This includes detailed operating plans showing pits, mineral stockpiles, overburden piles, tailings ponds, haul roads, and all related facilities. The Reclamation Plan must also address appropriate BMPs and provide for financial assurance in the amount necessary to reclaim those mining activities. The plan must be approved prior to any surface disturbance. A large portion of the contemplated Yellow Pine, West End, and Hangar Flats pits, TSF buttress, processing plant, truck shop, and other associated facilities are located on patented land. The Reclamation and Closure Plan (RCP; Tetra Tech 2019a) is under review by USFS, IDL and other agencies, and is intended to satisfy each agency's requirements for facilities under their jurisdiction. An updated RCP is in preparation, reflecting changes to the project layout contained in this FS, but broadly retaining the reclamation and restoration approach.
- State Historic Preservation Office – Approval of a historic/cultural resources assessment by the State Historic Preservation Office would be required. The Project is located within the Stibnite National Historic District; however, no designated historical buildings are present.
- Others – State requirements would also involve compliance with the Idaho Solid Waste Management Regulations and Standards, transportation safety requirements enforced by the Idaho Public Utilities Commission, and others.

20.3.7 Local and County Requirements

There are several other permits and approvals that would apply to the Project including:

- Conformance with the Valley County Comprehensive Plan;
- Issuance of building permits and conditional use permits by Valley County; and

- Sewer and water systems approval by Central District Health Department, and various other authorizations.

A key annual authorization by the Valley County Road Department is the Valley County Road Use Permit for summer and winter road maintenance. This permit addresses standard operating procedures for the County maintained road route to be used, snow removal, dust suppression, and seasonal load limits.

As the Project facilities lie outside incorporated towns, except for portions of the power line upgrades which are on already-existing utility rights-of-way, there are no applicable local approvals below the County level.

20.3.8 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process (**IJRP**). The IJRP involves an interagency Memorandum of Understanding (**MOU**) between involved state and federal agencies. Further, the IJRP addresses a process to achieve pre-analysis coordination in approving / administering exploration permits, interagency agreement on plan completeness, alternatives considered, draft and final permits, bonding during mine plan analysis, and interagency coordination related to compliance, permit changes and reclamation/closure for major mining projects. In Idaho, the Joint Review Process was established to be the basis for interagency agreement (state, federal, and local) on all permit review requirements. The focus of the IJRP is concurrent analysis timelines; this would include, for example, in the case of Stibnite Gold Project the NEPA process, NPDES permit, USACE 404 permit, State 401 Certification (i.e., water quality certification) of these latter two key permits, the State Cyanidation Permit, and the ESA Consultation. The IJRP is intended to play a key role in achieving two primary permitting goals: (1) increased communication and cooperation between the various involved governmental agencies, and (2) reduced conflict, delay, and costs in the permitting process.

The USFS, USACE, USEPA, IDL, IDEQ, the Idaho Governor's Office of Energy and Mineral Resources and Valley County signed a MOU for the SGP IJRP in September 2017. The MOU gives the agencies the framework to evaluate the PRO as they work together to prepare a single, joint EIS for the SGP under NEPA. The single EIS will be a USFS document, however all signatory agencies will collaborate in the preparation of the EIS, provide adequate resources to ensure satisfactory and timely performance, follow a mutually agreed updated schedule, and ensure that the public process meets the requirements of all cooperating agencies and NEPA.

20.4 SOCIAL AND COMMUNITY IMPACT

Midas Gold has strived to develop a project that respects and responds to the needs of all project stakeholders including local communities, tribal governments, and regional interests. This has been achieved through an iterative process of community engagement involving communication, listening and responding to stakeholders through all aspects and phases of Project planning and design. These activities include estimation of project economic impacts and communication of those impacts to the potentially affected communities, helping local communities plan for potential expansion of public services and infrastructure, developing community agreements to ensure long-term financial benefits beyond the Project lifespan, engagement with local tribal governments, and sponsorship and participation in community fundraisers and educational events. The public scoping and DEIS public comment phases of the NEPA process have also provided important feedback from the communities that will be affected by the Project. It is notable that significant comment-driven project changes, including modification of proposed public access through the project site, backfilling of Hangar Flats pit, and additional fisheries and water quality mitigation measures, were incorporated into Midas Gold's modifications of the Proposed Action, and either previously incorporated as alternatives in the DEIS or are proposed herein to further reduce Project environmental impacts, for adoption in the FEIS.

20.4.1 Economic Effects

Economic impacts of the Project include creation of direct, indirect and induced jobs and additional tax revenues for local communities, the state of Idaho and the nation. An economic model known as IMpact analysis for PLANning (**IMPLAN**) was constructed to estimate impacts within Valley and Adams Counties (regional impacts), the state of Idaho and the U.S. (Highland Economics, 2018). The IMPLAN model is based on estimates of expenditures related to labor, materials and services and allocation of expenditures to various geographical and retail sectors for different periods of the project. The IMPLAN model was reviewed and approved by the USFS for suitability in the NEPA process and supersedes a previous economic study reported in the PFS (M3, 2014).

The Project would directly employ approximately 594 people during construction, approximately 583 during operations, and 160 and 44 during reclamation and post-closure, respectively. For every direct hire, an additional 2 to 3 indirect or induced jobs would be created locally and statewide. Local jobs are anticipated to represent approximately 5% of the total local workforce with typical annual wages of around $70,000, well exceeding average local annual wages of approximately $35,000 per year.

In addition to job creation and support, the Project is estimated to create substantial tax revenues from business, property, and individual taxes on Midas Gold, its employees, suppliers and contractors and their employees, and from induced economic activity. Project expenditures and taxes paid by Midas Gold are included in the Project financial analysis (Section 22).

20.4.2 Community Agreements

In December 2018, Midas Gold entered into a Community Agreement with villages, cities and counties in the vicinity of the Project. This Agreement created a collaborative environment for engagement with these communities and provides a venue in which to identify and address opportunities and concerns associated with company and Project operations. To facilitate these interactions, the Community Agreement established the Stibnite Advisory Council, a panel composed of local residents appointed by each signatory community, project stakeholders and Midas Gold leadership. The Stibnite Advisory Council provides a forum for communication and dissemination of information between the communities and Midas Gold on such topics as safety and environment, employment and workforce training, business opportunities, housing and infrastructure and community and family support and sustainability.

The Community Agreement also established the Stibnite Foundation, a non-profit organization which identifies, evaluates, and funds projects to benefit the communities in the West Central Mountains. Decisions regarding projects to be supported are made by a Stibnite Foundation board, which comprise one representative from each community that has signed the Community Agreement. Long term financial stability of the Stibnite Foundation is ensured through creation of an endowment funded by Midas Gold through cash and equity grants at periodic intervals in conjunction with Project milestones including receipt of operating permits, commencement of construction, commencement of commercial production, CAPEX payback, and completion of reclamation.

To date, eight communities in the West Central Mountains have signed on to the Community Agreement including Adams County, Cascade, Council, Donnelly, Idaho County, New Meadows, Riggins and Yellow Pine. Valley County has recused itself from participation due to its potential conflict of interest as an approval authority for the Burntlog Route and sanitary waste facilities. The city of McCall declined participation in the Community Agreement until after the Draft EIS on the Project was issued.

20.4.3 Community Engagement

Midas Gold has undertaken a number of initiatives to act as a contributing member of the local community while providing transparency and accountability. Midas Gold Idaho, Inc. (MGII) was established as a local operating subsidiary to ensure the Project continues to meet the needs of the community as it is advanced. MGII board of directors is composed largely of independent local community leaders, former county commissioners, and a former mayor. Midas Gold participates in community fundraising, conducts educational outreach and Midas Gold employees are active participants with local boards and non-profit foundations.

To ensure that effected communities are prepared for future development of the Project, Midas Gold has participated in strategic planning with local public service and infrastructure stakeholders to identify and plan for potential issues. One of the most important aspects of the Project, the primary mine access road (Burntlog Route), was conceived in a community meeting held in Yellow Pine. Midas Gold has worked with Valley County landowners along the electrical transmission line right-of-way to inform them of improvements and negotiate access for collection of baseline data. Midas Gold has discussed potential improvements to key transportation corridors and intersections with the cities of McCall and Cascade, Valley County Road Department and Idaho Transportation Department. Midas Gold has worked with local school districts, fire departments and emergency response providers plan for future stresses on the community associated with the influx of workers and indirect job creation associated with the Project. Midas Gold has also worked with local outdoor recreation groups on issues including a snowmobile trail adjacent to the Project access road and a new public road through the mine site to access the Thunder Mountain recreation area.

20.4.4 Tribal Engagement

Midas Gold respects the sovereign treaty rights of Native American tribes and has engaged them in good faith through all phases of Project exploration, development and planning. Through early engagement with the Nez Perce Tribe (**NPT**) commencing in 2012, Midas Gold has undertaken measures to mitigate potential impacts of its exploration activities identified by the NPT and has allowed the NPT full access to the Site and shared baseline environmental data. More recently, Midas Gold has been engaged with the Shoshone-Bannock Tribes and has been undertaking efforts to educate Tribal representatives on its proposed plans to improve water quality, address legacy issues caused by prior mining companies and to collaborate on the re-establishment and enhancement of anadromous fisheries. Also, Midas Gold has funded and continues to provide funding for consultation between the Shoshone-Paiute tribe environmental group (Wings and Roots) and the Payette National Forest.

Despite best intentions to collaborate with the NPT on efforts to jointly develop measures to address legacy environmental issues at site for the last several years, on August 9, 2019, the NPT filed suit against Midas Gold in federal court alleging unpermitted water pollution discharges under the Clean Water Act in specified areas of the Project site controlled or owned by Midas Gold and the USFS and previously disturbed by prior operators and government agencies. In August 2020, Midas Gold brought litigation to include the USFS to the case in order to account for the claimed water pollution alleged to be occurring on Federal lands. As of December 2020, each of the lawsuits are ongoing.

20.5 CONSERVATION, RESTORATION, AND MITIGATION

Early restoration and mitigation are key aspects of the Stibnite Gold Project. In addition to the cleanup and restoration of legacy mining-related disturbance and reestablishment of upstream fish passage, Midas Gold plans to minimize, to the extent practicable, the Project's footprint and related impacts by using existing roads, locating facilities on previously disturbed ground and avoiding riparian areas. In combination with restoration of both project and legacy impacts, the Project seeks to provide a net environmental benefit, and leave the site restored with self-sustaining aquatic and terrestrial ecosystems.

This cleanup activity will start in advance of new mining operations, and will continue throughout the construction, operation and closure stages of the Project. Private investors, not the American taxpayer, will pay for the site cleanup, as the restoration work is a fundamental aspect of the Stibnite Gold Project as proposed.

20.5.1 Net Benefit Goal

Midas Gold believes strongly in environmental protection and has established a "net benefit" goal for the Stibnite Gold Project. In establishing the goal of net benefit to the environment, and as central principles to the Project development and operation, early in the design process Midas Gold focused on these key conservation, restoration, and mitigation principles:

1. Midas Gold would conduct mining, processing, and reclamation activities in an environmentally responsible manner.
2. Project infrastructure would be located on previously disturbed areas and sites wherever practicable.
3. Midas Gold would design, construct, operate, and close facilities to minimize impacts to aquatic and terrestrial wildlife, improve habitat through various projects across the Project site, protect anadromous and local aquatic populations, and remove impediments to fish passage.
4. Midas Gold would protect and improve local surface water and groundwater quality by removal and reuse of legacy mining materials, by sediment control and reforestation, and by properly managing water during project construction, operations, and closure.
5. Midas Gold would enhance, construct, or preserve ecologically diverse stream channels and wetlands to replace those affected by new mine development – ultimately providing stream and wetland functional value greater than what was replaced.

In achieving this net benefit goal, Midas Gold will provide Project restoration and mitigation projects that are both durable and additive; that is to say the environmental outcomes will be above and beyond that which would have occurred in the absence of the Project.

Designing the site restoration for a net benefit was guided by a similar hierarchy of priorities as that applied in wetland mitigation under the Clean Water Act:

- Avoidance: avoid an activity or disturbance to the degree practicable.
- Minimization: where a disturbance or activity cannot be avoided, minimize disturbance (e.g. utilizing previously disturbed ground to the degree practicable).
- Mitigation: where unavoidable impacts occur, mitigate for them in the interim or at conclusion (e.g. wetlands/stream restoration).

The measured identified below include avoidance, minimization, and compensatory mitigation under the Clean Water Act, environmental protection measures required under other regulations, and elective restoration projects Midas Gold has identified as beneficial. Taken together, they are intended to restore the site and produce a net environmental benefit, at no cost to taxpayers.

20.5.2 Avoidance and Minimization

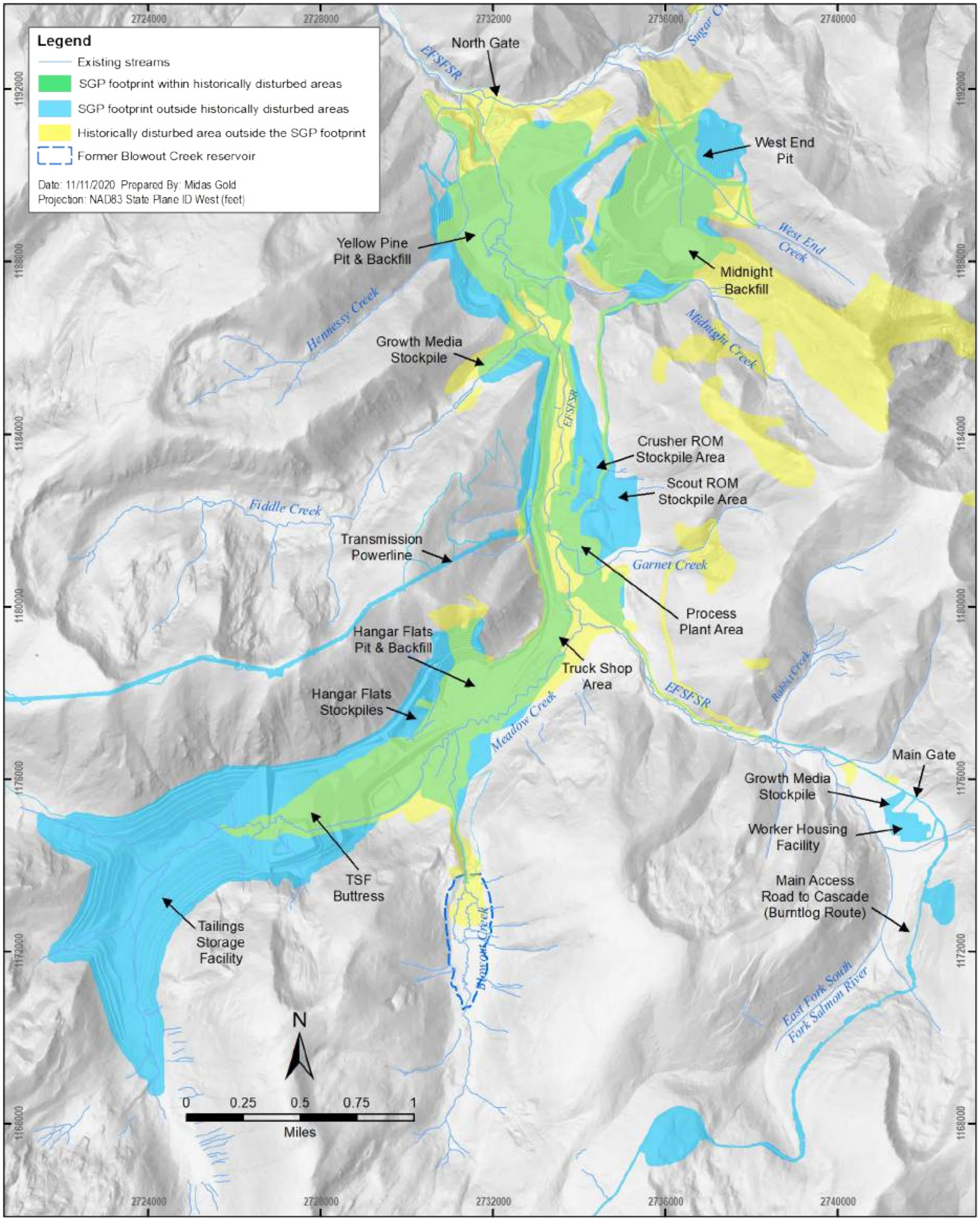
Midas Gold sought to conserve existing resources and avoid and minimize environmental impacts in selection of project facility locations, operating plans, and facility design features. Avoidance and minimization measures reduced project footprint, impacts to aquatic habitat, and the potential for water quality impacts.

20.5.2.1 Facility Siting

Careful thought and planning have gone into the Stibnite Gold Project design with specific effort made toward avoiding and minimizing incremental disturbance by locating facilities and infrastructure on previously disturbed and impacted areas, and improving conditions at the site, as shown on Figure 20-3. Key examples of this planning effort include:

- The TSF buttress is located at the SODA site, which is also the location of the historical tailings and spent heap leach ore storage facility, and has been sited to provide a substantial buttress to the Project's TSF;
- The process plant area encompasses portions of the former Stibnite town site, the current Stibnite camp area, and former contractor shop area;
- The Stibnite Gold Project truck shop and fuel storage area are located on the plant site area from previous heap leach operations;
- The Hangar Flats, West End and Yellow Pine open pits largely lie within areas already extensively disturbed by historical mining operations;
- The EFSFSR diversion approach is similar to that undertaken in prior operations and is situated within the currently disrupted portions of the river channel;
- The Burntlog Route would primarily follow an existing forestry road corridor mostly outside of valleys, avoiding having long sections of road directly adjacent to fish-bearing rivers (as is the case for the current access routes), thereby minimizing the risk of spills or sediment entering waterways;
- The power line would follow the existing and historically used power line corridor and right-of-way, with short exceptions to avoid wetlands and recently developed communities, and near and within the Project site to accommodate new site facilities (TSF and process plant location); and
- Several existing haul road corridors would be utilized to minimize new disturbance.

Figure 20-3 Project Facilities Locations Relative to Historical Disturbance



20.5.2.2 Responsible Operations

Midas Gold developed and currently utilizes several measures designed to minimize environmental impacts due to current activities at the Project site including:

- A Storm Water Pollution Prevention Plan (**SWPPP**) implemented as part of a Multi Sector General Permit (MSGP) to inhibit sediment or pollution from entering onsite streams.
- A Spill Prevention, Control and Countermeasures Plan (**SPCC**) that includes a site-specific spill prevention plan, fuel haul guidelines, fuel unloading procedures, inspections, secondary containment on all fuel storage tanks onsite, spill response kits staged along fuel haul routes, and staff training.
- An onsite Recycling SOP to reduce recyclable waste delivery to landfills.
- Midas Gold currently reduces fossil fuel energy consumption at the site by the targeted application of solar power. In operations, additional reduction would be accomplished through the use of line power over diesel generation for processing and mining, supplemented by solar, thereby reducing human emissions of greenhouse gases.
- Comprehensive surface water and groundwater monitoring programs to assess the effective implementation of BMPs.
- An annual Environmental Training Program for onsite staff and consultants, covering SWPPP, SPCC, Waste and Recycling management, Midas Gold's wastewater reuse plant, noxious weed overview, Threatened, Endangered, Sensitive, Candidate, and Proposed (**TESCP**) plants and wildlife overview, and operational requirements.
- An Operating Permit Compliance Training class for site management and supervisors to specifically cover operating permit constraints and limits to promote accountability with all levels of Project management.
- Additional SOPs and BMPs for Fuel Haulage, Drilling, Ground Water Protection, Drill Pad siting and helicopter supported drilling, Blasting, Water Diversion, Fish Protection and Salvage, Hazardous Material Handling, and Reclamation are just some of the various protection measures.

Going forward, Midas Gold would continue to build on their strong record by continuing to proactively evaluate BMPs and SOPs effectiveness, and adapt and improve them as appropriate, including a post-closure component. The programs above will be enhanced and continued, and additional ones added. Key operational measures to avoid and minimize impacts include water diversions to maintain water quality (keeping clean water clean), water reuse and treatment for Project-impacted water, diversion and tunnel fishway operations to minimize take of ESA-listed fish, and a fugitive dust control plan to monitor and mitigate dust generated by vehicle traffic on haul roads and access roads. Each of these are detailed in other sections of this report or in publicly available environmental permitting documents.

20.5.2.3 Facility and Site Design Features

Environmental modeling revealed additional potential impacts from facility operation, which Midas Gold will prevent with additional design features or facility configurations. Examples include:

- Installing a low-permeability cap on the TSF buttress to prevent water quality impacts;
- Low-flow pipes in perennial stream diversions, riparian plantings along restored and enhanced stream sections, and establishment of a lake along the restored EFSFSR at Yellow Pine pit; to prevent stream temperature increases;
- Diversion of Meadow Creek around Hangar Flats pit in a restored natural channel/floodplain corridor rather than a ditch or pipe to provide long term habitat for ESA listed fish species;

- Fish screens at pump intakes and the fishway control weir to protect ESA listed fish species;
- Utilization of a limestone resource at West End pit, and inclusion of a lime kiln on-site, reducing traffic to site and associated emissions related to transport of lime to site from offsite sources;
- Backfill of Hangar Flats, Midnight (a small satellite pit within West End pit), and Yellow Pine pits to eliminate pit lakes;
- Dark Skies-compliant lighting on facilities to reduce lighting impacts;
- Reuse (principally in ore processing) to reduce water consumption, and water treatment before discharge of excess contact water (including dewatering well water) to improve water quality; and,
- Discharge of treated water into Meadow Creek to augment stream flows to offset losses caused by Hangar Flats pit dewatering, thereby maintaining habitat quality for ESA listed fish species.

20.5.3 Legacy Material Cleanup and Restoration

Midas Gold will remove, reuse, reprocess, or isolate a variety of legacy materials from prior mining operations, in some cases in the normal course of remining a brownfield site, and as additional and elective cleanup measures. In addition to removals that will improve water quality, Midas Gold will repair a number of physical legacies that degrade fish habitat and limit fish migration. Several examples of legacy impact cleanup include:

- Removal of uncontained historical tailings in Meadow Creek valley, reprocessing them to remove metals in sulfides, and re-deposition into a lined TSF;
- Removal and reuse of spent heap leach ore from SODA, placing it underneath the TSF liner and above groundwater to eliminate it is a source of metal leaching;
- Removal and reuse of spent leach ore from Hecla Mining Company (**Hecla**) heap to eliminate it as a source of metal leaching, similar to the spent heap leach ore at SODA;
- Removal of historical development rock dumps from around the Yellow Pine and West End pit areas and relocation to a designed storage facility (pit backfill or TSF buttress) to eliminate uncontrolled water infiltration and potential metal leaching that could affect water quality;
- Removal of historical Hecla, Canadian Superior Mining Ltd. leach pads and residual infrastructure;
- Divert clean water around, and remove potentially contaminated materials from below the historical mill and smelter site to improve water quality;
- Remove (during mining) and plug remaining historical underground mine workings at Yellow Pine and Hangar Flats pits, including the Bailey Tunnel (former EFSFSR diversion) to improve water quality;
- Divert Hennessy Creek and Midnight Creek away from legacy dumps, preventing infiltration of creek water into legacy mined materials from affecting downstream water quality;
- Re-establishing short and long-term passage for Chinook salmon, steelhead, and bull trout through the Yellow Pine pit area on the EFSFSR – first with the tunnel fishway and ultimately with a permanent restoration across the backfilled pit. This would allow for upstream fish passage for the first time since 1938 and provide a head-start on re-establishing the previously abundant salmon runs, prior to permanent reestablishment of the EFSFSR over backfilled pit;
- Enhance (with riparian vegetation, engineered log jams, and boulder placement) the EFSFSR from Meadow Creek to Sugar Creek, and un-diverted portions of lower Meadow Creek, thereby increasing in-stream habitat diversity and pool quality and lowering stream temperatures;

- Restore, including with a connected floodplain, rather than simply divert, the historically straightened segment of Garnet Creek at the process plant site;
- Reforestation of burned areas in and around the Project site; and,
- Stabilize and restore Blowout Creek (and associated wetlands), the site of a 1965 dam failure and ongoing source of fine sediment, which would dramatically reduce the sediment currently available for transport at every major precipitation event and during spring runoff that affects downstream water quality.

Legacy feature removals would begin during initial project construction and continue concurrent to operations and through closure.

20.5.4 Compensatory Mitigation for Wetlands and Streams

As detailed in this Report, aspects of the current design of the Stibnite Gold Project entail the disturbance of property within the Stibnite Mining District and, in the case of the proposed power-line upgrade and Burntlog access route (see Section 18), outside of the District. While Project facilities and infrastructure would be located in areas of previous disturbance wherever practicable, in some cases disturbance of wetlands and streams would be unavoidable. Under current regulations, any person, firm, or agency planning to alter or work in "Waters of the U.S.", including the discharge of dredged or fill material, must first obtain authorization from USACE under Section 404 of the Clean Water Act (CWA; 33 United States Code [U.S.C.] 1344) and, if applicable, Section 10 of the Rivers and Harbors Act of 1899 (33 U.S.C. 403).

Under Section 404 of the Clean Water Act, after efforts at avoidance and minimization (such as Midas Gold's efforts to locate facilities on previously-disturbed uplands) have been exhausted, remaining unavoidable impacts to waters of the U.S. require compensatory mitigation – that is, replacement of their lost function – generally in advance of the disturbance taking place. Several means exist to mitigate disturbed aquatic resources, a common one is the use of a mitigation bank. According to the EPA "a mitigation bank is a wetland, stream, or other aquatic resource area that has been restored, established, enhanced, or (in certain circumstances) preserved for the purpose of providing compensation for unavoidable impacts to aquatic resources permitted under Section 404 or a similar state or local wetland regulation" (EPA, 2014). A second means is construction of replacement wetlands, either on-site or off-site but generally in the same drainage basin.

Owing to the combined effects of the Project sequence and resultant temporal loss of wetlands, limited valley-bottom land available, lack of established mitigation banks in the South Fork Salmon River basin, and the amount of Project wetland disturbance, complete compensatory mitigation via a single means is impractical for the Project. Midas Gold is pursuing a comprehensive approach to wetland and stream compensatory mitigation that entails on-site enhancement and restoration of both streams and wetlands, banking, and off-site projects such as stream habitat enhancements and replacement of culverts that presently impede fish passage. Midas Gold and the USACE are evaluating (scoring) the impacts and mitigation on the basis of function rather than strict acreage, wherein higher-quality habitat yields a higher score and thus either requires proportionally greater acreage of lesser habitat to mitigate, or less acreage of better habitat. Midas Gold's proposed onsite mitigation package will result in a net gain to total stream functional units on site, and a net overall gain to wetlands functional units basin-wide; additional credits from offsite programs and banking are necessary to offset losses incurred early in construction and operations when not enough mine facility acreage has been retired from use to enable sufficient on-site restoration credit. The mitigation plan, scoring system, and resultant gains are described in the Conceptual Mitigation Plan (CMP; Tetra Tech 2019b) for wetlands, which upon finalization and permit approval by USACE will become the Compensatory Mitigation Plan, and Stream Functional Assessment (Rio ASE 2019) for streams. Many of the compensatory mitigation measures are also closure and restoration projects and are summarized in the sections that follow. The potential costs associated with these activities are provided in Section 21 of this report.

20.6 CLOSURE AND RECLAMATION

Midas Gold considers site restoration, closure, and reclamation to be integral and important components of the Project. The overall purpose of the Project's net benefit goal is to reclaim legacy and new activity areas to stable and productive conditions for long-term, post-Project protection of land and water resources. The objective of this restoration work is to reestablish a sustainable fishery with enhanced habitat to support natural populations of salmon, steelhead, and bull trout; improve water quality; establish a productive and sustainable vegetative community; and enhance wildlife habitat, all contributing to a self-sustaining and productive ecosystem.

Closure, reclamation and restoration work at the site would include interim, concurrent, and final closure, reclamation and restoration of the site:

1. Interim reclamation is intended to provide shorter-term stabilization to prevent erosion of disturbed areas and stockpiles that would be removed or more fully and permanently reclaimed later.
2. Concurrent reclamation and restoration are designed to provide permanent, low-maintenance achievement of final reclamation and restoration goals on completed portions of the Project prior to the overall completion of mining activities throughout the mine site.
3. Final closure and reclamation and restoration would involve removing all structures and facilities; reclamation of those areas that have not been concurrently reclaimed such as the TSF and some DRSF and backfill surfaces; recontouring and improving drainages; creation of wetlands; reconstructing various stream channels; decommissioning of the EFSFSR diversion tunnel; growth media placement; planting and revegetation on disturbance areas; and reopening Stibnite Road (FR 50412) through the mine site.

Final reclamation and restoration of certain facilities could continue beyond the five-year closure, reclamation and restoration period – for example, the TSF reclamation will not begin until roughly five years after operations end, and the truck shop and Burntlog Route would be needed until the TSF is fully reclaimed.

Closure, reclamation and restoration activities are intended to achieve post-mining land uses of wildlife and fisheries habitat and dispersed recreation at the mine site. Dispersed recreation uses would be accessible by the reopening of Stibnite Road (FR 50412) (including establishment of a permanent public road through the backfilled Yellow Pine pit) that would facilitate recreational traffic and access to Thunder Mountain.

Some reclamation and restoration also entail mitigation or legacy feature removal, and may take place on private or public land, or on regulated facilities (i.e., TSF). Thus, closure, reclamation and restoration will be governed by standards and/or permit conditions from multiple agencies, including applicable USFS Land Resource Management Plan (LRMP) provisions, Idaho Department of Lands (IDL) regulations and standards, USACE 404 permit conditions, IDWR Dam Safety rules, IDEQ IPDES and Cyanidation permits, and EPA cleanup standards. Closure plans were developed to satisfy the most environmentally stringent of overlapping requirements.

Facility-specific closure, reclamation and restoration are described in greater detail in the following sections, and the overall reclamation plan identifying revegetated areas and restored streams is shown on Figure 20-4. General closure practices (common to all facilities of a given type) include:

- Where practicable, conduct site restoration activities concurrent to and in conjunction with exploration, construction and subsequent mining operations;
- Contouring artificial landforms to blend more naturally into the landscape;
- Cover reclaimed surfaces with soil/rock cover, and either growth medium (uplands) or seed bank (wetlands) material at a thickness appropriate to the location, or talus (coarse) rock on certain slopes appropriate to the steepness, aspect, and adjacent or analogous natural conditions;

- Revegetate reclaimed surfaces with native or adapted species appropriate to the topography, soil, and hydrologic conditions;
- Provide low-permeability liners (beneath) or caps (above) facilities as appropriate to isolate materials with geochemical concerns or prevent unacceptable loss of streamflow;
- Backfill open pits to the extent practicable;
- Protect the public and wildlife with vehicle barriers, exclusion fencing, signs, and closure of underground workings; and,
- Prevent the establishment and spread of noxious weeds.

Because closure, reclamation and restoration practices and technology evolve and improve over time, Midas Gold will take advantage of future opportunities to explore new reclamation techniques and, where appropriate, will implement practicable improved measures through adaptive management.

Figure 20-4 provides a shaded contour rendering of the final site topographic surface that illustrates the Project disturbance footprint, the reclaimed surface areas, the primary stream restoration and enhancement reaches, and the public roads that will remain following closure activities.

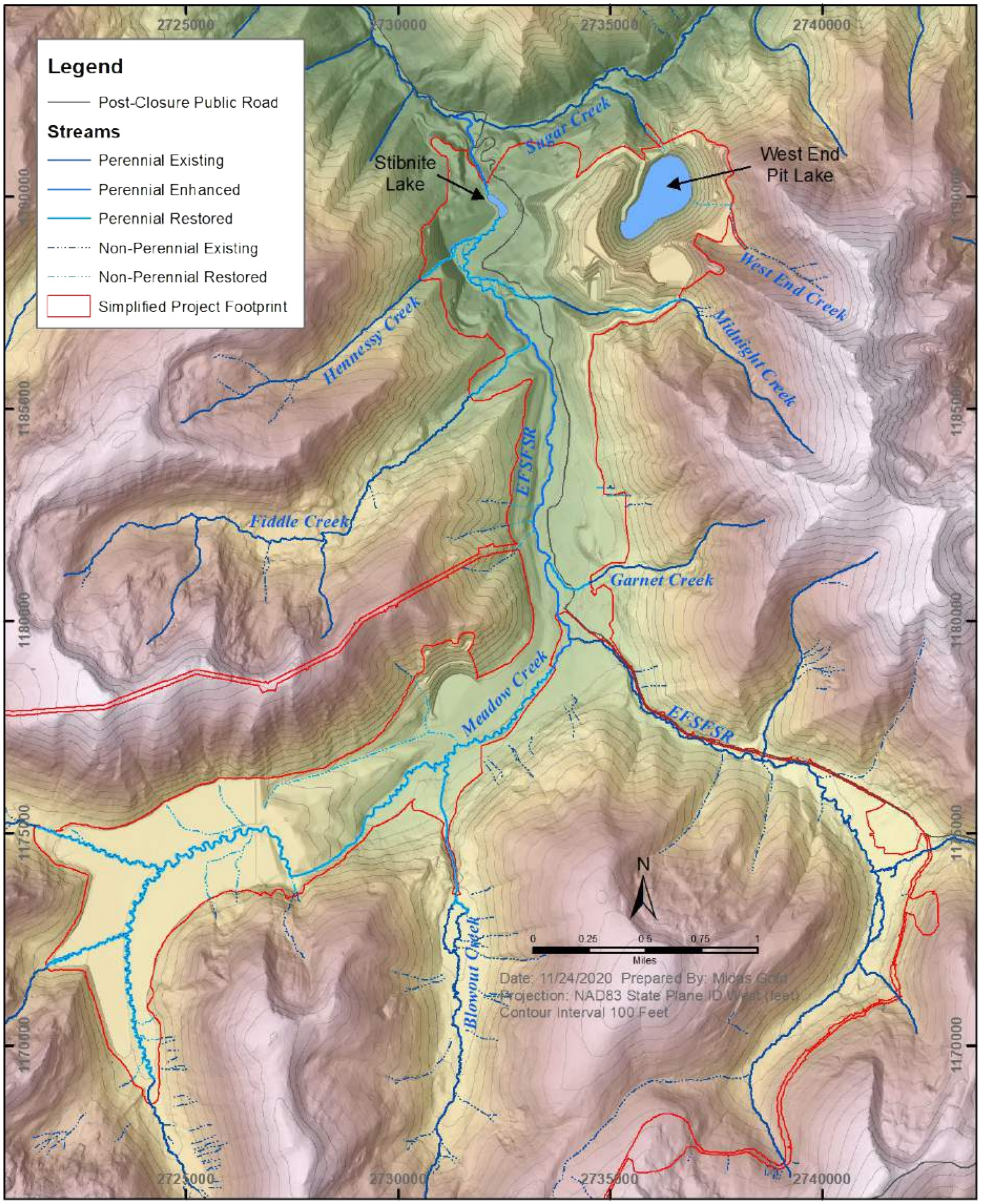
20.6.1 Tailings Storage Facility and Buttress

Midas Gold proposes to complete tailings reclamation and restoration within approximately 9 years after ore processing operations cease. After tailings consolidate sufficiently to use heavy equipment on top of the tailings, starting approximately 5 years after the end of deposition, Midas Gold would begin with placement of soil/rock cover material, then construct wetlands and restore Meadow Creek and its tributaries within appropriately sized lined floodplain corridors, place growth media, and revegetate the area.

Once ore processing operations have ceased, Midas Gold would begin removing the remaining supernatant pond through a combination of spray evaporators (similar to snowmaking misters) operated within the TSF boundary, and active water treatment that meets IPDES discharge limits, followed by discharge to the EFSFSR or Meadow Creek. Removal of the remaining supernatant water from the TSF would allow the surficial layers of the tailings to dry and gain strength, which would allow equipment to operate on the tailings surface for grading and the placement of a soil/rock cover. Cover placement and minor grading of tailings would occur as portions of the TSF allow equipment traffic, working inward from the facility perimeter beginning within 3 to 5 years from the end of deposition. The cover material would be sourced from unconsolidated overburden stored in the upper lifts of the adjacent TSF Buttress.

Midas Gold would restore meandering stream channels (Meadow Creek and tributaries) within a geosynthetic-lined stream and floodplain corridor across the top of the TSF. Pools and riffles would be constructed within the channel. Measures to create aquatic habitat would include side channels, oxbows, boulder clusters, root wads, and large woody debris. This would allow for the post-closure development of riparian habitat, convey water off the facility, and minimize potential interaction of surface water with the underlying tailings. Given the nature of the surface of the TSF, the constructed channel would have a shallow gradient.

Figure 20-4: Overall Site Closure



High-flow events would drive the overall channel and floodplain design, which would necessitate the construction of defined channels ranging from approximately 5 to 15 feet in bankfull width, with average bankfull depth reaching approximately 2 feet. A connected floodplain up to 200 feet wide would convey higher flows during a 100-year flood event.

Consolidation of the tailings would continue after surface reclamation, at gradually declining rates, and this consolidation water would mix with meteoric water on the cover, potentially leading to water quality impacts if discharged to streams. The lined stream corridors provide physical separation of these areas from Meadow Creek and its tributaries. The commingled water from the portions of the facility outside the lined corridors would be collected for treatment, and the TSF perimeter diversions would continue in service to divert hillside runoff away from the cover. Initially, collected flows would be routed first to equalization basins on the TSF surface, and then to a WTP for treatment and discharge. After flows decline to levels appropriate for passive treatment, they would be routed to a passive treatment facility and on to discharge to Meadow Creek below the buttress. Treatment would no longer be required after approximately 40 years; at which time the treatment facility would be decommissioned, and the treatment facility site and water storage basins reclaimed.

Final slopes of the TSF buttress would be variable, to blend with the surrounding terrain to the extent practicable, produce a permanent and stable landform, provide access for future maintenance on the TSF and buttress, and provide for non-erosive drainage across the reclaimed face of the buttress. Upon completion of final grading of the TSF buttress, a low permeability geosynthetic cover would be placed over the facility, which would be designed to limit infiltration through the cover into the underlying development rock. The geosynthetic cover would be overlain by an inert soil/rock layer and growth media and revegetated. Similar to that for the TSF, a lined channel and floodplain corridor would be established for Meadow Creek across the top of the closed buttress, with the stream corridor liner contiguous with the buttress cover. The channel would have a low gradient and wide floodplain across the top of the buttress, then drop more steeply to the valley floor near the south abutment. The steep channel segment would consist of a boulder chute (with underlying liner contiguous with the buttress cover) that would flow through an energy-dissipating basin at the toe of the TSF buttress before being discharged to a restored Meadow Creek on the valley bottom.

20.6.2 Hangar Flats Pit

Hangar Flats pit would be backfilled to the valley bottom elevation or slightly higher during mine operations. The already-established Meadow Creek diversion channel and floodplain corridor would be retained around Hangar Flats pit as the final configuration, and the segment of Meadow Creek between the toe of the TSF Buttress and the entrance to the Hangar Flats pit diversion would be restored along with adjacent riparian wetlands. At closure, growth media and seed bank material would be placed on the backfill surface, and the area revegetated with a combination of upland and wetland vegetation. Wetlands created on the backfill surface would be fed from reestablished intermittent and ephemeral streams that were diverted above the Hangar Flats pit highwall and the TSF Buttress during operations. Meadow Creek downstream of the Hangar Flats pit diversion, to the confluence with the EFSFSR, would be enhanced during mine operations with large woody debris, boulder cluster habitat structures, and riparian plantings.

Saturation of the Hangar Flats backfill and rebound of the alluvial groundwater is predicted to take approximately 2 years (i.e., by the end of mine Year 8) from the end of mining Hangar Flats pit.

20.6.3 Yellow Pine Pit

The Yellow Pine pit would be backfilled with West End pit development rock during operations, reaching the post-mine floodplain level by approximately Year 10, after which the EFSFSR and its nearby tributaries would be restored across the backfill. Portions of the highwalls on the east and west sides of the pit would remain above the backfilled portion of the pit and would not be reclaimed, enabling a wider restored floodplain in the middle of the backfill. The curved alignment of the valley restored in the backfill allows for a longer length and therefore flatter gradient, enabling a longer, flatter, and more sinuous EFSFSR channel to be constructed through the backfilled area than currently exists, maximizing fish habitat and facilitating fish passage. The channel and floodplain corridor atop the Yellow Pine pit backfill would be lined with low permeability geosynthetics. Above the stream corridor liner, a layer of relatively fine material would be placed to protect the liner from puncture, followed by coarse rock armor to prevent exposure via stream scour, followed by floodplain alluvium. Growth media and seedbank material will then be placed, and the area revegetated as appropriate. The lined corridor will be wide enough to accommodate future channel migration and evolution.

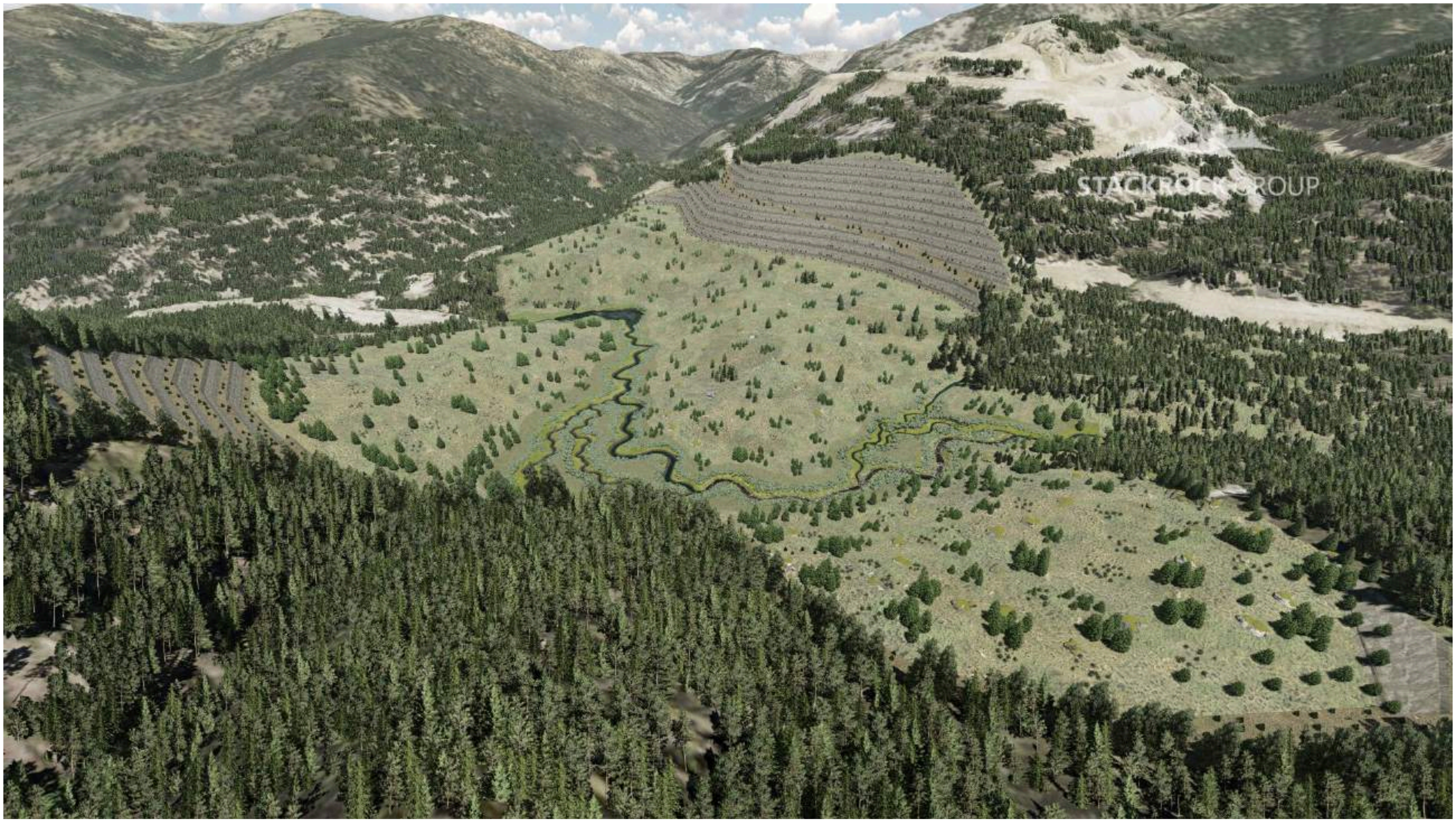
Stibnite Lake, of similar size to the current Yellow Pine pit lake, would be constructed within the lined corridor. The Stibnite Lake feature would reduce summer maximum stream temperatures leaving the site and replace the habitat functions of the current pit lake.

Hennessy Creek would cascade over the west highwall of the Yellow Pine pit to a restored section of low-gradient channel on the western edge of the reconstructed EFSFSR floodplain before joining the restored EFSFSR channel, and Midnight Creek would be restored across the southeastern portion of the EFSFSR floodplain, both forming “wall-based channels” that receive high-flow overflows from the main EFSFSR and sustain cold low flows year-round for juvenile fish rearing.

A road would be established over the backfilled Yellow Pine pit to allow public access through the reclaimed site and connect Stibnite Road (FR 50412) to Thunder Mountain Road (FR 50375, replacing segments of Stibnite Road (FR 50412) removed by mining. After restoration of the EFSFSR and Hennessy Creek across the backfill, closure of the EFSFSR tunnel, and construction of the permanent public access road, the Hennessy Creek diversion would be decommissioned and the area reclaimed, along with the adjacent operations-phase public access road. Similarly, remaining portions of the Midnight Creek diversion would be reclaimed to pre-mining conditions.

Figure 20-5 provides an isometric rendering of the backfilled and restored Yellow Pine pit area following closure activities.

Figure 20-5: Yellow Pine Pit Post Closure



20.6.4 West End Pit

Carbonate-rich development rock from the West End pit would be used to backfill the Yellow Pine pit prior to closure. This acid-neutralizing material would form the base over which the restored EFSFSR channel would be constructed (see Section 20.6.3). The sequence of mining Yellow Pine first and West End last facilitates backfilling the Yellow Pine and Hangar Flats pits, enabling permanent restoration of fish passage and preventing formation of large pit lakes at either Yellow Pine or Hangar Flats – but necessitates that the West End pit remain open. Owing to its location high in the drainage, with minimal upslope hydrologic catchment, the West End pit would gradually form a lake that is not expected to overflow or require water treatment. The pit would be properly signed to communicate the safety risk of the pit lake and pit high walls. Safety berms would also be employed, as appropriate.

West End Creek would be routed into the West End pit in a rock chute on the highwall adjacent to the upper legacy DRSF outslope, below which a pit lake is anticipated to form in the main portion of the West End pit. The up to 400-feet-deep West End pit lake will fill gradually, and lake levels will fluctuate seasonally and with longer-term climate variations; however, the lake is not expected to completely fill with water or spill due to the limited catchment area.

As a contingency to account for model and climate uncertainty, lake levels would be monitored after closure, and a threshold water level would be established, sufficient to contain the predicted runoff volume from a high-snowpack year without discharge. If water levels approach the threshold, either or both surface water diversions and water treatment could be implemented to prevent a discharge. For treatment, a temporary treatment unit would be mobilized to the site to treat and discharge the water until the lake level falls below the threshold level, thus preventing untreated discharge in potential subsequent wet weather years and enabling gradual and predictable water treatment rather than treatment at higher but variable and uncertain peak spring runoff rates.

The Midnight pit, an approximately 6-acre, 100-foot-deep satellite pit in the southern portion of the overall West End pit would be backfilled during operations with approximately 6 million tons of development rock from the West End pit. The backfill would be placed to achieve a mounded final reclamation surface to promote drainage away from the West End pit and prevent formation of a pit lake within Midnight pit. Portions of the backfill would be covered with growth media and revegetated, and the remainder covered with talus like development rock to mimic a natural talus slope.

The floor of the "pitlet", a sidehill pit southwest of the main West End pit, would be graded to drain, covered with growth media, and revegetated.

No backfilling would occur for the main West End pit. At closure, the remaining road into the pit and access to highwalls would be blocked with large boulders and/or earthen berms to deter motorized vehicle passage into the pit.

20.6.5 Plant Site and Related Infrastructure

Unless there is an ongoing beneficial use, the processing plant, maintenance facilities, office, shop buildings, and offsite Burntlog Maintenance Facility will be dismantled and recycled/salvaged to the extent practicable. All structures and facilities not necessary for post-closure water management (e.g., certain roads, culverts, pipelines, and water treatment facilities) or other beneficial use would be removed, and the affected areas reclaimed.

The materials from the dismantling or demolition of structures and facilities would be salvaged or disposed in the onsite private landfill(s) and/or in permitted offsite landfills. All reagents, petroleum products, solvents, and other hazardous or toxic materials would be removed from the site for reuse or would be disposed of according to applicable state and federal regulations. Sewage systems and septic tanks would be decommissioned. Foundations would be broken or fractured as required to prevent excessive water retention and covered in-place with an appropriate depth of soil-like material (approximately 2-ft thick combination of 1.5 feet of backfill and 0.5 feet of growth media) or would be removed and buried a minimum of 2 feet deep in the TSF buttress or pit backfill. Soil beneath fuel storage areas and chemical storage or processing buildings would be tested for contamination and removed and disposed of appropriately as needed. Following removal of facilities, the affected areas would be graded to restore drainage patterns and revegetated with approved seed mix

20.6.6 Burntlog Route

The Burntlog Route was created to avoid or bypass major sections of the Johnson Creek route and its important fishery and would be closed once all reclamation work has been completed, and significant fuel and reagent haulage has ceased. New sections of the Burntlog Route would be decommissioned/obliterated and upgraded sections would be returned to their pre-project width, while retaining the safer upgraded lines and grades. Upon removal, access to the site and the neighboring Thunder Mountain area would be re-established by constructing a public access road through the site connect the existing Stibnite and Thunder Mountain Roads (see Section 20.6.3).

20.6.7 Worker Housing Facility

The worker housing facility would be used during the initial 2 - 3 years of reclamation, restoration and closure activities but these activities would not require the full facility; consequently, a portion of it would be removed during the early years of closure. After the majority of closure activities are complete, the worker housing facility would be dismantled, salvaged and the area reclaimed and revegetated as described in Section 20.6.5.

20.6.8 Haul Roads

Strategic roads would initially be left in place during reclamation, restoration and closure. Other haul roads would be recontoured, ripped, and revegetated to the approximate pre-mining condition. Stream crossings would be restored in kind, and drainage facilities constructed for haul roads would be retained temporarily as necessary for sediment control and then reclaimed.

20.6.9 Powerline Corridor

The section of powerline from the Johnson Creek substation to the site would be retained in service during closure and post-closure water treatment, but substation components downsized to accommodate approximately 1 MW service. After closure activities that have significant power requirements have been completed, the section of the powerline from the Johnson Creek substation to the site will be disassembled, and the associated roads reclaimed to their pre-project state. Drainage stabilization and erosion control features would be installed. The upgraded powerline from Warm Lake to Yellow Pine would be left in place; Idaho Power would continue to maintain that line.

20.6.10 EFSFSR Tunnel and Underground Openings

Midas Gold would decommission and close underground facilities and underground support facilities, including the portals of the EFSFSR Tunnel and underground workings partially mined-out within the open pits. To prevent future access to underground workings, portals will be closed using a concrete block bulkhead, rockfill, or a combination of rockfill and low-permeability foam. The downstream (north) EFSFSR Tunnel portal would be closed with bulkheads inside the portals (where overhead cover was at least 3 times the tunnel height) or backfilled with clean rockfill starting inside the portals and working outward, and up against the portal headwalls. Surface swales would be installed to direct surface water around the backfilled portal, and the exterior backfill and surrounding disturbance would be graded to blend with adjacent topography, covered with growth media, and revegetated. At the EFSFSR Tunnel upstream (south) portal, the control weir would be left in place, and the fishway weir notch raised with concrete, creating an approximately 4-foot-high sill to exclude river water or alluvial groundwater, and low-permeability geofoam or similar would be installed inside the portal after the initial backfill or bulkhead, to prevent water entry. Then, the portal area would be filled, regraded, and revegetated as described for other openings.

20.6.11 Landfills

Onsite landfills will be closed per Idaho requirements for Non-Municipal Waste Landfills. The surface would be covered with development rock, alluvium, or till at least 12 inches thick, graded to promote drainage and prevent pooling of water and to match the surrounding surface topography. Following grading growth media would be placed on the covered landfill and the area would be revegetated. The final overall slope of the Fiddle landfill would be no greater than 3H:1V; landfills within pit backfill would be covered by backfill and reclaimed at the final backfill slope. Landfill access roads would be retained for monitoring and maintenance access until reclamation is complete. When surface reclamation and revegetation monitoring is completed, roads would be obliterated and reclaimed.

20.6.12 Temporary Closure

There are no periods of temporary or seasonal closure currently planned for the SGP. In the event of temporary suspension of activity, Midas Gold would notify the USFS, IDEQ, IDWR, IDL, and Valley County in writing with as much advanced warning as possible of the temporary stop of mining activities. This notification would include reasons for the shutdown and the estimated timeframe for resuming production.

During any temporary shutdown, Midas Gold would continue to implement operational and environmental maintenance and monitoring activities to meet permit stipulations and requirements for environmental protection. If ore processing is not occurring, and depending on the time of year, dewatering may be halted, and excess contact water collected from the various facilities may be allowed to remain in pits, stored in ponds, or transferred to the pits or TSF for temporary storage prior to water treatment or later reuse. In the case of a longer-term closure, mobilization of additional water treatment capacity may be necessary to allow discharge to the area streams and prevent filling of the TSF. In no case would the TSF design freeboard or reserved flood storage be exceeded. A plan would need to be developed, reviewed and approved by the appropriate regulatory authorities, and implemented at the time of any longer-term temporary closure.

20.6.13 Water Management Considerations

Post-mining water management would be a continuation of the operations-phase approach of separation of clean water from mine-affected water, and management of mine-impacted water to meet applicable water quality standards. This would include the following:

- During TSF early closure (prior to cover placement), diversions (including low-flow pipes) would be maintained to prevent upgradient clean water from running onto the facility and maintain cold stream temperatures. During this time, excess water inventory on the TSF would be reduced with a combination of mechanical evaporators and active treatment for discharge.
- After the cover is placed, tailings consolidation water, and runoff from the TSF cover that commingles with it, would be treated for approximately 40 years, first via active treatment and later by passive measures pending successful pilot studies. Seasonal equalization storage prior to treatment would be provided by shallow basins constructed on top of the TSF on either side of the stream corridor. As treatment flows diminish and portions of the cover are fully vegetated, diversions would be decommissioned, water storage basins covered and reclaimed, and offsite runoff would flow over the cover or into the restored sections of Meadow Creek.
- Early in closure, toe seepage (draindown) and contact surface runoff from the (partially or recently capped) buttress would be collected and treated along with TSF consolidation water, until diminishing to levels that allow passive treatment, evaporation, or re-infiltration.

- Stormwater from reclaimed surfaces that are not yet revegetated would be managed with BMPs and sedimentation ponds.

Costs for post-closure water management are included in Section 21.

20.7 ENVIRONMENTAL MONITORING AND REPORTING

Monitoring will measure the effects of Project activities and the success and efficiency of environmental management and mitigation measures. Monitoring will provide valuable information to Midas Gold, governmental regulatory agencies and other stakeholders regarding Project environmental performance. Information gained from monitoring will be used as the basis for adaptive management in designing additional or altering existing mitigation measures and operational activities, if necessary.

The general objectives for site environmental monitoring are:

- Confirm compliance with the approved ROD, as well as with other federal and state laws, regulations, and permit conditions;
- Provide data and information to develop, calibrate, and validate models used to support decisions (i.e., water balance, water quality predictions, etc.);
- Provide data and information that can provide for early detection of potential problems;
- Provide data and information to formulate and direct corrective actions, should they become necessary;
- Provide data to assist Midas Gold in avoiding and then minimizing harmful effects to water, air, wildlife, and other natural resources, consistent with the goal of avoidance and minimization, and to support adaptive management of site operations;
- Establish response protocols to prevent or mitigate environmental problems; and,
- Provide related monitoring information to local communities, Tribes, NGOs, agencies and other interested parties.

Certain environmental monitoring measures will be required under permits and other approvals from the USFS, USACE, EPA, IDEQ, IDL, Valley County, and other appropriate agencies. The Project will operate under federal, state and local permit approvals that will mandate practices and procedures to mitigate environmental impacts and to reclaim disturbed areas. These agencies will conduct routine inspections to ensure compliance with applicable monitoring and reporting regulations.

During construction and mine operations, elements of the baseline monitoring program discussed in Section 20.1.2 would continue, with water quality and fisheries monitoring sites added, subtracted, or relocated in some cases with the expansion of mine features. Additional water quality monitoring would be conducted at IPDES outfalls, and groundwater point-of-compliance wells. Geotechnical monitoring would be conducted for the TSF, buttress, tunnel, and pit highwalls – primarily in support of mine operations but also related to environmental performance. Similarly interrelated with operations, water and process flows would be metered as needed for process control and water management, balance, and treatment. Monitoring costs are factored into operations expenditures and staffing summarized in Section 21.

A summary of the water and restoration-related sampling program follows. More definitive plans are included in the PRO, Reclamation and Closure Plan and Conceptual Mitigation Plan, and costs are included in Chapter 21. Post-closure monitoring would include:

- water quantity measurements at USGS gages onsite;

- water quality monitoring as required by the SWPPP;
- water quality monitoring as required by the IPDES Permit;
- ongoing trend sampling for surface water and ground water;
- wetland/stream restoration monitoring required for CWA section 404 compensatory mitigation sites;
- reclamation/revegetation monitoring require by IDL reclaimed mine features; and
- monitoring associated with the State of Idaho Groundwater Rule and point(s) of compliance.

The primary purpose of this monitoring would be to determine if potential environmental changes would result from the Project. Further, the monitoring program is intended to evaluate the long-term effectiveness of conservation and mitigation measures outlined in the final ROD, USACE 404 permit and other permit approval documents.

Inspections of the TSF and DRSF would occur annually for the initial three years following closure, and after extreme events (100-year, 24-hour storm). After this initial monitoring, the contemplated schedule would be Years 5, 15, and 30. This would involve evaluation of the performance of the TSF for the following: geotechnical observations and recommendations, hydrologic monitoring, and water balance review. The actual routine and emergency monitoring and reporting requirements would be defined in the Cyanidation and Dam Safety permits.

The ongoing post-closure fisheries and aquatic biota (stream habitat) monitoring program would focus on evaluating species diversity and habitat conditions as they relate to the mitigation and conservation plans. The initial pre-Project environmental baseline program conducted during 2011-2014 will be the foundation that future potential impacts and long-term mitigation success are measured. Key components of the monitoring would include in-stream flow needs, adult salmon counts, fry escapement and winter survival, habitat characteristics, and construction monitoring. This program would demonstrate conservation and mitigation program effectiveness. Monitoring would occur in Years 5, 15, 30.

Ground water monitoring would focus on measuring any potential changes in exiting ground water conditions beneath the tailings impoundment system and throughout the upper EFSFSR basin. Sampling stations downstream of the tailings impoundment, as well as downstream of the three mine pits and at the downstream points(s) of compliance, would be indicative of potential ground water impacts associated with the mining operation.

All newly reclaimed and restored areas would be managed consistent with the Project's reclamation, mitigation and conservation principles. The sites would be examined according to the schedule beginning with the concurrent reclamation phase and proceeding through reclamation and post-closure. The success of re-vegetation would be monitored to ensure erosion is minimized and/or mitigated, and that native species re-establishment is occurring. Maintenance would be conducted on the site as necessary to promote species viability and re-colonization. Reclamation guarantees per 36 CFR Section 228A regulations would be provided by Midas Gold via reclamation bonding or other acceptable and established financial assurance mechanisms.

At the conclusion of "active closure", when construction of all final closure activities is complete, the post-closure program would be initiated. The contemplated schedule is Years 1 through 5, 15 and 30. Closure maintenance is planned for Years 5, 15 and 30, and would vary for each of the primary components listed.

Midas Gold would compile all reporting information into a single comprehensive "environmental monitoring and mitigation report", based on these schedules. The report would contain information about the following:

- surface water quality;
- ground water quality;

- aquatic biota;
- fisheries;
- tailings storage facility;
- reclamation / re-vegetation status; and
- mitigation and conservation.

The report would be kept on file by Midas Gold, and made available to appropriate federal, state and local agencies upon request or as required under their respective permits.

20.8 CLOSURE AND RECLAMATION COSTS, AND FINANCIAL ASSURANCE

Anticipated costs for closure and reclamation of the Stibnite Gold Project were developed utilizing the Standardized Reclamation Cost Estimator (**SRCE**) model currently used and developed in Nevada for mining specific projects, supplemented by site-specific costs and quantity estimates from the FS designs. This model has been utilized for mining projects on public and private land in Nevada and other western states for many years and is publicly available online through the Nevada Division of Environmental Protection.

Closure cost estimates were developed for planned self-performance of reclamation and restoration by Midas Gold or its contractors in accordance with the mine plan timeline. Cost for reclamation and closure and conservation/mitigation measures are provided in Section 21, for both concurrent reclamation/restoration (integrated with mining costs) and final reclamation/restoration. Bonding costs, priced based on third-party performance of reclamation and closure activities in the event Midas Gold is unable to self-perform, are not included in the FS since the bonding costs and form of financial assurance are not determined as yet.

As part of the approval of a Plan for the SGP, the PNF Forest Supervisor would require Midas Gold to post financial assurance to ensure that NFS lands and resources involved with the mining operation are reclaimed in accordance with the approved Plan and reclamation requirements (36 CFR Parts 228.8 and 228.13). This financial assurance would provide adequate funding to allow the USFS to complete reclamation and post-closure operation, maintenance activities, and necessary monitoring for as long as required to return the site to a stable and acceptable condition. The amount of financial assurance would be determined by the USFS and would “address all USFS costs that would be incurred in taking over operations because of operator default” (USFS, 2004). The financial assurance would be required in a readily available financial instrument such as a surety bond or trust funds. To ensure the bond can be adjusted as needed to reflect actual costs and inflation, there would be provisions allowing for periodic adjustment on bonds in the final Plan prior to approval. Calculation of the initial bond amount would occur following the record of decision, when enough information is available to adequately and accurately perform the calculation.

In addition to the USFS-held bond, mitigation under Section 404 of the CWA also requires financial assurance. The IDL would require a bond as part of their permitting authority and would hold the bond associated with IDEQ’s cyanidation permit. The Idaho Department of Water Resources (**IDWR**) is the state agency responsible for design review and approval of the TSF. IDWR also would hold a bond so that the TSF can be placed in a safe maintenance-free condition if abandoned by the owner. These assurances are separate from those required by the USFS but are similarly excluded from the FS as the permitting status of the Project does not permit their accurate estimation.

20.9 REFERENCES

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21 CAPITAL AND OPERATING COSTS

Estimation of capital and operating costs is essential to the evaluation of the economic viability of a prospective project. These factors, combined with revenue and other expense projections, form the basis for the financial analysis presented in Section 22. Capital (**CAPEX**) and operating (**OPEX**) costs for the SGP were estimated on the basis of the feasibility mine plan, plant design, estimates of materials and labor based on that design, analysis of the process flowsheet and predicted consumption of power and supplies, budgetary quotes for major equipment, labor requirements, and estimates from consultants and potential suppliers to the project.

21.1 CAPITAL COSTS

Estimated CAPEX, or capital expenditures, include four components: (1) the initial CAPEX to undertake the detailed design, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, and operations camp, and complete on and offsite environmental mitigation and remediation; (2) the sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing environmental mitigation activities; (3) the closure and reclamation CAPEX to close and rehabilitate on and off-site components of the Project, which includes post-closure water treatment; and (4) working capital to cover delays in the receipts from sales and payments for accounts payable and financial resources tied up in inventory. Initial and working CAPEX are the two main categories that need to be available to construct a mining project. Sustaining CAPEX is critical for evaluation of all-in sustaining costs (**ASIC**) and all-in costs (**AIC**). Closure and reclamation CAPEX consists of expenditures to close and rehabilitate the mine site after mineral processing and the attendant revenue have ceased. Table 21-1 summarizes the initial, sustaining and closure CAPEX for the Project.

Table 21-1: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Closure CAPEX ($000s)(1)	Total CAPEX ($000s)
Direct Costs	Mine Costs	84,019	118,968	-	202,987
	Processing Plant	433,464	49,041	-	482,505
	On-Site Infrastructure	190,910	83,892	-	274,802
	Off-Site Infrastructure	115,940	-	-	115,940
Indirect Costs		232,684	-	-	232,684
Owner's Costs, First Fills, & Light Vehicles		38,351	-	-	38,351
Offsite Environmental Mitigation Costs		14,397	-	-	14,397
Onsite Mitigation, Monitoring, and Closure Costs		3,474	23,484	98,052	125,010
Total CAPEX without Contingency		**1,113,239**	**275,385**	**98,052**	**1,486,677**
Contingency		149,708	20,354	1,244	171,306
Total CAPEX with Contingency		**1,262,948**	**295,739**	**99,296**	**1,657,982**

Notes:
(1) Closure assumes self-performed closure costs, which will differ for those assumed for financial assurance calculations required by regulators.

The CAPEX estimate includes direct mining equipment and pre-stripping costs, process plant costs, on-site infrastructure such as the TSF and the operations camp, and off-site infrastructure such as the power transmission line, the mine access road, the Stibnite Gold Logistics Facility (**SGLF**), and reclamation and closure costs. The initial CAPEX also includes indirect costs for detailed design and engineering, land acquisition, some environmental mitigation, and other costs. Initial CAPEX also includes an estimate of contingency based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements that may occur but are not included in the cost estimate. These cost elements include uncertainties concerning completeness, accuracy and characteristics or nature of material takeoffs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

The primary assumptions used to develop the CAPEX are provided below:

- The estimate is based on 3rd quarter 2020 costs.
- All cost estimates were developed and are reported in United States of America (US) dollars.
- Units of measure for this project are primarily in English customary units.
- At the time of this estimate, engineering was approximately 20-percent complete.
- Contingency during the pre-production period is specific to each major component of the Project as determined by the various consultants.
- Qualified and experienced construction contractors will be available at the time of Project execution.
- Borrow sources are available in the Meadow Creek valley or nearby within the Project boundary.
- Weather related delays in construction are not accounted for in the estimate. However, the engineering, procurement and construction management (**EPCM**) schedule does account for a ramp down in construction activity during the three winter months (December, January, and February).
- The oxygen plant is accounted for as an "over-the-fence" supply contract. Capital costs have been included for building a dedicated substation for the oxygen plant. Midas Gold will supply power and other utilities to the oxygen plant during operations as well as provide beds at the operations camp for its workers.
- Financial assurance costs associated with closure-related bonding are excluded from this estimate.
- No provision has been made for currency fluctuations.

21.1.1 Mine Capital Costs

The mine capital generally includes three components: the mining fleet, mine support equipment, and the cost of pre-stripping. Mine capital cost for mobile equipment was developed from the mine equipment list presented in Section 16. Mine capital costs including equipment and pre-production development are presented in Table 21-2.

Table 21-2: Mine Capital Cost Summary

Mining CAPEX Components	Pre-Production ($000s)	Sustaining ($000s)	Total CAPEX ($000s)
Mine Major Equipment (Leased)	44,013	105,424	149,437
Mine Support Equipment (Purchased)	18,538	13,543	32,082
Capitalized Preproduction Development (30%)	21,468	-	21,468
Total Mining CAPEX	**84,019**	**118,968**	**202,987**

Notes:
(1) Pre-production mining costs include environmental remediation costs as discussed in Section 21.1.6.1; the remaining 70% of preproduction development is included in OPEX as detailed in Table 21.5.
(2) All mine support equipment is purchased except for motor graders which are leased.

Midas Gold plans to lease the major mining equipment. The down payment, principal payment, and buyout portions of the leasing costs for the mining fleet are included in initial and sustaining CAPEX. During pre-production, 30% of the mining fleet OPEX is accounted as initial CAPEX. Lease rates were based on 60-month leases with equipment buyouts at the end of the lease period.

Lease rates for the major mine equipment were obtained from local major mine equipment vendors. Lease down payments, lease principal payments, and end of lease term buyout options are accounted for as capital costs.

Capital costs for each equipment type were estimated using vendor budgetary quotes or recent mining industry surveys. Equipment capital costs include estimates for freight, assembly, spare parts, initial tire purchase, fire suppression, equipment advance payments, and potential equipment modifications. For equipment that is planned to be leased, pay schedules are based on quotes provided by equipment manufacturers. The mining equipment purchase schedule is shown in Table 21-3.

Table 21-3: Mining Equipment Purchase Schedule

Equipment	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
28-yd^3 Hydraulic Shovel	2	-	-	2	-	-	-	-	-	-	-	-	-	-	-	-
28-yd^3 Wheel Loader	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
150-ton Haul Truck	18	-	-	16	-	-	2	-	-	-	-	-	-	-	-	-
600-Hp Track Dozer	8	-	2	1	1	-	-	-	-	2	1	-	1	-	-	-
Blasthole Drill	6	-	-	5	-	-	-	-	1	-	-	-	-	-	-	-
5-yd^3 Excavator	6	-	2	1	-	-	1	-	-	1	-	-	-	-	1	-
8-yd^3 Wheel Loader	5	-	2	-	-	-	1	-	1	-	-	-	1	-	-	-
45-ton Articulated Truck	16	-	9	3	-	-	-	-	-	2	-	-	-	-	1	1
215-Hp Track Dozer	6	-	2	1	-	-	-	-	-	-	2	-	-	-	1	-
Track Mounted Drill	2	-	1	1	-	-	-	-	-	-	-	-	-	-	-	-
18-ft Blade Motor Grader	4	-	1	-	1	-	-	-	-	1	-	-	-	1	-	-
14-ft Blade Motor Grader	2	-	-	1	-	-	-	-	-	-	1	-	-	-	-	-
9k gallon Water Truck	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
ANFO Truck	4	-	2	-	-	-	-	-	-	2	-	-	-	-	-	-
15-yd^3 Stemming Truck	3	-	1	-	-	-	-	1	-	-	-	1	-	-	-	-
100-ton Rock Spreader	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-
100-ton Lowboy Trailer	1	-	-	-	-	1	-	-	-	-	-	-	-	-	-	-
45-ton Fuel & Lube Truck	4	-	1	1	-	-	-	-	1	1	-	-	-	-	-	-
Mechanics Truck	4	-	1	1	-	-	-	-	-	1	-	1	-	-	-	-
Tire Service Truck	2	-	1	-	-	-	-	-	-	-	1	-	-	-	-	-
Flatbed Truck	3	-	2	-	-	-	-	-	-	-	-	-	-	-	1	-
6k-lb Forklift	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-
11k-lb Telehandler	2	-	1	-	-	-	-	-	-	-	-	1	-	-	-	-

There are certain capital costs associated with the mine that are included elsewhere in the estimate. These items include mine office buildings, shop facilities, mobile equipment that is not required by the mine, and all infrastructure costs (except for haul roads).

Table 21-4 summarizes the mine capital costs by year. The down payment, principal payments, and buyout costs for the mine major equipment are included as capital costs. Preproduction stripping is part of the mine capital cost but is shown separately to differentiate it from the cost of purchasing mine equipment.

Table 21-4: Life-of-Mine Mining Capital Cost Detail

Production Year	Mine Equipment		Capitalized Preproduction Consumables and Labor ($000s)	Total(1)(2) Mine Capital ($000s)
	Leased Major Equipment Down & Monthly Payments ($000s)	Other Support Equipment Capital Costs ($000s)		
Initial Capital				
-3	1,115	2,817	867	4,798
-2	8,869	10,188	6,174	25,232
-1	34,029	5,534	14,427	53,989
Sub-Totals	**44,013**	**18,538**	**21,468**	**84,019**
Sustaining Capital				
1	13,146	872	-	14,017
2	14,299	2,184	-	16,483
3	19,358	633	-	19,991
4	16,303	628	-	16,932
5	21,039	2,556	-	23,596
6	4,220	2,477	-	6,697
7	2,985	1,218	-	4,203
8	3,298	439	-	3,737
9	2,112	261	-	2,373
10	2,621	842	-	3,462
11	3,184	607	-	3,792
12	1,697	339	-	2,036
13	429	139	-	569
14	599	140	-	739
15	135	208	-	343
Sub-Totals	**105,424**	**13,543**		**118,968**
Totals	**149,437**	**32,082**	**21,468**	**202,987**

Notes:
(1) Mine preproduction development is shown as 30% capital cost and 70% operating expense.
(2) Lease down payments, principal payments and end of lease term buyout options shown as a capital cost.

Major mine equipment is leased in the year it is required for operation. The acquisition schedule for the leased major mine mobile equipment is provided in Section 16. The mine capital costs in Table 21-4 represent major mine equipment being leased throughout the mine life and bought out when the lease term has expired. Mine support equipment is purchased outright except motor graders and includes auxiliary equipment (e.g., water trucks, light plants, ANFO trucks), mine maintenance vehicles, and mine administration vehicles, such as pickup trucks for mine supervisors.

Table 21-4 also includes the mine support equipment capital costs. Mine support equipment pricing was priced from vendor quotes. The truck shop, truck wash, and truck shop warehouse are included in the Plant CAPEX.

Pre-stripping requirements were developed monthly to provide ore exposure for production in Year 1 and construction material for the TSF starter dam. A total of 28.5 Mst of development rock would be mined during preproduction from Yellow Pine open pit, West End open pit, SODA area and TSF borrow. Mining costs during pre-production were based on areas stripped, haul profiles, established equipment rates and estimated operator wages. The cost build-up assumes that pre-stripping activities will be conducted by an owner-operated fleet using leased equipment.

Table 21-5 shows the estimated development costs by year before start-up. The costs for topsoil stripping and storage are included in the mining costs. Development costs in preproduction Year -3 include costs for the haul road between the Yellow Pine pit and the TSF. The development costs are divided between capital (30%) and operating (70%) expenses, showing the detail for the Capitalized Pre-production Development shown in Table 21-2.

Table 21-5: Mine Pre-Production Expense

Period	Development Costs ($000s)	CAPEX 30% ($000s)	OPEX 70% ($000s)
Year -3	2,890	867	2,023
Year -2	20,581	6,174	14,407
Year -1	48,089	14,427	33,663
Totals	**71,561**	**21,468**	**50,093**

21.1.2 Plant Capital Costs

Capital costs for the processing plant were estimated using budgetary equipment quotes, material take-offs (**MTO**s) for concrete, steel, and earthwork, estimates from vendors and consultants, and estimates based on experience with similar projects of this type. The capital cost estimate for the plant is shown in Table 21-6. Some of the costs and quantity estimates used by M3 were supplied by other consultants.

Table 21-6: Plant Capital Cost Summary

Area Description	Initial ($000s)	Sustaining ($000s)	Total ($000s)
General /Standards/ Site Plan	33,985	-	33,985
Historical Tailings Re-Pulping	4,690	-	4,690
Primary Crusher	14,496	-	14,496
Crushed Ore Stockpile & Reclaim	18,312	-	18,312
Grinding and Classification	74,975	-	74,975
Pebble Crushing Circuit	7,231	-	7,231
Antimony Recovery	8,001	-	8,001
Gold Flotation	28,992	-	28,992
Pressure Oxidation	112,270	10,379	122,648
Slurry Cooling and Neutralization	27,667	-	27,667
POX Leach/CIP	17,080	-	17,080
Tailings/Oxide CIP	-	38,663	38,663
Carbon Handling & Refinery	15,025	-	15,025
Fresh Water System	8,415	-	8,415
Main Substation	13,333	-	13,333
Reagents	16,979	-	16,979
Limestone and Lime	30,049	-	30,049
Oxygen Plant	1,964	-	1,964
Total Plant CAPEX	**433,464**	**49,041**	**482,505**

21.1.2.1 Plant Capital Basis of Estimate

The capital cost estimate is based on the cost of equipment, material, labor, and construction equipment needed to complete the plant up to start-up. The accuracy of the CAPEX estimate at the feasibility level is -10% to +15%. Data for this estimate was obtained from numerous sources including:

- Feasibility-level design engineering consisting of flow sheets, general arrangement plans and cross sections, civil grading drawings, process and instrumentation diagrams (**P&ID**s), and electrical one-line drawings;
- Pressure oxidation engineering conducted by Hydromet;
- Topographical base information provided by Midas Gold from a 2009 aerial LiDAR survey augmented by a 2013 LiDAR survey for outlying areas for the mine access road;

- Budgetary equipment and materials quotations from vendors; and
- Construction labor rates were based on crew rates developed using the published prevailing shop wages from Davis-Bacon for July 24.

Below is a description of the pricing that was used by category.

Capital Equipment Pricing

Prices were solicited for all major equipment. Procurement packages of similar equipment were sent to three qualified suppliers to get budgetary quotations. Major capital equipment categories for this Project included electrical, mechanical, and piping. Accuracy of +/- 15% was requested from suppliers for this CAPEX. For some equipment generally under $100,000 in value, pricing data were taken from recent M3 projects.

Electrical Equipment

One-line electrical distribution diagrams were designed for each plant and ancillary area to determine the required number and size of transformers, switchgear, and motor control centers. These one-line drawings were sent to three qualified electrical suppliers for direct pricing. The intent is to maximize the use of prefabricated e-houses as much as possible to minimize on-site labor. Vendors supplied quotes for these e-houses. In some cases, the electrical rooms are too large to allow for the prefabricated option. In those cases, vendors provided quotes for the equipment only. Quotes were evaluated by the electrical engineer to ensure that the specifications for the equipment were met. In general, the price that was used in the capital cost estimate was based on the most suitable quote, not the lowest cost proposal.

Electrical bulk materials were factored by area and benchmarked from recent projects. The cost of electrical equipment was subtracted from the factors except in cases where the electrical costs were judged to be too low.

Mechanical Equipment

All major mechanical equipment was priced for the capital cost estimate by soliciting budgetary quotations, or in the case of minor equipment, from quotes or purchases from recent jobs. The vendors that were approached were generally the best-known suppliers of process equipment in the mining industry: Metso, FL Smidth, Outotec, Sandvik, Weir Warman, GIW, Goulds, Flowserve, Delkor Tennova, McClanahan, Konecranes, etc. Autoclave equipment prices were solicited from the main providers: Carpenteria Corsi, Morimatsu, Access Petrotec, Stebbins, Koch-Knight, DSB, Ekato, Lightnin, Hayward Gordon, Mikropul, Clean Gas Systems, Weir-Geho, Midwest Cooling Towers, Marley, Clayton, Cleaver-Brooks, and others. Operating data sheets (ODSs) were developed to provide duty specifications for each unique piece of major equipment in the Equipment Register. The ODSs were populated with process flows and data from the METSIM process simulation, from specifications in the Process Design Criteria, and from physical information derived from General Arrangement drawings. Vendors were provided capacities and flows (nominal and design), specific gravity and bulk density, slurry densities (percent solids), work and abrasion indices, materials of construction, and other information needed to receive a credible quote. All quotes were evaluated to determine if they met the duty specifications. In general, the price that was used in the capital cost estimate was based on the most suitable quote, not the lowest cost proposal. Mechanical equipment quotes were obtained for:

- jaw crusher;
- conveyors & stacker;
- reclaim apron feeders;
- SAG/ball mills with trammel screens;

- cone (pebble) crusher;
- hydrocyclones;
- flotation tank cells for both antimony and gold rougher and cleaner circuits;
- concentrate and neutralization thickeners;
- plate-and-frame filter presses;
- field-erected and shop-fabricated tanks;
- POX equipment including the autoclave, flash tanks, autoclave agitators, positive displacement feed pumps, steam generators, and Venturi scrubbers;
- conventional 7-ton carbon plant for gold recovery and carbon regeneration;
- screen plant for re-pulping Historical Tailings; and
- tailings, slurry, froth, and process pumps.

Piping, Pump, and Valve Quotes

A list of pumps was developed for all process areas. Operating data were tabulated for all pumps on this list including flow, total dynamic head, percent solids, slurry specific gravity, service, corrosivity, and pump style (horizontal centrifugal, vertical turbine, etc.). Requests for budgetary quotes were furnished to three or more pump suppliers for comparative quotes. A piping engineer reviewed the vendor submissions and technical information to select the appropriate equipment to include in the capital cost estimate.

Hydromet sized and specified the valves in the autoclave area. These valves were priced by known providers, Caldera, Ferguson, Salt Lake Windustrial, Control Distributors, Caltrol, Bray, and Rust. The total bill of materials for autoclave area valves is $11.7 million. Piping costs were based on MTOs from P&IDs and quotes received for carbon steel (including HDPE or rubber-lined), stainless steel, and HDPE pipe.

Structural Steel and Concrete Quantity Estimates

Structural steel and concrete quantities were based on MTOs. Dimensions were taken from design drawings and entered into a spreadsheet to provide quantities for estimation. The spreadsheet provided total quantities of each category of steel by plant area number. Concrete quantity totals were similarly compiled by type and plant area number.

Concrete & Structural Commodity Pricing

Unit pricing was solicited from four structural steel providers for the Project, which were adjusted for steel unit prices typical for current large EPCM jobs. These unit prices were applied by the estimator to the quantities provided in the MTOs.

A regional concrete supplier provided prices for supply of concrete predicated on the assumption that a batch plant would be set up on site and that aggregate would be available from site-furnished materials. A crushing and screening plant would also be needed to make the particle size gradations for concrete mix designs. The cost to house the batch plant operators was also included in the prices for the various strengths of concrete.

Instrumentation

Instrumentation materials costs were based on instrumentation lists derived from P&IDs developed for the feasibility study.

21.1.3 Infrastructure Costs

21.1.3.1 Onsite Infrastructure

The onsite Infrastructure includes site utilities and roads, auxiliary facilities, the TSF, water management systems, and the operations camp. Table 21-7 summarizes the direct costs for onsite infrastructure. The 300-bed operations camp would be formed from the 1,000-bed construction camp by removing 700 beds after start-up; the dining and housekeeping facilities, fresh water supply, power distribution, and wastewater treatment at the camp would remain. The direct costs are based mainly on budgetary quotations of from a local supplier of modular camps with specific experience at the Stibnite Gold site. The total direct cost of the operations plus construction camp facility, shown in Table 21-7, does not include the cost of catering or housekeeping.

Table 21-7: Onsite Infrastructure CAPEX Summary

Onsite Infrastructure	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Ancillary Facilities	26,602	-	26,602
Tailings Storage Facility / Reclaim System	69,313	63,294	132,608
Water Management	59,621	10,435	70,056
Mine-Impacted Water Treatment Plant	5,022	10,163	15,184
Permanent Camp	30,351	-	30,351
Total Onsite Infrastructure	**190,910**	**83,892**	**274,802**

The ancillary facilities include a variety of offices, shops, and warehouses that support the day-to-day operations of the mine and the plant. Table 21-8 lists the main ancillary facilities and their direct costs that were included in the initial CAPEX.

Table 21-8: Onsite Ancillary Facilities CAPEX

Onsite Ancillary Facilities	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Ancillaries General	2,032	-	2,032
Administration Building	1,198	-	1,198
Security Building	365	-	365
Medical & Emergency Services	2,161	-	2,161
Mine Ops/Mine Dry Building	462	-	462
Truck Shop/Truck Wash/Truck Warehouse	11,393	-	11,393
Reagents Warehouse	3,961	-	3,961
Plant Maintenance Building	2,656	-	2,656
Assay Lab	764	-	764
Fuel Station	1,392	-	1,392
Explosive Storage	219	-	219
Total Onsite Auxiliary Facilities	**26,602**	**-**	**26,602**

The capital components that make-up the tailings management system consist of the TSF embankment, the tailings impoundment and liner, tailings pumps, slurry pipeline system, water reclaim system, TSF under-liner drains, TSF surface water diversions, and the civil work that is required to route the tailings and reclaim water lines between the process plant and the TSF. Capital costs for the TSF and buttress water diversions, embankment and impoundment construction, liner, over-liner drain, and under-liner drain were estimated by Tierra Group. The water reclaim system consists of reclaim barge, pumps, head tank, pipeline, and process water storage tank, estimated by M3.

The TSF will be constructed in five stages. The Stage 1 TSF, constructed in Years -2 and -1, would be preceded by the construction of the TSF and buttress diversion channels. Stages 2 and 3 would be constructed over two years each, finishing in Years 2 and 5, respectively. Stages 4 and 5 would be completed in a single year, finishing in Years 8 and 11. The tailings and reclaim pipeline corridor must be relocated out of the footprint of the Hangar Flats pit in Year 3, resulting in additional sustaining CAPEX. Table 21-9 summarizes the direct CAPEX costs for the TSF.

Table 21-9: Tailings Storage Facility CAPEX

Tailings Storage Facility	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Surface Water Diversion	12,351	-	12,351
Embankment and Impoundment	28,756	61,370	90,126
Tailing Pipeline & Water Reclaim System	28,206	1,925	30,131
TSF, Diversion, and Reclaim System	**69,313**	**63,295**	**132,608**

Water management systems include pit dewatering; surface diversions (excluding the TSF diversion); contact water ponds, pumps, and piping; water treatment; and a diversion tunnel for the EFSFSR. The EFSFSR diversion includes the surface approaches and exit to the tunnel diversion around the Yellow Pine pit, fishway, freshwater intake and the diversion tunnel itself. The contact water management systems were estimated by M3 while the tunnel diversion was estimated by McMillen Jacobs Associates. CAPEX for water management systems is shown in Table 21-10 include initial and sustaining CAPEX. Initial CAPEX includes water management systems (excluding the TSF and buttress), pre-operation water treatment, and tunnel diversion around the Yellow Pine pit. Sustaining CAPEX costs are also estimated for water management modifications and water treatment required by the changes in the mining operation.

Table 21-10: Water Management CAPEX

Water Management Systems	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Water Treatment Plant	5,022	10,163	15,184
Dewatering, Contact Water Systems, & Diversions	29,527	10,435	39,962
Water Diversion Tunnel & Intake (MJA)	30,094	-	30,094
Water Management Totals	**64,642**	**20,598**	**85,240**

21.1.3.2 Offsite Infrastructure

The offsite infrastructure includes three main components: the mine access road, the public bypass road near the Yellow Pine pit, the power transmission line, the Burntlog Road Maintenance Facility, and the Stibnite Gold Logistics Facility, which includes administration offices, the production assay lab, the staging area for mine personnel transportation, and warehouse capacity. Table 21-11 summarizes the direct costs estimated for these five components.

The mine access roads are described in Section 18.2. The FS designs and cost estimates for the Burntlog Route and Public Bypass road were developed by Parametrix. The cost estimates include civil excavation costs, placement of aggregate base course and geotextile, emplacement of culverts, retaining walls, installation/upgrade of bridges, the installation of a storm water drainage system, and other minor costs.

The power supply infrastructure upgrades are described in Section 18.3. The cost for the power transmission line, communications, and substation upgrades was developed by HDR, in consultation with Idaho Power Company. Increasing the power supply includes upgrading seven substations, installation of a new switching station in Cascade and a substation at the Stibnite Gold Logistics Facility (**SGLF**), and construction of a new transmission line with under-built fiber optic communication line from Cascade to the mine site.

The Burntlog Road Maintenance Facility is designed for a location 4.4 miles from the junction of Warm Lake and Johnson Creek roads, as described in Section 18.4.2. The cost estimate includes a 7,500-square-foot maintenance building, a 7,100-square-foot aggregate storage building, a 4,300-square-foot equipment shelter, and an 825-square-foot sleeping quarters.

The SGLF is described in Section 18.4.1. The facility design includes administrative offices and an analytical laboratory, both of modular construction; a pre-engineered warehouse; and parking and transportation areas for employees bussed to the site. The estimated direct costs of these facilities do not include land acquisition costs. The land for the SGLF is owned by MGII.

Table 21-11: Offsite Infrastructure Summary

Off-Site Infrastructure	Initial ($000s)	Sustaining ($000s)	Total ($000s)
Mine Access Road	49,121	-	49,121
Public Bypass Road	2,426	-	2,426
Power Supply Infrastructure	52,641	-	52,641
Burntlog Road Maintenance Facility	4,449	-	4,449
Stibnite Gold Logistics Facility	7,303	-	7,303
Total Off-Site Infrastructure	**115,940**	**-**	**115,940**

21.1.4 Indirect Costs

Indirect costs are those costs that can generally not be tied to a specific work area, as summarized in Table 21-12.

Table 21-12: Indirect Capital Cost Summary

Indirect Cost Items	Cost ($000s)
Bussing	2,453
Mobilization - Plant Contractors	11,019
Freight	30,336
EPCM Contract	105,372
Temporary Construction Facilities	3,229
Temporary Construction Power	646
Construction Camp Operation Costs	24,025
Vender Representative Supervision	3,947
Start-up and Commissioning	2,631
Commissioning and Capital Spares	6,578
Consultant Indirect Estimates	29,936
Idaho Sales Tax	12,512
Total Indirect Costs	**232,684**

This category includes “other direct costs” that are related to construction that can’t be assigned directly to a work area including the following:

- bussing workers from the SGLF to the mine site during construction;
- mobilization of contractors is 0.5% of total direct cost without mine and mobile equipment and including quality assurance;

- EPCM contract, fee, temporary facilities, and support;
- temporary construction facilities;
- temporary construction power supply;
- construction camp operating costs;
- vendor representative supervision;
- start-up and commissioning;
- commissioning and capital spares; and
- Idaho State Sales Tax.

21.1.4.1 EPCM Costs

M3 breaks down estimated EPCM costs into various categories that total 16.7% of direct constructed field cost excluding mining pre-strip and mine equipment costs, as shown in Table 21-13.

Table 21-13: EPCM Capital Cost Summary

EPCM Components	Percentage of Total Direct Field Cost	Cost ($000s)
Management & Accounting	0.75%	4,843
Engineering	6.00%	36,278
Project Services	1.00%	6,457
Project Controls	0.75%	4,843
Construction Management	6.50%	41,973
EPCM Fee	1.50%	9,686
EPCM Temporary Facilities & Support	0.18%	1,291
EPCM Total	**16.68%**	**105,372**

21.1.4.2 Other Indirect Costs

Table 21-12 also includes indirect costs from other consultants for infrastructure engineering and construction, including the power transmission line, mine access roads, TSF, and water diversions. The indirect costs for these tasks were provided by the estimating entity, as detailed in Table 21-14.

Table 21-14: Consultants' Indirect Capital Cost Estimates

Consultants' Indirect Cost Estimates	Cost ($000s)
Tailings Construction (Tierra Group)	3,280
EFSFSR Diversion and Intake (McMillen Jacobs)	8,766
Access Road (Parametrix)	7,484
Public Bypass Road (Parametrix)	505
Power Supply Infrastructure (HDR)	9,901
Total Consultants' Indirect Estimates	**29,936**

21.1.5 Owner Costs

Owner costs were developed to cover specific functions relating to the construction of the Project. Owner costs exclude exploration and corporate costs and are summarized in Table 21-15.

Key staff, plant and equipment operators will be hired as early as three months prior to start-up for training, and preparation work. Senior staff and engineering personnel will also be hired several months prior to start-up as they become available. Environmental monitoring will continue through the construction period. Other Owner Cost items include:

- Owner's construction and administrative costs, including the Owners camp;
- plant mobile equipment and light vehicles;
- insurance, accounting and legal;
- furniture and office equipment;
- tools;
- staffing and operator training cost; and
- initial fills and wear steel spares.

Table 21-15: Owner Team Capital Costs

Owner Team Item	Total ($000s)
Stibnite Pre-Operations Team Salaries & Burden	7,820
SGLF Pre-Operations Team Salaries & Burden	3,633
Owner's Team Indirect Costs	4,535
Community Relations Costs	1,098
Land, Legal & Insurance Costs	8,027
First Fills	4,365
Mobile Equipment & Light Vehicles	8,874
Total Owner Costs	**38,351**

21.1.6 Environmental Mitigation, Reclamation, and Closure Costs

The Project site is located near the headwaters of the EFSFSR and has been environmentally impacted by historical mining activities. MGII has integrated environmental remediation and restoration activities with the operating plan and will be required to reclaim Project disturbance and accomplish both onsite and offsite stream and wetland compensatory mitigation to offset impacts to these resources attendant to the mining operation. Additionally, offsite road intersection improvements are included as mitigation for traffic impacts. Capital costs for these activities are summarized in Table 21-16. These costs are divided into three time periods: pre-operation (initial), operation (sustaining, i.e., for concurrent reclamation), and post-operation (closure).

Table 21-16: Mitigation, Reclamation, and Closure Costs

Environmental Mitigation and Reclamation	Initial ($000s)	Sustaining ($000s)	Closure ($000s)	Total ($000s)
Offsite Mitigation	14,397	-	-	14,397
Onsite Mitigation, Reclamation, and Closure	3,474	23,484	98,052	125,010
Total Mitigation and Reclamation Costs	**17,871**	**23,484**	**98,052**	**139,407**

Closure and reclamation costs were developed utilizing the Standardized Reclamation Cost Estimator (**SRCE**), discussed in Section 20, based on these activities being conducted by the operator, and do not include management and administration by outside entities. Costs were then incorporated into the overall Project cost model in the year that they occur.

Closure costs include items such as potential long-term water treatment, stream and wetland restoration, reclamation and reclamation maintenance, and long-term site monitoring such as surface and ground water monitoring, vegetation success monitoring, aquatic species and habitat monitoring, and chemical and physical stability. Water treatment during construction and operations is included in the water management cost discussed in Section 21.1.3.1. Bulk earthmoving of legacy materials accomplished by the mine fleet is included in the mining operations cost (Section 21.2.2).

Long-term closure and monitoring costs are factored from anticipated operational costs, experience from closure operations of similar projects, first principles construction costs, and standard unit costs. The schedule of costs for reclamation, closure, and post-closure are allocated along the life of mine and closure, based upon expected reclamation and closure related activities.

Reclamation bonding will be required by the permitting authorities before construction of the Project can be initiated. Bonding costs have not been included in the capital cost estimate because the structure and amount of the bonding requirement will be established in the future with permitting authorities.

21.1.7 Contingency

Contingency costs, as summarized in Table 21-17, are estimates of the costs that are not included in the CAPEX that can be expected to be spent during initial construction. The more engineering and construction execution planning that is done ahead of the estimate, the higher the accuracy of the CAPEX and thus, the lower the contingency costs. The total estimated contingency for this Project, 15.2% of the total initial CAPEX before sales tax, is considered typical for a feasibility-level study.

Table 21-17: Summary of Contingency Capital Costs

Contingency Components	Percent	Cost ($000s)
Plant Construction (M3)	15.0%	114,466
Tailings Facility (Tierra Group)	15.0%	6,166
Diversion Tunnel and Intake (MJA)	15.0%	4,514
Mine Access Road (Parametrix) - Segment A	10.0%	305
Mine Access Road (Parametrix) - Segment B	15.0%	1,213
Mine Access Road (Parametrix) - Segment C	15.0%	1,307
Mine Access Road (Parametrix) - Segment D	20.0%	3,080
Mine Access Road (Parametrix) - Segment E	20.0%	2,050
Access Road Maintenance Pre-Operations	15.0%	539
Public Bypass Road (Parametrix)	30.0%	728
Power Supply Upgrades (HDR)	17.0%	10,604
Pre-Operation Water Treatment Plant	15.0%	619
Road Intersection upgrades (Parametrix)	20.0%	350
Owner's Cost	15.0%	3,767
Contingency Total	**15.2%**	**149,708**

21.2 OPERATING COSTS

The average cash operating cost per short ton (**st**) of processed material before by-product credits, royalties, refining and transportation charges over the life-of-mine (**LOM**) and during the first four years of operations are summarized in Table 21-18. These cash costs include mine operations, process plant operations, and general and administrative costs (**G&A**). The average cash operating cost per ton of processed material after by-product credits but before royalties, refining and transportation charges over the LOM and during the first four years of operations are also provided, as are the all-in sustaining costs (**AISC**) and all-in costs (**AIC**). Total costs in each category are divided by the total tonnage of processed material or the total ounces produced to arrive at the values shown.

Table 21-18: Cash Costs, All-In Sustaining Costs, and All-In Costs

Total Production Cost Item	Years 1-4		LOM	
	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	9.71	156	8.22	205
Processing	13.13	211	12.76	318
G&A	3.54	57	3.43	85
Cash Costs Before By-Product Credits	**26.38**	**424**	**24.41**	**608**
By-Product Credits	(5.99)	(96)	(2.81)	(70)
Cash Costs After By-Product Credits	**20.40**	**328**	**21.60**	**538**
Royalties	1.69	27	1.09	27
Refining and Transportation	0.46	7	0.24	6
Total Cash Costs	**22.54**	**362**	**22.94**	**571**
Sustaining CAPEX	4.64	75	2.83	70
Salvage	-	-	(0.26)	(6)
Property Taxes	0.05	1	0.04	1
All-In Sustaining Costs	**27.23**	**438**	**25.54**	**636**
Reclamation and Closure(1)	-	-	0.95	24
Initial (non-sustaining) CAPEX(2)	-	-	11.65	290
All-In Costs	**-**	**-**	**38.14**	**950**

Notes:
(1) Defined as non-sustaining reclamation and closure costs in the post-operations period.
(2) Initial Capital includes capitalized preproduction.

21.2.1 Mine Operating Costs

Mine operating costs were developed based on first principles for the mine plan and equipment list presented in Section 16. The unit costs for labor were jointly developed by Midas Gold and M3. Table 21-19 summarizes the consumable and labor operating costs by the unit operations.

Table 21-19: Life-of-Mine Mining Cost Averages

Mining Function	Percentage	Unit Cost ($/st)
Drilling	9.8%	0.22
Blasting	12.5%	0.28
Loading	7.1%	0.16
Hauling	28.1%	0.63
Auxiliary	11.6%	0.26
Mine Development	17.9%	0.40
General Mine	4.0%	0.09
General Maintenance	4.0%	0.09
G&A	4.9%	0.11
Total for Material Mined	**100.0%**	**2.24**

Preproduction development costs (Table 21-4) are carried 30% as CAPEX and the remaining 70% as OPEX. Table 21-20 summarizes the total mine operating cost per year.

Table 21-20: Mine OPEX by Year

Year	Total ($000s)
-3	2,023
-2	14,407
-1	33,663
1	67,675
2	67,397
3	73,575
4	77,534
5	70,216
6	66,569
7	63,817
8	63,141
9	61,576
10	57,809
11	53,198
12	34,624
13	22,514
14	19,064
15	11,349
Total	**860,151**

Note: Mine preproduction development is shown as 30% capital cost and 70% operating expense.

The mine operating costs provided in Table 21-20 include:

1. Drilling, blasting, loading, and hauling of material from the mine to the crusher, stockpiles or development rock storage facilities. Maintenance of the development rock storage areas and stockpiles is included in the mining costs. Maintenance of mine mobile equipment is included in the operating costs.
2. Rehandling ore stockpiles to the crusher is included in the mining costs.
3. Mine supervision, mine engineering, geology and ore control are included in the G&A category.
4. Operating labor and maintenance labor for the mine mobile equipment are included.
5. Mine access road construction and maintenance are included. If mine haul trucks drive on the road, its cost and maintenance is included in the mine operating costs.
6. Relocation of SODA material and reprocessing of Historical Tailings is included.
7. Delivery of mine development rock to the tailings dam construction is included. However, placement and compaction of that material at the TSF is not included.
8. The cost of backfilling the Yellow Pine open pit, Hangar Flats open pit, and Midnight area of the West End open pit is included.
9. A general mine allowance is included that is intended to cover mine pumping costs and general operating supplies that cannot be assigned to one of the unit operations.
10. A general maintenance allowance is included that is intended to cover the general operating supplies of the maintenance group.

The mine is planned to work two 12-hour shifts per day for 365 days per year. Ten days (20 shifts) of lost time are assumed due to weather delays or other interruptions.

21.2.2 Plant Operating Costs

The process plant operating costs are summarized by the categories of labor, electric power, liners (wear steel), grinding media, reagents, maintenance parts and services, annual POX shutdown, oxygen, and supplies and services, as presented in Table 21-22.

Table 21-21: Process Plant OPEX Summary by Category

Plant Operation Cost Category	LOM Cost ($000s)	Cost ($/st)
Labor	197,965	1.72
Power	249,107	2.16
Liners	56,099	0.49
Grinding Media	124,421	1.08
Reagents	350,842	3.04
Maintenance Parts & Services	173,236	1.50
Annual POX Shutdown	52,000	0.45
Oxygen	80,803	0.70
Water Treatment Plant	3,157	0.03
Supplies & Services	47,649	0.41
Totals	**1,335,279**	**11.58**

The processing costs allocated by process area are provided in Table 21-22.

Table 21-22: Process Plant OPEX by Process Area

Process Area	LOM Cost ($000s)	Cost ($/st)
Crushing and Conveying	29,183	0.25
Grinding & Classification	402,852	3.49
Antimony Recovery	29,125	0.25
Gold Flotation	115,856	1.00
Pressure Oxidation	279,829	2.43
POX Discharge Cooling, HC & Neutralization	91,940	0.80
POX Leach-CIP Circuit	163,030	1.41
Tailings / Oxide Leach-CIP	45,802	0.40
Carbon Handling & Refinery	55,230	0.48
Tailings & Water Reclaim	43,353	0.38
Water Treatment	3,157	0.03
Fresh / Contact Water System	13,210	0.11
Ancillaries	62,712	0.54
Total Process Plant	**1,335,279**	**11.58**

21.2.2.1 Process Plant Labor Costs

The process plant operating and maintenance labor costs were derived from a staffing plan and are based on labor rates from an industry survey for this region and modified where necessary. The annual salaries include overtime and benefits for both salaried and hourly employees. The burden rate used is 35% for hourly staff and 40% for salaried staff to include a 5% average annual bonus provision. The process labor numbers of personnel and costs divided by process area are provided in Table 21-24.

Table 21-23: Process Labor Costs by Process Area

Labor Costs by Process Area	Number of Personnel	Annual Labor Cost ($000s)
Crushing and Conveying	8	691
Grinding	12	1,080
Antimony Recovery	8	691
Gold Flotation Operator	4	378
Pressure Oxidation	12	1,004
Tailings	4	346
Carbon Handling & Refinery	12	1,037
Ancillaries	25	2,669
Maintenance	61	5,440
Totals	**146**	**13,336**

21.2.3 General and Administrative Costs

General and Administrative (**G&A**) costs include management, accounting, human resources, environmental and safety compliance, laboratory, community relations, site residential camp, communications, insurance, legal, training, and other costs not associated with either mining or processing. The LOM G&A cost estimated for the Project are presented in Table 21-28.

Table 21-24: Life-of Mine General and Administration Cost Detail

Cost Item	LOM ($000s)
Labor & Fringes (G&A and Lab)	128,999
Accounting (excluding labor)	1,450
Safety (excluding labor)	1,450
Human Resources (excluding labor)	1,450
Security (excluding labor)	1,450
Laboratory (excluding labor)	10,875
Janitorial Services (contract)	2,900
Office Operating Supplies and Postage	1,450
Maintenance Supplies	14,418
Maintenance Labor, Fringes, and Allocations	3,625
Access Road Maintenance	20,942
Power	2,646
Propane	2,900
Phone/Communications	4,350
Licenses, Fees, and Vehicle Taxes	2,175
Legal	3,625
Insurances	36,975
Water Rights	2,175
Claim Payments	3,759
Property Tax	232
Subs, Dues, PR, and Donations	1,450
Travel, Lodging, and Meals	2,900

Cost Item	LOM ($000s)
Camp	83,563
Busing	2,654
Training	3,625
Stibnite Foundation Payments	15,817
Total	**357,857**

21.2.4 Labor Requirements

Labor for the Project was estimated for the mine, process plant, and G&A support. Labor rates were estimated using market surveys for the region and comparable wage rates from other mining operations in the area. Onsite personnel were assumed to be housed in a camp facility and working 12-hour shifts on a 14-day on, 14-day off work schedule except for salaried employees. A breakdown of the labor requirements stratified by function (mine, process, or G&A) and location (onsite or offsite) is presented in Table 21-25 with the annual estimated payroll for an average year.

Table 21-25: Estimated Labor Requirements

Labor Category	Number of Personnel			Average Annual Payroll ($000s)
	Low	Peak	Average	
Mine Operations Personnel - Hourly	113	192	172	16,287
Mine Personnel - Salaried	23	35	30	4,305
Mine Maintenance Personnel - Hourly	41	78	70	6,253
Mine Maintenance Personnel - Salaried	8	11	10	1,446
Process Operations Personnel - Hourly	98	98	98	8,429
Process Operations Personnel - Salaried	13	13	13	1,841
Process Maintenance Personnel - Hourly	56	56	56	5,114
Process Maintenance Personnel - Salaried	5	5	5	692
G&A Hourly Personnel - Onsite	19	19	19	1,442
G&A Salaried Personnel - Onsite	12	12	12	1,241
G&A Hourly Personnel - Offsite	19	19	19	1,211
G&A Salaried Personnel - Offsite	16	16	16	1,882
Labor Totals	**423**	**554**	**520**	**50,142**

21.2.5 Major Reagents, Fuel and Electricity Costs

Table 21-26 summarizes the unit costs for the major Project consumables (process reagents, diesel fuel and power). A more detailed list of the consumables for the Project is provided in Table 21-27.

Table 21-26: Cost Assumptions for Reagents and Power

Item	Unit	Cost Estimate	Comment
Diesel fuel	$ per gallon	1.77	Quote for off-road diesel delivered to site
Electricity	$ per kWhr	0.0554	Price rate quote
Lime	$ per st	130	OPEX for onsite production
Sodium Cyanide	$ per lb	3.00	Price quote delivered to site
Sodium Metabisulfite	$ per lb	0.62	Price quote delivered to site
Copper Sulfate	$ per lb	2.55	Price quote delivered to site

Reagent consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources where available, with an allowance for freight to site or from historical data from other projects.

Table 21-27: Life-of-Mine Reagent Costs by Process Area

Process Area	Reagent	Life-of-Mine ($000s)
Grinding	Lime	541
	Sodium Cyanide	1,399
	Copper Sulfate	21,965
Antimony Recovery	Lead Nitrate	5,906
	Aerophine 3418A	2,167
	Methyl Isobutyl Carbinol	622
	Sodium Cyanide	-
Antimony Cleaning	Sodium Cyanide	52
	3418A	9
	Lead Nitrate	8
	Flocculant	4
Gold Flotation	Flocculant	5,336
	Flocculant	400
	Copper Sulfate	8,514
	PAX	29,813
	3477	5,135
	MIBC	9,055
Pressure Oxidation	Hydrogen Peroxide	355
	Flocculant	-
	Limestone(1)	17,029
POX Discharge Cooling & Neutralization	Lime(1)	45,143
	Limestone(1)	10,896
POX Leach - CIP Circuit	Sodium Cyanide	119,853
	Lime for detox	127
	Carbon	18,166
	Sodium Metabisulfite	966
	Copper Sulfate	-
Tailing/Oxide Leach- CIP	Sodium Cyanide	5,699
	Lime - pH control for leach(1)	11,463
	Carbon	5,563
	Lime - for detox(1)	573
	Sodium Metabisulfite	4,082
	Copper Sulfate	-
Carbon Handling & Refinery	Sodium Hydroxide	2,261
	Nitric Acid	17,686
Total LOM Reagent Cost		**350,785**

Note:
(1) Limestone and lime costs include limestone comminution and lime kiln operating costs.

Wear parts consumption (liners) and grinding media were estimated on a pound/ton basis. The consumption rate and unit costs were used to calculate the annual costs and cost per unit of production. These consumption rates and costs are shown in Table 21-28.

Table 21-28: Life-of-Mine Wear Steel Cost

Wear Steel Category	Applicable Equipment	Life-of-Mine Costs ($000s)
Liners	Primary Crusher	1,966
	Pebble Crusher	2,664
	SAG Mill	41,969
	Ball Mill	9,499
Grinding Media	SAG Mill	73,039
	Ball Mill	51,382
Total LOM Wear Steel Cost		**180,520**

An allowance was made to cover the cost of maintenance for the facilities and all items not specifically identified. The allowance made as a percent of the direct capital cost of equipment for each area; the rate used was 5%.

21.2.6 All-In Sustaining and All-In Costs

AISC is used to evaluate the various operational and non-operational costs of an operation in terms of cost per ton ($/st) and cost per troy ounce of gold produced ($/oz Au). Cash costs for mining, process, and G&A are adjusted by subtracting the revenue from by-product credits and adding the “off-site” costs, such as royalties, transportation and refining charges, to arrive at the total cash costs. Sustaining capital costs and non-revenue-based taxes (most notably property taxes) are added to arrive at AISC. AIC includes the foregoing, but also includes capital costs from pre-operation (initial CAPEX) and from post-operation (reclamation and closure costs).

The AISC and AIC for the SGP for both the first four years of operation and for the life-of-mine (**LOM**) are presented in Table 21-18. Total costs in each category are divided by the total tonnage of processed material or the total ounces produced to arrive at the values shown.

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22 ECONOMIC ANALYSIS

The economic analysis presented in this Report uses a financial model that estimates cash flows on an annual basis for the life of the Project at the level of detail appropriate to the feasibility level of engineering and design. Annual cash flow projections are estimated over the LOM based on the CAPEX, OPEX, sales revenue and other cost estimates outlined in Section 21. CAPEX is estimated in four categories: initial, sustaining, closure and reclamation, and working, and are distributed in accordance with the estimated year of expenditure. OPEX estimates include labor, reagents, maintenance, supplies, services, and electrical power for each year. The sales revenue is based on payable metals contained in doré bullion and antimony concentrate produced by the ore processing plant. Other costs, such as royalties, taxes, and depreciation are estimated in accordance with the present stage of the Project.

The financial model results are presented in terms of Net Present Value (**NPV**), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (**IRR**) for the Project. Annual cash flow projections are estimated over the life-of-mine (**LOM**) based on the estimates of capital expenditures and production cost and sales revenue. The estimates of CAPEX and OPEX have been developed specifically for this Project, as presented in Section 21.

22.1 ASSUMPTIONS

Assumptions that were used to estimate the CAPEX and OPEX are presented in Section 21. Specific assumptions used in the construction of the financial model are provided below.

- A discount rate of 5% is applied to NPV calculations (**$NPV_{5\%}$**).
- Funding for the Project is assumed to be 100% equity funding with no financing costs except leasing of major mining equipment since this equipment would almost certainly be lease purchased.
- Revenue for doré and antimony concentrates is claimed in the same year as it is produced.
- Costs incurred prior to the start of construction are not included in the model and are considered “sunk costs”, except for tax purposes, where the aggregate expenditures accumulated prior to the construction start date are available to offset taxes.
- A 15-day delay in revenue from sales and a 15-day delay in payment of accounts payable are used in the formulation of working capital, which is recaptured at the end of mine life.
- An allowance of 5% is included in the financial model for salvage value of selected capital equipment, excluding buildings and tanks, which are included in the reclamation costs.
- Depreciation is calculated using the Modified Accelerated Cost Recovery System (**MACRS**) method in accordance with current U.S. Internal Revenue Service (**IRS**) regulations.
- Depletion is estimated for the financial model using the percentage method; a rate of 15% is used for gold and silver and 22% is used for antimony.

22.2 REVENUE

Revenue for the financial model is based on the grade and tonnage of mill feed from the mine plan (Table 22.1), using the plant recovery for the specific mineralization type to yield metal production figures (Table 22.2). The appropriate refinery or smelter treatment terms (Table 22.3) are applied to the payable metals (Table 22.4) using the metal prices presented in Table 22.5.

Table 22.1: Life of Mine Contained Metal by Deposit

Deposit	Ore Type	Ore Tons (kst)	Contained Metal Grade			Contained Metal Quantity		
			Gold (oz/st)	Silver (oz/st)	Antimony (%)	Gold (oz)	Silver (oz)	Antimony (klb)
Yellow Pine	High Sb	11,279	0.060	0.137	0.460	671,143	1,542,535	103,758
	Low Sb	41,463	0.049	0.045	0.009	2,047,125	1,880,672	7,859
Hangar Flats	High Sb	3,411	0.056	0.141	0.369	191,093	482,532	25,148
	Low Sb	5,696	0.039	0.048	0.018	223,364	273,486	2,104
West End	Oxide	5,235	0.016	0.025	-	82,506	133,256	-
	Mixed	28,483	0.030	0.043	-	854,621	1,236,261	-
	Low Sb	16,801	0.039	0.038	-	649,429	634,716	-
Historical Tailings	High Sb	2,962	0.034	0.084	0.166	100,011	247,418	9,817
Totals / Averages		**115,330**	**0.034**	**0.045**	**0.064**	**4,819,291**	**6,430,876**	**148,686**

Table 22.2: Recovered Metal Production

Deposit	Doré Bullion		Antimony Concentrate		
	Gold (koz)	Silver (koz)	Antimony (klb)	Gold (koz)	Silver (koz)
Yellow Pine	2,453	11	92,065	17	573
Hangar Flats	364	1	20,822	4	255
West End	1,333	839	0	0	0
Historical Tailings[1]	68	0	2,454	1	31
Totals Production	**4,217**	**852**	**115,342**	**21**	**858**

Annual metal production by deposit is illustrated on Figure 22.1.

Figure 22.1: Annual Metal Production by Deposit

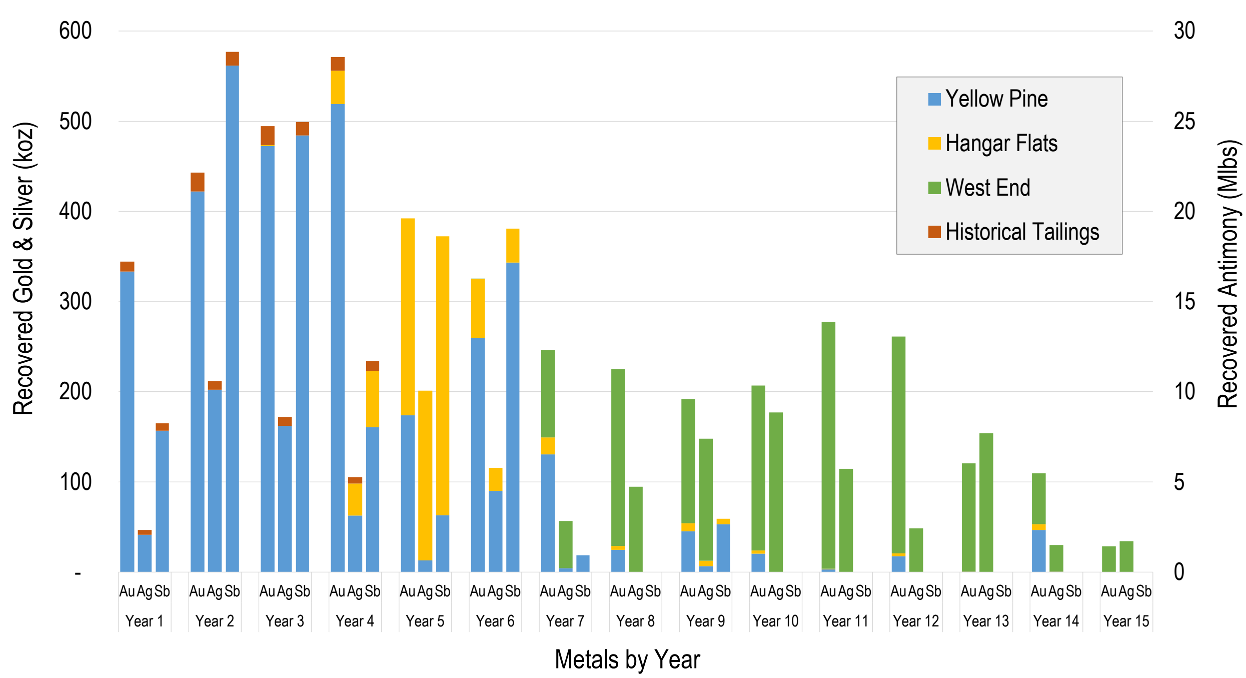


Table 22.3: Smelter Treatment Factors

Gold and Silver Bullion	
Gold Payability	99.5%
Silver Payability	98.0%
Refining Charge – Au (per troy ounce)	$1.00
Transportation Charge – Au (per troy ounce)	$1.15
Refining Charge – Ag (per troy ounce)	$0.50
Transportation Charge – Ag (per troy ounce)	$1.15
Antimony Concentrate	
Payable Antimony (%)	68%
Gold Payability (approximate)	
<5.0 g/t	0%
5.0 to <8.5 g/t	15-20%
8.5 to <10.0 g/t	20-25%
≥10.0 g/t	25%
Silver Payability (approximate)	
<300 g/t	0%
300 to <700 g/t	40-50%
≥700 g/t	50%
Transportation to Asia (per wet ton)	$156

Table 22.4: Payable Metals Production

Product	Gold (koz)	Silver (koz)	Antimony (klb)
Doré Bullion	4,196	835	-
Antimony Concentrate	4	134	78,433
Total Payable Metals	**4,200**	**968**	**78,433**

Table 22.5: Metal Price Cases

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(2) ($/lb)	
Case A	1,350	16.00	3.50	Lower bound case defined by the approximate 5-year trailing average gold price and consistent with the gold price used in the PFS (M3, 2014).
Case B (Base Case)	1,600	20.00	3.50	Base case derived from the weighted average of the 3-year trailing gold price (60%) and the 2-year gold futures price (40%).
Case C	1,850	24.00	3.50	Case corresponds to the approximate spot gold price at the effective date of this report.
Case D	2,100	28.00	3.50	Case corresponds with a gold price at approximately the peak 2020 spot price.
Case E	2,350	32.00	3.50	Upper bound case provides investors with insight into the revenues generated by the Project at a sustained elevated long-term gold price.

Notes:
(1) *The base case silver price was set at a gold-silver ratio ($/oz:$/oz) of 80:1 or $20/oz. The base case price was then varied similar to the way the gold price was varied (in this case by $4/oz Ag versus $250/oz Au) for the other cases.*
(2) *Antimony prices were assumed to be constant at $3.50/lb for all cases as antimony does not historically vary proportional to the gold and silver prices and is not expected to do so in the future. The $3.50/lb price was derived from a market study undertaken by an independent expert in antimony markets.*

22.3 CAPITAL COSTS

The details of the CAPEX estimate for the Project are summarized below and are presented in more detail in Section 21. For purposes of the financial model, CAPEX is broken into four categories: initial capital, sustaining capital, closure and reclamation capital, and working capital. Table 22.6 presents a summary of the initial, sustaining and closure and reclamation capital costs.

Table 22.6: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Closure CAPEX ($000s)(1)	Total CAPEX ($000s)
Direct Costs	Mine Costs	84,019	118,968	-	202,987
	Processing Plant	433,464	49,041	-	482,505
	On-Site Infrastructure	190,910	83,892	-	274,802
	Off-Site Infrastructure	115,940	-	-	115,940
Indirect Costs		232,684	-	-	232,684
Owner's Costs, First Fills, & Light Vehicles		38,351	-	-	38,351
Offsite Environmental Mitigation Costs		14,397	-	-	14,397
Onsite Mitigation, Monitoring, and Closure Costs		3,474	23,484	98,052	125,010
Total CAPEX without Contingency		**1,113,239**	**275,385**	**98,052**	**1,486,677**
Contingency		149,708	20,354	1,244	171,306
Total CAPEX with Contingency		**1,262,948**	**295,739**	**99,296**	**1,657,982**

Note:
(1) Closure assumes self-performed closure costs, which will differ for those assumed for financial assurance calculations required by regulators.

22.3.1 Initial Capital

The total initial CAPEX carried in the financial model for new construction and pre-production mine development is expended over a 3-year period. The initial CAPEX includes direct and indirect capital costs, owner's costs and contingency. The initial CAPEX would be expended in the years before production and a small amount carried over into the first production year.

22.3.2 Sustaining Capital

A schedule of CAPEX incurred during the production period was estimated and included in the financial analysis under the category of sustaining capital. The LOM sustaining capital is shown in Table 22.6. This capital will be expended over a 15-year period.

22.3.3 Reclamation and Closure

Reclamation and closure costs were estimated as shown in Table 22.6. The estimated costs do not include the revenue from the gold to be recovered from Historical Tailings as part of the Project legacy clean-up, nor does it include savings incurred from using the 7.3 million tons of spent heap leach ore in TSF construction, which is material that would otherwise have had to be obtained from other sources at additional cost.

22.3.4 Working Capital

A 15-day delay of receipt of revenue from sales is used for accounts receivable. A delay of payment for accounts payable of 15 days is also incorporated into the financial model. Working capital is estimated to be $7.5 million before production and an additional $18 million immediately after commencement of production but prior to receipt of revenue. Working capital also includes an allowance for capital tied up in parts inventory prior to its use. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.4 OPERATING COSTS

The average cash operating cost per short ton (st) of processed material before by-product credits, royalties, refining and transportation charges over the LOM and during the first four years of operations are summarized in Table 22.7. These cash costs include mine operations, process plant operations, and general and administrative costs (**G&A**). By-product revenue from silver and antimony can be "credited" as a deduction to the operating costs. The average cash operating cost per ton of processed material after by-product credits but before royalties, refining and transportation charges over the LOM and during the first four years of operations are also presented in Table 22.7.

Table 22.7: Operating Cost Summary

Cash Operating Cost Estimate	Years 1-4 Average		LOM Average		
	$/st milled	$/oz Au	$/st mined	$/st milled	$/oz Au
Mining OPEX(1)	9.71	156	2.37	8.22	205
Processing OPEX	13.13	211	-	12.76	318
General & Administrative OPEX	3.54	57	-	3.43	85
Cash Costs Before By-Product Credits(2)	**26.38**	**424**	**-**	**24.41**	**608**
By-Product Credits	(5.99)	(96)	-	(2.81)	(70)
Cash Costs After By-Product Credits(3)	**20.40**	**328**	**-**	**21.60**	**538**

Notes:
(1) Mining OPEX excludes capitalized stripping.
(2) Cash costs shown in this table are before royalties, refining, and transportation charges; cash costs that include these costs are presented in Table 22.8.
(3) By-product credits accrue from silver and antimony revenue.

22.5 ROYALTIES, DEPRECIATION AND DEPLETION

There is a 1.7% royalty that applies to gold revenue, as detailed in Section 4. The LOM reduction in Net Operating Income is estimated to be $114 million.

Depreciation is calculated using the MACRS method starting with the first year of production. The initial capital and sustaining capital used a 7-year life. The last year of production is the catch-up year for the assets that are not fully depreciated at that time.

The percentage depletion method was used in the evaluation. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. A rate of 15% is used for gold and silver and a rate of 22% is used for antimony.

22.6 TAXATION

22.6.1 Income Tax

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation and depletion. Deduction for depletion is used in the calculation of State income tax, but no deduction is taken for the federal income taxes paid. The combined effective tax rate was calculated as follows:

$$\text{Combined Effective Tax Rate} = \text{State Rate} + \text{Federal Rate} \times (100\% - \text{State Rate})$$

$$= 6.9\% + 21\% \times (100\% - 6.9\%) = 26.45\%$$

22.6.2 Idaho Mine License Tax

This is a tax for the privilege of mining or receiving royalties from mining operations. The tax rate is 1% of the value of ores mined or extracted and royalties received. The basis is the taxable income that is defined by the IRS.

22.7 TOTAL PRODUCTION COSTS

A detailed breakdown of the various measures of cash cost over the life of the mine are shown in Table 22.8. The costs are presented in $/st mined, $/st milled, and in $/oz Au. The table provides the cash costs before and after by-product credits; the total cash costs, which include royalties, refining and transportation charges; and All-In Sustaining Costs (**AISC**) that includes the Sustaining CAPEX, salvage, and property taxes for both the LOM and initial four years of operation. The All in Costs (**AIC**), that includes non-sustaining capital, is included for the LOM.

Table 22.8: Total Production Cost Summary

Total Production Cost Item	Years 1-4		LOM	
	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	9.71	156	8.22	205
Processing	13.13	211	12.76	318
G&A	3.54	57	3.43	85
Cash Costs Before By-Product Credits	**26.38**	**424**	**24.41**	**608**
By-Product Credits	(5.99)	(96)	(2.81)	(70)
Cash Costs After By-Product Credits	**20.40**	**328**	**21.60**	**538**
Royalties	1.69	27	1.09	27
Refining and Transportation	0.46	7	0.24	6
Total Cash Costs	**22.54**	**362**	**22.94**	**571**
Sustaining CAPEX	4.64	75	2.83	70
Salvage	-	-	(0.26)	(6)
Property Taxes	0.05	1	0.04	1
All-In Sustaining Costs	**27.23**	**438**	**25.54**	**636**
Reclamation and Closure(1)	-	-	0.95	24
Initial (non-sustaining) CAPEX(2)	-	-	11.65	290
All-In Costs	**-**	**-**	**38.14**	**950**

Notes:
(1) Defined as non-sustaining reclamation and closure costs in the post-operations period.
(2) Initial Capital includes capitalized preproduction.

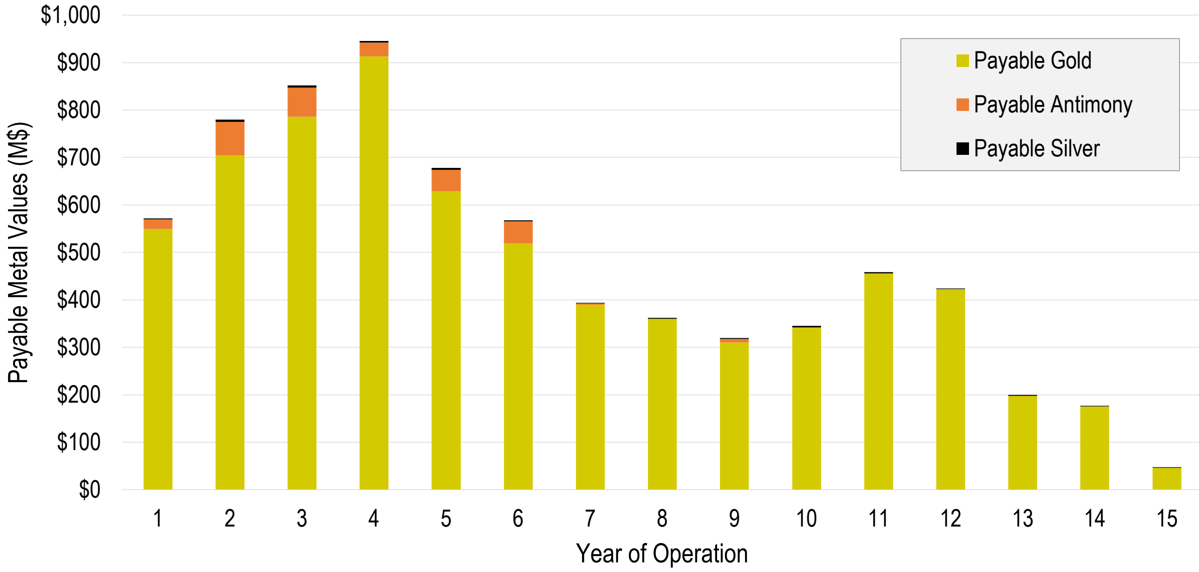
22.8 FINANCIAL MODEL RESULTS

The financial model results are presented in terms of NPV, IRR, and payback period in years for recovery of the capital expenditures. These economic indicators are presented on both pre-tax and after-tax bases. The NPV is presented both undiscounted (**$NPV_{0\%}$**) and at a 5% discount rate ($NPV_{5\%}$), as shown in Table 22.9. The primary metric for comparison of the cases is the after-tax net present value at a 5% discount rate (**$ATNPV_{5\%}$**).

Table 22.9: Financial Model Pre-Tax and After-Tax Indicators by Case

Parameter	Unit	Pre-tax Results	After-tax Results
Case A ($1,350/oz Au, $16.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	1,637	1,434
$NPV_{5\%}$	M$	896	771
Annual Average EBITDA	M$	223	-
Annual Average After-Tax Free Cash Flow	M$	-	189
IRR	%	17.3	16.2
Payback Period	Production Years	3.4	3.4
Case B ($1,600/oz Au, $20.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	2,667	2,232
$NPV_{5\%}$	M$	1,599	1,320
Annual Average EBITDA	M$	292	-
Annual Average After-Tax Free Cash Flow	M$	-	242
IRR	%	24.3	22.3
Payback Period	Production Years	2.9	2.9
Case C ($1,850/oz Au, $24.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	3,697	3,026
$NPV_{5\%}$	M$	2,301	1,864
Annual Average EBITDA	M$	360	-
Annual Average After-Tax Free Cash Flow	M$	-	295
IRR	%	30.4	27.7
Payback Period	Production Years	2.4	2.5
Case D ($2,100/oz Au, $28.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	4,726	3,815
$NPV_{5\%}$	M$	3,002	2,404
Annual Average EBITDA	M$	429	-
Annual Average After-Tax Free Cash Flow	M$	-	348
IRR	%	35.9	32.4
Payback Period	Production Years	2.2	2.2
Case E ($2,350/oz Au, $32.00/oz Ag, $3.50/lb Sb)			
$NPV_{0\%}$	M$	5,755	4,603
$NPV_{5\%}$	M$	3,704	2,943
Annual Average EBITDA	M$	498	-
Annual Average After-Tax Free Cash Flow	M$	-	400
IRR	%	41.0	36.9
Payback Period	Production Years	1.9	1.9

Figure 22.2: Payable Metal Value by Year for Case B in Millions of Dollars



The undiscounted cash flows for Case B, the base case, are depicted on Figure 22.3.

Figure 22.3: Undiscounted After-Tax Cash Flow for Case B

22.9 MINE LIFE

Using the current Mineral Reserve and the nominal design throughput of 22,050 stpd, the mine plan projects a 14.3-year production life. Construction is projected to require a three-year period after the permits are obtained and prior to the start of commercial operations. Closure is projected to take at least 35 years post-production, with some reclamation work occurring concurrently with operations, and the bulk of the closure activities and costs incurred in the first 10 years after operations cease. Some closure activities and long-term monitoring are anticipated to continue well after the reclamation period is complete to ensure that the closure designs continue to protect the environment and are performing in accordance with the design parameters.

22.10 SENSITIVITY ANALYSIS

The sensitivity of the financial model was tested with respect to metal prices or gold grade, initial CAPEX, and OPEX for each case. The value of each parameter was raised and lowered 20% to evaluate the impact of such changes on the NPV at a 5% discount rate. The results for the pre-tax $NPV_{5\%}$ (**$PTNPV_{5\%}$**) and after-tax $NPV_{5\%}$ (**$ATNPV_{5\%}$**) are presented in Table 22.10. After-tax sensitivities with respect to $NPV_{0\%}$, $NPV_{5\%}$, IRR, and payback in production years for the base case are presented in Table 22.11.

Table 22.10: Pre-Tax and After-Tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	$NPV_{5\%}$ (M$)					
		-20% Variance		0% Variance		20% Variance	
		Pre Tax	After Tax	Pre Tax	After Tax	Pre Tax	After Tax
Case A	CAPEX	1157	980	896	771	635	560
	OPEX	1228	1023			564	507
	Metal Price or Grade	97	88			1695	1396
Case B	CAPEX	1,859	1,527	1,599	1,320	1,338	1,113
	OPEX	1,931	1,568			1,266	1,069
	Metal Price or Grade	659	581			2,538	2,047
Case C	CAPEX	2,561	2,068	2,301	1,864	2,040	1,658
	OPEX	2,634	2,109			1,968	1,616
	Metal Price or Grade	1221	1026			3,380	2,694
Case D	CAPEX	3,263	2,607	3,002	2,404	2,742	2,200
	OPEX	3,337	2,649			2,669	2,158
	Metal Price or Grade	1,783	1,462			4,222	3,341
Case E	CAPEX	3,965	3,146	3,704	2,943	3,444	2,739
	OPEX	4,039	3,188			3,370	2,698
	Metal Price or Grade	2,344	1,897			5,064	3,986

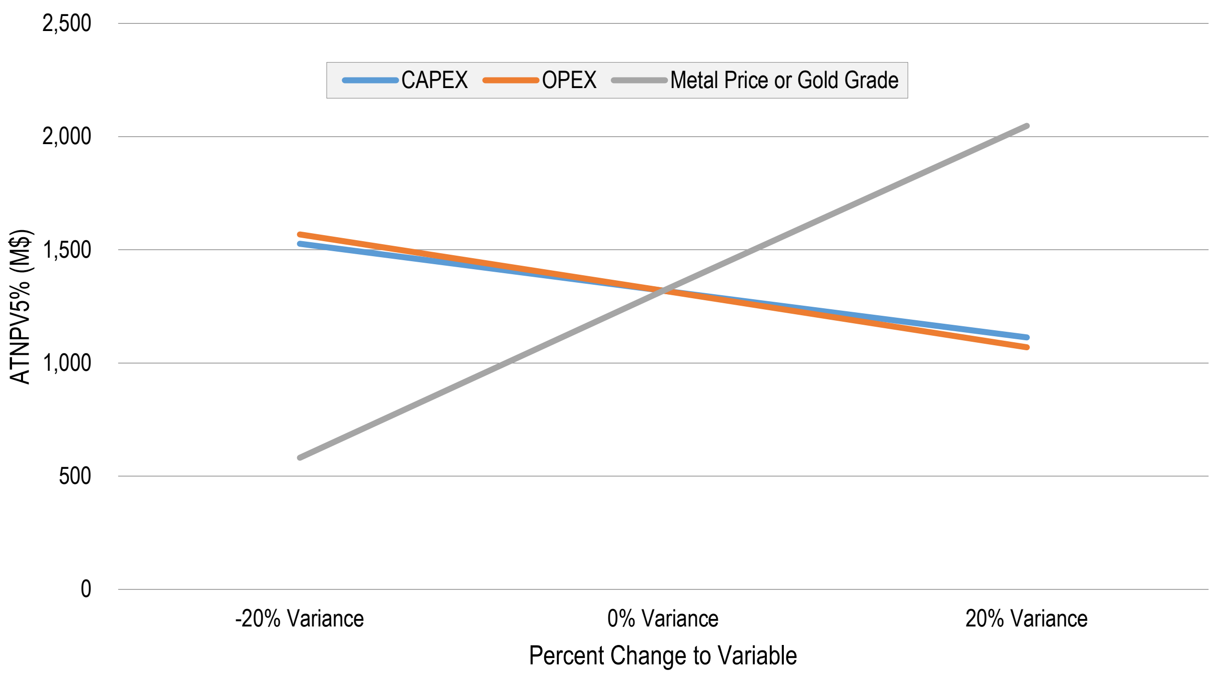
Table 22.11: Base Case After-Tax Sensitivity Analysis

Variance	$NPV_{0\%}$ (M$)	$NPV_{5\%}$ (M$)	IRR (%)	Payback (yrs)
Metal Prices or Gold Grade				
20%	3,289	2,047	29.4	2.3
10%	2,762	1,685	26.0	2.6
0%	**2,232**	**1,320**	**22.3**	**2.9**
-10%	1,701	954	18.3	3.2
-20%	1,164	581	13.7	3.7
Capital Cost				
20%	2,005	1,113	17.9	3.3
10%	2,119	1,216	20.0	3.1
0%	**2,232**	**1,320**	**22.3**	**2.9**
-10%	2,346	1,423	25.0	2.7
-20%	2,459	1,527	28.1	2.4

Variance	$NPV_{0\%}$ (M$)	$NPV_{5\%}$ (M$)	IRR (%)	Payback (yrs)
Operating Cost				
20%	1,850	1,069	19.8	3.1
10%	2,042	1,195	21.0	3.0
0%	**2,232**	**1,320**	**22.3**	**2.9**
-10%	2,422	1,444	23.5	2.8
-20%	2,610	1,568	24.6	2.7

The after-tax sensitivities for $NPV_{5\%}$ (Table 22.11) for Case B are illustrated on Figure 22.4.

Figure 22.4: Case B After-Tax $NPV_{5\%}$ Sensitivities



The $ATNPV_{5\%}$ of the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices and gold grades. For example, a 20% increase in gold price or gold grade leads raises the $ATNPV_{5\%}$ from $1,320 million to $2,047 million, a 55% increase. Similarly, a decrease of 20% in gold grade or gold price results in a 56% decrease in $ATNPV_{5\%}$.

All of the cases indicate that the Project is slightly more sensitive to changes in OPEX than it is to changes in CAPEX. For example, the change in $ATNPV_{5\%}$ for a 20% increase in CAPEX is -16%, whereas a 20% increase in OPEX causes a -19% change in $ATNPV_{5\%}$.

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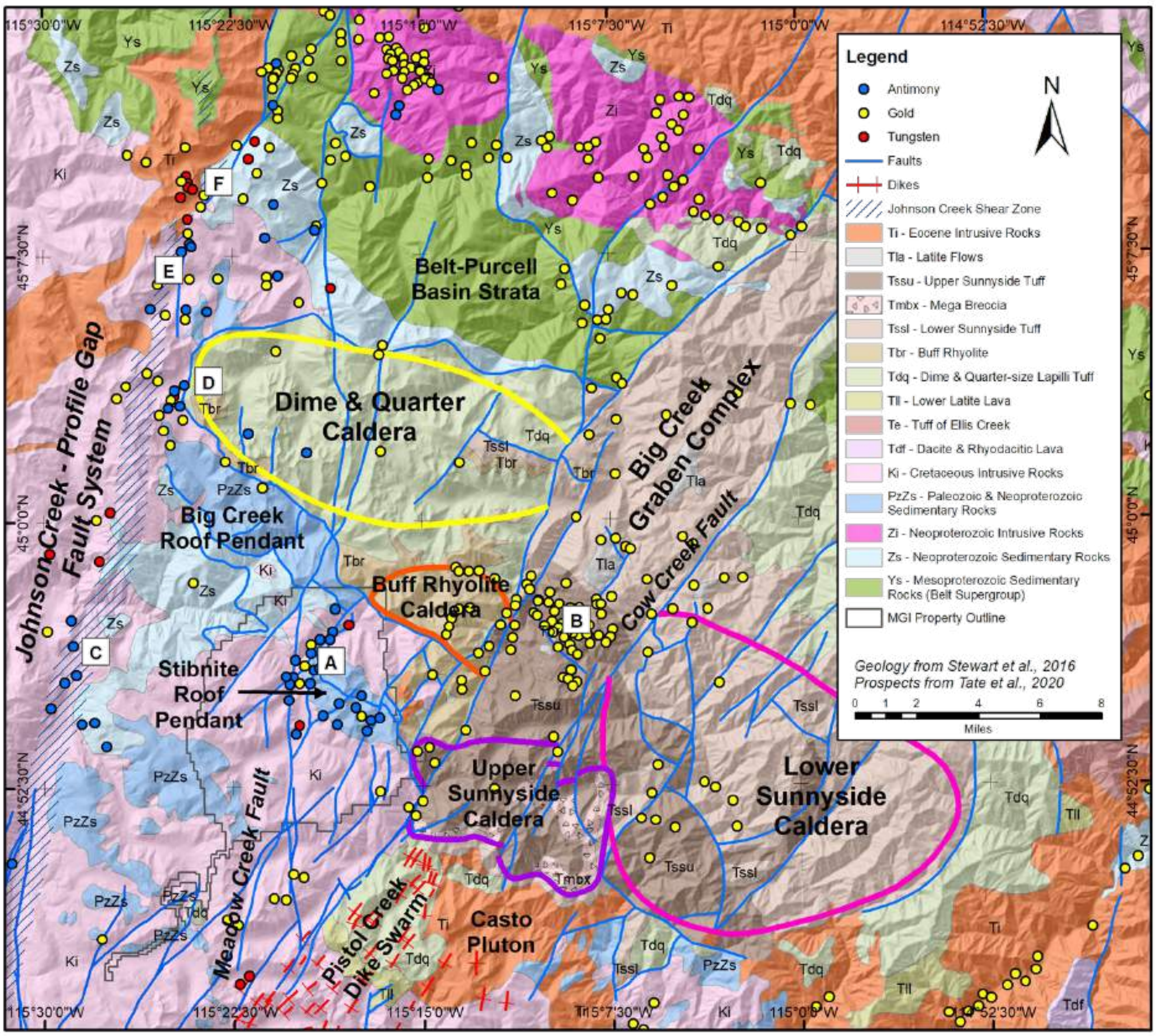
23 ADJACENT PROPERTIES

23.1 NEARBY PAST PRODUCERS AND MAJOR PROSPECTS

The Stibnite Gold Project is not impacted by adjacent properties. However, there are properties controlled by other parties to the east, west and north of the Project that have been past producers and continue to be considered major prospects. Figure 27-1 illustrates the location of these adjacent properties relative to Stibnite.

Significant past producing gold mines and major prospects from the Idaho Geological Survey Mines Database (2018 version) and Midas Gold files near Stibnite (A) include: Thunder Mountain (B); Golden Gate and Antimony Ridge (C); B&B and Red Mountain (D); Moscow and Ludwig (E); and, McRae and Independence (F).

Figure 27-1: Past Producing Mines and Major Prospects near Stibnite



The Thunder Mountain District (labeled B on map) had numerous placer mines and was the site of a major gold rush in the late 1880s and early 1990s. Later several of the larger lode mines in the area produced over 100,000 oz of Au. Recorded Sb production from Antimony Ridge (aka Babbitt Metal mine) south-southeast of the town of Yellow Pine (labeled C on map) includes ~40 tons mined in 1916-1917 and 400 tons mined from 1940-42 by the Bradley interests (Schrader and Ross, 1926; Shenon and Ross, 1936; La Heist, 1964). Small amounts of silver and gold were reported in some antimony ore (Thomson, 1919). Anaconda mapped and sampled the prospect in 1938 and reported high-grade antimony-gold mineralization in a series of parallel but discontinuous veins. In the 1950s-70s, the Oberbillig interests and lessees continued work including development of short adits and prospect pits and produced an undisclosed, but presumably small, amount of antimony from hand-cobbed stibnite veins. Amselco, Meridian, and TRV Minerals conducted extensive gold exploration in the 1980s-1990s outlining a large area of mineralized material containing gold and antimony. However, no NI 43-101-compliant mineral resources have been reported. Material from this prospect was mined in the 1960s-1970s and either shipped to out of state smelters or processed at the nearby Antimony Camp (Oberbillig) mill along the Johnson Creek flood plain. Former mill tailings indicate that several thousand tons have been processed; however, some tailings represent custom milling of ore from other deposits.

The Golden Gate prospect is located along a prominent ridge southeast of the town of Yellow Pine (labeled C on Figure 27-1) and approximately 9,000 tons of tungsten ore grading ~2 wt% WO_3 were mined from Golden Gate Hill in 1972 and 1980, although it is unclear if tungsten was recovered (Leonard, ca 1992).

Production of antimony, and possibly other metals such as mercury, from the former "B&B" underground and open pit mine near Profile Gap (located near D on Figure 27-1) probably did not exceed several hundreds of tons at an unknown grade (Leonard, 1965; Leonard et al., 1973).

Extensive exploration targeting gold were conducted in other areas by other operators during the 1980s-1990s to the north-northwest of Stibnite including drill campaigns at the Red Mountain (labelled D on Figure 27-1), Moscow (E), Ludwig (E), Independence (F) and McCrae (F) mines by Placer Dome, Freeport, Cambior, Amselco, St. Joe American Corporation, Kennecott, Coeur d'Alene Mines, Nerco Exploration, Freeport-McMoRan, Independence Mining Company, Meridian Gold, and others. Several of these former operators reported historical estimates of mineralized materials, but there are no current NI 43-101 compliant mineral resources reported for these prospects.

23.2 REFERENCES

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24 OTHER RELEVANT DATA AND INFORMATION

24.1 ANTIMONY INFORMATION

24.1.1 Introduction

The name “antimony” is derived from the Greek meaning “never found alone” illustrating its often-complex associations in nature. Antimony (**Sb**) is a silvery-white, shiny, soft, and brittle metal. The principal use of antimony today is as an oxide synergist in the flame-retardant chemical additive sector primarily in the form of antimony trioxide (**ATO**), antimony pentoxide (**APO**), and other forms. It is a semiconductor and has thermal conductivity lower than most metals. Due to its poor mechanical properties, pure antimony is only used in very small quantities; larger amounts are used for alloys and in antimony compounds.

24.1.2 History

World supplies of antimony have been dominated for over a century by one deposit; the Hsikwangshan deposit in Hunan, China, worked since the 16th Century and is still the world’s dominant source producing between 30,000 to 40,000 tonnes of contained antimony per annum (Confidential Private Report, 2018).

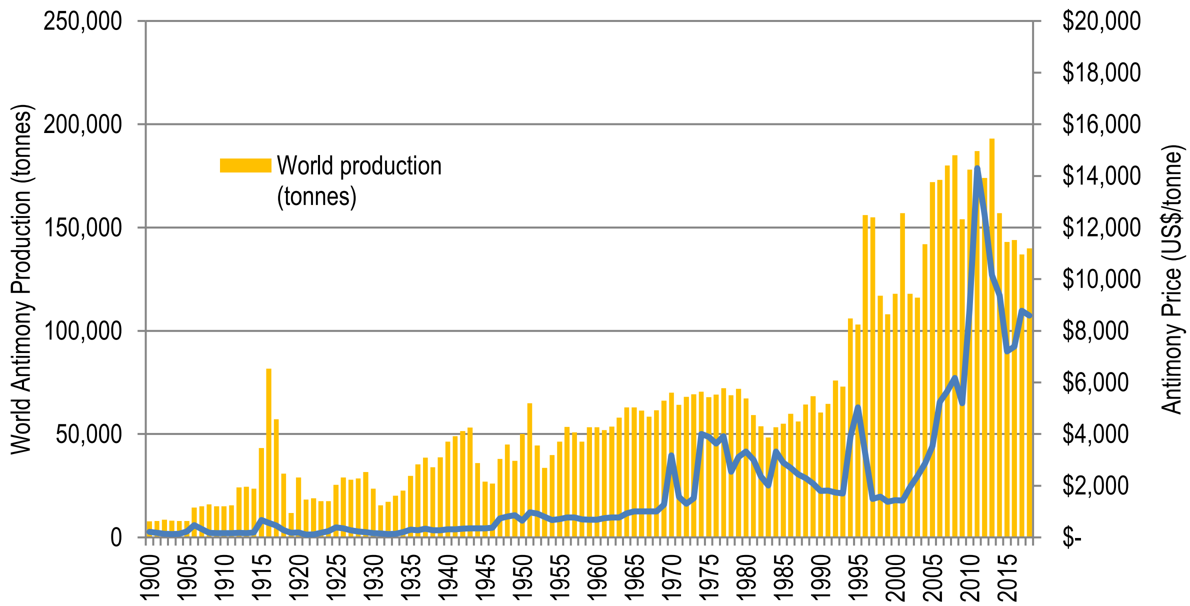
From 1897 to 1914, the average annual world production of antimony metal grew relatively steadily and rose rapidly in World War I due to its use in munitions. Peacetime demand declined and then jumped once more during World War II and the Korean War. During this period, the US government established assistance programs and set price supports to encourage US operations to produce antimony to supply war needs since Chinese production was unavailable. China dramatically increased its production in the late 1980s and 1990s to command 90% of production once more. Figure 24-1 illustrates antimony production and pricing since 1900. Production and consumption of antimony were weak in 2019 and 2020 due to the economic impacts of COVID-19 (Roskill, 2020), but are expected to rebound as business returns to normal.

24.1.3 Supply and Reserves

Global mine production declined from 2010-2017 and bottomed out in 2017, mostly tracking falling demand due to weak global economies. From 2017-2018 total primary global mine output averaged ~112,000 tpa with average byproduct production, primarily from lead smelters, of ~28,250 tpa. Secondary recycled output averaged 33,000 tpa. (Roskill, 2018; Confidential Report, 2018; USGS 2020). Mines and markets were severely impacted during 2019-2020 due to the Covid-19 crisis, with mines and processing facilities across China, the Russian Far East, and elsewhere shuttering for several months and some are still closed or operating at reduced output (Argus, 2020). China still has the world’s largest antimony resources and remains the leading primary producer in 2020, but declining reserve grades, market consolidation and increased environmental and regulatory controls across China have led to the closure of hundreds of smaller facilities and resulted in a significant decrease in output from over 80% of global production in 2010 to around 50% in 2020 (Confidential Private Report, 2019; Roskill 2020). Despite closure of many operations as of 2020, there are still roughly 71 antimony producers in China, of which Hsikwangshan Twinkling Star, Hunan Gold Corporation Limited, Guangxi China Tin Group, China Antimony Corporation, Guizhou Dongfeng Mining Group Co., Ltd. together contribute more than 80% of the country’s total production (Market Research, 2020). China has increased importation of antimony to counter its declining production and purchased interests in foreign operations in Bolivia, Canada, Tajikistan, and elsewhere. Despite new operations coming online, in the long-term supply is forecasted to be less than demand (Confidential Private Report, 2019).

Increasingly, antimony is being recovered from gold mining and processing operations as a by-product or co-product due to the lack of new discoveries of primary high-grade antimony deposits and the decline of antimony mine production from the Hunan area in China. Mine operations in Australia, Bolivia, Russia are moving in this direction, with processing facilities of various sizes and states of development either built or being developed in Russia, Oman, Vietnam, and Myanmar.

Figure 24-1: World Antimony Production and Price from 1900 – 2018



Source: Roskill, 2018; USGS, 2020

China still accounts for the majority of worldwide annual oxide production. Many countries have no primary production and import feedstock to facilities such as those in Belgium and France, the centers of European antimony oxide sector. China continues to be the world's largest producer of primary antimony metal and oxides (Table 24-1), but its share of world production has been declining, especially with increased development of Russian Far East and central Asian mines and facilities and government-driven cost, environmental, and trade pressures (Roskill, 2020; Confidential Report, 2018). In 2019, China produced 78,454 metric tonnes of antimony metal and 90,223 metric tonnes of antimony oxides (Metal News, 2020). China's share of global primary mine output is likely to fall from a current ~70% to less than 65% by 2030 (Confidential Report, 2018). Recent Chinese government actions to shutter illegal mines or require increased investment is having the effect of reducing production and increasing mining and smelting costs. Output from Russia and Tajikistan are steadily increasing to fill the gap left by China and causing global supply to rebound in 2018 and 2019. In 2018, Russian supply of antimony surpassed Tajikistan's output with Russian company Polyus, one of the world's leading gold producers, supplying by-product antimony from its Olimpiada mine equivalent to 15% of global antimony mine production. This operation has repeatedly curtailed antimony mine output and concentrate processing and sales to China due to shutdowns, plant issues, and a Covid-19 outbreak at its mine and plant in 2020 (Metals Bulletin, 2020a and 2020b).

Global reserves are estimated at 1.5 million tonnes (Table 24-1) which, at an estimated global production rate of approximately 0.2 million tonnes per year is estimated to be less than 10 years of production. Furthermore, while official Chinese statistics still report considerable reserves, independent estimates suggest that some of the mines associated with these reserves might be reaching economic exhaustion, particularly in the area of Lengshuijiang City, the center of antimony mining in China. Very few deposits have been explored or developed in recent years outside of those in Russia, Tajikistan, and the SGP in the US (Roskill, 2012, 2018; Confidential Report, 2018).

Table 24-1: Estimated Mine Production and Reserves of Antimony by Country

Country	Production (tonnes)							Reserves 2019[2]
	2013[1]	2014[1]	2015[1]	2016[1]	2017[1]	2018[2]	2019[2]	
Australia	3,062	3,680	3,926	5,004	4,294	2,170	2,000	140,000
Bolivia	5,053	4,186	3,843	2,669	2,844	3,110	3,000	310,000
Burma	7,000	4,234	5,777	2,780	3,060	2,640	3,000	-
Canada	177	5	1	-	-	-	-	-
China	152,104	140,389	120,732	107,535	101,000	89,600	100,000	480,000
Ecuador	-	-	-		579	50	50	-
Guatemala	159	-	-	25	-	25	25	-
Honduras	-	13	14	-	-	12	10	-
Iran	400	432	1,020	1,765	1,800	600	600	-
Kazakhstan	900	800	700	900	200	300	300	-
Kyrgyzstan	900	1,450	1,200	1,880	1,100	370	400	-
Laos	804	620	1,166	242	320	300	300	-
Mexico	294	266	90	166	240	260	300	18,000
Pakistan	89	127	114	21	15	28	30	26,000
Russia	6,520	6,400	7,420	6,620	6,120	30,000	30,000	350,000
South Africa	2,332	816	400	-	-	-	-	-
Tajikistan	7,307	7,000	6,800	12,700	12,500	15,200	16,000	50,000
Thailand	488	706	700	32				-
Turkey	4,512	3,013	1,917	2,700	4,750	2,400	3,000	100,000
United States	-	-	-	-	-	-	-	60,000
Vietnam	990	1,098	219	229	243	240	240	-
TOTALS	**193,091**	**175,235**	**156,039**	**145,268**	**139,065**	**147,305**	**159,255**	**1,534,000**

Notes: [1] British Geological Survey World Mineral Production (2018)
[2] USGS Mineral Commodity Summary, Antimony, 2020.

Chinese mines continue to produce and process the bulk of the world's primary antimony, with the majority of non-Chinese mine production being sold to China for primary processing. China appears unwilling (if not unable) to maintain its level of mine production given resource depletion, rising costs, environmental crackdowns, and resource conservation (Confidential Report, 2018). As a result, production in China is unlikely to increase over the next few years and could even fall in the face of government determination to limit environmental damage from smaller operations (Roskill, 2018, Confidential Report, 2018).

Russia and Tajikistan are the next largest producers of antimony after China, both ramping up production in recent years to fill the gap left by the drop in Chinese production. In 2018, Russian supply of antimony exceeded that of Tajikistan, but Russia has not been exporting antimony and is managing its trade in a similar manner to China, with strict government control.

In Tajikistan, Chinese interests have loaned significant money to the state-owned TALCO aluminum smelting complex, which is an aging and Soviet-era facility that has been processing ores from two antimony mining districts in the country with hopes of increasing production to roughly 10% of global output. However, Chinese interests will likely control future antimony output from operations given TALCO's debt load and proximity to Chinese industrial consumers (Financial Post, 2019).

Smaller sources of antimony supply outside China include Geopromining in Russia, Mandalay Resources in Australia, and operations in Bolivia, Mexico, Turkey, and Vietnam. The Consolidated Murchison mines in South Africa had been operating since the 1930s but closed their roasters in 2014 and completely shut down their antimony mining operations in 2015 for lack of reserves and high costs. Other shuttered operations include Beaver Brook in Canada, which is owned by Twinkling Star and is estimated to have two years of reserves left, and Hillgrove in New South Wales, Australia, which has had repeated unprofitable startups and problems with recovery.

24.1.4 Critical Minerals Status

The European Union, Great Britain, Canada the US, and Japan are among western powers to have compiled lists of strategic minerals in response to China's dominance of minerals used in electric vehicles, high-tech and defense applications. Antimony has been ranked consistently as No. 1 or No. 2 on their criticality lists for over a decade (BGS, 2012, 2015; EC, 2010, 2013; 2014, 2017, 2018; Hatayama and Tahara, 2014). In 2013, the U.S. Department of Defense (**DoD**) ranked antimony No. 2 in the list of strategic and non-fuel defense material shortfalls (US DoD, 2013) and continues to foresee shortfalls and has initiated efforts to locate sources other than Chinese or Russian through solicitations and contracts, including a Defense Logistics Agency (**DLA**) award to US Antimony Corporation to establish a new source of antimony trisulfide (used in primers) that meets the DLA stockpile specifications. Several US government studies have more recently warned that resource nationalism can become a tool for countries with raw materials supplies to use against countries that lack those supplies (OSTP, 2016; USGS, 2018).

The Obama Administration chartered a White House committee under the National Science and Technology Council on Critical and Strategic Minerals Supply Chains (CSMSC)[1]. The CSMSC was tasked with coordinating critical mineral policy development across the federal government. In 2010[2] and 2011[3], the Department of Energy issued its "Critical Mineral Strategy" reports that focused on managing supply risk and "taking steps to facilitate extraction, processing, and manufacturing here in the United States." The CSMSC also issued important reports in 2014[4] and in 2016[5] on identifying and monitoring Critical Minerals.

Also, on December 20, 2017, Executive Order 13817 A Federal Strategy to Ensure Secure and Reliable Supplies of Critical Minerals (EO) was issued. Several actions were required of Federal agencies to address critical minerals. Pursuant to the EO, the Secretary of the Interior, in coordination with the Secretary of Defense, and in consultation with the heads of other relevant executive departments and agencies, was tasked with developing and submitting a list of minerals defined as critical minerals to the Federal Register. The final list of critical minerals was published in the Federal Register on May 18, 2018 (83 FR 23295), citing 35 minerals or mineral material groups including antimony. In addition, and supporting this Federal Register list, the USGS released a comprehensive report on the 35 mineral commodities (USGS, 2018). In 2019, the Department of Commerce issued its "Federal Strategy to Secure Reliable Supplies of Critical Minerals"[6].

In 2019-2020, the US did not impose import tariffs on Chinese antimony, signaling its strategic importance. The importance of the antimony reserves and resources of the SGP was recognized by the September 10, 2020 announcement of the listing of the project to the High Priority Infrastructure Project (**HPIP**) Permitting Dashboard – the first mine development project in the U.S. to be listed. Information on HPIPs is published on the Council on Environmental Quality website and provides for enhanced coordination between federal agencies. The goals of

[1] https://obamawhitehouse.archives.gov/sites/default/files/microsites/ostp/NSTC/CSMSC%20Charter%202016-04-21%20signed.pdf

[2] https://energy.gov/sites/prod/files/edg/news/documents/criticalmaterialsstrategy.pdf

[3] https://energy.gov/sites/prod/files/DOE_CMS2011_FINAL_Full.pdf

[4] https://www.federalregister.gov/documents/2014/07/22/2014-17192/critical-and-strategic-materials-supply-chains

[5] https://www.whitehouse.gov/sites/whitehouse.gov/files/images/CSMSC%20Assessment%20of%20Critical%20Minerals%20Report%202016-03-16%20FINAL.pdf

[6] https://www.commerce.gov/news/reports/2019/06/federal-strategy-ensure-secure-and-reliable-supplies-critical-minerals

applying for the HPIP listing were to: ensure effective communications and timely permitting for the project; provide a domestic supply of critical minerals for national security; and, restore an abandoned and contaminated mine site.

24.1.5 Stockpiling

China's State Reserve Bureau (**SRB**) has been buying antimony metal in China and elsewhere (Confidential Report, 2014), but the details of these activities are not publicly available. The Fanya Minor Metals Exchange, a private rare metals exchange, was formed in 2011 and began warehousing minor metals in 2014. The Exchange went bankrupt in 2015 and 18,661 tonnes of antimony, equivalent to 13% of annual global production, was sold on August 25, 2019 under court order in one lot. Twinkling Star, the world's largest antimony producer, purchased the lot at approximately 19% below market (Reuters, 2019).

24.1.6 Smelting and Refining

Integration of mining and smelting or downstream processing has become rarer because of Chinese competition. As a result, smelter production capacity for antimony outside of China is negligible. Tri-Star Resources (Oman), a British corporation, and US Antimony Corp., a Montana-based US corporation, are in the early stages of attempting to integrate mining supply with processing facilities. Facilities in Belgium, France, Bolivia, and India are producing primary ATO and recycling antimony from lead-acid batteries. However, none use mined antimony except the Bolivian facility.

Tri-Star began construction of its smelter in Oman in 2015 and as of July 2020 it was operating at 50% capacity with full capacity output expected by March 2021, which would be approximately 12-15% of global ATO output. Oman Investment Authority (40% equity in the facility venture) filed for arbitration against Tri-Star and other parties in April 2020, clouding prospects for success of the smelter in attaining its goals.

Until 2018, the sole supplier of Mil Spec grade antimony trisulfide (primarily used in primers) to the U.S. military was China. U.S. Antimony operates a refining facility in Montana that produces ATO from material imported from Mexico where it is mined and processed into sodium antimonate. The company announced it is testing whether it can provide Military Specification (**Mil Spec**) products that meet DLA National Defense Stockpile requirements. U.S. Antimony reported on June 1, 2020 that it has added a flotation thickener circuit to its Puerto Blanco mill in Guanajuato, Mexico in an attempt to meet the DLA Mil Spec requirements.

24.1.7 Export Quotas

The Chinese central government has been developing production plans for strategic commodities including antimony Since the early 1990s (Morrison and Tang, 2012). In 2008, China's Ministry of Land and Resources issued a government directive with the stated goal of protecting and rationally utilizing China's valuable natural resources for the period 2008 to 2015 titled, "Guidelines for Development of National Mineral Resources 2008-2015". A similar directive was later issued and is still in force. This development plan designates antimony, along with rare earth elements, tungsten, and several other commodities as protected mineral commodities.

Exploration, production, processing, importing, and exporting of protected commodities is strictly controlled by the Chinese government. In December 2019 (Argus, 2019), the government announced the list of 11 antimony exporters that will be allowed to operate in 2020-2021, and it consists of four trading companies and seven producers; most of which have some level of government ownership and/or control.

The Chinese government takes an active role in limiting competition by the use of tariffs, export quotas, and occasionally bans exports completely. There has been extensive and continued litigation in the World Trade Organization between the European Union, the US, and China over its handling of and export restrictions on various commodities including antimony (Roskill, 2016). China has imposed export quotas for antimony and antimony products since 2009. China has not hesitated to exercise its control of such commodities as rare earth elements and antimony such as in 2010 when China cut off rare earths and antimony supplies to Japan in a diplomatic spat over fishing rights (New York Times, 2010). Between 2014 and 2017 (the latest date the data is available) Chinese imports of antimony have increased nearly 30% (Confidential Report, 2018).

24.1.8 Primary Antimony Uses

Metallurgical antimony (lead-antimony alloys) are used in lead-acid storage batteries for backup power and transportation and account for more than two-thirds of the use of metallurgical antimony, much of which is obtained from secondary recycling (USGS, 2018). Very high purity antimony metal is used with other materials as sputtering targets in electronics and semiconductor manufacturing for silicon wafers, for making infrared detectors, diodes, and other electronic components including thermocouple switches, capacitors, and in solder. Babbitt metal (an antimony alloy) bearings are used in engines to support moving mechanical parts and protect them from frictional degradation. Typical shipbuilding and marine users specify antimony-alloys for use in bearings and bushings for submerged propellers and turbines. Antimony is also used in nuclear reactors as startup neutron sources and in various neutron generating devices such as portable x-ray fluorescence spectrometers, down-hole probes used in the oil industry, and pipeline inspection equipment. Anti-friction, self-lubricating graphite bearings are impregnated with antimony to increase heat tolerance and are used in many “green” energy systems (wind generator bearings and hydroelectric dam turbines). “Antimony black” is finely ground metallic antimony used in bronzing in castings (Miller, 1973).

ATO and other antimony oxide compounds are utilized in large quantities in the manufacture of plastic housing for electronics, wire sheathing, wire insulation, and motherboards due its flame-retardant properties. ATO is used as a flame-retardant component in adhesives, paints, papers, plastics, and sealants and as a fire-retardant backing on rubber and textile upholstery, typically with bromine- or chlorine-based halogenated compounds (European Flame Retardants Association, 2006). Major markets for flame retardants include electronics, plastics, and fabrics used in making children’s clothing, aircraft and automobile seat covers, and bedding. Pure antimony sulfide will combust in the presence of oxygen and it is a primary ingredient in the manufacture of ammunition primers, detonators, smoke-generating munitions, and tracers. The rubber industry uses antimony as a vulcanizing agent (Miller, 1973; Gibson, 1998). Antimony is also used in ceramics and glassmaking; for example, with suitable stabilizers and coloring additives, antimony trioxide glass can be made opaque to all visible light except long-wave infrared rays (Miller, 1973) an important property in modern energy-efficient windows and shortwave reflective glass applications. Because antimony tends to bond with many elements it is an excellent decolorizing agent in optical glass production for use in photocopiers, camera lenses, binoculars, and iPad screens. Antimony is used as a phosphorescent agent in fluorescent light bulbs and many light-emitting diode (LED) applications. Sodium antimonite (NaO_3Sb) is used as a flame retardant, as well as for removing bubbles from glass (United States Antimony Corp., 2016). Other uses include as a component in the striking surface of safety matches; it also provides the “glitter” effects in fireworks. One potential key future growth area could be in computer phase-change memory, which is projected to lead to gigahertz transfer speeds (30x faster than flash) (Visual Capitalist, 2012).

24.1.9 The US Perspective

Before the industrial revolution and World War I, the US produced and consumed only minor amounts of antimony. Since then, the US has met some, but very little, of its demand from domestic mine production and recycling. Spikes in consumption during World War I, World War II, the Korean War, the Vietnam War, and, to some extent, during the Iraq conflict occurred due to demand for munitions production. Imports began to climb rapidly in the early 1980s with increased consumption due to the use of ATO in plastics and industrial uses as a flame retardant. The gap between production (which is essentially nil) and consumption in the US continues to widen. However, imports fell to ~22,000 tonnes in 2012 as the result of the effects of rising prices, the global financial crisis, and substitution. According to the USGS (2020), there has been no domestic mine production in the US for many years and there is only one processing facility in North America, in Montana, producing minor amounts of antimony metal and oxide from imported feedstock. US dependence on imports is 100%, outside of secondary antimony recovered at lead smelters as antimonial lead for use in lead-acid batteries, which amounted to 14% of overall consumption in 2019 (USGS, 2020). Given concerns over national security, antimony was added to the National Defense Stockpile in December 2018.

The estimated US domestic distribution of primary antimony consumption was as follows: metal products, including flame retardants, 35%; antimonial lead (for batteries) and ammunition, 29%; chemicals, 16%; Ceramics and glass, 12%; and others, 8% (USGS, 2018).

24.1.10 Outlook

Commodity market estimates for antimony demand vary widely from 1-2% growth per year (Roskill, 2020) up to 7.5% (Research and Markets, 2020, Syngene Research, 2020); the variance due in part to the opacity of the supply chain components and, to a lesser extent, lack of reliable reserve estimates in China, Russia, and former Soviet satellites. In addition, market forecasting is difficult due to market fluctuations and growth rates complicated by the economic and industrial impacts from Covid-19. Overall, antimony demand remains highly dependent on the level of consumption of antimony trioxide in the flame-retardants sector and antimony metal in lead-acid batteries; These two sectors account for nearly 80% of antimony consumption worldwide (USGS, 2018).

The rising demand for electric vehicles expected to drive the demand for metallurgical antimony alloys. Demand for antimony oxides use in flame retardants is also expected to increase. Increasing usage in fiberglass composite resins and their applications may contribute to a significant growth rate in global antimony market. Composites are rapidly replacing all conventional materials in many applications, such as aerospace, automobiles, construction, electrical, and electronics, due to their high strength, low cost, easy processability, and availability in various forms and shapes with excellent aesthetics; antimony provides heat resistance and fire retardant properties to the resins. Concern about the health effects of antimony in the fire retardants may affect its use, but there are few substitutes in many applications.

Based on the forecast for demand growth and China's falling production, it is estimated that there will be a supply deficit starting in 2020 rising to approximately 20,000 tonnes of additional annual primary mine production necessary through 2030 to meet worldwide demand (Confidential Report, 2019).

24.2 STIBNITE CONCENTRATE PROCESSING

The process design and flowsheet developed for this FS were based on producing antimony concentrate with the sale of the concentrate to an antimony smelter (suitable, currently operating antimony smelters are located in Asia or Oman). While a secondary processing facility could offer financial advantages over the base case, there are logistical and other complications that at current metal prices render this option unfeasible. If significant additional antimony mineral reserves are identified, or if antimony prices increase substantially, additional metallurgical testing, engineering, and cost estimating may be warranted. Were additional processing of antimony concentrates deemed warranted, the facility would likely be located off site; as a result, the current Project design of trucking concentrates offsite would not change.

For over 60 years the Sunshine hydrometallurgical plant in Kellogg, Idaho operated successfully by producing high purity base and precious metals, sodium antimonate, and metallic antimony ingots. The plant was built in 1942 as part of the U.S. critical metals program and recovered antimony from the tetrahedrite ores mined in Idaho's Silver Valley, but the plant closed in 2001 (Masters, 2007). The current owners of the plant have been utilizing the facility for refining precious metal concentrates, but we understand the owners are considering rehabilitating and restarting the hydrometallurgical plant (that can treat antimony-bearing tetrahedrite ores and potentially SGP antimony concentrates) for currently operating and potential future Silver Valley area mines.

24.3 PYRITE CONCENTRATE SALES

A preliminary market study for gold concentrate sales was completed by an independent leading industry participant. The participant's name has been withheld for confidentiality. In the study, the assumption was that the gold flotation concentrate would be shipped offsite to a regional processing facility located in Nevada where several autoclave and roaster plants are located. The direct sale of gold concentrate is not included in the economic cases presented in this report but rather, it is an opportunity for the project that would:

- Simplify the mineral processing done on-site by eliminating the POX and potentially eliminating cyanide leach circuits;
- Potentially eliminate the use of cyanide on-site; and
- Significantly decrease capital costs.

However, these benefits would be offset by reduced payability and significant transportation costs. In addition, there would less gold produced and loss of revenue due to the inability to produce gold from oxide ores present in all three deposits. It also is unlikely and contrary to industry practice for toll operations to agree to life-of-mine concentrate sales contracts covering the duration of an operation the size of the SGP, leaving the operation vulnerable to disruptions by its concentrate processor.

Treatment facilities in Nevada and elsewhere are capable of processing gold concentrate that could be produced at the SGP. This option would only require a milling and concentration circuit on-site and would eliminate all downstream processing facilities such as the POX plant, oxygen plant, cyanide-leaching facilities, cyanide destruction plant, and other associated operations. Significant CAPEX savings on the order of $200 million to $250 million would be possible. The elimination of cyanide use on-site may reduce the complexity of the tailings storage facility liner system design and eliminate the need and complexity for some permits (e.g. IDEQ Cyanidation Permit).

On May 9, 2018, Barrick Gold, which owns and operates (through the Nevada Gold Mines joint venture with Newmont) several roasters and autoclaves in Nevada, was granted a right of first refusal regardiing purchase of gold concentrates as part of a financing arrangement were such concentrates to be shipped off-site. If Barrick maintains a minimum of 10% ownership in Midas Gold, Barrick will maintain its right of first refusal regarding purchase of gold concentrates were Midas Gold to ship such off-site. As of August 26, 2020, Barrick owns ~11% of the issued and outstanding shares of the Corporation, and the right of first refusal is still in force at the time of this Technical Report.

24.4 PROJECT EXECUTION PLAN

24.4.1 Description

The Project Execution Plan (**PEP**) describes, at a high level, how the FS design presented in this document would be carried out. This plan contains an overall description of what the main work focuses are, Project organization, the estimated schedule, and where important aspects of the design would be carried out. Figure 24-2 provides a preliminary organizational chart based on this PEP and Figure 24-3 presents the Project schedule.

The PEP assumes an integrated strategy for engineering, procurement, and construction management (**EPCM**). The primary objective of the execution methodology is to deliver the Project at the lowest capital cost, on schedule, and consistent with the Project standards for quality, safety, and environmental compliance.

Figure 24-2: Project Organization Block Diagram

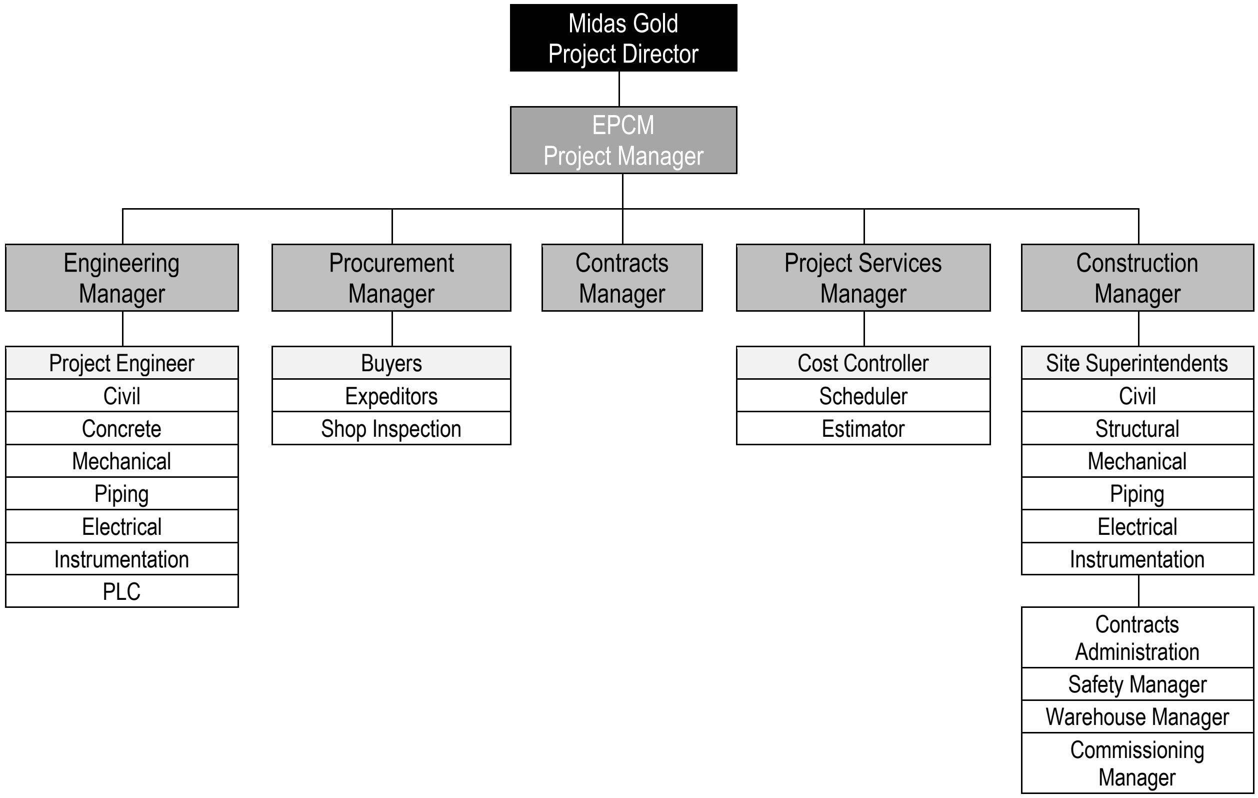
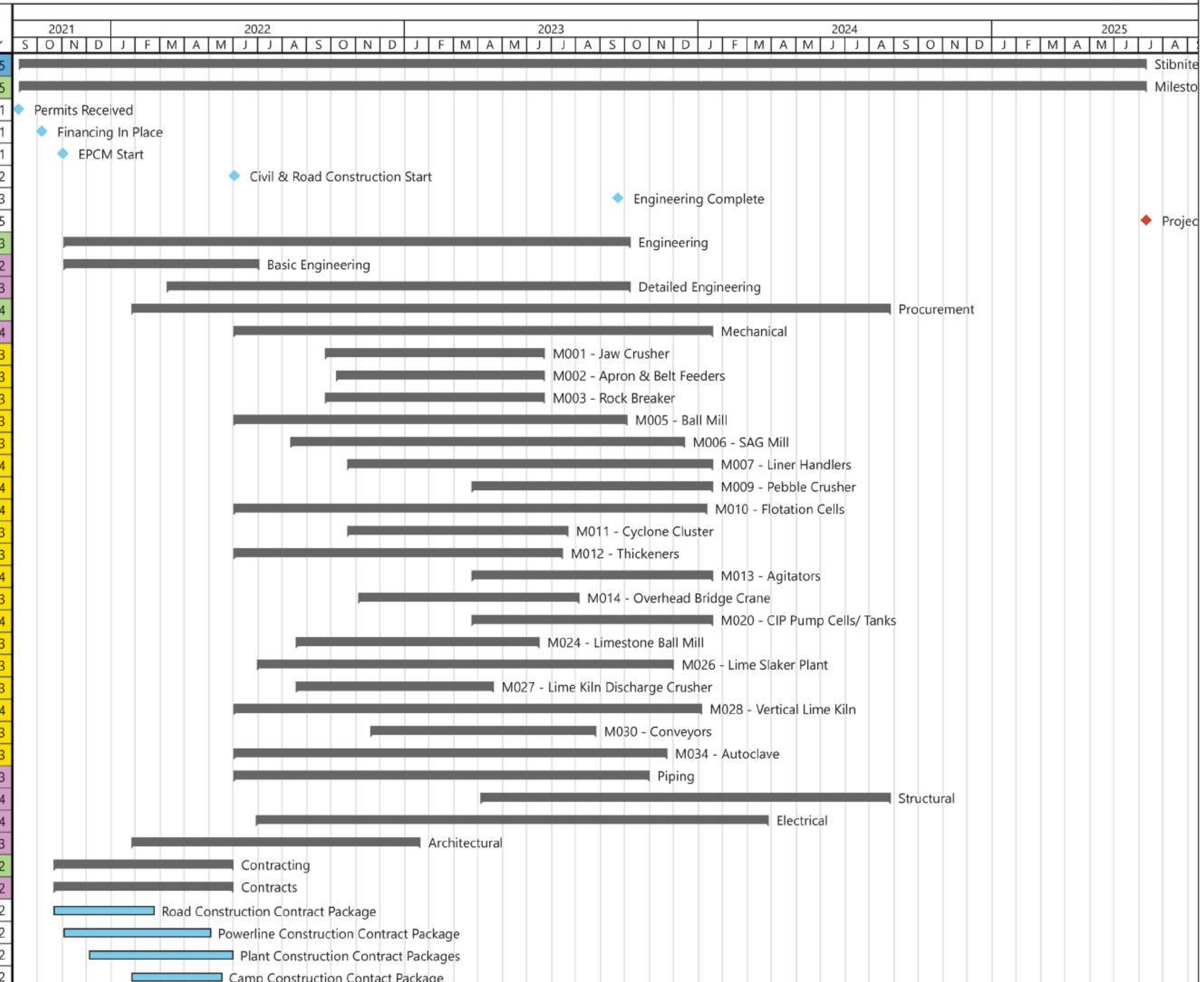


Figure 24-3: Stibnite Gold Project Summary Schedule

Stibnite Gold - EPCM Project Schedule for FS Update 2021 — M3-170045

Name	Start	Finish
Stibnite Gold - EPCM Project Schedule for FS...	Sep-07-21	Jul-10-25
Milestones	Sep-07-21	Jul-10-25
Permits Received	Sep-07-21	Sep-07-21
Financing In Place	Oct-05-21	Oct-05-21
EPCM Start	Nov-02-21	Nov-02-21
Civil & Road Construction Start	Jun-01-22	Jun-01-22
Engineering Complete	Sep-22-23	Sep-22-23
Project Complete	Jul-10-25	Jul-10-25
Engineering	Nov-02-21	Oct-06-23
Basic Engineering	Nov-02-21	Jul-01-22
Detailed Engineering	Mar-08-22	Oct-06-23
Procurement	Jan-25-22	Aug-26-24
Mechanical	May-31-22	Jan-18-24
M001 - Jaw Crusher	Sep-22-22	Jun-22-23
M002 - Apron & Belt Feeders	Oct-06-22	Jun-22-23
M003 - Rock Breaker	Sep-22-22	Jun-22-23
M005 - Ball Mill	May-31-22	Oct-02-23
M006 - SAG Mill	Aug-10-22	Dec-14-23
M007 - Liner Handlers	Oct-20-22	Jan-18-24
M009 - Pebble Crusher	Mar-23-23	Jan-18-24
M010 - Flotation Cells	May-31-22	Jan-11-24
M011 - Cyclone Cluster	Oct-20-22	Jul-21-23
M012 - Thickeners	May-31-22	Jul-14-23
M013 - Agitators	Mar-23-23	Jan-18-24
M014 - Overhead Bridge Crane	Nov-03-22	Aug-04-23
M020 - CIP Pump Cells/ Tanks	Mar-23-23	Jan-18-24
M024 - Limestone Ball Mill	Aug-17-22	Jun-15-23
M026 - Lime Slaker Plant	Jun-28-22	Nov-30-23
M027 - Lime Kiln Discharge Crusher	Aug-17-22	Apr-19-23
M028 - Vertical Lime Kiln	May-31-22	Jan-04-24
M030 - Conveyors	Nov-18-22	Aug-25-23
M034 - Autoclave	May-31-22	Nov-21-23
Piping	May-31-22	Oct-30-23
Structural	Apr-03-23	Aug-26-24
Electrical	Jun-27-22	Mar-27-24
Architectural	Jan-25-22	Jan-19-23
Contracting	Oct-19-21	May-31-22
Contracts	Oct-19-21	May-31-22
Road Construction Contract Package	Oct-19-21	Feb-21-22
Powerline Construction Contract Package	Nov-02-21	May-02-22
Plant Construction Contract Packages	Dec-03-21	May-31-22
Camp Construction Contact Package	Jan-25-22	May-16-22



Current
Progress
Critical
Milestones
Summary

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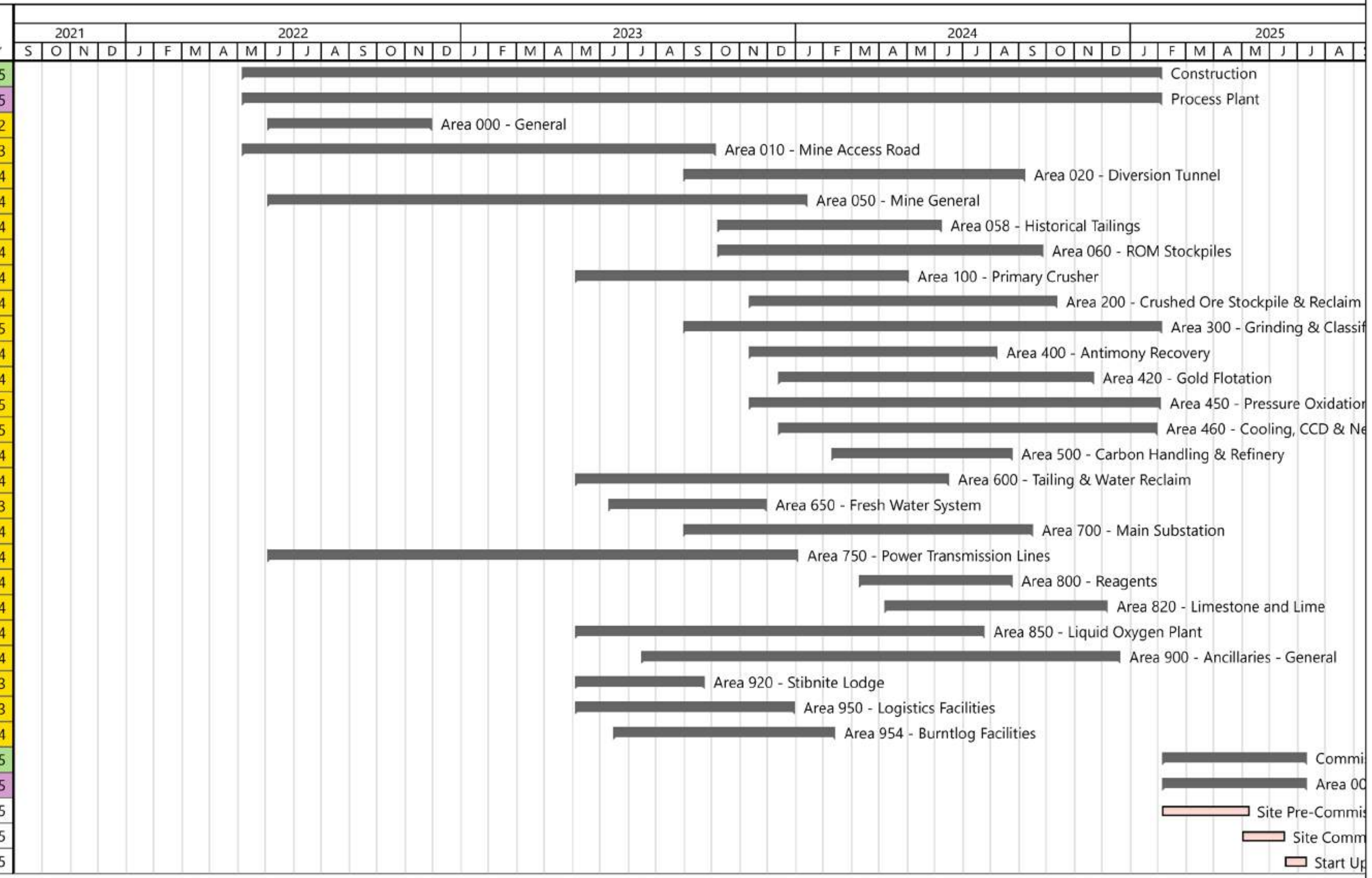
Stibnite Gold - EPCM Project Schedule for FS Update 2021 — **M3-170045**

Name	Start	Finish
Construction	May-04-22	Feb-03-25
Process Plant	May-04-22	Feb-03-25
Area 000 - General	Jun-01-22	Nov-28-22
Area 010 - Mine Access Road	May-04-22	Oct-04-23
Area 020 - Diversion Tunnel	Aug-30-23	Sep-06-24
Area 050 - Mine General	Jun-01-22	Jan-12-24
Area 058 - Historical Tailings	Oct-05-23	Jun-07-24
Area 060 - ROM Stockpiles	Oct-05-23	Sep-25-24
Area 100 - Primary Crusher	May-03-23	May-01-24
Area 200 - Crushed Ore Stockpile & Recl...	Nov-10-23	Oct-11-24
Area 300 - Grinding & Classification	Aug-30-23	Feb-03-25
Area 400 - Antimony Recovery	Nov-10-23	Aug-06-24
Area 420 - Gold Flotation	Dec-11-23	Nov-20-24
Area 450 - Pressure Oxidation	Nov-10-23	Feb-03-25
Area 460 - Cooling, CCD & Neutralization	Dec-11-23	Jan-29-25
Area 500 - Carbon Handling & Refinery	Feb-08-24	Aug-24-24
Area 600 - Tailing & Water Reclaim	May-03-23	Jun-14-24
Area 650 - Fresh Water System	Jun-08-23	Nov-28-23
Area 700 - Main Substation	Aug-30-23	Sep-14-24
Area 750 - Power Transmission Lines	Jun-01-22	Jan-03-24
Area 800 - Reagents	Mar-08-24	Aug-24-24
Area 820 - Limestone and Lime	Apr-05-24	Dec-05-24
Area 850 - Liquid Oxygen Plant	May-03-23	Jul-23-24
Area 900 - Ancillaries - General	Jul-14-23	Dec-19-24
Area 920 - Stibnite Lodge	May-03-23	Sep-22-23
Area 950 - Logistics Facilities	May-03-23	Dec-30-23
Area 954 - Burntlog Facilities	Jun-13-23	Feb-12-24
Commissioning	Feb-04-25	Jul-10-25
Area 000 - Site General	Feb-04-25	Jul-10-25
Site Pre-Commissioning	Feb-04-25	May-08-25
Site Commissioning	Apr-30-25	Jun-16-25
Start Up & Project Turn Over	Jun-17-25	Jul-10-25



Current
Progress
Critical
Milestones
Summary

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24.4.2 Objectives

The PEP has been established with the following objectives:

- Maintain the highest standard of safety and environmental performance to avoid and minimize incidents and accidents;
- Design and construct a process plant, together with the associated infrastructure, that is cost-effective, achieves performance specifications, and is built to high-quality standards;
- Design and operate the mine using proven methodologies and equipment;
- Optimize the Project schedule to achieve an operating plant in the most efficient and timely manner within the various constraints placed upon the Project; and
- Comply with the requirements of the conditions for the construction and operating license approvals.

24.4.3 Plan of Approach

24.4.3.1 Philosophy

This section describes the execution plan for advancing the Stibnite Gold Project from the current FS design stage to production. The PEP formally identifies and documents the key Project processes and procedures that are required to support the successful execution of the Project including:

- Develop a project schedule that encompasses Basic Engineering through procurement, construction, and commissioning;
- Consider significant Project logistics;
- Develop and implement site communications, construction infrastructure, and water supply for an early and efficient start-up;
- Plan for early construction mobilization;
- Develop practices and protocols that are protective of the environment and ensure compliance with permits and regulations;
- Develop an Environmental, Health and Safety Plan that is comprehensive yet concise so that contractors, construction managers, and members of Midas Gold's development team are safe during the construction phase of the Project;
- Develop and execute Project control procedures and processes;
- Perform constructability reviews;
- Implement Project accounting and cost control best practices;
- Issue a cost control plan and a control budget; and
- Oversee Project accounting.

Midas Gold has assumed an EPCM approach, utilizing multiple hard money and low unit cost prime contracts for Construction Management (**CM**), as the recommended method for executing the Project. The capital cost estimate is based on this methodology. Mine development pre-production work activities are envisioned to be performed by contractors selected through a pre-qualification and pre-tendering process, beginning with the water diversion tunnel, the site access road construction, and power transmission line.

Construction is planned to be performed by companies from the Rocky Mountain region wherever possible because the Project is located in an area with an abundance of qualified contractors. Some items affecting the Project are:

- Ability to start work that does not require engineering;
- Availability of construction and engineering resources;
- Experience of the qualified firms considered and their typical and proposed approach; and
- An approach that utilizes the best resources available (matching contractors to the size of each contract).

The EPCM approach provides for contracts that would include civil, concrete, structural steel, mechanical, piping, electrical, and instrumentation.

The majority of mechanical and electrical equipment required are designed to be procured within North America. Concrete and building construction materials are designed to be sourced locally, wherever possible. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment are designed to be sourced within the US and within the region to the extent practical.

24.4.3.2 Engineering

Engineering is designed to match the plant protocol for drawing titles, equipment numbers, and area numbers and produce drawings in the Imperial System of Units (English) and drawings and specifications for the FS are in English, which is planned for subsequent design and production.

Engineering control of the FS design is maintained through drawing lists, specification lists, equipment lists, pipeline lists, cable schedules, and instrument lists. Control of Engineering Requisitions for Quote would be performed through an anticipated purchase orders list. Progress would be tracked through the use of the lists mentioned.

Concrete reinforcing steel drawings use customary bar sizes available in the US, fully detailed to allow either site or shop fabrication.

Structural steel details use a software program such as TEKLA and mechanical steel details use software programs such as Inventor, TEKLA, or similar. This permits steel fabrication before installation contracts are awarded.

24.4.3.3 Procurement

Per Midas Gold's commitments, procurement will occur in expanding circles starting in Valley County, to the surrounding counties, Idaho, the greater U.S., and overseas. Given the number of large mines in the surrounding states and across the U.S., potential suppliers are abundant for many components and supplies.

Procurement of long delivery equipment and materials is scheduled with associated engineering tasks to ensure that the applicable vendor information is incorporated into the design drawings and that the equipment is delivered to the site at the appropriate time, in support of the Project schedule. Particular emphasis is placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

The EPCM contractor, acting as Agent for Midas Gold through the use of owner-approved purchase order forms would procure major process equipment. including all of the equipment on the equipment list and instruments on the instrument list. Some instruments are designed to be part of vendor equipment packages. Structural steel, electrical panels, electrical lighting, major cable quantities, specialty valves, and specialized piping are planned as vendor packages, leaving contractors responsible for the purchase of common materials only.

Equipment and bulk material suppliers would be selected via a competitive bidding process and construction contractors would be selected through a pre-qualification process followed by competitive bidding. The Project is anticipated to employ a combination of lump sum and unit price contracts as appropriate for the level of engineering and scope definition available at the time contract(s) are awarded.

Subject to Midas Gold's procurement commitments, sourcing for engineered equipment is planned on a world-wide basis and will be selected on the best delivered price and delivery schedule on a fit-for-purpose basis.

Equipment purchases are typically Free Carrier (**FCA**) at the point of manufacture or nearest shipping port for international shipments so that a logistics contractor can coordinate all shipments of equipment and materials for the Project and arrange for ocean and overland freight to the job site.

The receipt and storage of the major equipment and materials at site would be the responsibility of the EPCM contractor and issued to the contractors for installation at the appropriate time. Bulk piping and electrical materials and some minor equipment are intended to be supplied by the various construction contractors as part of their contracts. Each construction contractor would be responsible for the receipt, storage, and distribution of materials and minor equipment they purchased.

Recommended pre-qualified vendors for each major item of equipment would be established by The EPCM contractor for approval by Midas Gold. The EPCM contractor prepares the tender documents, issues the equipment packages for the bid, prepares a technical and commercial evaluation, and issues a letter of recommendation for purchase for approval by Midas Gold. With the assistance of the EPCM contractor, Midas Gold would conduct the commercial negotiations with the recommended vendor and advise the EPCM contractor of the negotiated terms for preparation of the purchase documents. When approved, the EPCM contractor would issue the purchase order, track the order, and expedite the engineering information and delivery of the equipment to the site.

24.4.3.4 Inspection

The EPCM contractor's responsibilities would include conducting QA/QC inspections for major equipment during the fabrication process to ensure the quality of manufacture and adherence to specifications. Levels of inspection for major equipment identified during the bidding stage may range from receipt and review of the manufacturer's quality control procedures to visits to the vendor's shops for inspection and witnessing of shop tests prior to shipment of the equipment. Inspectors close to the point of fabrication should be contracted to perform this service to minimize the travel cost for the Project. Some assistance may also be provided by the EPCM engineering design team.

24.4.3.5 Expediting

The EPCM contractor is also responsible for expediting the receipt of vendor drawings to support the engineering effort as well as the fabrication and delivery of major equipment to the site. Expediting reports issued at regular intervals outlining the status of each purchase order in order would alert the Project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action can then be taken to mitigate any delay. The logistics contractor coordinates and expedites the equipment and material shipments from point of manufacture to the site, including international shipments through customs.

24.4.3.6 Project Services

The EPCM contractor manages and controls of the various project activities to ensure that the team has appropriate resources to accomplish Midas Gold's objectives.

24.4.4 Construction

24.4.4.1 Construction Methodology

The grinding-flotation building and autoclave buildings are planned to be bridge-frame metal, moment-frame structures. The truck shop, the Historical Tailings reclaim building, maintenance shop, and warehouse buildings are currently planned as pre-engineered metal buildings or fabric-covered structures. Most of the ancillary buildings on the Stibnite Gold Project site are planned to be modular buildings including the offices, camp, and other ancillary facilities.

As currently designed, construction work is scheduled for approximately 36 months from mobilization to the commencement of commissioning. Assuming funding is in place, earthworks would commence shortly after Project permits have been released and as soon as a contractor can be mobilized to the field. The construction program is scheduled to start in Year -3. Initial construction work includes clearing and grubbing of the plant site, water diversion tunnel, mass earthwork for site development, access road, and in-plant roads. Concrete foundations for the process buildings and other support structures are designed to be constructed next.

24.4.4.2 Construction Management

The EPCM contractor, as Agent for the Owner, conducts Construction Management in accordance with the Contracting Plan using prime contracts for civil/concrete and structural, mechanical, electrical, piping, and instrumentation. The contracting plan emphasizes using local contractors for the construction work packages to minimize mobilization and travel costs. The EPCM contractor would pre-qualify local contractors and prepare tender documents to bid and select the most qualified contractor for the various work packages. Some work packages would include the design, supply, and erection for specific facilities which are specialized in nature. The EPCM team would be composed of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, supervising warehouse and material management functions, and performing cost control and schedule control functions. A resident construction manager directs these activities with a team of engineers, locally hired supervisors, and technicians. A commissioning team would do final checkout of the Project.

Construction progress is measured using ledgers for construction quantities to develop completion percent and hours earned by contractors. Surveyors measure civil quantities, yards of concrete placed, tons of steel erected, and similar measures for architectural, piping, and electrical quantities. Mechanical installations would be measured against the estimated installation hours from control estimates developed during detailed engineering.

Some site services would be contracted to third party specialists working under the direction of the resident construction manager. Construction service contracts include field survey and QA/QC testing services.

24.4.5 Contracting Plan

Contracting is an integral function in the Project's overall execution conducted in accordance with the EPCM contract. A combination of vertical, horizontal, and design-construct contracts may be employed as determined by the work to be performed, degree of engineering, and scope definition at the time of award. The FS contracting plan includes an on-site concrete batch plant using screened native colluvial and alluvial materials as aggregate. The civil contract would cover all clearing, grubbing, bulk excavation, engineered fill, grading, and possibly geomembrane lining of the TSF, ponds, and pipe trenches. The concrete placement contract includes concrete forming, rebar, placement, and stripping.

A list of proposed contract work packages has been developed to identify items of work anticipated to be assembled into a contract bid package. Certain work packages may be combined in a single package depending on how the project execution and timing, while larger bid packages may involve sub-contractors on certain components of the work. Table 24-2 represents the Proposed Contract Work Package list.

Table 24-2: Proposed Contract Work Package List

No.	Bid Packages:	Comments
1	Materials Testing	Soils, concrete & structural materials
2	Surveying	Establish control points, layout roadway, and plant site areas
3	Mine Access Road	Includes roadway, drainage, culverts, and retaining walls
4	Bridges and Stream Crossings	Multi-plate tunnels
5	Water Diversion Tunnel	Underground mine contractor
6	138 kV Power Transmission Line	Idaho Power to Yellow Pine Substation; a second contractor to erect power transmission line from Yellow Pine Substation to site
7	Construction Camp Installation	Possibly by provider of modular construction camp
8	Main Substation & Oxygen Plant Substation	Includes emergency generator installation & testing
9	Mine Pre-Stripping Contract	Includes starter dam construction
10	Field Electrical Distribution - Sub Station to Process Areas, Camp & Water Pumping	Overhead lines and duct banks from switchgear
11	Water Supply System - Yard Water Piping	Includes fire suppression
12	Septic System - Sewer Piping, Plant & Leach Field	Two septic systems required: process plant area and camp area
13	Clearing, Grubbing, Site Excavation, Engineered Backfill, Grading, Trenching, - all Areas	
14	Concrete Work - All Areas	
15	Structural Steel Buildings & Platforms	Includes roofing and siding installation
16	Architectural Finishes	In offices and larger frame structure buildings
17	Field Erected Tanks	Typically part of design-supply-erect contract
18	Mechanical Equipment	Crusher, conveyors, reclaim feeders, grinding mills, flotation cells, thickeners, pumps, mechanical steel, etc.
19	Process Piping & Field Instrumentation	
20	Instrumentation & Controls Programming	PLC programming, HMI screen development; I/O & communications.
21	Permanent Camp Installation	By camp provider

24.4.6 Project Schedule

A feasibility-level schedule has been developed based on the Project description with the objectives and philosophy documented in this report. The schedule includes engineering, contracts, procurement, construction, remaining site work, plant pre-commissioning, and commissioning activities and is presented on Figure 24-3.

The schedule assumes that permitting progress enables basic engineering to commence in Year -4 leading into detailed engineering so that procurement can begin in Q4 of Year -4. Construction would commence shortly thereafter in Q1 of Year -3.

- Mining equipment would need to be procured and assembled early starting in Q1 of Year -3 so that pre-stripping could commence in Q2 of Year -2.
- The 138-kV power transmission line would also need to start early commencing in Q2 of Year -4 and finishing at the end of Q3 of Year -1.
- The mine access road is scheduled so that it would commence in Q1 of Year -3 and continue through Q4 of Year -2 to enable transport of larger items to the project site.
- The Oxygen Plant contract procurement is currently designed to begin in Q1 of Year -3.
- The autoclave procurement and fabrication commence in Q3 of Year -4 so that they could be delivered, welded into a single shell, stress relieved, pressure tested, and installed by the end of Q2 of Year -1.

24.4.6.1 Construction Completion and Handover Procedure

The Construction Completion Procedure is part of the Construction Quality Plan as well as the project-specific Commissioning Plan. Contractors would enter into contractual agreements with Midas Gold to perform certain portions of the work, which includes quality control of their work.

The Commissioning Plan would be designed, developed, and implemented to ensure a step-by-step, documented process and procedure for all mechanical, process, electrical, and instrumentation completion, checkout, and pre-operational testing. Pre-operational testing and commissioning take place concurrent with mechanical completion. Pre-operational testing is currently scheduled to commence in Q2 of Year -1 and wet commissioning and start-up is scheduled to commence in Q4 of Year -1.

24.4.7 Quality Plan

A project-specific Quality Plan needs to be developed and implemented for the site. The Quality Plan would be designed to be a management tool for the EPCM contractor to maintain, through the construction contractors, the quality of construction and installation for every aspect of the Project. The plan consists of many different manuals and categories and is typically developed during the engineering phase for availability at the start of construction.

24.4.8 Commissioning Plan

The project-specific Commissioning Plan guides the transition of the constructed facilities from a status of “mechanically” or “substantially” complete to “operational” as defined by the subsystem list developed for the Project. The commissioning group systemically verifies the functionality of plant equipment, piping, electrical power, and controls. This test-and-check phase is conducted by discrete facility subsystems. The tested subsystems are combined until the plant is fully functional. Start-up, also a commissioning group responsibility, would progressively move the functional facilities to operational status and performance. In addition to these activities, the commissioning portion of the work also includes coordination of facilities operations training, maintenance training, and turnover of all compiled commissioning documentation in an agreed form.

24.4.9 Environmental, Health and Safety Plan

An Environmental Health and Safety Plan (**EHSP**) would be established for the construction of the SGP and any other authorized work at the Project site. The EHSP would cover all contractor personnel working and any other authorized work for the Project.

The EHSP specifies regulatory compliance requirements, training, certifications, and medical requirements necessary to complete the Project for personnel and contractors involved in the Project. The EHSP would include a comprehensive program of sampling and analyses to monitor environmental conditions during construction and mitigate potential adverse impacts. The plan would also include a sitewide Stormwater Management Pollution Prevention Plan (**SWPPP**) as a preventative measure and a Spill Control and Countermeasures Plan (**SPCC**). Along with the Operations Procedures, the EHSP would be required to be followed by all Contractor personnel working at the site.

24.4.10 Traffic Management Plan

In order to minimize the disruption along the mine access road and at the mine site, traffic to the site would need to be coordinated by a dispatcher located at the Cascade offsite facility. Midas Gold would develop a Traffic Management Plan to guide those traveling between the SGLF and the mine site. The plan would be developed in collaboration with the EPCM contractor, construction contractors, suppliers, and transportation companies.

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25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

Since inception, Midas Gold's vision for the Stibnite Mining District (**District**) has been to use modern mining to redevelop an abandoned, brownfield mine site and provide long-term employment and business opportunities for a rural area in Idaho, funded by an economically viable project. Restoration goals were established early on to address environmental impacts from over 100 years of historical mining activities and return the site to a fully functioning, self-sustaining ecosystem with improved water quality and habitat capable of supporting enhanced populations of fish, wildlife and flora. In addition to gold, the District also contains significant Mineral Reserves of antimony, which is on the U.S. Department of Interior's final list of 35 critical minerals.

Midas Gold submitted its PRO to regulators in September 2016. The plan laid out in the PRO was founded on Midas Gold's core values of safety, environment, community involvement, transparency, accountability, integrity and performance. Since filing the PRO, Midas Gold has continued to advance the Project along two parallel paths: (1) additional design and engineering studies in support of the FS; and (2) further environmental modeling and analysis in support of Project permitting. The Project envisioned in this FS achieves Midas Gold's vision of unifying environmental protection and restoration with modern mining operations in an economically attractive Project.

According to CIM definition standards for Mineral Resources and Mineral Reserves, a Feasibility Study is a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a prefeasibility study. Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves; these include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

25.2 INTERPRETATION

The QPs for this Technical Report have reviewed the contents of this Report and are of the opinion that it meets the requirements for a Feasibility Study. Individual QP responsibilities are provided in Section 2. The following subsections summarize the key interpretations for this Technical Report.

25.2.1 Surface Rights, Royalties, and Mineral Tenure

Midas Gold is vested with fee simple, mineral, or possessory record title to, or an option to purchase, the Stibnite Gold Project properties described in Section 4, subject to the royalties, agreements, limitations and encumbrances described in Section 4.

25.2.2 Geology and Mineralization

The understanding of the regional and local geology with regards to the lithology, structure, alteration and mineralization for each of the mineralized zones and deposit types discussed in Sections 7 and 8 is sufficient to estimate the Mineral Resources contained herein.

25.2.3 Exploration

The previous drilling exploration programs, along with the geologic mapping, geochemical and geophysical studies, and petrology and mineralogy research carried out to date, reasonably supports the potential for expansion of defined deposits, potential for discovery of high-grade underground mineable prospects, and the potential for discovery of new bulk mineable prospects as discussed in Section 9.

25.2.4 Drilling and Sampling

The drilling methods, recovery, collar survey, downhole survey, and material handling for the samples used in the Mineral Resource estimates for this Report are sufficient to support the Mineral Resource estimates contained in this Report, subject to the assumptions and qualifications contained in Sections 10 and 11.

25.2.5 Data Verification

The data used for estimating the Mineral Resources for the Hangar Flats, West End, Yellow Pine and Historical Tailings is adequate for the purposes of this Report and may be relied upon to report Mineral Resources based on the conditions and limitations set out in Section 12.

25.2.6 Metallurgy

The metallurgical testing conducted on samples from West End, Hangar Flats, Yellow Pine, and the Historical Tailings included extensive process mineralogy optimizations, batch and pilot plant test work, metallurgical variability testing on various ore types from each of the deposits and environmental stability testing of tailings. The confirmatory metallurgical testing and analysis detailed in Section 13 support the process flow sheet and its applicability to each of the deposits, demonstrating a single plant can process all ores from the Project as they are mined, subject to the conditions and limitations set out in Section 13.

25.2.7 Mineral Resources

The Mineral Resource estimates in Section 14 are accurate to within the level of estimate required for categorization as Inferred, Indicated, and Measured Mineral Resources, with the latter two categories suitable for use in a Feasibility Study, subject to the conditions and limitations set out in Section 14. Further, it can be reasonably expected that the majority of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. These estimates were performed consistent with industry best practices and demonstrate reasonable prospects for eventual economic extraction, as required by NI 43-101.

25.2.8 Mineral Reserves

Based on a thorough review of the designs, schedules, risks, and constraints of the Project detailed within this Report, it is the opinion of the QP responsible for Section 15 that the FS forms the basis for an economically viable Project after taking into account mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, governmental factors and other such modifying factors and supports the declaration of Mineral Reserves. Subject to the conditions and limitations contained in this Report, this FS demonstrates that, as of the effective date of this Report, extraction can be economically justified. The term ‘Mineral Reserve’ does not necessarily signify that all governmental approvals have been received; it does signify that there are reasonable expectations that such approvals will be granted.

25.2.9 Mine Plan and Schedule

The mine plan and schedule detailed in Section 16 have been developed to maximize mining efficiencies, while utilizing the current level of geotechnical, hydrological, mining and processing information available and are, subject to the conditions and limitations set out in Section 16, sufficient to support the declaration of Mineral Reserves.

25.2.10 Metallurgical Recovery

The recovery methods including the major unit operations detailed in Section 17 comprising primary crushing, SAG and ball mill grinding, antimony flotation (when warranted), bulk auriferous sulfide flotation, auriferous sulfide concentrate pressure oxidation, in-situ acid neutralization, cyanidation of the pressure oxidation residue, CIL processing (when warranted) of whole ore and/or flotation tailings, precious metal recovery to doré and tailings detoxification are sufficient to demonstrate recoveries to support the mine planning and economics detailed herein, and the declaration of Mineral Reserves.

25.2.11 Infrastructure

The onsite and offsite infrastructure, including power, access, ore processing, tailings management, and support facilities, detailed in Section 18 is designed and cost estimated to a level of detail that supports Project viability and the economics detailed herein.

25.2.12 Market Studies and Contracts

The doré and antimony concentrate market studies detailed in Section 19 are consistent with industry standards and market patterns and are similar to contracts found throughout the world. The metal prices selected for the five economic cases in this Report represent a probable range of scenarios that support a Feasibility Study level economic analysis.

25.2.13 Environment, Permits, and Social and Community Impacts

Section 20 summarizes the available information on: environmental studies, including modeling, conducted to date and the related known environmental issues associated with the Project; status of litigation related to legacy environmental issues; Project permitting requirements and status; Project-related social and community impacts, benefits, and community agreements; remediation of legacy impacts built into the design for and execution of the Project; requirements and plans for short-term and long-term water treatment. Additionally, mine closure, restoration, reclamation, and mitigation are discussed, and attendant costs are estimated to a level of detail that supports Project economic and technical viability to the level of a Feasibility Study and the economics detailed herein.

25.2.14 Capital and Operating Costs

The capital and operating costs detailed in Section 21, which were derived from several previous sections of the Report, are, subject to the conditions and limitations in this Report, designed and cost estimated to a level of detail that supports Project economic and technical viability to the level of a Feasibility Study and the economics detailed herein.

25.2.15 Financial Analysis

The financial analysis presented in Section 22 illustrates that the Project economics, subject to the conditions and limitations in this Report, are positive and can support declaration of Mineral Reserves and the demonstration of technical and economic viability to the level of a Feasibility Study.

25.3 CONCLUSIONS

This FS highlights the positive economics of the Stibnite Gold Project. The Project's exceptional grade and low strip ratio would place this Project in the lowest quartile of the global gold mining industry cost curve and, coupled with its large Mineral Reserve and capital expenditure profile, make the Stibnite Gold Project an economically attractive development project. The Project's economics are resilient at lower metal prices and also exhibit significant leverage to rising prices. The FS affirms that the Project can address legacy impacts left behind by previous mining operators, including the recovery, reprocessing and safe storage of historical tailings, relocation and/or reuse of legacy development rock and spent ore, stream restoration, improved water quality, restoration of fish passage, and reforestation. The FS demonstrates a positive local economic benefit to Idaho communities, bringing more than $1 billion in initial capital investment, approximately 550 direct jobs during 14+ years of operations, and hundreds of indirect and induced jobs, while generating significant taxes and other benefits to the local, state and national economies. The financial analysis presented in Section 22 demonstrates that the Stibnite Gold Project is financially viable and has the potential to generate robust economic returns based on the assumptions and conditions set out in this Report and this conclusion warrants continued work to advance the Project towards Basic Engineering and, ultimately, development once permitted.

The QPs of this Report are not aware of any unusual, significant risks or uncertainties that could be expected to affect the reliability or confidence in the Project based on the data and information available to date.

25.4 RISKS

As with most projects at the Feasibility Study level, there continue to be risks that could affect the economic potential of the Project. A number of risks and opportunities have been identified with respect to the Project; aside from industry-wide risks and opportunities (such as changes in capital and operating costs related to inputs like steel and fuel, metal prices, permitting timelines, etc.). External risks are, to a certain extent, beyond the control of Midas Gold and are much more difficult to anticipate and mitigate, although, in many instances, some risk reduction can be achieved. Table 25.1 identifies what are currently deemed to be the most significant internal Project risks, potential impacts, and possible mitigation approaches.

Table 25.1 Project Risks Identified Following the FS

Risk		Explanation / Potential Impact	Comments / Possible Risk Mitigation
General Risks Common to the Mining Industry			
GR1	CAPEX and OPEX	The ability to achieve the estimated CAPEX and OPEX costs are important elements of Project success. An increase in OPEX of 20% would reduce the after tax $NPV_{5\%}$ to approximately $1.15 billion versus $1.40 billion using current open pit designs. If OPEX increases, then the mining cut-off grade would increase and, all else being equal, the size of the optimized pit would reduce, yielding fewer mineable tons and less recoverable gold. Similarly, an increase in CAPEX by 20% would reduce the $NPV_{5\%}$ to approximately $1.20 billion using current open pit designs.	Additional engineering, cost estimating, and construction execution planning would increase the CAPEX and OPEX estimate's accuracy. Developing mine plans and schedules for higher CAPEX and OPEX cases would also help mitigate the financial impacts of higher CAPEX and OPEX cases.
GR2	Permit Acquisition or Delay	The ability to secure all of the permits to build and operate the Project is of paramount importance. Failure to secure the necessary permits could stop or delay the Project.	A thorough Environmental Impact Statement for the Project and a design that gives appropriate consideration to the environment and local community expectations and input is required and is in progress.
GR3	Ability to Attract Experienced Professionals	The ability of Midas Gold to attract and retain competent, experienced professionals is a key success factor for the Project. High turnover or the lack of appropriate technical and management staff at the Project could result in difficulties meeting Project goals.	The early search for, and retention of, professionals should help identify and attract critical people and mitigate this risk.
GR4	Falling Metal Prices	A drop in metal prices during the mine development process could have a negative impact on the profitability of the operation, especially in the critical first years.	Begin construction when the outlook is good for price improvement (or is stable through a high metal price environment) and have mitigating strategies, such as hedging or purchase of puts, and supporting analyses to address the risk of a downturn.
GR5	Change in Permit Standards, Processes, or Regulations	A change in standards, processes, or regulations could have a significant impact on Project schedules, operating cost and capital cost. Permit conditions could require design changes to the Project, increasing costs.	Participate in legislative and regulatory processes to ensure standards remain protective, fair and achievable.
GR6	Development or Construction Schedule	The Project development could be delayed or extended for a number of reasons, which could impact Project economics.	Opportunities exist to modify the construction activities schedule and delivery method such as accelerating construction of the new access road to build a greater percentage of the Project from that road versus undertaking appreciable early site construction from the existing road.

Risk		Explanation / Potential Impact	Comments / Possible Risk Mitigation
GR7	Geotechnical Engineering	The geotechnical nature of the open pit walls and infrastructure areas could impact the allowable pit slopes and design criteria, which could impact mineable tons, strip ratio and overall Project economics negatively.	Additional geotechnical studies and stability monitoring during construction and operations may improve understanding of geotechnics and reduce such risks.
Stibnite Gold Project Specific Risks			
PR1	Mineral Resource Modelling	Certain Mineral Resources were estimated with data that included historical sample data which introduces some level of risk and uncertainty. The risk is the level of certainty in the Mineral Resource estimates and whether they can be confirmed with additional drilling.	Additional drilling could be completed to reduce risk associated with use of historical data, especially in the West End Deposit. See Section 26 for additional drilling recommendations.
PR2	Clean Water Act Litigation	Delays related to the Clean Water Act litigation initiated by the Nez Perce Tribe (NPT).	Continue to engage NPT to determine if alternatives to litigation are mutually beneficial. Continue to work with regulatory authorities with EPA and USFS to establish an Administrative Order on Consent (AOC) that could limit the ability for the NPT lawsuit to move forward.
PR3	Metallurgical Recoveries	Lower metallurgical recoveries and revenue, increased processing costs, and/or changes to the processing circuit design, could all negatively impact the Project economics.	Pilot plant runs with appreciably larger samples were completed to support the Feasibility Study and increase the confidence of the recovery assumptions and overall process design, however, some residual metallurgical risk always remains until operations commence.
PR4	Water Management	Water management is a critical component of the Project. While a comprehensive site-wide water balance model and 3D groundwater model, along with storm runoff modeling and stream gage analysis, were used to design the ground and surface water diversion and interception systems, more information would help improve the accuracy of the water balance, optimize diversion channel and pond sizing, design treatment facilities, and advance comprehensive long-term closure designs.	Continue to collect and analyze on-site groundwater, surface water, and meteorological data to enhance hydrological knowledge of the site for improved water management and closure designs. Refine hydrologic and hydrogeologic predictive modeling during operations for more accurate long-term estimates and associated mitigation strategies.

Risk		Explanation / Potential Impact	Comments / Possible Risk Mitigation
PR5	Water Geochemistry	Metal leaching (**ML**) from development rock, groundwater quality and process water quality are such that contact runoff and pit dewatering water would have to be collected and treated (or reused) to achieve regulatory water discharge limits during operations and/or early closure, the TSF Buttress would require a low-permeability cap at closure, and the TSF would require long-term water treatment for approximately 25 years after operations end. If water quality standards are more stringent than assumed or water quality worse than predicted, additional measures to isolate mined materials from water interaction may be necessary, water treatment equipment complexity and treatment duration could increase, and CAPEX and/or OPEX would increase.	Perform treatability studies / technology confirmation steps for proposed contact water (operations and closure) and process water treatment (TSF closure). Continue to collect and analyze on-site groundwater and surface water quality data to enhance knowledge of the site for improved water management and closure designs. Complete cap/cover effectiveness studies to assess water quality impact of different capping technologies. Evaluate tailings consolidation in more detail to improve closure planning. Seek site-specific water quality standards reflective of high natural background concentrations. Refine geochemical predictive modeling during operations for more accurate long-term estimates and associated mitigation strategies.
PR6	Reclamation and construction material deficit	There may be insufficient materials that meet construction and/or reclamation specifications within the project footprint.	Conduct additional investigations to better define material volumes and characteristics.

25.5 OPPORTUNITIES

There are a number of significant opportunities that could improve the economics of the Project other than those factors that are common to the sector (such as increasing metal prices, falling input costs, etc.). The major opportunities that have been identified at this time are summarized in Table 25.3. Further information and assessments are needed before these opportunities could be included in the Project economics.

The opportunities in Table 25.3 are separated into general opportunities common to the mining industry, and Project-specific opportunities unique to the Stibnite Gold Project. The Project-specific opportunities are further categorized into three broad categories of potential to improve the Project Net Present Value (**$NPV_{5\%}$**); the categories, and a brief listing the opportunities, are provided below.

Opportunities that could improve the economics of the Project, including a number with potential to increase the $NPV_{5\%}$ by more than $100 million follow:

- Conversion of Mineral Resources not currently part of the Mineral Reserves, and associated opportunities, are summarized below and presented by deposit in Table 25.2:
  - In pit conversion of approximately 9.8 Mt of Inferred Mineral Resources grading 1.02 g/t Au occurring within the Mineral Reserve Pits and containing approximately 321 koz of gold, to Mineral Reserves, increasing Mineral Reserves and reducing the strip ratio;
  - Out of pit conversion of approximately 27.1 Mt of Measured and Indicated Mineral Resources grading 1.26 g/t occurring outside the current Mineral Reserve Pits containing approximately 1,098 koz of gold, to Mineral Reserves;
  - Out of pit conversion of approximately 26.2 Mt of Inferred Mineral Resources grading 1.09 g/t Au occurring outside the current Mineral Reserve Pits containing approximately 917 koz of gold, to Mineral Reserves;

Table 25.2: Stibnite Gold Project Mineral Resources exclusive of Mineral Reserves

Location	Mineral Resources within Reserve Pits			Mineral Resources outside Reserve Pits but within Resource Pits(3)					
Classification	***Inferred Mineral Resources***			***M&I Mineral Resources***			***Inferred Mineral Resources***		
Deposit	**Tonnage (000s)**	**Gold Grade (g/t)**	**Contained Gold (000s oz)**	**Tonnage (000s)**	**Gold Grade (g/t)**	**Contained Gold (000s oz)**	**Tonnage (000s)**	**Gold Grade (g/t)**	**Contained Gold (000s oz)**
Yellow Pine	714	0.78	18	5,109	1.25	205	2,499	1.01	81
Hangar Flats	58	1.87	3	17,333	1.36	756	12,166	1.12	437
West End	9,044	1.03	300	4,684	0.91	137	11,495	1.08	400
Total 2	***9,816***	***1.02***	***321***	***27,126***	***1.26***	***1,098***	***26,161***	***1.09***	***917***

Notes:

(1) All Mineral Resources have been estimated in accordance with Canadian Institute of Mining and Metallurgy and Petroleum ("CIM") definitions, as required under National Instrument 43-101 ("NI 43-101").

(2) Total of inferred Mineral Resources within Reserve Pits excludes Historical Tailings.

(3) Mineral resources exclusive of mineral reserves are reported based on a fixed gold cut-off grade of 0.45 g/t for sulfide and 0.40 g/t for oxide, and in relation to conceptual Mineral Resource pit shells and Mineral Reserve pits to demonstrate potential economic viability as required under NI43-101. Indicated mineral resources exclusive of mineral reserves are reported to demonstrate potential for future expansion should economic conditions warrant. Inferred mineral resources exclusive of mineral reserves are reported to demonstrate potential to increase in-pit production should inferred mineral resources be successfully converted to mineral reserves; mineralization lying outside of Mineral Resource pit shells is not reported as a mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability. These mineral resource estimates include inferred mineral resources that are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated.

- Opportunities for discovery of new deposits and to increase grade within known deposits include:
  - Conversion of unclassified material within the Mineral Reserve pits that is currently treated as development rock to Mineral Reserves, increasing Mineral Reserves and reducing strip ratios;
  - Discovery of additional antimony Mineral Resources and Mineral Reserves in the Hangar Flats and Yellow Pine deposits as improved continuity of stibnite vein arrays and/or additional discrete zones of higher-grade antimony mineralization;
  - Increased Mineral Resources and Mineral Reserves in West End due to improved continuity of higher-grade gold mineralization and through addition of fire assay information in areas where only cyanide assays were available for the current Mineral Resource estimates;
  - Potential for the definition of higher grade, higher margin underground Mineral Reserves at Scout, Garnet or Hangar Flats; and,
  - Discovery of other new deposits with attractive operating margins.

Exploration targets include conceptual geophysical targets, geochemical targets from soil, rock and trench samples, and results from widely spaced drill holes; as a result, the potential size and tenor of the targets are conceptual in nature. There has been insufficient exploration to define mineral resources on these prospects and this data may not be indicative of the occurrence of a mineral deposit. Such results do not provide assurance that further work will establish sufficient grade, continuity, metallurgical characteristics and economic potential to be classed as a category of mineral resource. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Medium potential benefit opportunity (potential to increase $NPV_{5\%}$ by \$10 to \$100 million) include:

- Metallurgical improvements that improve the Project economics;
- Secondary antimony processing to enhance payability;
- Federal action to subsidize antimony production and/or onshore processing through grants or low-cost loans for the Department of Defense, Defense Logistics Agency managed National Materials Stockpile program;
- Alternative (government or vendor) funding sources for off-site infrastructure;
- Utilizing preowned equipment to reduce CAPEX and development timelines; and
- Titanium versus brick and lead clad autoclave to reduce the size of the vessel and increase utilization.

Low Potential Benefit Opportunity (potential to increase $NPV_{5\%}$ by less than \$10 million) include:

- Tungsten recovery as a by-product;
- Reducing contact water treatment to a single-pass system;
- Designing low-permeability cap for the TSF Buttress without a geomembrane;
- Reducing length, height, and expense of retaining walls on Burntlog Route;
- Using the antimony credit in open pit optimization, increasing Mineral Reserves; and,
- Expansion of existing or construction of an additional antimony processing facility to produce marketable antimony-derived products in North America potentially reducing antimony concentrate shipping costs and increasing payability.

Table 25.3 Project Opportunities Identified Following the FS

Opportunity		Explanation	Potential Benefit
General Opportunities Common to the Mining Industry			
GO1	General Project Optimization	In the same way that overall CAPEX, OPEX, metallurgical recoveries, etc. are potential risks to the Project, they may also be opportunities.	Continued Value Engineering studies will be undertaken concurrent to Basic Engineering and will focus on improving the overall Project economics.
GO2	Rising Metal Prices	Increases in metal prices, especially gold, would increase revenue and Project economics.	Increased revenue enhances financial factors.
GO3	Reagent/Fuel Price Decreases	Reductions in reagent and consumable prices, especially fuel, power and cyanide, have the potential to decrease operating costs and enhance the Project economics.	Lower OPEX may lead to higher net revenue and enhanced Project economics.
GO4	Exploration Potential for Additional Deposits	As discussed in Section 9, the expansion of known Mineral Resources and the addition of new deposits may be possible with further drilling. Based on widely spaced drilling, soil sampling, rock chip sampling and geophysical results the Project area has several exploration targets that justify drilling and may or may not lead to the discovery of additional underground and/or open pit deposits.	The expansion of the Project's Mineral Resources could potentially lead to a longer Project life and/or greater operating flexibility and potentially the justification for a higher throughput. This becomes particularly important, as demonstrated by the economic margin from Yellow Pine vs. Hangar Flats or West End, if higher-grade Mineral Resources are defined that defer lower-grade Mineral Resources currently utilized in the economic analysis.
GO5	New Technology	Over a project life of decades, technological advances are likely. Examples of potential technological advances include autonomous equipment, CNG-powered haul trucks, improved flotation reagents, automation in plant and in vehicles, grade control improvements, water treatment, and advances in materials science particularly geosynthetics.	Technological advances may improve productivity; decrease CAPEX, OPEX, and closure cost; or decrease the likelihood or consequences of safety and environmental incidents.
Project Specific Opportunities with High Potential Benefit			
PO1	In-pit conversion of Inferred Mineral Resources to the Indicated category	Inferred Mineral Resources exist in each of the Project deposits, including material within the Mineral Reserve pits; these Mineral Resources are currently treated as development rock. Conversion of Inferred Mineral Resources within the Mineral Reserve pits to the Measured and Indicated Mineral Resources categories would increase Mineral Reserves, reduce strip ratios and improve overall Project economics.	In pit conversion of approximately 9.8 Mt of Inferred Mineral Resources grading 1.02 g/t Au occurring within the Mineral Reserve Pits containing approximately 321 koz of gold, to Mineral Reserves, would increase the Mineral Reserves and reduce the strip ratio.

	Opportunity	Explanation	Potential Benefit
PO2	Out-of-pit conversion of Mineral Resources to Mineral Reserves	Indicated and Inferred Mineral Resources occur adjacent to and below the Mineral Reserve pits but within the Mineral Resource pit shells; these Mineral Resources are currently assumed not to be mined. Additional drilling and/or a change in economic considerations has the potential to increase the grade and tonnage of the Mineral Reserves by supporting expanded pits.	Increases in Mineral Reserve tonnages, especially at higher grades, could improve the Project economics.
PO3	In-Pit Development Rock conversion to Mineral Resource	Zones within each of the Mineral Reserve pits are comprised of unclassified material based on a lack of drilling. Additional drilling within the pit limits could convert some of this material to Mineral Resources above cutoff, potentially increasing Mineral Reserves and reducing the strip ratios.	Increases in Mineral Reserve tonnages within the Mineral Reserve pits, especially at higher grades, could improve the Project economics.
PO4	Increase in Mineral Resources and Reserves in West End from CN Assay	Partial or spot gold fire assays are prevalent throughout the West End deposit, where available cyanide soluble gold (**AuCN**) assays do not adequately define the transition from oxide to sulfide gold and may underestimate the contained gold in the transition and sulfide portions of the West End deposit.	Additional drilling in the areas where AuCN assays have confirmed, but potentially under-predicted, the grades of gold mineralization could increase the quantity and grade of the Mineral Resources and increase the Mineral Reserves and reduce the strip ratio in the West End open pit if material currently classified as below cutoff grade (and therefore treated as development rock) becomes Mineral Reserves.
PO5	Potential Additional Antimony	Continuity of high-grade antimony mineralization within the Hangar Flats and Yellow Pine deposits is affected by structural complexity that may have led to underestimation of antimony grades in higher grade zones and overestimation of antimony tonnage in the Mineral Resource estimates for antimony.	Tightly spaced grade control drilling during mining may better delineate higher grade antimony zones, increasing grade, and allowing for increased selectivity of high-antimony materials, thereby reducing processing costs and gold losses to the antimony concentrate.
PO6	Potential for Scout Underground Mineral Reserve	Scout is an underground Au-Ag-Sb exploration prospect (see Section 9). It has been identified as a **conceptual** underground target ranging between 2-5 million tons potentially containing between 50-300 koz Au; 40-150 Mlbs Sb; and 300-1,500 koz Ag with target dimensions (true) of approximately 25 to 75 ft thick, 2,000 to 3,000 ft along strike and extending 250 to 300 ft down dip at grades ranging from 0.03-0.06 oz/st Au (1-2 g/t), 1-4-% Sb, and 0.15-0.30 oz/st Ag (5-25 g/t).	Addition of a high-grade underground Mineral Reserve at Scout could enhance Project economics by blending in a percentage of high-grade, high-margin feed early in the Project life and would help to smooth and extend the antimony concentrate production profile.
PO7	Potential for Garnet Underground Mineral Reserve	Garnet is is an underground Au/Ag exploration prospect (see Section 9). It has been identified as a **conceptual** underground target ranging between 1-2 million tons potentially containing between 250 to 500 koz Au with target dimensions (true) approximately 30-60 ft thick by 160–250 ft wide by 1,300-1,800 ft long down plunge at grades ranging from 0.15–0.23 oz/st Au (5-8 g/t).	Addition of a high-grade underground Mineral Reserve at Garnet could enhance Project economics by blending in a percentage of high-grade, high-margin feed early in the Project life, increasing annual gold production.

Opportunity		Explanation	Potential Benefit
Project Specific Opportunities with Medium Potential Benefit			
PO8	Secondary antimony processing	Secondary antimony processing of the antimony concentrates to produce a marketable antimony product (such as antimony trioxide, antimony metal, and sodium antimonite) has been tested on a preliminary basis with positive results (see Section 13) and could result in enhanced Project economics. These benefits increase as antimony prices increase due to the percentage payability for antimony concentrates versus stable costs for secondary processing. In addition, secondary antimony processing would largely eliminate any risk related to gold lost to antimony concentrates during flotation, since most of such gold could be recovered from leach residues after secondary antimony processing.	Secondary antimony processing would allow a significant portion of antimony products to be produced in the USA, reduce US reliance on offshore suppliers, as well as improve terms for payable metal. Additionally, in the current flow sheet, antimony flotation is performed prior to gold flotation and the antimony concentrate is shipped offsite for further processing. As a result, any gold lost to the antimony flotation circuit is also shipped offsite, resulting in the loss of gold or reduced payability. Secondary antimony processing at a nearby plant could allow the gold lost in the concentrate to be fed back into the POX Circuit, post-antimony processing, to recover some of the gold lost to the antimony concentrate.
PO9	Potential increased emphasis on domestic production and processing of antimony under Critical Mineral legislation and Executive Orders	Antimony was listed as a Critical Mineral by the US Department of Interior in 2018. Subsequently, presidential Executive Orders have been issued to promote domestic production and processing of Critical Minerals, including antimony. Such actions are intended to improve permitting predictability and timelines.	Additional Executive Orders to improve permitting timelines, reduce redundancies and promote production of critical minerals may improve timelines and economics for eventual development.
PO10	Preowned Equipment	If available at the time of construction decision, some major capital equipment components may be available as pre-owned items suitable for the Project, with some modifications to the equipment and/or Project.	If acquired on favorable terms, could reduce capital costs and lead times.
PO11	Alternative Autoclave Cladding	The FS design and cost estimate for the autoclave is based on a lead and brick lined pressure vessel, which is conventional for a pyrite-based gold concentrate; however, because the Stibnite concentrate will be treated with ground limestone to maintain constant free acid levels in the vessel there may be an opportunity to use titanium for the interior cladding.	Using titanium could reduce the size and CAPEX of the autoclave and could increase the utilization as inspections would likely be less frequent and maintenance less time consuming, which could increase annual gold production.
PO12	Alternative funding for off-site infrastructure	Government funding programs such as the Transportation Investment Generating Economic Recovery, or TIGER Discretionary Grant program, provides a unique opportunity for the DOT to invest in road, rail, transit and port projects that promise to achieve critical national objectives. Since 2009, Congress has dedicated more than $4.1 billion to fund projects that have a significant impact on the Nation, a region or a metropolitan area. Similarly, P3 (public-private partnerships) have been used for infrastructure development when the benefits extend to the broader community.	Alternative funding could reduce CAPEX and/or OPEX.

Opportunity		Explanation	Potential Benefit
Project Specific Opportunities with Low Potential Benefit			
PO13	Tungsten contribution	The YP open pit was mined in the early 1940s for its tungsten; the pit was the largest single source of tungsten for the WWII Allied war effort. Tungsten content remaining in the YP and HF deposits is unknown due to limited assay data and highly variable distribution.	The addition of a tungsten component to the overall value of the Project cannot be quantified until Mineral Resources are defined or production commences, and sufficient tungsten is identified in the production stream, but there remains a possibility that tungsten could contribute to the Project economics on an incremental basis.
PO14	Optimize mine-impacted water treatment approach	Treatment of mine-impacted water to stringent water quality standards for both arsenic and antimony is expected to require two-pass iron coprecipitation. If mine-impacted water quality is better than predicted, treatment effectiveness is improved upon in bench and pilot-scale testing, or discharge water quality standards revised to reflect elevated natural background levels, there is a potential to meet standards with a single-pass system. Additional opportunities exist, including re-use of process tanks, and passive treatment.	Optimization of the water treatment system could require less infrastructure (tanks/clarifiers) and may consume less reagents and power, reducing both CAPEX and OPEX associated with water treatment during operations and closure.
PO15	Designing low-permeability cap for the TSF Buttress without a geomembrane	A low-permeability cap is required for the TSF Buttress to protect downstream water quality, and such a cap would require a relatively high effectiveness provided by a geomembrane and relatively complex set of soil/rock/growth media layers above the geomembrane. Locating a local source of low-permeability soil, e.g., from silt layers of currently unknown continuity within the Hangar Flats pit alluvium, may enable design of a less complex and expensive cap for the facility.	Use of local materials, different or no geomembrane, and/or less complex section would reduce closure costs for the TSF Buttress, which affects both sustaining CAPEX (for concurrent reclamation of certain portions) and final closure cost.
PO16	Reducing length, height, and expense of retaining walls on Burntlog Route	The Burntlog access route includes significant retaining walls in both cut and fill sections, designed based on limited geotechnical data. Additional geotechnical field data prior to and during construction, refinements to the road line and grade, and substitution of slope stabilization measures for structural walls may reduce the length, height, and expense of walls.	A geotechnical drilling program is planned to be undertaken on the Burntlog Route before detailed design of the road commences, which could reduce wall requirements and CAPEX for the Burntlog Route.
PO17	Conversion of additional legacy waste materials to ore	There are several million tons of historical waste stored at Yellow Pine and West End that limited data suggests some may be above cut-off grade. This material is currently treated as development rock and therefore a cost center in the FS.	If sufficient tonnage and grade is defined through drilling, this material could be reprocessed, generating additional revenues and reducing strip ratios.

Exploration data for the target opportunities discussed in this section include geophysical data; geochemistry from soil, rock, and trench samples; and results from widely spaced drill holes. As a result, the potential size and tenor of the targets are conceptual. There has been insufficient exploration to define mineral resources on these prospects and these data may not be indicative of the occurrence of a mineral deposit. Such results do not assure that further work will establish sufficient grade, continuity, metallurgical characteristics, and economic potential to be classed as a category of mineral resource. Some of the targets include areas with inferred mineral resources. Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

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26 RECOMMENDATIONS

Based on the results of this Feasibility Study, it is recommended that the Project continue to advance. A detailed list of recommendations and work programs has been developed, including estimated costs, that would move the Project through to a construction decision. The estimated cost for completion of this phase is approximately $14 million, of which approximately $12.5 million is required for permitting.

Discretionary expenditures that would target certain opportunities identified in Section 25 that could enhance the FS case, but that are not required to make a construction decision, have also been provided. The estimates have been developed on the basis of some assumed success in each of these areas; were poor results to be received early in the evaluation of the opportunity, discretionary expenditures for this activity would be significantly less than indicated, while exceptional success or exceptional results in a particular area of activity could require higher expenditures than indicated. In addition, it is not likely that all discretionary activities would be undertaken before commencing construction; some, such as exploration and confirmatory drilling at West End, may wait for some time post-production due to the current mining schedule, which sees the West End deposit mined last.

The detailed recommendations have been grouped into logical discipline categories including:

- Mineral Resource evaluation and exploration;
- Field programs required prior to construction;
- Project optimization and Basic Engineering; and
- Environmental, regulatory affairs and compliance.

Table 26.1 summarizes the recommendations and work programs, and separates the costs associated with the work program into core and discretionary categories.

Table 26.1: Project Recommendations, Work Program and Budget

Recommendations and Work Program		Unit	Quantity	Estimated Costs ($000s)	
				Core	Discretionary
Mineral Resource Evaluation and Exploration					
R1	Selective, high-value drilling that targets converting in-pit Inferred Mineral Resources to Measured and Indicated Mineral Resources, with the goals of increasing the Mineral Reserves, increasing grade and/or reducing strip ratio, especially within the West End pit.	feet of drilling	15,000	-	3,000
R2	Selective, high value drilling targeting near-pit opportunities for additional Mineral Reserves, at all three deposits.	feet of drilling	10,000	-	2,000
R3	Selective testing of in-pit unclassified material for potential additional Mineral Reserves and lower strip ratio for all pits, but especially at Hangar Flats west of the MCFZ and at Yellow Pine east of the MCFZ.	feet of drilling	10,000	-	2,000
R4	Additional drilling of both Mineral Resources and in-pit unclassified material at West End for potential higher grades, additional Mineral Reserves, and/or lower strip ratio.	feet of drilling	15,000	-	3,000
R5	Exploratory surficial drilling along the Scout Fault system to test the continuity of the high-grade antimony mineralization and geotechnical/structural analysis to inform geological potential and construction of an exploration decline.	feet of drilling	10,000	-	2,000
R6	Discovery and definition of small tonnage, high grade Mineral Resources at Garnet, Upper Midnight, and/or other areas for potential high margin mill feed that could supplement early production.	feet of drilling	25,000	-	5,000
R7	Continued exploration including mapping, geochemical sampling, and drilling geared toward defining additional Mineral Resources.	Lump sum	1	-	4,000
Field and Laboratory Programs Required Prior to or Concurrent with Construction					
R8	Shallow sampling of alluvium and bedrock via test pits or hand-held auger drilling to better define concrete aggregate borrow sources.	Lump sum	1	-	100
R9	Geotechnical drilling along Burntlog Route to support detailed design of bridges, retaining walls, and confirm suitability of borrow areas.	Lump sum	1	660	-
R10	Pit slope geotechnical evaluation prior to pit development to validate Feasibility Study pit design criteria.	Lump sum	1	-	150
R11	Surficial sampling, drilling, and characterization of the limestone resource in the West End pit to better define the limestone deposit prior to commissioning of the ore processing plant and limestone processing facility.	Lump sum	1	-	500
R12	Consider additional and/or higher-energy geophysics to confirm the bedrock contact and overburden properties at the TSF and tunnel.	Lump sum	1	-	150

Recommendations and Work Program		Unit	Quantity	Estimated Costs ($000s)	
				Core	Discretionary
Project Optimization and Basic Engineering					
R13	Complete a study to assess the potential use of titanium cladding rather than brick for the interior lining of the autoclave.	Lump sum	1	-	100
R14	Consider working with US-based companies to refine antimony concentrate or develop high purity stibnite.	-	-	-	-
R15	Consider undertaking a study to further evaluate the economics of leasing and/or contracting out certain equipment and infrastructure such as: oxygen plant, lime plant, truck fleet, worker housing facility, water treatment plant, evaporators, and other miscellaneous construction equipment including gensets.	Lump sum	1	-	100
R16	Assess the potential to defer construction of the pyrite cleaner flotation circuit, thereby reducing Initial CAPEX.	Lump sum	1	-	50
R17	Assess the potential to eliminate the concentrate preheating circuit, thereby reducing CAPEX and OPEX.	Lump sum	1	-	50
R18	Complete mine impacted water treatability studies to optimize treatment process flowsheet.	Lump sum	1	-	250
R19	Update site-specific seismic hazard study to include most recent data.	Lump sum	1	120	-
Environmental, Regulatory Affairs and Compliance					
R20	Advance environmental and closure-related technical studies based on additional field and laboratory information generated to refine reclamation, closure and bonding cost estimates.	Lump sum	1	-	300
R21	Continue baseline data collection, environmental compliance and reclamation. Consider initiating snow course measurements at a variety of elevations.	Lump sum	1	730	50
R22	Continue to advance regulatory process including Federal Final EIS under NEPA, and ancillary Federal and State permits. Key outstanding ancillary permits and authorizations include wetlands/streams (with U.S. Army Corps of Engineers), water discharge (IPDES; IDEQ), cyanidation (IDEQ), dam safety (IDWR), and closure plans (USFS, IDL).	Lump sum	1	12,500	-
Totals				**14,010**	**22,800**

27 REFERENCES

For convenience, references throughout this Technical Report are provided at the end of the individual sections rather than compiled in this section.

Appendix I

Feasibility Study Contributors and Professional Qualifications

CERTIFICATE OF QUALIFIED PERSON

Richard K Zimmerman

I, Richard K Zimmerman, R.G., do hereby certify that:

1. I am currently employed as a Registered Professional Geologist by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of Carleton College and received a Bachelor of Arts degree in Geology in 1976. I am also a graduate of the University of Michigan and received a M.Sc. degree in Geology 1980.

3. I am a:

   - Registered Professional Geology in good standing with the State of Arizona (No. 24064)
   - Registered Member in good standing of the Society for Mining, Metallurgy and Exploration, Inc. (No. 3612900RM)

4. I have practiced geology, mineral exploration, environmental remediation, and project management for 41 years. I have worked for mining and exploration companies for 9 years and engineering consulting firms for 22 years. The past 10 years have been spent with M3 Engineering & Technology Corporation managing, planning, and constructing processing plants for base and precious metals.

5. I have read National Instrument 43-101 (NI 43-101) and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

6. I have read the definition of "qualified person" set out NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I am responsible for Sections 1, 2, 3, 4, 5, 6, 7, 8, 9, 18 (excluding 18.8), 19, 20 (excluding 20.8), 21 (excluding 21.1.1, 21.1.6, 21.2.1), 22, 23, 24, 25, 26, and 27 of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho" (the "Technical Report"), with an effective date of December 22, 2020, prepared for Midas Gold Corporation.

8. I visited the project site on March 7, 2013. My prior involvement with the property that is subject of the Technical Report during the pre-feasibility study.

9. As of the effective date of the technical report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all tests in section 1.5 of National Instrument 43-101.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them of the Technical Report for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated this 27th day of January 2021.

(Signed) (Sealed) "*Richard K Zimmerman*"
Signature of Qualified Person

Richard K Zimmerman, M.Sc., R.G., SME-RM No. 3612900RM
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Art S. Ibrado

I, Art S. Ibrado, PhD, PE, do hereby certify that:

1. I am employed as a project manager and metallurgist at M3 Engineering & Technology Corp., 2051 W Sunset Rd, Suite 101, Tucson, AZ 85704, USA.
2. I graduated with the following degrees:

   Bachelor of Science in Metallurgical Engineering, University of the Philippines, 1980
   Master of Science (Metallurgy), University of California, Berkeley, 1986
   Doctor of Philosophy (Metallurgy), University of California, Berkeley, 1993

3. I am a registered professional engineer in the State of Arizona (No. 58140) and a qualified professional (QP) member of the Mining and Metallurgical Society of America (MMSA).
4. I have worked as a metallurgist in the academic and research setting for five years, excluding graduate school research, and in the mining industry for 13 years before joining M3 Engineering in 2009.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that, by reason of my education, professional engineer registration, affiliation with a professional association (as defined in NI 43-101) and past relevant experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for Section 17 of the technical report titled ""Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho" (the "Technical Report"), with an effective date of December 22, 2020, prepared for Midas Gold Corporation.
7. I have not visited the Stibnite Gold property.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Midas Gold Corp. as independence is described in Section 1.5 of NI 43-101 and do not own any of their stocks or shares.
10. I have had no prior involvement with the Stibnite Gold property.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.

(Signed and Sealed) "*Art S. Ibrado*"
Art S. Ibrado, PhD, PE

CERTIFICATE OF QUALIFIED PERSON

Grenvil Marquis Dunn

I, Grenvil Marquis Dunn, C. Eng., do hereby certify that:

1. I am the Director of:

   Hydromet WA Pty Ltd
   Unit 1806, 8 Adelaide Terrace, East Perth 6004, Western Australia, Australia

2. I graduated with a BSc Eng. (Honors) at University of Cape Town in 1970.

3. I am a Professional Engineer (ECSA registration number 740596) in good standing in South Africa, and C. Eng in United Kingdom in the areas of Metallurgical and Chemical Engineering. I am also registered as Fellow of Institute of Chemical Engineers, MSAIMM, Member TMS.

4. I have worked as a Metallurgical and Chemical Engineer for a total of 40+ years. My experience includes Pressure Leaching Operations and Plant Design, Project Management, Design and Management of complex hydrometallurgy testwork.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho", dated effective December 22, 2020 (the "Technical Report") prepared for Midas Gold Corporation; and am responsible for Sections 13.9 and 13.10 Hydrometallurgy. I have not visited the project site.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January, 2021.

(Signed) *Grenvil Marquis Dunn*
Signature of Qualified Person

Grenvil Marquis Dunn
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Garth Kirkham

I, Garth Kirkham, P.Geo, do hereby certify that:

1. I am currently employed as a Consulting Geoscientist and Principal for:
   Kirkham Geosystems Ltd.
   6331 Palace Place
   Burnaby, BC, Canada V5E 1Z6

2. I am a graduate of the University of Alberta in 1983 with a BSc. I have continuously practiced my profession since 1988. I have worked on and been involved with NI 43-101 studies such as the Fenn Gib, Kutcho, Adi Nefas, Debarwa, Tahuehueto, Demir deposits.

3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (EGBC).

4. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I am responsible for Sections 10, 11, 12, and 14 of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho", dated effective December 22, 2020 prepared for Midas Gold Corporation.

6. I have had prior involvement with the property that is the subject of the Technical Report since 2014. I was also a contributing author for the Stibnite Gold Project Prefeasibility Study Technical Report.

7. I have visited the Stibnite Gold property on April 23-25, 2014, July 14-15, 2014, January 12-14, 2017, and July 30-Aug1, 2018.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.

(Signed) (Sealed) *"Garth Kirkham"*
Signature of Qualified Person

Garth Kirkham, P.Geo.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Christopher Martin

I, Christopher Martin, MIMMM, C.Eng. do hereby certify that:

1. I am currently employed as Principal Metallurgist by Blue Coast Metallurgy, Ltd, 1020 Herring Gull Way, Parksville, BC V9P 1R2.
2. I hold degrees in Mineral Processing Technology from Camborne School of Mines (BSc(Hons)) (1984) and Metallurgical Engineering from McGill University (1988).
3. I am a full professional member of the Institute of Minerals, Materials, and Mining, in good standing since 1990.
4. I have practiced my profession in plant operations, in flowsheet development, plant design and optimization since 1984.
5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Section 13.1, 13.2, 13.3, 13.4, 13.5, 13.6, 13.7, 13.8, 13.11, 13.12, and 13.13 of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho", dated effective December 22, 2020 prepared for Midas Gold Corporation.
7. I have worked with Midas Gold Corporation on the project since 2010, providing metallurgical support to the development of the PEA and PFS Studies during this time.
8. I have visited the Stibnite Gold property on August 25, 2011 for one day.
9. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.

(Signed) *"Christopher Martin"*
Signature of Qualified Person

Christopher Martin, MIMMM, C.Eng.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Chris J. Roos

I, Christopher (Chris) J. Roos P.E., do hereby certify that:

1. I am currently a Consulting Engineer with Value Consulting, Inc. with an office at 580 Sundance Pl. Castle Pines, Colorado, U.S.A.

2. I am a graduate of Montana Tech in 2007 and 2008 with B.S. and M.S. (Mining Engineering) degrees, respectively. I have practiced my profession continuously since graduation in 2008 with experience including site-based roles, head office technical support for operating sites and projects, consulting, and as a faculty member at Montana Technological University. My principal focus is mine optimization, design, scheduling, and cost estimation, primarily in surface metal mining.

3. I am a Licensed Professional Engineer (P.E.) (Mining) in the State of Nevada (License #020978) and I am a Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) (Member #04140903).

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I am responsible for Sections 15, 21.1.1, and 21.2.1 of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho", dated effective December 22, 2020 prepared for Midas Gold Corporation.

6. I visited the Stibnite Gold property on October 6, 2017.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.

(Signed) (Sealed) *"Chris J. Roos"*
Signature of Qualified Person

Chris J. Roos, P.E.
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Scott D. Rosenthal

I, Scott D. Rosenthal, P.E., do hereby certify that:

1. I am currently a Consulting Engineer with Value Consulting, Inc. with an office at 580 Sundance Pl. Castle Pines, Colorado, U.S.A.

2. I am a graduate of Montana Tech in 1982 with B.S. (Mining Engineering) and 2010 M.S. (Project Engineering and Management) degrees. I have practiced my profession continuously since graduation in 1982 with experience including site-based roles, head office technical support for operating sites and projects, consulting, and as a faculty member at Montana Technological University. My principal focus is mine equipment selection and cost estimation, primarily in surface metal mining.

3. I am a Licensed Professional Engineer (P.E.) (Mining) in the State of Nevada (License #8739) and I am a Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) (Member #2764600).

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I am responsible for Section 16 of the technical report titled “Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho”, dated effective December 22, 2020 prepared for Midas Gold Corporation.

6. I visited the Stibnite Gold property on October 6, 2017.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.

(Signed) (Sealed) *“Scott D. Rosenthal”*
Signature of Qualified Person

Scott D. Rosenthal, P.E.
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Peter E. Kowalewski

I, Peter E. Kowalewski P.E., do hereby certify that:

1. I am currently employed as a Principal Engineer by Tierra Group International, Ltd. ("Tierra Group") with an office at 111 East Broadway, Suite 220, Salt Lake City, Utah, U.S.A.

2. I am a graduate of the Colorado School of Mines in 1992 and 1997 with B.Sc. (Geological Engineering) and **M.E.** (Applied Mechanics) degrees, respectively. I have practiced my profession continuously since graduation in 1992, focusing on the civil, geotechnical, hydrologic, and hydraulic design of facilities primarily for the mining industry. My primary focus has been on the design, permitting, construction, operation, and closure of mine waste containment facilities such as tailings impoundments, heap leach facilities, waste rock storage facilities, and appurtenant structures such as ponds and channels.

3. I am a Licensed Professional Engineer **(P.E.)** (Civil) in multiple States, including the State of Idaho (Idaho License #15289). In addition, I am a Registered Member of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) (Member #4055322RM).

4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

5. I am responsible for Sections 18.8, 20.8, and 21.1.6 of the technical report titled "Stibnite Gold Project, Feasibility Study Technical Report, Valley County, Idaho", dated effective December 22, 2020 prepared for Midas Gold Corporation.

6. I previously participated in the preparation of the preliminary feasibility study (PFS) for the Stibnite Gold Project, providing support for the tailings storage facility (TSF) design, water management, and closure/reclamation. Concurrent to work on the FS, I provided permitting support related to the tailings and water storage pond designs.

7. I visited the Stibnite Gold property on March 7, 2013.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 27th day of January 2021.
(Signed) (Sealed) *"Peter E. Kowalewski"*
Signature of Qualified Person

Peter E. Kowalewski, P.E.
Print name of Qualified Person

Appendix II

Property Description & Location

Figure II.1: Land Status Map

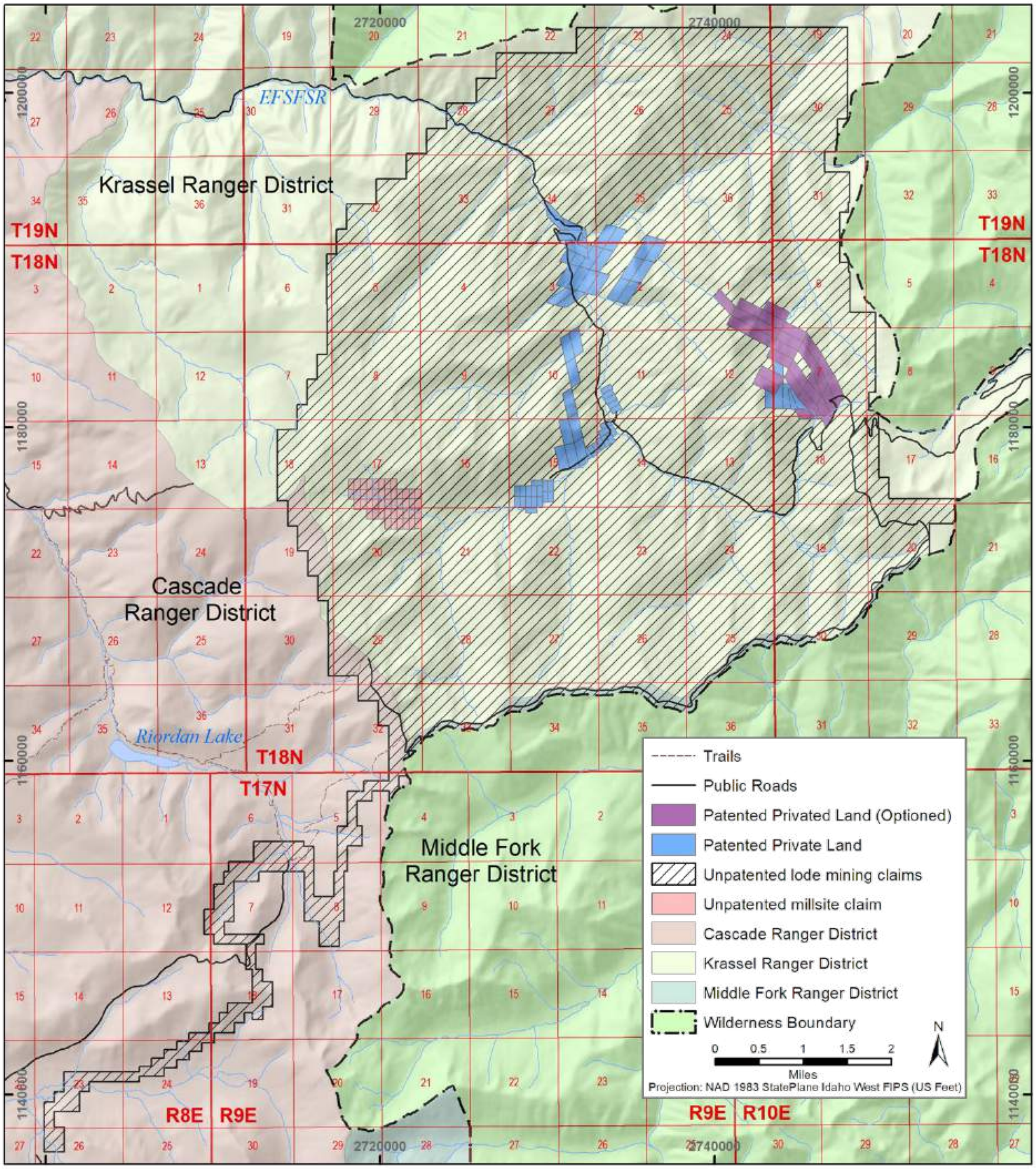

Table II.1: Mineral Concession Summary[5]

PATENTED CLAIMS							
Valley County Parcel ID	Owner	Number of Claims		Assessed Acres[4]	Assessed Hectares[4]	Property Tax 2020
		Lode	Millsite			
RP18N09E155300	IGRCLLC	-	16	80.00	32.37	$1,250.82
RP18N09E020026	IGRCLLC	6	-	129.82	52.54	$15.28
RP18N09E115495	IGRCLLC	-	14	53.57	21.68	$6,560.54
RP14N05E074475[1]	IGRCLLC[1]	-	-	25.06	10.14	$340.82
RP18N09E038995	SGC	4	-	81.63	33.03	$61.52
RP18N09E108995	SGC	5	-	102.8	41.60	$77.48
RP18N09E127345	SGC	6	-	99.87	40.42	$31.58
RP18N09E030005	SGC	11	-	218.90	85.59	$34.42
RP18N09E030020	SGC	6	-	81.17	32.85	$31.16
RP18N09E12255	SGC[2]	2	-	89.40	36.18	$67.40[2]
RP18N10E071525	SGC[2]	6	-	38.95	15.76	$29.36[2]
RP18N09E18150	SGC[2]	7	-	139.19	56.33	$104.92[2]
RP18N09E018435	SGC[2]	4	-	80.23	32.47	$60.46[2]
RP18N09E013840	SGC[2]	8	-	136.01	55.04	$102.52[2]
	Totals	**65**	**30**	**1,356.60**	**549.00**	**$8,403.62[3]**

UNPATENTED CLAIMS						
Owner	Claim Type	Number of Claims		Acres	Hectares	BLM Claims Fees
		Lode	Millsite			
IGRCLLC	Unpatented lode and millsite claims	1,464	46	28,317	11,460	$249,150
SGC[3]	Unpatented lode claims	8	-	165	67	$1,320
	Totals	**1,472**	**46**	**28,482**	**11,527**	**$250,470**

Notes:

1. *The Scott Valley parcel for the Stibnite Gold Logistics Facility is a 100% owned fee-simple parcel containing 25 acres more or less with no mineral rights and 2019 taxes of $340.82.*
2. *SGC has an option to purchase (OTP), but no ownership of these parcels. The owner pays property taxes for these parcels until the OTP is exercised.*
3. *Does not include taxes paid on OTP properties which are paid by owner.*
4. *Not all values may sum due to rounding errors. Assessed acreage may not correspond exactly to surveyed acreage reported in text.*
5. *This table summarizes the mineral rights held by Midas Gold Corp.'s wholly owned subsidiary, Idaho Gold Resources Company, LLC (**IGRCLLC**), and its wholly owned subsidiary, Stibnite Gold Company (**SGC**). For additional information on ownership see Section 4 in this Report.*

Table II.2: Mineral Concession Summary – Unpatented Claims Listing

Claim Name	IMC No.1	Claim Type	Owner
YP1	186740	Lode	SGC
YP2	186741	Lode	SGC
YP3	186742	Lode	SGC
YP4	186743	Lode	SGC
YP5	186744	Lode	SGC
YP6	186745	Lode	SGC
YP7	186746	Lode	SGC
YP8	186747	Lode	SGC
SF 1	189924	Lode	IGRCLLC
SF 2	189925	Lode	IGRCLLC
SF 3	189926	Lode	IGRCLLC
SF 4	189927	Lode	IGRCLLC
SF 5	189928	Lode	IGRCLLC
SF 6	189929	Lode	IGRCLLC
SF 7	189930	Lode	IGRCLLC
SF 8	189931	Lode	IGRCLLC
SF 9	189932	Lode	IGRCLLC
SF 10	189933	Lode	IGRCLLC
SF 11	189934	Lode	IGRCLLC
SF 12	189935	Lode	IGRCLLC
SF 13	189936	Lode	IGRCLLC
SF 14	189937	Lode	IGRCLLC
SF 15	189938	Lode	IGRCLLC
SF 16	189939	Lode	IGRCLLC
SF 17	189940	Lode	IGRCLLC
SF 18	189941	Lode	IGRCLLC
SF 19	189942	Lode	IGRCLLC
SF 20	189943	Lode	IGRCLLC
SF 21	189944	Lode	IGRCLLC
SF 22	189945	Lode	IGRCLLC
SF 23	189946	Lode	IGRCLLC
SF 24	189947	Lode	IGRCLLC
SF 25	189948	Lode	IGRCLLC
SF 26	189949	Lode	IGRCLLC
SF 27	189950	Lode	IGRCLLC
SF 28	189951	Lode	IGRCLLC
SF 29	189952	Lode	IGRCLLC
SF 30	189953	Lode	IGRCLLC
SF 31	189954	Lode	IGRCLLC
SF 32	189955	Lode	IGRCLLC
SF 33	189956	Lode	IGRCLLC
SF 34	189957	Lode	IGRCLLC
SF 35	189958	Lode	IGRCLLC
SF 36	189959	Lode	IGRCLLC
SF 37	189960	Lode	IGRCLLC
SF 89	190010	Lode	IGRCLLC
SF 90	190011	Lode	IGRCLLC
SF 91	190012	Lode	IGRCLLC
SF 92	190013	Lode	IGRCLLC
SF 93	190014	Lode	IGRCLLC
SF 94	190015	Lode	IGRCLLC
SF 38	189961	Lode	IGRCLLC
SF 39	189962	Lode	IGRCLLC
SF 40	189963	Lode	IGRCLLC
SF 41	189964	Lode	IGRCLLC
SF 42	189965	Lode	IGRCLLC
SF 43	189966	Lode	IGRCLLC
SF 44	189967	Lode	IGRCLLC
SF 45	189968	Lode	IGRCLLC
SF 46	189969	Lode	IGRCLLC
SF 47	189970	Lode	IGRCLLC
SF 48	189971	Lode	IGRCLLC
SF 49	189972	Lode	IGRCLLC
SF 50	189973	Lode	IGRCLLC
SF 52	189974	Lode	IGRCLLC
SF 53	189975	Lode	IGRCLLC
SF 54	189976	Lode	IGRCLLC
SF 55	189977	Lode	IGRCLLC
SF 56	189978	Lode	IGRCLLC
SF 57	189979	Lode	IGRCLLC
SF 58	189980	Lode	IGRCLLC
SF 59	189981	Lode	IGRCLLC
SF 61	189982	Lode	IGRCLLC
SF 62	189983	Lode	IGRCLLC
SF 65	189986	Lode	IGRCLLC
SF 66	189987	Lode	IGRCLLC
SF 67	189988	Lode	IGRCLLC
SF 68	189989	Lode	IGRCLLC
SF 69	189990	Lode	IGRCLLC
SF 70	189991	Lode	IGRCLLC
SF 73	189994	Lode	IGRCLLC
SF 74	189995	Lode	IGRCLLC
SF 75	189996	Lode	IGRCLLC
SF 76	189997	Lode	IGRCLLC
SF 77	189998	Lode	IGRCLLC
SF 78	189999	Lode	IGRCLLC
SF 79	190000	Lode	IGRCLLC
SF 80	190001	Lode	IGRCLLC
SF 81	190002	Lode	IGRCLLC
SF 82	190003	Lode	IGRCLLC
SF 83	190004	Lode	IGRCLLC
SF 84	190005	Lode	IGRCLLC
SF 85	190006	Lode	IGRCLLC
SF 86	190007	Lode	IGRCLLC
SF 87	190008	Lode	IGRCLLC
SF 88	190009	Lode	IGRCLLC
SFMS 11	190080	Millsite	IGRCLLC
SFMS 12	190081	Millsite	IGRCLLC
SFMS 13	190082	Millsite	IGRCLLC
SFMS 14	190083	Millsite	IGRCLLC
SFMS 15	190084	Millsite	IGRCLLC
SFMS 16	190085	Millsite	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 95	190016	Lode	IGRCLLC
SF 96	190017	Lode	IGRCLLC
SF 97	190018	Lode	IGRCLLC
SF 98	190019	Lode	IGRCLLC
SF 99	190020	Lode	IGRCLLC
SF 100	190021	Lode	IGRCLLC
SF 102	190023	Lode	IGRCLLC
SF 103	190024	Lode	IGRCLLC
SF 104	190025	Lode	IGRCLLC
SF 105	190026	Lode	IGRCLLC
SF 106	190027	Lode	IGRCLLC
SF 107	190028	Lode	IGRCLLC
SF 108	190029	Lode	IGRCLLC
SF 109	190030	Lode	IGRCLLC
SF 110	190031	Lode	IGRCLLC
SF 111	190032	Lode	IGRCLLC
SF 112	190033	Lode	IGRCLLC
SF 113	190034	Lode	IGRCLLC
SF 114	190035	Lode	IGRCLLC
SF 115	190036	Lode	IGRCLLC
SF 116	190037	Lode	IGRCLLC
SF 117	190038	Lode	IGRCLLC
SF 118	190039	Lode	IGRCLLC
SF 126	190041	Lode	IGRCLLC
SF 127	190042	Lode	IGRCLLC
SF 128	190043	Lode	IGRCLLC
SF 129	190044	Lode	IGRCLLC
SF 130	190045	Lode	IGRCLLC
SF 132	190047	Lode	IGRCLLC
SFMS 1	190070	Millsite	IGRCLLC
SFMS 2	190071	Millsite	IGRCLLC
SFMS 3	190072	Millsite	IGRCLLC
SFMS 4	190073	Millsite	IGRCLLC
SFMS 5	190074	Millsite	IGRCLLC
SFMS 6	190075	Millsite	IGRCLLC
SFMS 7	190076	Millsite	IGRCLLC
SFMS 8	190077	Millsite	IGRCLLC
SFMS 9	190078	Millsite	IGRCLLC
SFMS 10	190079	Millsite	IGRCLLC
SF 142	194747	Lode	IGRCLLC
SF 143	194748	Lode	IGRCLLC
SF 144	194749	Lode	IGRCLLC
SF 145	194750	Lode	IGRCLLC
SF 146	194751	Lode	IGRCLLC
SF 147	194752	Lode	IGRCLLC
SF 148	194753	Lode	IGRCLLC
SF 149	194754	Lode	IGRCLLC
SF 150	194755	Lode	IGRCLLC
SF 151	194756	Lode	IGRCLLC
SF 152	194757	Lode	IGRCLLC
SF 153	194758	Lode	IGRCLLC
SF 154	194759	Lode	IGRCLLC
SF 155	194760	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SFMS 17	190086	Millsite	IGRCLLC
SFMS 18	190087	Millsite	IGRCLLC
SFMS 19	190088	Millsite	IGRCLLC
SFMS 20	190089	Millsite	IGRCLLC
SFMS 21	190090	Millsite	IGRCLLC
SFMS 22	190091	Millsite	IGRCLLC
SFMS 23	190092	Millsite	IGRCLLC
SFMS 24	190093	Millsite	IGRCLLC
SFMS 25	190094	Millsite	IGRCLLC
SFMS 26	190095	Millsite	IGRCLLC
SFMS 27	190096	Millsite	IGRCLLC
SFMS 28	190097	Millsite	IGRCLLC
SFMS 29	190098	Millsite	IGRCLLC
SFMS 30	190099	Millsite	IGRCLLC
SFMS 31	190100	Millsite	IGRCLLC
SFMS 32	190101	Millsite	IGRCLLC
SFMS 33	190102	Millsite	IGRCLLC
SFMS 34	190103	Millsite	IGRCLLC
SFMS 35	190104	Millsite	IGRCLLC
SFMS 36	190105	Millsite	IGRCLLC
SFMS 37	190106	Millsite	IGRCLLC
SFMS 38	190107	Millsite	IGRCLLC
SFMS 39	190108	Millsite	IGRCLLC
SFMS 40	190109	Millsite	IGRCLLC
SFMS 41	190110	Millsite	IGRCLLC
SFMS 42	190111	Millsite	IGRCLLC
SFMS 43	190112	Millsite	IGRCLLC
SFMS 44	190113	Millsite	IGRCLLC
SFMS 45	190114	Millsite	IGRCLLC
SFMS 46	190115	Millsite	IGRCLLC
SF 133	194738	Lode	IGRCLLC
SF 134	194739	Lode	IGRCLLC
SF 135	194740	Lode	IGRCLLC
SF 136	194741	Lode	IGRCLLC
SF 137	194742	Lode	IGRCLLC
SF 138	194743	Lode	IGRCLLC
SF 139	194744	Lode	IGRCLLC
SF 140	194745	Lode	IGRCLLC
SF 141	194746	Lode	IGRCLLC
SF 187	194792	Lode	IGRCLLC
SF 188	194793	Lode	IGRCLLC
SF 189	194794	Lode	IGRCLLC
SF 190	194795	Lode	IGRCLLC
SF 191	194796	Lode	IGRCLLC
SF 63	199733	Lode	IGRCLLC
SF 64	199734	Lode	IGRCLLC
SF 71	199735	Lode	IGRCLLC
SF 72	199736	Lode	IGRCLLC
SF 101	199737	Lode	IGRCLLC
SF 125	199738	Lode	IGRCLLC
SF 131	199739	Lode	IGRCLLC
SF 192	199740	Lode	IGRCLLC
SF 193	199741	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 156	194761	Lode	IGRCLLC
SF 157	194762	Lode	IGRCLLC
SF 158	194763	Lode	IGRCLLC
SF 159	194764	Lode	IGRCLLC
SF 160	194765	Lode	IGRCLLC
SF 161	194766	Lode	IGRCLLC
SF 162	194767	Lode	IGRCLLC
SF 163	194768	Lode	IGRCLLC
SF 164	194769	Lode	IGRCLLC
SF 165	194770	Lode	IGRCLLC
SF 166	194771	Lode	IGRCLLC
SF 167	194772	Lode	IGRCLLC
SF 168	194773	Lode	IGRCLLC
SF 169	194774	Lode	IGRCLLC
SF 170	194775	Lode	IGRCLLC
SF 171	194776	Lode	IGRCLLC
SF 172	194777	Lode	IGRCLLC
SF 173	194778	Lode	IGRCLLC
SF 174	194779	Lode	IGRCLLC
SF 175	194780	Lode	IGRCLLC
SF 176	194781	Lode	IGRCLLC
SF 177	194782	Lode	IGRCLLC
SF 178	194783	Lode	IGRCLLC
SF 179	194784	Lode	IGRCLLC
SF 180	194785	Lode	IGRCLLC
SF 181	194786	Lode	IGRCLLC
SF 182	194787	Lode	IGRCLLC
SF 183	194788	Lode	IGRCLLC
SF 184	194789	Lode	IGRCLLC
SF 185	194790	Lode	IGRCLLC
SF 186	194791	Lode	IGRCLLC
SF 225	200328	Lode	IGRCLLC
SF 226	200329	Lode	IGRCLLC
SF 227	200330	Lode	IGRCLLC
SF 228	200331	Lode	IGRCLLC
SF 229	200332	Lode	IGRCLLC
SF 230	200333	Lode	IGRCLLC
SF 231	200334	Lode	IGRCLLC
SF 232	200335	Lode	IGRCLLC
SF 233	200336	Lode	IGRCLLC
SF 234	200337	Lode	IGRCLLC
SF 235	201078	Lode	IGRCLLC
SF 236	201079	Lode	IGRCLLC
SF 237	201080	Lode	IGRCLLC
SF 238	201081	Lode	IGRCLLC
SF 239	201082	Lode	IGRCLLC
SF 240	201083	Lode	IGRCLLC
SF 241	201084	Lode	IGRCLLC
SF 242	201085	Lode	IGRCLLC
SF 243	201086	Lode	IGRCLLC
SF 244	201087	Lode	IGRCLLC
SF 245	201088	Lode	IGRCLLC
SF 246	201089	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 194	199742	Lode	IGRCLLC
SF 195	199743	Lode	IGRCLLC
SF 196	199744	Lode	IGRCLLC
SF 197	199745	Lode	IGRCLLC
SF 198	199746	Lode	IGRCLLC
SF 199	199747	Lode	IGRCLLC
SF 200	199748	Lode	IGRCLLC
SF 201	199749	Lode	IGRCLLC
SF 202	199750	Lode	IGRCLLC
SF 203	199751	Lode	IGRCLLC
SF 204	199752	Lode	IGRCLLC
SF 205	199753	Lode	IGRCLLC
SF 206	199754	Lode	IGRCLLC
SF 207	199755	Lode	IGRCLLC
SF 208	199756	Lode	IGRCLLC
SF 209	199757	Lode	IGRCLLC
SF 210	199758	Lode	IGRCLLC
SF 211	199759	Lode	IGRCLLC
SF 212	199760	Lode	IGRCLLC
SF 213	199761	Lode	IGRCLLC
SF 214	199762	Lode	IGRCLLC
SF 215	199763	Lode	IGRCLLC
SF 216	199764	Lode	IGRCLLC
SF 217	199765	Lode	IGRCLLC
SF 218	199766	Lode	IGRCLLC
SF 219	199767	Lode	IGRCLLC
SF 220	199768	Lode	IGRCLLC
SF 221	199769	Lode	IGRCLLC
SF 222	199770	Lode	IGRCLLC
SF 223	200326	Lode	IGRCLLC
SF 224	200327	Lode	IGRCLLC
SF 270	201113	Lode	IGRCLLC
SF 271	201114	Lode	IGRCLLC
SF 272	201115	Lode	IGRCLLC
SF 273	201116	Lode	IGRCLLC
SF 274	201117	Lode	IGRCLLC
SF 275	201118	Lode	IGRCLLC
SF 276	201119	Lode	IGRCLLC
SF 277	201120	Lode	IGRCLLC
SF 278	201121	Lode	IGRCLLC
SF 279	201122	Lode	IGRCLLC
SF 280	201123	Lode	IGRCLLC
SF 281	201124	Lode	IGRCLLC
SF 282	201125	Lode	IGRCLLC
SF 283	201126	Lode	IGRCLLC
SF 284	201127	Lode	IGRCLLC
SF 285	201128	Lode	IGRCLLC
SF 286	201129	Lode	IGRCLLC
SF 287	201130	Lode	IGRCLLC
SF 288	201131	Lode	IGRCLLC
SF 289	201132	Lode	IGRCLLC
SF 290	201133	Lode	IGRCLLC
SF 291	201134	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 247	201090	Lode	IGRCLLC
SF 248	201091	Lode	IGRCLLC
SF 249	201092	Lode	IGRCLLC
SF 250	201093	Lode	IGRCLLC
SF 251	201094	Lode	IGRCLLC
SF 252	201095	Lode	IGRCLLC
SF 253	201096	Lode	IGRCLLC
SF 254	201097	Lode	IGRCLLC
SF 255	201098	Lode	IGRCLLC
SF 256	201099	Lode	IGRCLLC
SF 257	201100	Lode	IGRCLLC
SF 258	201101	Lode	IGRCLLC
SF 259	201102	Lode	IGRCLLC
SF 260	201103	Lode	IGRCLLC
SF 261	201104	Lode	IGRCLLC
SF 262	201105	Lode	IGRCLLC
SF 263	201106	Lode	IGRCLLC
SF 264	201107	Lode	IGRCLLC
SF 265	201108	Lode	IGRCLLC
SF 266	201109	Lode	IGRCLLC
SF 267	201110	Lode	IGRCLLC
SF 268	201111	Lode	IGRCLLC
SF 269	201112	Lode	IGRCLLC
SF 315	201158	Lode	IGRCLLC
SF 316	201159	Lode	IGRCLLC
SF 317	201160	Lode	IGRCLLC
SF 318	201161	Lode	IGRCLLC
SF 319	201162	Lode	IGRCLLC
SF 320	201163	Lode	IGRCLLC
SF 321	201164	Lode	IGRCLLC
SF 322	201165	Lode	IGRCLLC
SF 323	201166	Lode	IGRCLLC
SF 324	201167	Lode	IGRCLLC
SF 325	201168	Lode	IGRCLLC
SF 326	201169	Lode	IGRCLLC
SF 327	201170	Lode	IGRCLLC
SF 328	201171	Lode	IGRCLLC
SF 329	201172	Lode	IGRCLLC
SF 330	201173	Lode	IGRCLLC
SF 331	201174	Lode	IGRCLLC
SF 332	201175	Lode	IGRCLLC
SF 333	201176	Lode	IGRCLLC
SF 334	201177	Lode	IGRCLLC
SF 335	201178	Lode	IGRCLLC
SF 336	201179	Lode	IGRCLLC
SF 337	201180	Lode	IGRCLLC
SF 338	201181	Lode	IGRCLLC
SF 339	201182	Lode	IGRCLLC
SF 340	201183	Lode	IGRCLLC
SF 341	201184	Lode	IGRCLLC
SF 342	201185	Lode	IGRCLLC
SF 343	201186	Lode	IGRCLLC
SF 344	201187	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 292	201135	Lode	IGRCLLC
SF 293	201136	Lode	IGRCLLC
SF 294	201137	Lode	IGRCLLC
SF 295	201138	Lode	IGRCLLC
SF 296	201139	Lode	IGRCLLC
SF 297	201140	Lode	IGRCLLC
SF 298	201141	Lode	IGRCLLC
SF 299	201142	Lode	IGRCLLC
SF 300	201143	Lode	IGRCLLC
SF 301	201144	Lode	IGRCLLC
SF 302	201145	Lode	IGRCLLC
SF 303	201146	Lode	IGRCLLC
SF 304	201147	Lode	IGRCLLC
SF 305	201148	Lode	IGRCLLC
SF 306	201149	Lode	IGRCLLC
SF 307	201150	Lode	IGRCLLC
SF 308	201151	Lode	IGRCLLC
SF 309	201152	Lode	IGRCLLC
SF 310	201153	Lode	IGRCLLC
SF 311	201154	Lode	IGRCLLC
SF 312	201155	Lode	IGRCLLC
SF 313	201156	Lode	IGRCLLC
SF 314	201157	Lode	IGRCLLC
SF 360	201203	Lode	IGRCLLC
SF 361	201204	Lode	IGRCLLC
SF 362	201205	Lode	IGRCLLC
SF 363	201206	Lode	IGRCLLC
SF 364	201207	Lode	IGRCLLC
SF 365	201208	Lode	IGRCLLC
SF 366	201209	Lode	IGRCLLC
SF 367	201210	Lode	IGRCLLC
SF 368	201211	Lode	IGRCLLC
SF 369	201212	Lode	IGRCLLC
SF 370	201213	Lode	IGRCLLC
SF 371	201214	Lode	IGRCLLC
SF 372	201215	Lode	IGRCLLC
SF 373	201216	Lode	IGRCLLC
SF 374	201217	Lode	IGRCLLC
SF 375	201218	Lode	IGRCLLC
SF 376	201219	Lode	IGRCLLC
SF 377	201220	Lode	IGRCLLC
SF 378	201221	Lode	IGRCLLC
SF 379	201222	Lode	IGRCLLC
SF 380	201223	Lode	IGRCLLC
SF 381	201224	Lode	IGRCLLC
SF 382	201225	Lode	IGRCLLC
SF 383	201226	Lode	IGRCLLC
SF 384	201227	Lode	IGRCLLC
SF 385	201228	Lode	IGRCLLC
SF 386	201229	Lode	IGRCLLC
SF 387	201230	Lode	IGRCLLC
SF 388	201231	Lode	IGRCLLC
SF 389	201232	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 345	201188	Lode	IGRCLLC
SF 346	201189	Lode	IGRCLLC
SF 347	201190	Lode	IGRCLLC
SF 348	201191	Lode	IGRCLLC
SF 349	201192	Lode	IGRCLLC
SF 350	201193	Lode	IGRCLLC
SF 351	201194	Lode	IGRCLLC
SF 352	201195	Lode	IGRCLLC
SF 353	201196	Lode	IGRCLLC
SF 354	201197	Lode	IGRCLLC
SF 355	201198	Lode	IGRCLLC
SF 356	201199	Lode	IGRCLLC
SF 357	201200	Lode	IGRCLLC
SF 358	201201	Lode	IGRCLLC
SF 359	201202	Lode	IGRCLLC
SF 405	201248	Lode	IGRCLLC
SF 406	201249	Lode	IGRCLLC
SF 407	201250	Lode	IGRCLLC
SF 408	201251	Lode	IGRCLLC
SF 409	201252	Lode	IGRCLLC
SF 410	201253	Lode	IGRCLLC
SF 411	201254	Lode	IGRCLLC
SF 412	203035	Lode	IGRCLLC
SF 413	203036	Lode	IGRCLLC
SF 414	203037	Lode	IGRCLLC
SF 415	203038	Lode	IGRCLLC
SF 416	203039	Lode	IGRCLLC
SF 417	203040	Lode	IGRCLLC
SF 418	203041	Lode	IGRCLLC
SF 419	203042	Lode	IGRCLLC
SF 420	203043	Lode	IGRCLLC
SF 421	203044	Lode	IGRCLLC
SF 422	203045	Lode	IGRCLLC
SF 423	203046	Lode	IGRCLLC
SF 424	203047	Lode	IGRCLLC
SF 425	203048	Lode	IGRCLLC
SF 426	203049	Lode	IGRCLLC
SF 427	203050	Lode	IGRCLLC
SF 428	203051	Lode	IGRCLLC
SF 429	203052	Lode	IGRCLLC
SF 430	203053	Lode	IGRCLLC
SF 431	203054	Lode	IGRCLLC
SF 432	203055	Lode	IGRCLLC
SF 433	203056	Lode	IGRCLLC
SF 434	203057	Lode	IGRCLLC
SF 435	203058	Lode	IGRCLLC
SF 436	203059	Lode	IGRCLLC
SF 437	203060	Lode	IGRCLLC
SF 438	203061	Lode	IGRCLLC
SF 439	203062	Lode	IGRCLLC
SF 451	205314	Lode	IGRCLLC
SF 452	205315	Lode	IGRCLLC
SF 456	206796	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 390	201233	Lode	IGRCLLC
SF 391	201234	Lode	IGRCLLC
SF 392	201235	Lode	IGRCLLC
SF 393	201236	Lode	IGRCLLC
SF 394	201237	Lode	IGRCLLC
SF 395	201238	Lode	IGRCLLC
SF 396	201239	Lode	IGRCLLC
SF 397	201240	Lode	IGRCLLC
SF 398	201241	Lode	IGRCLLC
SF 399	201242	Lode	IGRCLLC
SF 400	201243	Lode	IGRCLLC
SF 401	201244	Lode	IGRCLLC
SF 402	201245	Lode	IGRCLLC
SF 403	201246	Lode	IGRCLLC
SF 404	201247	Lode	IGRCLLC
SF 464	206804	Lode	IGRCLLC
SF 465	206805	Lode	IGRCLLC
SF 466	206806	Lode	IGRCLLC
SF 467	206807	Lode	IGRCLLC
SF 468	206808	Lode	IGRCLLC
SF 469	206809	Lode	IGRCLLC
SF 470	206810	Lode	IGRCLLC
SF 471	206811	Lode	IGRCLLC
SF 472	206812	Lode	IGRCLLC
SF 473	206813	Lode	IGRCLLC
SF 474	206814	Lode	IGRCLLC
SF 475	206815	Lode	IGRCLLC
SF 476	206816	Lode	IGRCLLC
SF 477	206817	Lode	IGRCLLC
SF 478	206818	Lode	IGRCLLC
SF 479	206819	Lode	IGRCLLC
SF 480	206820	Lode	IGRCLLC
SF 481	206821	Lode	IGRCLLC
SF 482	206822	Lode	IGRCLLC
SF 483	206823	Lode	IGRCLLC
SF 484	206824	Lode	IGRCLLC
SF 485	206825	Lode	IGRCLLC
SF 486	206826	Lode	IGRCLLC
SF 487	206827	Lode	IGRCLLC
SF 488	206828	Lode	IGRCLLC
SF 489	206829	Lode	IGRCLLC
SF 490	206830	Lode	IGRCLLC
SF 491	206831	Lode	IGRCLLC
SF 492	206832	Lode	IGRCLLC
SF 493	206833	Lode	IGRCLLC
SF 494	206834	Lode	IGRCLLC
SF 495	206835	Lode	IGRCLLC
SF 496	206836	Lode	IGRCLLC
SF 497	206837	Lode	IGRCLLC
SF 498	206838	Lode	IGRCLLC
SF 499	206839	Lode	IGRCLLC
SF 500	206840	Lode	IGRCLLC
SF 501	206841	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 457	206797	Lode	IGRCLLC
SF 458	206798	Lode	IGRCLLC
SF 459	206799	Lode	IGRCLLC
SF 460	206800	Lode	IGRCLLC
SF 461	206801	Lode	IGRCLLC
SF 462	206802	Lode	IGRCLLC
SF 463	206803	Lode	IGRCLLC
SF 509	206849	Lode	IGRCLLC
SF 510	206850	Lode	IGRCLLC
SF 511	206851	Lode	IGRCLLC
SF 512	206852	Lode	IGRCLLC
SF 513	206853	Lode	IGRCLLC
SF 514	206854	Lode	IGRCLLC
SF 515	206855	Lode	IGRCLLC
SF 516	206856	Lode	IGRCLLC
SF 517	206857	Lode	IGRCLLC
SF 518	206858	Lode	IGRCLLC
SF 519	206859	Lode	IGRCLLC
SF 520	206860	Lode	IGRCLLC
SF 521	206861	Lode	IGRCLLC
SF 522	206862	Lode	IGRCLLC
SF 523	206863	Lode	IGRCLLC
SF 524	206864	Lode	IGRCLLC
SF 525	206865	Lode	IGRCLLC
SF 526	206866	Lode	IGRCLLC
SF 527	206867	Lode	IGRCLLC
SF 528	206868	Lode	IGRCLLC
SF 529	206869	Lode	IGRCLLC
SF 530	206870	Lode	IGRCLLC
SF 531	206871	Lode	IGRCLLC
SF 532	206872	Lode	IGRCLLC
SF 533	206873	Lode	IGRCLLC
SF 534	206874	Lode	IGRCLLC
SF 535	206875	Lode	IGRCLLC
SF 536	206876	Lode	IGRCLLC
SF 537	206877	Lode	IGRCLLC
SF 538	206878	Lode	IGRCLLC
SF 539	206879	Lode	IGRCLLC
SF 540	206880	Lode	IGRCLLC
SF 541	206881	Lode	IGRCLLC
SF 542	206882	Lode	IGRCLLC
SF 543	206883	Lode	IGRCLLC
SF 544	206884	Lode	IGRCLLC
SF 545	206885	Lode	IGRCLLC
SF 546	206886	Lode	IGRCLLC
SF 547	206887	Lode	IGRCLLC
SF 548	206888	Lode	IGRCLLC
SF 549	206889	Lode	IGRCLLC
SF 550	206890	Lode	IGRCLLC
SF 551	206891	Lode	IGRCLLC
SF 552	206892	Lode	IGRCLLC
SF 553	206893	Lode	IGRCLLC
SF 599	206939	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 502	206842	Lode	IGRCLLC
SF 503	206843	Lode	IGRCLLC
SF 504	206844	Lode	IGRCLLC
SF 505	206845	Lode	IGRCLLC
SF 506	206846	Lode	IGRCLLC
SF 507	206847	Lode	IGRCLLC
SF 508	206848	Lode	IGRCLLC
SF 554	206894	Lode	IGRCLLC
SF 555	206895	Lode	IGRCLLC
SF 556	206896	Lode	IGRCLLC
SF 557	206897	Lode	IGRCLLC
SF 558	206898	Lode	IGRCLLC
SF 559	206899	Lode	IGRCLLC
SF 560	206900	Lode	IGRCLLC
SF 561	206901	Lode	IGRCLLC
SF 562	206902	Lode	IGRCLLC
SF 563	206903	Lode	IGRCLLC
SF 564	206904	Lode	IGRCLLC
SF 565	206905	Lode	IGRCLLC
SF 566	206906	Lode	IGRCLLC
SF 567	206907	Lode	IGRCLLC
SF 568	206908	Lode	IGRCLLC
SF 569	206909	Lode	IGRCLLC
SF 570	206910	Lode	IGRCLLC
SF 571	206911	Lode	IGRCLLC
SF 572	206912	Lode	IGRCLLC
SF 573	206913	Lode	IGRCLLC
SF 574	206914	Lode	IGRCLLC
SF 575	206915	Lode	IGRCLLC
SF 576	206916	Lode	IGRCLLC
SF 577	206917	Lode	IGRCLLC
SF 578	206918	Lode	IGRCLLC
SF 579	206919	Lode	IGRCLLC
SF 580	206920	Lode	IGRCLLC
SF 581	206921	Lode	IGRCLLC
SF 582	206922	Lode	IGRCLLC
SF 583	206923	Lode	IGRCLLC
SF 584	206924	Lode	IGRCLLC
SF 585	206925	Lode	IGRCLLC
SF 586	206926	Lode	IGRCLLC
SF 587	206927	Lode	IGRCLLC
SF 588	206928	Lode	IGRCLLC
SF 589	206929	Lode	IGRCLLC
SF 590	206930	Lode	IGRCLLC
SF 591	206931	Lode	IGRCLLC
SF 592	206932	Lode	IGRCLLC
SF 593	206933	Lode	IGRCLLC
SF 594	206934	Lode	IGRCLLC
SF 595	206935	Lode	IGRCLLC
SF 596	206936	Lode	IGRCLLC
SF 597	206937	Lode	IGRCLLC
SF 598	206938	Lode	IGRCLLC
SF 765	206984	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 721	206940	Lode	IGRCLLC
SF 722	206941	Lode	IGRCLLC
SF 723	206942	Lode	IGRCLLC
SF 724	206943	Lode	IGRCLLC
SF 725	206944	Lode	IGRCLLC
SF 726	206945	Lode	IGRCLLC
SF 727	206946	Lode	IGRCLLC
SF 728	206947	Lode	IGRCLLC
SF 729	206948	Lode	IGRCLLC
SF 730	206949	Lode	IGRCLLC
SF 731	206950	Lode	IGRCLLC
SF 732	206951	Lode	IGRCLLC
SF 733	206952	Lode	IGRCLLC
SF 734	206953	Lode	IGRCLLC
SF 735	206954	Lode	IGRCLLC
SF 736	206955	Lode	IGRCLLC
SF 737	206956	Lode	IGRCLLC
SF 738	206957	Lode	IGRCLLC
SF 739	206958	Lode	IGRCLLC
SF 740	206959	Lode	IGRCLLC
SF 741	206960	Lode	IGRCLLC
SF 742	206961	Lode	IGRCLLC
SF 743	206962	Lode	IGRCLLC
SF 744	206963	Lode	IGRCLLC
SF 745	206964	Lode	IGRCLLC
SF 746	206965	Lode	IGRCLLC
SF 747	206966	Lode	IGRCLLC
SF 748	206967	Lode	IGRCLLC
SF 749	206968	Lode	IGRCLLC
SF 750	206969	Lode	IGRCLLC
SF 751	206970	Lode	IGRCLLC
SF 752	206971	Lode	IGRCLLC
SF 753	206972	Lode	IGRCLLC
SF 754	206973	Lode	IGRCLLC
SF 755	206974	Lode	IGRCLLC
SF 756	206975	Lode	IGRCLLC
SF 757	206976	Lode	IGRCLLC
SF 758	206977	Lode	IGRCLLC
SF 759	206978	Lode	IGRCLLC
SF 760	206979	Lode	IGRCLLC
SF 761	206980	Lode	IGRCLLC
SF 762	206981	Lode	IGRCLLC
SF 763	206982	Lode	IGRCLLC
SF 764	206983	Lode	IGRCLLC
SF 622	207029	Lode	IGRCLLC
SF 623	207030	Lode	IGRCLLC
SF 624	207031	Lode	IGRCLLC
SF 625	207032	Lode	IGRCLLC
SF 626	207033	Lode	IGRCLLC
SF 627	207034	Lode	IGRCLLC
SF 628	207035	Lode	IGRCLLC
SF 629	207036	Lode	IGRCLLC
SF 630	207037	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 766	206985	Lode	IGRCLLC
SF 767	206986	Lode	IGRCLLC
SF 768	206987	Lode	IGRCLLC
SF 769	206988	Lode	IGRCLLC
SF 770	206989	Lode	IGRCLLC
SF 771	206990	Lode	IGRCLLC
SF 772	206991	Lode	IGRCLLC
SF 773	206992	Lode	IGRCLLC
SF 774	206993	Lode	IGRCLLC
SF 775	206994	Lode	IGRCLLC
SF 776	206995	Lode	IGRCLLC
SF 777	206996	Lode	IGRCLLC
SF 778	206997	Lode	IGRCLLC
SF 779	206998	Lode	IGRCLLC
SF 780	206999	Lode	IGRCLLC
SF 781	207000	Lode	IGRCLLC
SF 782	207001	Lode	IGRCLLC
SF 783	207002	Lode	IGRCLLC
SF 784	207003	Lode	IGRCLLC
SF 785	207004	Lode	IGRCLLC
SF 786	207005	Lode	IGRCLLC
SF 787	207006	Lode	IGRCLLC
SF 600	207007	Lode	IGRCLLC
SF 601	207008	Lode	IGRCLLC
SF 602	207009	Lode	IGRCLLC
SF 603	207010	Lode	IGRCLLC
SF 604	207011	Lode	IGRCLLC
SF 605	207012	Lode	IGRCLLC
SF 606	207013	Lode	IGRCLLC
SF 607	207014	Lode	IGRCLLC
SF 608	207015	Lode	IGRCLLC
SF 609	207016	Lode	IGRCLLC
SF 610	207017	Lode	IGRCLLC
SF 611	207018	Lode	IGRCLLC
SF 612	207019	Lode	IGRCLLC
SF 613	207020	Lode	IGRCLLC
SF 614	207021	Lode	IGRCLLC
SF 615	207022	Lode	IGRCLLC
SF 616	207023	Lode	IGRCLLC
SF 617	207024	Lode	IGRCLLC
SF 618	207025	Lode	IGRCLLC
SF 619	207026	Lode	IGRCLLC
SF 620	207027	Lode	IGRCLLC
SF 621	207028	Lode	IGRCLLC
SF 662	207074	Lode	IGRCLLC
SF 663	207075	Lode	IGRCLLC
SF 664	207076	Lode	IGRCLLC
SF 665	207077	Lode	IGRCLLC
SF 666	207078	Lode	IGRCLLC
SF 667	207079	Lode	IGRCLLC
SF 668	207080	Lode	IGRCLLC
SF 669	207081	Lode	IGRCLLC
SF 670	207082	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 631	207038	Lode	IGRCLLC
SF 632	207039	Lode	IGRCLLC
SF 633	207040	Lode	IGRCLLC
SF 634	207041	Lode	IGRCLLC
SF 635	207042	Lode	IGRCLLC
SF 636	207043	Lode	IGRCLLC
SF 637	207044	Lode	IGRCLLC
SF 638	207045	Lode	IGRCLLC
SF 641	207046	Lode	IGRCLLC
SF 642	207047	Lode	IGRCLLC
SF 643	207048	Lode	IGRCLLC
SF 644	207049	Lode	IGRCLLC
SF 645	207050	Lode	IGRCLLC
SF 646	207051	Lode	IGRCLLC
SF 647	207052	Lode	IGRCLLC
SF 648	207053	Lode	IGRCLLC
SF 649	207054	Lode	IGRCLLC
SF 803	207055	Lode	IGRCLLC
SF 804	207056	Lode	IGRCLLC
SF 805	207057	Lode	IGRCLLC
SF 806	207058	Lode	IGRCLLC
SF 807	207059	Lode	IGRCLLC
SF 808	207060	Lode	IGRCLLC
SF 809	207061	Lode	IGRCLLC
SF 650	207062	Lode	IGRCLLC
SF 651	207063	Lode	IGRCLLC
SF 652	207064	Lode	IGRCLLC
SF 653	207065	Lode	IGRCLLC
SF 654	207066	Lode	IGRCLLC
SF 655	207067	Lode	IGRCLLC
SF 656	207068	Lode	IGRCLLC
SF 657	207069	Lode	IGRCLLC
SF 658	207070	Lode	IGRCLLC
SF 659	207071	Lode	IGRCLLC
SF 660	207072	Lode	IGRCLLC
SF 661	207073	Lode	IGRCLLC
SF 829	207119	Lode	IGRCLLC
SF 830	207120	Lode	IGRCLLC
SF 831	207121	Lode	IGRCLLC
SF 832	207122	Lode	IGRCLLC
SF 833	207123	Lode	IGRCLLC
SF 848	207124	Lode	IGRCLLC
SF 849	207125	Lode	IGRCLLC
SF 850	207126	Lode	IGRCLLC
SF 851	207127	Lode	IGRCLLC
SF 852	207128	Lode	IGRCLLC
SF 853	207129	Lode	IGRCLLC
SF 854	207130	Lode	IGRCLLC
SF 855	207131	Lode	IGRCLLC
SF 868	207132	Lode	IGRCLLC
SF 869	207133	Lode	IGRCLLC
SF 870	207134	Lode	IGRCLLC
SF 871	207135	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 671	207083	Lode	IGRCLLC
SF 672	207084	Lode	IGRCLLC
SF 673	207085	Lode	IGRCLLC
SF 674	207086	Lode	IGRCLLC
SF 675	207087	Lode	IGRCLLC
SF 676	207088	Lode	IGRCLLC
SF 677	207089	Lode	IGRCLLC
SF 678	207090	Lode	IGRCLLC
SF 679	207091	Lode	IGRCLLC
SF 680	207092	Lode	IGRCLLC
SF 681	207093	Lode	IGRCLLC
SF 682	207094	Lode	IGRCLLC
SF 683	207095	Lode	IGRCLLC
SF 684	207096	Lode	IGRCLLC
SF 685	207097	Lode	IGRCLLC
SF 686	207098	Lode	IGRCLLC
SF 687	207099	Lode	IGRCLLC
SF 688	207100	Lode	IGRCLLC
SF 689	207101	Lode	IGRCLLC
SF 690	207102	Lode	IGRCLLC
SF 691	207103	Lode	IGRCLLC
SF 692	207104	Lode	IGRCLLC
SF 693	207105	Lode	IGRCLLC
SF 694	207106	Lode	IGRCLLC
SF 695	207107	Lode	IGRCLLC
SF 696	207108	Lode	IGRCLLC
SF 697	207109	Lode	IGRCLLC
SF 698	207110	Lode	IGRCLLC
SF 699	207111	Lode	IGRCLLC
SF 700	207112	Lode	IGRCLLC
SF 701	207113	Lode	IGRCLLC
SF 702	207114	Lode	IGRCLLC
SF 703	207115	Lode	IGRCLLC
SF 826	207116	Lode	IGRCLLC
SF 827	207117	Lode	IGRCLLC
SF 828	207118	Lode	IGRCLLC
SF 790	207193	Lode	IGRCLLC
SF 791	207194	Lode	IGRCLLC
SF 792	207195	Lode	IGRCLLC
SF 793	207196	Lode	IGRCLLC
SF 794	207197	Lode	IGRCLLC
SF 795	207198	Lode	IGRCLLC
SF 796	207199	Lode	IGRCLLC
SF 797	207200	Lode	IGRCLLC
SF 798	207201	Lode	IGRCLLC
SF 799	207202	Lode	IGRCLLC
SF 800	207203	Lode	IGRCLLC
SF 801	207204	Lode	IGRCLLC
SF 802	207205	Lode	IGRCLLC
SF 810	207206	Lode	IGRCLLC
SF 811	207207	Lode	IGRCLLC
SF 812	207208	Lode	IGRCLLC
SF 813	207209	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 872	207136	Lode	IGRCLLC
SF 873	207137	Lode	IGRCLLC
SF 874	207138	Lode	IGRCLLC
SF 875	207139	Lode	IGRCLLC
SF 920	207140	Lode	IGRCLLC
SF 921	207141	Lode	IGRCLLC
SF 1356	207145	Lode	IGRCLLC
SF 1357	207146	Lode	IGRCLLC
SF 1358	207147	Lode	IGRCLLC
SF 704	207174	Lode	IGRCLLC
SF 705	207175	Lode	IGRCLLC
SF 706	207176	Lode	IGRCLLC
SF 707	207177	Lode	IGRCLLC
SF 708	207178	Lode	IGRCLLC
SF 709	207179	Lode	IGRCLLC
SF 710	207180	Lode	IGRCLLC
SF 711	207181	Lode	IGRCLLC
SF 712	207182	Lode	IGRCLLC
SF 713	207183	Lode	IGRCLLC
SF 714	207184	Lode	IGRCLLC
SF 715	207185	Lode	IGRCLLC
SF 716	207186	Lode	IGRCLLC
SF 717	207187	Lode	IGRCLLC
SF 718	207188	Lode	IGRCLLC
SF 719	207189	Lode	IGRCLLC
SF 720	207190	Lode	IGRCLLC
SF 788	207191	Lode	IGRCLLC
SF 789	207192	Lode	IGRCLLC
SF 858	207238	Lode	IGRCLLC
SF 859	207239	Lode	IGRCLLC
SF 860	207240	Lode	IGRCLLC
SF 861	207241	Lode	IGRCLLC
SF 862	207242	Lode	IGRCLLC
SF 863	207243	Lode	IGRCLLC
SF 864	207244	Lode	IGRCLLC
SF 865	207245	Lode	IGRCLLC
SF 866	207246	Lode	IGRCLLC
SF 867	207247	Lode	IGRCLLC
SF 876	207248	Lode	IGRCLLC
SF 877	207249	Lode	IGRCLLC
SF 878	207250	Lode	IGRCLLC
SF 879	207251	Lode	IGRCLLC
SF 880	207252	Lode	IGRCLLC
SF 881	207253	Lode	IGRCLLC
SF 882	207254	Lode	IGRCLLC
SF 883	207255	Lode	IGRCLLC
SF 884	207256	Lode	IGRCLLC
SF 885	207257	Lode	IGRCLLC
SF 886	207258	Lode	IGRCLLC
SF 887	207259	Lode	IGRCLLC
SF 888	207260	Lode	IGRCLLC
SF 889	207261	Lode	IGRCLLC
SF 890	207262	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 814	207210	Lode	IGRCLLC
SF 815	207211	Lode	IGRCLLC
SF 816	207212	Lode	IGRCLLC
SF 817	207213	Lode	IGRCLLC
SF 818	207214	Lode	IGRCLLC
SF 819	207215	Lode	IGRCLLC
SF 820	207216	Lode	IGRCLLC
SF 821	207217	Lode	IGRCLLC
SF 822	207218	Lode	IGRCLLC
SF 823	207219	Lode	IGRCLLC
SF 824	207220	Lode	IGRCLLC
SF 825	207221	Lode	IGRCLLC
SF 834	207222	Lode	IGRCLLC
SF 835	207223	Lode	IGRCLLC
SF 836	207224	Lode	IGRCLLC
SF 837	207225	Lode	IGRCLLC
SF 838	207226	Lode	IGRCLLC
SF 839	207227	Lode	IGRCLLC
SF 840	207228	Lode	IGRCLLC
SF 841	207229	Lode	IGRCLLC
SF 842	207230	Lode	IGRCLLC
SF 843	207231	Lode	IGRCLLC
SF 844	207232	Lode	IGRCLLC
SF 845	207233	Lode	IGRCLLC
SF 846	207234	Lode	IGRCLLC
SF 847	207235	Lode	IGRCLLC
SF 856	207236	Lode	IGRCLLC
SF 857	207237	Lode	IGRCLLC
SF 911	207283	Lode	IGRCLLC
SF 912	207284	Lode	IGRCLLC
SF 913	207285	Lode	IGRCLLC
SF 914	207286	Lode	IGRCLLC
SF 915	207287	Lode	IGRCLLC
SF 916	207288	Lode	IGRCLLC
SF 917	207289	Lode	IGRCLLC
SF 918	207290	Lode	IGRCLLC
SF 919	207291	Lode	IGRCLLC
SF 922	207292	Lode	IGRCLLC
SF 923	207293	Lode	IGRCLLC
SF 924	207294	Lode	IGRCLLC
SF 925	207295	Lode	IGRCLLC
SF 926	207296	Lode	IGRCLLC
SF 927	207297	Lode	IGRCLLC
SF 928	207298	Lode	IGRCLLC
SF 929	207299	Lode	IGRCLLC
SF 930	207300	Lode	IGRCLLC
SF 931	207301	Lode	IGRCLLC
SF 932	207302	Lode	IGRCLLC
SF 933	207303	Lode	IGRCLLC
SF 934	207304	Lode	IGRCLLC
SF 935	207305	Lode	IGRCLLC
SF 936	207306	Lode	IGRCLLC
SF 937	207307	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 891	207263	Lode	IGRCLLC
SF 892	207264	Lode	IGRCLLC
SF 893	207265	Lode	IGRCLLC
SF 894	207266	Lode	IGRCLLC
SF 895	207267	Lode	IGRCLLC
SF 896	207268	Lode	IGRCLLC
SF 897	207269	Lode	IGRCLLC
SF 898	207270	Lode	IGRCLLC
SF 899	207271	Lode	IGRCLLC
SF 900	207272	Lode	IGRCLLC
SF 901	207273	Lode	IGRCLLC
SF 902	207274	Lode	IGRCLLC
SF 903	207275	Lode	IGRCLLC
SF 904	207276	Lode	IGRCLLC
SF 905	207277	Lode	IGRCLLC
SF 906	207278	Lode	IGRCLLC
SF 907	207279	Lode	IGRCLLC
SF 908	207280	Lode	IGRCLLC
SF 909	207281	Lode	IGRCLLC
SF 910	207282	Lode	IGRCLLC
SF 958	207328	Lode	IGRCLLC
SF 959	207329	Lode	IGRCLLC
SF 960	207330	Lode	IGRCLLC
SF 961	207331	Lode	IGRCLLC
SF 962	207332	Lode	IGRCLLC
SF 963	207333	Lode	IGRCLLC
SF 964	207334	Lode	IGRCLLC
SF 965	207335	Lode	IGRCLLC
SF 966	207336	Lode	IGRCLLC
SF 967	207337	Lode	IGRCLLC
SF 968	207338	Lode	IGRCLLC
SF 969	207339	Lode	IGRCLLC
SF 970	207340	Lode	IGRCLLC
SF 971	207341	Lode	IGRCLLC
SF 972	207342	Lode	IGRCLLC
SF 973	207343	Lode	IGRCLLC
SF 974	207344	Lode	IGRCLLC
SF 975	207345	Lode	IGRCLLC
SF 976	207346	Lode	IGRCLLC
SF 977	207347	Lode	IGRCLLC
SF 978	207348	Lode	IGRCLLC
SF 979	207349	Lode	IGRCLLC
SF 980	207350	Lode	IGRCLLC
SF 981	207351	Lode	IGRCLLC
SF 982	207352	Lode	IGRCLLC
SF 983	207353	Lode	IGRCLLC
SF 984	207354	Lode	IGRCLLC
SF 985	207355	Lode	IGRCLLC
SF 986	207356	Lode	IGRCLLC
SF 987	207357	Lode	IGRCLLC
SF 988	207358	Lode	IGRCLLC
SF 989	207359	Lode	IGRCLLC
SF 990	207360	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 938	207308	Lode	IGRCLLC
SF 939	207309	Lode	IGRCLLC
SF 940	207310	Lode	IGRCLLC
SF 941	207311	Lode	IGRCLLC
SF 942	207312	Lode	IGRCLLC
SF 943	207313	Lode	IGRCLLC
SF 944	207314	Lode	IGRCLLC
SF 945	207315	Lode	IGRCLLC
SF 946	207316	Lode	IGRCLLC
SF 947	207317	Lode	IGRCLLC
SF 948	207318	Lode	IGRCLLC
SF 949	207319	Lode	IGRCLLC
SF 950	207320	Lode	IGRCLLC
SF 951	207321	Lode	IGRCLLC
SF 952	207322	Lode	IGRCLLC
SF 953	207323	Lode	IGRCLLC
SF 954	207324	Lode	IGRCLLC
SF 955	207325	Lode	IGRCLLC
SF 956	207326	Lode	IGRCLLC
SF 957	207327	Lode	IGRCLLC
SF 1003	207373	Lode	IGRCLLC
SF 1004	207374	Lode	IGRCLLC
SF 1005	207375	Lode	IGRCLLC
SF 1006	207376	Lode	IGRCLLC
SF 1007	207377	Lode	IGRCLLC
SF 1008	207378	Lode	IGRCLLC
SF 1009	207379	Lode	IGRCLLC
SF 1010	207380	Lode	IGRCLLC
SF 1011	207381	Lode	IGRCLLC
SF 1012	207382	Lode	IGRCLLC
SF 1013	207383	Lode	IGRCLLC
SF 1014	207384	Lode	IGRCLLC
SF 1015	207385	Lode	IGRCLLC
SF 1016	207386	Lode	IGRCLLC
SF 1017	207387	Lode	IGRCLLC
SF 1018	207388	Lode	IGRCLLC
SF 1019	207389	Lode	IGRCLLC
SF 1020	207390	Lode	IGRCLLC
SF 1021	207391	Lode	IGRCLLC
SF 1022	207392	Lode	IGRCLLC
SF 1023	207393	Lode	IGRCLLC
SF 1024	207394	Lode	IGRCLLC
SF 1025	207395	Lode	IGRCLLC
SF 1026	207396	Lode	IGRCLLC
SF 1027	207397	Lode	IGRCLLC
SF 1028	207398	Lode	IGRCLLC
SF 1029	207399	Lode	IGRCLLC
SF 1030	207400	Lode	IGRCLLC
SF 1031	207401	Lode	IGRCLLC
SF 1032	207402	Lode	IGRCLLC
SF 1033	207403	Lode	IGRCLLC
SF 1034	207404	Lode	IGRCLLC
SF 1035	207405	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 991	207361	Lode	IGRCLLC
SF 992	207362	Lode	IGRCLLC
SF 993	207363	Lode	IGRCLLC
SF 994	207364	Lode	IGRCLLC
SF 995	207365	Lode	IGRCLLC
SF 996	207366	Lode	IGRCLLC
SF 997	207367	Lode	IGRCLLC
SF 998	207368	Lode	IGRCLLC
SF 999	207369	Lode	IGRCLLC
SF 1000	207370	Lode	IGRCLLC
SF 1001	207371	Lode	IGRCLLC
SF 1002	207372	Lode	IGRCLLC
SF 1048	207418	Lode	IGRCLLC
SF 1049	207419	Lode	IGRCLLC
SF 1050	207420	Lode	IGRCLLC
SF 1051	207421	Lode	IGRCLLC
SF 1052	207422	Lode	IGRCLLC
SF 1053	207423	Lode	IGRCLLC
SF 1054	207424	Lode	IGRCLLC
SF 1055	207425	Lode	IGRCLLC
SF 1056	207426	Lode	IGRCLLC
SF 1057	207427	Lode	IGRCLLC
SF 1058	207428	Lode	IGRCLLC
SF 1059	207429	Lode	IGRCLLC
SF 1060	207430	Lode	IGRCLLC
SF 1061	207431	Lode	IGRCLLC
SF 1062	207432	Lode	IGRCLLC
SF 1063	207433	Lode	IGRCLLC
SF 1064	207434	Lode	IGRCLLC
SF 1065	207435	Lode	IGRCLLC
SF 1066	207436	Lode	IGRCLLC
SF 1067	207437	Lode	IGRCLLC
SF 1068	207438	Lode	IGRCLLC
SF 1069	207439	Lode	IGRCLLC
SF 1070	207440	Lode	IGRCLLC
SF 1071	207441	Lode	IGRCLLC
SF 1072	207442	Lode	IGRCLLC
SF 1073	207443	Lode	IGRCLLC
SF 1074	207444	Lode	IGRCLLC
SF 1075	207445	Lode	IGRCLLC
SF 1076	207446	Lode	IGRCLLC
SF 1077	207447	Lode	IGRCLLC
SF 1078	207448	Lode	IGRCLLC
SF 1079	207449	Lode	IGRCLLC
SF 1080	207450	Lode	IGRCLLC
SF 1081	207451	Lode	IGRCLLC
SF 1082	207452	Lode	IGRCLLC
SF 1083	207453	Lode	IGRCLLC
SF 1084	207454	Lode	IGRCLLC
SF 1085	207455	Lode	IGRCLLC
SF 1086	207456	Lode	IGRCLLC
SF 1087	207457	Lode	IGRCLLC
SF 1088	207458	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1036	207406	Lode	IGRCLLC
SF 1037	207407	Lode	IGRCLLC
SF 1038	207408	Lode	IGRCLLC
SF 1039	207409	Lode	IGRCLLC
SF 1040	207410	Lode	IGRCLLC
SF 1041	207411	Lode	IGRCLLC
SF 1042	207412	Lode	IGRCLLC
SF 1043	207413	Lode	IGRCLLC
SF 1044	207414	Lode	IGRCLLC
SF 1045	207415	Lode	IGRCLLC
SF 1046	207416	Lode	IGRCLLC
SF 1047	207417	Lode	IGRCLLC
SF 1093	207463	Lode	IGRCLLC
SF 1094	207464	Lode	IGRCLLC
SF 1095	207465	Lode	IGRCLLC
SF 1096	207466	Lode	IGRCLLC
SF 1097	207467	Lode	IGRCLLC
SF 1098	207468	Lode	IGRCLLC
SF 1099	207469	Lode	IGRCLLC
SF 1100	207470	Lode	IGRCLLC
SF 1101	207471	Lode	IGRCLLC
SF 1102	207472	Lode	IGRCLLC
SF 1103	207473	Lode	IGRCLLC
SF 1104	207474	Lode	IGRCLLC
SF 1105	207475	Lode	IGRCLLC
SF 1106	207476	Lode	IGRCLLC
SF 1107	207477	Lode	IGRCLLC
SF 1108	207478	Lode	IGRCLLC
SF 1109	207479	Lode	IGRCLLC
SF 1110	207480	Lode	IGRCLLC
SF 1111	207481	Lode	IGRCLLC
SF 1112	207482	Lode	IGRCLLC
SF 1113	207483	Lode	IGRCLLC
SF 1114	207484	Lode	IGRCLLC
SF 1115	207485	Lode	IGRCLLC
SF 1116	207486	Lode	IGRCLLC
SF 1117	207487	Lode	IGRCLLC
SF 1118	207488	Lode	IGRCLLC
SF 1119	207489	Lode	IGRCLLC
SF 1120	207490	Lode	IGRCLLC
SF 1121	207491	Lode	IGRCLLC
SF 1122	207492	Lode	IGRCLLC
SF 1123	207493	Lode	IGRCLLC
SF 1124	207494	Lode	IGRCLLC
SF 1125	207495	Lode	IGRCLLC
SF 1126	207496	Lode	IGRCLLC
SF 1127	207497	Lode	IGRCLLC
SF 1128	207498	Lode	IGRCLLC
SF 1129	207499	Lode	IGRCLLC
SF 1130	207500	Lode	IGRCLLC
SF 1131	207501	Lode	IGRCLLC
SF 1132	207502	Lode	IGRCLLC
SF 1133	207503	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner	Claim Name	IMC No.1	Claim Type	Owner
SF 1089	207459	Lode	IGRCLLC	SF 1134	207504	Lode	IGRCLLC
SF 1090	207460	Lode	IGRCLLC	SF 1135	207505	Lode	IGRCLLC
SF 1091	207461	Lode	IGRCLLC	SF 1136	207506	Lode	IGRCLLC
SF 1092	207462	Lode	IGRCLLC	SF 1137	207507	Lode	IGRCLLC
SF 1138	207508	Lode	IGRCLLC	SF 1183	207553	Lode	IGRCLLC
SF 1139	207509	Lode	IGRCLLC	SF 1184	207554	Lode	IGRCLLC
SF 1140	207510	Lode	IGRCLLC	SF 1185	207555	Lode	IGRCLLC
SF 1141	207511	Lode	IGRCLLC	SF 1186	207556	Lode	IGRCLLC
SF 1142	207512	Lode	IGRCLLC	SF 1187	207557	Lode	IGRCLLC
SF 1143	207513	Lode	IGRCLLC	SF 1188	207558	Lode	IGRCLLC
SF 1144	207514	Lode	IGRCLLC	SF 1189	207559	Lode	IGRCLLC
SF 1145	207515	Lode	IGRCLLC	SF 1190	207560	Lode	IGRCLLC
SF 1146	207516	Lode	IGRCLLC	SF 1191	207561	Lode	IGRCLLC
SF 1147	207517	Lode	IGRCLLC	SF 1192	207562	Lode	IGRCLLC
SF 1148	207518	Lode	IGRCLLC	SF 1193	207563	Lode	IGRCLLC
SF 1149	207519	Lode	IGRCLLC	SF 1194	207564	Lode	IGRCLLC
SF 1150	207520	Lode	IGRCLLC	SF 1195	207565	Lode	IGRCLLC
SF 1151	207521	Lode	IGRCLLC	SF 1196	207566	Lode	IGRCLLC
SF 1152	207522	Lode	IGRCLLC	SF 1197	207567	Lode	IGRCLLC
SF 1153	207523	Lode	IGRCLLC	SF 1198	207568	Lode	IGRCLLC
SF 1154	207524	Lode	IGRCLLC	SF 1199	207569	Lode	IGRCLLC
SF 1155	207525	Lode	IGRCLLC	SF 1200	207570	Lode	IGRCLLC
SF 1156	207526	Lode	IGRCLLC	SF 1201	207571	Lode	IGRCLLC
SF 1157	207527	Lode	IGRCLLC	SF 1202	207572	Lode	IGRCLLC
SF 1158	207528	Lode	IGRCLLC	SF 1203	207573	Lode	IGRCLLC
SF 1159	207529	Lode	IGRCLLC	SF 1204	207574	Lode	IGRCLLC
SF 1160	207530	Lode	IGRCLLC	SF 1205	207575	Lode	IGRCLLC
SF 1161	207531	Lode	IGRCLLC	SF 1206	207576	Lode	IGRCLLC
SF 1162	207532	Lode	IGRCLLC	SF 1207	207577	Lode	IGRCLLC
SF 1163	207533	Lode	IGRCLLC	SF 1208	207578	Lode	IGRCLLC
SF 1164	207534	Lode	IGRCLLC	SF 1209	207579	Lode	IGRCLLC
SF 1165	207535	Lode	IGRCLLC	SF 1210	207580	Lode	IGRCLLC
SF 1166	207536	Lode	IGRCLLC	SF 1211	207581	Lode	IGRCLLC
SF 1167	207537	Lode	IGRCLLC	SF 1212	207582	Lode	IGRCLLC
SF 1168	207538	Lode	IGRCLLC	SF 1213	207583	Lode	IGRCLLC
SF 1169	207539	Lode	IGRCLLC	SF 1214	207584	Lode	IGRCLLC
SF 1170	207540	Lode	IGRCLLC	SF 1215	207585	Lode	IGRCLLC
SF 1171	207541	Lode	IGRCLLC	SF 1216	207586	Lode	IGRCLLC
SF 1172	207542	Lode	IGRCLLC	SF 1217	207587	Lode	IGRCLLC
SF 1173	207543	Lode	IGRCLLC	SF 1218	207588	Lode	IGRCLLC
SF 1174	207544	Lode	IGRCLLC	SF 1219	207589	Lode	IGRCLLC
SF 1175	207545	Lode	IGRCLLC	SF 1220	207590	Lode	IGRCLLC
SF 1176	207546	Lode	IGRCLLC	SF 1221	207591	Lode	IGRCLLC
SF 1177	207547	Lode	IGRCLLC	SF 1222	207592	Lode	IGRCLLC
SF 1178	207548	Lode	IGRCLLC	SF 1223	207593	Lode	IGRCLLC
SF 1179	207549	Lode	IGRCLLC	SF 1224	207594	Lode	IGRCLLC
SF 1180	207550	Lode	IGRCLLC	SF 1225	207595	Lode	IGRCLLC
SF 1181	207551	Lode	IGRCLLC	SF 1226	207596	Lode	IGRCLLC
SF 1182	207552	Lode	IGRCLLC	SF 1227	207597	Lode	IGRCLLC
SF 1228	207598	Lode	IGRCLLC	SF 1273	207643	Lode	IGRCLLC
SF 1229	207599	Lode	IGRCLLC	SF 1274	207644	Lode	IGRCLLC
SF 1230	207600	Lode	IGRCLLC	SF 1275	207645	Lode	IGRCLLC
SF 1231	207601	Lode	IGRCLLC	SF 1276	207646	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1232	207602	Lode	IGRCLLC
SF 1233	207603	Lode	IGRCLLC
SF 1234	207604	Lode	IGRCLLC
SF 1235	207605	Lode	IGRCLLC
SF 1236	207606	Lode	IGRCLLC
SF 1237	207607	Lode	IGRCLLC
SF 1238	207608	Lode	IGRCLLC
SF 1239	207609	Lode	IGRCLLC
SF 1240	207610	Lode	IGRCLLC
SF 1241	207611	Lode	IGRCLLC
SF 1242	207612	Lode	IGRCLLC
SF 1243	207613	Lode	IGRCLLC
SF 1244	207614	Lode	IGRCLLC
SF 1245	207615	Lode	IGRCLLC
SF 1246	207616	Lode	IGRCLLC
SF 1247	207617	Lode	IGRCLLC
SF 1248	207618	Lode	IGRCLLC
SF 1249	207619	Lode	IGRCLLC
SF 1250	207620	Lode	IGRCLLC
SF 1251	207621	Lode	IGRCLLC
SF 1252	207622	Lode	IGRCLLC
SF 1253	207623	Lode	IGRCLLC
SF 1254	207624	Lode	IGRCLLC
SF 1255	207625	Lode	IGRCLLC
SF 1256	207626	Lode	IGRCLLC
SF 1257	207627	Lode	IGRCLLC
SF 1258	207628	Lode	IGRCLLC
SF 1259	207629	Lode	IGRCLLC
SF 1260	207630	Lode	IGRCLLC
SF 1261	207631	Lode	IGRCLLC
SF 1262	207632	Lode	IGRCLLC
SF 1263	207633	Lode	IGRCLLC
SF 1264	207634	Lode	IGRCLLC
SF 1265	207635	Lode	IGRCLLC
SF 1266	207636	Lode	IGRCLLC
SF 1267	207637	Lode	IGRCLLC
SF 1268	207638	Lode	IGRCLLC
SF 1269	207639	Lode	IGRCLLC
SF 1270	207640	Lode	IGRCLLC
SF 1271	207641	Lode	IGRCLLC
SF 1272	207642	Lode	IGRCLLC
SF 1318	207688	Lode	IGRCLLC
SF 1319	207689	Lode	IGRCLLC
SF 1320	207690	Lode	IGRCLLC
SF 1321	207691	Lode	IGRCLLC
SF 1322	207692	Lode	IGRCLLC
SF 1323	207693	Lode	IGRCLLC
SF 1324	207694	Lode	IGRCLLC
SF 1325	207695	Lode	IGRCLLC
SF 1326	207696	Lode	IGRCLLC
SF 1327	207697	Lode	IGRCLLC
SF 1328	207698	Lode	IGRCLLC
SF 1329	207699	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1277	207647	Lode	IGRCLLC
SF 1278	207648	Lode	IGRCLLC
SF 1279	207649	Lode	IGRCLLC
SF 1280	207650	Lode	IGRCLLC
SF 1281	207651	Lode	IGRCLLC
SF 1282	207652	Lode	IGRCLLC
SF 1283	207653	Lode	IGRCLLC
SF 1284	207654	Lode	IGRCLLC
SF 1285	207655	Lode	IGRCLLC
SF 1286	207656	Lode	IGRCLLC
SF 1287	207657	Lode	IGRCLLC
SF 1288	207658	Lode	IGRCLLC
SF 1289	207659	Lode	IGRCLLC
SF 1290	207660	Lode	IGRCLLC
SF 1291	207661	Lode	IGRCLLC
SF 1292	207662	Lode	IGRCLLC
SF 1293	207663	Lode	IGRCLLC
SF 1294	207664	Lode	IGRCLLC
SF 1295	207665	Lode	IGRCLLC
SF 1296	207666	Lode	IGRCLLC
SF 1297	207667	Lode	IGRCLLC
SF 1298	207668	Lode	IGRCLLC
SF 1299	207669	Lode	IGRCLLC
SF 1300	207670	Lode	IGRCLLC
SF 1301	207671	Lode	IGRCLLC
SF 1302	207672	Lode	IGRCLLC
SF 1303	207673	Lode	IGRCLLC
SF 1304	207674	Lode	IGRCLLC
SF 1305	207675	Lode	IGRCLLC
SF 1306	207676	Lode	IGRCLLC
SF 1307	207677	Lode	IGRCLLC
SF 1308	207678	Lode	IGRCLLC
SF 1309	207679	Lode	IGRCLLC
SF 1310	207680	Lode	IGRCLLC
SF 1311	207681	Lode	IGRCLLC
SF 1312	207682	Lode	IGRCLLC
SF 1313	207683	Lode	IGRCLLC
SF 1314	207684	Lode	IGRCLLC
SF 1315	207685	Lode	IGRCLLC
SF 1316	207686	Lode	IGRCLLC
SF 1317	207687	Lode	IGRCLLC
SF 1362	214713	Lode	IGRCLLC
SF 1363	214714	Lode	IGRCLLC
SF 1364	214715	Lode	IGRCLLC
SF 1365	214716	Lode	IGRCLLC
SF 1366	214717	Lode	IGRCLLC
SF 1367	214718	Lode	IGRCLLC
SF 1368	214719	Lode	IGRCLLC
SF 1369	214720	Lode	IGRCLLC
SF 1370	214721	Lode	IGRCLLC
SF 1371	214722	Lode	IGRCLLC
SF 1372	214723	Lode	IGRCLLC
SF 1373	214724	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1330	207700	Lode	IGRCLLC
SF 1331	207701	Lode	IGRCLLC
SF 1332	207702	Lode	IGRCLLC
SF 1333	207703	Lode	IGRCLLC
SF 1334	207704	Lode	IGRCLLC
SF 1335	207705	Lode	IGRCLLC
SF 1336	207706	Lode	IGRCLLC
SF 1337	207707	Lode	IGRCLLC
SF 1338	207708	Lode	IGRCLLC
SF 1339	207709	Lode	IGRCLLC
SF 1340	207710	Lode	IGRCLLC
SF 1341	207711	Lode	IGRCLLC
SF 1342	207712	Lode	IGRCLLC
SF 1343	207713	Lode	IGRCLLC
SF 1344	207714	Lode	IGRCLLC
SF 1345	207715	Lode	IGRCLLC
SF 1346	207716	Lode	IGRCLLC
SF 1347	207717	Lode	IGRCLLC
SF 1348	207718	Lode	IGRCLLC
SF 1349	207719	Lode	IGRCLLC
SF 1350	207720	Lode	IGRCLLC
SF 1351	207721	Lode	IGRCLLC
SF 1352	207722	Lode	IGRCLLC
SF 1353	207723	Lode	IGRCLLC
SF 1354	207724	Lode	IGRCLLC
SF 1355	207725	Lode	IGRCLLC
SF 453 A	211429	Lode	IGRCLLC
SF 1356	214707	Lode	IGRCLLC
SF 1357	214708	Lode	IGRCLLC
SF 1358	214709	Lode	IGRCLLC
SF 1359	214710	Lode	IGRCLLC
SF 1360	214711	Lode	IGRCLLC
SF 1361	214712	Lode	IGRCLLC
SF 1407	214758	Lode	IGRCLLC
SF 1408	214759	Lode	IGRCLLC
SF 1409	214760	Lode	IGRCLLC
SF 1410	214761	Lode	IGRCLLC
SF 1411	214762	Lode	IGRCLLC
SF 1412	214763	Lode	IGRCLLC
SF 1413	214764	Lode	IGRCLLC
SF 1414	214765	Lode	IGRCLLC
SF 1415	214766	Lode	IGRCLLC
SF 1416	214767	Lode	IGRCLLC
SF 1417	214768	Lode	IGRCLLC
SF 1418	214769	Lode	IGRCLLC
SF 1419	214770	Lode	IGRCLLC
SF 1420	214771	Lode	IGRCLLC
SF 1421	214772	Lode	IGRCLLC
SF 1422	214773	Lode	IGRCLLC
SF 1423	214774	Lode	IGRCLLC
SF 1424	214775	Lode	IGRCLLC
SF 1425	214776	Lode	IGRCLLC
SF 1426	214777	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1374	214725	Lode	IGRCLLC
SF 1375	214726	Lode	IGRCLLC
SF 1376	214727	Lode	IGRCLLC
SF 1377	214728	Lode	IGRCLLC
SF 1378	214729	Lode	IGRCLLC
SF 1379	214730	Lode	IGRCLLC
SF 1380	214731	Lode	IGRCLLC
SF 1381	214732	Lode	IGRCLLC
SF 1382	214733	Lode	IGRCLLC
SF 1383	214734	Lode	IGRCLLC
SF 1384	214735	Lode	IGRCLLC
SF 1385	214736	Lode	IGRCLLC
SF 1386	214737	Lode	IGRCLLC
SF 1387	214738	Lode	IGRCLLC
SF 1388	214739	Lode	IGRCLLC
SF 1389	214740	Lode	IGRCLLC
SF 1390	214741	Lode	IGRCLLC
SF 1391	214742	Lode	IGRCLLC
SF 1392	214743	Lode	IGRCLLC
SF 1393	214744	Lode	IGRCLLC
SF 1394	214745	Lode	IGRCLLC
SF 1395	214746	Lode	IGRCLLC
SF 1396	214747	Lode	IGRCLLC
SF 1397	214748	Lode	IGRCLLC
SF 1398	214749	Lode	IGRCLLC
SF 1399	214750	Lode	IGRCLLC
SF 1400	214751	Lode	IGRCLLC
SF 1401	214752	Lode	IGRCLLC
SF 1402	214753	Lode	IGRCLLC
SF 1403	214754	Lode	IGRCLLC
SF 1404	214755	Lode	IGRCLLC
SF 1405	214756	Lode	IGRCLLC
SF 1406	214757	Lode	IGRCLLC
SF 1452	214803	Lode	IGRCLLC
SF 1453	214804	Lode	IGRCLLC
SF 1454	214805	Lode	IGRCLLC
SF 1455	214806	Lode	IGRCLLC
SF 1456	214807	Lode	IGRCLLC
SF 1457	214808	Lode	IGRCLLC
SF 1458	214809	Lode	IGRCLLC
SF 1459	214810	Lode	IGRCLLC
SF 1460	214811	Lode	IGRCLLC
SF 1461	214812	Lode	IGRCLLC
SF 1462	214813	Lode	IGRCLLC
SF 1463	214814	Lode	IGRCLLC
SF 1464	214815	Lode	IGRCLLC
SF 1465	214816	Lode	IGRCLLC
SF 1466	214817	Lode	IGRCLLC
SF 1467	214818	Lode	IGRCLLC
SF 1468	214819	Lode	IGRCLLC
SF 1469	214820	Lode	IGRCLLC
SF 1470	214821	Lode	IGRCLLC
SF 1471	214822	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1427	214778	Lode	IGRCLLC
SF 1428	214779	Lode	IGRCLLC
SF 1429	214780	Lode	IGRCLLC
SF 1430	214781	Lode	IGRCLLC
SF 1431	214782	Lode	IGRCLLC
SF 1432	214783	Lode	IGRCLLC
SF 1433	214784	Lode	IGRCLLC
SF 1434	214785	Lode	IGRCLLC
SF 1435	214786	Lode	IGRCLLC
SF 1436	214787	Lode	IGRCLLC
SF 1437	214788	Lode	IGRCLLC
SF 1438	214789	Lode	IGRCLLC
SF 1484	214835	Lode	IGRCLLC
SF 1439	214790	Lode	IGRCLLC
SF 1440	214791	Lode	IGRCLLC
SF 1441	214792	Lode	IGRCLLC
SF 1442	214793	Lode	IGRCLLC
SF 1443	214794	Lode	IGRCLLC
SF 1444	214795	Lode	IGRCLLC
SF 1445	214796	Lode	IGRCLLC
SF 1446	214797	Lode	IGRCLLC
SF 1447	214798	Lode	IGRCLLC
SF 1448	214799	Lode	IGRCLLC
SF 1449	214800	Lode	IGRCLLC
SF 1450	214801	Lode	IGRCLLC
SF 1451	214802	Lode	IGRCLLC

Claim Name	IMC No.1	Claim Type	Owner
SF 1472	214823	Lode	IGRCLLC
SF 1473	214824	Lode	IGRCLLC
SF 1474	214825	Lode	IGRCLLC
SF 1475	214826	Lode	IGRCLLC
SF 1476	214827	Lode	IGRCLLC
SF 1477	214828	Lode	IGRCLLC
SF 1478	214829	Lode	IGRCLLC
SF 1479	214830	Lode	IGRCLLC
SF 1480	214831	Lode	IGRCLLC
SF 1481	214832	Lode	IGRCLLC
SF 1482	214833	Lode	IGRCLLC
SF 1483	214834	Lode	IGRCLLC

Appendix III

Financial Modeling Results

Stibnite Gold Project

Midas Gold Corporation

Mining		Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Yellow Pine																				
High Antimony	kt	11,279	-	-	36	848	3,477	4,022	1,497	458	839	100	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.06	-	-	0.07	0.07	0.05	0.06	0.06	0.04	0.07	0.06	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.14	-	-	0.05	0.11	0.15	0.14	0.14	0.06	0.16	0.08	-	-	-	-	-	-	-	-
Antimony grade	%	0.46%	0.00%	0.00%	0.13%	0.53%	0.49%	0.38%	0.39%	0.26%	0.89%	0.53%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	671	-	-	2	57	190	245	95	17	58	6	-	-	-	-	-	-	-	-
Contained Silver	kozs	1,543	-	-	2	96	518	552	205	28	133	8	-	-	-	-	-	-	-	-
Contained Antimony	klbs	103,758	-	-	96	8,971	33,981	30,599	11,730	2,409	14,912	1,061	-	-	-	-	-	-	-	-
Low Antimony	kt	41,463	-	-	776	7,075	5,454	9,076	10,958	4,837	2,578	709	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.05	-	-	0.03	0.05	0.06	0.05	0.06	0.04	0.05	0.05	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.05	-	-	0.04	0.04	0.06	0.05	0.04	0.04	0.04	0.03	-	-	-	-	-	-	-	-
Antimony grade	%	0.01%	0.00%	0.00%	0.00%	0.00%	0.01%	0.02%	0.01%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	2,047	-	-	24	323	310	437	609	177	131	37	-	-	-	-	-	-	-	-
Contained Silver	kozs	1,881	-	-	27	306	301	452	476	188	109	21	-	-	-	-	-	-	-	-
Contained Antimony	klbs	7,859	-	-	7	202	1,498	2,949	2,730	284	160	30	-	-	-	-	-	-	-	-
Total Yellow Pine	kt	52,742	-	-	813	7,924	8,931	13,098	12,455	5,295	3,417	809	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.05	-	-	0.03	0.05	0.06	0.05	0.06	0.04	0.06	0.05	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.06	-	-	0.04	0.05	0.09	0.08	0.05	0.04	0.07	0.04	-	-	-	-	-	-	-	-
Antimony grade	%	0.11%	0.00%	0.00%	0.01%	0.06%	0.20%	0.13%	0.06%	0.03%	0.22%	0.07%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	2,718	-	-	26	380	500	682	704	194	189	43	-	-	-	-	-	-	-	-
Contained Silver	kozs	3,423	-	-	29	402	819	1,004	682	217	242	29	-	-	-	-	-	-	-	-
Contained Antimony	klbs	111,617	-	-	102	9,173	35,478	33,548	14,459	2,693	15,072	1,091	-	-	-	-	-	-	-	-
Waste	kt	109,863	-	2,631	15,846	23,824	22,763	19,176	6,977	9,327	8,944	374	-	-	-	-	-	-	-	-
Hangar Flats																				
High Antimony	kt	3,411	-	-	-	-	-	1	771	2,325	314	-	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.06	-	-	-	-	-	0.02	0.05	0.06	0.05	-	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.14	-	-	-	-	-	0.19	0.15	0.15	0.07	-	-	-	-	-	-	-	-	-
Antimony grade	%	0.37%	0.00%	0.00%	0.00%	0.00%	0.00%	0.29%	0.36%	0.40%	0.18%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	191	-	-	-	-	-	0	38	138	15	-	-	-	-	-	-	-	-	-
Contained Silver	kozs	483	-	-	-	-	-	0	113	348	21	-	-	-	-	-	-	-	-	-
Contained Antimony	klbs	25,148	-	-	-	-	-	8	5,588	18,451	1,100	-	-	-	-	-	-	-	-	-
Low Antimony	kt	5,696	-	-	-	-	-	61	1,267	3,428	940	-	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.04	-	-	-	-	-	0.03	0.04	0.04	0.04	-	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.05	-	-	-	-	-	0.04	0.06	0.05	0.05	-	-	-	-	-	-	-	-	-
Antimony grade	%	0.02%	0.00%	0.00%	0.00%	0.00%	0.00%	0.01%	0.02%	0.02%	0.01%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	223	-	-	-	-	-	2	47	135	39	-	-	-	-	-	-	-	-	-
Contained Silver	kozs	273	-	-	-	-	-	3	72	156	43	-	-	-	-	-	-	-	-	-
Contained Antimony	klbs	2,104	-	-	-	-	-	9	608	1,249	237	-	-	-	-	-	-	-	-	-
Total Hangar Flats	kt	9,107	-	-	-	-	-	63	2,038	5,753	1,254	-	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.05	-	-	-	-	-	0.03	0.04	0.05	0.04	-	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.08	-	-	-	-	-	0.05	0.09	0.09	0.05	-	-	-	-	-	-	-	-	-
Antimony grade	%	0.15%	0.00%	0.00%	0.00%	0.00%	0.00%	0.01%	0.15%	0.17%	0.05%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	414	-	-	-	-	-	2	85	273	54	-	-	-	-	-	-	-	-	-
Contained Silver	kozs	756	-	-	-	-	-	3	185	504	64	-	-	-	-	-	-	-	-	-
Contained Antimony	klbs	27,252	-	-	-	-	-	18	6,197	19,700	1,338	-	-	-	-	-	-	-	-	-
Waste		22,119	-	-	-	-	-	2,532	9,749	9,159	678	-	-	-	-	-	-	-	-	-
West End																				
Low Antimony	kt	16,801	-	-	-	-	-	-	-	-	13	743	2,478	395	2,208	5,309	5,656	-	-	-
Gold grade	oz/t	0.04	-	-	-	-	-	-	-	-	0.03	0.04	0.04	0.04	0.03	0.04	0.04	-	-	-
Silver grade	oz/t	0.04	-	-	-	-	-	-	-	-	0.00	0.02	0.01	0.04	0.04	0.05	0.04	-	-	-
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	649	-	-	-	-	-	-	-	-	0	31	87	14	66	221	229	-	-	-
Contained Silver	kozs	635	-	-	-	-	-	-	-	-	0	16	32	16	86	246	238	-	-	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Stibnite Gold Project

Midas Gold Corporation

Oxide Feed	kt	5,670	-	-	24	-	-	18	-	30	886	2,481	1,143	593	391	97	7	-	-	-
Gold grade	oz/t	0.02	-	-	0.02	-	-	0.02	-	0.01	0.02	0.02	0.02	0.02	0.02	0.02	0.01	-	-	-
Silver grade	oz/t	0.03	-	-	0.02	-	-	0.01	-	0.03	0.02	0.02	0.03	0.03	0.03	0.06	0.09	-	-	-
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	89	-	-	0	-	-	0	-	0	14	39	18	9	6	2	0	-	-	-
Contained Silver	kozs	144	-	-	1	-	-	0	-	1	17	56	34	19	11	6	1	-	-	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mix Sulfide & Oxide	kt	28,483	-	-	16	-	-	17	-	57	990	4,521	5,348	5,184	6,258	5,076	1,016	-	-	-
Gold grade	oz/t	0.03	-	-	0.02	-	-	0.02	-	0.04	0.03	0.02	0.03	0.03	0.03	0.03	0.04	-	-	-
Silver grade	oz/t	0.04	-	-	0.04	-	-	0.02	-	0.05	0.03	0.03	0.03	0.05	0.05	0.05	0.05	-	-	-
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	855	-	-	0	-	-	0	-	2	25	112	165	162	189	159	40	-	-	-
Contained Silver	kozs	1,236	-	-	1	-	-	0	-	3	31	132	169	244	336	272	47	-	-	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total West End	kt	50,953			40	-	-	34	-	87	1,889	7,745	8,970	6,172	8,857	10,482	6,678	-	-	-
Gold grade	oz/t	0.03			0.02	-	-	0.02	-	0.03	0.02	0.02	0.03	0.03	0.03	0.04	0.04	-	-	-
Silver grade	oz/t	0.04			0.03	-	-	0.01	-	0.04	0.03	0.03	0.03	0.05	0.05	0.05	0.04	-	-	-
Antimony grade	%	0.00%			0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	1,593	-	-	1	-	-	1	-	3	39	182	270	186	261	382	269	-	-	-
Contained Silver	kozs	2,015	-	-	1	-	-	0	-	4	48	203	236	279	433	524	285	-	-	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste		147,309	-	-	2,051	380	2,062	372	2,099	4,698	14,704	24,565	25,713	30,143	25,289	11,818	3,416	-	-	-
Historical Tailings	kt	2,962	-	-	-	477	916	916	653	-	-	-	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.03	-	-	-	0.03	0.03	0.03	0.03	-	-	-	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.08	-	-	-	0.08	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-
Antimony grade	%	0.17%	-	-	-	0.17%	0.17%	0.17%	0.17%	-	-	-	-	-	-	-	-	-	-	-
Contained Gold	kozs	100	-	-	-	16	31	31	22	-	-	-	-	-	-	-	-	-	-	-
Contained Silver	kozs	247	-	-	-	40	77	77	55	-	-	-	-	-	-	-	-	-	-	-
Contained Antimony	klbs	9,817				1,580	3,036	3,036	2,164	-	-	-	-	-	-	-	-	-	-	-
Waste		5,752	-	-	5,752	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant		**Total**	**Year -3**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**
Yellow Pine																				
High Antimony	kt	11,279				825	3,049	3,287	884	693	1,700	98	-	742	-	-	-	-	-	-
Gold grade	oz/t	0.06				0.07	0.06	0.07	0.08	0.05	0.05	0.06	-	0.01	-	-	-	-	-	-
Silver grade	oz/t	0.14				0.12	0.16	0.15	0.17	0.08	0.12	0.08	-	0.05	-	-	-	-	-	-
Antimony grade	%	0.46%				0.53%	0.52%	0.42%	0.51%	0.26%	0.56%	0.54%	0.00%	0.21%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	671				58	183	223	71	32	86	6	-	11	-	-	-	-	-	-
Contained Silver	kozs	1,543				96	500	489	153	56	204	8	-	36	-	-	-	-	-	-
Contained Antimony	klbs	103,758				8,784	31,499	27,510	9,026	3,658	19,133	1,051	-	3,098	-	-	-	-	-	-
Gold Bullion Recovery	%	88.8%				89.5%	88.7%	88.9%	88.8%	88.4%	88.8%	89.1%	80.3%	86.5%	80.3%	80.3%	80.3%	80.3%	80.3%	80.3%
Silver Bullion Recovery	%	0.0%				0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recovered Gold	kozs	596				52	162	198	63	29	77	5	-	9	-	-	-	-	-	-
Recovered Silver	kozs	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Antimony - Gold Recovery	%	2.5%				2.7%	2.6%	2.3%	2.6%	1.8%	2.8%	2.7%	0.9%	1.6%	0.9%	0.9%	0.9%	0.9%	0.9%	0.9%
Antimony - Silver Recovery	%	37.1%				41.3%	40.1%	32.9%	39.7%	21.6%	43.5%	41.8%	2.2%	17.5%	2.2%	2.2%	2.2%	2.2%	2.2%	2.2%
Antimony Recovery	%	88.7%				89.3%	89.1%	88.0%	89.1%	86.1%	89.7%	89.4%	83.0%	85.5%	83.0%	83.0%	83.0%	83.0%	83.0%	83.0%
Antimony Concentrate	kt	71				6.0355	22	19	6	2	13	1	-	2	-	-	-	-	-	-
Antimony Concentrate Grade	%	65.0%				65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%
Recovered Gold	kozs	17				2	5	5	2	1	2	0	-	0	-	-	-	-	-	-
Recovered Silver	kozs	573				40	201	161	61	12	89	3	-	6	-	-	-	-	-	-
Recovered Antimony	klbs	92,065				7,846	28,079	24,202	8,040	3,151	17,160	940	-	2,648	-	-	-	-	-	-

Stibnite Gold Project
Midas Gold Corporation

Low Antimony	kt	41,463	5,785	4,105	4,391	6,535	3,120	4,518	3,981	1,374	1,989	1,129	172	978	45	3,342	-
Gold grade	oz/t	0.05	0.05	0.07	0.07	0.08	0.05	0.04	0.03	0.02	0.02	0.02	0.02	0.02	0.02	0.02	-
Silver grade	oz/t	0.05	0.05	0.06	0.06	0.05	0.04	0.04	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	-
Antimony grade	%	0.01%	0.00%	0.02%	0.02%	0.02%	0.01%	0.01%	0.01%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	2,047	308	281	297	501	159	199	138	27	39	22	3	19	1	52	-
Contained Silver	kozs	1,881	290	255	255	341	139	198	153	40	58	33	5	28	1	84	-
Contained Antimony	klbs	7,859	165	1,280	1,761	2,205	348	687	666	130	188	107	16	93	3	210	-
Gold Bullion Recovery	%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%	90.7%
Silver Bullion Recovery	%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%	0.6%
Recovered Gold	kozs	1,857	280	255	269	454	144	181	125	25	36	20	3	18	1	47	-
Recovered Silver	kozs	11	2	2	2	2	1	1	1	0	0	0	0	0	0	1	-
Antimony - Gold Recovery	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony - Silver Recovery	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony Recovery	%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Total Yellow Pine	kt	52,742	6,610	7,154	7,678	7,419	3,813	6,217	4,079	1,374	2,732	1,129	172	978	45	3,342	-
Gold grade	oz/t	0.05	0.06	0.06	0.07	0.08	0.05	0.05	0.04	0.02	0.02	0.02	0.02	0.02	0.02	0.02	-
Silver grade	oz/t	0.06	0.06	0.11	0.10	0.07	0.05	0.06	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	-
Antimony grade	%	0.11%	0.07%	0.23%	0.19%	0.08%	0.05%	0.16%	0.02%	0.00%	0.06%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	2,718	367	464	520	572	192	286	144	27	50	22	3	19	1	52	-
Contained Silver	kozs	3,423	387	756	744	494	194	403	161	40	94	33	5	28	1	84	-
Contained Antimony	klbs	111,617	8,949	32,778	29,271	11,232	4,006	19,820	1,718	130	3,286	107	16	93	3	210	-
Gold Bullion Recovery	%	90.2%	90.5%	89.9%	89.9%	90.5%	90.3%	90.1%	90.6%	90.7%	89.8%	90.7%	90.7%	90.7%	90.7%	90.7%	0.0%
Silver Bullion Recovery	%	0.3%	0.5%	0.2%	0.2%	0.4%	0.4%	0.3%	0.6%	0.6%	0.4%	0.6%	0.6%	0.6%	0.6%	0.6%	0.0%
Recovered Gold	kozs	2,453	332	417	467	517	173	257	130	25	45	20	3	18	1	47	-
Recovered Silver	kozs	11	2	2	2	2	1	1	1	0	0	0	0	0	0	1	-
Antimony - Gold Recovery	%	0.6%	0.4%	1.0%	1.0%	0.3%	0.3%	0.8%	0.1%	0.0%	0.3%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony - Silver Recovery	%	16.7%	10.3%	26.5%	21.6%	12.3%	6.2%	22.1%	2.1%	0.0%	6.7%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony Recovery	%	82.5%	87.7%	85.7%	82.7%	71.6%	78.7%	86.6%	54.7%	0.0%	80.6%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony Concentrate	kt	71	6	22	19	6	2	13	1	-	2	-	-	-	-	-	-
Antimony Concentrate Grade	%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	0.0%	65.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recovered Gold	kozs	17	2	5	5	2	1	2	0	-	0	-	-	-	-	-	-
Recovered Silver	kozs	573	40	201	161	61	12	89	3	-	6	-	-	-	-	-	-
Recovered Antimony	klbs	92,065	7,846	28,079	24,202	8,040	3,151	17,160	940	-	2,648	-	-	-	-	-	-
Hangar Flats																	
High Antimony	kt	3,411	-	-	-	370	2,266	592	-	-	183	-	-	-	-	-	-
Gold grade	oz/t	0.06	-	-	-	0.07	0.06	0.04	-	-	0.02	-	-	-	-	-	-
Silver grade	oz/t	0.14	-	-	-	0.18	0.16	0.08	-	-	0.06	-	-	-	-	-	-
Antimony grade	%	0.37%	0.00%	0.00%	0.00%	0.51%	0.41%	0.20%	0.00%	0.00%	0.11%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	191	-	-	-	24	140	24	-	-	3	-	-	-	-	-	-
Contained Silver	kozs	483	-	-	-	67	356	48	-	-	12	-	-	-	-	-	-
Contained Antimony	klbs	25,148	-	-	-	3,764	18,651	2,313	-	-	420	-	-	-	-	-	-
Gold Bullion Recovery	%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%
Silver Bullion Recovery	%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%	0.1%
Recovered Gold	kozs	165	-	-	-	21	121	20	-	-	3	-	-	-	-	-	-
Recovered Silver	kozs	0	-	-	-	0	0	0	-	-	0	-	-	-	-	-	-
Antimony - Gold Recovery	%	1.9%	0.2%	0.2%	0.2%	2.1%	2.0%	1.1%	0.2%	0.2%	0.9%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
Antimony - Silver Recovery	%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%	52.8%
Antimony Recovery	%	82.8%	80.0%	80.0%	80.0%	83.1%	82.9%	81.4%	80.0%	80.0%	81.2%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%
Antimony Concentrate	kt	19	-	-	-	3	14	2	-	-	0	-	-	-	-	-	-
Antimony Concentrate Grade	%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%	54.1%
Recovered Gold	kozs	4	-	-	-	1	3	0	-	-	0	-	-	-	-	-	-
Recovered Silver	kozs	255	-	-	-	35	188	25	-	-	6	-	-	-	-	-	-
Recovered Antimony	klbs	20,822	-	-	-	3,129	15,470	1,883	-	-	341	-	-	-	-	-	-

Low Antimony	kt	5,696	-	-	22	261	1,993	1,239	727	241	349	198	30	171	6	461	-
Gold grade	oz/t	0.04	-	-	0.06	0.07	0.05	0.04	0.03	0.02	0.02	0.02	0.02	0.02	0.02	0.02	-
Silver grade	oz/t	0.05	-	-	0.08	0.09	0.06	0.05	0.04	0.03	0.03	0.03	0.03	0.03	0.02	0.02	-
Antimony grade	%	0.02%	0.00%	0.00%	0.01%	0.06%	0.03%	0.01%	0.01%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	223	-	-	1	17	106	50	21	5	7	4	1	3	0	7	-
Contained Silver	kozs	273	-	-	2	23	112	60	32	8	12	7	1	6	0	12	-
Contained Antimony	klbs	2,104	-	-	6	312	1,127	365	171	24	35	20	3	17	0	25	-
Gold Bullion Recovery	%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%	88.9%
Silver Bullion Recovery	%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
Recovered Gold	kozs	199	-	-	1	15	95	45	19	4	6	4	1	3	0	6	-
Recovered Silver	kozs	1	-	-	0	0	0	0	0	0	0	0	0	0	0	0	-
Total Hangar Flats	kt	9,107	-	-	22	631	4,259	1,831	727	241	532	198	30	171	6	461	-
Gold grade	oz/t	0.05	-	-	0.06	0.07	0.06	0.04	0.03	0.02	0.02	0.02	0.02	0.02	0.02	0.02	-
Silver grade	oz/t	0.08	-	-	0.08	0.14	0.11	0.06	0.04	0.03	0.04	0.03	0.03	0.03	0.02	0.02	-
Antimony grade	%	0.15%	0.00%	0.00%	0.01%	0.32%	0.23%	0.07%	0.01%	0.00%	0.04%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	414	-	-	1	42	246	74	21	5	10	4	1	3	0	7	-
Contained Silver	kozs	756	-	-	2	90	468	108	32	8	23	7	1	6	0	12	-
Contained Antimony	klbs	27,252	-	-	6	4,076	19,778	2,678	171	24	455	20	3	17	0	25	-
Gold Bullion Recovery	%	87.8%	0.0%	0.0%	88.9%	87.5%	87.6%	88.2%	88.9%	88.9%	88.2%	88.9%	88.9%	88.9%	88.9%	88.9%	0.0%
Silver Bullion Recovery	%	0.1%	0.0%	0.0%	0.2%	0.1%	0.1%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.0%
Recovered Gold	kozs	364	-	-	1	36	216	65	19	4	9	4	1	3	0	6	-
Recovered Silver	kozs	1	-	-	0	0	1	0	0	0	0	0	0	0	0	0	-
Antimony - Gold Recovery	%	0.9%	0.0%	0.0%	0.0%	1.3%	1.1%	0.3%	0.0%	0.0%	0.3%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony - Silver Recovery	%	33.7%	0.0%	0.0%	0.0%	39.3%	40.1%	23.5%	0.0%	0.0%	26.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony Recovery	%	76.4%	0.0%	0.0%	0.0%	76.8%	78.2%	70.3%	0.0%	0.0%	75.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Antimony Concentrate	kt	19	-	-	-	3	14	2	-	-	0	-	-	-	-	-	-
Antimony Concentrate Grade	%	54.1%	0.0%	0.0%	0.0%	54.1%	54.1%	54.1%	0.0%	0.0%	54.1%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Recovered Gold	kozs	4	-	-	-	1	3	0	-	-	0	-	-	-	-	-	-
Recovered Silver	kozs	255	-	-	-	35	188	25	-	-	6	-	-	-	-	-	-
Recovered Antimony	klbs	20,822	-	-	-	3,129	15,470	1,883	-	-	341	-	-	-	-	-	-
West End																	
Low Antimony	kt	16,801	-	-	-	-	-	2	550	2,039	320	1,755	4,719	5,264	28	2,124	-
Gold grade	oz/t	0.04	-	-	-	-	-	0.03	0.05	0.04	0.04	0.03	0.04	0.04	0.02	0.02	-
Silver grade	oz/t	0.04	-	-	-	-	-	0.00	0.02	0.01	0.04	0.04	0.05	0.04	0.03	0.03	-
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	649	-	-	-	-	-	0	28	80	13	58	211	222	0	36	-
Contained Silver	kozs	635	-	-	-	-	-	0	13	28	13	71	225	222	1	63	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion Recovery	%	85.3%	97.3%	97.3%	97.3%	97.3%	97.3%	81.4%	82.2%	86.2%	77.3%	80.4%	84.5%	88.9%	79.8%	79.8%	97.3%
Silver Bullion Recovery	%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.3%	0.3%	0.2%	0.3%	0.3%	0.2%	0.2%	0.3%	0.3%	0.2%
Recovered Gold	kozs	554	-	-	-	-	-	0	23	69	10	47	178	197	0	29	-
Recovered Silver	kozs	2	-	-	-	-	-	0	0	0	0	0	1	0	0	0	-
Oxide Feed	kt	5,235	-	-	-	-	-	-	138	338	185	125	31	7	-	2,123	2,288
Gold grade	oz/t	0.02	-	-	-	-	-	-	0.02	0.02	0.02	0.02	0.02	0.01	-	0.02	0.01
Silver grade	oz/t	0.03	-	-	-	-	-	-	0.03	0.04	0.03	0.03	0.08	0.09	-	0.02	0.02
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	83	-	-	-	-	-	-	3	7	4	2	1	0	-	32	34
Contained Silver	kozs	133	-	-	-	-	-	-	4	15	6	4	2	1	-	46	55
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion Recovery	%	85.6%	1.2%	1.2%	1.2%	1.2%	1.2%	1.2%	89.4%	88.2%	88.2%	87.4%	87.5%	82.9%	1.2%	84.9%	84.9%
Silver Bullion Recovery	%	63.3%	25.4%	25.4%	25.4%	25.4%	25.4%	25.4%	65.0%	64.4%	64.4%	64.1%	64.1%	62.1%	25.4%	62.9%	63.0%
Recovered Gold	kozs	71	-	-	-	-	-	-	3	6	3	2	1	0	-	27	29
Recovered Silver	kozs	84	-	-	-	-	-	-	3	10	4	3	2	0	-	29	34

Mix Sulfide & Oxide	kt	28,483	-	-	-	-	-	-	2,556	4,058	4,304	4,844	3,098	1,630	7,993	-	-
Gold grade	oz/t	0.03	-	-	-	-	-	-	0.03	0.04	0.03	0.03	0.04	0.03	0.02	-	-
Silver grade	oz/t	0.04	-	-	-	-	-	-	0.03	0.03	0.05	0.06	0.06	0.05	0.03	-	-
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	855	-	-	-	-	-	-	81	143	148	164	120	55	143	-	-
Contained Silver	kozs	1,236	-	-	-	-	-	-	81	139	215	285	184	78	253	-	-
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion Recovery	%	82.8%	16.1%	16.1%	16.1%	16.1%	16.1%	16.1%	87.4%	84.7%	84.1%	81.5%	79.2%	78.3%	83.5%	16.1%	16.1%
Silver Bullion Recovery	%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%	60.9%
Recovered Gold	kozs	708	-	-	-	-	-	-	71	121	124	134	95	43	120	-	-
Recovered Silver	kozs	753	-	-	-	-	-	-	50	85	131	174	112	47	154	-	-
Total West End	kt	50,519	-	-	-	-	-	2	3,244	6,436	4,809	6,723	7,848	6,901	8,021	4,247	2,288
Gold grade	oz/t	0.03	-	-	-	-	-	0.03	0.03	0.04	0.03	0.03	0.04	0.04	0.02	0.02	0.01
Silver grade	oz/t	0.04	-	-	-	-	-	0.00	0.03	0.03	0.05	0.05	0.05	0.04	0.03	0.03	0.02
Antimony grade	%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Gold	kozs	1,587	-	-	-	-	-	0	112	230	165	225	332	277	144	68	34
Contained Silver	kozs	2,004	-	-	-	-	-	0	99	182	234	360	411	300	253	109	55
Contained Antimony	klbs	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion Recovery	%	84.0%	0.0%	0.0%	0.0%	0.0%	0.0%	81.4%	86.2%	85.3%	83.7%	81.3%	82.6%	86.8%	83.4%	82.2%	84.9%
Silver Bullion Recovery	%	41.9%	0.0%	0.0%	0.0%	0.0%	0.0%	0.3%	53.0%	52.0%	57.6%	49.1%	27.8%	16.1%	60.7%	26.8%	63.0%
Recovered Gold	kozs	1,333	-	-	-	-	-	0.051	97	196	138	183	274	240	120	56	29
Recovered Silver	kozs	839	-	-	-	-	-	0	52	95	135	177	114	48	154	29	34
Bradley Tails	kt	2,962	477	916	916	653	-	-	-	-	-	-	-	-	-	-	-
Gold grade	oz/t	0.03	0.03	0.03	0.03	0.03	-	-	-	-	-	-	-	-	-	-	-
Silver grade	oz/t	0.08	0.08	0.08	0.08	0.08	-	-	-	-	-	-	-	-	-	-	-
Antimony grade	%	0.17%	0.2%	0.2%	0.2%	0.2%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
Contained Gold	kozs	100	16	31	31	22	-	-	-	-	-	-	-	-	-	-	-
Contained Silver	kozs	247	40	77	77	55	-	-	-	-	-	-	-	-	-	-	-
Contained Antimony	klbs	9,817	1,580	3,036	3,036	2,164	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion Recovery	%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	67.7%	0.0%	0.0%	0.0%
Silver Bullion Recovery	%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%	0.0%	0.0%	0.0%
Recovered Gold	kozs	68	11	21	21	15	-	-	-	-	-	-	-	-	-	-	-
Recovered Silver	kozs	0	0	0	0	0	-	-	-	-	-	-	-	-	-	-	-
Antimony - Gold Recovery	%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.5%	0.0%	0.0%	0.0%
Antimony - Silver Recovery	%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	12.5%	0.0%	0.0%	0.0%
Antimony Recovery	%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	25.0%	0.0%	0.0%	0.0%
Antimony Concentrate	kt	2	0	1	1	1	-	-	-	-	-	-	-	-	-	-	-
Antimony Concentrate Grade	%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	50.0%	0.0%	0.0%	0.0%
Recovered Gold	kozs	1	0	0	0	0	-	-	-	-	-	-	-	-	-	-	-
Recovered Silver	kozs	31	5	10	10	7	-	-	-	-	-	-	-	-	-	-	-
Recovered Antimony	klbs	2,454	395	759	759	541	-	-	-	-	-	-	-	-	-	-	-

Payable Metals		**Total**	**Year -3**	**Year -2**	**Year -1**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**
Doré Metals																				
Payable Gold	kozs	4,196				341	436	487	566	387	321	245	224	191	206	276	260	120	109	28
Payable Silver	kozs	835				2	2	2	2	1	1	52	93	133	173	112	47	151	29	34
Antimony Concentrate Payable Metals																				
Antimony Concentrate	kt	93				6	22	19	10	17	15	1	-	2	-	-	-	-	-	-
Payable Gold	kozs	3.73				0.30	0.86	1.05	0.51	0.58	0.39	0.03	-	-	-	-	-	-	-	-
Payable Silver	kozs	134				-	-	-	39.24	80.25	11.17	-	-	2.91	-	-	-	-	-	-
Payable Antimony	klbs	78,433				5,604.03	19,609.67	16,973.35	7,962.41	12,661.91	12,949.31	639.29	-	2,032.55	-	-	-	-	-	-
Revenues																				
Metal Prices		$000																		
Gold	$/oz	$1,600.00				$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00	$1,600.00
Silver	$/oz	$20.00				$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00
Antimony	$/lb	$3.50				$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50	$3.50
Doré		$000																		
Gold		$6,713,596				$545,633	$697,600	$778,800	$905,102	$619,110	$513,301	$391,574	$358,376	$305,528	$329,199	$441,921	$415,609	$192,147	$174,257	$45,441
Silver		$16,691				$36	$33	$33	$44	$28	$27	$1,043	$1,858	$2,655	$3,468	$2,243	$949	$3,015	$586	$674
Refining/Transport Cost																				
Gold		$9,021				$733	$937	$1,047	$1,216	$832	$690	$526	$482	$411	$442	$594	$558	$258	$234	$61
Silver		$1,377				$3	$3	$3	$4	$2	$2	$86	$153	$219	$286	$185	$78	$249	$48	$56
Antimony Concentrate		$000																		
Gold		$5,966				$488	$1,383	$1,682	$821	$926	$620	$47	$0	$0	$0	$0	$0	$0	$0	$0
Silver		$2,671				$0	$0	$0	$785	$1,605	$223	$0	$0	$58	$0	$0	$0	$0	$0	$0
Antimony		$274,514				$19,614	$68,634	$59,407	$27,868	$44,317	$45,323	$2,238	$0	$7,114	$0	$0	$0	$0	$0	$0
Treatment/Transport Cost		$15,154				$1,053	$3,662	$3,174	$1,575	$2,739	$2,447	$118	$0	$385	$0	$0	$0	$0	$0	$0
Total Revenues		$6,987,886				$563,981	$763,047	$835,698	$931,826	$662,411	$556,354	$394,171	$359,599	$314,340	$331,939	$443,384	$415,921	$194,655	$174,560	$45,999

Case B
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,600/oz
Silver - $20/oz
Antimony - $3.50/lb

Operating Cost	$000	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15
Mining	$860,151	$2,023	$14,407	$33,663	$67,675	$67,397	$73,575	$77,534	$70,216	$66,569	$63,817	$63,141	$61,576	$57,809	$53,198	$34,624	$22,514	$19,064	$11,349
Process Plant	$1,332,063				$88,809	$96,643	$101,116	$99,642	$96,688	$95,935	$92,503	$90,989	$92,893	$91,388	$92,312	$92,447	$90,096	$81,083	$29,521
Water Treatment Plant	$3,157				$0	$0	$0	$589	$1,235	$779	$70	$0	$0	$0	$0	$0	$0	$0	$484
G&A	$358,607	$250	$250	$250	$24,475	$25,931	$26,813	$27,125	$25,095	$24,348	$23,986	$23,741	$23,486	$23,517	$24,193	$24,425	$23,986	$23,486	$13,252
Total Operating Cost	$2,553,979	$2,273	$14,657	$33,913	$180,958	$189,970	$201,504	$204,890	$193,234	$187,632	$180,375	$177,872	$177,955	$172,713	$169,703	$151,495	$136,596	$123,632	$54,607
Royalty	$114,079				$9,272	$11,867	$13,250	$15,380	$10,526	$8,725	$6,649	$6,084	$5,187	$5,589	$7,503	$7,056	$3,262	$2,958	$771
Property Taxes	$4,354				$315	$427	$389	$380	$317	$366	$346	$389	$137	$204	$167	$230	$230	$230	$230
Salvage Value	-$27,240				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$4,086	-$4,086
Reclamation/Closure	$0				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Cost	$2,645,172	$2,273	$14,657	$33,913	$190,545	$202,264	$215,143	$220,651	$204,077	$196,722	$187,369	$184,345	$183,278	$178,506	$177,372	$158,781	$140,088	$122,734	$51,522
Net Operating Income - EBITDA	$4,342,714	-$2,273	-$14,657	-$33,913	$373,436	$560,783	$620,555	$711,175	$458,334	$359,632	$206,801	$175,254	$131,062	$153,433	$266,012	$257,141	$54,567	$51,826	-$5,523
Depreciation																			
Initial Capital	$1,218,935			$1,235	$1,202,454	$4,120	$4,120	$4,120	$2,885	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Equipment Lease	$149,437				$21,355	$36,597	$26,137	$18,665	$13,345	$13,330	$13,345	$6,665	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital	$269,556				$19,508	$15,973	$14,554	$9,590	$7,426	$13,766	$18,937	$15,847	$13,548	$10,676	$12,030	$12,322	$8,297	$4,861	$3,918
Total Depreciation	$1,635,847	$0	$0	$1,235	$1,243,317	$56,691	$44,811	$32,375	$23,656	$27,096	$32,282	$22,512	$13,548	$10,676	$12,030	$12,322	$8,297	$4,861	$3,918
Interest																			
Capital Equipment Lease Interest	$17,593	$0	$594	$2,543	$3,843	$3,088	$2,488	$1,633	$716	$455	$553	$421	$381	$336	$246	$135	$97	$51	$13
Total Interest	$17,593	$0	$594	$2,543	$3,843	$3,088	$2,488	$1,633	$716	$455	$553	$421	$381	$336	$246	$135	$97	$51	$13
Net Income after Depreciation & Interest	$2,689,274	-$2,273	-$15,251	-$37,690	-$873,724	$501,004	$573,257	$677,167	$433,962	$332,082	$173,967	$152,322	$117,133	$142,421	$253,736	$244,684	$46,173	$46,914	-$9,454
Idaho Mine License Tax	$27,269		$0	$0	$0	$3,786	$4,411	$5,334	$3,299	$2,445	$1,144	$983	$693	$925	$1,871	$1,822	$273	$277	$0
Idaho Corporate Income Tax	$106,600		$0	$0	$0	$0	$0	$17,380	$22,256	$16,341	$7,338	$6,219	$4,212	$5,820	$12,370	$12,030	$1,304	$1,331	$0
Federal Income Tax	$300,879		$0	$0	$0	$0	$0	$49,055	$62,819	$46,122	$20,710	$17,553	$11,889	$16,426	$34,914	$33,954	$3,681	$3,756	$0
Net Income after Taxes	$2,254,526	-$2,273	-$15,251	-$37,690	-$873,724	$497,218	$568,846	$605,398	$345,588	$267,174	$144,774	$127,567	$100,339	$119,250	$204,581	$196,878	$40,915	$41,551	-$9,454
Cash Flow																			
Net Operating Income after Interest	$4,325,121	-$2,273	-$15,251	-$36,456	$369,592	$557,695	$618,067	$709,542	$457,618	$359,177	$206,248	$174,833	$130,681	$153,097	$265,767	$257,006	$54,470	$51,775	-$5,536
Working Capital																			
Account Receivables	$0				-$23,177.31	-$8,180.79	-$2,985.65	-$3,950.46	$11,071.82	$4,358.51	$6,665	$1,420.76	$1,859.96	-$723.23	-$4,579.95	$1,128.61	$9,093.13	$826	$5,283
Accounts Payable	$0				$7,437	$370.35	$473.97	$139.19	-$479.05	-$230.22	-$298	-$102.89	$3.40	-$215.40	-$123.71	-$748.25	-$612.30	-$533	-$2,837
Inventory (Parts)	$0			-$7,500	-$7,500	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$15,000
Total Working Capital	$0	$0	$0	-$7,500	-$23,241	-$7,810	-$2,512	-$3,811	$10,593	$4,128	$6,367	$1,318	$1,863	-$939	-$4,704	$380	$8,481	$293	$17,447
Capital Expenditures																			
Initial Capital	$1,218,935	$199,033	$448,961	$570,941	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Equipment Lease	$149,437	$1,115	$8,869	$34,029	$13,146	$14,299	$19,358	$16,303	$21,039	$4,220	$2,985	$3,298	$2,112	$2,621	$3,184	$1,697	$429	$599	$135
Sustaining Capital	$289,610	$0	$0	$0	$19,508	$19,278	$20,710	$14,179	$10,856	$61,386	$1,668	$12,415	$716	$5,145	$16,198	$1,671	$2,475	$1,629	$2,481
Total Capital Expenditures	$1,657,982	$200,148	$457,831	$604,970	$32,653	$33,576	$40,067	$30,482	$31,895	$65,606	$4,653	$15,712	$2,828	$7,766	$19,382	$3,368	$2,905	$2,228	$2,616
Cash Flow before Taxes	$2,667,138	-$202,421	-$473,082	-$648,925	$313,698	$516,308	$575,488	$675,249	$436,316	$297,699	$207,962	$160,439	$129,716	$144,393	$241,681	$254,018	$60,047	$49,840	$9,295
Cummulative Cash Flow before Taxes		-$202,421	-$675,503	-$1,324,428	-$1,010,729	-$494,421	$81,067	$756,316	$1,192,632	$1,490,331	$1,698,293	$1,858,731	$1,988,447	$2,132,840	$2,374,521	$2,628,539	$2,688,585	$2,738,425	$2,747,720
Taxes	$434,748	$0	$0	$0	$0	$3,786	$4,411	$71,769	$88,374	$64,908	$29,192	$24,754	$16,794	$23,171	$49,155	$47,806	$5,259	$5,364	$0
Cash Flow after Taxes	$2,232,391	-$202,421	-$473,082	-$648,925	$313,698	$512,522	$571,077	$603,480	$347,942	$232,791	$178,769	$135,684	$112,922	$121,222	$192,525	$206,212	$54,788	$44,476	$9,295
Cummulative Cash Flow after Taxes		-$202,421	-$675,503	-$1,324,428	-$1,010,729	-$498,208	$72,870	$676,350	$1,024,292	$1,257,083	$1,435,853	$1,571,537	$1,684,459	$1,805,681	$1,998,206	$2,204,418	$2,259,205	$2,303,682	$2,312,977
								-	-	-	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes		$000
NPV @ 0%	0.0%	$2,667,138
NPV @ 5%	5.0%	$1,598,616
NPV @ 7%	7.0%	$1,290,396
NPV @ 10%	10.0%	$919,133
IRR		24.3%
Payback	Years	2.9

Economic Indicators after Taxes		$000
NPV @ 0%	0.0%	$2,232,391
NPV @ 5%	5.0%	$1,319,814
NPV @ 7%	7.0%	$1,054,337
NPV @ 10%	10.0%	$733,218
IRR		22.3%
Payback	Years	2.9

Case B
Expenses and Results

Stibnite Gold Project
Midas Gold Corporation

Gold - $1,600/oz
Silver - $20/oz
Antimony - $3.50/lb

Operating Cost	$000	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30
Mining	$860,151	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$1,332,063	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Water Treatment Plant	$3,157	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
G&A	$358,607	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Operating Cost	$2,553,979	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Royalty	$114,079	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Property Taxes	$4,354	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	-$27,240	-$4,086	-$4,086	-$2,724	-$2,179	-$2,179	-$1,907	-$1,907	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation/Closure	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Production Cost	$2,645,172	-$4,086	-$4,086	-$2,724	-$2,179	-$2,179	-$1,907	-$1,907	$0	$0	$0	$0	$0	$0	$0	$0
Net Operating Income - EBITDA	$4,342,714	$4,086	$4,086	$2,724	$2,179	$2,179	$1,907	$1,907	$0	$0	$0	$0	$0	$0	$0	$0
Depreciation																
Initial Capital	$1,218,935	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Equipment Lease	$149,437	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital	$269,556	$3,631	$4,211	$4,620	$4,746	$5,455	$6,194	$6,940	$8,853	$9,722	$8,528	$7,124	$5,796	$4,695	$3,522	$2,186
Total Depreciation	$1,635,847	$3,631	$4,211	$4,620	$4,746	$5,455	$6,194	$6,940	$8,853	$9,722	$8,528	$7,124	$5,796	$4,695	$3,522	$2,186
Interest																
Capital Equipment Lease Interest	$17,593	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Interest	$17,593	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net Income after Depreciation & Interest	$2,689,274	$455	-$125	-$1,896	-$2,567	-$3,275	-$4,287	-$5,034	-$8,853	-$9,722	-$8,528	-$7,124	-$5,796	-$4,695	-$3,522	-$2,186
Idaho Mine License Tax	$27,269	$5	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Idaho Corporate Income Tax	$106,600	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Federal Income Tax	$300,879	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Net Income after Taxes	$2,254,526	$450	-$125	-$1,896	-$2,567	-$3,275	-$4,287	-$5,034	-$8,853	-$9,722	-$8,528	-$7,124	-$5,796	-$4,695	-$3,522	-$2,186
Cash Flow																
Net Operating Income after Interest	$4,325,121	$4,086	$4,086	$2,724	$2,179	$2,179	$1,907	$1,907	$0	$0	$0	$0	$0	$0	$0	$0
Working Capital																
Account Receivables	$0	$1,890	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Accounts Payable	$0	-$2,244	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Inventory (Parts)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Working Capital	$0	-$354	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Expenditures																
Initial Capital	$1,218,935	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Equipment Lease	$149,437	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital	$289,610	$2,408	$6,564	$7,299	$5,930	$8,493	$7,488	$9,218	$16,451	$6,391	$4,571	$2,225	$2,203	$989	$846	$18,219
Total Capital Expenditures	$1,657,982	$2,408	$6,564	$7,299	$5,930	$8,493	$7,488	$9,218	$16,451	$6,391	$4,571	$2,225	$2,203	$989	$846	$18,219
Cash Flow before Taxes	$2,667,138	$1,324	-$2,478	-$4,575	-$3,751	-$6,314	-$5,581	-$7,311	-$16,451	-$6,391	-$4,571	-$2,225	-$2,203	-$989	-$846	-$18,219
Cummulative Cash Flow before Taxes		$2,749,044	$2,746,565	$2,741,990	$2,738,240	$2,731,925	$2,726,344	$2,719,033	$2,702,582	$2,696,191	$2,691,620	$2,689,395	$2,687,192	$2,686,203	$2,685,357	$2,667,138
Taxes	$434,748	$5	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cash Flow after Taxes	$2,232,391	$1,319	-$2,478	-$4,575	-$3,751	-$6,314	-$5,581	-$7,311	-$16,451	-$6,391	-$4,571	-$2,225	-$2,203	-$989	-$846	-$18,219
Cummulative Cash Flow after Taxes		$2,314,296	$2,311,818	$2,307,243	$2,303,492	$2,297,178	$2,291,596	$2,284,285	$2,267,834	$2,261,443	$2,256,872	$2,254,647	$2,252,444	$2,251,455	$2,250,609	$2,232,391
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