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Stibnite Gold Project

Prefeasibility Study Technical Report

Valley County, Idaho

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MIDAS GOLD

DATE AND SIGNATURES PAGE

The effective date of this Report is December 8, 2014. The issue date of this Report is December 15, 2014. See Appendix I, Prefeasibility Study Contributors and Professional Qualifications, for certificates of Qualified Persons.

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1 SUMMARY

1.1 INTRODUCTION AND PURPOSE

Redevelopment of the Stibnite Gold Project (**Project**) has the potential to clean up an existing brownfield site, one that has been extensively mined for more than 80 years, and which could become one of the largest gold producers in the United States. This billion-dollar Project could create more than 700 jobs in Idaho during the first three years of construction and nearly 1,000 jobs in Idaho during 12 years of Project operations, while generating significant taxes and other benefits to the local, state and federal economies. This preliminary feasibility study (**PFS**) and related Technical Report (**Report**) provide 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.

Key considerations for the 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 society's present day needs for economic prosperity while remaining protective of the environment, and enhancing the ability of future generations to sustain their own needs.
- The Project is designed to ensure ongoing positive local and regional financial and social benefits through taxation, employment, and business opportunities in a region where the economy has suffered for more than a decade, resulting in some of the highest unemployment and lowest annual wages in Idaho.
- From the beginning, the Project has been designed for what will remain after closure. The plan for closure is protective of the environment and incorporates inherently stable, secure features that will provide the foundation for a naturally sustainable ecosystem.
- The Project design incorporates cleanup and repair of extensive historical mining-related impacts; much of the cleanup and repair would occur during initial construction and early operations.
- The new facilities contemplated for the Project are tightly constrained and, to a large extent, placed in historically impacted areas in order to minimize the incremental Project footprint.
- Salmon and other fishery enhancements are integral to the Project design. Removal of man-made barriers and reconstruction of natural habitat would allow salmon and other 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 potential, Mineral Resources, Mineral Reserves, mining method, process method, infrastructure, social and economic benefits, environmental protection, cleanup and repair of historical impacts, reclamation and closure concepts, capital and operating costs and an economic analysis for the Project.

1.2 KEY RESULTS

The Project consists of rehabilitating an existing brownfields site in an area of significant historical mining, including removal and reprocessing of the historic gold-silver-antimony tailings (**Historic Tailings)**, and mining the Yellow Pine, Hangar Flats and West End gold-silver-antimony 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. Midas Gold's plans for decommissioning the site include progressive and concurrent remediation and reclamation activities, beginning at the start of construction and continuing beyond the operations phase, through Project reclamation and closure.

The Stibnite Gold Project, as contemplated in the PFS, comprises:

- A design that minimizes Project footprint, and locates facilities within already impacted areas, and which incorporates a number of approaches to risk reduction, such as an improved access road to the site that avoids all major waterways.
- An extensive reclamation and remediation program for historical impacts to the site including, but not limited to, the recovery and reprocessing of Historic Tailings, restoration of fish passage during and after operations, removal of historical waste rock to an engineered waste rock storage facility, repair of the Blowout Creek channel that is a source of significant sedimentation, reforestation of impacted areas, stream channel repairs, etc. Many of these activities will occur during construction and/or relatively early in the mine life.
- Four deposits, including Historic Tailings, with combined Indicated Mineral Resources of 115.2 million short tons (**Mst**) or 104.5 million metric tonnes (**Mt**) grading 0.048 troy ounces per short ton (**oz/st**) Au or 1.63 grams per metric tonne (**g/t**) Au, 0.077 oz/st (2.65 g/t) Ag, and 0.07% Sb. The aggregate Indicated Mineral Resources contain 5.46 million oz (**Moz**) Au, 8.90 Moz Ag, and 155.2 million pounds (**Mlbs**) Sb.
- Combined Probable Mineral Reserves of 98.07 Mst (88.96 Mt) grading 0.047 oz/st (1.60 g/t) Au, 0.071 oz/st (2.43 g/t) Ag, and 0.07% Sb. Total contained metal in the Probable Mineral Reserves includes 4.58 Moz Au, 6.96 Moz Ag, and 137.0 Mlbs Sb.
- The four deposits combined also contain additional Inferred Mineral Resources of 10.8 Mst (9.8 Mt) grading 0.032 oz/st (1.10 g/t) Au, 0.049 oz/st (1.67 g/t) Ag, and 0.04% Sb that is not utilized in the PFS. The combined Inferred Mineral Resources contain 347 koz Au, 523 koz Ag, and 9.5 Mlbs Sb. ***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.***
- The mineralization is primarily hosted in sulfides, with modest amounts of oxides, both of which can be treated with different extraction processes in the same plant. Sulfide mineralization would be milled and treated with bulk flotation, or with sequential flotation when sufficient antimony is present, to produce two products: (1) an antimony concentrate for off-site shipment to a smelter, and (2) a gold concentrate. In both cases, sulfide gold concentrates would be further processed on-site using pressure oxidation (**POX**) followed by vat leaching to produce gold-silver doré. The oxide material would be milled and then vat leached to recover gold and silver only, with a significant portion processed during down times for the POX circuit.
- Production is recommended to average 8.05 Mst of ore fed to the crusher per year (22,050 short tons per day (**st/d**)) with an average strip ratio of 3.5:1; in metric units, this equates to 7.30 Mt and 20,000 t/d. With this production rate, the mine life would be approximately 12 years. The average mill feed gold grade for the Project is approximately 0.047 oz/st (1.60 g/t) containing 4,575 thousand ounces (**koz**) of gold with significant sliver and antimony credits.
- Payable metals for the Project total 4,006 koz of gold, 1,467 koz of silver, and 67,900 thousand lbs (**klbs**) of antimony for the Project life of mine.

- Total capital cost would be approximately $1,125 million, including start-up capital costs of $970 million, sustaining capital costs of $99 million, and closure costs of $56 million.
- Using the Base Case economic factors detailed in Section 22, the financial model yields a pre-tax net present value at a 5% discount rate (**$PTNPV_{5\%}$**) of $1,093 million and an after tax net present value at a 5% discount rate (**$ATNPV_{5\%}$**) of $832 million. As currently designed, the Project's Internal Rate of Return (**IRR**) is 19.3% with a payback period of approximately 3.4 production years.
- The $ATNPV_{5\%}$ for the Project is most sensitive to changes in revenue, which is manifested as changes in metal prices, gold grades, or gold recovery. For example, a 20% increase in gold price or gold grade raises the $ATNPV_{5\%}$ from $832 million to $1,369 million, a 63% increase for the base Case. Similarly, a decrease of 20% in gold grade, gold recovery, or gold price results in a 71% decrease in $ATNPV_{5\%}$ for the Base Case.
- A number of risks and opportunities have been identified, including the potential for additional gold production from within the current pit outlines, as well as some from outside, either of which could significantly enhance the economic outcomes for the Project.
- The closure concept for the Project envisions removal or demolishing of onsite facilities, comprehensive reclamation and reforestation of disturbed areas, permanent establishment of fish passage through the site, and a sustainable environment.

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.

1.3 REGULATORY INFORMATION

This Report has been prepared based on the results of a PFS 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.

This Report has been prepared under the direction of Independent Qualified Persons (**QP**) and 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. This Report supersedes and replaces the technical report entitled '*Preliminary Economic Assessment Technical Report for the Golden Meadows Project, Idaho'* prepared by SRK Consulting (Canada) Inc. and dated September 21, 2012 (**PEA**) and that report should no longer be relied upon.

For readers to fully understand the information in this Report, they should read the Report (to be available on SEDAR or at www.midasgoldcorp.com by the end of 2014) 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.

1.4 RELIANCE ON OTHER EXPERTS

Certain sections of the Report rely 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 Report have reviewed the information and conclusions provided that they are responsible for and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report.

1.5 PROPERTY DESCRIPTION AND LOCATION

The Stibnite Gold Project is located in central Idaho, USA. The Project lies approximately 100 miles (**mi**) northeast of Boise, Idaho, 38 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho. Figure 1.1 illustrates the location of the Project.

The Hangar Flats, West End, and Yellow Pine deposits, along with the Historic Tailings, lie within mineral concessions controlled by Midas Gold, as are other exploration prospects and targets identified in this Report. Mineral rights controlled by Midas Gold include patented lode claims, patented mill site claims, unpatented federal lode claims, and unpatented federal mill site claims 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 Stibnite Gold Project is located approximately 152 road-miles northeast of Boise, Idaho in an area of deeply incised drainage related to the East Fork of the South Fork of the Salmon River (**EFSFSR**) at an elevation of ~ 6,500 feet (**ft**) with nearby mountains rising to an elevation of approximately 7,800 to 8,900 ft.

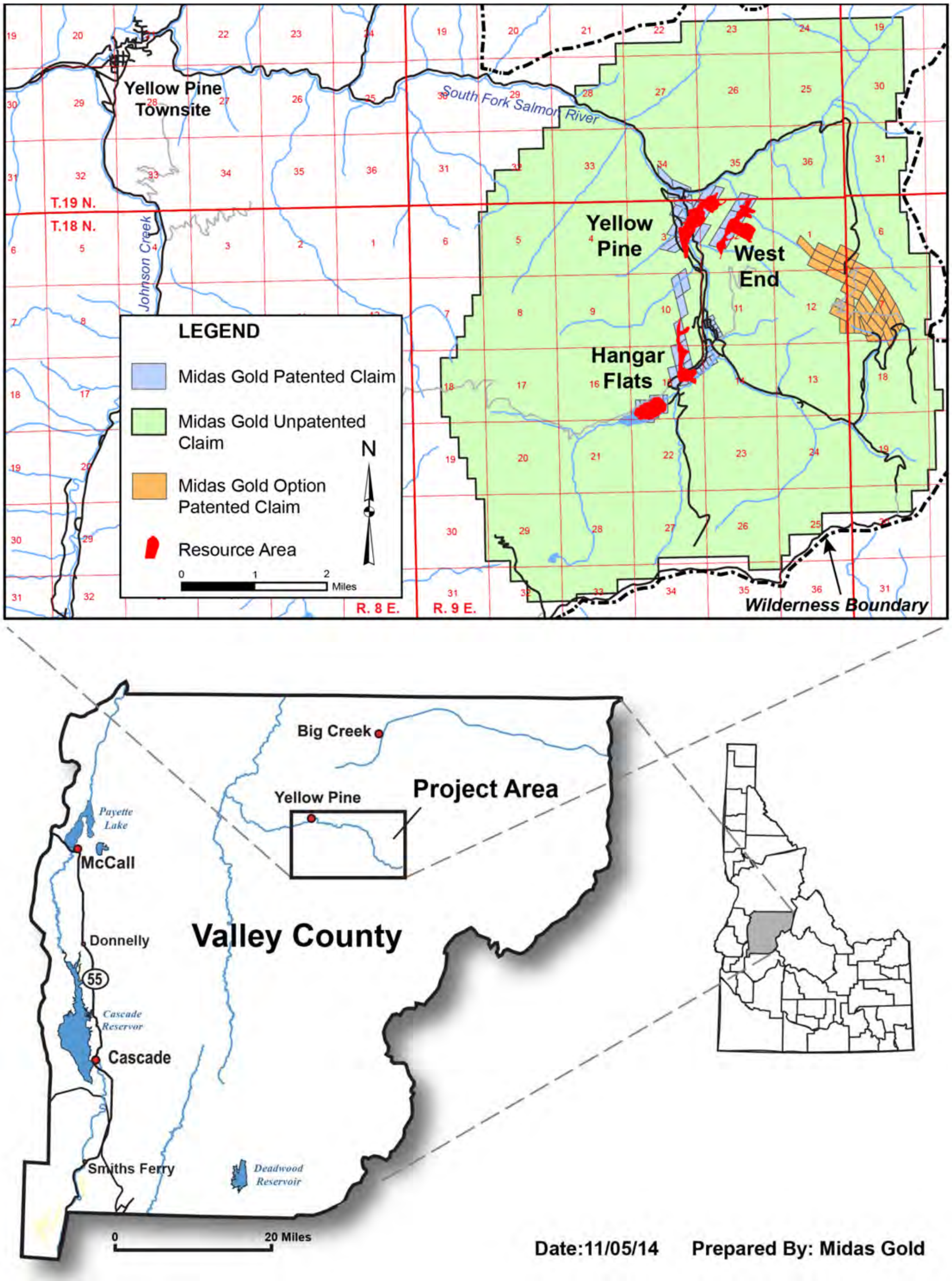
The climate is characterized by moderately cold winters and mild summers. Most precipitation occurs as snowfall in the winter and rain during the spring. The local climate allows for year round operations, as evidenced by historical production over extended periods, and climate information.

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. The closest rail is in Cascade, while the closest access for sea transportation is on the west coast of the US and Canada, or via the inland port of Lewiston, ID.

Power-lines 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, along an existing and previously used right-of-way.

Midas Gold has four permanent and three temporary water rights in the District.

Figure 1.1: Location Map of the Stibnite Gold Project



1.7 HISTORY

The Project is located in a past-producing area near the historical town of Stibnite. Since the late 1920s, gold, silver, antimony, tungsten, and mercury mineralized materials have been mined in the area by both underground and, later, open pit methods, creating numerous open pits, underground workings, large-scale waste rock dumps, heap leach pads, spent heap leach ore piles, tailings depositories, a mill site, three town sites, an airstrip, and other disturbances, some of which still exist today. Antimony-tungsten-gold sulphide milling operations ceased in 1952 as a result of lower metal prices following the end of the Korean War, while mercury operations on the Cinnabar claims continued until 1963. Exploration recommenced in 1974, followed by open pit mining and seasonal on-off heap leaching from 1982 to 1997. Midas Gold commenced its exploration activities in 2009.

Table 1.1 summarizes the approximate historical production for the Project by area; additional details are provided in Section 6.

Table 1.1: Estimated Historical Metal 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
Yellow Pine	1938 - 92	6,493,838	479,517	1,756,928	40,257	856,189
West End	1978 - 97	8,156,942	454,475	149,760	-	-
Totals		**14,954,633**	**985,602**	**2,088,551**	**44,015**	**856,256**

Note:
(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.

1.8 GEOLOGICAL SETTING AND MINERALIZATION

The Project area is underlain by pre-Cretaceous "basement" sediments, the Cretaceous-age Idaho Batholith (granitic), Tertiary-age intermediate to felsic intrusions and volcanics, younger unconsolidated sediments derived from erosion of the older sequences and glacial materials.

Large, north-south striking, steeply dipping to vertical structures exhibiting pronounced gouge and multiple stages of brecciation occur in the central and eastern portions of the property and are often associated with east-west and northeast-southwest trending splays and dilatant structures.

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), with gold almost exclusively in solid solution in these minerals.

Antimony mineralization occurs primarily associated with the mineral stibnite (Sb_2S_3). Zones of 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).

Metasediment-hosted mineralization has a similar sulfide suite and similar geochemistry to the intrusive hosted mineralization, but with higher carbonate content in the gangue and a much more diverse suite of late stage minerals.

1.9 DEPOSIT TYPES

The origin of the wide variety of mineralization occurrences at the Stibnite Gold Project is attributed to deep-seated intrusives and associated high temperature and high pressure processes to shallow lower temperature, lower pressure hydrothermal processes.

1.10 EXPLORATION

The District has been the subject of exploration and development activities for nearly 100 years. Numerous prospects have been discovered through the years using a variety of methods. Some of these prospects were developed into mines and others remain undeveloped; further, new ones may be discovered as the Project advances and the nature of mineralization previously exploited is better understood.

Midas Gold's analysis of historical data and its exploration since 2009 has identified a number of key exploration opportunities:

- There is potential at each of the Hangar Flats, West End and Yellow Pine deposits to increase Mineral Resources and Mineral Reserves at grades higher than cut-off, this potential includes conversion of currently Inferred Mineral Resources to higher confidence levels, conversion of currently unclassified material within the economic pits, and expansion potential immediately adjacent to the existing Mineral Resources and Mineral Reserves that could result in increased Mineral Reserves and reduced strip ratios;
- There is good potential to delineate high grade, Au +/- Sb, near surface underground mineral deposits at prospects such as Scout, Garnet and Upper Midnight (based on varying degrees of drilling already completed) that could provide supplemental early mine life, higher margin, mill feed;
- There is potential for the discovery and definition of additional mineral deposits along the main mineralized trends, such as between Hangar Flats and Yellow Pine, based on exploration and drilling completed to date;
- A number of other prospects have been defined to varying degrees, up to and including detailed drilling, that indicate potential for bulk tonnage disseminated Au deposits similar to those containing the current Mineral Resources – these include the Rabbit and Ridgetop-Cinnamid prospects; and
- A number of prospects, such as Mule, have different geologic settings to those discussed above but which could potentially develop into significant mineral deposits.

Note: There has been insufficient exploration to define Mineral Resources on these prospects and it is uncertain whether further exploration will result in the targets being delineated as either Mineral Resources or Mineral Reserves.

1.11 DRILLING

The Project area, including the three main deposits, has been drilled by numerous operators, totaling 773,744 ft in 2,606 drill holes, of which Midas Gold drilled 550 holes, totaling over 326,275 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. All Midas Gold holes were surveyed and recoveries were generally good to excellent. Industry standard QA/QC procedures were used by Midas Gold, including sample security, blanks, standards and duplicates and these procedures were verified by the Independent QP.

1.12 DATA VERIFICATION

Extensive data verification programs have been undertaken by numerous independent consultants for Midas Gold and by Midas Gold personnel, as discussed in previous NI 43-101 technical reports (SRK, 2011; SRK, 2012) and discussed in this Report. These verification programs have been essential in ensuring that the datasets used for the Mineral Resource estimates are validated and verified as adequate for the estimation of Mineral Resources for each of the respective deposits. 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 Historic Tailings deposits is adequate for this purpose and may be relied upon to report the Mineral Resources and Mineral Reserves contained in this Report.

1.13 MINERAL PROCESSING AND METALLURGICAL TESTING

Subsequent to the test work program undertaken for the 2012 PEA and other historical testing undertaken by prior owners and operators, a total of seven flowsheet development composites and 114 variability composites were prepared for metallurgical testing in support of the PFS from the more than 800 samples collected from the Project. Mineralogical work confirmed that the gold is mostly present in both pyrite and (to a much lesser extent) arsenopyrite, at concentrations that are usually high enough to economically justify flotation concentration followed by POX of the sulfides and cyanidation of the released gold. Oxide zones, mostly in the West End Deposit, contained very fine-grained, discrete gold available to direct cyanidation. Antimony occurs as stibnite, which is typically coarse-grained when occurring in higher-grade samples.

After the PEA related testing, grindability testing was conducted on all deposits, including two JK Drop Weight tests, 22 JK SAG mill characterization (**SMC**) tests, 10 crusher work index and abrasion index tests, 8 rod mill work index, and 24 ball mill work index tests. All composites indicate medium hardness (ball mill work index 13.0 to 14.1 kWh/t) and are amenable to semi-autogenous grinding (**SAG**) milling, though West End is somewhat more resistant to SAG milling, and Yellow Pine appears to be slightly more resistant to ball milling.

Over 300 metallurgical tests were completed on samples from the Yellow Pine, Hangar Flats, West End and Historic Tailings deposits as part of the PFS; in addition, more than 130 tests were completed for the PEA and numerous test programs were completed by prior owners and operators. Despite some mineralogical differences between the deposits, developmental metallurgical testwork has been able to identify a single, modular flowsheet that proved successful when applied to each of the deposits, making it possible to design a single plant that can process all ores from the Project as they are mined. This plant would, when antimony grades are high enough, float off the stibnite to create a saleable antimony concentrate, and then all ores (whether or not antimony is pre-floated) would be subject to bulk flotation of sulfides to produce an auriferous concentrate. Limited testwork on the Historic Tailings showed that they could be successfully co-processed through either flowsheet with the early production Yellow Pine ores.

At most times, the rougher flotation concentrates are expected to meet the POX sulfur content requirements and not require further cleaning, although West End concentrates require additional processing to reject carbonate-bearing (**CO_3**) minerals from the gold concentrates to produce a POX friendly concentrate.

Developmental leaching test work was also undertaken on the West End oxide ores, as well as on select flotation tailings produced from partially oxidized mineralization from Hangar Flats and West End. West End oxide leach studies indicate that 96% of the extracted gold leaches in the first six hours, with another 2% leached over the final 18 hours. Leach studies on the flotation tailings from Hangar Flats and West End indicate that any leachable gold in the flotation tailings is also fast leaching and could contribute significantly to gold recovery. Leach studies on the flotation tailings from Yellow Pine suggest little incremental recovery, but leaching them would provide additional assurance against losses of cyanide-soluble gold.

The projected overall recoveries for each deposit are shown on Figure 1.2 and Figure 1.3.

Figure 1.2: Gold and Silver Recoveries to Doré

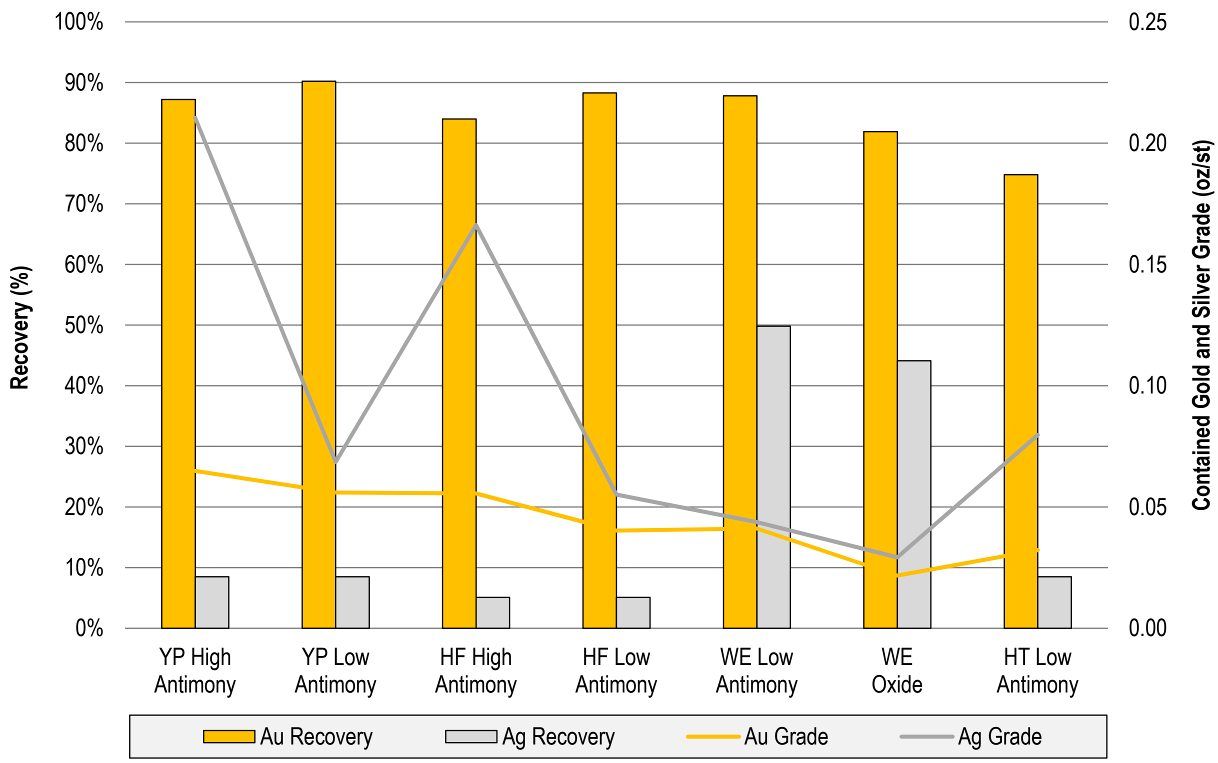
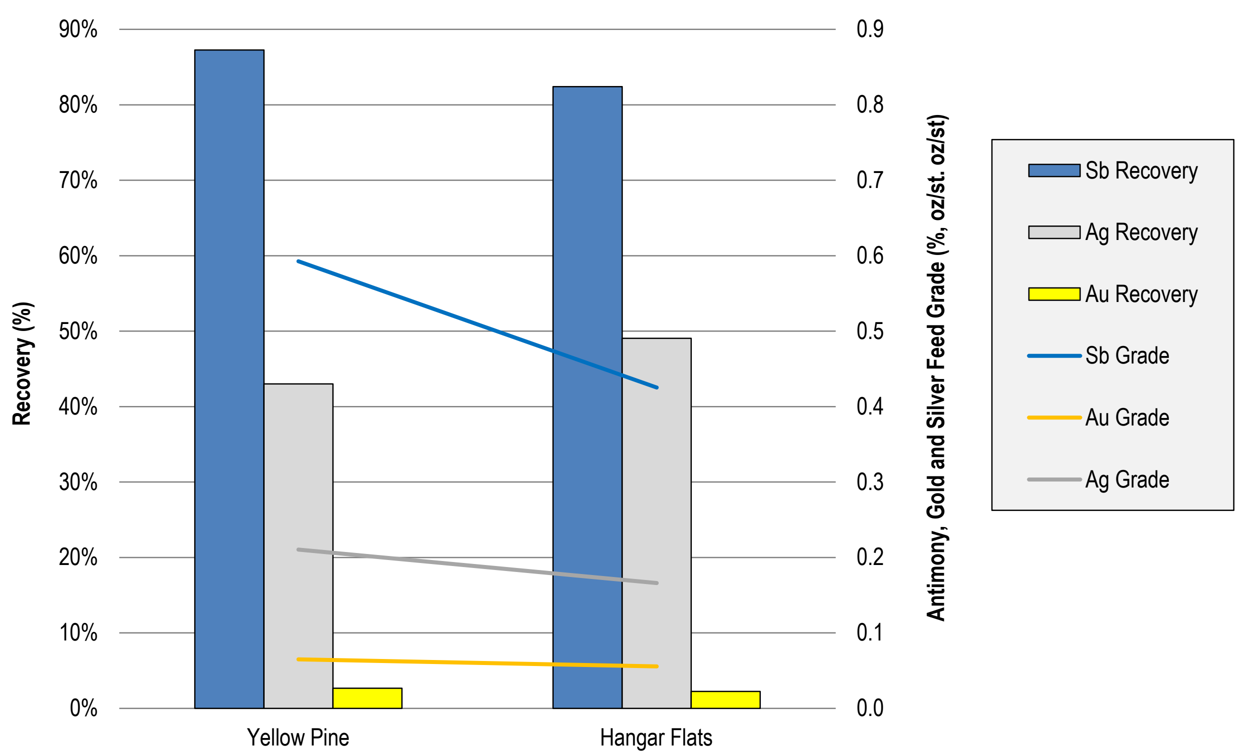


Figure 1.3: Antimony Concentrate Recoveries



1.14 MINERAL RESOURCE ESTIMATES

The Mineral Resource estimates for Hangar Flats, West End and Yellow Pine deposits, and the Historic Tailings, were prepared to industry standards and best practices using commercial mine-modeling and geostatistical software by third party consultants and verified by an Independent QP.

The Mineral Resources were initially calculated using a gold price of $1,400/oz and parameters defined in Section 14; based on this, the open pit sulfide cut-off grade was calculated as approximately 0.016 oz/st (0.55 g/t) Au and the open pit oxide cut-off grade calculated as approximately 0.010 oz/st (0.35 g/t) Au. However, Midas Gold elected to report its Mineral Resources at a 0.022 oz/st (0.75 g/t) Au sulfide cut-off grade and 0.013 oz/st (0.45 g/t) Au oxide cut-off grade, which is equivalent to utilizing the cost assumptions stated in Section 14 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides. The consolidated Mineral Resource statement for the Project is shown in Table 1.2.

Table 1.2: Consolidated Mineral Resource Statement for the Stibnite Gold Project

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	21,389	1.60	1,103	4.30	2,960	0.11	54,180
West End	35,974	1.30	1,501	1.35	1,567	0.008	6,563
Yellow Pine	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	***104,506***	***1.63***	***5,464***	***2.65***	***8,904***	***0.07***	***155,169***
Inferred							
Hangar Flats	7,451	1.52	363	4.61	1,105	0.11	18,727
West End	8,546	1.15	317	0.68	187	0.006	1,083
Yellow Pine	9,031	1.31	380	1.50	437	0.03	5,535
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	***25,168***	***1.32***	***1,066***	***2.15***	***1,743***	***0.05***	***25,908***

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 in order 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 Historic Tailings Mineral Resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 1.3. Antimony zones are reported only if they lie within gold Mineral Resource estimates.

Table 1.3: Antimony Sub-Domains Consolidated Mineral Resource Statement

Classification	Tonnage (kt)	Gold Grade (g/t)(3)	Contained Gold (koz)	Silver Grade (g/t)(3)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Total Indicated	12,564	1.98	800	6.23	2,518	0.50	138,218
Total Inferred	1,735	1.74	97	6.88	384	0.60	22,959

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. 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.*
(2) *Open pit antimony sulfide Mineral Resources are reported at a cutoff grade 0.1% antimony within the overall 0.75 g/t Au cutoff.*
(3) *Includes contributions from Hangar Flats, Yellow Pine and Historic Tailings. See Section 14 for details.*

1.15 MINERAL RESERVE ESTIMATES

The qualified person (**QP**) for the estimation of the Mineral Reserve was John M. Marek, P.E. of Independent Mining Consultants, Inc. The Mineral Reserves were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mr. Marek 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 Mineral Reserve at a higher level of risk than any other North American developing projects.

The Mineral Reserve was developed by allowing only 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; further blocks needed to be economic based on gold content alone before being categorized as a Mineral Reserve. A series of floating cones were developed by varying the gold price from $200/oz to $1,500/oz and then evaluated at a $1,200/oz price for gold without changing the size of the cone; for Yellow Pine, an $800/oz cone was selected as optimal, while $1,100/oz cones were selected for Hangar Flats and West End.

Based on the longer-term nature of the Project, cutoff grades for Mineral Reserves were developed assuming long term metal prices of $1,350/oz gold, $22.50/oz silver, and $4.50/lb antimony for material lying within the cones selected above. Confidence classification was based on gold estimation.

The cut-off grade is defined by a term called "Net of Process Revenue" (**NPR**) which takes into account final PFS processing recoveries, processing costs, and smelter terms (see Section 15), with any block with a NPR greater than zero meets the requirement for internal cutoff grade. The processing costs for ore range from $9.07/st for oxides to $17.00/st for high antimony sulfides with an additional $3.40/st of ore for G&A. Therefore the NSR equivalent of the cut-off grade range is: $12.47/st – $20.40/st. The Mineral Reserves are summarized in Table 1.4.

Table 1.4: Stibnite Gold Project Probable Mineral Reserve Estimate (Imperial & Metric Units)

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony	Silver
Imperial Units	**(kst)**	**(oz/st)**	**(%)**	**(oz/st)**	**(koz)**	**(klbs)**	**(koz)**
Yellow Pine	43,985	0.057	0.098	0.090	2,521	86,376	3,973
Hangar Flats	15,430	0.045	0.132	0.086	690	40,757	1,327
West End	35,650	0.035	0.000	0.040	1,265	-	1,410
Historic Tailings	3,001	0.034	0.165	0.084	102	9,903	252
Total Probable Mineral Reserve(1)	**98,066**	**0.047**	**0.070**	**0.071**	**4,579**	**137,037**	**6,962**
Metric Units	**(kt)**	**(g/t)**	**(%)**	**(g/t)**	**(t)**	**(t)**	**(t)**
Yellow Pine	39,903	1.97	0.098	3.10	78.4	39,179	123.6
Hangar Flats	13,998	1.53	0.132	2.95	21.5	18,487	41.3
West End	32,341	1.22	0.000	1.36	39.3	-	43.9
Historic Tailings	2,722	1.17	0.165	2.88	3.2	4,492	7.8
Total Probable Mineral Reserve(1)	**88,964**	**1.60**	**0.070**	**2.43**	**142.4**	**62,159**	**216.5**

Notes:
(1) Metal prices used for Mineral Reserves: $1350/oz Au, $22.50/oz Ag, $4.50/lb Sb.
(2) Block MUST be economical based on gold value only in order to be included as ore in Mineral Reserve.
(3) Numbers may not add exactly due to rounding.

Mineral Reserves exclude approximately 10.8 Mst with average grades of 0.032 oz/st (1.10 g/t) Au, 0.049 oz/st (1.67 g/t) Ag and 0.05% Sb that are Inferred Mineral Resources that lie within the Mineral Reserve pit limits; conversion of some or all of these tons would increase payable metal and reduce strip ratios. **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Inferred Mineral Resources 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.**

1.16 MINING

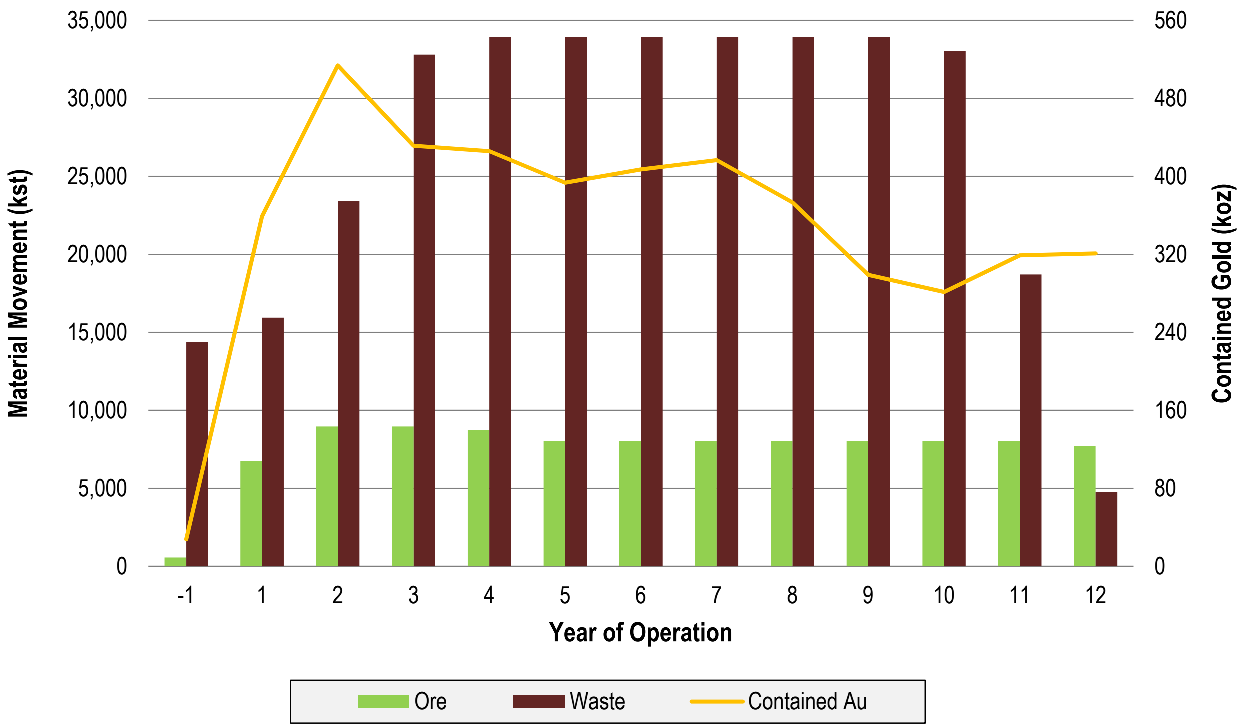
The mine plan developed for the Project incorporates the mining of the three *in situ* Mineral Deposits: Yellow Pine, Hangar Flats, and West End and their related waste rock, and the re-mining of Historic Tailings along with its cap of spent heap leach ore (**SODA**). Ore from the three pits would be sent to a centrally located crusher while the Historic Tailings would be fed by slurry into the process plant's grinding circuit. Waste rock would be sent to four distinct destinations: the tailings storage facility (**TSF**), the main Waste Rock Storage Facility (Main **WRSF**), the West End Waste Rock Storage Facility (West End WRSF), and to the Yellow Pine pit as backfill. The general sequence of mining would be the Yellow Pine deposit first, Hangar Flats second, and West End third. This planned sequence is driven by the need to backfill the Yellow Pine pit with waste rock from the West End pit in order to restore the original gradient of the EFSFSR while using environmentally appropriate carbonate-rich material for such backfill. This order generally follows a sequence of mining gold ounces from highest grade to lowest grade, and lowest cost to highest cost. The Historic Tailings, which lie within the footprint of the Main WRSF, would be removed during the first four years of the mine schedule to make the necessary space for the Main WRSF.

Mining at the Stibnite Gold Project would be accomplished using conventional open pit hard rock mining methods. Mining is planned to deliver 8.05 Mst of ore to the crusher per year (22,050 st/d), with stockpiling by ore type (low antimony sulfide, high antimony sulfide and oxide). Batches of oxide and sulfide material would be sent to the crusher; the oxide feed would be vat leached while the sulfide material would be floated to produce up to two concentrates: (1) an antimony concentrate, when there is sufficient antimony to justify recovering it, to be sent offsite and (2) a gold-bearing sulfide concentrate that would be oxidized in an autoclave and then sent to agitated leach tanks for gold-silver leaching.

The PFS mine plan schedules 98.066 Mst of ore to be fed to the processing plant from Yellow Pine, Hangar Flats and West End pits. The mining sequence requires the waste stripping to average 3.5:1 (waste rock: ore) for the first 3 years; then the stripping ratio would grow to 4.2:1 for years 4 through 9 after which it would drop to an average of 2.4:1 for the final 3 years. During the first four years, 3.0 Mst of Historic Tailings would be fed to the processing plant at a stripping ratio of 2.0:1 (SODA:tailings). The life-of-mine (**LOM**) strip ratio averages 3.5:1.

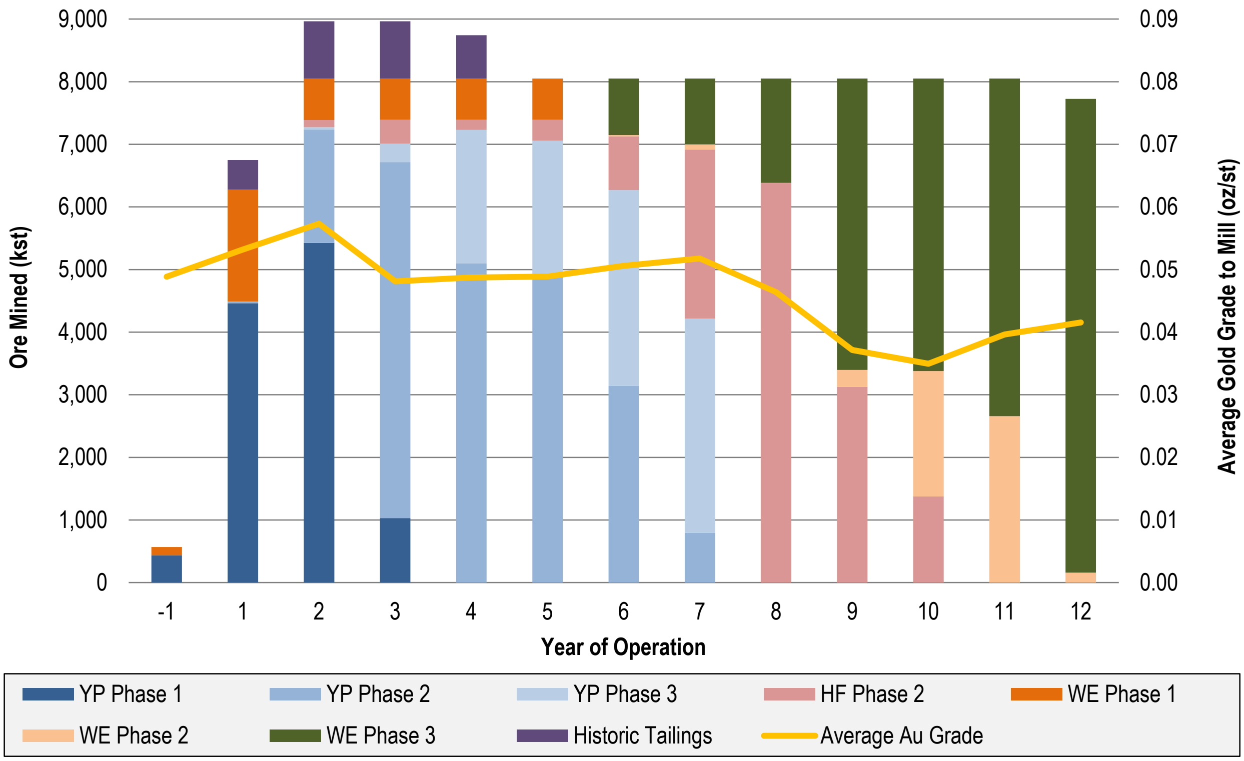
Figure 1.4 is a graphical depiction of the ore and waste rock movements from the mining phases by period and the contained gold ounces for the potential mine schedule for the Stibnite Gold Project; preproduction material from Year -1 would be processed in Year 1.

Figure 1.4: District Ore and Waste Movements and Ounces of Contained Gold Mined by Year



A summary of the mill feed by deposit is provided on Figure 1.5. This figure represents the Mineral Reserve because the Probable Mineral Reserve corresponds to the total ore processed in the mine.

Figure 1.5: Ore Mining Schedule by Deposit and Phase



A summary of the mill feed statistics by ore type is provided in Table 1.5

Table 1.5: LOM Mill Feed Statistics by Ore Type

Item	Unit	Value
General LOM Production Statistics		
Waste Rock Mined	Mst	346.7
Ore Mined	Mst	98.1
Strip Ratio (waste rock tons : ore tons).	st:st	3.5:1
Daily Mill Throughput	st/d	22,050
Annual Mill Throughput	Mst	8.05
Mine Life	production years	12
LOM Average Mill Head Grade		
Tonnage Milled	Mst	98.1
Gold Feed Grade	oz/st Au	0.047
Silver Feed Grade	oz/st Ag	0.071
Antimony Feed Grade	% Sb	0.070
Oxide Ore		
Tonnage Milled	Mst	10.7
Gold Feed Grade	oz/st Au	0.025
Silver Feed Grade	oz/st Ag	0.030
Antimony Feed Grade	% Sb	-

Item	Unit	Value
High Antimony Ore		
Tonnage Milled	Mst	11.0
Gold Feed Grade	oz/st Au	0.061
Silver Feed Grade	oz/st Ag	0.193
Antimony Feed Grade	% Sb	0.528
Low Antimony Ore (includes Historic Tailings)		
Tonnage Milled	Mt	76.3
Gold Feed Grade	oz/st Au	0.048
Silver Feed Grade	oz/st Ag	0.059
Antimony Feed Grade	% Sb	0.014

Mining would be performed with up to eighteen 200 st class haul trucks loaded by up to four 23.5 cubic yard front end loaders. The trucks would be light-body versions with an actual haulage capacity of 220 st. Blast holes would be 7-7/8" in diameter drilled by up to four drill rigs. An auxiliary fleet comprising dozers, motor graders water trucks and other ancillary equipment is also included in equipment requirements.

The overall gold recoveries to doré are expected to average approximately 90% from Yellow Pine, 87% from Hangar Flats, 86% from West End, and 75% from the Historic Tailings. When processing material containing more than 0.1% Sb, antimony recoveries are expected to average 82% for Hangar Flats and 87% for Yellow Pine, with minor gold and silver contained in the antimony concentrate.

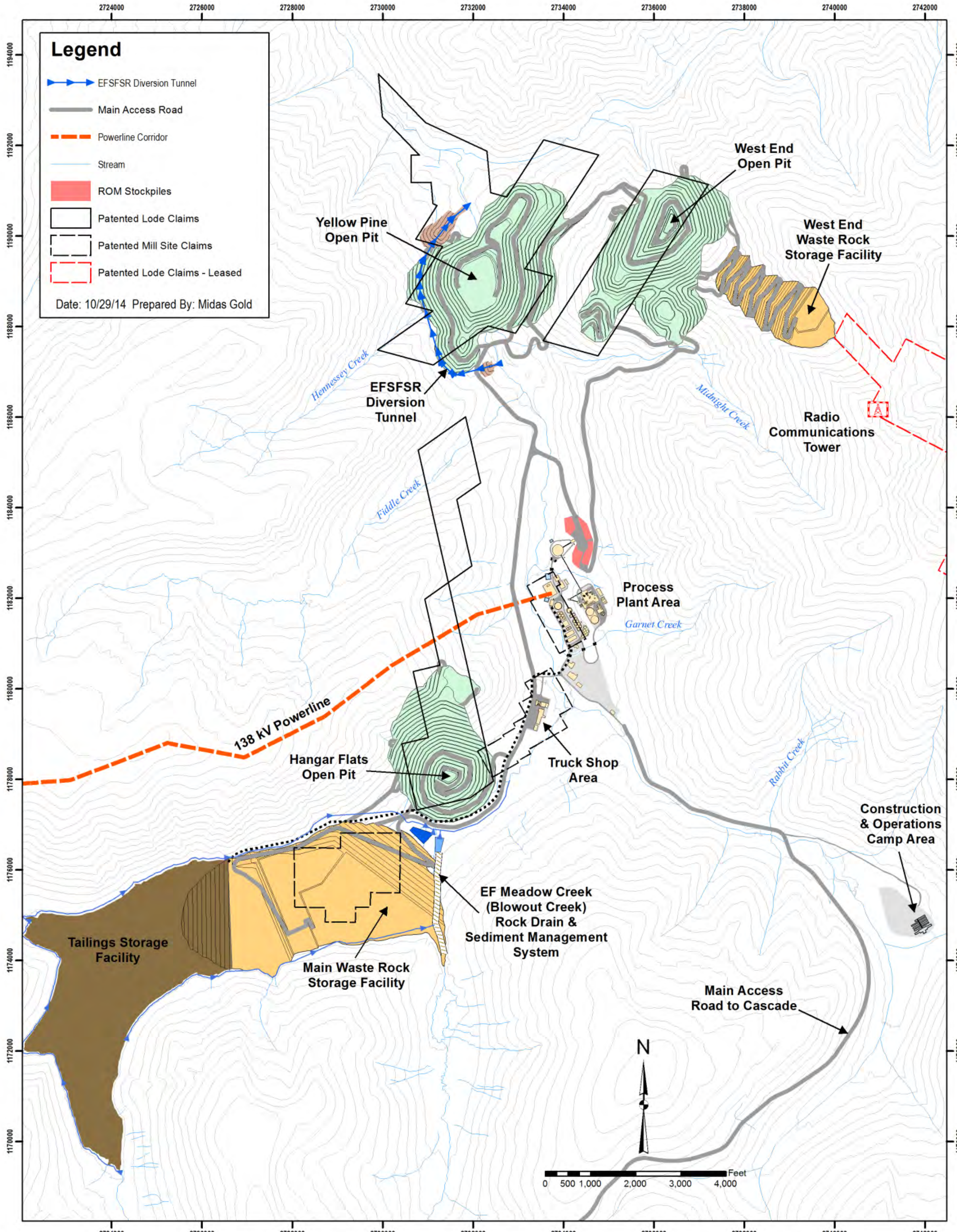
Figure 1.6 is a general overview of the mine site at the end of mine life prior to closure and reclamation.

1.17 RECOVERY

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 22,050 st/d, or 8.05 Mst/y. Additionally, the Historic Tailings would be reprocessed early in the mine life to recover precious metals and antimony, and to provide space for the Main WRSF.

The overall gold recoveries to doré are expected to average approximately 90% from Yellow Pine, 87% from Hangar Flats, 86% from West End, and 75% from the Historic Tailings. When processing material containing more than 0.1% Sb, antimony recoveries are expected to average 82% for Hangar Flats and 87% for Yellow Pine, with minor gold and silver contained in the antimony concentrate.

Figure 1.6: Overall Site Layout



1.18 PROCESS OPERATION COMPONENTS

Run-of-mine (**ROM**) material would be crushed and milled, then flotation would be used to recover antimony as a stibnite flotation concentrate (with some silver and minor gold) when there is sufficient antimony to justify it. For all sulfide ore, an auriferous bulk sulfide flotation concentrate would be produced and oxidized in an autoclave. The autoclave residue and flotation tailings would be processed through conventional cyanidation and, doré bars produced containing gold and silver. Historic Tailings would be introduced into the ball mill during the first 3 - 4 years of operation. Tailings from the operation would be deposited in a geomembrane-lined TSF. 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 1,225 short tons per hour (**st/h**).
- **Grinding Circuit** – The grinding circuit incorporates a single semi-autogenous (**SAG**) mill, single ball mill design with an average utilization of 92%, yielding an instantaneous design-throughput of 998.5 st/h. When Historic 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 ($\mathbf{P_{80}}$) 75 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 concentrate.
- **Pressure Oxidation Circuit** – Two concentrate surge tanks would be pumped to the autoclave feed tank, which would feed the autoclave. The autoclave is designed to provide one hour of retention time at 428 degrees Fahrenheit to oxidize the sulfides and liberate the precious metals. Autoclave discharge would be processed through flash vessels and gas discharge is processed through a scrubber. Slurry discharge from the flash vessels would be processed through the basic ferric sulfate (**BFS**) re-leach tanks to stabilize the solids prior to cyanide leaching.
- **Oxygen Plant** – An oxygen plant producing 670 st/d of gas at 95 percent oxygen and a gauge pressure (**psig**) of 570 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.
- **Oxidized Concentrate Processing** – Post-POX, the concentrate stream would be conditioned with lime and leached for 24 hours and discharged to a six stage pump-cell carbon-in-pulp (**CIP**) circuit for precious metal recovery from this high grade stream. The CIP tailings would be discharged to the flotation tailings leach circuit for extended retention time and to minimize reagent costs for the tailings leach system.
- **Oxide Carbon-in-Leach and Tailings Detoxification** – A (**CIL**) circuit was included in the design of the process plant to recover gold from non-refractory material in the flotation tailings, and in oxide material from the West End deposits that would be processed during oxidation circuit scheduled maintenance periods.
- **Carbon Handling** – Loaded carbon from the CIP circuit would be processed through a conventional carbon handling circuit.
- **Gold Room –** Precious metals would be recovered from the strip solution by electrowinning.
- **Tailings –** 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.

1.19 PROJECT INFRASTRUCTURE

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 streams, creeks and rivers, and this route respects the advice and privacy of community members close to the Project location.

Onsite and Offsite Facilities

In an effort to reduce traffic to and from the Project site and to reduce housing requirements at the site, administrative offices for Project would be located in or near the town of Cascade (the **Cascade Complex**). The Cascade Complex would include offices for some managers, safety and environmental services, human resources, purchasing, and accounting personnel. The Cascade Complex would also have a small warehouse, a parking area for trucks to check-in and assemble prior to traveling to the Project site and the main assay laboratory.

Midas Gold currently has an on-site facility capable of housing approximately 60 and feeding 125 workers per 12 hour shift. To manage the estimated peak construction workforce of 1,000-persons, the existing exploration camp would be relocated and expanded to provide the necessary accommodations. The operations camp would be developed by upgrading, and downsizing, the construction camp to meet the needs of the operations staff that would peak at over 500 persons.

Power Supply and Transmission

Grid power was selected as the best alternative for the electrical power supply for the Project based on its low operating cost and likely lowest environmental impact. In order provide the necessary power, the existing grid system would need to be upgraded to support the full anticipated 50 megawatt (**MW**) load of the Project. The upgrades would include an upgrade of approximately 42 mi of 69 kilovolt (**kV**) lines to 138 kV, new 138 kV substations at Lake Fork, Cascade, and Warm Lake, as well as measures to strengthen the voltages on the IPCo system. In addition, IPCo would re-supply small consumers between Warm Lake and Yellow Pine via a replacement 12.5 kV line. Construction power supply would be provided by three diesel generators that would then be used as emergency backup for the remainder of the operations of the Project.

Water Management and Supply

Water management infrastructure would be needed for surface water and sediment management and to provide water supply for both personnel and the operations. The PFS provides the framework for a comprehensive approach to water management at the Project site, addressing water management objectives for construction, operation, and post-closure. Key elements include segregation of process water, contact water, untreated stormwater, and sanitary waste from the environment, provision for fish passage around and then through the Yellow Pine pit during operations and after closure respectively, clean-up of legacy issues in the Project area, and reclamation and closure of the site to achieve acceptable and sustainable water quality.

Waste Management

Mine waste requiring on-site management includes waste rock from the three open pits, flotation and POX tailings from ore processing, and historic mine waste (spent heap leach ore from SODA and the Hecla heap, as well as historical waste dumps) exposed during construction and mining. The existing Historic Tailings would be reprocessed, and subsequently commingled with the rest of the tailings. A single TSF would be constructed to retain all tailings from the processing of the various ore types. The TSF would consist of a rockfill dam and a geosynthetic-lined impoundment that would be constructed in stages throughout the Project life. A majority of the waste rock would be deposited in the main WRSF located downstream of the TSF dam and would act as a buttress (enhancing dam stability), used as rockfill in TSF construction, or placed as backfill within mined-out areas of the pits to facilitate closure and reclamation. Current test work indicates no need for special handling of any of the waste materials. Spent ore and waste rock from previous on-site operations would be used as a construction material in the TSF. With SODA material included, the TSF dam and WRSF combined would hold 210 Mst of waste rock and overburden. Most of the waste from the West End pit would be used to backfill portions of the West End and Yellow Pine pits, with the remainder placed at the TSF and West End WRSF.

A geochemical characterization program was carried out for mine waste rock materials, including the spent ore on the SODA, which provides a basis for assessment of the potential for metal leaching and acid rock drainage, prediction of contact water quality, and evaluation of options for design, construction, and closure of the mine facilities. The results of the static geochemical test work demonstrate that the bulk of the Project waste rock material is likely to be net neutralizing and presents a low risk for acid generation, while there is still a potential to leach some constituents under the neutral to alkaline conditions (i.e. arsenic and antimony) both of which are currently elevated in ground and surface waters due to the naturally high geochemical background of these metals in the District and impacts from past mining activities. Similarly, bulk flotation tailings are expected to generate neutral pH drainage and require no special disposal considerations to prevent acidic drainage, and POX tailings will be blended with the bulk flotation tailings in order to benefit from their buffering capacity.

1.20 MARKET STUDIES AND CONTRACTS

The economic analysis completed for this PFS assumed that gold and silver production in the form of doré with payabilities, refining and transport charges as provided in Table 1.6.

Table 1.6: Doré 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

Table 1.7 summarizes the antimony concentrate payables and transportation charge assumptions for this PFS.

Table 1.7: 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

The metal prices selected for the four economic cases in this Report are shown in Table 1.8.

Table 1.8: Assumed Metal Prices by Case

Case	Metal Prices			Basis
	Gold ($/oz)	Silver(1) ($/oz)	Antimony(1) ($/lb)	
Case A	1,200	20.00	4.00	Lower-bound case that reflects the lower prices over the past 36 months and spot on December 1, 2014.
Case B (Base Case)	**1,350**	**22.50**	**4.50**	Approximate 24-month trailing average gold price as of December 1, 2014.
Case C	1,500	25.00	5.00	Approximate 48-month trailing average gold price as of December 1, 2014.
Case D	1,650	27.50	5.50	An upside case to show Project potential at metal prices approximately 20% higher than the base case.

Note:
(1) Prices were set at a constant gold:silver ratio ($/oz:$/oz) of 60:1 and a constant gold:antimony ratio ($/oz:$/lb) of 300:1 for simplicity of analysis, although individual price relationships may not be as directly correlated over time. Historic gold:silver ratios have averaged around 60:1.

1.21 ENVIRONMENTAL STUDIES

The Project area has been mined extensively for tungsten, antimony, mercury, gold, and silver since the early 1900s, providing strategic metals to the United States during war time critical minerals shortages, generating substantial economic benefit to the local counties and the State of Idaho, and providing much needed jobs and support to local businesses for nearly 100 years. These various historic mining efforts have left significant legacy environmental impacts that persist to this day, although multiple cleanup efforts undertaken by federal and state agencies and private entities have mitigated some of those historic impacts. Historic mining impacts have been compounded by extensive forest fires and subsequent damage from soil erosion, landslides and debris flows and resultant sediment transport.

In conjunction with the redevelopment of the Project area outlined in the PFS, Midas Gold has developed a plan to restore much of the site by removing existing barriers to fish migration and re-establishing salmon and steelhead fish passage, removing and reprocessing unconstrained historic tailings, reusing historic spent ore material for construction, restoring stream channels, and implementing sediment control projects such as repairing on Blowout Creek, as well as extensive reforestation of the Project area. Midas Gold has endeavored to minimize the Project's footprint and related impacts by siting facilities and roads on previously disturbed ground and away from riparian areas, provided for a new access road that avoids rivers and large waterways, and would connect to grid power to minimize fossil fuel consumption and haulage.

Baseline Studies and Existing Conditions

An extensive set of baseline data demonstrating historic and existing conditions exists for the Project site, including those collected by contractors for the US Forest Service (**USFS**) and the US Environmental Protection Agency (**EPA**) that determined there were no unacceptable risks to the environment or human health and that there were no populations (fish, wildlife, or human) shown as having a "likely" risk. In 2001, the EPA and the Bureau of Environmental Health and Safety, Division of Health, Idaho Department of Health and Welfare, determined the risk to be too low for listing on the National Priorities List. In 2009 and 2010, contractors to Midas Gold conducted Phase I and Phase II Environmental Site Assessments, as prescribed by ASTM International (**ASTM**) Standard Practices; these assessments determined that there were no imminent threats to human health or the environment, but that there was a number of pre-existing significant and moderate recognized environmental conditions.

In 2011, Midas Gold retained environmental consulting firms to conduct technical adequacy audits of all existing environmental information and to develop individual work plans to conduct an environmental baseline collection program. These workplans were developed with input from involved state and federal agencies in order to establish the existing environmental conditions, identify and quantify environmental risks and liabilities, and monitor for potential impacts from onsite activities. Work programs commenced in 2011 and will continue into 2015 and beyond to ensure an adequate baseline accurately describe the existing environment at the “brownfield site”, and allow for a "full and fair" discussion of all potentially significant environmental impacts in the event that the Stibnite Gold Project moves forward.

Consent Decrees

Several of the patented lode and mill site claims acquired by Midas Gold are subject to consent decrees entered in the US District Courts involving or pertaining to environmental liability and remediation responsibilities with respect to the affected properties, which provide regulatory agencies access and the right to conduct remediation activities and also require 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.

Permitting

Should a decision be made to file a Plan of Operations (**PoO**), approval of any Final PoO / Reclamation Plan for the Project would require an environmental analysis in compliance with the National Environmental Policy Act (**NEPA**), which requires federal agencies to study and consider the likely environmental impacts of the proposed action before taking whatever federal action is necessary for the Project to proceed. An EIS serves as an "overarching” permit requirement, as well as that for water discharge; waste and tailings placement and endangered species authorization. The EIS Record of Decision (**ROD**) effectively drives the entire permitting process, since a favorable ROD is required before these important clearances can be obtained. State and local permitting processes would be integrated and proceed concurrent with the EIS, and include air quality, cyanide, land application of water, groundwater, water rights, dam safety, reclamation, building permits, sewer and water systems, etc. Midas Gold believes it will be beneficial to have all permit processes integrated into the Idaho Joint Review Process (**IJRP**) and that the IJRP would play a key role in increased communication and cooperation between the various involved governmental agencies, and reduced conflict, delay, and costs in the permitting process. Midas Gold’s objective is to make the Project a fully integrated, sustainable, and socially and environmentally responsible operation through open communications and accessibility.

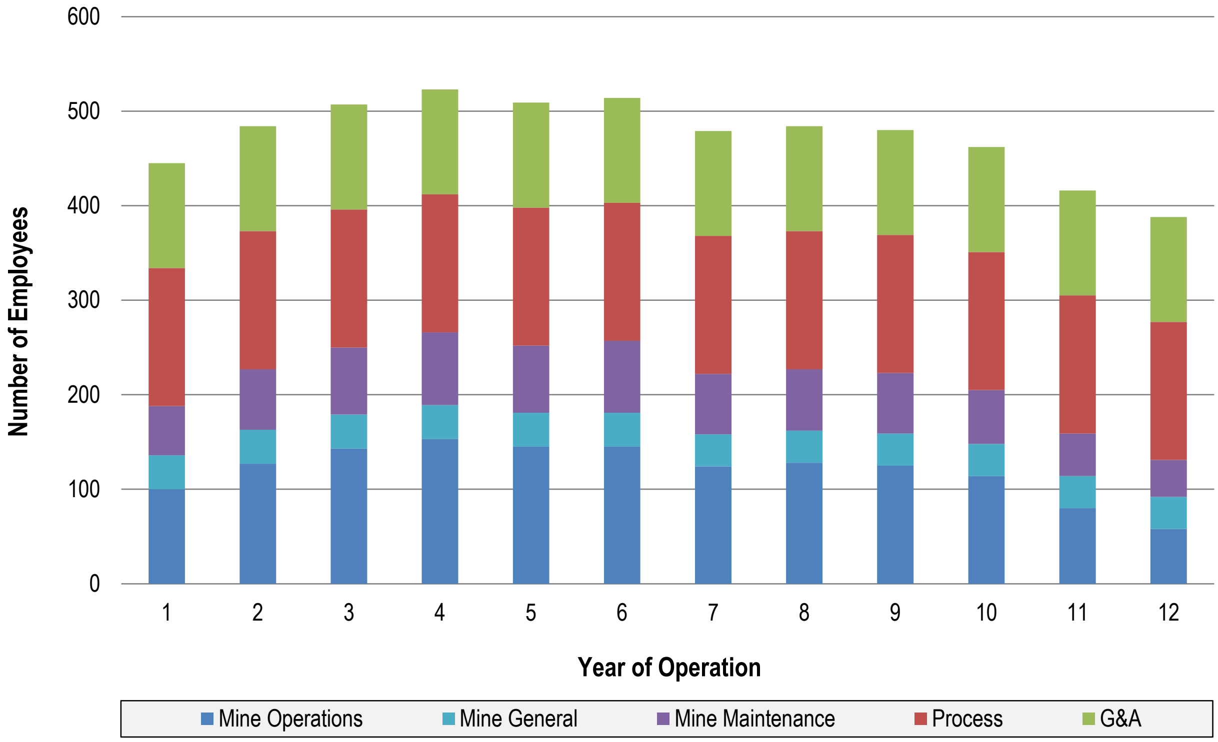
1.22 SOCIAL IMPACTS

Employment

Populations continue to grow in Valley and Adams Counties, but jobs are not keeping pace; unemployment rates in these counties are some of the highest in Idaho, while wages average only $27,433/year. The Project could do much to improve this situation, with current mining jobs in Idaho averaging $72,500/year and the Project offering an approximate average of some 400 direct and 321 indirect and induced jobs in Idaho generating aggregate annual payrolls of $48 million/year during the 3-year construction period (plus additional out-of-state contractors for specialized construction functions) and an approximate average of some 500 direct and 439 indirect and induced jobs generating aggregate annual payrolls of $56 million/year during the 12-year operating period.

Operations are scheduled for 365 days/year; a breakdown of the annual staffing requirements to operate and maintain the mine, processing plant, and appurtenant facilities and functions for the five functional work areas is provided on Figure 1.7. Whenever possible, the work force was segregated between the mine site and the Cascade Complex to limit the number of personnel at the mine site that require residential support and transportation to and from site.

Figure 1.7: Annual Direct Employment by Department



Taxes

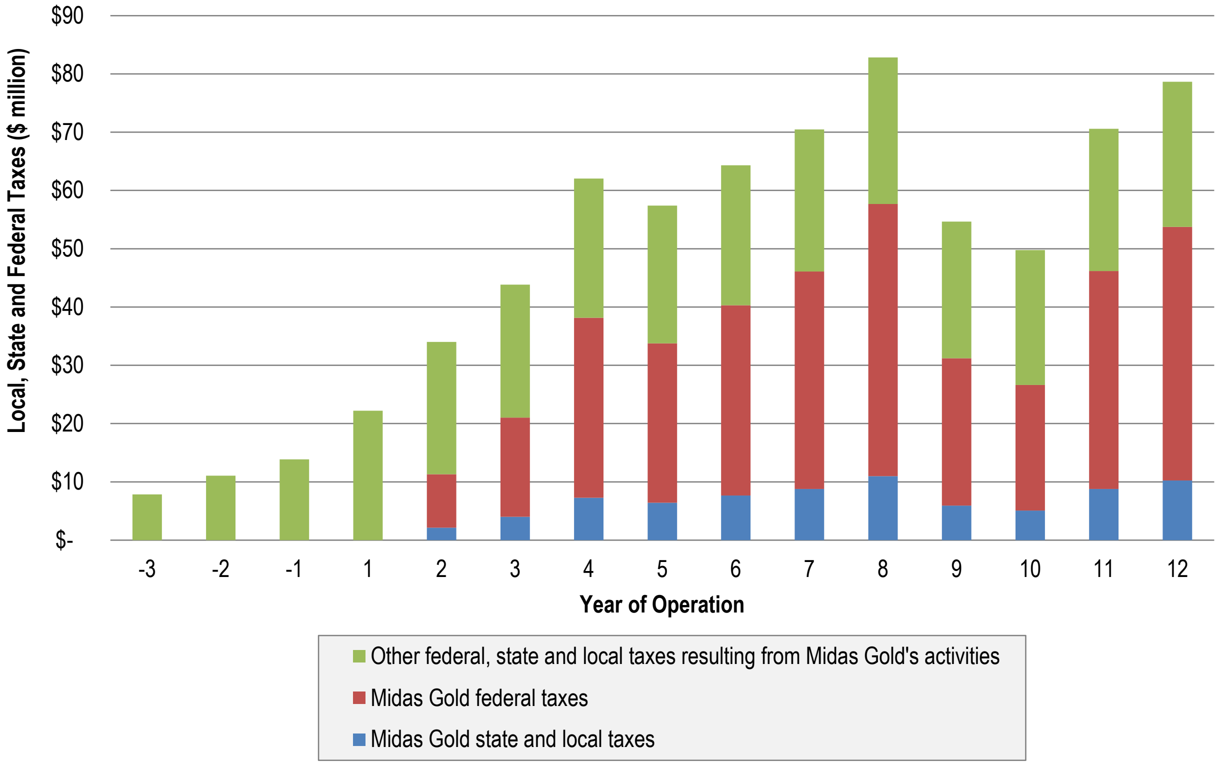
To estimate the potential economic impacts from the Project, an economic impact model known as IMpact analysis for PLANning (IMPLAN) was constructed (Peterson, 2014). The IMPLAN model was used to estimate direct, indirect and induced taxes, that would be paid by other taxpayers (other than Midas Gold), and the tax estimates were combined with the direct federal, state and local taxes that would be paid by Midas Gold (see Section 22 for details on the PFS financial model and tax calculations) to develop an estimate for the overall taxes generated by the Project. Figure 1.8 presents a plot of estimated annual direct, indirect and induced taxes associated with the Project paid by both Midas Gold and other taxpayers to federal, state and local governments.

Taxes that would be paid directly by Midas Gold over the life of the Project, based on the assumptions in the PFS, are estimated at approximately $329 million in federal corporate income taxes, and $86 million in state corporate income and mine license taxes.

Additional indirect and induced taxes that result from Midas Gold's activities that would be paid by other taxpayers, based on the assumptions in the PFS, are estimated at approximately $177 million in federal taxes (including payroll, excise, income and corporate), and $131 million in state and local taxes (including property, sales, excise, personal, corporate, and other).

Total direct, indirect and induced taxes are therefore estimated at $506 million in federal taxes and $218 million in state and local taxes, representing a significant contribution to the economy during the 15 year construction and operating life of the Project.

Figure 1.8: Chart of Estimated State and Federal Taxes



Environmental Mitigation and Remediation

Midas Gold has made considerable effort to design the Project restoration of the site through the incorporation of specific mitigation and remediation components, including re-establishing fish passage, removal and reprocessing of unconstrained Historic Tailings, removal of unconstrained historical waste rock, reuse of historical spent ore piles for construction, stream channel restoration projects, and sediment control. The mitigation and remediation activities and costs are summarized in Section 20 and Section 21, respectively. Additionally, the Project design team has optimized siting of facilities wherever possible to avoid riparian areas, limit stream crossings, position facilities on previously disturbed ground, move major access routes away from large waterways, minimize the number of people on site to limit traffic, and re-establish historic line power to the site to minimize fuel haulage and reduce greenhouse gas emissions. In some cases, disturbance of albeit already impacted wetlands and streams would be unavoidable, which disturbance Midas Gold intends to address through a mitigation bank or similar entity as well as through onsite replacement and restoration of existing wetlands. Midas Gold would continue to build on its strong record by continuing to proactively evaluate Best Management Practices (**BMPs**) and Standard Operating Procedures (**SOPs**) effectiveness, including a post-closure component.

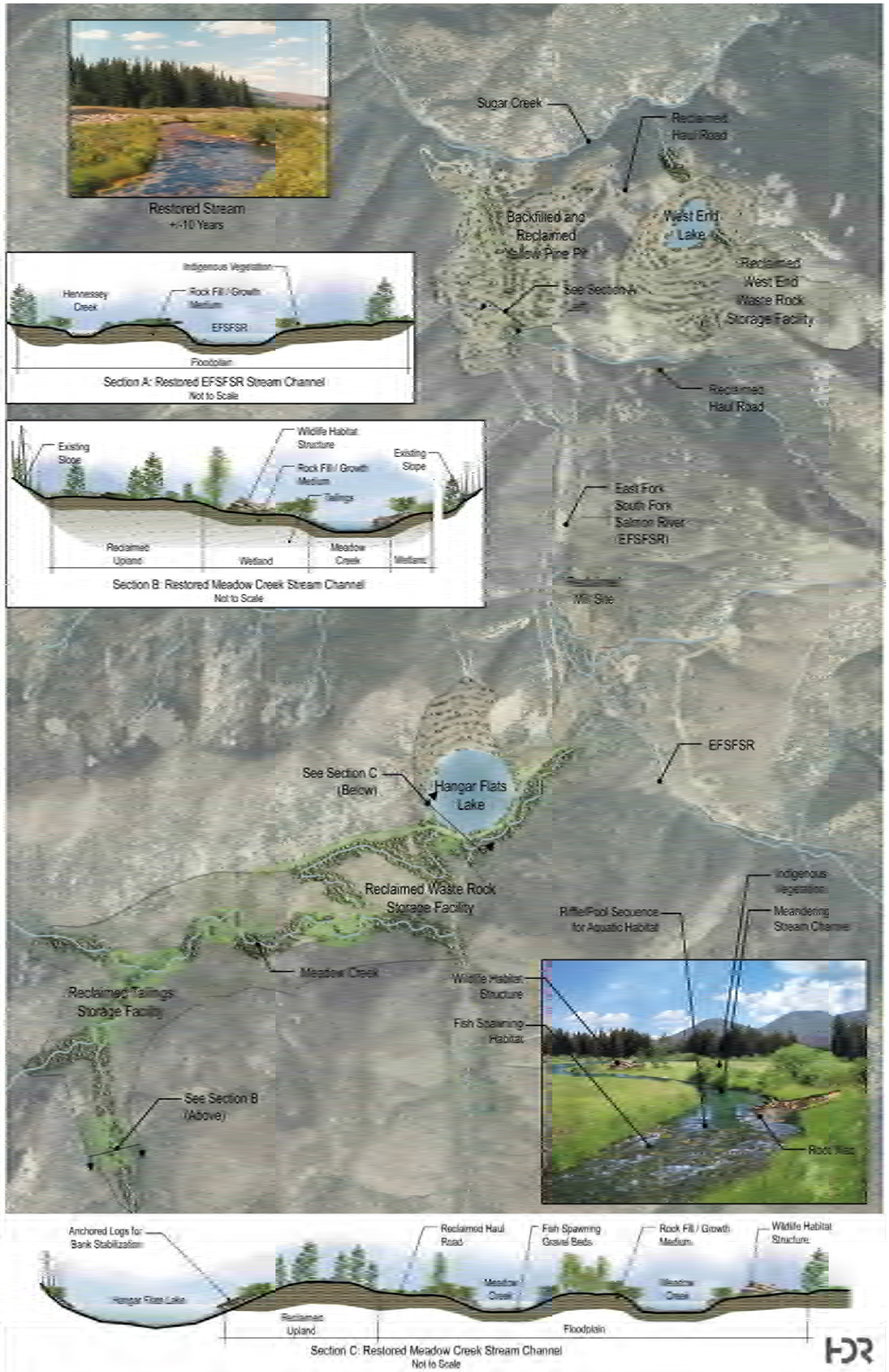
A critical goal for Midas Gold has been the incorporation of fisheries protection and habitat restoration components aimed at achieving a sustainable anadromous fishery, including passage of migrating salmon, steelhead, and trout to the headwaters of the EFSFSR both during and after operations for the first time since 1938. Upon closure, new enhanced wetlands and spawning grounds would be established to assist in the return of fish migration and reestablishment of a health riparian zone along the rebuilt stream channel. Midas Gold has also incorporated efforts to improve water quality by removing historical tailings, spent ore and waste rock and respectively reprocessing, reusing and relocating these materials, as well as developing sediment control features for Blowout Creek, currently a major contributor of sediment, and replanting historically disturbed and forest fire affected areas to reduce sedimentation.

Closure

During construction, operations and once operations cease, extensive reclamation would be completed, creating enhanced surface water systems and suitable fisheries habitat. Midas Gold has identified 17 priority Project conservation components that form the basis of the overall conservation strategy that are summarized in Section 20; Figure 1.9 presents a site-wide illustration of the overall closure strategy. These components include: construction of the new Burntlog Road (which effectively moves the primary transportation route away from the Johnson Creek fishery), backfilling the Yellow Pine pit with environmentally appropriate material to create a stable hydrogeologic gradient suitable to the current conditions, closure of historic mine workings on USFS lands, ongoing wetlands and stream habitat enhancement, permanent restoration of fish passage up the EFSFSR, post-closure wetlands and stream habitat enhancement on top of the Meadow Creek TSF surface and reforestation of the Project area. The conservation commitment to restore the site through implementation of these measures is discussed in greater detail in Section 20, while closure costs are detailed in Section 21.

When operations cease, mobile and salvageable equipment would be removed, and foundations broken up, covered and re-vegetated (Figure 1.9). The objective is for the development of a self-sustaining natural environment that has addressed many of the historical impacts and supports a healthy fish and wildlife population. Post-closure monitoring is planned for an extended period to ensure that these objectives have been met.

Figure 1.9: Conceptual Post Closure Reclamation



1.23 ECONOMIC ANALYSIS

Capital and operating cost estimates were developed based on Q3 2014, un-escalated U.S. dollars. Vendor quotes were obtained for all major equipment. Most costs were developed by first principles, although some were estimated based on factored references and experience with similar projects.

Capital Costs

The estimated capital expenditure or capital costs (**CAPEX**) for the Project consists of four components: (1) the initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, operations camp, and on and off site environmental mitigation; (2) the sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing 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 costs. Initial and working CAPEX are the two main categories that need to be available to construct the Project. Table 1.9 summarizes the initial, sustaining and closure CAPEX for the Project.

Table 1.9: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)(2)	Closure CAPEX ($000s)(2)	Total CAPEX ($000s)
Direct Costs	Mine Costs	47,552(1)	35,346	-	82,898
	Processing Plant	336,219	1,579	-	337,798
	On-Site Infrastructure	149,245	39,937	-	189,182
	Off-Site Infrastructure	80,327	-	-	80,327
Indirect Costs		176,687	4,275	-	180,962
Owner's Costs		26,806	-	-	26,806
Environmental Mitigation Costs		10,606	8,165	-	18,771
Closure Bonding, Closure and Reclamation Costs		762	9,185	56,542	66,489
Total CAPEX without Contingency		**828,204**	**98,488**	**56,542**	**983,233**
Contingency		142,050	-	-	142,050
Total CAPEX with Contingency		**970,254**	**98,488**	**56,542**	**1,125,283**

Note:
(1) Initial mining CAPEX includes environmental remediation costs as discussed in Section 21.
(2) Contingency included in line items.

Mitigation costs only refer to relocation of a certain portion of the readily identifiable and quantified waste from historical mining activities; other costs related to recovery and reprocessing of Historic Tailings and relocation of unquantified waste rock at West End and Yellow Pine are included in operating costs and are partially offset by recovery of gold and antimony from the Historical Tailings.

Operating and All-In Costs

The cash operating costs include mine operating costs, process plant operating costs, general and administrative (**G&A**) costs, while total cash costs include smelting and refining charges, transportation charges, and royalties. A detailed breakdown of the summary of the operating costs (**OPEX**) costs is presented in Table 1.10. The details that comprise the OPEX are provided Section 21. The All-In Sustaining Costs (**AISC**) are also provided in the table, as well as the All-In Costs (**AIC**), which include non-sustaining capital and closure and reclamation costs.

Table 1.10: Operating Cost, AISC and AIC Summary

Total Production Cost Item	LOM			Years 1-4	
	($/st mined)	($/st milled)	($/oz Au)	($/st milled)	($/oz Au)
Mining	2.00	9.08	222	10.04	222
Processing	-	14.45	354	14.10	312
G&A	-	3.13	77	3.01	67
Cash Costs Before By-Product Credits	**-**	**26.65**	**653**	**27.15**	**601**
By-Product Credits	-	-3.45	-85	-5.32	-118
Cash Costs After of By-Product Credits	**-**	**23.20**	**568**	**21.83**	**483**
Royalties	-	0.94	23	0.34	23
Refining and Transportation	-	0.25	6	1.04	8
Total Cash Costs	**-**	**24.38**	**597**	**23.20**	**513**
Sustaining CAPEX	-	1.00	24	0.52	11
Salvage	-	-0.27	-7	0.00	0
Property Taxes	-	0.04	1	0.04	1
All-In Sustaining Costs	**-**	**25.15**	**616**	**23.76**	**526**
Reclamation and Closure(1)	-	0.58	14	-	-
Initial (non-sustaining) CAPEX(2)	-	9.89	242	-	-
All-In Costs	**-**	**35.62**	**872**	**-**	**-**

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

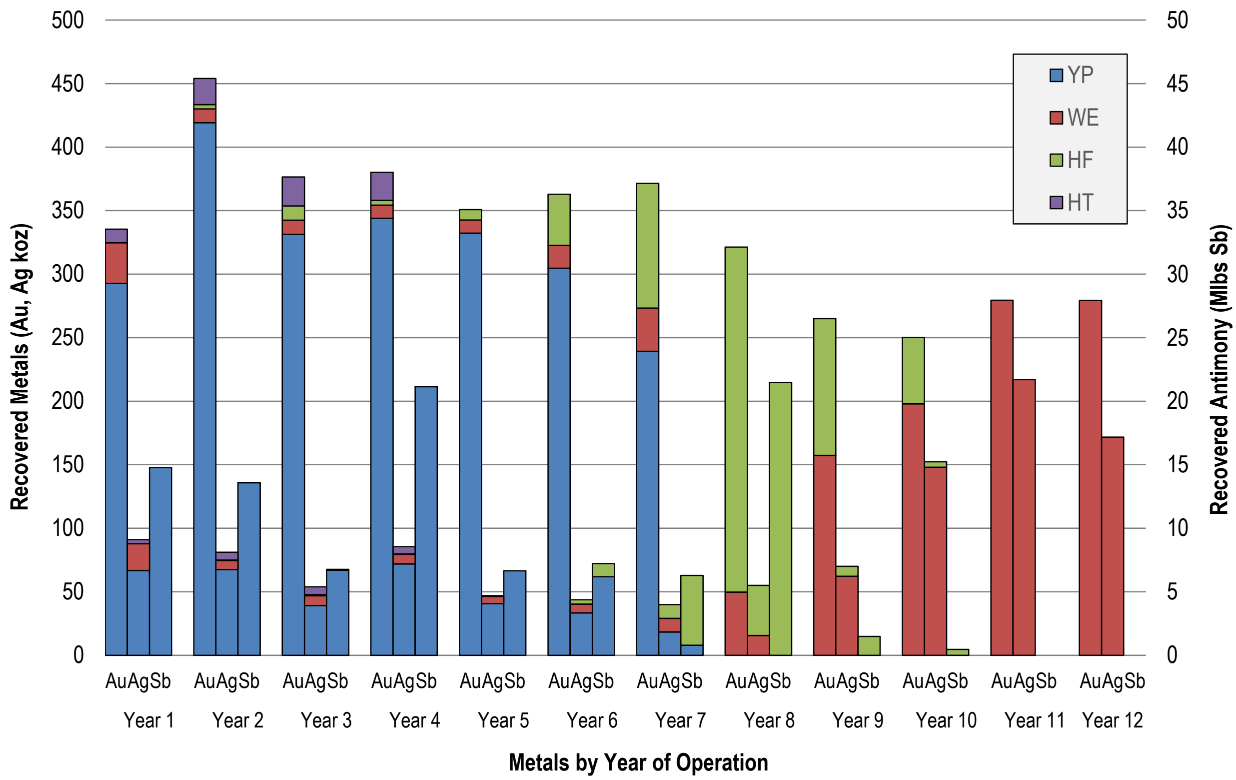
Metal Production

Recovered metal production by deposit is summarized in Table 1.11 and illustrated on an annual basis on Figure 1.10.

Table 1.11: Recovered Metal Production

Product by Deposit	Gold (koz)	Silver (koz)	Antimony (klbs)
Doré Bullion			
Yellow Pine	2,263	338	-
Hangar Flats	597	68	-
West End	1,090	681	-
Historic Tailings	72	20	-
Doré Bullion Recovered Metal Totals	**4,023**	**1,107**	**-**
Antimony Concentrate			
Yellow Pine	12	611	69,822
Hangar Flats	5	349	30,030
Antimony Concentrate Recovered Metal Totals	**17**	**960**	**99,852**
Total Recovered Metals	**4,040**	**2,067**	**99,852**

Figure 1.10: Annual Recovered Metals by Deposit



Economic Analysis

The economic model described herein 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 PFS 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 1.8 and off-site costs and payables used are in Table 1.6 and Table 1.7. There is no guarantee that any of the metal prices used in the four cases are representative of future metals prices. The constant parameters for all cases are shown in Table 1.12.

Table 1.12: Economic Assumptions used in the Economic Analyses (all Cases)

Item	Unit	Value
Net Present Value Discount Rate	%	5
Federal Income Tax Rate	%	35
Idaho Income Tax Rate	%	7.4
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

Item	Unit	Value
Depreciation Term	Years	7
Equity Finance	%	100
Capital Contingency (Overall)	%	17.2

The results of the economic analyses are shown in Table 1.13. Based on the assumptions made in this PFS, the $ATNPV_{5\%}$ is estimated to be \$832 million yielding an after-tax IRR of 19.3%. The $ATNPV_{5\%}$ and IRR increases considerably with the Case C metal prices and decreases with the Case A metal prices. The $PTNPV_{5\%}$ for Case B was estimated to be \$1,093 million with an IRR of 22.0%.

Table 1.13: Economic Results by Case

Parameter	Unit	Pre-tax Results	After-tax Results
Case A (\$1,200/oz Au, \$20.00/oz Ag, \$4.00/lb Sb)			
$NPV_{0\%}$	M\$	1,286	1,041
$NPV_{5\%}$	M\$	662	513
IRR	%	16.2	14.4
Payback Period	Production Years	4.0	4.1
Case B (\$1,350/oz Au, \$22.50/oz Ag, \$4.50/lb Sb)			
$NPV_{0\%}$	**M\$**	**1,915**	**1,499**
$NPV_{5\%}$	**M\$**	**1,093**	**832**
IRR	**%**	**22.0**	**19.3**
Payback Period	**Production Years**	**3.2**	**3.4**
Case C (\$1,500/oz Au, \$25.00/oz Ag, \$5.00/lb Sb)			
$NPV_{0\%}$	M\$	2,543	1,929
$NPV_{5\%}$	M\$	1,524	1,129
IRR	%	27.2	23.4
Payback Period	Production Years	2.6	2.9
Case D (\$1,650/oz Au, \$27.50/oz Ag, \$5.50/lb Sb)			
$NPV_{0\%}$	M\$	3,171	2,344
$NPV_{5\%}$	M\$	1,955	1,414
IRR	%	31.9	27.0
Payback Period	Production Years	2.2	2.5

The contribution to the Project economics, by metal, is about 94% from gold, 5% from antimony, and less than 1% from silver. The undiscounted after-tax cash flow for Case B is presented in 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

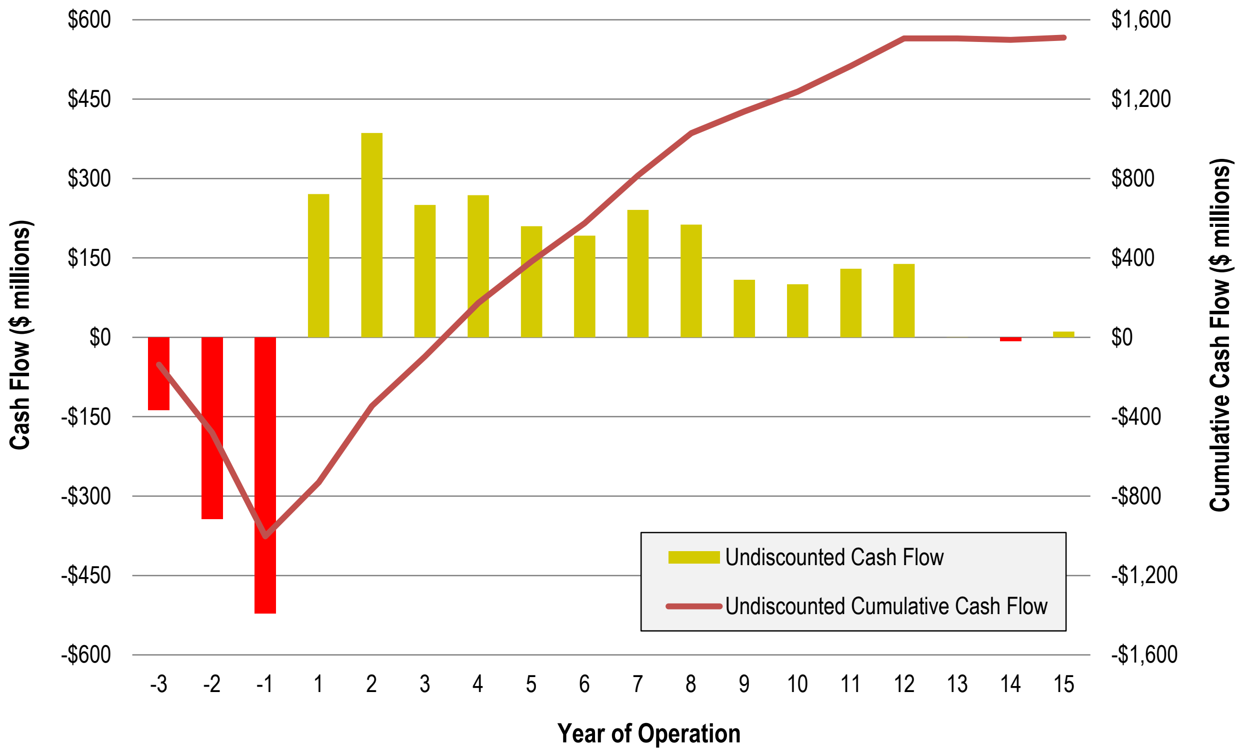
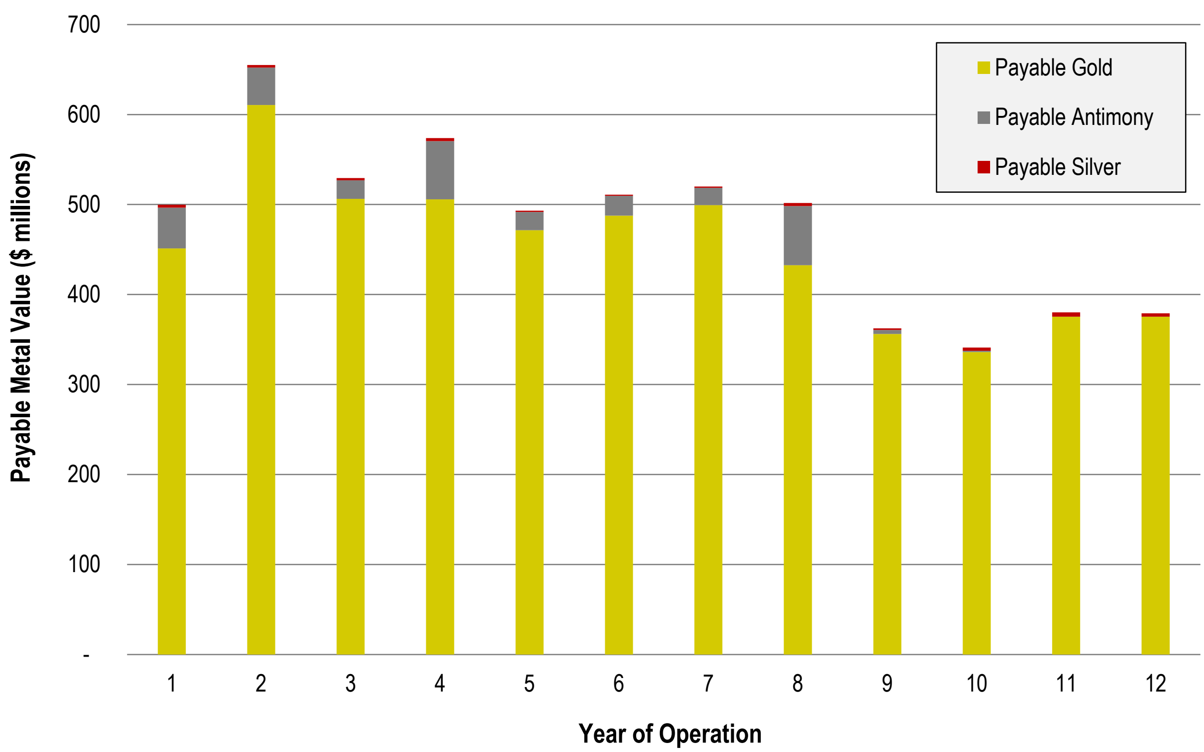


Figure 1.12: Payable Metal Value by Year for Case B



Mine Life

Using the current Mineral Reserve and the nominal design throughput of 22,050 st/d, the mine plan projects a 12 year production life. Construction is projected to require a three-year period after the permits are obtained and prior to the start of operations. Closure is projected to take at least 10 years post-production, with some reclamation work occurring concurrently with operations, and the bulk of the closure activities and costs incurred in the first 3 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.

Sensitivity Analysis

Sensitivity analyses were performed using metal prices, mill head grade, CAPEX, and OPEX as variables. The value of each variable was changed plus and minus 20% independently while all other variables were held constant. The results of the sensitivity analyses are shown in Table 1.14 and Table 1.15.

Table 1.14: Pre-tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	$PTNPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	862	662	463
	OPEX	1,017	662	308
	Metal Price or Grade	-27	662	1,352
Case B (Base Case)	**CAPEX**	**1,292**	**1,093**	**894**
	OPEX	**1,447**	**1,093**	**739**
	Metal Price or Grade	**318**	**1,093**	**1,869**
Case C	CAPEX	1,723	1,524	1,325
	OPEX	1,878	1,524	1,170
	Metal Price or Grade	662	1,524	2,386
Case D	CAPEX	2,154	1,955	1,755
	OPEX	2,309	1,955	1,600
	Metal Price or Grade	1,007	1,955	2,902

Table 1.15: After-tax $NPV_{5\%}$ Sensitivities by Case

Case	Variable	$ATNPV_{5\%}$ (M$)		
		-20% Variance	0% Variance	20% Variance
Case A	CAPEX	676	513	346
	OPEX	760	513	239
	Metal Price or Grade	-30	513	1,012
Case B (Base Case)	**CAPEX**	**980**	**832**	**674**
	OPEX	**1,057**	**832**	**577**
	Metal Price or Grade	**244**	**832**	**1,357**
Case C	CAPEX	1,266	1,129	982
	OPEX	1,341	1,129	903
	Metal Price or Grade	513	1,129	1,696
Case D	CAPEX	1,548	1,414	1,277
	OPEX	1,623	1,414	1,200
	Metal Price or Grade	770	1,414	2,035

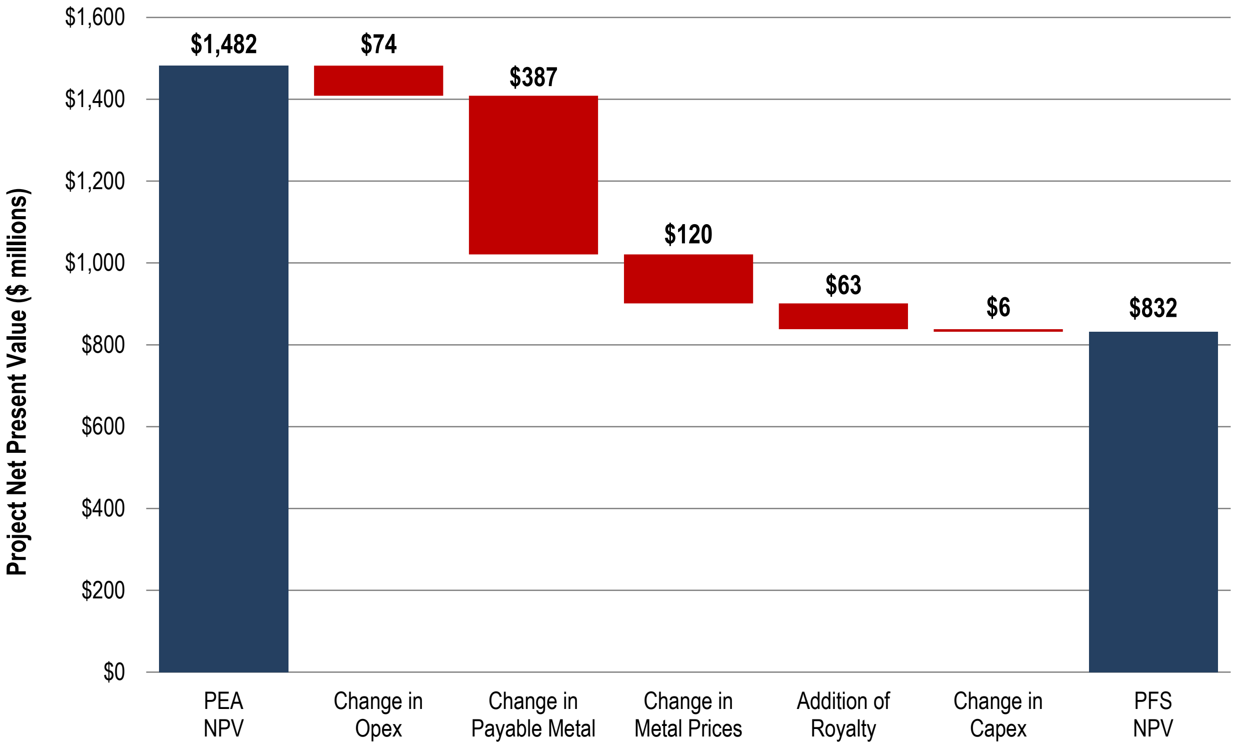
1.24 COMPARISON OF 2012 PEA TO PFS

The estimated PFS LOM CAPEX is $57 million less than the estimated PEA (SRK, 2012) LOM CAPEX. Principle causes of the reduction can be attributed to decreases in mining costs related leasing the mining fleet, and the decision to eliminate a portion of the Hangar Flats deposit from the LOM plan thereby reducing the mine life and total tons moved. Additional reductions include: a lower Project contingency resulting from more detailed engineering and designs; a reduction of owner's costs; the elimination of an acidulation circuit; and a slightly smaller tailings storage facility due to less material being processed.

Compared to the PEA, the PFS LOM unit operating costs have increased. The principle changes include: a reduction in by-products credits; an increase in cash costs in mining resulting from leasing major pieces of equipment; an increase in processing costs resulting from higher grinding media consumption and higher power costs; and the addition of a 1.7% royalty that applies to gold revenue.

Many factors have influenced the $ATNPV_{5\%}$ from the $1,482 million reported in the PEA to the $832 million reported in this PFS. Significant changes include a decrease in payable metal, decrease in metal prices, increases to OPEX and the addition of a royalty. The decrease in payable metal is partially a result of changing from using Mineral Resources in the PEA to Mineral Reserves (i.e. Inferred Mineral Resources are excluded, as required for a PFS under NI 43-101) in addition to other changes in the Mineral Resource estimates for at each of the deposits, as discussed in Section 14. The decrease in metal prices during the intervening time further exacerbated the reduction in LOM revenue. Changes in $ATNPV_{5\%}$ relative to the 2012 PEA are summarized on Figure 1.13.

Figure 1.13: Changes in LOM After-Tax $NPV_{5\%}$ from PEA to PFS



1.25 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, for which additional information is required in order to mitigate:

- Use of historical data in Mineral Resource estimates, which could affect these estimates;
- Limited geotechnical data which could change pit slopes or foundation conditions in infrastructure areas;
- Loss of gold into antimony concentrates;
- Water management and chemistry, which could affect diversion and closure designs and/or the need for long term water treatment; 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 follow:

- In pit conversion of Inferred Mineral Resources to Mineral Reserves, increasing Mineral Reserves and reducing strip ratio;
- Out of pit conversion of Inferred Mineral Resources to Mineral Reserves adjacent to the current Mineral Reserves, resulting in increased Mineral Reserves in close proximity to planned pits;

- In pit conversion of unclassified material currently treated as waste rock to Mineral Reserves, increasing Mineral Reserves and reducing strip ratios;
- Improved continuity of higher grade gold mineralization in the Yellow Pine pit, particularly around the area with excluded or limited Bradley drilling, increasing grade of the Mineral Reserves;
- Additional fire assay information at West End in areas where only cyanide assays were available, potentially increasing grade and Mineral Reserves;
- Potential additional antimony mineralization and/or grade in areas where Bradley data was eliminated and/or areas where antimony was not assayed, increasing by-product credits;
- Potential for the definition of a higher grade, higher margin underground Mineral Reserve at Scout and Garnet; and
- Discovery of other new deposits with attractive operating margins.

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

1.26 CONCLUSIONS AND RECOMMENDATIONS

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 PFS.

The financial analysis presented in Section 22 of the PFS 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 PFS demonstrate that the Project is technically and environmentally viable. These conclusions warrants continued work to advance the Project to the next level of study, which is a Feasibility Study (**FS**), by conducting the work indicated in the recommendations section of this Report. These recommendations form a single phase that will move the Project through to completion of a FS and, if so desired, through the regulatory process for mine development. Total estimated costs for completion of this single phase are $22.3 million. While additional information is required for a complete assessment of the Project, at this point there do not appear to be any fatal flaws. The PFS has achieved its original objective of providing a review of the potential economic viability of the Project to standards appropriate for a PFS.

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.

An additional $22.5 million is identified as discretionary expenditures that would target a number of the opportunities identified in Section 25 of this PFS Report that could enhance the PFS case but that are not required in order to complete a FS or permitting.

Table 1.16: Project Development Work Program Budget

Recommendations and Work Program	Estimated Costs ($000s)	
	Core	Discretionary
Mineral Resource Evaluation and Exploration	3,700	21,200
Field Programs Required for FS	1,900	-
Metallurgical Testing Required for FS	2,400	1,300
FS-Level Engineering	3,500	-
Environmental, Regulatory Affairs and Compliance	10,800	-
Totals	**22,300**	**22,500**

SECTION 2 TABLE OF CONTENTS

SECTION 2 LIST OF TABLES

2 INTRODUCTION

This prefeasibility study technical report (PFS or Report) was commissioned by Midas Gold, for its Stibnite Gold 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 that owns and operates the Project through its wholly-owned subsidiaries, Midas Gold, 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.

2.1 PURPOSE OF REPORT

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 Stibnite Gold Project key considerations 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 taxation, employment, and business opportunities in a region where the economy has suffered for more than a decade, resulting in some of the highest unemployment and lowest annual wages in Idaho.
- From the beginning, the Project has been designed for what will remain after closure. The plan for closure is protective of the environment and incorporates inherently stable, secure features that will provide the foundation for an evolution through time 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.
- The new facilities contemplated for the Project are tightly constrained and, to a large extent, placed in historically impacted areas in order to minimize the incremental Project footprint.
- Salmon and other fishery enhancement is integral to the Project design. Removal of man-made barriers and reconstruction of natural habitat would allow salmon and other 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 method, process method, infrastructure, social and economic benefits, environmental protection, repair of historical impacts, reclamation and closure concepts, capital and operating costs and an 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 Preliminary Economic Assessment (PEA) Technical Report for the Golden Meadows Project Idaho, dated September 21, 2012 (SRK, 2012); the information in the PEA should no longer be relied upon.

2.2 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 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 PFS is based on information collected by the Qualified Persons (each a QP) during their site visits. In addition, a number of meetings were conducted between M3 and Midas Gold. This Prefeasibility 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. to Power Engineers concerning power supply for the Project.
- Technical and economic information subsequently 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 PFS 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.

Table 2.1: List of Qualified Persons

Qualified Person	Company	Section Responsibility	Site Visit Date	Site Visit Review
Conrad E. Huss, P.E.	M3 Engineering & Technology Corp.	1, 2, 3, 4, 5, 6, 18.3, 18.4.2, 18.6, 18.7, 18.8, 19, 21.1, 21.2.2, 22, 23, 24, 25, 26, 27	1	
Garth D. Kirkham, P. Geo	Kirkham Geosystems Ltd.	7, 8, 9, 10, 11, 12, 14	April 23-25, 2014, and July 14-15, 2014	On both visits toured core logging and storage facilities. Site visit entailed inspection of the shops, offices, reclaimed drill sites, the Yellow Pine, Hanger Flats and West End mineral resource areas along with the outcrops, historic drill collars and areas of potential disturbance for potential future mining operations
Christopher J. Martin, C.Eng.	Blue Coast Metallurgy Ltd.	13	August 25, 2011	General site visit
John M. Marek, P.E.	Independent Mining Consultants Inc.	15, 16, 21.1.2, 21.2.3	September 16-17, 2013	Reviewed Project geology, terrain, and operational constraints at site. Visited drill core handling facility to review logging, sampling, and handling procedures.
Allen R. Anderson, P.E.	Allen R. Anderson Metallurgical Engineer Inc.	17	2	
Richard C. Kinder, P.E.	HDR Engineering Inc.	18.2	October 12, 2012	Route survey, where practicable, of access road options evaluated.
Peter E. Kowalewski, P.E.	Tierra Group International Ltd.	18.9, 18.10, 18.11, 18.12, 20	March 7, 2013	General site visit.

Notes:
1) Conrad Huss has not visited the site but has relied on observations of Bill Oppenheimer, Tim Burns, Justine Crabtree, and Justin Nail, all from M3, who visited the site on June 17, 2013 to inspect the proposed location of plant facilities and infrastructure in relation to the existing topography and improvements on the site.
2) Allen Anderson has not visited the site.

2.3 ABBREVIATIONS, UNITS, AND TERMS OF REFERENCE

This PFS is intended for the use of Midas Gold for the further advancement of the Stibnite Gold Project toward the feasibility study phase. 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 2.3.4. All monetary values are in U.S. dollars ($) unless otherwise noted.

2.3.1 Mineral Resources

As required by NI43-101, the Mineral Resources and Mineral Reserves in this Report have been classified according to the "*CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines*" (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.

> *"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."*

> *"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."*

2.3.2 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."*

2.3.3 Glossary

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

Table 2.2: Glossary

Term	Definition
Albion	A patented metallurgical process developed by Glencore that involves recovering metals using a combination of ultrafine grinding and oxidative leaching at atmospheric pressure.
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.
Cascade Complex	administrative offices for Project located in or near the town of Cascade offices for managers, safety and environmental services, human resources, purchasing, and accounting personnel
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 (CoG)	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.
Historic 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 combine with oxygen.
Project	A collaborative enterprise, involving research or design, that is carefully planned to achieve a particular aim i.e.

Term	Definition
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 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 characteristics (usually grade).

2.3.4 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
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
~	approximately
ARD	acid rock drainage
As	arsenic
AT	after tax
$ATNPV_{5\%}$	after-tax net present value at a 5% discount rate
Au	gold
AuCN	assays that determine the cyanide soluble gold content
AuFA	assays that determine the total gold content using the fire assay technique
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

Abbreviation	Unit or Term
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
COC	chain of custody
CoG	cut-off grade
CSAMT	controlled source audio magneto-tellurics geophysical survey method
°	degree (degrees)
dia.	diameter
EFMC	East Fork of Meadow Creek, commonly known as “Blowout Creek”
EFSFSR	East Fork of the South Fork of the Salmon River
EGL	effective grinding length
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
°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	grams
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

Abbreviation	Unit or Term
g/st	grams per short ton
g/t, gpt	grams per metric tonne
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
ICP MS	inductively coupled plasma mass spectrometry, an analytical method for assaying
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
kg	kilograms
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	liters
lb	pounds
LiDAR	Light Detection And Ranging distance measuring technology
LLDPE	linear low density polyethylene plastic

Abbreviation	Unit or Term
LOM	life-of-mine
m	meters
Ma	million years
MACRS	Modified accelerated cost recovery system
MBR	membrane bioreactor
MCFZ	Meadow Creek fault zone
mg/L	milligrams/liter
mi	miles
mi^2	square miles
MIBC	Methyl isobutyl carbinol
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
MFZ	Mule fault zone
MW	Megawatts or million watt (where context indicates)
MWMP	Meteoric Water Mobility Procedure (Nevada)
mV	Millivolt or 10^{-3} volts
MVA	megavolt amperes
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
NSR	net smelter return
OHWM	ordinary high water mark
OPEX	operating expenditures
oz	troy ounces
oz/st	troy ounces per short ton
%	percent
P_{80}	80% passing a certain size
PAX	Potassium amyl xanthate
PEA	Preliminary Economic Assessment as defined in NI43-101
PFS	Preliminary Feasibility Study as defined in NI43-101
PLC	programmable logic controller
PMF	probable maximum flood
PoO	Plan of Operations

Abbreviation	Unit or Term
POX	pressure oxidation
ppb	parts per billion
ppm	parts per million
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
SMC	Sag Mill Comminution
SVFZ	Scout Valley fault zone
SG	specific gravity
SIMS	secondary ion mass spectrometry
SODA	spent ore disposal area
SOG	sale-of-gas
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
SPLP	synthetic precipitation leachate procedure
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
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

Abbreviation	Unit or Term
VHF	very high frequency
VLF-EM	very low frequency electro-magnetic geophysical survey
W	watts, where context indicates
W	tungsten, where context indicates
WAD cyanide	weak acid dissociable cyanide
WE	West End
WEFZ	West End fault zone
WEO	Est End oxide mineralization
WES	West End sulfide mineralization
WRSF	Waste rock storage facility
XRD	x-ray diffraction
XRF	x-ray fluorescence
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 & Regulatory Abbreviations

Abbreviation	Agency Name & 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
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

Abbreviation	Agency Name & Related Act or Regulation or Term
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
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)
NMFS	National Marine Fisheries Service, a division of the National Oceanic and Atmospheric Administration, U.S. Dept. of Commerce
NOAA	US National Oceanic and Atmospheric Administration, U.S. Dept. of Commerce
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	storm water 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
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.
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
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.
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
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
Pegasus	Pegasus Gold Corporation
Pioneer	Pioneer Metals Corporation
Ranchers	Rancher's Exploration Company
SGS	SGS Minerals Inc.
SMI	Stibnite Mines Inc., a subsidiary of MinVen and later Dakota
SRK	SRK Consulting (Canada), Inc.
Strata	Strata, a professional services corporation
Superior	Canadian Superior Mining (U.S.) Ltd.

Abbreviation	Company Name
URS	URS Corporation
Vista	Vista Gold Corp.
Vista US	Vista Gold US Inc., a subsidiary of Vista

Table 2.6: 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.

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

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3 RELIANCE ON OTHER EXPERTS

The Stibnite Gold Project 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 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.

3.1 PROPERTY OWNERSHIP AND TITLE

Legal review of the Stibnite Gold Project property ownership and title was completed by the Idaho law firm Givens Pursley LLP (Givens Pursley). Givens Pursley commissioned multiple Landman Reports for the Project that cover various patented and unpatented mining claims. The Landman Reports were completed in accordance with reasonable industry standards by person(s) having appropriate training, experience and expertise. Givens Pursley's review of these Landman Reports and related examinations are summarized in various title opinion documents, the most recent of which were completed in 2013. Givens Pursley concluded that 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 of this Report, subject to the royalties, agreements, limitations and encumbrances described in Section 4.4.

3.2 AUTOCLAVE DESIGN AND SIZING

Technical assistance for the design and sizing of the autoclave was provided by Mr. Herman Pieterse, engineering consultant, who has 25 years of experience in autoclave operation and design. Mr. Pieterse provided significant input concerning the design, operation, and sizing of the autoclave and appurtenant equipment for pressure oxidation of sulfide concentrates for the liberation of gold. The pressure oxidation process is critical to the recovery of gold from the Stibnite Gold mineralization.

3.3 WATER RIGHTS

Mr. Terry Scanlan, P.E., P.G. of SPF Water Engineering, LLC performed a comprehensive review of Midas Gold's water rights portfolio. The water rights held by Midas Gold are summarized in Section 5.5.2 of this Report.

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4 PROPERTY DESCRIPTION AND LOCATION

4.1 MINERAL TITLE

Midas Gold's property holdings consist of patented lode claims, patented mill site claims, unpatented federal lode claims and unpatented federal mill site claims (collectively, "Claims") which cover approximately 27,104 acres (approximately 42 mi^2) as shown on Figure 4.1. Appendix II presents a detailed mineral concession summary, a land status map, and complete tables listing the Claims. No significant flaws or title issues have been identified in multiple formal title reviews of the Claims performed by a qualified, independent, title examiner. 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.

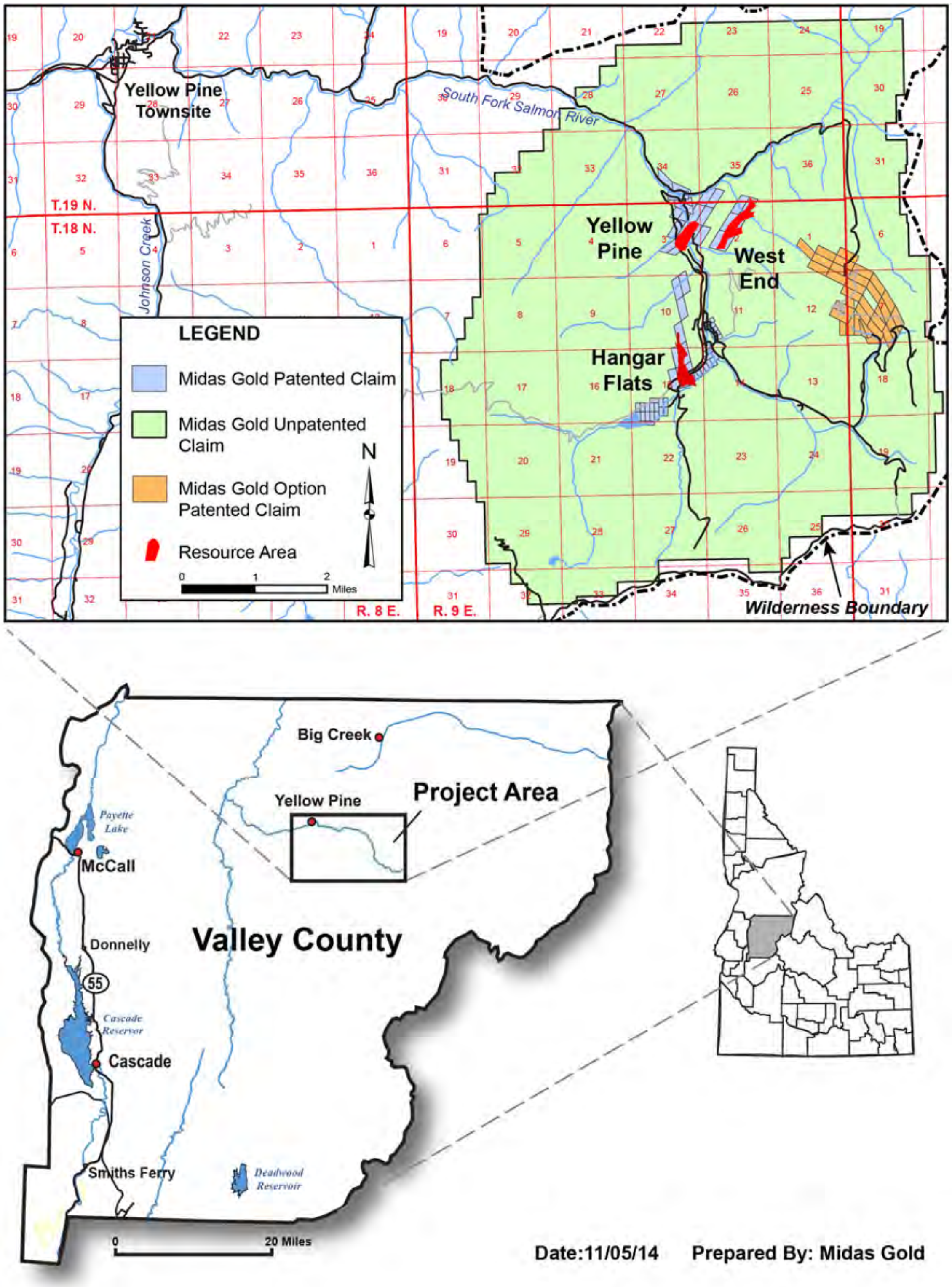
4.2 LOCATION

The Project is located in central Idaho approximately 98 mi northeast of Boise, Idaho, 40 mi east of McCall, Idaho, and approximately 10 mi east of Yellow Pine, Idaho (Figure 4.1) in all or part of the following sections (Boise Meridian):

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

The Project area elevations range from approximately 6,500 ft to over 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 1181270 ft US N and 1103 2734259 ft US W.

Figure 4.1: Project Location Map



4.3 NATURE AND EXTENT OF TENURE

The following description was updated to July 30, 2014. Claim groups under full Midas Gold ownership are discussed in this section while those with encumbrances are detailed in section 4.4.

Midas Gold acquired 229 federal unpatented claims by purchase from previous owners in 2009 and 2011. These include 46 federal mill site claims which carry surface use rights but no mineral rights, and 183 federal unpatented lode-mining claims. In addition to the purchased claims, Midas Gold acquired, by staking on its own behalf, an additional 238 federal unpatented lode mining claims in 2009, 921 federal unpatented lode-mining claims in 2011, and one federal unpatented lode-mining claim in 2012 (re-staked to correct a BLM clerical error). A complete list of active claims is included in Appendix II. Federal unpatented claims total approximately 25,762 acres. Maintenance of unpatented federal claims requires that Midas Gold provide a list of claims and serial numbers to the Bureau of Land Management (**BLM**) with annual maintenance fees of $155 for each lode or mill site claim on or before September 1st each year. This was completed for the most recent filing year on August 4, 2014, and an Affidavit of Satisfaction was subsequently recorded in Valley County on August 21, 2014. There is no underlying royalty on these federal lode and mill site claims other than the Franco-Nevada Corporation (**Franco-Nevada**) royalty detailed in Section 4.4. None of the Claims are subject to back-in rights.

The ownership of the Yellow Pine Deposit was conveyed to Midas Gold in 2011 by way of a company merger between a subsidiary of Midas Gold Corp. and a subsidiary of Vista Gold Corp. (**Vista**) agreed to February 22, 2011. As a result of the combination, Midas Gold, Inc. (**MGI**) 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, Midas Gold purchased 6 patented lode claims in the eastern area of the Project. This group of claims is referred to as the Fern claim group, totaling approximately 100 acres.

4.4 ROYALTIES, OPTION AGREEMENTS AND ENCUMBRANCES

4.4.1 Royalties and Option Agreements

On June 11, 2009, Midas Gold acquired an option to purchase the Meadow Creek group of patented lands from Bradley Mining Co. (**Bradley**) by direct purchase of nine patented mining claims, totaling approximately 184 acres. These lands are subject to a 5% net smelter return (**NSR**) royalty interest (the **Oberbillig Royalty**) to the Oberbillig Group, a group of beneficiaries to the royalty. However, on May 27, 2009, Midas Gold entered into the Oberbillig Royalty agreement whereby it purchased the full 5% NSR royalty from the Oberbillig Group. Midas Gold has an underlying promissory note in respect to the agreement in favor of the Oberbillig Group, which has not reached maturity or been settled as of the date of this Report and is secured by the Oberbillig Royalty. The principal balance on the promissory note is $160,000 as of the date of this Report. The promissory note accrues interest at 3% per annum and matures on June 2, 2015. Property taxes for this and other patented claim groups are included in Appendix II.

Midas Gold secured a purchase agreement from the J.J. Oberbillig Estate on June 2, 2009 to acquire 30 patented federal mill site claims totaling approximately 149 acres, which include both surface and mineral rights and six patented federal lode claims totaling approximately 124 acres. The majority of the mineralization constituting the West End Deposit is located within these 6 patented lode claims. The surface right for portions of six of the patented federal mill site claims was granted to Hecla Mining Company (**Hecla**), however the mineral rights, and the right to explore and mine were retained by the J.J. Oberbillig Estate. With respect to the purchase option agreement described above, Midas Gold currently owes a final annual payment of $40,000 to the J.J. Oberbillig Estate, due June 2, 2015.

On May 3, 2011, Midas Gold entered into an option to purchase 27 patented lode claims totaling approximately 485 acres from the J.J. Oberbillig Estate (the Cinnabar option claims). The total purchase price of these claims is $750,000. To date, Midas Gold has paid Oberbillig a total of $450,000 ($150,000 at closing and $100,000 for the first three of five one-year extensions). All monies spent to date apply toward the purchase price should Midas Gold decide to exercise the option. Two more one-year extensions are available with the final extension expiring on May 1, 2017. Midas Gold is currently responsible for property taxes on these claims. Property tax information for all claim groups is included in Appendix II.

Effective May 9, 2013, Midas Gold granted a 1.7% NSR royalty on future gold production to Franco-Nevada, but not 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 at this time. Midas Gold also retains an option to re-acquire one-third of the royalty for $9.0 million (this option expires May 9, 2016).

4.4.2 Consent Decrees under CERCLA

Several of the patented lode and mill site claims acquired by Midas Gold comprising part of the West End Deposit, and the Cinnabar claims held under option 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 the regulatory agencies that were party to the agreement access 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.

The mineral properties held by Midas Gold's subsidiary, Idaho Gold Resources and that portion of Midas Gold's mineral properties acquired from Bradley pursuant to the Bradley Mining Agreement (i.e. collectively, the Hangar Flats Deposit and Yellow Pine Deposit) are subject to a consent decree that was entered in two United States District Court cases (United States v. Bradley Mining Co., No. 3:08-CV-03986 TEH (N.D. Cal.) and United States v. Bradley Mining Co., No. 3:08-CV-05501 TEH (N.D. Cal)). The first case concerned Bradley's Sulfur Bank Mercury Mine Superfund Site in Lake County, California while the second case was related to the Stibnite Mine site in Valley County, Idaho (part of the Project). On December 7, 2011, these two cases were consolidated into one case (United States v. Bradley Mining Co., No. 3:08-CV-03986 TEH (N.D. Cal.)). A proposed consent decree was lodged on February 14, 2012 and approved on April 19, 2012 after appropriate public comment. The consent decree includes a financial order against Bradley and related terms. The consent decree also states that if Environmental Protection Agency (**EPA**) or the 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 agrees to cooperate with EPA's or the Forest Service's efforts to secure such governmental controls.

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

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

A number of environmental studies and regulatory investigations in the District have identified numerous areas of potential environmental degradation related to historic mining. For detailed ownership and mine development history in the District refer to Section 6 of this Report. In 2009 and 2010, Midas Gold and Vista US contracted Millennium Science & Engineering, Inc. (**MSE**) to conduct Phase I and Phase II Environmental Site Assessments (**ESA**s), as prescribed by the American Society for Testing and Materials (**ASTM**) Standard Practices for Environmental Site Assessments. The results of the ESAs indicate that overall water quality in all drainages is good given the duration and extent of mining. MSE's Phase I ESA identified 88 potential or known Recognized Environmental Concerns (**REC**s) which included several redundant items (e.g. RECs that span both patented and unpatented property boundaries are counted more than once). Based strictly on location or legacy site features, there are approximately 24 distinct RECs that Midas Gold continues to evaluate on an individual basis. There are also some non-ASTM (e.g., geotechnical) issues that are counted in MSE's REC total. The following sections describe the existing historical liabilities; Figure 4.2 provides the general location and extents of these liabilities.

4.5.1 Spent Ore Disposal Area and Historic Tailings

Bradley placed an estimated 4,000,000 cubic yards of mill tailings in the upper Meadow Creek Valley from 1946 to 1952. At the time, Meadow Creek was diverted around the tailings, however, in 1959, Bradley was ordered to breach the diversion and allow the creek to resume a more natural course through the tailings. Over the next 20 years an estimated 10,000 cubic yards of tailings were eroded and carried downstream.

In the 1980s, Canadian Superior Mining (**Superior**) was required to mitigate these historic tailings by constructing a new Meadow Creek diversion channel and stabilizing the tailings by covering them with neutralized ore from their on-off heap leach operations, creating the Spent Ore Disposal Area (**SODA)**. Superior and their successors placed an estimated 6,050,000 tons of spent ore here between 1982 and 1994. The Meadow Creek diversion was moved again circa 2000.

The majority of the historic tailings are located below the water table and likely continue to leach metals. In addition, the upstream wetland lying west of the historic tailings and SODA (formerly a water storage area related to Bradley's operations) is also underlain by tailings.

4.5.2 Hangar Flats / Former Meadow Creek Mine

The first claims were staked in this area in 1914, but significant development of the underground Meadow Creek Mine began c. 1927 and the adjacent Bradley (née, Yellow Pine Company) mill beginning production in 1932. The underground mine closed in 1938 but the mill continued to operate processing ore from the Yellow Pine Pit located 2.5 miles to the north. In 1945, the crusher was moved from the mill to an in-pit location at Yellow Pine and, in 1949, a smelter was added adjacent to the mill at Meadow Creek. The mill closed in 1952 and, by 1957, was dismantled or abandoned. Up until 1946, mill tailings were placed in impoundments adjacent to the mill or pumped directly into Meadow Creek during the winter months. The US Forest Service (**USFS**) has performed several remedial actions in this area, including removal of some historic tailings and the smelter stack in 2003, re-channelization of lower Meadow Creek in 2005, and covering the 2003 impoundments with clean fill in 2009. In spite of these actions, historic tailings remain buried over much of the area, including under the airstrip and adjacent to Meadow Creek.

Later heap leach gold operations also operated in the area of the former Meadow Creek Mine, mill and related facilities, in what is now the area of the current Hangar Flats deposit. Superior and their successors operated on-off heap leach pads and an adjacent process plant from 1982-1997, processing ore from the West End area. The empty leach pads still exist but have been covered by fill. The former process plant and related facilities (site of the current Midas Gold helicopter hanger) have been removed, however, subsurface impacts remain including a former diesel fuel release. Hecla also had a gold heap operation here from 1988-1992 processing ore from the Homestake area of the Yellow Pine deposit. The loaded heap, underdrain system, and infiltration galleries remain but Hecla's former processing facilities have been removed.

4.5.3 Garnet Pit

The Garnet Pit was a short-lived open pit mine, operated by Stibnite Mines Inc. (**SMI**) in the mid-1990s, and located just east of Midas Gold's current camp and core shed area. The ore was processed on the Superior leach pads. The open pit and associated waste rock storage facilities cover an area of approximately 5.5 acres.

4.5.4 Former Stibnite Town Site

During World War II, Stibnite was an incorporated town with a peak population of over 750 Bradley employees and their families. The town site area shown on Figure 4.2 included several employee homes, the recreation center, the hospital, the school, the automobile service station, and the municipal waste landfill. Other "neighborhoods" existed in Stibnite outside this area, including Fiddle Creek and Midnight Creek and there was an earlier landfill location approximately ½ mile to the east.

4.5.5 Former EFSF Haul Road and Adjacent Areas

The area from the former Stibnite town site to the Yellow Pine Pit is the site of many legacy environmental features. The Monday Camp, adjacent to the Yellow Pine Pit, was the site of Bradley's truck repair, machine, and maintenance shops. Fuel oil storage was significant and petroleum contamination is likely to be encountered in the vicinity. Other areas along the now-reclaimed haul road were the site of the former Bradley saw mill, various man camps, the former SMI pilot plant, and other activity. Midas Gold's current exploration camp / core shed area was once the site of the SMI crusher, staging area, shop, and fuel depot. Hecla also maintained a camp and an equipment staging area here.

4.5.6 Yellow Pine Pit and Homestake

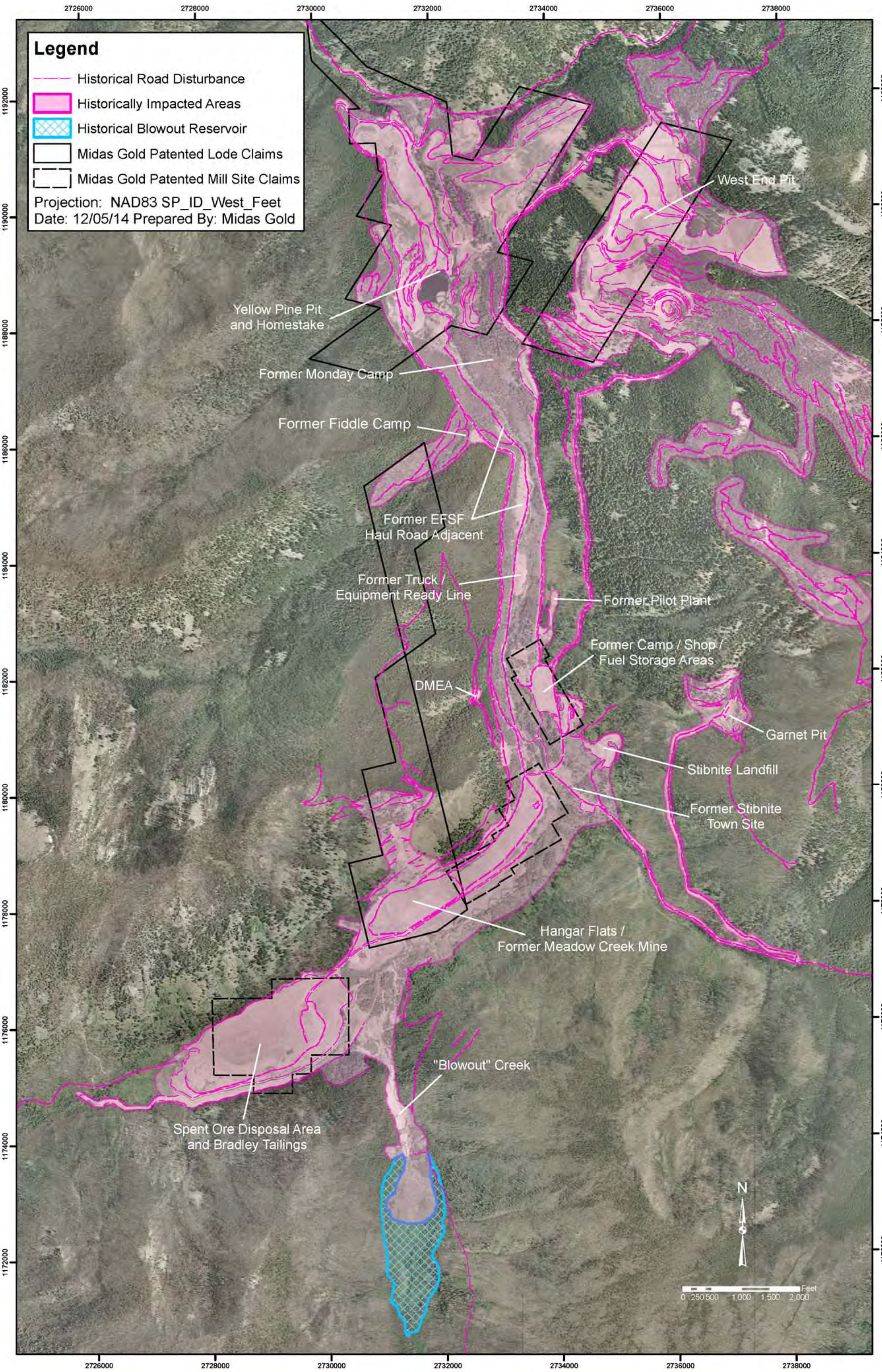
Open pit mining in the Yellow Pine Pit began in 1937. During World War II and the Korean War, the pit provided an estimated 90% of the antimony and 50% of the tungsten needed for U.S. war efforts. In 1943, the 3,500 foot Bailey Tunnel was driven to divert the East Fork of the South Fork of the Salmon River (**EFSFSR**) away from the open pit. Mining ceased in 1952 until Hecla returned to mine the adjacent Homestake area from 1988-1992.

The open pit and subsequently the pit lake has been a barrier to fish passage since 1938. The pit lake has also acted as a sediment trap and holds legacy tailings eroded from the Meadow Creek Valley upstream. There are numerous waste rock storage facilities throughout this area and spent ore has been placed here and elsewhere in the District (e.g., as road surfacing). A waste repository was also created on the west rim of the pit in 2003 during a Forest Service removal action related to the former smelter stack and other remediation actions in the Hangar Flats area.

4.5.7 West End

West End is a series of open pits and waste rock storage facilities. Superior began work in the area in 1982. They and their successors (Pioneer Metals Corp., Pegasus Gold Corp, Dakota Mining, and SMI) continued mining until 1997. The area remains sparsely vegetated. The upper reach of West End Creek is in a large diameter culvert pipe beneath a waste rock storage facility.

Figure 4.2: Legacy Environmental Liabilities



4.5.8 East Fork of Meadow Creek

A small reservoir was constructed on the East Fork of Meadow Creek (commonly referred to as Blowout Creek) in 1931 for electric power generation. The dam was raised in 1949 to accommodate increased water demand created by the new smelter. In June 1965, high runoff caused a catastrophic failure of the dam. Damage at the time was significant and the drainage remains a major source of sedimentation due to active head cutting and erosion during high flow events. The head cutting has also lowered the water table in the upper valley, reducing the quality of the wetlands in that area.

4.5.9 DMEA

As the first generation of active mining ceased in 1952, Bradley was awarded two contracts by the Defense Minerals Exploration Administration (**DMEA**) and performed exploration work through 1955. This work included 4,900 ft of underground workings with an associated adit and waste rock storage facility at the location shown on Figure 4.2. A creek runs through the waste rock storage facility.

4.5.10 Underground Workings

In addition to the disturbed areas show on Figure 4.2, several underground workings also exist. There are numerous tunnels, adits, shafts, and associated waste rock storage facilities. Major historic underground workings include the Meadow Creek Mine, the Cinnabar Tunnel, the Monday Tunnel, the North Tunnel, the Bailey Tunnel (former EFSFSR water diversion), and DMEA workings; while on the adjacent optioned claims, the Cinnabar Mine has extensive underground workings and related surface disturbance.

4.5.11 Past Remediation

In the past, regulatory actions under U.S. Comprehensive Environmental Response, Compensation, and Liability Act (**CERCLA**), U.S. Resource Conservation and Recovery Act (**RCRA**), and state law have been taken by the EPA, USFS, the Idaho Department of Lands (**IDL**) and the Idaho Department of Environmental Quality (**IDEQ**) against historic mining operators. These agencies along with several past owners and operators have conducted certain remediation of historic mining activities at the Project site. The primary projects include the following:

- stabilizing and covering with spent ore the 551 acres historic tailings pile at the upper-south end of the Meadow Creek valley;
- re-routing the Meadow Creek diversion and reconstructing it to accommodate a 500-year flood event;
- capping some of the old heap leach facilities;
- removal of nearly all above ground historic mining structures and facilities including the former Meadow Creek mill and smelter;
- removal of leaking underground fuel storage tanks left from previous owners; and
- EPA conducted and paid for the clean-up of the smelter stack, assay labs and other sites at Stibnite well after 2001.

The Stibnite site was considered and rejected for listing on the National Priorities List (**NPL**) in September 2001.

Midas Gold may conduct environmental enhancement and reclamation opportunities in the future as part of an operating plan, including potentially improving or re-establishing key fish passage routes.

4.6 PERMIT REQUIREMENTS

4.6.1 Exploration Permits and Status

The Midas Gold exploration programs completed 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. 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.

Midas Gold is currently conducting exploration in the Stibnite mining district on patented property under an annual IDL Notice of Exploration. Midas Gold currently has an exploration Plan of Operations (**PoO**) filed with the USFS under POO-2014-049059 and is awaiting approval.

4.6.2 Exploration Compliance Evaluation

Midas Gold is 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 activities on the site.

4.6.3 Mine Development Permits

The environmental permitting process for the development of a mine within the Project boundaries would primarily involve: 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 PoO/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.

A detailed list of applicable permits, licenses, and approvals is listed in Table 4.1. The National Environmental Policy Act (**NEPA**) requires federal agencies to study and consider the likely environmental effects of any project before allowing it to proceed. In the case of the development of a mine within the Project boundaries, the major federal action necessary to move the project forward is the approval of the final Plan of Operations/Reclamation Plan by the USFS (Forest Supervisor) in conjunction with the IDL and other cooperating agencies. This would be done by the preparation, review, and completion of a decision (Record of Decision or **ROD**) on the Environmental Impact Statement (**EIS**). The EIS would likely be prepared by a qualified third-party environmental contractor under the guidance of the USFS. The cost of the EIS would be borne by Midas Gold.

A more detailed description of all the permits and authorizations listed in Table 4.1 is provided in Section 20. Some ancillary permits and licenses, not listed in the table that may be required are conditional use permits, rights of way and Federal Communications Commission (**FCC**) communications licenses.

Table 4.1: List of Permits, Licenses, and Approvals Required for the Project

Permit, License, or Approval	Purpose of Authorization	Timing
USFS		
Approval Final Plan of Operations (36 CFR 228 Subpart A)	To allow for locatable mineral exploration and development. This PoO must be consistent with the forest plan. Approval follows the EIS ROD and incorporation of specified requirements and mitigation and monitoring in the final plan so as to minimize or eliminate effects on surface resources. A reclamation plan and financial assurance are a primary element of the final PoO. A waste management plan would also be important.	To be filed after the EIS ROD. Requires input from the final feasibility study.
Road Use Permit	To specify operation and maintenance responsibilities on forest roads. Valley County is also involved as a primary agency. The Road Use Permit would be incorporated into the final PoO.	Required annually for current exploration and final Plan of Operations.
Mineral Material Permit	To allow Midas Gold to collect and use borrow materials from national forest lands.	Would be needed during construction.
Timber Sale Contract	To allow Midas Gold to harvest commercial timber from the Project area (construction clearing). The Timber Sale Contract would be incorporated into the final PoO.	This is a pre-construction need.
Cultural Resource Clearance	To obtain joint approval from the USFS and State Historic Preservation Officer prior to construction.	This is a pre-construction authorization.
Monitoring Plans	Part of the final PoO to assure compliance with state and federal environmental standards.	Required for construction, operations, reclamation and post-closure.
Plan of Operations Review	To ensure consistency with design of plant processing, waste management, water treatment, access roads and other facilities, operational requirements as described in the ROD, final PoO, and other permit approvals.	This is an annual requirement.
U.S. Environmental Protection Agency		
NPDES Permit (water discharge) (EPA)	Required under Section 402 of the Clean Water Act for point source discharges to waters of the US, total maximum daily loads must be considered in this permit. Also required for all "new sources". Section 401 Certification of NPDES and Corps 404 Permits by IDEQ is also required.	Application must be filed 180 days prior to discharging.
Storm Water Pollution Prevention Plan (EPA)	Required to minimize or mitigate the effects of storm water discharges; also includes snowmelt runoff and surface runoff and drainage.	This is a pre-construction requirement.
Section 311 Contingency Plan (EPA)	Required to develop a spill prevention plan for above-ground fuel storage capacity in excess of 1,320 gallons (5,000 liters), or below-ground storage greater than 42,000 gallons (159,000 liters).	Needed during construction, operations and post-closure.
Drinking Water Act Underground Injection Control Permit (EPA)	To regulate subsurface emplacement of wastewater by well injection; may also apply to land application of wastewater.	This permit may be required for operations.

Permit, License, or Approval	Purpose of Authorization	Timing
Prevention of Significant Deterioration Determination (EPA)	Initial analysis to demonstrate the mine would not emit more than 250 tons (230 tonnes) of fugitive dust per year. This is a preconstruction authorization.	To be evaluated as part of the EIS.
Permit to Construct/Permit to Operate (Jointly with IDEQ)	Required to be in compliance with applicable requirements of the Clean Air Act to the extent the permit specifically includes these requirements, or includes a determination that specific requirements do not apply.	To be filed during the EIS/pre-construction phase.
USACE		
Section 404 Dredged and Fill Permit	Required for discharge of dredged or fill material into waters of the U.S. including wetlands.	Concurrent with the EIS; ROD required.
NOAA Marine Fisheries / USFWS		
Fish and Wildlife Endangered Species Act Consultation	Required to demonstrate that the proposed action would not likely jeopardize threatened and endangered fish and wildlife species.	Concurrent with the EIS; ROD required.
Mine Safety and Health Administration (MSHA)		
Safety Plan	This plan is developed to assure the health and safety of the nation's miners. The plan addresses accident protection, communications, prohibition of alcohol and drugs, and other considerations.	Required for active mining operations.
Executive Orders (USFS, EPA, USFWS, and USACE)		
E.0.11988; E.0.11990	These two executive orders deal with the protection of floodplains and floodplain management, and the protection of wetlands.	Concurrent with the EIS.
Indian Tribes		
Native American Consultation	Required to ensure no significant impacts on Nez Perce, Shoshone-Bannock, and Shoshone-Paiute traditional cultural values, practices, properties, or human remains.	Concurrent with the EIS.
IDEQ		
Rules Governing Cyanidation Facilities (Jointly with IDL)	Required to construct and operate cyanide processing facilities; rules also address performance bonds for reclamation and permanent closure of these operations.	To be filed after EIS scoping.
Wastewater Land Application Permit	To regulate the application of industrial/municipal wastewater to plan for the purpose of treatment; used to meet zero discharge requirements.	To be filed after EIS scoping.
Ground Water Rule	Required to protect local ground water resources and quality to be maintained at or near background levels (sites-specific standards).	To be filed after PDEIS.
Air Quality Tier Permit	Required to operating permit for major stationary sources of air pollutants.	To be filed concurrent with EIS.

Permit, License, or Approval	Purpose of Authorization	Timing
Idaho Department of Water Resources (IDWR)		
Water Rights	These permits are required prior to the diversion and use of surface or ground.	Additional to be filed after PFS.
Stream Channel Alteration Permit	Required to divert a stream around mining facilities (i.e. EFSFSR around Yellow Pine open pit)	To be filed prior to PDEIS.
Dam Safety Permit	Required for the construction of a tailings dam; includes detailed design and specifications.	To be filed prior to PDEIS.
IDL		
Reclamation Plan	Describes plans for closing and reclaiming components of the operation located on patented (fee) ground. This plan and associated financial assurance closely interface with the final PoO for the USFS.	To be filed prior to PDEIS.
State of Idaho – Historic Preservation Office		
State Historic Preservation Officer Consultation	The final PoO must receive clearance by the State Historic Preservation Officer.	Approval required prior to construction.
Valley County Planning Department		
Consistent with Comprehensive Plan	The final PoO (proposed Project) must be consistent with the goals and objectives of the Valley County Comprehensive Plan.	Approval required prior to construction.
Valley County Building Department		
Individual Building Permits	A number of building permits would need to be obtained for the primary Project components	Approval required Prior to construction.
Valley County Road Department		
Road use Permits	Road use permits would be required by the county annually; a user fee to pay for necessary road maintenance would be negotiated, including special conditions for winter use.	Annual application required.
Other Consultations/Clearances		
Other Consultations	These include, among others: Executive Order 12898 Environmental Justice, Migratory Bird Treaty Act, Bald Eagle and Golden Eagle Protection Act, USFS Mining Regulations (36 CFR 228A).	Part of the EIS process.

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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 with narrow, flat valleys at an elevation of approximately 6,500 ft and nearby mountains rise to an elevation of approximately 7,800 to 8,900 ft. The land is heavily wooded with fir and pine trees and underbrush is 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: SRK, 2012

Photograph 5.2: View Looking West up Meadow Creek

Source: SRK, 2012

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 32.19 inches per year. Average 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, a city with a population of more than 600,000 people in its metropolitan area. Figure 5.1 shows a map of current access routes.

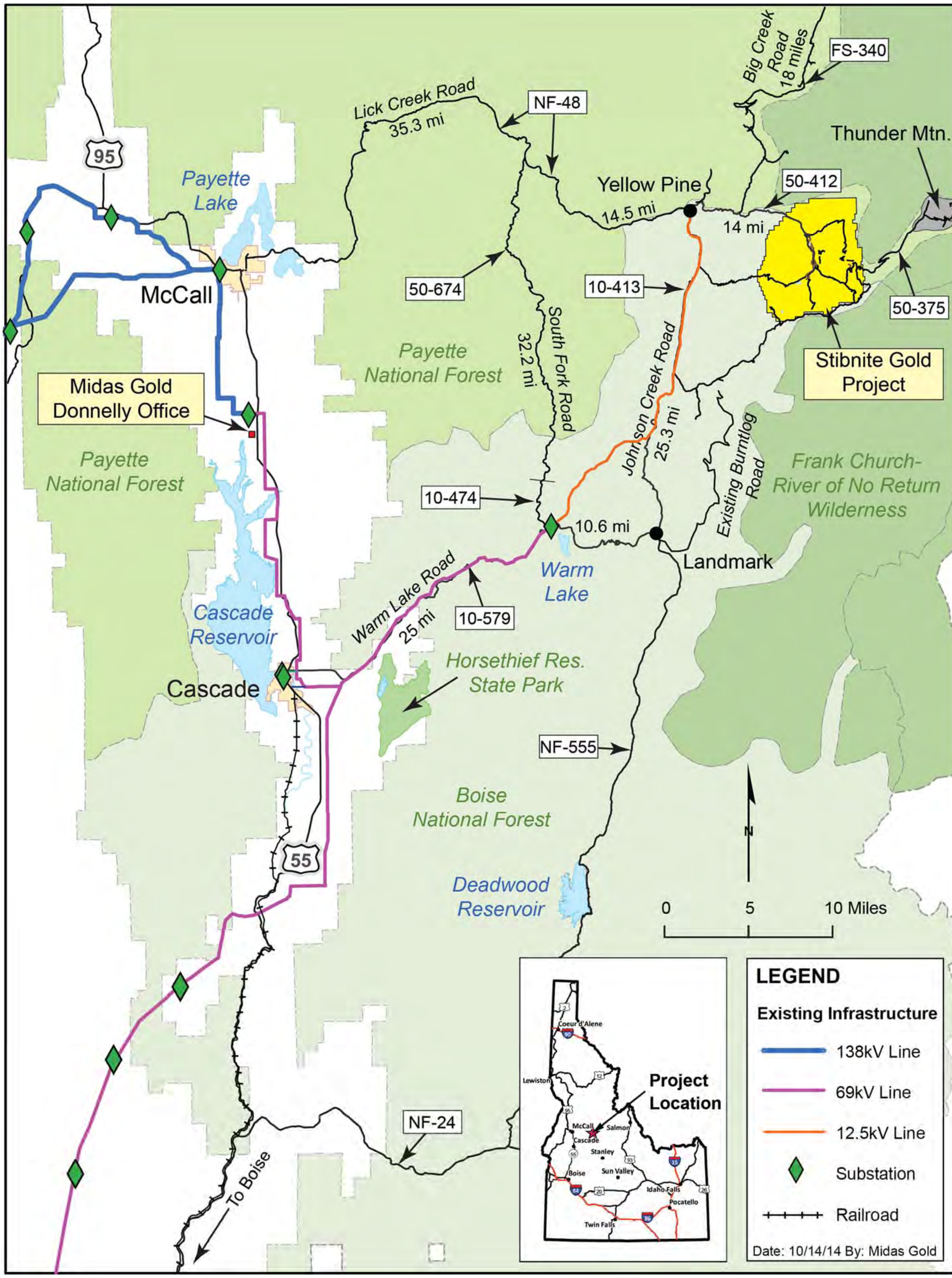
The primary access to the Project area is via:

- 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)
- Yellow Pine to Stibnite – single-lane, unpaved Forest Service 50-412 Road (14 mi)

The primary access (the "**Johnson Creek Route**") measures approximately 84 mi from Cascade to Stibnite and is not available at certain times of the year when Johnson Creek Road is impassable due to snow. Alternate, low elevation, year-round access is available by traveling from Cascade along the Warm Lake Road and turning north on the South Fork Road 10.6 mi west of Landmark and then turning east onto the East Fork Road (NF-48) towards Yellow Pine and onto Stibnite (the "**South Fork Route**"). The distance from Cascade to Stibnite is approximately 86 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 63 mi. and approximately 93 mi from Cascade to Stibnite via McCall. A grass airstrip is located at Johnson Creek, about 3 mi south of the town of Yellow Pine and a 2,300 ft long improved gravel airstrip is located at Stibnite.

Figure 5.1: Site Access and Existing Pertinent Regional Infrastructure



5.4 SUFFICIENCY OF SURFACE RIGHTS

Midas Gold currently controls 27,105 acres of land through a combination of 1,492 patented and unpatented 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. Approvals for such development come from approval of the USFS Plan of Operations (**PoO**), which will come in the form of a Record of Decision following completion of an anticipated Environmental Impact Statement, along with a mined land reclamation plan from the IDL. Additional information on Midas Gold patented and unpatented claims are 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 power lines are located in the town of Yellow Pine, roughly 10 mi to the northwest. The power line to Yellow Pine would be insufficient to support a mining operation. Power lines 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 power transmission line route is addressed later in Section 18 of this Report.

5.5.2 Water Supply

Midas Gold has 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). One of these Water Rights (77-7293) is currently shown in the IDWR database as owned by Bradley, however, Midas Gold has provided title information to IDWR, which has acknowledged that the documentation is sufficient, and is in the process of updating the database listing.

Table 5.2: Water Rights Summary

Water Right ID	Type	Source	Location	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 (Hennessey Creek)	SW¼ of the NE¼, Section3, T18N, R9E	Mining	0.25	20.0
Source: IDWR, 2014						

Midas Gold's 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 are included in Section 20.

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 operates the Cascade Branch rail line on approximately 100 mi of track between Payette, Idaho and Cascade with a switch yard in Emmett, Idaho. Track runs from Cascade to Payette connecting the line to the Union Pacific Railroad which is capable of reaching ports in California, Oregon, Washington and British Columbia. The INPR also operates a tourist train, the Thunder Mountain Line, on its Cascade Branch, which runs from Horseshoe Bend to Banks, Idaho.

Active freight service to Cascade ceased in the mid-1990s, but INPR continues to maintain the line to Cascade and conducts annual maintenance and inspections. INPR owns land at the terminus of the rail line for switching and transload 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 trans-load and intermodal services facility in the southeast of Boise. Though construction of the proposed facility has not progressed beyond the initial letter of intent, when 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 Hanjin Shipping on a weekly basis.

Additionally, The 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 trans-loading 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 Hanjin 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. 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 surface grade 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 off-site on a nightly basis to a Midas Gold-owned co-located server at the Syringa Networks data center adjacent to the Boise airport.

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 colluvium/alluvium/bedrock contact, which is consistent with infrastructure siting by previous mine-operators.

The process plant site selected to support the 2012 PEA was located on the southeast side of the confluence of Meadow Creek and the EFSFSR. While the PEA plant area is relatively flat and open, centrally located, with minimal underlying mineral resource potential, the area possesses pervasive wetlands, challenging foundation conditions, and is in very close proximity to sensitive jurisdictional waters. Following publication of the PEA, alternative process plant sites were identified by the Midas Gold team; consequently, a more comprehensive study was deemed appropriate to arrive at the preferred layout. 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.

Following this process, 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; a simplified version of the site layout is provided on Figure 5.2.

5.5.7 Potential Tailings Storage Area

Approximately 98 million tons of mineralized material are expected to be processed during the 12-year mine life of the Project as contemplated in this PFS. 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 five locations that could provide sufficient storage capacity to contain the expected tailings quantities. An additional 14 smaller sites that could contain a portion of the required tailings storage in a second, separately managed facility, were also identified (SRK, 2012).

The preferred tailings site, based on considerations such as: topography, hydrology, use of previously disturbed areas, environmental management and closure considerations, proximity to the processing plant, and expected cost, was determined to be in the Meadow Creek Valley. The valley has sufficient capacity for both tailings and waste rock, and a significant portion of the area has been previously disturbed by historical mining operations. This site was identified and incorporated into the PEA and continues to be the preferred tailings storage site since it, among other things, keeps incremental disturbance to a minimum by overlapping on pre-existing historically disturbed areas used previously for tailings disposal. A comprehensive description of the Meadow Creek TSF is provided in Section 18; a simplified figure showing the location of the TSF is presented on Figure 5.2.

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 and is controlled by the USFS.

5.5.8 Potential Waste Rock Disposal Area

There are several locations on the Stibnite Gold Project site where uneconomic mineralized material or waste 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 waste rock is in the Meadow Creek Valley downstream of the TSF that would, in addition, provide a robust geotechnical stability buttress for the TSF. This site is preferred since it, among other things, keeps incremental disturbance to a minimum by overlapping on pre-existing historically disturbed areas used previously for tailings disposal and spent heap leach ore disposal, and keeps the waste rock and tailings within the same area. The preferred storage location for the West End waste rock is above the existing West End waste rock storage facility and, mostly, in the mined-out Yellow Pine open pit, which would enable the EFSFSR to be reestablished to its original gradient, facilitating long-term fish passage to the headwaters of the EFSRSR and Meadow Creek. Some of the proposed waste rock storage land is owned by Midas Gold and comprises patented mining claims; the rest of the land in the valley is Federal land and is controlled 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 waste rock storage areas; a simplified layout is provided on Figure 5.2.

5.5.9 Labor

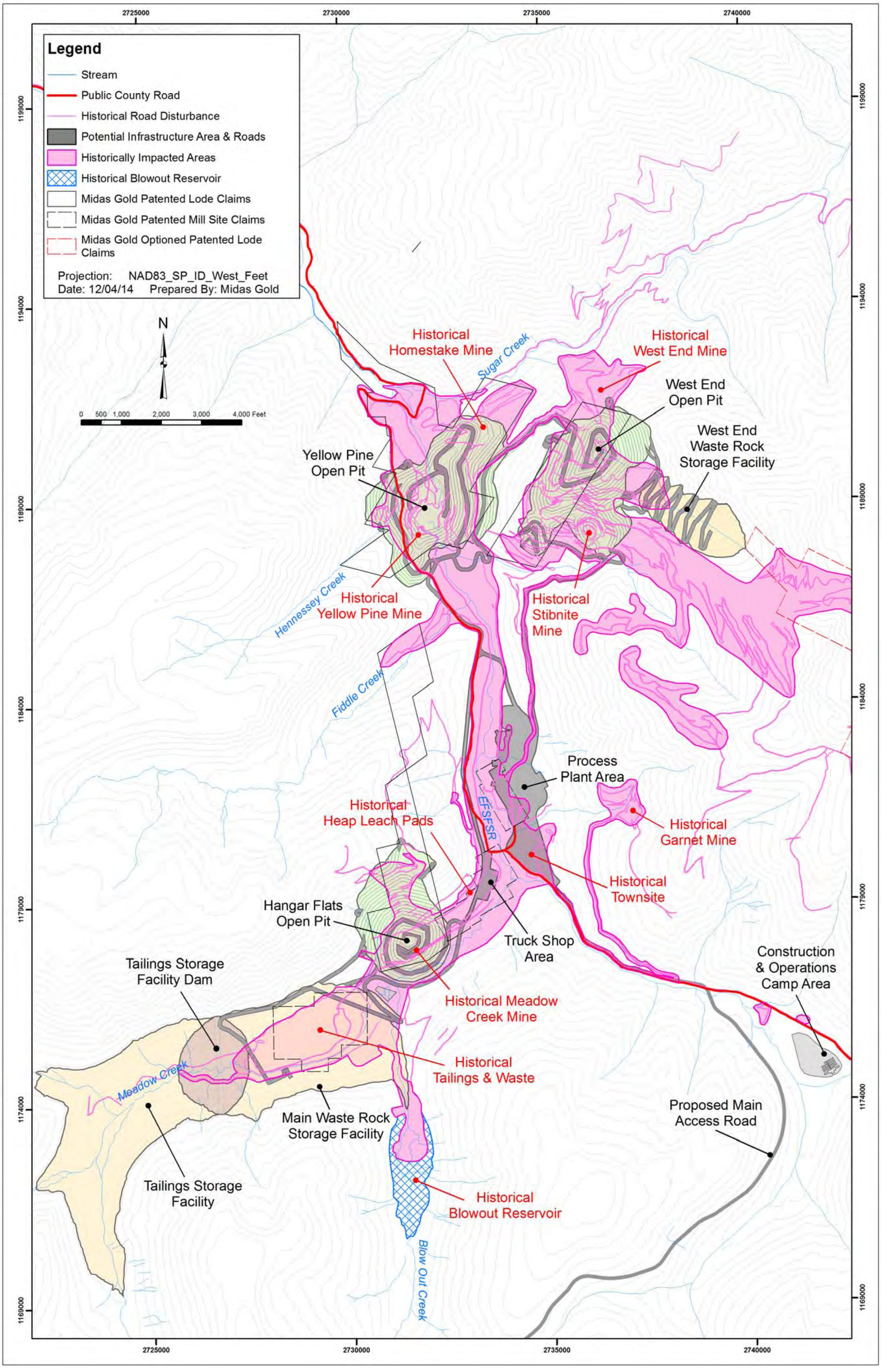
Yellow Pine, which is the nearest town, is located approximately 14 mi to the west of the Project. It has a population of approximately 60 people during the summer months, up to 40 in the winter, and limited services such as a general store, a restaurant, and a few lodging facilities. The nearby Valley 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 and mining professionals, as well as local laborers and equipment operators, would be identified from within Valley County and adjacent Adams County, where feasible, with additional workers sourced throughout Idaho if necessary.

Based on the currently envisioned Project, Midas Gold would likely become the largest employer in Valley County and Adams County, paying higher salaries than any other industry except the federal government. These two counties have some of the highest unemployment rates in Idaho, which is nearly double the rate in Boise. Midas Gold jobs would revitalize the local manufacturing sector and provide an important complement to the region's recreation industry. In addition to the long-term operations-related employment opportunities, Midas Gold would also employ a large number of construction workers during the construction phase, which would bolster the slumping real estate industry. The property and sales taxes generated from the mining operations would help support the region's schools and infrastructure, which have been under recent economic stress. The infusion of new economic activity would likely help support every industry in the regional economy.

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

Figure 5.2: General Site Layout



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6 HISTORY

6.1 OWNERSHIP AND ROYALTIES

During the first half of the 20th century, two major landowners were working the Stibnite Mining District. The eastern part was partially consolidated by United Mercury Mines, whereas the western part was controlled by Bradley. Bradley production was initially from the underground Meadow Creek mine (c. 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 district led to the Oberbillig family receiving royalties on some of the claims mined by Bradley. Mining claims associated with the Meadow Creek Mine and Yellow Pine Mine (first staked in 1914 and 1923, respectively) were patented during this period. Bradley operated Yellow Pine until 1952 and optioned the nearby Cinnabar mine in the late 1950s. Mining operations ceased after a worldwide collapse in antimony and mercury prices following the end of the Korean War, while milling and smelting continued from stockpiled ores, as well as antimony-bearing materials from the Coeur d'Alene district, as well as tungsten ores from the Springfield and Ima tungsten mines. The former mill and smelter were subsequently dismantled and the Stibnite town site abandoned with many of the cabins and other buildings comprising the town site and other facilities moved elsewhere.

Aside from minor mining and processing of stibnite ores from the Stibnite District and mercury mining at the nearby Cinnabar mine in the 1960s, the Stibnite/Yellow Pine district lay dormant until the early 1970s, when a sharp rise in gold prices and the advent of heap-leach processing technology for oxide gold ores revitalized exploration in the District. Operators who conducted exploration and/or minerals 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, Barrick Gold Corporation (**Barrick**, and formerly American Barrick Resources), and SMI.

Hecla delineated a small oxide resource at 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 they brought that area into production in 1982. Superior was ultimately acquired by the Superior Oil Company of Houston, Texas, which, in turn, was acquired by Mobil Oil. Mobil sold the West End Mine in 1986 to a 50/50 joint venture of Pioneer and MinVen, both small 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 Mining Company (**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.

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 Oberbillig Family Estate. These Oberbillig Estate patented lands and the 5% NSR royalty interest on the Bradley Estate are currently the subject of purchase option agreements with Midas Gold (both Promissory Notes mature June 2, 2015). Midas Gold will purchase the royalty which would, in effect, have Midas Gold pay the royalty to itself.

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, MGI entered into a combination agreement with Vista US and IGR whereby these entities became wholly owned subsidiaries of Midas Gold. Midas Gold made final payment under the Option to Purchase on November 28, 2012 and now holds 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 both the Bradley and Oberbillig Family 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 100%.

On May 1, 2011, MGI's wholly owned subsidiary, MGIAC, entered into an option agreement with the owners of a number of patented and unpatented mineral claims comprising the former Cinnabar Mine property JJO, LLC a limited liability company and the personal representative of the estate of J.J. Oberbillig, whereby MGIAC had the right but not the obligation to acquire these claims over a period extending to May 1, 2017 in exchange for certain payments. MGIAC has made all payments required to date under the option agreement and the option agreement remains in effect.

MGI subsequently completed staking of additional claims and corrected claim deficiencies between 2009 and 2012.

The entire property (excluding the Cinnabar group of claims) is subject to the May 9, 2013 1.7% gold only NSR royalty held by Franco-Nevada Corporation. Midas Gold's subsidiaries have a one-time right to repurchase one third of the royalty for US$9 million before May 9, 2016 thereby reducing the royalty to 1.13%.

6.2 PAST EXPLORATION AND DEVELOPMENT

There have been two major periods of exploration and development operations in the District prior to Midas Gold gaining control, one spanning from the early 1900s through the 1950s and another during the period from the early 1970s through the mid-1990s. 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 to the Thunder Mountain District 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).

Between the 1920s and late 1990s, 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.

6.2.1 Hangar Flats Deposit

Gold and antimony mineralization were discovered in 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 modern style exploration core 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 core 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. Figure 6.1 shows the processing facility and tailings pond for the Meadow Creek Mine during this time period. Most of the historic underground maps, tunnel assays, drill logs, and drill assay results can be found in Midas Gold's files or the Idaho Geological Survey archives.

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, all post-operations. 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).

Figure 6.1: Bradley Mining Company Processing Plant and Tailings Pond

Source: Photograph circa 1942, courtesy of Robin McRae

From 1951 through 1954, the Defense Minerals Exploration Administration (**DMEA**) carried out an underground exploration program immediately to the 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 from underground stations was also carried out. Detailed drill logs and systematic assaying were well documented.

In the late 1970s, Ranchers acquired 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 historic 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. Figure 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 of the project in 1987. Barrick optioned the property in the early 1990s 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 in 2003 (Pincock, Allen and Holt, 2003) and a Preliminary Assessment by the same group in 2006 (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 by purchasing IGR in 2011.

Figure 6.2: Bradley Mining Company Open Pit Mine

Source: Photograph circa 1942, courtesy of Robin McRae

6.2.3 West End Deposit

Gold mineralization was first discovered along the West End Fault by Bradley interests in the late 1930s or early 1940s; during this time Bradley's exploration focused on replacement of reserves at their Yellow Pine mining operation. Subsequent work sponsored by the USGS outlined a large biogeochemical and 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 mid-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 Environmental Impact Statement, five heap-leach pads were constructed, and a 2,000 - 3,000 st/d oxide mining operation began in 1982 (Figure 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 once again 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).

Figure 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 mid- to late-1990s. Between 1982 and 1994 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.

6.3 HISTORICAL MINERAL RESOURCE AND RESERVE ESTIMATES

Through the years after historic mining ceased in the 1950s, 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. These estimates are available in Midas Gold files, but were completed prior to 1995 and were not prepared in accordance with the requirements of Sections 1.2 and 1.3 of NI 43-101. There are no historic Mineral Resource or Mineral Reserve estimates that compare with the Mineral Resource estimates of this Report. Historic 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 (**PAH**) 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 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 and the PEA.

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 and, 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 below.

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
Yellow Pine	1938 - 92	6,493,838	479,517	1,756,928	40,257	856,189
West End	1978 - 97	8,156,942	454,475	149,760	-	-
Totals		**14,954,633**	**985,602**	**2,088,551**	**44,015**	**856,256**

Note:
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.

6.4.1 Hangar Flats Deposit

Gold, silver, antimony, and tungsten 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
TOTAL		**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.*

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, from 1989 to 1992 gold was produced from open pit operations 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 Records

Company Name	Production Year	Tons Mined (st)	Recovered Au (oz)	Recovered Ag (oz)	Recovered Sb (st)	Recovered WO_3 (Units)[2]
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

Company	Production	Tons Mined	Recovered	Recovered	Recovered	Recovered
Name	Year	(st)	Au (oz)	Ag (oz)	Sb (st)	WO_3 (Units)[2]
Bradley	1951[1]	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	-	-	-
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. *Re-processing tailings.*
2. *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.*

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	Production	Tons Mined	Recovered	Recovered	Recovered	Recovered
Name	Year	(st)	Au (oz)	Ag (oz)	Sb (st)	WO3 (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 (In Garnet Creek 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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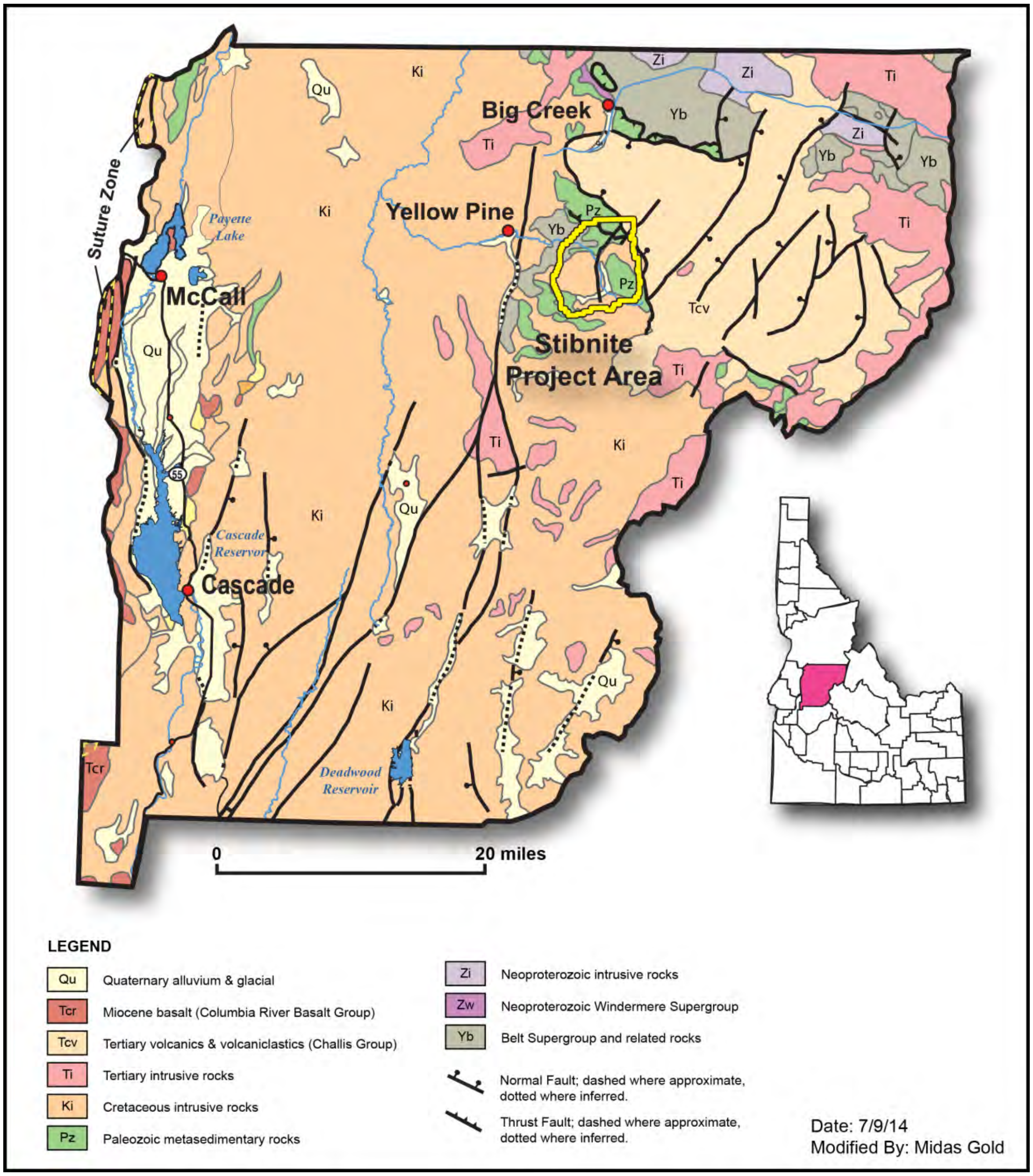
7 GEOLOGICAL SETTING AND MINERALIZATION

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 rocks that are part of the pre-Cretaceous "basement," the Cretaceous Idaho Batholith, Tertiary intrusions and volcanics, and younger unconsolidated sediments derived from erosion of the older sequences and glacial materials (Figure 7.1).

Figure 7.1: Valley County, Idaho Regional Geologic Map



Source: Modified from Lewis, 2002

The pre-Cretaceous basement rocks record the development and subsequent tectonic overprinting of the western Laurentian continental margin which formed during a protracted rifting event from Neoproterozoic through middle Paleozoic time. This rifting event was accompanied by deposition of rift and passive margin sediments along the western edge of ancestral North America. Poorly preserved remnants of the rift and subsequent passive margin sedimentary sequences are exposed in the region as discontinuous roof pendants in a broad northwesterly trending belt adjacent to or as inliers within the Idaho Batholith extending from southeast Idaho to at least as far north as northeast Washington and beyond (Lund, et al., 2003; Lewis, et al., 2012).

These rocks record a long and varied sedimentary record spanning Proterozoic through Paleozoic time and 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). 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, recent regional mapping in conjunction with more definitive age determinations using high resolution detrital zircon dating methodologies suggests the youngest metasedimentary rocks within the Project area are correlative in part to rocks exposed in southeast Idaho in the Bayhorse region and in the northwestern Panhandle of Idaho and record the Neoproterozoic rifting event and development of a passive continental margin (Lewis, et al., 2014). This mapping and dating is being conducted by the Idaho Geologic Survey (**IGS**). Research is ongoing and findings will be published when necessary and appropriate by the researchers involved. Mapping and dating reported here are the results of this ongoing research.

Although difficult to document at the local scale, regional mapping indicates pre-Cretaceous basement rocks in the region underwent several periods of deformation, likely including the Cretaceous-Tertiary Sevier and Laramide orogenies. Each subsequent orogeny resulted in progressively eastward contraction of the miogeoclinal sequence and underlying, older rift-related units. The Salmon River Suture Zone, situated west of the Project area (Figure 7.1), marks the transition zone between Precambrian continental crust of North American affinity to the east and accreted Neoproterozoic to Paleozoic oceanic crust to the west, as defined by various petrologic and geochemical studies as well as isotope values and geophysical models (Piccoli and Hyndman, 1985; Kleinkopf, 1988; Lund and Snee, 1988; Strayer, et al., 1989).

After rifting and development of the passive margin, regional folding and faulting in the early Paleozoic was followed by extensive early Mesozoic folding, extensive west to east thrust faulting in the middle and late Mesozoic, and late Mesozoic normal faulting (Lund, et al., 2003). The Idaho Batholith intruded the sedimentary sequences in mid-to-late Cretaceous. 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 eastern margin is overprinted by younger Tertiary caldera complexes (Fisher, et al., 1992). Intrusive activity and volcanism continued through the Tertiary during uplift as the batholith was unroofed. During the Eocene the Challis volcanics blanketed the region to the east. Eocene, and later Miocene, Basin and Range normal faulting reactivated pre-existing Cretaceous structures resulting in a series of normal fault-bounded basins. To the west of the Project area, evidence of widespread extensional deformation is concentrated in the Late Cretaceous Western Idaho Shear Zone, resulting in the development of the Long Valley basin near the towns of New Meadows, McCall, Donnelly and Cascade. The area affected by the Western Idaho Shear Zone displays two orientations of steep faults: one set of normal faults strikes north-south and is parallel to fabrics within the suture and the other sets strike east-west and northeast and accommodate components of both normal and strike-slip movement. Similar structural trends are evident in the area surrounding the Project. 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 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 characteristics to the Johnson Creek Profile Gap structure.

Regionally, the Atlanta Lobe of the Idaho Batholith shows a progression from early mantle-derived metaluminous magmatism from 98 mega-annum (Ma [or million years]) to 87 Ma, followed by more voluminous crustal-contaminated peraluminous magmatism from 83 Ma to 67 Ma, which is attributed to crustal thickening, resulting from either subduction processes or terrane collision (Gaschnig, Vervoot, Lewis, and Tikoff, 2011; Lund, 1999).

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. 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 Project 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.

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 sedimentary fluvial deposits in floodplains often composed of a mixture of angular clasts from adjacent bedrock sources combined with more rounded reworked glacial deposits.

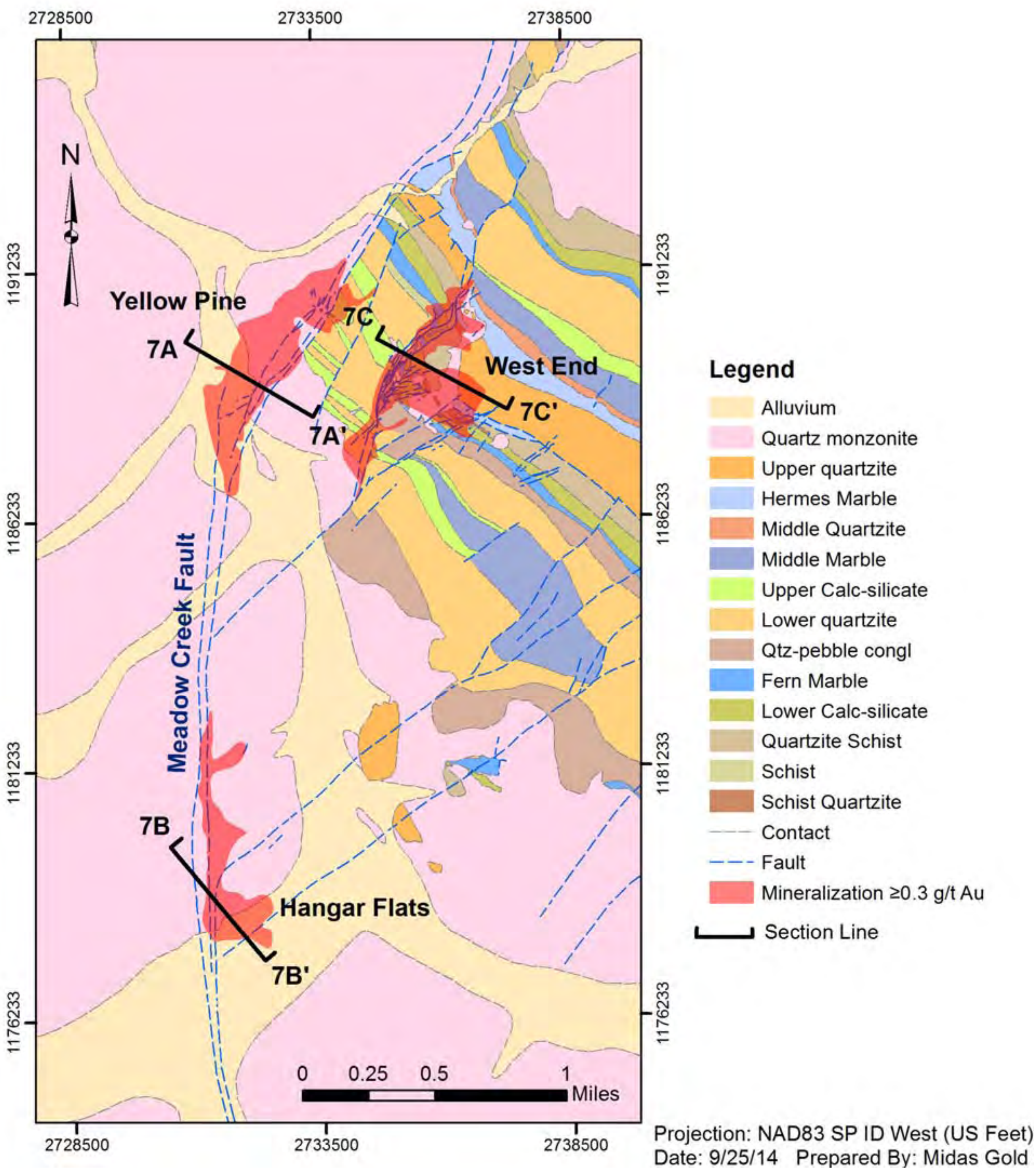
7.2 LOCAL GEOLOGY

7.2.1 Lithology

The Yellow Pine Deposit is hosted by intrusive phases associated with the Atlanta Lobe of the Idaho Batholith and by down-dropped blocks of metasedimentary rocks. The Hangar Flats Deposit is hosted by intrusive phases associated with the Atlanta Lobe of the Idaho Batholith. Other post-mineralization intrusive igneous rocks associated with the Challis Volcanics also occur within the Yellow Pine and Hangar Flats Deposits. The West End Deposit is hosted by metasedimentary rocks of the Stibnite roof pendant located within the Atlanta Lobe of the Idaho Batholith. Figure 7.2 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 7.2: Local Geology of the Stibnite Mining District



Quartz Monzonite

The dominant type of intrusive rock exposed in the District and intersected in drilling consists of Cretaceous, light to medium gray, equigranular, medium- to coarse-grained, granodiorite and quartz monzonite with distinctly peraluminous bulk rock geochemical compositional characteristics (Photograph 7.1). 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. Zircon rims from an unaltered sample of biotite quartz monzonite from the Hangar Flats Deposit (drill hole MGI-10-21, 171-174 ft) were dated by U-Pb methods with LA-ICPMS to have an age of 91.2 Ma ± 2.2 Ma as reported by the IGS (Lewis, et al., 2014).

Photograph 7.1: Example of Quartz Monzonite in HQ Core

Alaskite

Alaskites occur as dikes, sills and segregations and range in width from less than 1 inch to over 30 ft. The alaskites are relatively siliceous, are typically fine-grained, sucrosic textured, and can be distinguished from the quartz monzonite by the lack of biotite or other mafic minerals (Photograph 7.2). The alaskite dikes can be coarsely crystalline to pegmatitic locally. Some alaskite dikes are unaltered and clearly crosscut altered quartz monzonite, and others are altered and cut unaltered quartz monzonite suggesting there may be several different ages of intrusions with similar mineralogy. 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. Zircon rims from an altered and mineralized sample of alaskite from Hangar Flats Deposit drill core (MGI-10-20, 240 - 243 ft) were dated by the IGS and produced a U-Pb age of 87.9 Ma ± 4.9 Ma (Lewis, et al., 2014). Zircon tips from another drill core sample from the Yellow Pine Deposit (MGI-12-306 at 550 ft) were analyzed by ID-TIMS and produced a U-Pb age of 83.6 Ma ± 0.1 Ma (Gillerman, et al., 2014).

Hydrothermal alteration and the character of sulfide mineralization is the same as that described for the quartz monzonite. Sulfide mineralization occurs as veinlets, veins, stockworks, fissure filling, fault breccias, and massive sulfide veins or lenses. The biotite and magnetite are replaced by sulfides and/or pyrite as disseminations. Xenoliths of both unaltered and altered quartz monzonite have been observed within the alaskite dikes.

Photograph 7.2: Example of Alaskite in HQ Core

Pegmatite

Pegmatite dikes are coarsely crystalline consisting of large euhedral grains of interlocking potassium feldspar and quartz (Photograph 7.3). 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.

Sulfide mineralization locally occurs as veinlets and up to several inches wide massive sulfide veins cutting through the pegmatite dikes.

Photograph 7.3: Example of Pegmatite in HQ Core

Biotite Granite

Biotite granite is exposed in several areas in the District and a large northeast-trending body is exposed and cut 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 (Photograph 7.4). Muscovite is present but in smaller quantities than biotite. The biotite granite crosscuts both the quartz monzonite and the metasedimentary sequence. Recent preliminary U-Pb LA-ICPMS isotopic dating by the IGS on zircons from outcrops of the Stibnite Stock in the Stibnite Pit produced an age of 84.9 Ma ± 2.0 Ma; a more precise age was recently reported from drill core from the Stibnite Stock in hole MGI-10-37 at 50ft., producing a concordant age of 85.7 ± 0.1 Ma with ID-TIMS methods (Gillerman, et al., 2014). Clasts of the biotite granite occur in mineralized breccias in the West End Deposit suggesting mineralization at least locally post-dates the stock and is consistent with recently reported $^{40}Ar/^{39}Ar$ isotopic dating of potassium feldspar selvages on quartz veins cutting the Stibnite Stock; the feldspar was dated at 50 Ma ± 0.4 Ma (Gillerman, et al., 2014).

Photograph 7.4: Example of the Biotite Granite in HQ Core with Local Fe Oxide

Granite

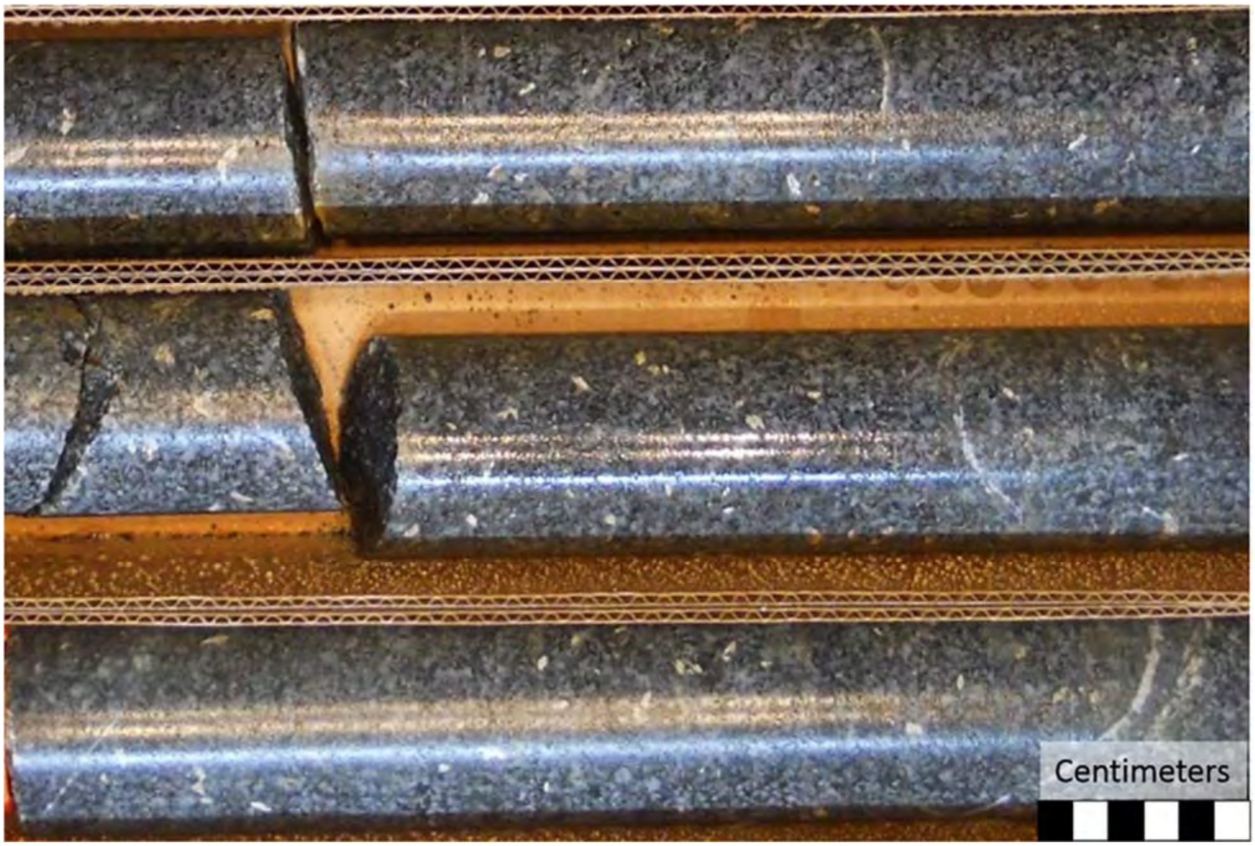
The granite is phaneritic, fine- to medium-grained, equigranular and typically light gray to white (Photograph 7.5). Principal components include feldspar, quartz, and fine-grained mica with accessories of magnetite, hematite, garnet, and sulfides (pyrite, arsenopyrite, and stibnite). Contacts with quartz monzonite are often gradational, which distinguishes the phyllosilicate-poor granites from the alaskites. No reliable isotopic dates have yet been determined for the granites. 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. It likely represents a stock-like body based on three-dimensional interpretations of drill.

Photograph 7.5: Example of the Granite in HQ Core

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 (Photograph 7.6). Diorite clasts are observed as inclusions within quartz monzonite and occasionally crosscutting the quartz monzonite as well as the metasediments suggesting at least several different ages for the intrusions with dioritic composition. 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.

Photograph 7.6: Example of the Diorite in HQ Core



Rhyolite

Several rhyolite dikes are found within the district and are associated with the MCFZ. In 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 in color when fresh, and weather to a distinctive green-brown mottled color due to weathering of magnetite and or sulfides forming iron-oxide stains (Photograph 7.7). 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.

Photograph 7.7: Example of the Rhyolite in HQ Core

Latite and Trachyte Porphyries

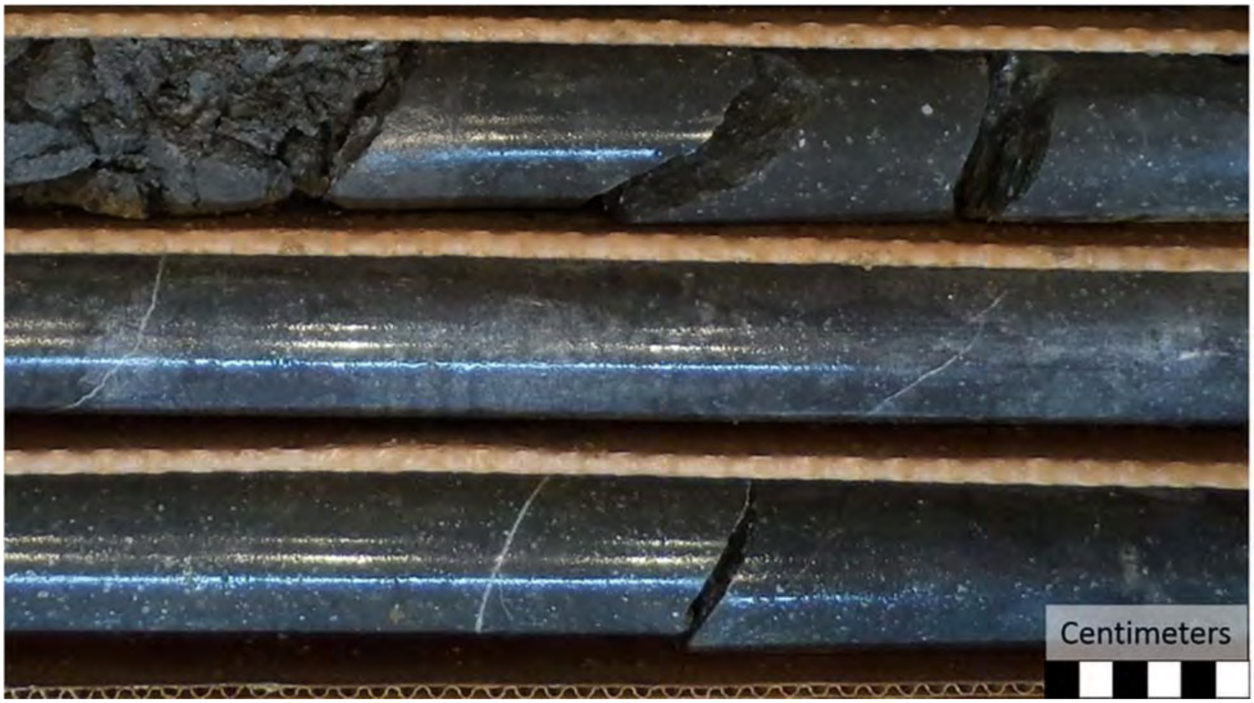
Porphyritic dikes of variable composition, but typically latite and trachyte, are common in faults throughout the district and occur as small plugs and sills in the eastern part of the Project area. The Pistol Creek Dike Swarm, located just southeast of the District and the Smith Creek Dike Swarm in 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 in color when fresh and weather to an olive green to orange-gray color and often make a sticky, clay-rich soil likely due to alteration of devitrified glasses (Photograph 7.8). 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 in drill hole MGI-13-383 sampled by the IGS at 285 ft 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.

Photograph 7.8: Example of Latite in NQ Core

<u>Diabase</u>

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 crosscutting relationships, the dikes are likely Eocene or younger. They typically are brecciated and heavily fractured when they occur within structures. They are typically aphanitic to very finely porphyritic in texture, medium to dark green in color when fresh, containing 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 (Photograph 7.9). 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.

Photograph 7.9: Example of Diabase in NQ Core

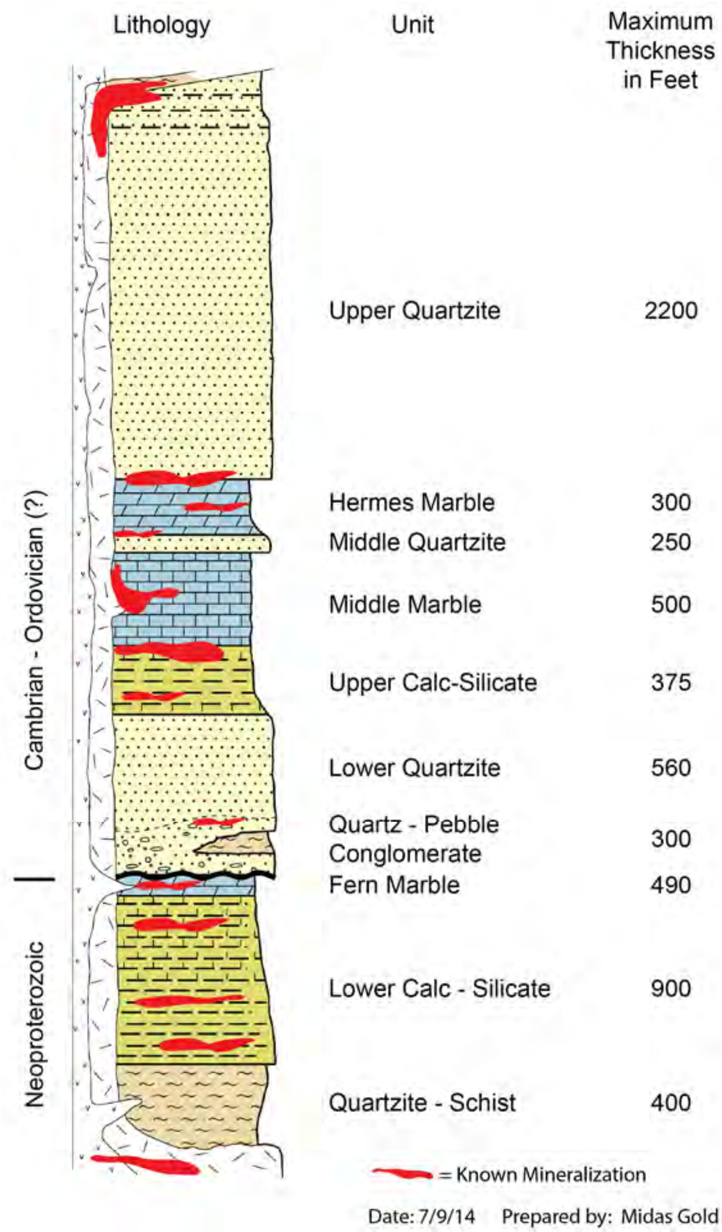


Metasedimentary Rocks

Early workers believed that the rocks of the roof pendant were Proterozoic in age, partly because of their proximity to the Belt sedimentary basin, however, recent work has determined they 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 Mezo- and Neo-Proterozoic (Lewis, et al., 2014).

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 7.3). 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.

Figure 7.3: Stibnite Roof Pendant Stratigraphy



Source: modified from Smitherman, 1985

Quartzite-Schist

This unit is up to 460 ft thick and is apparently the oldest unit exposed in the immediate Project area. Exposures are confined to two northwest trending belts: one along the northeast roof pendant border and one extending through the center of the roof pendant. The lower contact of the northeast belt is with the Idaho Batholith and a major fault forms the lower contact of the central belt. Schistosity is moderately developed with 4-inch- to 4-foot-thick interbeds of quartzite and schist forming distinct compositional banding likely reflecting original lithologic bedding (Photograph 7.10). 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. The aluminous quartz-mica schist consists of quartz-muscovite-biotite ± andalusite + sillimanite + chlorite. The quartz biotite schist is 80% fine-grained quartz, 10% biotite grains (in biotite rich layers) which define a foliation, and 2% to 3% almandine garnet porphyroblasts. The micaceous quartzite contains over 90% quartz and 5% to 10% muscovite, which has developed a weak schistosity. Traces of biotite, sphene, zircon, tourmaline, and opaque minerals occur as accessories. 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).

Photograph 7.10: Example of the Quartzite Schist in HQ Core

Lower Calc-Silicate

This unit is 165 ft to 900 ft thick, consisting 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 (Photograph 7.11). The dark layers are composed of fine oligoclase, microcline, and quartz. Xenoblastic epidote constitutes 20% to 90% of the calc-silicate layers, with minor hornblende, actinolite, and scapolite. The calcareous calc-silicate rock contains interlayers of calcite marble and calc-silicate rocks. Calc-silicate minerals include xenoblastic diopside, pale green tremolite and actinolite, and minor scapolite. Epidote occurs as very fine grains between the calcite and quartz rich layers and as coarse grains with tremolite and actinolite intergrown with pyrite. Accessory minerals include phlogopite, rare sphene, and allanite. Locally the rocks have been altered to a coarse-grained skarn assemblage of garnet-epidote-diopside, calcite, pyrite, and iron oxide.

Photograph 7.11: Example of the Lower Calc-Silicate in HQ Core

Fern Marble

The Fern marble overlies the lower calc-silicate and reaches a maximum thickness of about 500 ft. Fresh marble is light gray to blue gray and weathers to a light yellow to white leaving sucrosic-textured outcrops and poorly developed sandy soils (Photograph 7.12). The marble consists of coarse dolomite grains, rare quartz grains, and traces of brown amorphous material that may reflect the former presence of carbon residues. Green gray calc-silicate marble is locally common within 500 ft of the batholith contact. One specimen from the West End Pit is composed of 60% green diopside, 40% colorless to green tremolite/ actinolite, and rare phlogopite, forsterite, and dolomite or calcite.

Photograph 7.12: Example of the Fern Marble in HQ Core with local Fe Oxide

Quartz-Pebble Conglomerate

The quartz-pebble conglomerate is a coarse-grained, pebbly quartzite unit, which contains lenses of pebble conglomerates and bodies of quartz-mica schist (Photograph 7.13). The contact with the Fern marble is well exposed and likely represents an unconformity. The quartz pebbles are coarse, irregular to polygonal grains with flattened quartz grains and muscovite as the matrix. Small schist lenses occur locally and consist of quartz-muscovite, biotite, sillimanite, and andalusite. The unit is thickest and best exposed in the area near the Fern Mine and thins and appears to pinch out towards the West End Deposit. The unit clearly crosscuts the Fern marble along an unconformity surface well exposed in the Fern Mine area on the east side of the District. Detrital zircon dated using Laser Ablation Inductively Coupled Plasma Mass Spectrometry (LA-ICP-MS) 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).

Photograph 7.13: Example of the Quartz-Pebble Conglomerate in HQ Core with Fe Oxide

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 (Photograph 7.14). 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 to very coarse-grained quartz. Muscovite grains make up to 5% to 10%, quartz is up to 85%, and andalusite may be 2%.

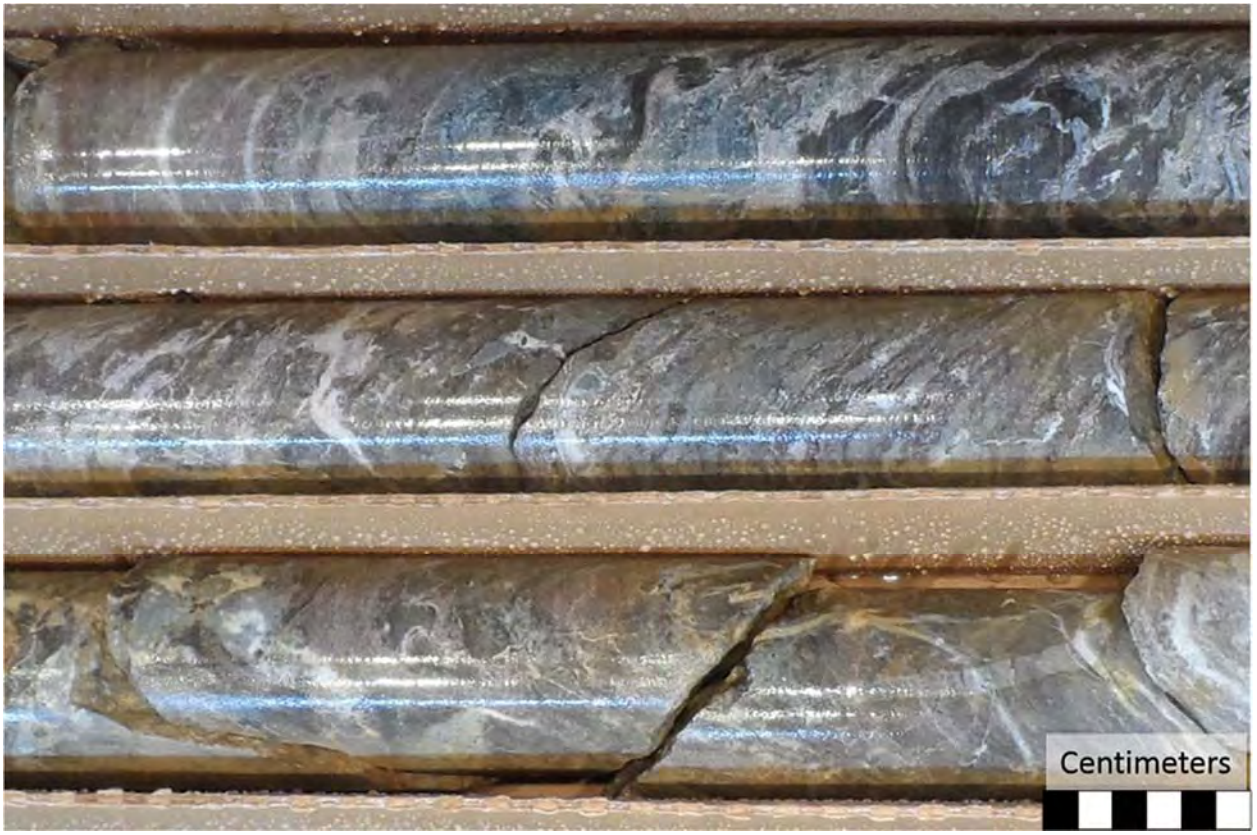
Photograph 7.14: Example of the Lower Quartzite in HQ Core

Upper Calc-Silicate

The upper calc-silicate consists of biotite, plagioclase, calc-silicate rock (Photograph 7.15). 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 sub-units.

The lower plagioclase calc-silicate rock is dense, laminated, dark gray, and weathers to gray or red-brown. Thin sections show 70% plagioclase, 10% diopside, 10% tremolite/actinolite, and 5% fine-grained biotite. The middle plagioclase-biotite rock is similar to the lower unit with the addition of plagioclase-biotite layers. The upper unit is a massive calcareous, plagioclase calc-silicate rock with 35% labradorite, 30% scapolite, 30% diopside, and minor calcite. The uppermost unit is a laminated calc-silicate and calcitic marble rock. This unit varies up to 195 ft in thickness on the northern limb of the syncline. The calc-silicate layers form thin (0.4 inch) ribs above the easily weathered marble layers. Minor interbedding folds are common. The calc-silicate layers are about 50% scapolite and 50% fine diopside grains. The marble layers are approximately 0.6 inch thick and contain over 95% calcite with minor scapolite and diopside.

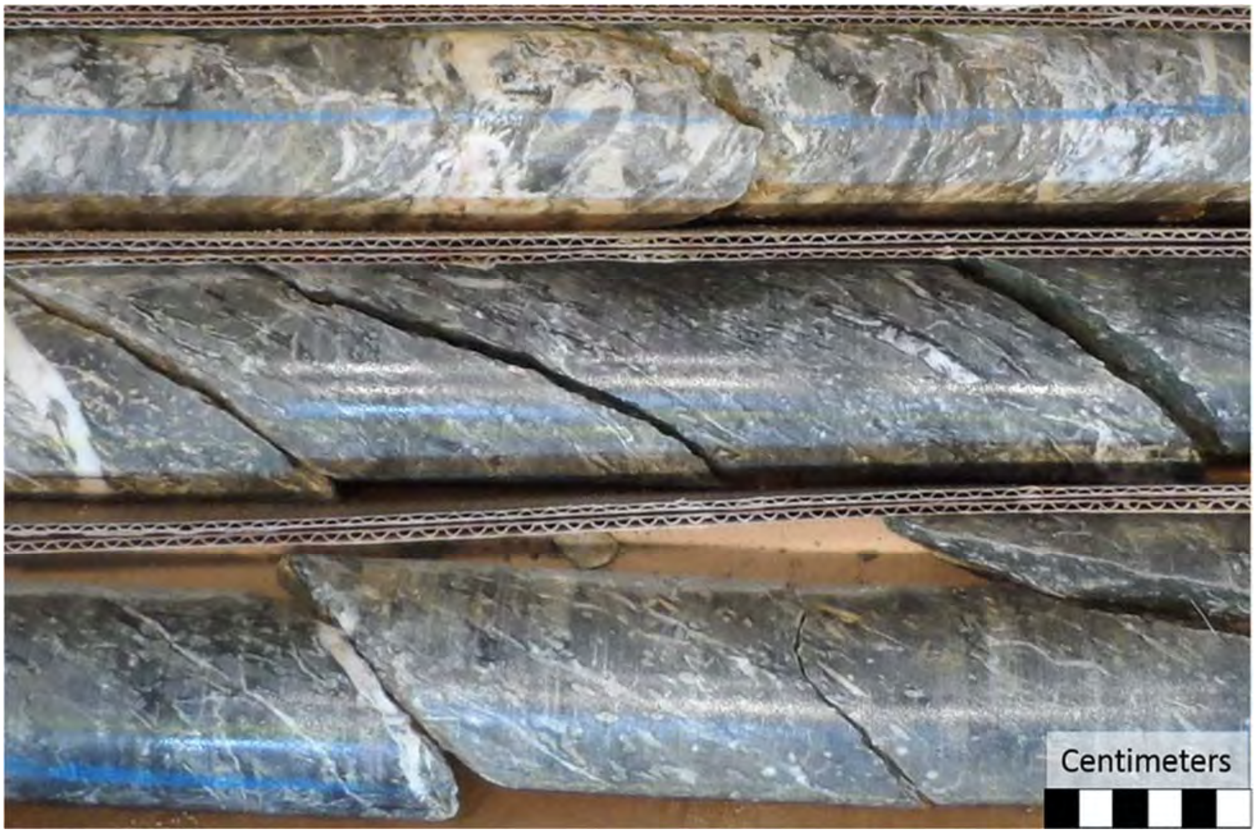
Photograph 7.15: Example of the Upper Calc-Silicate in HQ Core



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 (Photograph 7.16). The rock is 80% to 99% calcite with minor biotite, diopside, and graphite.

Photograph 7.16: Example of the Middle Marble in HQ Core



Middle Quartzite

A 30 ft- to 250 ft-thick quartzite unit 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. The rock is locally very porous due to hydrothermal leaching, and sparse alunite grains probably formed during hydrothermal activity. Carbonate cement is locally present, as well as rare biotite schist bodies near the lower contact. Stratigraphic relationships are important for identifying this unit, as the texture can often be similar to the Lower Quartzite, which can be seen in Photograph 7.14.

Hermes Marble

The Middle Quartzite is overlain by 195 ft to 295 ft of dolomite marble. The lower 195 ft consist of a light gray massive dolomite marble (Photograph 7.17). This contains 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. Throughout its outcrop area and in underground workings and drill holes within the Cinnabar Mine complex east of the District, the Hermes is often silicified and converted to maroon to grey-red jasperoids.

Photograph 7.17: Example of the Hermes Marble in HQ Core

Upper Quartzite

Overlying the Hermes Marble is a quartzite unit with minor siltite. Thickness varies from 1400 ft to 2200 ft. The unit forms large, bold outcrops of cliffs and ridges. In thin section 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% biotite and minor muscovite. Stratigraphic relationships are important for identifying this unit, as the texture can often be similar to the Lower Quartzite, which can be seen in Photograph 7.14. 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

Regional- and district-scale structural trends are broadly parallel to the trace of the relict rifted western edge of the continent, suggesting it was a fundamental control on the geometry of the miogeocline, subsequent contractional orogenic events and development of the suture zone. Lund, et al. (2003) suggested that the rifted margin contained two segments, interpreting the variability between pendant stratigraphy as reflecting the effects of northwest-striking asymmetric extensional segments divided by northeast-striking transform and transfer segments. These earlier large scale crustal features controlled the provenance and spatial distribution of sedimentary lithologies and also likely played a role in where subsequent intrusive and volcanic activity developed with pre-existing zones of weakness providing conduits for ascending magmas and circulation of hydrothermal fluids (Georgis, Tikoff, Kelso and Markley, 2004).

Several major regional scale structural features cut through the Project area along with smaller subsidiary structures. Historic surface and underground mining records, field mapping, data from oriented drill core and geophysical surveys indicate three dominant trends within the district that are similar to those found in more well studied areas to the west in the Long Valley area. Structural elements show a wide variety of characteristics including thrust, low angle normal, high angle reverse and normal, and strike slip movement.

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**); 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 and are recessive weathering and often are found under or along the flanks of glacially carved valleys. Interpretation of kinematic indicators in underground and surface exposures and oriented drill core suggest these faults had early high angle reverse movement followed by right lateral displacement, but due to structural complexity variations in sense and amount of relative displacement are common. These north-south faults are often associated with east-west and northeast-southwest trending splays and dilatant structures and locally appear to truncate the northeast-trending features, but due to lack of exposure the relationships are unclear.

Large, northwest-southeast trending geophysical features occur cutting through and within the metasedimentary rocks of the roof pendant and continue to the northwest across batholith rocks and across 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. A distinct "break" in rather continuous mineralization in the main Yellow Pine deposit area and the Homestake area may be related to one of these northwest structural features.

The MCFZ is the dominant structure associated with the Hangar Flats Deposit. A jog in the fault occurs adjacent to the main deposit and kinematic indicators (from historic operators with access to the underground workings and from Midas Gold geotechnical drilling and oriented core studies) show an early reverse sense of movement followed by right lateral strike slip displacement. This jog likely created a dilatant zone allowing hydrothermal fluids to pervasively alter and mineralize the area near the bend. A pronounced and pervasive set of northeast to east-northeast striking, shallow northwest dipping joints and an alaskite dike swarm occurring adjacent to the MCFZ are likely reflecting the presence of dilatant splays generated during movement along the MCFZ.

Mineralization in the Yellow Pine Deposit is also structurally controlled and localized by the MCFZ, a generally north to northeast striking, steeply west-northwest dipping, complex fault zone; and north striking gently west dipping conjugate splay or cross structures associated with the MCFZ. The main body of mineralization in the Yellow Pine pit area is associated with a dilatant bend in the MCFZ, where its strike changes from a linear north-south trend to a more north-easterly trend. Early reverse movement and later right lateral strike slip movement along this fault created a large area of fracturing and open space allowing hydrothermal fluids to pervasively alter and mineralize the rocks within the area of the bend. Historic operators mapped several large faults here and they are discussed in the mineral resources section of this Report.

The West End Fault Zone (**WEFZ**) is the predominant structure associated with the West End Deposit. The main fault zone consists of three high angle faults, all striking along an azimuth of approximately 030° and dipping 50° to 75° to the southeast. The width of the fault zone as measured between the footwall and the hanging wall faults varies from 100 ft to 295 ft. Several subsidiary structures exist on the northern and southern ends of the deposit both west and east of the primary WEFZ, but are poorly defined at present and are not well exposed. Several east-northeast striking structures appear to splay off the primary structural zone and include the Splay Fault, Stibnite Fault, and Northeast Extension Fault structures. The subsidiary structures have strikes ranging from azimuth 060° to 090° and dip steeply north and south. Based on the relative offsets of the metasediments, and kinematic indicators, the WEFZ has experienced right lateral and probably normal (down to east) offset.

7.2.3 Alteration

Intrusive Rocks

Mineralization in intrusive rocks from the Yellow Pine and Hangar Flats deposit were described by White (1940) and Lewis (1984). Lewis subdivided alteration of granitic rocks as an early sodium metasomatism followed by potassium metasomatism and this was subsequently followed by multistage potassium alteration, which he described as hydrothermal in origin. Lewis's work outlined four phases of hydrothermal potassium alteration following the metasomatism. Alteration is not typically texture destructive at hand specimen scale and many of the primary textural relations of the typical quartz monzonite host rock still appear to be evident. However, microscopic examination indicates this early alteration is highly pervasive.

The earliest gold bearing alteration phase is typified by hydrothermal replacement of plagioclase and microcline by adularia, quartz flooding, and alteration of biotite to sericite. The adularia replacement of plagioclase is pervasive and is associated with distinct geochemical changes, including increased potassium content associated with sodium depletion, and often calcium and magnesium depletion as well, presumably due to the destruction of sodium-bearing feldspar phases and replacement with potassium-rich feldspars and replacement of biotite with sericite. This phase is associated with the introduction of very fine-grained, disseminated, euhedral hydrothermal pyrite. Arsenopyrite, pyrrhotite, or other sulfides are rare. This early pyrite is disseminated throughout the matrix of the quartz monzonite, but is concentrated in biotite and to a lesser extent in the feldspar phases.

The second gold bearing alteration phase also includes adularia replacing plagioclase and microcline, addition of quartz, and sericitization of biotite, but also includes introduction of pyrite, arsenopyrite, and minor pyrrhotite. By far, pyrite is the dominant sulfide phase, making up over 90% of the sulfides, and the sulfides exhibit distinctive microcrystalline textures. The sulfides occur as elongated "blebs" of sulfides in the sericite which replaces the biotite. These sulfide "blebs" may be randomly oriented or more typically are oriented parallel to the original cleavage in the micas. The arsenopyrite is typically represented by euhedral grains surrounded by pyrite. In small irregularly shaped patches, minor amounts of pyrrhotite may occasionally be present. Characterization studies performed during metallurgical work indicate the earlier sulfides are likely higher temperature, have more arsenic and more gold and that later sulfides often have developed at the expense of the earlier sulfides and document various morphological changes over time (Martin and Palko, 2011a; Martin and Palko, 2011b; Palko, 2012). Carbonates (dolomite and calcite) were introduced and usually occur as partial replacements of adularia or plagioclase or in the sericitized groundmass.

The third phase of potassium alteration represents a period when sulfides were not precipitated in significant quantities and the majority of the alteration occurs as the coarse-grained sericite replacement of adularia.

The fourth alteration event is distinguished by open space filling and is represented by dolomite, calcite, quartz, and sericite precipitating in small cavities and along fractures and as fissure-filling veinlets with pyrite and arsenopyrite. Coarse-grained stibnite veins are commonly associated with this stage.

Metasedimentary Rocks

Secondary silica, as veins and disseminations, is the most pervasive alteration of the mineralized material. Silica has replaced and permeated the rock within and near the major fault zones; in places silica is over 90% of the rock mass. Quartz occurs in stockwork veins and veinlets, as disseminations, in coliform bands, and with reticulate textures. Vugs formed by leaching of feldspar grains or formed in extensional fractures are either partially filled with euhedral quartz + scheelite + gold and commonly with manganese and iron oxides, or they are completely filled with secondary quartz, forming a quartz-eye pattern in the rock (Cookro, 1989). As noted earlier, higher temperature quartz veins are cut by veins with distinctive lower temperature assemblages and fluid inclusions (Cookro, et al., 1987).

Plagioclase is commonly sericitized, although not necessarily in the mineralized zones. Feldspar grains are commonly argillized in the fault zones. Clay minerals that formed from the alteration of feldspar grains are commonly leached out, and the resulting void is filled with euhedral clear quartz. Altered silicates were an important source of secondary silica.

Calcite, dolomite and locally ankerite and siderite, occur as discrete grains and as coarsely crystalline carbonate in veins and cementing breccia fragments. Carbonates are sometimes dark in color and contain manganese and iron oxides and abundant opaque minerals, which include fine grains of sulfides (Cookro, et al., 1987) indicating they are likely hydrothermal in origin.

In the West End Deposit, gold concentrations occur within fractured, metasedimentary rocks. Although calc-silicates are the most favorable host rocks, all lithologies host mineralization. Within the oxidized zone, gold is tied up in iron oxides and hydroxide oxidation products of primary sulfides. Silica and potassium feldspar flooding and veining are manifestations of alteration associated with gold mineralization.

7.2.4 Mineralization

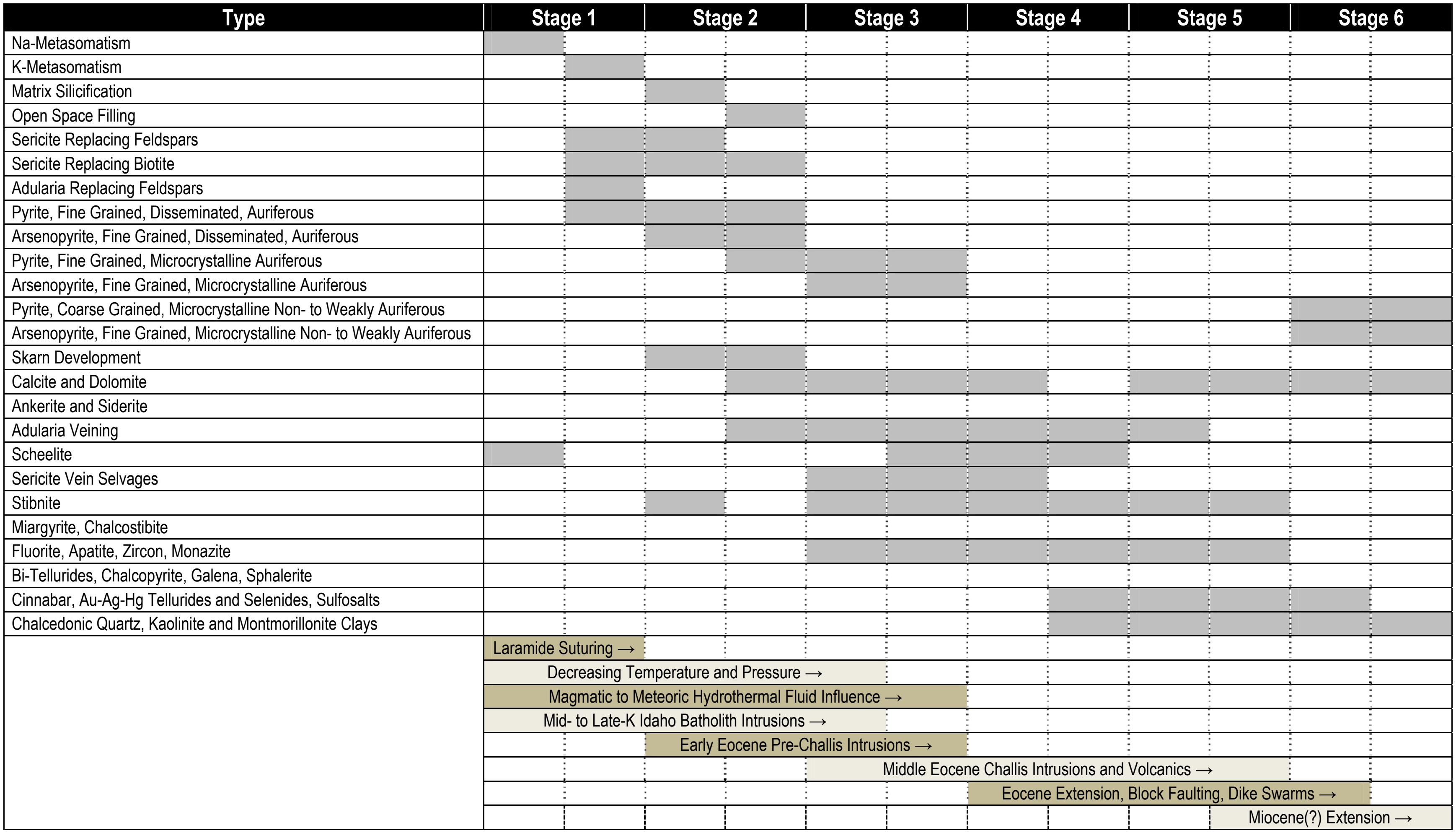
Intrusive hosted precious metals mineralization typically occurs in structurally prepared zones in association with very fine-grained disseminated arsenical pyrite (FeS_2) and to a lesser extent arsenopyrite (FeAsS). Base metal sulfides are uncommon. Mineralogical studies of sulfide morphology and mineral chemistry were completed for metallurgical process flow sheet testing using x-ray diffraction (**XRD**), dynamic secondary ion mass spectrometry (**SIMS**), QEMSCAN®, mineral liberation analyzer (**MLA**), and petrographic studies (Martin and Palko, 2011a; Martin and Palko, 2011b; Martin and Palko, 2011c). These studies, combined with past academic research (White, 1940; Cooper, 1951; Lewis, 1984; Cookro, et al., 1987) indicate that there are multiple periods of pyrite development and associated precious metals mineralization. Arsenical pyrite is the primary host for gold mineralization, and gold only rarely occurs as discrete particles and, if so, typically only in rare sub-micron size particles, but the vast majority of the gold instead occurs in solid solution within the pyrite crystal lattice. Arsenopyrite is the only other significant gold-bearing sulfide mineral in the intrusive hosted deposits. Base metals (except for arsenic, antimony, and tungsten) 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 associated with 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 [$(Pb)_{10}(Ag)_{3\text{-}8}(Sb)_{11\text{-}16}(S)_{28}$]. There is a weak, but persistent association of volumetrically small, typically <0.25%, base metal mineralization 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 typically and essentially exclusively associated with the mineral scheelite ($CaWO_4$). Observations suggest tungsten occurs late in the paragenesis, but precedes the stibnite mineralization 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) and coloradoite (HgTe) and to a lesser extent tiemannite (HgSe) and amalgam (HgAg).

Figure 7.4: Stibnite – Yellow Pine District Paragenesis



Source: modified from Lewis, 1984; Cookro, et al., 1988; Blue Coast Metallurgy, 2012

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.

The paragenesis of mineralization in the district and immediate area has been described by various workers including: Currier (1935); White (1940); Leonard and Marvin (1982); Lewis (1984); Cookro, et al. (1987). Figure 7.4 graphically outlines the primary stages of alteration and mineralization as currently interpreted. Additional studies by Midas Gold contractors and consultants and academic researchers from the IGS and USGS are ongoing.

7.3 MINERALIZED ZONES

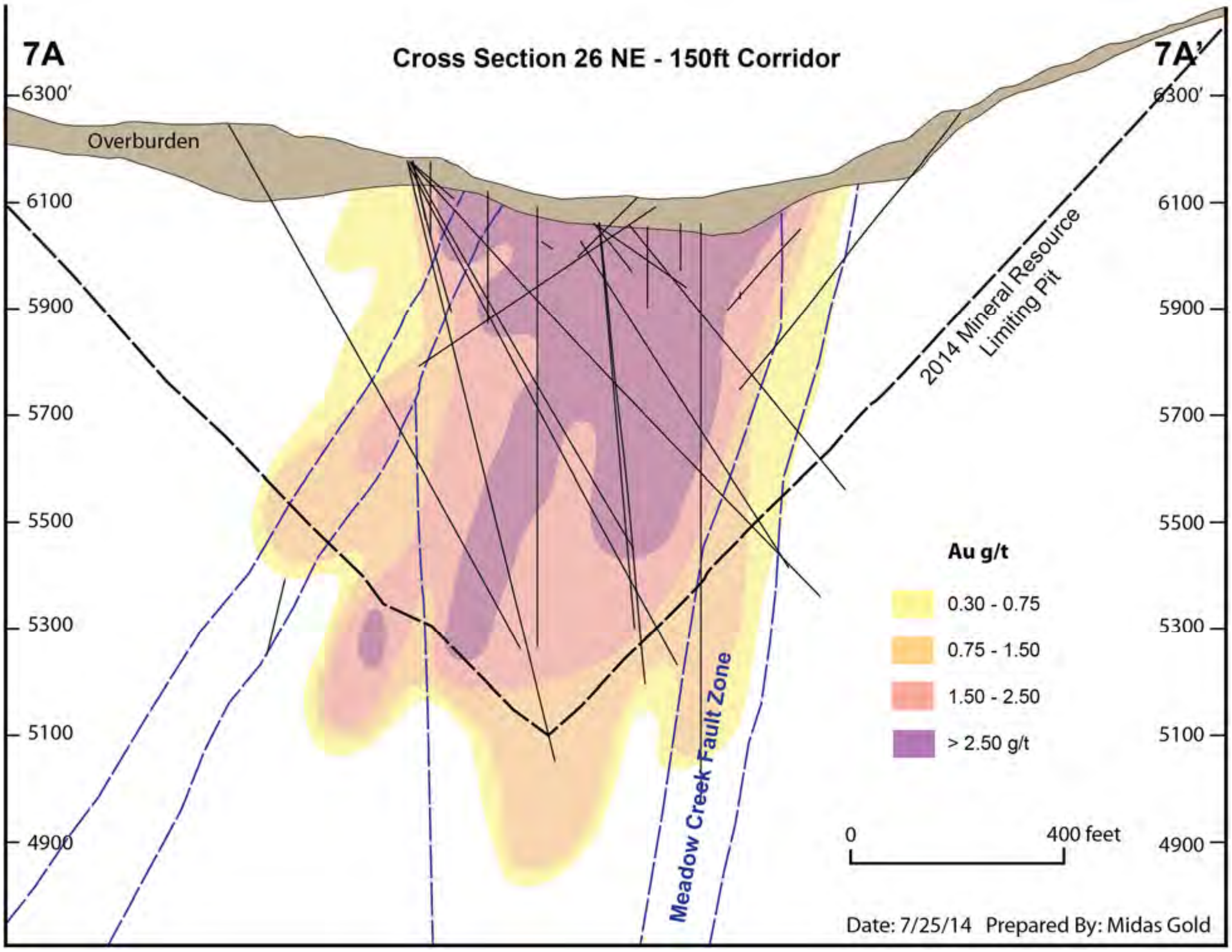
7.3.1 Yellow Pine Deposit

Mineralization of the Yellow Pine Deposit is structurally controlled and localized by the MCFZ and related structures. Mineralization styles, intensity of mineralization, widths and intensity of alteration vary relative to distance from the bend in strike of the MCFZ. Variography and stereonet plots of observed outcropping structures, mineralized features and data from modeling of oriented and un-oriented drill core, along with compilation of historic open pit and underground geologic information, has defined a series of domains outlining areas with common characteristics. Gold and antimony have different geochemical signatures, geometries, and locally used different structures during deposition. Structures and fractures open to circulating hydrothermal fluids during gold deposition were not necessarily open for antimony deposition. The deposit shows some apparent zonation with gold occurring throughout the deposit footprint, but with antimony and tungsten primarily in the central and southern portions of the deposit.

The dominant fault directions mapped underground and in the open pits by Bradley Mining Company geologists in the 1938-1952, by White in 1940-41, by Cooper 1950-1951, and also observed along the former open pit benches by Midas Gold geologists in 2012, trend north-south, northeast, and east-northeast. However, the controls for antimony mineralization show more northwesterly trends. The different geometries of antimony and gold distribution suggest different controls for mineralization – antimony is more strongly influenced by northwest fracturing and gold is more strongly influenced by northeast and east-northeast structures. White (1940) interpreted all strike-slip faulting as post-mineral whereas Cooper (1951) suggested there was significant post-mineralization movement between periods of early gold mineralization and later antimony-tungsten mineralization. Midas Gold's current interpretations on the relative timing of gold versus antimony mineralization are similar to those interpreted by Cooper (1951).

Mineralization at the south end of the Yellow Pine Deposit exhibits strong, steeply west- and east-dipping north-south oriented structural controls and occurs in a narrow 80-ft- to 165-ft-wide corridor along the footwall (east side) of the MCFZ. In the central domain of the deposit, numerous structural elements intersect and mineralization occur along east to east-northeast striking and west to west-northwest striking, north-dipping dilatant structures within a larger northeast-trending structural corridor. Both northeast and north-south striking structural elements may control mineralization along with the dilatant structures which occur at relatively high angles to the main shear zone. Mineralization in this area appears pipe-like in cross sections, but in long sections exhibits pronounced northeast and northwest plunges reflecting the interplay of the primary northeast structures and secondary splays and dilatant features which occur at high angles to the main MCFZ. The multiple structural features provided significant pathways to mineralizing hydrothermal solutions and the mineralization here is the highest grades and ranges from 165 ft to over 650 ft in true thickness and can be traced down dip for over 1,300 ft (Figure 7.5).

Figure 7.5: Yellow Pine Mineralized Zone



Note:
The Yellow Pine mineralized zone was mined by underground methods between approximately 1939 and 1948. For simplicity, the underground workings are not shown on the cross section but are quantified in Section 14 of this Report.

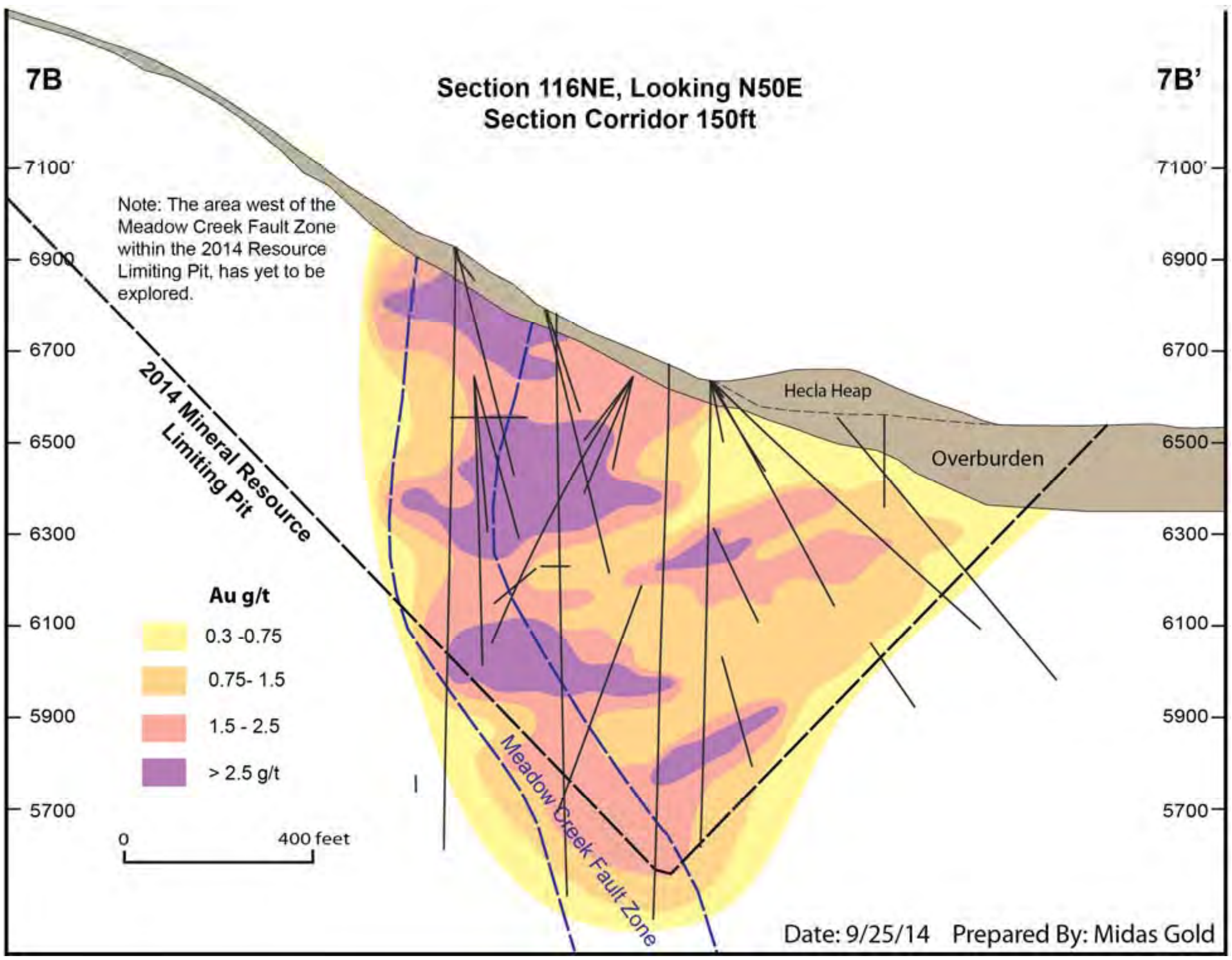
In the area of the former Homestake pit, mined by Hecla in the late 1980's and early 1990's, mineralization is more tabular and narrower than areas to the southwest (in the central domain) and is associated with multiple, north-striking, shallow west-dipping structures intersecting the main MCFZ as well as east to east-northeast striking and west to west-northwest striking, north-dipping dilatant structures.

Multiple stages of movement in the MCFZ are described in the historic literature (White, 1940; Cooper, 1951) and are evident in pit walls and in Midas Gold drill core, with the latest event marked by extensive gouge and brecciation. Various kinematic indicators suggest the latest movement involved right lateral and high angle reverse movement. The kinematics of this system have created the dilatational zones of mineralization in the Yellow Pine and Homestake pit areas. The bounding faults often contain lenses of previously mineralized material caught up in the faults during subsequent phases of deformation.

7.3.2 Hangar Flats Deposit

Mineralization in the Hangar Flats Deposit is entirely intrusive hosted, and structurally controlled and localized by the MCFZ, a generally north trending, steeply west-dipping complex fault zone with ancillary structures. 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, all the way to the rim of the Thunder Mountain Caldera. Past production and currently defined mineralized zones occur along variably north-plunging tabular to pipe-like bodies at the intersection of the main north-south structural feature and northeast to southwest and east to west trending steeply dipping conjugate structures and northeast trending, shallow northwest dipping (±30°) dilatant splays. The mineralized zones range in true thickness from 16 ft to over 330 ft, and can be traced several hundred feet down dip. They occur as stacked ellipsoidal lenses along the footwall to the main MCFZ which is a thick, 80-165 ft wide zone of clay gouge and heavily broken and brecciated ground. At Hangar Flats the mineralized zones become thinner, less continuous, and lower grade away from the main MCFZ (Figure 7.6).

Figure 7.6: Hangar Flats Mineralized Zone



Note:

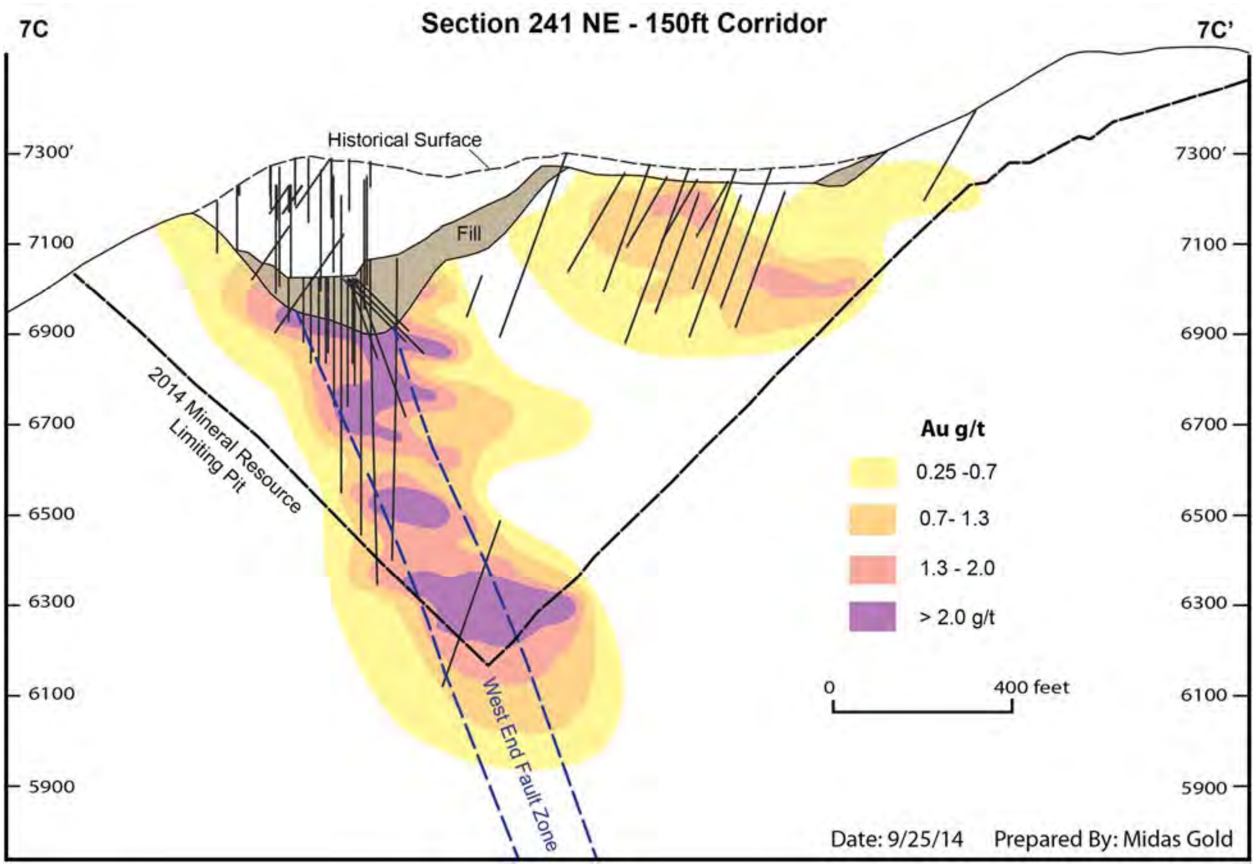
The Hangar Flats deposit was mined by underground methods between approximately 1926 and 1938. For simplicity, the underground workings are not shown on the cross section but are quantified in Section 14 of this Report.

Multiple stages of movement are described from underground mapping in the historic literature within unpublished company files and are evident in Midas Gold drill core, with the latest event marked by gouge and brecciation. Sulfide mineralized fragments have been rotated and then re-mineralized indicating several periods of movement coincident with at least some of the stages of sulfide mineralization. Various kinematic indicators suggest the latest movement along the MCFZ, at least some post-mineralization, involved right lateral and high angle reverse (i.e. west side up relative to east side) movement.

7.3.3 West End Deposit

Within the WEFZ gold mineralization occurs preferentially where the northwest-striking, northeast-dipping calc-silicate units are cut by any of the WEFZ or subsidiary faults, but all rock types host mineralization. Mineralized zones occur as stacked ellipsoidal bodies plunging along the intersection of favorable lithologic units and structural zones. 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, more significantly mineralized. Open space fill quartz veins are closely associated with the faults and are indicative of higher grade zones of mineralization. In addition to sulfide mineralization, open fractures along the WEFZ and subsidiary faults have allowed for oxide formation at depth from meteoric infiltration (Figure 7.7).

Figure 7.7: West End Mineralized Zone



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8 DEPOSIT TYPES

8.1 DEPOSIT MODELS

The origin of the wide variety of mineralization occurrences at Hangar Flats, West End, and Yellow Pine deposits is enigmatic and past workers have attributed the District metal endowment to deep-seated intrusives and associated deep high temperature and high pressure processes as well as shallower lower temperature, lower pressure hydrothermal processes within an epithermal environment. However, there is no single deposit model applicable to the deposits within the District that have been discovered to date. Within the Project area, the focus of past exploration and development for Au-Ag-Sb-W-Hg has been from both disseminated deposits extracted using conventional open pit methods and higher grade structurally controlled 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 associated with sheeted veins, stockworks, endoskarns, and complex polymictic breccias. In the metamorphosed sedimentary rocks, mineralization occurs associated 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.

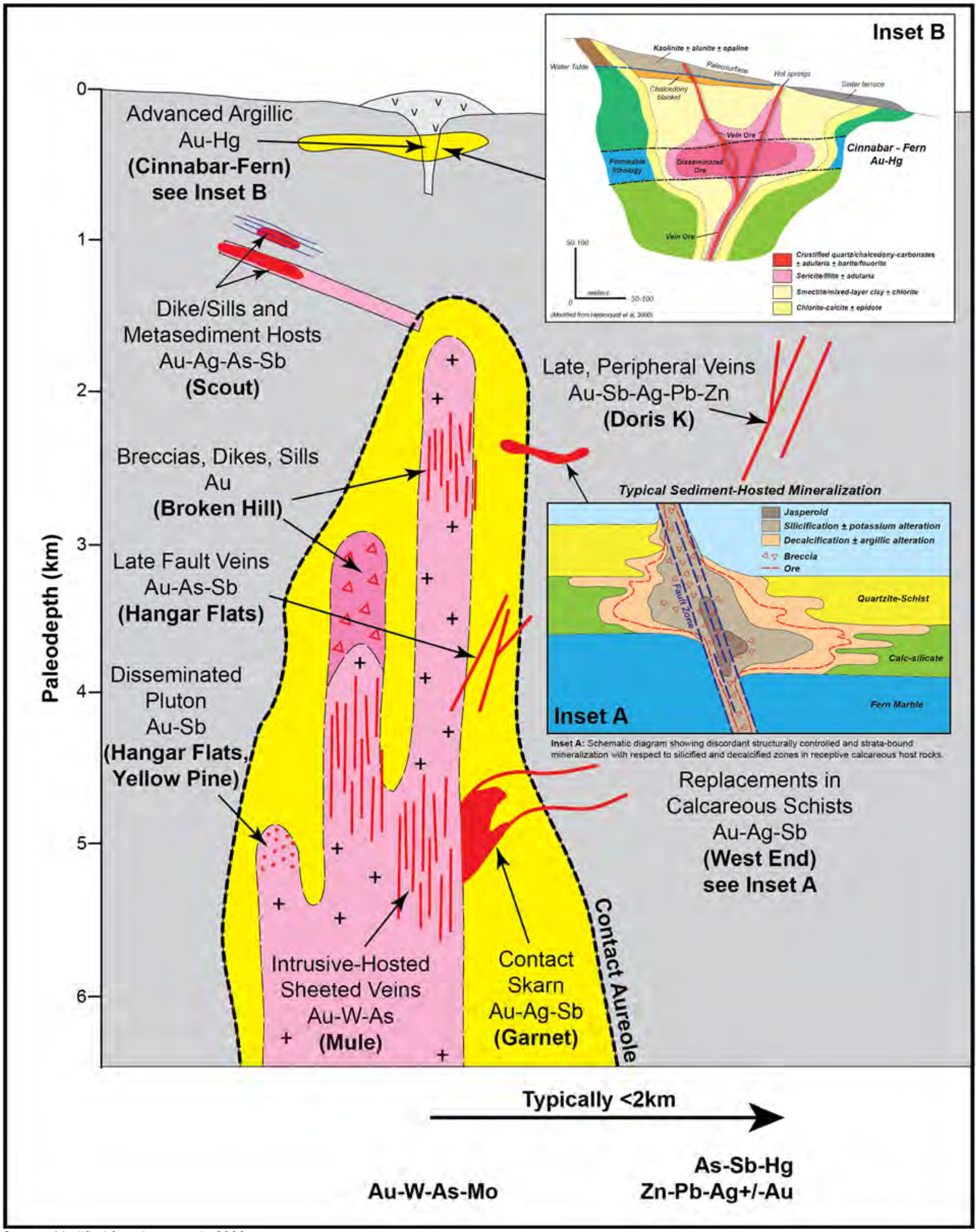
Field observations, petrographic studies, metallurgical studies, and process mineralogy studies indicate that there were likely multiple stages of mineralization, possibly separated by extended time periods. Early higher temperature, precious metal-rich mineralization with a potential magmatic fluid source was overprinted by younger, lower temperature Au and Sb-Ag mineralization; this was again overprinted by later epithermal mineralization involving meteoric water input into the hydrothermal system with a distinctly different style and geochemical signature.

The gold mineralization at the Hangar Flats and Yellow Pine deposits occurs in intrusive rocks associated with the Atlanta Lobe of the Idaho Batholith. Strong mineralization is localized along an overall north to south striking fault zone and also along northeast striking splay faults and dilatational fault jogs. Dilatant zones have generally provided conduits for movement of mineralizing hydrothermal fluids. Multiple episodes of fracturing have allowed multiple episodes of hydrothermal mineralization.

The gold mineralization at the West End Deposit occurs in metasedimentary rocks intruded by the Idaho Batholith and also within the intrusive rocks. The metasediments occur as pendants and xenoliths within the intrusive rocks. Strong mineralization is localized along a northeast striking fault zone and splay faults that strike northeast and east. Pull-apart fracturing along dilatant northeast fault jogs and splays provided conduits for movement of mineralizing hydrothermal fluids. Multiple episodes of fracturing allowed multiple episodes of hydrothermal mineralization.

A schematic of the geologic setting for the various deposits and exploration prospects is shown on Figure 8.1. Based on the nature and scale of the hydrothermal alteration systems present, the deposits are interpreted to be related to intrusive activity. This figure (modified from Lang et al., 2000) illustrates the spatial relationships of each major deposit type, the intrusion(s), and the associated hydrothermal systems.

Figure 8.1 Mineralization Model for Intrusive-Related Gold Systems in the Stibnite Mining District



Source: Modified from Lang et al., 2000

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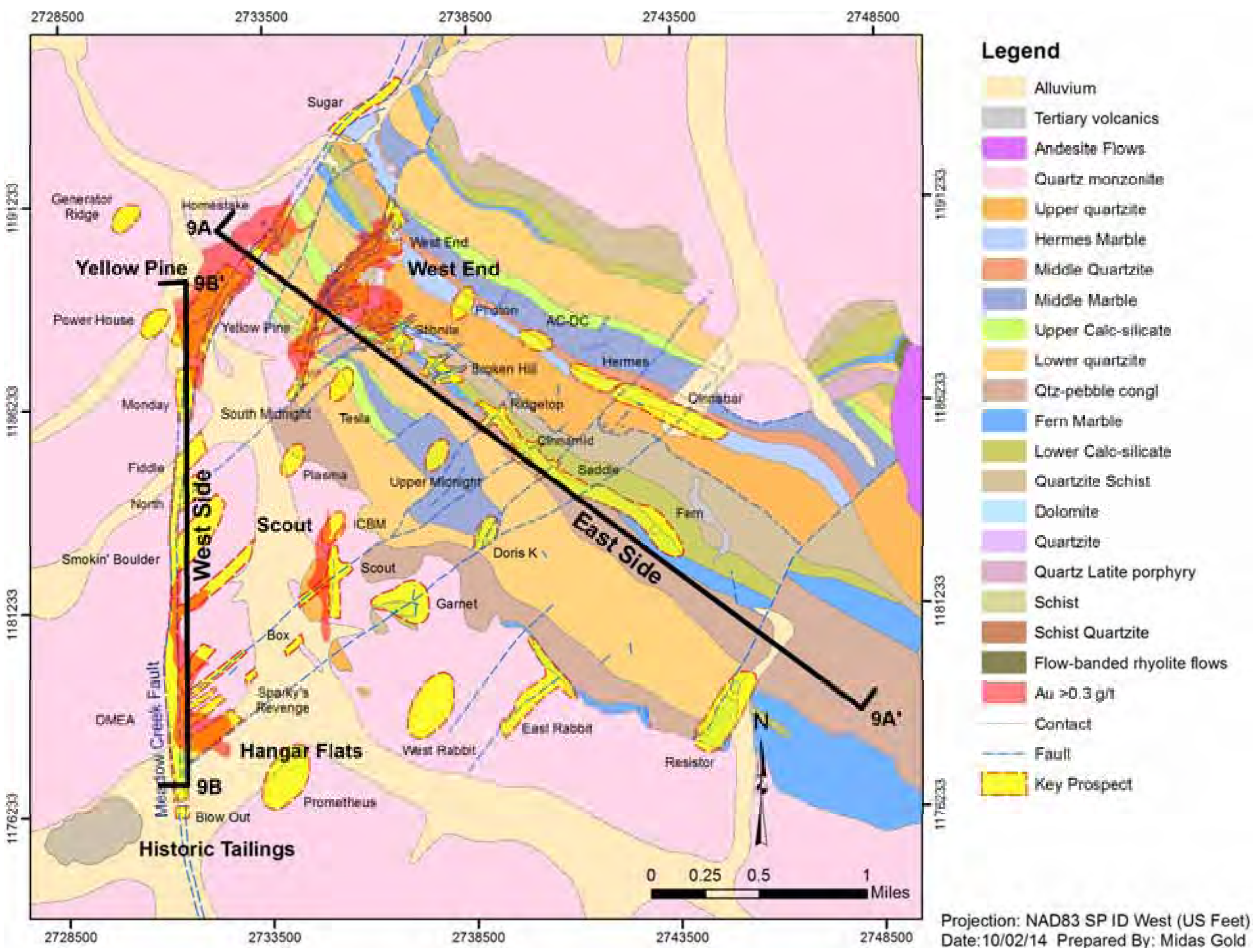
9 EXPLORATION

9.1 EXPLORATION POTENTIAL

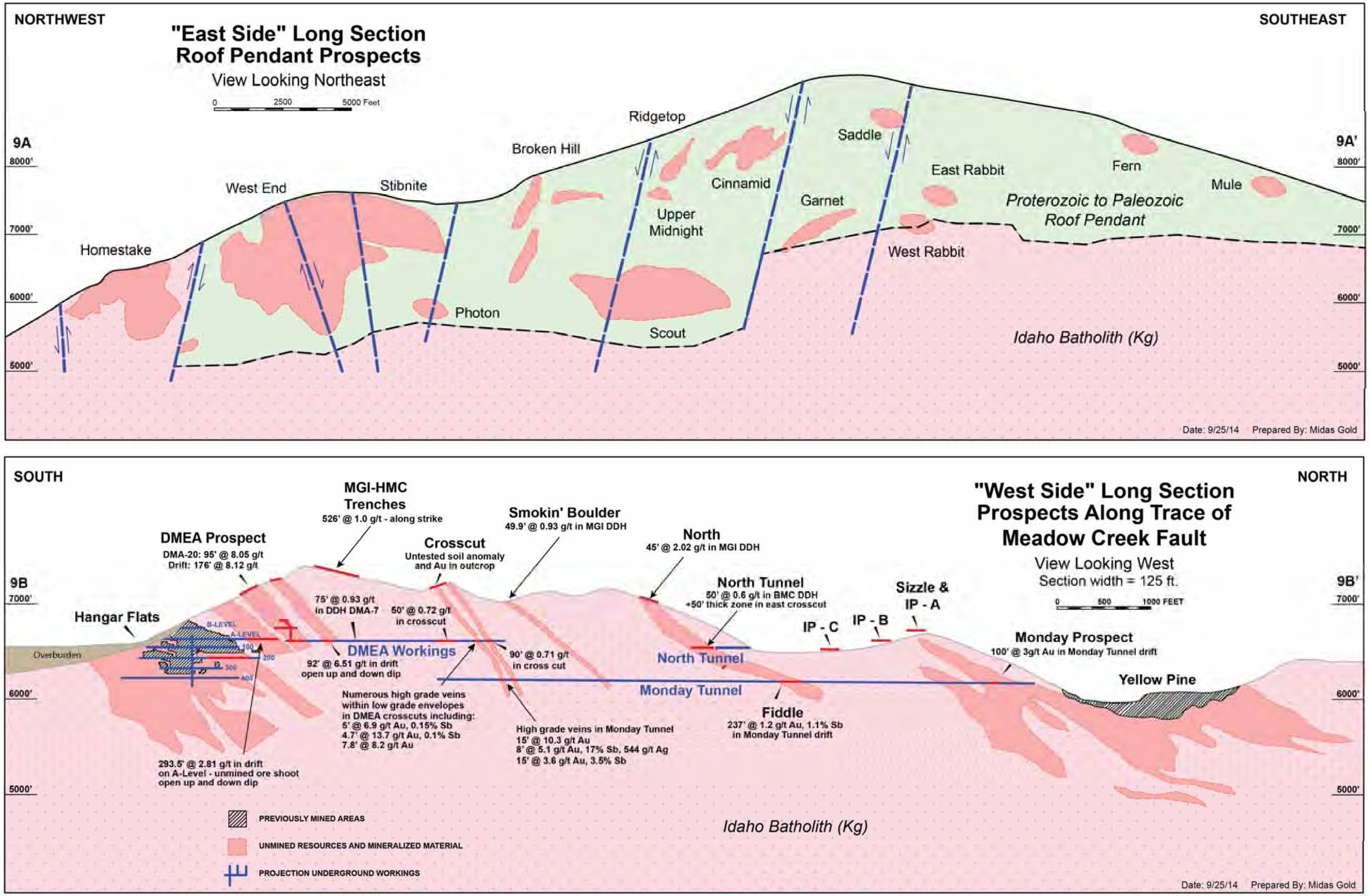
Numerous prospects have been discovered during exploration and development activities in the Districts over the past nearly 100 years using a variety of methods; some of these prospects were developed into mines while others remain undeveloped. Besides pit expansion possibilities around the main deposits, 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. Midas Gold has developed an extensive pipeline of exploration targets, which are summarized below and shown on Figure 9.1. The long sections associated with the District geology map are shown in Figure 9.2.

Figure 9.1: District Geology Map Showing Regional Prospects



The exploration targets discussed herein include more advanced prospects that have had past production and/or adequate drilling to infer good potential for high grade mineralization that might be exploited via underground mining methods (such as Scout and Garnet). In addition less advanced, but still promising underground prospects (Upper Midnight and Doris K) that have received less drilling, still have strong indications of potential high-grade mineralization. There are also more advanced prospects that have had enough drilling to infer good potential for disseminated mineralization that might be exploited via bulk tonnage, open pit mining methods. These prospects are located along the Broken Hill-Saddle trend, where past exploration focused on the search for leachable oxide ores.

Figure 9.2: East Side and West Side Long Sections through the District



Other areas with potential for new discoveries lie between the known deposits, such as the string of prospects that lie between the Yellow Pine and Hangar Flats deposits namely, Monday Tunnel, Sizzle, Fiddle, North, Smokin' Boulder and North DMEA. Midas Gold has also delineated several new prospects that have had little systematic work or were only recently discovered namely, Mule, Volt, East and West Rabbit and others.

Exploration data for the target areas discussed above and below 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 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.

9.2 GRIDS AND SURVEYS

Numerous local grids have been used on site since the 1920s and control points have been re-established where possible and practical. Original errors in earlier surveys are known to exist, but most data have been found to be reliable to within approximately 6-10 ft. Some datasets are less reliable, particularly historic soil and ground geophysical surveys. Midas Gold completed topographic and aerial photographic surveys in 2009 and 2012 for geodetic control. All Midas Gold drill locations are surveyed with survey grade instruments and typically have a level of precision of ± 10 inches. Rock and soil samples are usually surveyed with hand held GPS instruments and have been determined to usually be reliable to within ±10-20 ft.

9.3 GEOLOGIC MAPPING

The Project area has been mapped by numerous past workers and by Midas Gold staff. Mapping was typically completed by past operators along many roads, previously operating open pits and underground workings that are now reclaimed, covered and/or inaccessible, as well as in cross country traverses. Where possible and practical, map data have been field checked by Midas Gold geologists and found to be reliable for the needs of the Project. Midas Gold staff have remapped areas, where needed, to obtain additional information.

9.4 GEOCHEMICAL SAMPLING

Past operators collected and analyzed thousands of soils, rock chips, 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, lab methods and/or QA/QC. However, the geochemical data are considered reliable enough to utilize for basic exploration purposes. In areas of exploration interest, Midas Gold staff have collected samples using current industry standard protocols and certified analytical laboratories to verify past work and/or expand upon it.

9.5 GEOPHYSICS

Midas Gold contractors completed a helicopter-supported 222 line-mi aeromagnetics survey in 2009, covering 33 mi^2, followed by a more detailed 595 line-mi airborne electromagnetic (**EM**) and magnetics survey covering a larger area in 2011. The data was filtered, gridded, post-processed and integrated with geologic and geochemical data to generate and evaluate target areas. Contractors also completed induced polarization – resistivity surveys (**IP**) along 13 lines, totaling 13 line-mi, and Controlled Source Audio Magnetotellurics surveys (**CSAMT**) along 13 lines totaling 31 line-mi over the central part of the District. 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.

9.6 PETROLOGY, MINERALOGY AND RESEARCH

Extensive characterization of mineralogy has been completed as part of Midas Gold's metallurgical and mineralization characterization testing program using conventional petrographic and near infrared spectrometry methods, as well as Quantitative Evaluation of Minerals by Scanning (**QEMSCAN**) electron microscopy. The Idaho Geologic Survey has been conducting radiometric dating studies of intrusive- and metasedimentary-hosted mineralization to provide information on the approximate ages of mineralization and detrital zircon studies to evaluate age and provenance of metasedimentary rocks. This research is ongoing and has not yet been published, but preliminary data have been made available to Midas Gold.

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

All three major deposits with Mineral Resources reported herein remain open to expansion and potential is described in the following sections.

9.7.1 Yellow Pine

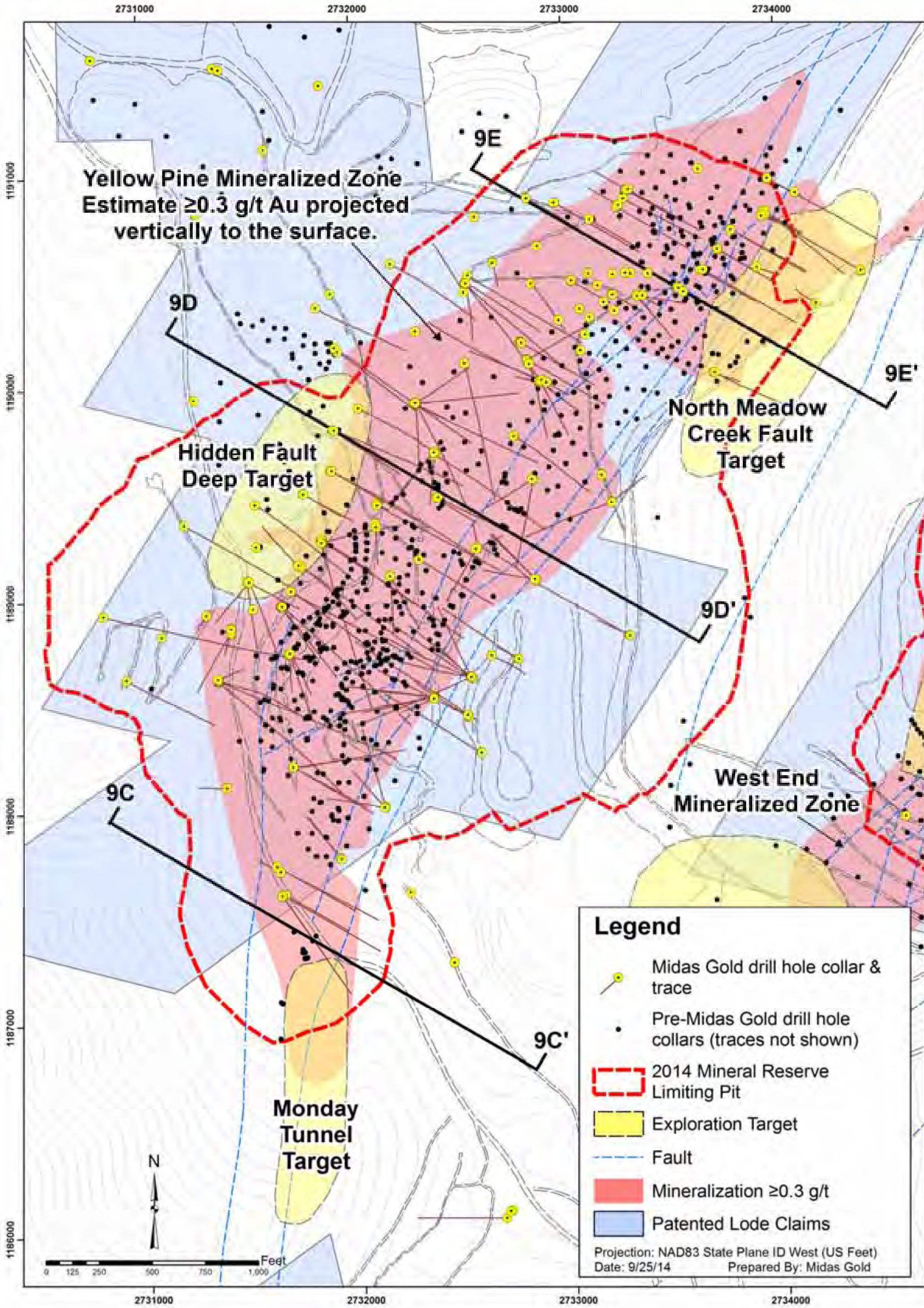
The Yellow Pine Deposit is open at depth and along strike in the north, northeast and southwest directions (Figure 9.3). Targets are defined by mineralized holes drilled by both Midas Gold and pre-Midas Gold operators. Highlights of some of the holes defining these targets are tabulated in Table 9.1 and the areas shown on Figure 9.3.

Table 9.1: Significant Drill Intercepts within the Yellow Pine Expansion Targets

Target	Operator	Drill Hole ID	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold[1,2] (g/t)
Hidden Fault Deep Zone	Midas Gold	MGI-11-187	-72.5	120	633	964	331	1.24
Hidden Fault Deep Zone	Midas Gold	MGI-12-224	-79	120	560	766	216	1.21
North Meadow Creek Fault	Bradley Mining Co.	B-043	-45	105	140	185	45	5.86
North Meadow Creek Fault	Hecla Mining Co.	89-02GT	-45	122	200	320	120	1.57
North Meadow Creek Fault	Midas Gold	MGI-11-082	-50	117	498	632	134	1.89
North Meadow Creek Fault	Midas Gold	MGI-13-307	-60	130	660	822	162	5.42
Monday Tunnel	Midas Gold	MGI-11-140	-50	145	300	395	95	2.80

Note:
(1) Selected intercepts composited with length weighted averages with cut-off grade 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.
(2) All gold and silver grades denoted g/t herein and in subsequent sections of this Report are reported in units of grams per metric tonne. Grades denoted oz/st are reported in units of troy ounces per short ton.

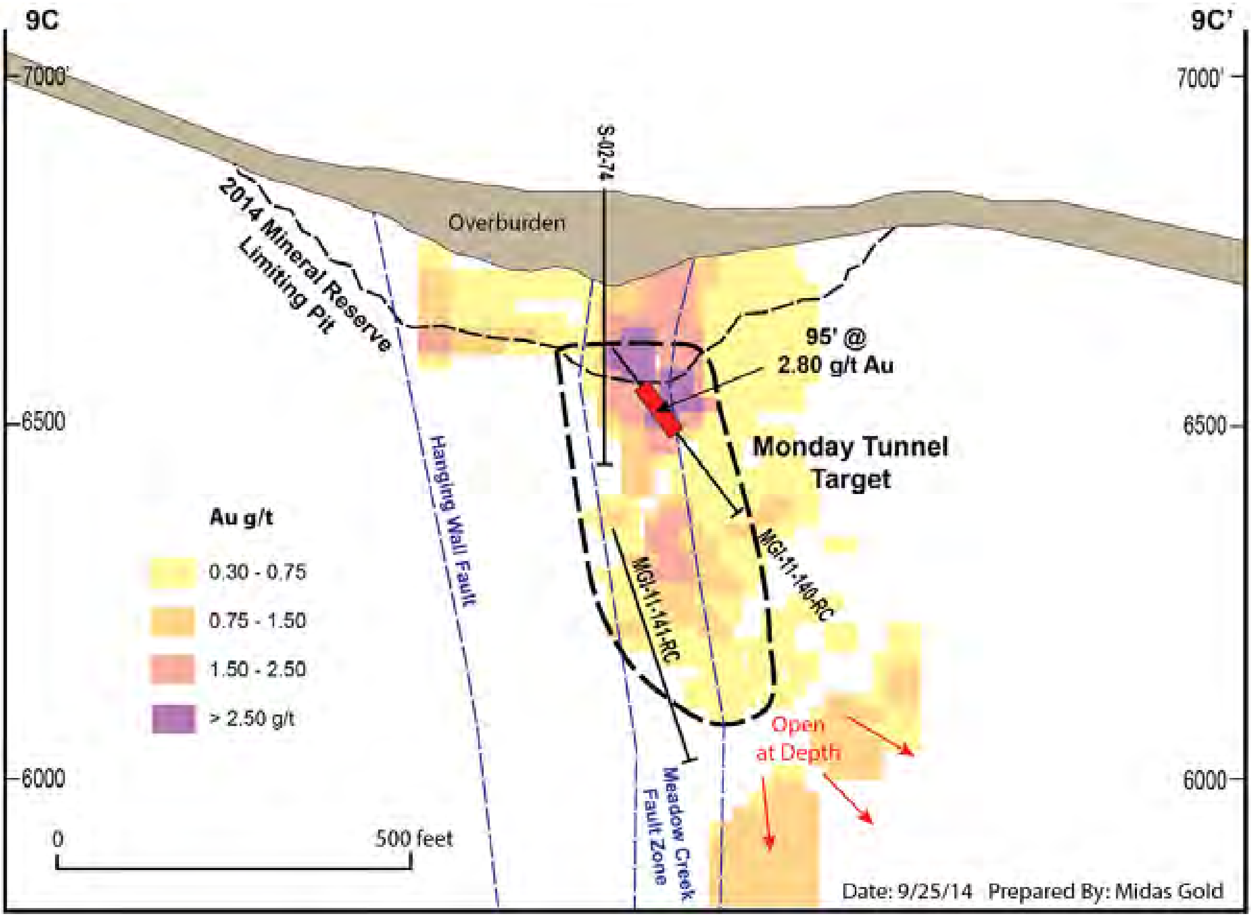
Figure 9.3: Plan Map of Yellow Pine Showing Potential Expansion Targets



Monday Tunnel Target

The continuation of the Meadow Creek Fault, south of the main Yellow Pine Deposit, has been the subject of limited drilling by Bradley Mining Company and Midas Gold. The most significant Midas Gold intercept in this area returned 95 ft averaging 2.80 g/t gold below the 2014 Mineral Reserve Limiting Pit (Figure 9.4) and the zone remains open along strike and down dip.

Figure 9.4: Cross Section of the Monday Tunnel Target with an 80 ft Corridor

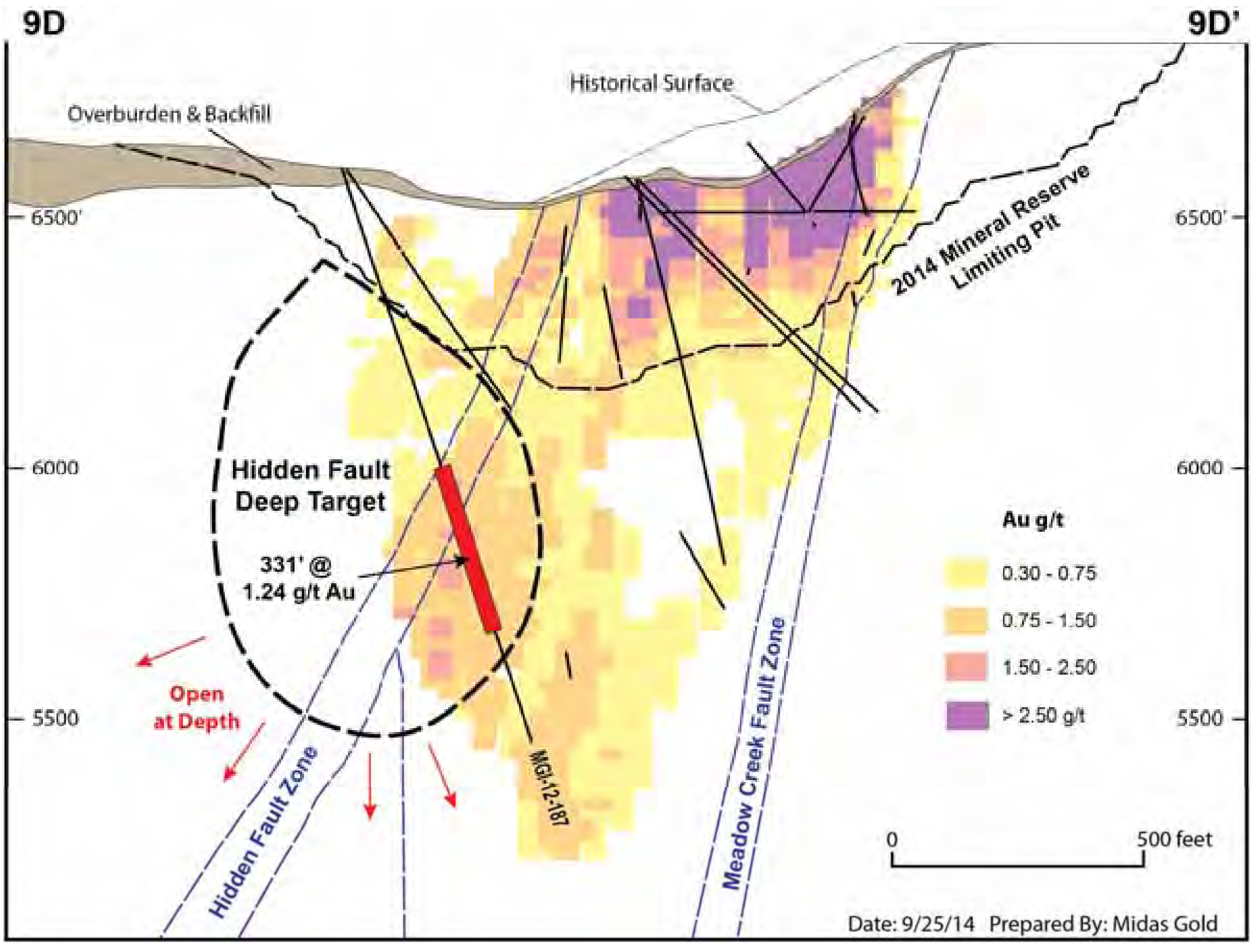


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Hidden Fault Deep Target

This area is located at the northwest edge of the Yellow Pine Deposit (Figure 9.5), along the trace of the Hidden Fault. The most significant Midas Gold intercept returned 331 ft averaging 1.24 g/t Au (Figure 9.5) and the zone remains open.

Figure 9.5: Cross Section of the Hidden Deep Fault Target with an 80 ft Corridor

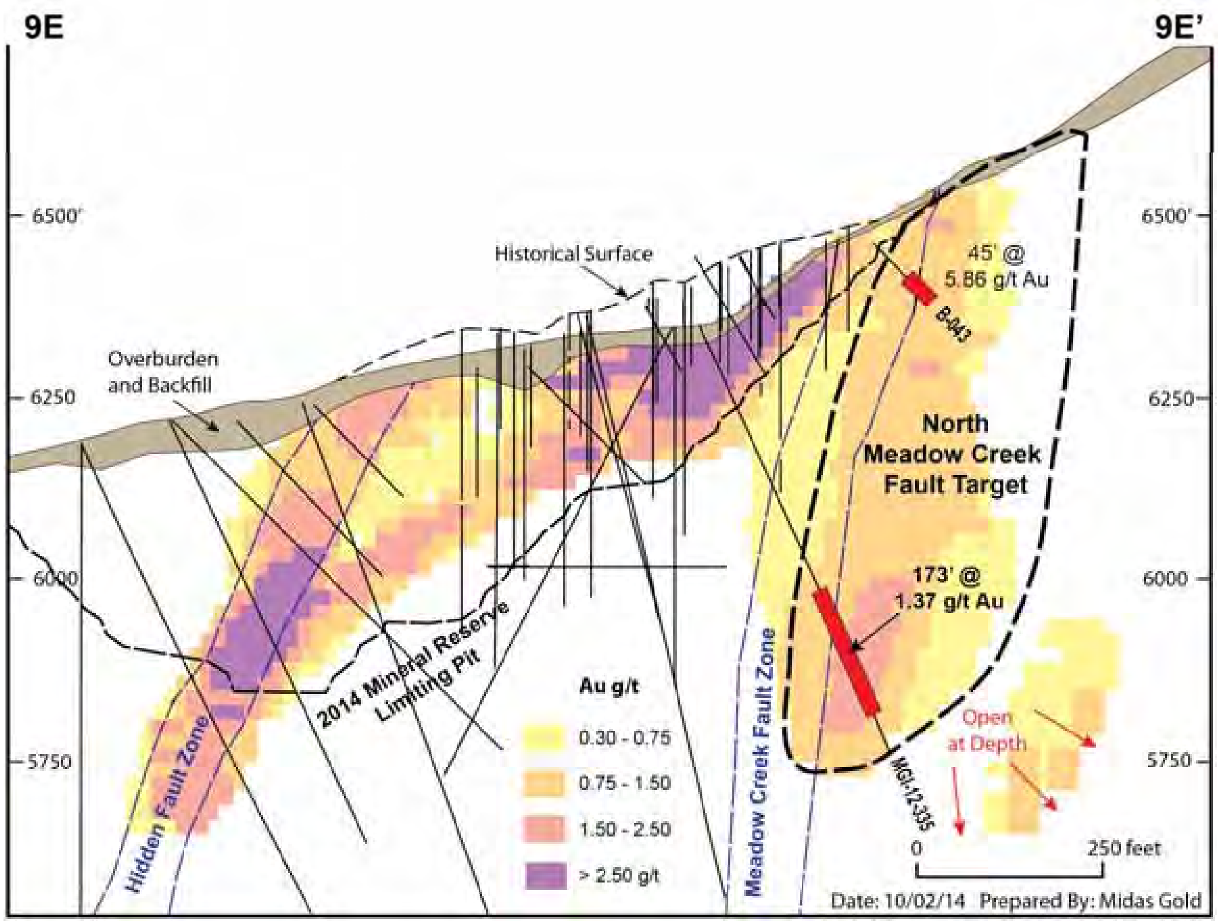


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

North Meadow Creek Fault Target

This target lies on the northeast side of the Yellow Pine Deposit and is defined by several holes drilled by pre-Midas Gold operators and Midas Gold (Figure 9.6). Significant drill intercepts include 45 ft averaging 5.86 g/t gold and 173 ft averaging 1.37 g/t gold.

Figure 9.6: Cross Section of the North Meadow Creek Fault Target with a 250 ft Corridor



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

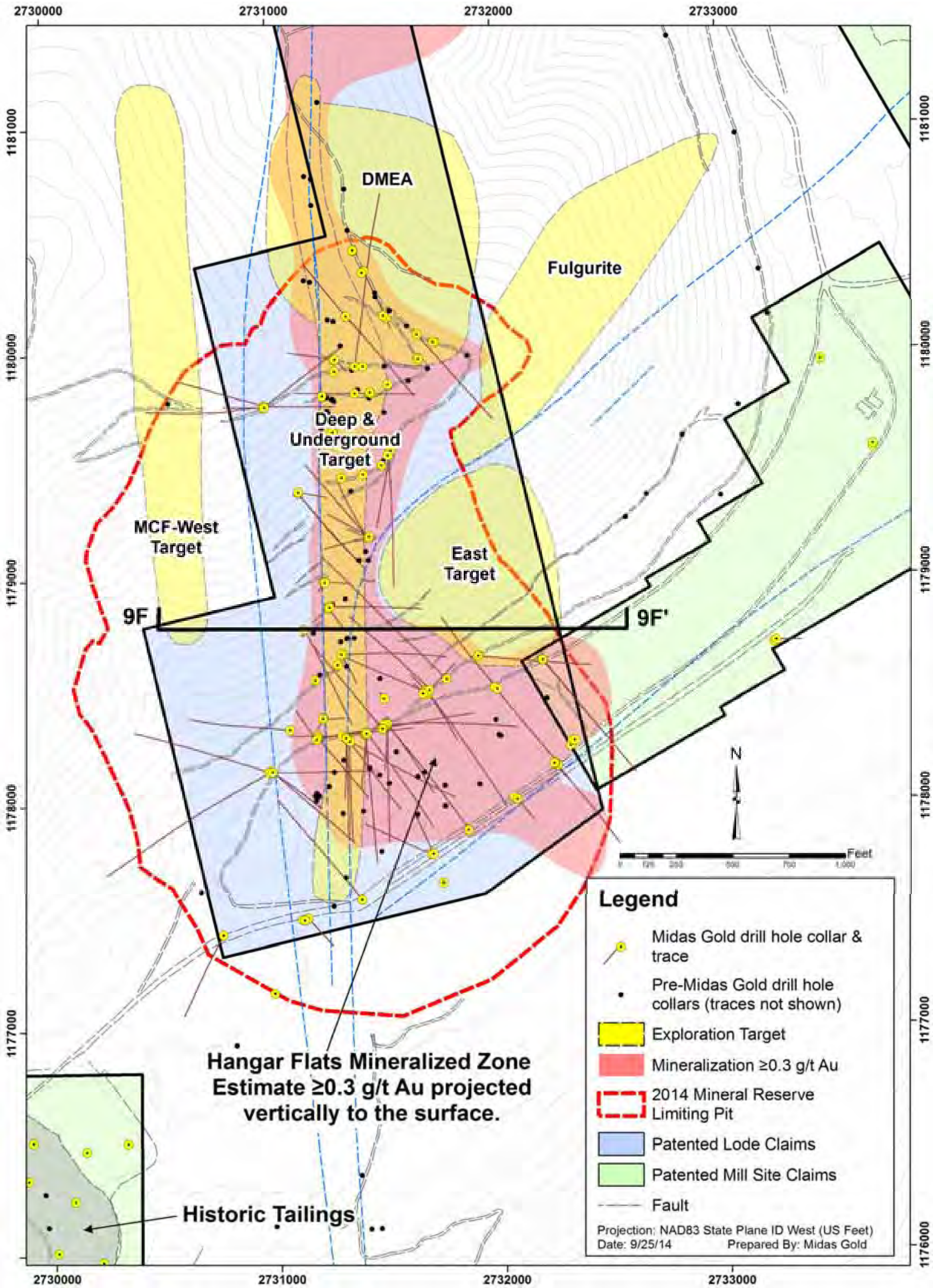
9.7.2 Hangar Flats

The Hangar Flats Deposit formed along the Meadow Creek Fault zone (**MCFZ**) and the 3,000(+) ft long corridor north, east and west of the main deposit is inadequately drill-tested outside of the known deposit (Figure 9.7).

Hangar Flats Deep Target

Historic sampling and production records from the former Meadow Creek Mine define a zone of high grade gold-antimony mineralization in a 30-330 ft wide corridor along the eastern boundary of the MCFZ that remains open along strike and down dip. Figure 9.8 shows drill hole MGI-12-203, which intersected multiple high grade intercepts, the most significant included 121 ft grading 2.96 g/t Au.

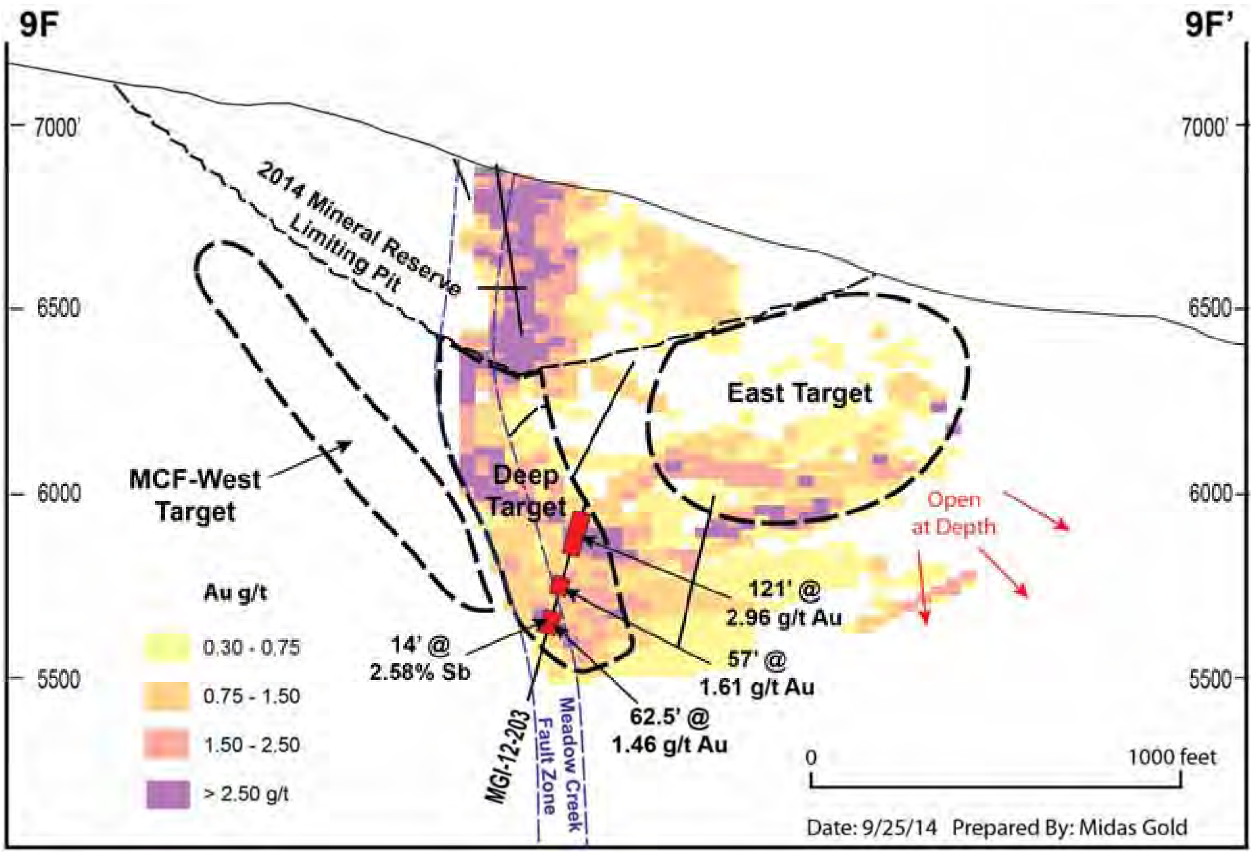
Figure 9.7: Plan map Showing the Hangar Flats Expansion Targets



DMEA Workings Target

This target lies beneath the northern part of the Hangar Flats PFS Mineral Reserve Limiting Pit and represents the northern extension of the "Hangar Flats Deep Target". The MCFZ is poorly tested over a distance of at least 1,000 ft at DMEA, which has been explored by a network of tunnels driven along the MCFZ. The underground workings were extensively mapped and sampled in the 1950s, which indicated the presence of north-east trending high grade vein systems. Pre-Midas Gold underground channel samples, from crosscuts, reportedly intersected a large zone of mineralization with a length-weighted average grade of 6.5 g/t Au over 92 ft and 1.56 g/t Au over 300 ft collected perpendicular to the MCFZ, while underground drill holes intersected significant high grade intercepts (Table 9.2).

Figure 9.8: Cross Section of Hangar Flats with a 160 ft Corridor



Note:
Potential mineralization reported here as prospects may be partially included within our resources discussed in Section 14 of this Report.

Table 9.2: Significant Drill Intercepts within the Hangar Flats Expansion Targets

Target	Operator	Drill Hole ID	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Antimony (%)
Hangar Flats Deep	Midas Gold	MGI-11-058	-90	-	588	713	125	3.13	1.45(1)
Hangar Flats Deep	Midas Gold	MGI-12-193	-88	324	921	1170	249	1.55	2.54
Hangar Flats Deep	Midas Gold	MGI-12-165	-79	320	825	870	45	1.40	1.20(2)
					915	960	45	1.29	1.12(3)
Hangar Flats Deep	Midas Gold	MGI-11-103	-70	140	605	644	39	4.84	-
					695	847	135	3.51	-

Target	Operator	Drill Hole ID	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold (g/t)	Antimony (%)
Hangar Flats Deep	Midas Gold	MGI-12-203	-65	320	696	817	121	2.96	
					912.5	970	57.5	1.61	0.26(4)
					1012	1074.5	62.5	1.46	2.58(5)
DMEA Workings	Midas Gold	MGI-12-331	-89	291	334	418	84	3.65	-
					441	480	39	1.36	0.10
					551	584	33	1.05	-
					657	696	39	2.71	-
DMEA Workings	Midas Gold	MGI-09-07	-70	90	679	836	157	5.09	0.30
					853	926	73	4.89	-
DMEA Workings (UG)(6)	Bradley Mining	DMA-20	-44	320	10	135	125	6.62	0.51(7)
DMEA Workings	Midas Gold	MGI-11-080	-90	-	619	654	35	2.61	-
DMEA Workings	Midas Gold	MGI-12-181	-90	-	520	560	40	2.66	-
DMEA Workings	Midas Gold	MGI-12-197	-90	-	838	889	51	2.59	-
					940	971	31	1.58	-
Underground Drill Target	Midas Gold	MGI-12-192	-83	280	344	434	90	2.88	-
					543	612	69	1.03	0.09
					1006	1300	294	1.57	2.76
MCFZ West	Midas Gold	MGI-11-099	-75	310	1507	1659	152	1.34	0.83(8)

Notes:
(1) Sb over 83.5 foot interval.
(2) Sb over 25 foot interval.
(3) Sb over 40 foot interval.
(4) Sb over 17.5 foot interval.
(5) Sb over 14 foot interval.
(6) UG = Underground drill hole.
(7) Sb over 95 foot interval.
(8) Sb over 115 foot interval.

Underground Drill Target

High grade gold-antimony mineralization has been intersected over a 2,000 ft strike length and over a 1,000 ft vertical extent and remains open to expansion at depth. 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 Au and 2.76% Sb illustrating this potential (Table 9.2).

MCFZ West Target

A geotechnical hole (MGI-11-099), drilled across the western limits of the conceptual PEA pit, intercepted significant mineralization (Table 9.2) and, based on geophysical surveys and oriented core data, mineralization is interpreted to be open and possibly extends along strike and up and down dip (Figure 9.8).

HF East Target

A large area east of the Hangar Flats Deposit has not been drill tested and is open to expansion. The area is underlain by mineralized low angle faults exposed east of the deposit which are known to control mineralization in the main portion of the Hangar Flats Deposit (Figure 9.8).

9.7.3 West End

There is potential to expand the West End Deposit at depth and along strike to the northeast and southwest. Highlights of significant drill intercepts from these areas are listed in Table 9.3 and the areas shown on Figure 9.9.

Figure 9.9: Plan Map of West End Showing Potential Expansion Zones

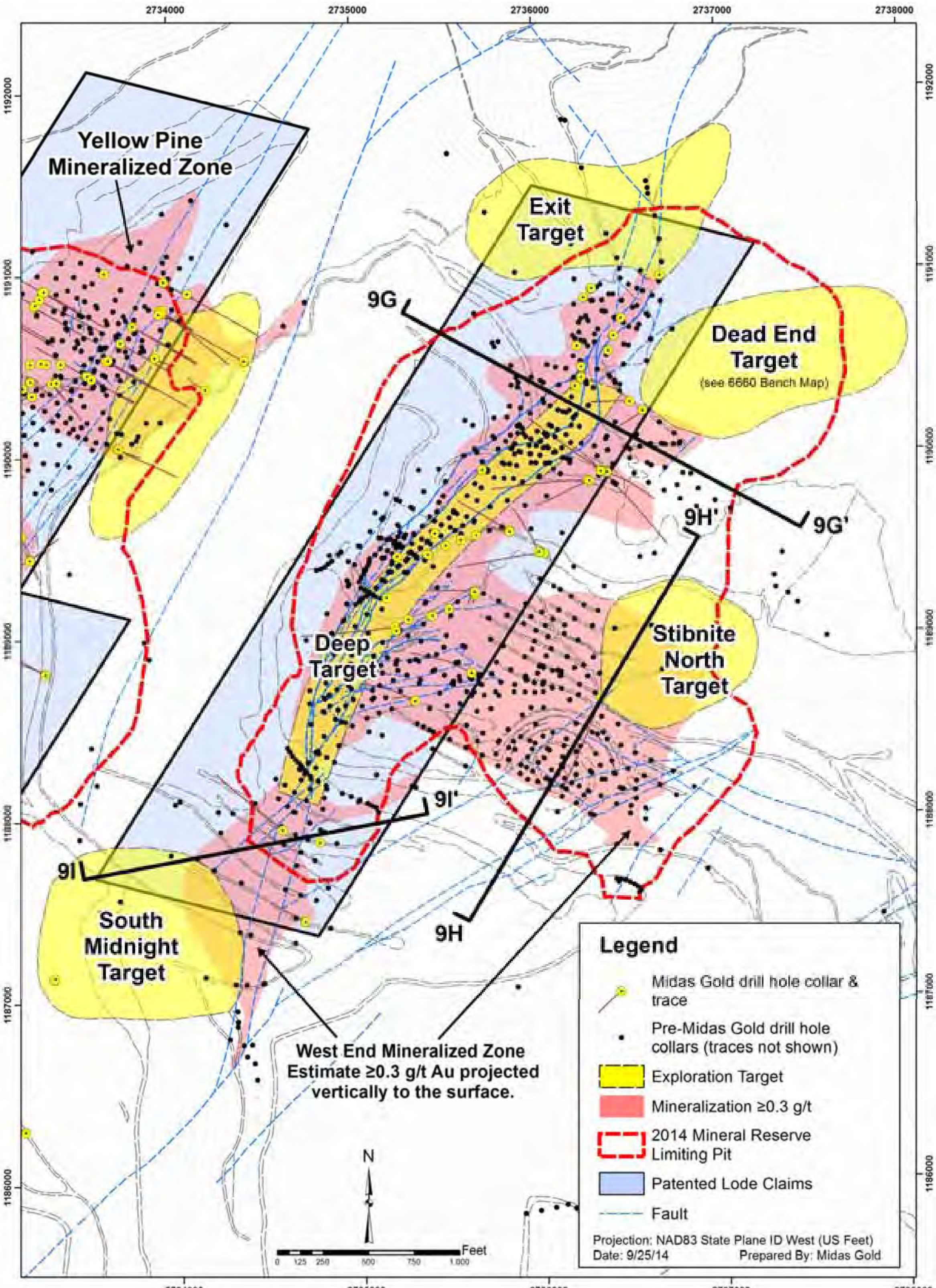


Table 9.3: Significant Drill Intercepts within the West End Expansion Targets

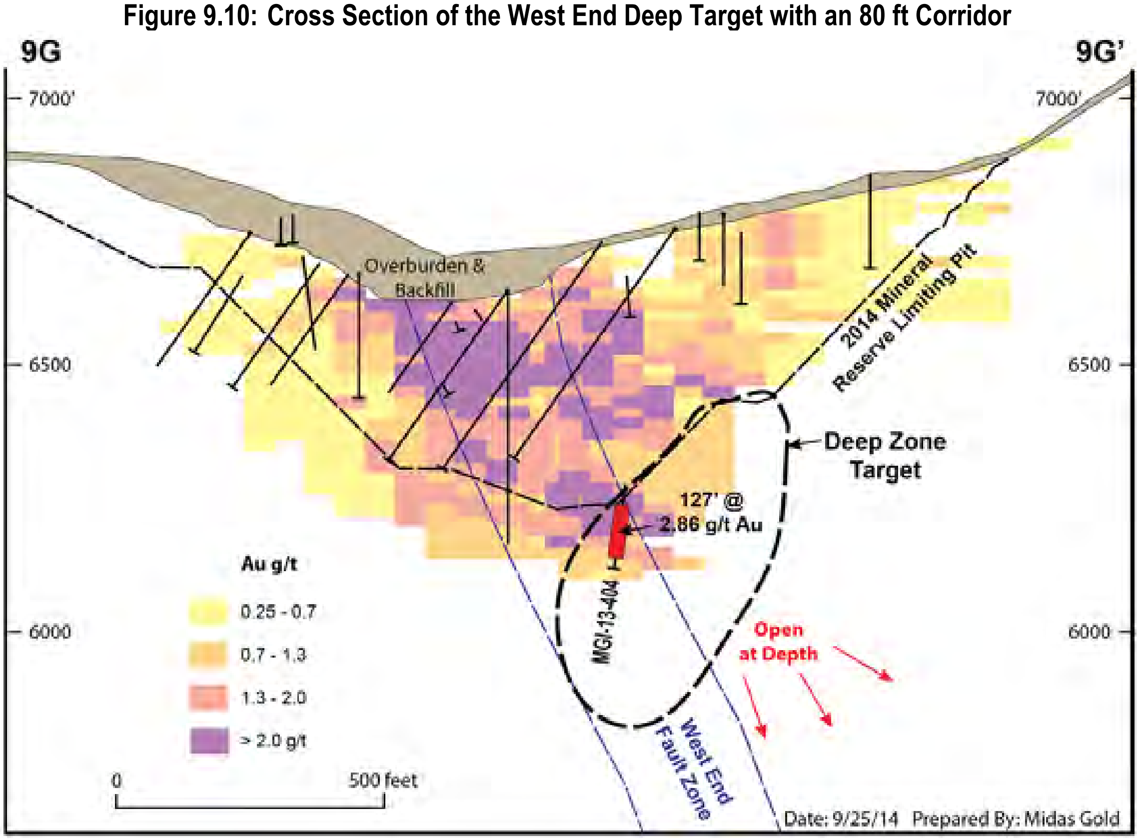
Target	Operator	Drill Hole ID	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold[1] (g/t)
West End Deep	Midas Gold	MGI-13-404	-75	230	521	648	127	2.86
West End Deep	Midas Gold	MGI-11-139	-66	260	835	1065	230	2.27
West End Deep	Midas Gold	MGI-12-290	-71	298	762	906	144	1.07
Dead End	Pioneer	W-110	-55	80	125	180	55	1.04
Dead End	Superior	WER83-23	-90	NA	110	340	230	1.10
Stibnite North	Pioneer	89-78	-70	300	250	405	155	3.46
Stibnite North	Midas Gold	MGI-10-37	-45	202.5	418	615	197	1.79
Stibnite North	Pioneer	89-57	-70	300	415	510	95	3.16
Stibnite North	Pioneer	89-75	-70	300	460	630	170	1.4

Note:
(1) Selected intercepts composited with length weighted averages with cut-off grade 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.

West End Deep Target

This target consists of a poorly explored area 330 ft wide and extending approximately 2,100 ft along strike beneath the Mineral Reserve Limiting Pit (Figure 9.10). The area is defined by three Midas Gold drill holes, with the most significant intercepts being 2.25 g/t Au over 230 ft and 2.86 g/t Au over 127 ft in MGI-11-139 and MGI-13-404, respectively.

Figure 9.10: Cross Section of the West End Deep Target with an 80 ft Corridor

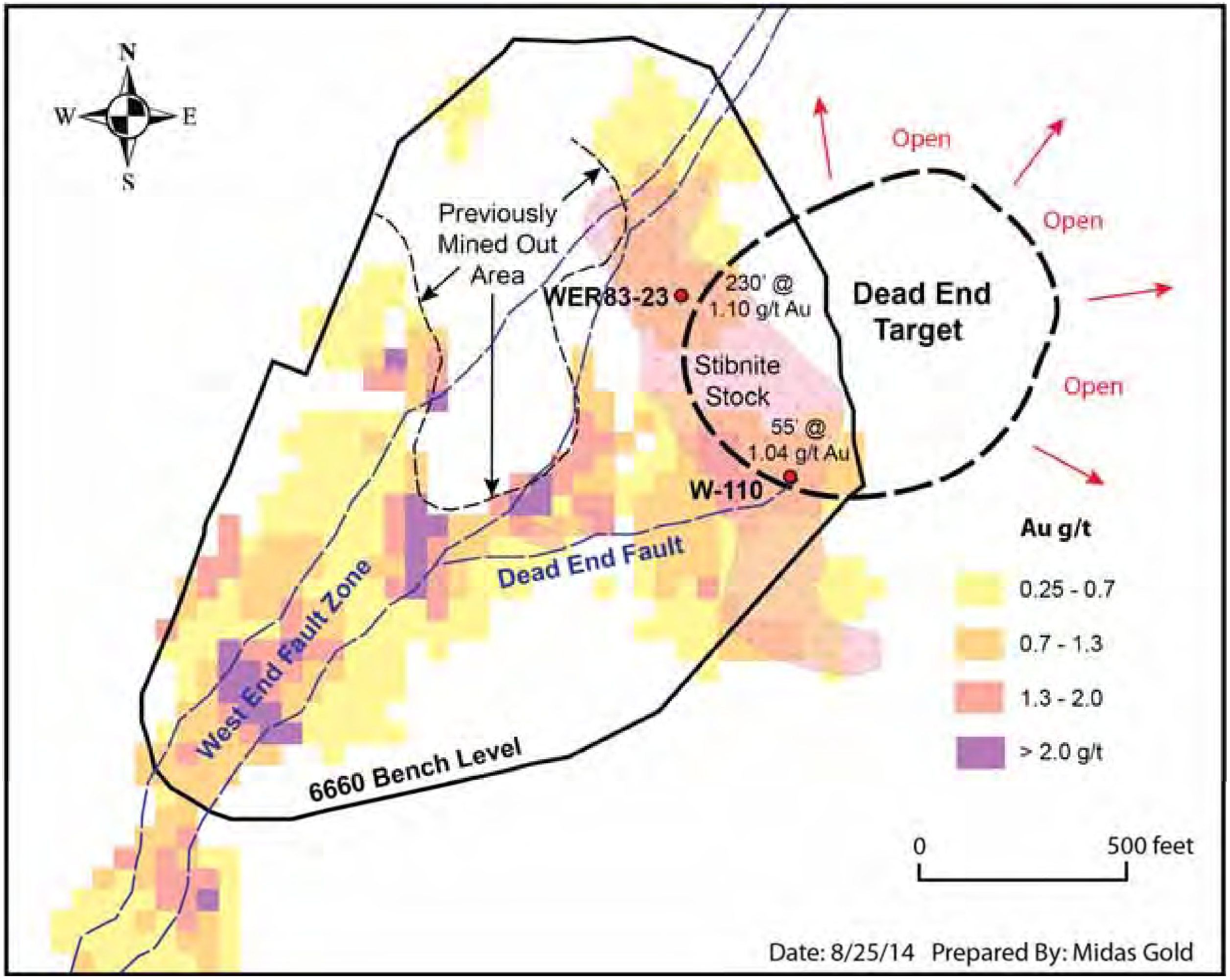


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Dead End Fault Target

This target is approximately 260 ft wide and extends over a strike length of 1,000 ft. The target is defined by holes drilled by pre-Midas Gold operators, with the most significant intercept being 230 ft with an average grade of 1.1 g/t Au along the east-northeast striking Dead End Fault that originates at a bend in the West End Fault system (Figure 9.11).

Figure 9.11: Level Section of the Dead End Fault Target with a 30 ft Thickness

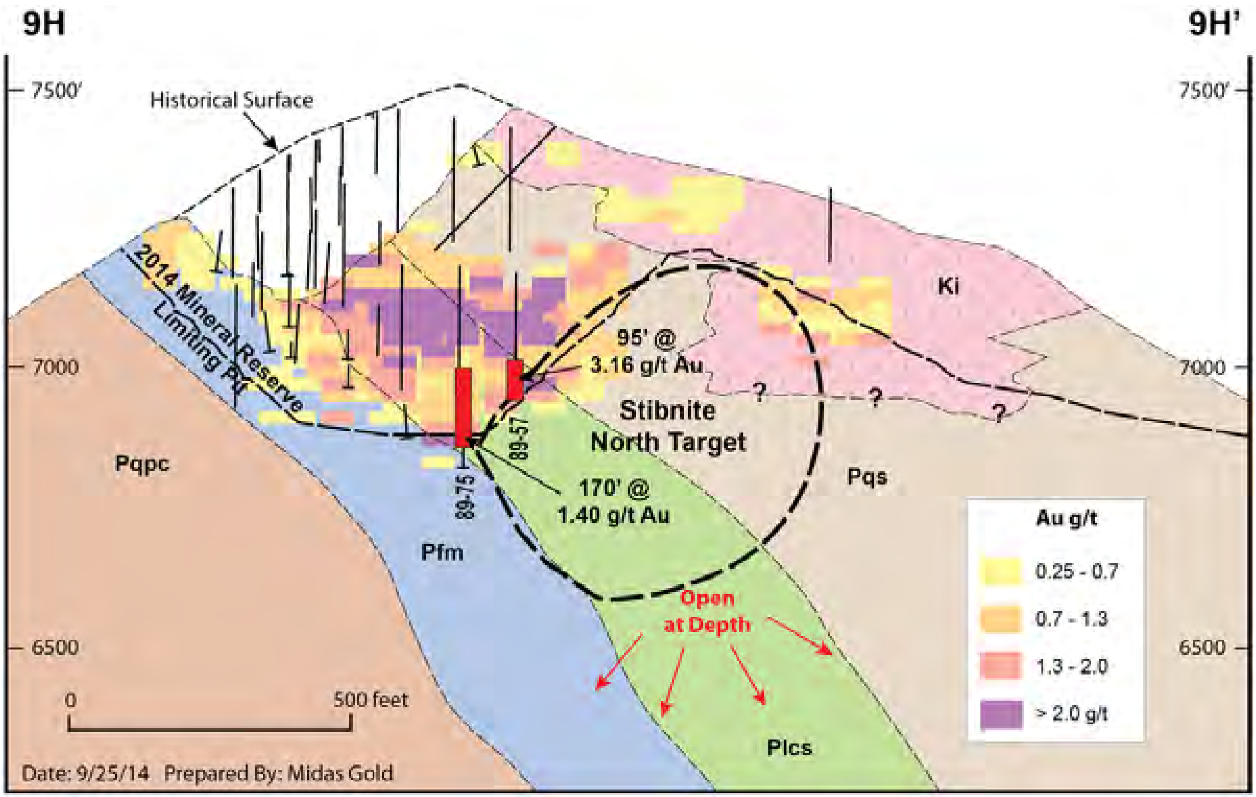


Note:
Potential mineralization reported here as a prospect may be partially included within the mineral resources discussed in Section 14 of this Report.

Stibnite North Target

The Stibnite North target is defined by Midas Gold and Pioneer Metals drill holes, with the most significant intercept being 155 ft with an average grade of 3.46 g/t Au in hole 89-78 (Table 9.3 and Figure 9.12). Mineralization may continue down dip and along strike within favorable faults and lithologies extending past the Mineral Reserve Limiting Pit.

Figure 9.12: Cross Section of the Stibnite North Target with an 80 ft Corridor

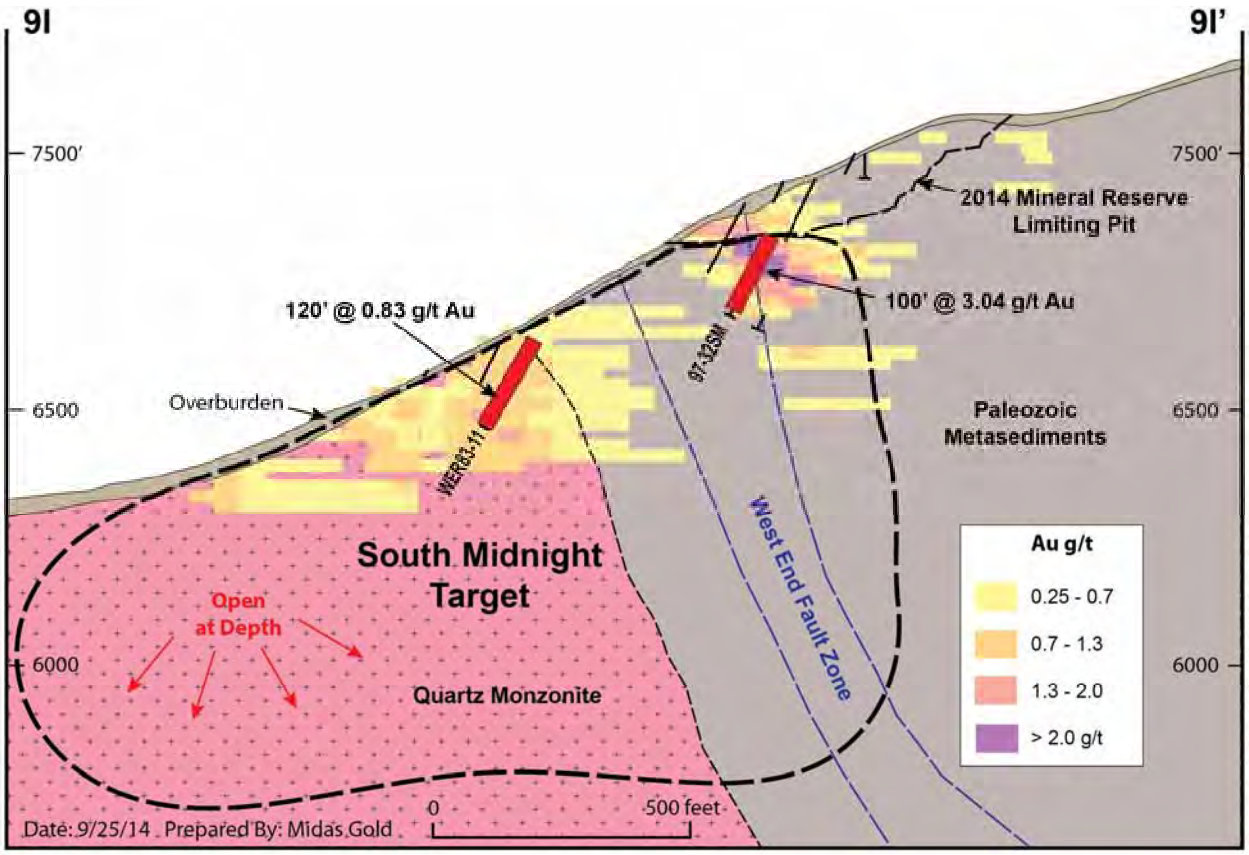


Note:
Potential mineralization reported here as a prospect may be partially included within our resources discussed in Section 14 of this Report.

South Midnight Target

This target is located at the southwest end of the West End Deposit. The area is identified by holes drilled by pre-Midas Gold operators. The most significant intercept was 100 ft averaging 3.04 g/t Au in hole 97-32SM (Figure 9.13). Mineralization roughly parallels the intrusive-metasediment contact and is open to the south.

Figure 9.13: Cross Section of the South Midnight Target with an 80 ft Corridor



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

Exit Target

This target is located northwest of the main West End Fault Zone, but also includes an extension of the fault to the east-northeast. The area is identified by a strong surface soil and rock chip Au anomaly over an area of approximately 1,300 ft by 1,300 ft. Canadian Superior identified an apparently continuous zone of east-west trending mineralization over 360 ft wide averaging 0.72 g/t Au in chip samples from road cuts. Mapping and sampling by Midas Gold geologists confirmed portions of this anomaly where the road cuts were still accessible, the balance having been buried under waste rock storage facilities. This area has been inadequately tested by past drilling and warrants further study.

9.8 PROSPECTS FOR DISCOVERY OF HIGH GRADE, UNDERGROUND MINEABLE POTENTIAL

9.8.1 Scout

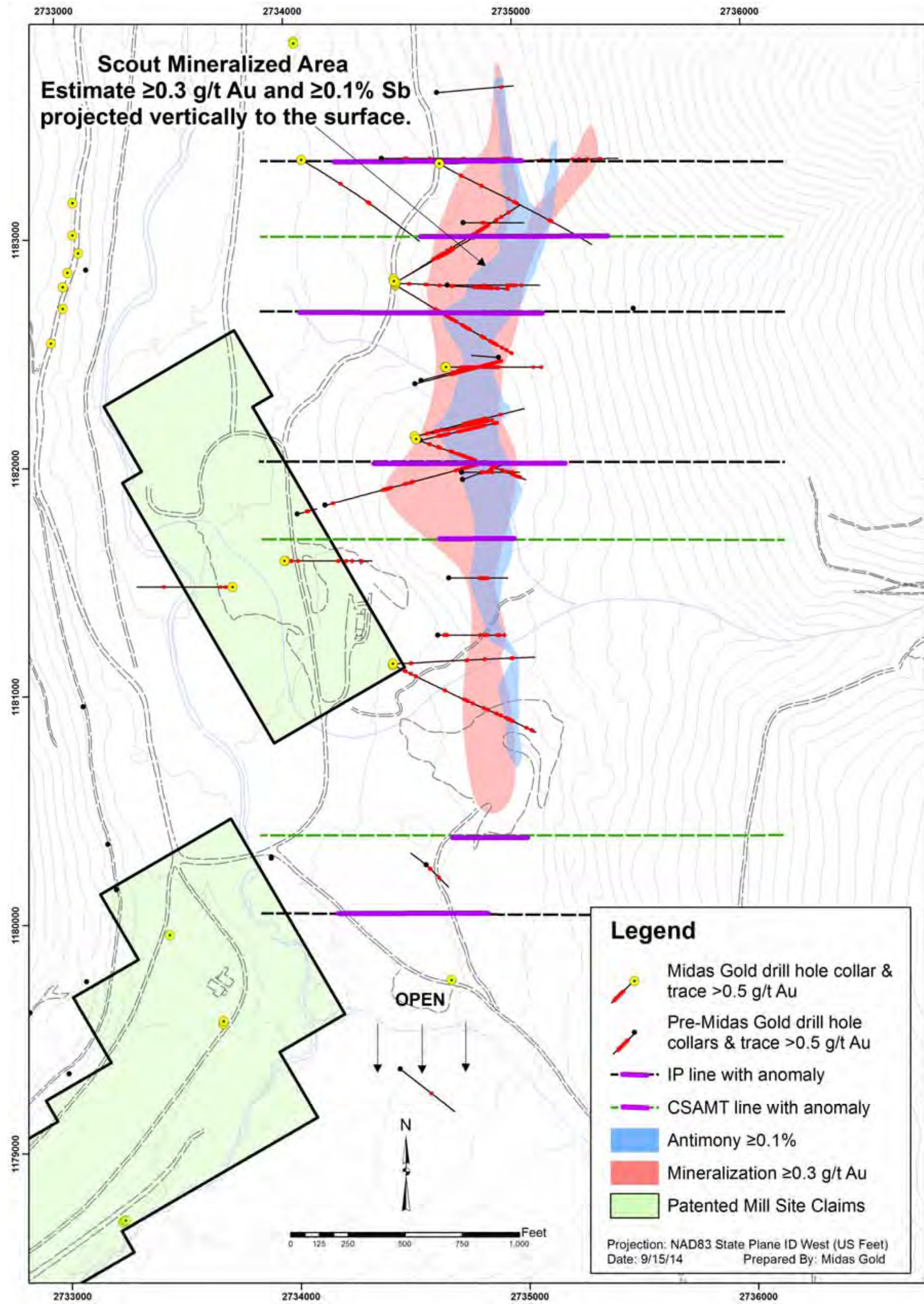
Scout is a potentially underground mineable Au-Ag-Sb exploration prospect (Figure 9.14) discovered in the 1930s by Bradley interests and further evaluated during Strategic Minerals investigations in the 1940s. Detailed exploration by other operators followed between 1947 and 1990. Pre-Midas Gold drilling includes 18 holes totaling 6,912 ft. Midas Gold work includes IP and CSAMT surveys, mapping, rock and stream sediment sampling along with the completion of 21 drill holes totaling 15,629 ft; Table 9.4 lists significant drill intercepts. Host rocks include 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, in addition to east-west and southwest-northeast trending faults. Drilling to date has been deemed insufficient to produce a mineral resource estimate, but does suggest a potential underground exploration target. The dimensions of the potential target, as determined by simple polygonal estimation methods from drilling, and further defined by trenches and geophysical surveys, outlines a **conceptual** potential underground target in the range of 2-5 million tons containing between 50,000-300,000 oz Au; 40,000,000-150,000,000 lbs Sb; and 300,000-1,500,000 oz Ag with target dimensions of approximately 25-75 ft thick (true), 2,000-3,000 ft along strike and extending 250-300 ft down dip at grades ranging from 1-2 g/t Au, 1-4% Sb, and 5-25 g/t 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 coincident with an IP and CSAMT anomaly. **Exploration data for the Scout target 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 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.**

Table 9.4: Selected Drill Intercepts from the Scout Prospect

Drill Hole ID	Operator	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Au[1] (g/t)	Sb[1] (%)	Ag (g/t)
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	1.06	7.36
MGI-12-244	Midas Gold	-45	77	305.4	429.1	123.7	2.37	0.5	5.88
				including					
				331.4	363.8	32.5	5.7	1.46	15.3
MGI-12-249	Midas Gold	-53	115	311.7	862.5	550.9	0.78	2.02	14.8
				including					
				419.3	490.8	71.5	0.82	4.63	43.5
MGI-12-302	Midas Gold	-45	120	495.1	534.4	39.4	4.55	1.71	4.65
				651.6	677.5	25.9	1.68	2.86	8.42
MGI-12-345	Midas Gold	-44.6	116	764.1	816.9	52.8	1.68	5.42	48.0
MGI-12-347	Midas Gold	-50	90	771.3	784.4	13.1	5.96	12.3	114.6
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:
(1) Selected intercepts composited with length weighted averages with cut-off grade 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 9.14: Plan map of the Scout Prospect



9.8.2 Garnet

The Garnet target is a potential underground mineable exploration prospect. It is the site of past underground exploration in the 1920s and a small open pit exploiting oxide heap leach ores in the mid-1990s. El Paso Oil and Gas discovered a broad zone of outcropping high-grade gold mineralization here in the mid-1970s and, between 1974 and 1995, four other companies explored the prospect. Pre-Midas Gold drilling includes 105 RC, core and air track percussion holes totaling 16,261 ft. The length-weighted average grade of pre-Midas Gold down-hole drill composites (using cutoff detailed in Table 9.5) is 5.3 g/t Au. Highlights of some of the drill intercepts in the unmined portions of the prospect are tabulated in Table 9.5. In 1995, Stibnite Mines, Inc. operated an open pit mine for one season within the prospect area. For approximate historic production records, see Section 6 of this Report. Fire assay grades of the material mined, where fire assayed, were approximately twice the cyanide leachable head grade. Mineralization occurs in sulfide and silica impregnated carbonates within a north plunging body developed at the intersection of a north-south striking, steeply to moderately west-dipping fault zone within two granite sills and an east-west striking, north-dipping dolomite unit. Midas Gold work includes mapping and rock, soil, and stream sediment sampling, but no drilling.

The conceptual target generally trends north-northwesterly from the Garnet Pit (Figure 9.15 and Figure 9.16). The dimensions of mineralized material located beneath and beyond the boundaries of the former open pit, as determined by simple polygonal estimation methods from historic drilling (Figure 9.15, Figure 9.16) and geophysical data, outlines a conceptual potential underground target in the 1-2 million ton range containing 250-500 koz Au approximately 30-60 ft thick (true) by 160-250 ft wide by 1,300-1,800 ft long down plunge at grades ranging from 5 g/t Au to 8 g/t Au. **Exploration data for the Garnet target 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 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.**

Table 9.5: Pre-Midas Gold Drill Intercepts within the Garnet Prospect

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 80-90% of the reported intercept lengths.

Figure 9.15: Plan map of Grade x Thickness at the Garnet Prospect

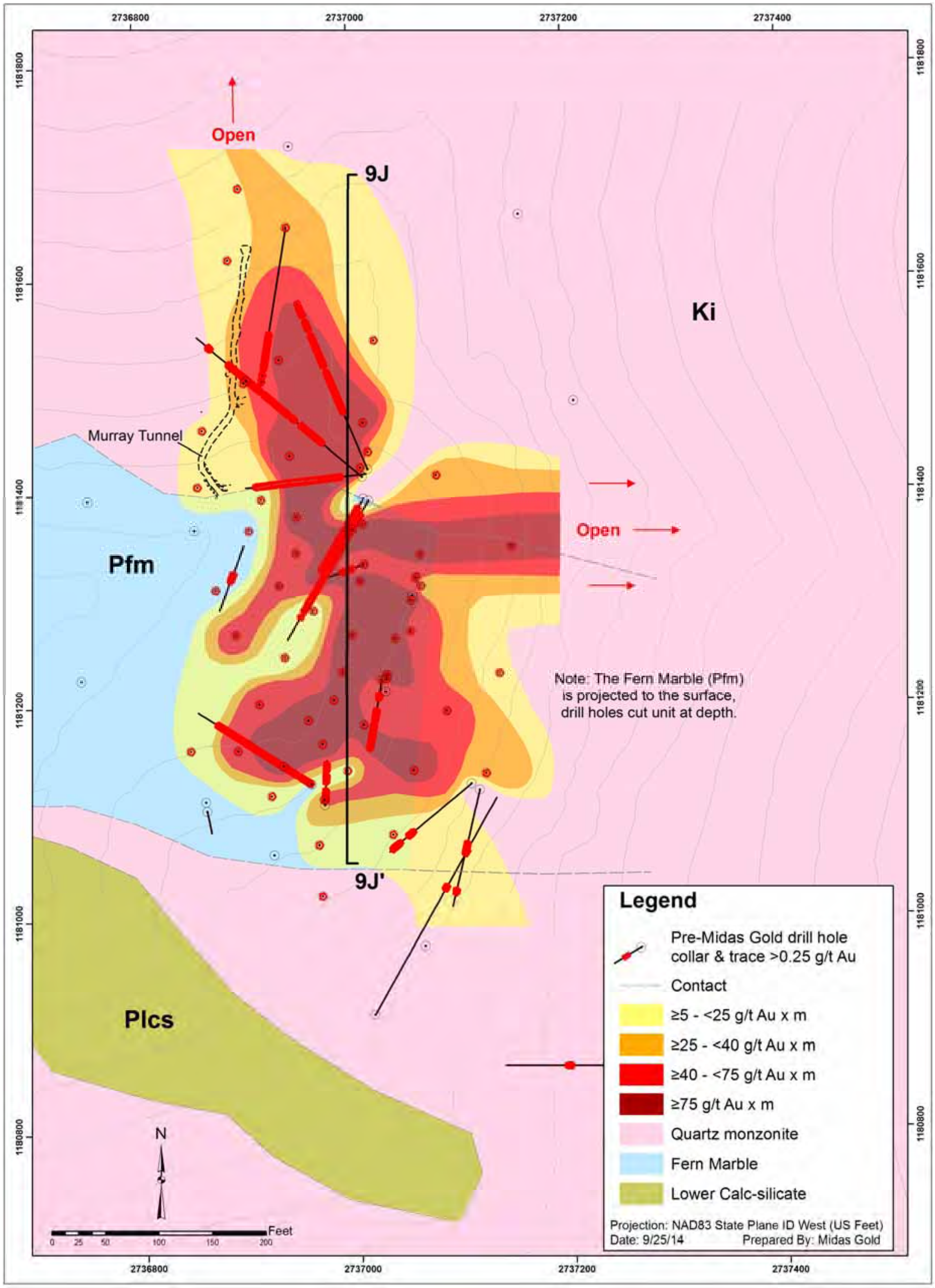
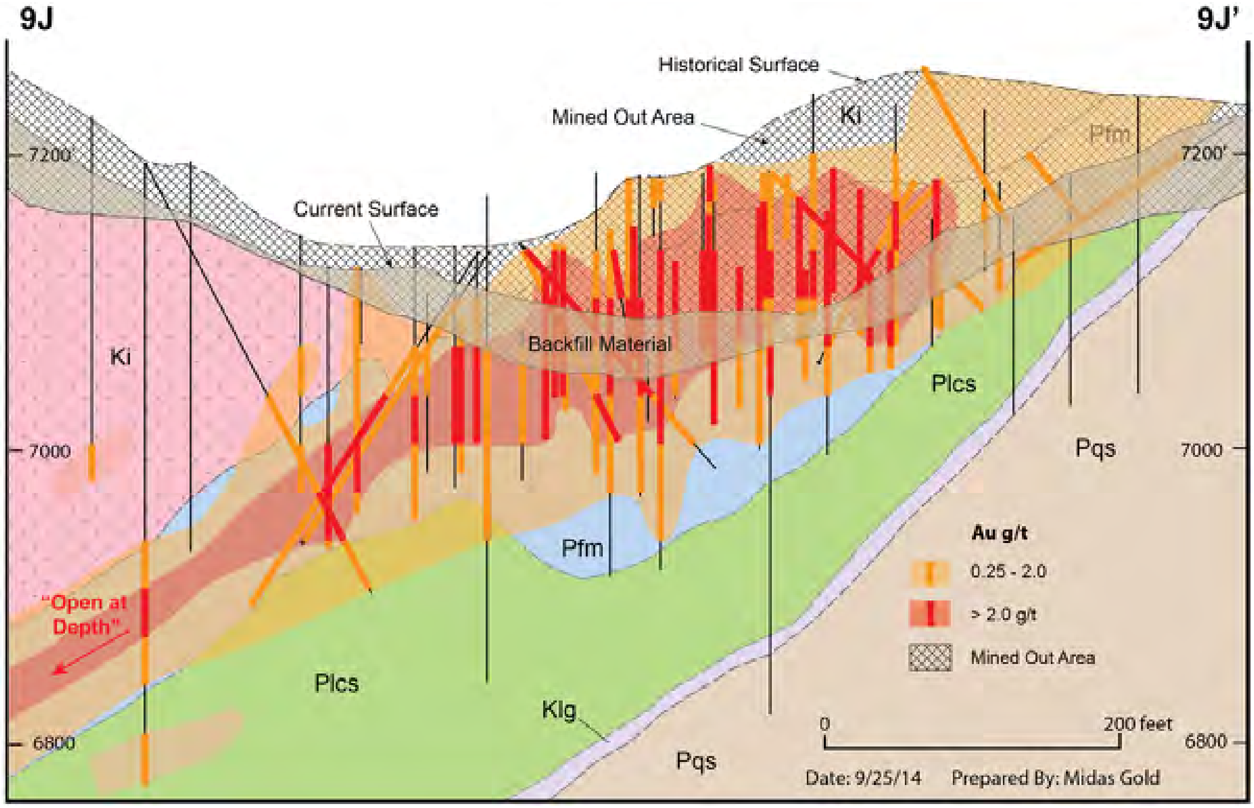


Figure 9.16: Cross Section of the Garnet Prospect with a 150 ft Corridor



9.8.3 Upper Midnight and Doris K

The Upper Midnight and Doris K prospects are located north-northeast and northeast of the Garnet Prospect, respectfully; they were originally located prior to World War II and re-discovered in the early 1970s when El Paso and Superior sampled and defined numerous large gold-in-soil and rock chip anomalies. In 1976, sampling of a black carbonate outcrop at Upper Midnight returned high grade gold assays, which were followed up by air track, core and RC drilling that confirmed the presence of a steeply southeast-dipping, northeast-striking 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 grade of 8.33 g/t Au (Table 9.6). In the early 1990s, a large soil, ground magnetics and Very Low Frequency Electro-Magnetic (**VLF-EM**) survey was completed over both the Upper Midnight and Doris K prospects and outlined several coincident magnetic anomalies and strong conductive features spatially associated with the anomalous geochemical features. The Doris K prospect also saw significant soil, rock, and trench sampling, but has not been drill tested. Neither prospect has any recorded historic production.

Between 2010 and 2013, Midas Gold collected stream sediments, soils, and outcrop samples covering Upper Midnight and Doris K to confirm and expand on past exploration work. At Upper Midnight, 60 rock chip samples outlined anomalous gold values over a broad area including 3 ft chip samples of 5.24 g/t Au within brecciated quartzites and 2.79 g/t Au in altered carbonates within a large 400 ft by 500 ft soil anomaly (> 0.1 g/t Au). A total of 46 rock samples were collected at the Doris K prospect with 19 of the 46 samples resulting in values > 0.1 g/t Au. High values range up to a maximum of 15.7 g/t Au within the brecciated quartzite and 13.55 g/t Au within the altered carbonates. These higher grade samples were taken within 150 ft of each other and within a historical 300 ft by

250 ft gold-in-soils anomaly near a historical adit. Ground geophysical (CSAMT and IP-Resistivity) and airborne EM surveys produced anomalous responses over known or suspected mineralized zones in these areas and adjacent to them.

Mineralization at both prospects occurs in quartzites, carbonates and calc-silicates near the intersections with northeast and northwest striking faults. The prospects lie on the flanks of a large, northwest-trending, marble-cored, synform; Upper Midnight on the overturned northeast limb and Doris K along the hinge. The conceptual targets consist of sediment-hosted, structurally and stratigraphically controlled, high grade, potentially underground mineable gold deposits. The Upper Midnight target trends north-northeast, and lies at the intersection of faults and the Middle Marble unit. Due to steep slopes and small footprint, there is little to no open pit potential, but the mineralized zone, as defined by soils, rocks, trenches and drilling, could extend along strike for a strike 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 potentially underground mineable target. Doris K is situated within a northeast trending zone of siliceous stockwork veining and breccias cutting quartzite and a friable recrystallized limestone unit. The breccia zone is approximately 50 ft-150 ft wide and is traceable for at least 500 ft in a northeast-southwest direction and is exposed over 200 ft of vertical extent across the nose of the synform. While less advanced than Upper Midnight, based on surface sampling to date, Doris K could represent another high grade, potentially underground mineable deposit.

Table 9.6: Pre-Midas Gold Drill Intercepts at 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 cut-off grade 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.

9.9 PROSPECTS FOR DISCOVERY OF NEW BULK MINEABLE POTENTIAL

9.9.1 Broken Hill - Saddle Trend

The Broken Hill, Ridgetop, Cinnamid, Saddle and Fern prospects are large tonnage, potentially bulk-mineable exploration targets that are located on a 2.5 mile long, northwest trending zone of continuous alteration and mineralization that also includes the Stibnite and West End deposits to the northwest and the former Fern Mine to the southeast. These prospects are shown in the East Side long section on Figure 9.2. Ridgetop and Cinnamid were discovered during soil sampling in 1990-1991; follow-up included systematic road cut rock chip sampling and mapping, with a total of 775 rock chip samples, followed up with drilling of 74 holes in Ridgetop and Cinnamid between 1992 and 1996. Drill hole 92-47 intersected 74.8 ft grading 2.13 g/t Au at Ridgetop and drill hole 92-49 intersected 52.5 ft averaging 4.15 g/t Au at Cinnamid. Highlights of historical drilling are tabulated in Table 9.7. Broken Hill was discovered by rock chip sampling in 1991 and followed with limited trenching and drilling. The Saddle prospect was discovered during soil sampling in the early 1980s and follow up rock chip sampling which outlined a broad area, approximately 1,500 ft in length by up to 500 ft in width, of anomalous gold in silicified Fern

Marble. Based on surface rock-chip and soil sampling between the Cinnamid and Saddle prospects, there appears to be continuity of gold mineralization at surface, in the area separating the two prospects over a distance of about 2,000 ft. A water monitor well drilled in 1996 confirmed the strong gold-in-soils anomaly at Saddle, intersecting 120 ft of 0.92 g/t Au in oxidized Fern Marble (Table 9.7). Since most analytical work from drill sample assays along the trend utilized CN-soluble assay methods, total gold grades may be significantly under reported. 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, suggesting these structures have additional potential along strike where no past work has been completed.

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 in reactive siltites, calc-silicates, Fern Marble and in a sequence of interbedded quartzites and schists) along the trend. Ridgetop and Cinnamid have been drilled extensively by previous operators, who focused on shallow oxide mineralization, while a nominal number of holes have been drilled at Broken Hill and only a single water monitoring well was drilled at Saddle. However, excellent potential exists along the entire trend to discover additional sulfide mineralization. On the southeast end of the trend, between the Ridgetop and Saddle prospects, 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 quartz-schists sequences 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), defining a conceptual target ranging from 4-10 million tons at grades between 1-2 g/t Au. All previously drilled mineralization remains open to expansion in along strike and down dip. Given the trend is over 2.5 miles long and is only drill tested over a short section of the trend, there is considerable upside potential. **Exploration data for the target area 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 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.**

The Fern Prospect is an epithermal, carbonate-hosted, Au-Ag exploration target at the southeast end of the Broken Hill-Ridgetop-Cinnamid-Saddle trend that was initially explored for gold in the early 1900s and later was prospected for mercury during the Thunder Mountain gold rush (Larsen and Livingston, 1920). There was minor Hg production after the turn of the century from ores developed from open cuts and from underground workings over a total elevation of approximately 1,085 ft. In the 1940s, the USBM conducted mercury exploration, including extensive trenching. In the 1950s, additional trenching was completed under the DMEA program and, 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 1983-1984, rock chip, soil sampling, detailed mapping and two geophysical IP lines were completed by Canadian Superior Mining, who drilled three follow-up 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. There is potential to discover a low-tonnage, high-grade, potentially underground mineable gold deposit and/or mineralization amenable to open pit extraction methods where low-grade gold mineralization would be associated with replacement bodies in the Fern Marble. Twenty-nine systematic outcrop chip samples, taken by Pioneer Metals in 1987 (near an old adit driven adjacent to a northeast trending jasperoid breccia) ranged from 0.87 g/t Au to 42.7 g/t Au and averaged 14.3 g/t Au. Holes drilled nearby at the time did not adequately test the mineralized structure but, nevertheless, the holes did cut significant mineralization (Table 9.7), suggesting the possibility of a larger, open-pit target, as well as underground potential from the main structure. Midas Gold work has included mapping, rock, soil and stream sediment sampling. Midas Gold collected a total of 11 systematic chip samples that averaged 19 g/t Au across a northeast trending, northwest dipping, 25-65 ft wide breccia. Mapping traced the zone of mineralization for approximately 175 ft along strike and over 130 ft of vertical relief up slope, beyond which it is lost under cover in both directions.

Table 9.7: Significant 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.

9.9.2 Hermes Trend

Similar to the Broken Hill-Saddle trend, the Hermes Trend prospects are located along a northwest-southeast zone that follows the trace of the Hermes Marble unit over a distance of 2.5 miles from the West End Deposit to the Cinnabar Mine complex; the trend includes the Photon, AC/DC, and Hermes prospects.

On the northwestern end of the trend, the "Northeast Extension" portion of the West End Deposit lies within highly silicified Hermes Marble, where a series of northeast trending faults cuts the reactive and receptive unit. The Photon Prospect is a new discovery made by Midas Gold in 2013 that is comprised of a large, multi-element soil anomaly at the intersection and extension of the Broken Hill fault system and the Hermes Marble. Farther to the southeast, at the AC/DC Prospect, Midas Gold has developed a large soil anomaly and identified alteration in quartzites where anomalous rock chips in the area suggest a similar setting to the Photon Prospect. At the far southeast end of the trend, at the Hermes Prospect, a limited drilling program was completed in 1990 following up on a soil and rock chip anomaly where drill intercepts included 45 ft of 0.74 g/t Au, 15 ft of 0.7 g/t Au, 40 ft of 0.45 g/t Au, and 20 ft of 0.63 g/t Au. Many of the holes were only assayed utilizing CN-soluble methods and the lack of fire assays below the redox boundary may have significantly under reported total gold values for these areas. Other exploratory work along the Hermes trend includes extensive development work and drilling at the past producing Cinnabar Mine complex, which was operated intermittently between 1921 and 1958. During World War II, it was among the leading quicksilver (Hg) producers in the U.S. and was the second largest mercury producer in Idaho. Midas Gold work includes mapping, rock and soil sampling and limited ground geophysical surveys along the northwestern end of the trend.

Along the Hermes trend, silicification in the marble is pervasive, especially adjacent to quartzite contacts, and mineralization occurs in stratiform silicified zones where reactive carbonates have been converted to jasperoids, as well as in cross-cutting breccias and silica veined zones. The mineralized areas are typically elongated parallel to bedding strike, but are thickest and best developed at the intersection of favorable beds and northeast-trending fault structures, as seen at Photon. The current surface area of the soil and rock chip geochemical anomaly measures approximately 1,500 ft long and is up to 600 ft wide.

9.9.3 Mule

The Mule prospect is a potentially 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 lies adjacent to the contact of the Stibnite roof pendant and volcanic stratigraphy associated with the Tertiary Thunder Mountain Caldera and can be seen on the East Side long section on Figure 9.2. Old surface cuts and minor underground workings from the 1920s era were rediscovered in the late 1980s. Subsequently, Pioneer excavated three trenches in 1987 around the northern end of some of the open cuts. These trenches cut a vein system that trends N30°E and dips 30°W. The first trench included a 1-2 ft wide vein which averaged 51 g/t CN-leachable Au within a 40 ft wide altered zone that averaged 0.58 g/t CN-leachable Au (excluding the vein intercept). The second trench, located approximately 175 ft North of the first, included a 2 ft wide vein which assayed 6.03 g/t CN-leachable Au within a 158 ft wide zone (true width ~75% of this) that averaged 0.4 g/t CN-leachable Au (excluding the vein). A third trench, 100 ft south of the first trench, did not intersect the vein nor cut altered rocks and no assays were reported, but appears to be situated too far to the west to intercept the vein based on the projections from the northern trenches. Midas Gold's 2011 airborne magnetic and EM surveys outlined a large, several mile long, N-S trending, geophysical feature running through this area and 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 550-sample reconnaissance type soil grid was established over the area and outlined two large soil anomalies near the old trenches and another anomaly farther to the south. A total of 18 rock samples, from the limited bedrock exposures within and around these soil anomalies, were collected from the prospects and consistently indicated the presence of narrow, but high grades of gold in veins within broader zones of silicified intrusive rocks.

Unlike mineralization elsewhere in the District, neither arsenic nor antimony appear to be particularly anomalous, nor associated with gold mineralization in this prospect. Sampling by Midas Gold outlined two large soil and rock chip anomalies associated with sericitized and silicified granite and high grade Au-veins and silicified Au-bearing intrusives. The largest of the anomalies in the south, is at least 1,500 ft long in a N-S direction and 750 ft wide in an E-W direction and also continues to be open to the south. The northern anomaly covers an area of approximately 350 ft by 750 ft, which is associated with the area of past historic trenching abuts against transported cover areas. The narrow, but high grades of gold in veins and silicified zones in the intrusive rocks could represent possible underground exploration targets while the historic trench results and large soil anomalies suggest two broad areas that might host potentially bulk tonnage mineralization amenable to open pit mining.

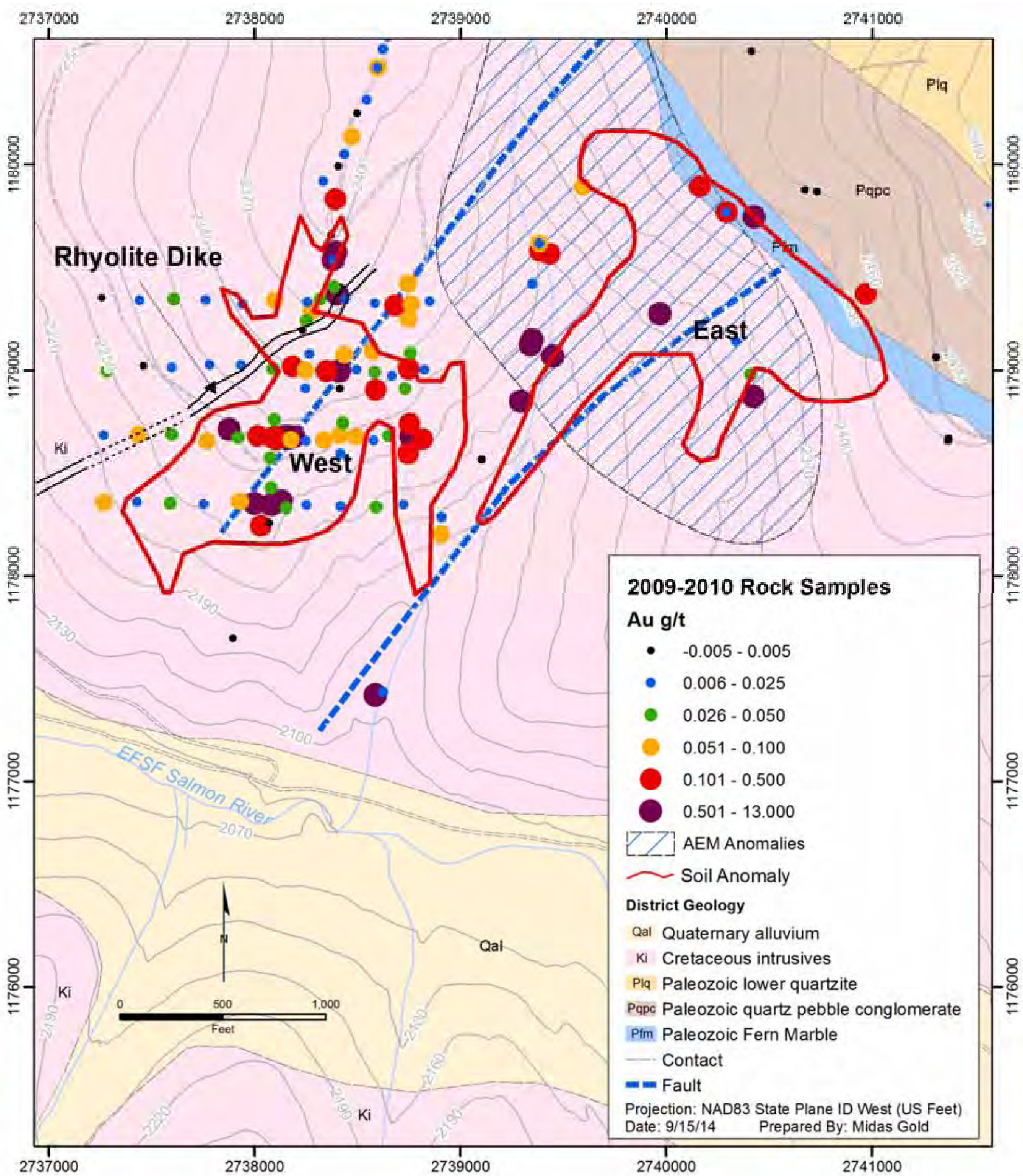
9.9.4 Rabbit

The Rabbit prospect consists of large, coincident, multi-lobed soil, rock chip geophysical anomalies situated east of the Hanger Flats Deposit and southeast of, and along strike from, the Garnet Prospect. The area was targeted after compilations suggested a setting similar to the nearby Garnet Prospect. Figure 9.17 shows soil and rock chip values and the two targets identified on the prospect. 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 an epithermal environment. Midas Gold work included mapping, stream sediment, rock chip, soil and test pit sampling and two lines of IP-resistivity.

The intrusive-hosted West Rabbit Prospect was first discovered in the 1920s when a short (~250 ft long) adit was driven into altered quartz monzonite and minor placer workings were excavated in the creek below the adit. The

NE-trending lode is reported to be over 100 ft wide and to extend over a vertical range of 100 ft. Minor prospecting was completed by modern explorers, but the prospect has reportedly never been drilled. The West Rabbit area is underlain by anomalous soils and rocks (Figure 9.17) that outline a conceptual intrusive-hosted target zone roughly 825 ft wide x 1,475 ft long, with over 500 ft of vertical relief. The dolomite-hosted East Rabbit Prospect was discovered by Midas Gold geologists in 2010 (Figure 9.17) based on anomalous soils and rocks (Figure 9.17) that outline a conceptual target zone roughly 650 ft wide x 1,975 ft long, and with over 600 ft of vertical relief.

Figure 9.17: Plan map of the Soil Sample Geochemistry at the Rabbit Prospect



9.9.5 Meadow Creek Fault Zone Trend

The MCFZ trend consists of a ~2 mile long string of prospects aligned along the MCFZ and associated cross structures; at the southern end lies the Hangar Flats Deposit and at the northern end, the Yellow Pine Deposit. The prospects along this trend can be seen in the West Side long section on Figure 9.2. Targets vary from open-pit to underground exploration prospects. The North Tunnel, sunk in the Fiddle Creek drainage in the 1920s, was a short exploration tunnel, with minor production, that was later re-opened in the 1940s to complete a small underground drilling program. The DMEA tunnels were driven westward, towards the MCFZ, between the North Tunnel and the Meadow Creek Mine workings and discovered high-grade mineralization during underground sampling and drilling, but it was not exploited. Minor additional historic exploration occurred along the MCFZ Trend from Yellow Pine to Hangar Flats, including ground-based geophysical surveys, soil grids, trenches, pits and rock sampling.

The MCFZ Trend hosts mineralization in both high-grade, Au-Sb-Ag-W vein systems and disseminated intrusive Au-Sb-Ag mineralization. The Idaho batholith is the predominant rock 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 and 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. The main MCFZ has been mapped underground as north-south trending and steeply dipping, with moderately-dipping, northeast and east-west striking structures intersecting it. Pre-Midas Gold underground mapping at the DMEA, Monday and North tunnels outlined extensive zones of Au-Sb-W mineralization and demonstrates 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 0.7 g/t Au was reported just east of the main MCFZ trend. Midas Gold drilling in this area resulted in an intercept 45 ft of 2.02 g/t. 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. Representative intercepts, when taken in aggregate (Table 9.8) are not considered underground grade at this time but high grade vein systems within these intervals with values >5 g/t Au along with significant W, Sb and Ag values, indicate the potential for high grade discoveries Figure 9.2. Broad soil and ground geophysical anomalies covering the projected surface expression of these vein and shear systems in the North DMEA area, and along the trend, suggest continuity of mineralization from these underground zones up to the surface.

From 2009 through 2012, Midas Gold completed over 31 line-mi of ground geophysical surveys, including IP-Resistivity and CSAMT, along with soil, rock and trench sampling and drilling in the DMEA and Fiddle Creek areas (Table 9.8). Additional work included a stream-sediment sampling program and several soil grids along the MCFZ trend. Drill intercepts were encouraging, especially in light of the fact that they did not drill into the main portion of the IP anomaly.

Table 9.8: Drill Intercepts within the Meadow Creek Fault Trend

Drill Hole ID	Operator	Collar Dip (°)	Collar Azimuth (°)	From (ft)	To (ft)	Interval (ft)	Gold[1] (g/t)
FC-1	USBM	+1	118	15	65	50	0.60
				290	345	55	0.92
MGI-10-39	Midas Gold	-45	83	245	290	45	2.02
DMA-05	USBM	+3	260	15	70	55	0.62
DMA-06	USBM	-1	157	35	70	35	0.71
				315	345	30	0.46
DMA-07	USBM	0	89	5	80	75	0.93
				235	270	35	0.51
DMA-08	USBM	0	270	10	100	90	0.78
DMA-09	USBM	0	270	105	170	65	0.50
DMA-10	USBM	0	106	0	55	55	0.76

Note:
(1) Drill hole composites over 0.5 g/t Au reported, >30 ft composite length and <15 ft of internal waste below 0.5 g/t Au.

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10 DRILLING

10.1 INTRODUCTION

The District has been drilled by numerous operators over the past 85 years. Table 10.1 below 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.

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	768	147,936	242	152,009	1,010	299,945
Hangar Flats	116	30,377	141	105,708	257	136,085
West End	889	208,260	55	38,907	944	247,167
Historic Tailings	25	1,512	47	3,786	72	5,298
Scout	18	6,912	21	15,629	39	22,541
Other	240	52,472	44	10,235	284	62,707
Totals	**2,056**	**447,469**	**550**	**326,275**	**2,606**	**773,744**

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 confirmation and definition, metallurgy, geotechnical engineering and condemnation drilling. The location of each mineralized area along with their associated drill hole collars can be found on Figure 10.1.

The Yellow Pine mineralized area has been drilled by 10 operators over the past 75 years and the total Yellow Pine database comprises approximately 299,945 ft of drilling in 1,010 holes. Drilling employed a variety of methods including core, RC, rotary, and air track (Table 10.1 and Figure 10.2). 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 85 years totaling approximately 136,085 ft of drilling in 257 holes (Table 10.1 and Figure 10.3). Drilling employed a variety of methods including surface and underground core, RC, rotary, and sonic. The pre-Midas Gold drilling was primarily performed in conjunction with underground mining operations.

The West End mineralized area has been drilled by six operators over the past 74 years and the total West End database comprises approximately 247,167 ft of drilling in 944 holes (Table 10.1 and Figure 10.4). 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 Historic Tailings area has been drilled by 2 operators over the past 20 years and the total Historic Tailings database comprises approximately 5,298 ft of drilling in 72 holes. Drilling employed a variety of methods including RC, sonic, and auger (Table 10.1 and Figure 10.5). Pre-Midas Gold drilling was conducted for well construction.

The Scout prospect has been drilled by 5 operators over the past 60 years and the total Scout database comprises approximately 22,541 ft of drilling in 39 holes. Drilling employed a variety of methods including core, RC, and air track (Table 10.1 and Figure 10.6). All drilling at Scout has been conducted as exploration drilling.

Figure 10.1: Mineralized Areas Showing All Drill Hole Collars

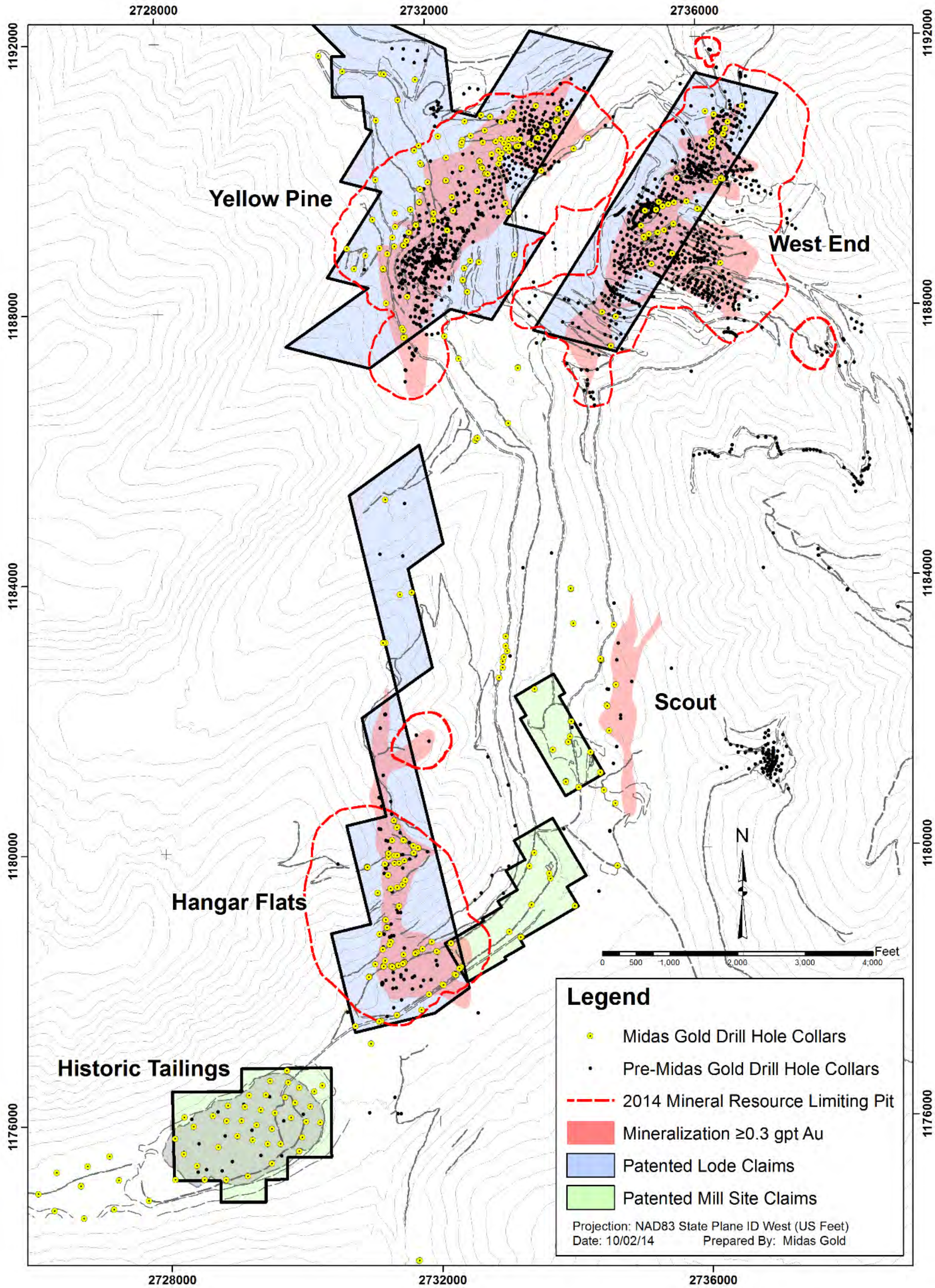


Figure 10.2: Yellow Pine Mineralized Area Showing All Drill Holes

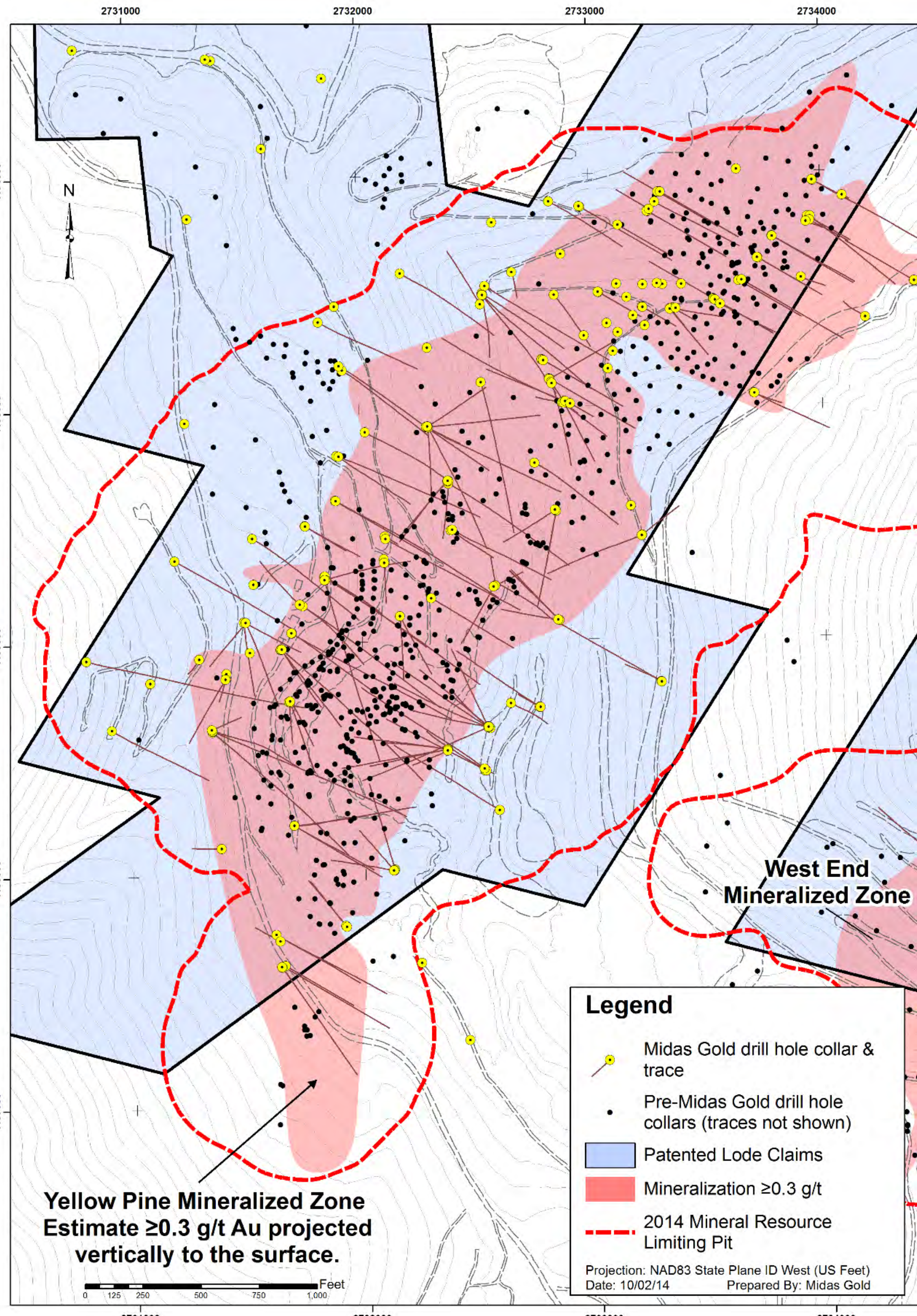


Figure 10.3: Hangar Flats Mineralized Area Showing All Drill Holes

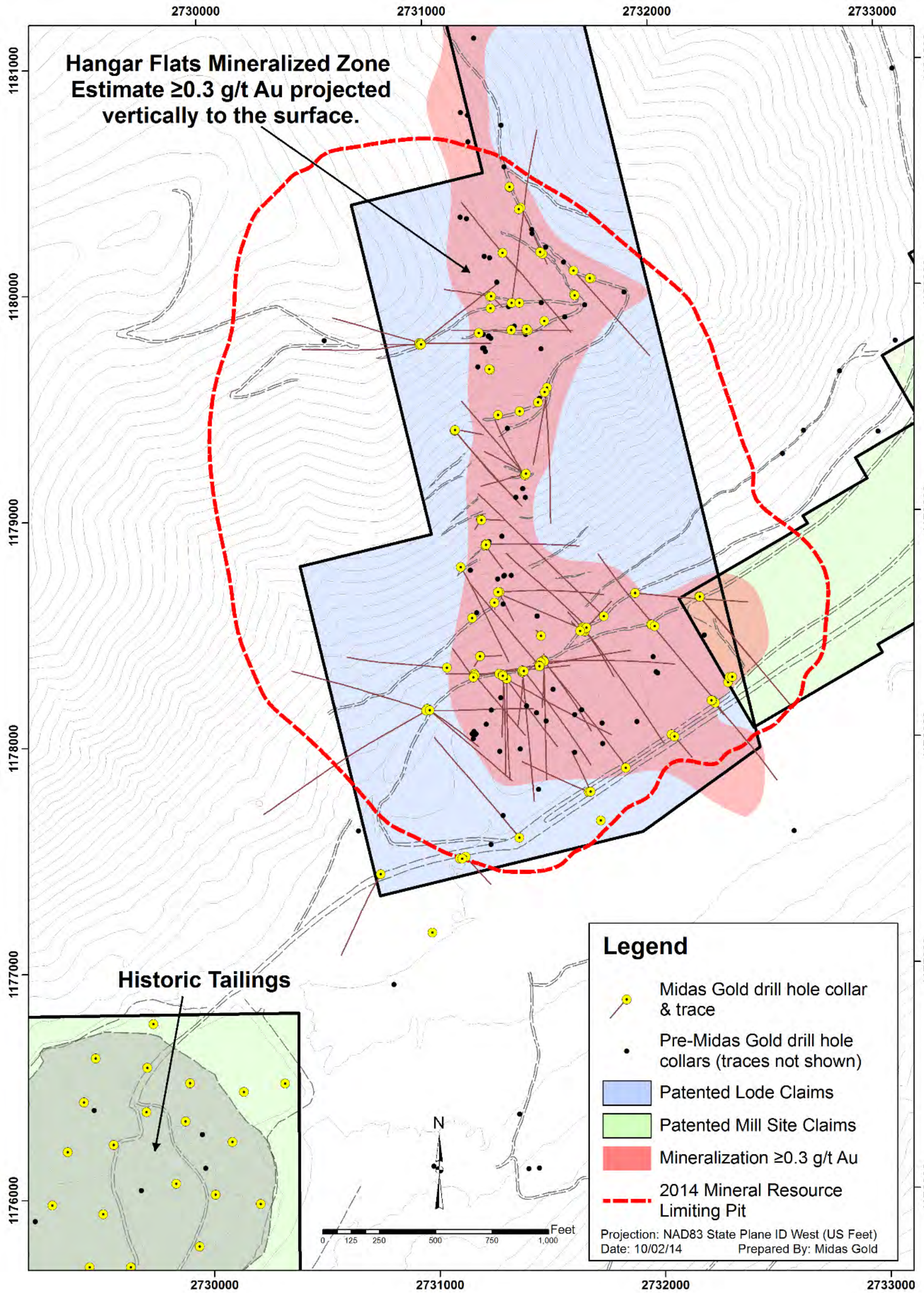


Figure 10.4: West End Mineralized Area Showing All Drill Holes

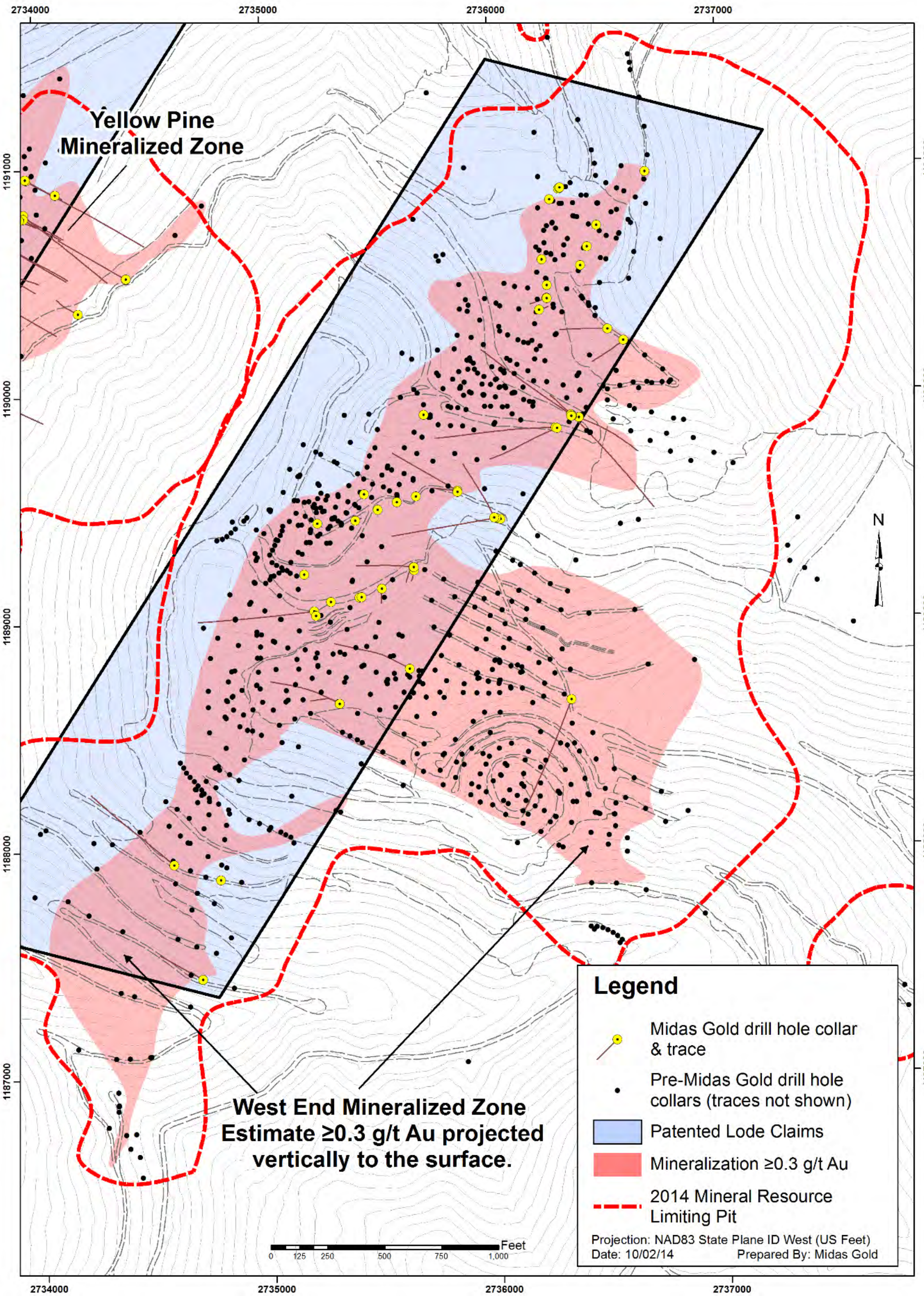


Figure 10.5: Historic Tailings Area Showing All Drill Holes

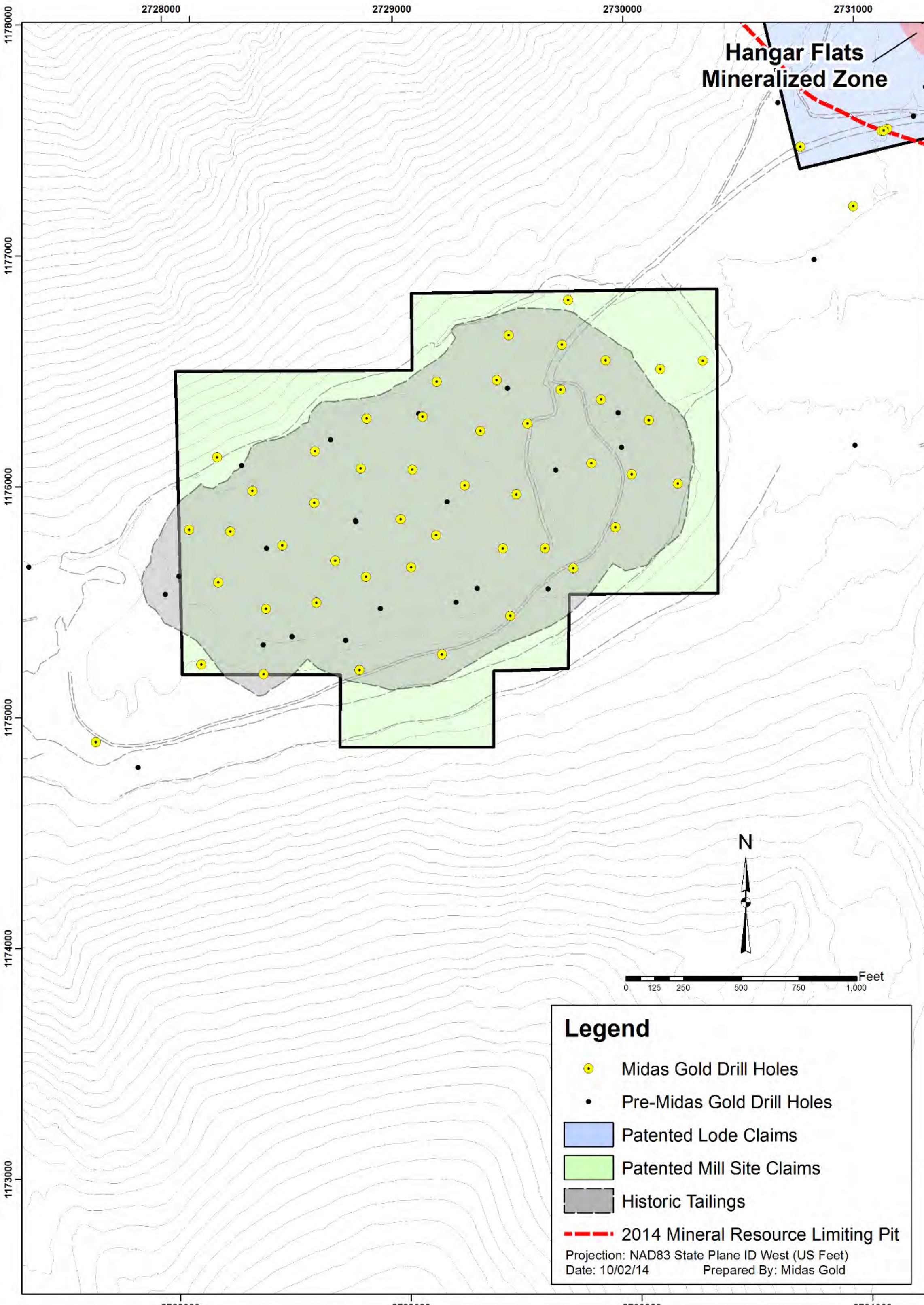
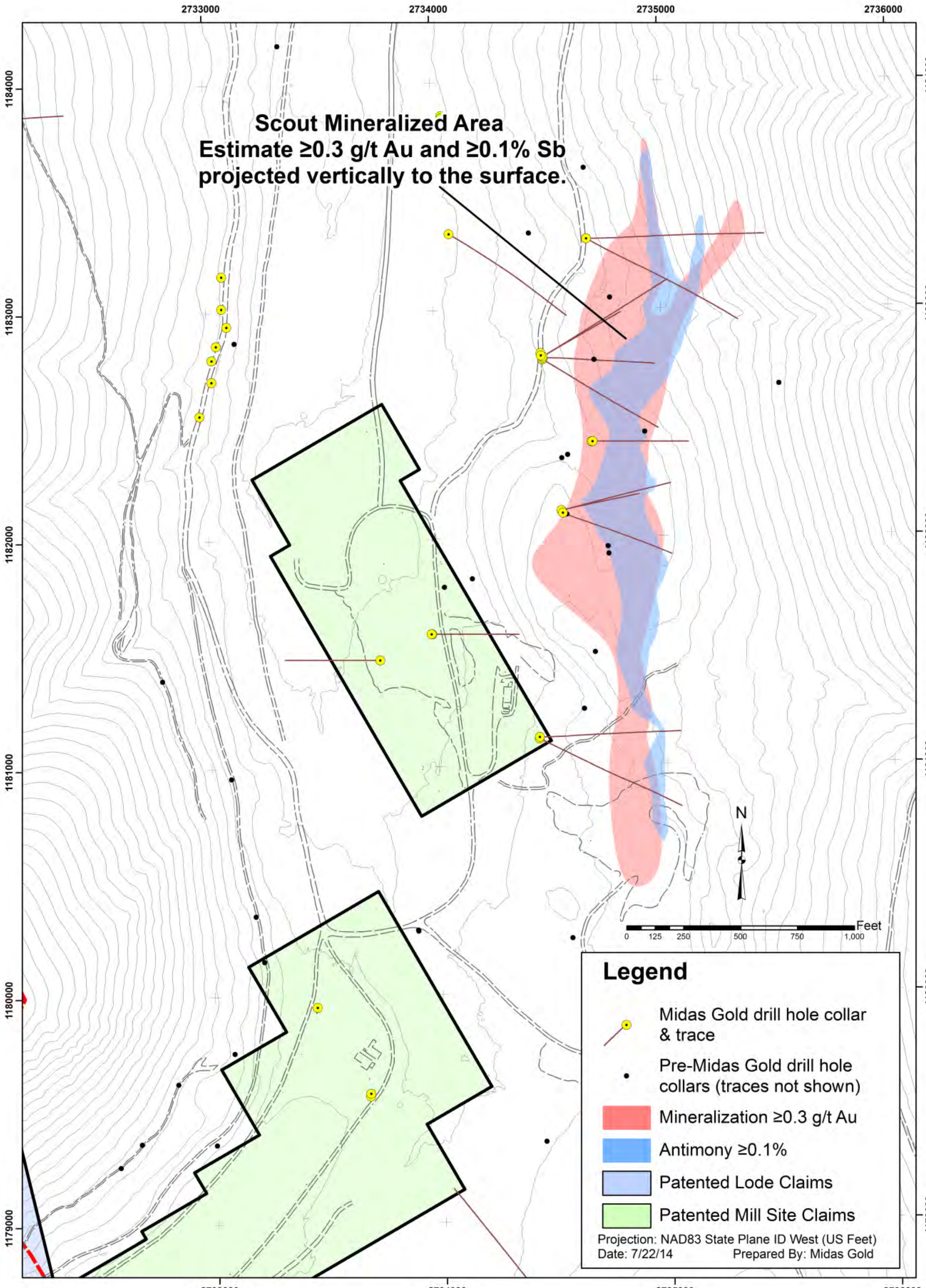


Figure 10.6: Scout Prospect Showing All Drill Holes



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. Twenty-nine percent of pre-Midas Gold holes were drilled vertically, 46% were drilled on a westerly to north westerly azimuth, the remaining were drilled on a southwest or southeast or easterly azimuth. Midas Gold holes were drilled on several different azimuths; 31% were vertical holes, 26% were drilled on a southeast azimuth, 19% on a northwest azimuth, 14% were drilled easterly, and 10% on a southerly azimuth.

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, and percussion holes. This chapter presents a discussion on pre-Midas Gold drilling followed by a discussion of Midas Gold drilling.

10.3 PRE-MIDAS GOLD DRILLING

The extent of pre-Midas Gold drilling varies significantly across the district and is broken out by individual areas (Yellow Pine, Hangar Flats, West End, Historic Tailings, and Scout) for further discussion.

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 database. 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 showing 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 often a space was reserved for assay values (Au, Ag, Sb, and W).

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 Chapter 11.

10.3.1 Yellow Pine

Past drilling within the Yellow Pine mineralized area was conducted with multiple methods by a number of different companies (Table 10.2). The historic Bradley and USBM drilling 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, comparable 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, 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-1980s, prior to that time there was no hole-abandonment remediation required for previous drilling.

Historic files do not always describe in detail the methods used for locating holes. However, 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. 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 significant 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.

Table 10.2: Pre-Midas Gold Drill Holes in the Yellow Pine Deposit

Year	Operator	Type of Drilling	Holes	Feet
1939	USBM	Core	6	1,331
1940	Bradley	Core	248	52,867
	USBM	Core	46	14,759
1946	Bradley	Core	17	3,411
1949	Bradley	Core	2	870
1950	Bradley	Churn	9	1,386
		Core	3	825
1951	Bradley	Churn	6	272
		Core	14	4,133
1952	Bradley	Core	1	371
1953	Bradley	Core	3	1,068
1954	Bradley	Churn	10	894
1973	Twin Rivers Resources	Core	1	229
	Ranchers	Core	6	820
1974	El Paso	Core	1	400
1978	El Paso	RC	1	50
	Superior	Air Track	4	158
		Rotary	16	2,027
1982	Ranchers	Core	63	12,196
1983	Ranchers	Rotary	26	5,580
1984	Ranchers	Core	9	1,193
		RC	55	7,850
1986	Pioneer	Air Track	4	275
		Percussion	5	845
		RC	2	450
1987	Hecla	RC	29	1,080
1988	Hecla	RC	63	13,584
	Pioneer	RC	1	380
1989	Hecla	Core	2	593
		RC	21	2,150
1991	SMI	RC	71	2,167
1992	Barrick	Core	14	11,427
		RC	3	1,655
1997	SMI	RC	6	640
		Totals	**768**	**147,936**

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

10.3.2 Hangar Flats

Past drilling within the Hangar Flats mineralized area was conducted with multiple methods by a number of different companies (Table 10.3). Drill core sizes, by pre-Midas Gold operators included AX, EX, BX and NX and were reduced as drilling conditions required. 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.

Historic files do not always describe in detail the methods used for locating holes. However, the operation was an active mine, and 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 historic 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.

Table 10.3: Pre-Midas Gold Drill Holes in the Hangar Flats Deposit

Year	Operator	Type of Drilling	Holes	Feet
1929	Bradley	Core	10	5,586
1940	Bradley	Core	28	6,207
1946	Bradley	Core	1	250
1947	Bradley	Core	3	961
1948	Bradley	Core	7	2,765
1952	USBM	Core	4	1,141
1953	Bradley	Core	4	2,448
	USBM	Core	8	2,528
1954	Bradley	Core	1	703
	USBM	Core	11	1,752
1955	USBM	Core	4	357
1974	El Paso	Core	1	399
		Rotary	1	200
1975	El Paso	Core	3	833
1982	Superior	Air Track	8	412
1988	Hecla	RC	5	935
1989	Hecla	RC	17	2,900
		Totals	**116**	**30,377**

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

10.3.3 West End

Past drilling within the West End mineralized area was conducted with multiple methods by a number of different companies, all of which were reputable industry operators or contractors (Table 10.4). Core drilling was much less common than RC and Air Track drilling, but core sizes included HQ and NX. 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. Sample was collected through both center return hammers and also with conventional above-hammer RC interchanges, and then traveled up the center of the drill string so that minimal down hole and cross 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 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.

Table 10.4: Pre-Midas Gold Drill Holes in the West End Deposit

Year	Operator	Type of Drilling	Holes	Feet
1940	Bradley	Core	2	370
1973	Twin Rivers Resources	Core	4	1,167
1975	Superior	Core	2	607
1976	Superior	Core	17	6,661
		RC	12	1,080
1977	Superior	Air Track	58	4,995
		Core	24	6,618
1978	El Paso	RC	1	100
	Superior	Air Track	92	8,990
		RC	50	9,608
1981	El Paso	Air Track	35	1,660
	Superior	RC	9	1,750
1982	Superior	Air Track	26	1,131
1983	Superior	RC	45	11,219
		Rotary	28	3,124
1984	Superior	RC	12	2,653
1986	Pioneer	RC	31	6,913
1987	Pioneer	Air Track	8	470
		RC	76	16,670
1988	Pioneer	RC	48	20,180
1989	Pioneer	RC	79	32,939
1990	Pioneer	RC	32	9,200
1991	Pioneer	RC	32	11,615
1992	Pioneer	RC	42	11,240
1995	SMI	Core	2	305
1996	SMI	Core	1	374
		RC	59	20,780
1997	SMI	RC	62	15,840
		Totals	**889**	**208,260**

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

10.3.4 Historic Tailings

Pre-Midas Gold drilling within the Historic Tailings area was conducted primarily for water quality monitoring purposes. Stibnite Mines Inc. is the only known operator to have drilled in this area and they used both RC and auger drilling techniques. They drilled 25 holes totaling 1,512 ft in 1994 and 1996.

10.3.5 Scout

Past drilling in the Scout area was conducted with multiple methods by a number of different companies (Table 10.5). Bradley generally drilled AX, EX and BX core while Pioneer and El Paso drilled BQ, BX, NX, and HQ. According to existing drill logs the overburden thickness in this area is variable and in some instances operators were forced to abandon drill holes as a result. There was no hole-abandonment remediation required at the time of the previous drilling.

Historic 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. However, most collars were typically not preserved.

Table 10.5: Pre-Midas Gold Drill Holes in the Scout Area

Year	Operator	Type of Drilling	Holes	Feet
1947	Bradley	Core	3	660
1948	Bradley	Core	1	405
1954	Bradley	Core	4	1,532
1955	Bradley	Core	2	1,123
1974	El Paso	Core	3	1,148
1975	El Paso	Core	2	1,289
1978	Superior	Air Track	1	40
1990	Pioneer	RC	2	715
		Totals	**18**	**6,912**

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

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 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 1927 Idaho State Plane.

Midas Gold has used two separate methods for grid conversion from historic coordinate systems. From the Project inception until 2013, coordinates were converted by first converting historic coordinates into the Hecla grid, then into 1927 Idaho State Plane, and finally into Universal Transverse Mercator North American Datum of 1983 (**UTM NAD83**) using software (Trimble Surveying Pathfinder and Aspen Software, Golden Software Surfer and/or ESRI ArcGIS). 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. Converted UTM collar coordinates were then plotted and compared to historic maps for hole placement and consistency. Several errors were addressed and in every instance the source of the error was discovered and corrected. The Midas Gold database uses the converted coordinates for plotting purposes, but retains original information for any future verification purposes.

10.4 MIDAS GOLD DRILLING

In addition to the drilling performed by previous operators, Midas Gold has drilled 550 holes totaling approximately 326,274 ft of drilling distributed throughout the district during 2009 through 2013 (Table 10.6). Out of these holes, 506 were drilled within the current mineralized areas totaling approximately 316,039 ft. Drilling within the deposits by Midas Gold was planned to satisfy three main goals: (1) exploration of the Hangar Flats and West End mineralized areas (2) to confirm pre-Midas Gold drilling data and (3) to add infill drilling in the main deposit areas. Holes were drilled with RC, core, sonic, air track, and auger both as vertical and angle holes. Drill core sizes were generally HQ and NQ with a few PQ holes. Drill hole azimuths varied in all directions with dips ranging from -20° to -90°. The RC drill typically used a down-hole hammer with a 5.5-inch bit. Samples were collected through both center return hammers and also with conventional above-hammer RC interchanges, and then traveled up the center of the drill string so that minimal down hole and cross contamination could occur.

At Yellow Pine, the drilling has defined a large zone of anomalous gold, antimony and silver mineralization within the MCFZ and adjacent intrusive and, local sedimentary units. The mineralization is interpreted to follow two main orientations controlled by the MCFZ with related fractures and faults, and east-striking, high angle intersecting faults. The drill holes are generally located in a wide range of orientations with approximately 80 – 160 ft spacing within the deposit. They are typically oriented to the southeast or northwest, inclined steep to moderate. This orientation provides an oblique angle of intersection between the predominant planes of mineralization and the drill hole. Based on the wide range of drill hole orientations, some of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is equal to or less than the true thickness of mineralization, as mineralization is broadly disseminated over significant widths and many drill holes bottom in mineralization (refer to Figure 7.5).

At Hangar Flats, drilling has defined a large zone of anomalous gold, antimony and silver mineralization within the MCFZ and adjacent intrusive units. The mineralization is interpreted to follow two main orientations controlled by both the MCFZ and northeast striking low angle splay faults. The drill holes are generally located in a wide range of orientations with approximately 100 - 210 ft spacing. The holes typically bear to the east or west, with some to the south, and are generally steeply to moderately inclined. This orientation provides an oblique angle of intersection between the predominant planes of mineralization and the drill hole. Based on the wide range of drill hole orientations, most of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is greater than the true thickness of mineralization (refer to Figure 7.6).

At West End, drilling has defined a large zone of anomalous gold mineralization within the WEFZ and adjacent lithologic units. The mineralization is interpreted to follow two main orientations controlled by both the fault planes and stratigraphy. The drill holes are generally arranged in parallel at 65 - 100 ft spacing on section lines and inclined steeply to the northwest along parallel sections 100 ft apart. This orientation provides a high angle of intersection between the predominant structural plane of mineralization and the drill hole. Based on the wide range of drill hole orientations, most of the sample lengths do not represent true thickness of mineralization. In general, the drill hole intercept length is greater than the true thickness of mineralization (refer to Figure 7.7).

In the Historic 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 cross-cut the tailings perpendicular to the body. Due to the horizontal body of tailings being drilled by vertical holes, drill hole intercepts represent true thickness.

At Scout, drilling has defined a north/south trending near vertical zone of antimony, gold and silver mineralization within and adjacent to the near vertical Scout fault. Mineralization is hosted within fracture zones and is not constrained to a specific lithology. The majority of drilling is widely spaced (approximately 275 – 400 ft) and is oriented to the east in attempts to drill across the main mineralized zone to obtain true thickness. In general, the drill hole intercept length is greater than the true thickness of mineralization.

Table 10.6: Drilling by Area Completed by Midas Gold

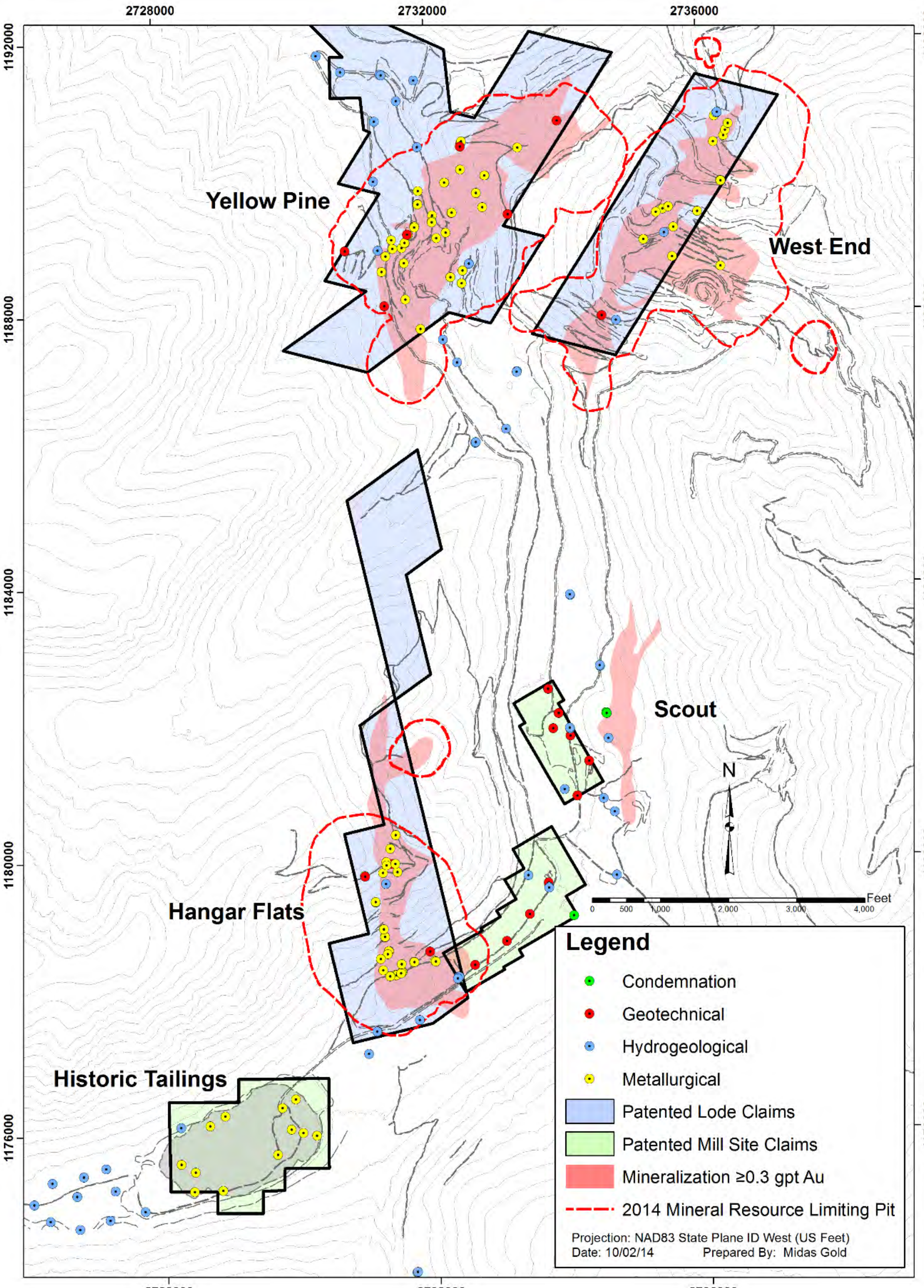
Hole Type	Year	# Holes	Feet
Yellow Pine			
Core	2011 - 2013	168	121,233
RC	2011 - 2012	62	29,312
Sonic	2011 - 2012	9	1,050
Air Lift	2012	3	414
	Totals	**242**	**152,009**
West End			
Core	2010, 2012 - 2013	34	28,837
RC	2011 - 2012	18	9,493
Air Lift	2012 - 2013	3	577
	Totals	**55**	**38,907**
Hangar Flats			
Core	2009 - 2013	102	86,798
RC	2012	23	15,675
Sonic	2011 - 2012	10	2,286
Air Lift	2012	6	949
	Totals	**141**	**105,708**
Historic Tailings			
Sonic	2011 - 2012	4	520
Air Lift	2012	1	60
Auger	2013	42	3,206
	Totals	**47**	**3,786**
Scout			
RC	2011 - 2012	5	4,310
Core	2012 - 2013	16	11,319
	Totals	**21**	**15,629**

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

10.5 GEOTECHNICAL AND HYDROLOGICAL DRILLING

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. More recently Midas Gold drilled a total of 13 bedrock geotechnical holes, 10 soil geotechnical holes, and 62 holes for water monitor well installations (Figure 10.7). Four other core holes were also used for multi-level sampler installations for hydrogeological purposes. Technical consulting firms were contracted to plan and manage the geotechnical and hydrogeological programs. SRK Consulting was hired to oversee the geotechnical and hydrogeological drill campaign in 2011 - 2012 and MWH Global took over the hydrogeological campaign in 2012 - 2013.

Figure 10.7: Hydrological, Geotechnical, Metallurgical, and Condemnation Drilling



Several of the holes drilled for geotechnical analysis in soils were also used to install water monitor wells; similarly, several of the holes drilled for water monitoring wells were also used for geotechnical analysis. For example, the hydrogeological holes shown on Figure 10.7 around the Historic Tailings area are currently being used for water monitoring purposes and are shown as such; however, these holes were drilled specifically to evaluate the geotechnical conditions beneath the potential Tailings Storage Facility. Geotechnical holes included bedrock core holes and overburden soil holes drilled with core, sonic, air track, and auger rigs. Water monitor wells were drilled with core, sonic, and RC rigs.

Core for the 13 bedrock geotechnical holes was drilled with split tubes and oriented using a Reflex ACT II tool. Whenever the drill was operating, geologists were onsite to log geotechnical data as the core was retrieved from the hole. Piezometers were installed down six of these holes and are presently collecting down hole data (Figure 10.7).

Monitor well holes included bedrock and alluvial wells. Well installation was carried out by SRK Consulting and MWH Global. Many of the water monitor wells were also logged for soil geotechnical purposes.

Ten auger holes were drilled in the vicinity of the potential plant site. These holes were drilled and logged solely for geotechnical purposes to aid in infrastructure construction considerations.

10.6 METALLURGICAL DRILLING

Midas Gold drilled 11 holes specifically to collect samples for metallurgical sampling. Core size for these holes was PQ but was sometimes reduced to HQ as drilling conditions required. Quartered core from these holes was assayed for use in mineral resource estimation and the remainder submitted or retained for metallurgical work with a portion archived in the Midas Gold core storage facilities.

Additionally, samples were taken from 99 other core and auger holes to be used for metallurgical testing (Figure 10.7). These holes were generally drilled with HQ core and were selected to provide representative samples from each of the advanced stage deposits (Yellow Pine, West End, Hangar Flats, and Historic Tailings).

10.7 CONDEMNATION DRILLING

Condemnation drilling completed by Midas Gold focused on two specific areas of potential infrastructure and consisted of 4 holes (Figure 10.7). These holes were drilled with HQ tooling. Three holes were drilled at Scout from the same drill pad near the current shop and camp as dual-purpose condemnation/exploration testing a potential deposit. A 9-hole exploration drill program followed in this area. A single hole was also drilled south of Scout within potential infrastructure areas for condemnation.

10.8 GEOLOGICAL LOGGING

Geologic logging performed by Midas Gold utilized paper log sheets in 2009 - 2010 and digital logging methods in 2011 - 2013. 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 - 2013, preliminary core logging was completed on site and detailed logging was completed at the core logging facility in Lake Fork, ID. Preliminary geological logging performed at Stibnite after core was received from drillers identified general geology and alteration for hole-tracking and daily reporting purposes. This was followed up with detailed geologic logging 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.

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.

10.9 DRILLING RECOVERY

In general, both RC and core recovery was good for all drilling completed by Midas Gold. Core recovery averaged 92.6 %, 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 significant 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 resource estimation.

10.10 ROCK QUALITY DESIGNATION

Rock Quality Designation (**RQD**) is a measure of naturally occurring fractures in a rock and was applied to all core drilled by Midas Gold since the beginning of the 2010 drilling program, starting with hole MGI-10-12. Approximately 95% of Midas Gold's core drill holes were measured for RQD between 2009 and 2013. 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.

10.11 DRILL HOLE COLLAR SURVEYS

During the 2009-2013 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. Between 2009 - 2012, a professional surveyor was used to survey the final collar locations. In 2013, a high-precision GPS was utilized by onsite geologists to survey final drill hole collar coordinates with an estimated accuracy of ± 2 ft.

10.12 DOWN HOLE SURVEYS

Down hole surveys were performed on core holes using a number of survey instruments including: an acid etch clinometer, tropari, and 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 entered into GEMS Logger by the logging geologist for each individual hole. Magnetic declination corrections were applied by GEMS logger based on the declination correction entered by the logging geologist. Declination corrections were modified at least annually based on changing magnetic declination.

10.13 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.

In general, the deposits show broadly disseminated mineralization with a few major structural boundaries. As these are not vein deposits, the term "true thickness" is difficult to apply due to the entirety of the rock mass being potential ore-grade material. However, an attempt was made to drill across major structures to test their effect on mineralization. Based on the wide range of drill hole orientations, many of the sample 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. At Yellow Pine, the drill hole intercept length is generally less than true thickness.

10.14 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 Lake Fork, ID. Once in Lake Fork, core and cuttings were stored within the Midas Gold core logging facility which is supervised during the work day and locked when vacant (nights and weekends). After core was logged and sampled, the remaining halved core was stored either in Midas Gold's Lake Fork buildings, in Midas Gold's Cascade warehouse, 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 at the Lake Fork facility. Rejects are stored inside of the chain-link fence at the warehouse in Cascade. Additionally, some core being held for future metallurgical sampling is being stored in a secure refrigerated truck behind the chain-link fence in Cascade. All storage locations remain locked when no Midas Gold personnel are present and have restricted access. In Cascade, both the fence and the warehouse remain locked.

Throughout 2014 Midas Gold relocated their Lake Fork facilities. Employee office facilities were moved to their current location in Donnelly, ID. Consequently all core, pulp, rejects, and additional samples at their Lake Fork facility have been moved to secure indoor and outdoor storage at the Cascade warehouse detailed above. All storage locations remain locked when no Midas Gold personnel are present.

Going forward, Midas Gold intends for all samples to be cut and logged at their onsite facilities and stored at their indoor and outdoor Cascade warehouse facilities.

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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.

11.1 SAMPLING METHODS

Throughout the last 85 years, multiple drilling and sampling methods have been used across the district by pre-Midas Gold operators as well as Midas Gold. Sampling methods have varied based on the era and the type of drilling.

11.1.1 Pre-Midas Gold Sampling

Early operators generally sampled drill core and sludge while later operators drilled and sampled either core or reverse circulation chips. Later operators used modern wire-line core drilling methods resulting in better core recovery. Reverse Circulation (**RC**) drill holes were drilled under both wet and dry conditions and samples collected from a cyclone or similar splitter. Sample lengths, regardless of the drilling campaign, 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 rig by the rig geologist and typically included 1 certified standard, 1 blank and 1 cyclone splitter reject every 100 ft. (i.e. every 20th sample). Sample bags were placed into larger rice bags which were placed into bulk storage sacks and shipped to Lake Fork, ID for shipping to the laboratory. Pre-numbered bar codes were utilized for sample numbering.

11.1.3 Core Drill Sampling

From the beginning of the core drilling program in 2009 through 2011, core was generally sampled on 5 ft intervals with sample breaks made at significant changes in lithology or intensity of alteration and/or mineralization. Beginning in 2012, sample intervals for core were based on the logging geologist's interpretation of the intensity of mineralization for example; 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 40 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.

Auger samples were collected in a split tube and split 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 in 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.

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 accompanied by 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 Lake Fork 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 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 Lake Fork 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 Lake Fork 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 taped shut with tamper-proof security tape. 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 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 3,196 intervals. Four hundred ninety-six 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 0.98%, indicating there was no significant difference 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. 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 2014. A total of four labs have been used in the United States and Canada for primary and check assays.

Table 11.2: Analytical Laboratories Used by Midas Gold

Laboratory	Location	Certification/ Accreditation	Use	Year
ALS	Elko, Reno, and Winnemucca, NV, USA; Vancouver, BC, Canada	ISO 17025:2005 ISO 9001:2008	Primary Lab 2009-Present	2009 - 2014
American Analytical Services (AAS)	Osburn, ID, USA	ISO 17025	Check Assays	2010, 2012-2013
Inspectorate	Reno, NV, USA	ISO 9001:2008	Check Assays	2009, 2012-2014
SGS Canada, Inc.	Vancouver, BC, Canada	CAN-P-1579 17025:2005	Check Assays	2014

11.4.2 Metallurgical and Geochemical Laboratories

Table 11.3 summarizes the laboratories used by Midas Gold for analysis from 2010 to 2014. A total of eight labs have been used in the United States and Canada for metallurgical and geochemical testing.

Table 11.3: Metallurgical and Geochemical Testing Laboratories Used by Midas Gold

Laboratory	Location	Certification/Accreditation	Use	Year
SGS Canada, Inc.	Lakefield, ON, Canada; Vancouver, BC, Canada	CAN-P-1579, CAN-P-1587, CAN-P-4E (ISO/IEC 17025:2005)	Primary Metallurgical Testing Lab	2010-2014
Kingston Process Metallurgy, Inc.	Kingston, ON, Canada	n/a	Metallurgical Testing	2013-2014
Pocock Industrial, Inc.	Salt Lake City, UT, USA	n/a	Metallurgical Testing	2013-2014
McClelland Laboratories	Sparks, NV, USA	EPA ID #: NV00933	Geochemical Testing	2012-2014
Western Environmental Testing Laboratory	Sparks, NV, USA	EPA ID #: NV000925	Geochemical Testing	2012-2014
SVL Analytical	Kellogg, ID, USA	EPA ID #: ID000019	Geochemical Testing	2013-2014
Inter-Mountain Laboratories	Sheridan, WY, USA	EPA ID #: WY00005	Geochemical Testing	2014
Dynatec Labs	Fort Saskatchewan, Alberta, Canada	ISO/IEC 17025; 2005	Metallurgical Testing	2012

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 being pulverized to 70% passing a ¼ inch mesh (6 mm) and dried at a maximum of 140 degrees Fahrenheit (60 degrees Celsius). Dried material was split and crushed 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 was done by a 4-acid digestion followed by inductively coupled plasma atomic emission spectroscopy (**ICP-AES**) for 33 elements with an Hg add-on. 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 mixed well and fused in an auto fluxer. A disc was prepared from the melt and analyzed using X-ray fluorescence spectroscopy with a lower detection limit of 0.01% (100 ppm) and an upper reporting limit of 50%. 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.

All gold assays were performed using a 30 g fire assay charge followed by an atomic absorption spectroscopy (**AAS**) 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 a 4-acid digestion followed by 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

The Midas Gold database administrator has multiple measures in place to check the database for errors. Interval verification tools are run regularly to check for intervals that are overlapping or out of sequence. Digital assay data are received from the primary assay laboratory and are imported directly into the database. 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.

As part of the development of the PEA, SRK checked 88% (28,692 records) of the assay intervals in the database versus original data files delivered by ALS, from the first drill hole, MGI-09-01, to MGI-12-210 (210 total holes) and found 14 errors in assay data which were all corrected. Also as part of the PEA, SRK verified the core logging information in the database of 21 random holes versus the original logs as filled out by the core logger. Post PEA database verifications are summarized in Section 12.

11.7 QUALITY ASSURANCE AND QUALITY CONTROL

Pre-Midas Gold operators conducted various QA/QC programs for both their drilling and mine assay operations. Some records of QA/QC measures may not have survived to be reviewed by Midas Gold. However Section 11.7.1 details the records that Midas Gold has collected and catalogued.

Midas Gold exercised strict and rigorous QA/QC protocols throughout the different drilling campaigns. Periodically these protocols were assessed for adequacy and improved accordingly.

11.7.1 QA/QC Pre-Midas Gold

Pre-Midas Gold operators had varying QA/QC programs, but not all records have survived. QA/QC data available for each operator from existing records are detailed in Table 11.4, where insertion rate is detailed for drilling conducted by that operator. Historic reports indicate that Bradley used duplicates and standards as QA/QC measures at Hangar Flats, but exact insertion rates are unknown.

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) Percentages 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 (2009-2012)

Midas Gold initially created a QA/QC program in 2009 to provide adequate confidence in the data collection and processing. As part of the development of the PEA, SRK consulting examined the performance of Midas Gold's QA/QC program from 2009 through June 2012. SRK determined that performance of certified blanks was good but the locally sourced in-house blanks performed poorly. No significant bias was determined to be present in core duplicates. RC field rejects were determined to exhibit no significant bias and good reproducibility. No evidence of systematic analytical bias was observed in standards.

11.7.3 QA/QC by Midas Gold (2012-2013)

Following review by SRK in 2012, Midas Gold revised its QA/QC program. New protocols incorporated additional blank materials and certified antimony standards, discontinued use of non-certified reference materials, increased insertion rates for antimony standards, increased use of second lab check assays, and initiated use of blind reject samples. Table 11.5 shows the insertion rates of various QA/QC measures used in Midas Gold drilling since the beginning of 2012, which may overlap slightly with the period reported in the PEA. QA/QC measures are described in sections below. Insertion rates were determined within mineralized zones that were defined by gold grades greater than 0.5 g/t.

Table 11.5: Midas Gold QA/QC Measures and Insertion Rates within Mineralized Zones

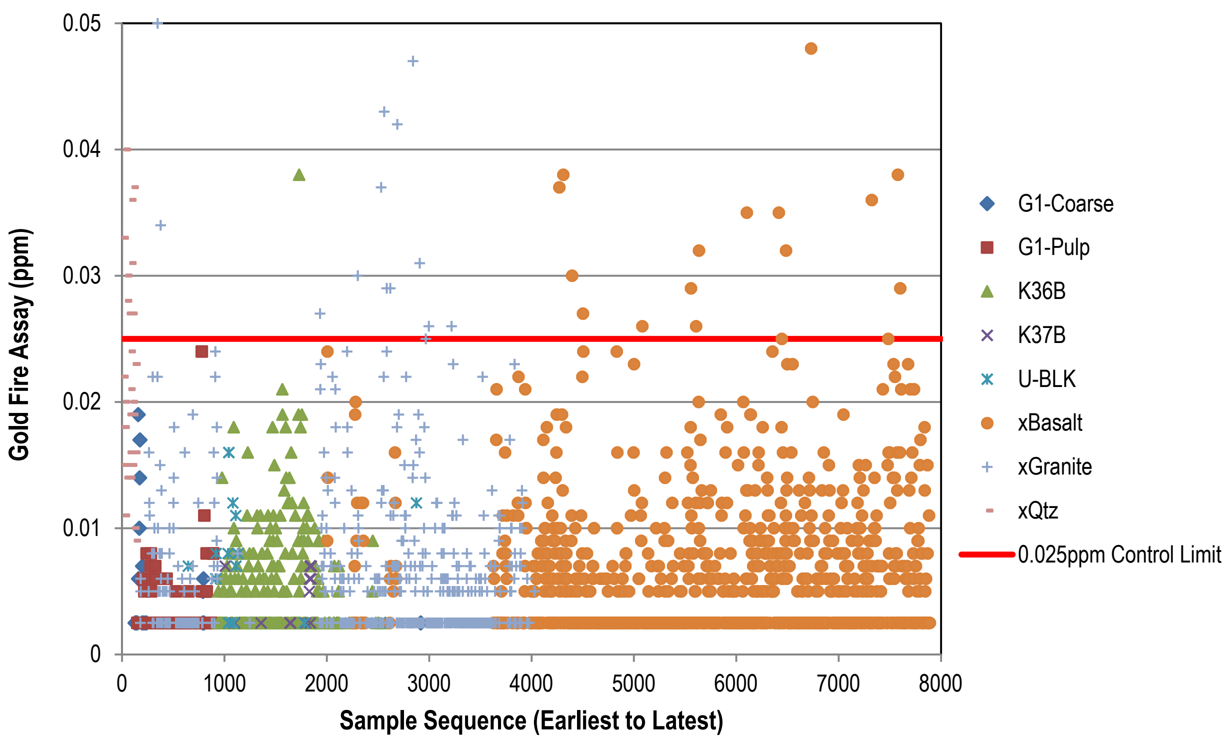
Year	Deposit	Assays	Field Duplicates	Pulp Duplicates	Check	Reject	Standard	Blank	Totals
2012	Yellow Pine	1300	2.77%	5.38%	21.00%	3.62%	6.31%	4.92%	44.00%
	Hangar Flats	364	3.30%	3.57%	7.42%	4.40%	7.14%	5.77%	31.60%
	West End	406	4.93%	5.42%	14.53%	5.17%	5.42%	4.68%	40.15%
	Scout	643	4.98%	5.44%	1.87%	0.00%	6.53%	4.67%	23.49%
2013	Yellow Pine	1117	4.66%	5.64%	3.67%	3.22%	9.13%	4.39%	30.71%
	Hangar Flats	194	5.15%	5.15%	3.61%	3.61%	9.79%	4.12%	31.43%
	West End	64	4.69%	7.81%	6.25%	6.25%	4.69%	7.81%	37.50%
	Historic Tailings	557	0.00%	4.85%	8.44%	0.00%	7.36%	1.97%	22.62%
	Scout	9	0.00%	0.00%	0.00%	44.44%	0.00%	0.00%	44.44%

11.7.4 Blanks QA/QC

Midas Gold used a total of 2,374 blanks in the sample stream, 312 of which were certified (Figure 11.1). Non-certified in house blanks were composed of locally sourced, unmineralized quartzite, basalt, and granite.

Upon evaluation, blanks reporting values below 0.025 ppm Au were considered satisfactory. Certified blanks reported 100% of values under this limit and non-certified blanks reported 86% of values under this limit. The only blank material utilized since the PEA was locally sourced Miocene basalt which exhibited a failure rate of 1.1%, with two samples assaying and re-assaying above 0.5 ppm gold.

Figure 11.1: Blank Performance – Gold



11.7.5 Standard Reference Materials QA/QC

In post-PEA drilling, Midas Gold began to decrease the use of non-certified gold standards, as well as increase use of certified antimony standards. Insertion rate of standards typically exceeded 5% for drilling within all deposits. Midas Gold used a total of 1,588 certified gold standards, 1,021 non-certified gold standards, and 499 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 95% of values within satisfactory limits, non-certified gold standards reported 96% of values within satisfactory limits, and certified antimony standards reported 99% of values within satisfactory limits.

Figure 11.2: Certified Gold Standards

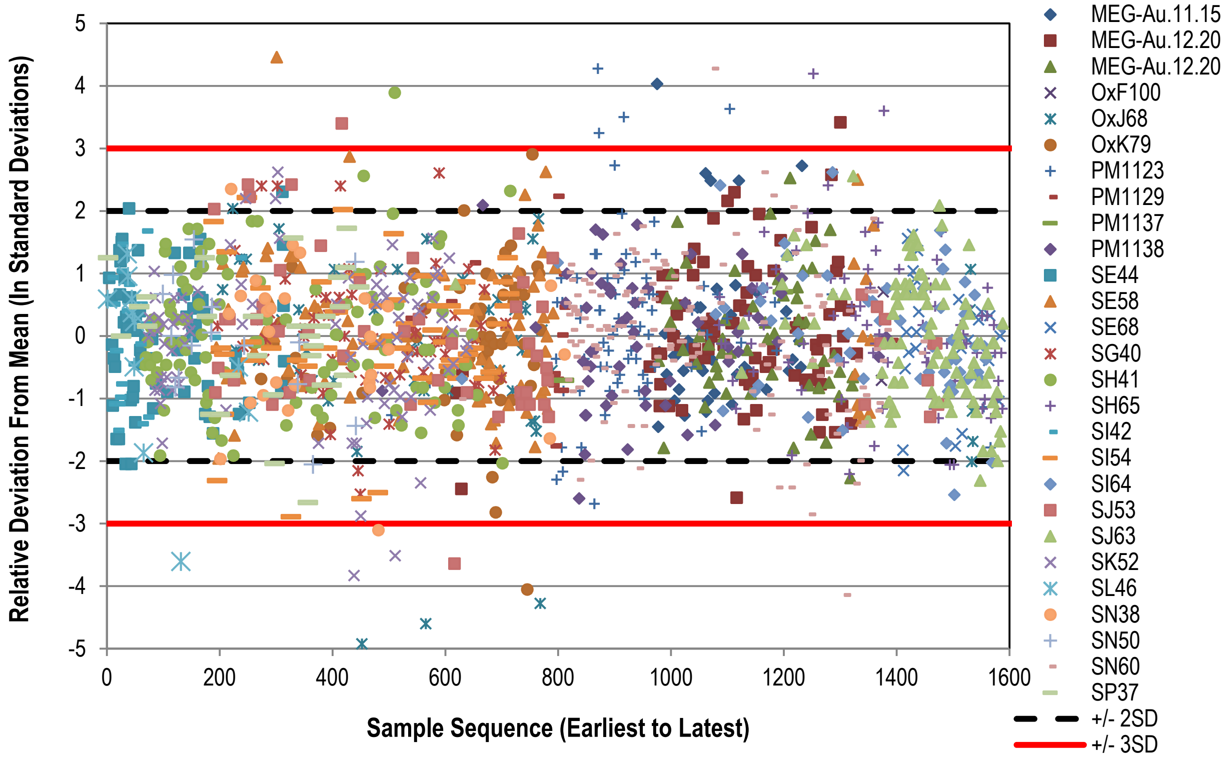
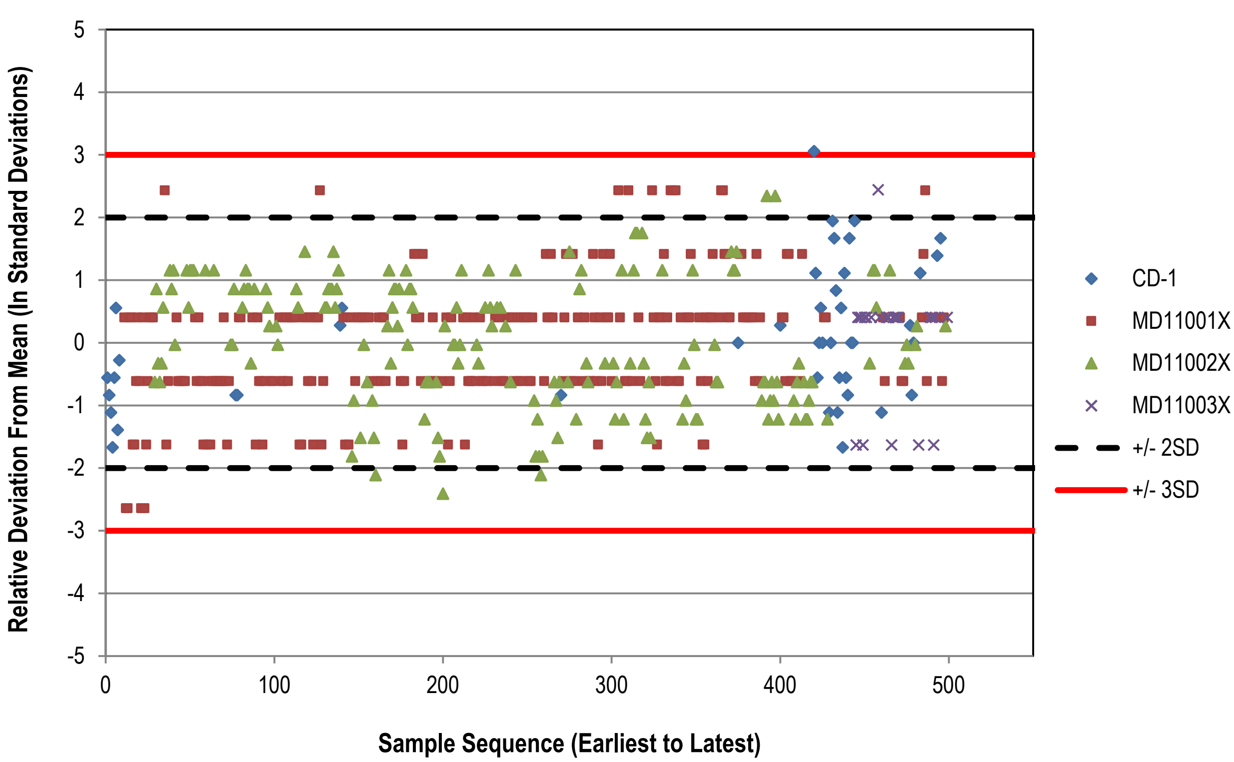


Figure 11.3: Certified Antimony Standards

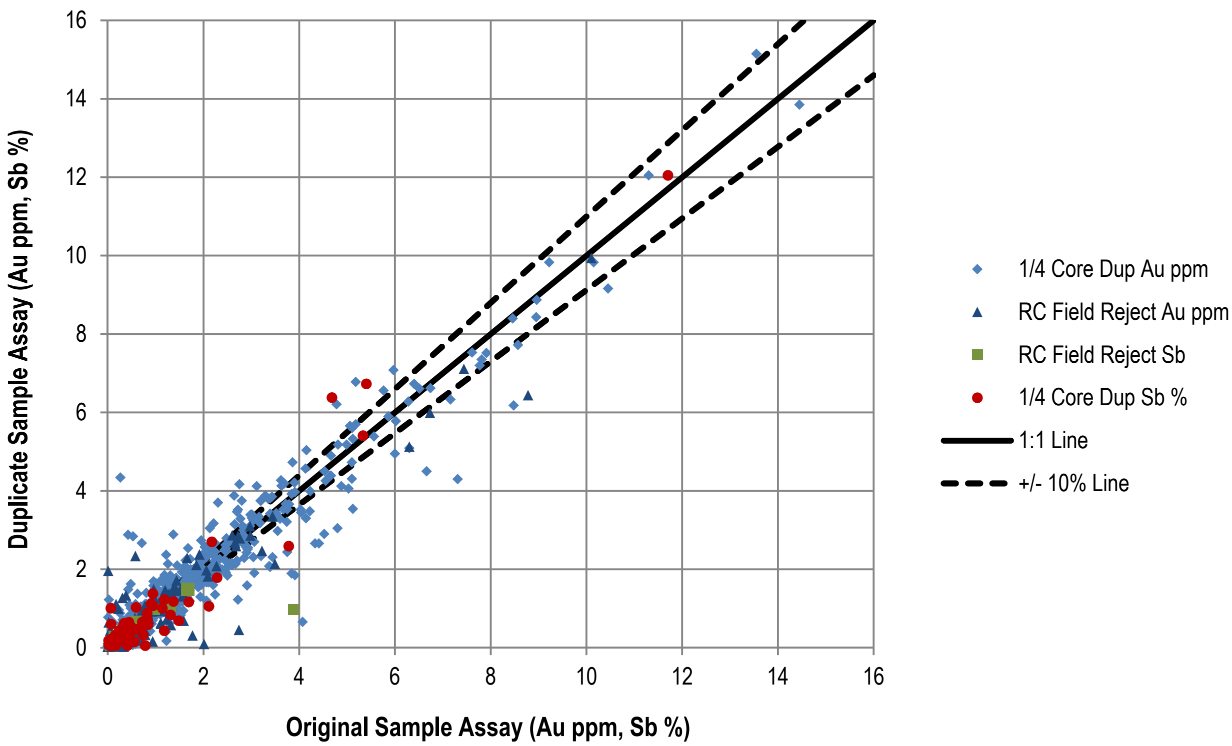


11.7.6 Field Duplicates QA/QC

Midas Gold generated 1,778 quarter core duplicates from core holes of which 1,055 were above 0.025 ppm by gold fire assay and 102 were above 0.05% antimony. Reproducibility for quarter core duplicates was fair for both gold and antimony with a RMS CV of 21.1% for gold and 29.7% for antimony however the correlation coefficients for both are excellent at 0.94 (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 warrants investigation in the future and should be considered for planning of future drill programs. As there are only 19 data points with which to compare for the antimony and the removal of one outlier bringing the correlation coefficient from 0.33 to 0.97, it appears that there is good agreement. Future anomalous values should be re-assayed.

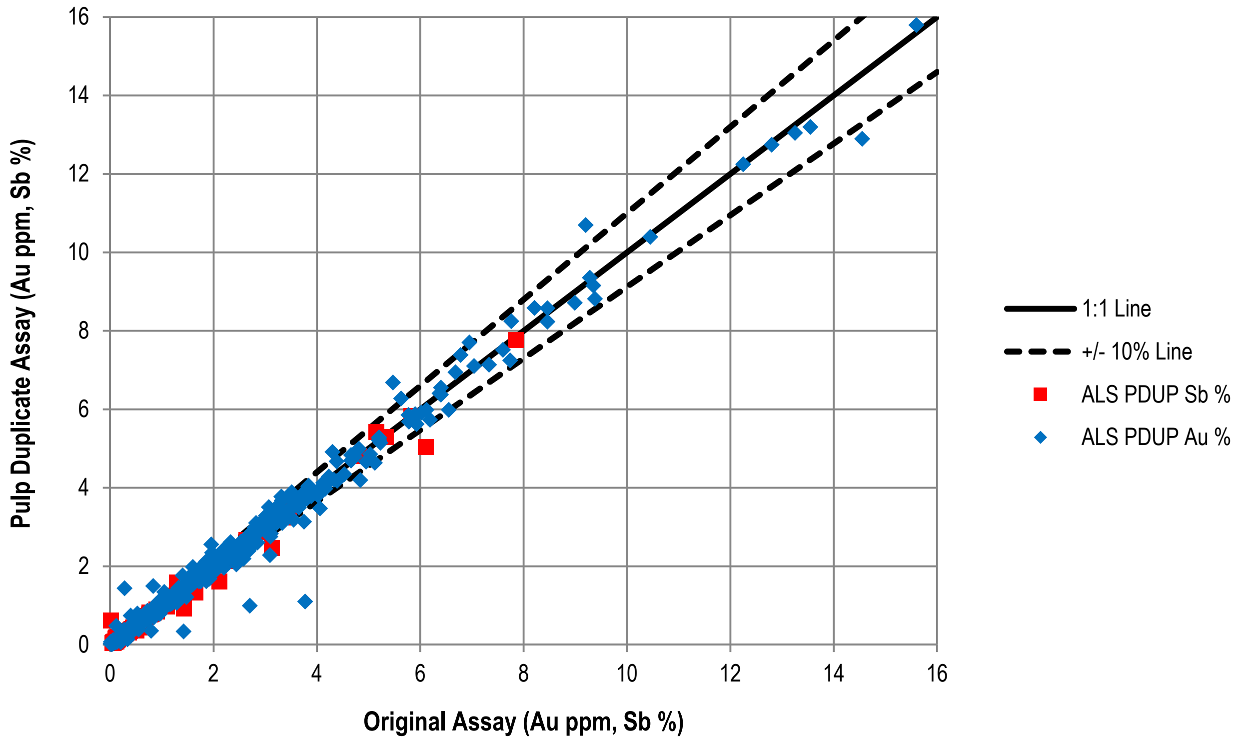
Figure 11.4: Field Duplicates



11.7.7 Pulp Duplicates QA/QC

ALS prepared one pulp duplicate for every twenty samples submitted. A total of 2,896 pulp duplicates were produced and assayed of which 1,646 were above 0.025 ppm for gold and 143 were above 0.05% antimony. Reproducibility for pulp duplicates was excellent for gold with an RMS CV of 6.8% and reproducibility was good to moderate for antimony with an RMS CV of 10.3%. Figure 11.5 shows scatter plots of the original assay values versus the pulp duplicate values.

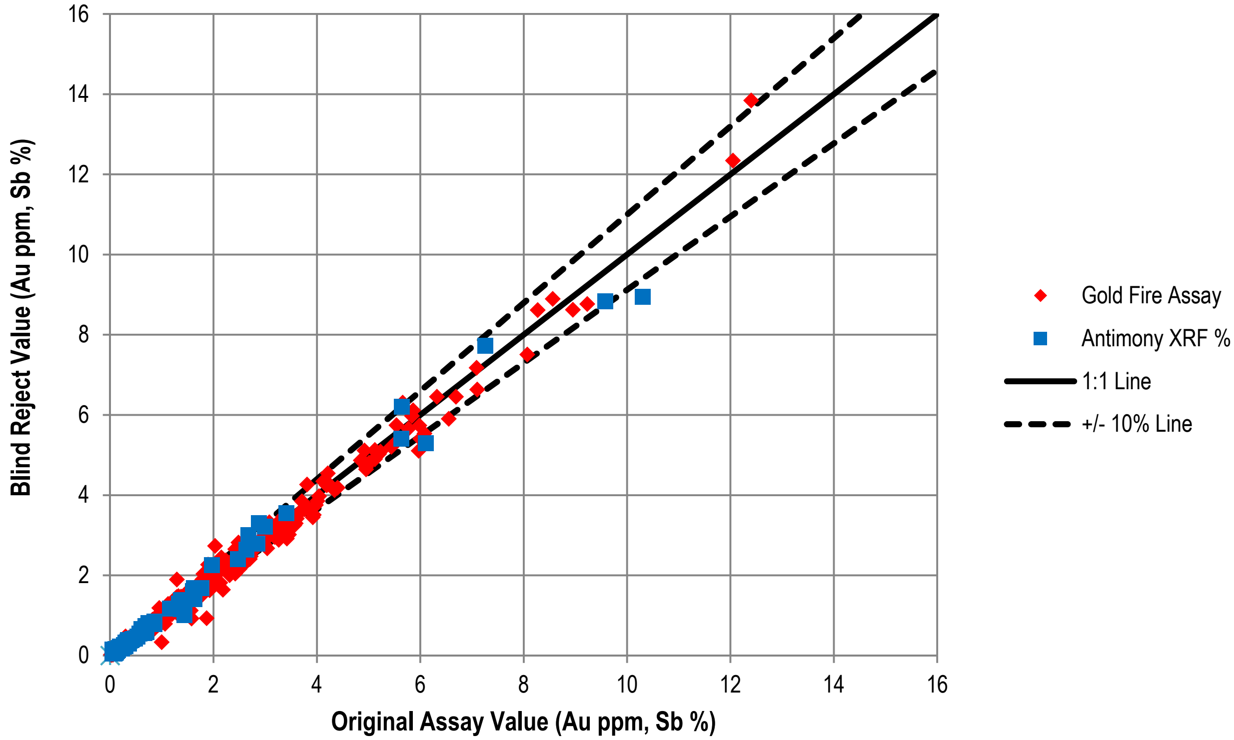
Figure 11.5: ALS Pulp Duplicates



11.7.8 Check Assays QA/QC

Midas Gold re-submitted 822 rejects with new sample numbers to ALS for assay to test for reproducibility and consistency (blind rejects). Out of the submitted rejects, 757 were above 0.025 ppm by gold fire assay and 99 were above 0.05% antimony by x-ray fluorescence (**XRF**). Within these parameters, the RMS CV for gold was 5.5% and the RMS CV for antimony was 9.4%, both values showing good reproducibility. A scatterplot of these values is shown on Figure 11.6.

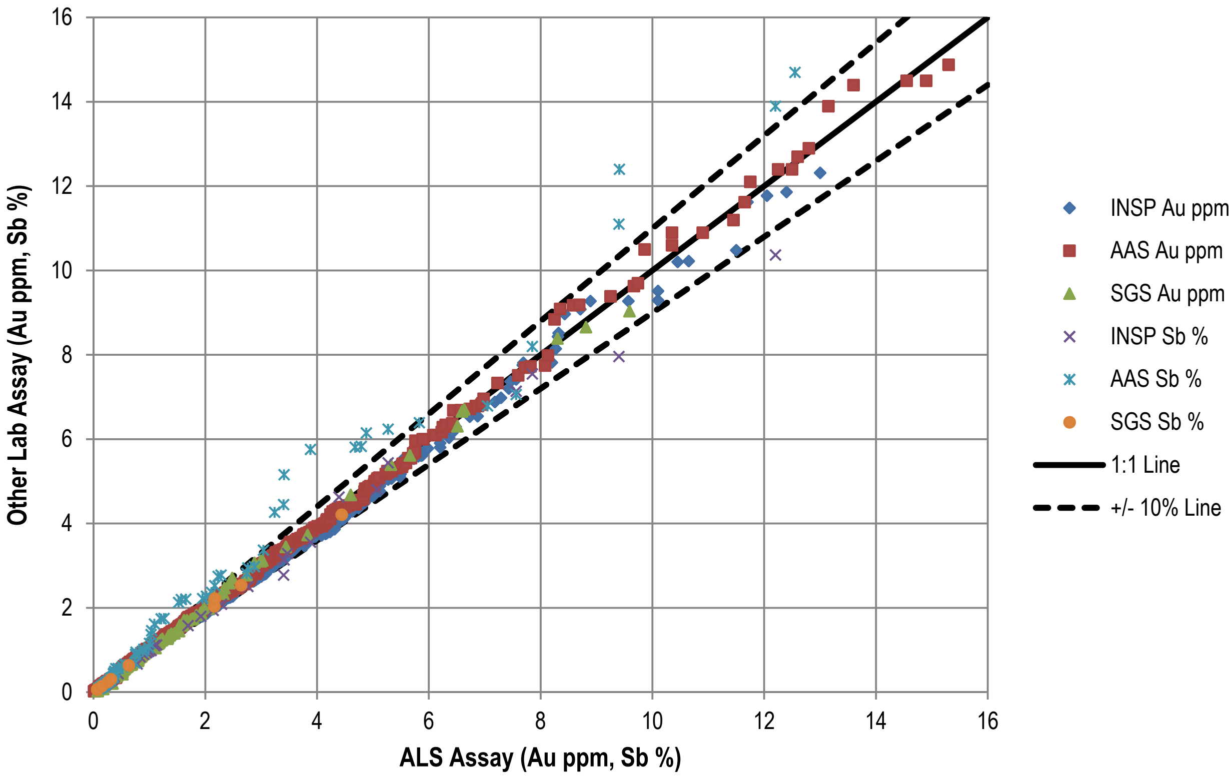
Figure 11.6: Blind Rejects Assays



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 988 pulps were submitted to Inspectorate for gold fire assay of which 957 were above 0.025 ppm. The average percent difference between the Inspectorate assay and the reported ALS assay was -4.73%. Of these samples, 97 were also assayed for antimony of which 51 exceed 0.05% antimony. The average percent difference between ALS and Inspectorate antimony assays for these samples was -5.77%. A total of 1,003 pulps were submitted to AAS for gold fire assay of which 904 were above 0.025 ppm and 80 samples were assayed for antimony that exceeded 0.05%. The average percent difference between the AAS assay and the reported ALS assay was 4.44% for gold and 22.96% for antimony. Removal of samples outliers reduces the average difference to 9%. It appears that there may have been sample numbering issues or possibly poor assay methods by the lab itself.

SGS analyzed 92 samples of which 53 were assayed for gold only and 39 were assayed for gold and antimony. Ninety-one samples were above 0.025 ppm gold and 13 samples were above 0.05% antimony. The average percent difference between the SGS assay and the reported ALS assay for gold was 1.49% and for antimony was -0.08%. Figure 11.7 shows the QQ plot of umpire laboratory check assays of pulps.

Figure 11.7: QQ Plot of Pulp Check Assays



11.7.9 Work Order Evaluation and Corrective Actions

Assay shipments containing samples, duplicates, standards and blanks are grouped as work orders. Beginning in 2012, each standard and blank within ALS work orders was systematically evaluated using the criteria discussed in Sections 11.7.4 and 11.7.5. Work orders for 2011 were retroactively evaluated. Upon evaluation, several work orders contained standards or blanks which failed. 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 by year.

Table 11.6: Work Orders and Revisions by Year

Year	Work Orders	Flagged Work Orders	Work Orders with Original Results Confirmed	Revised Work Orders
2011	189	27	23	4
2012	325	52	42	10
2013	82	6	3	3

11.8 CONCLUSIONS

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. This assumption of validity is based on various reviews including analysis and inspection of original drill logs, assay certificates, paired data analysis between pre-Midas Gold drilling and Midas Gold drilling, 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.

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.

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12 DATA VERIFICATION

12.1 INTRODUCTION

Data verification programs have been undertaken by numerous independent consultants and by Midas Gold personnel, as discussed in previous NI 43-101 technical reports (SRK, 2011; SRK, 2012) and performed subsequently. This section summarizes the verification work performed on data and practices for both historical and current data. The Independent Qualified Person (**QP**), Garth Kirkham, P. Geo., believes that the datasets used for the mineral resource estimates are validated and verified as adequate for the estimation of mineral resources for each of the respective deposits.

The QP visited the Lake Fork, Idaho offices and facilities on April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas on July 13 - 16, 2014.

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 recoveries appeared to be very good.

The site visit entailed inspection of the shops, offices, reclaimed drill sites, the Yellow Pine, Hanger Flats and West End mineral resource areas along with the outcrops, historic 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.

Kirkham is confident that the data and results are valid based on the site visit 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 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

Midas Gold professional personnel have constructed and maintained the drill hole and geologic solids databases in-house since Project inception. A database geologist is supervised by an on-site resource geologist who is responsible and accountable for all of the data stored in the drill hole database and GEMCOM project directories. At intervals, Midas Gold has augmented, revised, and corrected its database in the following respects:

1. new drilling information from ongoing campaigns;
2. addition of cyanide soluble gold assays for West End drill holes completed by Midas Gold;
3. addition of QA/QC from previous gaps and ongoing Midas Gold drilling;
4. pre-Midas Gold drilling collar coordinate revisions and minor changes to Midas Gold hole collars;
5. corrections to Barrick Gold Corporation (Barrick) down hole surveys;

6. addition of lithology codes for certain pre-Midas Gold drill holes;
7. revision of below-detection assay value assignments;
8. assay precision changes;
9. naming of hole pre-collars and elimination of duplicates; and
10. 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. In addition, Midas Gold routinely electronically verifies assay records in the drill hole database against original electronic laboratory certificates. 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.

12.3 PRE-MIDAS GOLD DRILL HOLE DATA

Historical drill holes on the Stibnite Gold property comprise 58% of the drill hole database by length and utilized a range of drilling, sampling and assaying methods over an 80 year period. Some historical drill data sets are characterized by factors which may impact the accuracy of the assay results including small diameter core, poor core recovery, assaying of sludge or sludge + core, reverse circulation or rotary drilling methods, disparate assay and analytical methods, and other factors. Midas Gold and its contractors have completed numerous projects to assess the accuracy of the historic drill hole data and evaluate what data sets are appropriate for estimation of mineral resources.

Midas Gold and previous operators on the property have conducted extensive confirmation drilling programs which provide the basis for statistical and graphical comparisons. Generally, confirmation drilling has tested areas previously drilled by the historical operators, but twinning of specific holes has been limited due to logistical restrictions based on post-mining topography and lack of access to historical underground development workings. Results of the statistical comparisons are typically reviewed using quantile-quantile plots, histograms and descriptive statistics.

Paired sample analysis has been employed at all three bedrock deposits to assess population characteristics of assay data for different drill programs. The analysis selects samples or composites from two drill data sets within a specified radius of one another (typically 10 to 30 m) and compares either: (a) all possible sample pairings; or (b) only the nearest unique sample pairings. The methodology comparing all sample pairings accentuates any clustering in the input data. A comparison of nearby samples selects sample groups representative of similar regions of the deposit by flagging all data points within a specified radius of samples in the comparative data set. Unlike paired sample analysis for all possible pairings, the populations for comparison can have different numbers of data points.

Nearest neighbor cell declustering utilizes a nearest neighbor estimation to assign grades to small blocks (typically 3 to 5 m cubes) within a specified anisotropic search radius from different sample or composite data sets. Blocks receiving an estimate from both data sets are compared statistically.

In an effort to achieve a more realistic cell size, data comparisons within drill "panels" takes the length weighted average grade of samples occurring within rectangular blocks or panels (typically 42 x 42 x 6 m) aligned approximately parallel to mineralization and orthogonal to drill holes.

Kriged blocks in common use standard linear geostatistical estimation methods to estimate mineral resource model block grades using the different drill data sets. Blocks estimated from each dataset are compared statistically.

12.3.1 Hangar Flats

Data evaluation at Hangar Flats focused on the pre-1955 historical drill holes consisting of three campaigns which comprise approximately 22% of the overall Hangar Flats database. These include the 1929 BMC-series underground core holes, circa 1950s MC-series underground and surficial core holes drilled by the Bradley, and circa 1950s DMA-series underground holes drilled by the USBM. These historical drill holes were BX-EX diameter, have core recoveries <50% and assayed sludge only.

12.3.1.1 Comparison of Pre-1955 Drilling Campaigns

The 1929 BMC-series underground drill holes at Hangar Flats are excluded from the mineral resource estimation process due to insufficient supporting information, poor core recoveries and uncertain assay methodologies of the period. The 1950s DMA series holes were reviewed graphically relative to Midas Gold drill holes and selected holes were excluded if assays were not corroborated by Midas Gold drilling or if assays did not adequately define the limits of the mineralized zones. Some MC-series drill holes were excluded from the mineral resource estimation dataset due to incomplete sample data. The remaining 38 MC-series and 22 DMA-series holes were evaluated relative to the combined Midas Gold and Hecla assay data using paired sample analysis methods, as summarized in Table 12.1. Results for gold indicate good agreement between data sets within 5 m but substantial high bias for pairs within 10 m. High bias in the historical underground holes is attributed to different orientations of the drill holes, with many Midas Gold drill holes drilled at a low angle to the MCFZ, while underground holes were drilled in more favorable orientations across relatively narrow high-grade zones in the underground development workings.

Table 12.1: Summary of Pre-1955 Drilling Campaign Evaluations for Hangar Flats

Method	Search Radius (m)	# Pairs		# Holes		Mean Gold Grade (g/t)		Mean Antimony Grade (%)	
		Gold	Antimony	Modern	Historic	Modern	Historic	Modern	Historic
Paired samples (nearest)	5	96	-	16	14	1.61	1.69	-	-
Paired samples (nearest)	10	376	167	31	25	1.22	1.62	0.08	0.34
Paired samples (all)	10	3,653	-	35	31	0.97	1.53	-	-

Paired sample analysis for antimony indicates substantial high bias in the historic data sets, which is potentially also attributed to drill hole orientation. Because they are more favorably oriented to the steeply dipping mineralization in the MCFZ than the Midas Gold and Hecla data, and also because they comprise only 11% of the data set at Hangar Flats, the pre-1955 MC- and DMA-series drill holes are retained for the purposes of mineral resource estimation for gold and antimony but are subjected to a strategy which limits their influence, effectively mitigating the impact of the potential assay bias with tighter spatial constraints on mineralization.

12.3.2 West End

The West End database contains 889 historical drill holes completed by a variety of operators using numerous drilling methods. Of these, 260 are air-track and rotary holes that are omitted from the dataset used for resource estimation. The remaining historical holes consist primarily of RC holes drilled by Pioneer and Superior, and core or RC holes drilled by Superior. Drill holes were evaluated using the paired sample analysis and blocks-in-common methods with results summarized in Table 12.2. With the exception of one analysis with very few samples, all campaigns yield means within +/- 10%, demonstrating that the fire-assay data for the historical campaigns is unbiased relative to the Midas Gold data and is suitable for the purposes of resource estimation.

Table 12.2: Summary of Post-1973 Drilling Campaign Evaluations for West End

Method	Campaigns	Search Radius (m)	# Pairs Gold	# Holes		Mean Gold Grade (g/t)	
				Modern	Historic	Modern	Historic
Paired sample analysis	Midas Gold vs Superior	10	497	2	5	1.76	1.91
Paired sample analysis	Midas Gold vs Pioneer	10	4,543	27	25	0.56	0.53
Paired sample analysis	Midas Gold vs Superior	10	602	7	7	1.07	0.67
Kriged blocks in common	Midas Gold vs Pioneer	40	27,968	-	-	0.625	0.562
Kriged blocks in common	Midas Gold vs Superior	40	19,568	-	-	0.731	0.693

12.3.3 Yellow Pine

The historic Yellow Pine drill hole database contains 768 historical drill holes which can be broadly divided into pre-1953 small diameter core holes and post-1973 core and RC drill holes.

12.3.3.1 Comparison of Post-1973 Drilling Campaigns

The post-1973 drill holes within the central region of the Yellow Pine Deposit were drilled by Ranchers and Barrick. Within the Homestake area of the Yellow Pine Deposit, north of 4,976,600 m, historical data sets consist primarily of drilling by Hecla and Ranchers, with a small number of Superior drill holes. Comparison results for gold are summarized in Table 12.3 and generally indicate good agreement of both Ranchers and Barrick gold assays with Midas Gold data. Ranchers antimony assays compare well to Midas Gold but Barrick antimony appears to be low-biased, presumably due to assaying of antimony on longer 12.19 m (20 ft) intervals. A comparison of the Hecla 1980s RC drill holes with Midas Gold data illustrates varying degrees of high-bias in the Hecla gold data set. To further evaluate this difference, Hecla data was compared to Ranchers data as the drilling covers a similar area and the Ranchers data shows good agreement with the Midas Gold data. These comparisons indicate a persistent high-bias in the Hecla data with respect to both Midas Gold and Ranchers data.

To quantify the potential impact of high-biased Hecla data on the mineral resource estimate, a model sensitivity analysis was run for the Homestake domain only and indicated a ~3% increase in contained gold ounces (~11,000 oz Au) with inclusion of the Hecla data versus the Hecla data removed. It was decided to retain the Hecla data in the final dataset for mineral resource estimation because the change in contained metal is not significant, and because removal of the data would force the model to extrapolate grade across greater distances.

Table 12.3: Summary of Post-1973 Drilling Campaign Evaluations for Yellow Pine

Campaigns	Method	Search Radius (m)	# Pairs Gold	Mean Gold Grade (g/t)	
				Midas Gold	Historic
Midas Gold vs Post-1973 (all)	Paired samples	5	251	2.08	2.32
Midas Gold vs Post-1973 (Central Yellow Pine)	Nearby samples	20	566 / 798	2.55	2.11
	Nearby samples	40	1,526 / 2,050	2.26	2.32
Midas Gold vs Barrick (Central Yellow Pine)	Nearest neighbor declustering	20	16,573	1.83	1.78
	Drill panel comparison	42 x 42 x 6	92	2.26	2.49
Midas Gold vs Ranchers (Central Yellow Pine)	Nearest neighbor declustering	20	14,732	2.40	2.41
	Drill panel comparison	42 x 42 x 6	83	2.45	2.39
Midas Gold vs Ranchers (Homestake area)	Nearby samples	20	369 / 185	1.38	1.12
	Nearby samples	40	551 / 298	1.16	1.05
	Nearest neighbor declustering	20	5,306	1.34	1.27
	Drill panel comparison	20	16	1.00	1.13
Midas Gold vs Hecla (Homestake area)	Nearby samples	20	459 / 271	1.45	1.41
	Nearby samples	40	760 / 447	1.41	1.57
	Nearest neighbor declustering	20	7,930	1.12	1.50
	Drill panel comparison	42 x 42 x 6	34	1.36	2.19
Ranchers vs Hecla (Homestake area)	Declustered mean grade	N/A	569 / 688	1.17	1.38
	Kriged blocks in common	60 x 45 x 25	3,259	1.19	1.52

12.3.3.2 Comparison of Pre-1953 Drilling Campaigns

The pre-1953 drill holes at Yellow Pine consist of Bradley and USBM drill holes, primarily drilled in the 1940s. Both the Bradley and USBM drill holes are commonly characterized by poor core recoveries associated with small diameter core and drilling technology utilized at the time, unverifiable surveyed drill hole positions, absence of a documented quality control program and sampling of sludge only (Bradley) or sludge + core (USBM). For the USBM drill holes, the Midas Gold database contains the weighted average grade based on the dry weight of core and sludge as calculated by the USBM and preserved on historic log-sheets. The averaging method applies a weighted average grade, but does not incorporate the theoretical recovery based on the hole diameter and drill bit annulus, as was standard practice at the time. Midas Gold, Barrick and Ranchers drill hole data show good agreement for gold, and Midas Gold and Ranchers show good agreement for antimony and these data sets are respectively utilized for evaluation of the these metals in the pre-1953 drill holes.

A total of 187 pre-1953 drill holes were flagged by Midas Gold and removed from the dataset including the 1920s Bradley churn drill holes, holes missing critical collar or assay data, and holes for which positions could not be verified on historic maps and cross sections. The remaining 179 pre-1953 drill holes were statistically evaluated on an overall basis, within specific regions of the deposit, and within sub-groups based on period and drill hole series, and surface versus underground collar positions.

The USBM drill holes consist of 52 surficial holes drilled in 1939 and 1940. As summarized in Table 12.4, the USBM drilling campaigns generally compare well to Midas Gold and post-1973 campaigns for both gold and antimony. For the central region of Yellow Pine, comparison methods indicate historic assays are within +/- 10% of post-1973 data. Within the southern region of the Yellow Pine Deposit, comparisons indicate both high- and low-bias using different methods, which is attributed to the limited number of samples and spatial bias rather than any persistent analytical bias. USBM drill holes were therefore retained in the dataset and used for the purposes of mineral resource estimation.

Table 12.4: Summary of Pre-1953 Drilling Campaign Evaluations for Yellow Pine

Campaigns	Comparisons	Method	Search Radius (m)	# Pairs Gold	Mean Gold Grade (g/t)	
					Midas Gold	Historic
USBM	Modern vs USBM (All)	Paired samples	5	61	1.65	1.61
		Nearby samples	40	2,808 / 866	2.35	2.12
	Modern vs USBM (YP South)	Nearby samples	20	33 / 62	2.59	0.84
		Nearby samples	40	58 / 152	2.54	0.78
	Midas Gold vs USBM (YP South)	Nearest neighbor declustering	20	3,598	0.63	1.40
		Drill panel comparison	42 x 42 x 6	6	0.79	1.03
	Midas Gold vs USBM (YP Central)	Nearest neighbor declustering	20	12,643	2.15	2.30
		Drill panel comparison	42 x 42 x 6	49	2.42	2.53
	Modern vs USBM (YP Central)	Nearby samples	20	916 / 737	2.34	2.37
		Nearby samples	40	2,106 / 1,132	2.33	2.34
Bradley Drill Holes	Modern vs Bradley (All)	Paired samples	5	125	2.01	2.76
	Modern vs B-series (All)	Nearby samples	20	1,338 / 462	2.47	2.52
	Modern vs B-series (YP Central)	Nearby samples	20	1,289 / 410	2.48	2.65
		Nearby samples	40	2,724 / 746	2.37	2.30
	Midas Gold vs Bradley (YP Central)	Nearest neighbor declustering	20	21,753	2.45	2.77
	Midas Gold vs Bradley (YP South)	Nearest neighbor declustering	20	1,438	1.59	1.65
		Drill panel comparison	42 x 42 x 6	3	1.94	1.97
	Midas Gold vs Bradley (Homestake)	Nearest neighbor declustering	20	3,709	1.12	1.8
	Midas Gold & Barrick vs Bradley 1940s surf. (YP Central)	Nearest neighbor declustering	20	33,361	2.50	2.57
		Drill panel comparison	42 x 42 x 6	169	2.55	2.84
	Midas Gold & Barrick vs Bradley 1950s surf. (YP Central)	Nearest neighbor declustering	20	6,166	2.27	1.95
		Drill panel comparison	42 x 42 x 6	12	2.20	2.00
Bradley Underground Drill Holes	Modern vs T-series UG (YP Central)	Nearby samples	20	824 / 899	2.74	2.94
		Nearby samples	40	1,802 / 1,371	2.25	2.74
	Midas Gold & Barrick vs Bradley 1940s U.G. (YP Central)	Nearest neighbor declustering	20	17,404	2.58	2.77
		Drill panel comparison	42 x 42 x 6	44	2.47	2.59
	Midas Gold & Barrick vs Bradley 1950s U.G. (YP Central)	Nearest neighbor declustering	20	7,508	2.39	2.92
		Drill panel comparison	42 x 42 x 6	16	2.83	3.16
	Bradley Surf. vs Bradley U.G. (YP Central)	Nearby samples	10	64 / 97	3.27	3.39
		Nearby samples	20	170 / 296	2.19	2.71

The Bradley drill holes can be subdivided into the T-series (underground) and B-series (surficial and underground), and into those drilled in 1940 versus those drilled in the late 1940s and early 1950s. When compared to all Bradley holes not omitted previously, the post-1973 data is low-biased by between 5% and 40% within the central and southern regions of the deposit for gold while the Bradley drill holes are substantially high-biased in the Homestake (northern) region. A comparison of only the surficial holes reduces the apparent assay bias in the Bradley data considerably and some comparison methods indicate a low bias relative to the modern data sets. The underground drill holes consistently show good agreement with modern data at shorter distances with an increasingly positive bias at larger comparative distances. This same relationship is demonstrated relative to the Bradley surficial drill holes, which is attributed to the location of these holes being within the higher-grade regions of the deposit rather than to any persistent analytical or sampling bias in the underground drill holes.

For antimony, the Bradley drill holes show a consistently high-bias relative to Midas Gold and Ranchers drill holes using both statistical methods and graphical comparison. This is attributed to assay tailing within sludges below mineralized zones, which is much more significant with antimony than with gold.

Based on the results discussed above, the surficial Bradley and USBM holes were retained for use in the gold mineral resource estimate within the central and southern regions of the deposit; however, the Bradley drill holes in the Homestake region were removed. The underground drill holes were also retained, but their range of influence was restricted to 12 m in the gold mineral resource estimate. In addition, the Bradley drill holes are not utilized in the antimony estimate at all. To quantify the potential impact of pre-1953 data on the mineral resource estimate, model sensitivities were run with various combinations of historic data. The sensitivity incorporating the final data set used for the purposes of mineral resource estimation indicates a 4% increase in total contained gold when compared to using only the post-1973 data, which is well within acceptable limits.

12.3.4 Historic Tailings

The Historic Tailings database contains 25 historic auger drill holes drilled by Stibnite Mines Inc. (SMI) in the 1990s, in addition to drilling completed by Midas Gold. The historical holes have cyanide assays only and were not utilized in the mineral resource estimate.

12.4 CONCLUSIONS

Kirkham visited the Lake Fork, ID offices and facilities on April 23 - 25, 2014 and subsequently visited the site, facilities and surrounding areas on July 13 - 16, 2014. During these visits, no issues were identified and all procedures and protocols were to industry standards as expected for a North American operation at the pre-feasibility stage of development.

The datasets employed for use in the mineral resource estimates are a mix of historic data and current, modern data. There is always a concern with respect to validity of the historic data and extensive validation verification must be performed in order to insure that the historic data may be relied upon.

Kirkham reviewed extensive validation and verification procedures and results performed by external consultants and by Midas Gold in order to ensure validity of the Mineral resource estimates and for classification purposes. The methods and procedures performed by Midas Gold were carried-out with great care, and were supervised and approved by Kirkham, which entailed detailed analysis and resulted in sub-sets of data being excluded or, in some cases, being flagged so as reduce their influence due to any potential bias.

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

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13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Metallurgical testing has been conducted on samples from the Yellow Pine, Hangar Flats, West End and the Historic Tailings deposits. This work has included extensive mineralogical studies and developmental metallurgical test work on various ore types from each of the deposits. Despite the differences in the deposits, developmental metallurgical testwork has been able to identify a flowsheet that proved successful when applied to each of the deposits, making it possible to design a single plant that can process all ores from the Project as they are mined.

An auriferous pyrite recovery flotation circuit was developed to process the low antimony sulfide ores of all three deposits. This circuit consisted of roughers and a single stage of cleaning with scavenging, with scavenger tailings operating with the option to flow closed circuit back to the primary ball mill (preferred option). At times, the rougher concentrates may meet the pressure oxidation (POX) requirements and not require further cleaning.

When antimony-rich sulfide ores are being processed, a smaller antimony recovery circuit would be operated ahead of pyrite flotation. To produce saleable stibnite concentrate, this circuit requires roughing followed by two stages of cleaning, with the antimony 1st cleaner tailings recombining with the rougher tailings to feed the pyrite circuit. The antimony 2nd cleaner tailings are returned to the 1st cleaner feed.

Additionally, test work was initiated to assess the ability to reject carbonate-bearing (CO_3) minerals from the gold concentrates of the carbonate-rich West End Deposit, which interfere with the ability to pressure oxidize the concentrates without pre-acidulation. West End sulfide ores can successfully produce concentrate with a POX friendly carbonate to sulfur ratio through one stage of cleaning; to maintain gold recovery, a scavenger and recirculation of scavenger tailings to the primary mill for reprocessing is recommended.

Developmental leaching test work was also undertaken on the West End oxide ores as well as on select flotation tailings produced from partially oxidized mineralization from Hangar Flats and West End. West End oxide leach studies indicate that 96% of the extracted gold leaches in the first six hours. Leach studies on the flotation tailings from Hangar Flats and West End indicate that gold in the flotation tailings are also fast leaching and could contribute substantially to gold recovery.

Below can be found a description of the grindability of the materials and the mineralogy of the deposits. Following this, for each deposit, a brief reference to past operations and testwork on ores and samples from each deposit is made followed by a summary of the metallurgical test data.

13.2 SAMPLE SELECTION AND COMPOSITE PREPARATION

Approximately 800 core samples from the Yellow Pine, Hangar Flats and West End deposits were delivered from the site to SGS Vancouver for mineralogical and metallurgical studies during 2013 and 2014. Sonic and auger samples from the Historic Tailings Deposit and geochemical laboratory assay reject samples from the West End Deposit were also tested. The sources of the samples from the deposits are shown on Figure 13.1, Figure 13.2 and Figure 13.3.

Figure 13.1: Sources of Samples for Yellow Pine and West End Metallurgical Testing

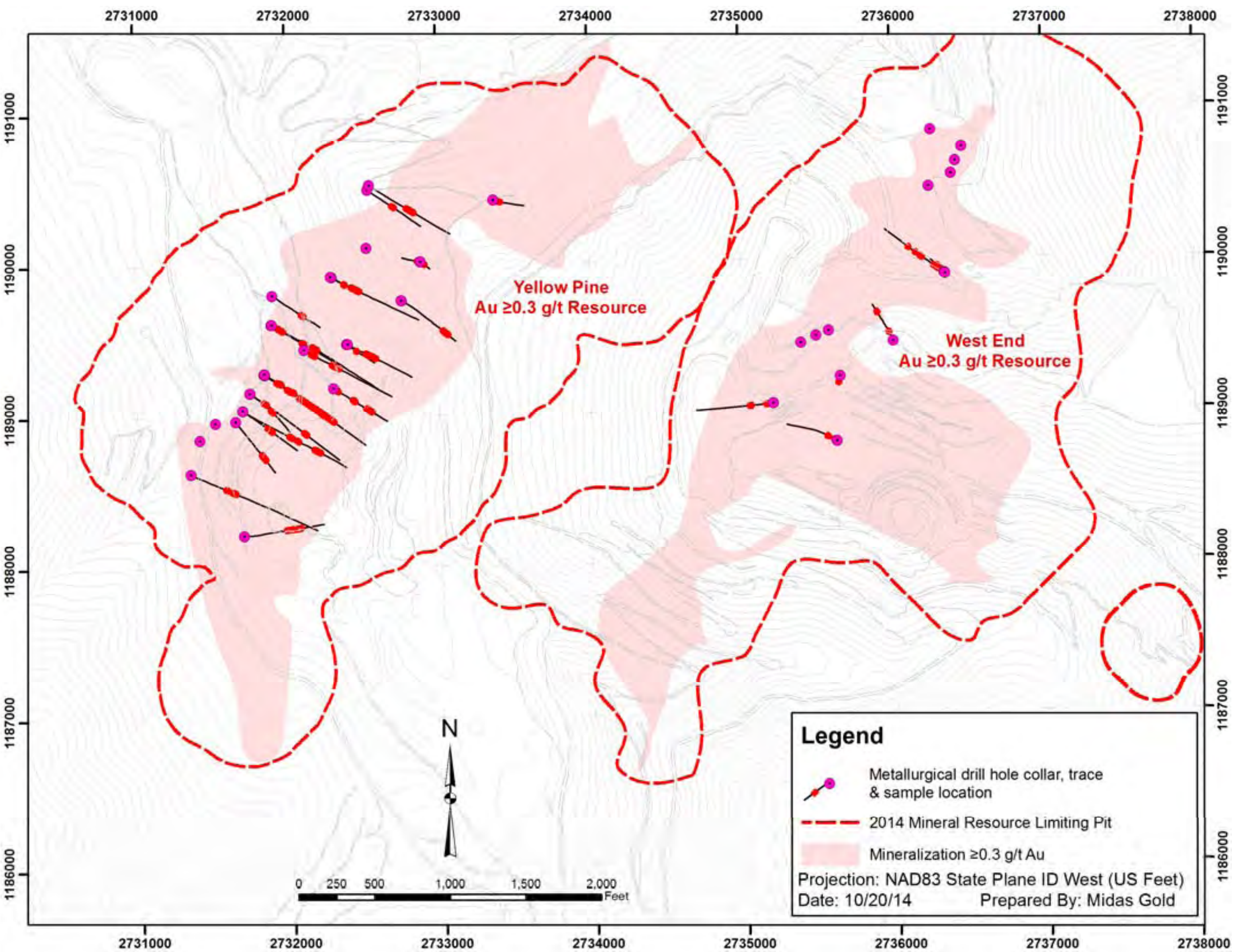


Figure 13.2: Sources of Samples for Hangar Flats Metallurgical Testing

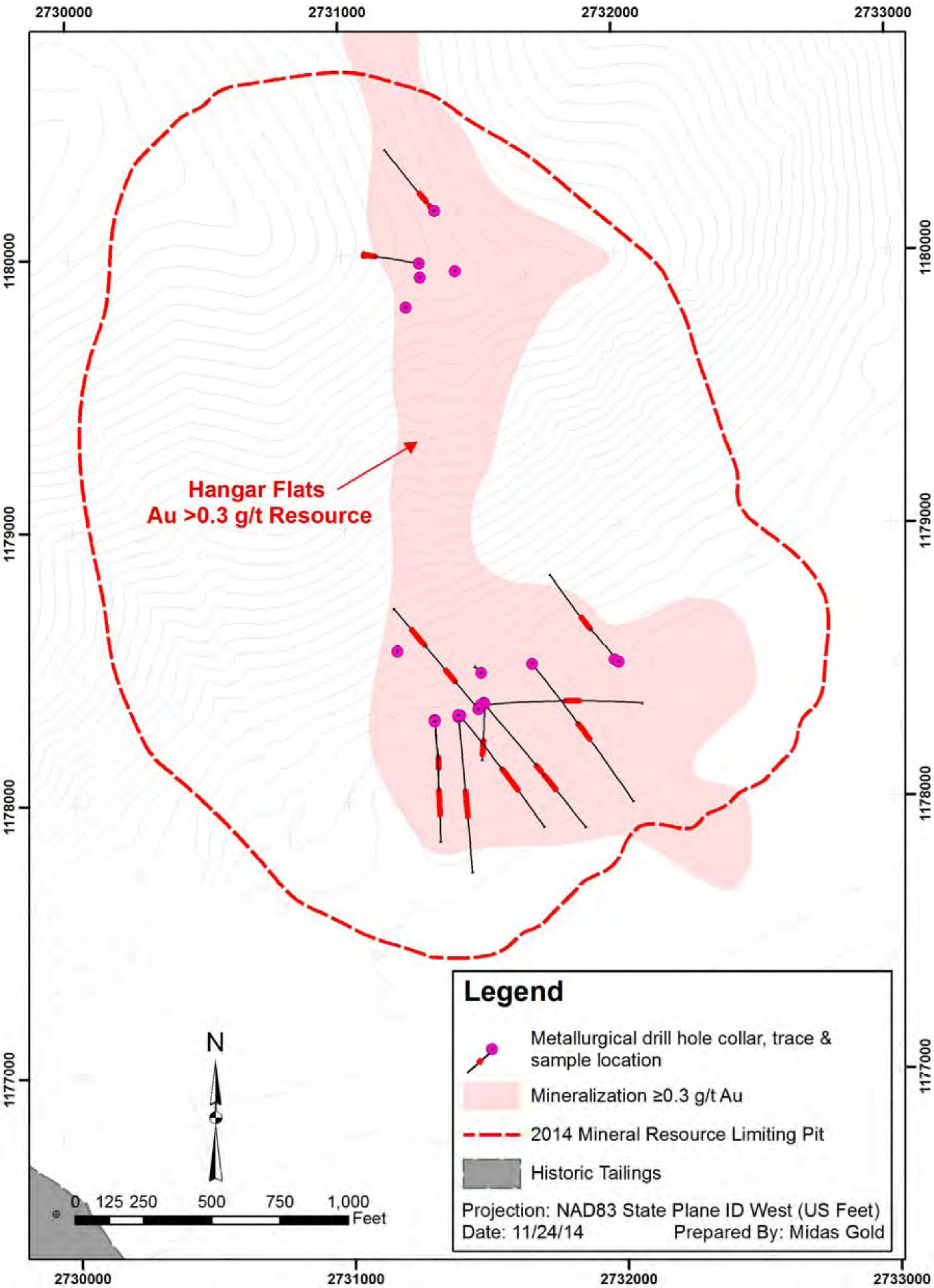
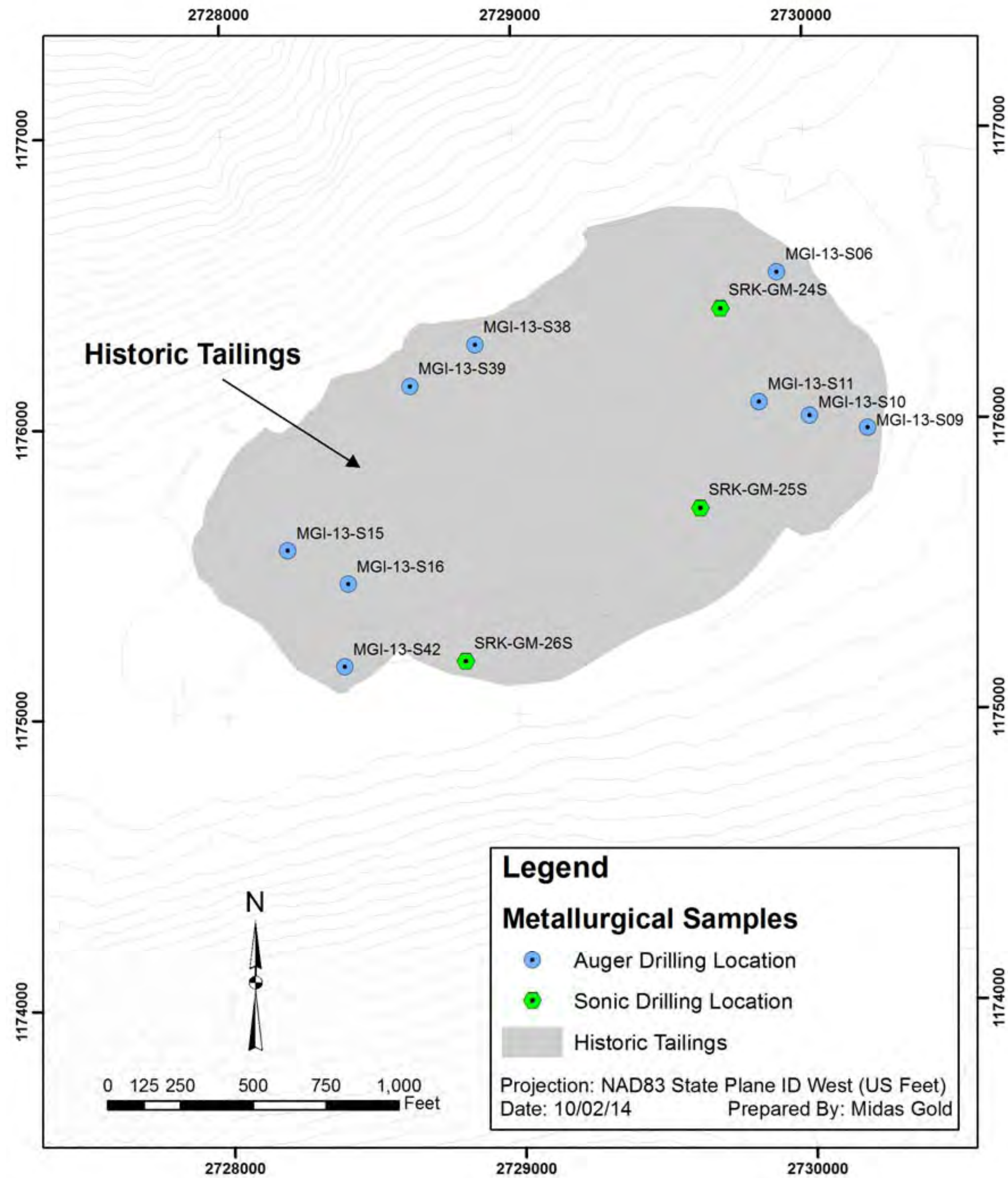


Figure 13.3: Sources of Samples for Historic Tailings Metallurgical Testing



From these samples, a variety of composites were created for various testing and characterization purposes (Table 13.1). A few composites were used that were remaining from the PEA program, as were some of the concentrate products tested. In addition, some 114 variability composites were created, including 43 from Yellow Pine, 11 from the Yellow Pine early production zone, 27 from Hangar Flats, 12 from the Historic Tailings Deposit and 16 from West End, in part to assess the variability in bulk mineralogy across the deposits.

Table 13.1: Primary Metallurgical Composites for Testing

Comp. ID	Description	No. of Tests	Purpose	Head Grades					
				Au (g/t)	Ag (g/t)	As (%)	Sb (%)	$S_{(t)}$ (%)	W (%)
SubA1	HF Global	12	Used PEA concentrate for testing	3.49	6.3	0.690	0.190	1.34	n/a
SubA2	WE Global	12	Used PEA concentrate for testing	2.05	2.5	0.330	0.010	0.89	n/a
SubA3a	YP High Sb	3	Used PEA concentrate for testing	2.87	1.5	0.450	0.190	1.47	n/a
SubA3b	YP Low Sb	9	Used PEA concentrate for testing	2.18	2.2	0.370	0.010	0.97	n/a
I	HF high tungsten	7	PEA Comp - Scoping tungsten recovery	2.31	185.0	0.180	13.00	6.06	1.600
YP104	YP	7	Scoping tungsten recovery	1.51	15.4	0.180	0.820	1.41	0.420
P2	Dup Comp P (YP)	16	PEA comps for production of Au concentrate	2.25	n/a	0.370	0.000	1.08	n/a
YPH	YP High Sb	39	Flowsheet development, locked cycle tests	2.14	4.1	0.244	0.380	1.03	n/a
YPL	YP Low Sb	25	Flowsheet development, locked cycle tests	1.82	2.8	0.353	0.043	0.99	n/a
HFH	HF High Sb	42	Flowsheet development, locked cycle tests	1.96	8.6	0.390	0.500	1.03	n/a
HFL	HF Low Sb	26	Flowsheet development, locked cycle tests	1.60	2.3	0.460	0.035	0.81	n/a
WES	WE Sulfide	23	Flowsheet development, locked cycle tests	1.79	1.7	0.218	0.027	0.65	n/a
WEO	WE Oxide	11	Oxide leach development	0.91	1.3	n/a	n/a	0.04	n/a
S06	HT High Sb	4	Flotation and leach confirmation testing	1.44	<10	0.099	0.871	0.61	0.070
S24	HT Low Au	9	Scoping flotation and leach testwork	0.98	4.0	0.092	0.140	0.43	0.016
S25	HT Avg Au	9	Scoping flotation and leach testwork	1.12	3.0	0.150	0.160	0.36	0.029
S26	HT High Au	10	Scoping flotation and leach testwork	1.51	3.8	0.180	0.220	0.29	0.023
HTL	HT Low Au	4	Flotation and leach confirmation testing	0.78	<10	0.091	0.074	0.18	0.012
HTM	HT Avg Au	6	Flotation and leach confirmation testing	1.17	<10	0.150	0.230	0.37	0.033
HTH	HT High Au	6	Flotation and leach confirmation testing	1.31	<10	0.170	0.170	0.28	0.021
EP1	Early YP Low Sb	3	Flotation and leach confirmation testing	2.87	n/a	0.602	<0.01	1.36	n/a
EP2	Early YP Low Sb	3	Flotation and leach confirmation testing	0.90	n/a	0.518	<0.01	1.29	n/a
EP3	Early YP Low Sb	3	Flotation and leach confirmation testing	2.94	n/a	0.226	0.070	1.01	n/a
EP4	Early YP Low Sb	3	Flotation and leach confirmation testing	2.40	n/a	0.264	<0.01	0.91	n/a
EP5	Early YP High Sb	4	Flotation and leach confirmation testing	2.53	n/a	0.577	0.540	1.41	n/a
EP6	Early YP High Sb	3	Flotation and leach confirmation testing	2.97	n/a	0.588	0.180	1.66	n/a
EP7	Early YP High Sb	3	Flotation and leach confirmation testing	3.35	n/a	0.316	0.930	1.74	n/a
EPL	Early YP Low Sb	2	Confirmatory flotation, locked cycle tests	2.22	3.4	0.425	0.030	1.21	n/a
EPH	Early YP High Sb	2	Confirmatory flotation, locked cycle tests	3.09	n/a	0.462	0.650	1.71	n/a
EPLB	EPL-HTM Blend	4	Confirmatory flotation, locked cycle tests	2.14	3.7	0.364	0.060	1.05	n/a
EPHB	EPH-HTM Blend	3	Confirmatory flotation, locked cycle tests	2.95	4.6	0.407	0.530	1.45	n/a

Note: YP = Yellow Pine, HF = Hangar Flats, WE = West End, HT = Historic Tailings, EP = Yellow Pine Early Production Zone

13.3 GRINDING CHARACTERIZATION

A total of twenty two SMC, twenty four Bond Ball Mill Work Index, eight Bond Rod Mill Work Index, ten abrasion index and ten crusher work index tests were conducted on composites to support the PEA and PFS metallurgical programs, taken from samples around each of the deposits. All the work was conducted by SGS Lakefield and SGS Vancouver; the results from these tests are provided in Table 13.2. All three deposits have average grindability characteristics of which, Yellow Pine is most resistant to ball milling and West End is the least amenable to SAG milling (Ratnayake, 2013a; Gajo, 2014b).

Table 13.2: Grinding Characterization Samples

Test	Units	Yellow Pine			Hangar Flats			West End		
		No. of Tests	Avg.	St. Dev.	No. of Tests	Avg.	St. Dev.	No. of Tests	Avg.	St. Dev.
JK Drop Weight SAG Testing										
A x b	N/A	0	n/a	n/a	1	123.2	n/a	1	63.4	n/a
ta	N/A	0	n/a	n/a	1	1.5	n/a	1	0.37	n/a
SMC Testing										
A x b	N/A	9	87.5	17.5	7	150.4	54	6	50	18.3
ta	N/A	9	0.86	0.18	7	1.5	0.53	6	0.49	0.18
Crusher and Mill Index Testing										
Crusher WI	kWh/Mt	2	5.1	0.07	5	7.4	1.6	3	11.6	1.9
Abrasion Index	N/A	2	0.26	0.003	5	0.22	0.02	3	0.24	0.11
Bond Rod Mill WI	kWh/Mt	2	10.9	0.21	4	10.9	0.93	2	14.7	2.1
Bond Ball Mill WI	kWh/Mt	10	14.1	0.8	7	13.3	0.58	7	13.0	0.64

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.

Full gold deportment studies were conducted on twelve samples (four from each deposit), while 140 samples were subjected to bulk mineralogical analysis using QEMSCAN (62, 50 and 28 from Yellow Pine, Hangar Flats and West End, 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. Any discrete gold occurrences are very fine, typically ranging from 1 to 10 microns (µm) in size.

The vast majority of the gold hosted within the three deposits occurs as solid-solution gold, atomically dispersed within the sulfides and it seems likely that only the materials where the sulfides have been completely destroyed (true-oxide materials) host no solid-solution gold at all. 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 not 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 porous pyrite, fine pyrite and arsenopyrite. The coarse crystalline sulfides contain relatively little gold.

Antimony occurs as stibnite, which is typically coarse-grained when occurring in higher-grade samples. At head grades above 0.1% antimony, the stibnite mean grain size is typically 15 - 25 microns. As the antimony grade drops, the respective stibnite grain size drops markedly.

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

The host rock bulk mineralogy is shown in Table 13.4, which describes the median, 10th percentile and 90th percentile of each of the major components in a total of 140 samples analyzed by QEMSCAN to date. Some key features of these data include:

- Significant variability occurs in the modal mineralogy in each of the deposits;
- West End tends to be poorer in sulfides, so mass pull would be lower to the pre-oxidation circuit;
- West End has a hard quartzite component, which may be the cause of the harder grindability data;
- Clays are best represented in these data by the illite/muscovite category, and tend to be richest in the Hangar Flats Deposit; and
- Carbonates are richest in the West End Deposit.

Table 13.4: Distribution of QEMSCAN Modal Abundances

Deposit	Yellow Pine			Hangar Flats			West End		
Modal Abundance Percentile	**10th**	**50th**	**90th**	**10th**	**50th**	**90th**	**10th**	**50th**	**90th**
Pyrite/Arsenian Pyrite	1.1	2.2	3.0	0.6	1.8	3.0	0.3	0.8	2.2
Arsenopyrite	0.4	0.9	2.2	0.1	1.1	2.9	0.0	0.4	1.5
Gold-bearing Sulfides	1.7	3.1	5.1	1.1	3.1	5.2	0.4	1.2	3.4
Galena	0.0	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0
Stibnite	0.0	0.0	0.9	0.0	0.0	2.5	0.0	0.0	0.0
Quartz	28.4	35.2	43.0	30.3	36.6	47.3	17.8	32.5	73.4
Feldspar	31.4	41.8	51.0	23.8	36.5	46.7	3.5	16.9	40.2
Illite/Muscovite	5.8	10.9	18.3	7.8	16.0	23.4	5.5	12.3	27.4
Chlorite	0.0	0.0	0.1	0.0	0.1	0.2	0.1	0.2	0.9
Clays	0.0	0.4	1.5	0.7	1.4	2.0	0.5	1.4	4.9
Other Silicates	0.2	0.4	0.8	0.1	0.3	0.9	0.5	2.6	5.0
Oxides	0.3	0.6	1.0	0.2	0.4	1.1	0.3	0.7	1.6
Carbonates	1.9	3.5	7.2	0.1	1.7	4.6	2.5	13.2	41.1
Apatite	0.1	0.3	0.7	0.0	0.5	0.6	0.1	0.2	0.3
Other	0.1	0.2	1.1	0.1	0.7	2.0	0.0	0.1	3.7

13.5 YELLOW PINE DEPOSIT

13.5.1 Historical Metallurgy

The Yellow Pine Deposit has been mined intermittently since 1938. In the earlier years, flotation was employed to produce, during different historical periods, gold, antimony and tungsten concentrates, with milling rates reaching 2,200 short tons per day (st/d) prior to shutdown in 1951. In the late 1980s, a modest heap leach operation was commissioned to process Homestake oxide ores (located to the northeast of the main Yellow Pine Deposit). This was expanded with the construction of a new heap in 1990 with recoveries in the order of 80%. Heap leach operations phased down in 1991 and were discontinued in 1992 (Mitchell, 2000).

Several programs of testing have been conducted on Yellow Pine sulfide samples since the 1970s that included flotation of sulfide ores and pre-oxidation followed by cyanidation of the oxidized sulfide concentrates. Key programs were conducted at Hazen Research, Bacon, Donaldson and Sherritt Gordon in 1983 (Sherritt Gordon Mines Limited, 1983); at Lakefield Research, Mountain States R&D and the University of Idaho in 1987 (Rollwagen, 1987; Brackebusch, 1987) and at Lakefield Research and the University of Idaho in 1993 (Jackman, 1993; Harrington, Bartlett, & Prisbrey, 1993). Aside from conventional roasting and autoclaving, bio-oxidation was tested in 1987 and 1993 and various more novel processes in the 1980s.

13.5.2 Flotation

The focus of this Project is on maximizing gold recovery to a gold concentrate for auto-thermic pressure oxidation and doré production, while generating a saleable grade antimony concentrate (generally considered at least 50% Sb) from ores which contain economically significant antimony (assuming a feed cut-off of ~0.1% Sb).

The PEA results demonstrated that saleable grade antimony concentrates and oxidation-ready gold concentrates could be made from Yellow Pine's antimony-bearing ores ground to a product size of 80% passing (**P_{80}**) 100 µm. Using sodium cyanide to depress the pyrite and arsenopyrite in the antimony circuit, the optimum test yielded a 51% antimony concentrate at an antimony recovery of 75% with just 1% of the gold lost to the antimony concentrate. Subsequent gold flotation recovered roughly 90% of the gold to a rougher concentrate assaying 27 grams per metric tonne (**g/t**) gold (Au) and 12% sulfur (S).

For the prefeasibility program, flowsheet development was conducted on two composites; the Yellow Pine High Antimony (**YPH**) composite and the Yellow Pine Low Antimony (**YPL**) composite. These were each blended to represent the average feed grades of gold, antimony and sulfur for the antimony-rich material, and gold and sulfur for the antimony-poor material expected over the four phases of mining as set forth in the PEA. Confirmatory testing was also conducted on two major composites broadly representing key early production material (Early Production High antimony [**EPH**] and Low antimony [**EPL**]) and seventeen variability composites sourced from points around the Yellow Pine Deposit (Gajo, 2014a; Gajo, 2014b).

Most of the grind optimization testwork was completed on the low antimony composites, which represents material that is more plentiful in the deposit. This identified a grind product P_{80} of 75 µm to be the preferred target. This grind target was applied to the YPH composite ore and compared to the results of the same test at a target P_{80} of 100 µm. The finer grind was found to benefit the recovery of gold and sulfur in YPH while improving the selectivity against gangue minerals in both circuits. Reagent optimization testwork was then undertaken to:

1. On the YPH composite: produce concentrates of both stibnite (antimony) and pyrite/arsenopyrite (gold) while minimizing reagent use and maximizing recoveries within the respective circuits. The antimony flowsheet goals were to produce a concentrate grading 50% Sb while recovering a minimum of 70% of the antimony and rejecting the maximum amount of gold from the circuit. The gold flowsheet goals were to recover the maximum amount of gold to a concentrate initially grading 10% sulfur but later determined by

the autoclave design group to be sufficient at 5% sulfur for auto-thermic autoclaving. For the most part, gold rougher flotation achieved the 5% goal but cleaning flowsheets were also developed in the case that higher sulfur grades were needed for pre-oxidation. In the final flowsheet, ore was ground in a fully stainless steel environment to a P_{80} of 75 µm with 200 g/t lime and 75 g/t sodium cyanide; stibnite was floated using 355 g/t lead nitrate and 15 g/t Aerophine 3418A, then cleaned without regrinding using 10 g/t sodium cyanide in each of two stages of cleaning. Methyl isobutyl carbinol (**MIBC**) was used as the frother. The baseline flowsheet included subsequent pyrite/arsenopyrite rougher flotation, conducted using 400 g/t copper sulfate and 200 g/t potassium amyl xanthate (**PAX**).

2. On the YPL composite; grind optimization testwork was completed by testing the flotation response of gold at primary grind product targets of P_{80} 55, 75, 100, 125 and 180 µm. There were improvements to the recovery of gold with each successively finer grind until 75 µm was achieved, after which the recovery reached a plateau. This identified a grind P_{80} of 75 µm to be the preferred target. In the final flowsheet, ore was ground to a P_{80} of 75 µm with pyrite/arsenopyrite rougher flotation conducted using 200 g/t copper sulfate and 125 g/t PAX. MIBC was used as the frother.

These flowsheets were then used to test the response to antimony rougher flotation of ten Yellow Pine high antimony variability samples, and gold rougher flotation on a suite of nineteen variability samples. Results from those tests are shown below in Table 13.5 and Table 13.6.

Table 13.5: Yellow Pine Antimony Rougher Flotation Recoveries

Composite/ Test ID	Feed Grade (calc)				Concentrate Grade				Recovery			
	Au (g/t)	As (%)	Sb (%)	S (%)	Au (g/t)	As (%)	Sb (%)	S (%)	Au (%)	As (%)	Sb (%)	S (%)
YPH-CF22	1.98	0.25	0.33	1.04	12.56	1.38	10.71	9.63	17.0	14.6	87.3	24.9
YPH-BF13	2.11	0.25	0.39	1.06	7.56	0.71	10.50	6.92	12.0	9.4	89.3	21.8
YP105	1.66	0.23	0.76	1.63	2.66	0.26	9.31	6.15	9.5	6.5	73.1	22.4
YP107	1.21	0.39	0.11	1.24	1.52	0.49	2.57	2.42	3.0	3.0	55.3	4.7
HSTK139	2.27	0.33	0.29	1.64	3.10	0.34	4.52	3.15	7.4	5.7	84.8	10.4
HSTK144	1.82	0.28	0.08	1.05	5.77	0.97	2.27	3.22	8.9	9.6	76.5	8.6
EP5	2.66	0.55	0.44	1.31	9.75	1.75	10.44	7.51	13.8	12.1	89.5	21.6
EP6	3.29	0.55	0.17	1.67	16.52	2.95	7.85	10.10	8.3	8.9	76.7	10.0
EP7	3.52	0.31	0.97	1.69	10.64	0.64	19.13	10.26	14.7	9.9	96.0	29.5
EPH	2.51	0.46	0.68	1.58	5.38	0.89	13.24	7.53	10.2	9.4	92.6	22.7
Average	**2.30**	**0.36**	**0.42**	**1.39**	**7.55**	**1.04**	**9.05**	**6.69**	**10.5**	**8.9**	**82.1**	**17.7**

Rougher recoveries include the gold recovered to the antimony rougher concentrate, much of which is ultimately diverted to the gold circuit feed through the antimony cleaner tailings.

Table 13.6: Yellow Pine Gold Rougher Flotation Recoveries

Composite/ Test ID	Head Grade (calc)			Concentrate Grade			Unit Recovery		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)
EP1	2.67	0.55	1.25	15.13	3.09	7.40	89.6	88.7	93.9
EP2	0.97	0.47	1.20	5.47	2.82	7.28	90.5	95.9	97.2
EP3	2.71	0.21	0.92	20.59	1.55	7.02	94.5	94.0	95.2
EP4	2.31	0.25	0.89	17.85	1.90	6.84	96.2	95.1	96.1
EP5	2.66	0.55	1.31	11.74	2.42	5.39	93.2	91.7	95.4
EP6(1)	3.29	0.55	1.67	14.35	2.29	7.20	94.5	91.8	95.3
EP7(1)	3.52	0.31	1.69	14.09	1.28	5.67	93.5	90.4	95.0
EPH-CL1(1)	2.51	0.46	1.58	11.64	2.15	6.54	92.1	92.9	95.6
EPH-LCT(1)	3.24	0.45	1.59	16.01	2.26	6.92	95.0	93.6	96.4
EPL	1.73	0.40	1.13	8.65	1.98	5.80	92.0	92.4	94.9
HSTK139(1)	2.27	0.33	1.64	9.03	1.31	6.79	89.1	88.5	96.0
HSTK144(1)	1.82	0.28	1.05	7.67	1.17	4.66	90.7	89.1	95.2
YP107(1)	1.21	0.39	1.24	4.05	1.34	4.44	79.6	80.3	86.7
YP108	1.89	0.48	1.25	11.35	2.78	7.56	93.8	91.1	94.6
YP119	2.72	0.45	1.55	10.13	1.62	5.95	91.1	87.3	93.7
YP130	2.28	0.38	0.86	16.40	2.68	6.30	95.8	94.7	98.0
YPH-CF22(1)	1.98	0.25	1.04	9.47	1.22	4.46	92.1	89.5	91.6
YPH-LCT1(1)	2.11	0.25	1.06	11.00	1.29	4.85	94.4	90.3	95.4
YPL	1.86	0.36	1.01	14.03	2.69	7.65	94.3	92.5	94.8
Averages	**2.30**	**0.39**	**1.26**	**12.03**	**1.99**	**6.25**	**92.2**	**91.1**	**94.8**

Note:
(1) High Sb samples.

In this study, only limited locked cycle cleaner testing has been performed as (a) antimony is not the primary metal of interest in the study and (b) gold flotation for the most part would probably not include closed circuit cleaning. However, to demonstrate the recovery of antimony in closed circuit cleaning, and to explore the potential for higher recovery of gold to the gold concentrates, the key composites have been tested in locked cycle mode. In the case of the YPH and EPH composites, for locked cycle testing, in addition to antimony cleaning as described earlier, the gold rougher concentrate was then open-circuit cleaned in a single stage with 10 g/t PAX followed by scavenging with a further 10 g/t PAX. The pyrite/arsenopyrite cleaner scavenger tailings were directed to final tailings. The results from a six cycle locked cycle test on each of the YPH and EPH composites are presented in Table 13.7, along with the results of one batch YPH test which produced concentrate meeting the revised pressure oxidation (**POX**) feed target.

Antimony cleaner recoveries were 86% and 91% to concentrates assaying 57% and 62% antimony and 9 and 11 g/t gold. The gold lost to the antimony product was 2.5% and 3.1%. The gold cleaner concentrates assayed 25.0 and 34.4 g/t Au, and 11.1% and 15% S, respectively. Overall gold recoveries to the cleaner concentrates were 88.3% and 88.7%. The gold rougher concentrates assayed from 11 g/t to 16 g/t Au, and from 4.9% to 6.9% S. Overall gold recoveries to the locked cycle test rougher concentrates were 92% for both tests, the batch test recovery was quite low but expected to increase significantly with the closed cycle operation of the antimony circuit.

Table 13.7: Antimony and Gold Flotation from Yellow Pine High Antimony Samples

High Sb Samples	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
Yellow Pine High Sb											
LCT Sb Final Concentrate	11.4	0.57	9.28	0.19	57.23	28.10	n/a	2.5	0.4	86.3	15.1
LCT Au Cleaner Concentrate	149.0	7.4	25.03	2.84	0.48	11.08	3.55	88.3	83.8	9.5	78.1
LCT Au Rougher Concentrate	353.6	17.7	11.00	1.29	0.24	4.85	2.59	92.1	90.0	11.5	81.0
LCT Au Rougher Tailings	1638	81.8	0.14	0.03	0.01	0.05	0.80	5.4	9.6	2.2	3.9
LCT Au Rougher + Cleaner Tailings	1843	92.0	0.21	0.04	0.02	0.08	0.92	9.2	15.7	4.2	6.8
BT Sb Rougher Concentrate	76.1	3.8	8.19	0.50	8.73	6.50	n/a	14.7	7.4	81.8	23.3
BT Au Rougher Concentrate	302	15.1	11.06	1.42	0.38	5.02	1.25	78.5	83.2	14.2	71.3
BT Au Rougher Tailings	1625	81.1	0.18	0.03	0.02	0.07	n/a	6.9	9.4	4.0	5.4
Early Production High Sb											
LCT Sb Final Concentrate	19.1	0.95	10.50	0.33	62.10	26.60	n/a	3.1	0.7	91.1	15.9
LCT Au Cleaner Concentrate	168.1	8.3	34.40	4.78	0.59	15.00	2.95	88.7	87.9	7.6	78.7
LCT Au Rougher Concentrate	374.9	18.6	16.00	2.26	0.29	6.92	n/a	92.0	92.9	8.3	81.0
LCT Au Rougher Tailings	1619	80.4	0.20	0.04	0.01	0.06	0.85	4.9	6.4	0.6	3.0
Note: LCT - Locked Cycle Test, BT - Batch Test											

In locked cycle testing of the YPL and EPL composites, the rougher concentrates were cleaned in a single stage with 10 g/t PAX added to the last one-third of the bank. Cleaner tailings were reground with 10 g/t copper sulfate and floated with 10 g/t PAX to scavenge an additional 2% of gold from the cleaner tailings. The gold flotation results from six cycle locked cycle tests on the YPL and EPL composites are presented in Table 13.8. The final cleaner concentrate gold grades were 35.4 g/t and 28.7 g/t and the sulfur grades were 19.1% and 14.6%. In both cases, the overall gold recovery to the cleaner concentrate was just over 92%. The rougher concentrate gold grade was 10.7 g/t and the sulfur grade was 5.7%, 93.3% of the gold reported to the rougher concentrate.

Table 13.8: Gold Flotation from Low-Antimony Yellow Pine Samples

Low Sb Samples	Weight		Assays				Distribution		
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)
Yellow Pine Low Sb									
LCT Au Cleaner + Scavenger Concentrate	89.9	4.8	35.4	6.89	19.1	1.20	92.3	89.8	93.3
LCT Au Rougher + Scavenger Tailings	1766	95.2	0.15	0.04	0.07	n/a	7.7	10.2	6.7
BT Au Rougher Concentrate	577.6	14.5	10.7	2.07	5.7	1.85	93.3	92.8	94.2
BT Au Rougher Tailings	3414	85.5	0.13	0.03	0.06	n/a	6.7	7.2	5.8
Early Production Low Sb									
Au Cleaner + Scavenger Concentrate	133.1	7.1	28.7	4.99	14.6	1.20	92.2	88.5	94.4
Au Rougher + Scavenger Tailings	1730	92.9	0.19	0.05	0.07	1.50	7.8	11.5	5.6
Note: LCT - Locked Cycle Test, BT - Batch Test									

The final antimony concentrate and gold rougher and cleaner concentrates from the Yellow Pine Low and Yellow Pine High tests were subjected to full Inductively Coupled Plasma, mercury, halides and whole rock analyses, the results from which are shown in Table 13.9.

Table 13.9: ICP, WRA, and Halide Analysis of Yellow Pine Concentrates

Analyte	Units	YPH Sb Cleaner Con	YPH Au Cleaner Con	YPH Au Rougher Con	YPL Au Cleaner Con	YPL Au Rougher Con	Analyte	Units	YPH Sb Cleaner Con	YPH Au Cleaner Con	YPH Au Rougher Con	YPL Au Cleaner Con	YPL Au Rougher Con
Al	%	0.55	5.54	7.43	5.04	8.16	CO_3	%		3.55	1.25	1.20	1.85
As	ppm	4120	31000	13700	66000	21000							
Ba	ppm	50	510	660	330	610	LOI	%	49.60	9.82	4.67	18.70	7.27
Be	ppm	<5	6	<5	10	7	Al_2O_3	%	1.16	9.78	14.30	9.85	16.00
Ca	%	0.2	1.7	0.7	0.6	1.1	CaO	%	0.3	2.2	0.9	0.9	1.4
Cd	ppm	20	<10	0	<10	1	Cr_2O_3	%	0.010	1.020	0.440	0.870	0.420
Cr	ppm	60	7510	3000	110	2800	Fe_2O_3	%	4.070	19.000	8.730	30.800	10.500
Co	ppm	<10	80	36	5810	37	K_2O	%	0.8	6.0	8.1	4.8	8.1
Cu	ppm	280	970	680	1050	480	MgO	%	0.12	0.98	0.98	0.76	1.09
Fe	%	2.64	14.30	6.01	22.00	7.29	MnO	%	0.014	0.144	0.060	0.100	0.070
K	%	0.6	5.3	6.7	4.0	6.6	Na_2O	%	n/a	n/a	0.190	0.120	0.170
La	ppm	10	100	59	100	90	P_2O_5	%	0.06	0.22	0.12	0.13	0.15
Li	ppm	20	<10	10	<10	10	SiO_2	%	5.87	42.70	58.00	30.50	52.50
Mg	%	0.08	0.64	0.44	0.37	0.54	TiO_2	%	0.06	0.99	0.51	1.50	0.75
Mn	ppm	100	1190	490	660	500	V_2O_5	%	0.002	0.019	<0.01	0.020	0.020
Mo	ppm	30	130	67	90	50	SUM	%	62.0	92.9	97.0	99.1	98.4
Ni	ppm	<10	3560	1450	2560	1320							
P	%	0.02	0.10	0.06	0.05	0.06	Hg	ppm	252.0	5.23	3.01	11.90	3.72
Pb	ppm	2290	680	599	30	76	Se	ppm	n/a	n/a	3	n/a	2
Sb	ppm	581952	4540	3140	3600	951	Bi	ppm	n/a	n/a	0.2	n/a	0.2
Sc	ppm	<5	6	<5	<5	<5	Ag	ppm	353.0	n/a	9.7	n/a	9.8
Sn	ppm	<50	<50	7	<50	9							
Sr	ppm	20	230	140	130	170	F	%	n/a	n/a	0.1450	n/a	0.1310
Ti	%	0.04	0.64	0.31	0.83	0.46	Cl	ppm	n/a	n/a	150	n/a	150
V	ppm	<10	110	60	110	64							
W	ppm	100	1840	408	230	112							
Y	ppm	<5	37	21	20	27							
Zn	ppm	2600	190	85	390	266							

13.5.3 Gold Concentrate Upgrading for Sales

Limited batch upgrading testing of gold concentrates from the two Yellow Pine composites for third-party sales indicated that it is possible to clean from rougher concentrates grading 11 g/t Au to concentrates grading 48 - 49 g/t Au in three stages of cleaning. Cleaning losses from the YPH tests were only 4.1%, however 14.7% of the gold was tied up in the antimony rougher concentrate, much of which would be rejected back to the gold circuit feed in locked cycle operation and its behavior in gold cleaning is unknown, while 15% in the gold was lost in cleaning the YPL rougher concentrate to a grade of 48 g/t.

13.5.4 Leaching of Flotation Tailings

Cyanidation leaches were conducted on the rougher or combined rougher/cleaner flotation tailings from tests on thirteen variability composites from Yellow Pine. A cyanide concentration of 1.25 g/L was used and for most tests the leach time was 48 hours; however for tests where kinetics samples were taken, the leach was complete within 10 hours (Gajo, 2014b). The results are summarized in Table 13.10.

Table 13.10: Yellow Pine Flotation Tailings Leach Extractions

Composite ID	Leach Feed Type	Gold Grade		Gold Recovery (%)
		Feed (g/t)	Residue (g/t)	
EP1	Rougher Tailings	0.28	0.25	11.8
EP2	Rougher Tailings	0.13	0.12	9.7
EP3	Rougher Tailings	0.20	0.18	10.7
EP4	Rougher Tailings	0.13	0.12	8.6
EP5*	Rougher Tailings	0.19	0.17	11.6
EP6*	Rougher Tailings	0.18	0.17	7.6
EP7*	Rougher Tailings	0.25	0.23	6.5
YP107* (cleaner)	Rougher + Cleaner Tailings	0.38	0.36	4.2
YP108	Rougher Tailings	0.13	0.11	16.9
YP119	Rougher Tailings	0.33	0.30	10.0
YP130	Rougher Tailings	0.11	0.09	14.3
HSTK144 (cleaner)(1)	Rougher + Cleaner Tailings	0.26	0.24	6.3
HSTK139*	Rougher Tailings	0.20	0.18	11.9
	Averages	**0.21**	**0.19**	**10.0**

Note:
(1) High Sb Samples.

13.5.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec Metallurgical Technologies (**Dynatec**) on Yellow Pine gold flotation concentrate produced during that program. Neither regrinding nor acidulation of the concentrate was employed prior to oxidation testing. The concentrate, grading 28.9 g/t Au, 12% S and 1.6% CO_3, was tested at 200, 215 and 230 °C each with samples taken at 40, 60 and 80 minutes. Sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for the two highest temperatures tested (Masters, 2012).

The residues were then leached at McClelland Laboratories through bottle roll leaching "as is" and after regrinding to P_{80} 45 µm. The results indicated that the gold was readily leached into solution, with recoveries of 96% for the "as is" leach and 99% for the reground residues. The silver did not leach well. Cyanide consumption in the leaches ranged from 3.9 kilograms per metric tonne (**kg/t**) for the "as is" leach and 4.3 kg/t for the reground residue leach (McClelland, 2012).

In the PFS phase of testwork, confirmation pressure oxidation tests were conducted at SGS Lakefield on bulk cleaner concentrate produced from a 13 cycle locked cycle test on low antimony Yellow Pine Composite P2 (Jackman, 2014a). One oxidation test was also conducted on remaining Sub-A3 concentrate from the PEA study (Jackman, 2014b). In total, four pressure oxidation tests were conducted at 220°C under 75 psi oxygen (O_2) for 60 minutes, with two tests acidulating the concentrates prior to POX and the final two using no acidulation to better mimic plant conditions as recommended by Dynatec. Sulfide oxidation averaged 97.5% in the acidulated tests and averaged 98.7% in the non-acidulated tests. The resulting slurries were then conditioned in a hot cure tank at 95 °C for 2 hours prior to being forwarded on for further testing. Hot cure solution assays showed similar average arsenic in the test samples at 2.9 ppm acidulated and 2.8 ppm non-acidulated. These results confirmed the recommended autoclave conditions as presented by Dynatec in the PEA program and that acidulation was not needed for the Yellow Pine concentrates.

Cyanide leaching of the PFS program POX slurries was completed at SGS Lakefield and evaluated baseline cyanidation extraction and effect of use of flotation tailings in slurry neutralization (Jackman, 2014a; Jackman, 2014b). Residue samples from three of the PFS program pressure oxidation runs underwent six cyanidation leaches, without regrinding, to determine the extraction of gold. For the three parallel straight cyanidation leaches after washing, the average gold extraction was 98.9% and silver extraction was negligible, while consumptions averaged 0.57 kg NaCN per tonne of POX solids and 21.7 kg lime per tonne of POX solids. There appears to have been anomalously high lime consumption for the Sub-A3 POX cyanidation, so it is possible that the actual lime consumption would be lower. It was noted that the recovery of silver may be enhanced by using different treatment methods on the POX discharge such as a lime boil rather than hot cure.

The effect of using flotation tailings versus limestone for stage 1 neutralization (to pH 4.5) prior to cyanidation on partially washed POX discharge slurry (34% PLS and 66% distilled, deionized water) was also evaluated. It was found that essentially all of the equivalent mass of tailings from the flotation test would be required to bring the washed POX slurry to a pH of 4.5 in stage one of neutralization, as compared to 174 g of limestone per liter. The subsequent amount of lime needed to bring both slurries from pH 4.5 up to pH 10.0 was similar at 31.6 and 25.7 g/L. Both neutralization approaches yielded 97.8% gold extraction in subsequent cyanidation.

13.5.6 Yellow Pine Oxides

Leaching of Yellow Pine oxides was not tested in the PFS program, but has been studied in the past (in addition to the heap leach operation previously described); using a grind of P_{100} 150 µm, those tests yielded gold extractions of 84% to 89% (Albert, 1997). It is generally believed that almost all Yellow Pine oxides have already been processed so existing oxide mineralized material is minimal.

13.6 HANGAR FLATS DEPOSIT

13.6.1 Historical Metallurgy

Hangar Flats ores were milled through the Meadow Creek mill from 1932 to 1938. Only high-grade antimony ore was floated during that period, assaying over 4% antimony in the mill feed.

Of all the testwork completed on ores from the area in the past few decades, no testwork has been identified that dealt specifically with the Hangar Flats/Meadow Creek area ores.

13.6.2 Flotation

The Hangar Flats High antimony (**HFH**) composite was blended to represent the average feed grade of gold, antimony and sulfur expected over the four phases of mining as set forth in the PEA.

Grind optimization testwork was completed on the other, low antimony composites, and applied to the HFH composite. As mentioned in the Yellow Pine antimony-gold bearing section, early testwork on the new high antimony composites (both Hangar Flats and Yellow Pine) indicated the need for stainless steel grinding to maintain higher pulp potentials for good stibnite flotation (Gajo, 2014a).

In the final flowsheet on Hangar Flats antimony-rich material, ore was ground to a product size of P_{80} 75 µm with 200 g/t lime and 75 g/t sodium cyanide, stibnite was floated using 355 g/t lead nitrate and 15 g/t Aerophine 3418A, then cleaned without regrinding using 10 g/t sodium cyanide in the first of two stages of cleaning and 5 g/t Aerophine 3418A in the final stage of cleaning. The subsequent pyrite/arsenopyrite rougher flotation was conducted using 275 g/t copper sulfate and 150 g/t PAX then cleaned in a single stage with 10 g/t PAX followed by scavenging with a further 5 g/t PAX. MIBC was used as the frother throughout. Results of the batch antimony rougher flotation tests conducted with this flowsheet on six Hangar Flats antimony-bearing samples are shown in Table 13.11 below. The average antimony rougher flotation recovery was 84%.

Table 13.11: Hangar Flats Antimony Rougher Flotation Results

Composite/ Test ID	Feed Grade (calc)				Concentrate Grade				Recovery			
	Au (g/t)	As (%)	Sb (%)	S (%)	Au (g/t)	As (%)	Sb (%)	S (%)	Au (%)	As (%)	Sb (%)	S (%)
HFH-BF17	1.86	0.39	0.48	1.06	2.80	0.47	8.55	4.35	7.0	5.5	83.1	18.9
HFH-CF22	1.83	0.36	0.45	1.07	2.91	0.47	9.85	5.40	5.7	4.6	79.4	18.1
HF33	1.20	0.42	0.14	2.19	1.73	0.59	4.42	3.87	3.3	3.2	75.0	4.0
HF38	1.43	0.39	0.17	0.82	4.04	1.04	4.39	3.59	8.5	7.1	77.3	11.8
HF43	2.61	0.24	0.88	1.63	3.61	0.27	12.74	6.41	8.5	7.2	89.3	24.2
Comp A(1)	2.56	0.50	0.40	1.29	7.15	1.23	6.19	5.07	16.4	14.6	90.6	23.2
Average	**1.91**	**0.38**	**0.42**	**1.35**	**5.10**	**1.07**	**7.87**	**5.51**	**10.9**	**9.8**	**83.9**	**19.0**

Note:
(1) Contained 52% Hangar Flats material.

The Hangar Flats Low antimony (**HFL**) composite for PFS testwork was blended to represent the average feed grades of gold and sulfur expected over the four phases of mining as set forth in the PEA. This composite had a target feed sulfur grade of 0.8% which brings it just bordering the transitional ore range. It was expected to have slightly lower gold flotation recoveries and elevated tailings leach gold extractions than higher sulfide containing ores.

Applying the PEA flowsheet to the HFL composite resulted in the production of a rougher concentrate grading 12.8 g/t Au, 7.4% S and recovered 81.4% of the gold into that rougher concentrate. A single stage of cleaning upgraded the concentrate to 28.3 g/t Au and 16.9% S with a cleaner stage recovery of 94.3% of the gold for an overall gold recovery of 76.8%.

Grind optimization testwork was completed by testing the flotation response of gold at primary grind targets P_{80} of 55, 75, 100, 125 and 150 µm. There did not to appear to be any improvement to the recovery of gold with each successively finer grind so the previously identified grind P_{80} of 75 µm preferred for Yellow Pine and West End (these two deposits containing the majority of recoverable gold in the Project) was chosen as the target for the Hangar Flats program.

Reagent optimization testwork was then undertaken to identify the maximum amount of gold to be recovered from the HFL composite with the optimum dosages of reagents. It was found with the new composite that the sodium silicate use could be discontinued, however copper sulfate dosage required doubling from 140 g/t to 250 g/t and xanthate was increased by 35% from 140 g/t to 200 g/t to maximize the rougher recovery of gold. The rougher concentrate was cleaned in a single stage with 60 g/t PAX, added in one-third increments through the cleaner. Cleaner tailings were reground with 10g/t copper sulfate and floated with 10 g/t PAX to recover an additional 2% of gold from the cleaner tailings. MIBC was used as the frother throughout. It is recommended that for the plant environment, a cyclone be used to return the coarse fraction of the cleaner tailings to the primary ball mill for reprocessing with the rougher stream. Batch rougher testwork on the HFL composite using this flowsheet produced a rougher concentrate which averaged 9.8 g/t Au and 5.4% S at a mass pull of 14% and recovery of 85.8% of the gold.

Table 13.12: Hangar Flats Gold Rougher Flotation Results

Composite/ Test ID	Head Grade (calc)			Concentrate Grade			Unit Recovery		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)
HFH-LCT1(1)	1.85	0.39	1.06	9.34	1.67	4.75	92.1	77.3	95.5
HFH-CF22	1.83	0.36	1.07	11.48	1.88	6.00	91.4	74.7	94.3
HF33(1)	1.20	0.42	2.19	4.80	1.87	11.35	73.7	82.2	95.8
HF38(1)	1.32	0.40	0.84	6.69	2.08	4.21	91.3	92.0	93.4
HF43(1)	2.99	0.23	1.61	13.08	0.86	5.78	90.1	77.2	89.5
HFL-CF10	1.58	0.44	0.80	8.73	2.04	4.78	86.1	72.3	93.6
HF28.1	1.14	0.55	1.13	5.37	2.62	5.46	92.2	92.9	94.3
HF31	1.46	0.59	1.42	4.15	1.62	4.46	74.2	72.4	82.3
HF42.1	0.97	0.39	0.76	6.17	2.56	5.00	91.2	93.9	94.4
Average (Sulfide)	**1.59**	**0.42**	**1.21**	**7.76**	**1.91**	**5.75**	**86.9**	**81.6**	**92.6**
HF32(2)	1.38	0.78	0.05	5.10	1.30	0.20	33.4	14.7	31.3
HF29.3(2)	1.18	0.39	0.16	4.20	0.90	0.60	37.1	23.5	39.8
HF29.2(2)	1.55	0.85	0.25	6.00	1.20	0.70	41.7	14.5	31.6
Comp F(2)	1.10	0.56	0.60	5.90	3.00	5.60	49.0	48.0	84.0
HF30.2(2)	1.64	0.70	0.74	9.10	2.50	5.40	61.0	39.5	79.7
Average (Oxide-Transition)	**1.37**	**0.66**	**0.36**	**6.06**	**1.78**	**2.50**	**44.4**	**28.0**	**53.3**

Notes: *
(1) *High Sb samples*
(2) *Oxide and transitional material samples*

Gold rougher flotation results from tests on nine sulfide samples, using the PFS flowsheet are shown in Table 13.12. The average gold recovery from the Hangar Flats sulfide samples was 87%. Data from five oxide-transition samples are also shown. These yielded inferior flotation recoveries, but gold invariably leached well from the tailings of these samples, averaging 80% extraction of remaining gold (Ratnayake, 2013c; Gajo, 2014c; Gajo, 2014b).

The results from a six cycle locked cycle test on the antimony-rich HFH composite are presented in the following table (Table 13.13). Antimony cleaner recovery was 80% to a concentrate assaying 58% antimony and 4.9 g/t gold, while the gold lost to the antimony product was 1.7%. This compared with 83% in the antimony batch rougher test suggests a cleaner stage recovery of 97%. The gold cleaner concentrate assayed 28.2 g/t gold and 14.5% S and had an overall gold recovery of 86.3% with a cleaner stage recovery of gold of 95.3%. The gold rougher concentrates from the locked cycle and batch tests assayed from 9.3 to 9.5 g/t Au and 4.8% to 5.3% S. Overall gold recovery to the locked cycle test gold rougher concentrate was 90.6%.

Some 14% of the antimony reported to the gold concentrate indicating possible upside in this recovery to the antimony concentrate with further refining of the antimony rougher flotation circuit.

Table 13.13: Antimony and Gold Flotation from the Hangar Flats High Antimony Composite

Hangar Flats High Sb	Weight		Assays					Distribution			
	Dry	%	Au (g/t)	As (%)	Sb (%)	S (%)	CO_3 (%)	Au (%)	As (%)	Sb (%)	S (%)
LCT Sb Final Concentrate	12.5	0.62	4.93	0.15	58.06	25.83	n/a	1.7	0.2	80.4	15.3
LCT Au Cleaner Concentrate	113.4	5.7	28.24	4.69	1.10	14.52	2.40	86.3	68.2	13.8	77.9
LCT Au Rougher Concentrate	359.7	18.0	9.34	1.67	0.40	4.75	2.06	90.6	77.1	16.0	80.9
LCT Au Rougher Tail	1629	81.4	0.18	0.11	0.02	0.05	1.35	7.8	22.7	3.6	3.9
LCT Au Rougher + Cleaner Tail	1875	93.7	0.24	0.13	0.03	0.08	1.42	12.0	31.6	5.8	6.8
BT Sb Rougher Concentrate	58.3	2.9	3.42	0.46	13.20	6.45	n/a	5.9	3.5	83.4	18.0
BT Au Rougher Concentrate	304	15.3	9.49	1.80	0.40	5.31	2.20	85.8	73.0	13.1	77.3
BT Au Rougher Tail	1630	81.8	0.17	0.11	0.02	0.06	n/a	8.2	23.5	3.5	4.7
Note: LCT - Locked Cycle Test, BT - Batch Test											

In locked cycle testing the HFL composite, the rougher concentrates were cleaned in a single stage with 50 g/t PAX added in two doses of 25g/t each down the bank, followed by scavenger flotation with an additional 10 g/t PAX. MIBC was used as frother throughout. Cleaner tailings were returned to the ball mill for reprocessing with fresh feed, which improved the rougher gold recovery over batch testing.

The gold flotation results from a six cycle locked cycle test on the HFL composite as well as a batch rougher without closed circuit cleaning are presented in Table 13.14. The final cleaner concentrate gold grade was 28.3 g/t and the sulfur grade was 14.9% with overall gold recovery to the cleaner concentrate just over 82%. The rougher concentrate gold grade was 10.5 g/t and the sulfur grade was 5.9% with 79.3% of the gold reporting to the rougher concentrate.

Table 13.14: Gold Flotation from Hangar Flats Low Antimony Composite

Hangar Flats Low Sb	Weight		Assays				Distribution		
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)
LCT Au Cleaner + Scav Conc	88.2	4.8	28.3	6.13	14.91	1.15	82.9	63.3	89.7
LCT Au Rougher + Scav Tail	1740	95.2	0.30	0.18	0.087	n/a	17.1	36.7	10.3
BT Au Rougher Concentrate	253.6	12.7	10.5	2.60	5.9	1.20	79.3	64.3	92.4
BT Au Rougher Tail	1745	87.3	0.40	0.21	0.07	n/a	20.7	35.7	7.6
Note: LCT - Locked Cycle Test, BT - Batch Test									

Antimony cleaner concentrates and gold rougher and cleaner concentrates from the tests on HFH and HFL samples were subjected to full ICP, mercury, halide and whole rock analysis scans, the results from which are shown in Table 13.15.

Table 13.15: ICP, WRA and Halide Analysis of Hangar Flats Concentrates

Analyte	Units	HFH Sb Cleaner Con	HFH Au Cleaner Con	HFH Au Rougher Con	HFL Au Cleaner Con	HFL Au Rougher Con	Analyte	Units	HFH Sb Cleaner Con	HFH Au Cleaner Con	HFH Au Rougher Con	HFL Au Cleaner Con	HFL Au Rougher Con
Al	%	0.56	6.24	7.52	6.07	0.45	CO_3	%		2.40	2.20	1.15	1.20
As	ppm	1420	48600	18900	57800	>10000							
Ba	ppm	70	390	650	330	88	LOI	%	54.50	15.70	6.70	17.00	7.71
Be	ppm	<5	7	6	8	<5	Al_2O_3	%	1.09	12.50	14.40	11.50	16.40
Ca	%	0.6	1.0	1.2	0.6	0.6	CaO	%	0.80	1.50	1.52	0.90	1.13
Cd	ppm	<10	<10	0	<10	<10	Cr_2O_3	%	0.01	0.37	0.29	0.00	0.31
Cr	ppm	50	2400	2030	4690	2020	Fe_2O_3	%	1.42	23.70	9.79	26.50	11.20
Co	ppm	<10	50	31	70	34	K_2O	%	0.80	5.40	6.32	4.50	6.28
Cu	ppm	460	640	440	930	442	MgO	%	0.47	1.17	1.25	0.74	1.15
Fe	%	0.97	15.60	6.80	18.50	7.41	MnO	%	0.04	0.08	0.07	0.00	0.06
K	%	0.7	4.2	5.3	3.7	0.3	Na_2O	%	n/a	n/a	0.60	n/a	0.85
La	ppm	<10	100	89	100	40	P_2O_5	%	0.05	0.17	0.17	0.11	0.17
Li	ppm	<10	20	20	10	<10	SiO_2	%	6.59	42.00	54.90	35.40	51.40
Mg	%	0.30	0.67	0.64	0.45	0.23	TiO_2	%	0.05	1.39	0.79	1.54	0.97
Mn	ppm	290	610	520	640	393	V_2O_5	%	0.00	0.02	0.01	0.02	0.01
Mo	ppm	<10	40	37	70	48	SUM	%	65.8	104.0	96.8	98.2	97.6
Ni	ppm	<10	1080	969	2200	943							
P	%	0.02	0.07	0.06	0.05	0.07	Hg	ppm	342.00	33.10	15.30	67.60	38.00
Pb	ppm	1140	580	545	<20	22	Se	ppm	n/a	n/a	3.00	n/a	2
Sb	ppm	579566	11000	3280	5260	1830	Bi	ppm	n/a	n/a	0.2	n/a	0.2
Sc	ppm	<5	6	<5	6	<5	Ag	ppm	684.0	n/a	13.00	n/a	12.30
Sn	ppm	<50	<50	6	<50	<50							
Sr	ppm	40	220	250	200	81	F	%	n/a	n/a	0.1390	n/a	0.2280
Ti	%	0.03	0.79	0.47	0.92	<0.01	Cl	ppm	n/a	n/a	150.0	n/a	<50
V	ppm	<10	110	72	110	23							
W	ppm	<50	<50	66	250	<50							
Y	ppm	<5	17	18	21	10							
Zn	ppm	1540	210	76	330	144							

13.6.3 Gold Concentrate Upgrading for Sales

Limited batch testing on the HFH and HFL composites of gold concentrate upgrading to explore the option of direct sales of gold concentrate yielded concentrates grading 48.7 g/t and 37.2 g/t Au, and 27.7% and 24.7% S respectively. Cleaner losses were 8.3% and 18.5% for the HFH and HFL composites. Further testing may improve this, although the mineralogy points to the finest disseminated pyrite and arsenopyrite (*i.e.* those preferentially rejected in cleaning) being the most enriched in gold, so cleaner performance to high-grade samples may always be relatively poor.

13.6.4 Leaching of Flotation Tailings

The tailings from bulk flotation tests conducted on HFH and HFL composites were subjected to scoping cyanide leaching tests (Gajo, 2014c). For the high antimony composite (HFH) cyanidation extracted 36% of the gold in the tailings, with the recovery effectively representing roughly 0.12 g/t recoverable gold. For the low antimony sample, which at 0.81% sulfur in feed is considered bordering transitional, 68% of the gold remaining in the tailings was extracted, representing a highly economic 0.19 g/t recoverable gold. Leaching of flotation tailings on six additional sulfide variability samples showed a range of recoveries from 7% to 19%, extracting from 0.01 to 0.06 g/t of gold.

Based on the scoping results, a developmental leach study was performed which utilized the combined rougher + cleaner tailings from the HFL locked cycle test. Results of a matrix of tests evaluating three different feed percent solids and three different cyanide concentrations identified the optimum gold extraction from the HFL tailings as being 53.4% of the gold contained in the tailings. A batch agitated tank carbon-in-pulp (**CIP**) leach was then conducted at 45% solids and 0.25 kg cyanide per tonne of tailings, which resulted in a gold extraction of 53.9%.

The combined gold recoveries from the locked cycle test and tailings leach for the HFH and HFL composites were calculated to be 91% and 92%, respectively.

13.6.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec in Fort Saskatchewan, Alberta on Hangar Flats gold flotation concentrates produced from the global Sub-A1 composite. Neither regrinding nor acidulation was used in these tests. The concentrate, grading 27.1 g/t Au, 11.8% S and 1.8% CO_3, was tested at 200, 215 and 230 °C each with samples taken at 40, 60 and 80 minutes. Sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for the two highest temperatures tested (Masters, 2012).

The residues were leached at McClelland Laboratories through bottle roll leaching “as is” and after regrinding to P_{80} 45µm. The total residence time in the leach was 72 hours. The results indicated that the gold was readily leached, with recoveries of 96% “as is” and 97% on reground residues. The silver extraction was less than 1% in both cases. Cyanide consumption in the tests was lower than with Yellow Pine, at 3 kg/t for the “as is” leach and 3.2 kg/t for the reground residue leach (McClelland, 2012).

In the PFS phase of testwork, a confirmation pressure oxidation test was completed on the remaining Sub-A1 concentrate that had been used for the Dynatec PEA study (Jackman, 2014b), using conditions recommended in Dynatec’s report. The oxidation products were forwarded on for neutralization, CIP, cyanide destruction (CND), and environmental studies. Tests on the samples were conducted at 220 °C under 75 pounds per square inch (**psi**) of O_2 for 60 minutes, and included acidulation of the concentrates to pH1.8 prior to POX per internal SGS operating procedures. Sulfide oxidation averaged 95.5% in the tests, with POX solutions containing 2.57 ppm arsenic and 0.7 mg/L antimony. The results confirmed the Dynatec autoclave operating conditions, but future lab tests should use the full test recommendations from Dynatec and not acidulate the feed.

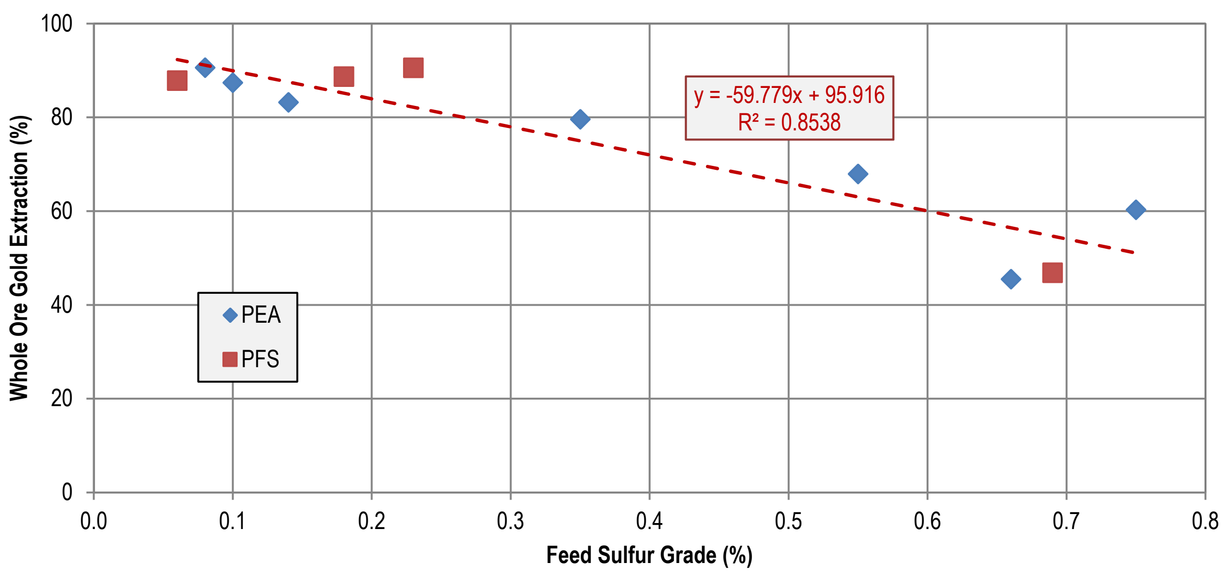
The samples from the three PFS program pressure oxidation runs were combined, fully washed and underwent a single CIP cyanidation leach to determine the extraction of gold (Jackman, 2014b). There were no regrinding tests performed. The gold extraction from the test was 97.8% and silver extraction was about 9%, while consumptions averaged 0.28 kg NaCN per tonne of POX solids and lime averaged 11.8 kg lime per tonne of POX solids. Again, cyanide consumption was lower with Hangar Flats POX residue than was seen with Yellow Pine.

13.6.6 Hangar Flats Oxides

There is a small amount of oxide mineralized material in the Hangar Flats Deposit and no past processing or testing reports have been found to indicate it has been worked or studied. Furthermore, no mineralogical gold balancing studies have been conducted on oxide samples from Hangar Flats, but it is believed that the gold is fine and mostly discrete in the true oxide materials.

During the PEA phase, whole ore cyanidation leaching was conducted on various oxide and transition samples with a standard diagnostic cyanidation procedure. Leach extraction was found to correlate quite closely to sulfide content. Cyanide consumption was 1.9 kg/t, while the lime consumption averaged 3.1 kg/t in these tests (Ratnayake, 2013c). An additional four oxide and transition samples were tested in the PFS phase to build upon that data set, and the combined results are shown on Figure 13.4. Consumption of cyanide for the new samples averaged 1.8 kg/t, while the lime consumption averaged 3.3 kg/t, quite similar to the PEA sample leaches (Gajo, 2014b).

Figure 13.4: Leach Gold Extraction *vs* Sulfur Grade for Hangar Flats Oxide and Transition Samples



13.7 WEST END DEPOSIT

13.7.1 Historical Metallurgy

Oxide ores from the West End Deposit were treated by heap leaching from 1982 to 1996. The heap leach process on ores (typically assaying roughly 1.3 g/t gold) involved crushing to minus 1.25 inches (32 mm) and a heap leach cycle of 50 days. Based on the information currently available, West End sulfides have never been processed commercially.

Testing conducted in the late 1970s and 1980s at Britton Research and Coastech Research on the sulfide ores yielded total recoveries in the 70% to 80% range through a combination of flotation, sulfide oxidation (roasting or bioleaching) and cyanidation of both the flotation tailings and oxidation residues (Britton, 1978; Broughton, 1987). Limited testing on oxide ores at Kappes, Cassiday and Associates occurred in 1997 which consisted of basic bottle roll extractions and all achieved between 84% and 89% extraction of gold (Albert, 1997).

13.7.2 Flotation

The flotation of West End sulfides is relatively straightforward, however very little flotation optimization work was conducted during the PEA phase. Most of the work constituted flotation from two global composites, namely A2 and Sub-A2, to create concentrate for downstream oxidation testing. The moderate levels of muscovite/illite in the composites facilitated the production of what was deemed a POX-ready flotation concentrate after roughing alone, recovering 83.6% and 81.8% of the feed gold to rougher concentrates assaying 9.4% and 9.9% sulfur respectively. This was achieved by a bulk sulfide float at a grind P_{80} target of 100 µm with 200 g/t copper sulfate and 110 g/t PAX. A single stage of cleaning of the A2 concentrate was able to reach grades of 30.7 g/t Au and 20.8% S, at a cleaner stage gold recovery of 93.2% for an overall gold recovery of 77.9%.

It was found in the PEA program oxidation study that the level of carbonates in the West End Sub-A2 concentrates necessitated the use of an acidulation step prior to autoclaving and this forced a change in the concentrate criteria for the PFS to include the carbonate/sulfur ratio (M3 has set a target of less than 0.9:1). Therefore, the primary recommendation from the PEA program was to investigate cleaning of the concentrates to reject carbonates in order to negate the need for the costly acidulation step.

The West End Sulfide (**WES**) composite for PFS testwork was blended to represent the average feed grade of gold expected over the four phases of mining as set forth in the PEA, but lacked a specific sulfur grade target. This composite's resulting sulfur grade of 0.65% brings it squarely into the transitional ore range. It was expected to have lower gold flotation recoveries and elevated tailings leach gold extractions than the higher sulfide containing ores. Applying the PEA flowsheet to the WES composite resulted in the production of a rougher concentrate grading 11.0 g/t Au, 5.1% S and 8.7% CO_3; and recovered 79.7% of the gold into that rougher concentrate at a mass pull of 12.2%. A single stage of cleaning upgraded the concentrate to 27.2 g/t Au, 13.2% S and 8.6% CO_3 with a cleaner stage recovery of 93.1% of the gold for an overall gold recovery of 74.2%. The carbonate to sulfur ratio was taken from 1.71 to 0.65 in one stage of cleaning.

Grind optimization testwork was completed at primary grind product targets of P_{80} 55, 75, 100, 150 and 180 µm. There was improvement to the recovery of gold with each successively finer grind. The recovery of gold in the 150 µm to 180 µm range was similar, but improved in the 75 µm -100 µm range, and again with the 55 µm grind. Due to concerns with the capital and operating costs involved in grinding to 55 µm, as well as the applicability of the 75 µm target to Yellow Pine, the grind product of P_{80} 75 µm was also chosen as the target for the WES program.

PFS reagent optimization testwork was then undertaken to identify the maximum amount of gold to be recovered while rejecting the most carbonates (Gajo, 2014a). Analysis of the PEA program data showed that the Yellow Pine and Hangar Flats concentrates had a carbonate to sulfur ratio of 0.13 and 0.15, respectively, while the West End concentrate was 1.14:1 for the carbonate to sulfur ratio. This was higher than the 0.9:1 limit imposed by M3 for the PFS. Through the batch flotation optimization tests, it was found that in a single stage of cleaning, the carbonate to sulfur ratio in the cleaner concentrate could be successfully reduced by two thirds as compared to the rougher concentrate ratio while maintaining a cleaner stage recovery of 95%.

In the final flowsheet, the feed was ground to a P_{80} of 75 µm with pyrite/arsenopyrite flotation conducted using 200 g/t copper sulfate and 175 g/t PAX, then cleaned in a single stage with 50 g/t copper sulfate and 50 g/t PAX. Cleaner tailings were reground with 25 g/t copper sulfate and floated with 25 g/t PAX to recover an additional 3% of gold from the cleaner tailings. It was recommended that for the plant environment, a cyclone be used to return the coarse fraction of the cleaner tailings to the ball mill for reprocessing with the rougher stream. Batch testwork on the

WES composite based on that flowsheet produced an average rougher concentrate of 9.6 g/t Au, 4.5% S and 9.3% CO_3 at a mass pull of 13.6% and recovery of 79.3% of the gold, nearly the same as the PEA flowsheet recovery. Single stage cleaning plus regrind and scavenging of the cleaner tailings produced an average concentrate grading 28.3 g/t Au, 14.0% S and 8.8% CO_3 at a cleaner stage recovery of 94.9% and an overall gold recovery of 75.2%, slightly exceeding the PEA flowsheet recovery by 1%. This drops the carbonate to sulfur ratio from 2.0 to 0.63 with a single stage of cleaning.

The results of the subsequent six-cycle locked cycle tests, which utilized the recycle of cleaner tailings back into primary grind for additional processing, are presented in Table 13.16 with a comparative batch test. The cleaner concentrate gold grade was 29.5 - 29.7 g/t, the sulfur grade was 14.4 - 15.0% and the carbonate was 7.9 - 8.2%, both having a comfortable carbonate to sulfur ratio of 0.55 or less in the concentrate. The batch rougher tests that this LCT was designed from achieved an average 79.3% rougher recovery, which gives this test a cleaner stage gold recovery of 97.8%. The locked cycle test tailings were forwarded for cyanidation testing, detailed later in this section. It is expected that a portion of the gold losses to tailings are associated with non-sulfide gangue minerals and would be available for leach recovery.

Table 13.16: Gold Flotation of Sulfide Concentrate from West End Sulfide Samples

West End Sulfide	Weight		Assays				Distribution			CO_3:S Ratio
	Dry	%	Au (g/t)	As (%)	S (%)	CO_3 (%)	Au (%)	As (%)	S (%)	
LCT Au Cleaner + Scav Conc	80.1	4.4	29.70	3.87	14.37	7.89	77.5	81.3	93.0	0.55
LCT Au Rougher + Scav Tail	1739	95.6	0.40	0.04	0.05	9.14	22.5	18.7	7.0	
BT Au Cleaner + Scav Conc	82.3	4.1	29.53	4.20	15.01	8.16	73.5	78.0	92.4	0.54
BT Au Rougher Concentrate	261.4	13.1	10.13	1.41	4.85	9.24	80.0	83.1	94.8	1.91
BT Au Rougher Tail	1739	86.9	0.38	0.04	0.04		20.0	16.9	5.2	
BT Au Rougher + Scav Tail	1918	95.9	0.46	0.05	0.05		26.5	22.0	7.6	
Note: LCT - Locked Cycle Test, BT - Batch Test										

Gold cleaner concentrates from the test were subjected to full ICP, mercury and whole rock analysis scans, the results from which are shown in Table 13.17.

Table 13.17: ICP, WRA and Halide Analysis of West End Sulfide Gold Cleaner Concentrate

Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con	Analyte	Units	WES Au Cleaner Con
Al	%	4.81	Mn	ppm	690	LOI	%	16.50	CO_3	%	7.69
As	ppm	37500	Mo	ppm	70	Al_2O_3	%	9.09	CO_3:S	none	0.55
Ba	ppm	310	Ni	ppm	2210	CaO	%	4.70			
Be	ppm	6	P	%	0.01	Cr_2O_3	%	<.001	Hg	ppm	19.00
Ca	%	3.3	Pb	ppm	20	Fe_2O_3	%	24.80	Se	ppm	n/a
Cd	ppm	<10	Sb	ppm	380	K_2O	%	4.50	Bi	ppm	n/a
Cr	ppm	4300	Sc	ppm	7	MgO	%	1.93	Ag	ppm	n/a
Co	ppm	170	Sn	ppm	<50	MnO	%	<.001			
Cu	ppm	990	Sr	ppm	130	Na_2O	%	n/a	F	%	n/a
Fe	%	17.40	Ti	%	0.26	P_2O_5	%	0.03	Cl	ppm	n/a
K	%	3.7	V	ppm	90	SiO_2	%	36.00			
La	ppm	40	W	ppm	100	TiO_2	%	0.44			
Li	ppm	30	Y	ppm	17	V_2O_5	%	0.02			
Mg	%	1.16	Zn	ppm	1430	SUM	%	98.0			

13.7.3 Gold Concentrate Upgrading for Sales

Limited testing using the WES composite on upgrading the concentrate for shipment to third party roasters and autoclaves has also been conducted. While gold grades of up to 52 g/t were produced, cleaner losses were about 18% at this grade. Further testing would be needed to improve the stage recovery of gold, and further stages of cleaning may be able to increase the grade of the concentrates as well.

13.7.4 Leaching of Flotation Tailings

A developmental leach study was performed which utilized the combined rougher + cleaner tailings from the WES locked cycle test (Gajo, 2014c). Results from tests evaluating three different feed percent solids and three different cyanide concentrations identified the optimum gold extraction from the WES tailings as 61% of the gold contained in the tailings. A batch agitated tank CIP leach was then conducted at 45% solids and 0.25 kg cyanide per tonne of tailings, which resulted in a gold extraction of 55.3%. The combined gold recovery from the locked cycle test and tailings leach for the WES composite is calculated to be 90%.

Leaching tests have also been conducted on the flotation tailings from the 29 variability samples (Table 13.18). Results from these are shown in the variability Section (13.7.7).

13.7.5 Pressure Oxidation of Concentrates and Cyanide Leaching of Residues

During the PEA phase, three pressure oxidation tests were run at Dynatec in Fort Saskatchewan, Alberta on West End gold flotation concentrates produced from the global Sub-A2 composite during that program. No regrinding of the concentrates was conducted. An acid demand test indicated that acidulation of the concentrates would be needed prior to oxidation testing, in the order of 180 kg of sulfuric acid per tonne of concentrate in testwork. The concentrate, grading 19.7 g/t Au, 9.4% S and 10.5% CO_3, was tested twice at 215°C (acidulated and with acid addition to POX) and 230°C, each with samples taken at 40, 60 and 80 minutes. Results were similar to past testing in that sulfur oxidation was very rapid with oxidation essentially complete within 40 minutes for all conditions tested (Masters, 2012).

The pressure oxidation residues were leached at McClelland Laboratories through bottle roll leaching "as is" and after regrinding to P_{80} 45 μm. The total residence time in the leach was 72 hours. The results indicated that the gold was readily leached into solution, with recoveries of 98% "as is" and 98.2% on reground residues. The silver extraction was 7.5% and 8.0%, respectively. Cyanide consumption in the tests was similar to that of Yellow Pine, at 4.3 kg/t for the "as is" leach and 4.0 kg/t for the reground residue leach, regrinding was deemed not beneficial to the leach (McClelland, 2012).

In the PFS phase of testwork, a confirmation pressure oxidation test was completed on the remaining Sub-A2 concentrate that had been used for the Dynatec PEA study (Jackman, 2014b). The pressure oxidation conditions used were recommended by Dynatec's final report. The oxidation products were forwarded on for neutralization, CIP, CND and environmental studies.

In total, at SGS, three pressure oxidation tests were conducted at 220°C under 75 psi of O_2 for 60 minutes, including acidulation of the concentrates to pH 1.8 prior to POX. Sulfide oxidation averaged 99% in the tests with POX solutions containing 3.08 ppm arsenic and <0.5 mg/L antimony. The results supported the Dynatec autoclave recommendations, but future lab tests should confirm assumptions that acidulation is not needed with better cleaning of concentrates from West End materials.

The samples from the two PFS program pressure oxidation runs were combined, fully washed and underwent a single CIP cyanidation leach, without regrinding, to determine the extraction of gold. The gold extraction from the test was 98.3% while silver extraction was negligible; consumptions were 0.13 kg of NaCN and 10.3 kg of lime per tonne of POX solids.

13.7.6 Whole Ore Cyanidation

Thirty variability samples in total have been subjected to whole ore leaching. For the thirteen samples from the PEA phase of testwork, the kinetics were fast, generally leaching greater than 75% of the total amount in the first 5 hours of the leach. The average extraction of all transitional and oxide samples in the PEA study was 49.9%, ranging from 13.6% to 93.9%. Cyanide consumption ranged from 0.3 - 1.4 kg/t, while the lime consumption varied 0.24 - 1.5 kg/t in these tests (Ratnayake, 2013c). An additional fifteen oxide and transition samples were tested in the PFS phase to build upon that data set, showing faster yet leaching kinetics of greater than 90% in the first 5 hours. All the tests were conducted at the PEA grind size of 80 percent passing 100 microns. The average extraction of all transitional and oxide samples in the PFS study was 45.2%, ranging from 11.6% to 84.4%. Consumption of cyanide for the new samples averaged 0.45 kg/t, while the lime consumption averaged 1.37 kg/t, quite similar to the PEA sample leaches (Gajo, 2014b).

Two additional whole ore global composites, WES and West End Oxide (**WEO**) were also tested and achieved similar kinetic results, showing greater than 96% of the extraction happening in the first 6 hours of the leach (Gajo, 2014c). Extractions of 30% and 73% were achieved on these transitional (WES) and oxide (WEO) composites. Reagent consumption in these tests was much lower than in the bottle rolls, at just 0.02 - 0.19 kg/t for cyanide and 0.64 - 1.21 kg/t for lime.

13.7.7 Variability Testing

Variability testing has been conducted on twenty-nine West End samples spanning the entire spectrum of oxidation from true sulfide to essentially pure oxide (Ratnayake, 2013c; Gajo, 2014b). The results from flotation, flotation tailings leaching and whole ore leaching tests at 100 µm P_{80} grind are summarized in Table 13.18 (PFS samples below the double line). These results have been used for metallurgical forecasting purposes, as described later in this section. The average whole ore leach gold extraction of sixteen West End oxide samples (defined here as having less than 0.35% S in the feed) is 61.8%. The average flotation gold recovery of four sulfide samples (defined here as having greater than 0.85% S in the feed) is 81.4%. For the transition range samples, with feed sulfur grades from 0.35 to 0.85%, the best recoveries are achieved with a flotation of sulfides followed by leaching of the tailings, which averaged 89.9% total recovery of gold.

Table 13.18: Variability Testing on West End Samples

Composite ID	Head Grade (calc)			Concentrate Grade			Flotation Recovery			Total Extraction, %		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)	Whole Ore Leach	Tailings Leach	Float + Tailings Leach
WE2	1.63	0.12	0.61	12.61	0.92	4.64	97.9	93.0	95.7	16.0	64.6	99.3
WE11	1.67	0.62	1.44	10.52	3.91	9.46	92.3	93.1	96.4	13.6	38.5	95.3
WE14	2.35	0.48	1.03	8.37	1.86	4.32	79.2	85.5	93.2	23.3	47.3	89.0
WE15	1.88	0.26	1.55	6.44	0.86	5.65	90.2	88.5	96.2	14.6	26.1	92.8
WE17	0.74	0.04	0.35	5.32	0.32	3.16	74.5	78.6	94.8	15.7	25.9	81.1
WE20	2.78	0.34	0.05	10.75	0.93	0.35	37.8	26.4	65.3	93.9	94.3	96.5
WE21	1.18	0.08	0.07	9.13	0.37	0.60	60.9	37.1	71.8	91.6	83.1	93.4
WE22	1.53	0.07	0.02	8.56	0.22	0.09	47.4	28.3	45.8	85.5	77.0	87.9
WE23	0.92	0.05	0.01	6.05	0.19	0.07	44.2	25.5	34.5	80.0	74.4	85.7
WE24/25	1.25	0.13	0.33	8.09	0.71	3.06	66.9	56.4	94.6	46.0	85.7	95.3

Composite ID	Head Grade (calc)			Concentrate Grade			Flotation Recovery			Total Extraction, %		
	Au (g/t)	As (%)	S (%)	Au (g/t)	As (%)	S (%)	Au (%)	As (%)	S (%)	Whole Ore Leach	Tailings Leach	Float + Tailings Leach
WE26	0.64	0.03	0.10	3.67	0.18	0.92	53.4	51.4	82.5	68.7	84.5	92.8
WE27	0.63	0.03	0.16	3.90	0.22	1.46	57.1	63.3	83.3	56.1	76.1	89.7
WE28	1.72	0.05	0.10	20.00	0.50	1.77	50.7	45.8	80.4	66.5	69.7	85.1
WE29	0.52	0.07	0.32	6.90	1.00	4.91	82.0	81.9	94.2	11.3	8.6	83.5
WE30	0.30	0.11	0.18	1.90	0.50	3.54	27.2	20.1	84.2	84.4	91.5	93.8
WE33	0.37	0.07	0.17	2.70	0.40	1.55	75.4	66.8	89.6	29.0	40.0	85.3
WE34	0.82	0.09	0.14	7.10	0.70	1.70	59.2	51.9	80.7	52.8	69.9	87.7
WE35	0.64	0.22	0.55	2.30	0.90	3.46	49.1	54.4	87.4	78.0	89.6	94.7
WE38	1.39	0.21	0.17	8.60	1.20	1.94	48.3	44.5	89.1	72.7	89.9	94.8
WE39	1.45	0.17	0.31	16.50	2.30	4.50	73.5	85.3	93.9	46.5	64.8	90.7
WE40	0.76	0.13	0.39	13.60	2.40	7.74	82.5	82.9	92.6	14.8	0.0	82.5
WE41	3.16	0.31	0.89	17.10	1.70	6.63	63.7	65.8	87.2	37.2	61.9	86.2
WE42	1.53	0.24	0.57	13.80	2.20	5.66	85.2	84.3	93.7	17.2	24.5	88.9
WE43	1.68	0.18	0.41	34.50	4.10	11.10	66.7	74.9	88.2	36.9	71.6	90.5
WE44	1.70	0.11	0.41	31.10	2.60	12.30	55.6	70.1	90.6	57.1	75.7	89.2
WE45	1.34	0.03	0.42	17.20	0.40	7.37	65.3	55.8	88.8	43.8	79.1	92.8
WE47	0.90	0.11	0.29	23.00	2.90	7.96	82.9	84.7	90.0	29.4	44.7	90.5
WEO	0.91	n/a	0.04	n/a	n/a	n/a	n/a	n/a	n/a	73.6	n/a	n/a
WES	1.62	0.24	0.67	10.00	1.50	5.10	79.7	83.7	93.4	29.7	55.3	90.9

The actual use of leach, flotation and flotation in combination with tailings leach would be driven by project economics and thus, the cut-off grades of sulfur for samples processed in each manner may change.

13.8 HISTORIC TAILINGS REPROCESSING

Approximately 2.7 million tonnes of historic tailings, produced and deposited during the 1930s through the 1950s by the Bradley Mining Company, are located in the Meadow Creek Valley. These tailings average approximately 1.19 g/t gold, 2.92 g/t silver and 0.17% antimony; consequently, a test program was completed to assess whether processing them is economically feasibility; based on the information currently available, no testwork has been completed in the past evaluating the reprocessing of the tailings. The testwork program considered blending the tailings with early production Yellow Pine material as that timing coincides with when they would have to be processed prior to development of the waste rock storage facility that is planned for that area.

Particle size analyses of the seven composites tested show an average P_{80} of 193 µm with a range from 109 µm to 323 µm and a gold head grade ranging from 0.78 g/t to 1.51 g/t.

Table 13.19: Head Grade and Particle Size Analyses of Historic Tailings Composites

Head Grade	Comp 1 (24S)	Comp 2 (25S)	Comp 3 (26S)	HTL	HTM	HTH	S06
Au, g/t	0.98	1.12	1.51	0.78	1.17	1.31	1.44
Ag, g/t	4.00	3.00	3.80	-	-	-	< 10
As, %	0.09	0.15	0.18	0.09	0.15	0.17	0.10
Sb, %	0.14	0.16	0.22	0.07	0.23	0.17	0.87
S, %	0.43	0.36	0.29	0.18	0.37	0.28	0.61
PSA P_{80}, µm	323	142	109	139	276	116	245

Testwork completed where the historic tailings material was blended into the early production Yellow Pine material at a ratio of 15% of the total feed to the ball mill used single point calibration data to determine relative work indices and laboratory grind times. This scoping level data indicates that blending the historic tailings material with fresh ore may reduce the operating work index of the total feed to the grinding circuit by 10 - 14% (Gajo, 2014b).

13.8.1 Flotation

Two courses of testwork have been undertaken at SGS on the Historic Tailings, the first being a scoping study into the response of the tailings to the standard flotation procedure that was conducted on the oxide-transitional materials of the three primary ore deposits (McCarley, 2013). The second study applied the Yellow Pine flotation flowsheets for low antimony and high antimony material to four different composites of the historic tailings (McCarley, 2014), which culminated in a locked cycle testwork program to evaluate the effect of a 15% blend of historic tailings with fresh, Early Production Yellow Pine material on flotation response (Gajo, 2014b).

The scoping study was conducted on three single sonic core composites (Comp 1, Comp 2 and Comp 3) which were spatially separated in the repository. Rougher flotation was conducted at 100 µm and 75 µm P_{80} grind targets, at natural pH, with 200 g/t copper sulfate as the activator, 110 g/t PAX as the collector and froth collection of 29 minutes. An additional set of tests, done at 75 µm grind, included sodium sulfide (Na_2S) in a conditioning step after the grind. Gold recovery in all samples was improved with the finer 75 µm P_{80} target in the grind over the 100 µm target while sulfidizing with Na_2S improved the performance of flotation for Composites 1 and 2 but had no discernible effect on Composite 3. The best performance achieved with the 75µm tests and no sulfidizing were:

- Composite 1: Concentrate assaying 5.23 g/t Au and 2.1% S at 84.3% gold recovery;
- Composite 2: Concentrate assaying 4.26 g/t Au and 1.4% S at 78.7% gold recovery; and
- Composite 3: Concentrate assaying 2.62 g/t Au and 0.45% S concentrate at 64.2% gold recovery.

Rougher mass pulls ranged from 17 to 37%. Kinetic cleaner testing was then conducted with 10 g/t PAX. The cleaner stage gold recoveries obtained in the tests without sodium sulfide were: Composite 1 with 94.4%, Composite 2 with 77.3% and Composite 3 with 85.5%.

The second flotation program for historic tailings evaluated composites which had been chosen considering the spatial relationship of the core holes in the repository, and also by gold grade. The composites are Historic Tailings High gold, 1.31 g/t (**HTH**), Historic Tailings Average gold, 1.17 g/t (**HTM**), and Historic Tailings Low gold, 0.78 g/t (**HTL**). An additional, single-hole variability composite (composite # S06) was also tested as a high antimony sample (0.87% Sb). The purpose of the testwork was to gauge the response of the tailings to the established Yellow Pine flotation flowsheet. Later testwork in the PFS study included flotation of fresh Yellow Pine ore samples blended with the HTM composite and evaluation of the flotation response.

The composites were subjected to kinetic rougher flotation testing on either the Yellow Pine High antimony or Low antimony flowsheet, based on their antimony content. Antimony circuit antimony recovery from the high-antimony composites HTM, HTH, and S06 were 37%, 12% and 49%, respectively. Gold losses to the rougher concentrate were all below 3.2%. The overall results of the kinetic tests for each composite were:

- HTL at 6.86 g/t Au, 1.6% S and 54.5% gold recovery;
- HTM with 15.9 g/t Au, 5.42% S and 71.1% gold recovery;
- HTH at 2.41 g/t Au, 0.7% S and 36.6% gold recovery; and
- S06 at 14.1 g/t Au, 6.2% S and 75% gold recovery.

The kinetics data suggest gold flotation was near completion for the HTM and S06 composites using the standard residence time, but flotation was slow in HTL and HTH and not complete (the latter was reported to be very viscous in the cell).

Cleaner tests were then conducted on the composites to evaluate the effect of the Yellow Pine flowsheet on cleaning of the historic tailings. Composite HTH was floated at 30% solids rather than at 35% and it was noted that it was still quite thick but did not appear to be as rich in slimes. It is felt that when blended into fresh ore, this influence on viscosity will not be observed in flotation, but further testwork is recommended to confirm such. A summary of the cleaner flotation test results are given in Table 13.20.

Table 13.20: Historic Tailings Cleaner Test Results with Yellow Pine Flowsheet

Parameter	HTL	HTM	HTH	S06	Parameter	HTL	HTM	HTH	S06
Feed P_{80}	81	71	61	68	-	-	-	-	-
Antimony 2nd Cleaner Concentrate					*Gold Final Concentrate*				
Mass Pull (%)	-	0.13	0.08	0.17	Mass Pull (%)	3.89	2.81	2.83	2.73
Grade					Grade				
Au, g/t	-	7.62	6.01	3.30	Au, g/t	11.5	25.7	12.4	32.9
As, %	-	0.35	0.60	0.18	As, %	1.21	2.98	1.70	3.07
Sb, %	-	50.4	6.79	60.2	Sb, %	-	1.44	1.48	10.4
S, %	-	20.7	3.67	24.8	S, %	2.85	8.94	4.47	14.8
Recovery					Recovery				
Au, %	-	0.94	0.53	0.45	Au, %	55.4	68.7	36.6	70.8
As, %	-	0.30	0.30	0.24	As, %	51.8	55.3	28.0	64.1
Sb, %	-	27.5	3.44	13.3	Sb, %	-	17.0	24.9	35.9
S, %	-	7.16	1.16	6.96	S, %	61.1	67.0	47.3	64.9

Both HTM and S06 yielded antimony concentrates that were at or above the 50% grade target for antimony content; all three antimony final concentrates held the gold losses to less than 1% but antimony recovery was poor in all three tests. In the case of S06 this resulted in a high antimony grade of nearly 10.5% in the final gold concentrate, while in the other tests the antimony remained low, below 1.5%. Gold concentrates from both HTM and S06 were above 5% sulfur in rougher flotation. Since the historic tailings are proposed to be blended into the Yellow Pine mill feed at a 15% ratio, this is not expected to have a significant impact on the overall concentrate sulfur grade produced from the blend.

13.8.2 Leaching Studies

All seven historical tailings composites were leached by a standard cyanidation procedure to determine the overall recovery of gold from whole samples. In the scoping program on the first three composites, one set of tests was completed at a grind P_{80} target of 100 µm, in line with the oxide-transition testwork. A second set of tests was completed at the same grind target and using oxygen addition to the leaches. A third set of tests was then completed at the new PFS grind P_{80} target of 75 µm with oxygen. For the four developmental program composites, a single set of leaches was conducted at the P_{80} target of 75 µm with oxygen.

The results of the leaches on the first three composites in the scoping program mirrored the flotation results in that Composite 3 was now best performing (poorest flotation), followed by Composite 2 and then finally Composite 1 (best flotation). Neither the use of oxygen nor the finer grind appeared to benefit the extraction of gold from any of the samples. Leach kinetics showed that most of the gold was leached within the first 24 hours of the test.

In the developmental study leaches, the extractions of gold from the feed also mirror the flotation results; where HTL floated 55 - 60% of the gold, approximately 36% leached, indicating that much of what did not float is associated with non-sulfide gangue minerals. There is a similar pattern for HTM, with 73 - 77% floating and 28% leaching, in HTH, with 58 - 69% floating and 49% leaching and in S06, with 79 - 83% floating and 26% leaching.

Select tailings from the flotation tests were also subjected to the standard cyanidation leaches, with the results given in Table 13.21. For tests HTM and S06 the tailings leach was conducted on rougher tailings alone, while for HTL and HTH the leaches were conducted on a combination of rougher and cleaner tailings.

Table 13.21: Flotation Tailings Leach Results on Historic Tailings Composites

Comp ID	P_{80} (µm)	Average Dissolved O_2 (mg/L)	Reagent Consumed		Gold Head Grade		Gold Residue (g/t)	Gold Extraction	
			NaCN (kg/t)	Lime (kg/t)	Calc (g/t)	Direct (g/t)		(%)	(g/t)
Comp 1	92.0	33.7	0.35	0.83	0.25	0.25	0.19	26.0	0.06
Comp 2	81.0	28.6	0.31	0.86	0.38	0.29	0.24	37.2	0.05
Comp 3	77.0	31.5	0.51	1.89	1.09	1.07	0.47	57.0	0.60
HTL	81.0	26.6	0.20	0.61	0.34	0.37(1)	0.22	35.6	0.15
HTM	71.0	23.9	0.16	0.54	0.28	0.28	0.23	16.5	0.05
HTH	61.0	13.7	0.77	0.86	0.59	0.58(1)	0.41	31.7	0.17
S06	68.0	25.8	0.23	0.64	0.27	0.27	0.21	22.4	0.06
Note: (1) Estimated from Rougher + Cleaner tailings assays									

When combined with the amount of gold recovered through flotation, the overall gold recovery of the historic tailings becomes 84.3%, 87.6% and 73.7% for Composites 1, 2 and 3, respectively. It becomes 71.3%, 75%, 56.7% and 80.6% for Composites HTL, HTM, HTH and S06, respectively. Including gold recovered to the antimony circuit, HTM, HTH and S06 overall gold recovery becomes 78.2%, 60.8% and 81% respectively.

Testwork was conducted on blends of Yellow Pine fresh material and Historic Tailings, with the aim of exploring if the two feed types could be effectively co-mingled for processing. This testwork included a brief batch test program on the EPH and EPL composites, each time blended with 15% of average gold grade (HTM2) Historic Tailings material (Gajo, 2014b). This culminated in a locked cycle test on the blended EPH/Historic Tailings composite. As the EPL/Historic Tailings blend reached target sulfur grade by roughing alone, this was not tested in locked cycle mode, rather by a 4 kg batch test. 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 very high and sulfur grades remaining above the threshold for POX.

Table 13.22: Flotation of Blended Yellow Pine Early Production Feed and Historic 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%) & Historic 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%) & Historic 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 tailings were leached using the standard Yellow Pine leach procedure, with 8% of the gold in the EPH Blend flotation tailings and 20% of the gold in the EPL Blend tailings being leached – the latter being slightly better than might have been expected from a pure weighted average of the individual metallurgy of the two components.

Overall, the above evidence suggests that the Yellow Pine and Historic Tailings materials can be successfully co-processed.

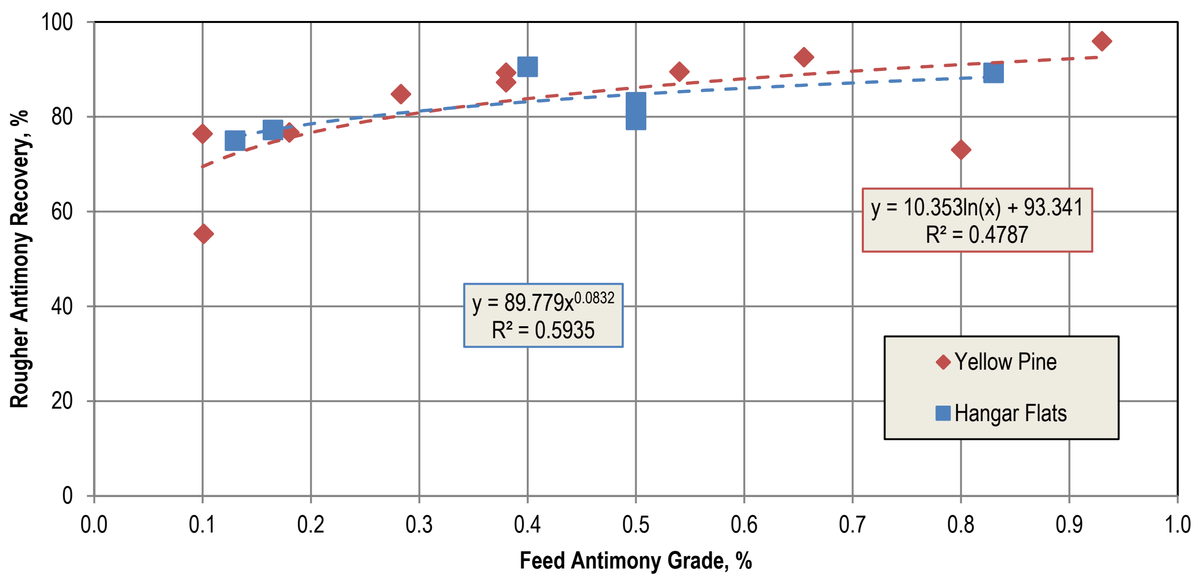
13.9 METALLURGICAL PREDICTION

13.9.1 Antimony Flotation

The PFS metallurgical testing program employed rougher flotation tests to describe the variability in antimony recovery, with the use of a small number of locked cycle tests to describe how well the antimony floated to the rougher concentrate and was upgraded by closed-circuit (locked cycle) cleaner flotation to saleable grade.

The rougher flotation results have been tabulated in previous sections as Table 13.5 and Table 13.11. Antimony recovery is linked to head grade for both Yellow Pine and Hangar Flats as shown on Figure 13.5, and equations linking head grade to a prediction of recovery are shown for both Yellow Pine and Hangar Flats.

Figure 13.5: Antimony Rougher Metallurgical Prediction



Three antimony flotation locked cycle tests were completed, the results from which have been described earlier in this section and are summarized below. Cleaner circuit performance is relatively consistent between the three tests yielding a concentrate assaying 59% Sb at roughly 96% cleaner stage recovery. Accordingly, antimony cleaner recoveries of 98.1% (the mean of the two tests) for Yellow Pine and 96.7% for Hangar Flats has been applied to the Sb rougher recoveries from the above equations to link antimony feed grade to recovery to final concentrate. A constant concentrate grade of 59% Sb has been assumed.

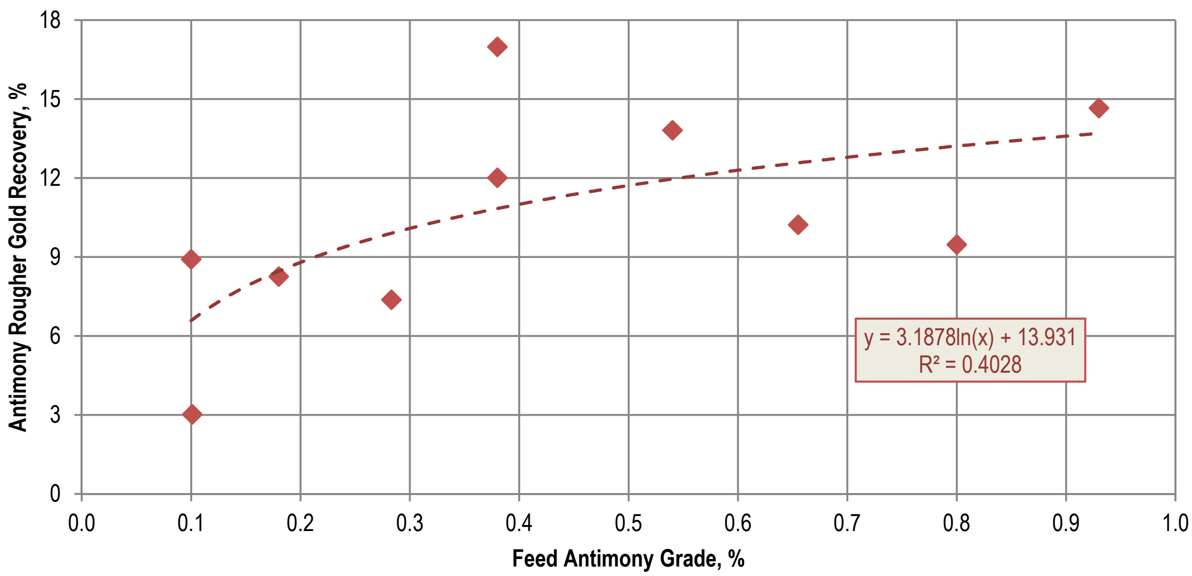
13.9.2 Flotation and Direct Leaching of Gold

13.9.2.1 Yellow Pine

The recovery of gold to the combined Yellow Pine rougher concentrates has been characterized for both the antimony-rich and antimony-poor samples using the data from sixteen variability tests conducted on early production samples, samples from the 2014 mini-variability program and on the major PFS composites (see Table 13.1).

While the use of the sequential antimony-gold flotation circuit has no adverse effect on overall gold recovery, the gold recovered to the gold concentrate itself is lower due to misplacement to the antimony final concentrate. In Yellow Pine, this is linked to the antimony head grade.

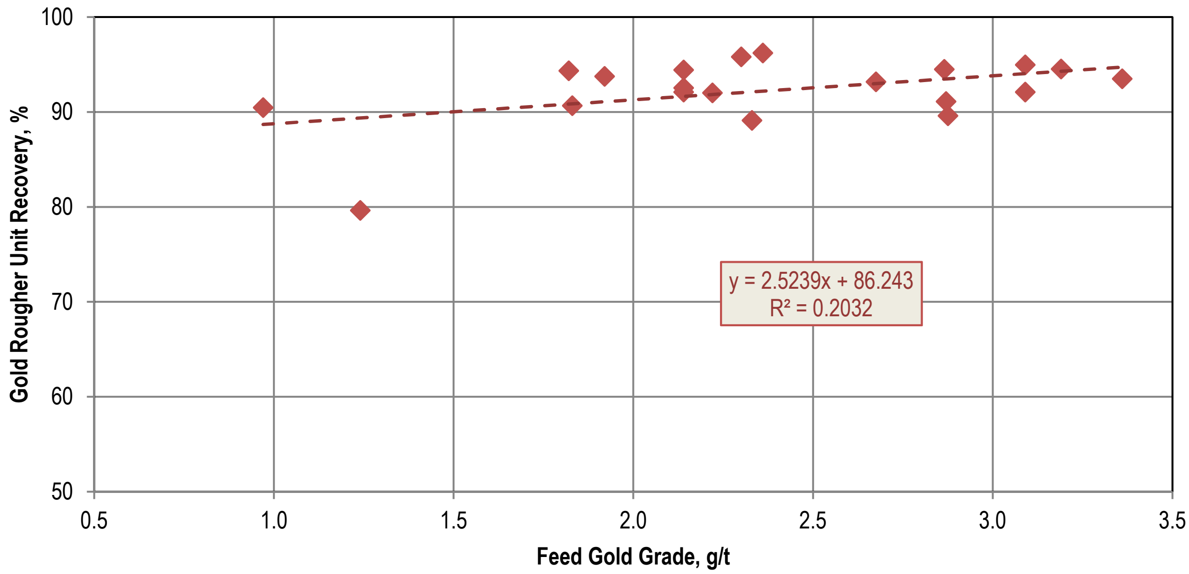
Figure 13.6: Yellow Pine Antimony Head Grade *vs* Antimony Rougher Gold Loss



In the final prediction equation, this value must be multiplied by a factor of 0.237 to account for the rejection of some of the rougher gold in cleaning that was observed in locked cycle flotation.

Unlike in the PEA, the PFS-optimized flowsheets demonstrated no systematic differential in overall gold recovery with the use of the two flowsheets, so all the rougher flotation data has been gathered into a single dataset. There is evidence, albeit weak, of a gold head grade/rougher recovery relationship so a simple linear regression has been used in preference to using a fixed recovery.

Figure 13.7: Yellow Pine Gold Head Grade *vs* Rougher Unit Recovery



The regression chart and equation shown on Figure 13.7 provide the flotation recovery of gold in samples with a feed grade of less than 4 g/t gold as a function of head grade.

Sulfur recovery appears fixed and independent of head grade, averaging 94.7%, and in the low antimony samples the concentrate sulfur grade also appears to be independent (Figure 13.8). In the high antimony samples there is a trend favoring higher concentrate grades as a consequence of higher head grades, and this has been used to predict the concentrate sulfur grade to the concentrate in those samples (Figure 13.9).

Figure 13.8: Yellow Pine Sulfur Head Grade *vs* Rougher Concentrate Recovery and Grade

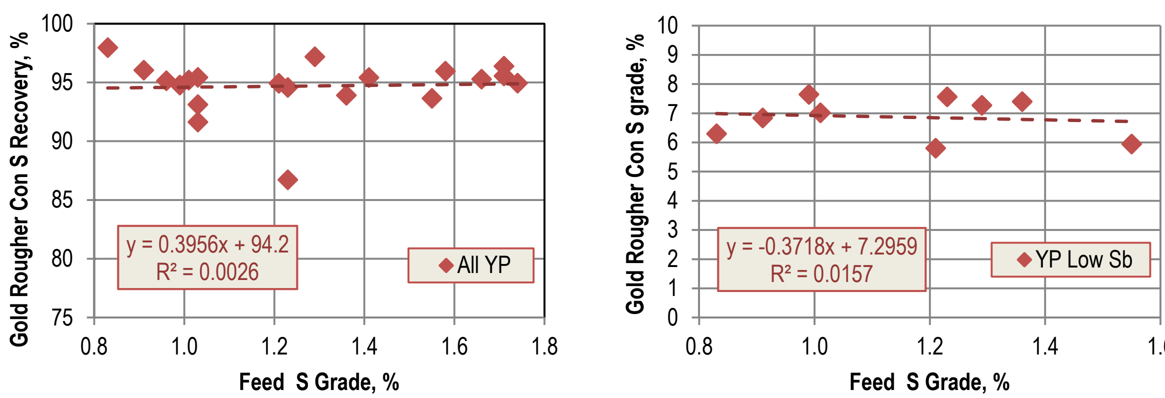
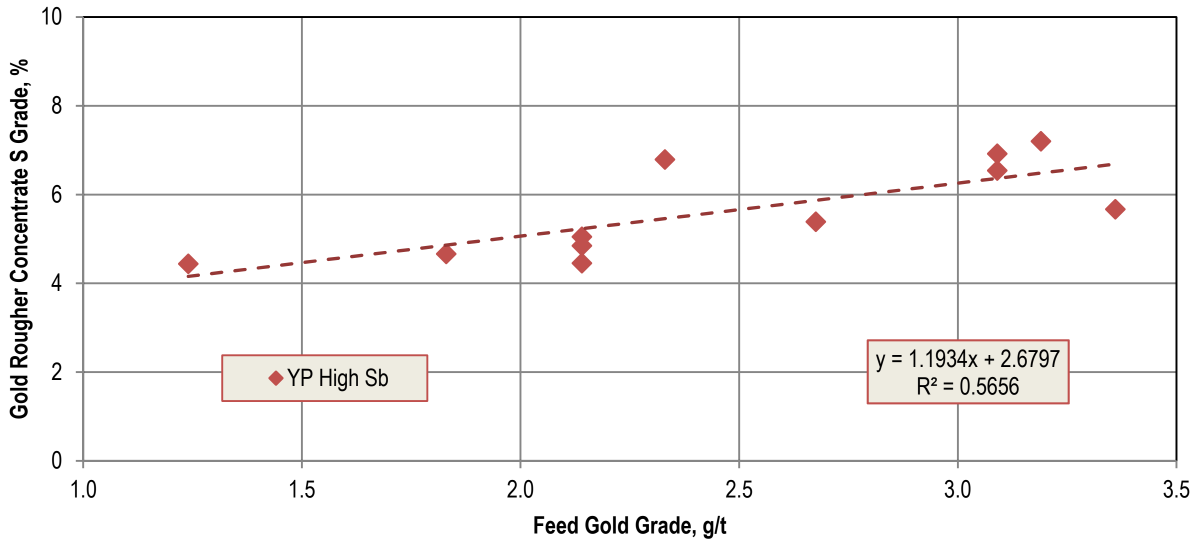


Figure 13.9: Yellow Pine High Antimony Sample Gold Head Grade *vs* Rougher Concentrate Sulfur Grade



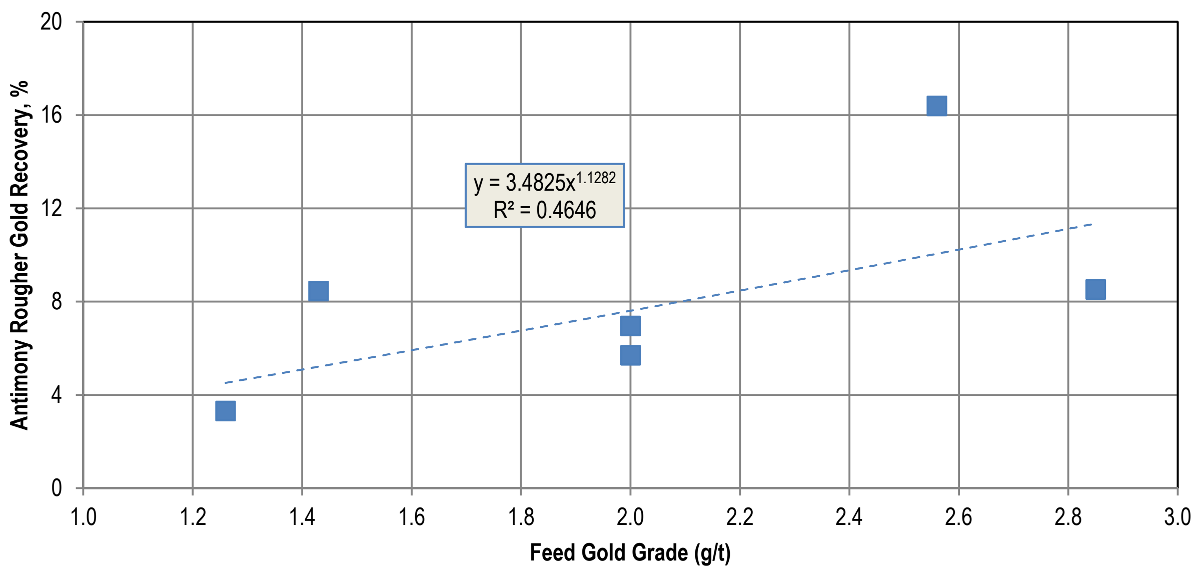
Based on the limited data available on the carbonate assay in the flotation concentrates, a factor of 0.63 x head carbonate grade has been used to predict the carbonate grade in the Yellow Pine gold concentrates.

Leaching the tailings yielded generally very poor recoveries of additional gold. A constant leach extraction reflecting the average of the dataset (10% of the gold in the tailings) has been assumed for metallurgical forecasting purposes.

13.9.2.2 Hangar Flats

In Hangar Flats, gold misplacement to the antimony rougher concentrate is best predicted using the feed gold grade.

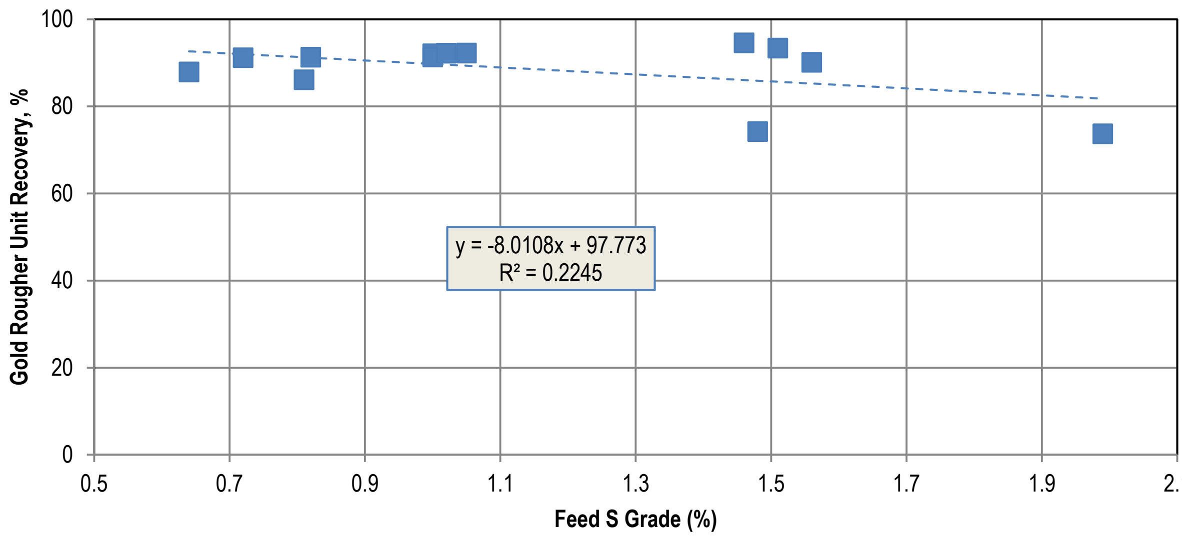
Figure 13.10: Hangar Flats Gold Head Grade *vs* Antimony Rougher Gold Loss



In the final prediction equation, this value must be multiplied by a factor of 0.239 to account for the rejection of some of the rougher gold in cleaning that was observed in locked cycle flotation.

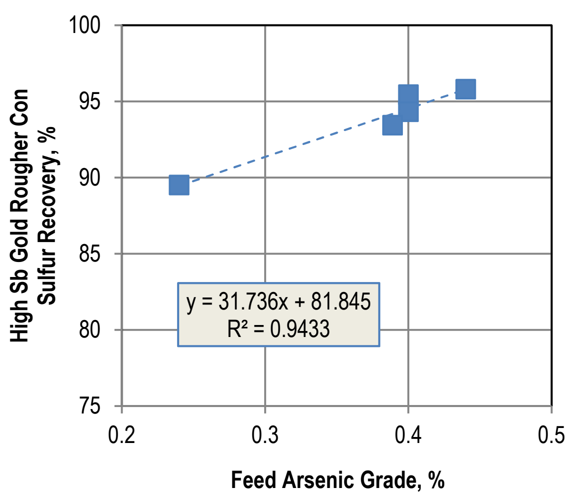
As with Yellow Pine, gold recovery from sulfide Hangar Flats samples appeared to be unrelated to the use of the bulk flotation or Sb-Au sequential flowsheet. A sulfur head grade/gold rougher recovery relationship has been used to predict gold recovery in preference to using a fixed recovery (for all low and high antimony samples). Although this negative correlation is somewhat counterintuitive, the limited data available meant that the two poor acting samples could not be ignored and the relationship has accordingly been adopted.

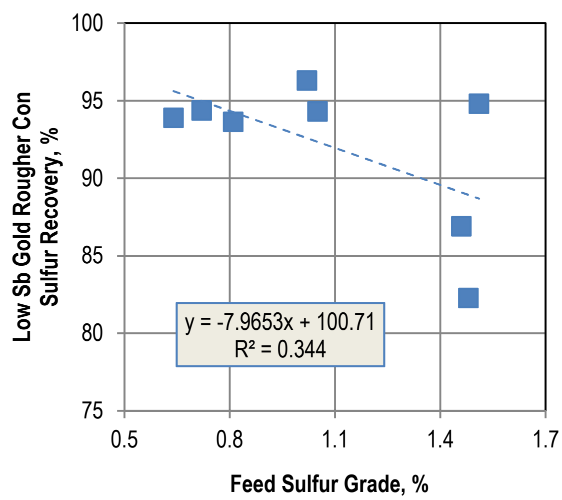
Figure 13.11: Hangar Flats Feed Sulfur Head Grade *vs* Gold Rougher Flotation Gold Recovery



Lower recoveries can be expected with gouge samples influenced by the Meadow Creek Fault, as evidenced by the two low performers in the chart, which are expected to make up approximately 15% of the Hangar Flats Mineral Resource. It may be beneficial in the future to segregate out the projections for gouge-influenced samples.

Figure 13.12: Hangar Flats Gold Rougher Concentrate Sulfur Recovery Prediction

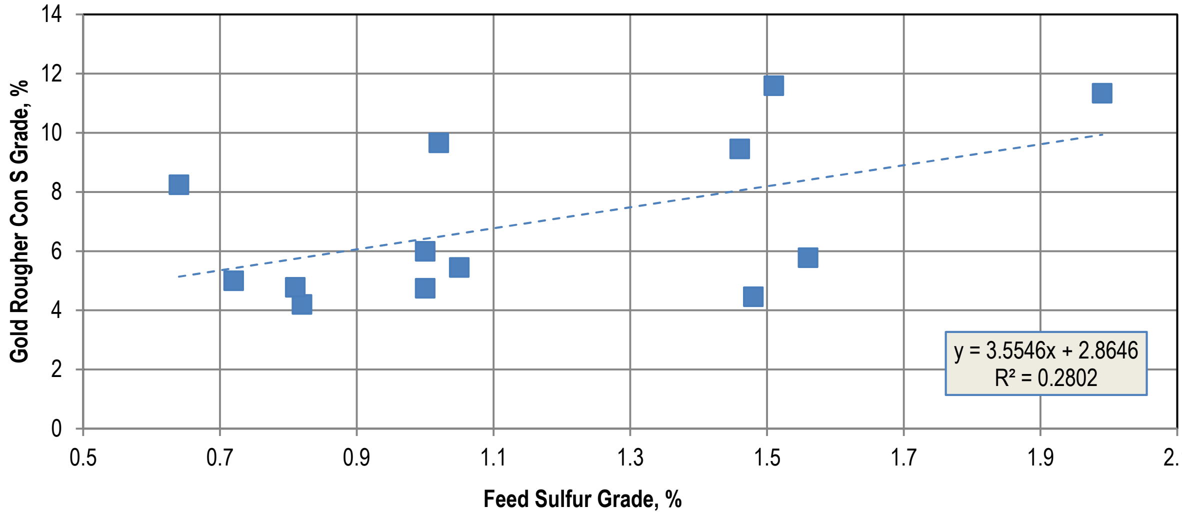




Sulfur recovery to the gold concentrate in High Sb samples appears strongly tied to the feed arsenic grade while for the Low Sb samples it is more closely tied to feed sulfur grade (Figure 13.12). The influence of the gouge material is again seen with lower recovery in higher sulfur samples.

Sulfur grade in the rougher concentrates of the full suite of samples is best modeled with the feed sulfur grade as shown on Figure 13.13.

Figure 13.13: Hangar Flats Sulfur Head Grade *vs* Rougher Sulfur Grade

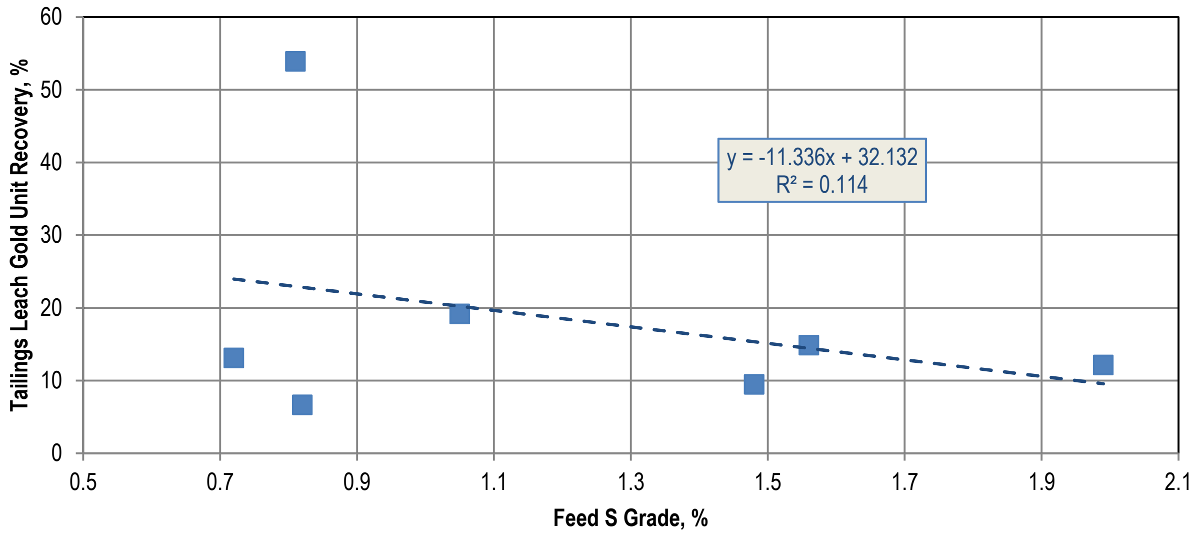


Based on the limited data available on the carbonate assay in the flotation concentrates, a factor of 0.74 x head carbonate grade has been used to predict the carbonate grade in the Hangar Flats gold rougher concentrates.

In the final years of operation, the planned mill feed is dominated by West End material with a minor component of material from Hangar Flats. The processing of the carbonate-rich West End material would prompt the need to clean the rougher concentrates to achieve the target 0.9:1 ratio of carbonate to sulfur. There would also be occasional times where the Hangar Flats concentrate sulfur grade would not meet the 5% sulfur requirement for POX (gouge material and some transitional ores). The effect of cleaning on overall grades and recoveries has been established by comparing the recoveries by locked cycle testing of the HFH and HFL with commensurate rougher testing. The HFH and HFL locked cycle concentrates assayed 14.5% and 14.9% sulfur respectively, at cleaner stage gold recoveries of 95.3% and 96.2%. For the sake of mass balancing these have been assumed to be constant at the average (95.8%) gold recovery and (14.7%) sulfur grade through the years when cleaner flotation would be operating. Sulfur recovery to the cleaner concentrate is also assumed fixed at 96.1%. Based on limited data, carbonate grade does not appear to change with cleaning and should be assumed to remain the same as measured in the rougher concentrate.

The leach response from the Hangar Flats flotation tailings has not been well-established in the test program – tailings from some poor-floating Hangar Flats samples leached well (the transition samples) and the HFH and HFL composites both yielded economic leach recoveries from their tailings while some other poor-floating samples leached poorly. A loose relationship exists however, between feed sulfur grade and the recovery of gold from the tailings.

Figure 13.14: Hangar Flats Feed Sulfur Grade *vs* Flotation Tailings Gold Leach Unit Recovery



13.9.2.3 West End

Two flowsheets would be employed to treat West End ores: cleaner flotation plus tailings leaching, and direct whole ore cyanidation (i.e. bypassing the flotation circuit). Process selection would be driven by the differential in recovery from the two options traded against the processing cost differential.

Gold flotation recoveries are highest when the gold is in solid solution in the host sulfides, and at their lowest when the sulfides have been totally destroyed by oxidation. Owing to its grain size, this free gold in oxide samples does not float well. Accordingly, the PEA variability sample flotation recovery shows an inverse correlation with the geochemical leach recovery.

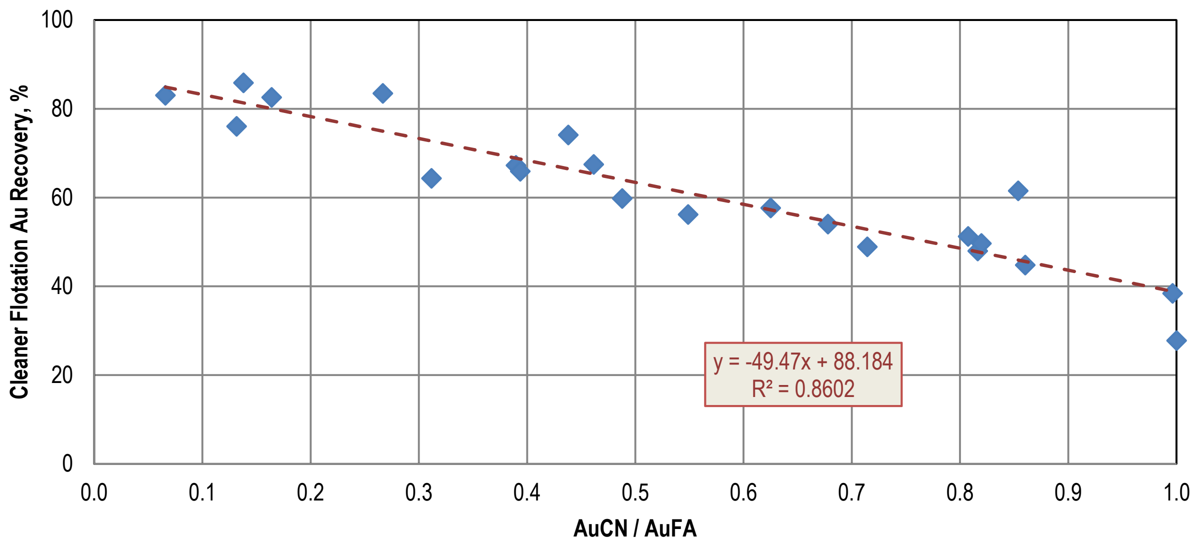
The most recent PFS variability testwork was conducted on the optimized flowsheets at a primary grind P_{80} of 75 µm, and as compared to the PEA primary grind of P_{80} at 100 µm, showed improved flotation and tailings leach recoveries which needed to be accounted for in the current PFS metallurgical projections. PFS cleaning of the rougher concentrates at this finer grind was able to produce a POX ready flotation concentrate at slightly improved recovery as compared to the PEA rougher concentrate recovery. The mean difference in recoveries between the PEA rougher and PFS cleaner flowsheets for flotation is shown below:

Table 13.23: Comparison of PEA and PFS Flowsheet Flotation Recoveries to POX Feed

Sample	PEA (%)	PFS (%)	Difference (%)
WE41	63.7	62.8	-0.9
WE44	55.6	56.7	1.1
WE47	82.9	84.5	1.6
Average	**67.4**	**68.0**	**0.6**

It must be stressed that the above 0.6% improvement in recovery is directly related to cleaned POX-ready concentrate, and that no account therefore needs to be made for cleaner losses in using this number to convert from the AuCN ratio-based equations to POX ready concentrate. This negates the PEA projection requirement for a 0.975 cleaning performance factor and is incorporated into the equation as shown on Figure 13.15.

Figure 13.15: West End Flotation Gold Recovery *vs* Whole Ore Geochemical Leach Gold Extraction



The sulfur grades of the cleaner concentrates from three batch tests and one locked cycle test on WE41, WE44, WE47 and WES respectively were 12.5%, 12.3%, 13.1% and 14.4%, despite highly variable sulfide sulfur head grades of 0.91%, 0.41%, 0.26% and 0.68% respectively. Accordingly, it has been assumed that the sulfur grade in the concentrate would be a constant, and the average of the tests (13.0%) has been used. The mass pull has been calculated by balancing the sulfur grades in the feed and concentrate, together with the sulfur recovery – which is assumed to be 93% based on the locked cycle test (the mean batch sulfur recoveries from the batch tests averaged 94%).

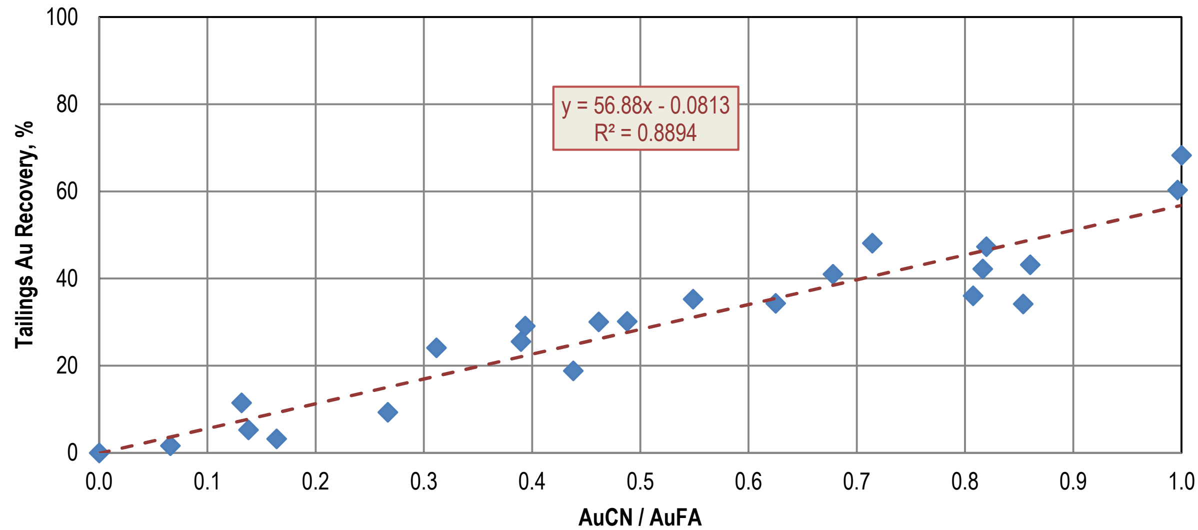
Tailings from the float are for the most part leachable, and correlate well with the geochemical gold cyanide (**AuCN**) / gold fire (**AuFA**) assays. Due to the finer primary grind size in the PFS flowsheet, now at 75 µm instead of 100 µm, the leachability of the gold in tailings has improved. The mean difference in recoveries between the PEA and PFS flowsheets for tailings leaching is shown in Table 13.24.

Table 13.24: Comparison of PEA and PFS Flowsheet Flotation Tailings Leach Gold Extraction

Sample	PEA (%)	PFS (%)	Difference (%)
WE41	22.5	25.6	3.1
WE44	33.6	33.5	-0.1
WE47	7.6	9.6	2.0
Average	**21.2**	**22.9**	**1.7**

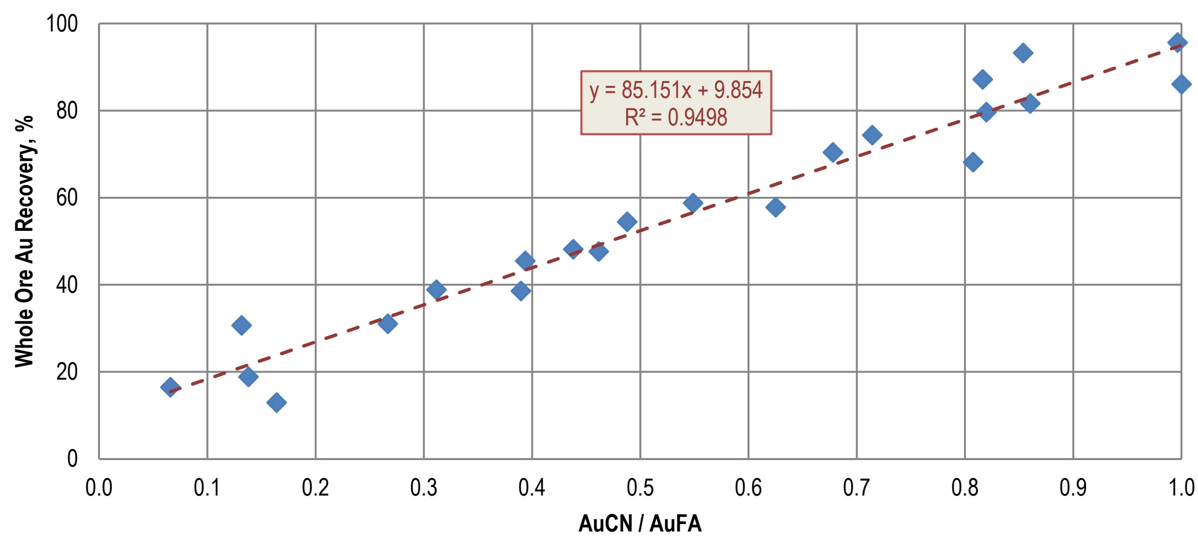
This data analyzed for the prediction equation includes the point (0, 0); it is felt that it is a legitimate point since zero extraction from the AuCN tests would indicate zero extraction from a laboratory leach of tailings. Also, as noted above, moving from the 100 µm PEA flowsheet to the 75 µm PFS flowsheet improves the leach recovery by an average of 1.7%, which is also incorporated into the analysis. The respective chart and equation developed is shown on Figure 13.16. It has been assumed for the sake of metallurgical projections that the gold lost in cleaning will subsequently leach to the same recovery as the gold in the rougher tailings.

Figure 13.16: West End Flotation Tailings Leach *vs* Whole Ore Geochemical Leach Gold Extraction



The extraction of gold from whole ore leaching is best defined through the modeled cyanide gold assay (AuCN) diagnostic geochemical data, adjusted to fit expected actual commercial leaching performance. The AuCN assays are compared with AuFA assays to evaluate the maximum percentage of gold that is leachable. This short, intensive, cold cyanide leach on material of a nominal 100 µm grind (but in reality quite pulverized), tended to yield recoveries somewhat higher than metallurgical laboratory bottle roll data used in the regression study (Table 13.18). The equation on Figure 13.17 was developed to arrive at a link between the geochemical and metallurgical laboratory recoveries.

Figure 13.17: West End Metallurgical Laboratory Whole Ore Leach *vs* Geochemical Leach Gold Extraction

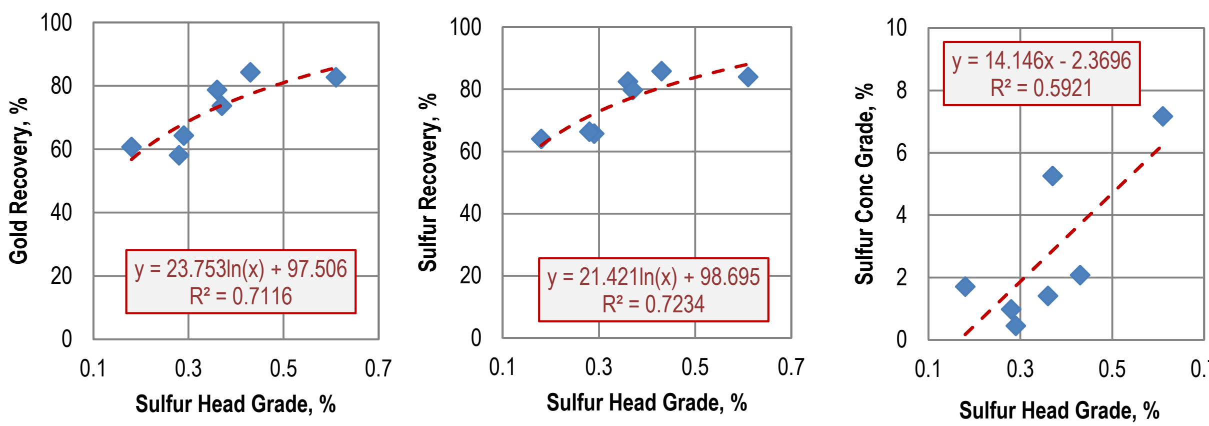


It is assumed, for the sake of the PFS projections, that leach recovery of gold in whole ore will improve by the same average (1.7%) as in the flotation tailings with the finer grind and this has been incorporated into the above analysis.

13.9.2.4 Historic Tailings Reprocessing

As the mine plan does not attempt to characterize the Historic Tailings material to be mined on a year-by-year basis, the metallurgical forecast for processing the Historic Tailings would also include a single performance parameter for the duration of the mine life. Based on tests on seven Historic Tailings samples, both gold and sulfur recovery are closely related to the sulfur head grade. Similarly the concentrate sulfur grade is linked to head grade. Using these correlations and a mine plan head grade of 0.33% sulfur, forecasts of gold recovery (71.2%), sulfur recovery (74.9%) and sulfur concentrate grade (2.3%) have been made.

Figure 13.18: Historic Tailings Sulfur Head Grade *vs* Process Parameters



Testwork on a blend containing 15% Historic Tailings and 85% Yellow Pine Early Production material showed there to be no adverse effects from blending these materials, such that the Project metallurgy of the blend would reflect the weighted mean metallurgical response of the two components.

13.9.3 Silver

The silver metallurgical forecasting equations have been summarized in Table 13.25.

Table 13.25: Silver Recovery Predictions

Deposit	Antimony Concentrate	Gold Concentrate	POX-CIL(2)	Tailings Leach	Whole Ore Leach
Yellow Pine Hi Sb	43%	30%	4%	10%	n/a
Yellow Pine Low Sb	n/a	73%	4%	10%	n/a
Hangar Flats High Sb	50%	30%	5%	6%	n/a
Hangar Flats Low Sb	n/a	80%	5%	6%	n/a
West End Sulfide	n/a	(1)	4%	59%	n/a
West End Mixed	n/a	(1)	4%	59%	n/a
West End Oxide	n/a	n/a	n/a	n/a	52%
Historic Tailings	n/a	73%	4%	10%	n/a

Note:
(1) *West End Ag recovery: 0.912 x gold recovery - 1.05*
(2) *Percent of Ag reporting to POX that is recovered.*

13.9.4 POX Leach Loss

There has been limited testwork completed defining the gold loss in oxidation and leaching of the flotation concentrates produced. It is recommended to apply a factor of 0.978 for Yellow Pine, 0.970 for Hangar Flats and 0.982 for West End to account for the gold remaining in the oxidized solids after leach.

13.9.5 Soluble Gold Loss

All barren solutions from the carbon elution and electrowinning circuits are considered internal recycling streams and should not contribute to gold losses. However, slurry streams which have passed through cyanide detoxification and are sent to the tailings facility will contain small amounts of soluble gold in solution unrecovered in the potential oxidized concentrate CIP and whole ore/flotation tailings carbon-in-leach (**CIL**) adsorption circuits and represent the only source of solution gold loss.

This soluble gold loss should be calculated from the steady state tailings pond solution volume plus pore solution in the settled solids and should not include reclaimed process water from the tailings facility which is also recycled. The calculation is performed with data sourced over a set time period and gives the cumulative mass of soluble gold loss, which as a fraction of the cumulative mass of gold fed to the leach gives the percent soluble gold loss over the set time period.

Limited test data indicates that the CIP adsorption efficiency averages approximately 94.9% and CIL adsorption efficiencies are greater than 98.5%. Further testwork is planned on the carbon and detoxification circuits in the feasibility testwork program. For the purposes of projecting PFS level metallurgical recoveries, losses due to carbon adsorption inefficiency in this program are only estimates. Soluble gold losses estimated against industry benchmarks are projected for this case at less than 1%, and possibly down to 0.5 - 0.2%, by mass, relative to gold in POX residues and oxide solids fed to leach (Ford, 2014).

For projection purposes, it is recommended to use a 0.8% loss in the calculations for all deposits (a factor of 0.992).

13.9.6 Metallurgical Summary

For convenience, all metallurgical recovery predictions for gold, silver, and antimony that have been developed in the preceding Sections, and that will form the basis of the PFS metallurgical recovery predictions, are provided in Table 13.26. The equations include all losses such that payable metal factors can be applied to these metallurgical recoveries. The only exceptions are the sulfur metallurgical regressions as these do not affect actual antimony, gold and silver metallurgy – just the mass balance of the circuit itself.

Table 13.26: PFS Metallurgical Recovery Prediction Equations Through POX and Leach

Deposit	Ore Type	Circuit	Metal	Metallurgical Recovery Equation (%)
West End	Oxide	Oxide	Au	Leach extraction = (85.151 x AuCN / AuFA + 9.8540) x 0.992
			Ag	Leach extraction = (Feed Ag Grade x 0.52) x 0.992
	Sulfide/ Mixed	Gold	Au	Cleaner flotation recovery = (-49.47 x AuCN / AuFA + 88.184) x 0.982 x 0.992
				Cleaner tailings leach extraction = (56.88 x AuCN / AuFA - 0.0813) x 0.992
			Ag	Cleaner flotation recovery = (0.912 x Cleaner Au Recovery - 1.05) x 0.04 x 0.992
				Cleaner tailings leach extraction = (100 – Cleaner Flotation Recovery) x 0.59 x 0.992
Yellow Pine	High Antimony Sulfide	Antimony	Sb	Antimony Cleaner Flotation Recovery = (10.353 x ln(Feed Sb Grade) + 93.341) x 0.981
			Ag	Antimony Cleaner Flotation Recovery = 43%
			Au	Antimony Cleaner Flotation Gold Loss = (3.1878 x ln(Feed Sb Grade) + 13.931) x 0.237
		Gold	Au	Rougher Flotation Gold Recovery = ((2.5239 x Feed Au Grade + 86.243) / 100 x (100 - Sb Cleaner Flotation Au Loss)) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.10) x 0.992
			Ag	Rougher Flotation Silver Recovery = (100- Antimony Cleaner Ag) x 0.30 x 0.04 x 0.992
				Flotation Tailings Leach Extraction = [100 – Antimony Cleaner Ag – ([100 – Antimony Cleaner Ag] x 0.30)] x 0.10 x 0.992
	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (2.5239 x Feed Au Grade + 86.243) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.10) x 0.992
			Ag	Rougher Flotation Silver Recovery = 2.9% x 0.992
				Flotation Tailings Leach Extraction = 2.7% x 0.992
Hangar Flats	High Antimony Sulfide	Antimony	Sb	Antimony Cleaner Flotation Recovery = (89.779 x (Feed Sb Grade ^ 0.0832)) x 0.967
			Ag	Antimony Cleaner Flotation Recovery = 50%
			Au	Antimony Cleaner Flotation Gold Loss = (3.4825 x (Feed Au Grade) ^ 1.1282) x 0.239
		Gold	Au	Rougher Flotation Gold Recovery = ((-8.0108 x Feed S Grade + 97.773) / 100 x (100 - Sb Cleaner Flotation Au Loss)) x 0.970 x 0.992
				Flotation Tailings Leach Extraction = ((-11.336 x Feed S Grade + 32.132)/100 x Au in Tailings) x 0.992
			Ag	Rougher Flotation Silver Recovery = (100 – Antimony Cleaner Ag) x 0.30 x 0.05 x 0.992
				Flotation Tailings Leach Extraction = [100 – Antimony Cleaner Ag – ([100 – Antimony Cleaner Ag] x 0.30)] x 0.06 x 0.992
	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (-8.0108 x Feed S Grade + 97.773) x 0.970 x 0.992
				Flotation Tailings Leach Extraction = ((-11.336 x Feed S Grade + 32.132) x Au in Tailings) x 0.992
			Ag	Rougher Flotation Silver Recovery = 80% x 0.08 x 0.992
				Flotation Tailings Leach Extraction = 4% x 0.992
Historic Tailings	Low Antimony Sulfide	Gold	Au	Rougher Flotation Gold Recovery = (23.753 x ln(Feed S Grade) + 97.506) x 0.978 x 0.992
				Flotation Tailings Leach Extraction = (Au in Tailings x 0.20) x 0.992
			Ag	Rougher Flotation Silver Recovery = 2.9% x 0.992
				Flotation Tailings Leach Extraction = 2.7% x 0.992

<u>Notes:</u> *AuCN / AuFA ratio is expressed as a decimal not as a percent; Feed Au Grade and Feed Ag Grade are expressed in g/t; Antimony Cleaner Ag, Cleaner Au Recovery, Feed Sb Grade, Feed S Grade, Au in Tailings and Sb Cleaner Flotation Au Loss are expressed in %.*

13.10 METALLURGICAL OPPORTUNITIES

13.10.1 Tungsten Recovery

While a tungsten resource remains undefined, portions of the Yellow Pine, Hangar Flats and West End deposits contain potentially recoverable tungsten in the form of scheelite (Blake, 2012). Historically, from 1940 through 1945, tungsten was recovered into two concentrates from Yellow Pine ore by flotation (high-grade concentrate) and tabling (low-grade concentrate) after the completion of sulfide flotation. High-grade materials were placer-mined which included ultra-violet (**UV**) hand sorting of trommel oversize and jigging of undersize. Cleaning of the various jig middlings included magnetic separation and tabling. A tungsten flotation recovery circuit was also started up to treat the sulfide flotation tailings (Mitchell, 2000).

A scoping test program investigating the recovery of tungsten was conducted on two small variability composites, a high-grade sample from Hangar Flats assaying 1.6% tungsten, and a lower grade sample from Yellow Pine assaying 0.42% tungsten. The limited program included a gravity release study, centrifugal gravity separation, tabling and flotation scoping testwork, in each case of gold-bearing sulfide flotation tailings (Gajo, 2014b).

Sulfide flotation of the Hangar Flats and Yellow Pine samples left 76.3% and 65.5% of the tungsten in the tailings. In both cases close to 80% of this tungsten was in the minus 53 µm fraction. Due to its fine size, gravity recovery work was not successful. The most effective rougher recovery method in both cases was scheelite flotation, with 95% and 76% recovery from the Hangar Flats and Yellow Pine samples. Mass pull rates were high making this no more than a pre-concentration step, and the limited testing of further concentration (cleaning) by gravity on flotation concentrates was not successful.

13.10.2 Finer Primary Grind

It was noted in the Master Composites summary sections that a finer grind was found to be beneficial to gold recovery for the West End Sulfide composite, which according to the PFS potential mining schedule represents approximately 29% of the mill feed to the plant over the life of the mine. For the WES composite, it was noted that the reduction in grind size from a P_{80} of 84 µm to 56 µm improved gold recovery in rougher flotation by about 4%. A comparison between the extra power and/or equipment required to achieve this finer grind and the increased amount of gold recovered would need to be studied (Gajo, 2014a).

It should be noted that a finer grind size did not appear to be any benefit to the Yellow Pine low antimony ores nor the Hangar Flats low antimony ores, although the data from the Hangar Flats low antimony tests is weak and further testwork is recommended. The effect of finer grind on high antimony ores was also not tested and should be evaluated to gauge the effect of a finer grind on possible sliming of the antimony flotation circuit.

13.10.3 Elimination of POX Countercurrent Decantation Circuit

Cyanide leaching Yellow Pine POX slurries was completed at SGS Lakefield and included a study into the effect of degree of slurry washing on cyanidation and gold recovery (Jackman, 2014a). Samples from three of the pressure oxidation runs underwent five cyanidation leaches to determine the extraction of gold. There were no regrinding tests performed. Three of the tests were straight bottle roll cyanidation leaches after 100% of hot cure solution was washed out of the solids, while two of them evaluated the effect of the degree of hot cure slurry washing on leach recovery.

Tests evaluating the effect of degree of slurry washing on gold extraction and reagent consumptions showed that there was less than 1% decrease in gold extraction when moving from fully washed slurry through to unwashed (Table 13.27). However, throughout that change, the cyanide consumption increased 7-fold and lime consumption

29-fold. The balance of capital and operating costs for washing thickeners versus reagent consumption with less washing is recommended to be studied further in the future.

Table 13.27: Summary of Slurry Washing Effects on Cyanidation

POX Discharge Slurry Washing Conditions	Reagent Consumption		Extraction
	NaCN (kg/t)	CaO (kg/t)	Au (%)
100% Water	0.34	3.9	97.9
34% POX PLS / 66% DI Water	1.29	59.4	97.8
100% POX PLS	2.45	114.0	97.2

13.11 ALTERNATIVE PROCESSES

13.11.1 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 separate party. The two options evaluated 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.

13.11.1.1 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. Results of 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.

13.11.1.2 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 three 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 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 appears 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.

13.11.1.3 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 should 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.

13.11.2 Gold Concentrate Processing

The refractory nature of the gold-bearing minerals in Midas Gold ores necessitates the oxidation of the sulfides in order to make the gold amenable to cyanide leaching. Various options are available commercially and each was researched as a possible method for Midas Gold at one point or another. A brief summary of each of the processes is provided below, along with any relevant testwork that has been completed.

13.11.2.1 Roasting

Historic roasting of antimony and refractory gold concentrates in the 1950s at the Yellow Pine smelter is described by Huttl (1952, *via* Mitchell, 2000) and summarized below. Assays of the concentrates sent to roasting are summarized in Table 13.28 (converted to metric units).

Table 13.28: Assays of Historic Concentrates Sent to Roaster

Sample	Head Grades				
	Au (g/t)	Ag (g/t)	Sb (%)	As (%)	S (%)
Gold Concentrate	85.7	102.9	4.0	9.0	35.0
Antimony Concentrate	20.6	582.9	46.0	1.8	22.0

The site had an eight hearth roaster with a temperature range of 370ºC - 730ºC which only needed a burner on one hearth due to the high sulfur content and "self-roasting" nature of the concentrates. Calcines were cooled on a rotary cooling conveyor and stored in bins; the calcines were reported to assay 1.5% S, 0.5% As and 3.0% Sb and fed the on-site smelter for doré production.

No roasting studies were conducted during the PEA or PFS program, but testwork has been completed on concentrates from Yellow Pine and West End in the past. In 1978, roasting of West End sulfide concentrates was conducted for Superior by Britton Research Ltd. The report indicated that roasting with an excess of lime at 700 - 750ºC fumed off 2.2 - 4.5% of the sulfur and 14.1 - 17.2% of the arsenic, cyanidation of the calcines was able to extract 75.2 - 76.6% of the gold in 72 hours. Reagents consumed ranged 18.5 - 21.0 kg/t for cyanide and no lime in the cyanidation tests. It was noted that roasting was not complete in either case and that more complete roasting should lead to "a marked reduction in the cyanide consumption" (Britton, 1978).

Roasting of Yellow Pine concentrates was conducted in 1987 for Hecla by Lakefield Research. The testwork was conducted at 570 - 670ºC with equal mass of nepheline syenite. There was greater than 95% oxidation of sulfides and elimination of 77 - 87% of arsenic and 86 - 96% of antimony in roasting. An acid leach of the calcines was conducted followed by cyanidation, which extracted between 62 - 74% of the gold from the calcines. Reagents consumed were 13.1 - 25.8 kg/t for cyanide and 8.9 - 13.0 kg/t for lime in the cyanidation tests. The best gold extraction was achieved at the 570ºC roasting temperature (Rollwagen, 1987).

13.11.2.2 Albion

The Albion process has not been tested on any concentrates to date. While the energy requirements for ultra-fine grinding are typically high, the site has access to relatively cheap hydroelectric power which, when combined with the lower reagent costs, may make this a feasible alternative. However, Albion was not envisioned to result in appreciable capital or operating cost savings, it was not envisioned to improve metallurgical recoveries, and it was not felt to be a technology that was sufficiently proven in a commercial setting to pursue at this time.

13.11.2.3 Biological Oxidation

Biological oxidation of the refractory sulfides prior to cyanidation for gold recovery has been tested in the West End and Yellow Pine sulfide concentrates and ores since the late 1980s and most recently on a PEA scale for the current Project.

Past Testwork

Highly refractory gold-bearing sulfide concentrates from West End, tested by Coastech Research Inc. in 1987 (Broughton, 1987) yielded 83.6% and 78% extraction of gold and silver respectively, after 85% of the sulfides were oxidized.

Yellow Pine concentrate and ore were tested by Giant Bay Biotech Inc. in 1987 (Hackl, 1987). With 93% of the sulfides oxidized, gold extraction by cyanidation improved from 10.3% to 97.2%. A continuous bio-oxidation run was then operated, the best conditions proving to be three stages of bio-oxidation over 5.2 days at 32% solids by weight and recycle of as much bio-leachate as was practical. Under those conditions they were able to achieve 91% gold recovery with cyanide consumptions in the 1.9 - 2.6 kg/t range. They recommended either a longer retention time or finer grind to further improve gold extraction.

Yellow Pine ore was tested in a heap bioleach for Hecla by the University of Idaho in 1992 (Harrington, Bartlett, & Prisbrey, 1993). The tests were completed in columns representing seven different size fractions, each utilizing cultured host rock bacterial colonies. Cyanidation was then performed on the oxidized material after size reduction to nominal minus 53 µm. Up to 72% of the gold was leachable after 360 days of bio-oxidation in the heap. Models predicted bio-oxidation would be required for two years to achieve over 90% extraction.

Current Testwork

Portions of the PEA sulfide concentrates from Yellow Pine, Hangar Flats and West End that underwent the pressure oxidation testwork were also tested through a bio-oxidation study at SGS Lakefield (Jackman, 2014b) for evaluation of applicability of the BiOx® process.

Data gathered from the study was provided to Biomin South Africa (Pty) Ltd (**Biomin**) for final evaluation and interpretation along with an order of magnitude capital and operating expense study (Olivier, 2014). The tests were conducted at 13% solids, between 38 and 42°C and a pH range of 1.4 - 1.6 controlled with acid and limestone additions. Cyanidation was conducted on rinsed residues as a CIP bottle roll at 35% solids over 48 hours at a maintained NaCN concentration of 1 g/L.

Acidulation of the concentrates was performed at the start of each test to ensure that the bacterial inoculum remained viable. This required an average of 78 kg/t sulfuric acid for Yellow Pine and Hangar Flats concentrates and 225 kg/t sulfuric acid for the high carbonate West End concentrates. Once started, the tests were acid producing and to maintain pH each consumed a maximum of 182 kg/t of limestone over the thirty-day runs and averaged 153 kg/t at twenty days and 93 kg/t at ten days.

Levels of microbial activity and oxidation of sulfides were monitored through electromotive force (**EMF**) potential readings on a silver/silver chloride electrode, in millivolts (**mV**), throughout the study. Readings of over 550 mV indicated increased microbial activity, while readings over 700 mV indicated active oxidation of sulfides by the bacteria. Results of bio-oxidation for all three concentrates were similar (see Table 13.29), with greater than 90% oxidation of sulfide achieved in the first ten days of batch testing and greater than 95% by the twentieth day (15^{th} in the case of West End concentrate). Extraction of gold was highest with West End, achieving 95.5% extraction after ten days of bio-oxidation and 97.8% after thirty days; silver extractions ranged from 75 - 79%. Next highest was Yellow Pine, achieving 92.1% gold extraction after ten days of bio-oxidation and 96.9% after thirty days; silver extractions ranged from 73 - 93%. Hangar Flats had the lowest gold extractions, achieving 86.9% gold extraction after ten days of bio-oxidation and 94.2% after thirty days; silver extractions ranged from 79 - 86%.

Table 13.29: Summary of Batch BiOx® Testwork Results

Test No.	Time (days)		$S^=$ Oxidation (%)	Recovery in CN Solution + Carbon	
	EMF >550 mV	EMF >700 mV		Au	Ag
Hangar Flats Concentrate					
Bio 1R	4	1	50.1	55.4	58.8
Bio 2	10	7	91.8	86.9	79.1
Bio 3	14	10	92.7	92.4	83.8
Bio 4	20	18	97.1	92.4	86.3
Bio 5	25	21	97.4	93.1	83.0
Bio 6	30	28	97.9	94.2	86.2
West End Concentrate					
Bio 7	4	0	26.9	56.8	42.4
Bio 8	8	6	93.3	95.5	80.8
Bio 9	14	11	98.8	97.5	79.4
Bio 10	19	16	98.4	96.4	74.7
Bio 11	24	21	99.0	97.8	77.2
Bio 12	29	28	99.0	97.8	77.1
Yellow Pine Concentrate					
Bio 13R	3	1	53.1	63.1	75.8
Bio 14	9	6	91.6	92.1	92.9
Bio 15	13	10	93.9	94.6	91.5
Bio 16	20	18	99.0	97.2	79.4
Bio 17	25	23	99.3	95.9	73.6
Bio 18	30	28	99.2	96.9	76.2

In leaching, the maximum cyanide consumption in the tests was an average of 7 kg/t, with the 20 day tests averaging 4.5 kg/t and the 10 day tests averaging 5.6 kg/t. The maximum lime consumption was an average of 40.6 kg/t, with the 20-day tests averaging 33.3 kg/t and the 10-day tests averaging18.1 kg/t.

Biomin's analyses of the data resulted in a recommendation that a BiOx® facility would require a 5-day residence time at 25% solids in the feed for the bioreactors to achieve 95% sulfide oxidation. The design assumes an average 13.8% sulfide sulfur in the feed, operation at 40 °C, nutrient feed at 3.9 kg/t of concentrate and use 67 kg/t of 75% pure limestone to maintain pH. A small amount of sulfuric acid would also be required at commissioning. The reactors would require external cooling to maintain temperature and blowers to provide air to the reactors.

Overall the residence time, oxidation and cyanidation findings were similar to the Hecla/Giant Bay study from 1987.

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14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

The Mineral Resource Statement presented herein represents the third mineral resource evaluation prepared for Midas Gold by qualified independent consultants for the Project in accordance with the Canadian Securities Administrators' National Instrument 43-101 and includes the maiden resource estimate for the historic tailings deposit.

This section describes the mineral resource estimation methodology and summarizes the key assumptions used. 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" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. **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. There is also no certainty that inferred mineral resources will be converted to the measured and indicated categories through further drilling, or into mineral reserves, once economic considerations are applied.**

The mineral resource evaluation reported herein is current and supersedes earlier mineral resource estimates completed for Midas Gold including:

- Technical Report on Mineral Resources for the Golden Meadows Project, Valley County, Idaho, dated June 6, 2011 (SRK, 2011).
- Preliminary Economic Assessment Technical Report for the Golden Meadows Project Idaho, September 21st, 2012 (SRK, 2012).

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 2009-2014, 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.

The general mineral resource estimation methodology for all deposits involved the following procedures:

- review of the geologic model and structural controls on mineralization;
- database verification;
- validation and verification of historic databases;
- data exploration, compositing and capping;
- 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 economic extraction;” and
- preparation of the mineral resource statement.

Detailed mineral resource evaluation methodologies are discussed in subsequent sections for Hangar Flats (Section 14.2), West End (Section 14.3), Yellow Pine (Section 14.4), and Historic Tailings (Section 14.5). An assessment of reasonable prospects for eventual economic extraction and mineral resource statements are presented in Sections 14.6 and 14.7, respectively.

14.2 HANGAR FLATS

14.2.1 Mineral Resource Estimation Procedures

The mineral resource estimate for Hangar Flats is based on the validated and verified drill hole database, interpreted geologic units and fault structures, digitized underground historic workings, and light detection and ranging (**LiDAR**) topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ version 14; geostatistical analysis was completed using Isaaks & Co.’s SAGE2001™ software package.

14.2.2 Drill Hole Database

The drill hole database was supplied by Midas Gold as an Excel Workbook that contained collar locations surveyed in UTM grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold, antimony, and silver, and geologic intervals with rock types. The database provided for the mineral resource estimate contained data for 256 separate drill holes representing both historic 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. Some pre-1953 drill holes were used to guide construction of estimation domains but were not used in the mineral resource estimates, including four Bradley Mining Company (Bradley) MC-series holes with sample lengths >6 m, all 1929 BMC-series holes with incomplete supporting documentation, and six DMA-series holes for which grades were not corroborated by Midas Gold confirmation drilling. After removal of select drill holes and non-bedrock assay intervals, the final database used for mineral resource estimation contained 219 drill holes with gold assays, of which 169 also have antimony assays and 144 have silver assays (as shown in Table 14.1). The vast majority of assay lengths are 1.52 m (5 ft) with sample intervals as long as 9.1 m with an overall average of 1.6 m. Modern era drill holes (post-1953) were typically drilled at a spacing of 30 to 50 m.

Table 14.1: Drill Hole Information used in the Hangar Flats Mineral Resource Estimate

Company	Gold			Silver			Antimony		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
Bradley	38	1,629	2,900	0	0	0	33	1,412	2,504
El Paso	3	112	364	2	89	274	2	46	128
Hecla	22	715	1,106	8	185	299	0	0	0
Midas Gold	134	18,323	31,386	134	18,323	31,386	134	18,291	31,328
USBM	22	807	1,225	0	0	0	0	0	0
All	219	21,586	36,981	144	18,597	31,959	169	19,749	33,959

Note: Drill hole information includes un-sampled intervals.

14.2.3 Geologic Modeling

Mineralization in the Hangar Flats deposit occurs in intrusive rocks associated with the Atlanta Lobe of the Idaho Batholith consisting of quartz monzonite and alaskite compositions. Mineralization is localized along an overall north to south striking fault zone and also along northeast striking splay faults and dilatational fault jogs. The interaction of these structural sets, one steeply dipping and one shallowly dipping, provided the ground preparation favorable for deposition of gold and antimony. Post-mineral dikes intrude the Idaho Batholith and consist of rhyolite and diabase.

The geologic model for Hangar Flats is based on a generalized single rock type model of quartz monzonite with solids representing the post-mineralization intrusive dikes and historic underground workings. Modeled structures include the Meadow Creek Fault Zone (**MCFZ**) solid (representing zones of breccia, gouge, and cataclasite), and subsidiary northeast striking splay faults. Midas Gold provided a topographic surface derived from 1 m gridded LiDAR flown in 2009, and a surface representing the current top of bedrock based on drill hole data. Historic underground workings of the Meadow Creek Mine (consisting of levels, raises, shafts, adits, and stopes) were modeled from geo-rectified historic maps in both plan views and section views.

Figure 14.1 depicts a plan view of the Hangar Flats geologic model. The historic underground workings were also modeled and the resulting solids were used to remove mined-out volumes from the reported mineral resource.

The MCFZ is the principal structure controlling mineralization. The MCFZ varies in width from 40 to 100 m and varies in dip from 80 degrees west to 45 degrees east. Gold mineralization and antimony mineralization form a corridor around the eastern boundary of the MCFZ at the intersections of the MCFZ and numerous low angle faults. The geometry and spatial extents of mineralization on the west side of the MCFZ is uncertain due to very low density of drilling. The primary occurrence of gold is similar to Yellow Pine, within quartz-sulfide veining, irregular masses within breccia and disseminated in the country rock as sulfides replacing biotites. As discussed in Section 7, antimony mineralization occurred later than gold but utilized many of the same structures.

14.2.4 Estimation Domain Modeling

The Hangar Flats gold and silver estimates utilize a grade shell and three estimation domains to define regions with different structural controls on mineralization. The antimony estimate utilizes an antimony shell. Solids representing dikes and historic underground workings were used for density assignment and data filtering only.

A gold grade shell was constructed based on an indicator Kriging estimate at a 0.25 g/t gold cutoff grade. Silver also utilizes the gold shell as silver correlates strongly with gold. A 0.1% antimony shell was manually constructed based on the underground workings, assay composites, and the geologic model. While some minor oxidation is observed within Hangar Flats, all material is assumed to be sulfide for the purpose of the mineral resource estimation.

Three estimation domains were generated for gold based on geologic interpretations, structural interpretations and the 0.25 g/t gold shell (Figure 14.2). Domain 1 is generally controlled by the MCFZ and contains steeply dipping gold mineralization. Domain 2 is bounded by Domain 1 on the west side and subdivides moderately dipping mineralization from mineralization in the fault corridor. The final gold domain contains all other blocks estimated in the model but not within the gold shell. Silver was also estimated within the gold domains. The antimony shell occurs entirely within the gold shell and contains nearly all of the potentially economic antimony mineralization. Descriptive statistics for raw assay data within the final estimation domains are shown in Table 14.2.

Variations in the mean grade of gold, antimony and silver across the domain and shell boundaries were examined using contact plots to determine domain boundary treatment during estimation. As a result, the boundary between Domains 1 and 2 is treated as a soft boundary and the antimony shell is treated as a hard boundary during estimation.

Figure 14.1: Geologic Model for Hangar Flats

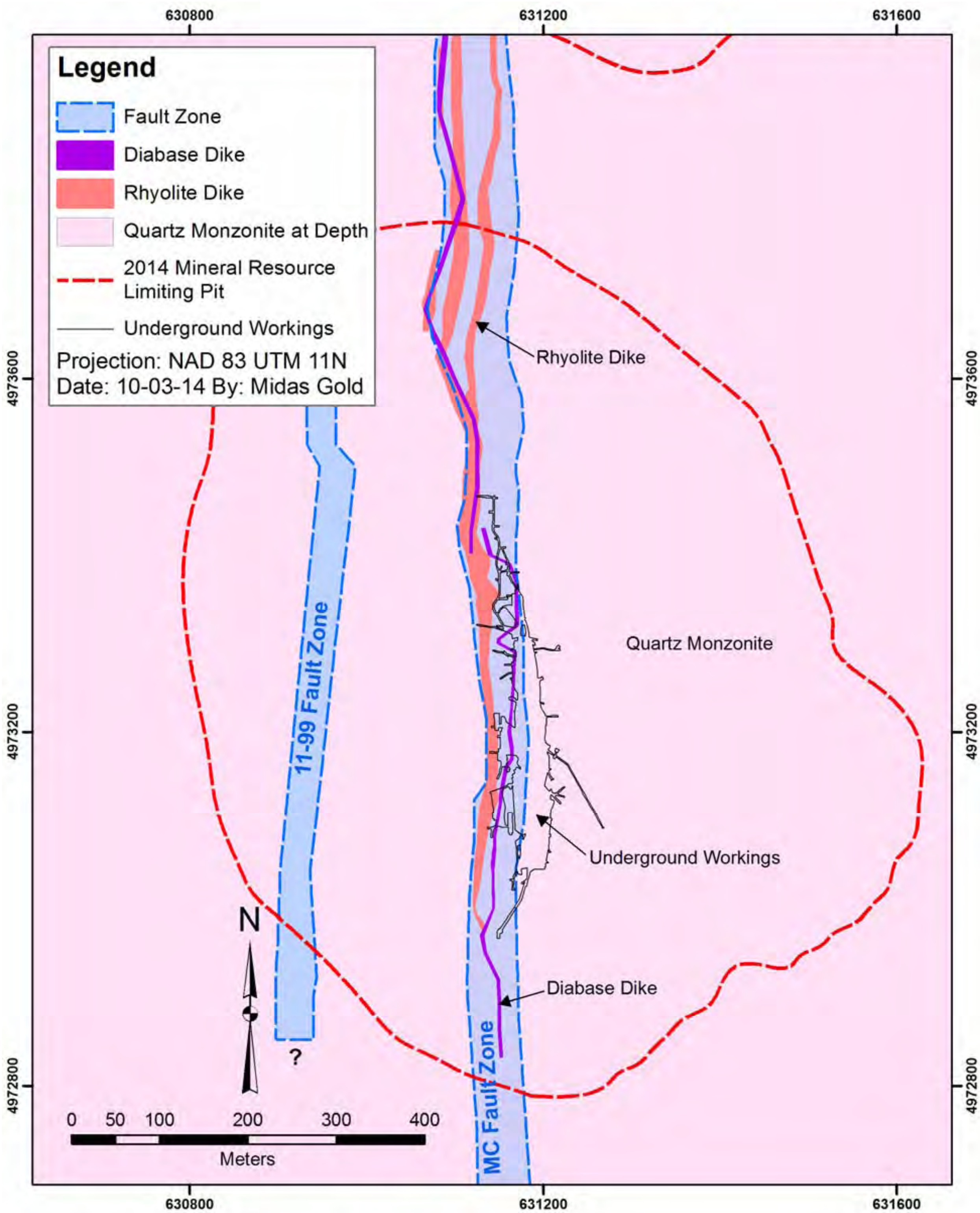


Figure 14.2: Estimation Domains for Hangar Flats

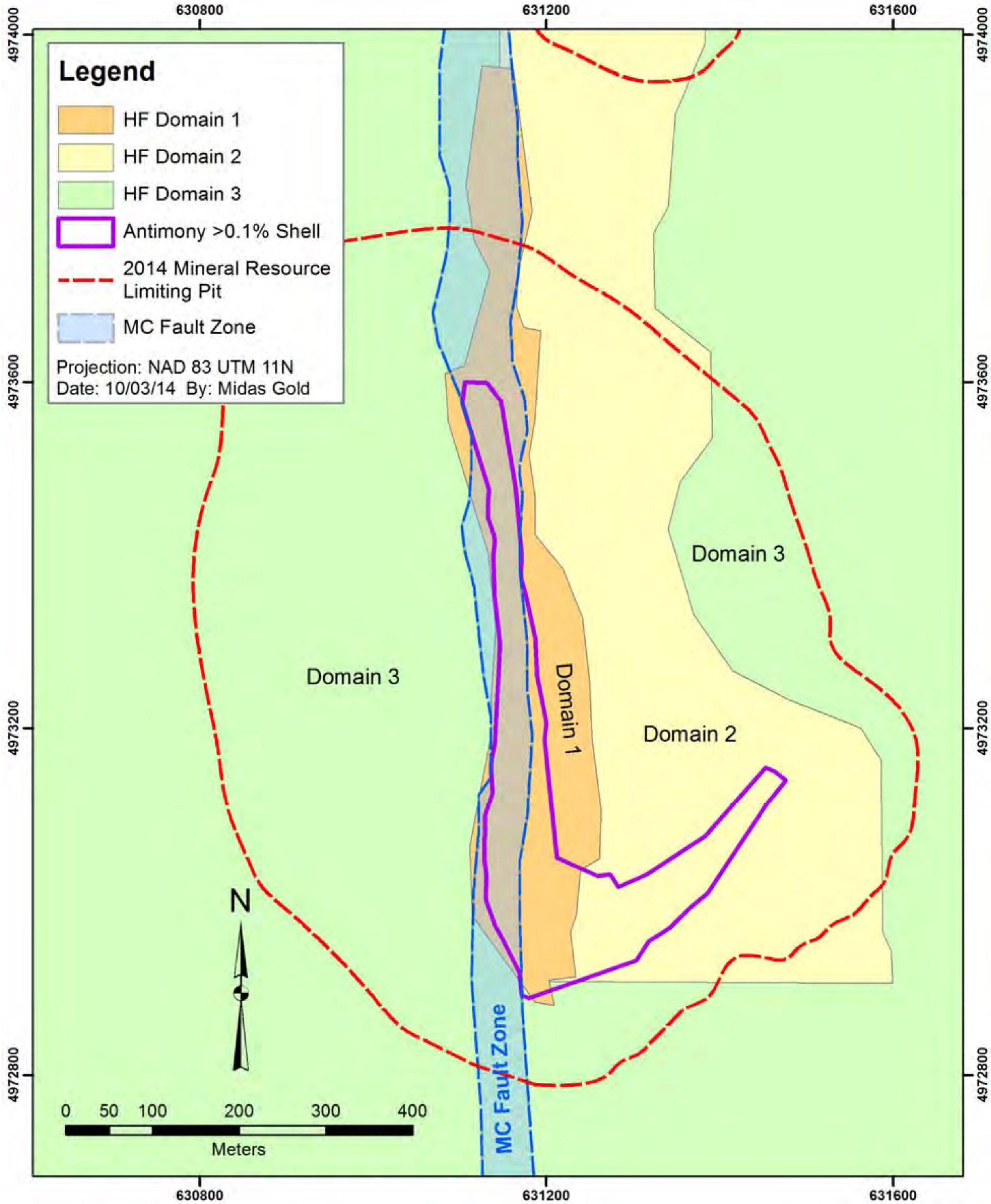


Table 14.2: Hangar Flats Raw Assay Descriptive Statistics by Estimation Domain

Statistic	Au (g/t) by Domain				Ag (g/t) by Domain				Sb (%) Relative to Shell		
	All	1	2	3	All	1	2	3	All	Sb Shell	Outside
Number	21,142	5,887	11,387	3,868	18,315	5,200	10,204	2,911	19,583	4,729	14,854
Mean	0.600	1.221	0.464	0.054	2.5	6.9	0.8	0.7	0.102	0.388	0.011
Standard Deviation	1.365	1.958	1.069	0.207	35.4	64.7	7.1	13.3	0.854	1.696	0.107
Minimum	0.003	0.003	0.003	0.003	0.0	0.3	0.0	0.3	0.000	0.000	0.000
Lower Quartile	0.005	0.034	0.008	0.003	0.3	0.3	0.3	0.3	0.001	0.005	0.001
Median	0.052	0.397	0.065	0.003	0.3	0.8	0.3	0.3	0.003	0.011	0.002
Upper Quartile	0.507	1.605	0.377	0.034	0.9	2.7	0.6	0.3	0.006	0.164	0.004
Maximum	24.8	24.8	17.85	5.143	3,160	3,160	679	709	35.00	35.00	8.290
Coefficient of Variation	2.276	1.603	2.305	3.834	14.0	9.4	8.6	20.1	8.395	4.376	10.009
95% Percentile	3.12	5.22	2.37	0.21	4.8	14	2.9	0.5	0.300	1.626	0.014
98% Percentile	5.12	7.376	3.909	0.549	10	55	4.3	1.2	0.947	4.200	0.082
99% Percentile	6.857	8.874	5.123	0.895	30.0	103.0	5.6	3.2	2.022	7.033	0.173

Note: Drill hole information excludes un-sampled intervals

14.2.5 Compositing

Gold, antimony and silver were composited downhole on 3.048 m (10 ft) intervals across geologic and domain boundaries. Composites associated with specific pre-1953 drill holes (as previously discussed), those falling within the rhyolite solid and those <0.61 m (2 ft) in length were removed and are not utilized in the estimation nor were they included in the statistical analysis. Pre-1953 historical data retained for the mineral resource estimate comprise 11% of the total data set. Table 14.3 shows statistics for the raw, un-capped composites.

Table 14.3: Hangar Flats Raw Composite Statistics by Estimation Domain

Statistic	Au (g/t) by Domain				Ag (g/t) by Domain				Sb (%) Relative to Shell		
	All	1	2	3	All	1	2	3	All	Sb Shell	Outside
Mean	0.577	1.185	0.452	0.051	2.493	6.988	0.817	0.442	0.085	0.325	0.009
Standard Error	0.011	0.030	0.012	0.004	0.399	1.402	0.061	0.055	0.006	0.023	0.001
Median	0.091	0.504	0.107	0.003	0.250	0.920	0.250	0.250	0.003	0.019	0.002
Standard Deviation	1.171	1.677	0.893	0.175	39.172	73.242	4.406	2.287	0.576	1.133	0.053
Minimum	0.003	0.003	0.003	0.003	0.010	0.250	0.010	0.250	0.000	0.000	0.000
Maximum	14.74	14.74	13.12	2.58	3,160	3,160	236.7	70.03	25.54	25.54	1.65
Count	11,132	3,106	5,802	2,224	9,660	2,729	5,202	1,729	10,222	2,478	7,744
Coefficient of Variation	2.03	1.41	1.97	3.41	15.71	10.48	5.39	5.17	6.74	3.48	6.14

14.2.6 Evaluation of Outliers

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 through 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 14.4. Descriptive statistics for capped composites are show in Table 14.5.

Table 14.4: Hangar Flats Composite Capping Grades by Estimation Domain

Statistic	Au (g/t) by Domain		Ag (g/t) by Domain		Sb (%)
	Domain 1	Domain 2	Domain 1	Domain 2	Sb Shell
Number	3,106	5,802	2,729	5,202	2,478
Maximum Value	14.7	13.12	3,160	236.66	25.54
Cap Value	10	10	225	11	8
Number Capped	9	1	8	14	10
Mean Uncapped	1.19	0.45	6.99	0.82	0.33
Mean Capped	1.18	0.45	4.91	0.71	0.31
Metal Removed	<0.5%		>30%		6%

Table 14.5: Hangar Flats Descriptive Statistics for Capped Composites

Statistic	Au (g/t) by Domain		Ag (g/t) by Domain		Sb (%)
	Domain 1	Domain 2	Domain 1	Domain 2	Sb Shell
Mean	1.181	0.452	0.723	0.641	0.307
Standard Error	0.030	0.012	0.048	0.018	0.018
Median	0.504	0.107	0.250	0.250	0.019
Standard Deviation	1.651	0.886	3.988	1.475	0.905
Minimum	0.0025	0.0025	0.01	0.01	0.0002
Maximum	10.00	10.00	236.66	70.03	8.00
Count	3,106	5,802	6,931	6,931	2,478
Coefficient of Variation	1.40	1.96	0.551	2.30	2.95

14.2.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipses and establish spatial correlation and sample weighting for the estimates. The nugget effect was derived using down-hole correlograms. Correlograms were developed for gold within estimation Domains 1 and 2, for antimony within the antimony shell, and for silver within Domain 1 and Domains 2 + 3 combined, as silver occurs both east and west of the MCFZ. Gold and silver correlograms demonstrate spatial correlation along primary structural trends while antimony is somewhat oblique. Gold and antimony correlograms were verified using indicator variograms. The correlogram parameters are summarized in Table 14.6.

Table 14.6: Correlogram Models for Hangar Flats

Element	Estimation Domain	Ellipse Axes Azimuth/Plunge			Nugget C0	Sill C1 and C2	Modeled Ranges Model 1 / Model 2 (m)			Type
		1st	2nd	3rd			1st	2nd	3rd	
Au	1-MCFZ	300/78	355/-7	264/-10	0.248	0.393	36/75	33/538	20/48	Exp
						0.359				
	2-Splay Faults	321/70	350/-18	77/9	0.193	0.439	19/176	8/545	55/253	Exp
						0.367				
Sb	1- Sb Shell	295/13	26/2	125/77	0.365	0.454	11/42	11/256	38/68	Exp
						0.182				
Ag	1	348/-3	33/86	258/3	0.414	0.48	20/360	52/27	12/219	Exp
						0.128				
	2 & 3	157/27	333/65	66/1	0.274	0.726	149	65	77	Exp

Note: Negative plunge is downward

14.2.8 Block Model Parameters and Grade Estimation

The Hangar Flats block model used 12.192 x 12.192 x 6.096 m (40 x 40 x 20 ft) blocks with coordinates defined in Table 14.7. The selected block size is approximately 25% of the median spacing of modern era drill holes and 35% of the median spacing of all drill holes and is consistent with conceptual mining bench heights. Blocks were discretized into a 4 x 4 x 2 array of points.

Table 14.7: Block Model Definition for Hangar Flats

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Hangar Flats	12.192	12.192	6.096	630,509.5	4,972,588	1,569.7	107	150	138	0
Note: Block centroid, NAD83 Zone 11N Datum										

The Hangar Flats database contains 988 density measurements for different rock types ranging from 2.01 g/cm^3 to 5.46 g/cm^3; the majority of the measurements were done onsite using the water displacement method, supported by independent third party estimates for verification. Density measurements were grouped using the geologic model wireframes and density values were calculated for each rock type represented in the geologic model after capping outliers at +/-2 standard deviations. Weighted densities were applied to the block model based on the percentage of block volume within each wire-frame. The amount of potential ore material (not rhyolite, voids or overburden) was also assigned to each block for use in mineral resource reporting. Density assignment values are shown in Table 14.8.

Table 14.8: Hangar Flats Density Assignment Values

Rock Model Unit	Bulk Density (g/cm^3)	Bulk Density (lbs/ft^3)
QM/AK/GRAN	2.63	164.2
Meadow Creek Fault	2.60	162.3
11-99 Fault	2.63	164.2
Diabase	2.61	162.9
Gouge	2.55	159.2
Rhyolite	2.54	158.6
Overburden	1.75	109.2

The Hangar Flats mineral resource estimate was completed for gold, antimony and silver using the estimation domains and shells discussed previously. Gold was estimated using ordinary Kriging within the 0.25 g/t Au grade shell in three passes. To mitigate the risk associated with use of historical data, the first pass used only post-1953 drill holes, whereas the second and third passes utilized all data. The influence of pre- vs. post-1953 data was calculated for each block using the same correlogram weighting as used for gold. Pass one for gold estimation was limited to a search based on correlogram ranges at 80% of the sill, the second pass was expanded from the first and the third pass using relaxed sample requirements and ellipse anisotropy. The boundary between Domains 1 and 2 was treated as a soft boundary with no restrictions on composite selection. Composites representing rhyolite, overburden, voids and backfill were not utilized in the estimates. A nearest neighbor estimate was also performed and used for verification purposes. Table 14.9 shows the search and sample selection parameters for the estimates.

Table 14.9: Estimation Parameters for Hangar Flats

Metal	Gold						Silver		Antimony		
Domain/Shell	Domain 1			Domain 2			Domain 1	Domain 2	Inside Sb Shell		Outside Sb Shell
Pass	1	2	3	1	2	3	1	2	1	2	1
Method	OK	OK	OK	OK	OK	OK	OK	OK	OK	IDS	IDS
Principal Axis Azimuth / Plunge(1)	350 / -5	350 / -5	350 / -5	350 / -18	350 / -18	350 / -18	348 / -3	157 / 27	26 / 2	26 / 2	0 / 0
Intermediate Axis Azimuth / Plunge(1)	80 / 6	80 / 6	80 / 6	77 / 9	77 / 9	77 / 9	78 / 3	337 / 63	125 / 77	125 / 77	90 / 0
Minor Axis Azimuth / Plunge(1)	120 / -82	120 / -82	120 / -82	142 / -70	142 / -70	142 / -70	123 / -86	247 / 0	295 / 13	295 / 13	0 / -90
Principal Axis Search Distance (m)	60	100	150	90	130	150	100	150	40	80	50
Major / Intermediate / Minor Axis	1 : 0.20 : 0.40	1 : 0.20 : 0.40	1 : 0.30 : 0.60	1 : 0.55 : 0.45	1 : 0.55 : 0.45	1 : 0.65 : 0.50	1 : 0.35 : 0.65	1 : 0.44 : 0.52	1 : 0.5 : 0.25	1 : 0.5 : 0.25	1 : 1 : 1
Search Type	4-Sector	4-Sector	1-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	4-Sector	Spherical, 1-Sector
Composite Restrictions	Domain 1&2, Post-1953	Domain 1&2	Domain 1&2	Domain 1&2, Post-1953	Domain 1&2	Domain 1&2	N/A	N/A	Domain Sb, Post-1953	Domain Sb	Outside
Maximum Composites / Sector	2	2	12	2	2	2	2	3	2	2	8
Minimum Composites	5	3	2	5	3	2	3	4	3	3	3
Minimum No. of Holes	2	2	1	2	2	1	N/A	N/A	2	2	2
Maximum Composites / Hole	4	4	6	4	4	6	N/A	N/A	4	4	4

Note:
(1) Negative plunge is downward.

Antimony was also estimated in three passes. The first pass used ordinary Kriging, and extended to a range of nearly 90% of the sill. The second pass extended the search to twice the first pass range and utilized inverse distance squared weighting for estimation of blocks that were not estimated in the first pass. The third pass estimated blocks outside of the antimony domain by inverse distance squared interpolation with an omni-directional ellipsoid search and isotropic weighting. Pre-1953 data was filtered for use, depending upon pass, in the same manner as the gold estimate.

Silver was estimated using ordinary Kriging within domains 1 and 2, separately. The boundary between the domains was treated as soft for silver estimation. Only blocks that received a gold estimate were estimated for silver.

14.2.9 Block Model Validation

The block model for the Hangar Flats mineral resource was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was checked by reviewing the block estimate relative to the geologic model, domain boundaries and grade shell. Block model variables were checked to ensure that they fall within appropriate ranges. Global bias was assessed by comparing the estimated grade to the nearest neighbor estimate. Local bias was assessed on swath plots in the X, Y and Z directions as shown on Figure 14.3, Figure 14.4, and Figure 14.5, respectively.

Change of support for gold was assessed using a Hermitian Correction model (**HERCO**) and indicates -1% to +3% bias in contained metal for Domains 1 and 2 between the Kriged estimate and the theoretical grade-tonnage distribution at a cutoff grade of 0.75 g/t Au indicating the Kriged estimate yields an appropriate level of smoothing. Relative to the 2012 PEA estimate, the new estimate yields an 18% increase in indicated gold ounces and a 15% increase in antimony pounds, consistent with the addition of new drill holes to the estimation database.

Figure 14.3: North-South Gold Swath Plot for Hangar Flats

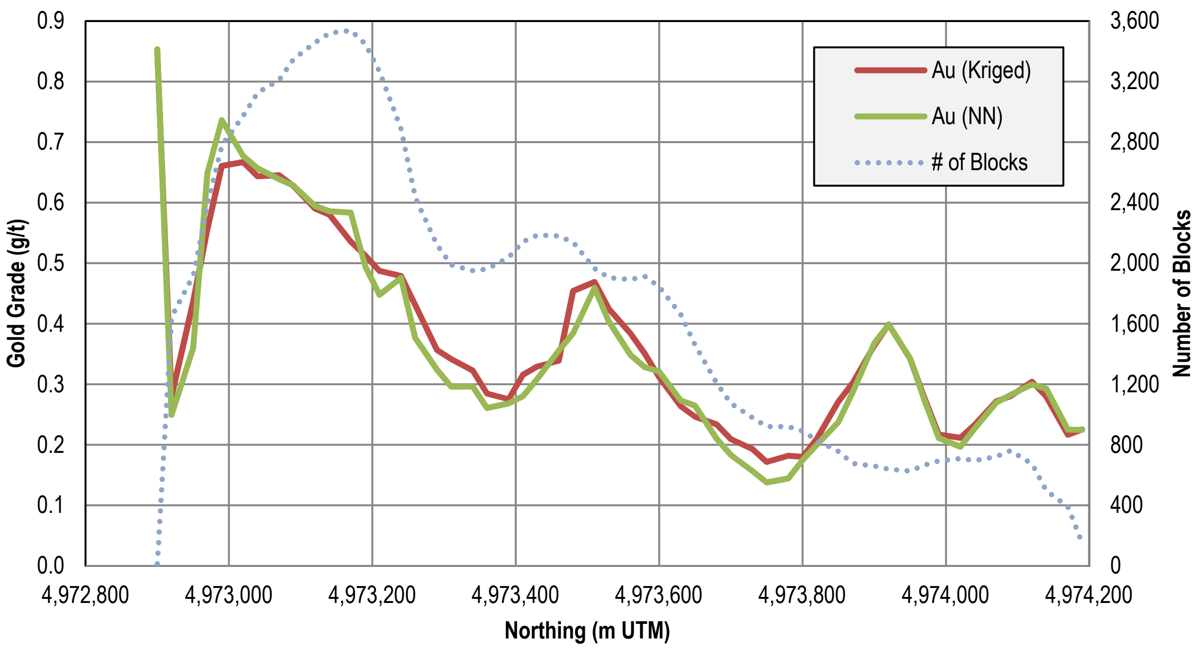


Figure 14.4: East-West Gold Swath Plot for Hangar Flats

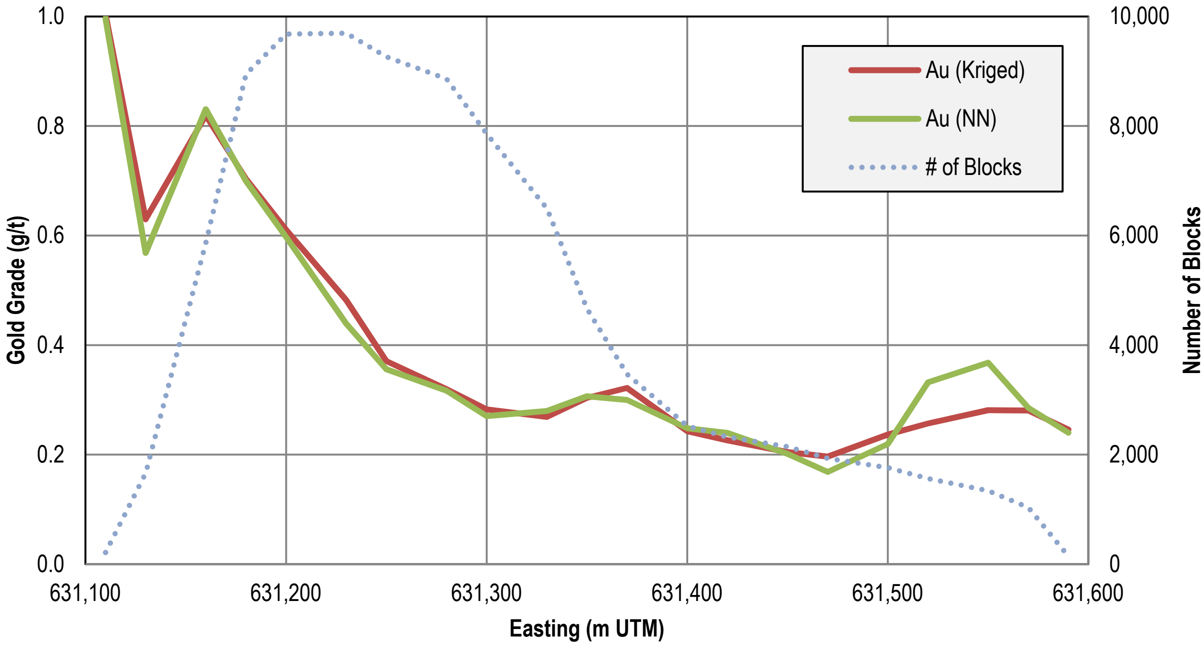
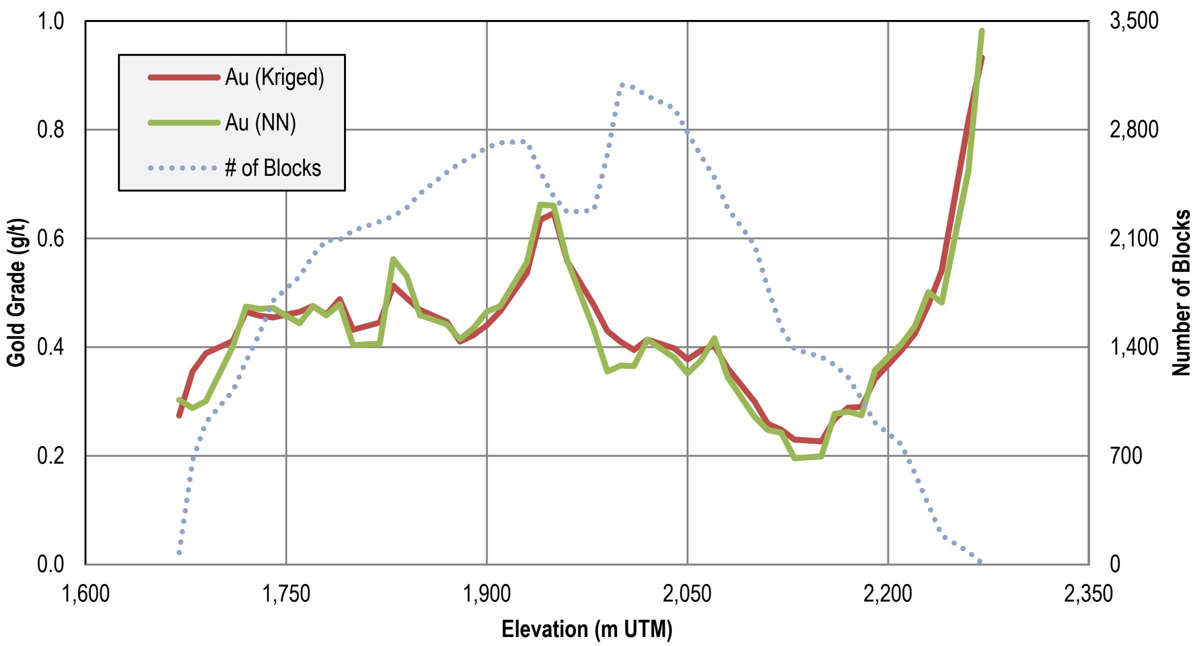


Figure 14.5: Elevation Gold Swath Plot for Hangar Flats



14.2.10 Hangar Flats Mineral Resource Classification

Confidence criteria used to guide mineral resource classification include composite distance, number of drill holes used, number of composites, influence of post-1953 data in the estimate, and single block Kriged results. Blocks eligible for indicated classification for gold are those estimated in the first estimation pass with a minimum anisotropic distance of <50 m, or those estimated in the second pass meeting the above criteria with composites in at least two sectors. Only blocks in Domains 1 or 2 are eligible for indicated classification. Antimony mineral resources require at least three composites from two drill holes with a minimum distance of 40 m to be eligible for indicated classification. Antimony in blocks meeting the gold criteria, but not the antimony criteria, are not included in the antimony mineral resource estimate. Single block Kriged results support the gold classification strategy. Final classification was applied following manual smoothing of the results on 6 m plan sections to produce a model with reasonably contiguous zones of inferred and indicated blocks.

14.3 WEST END

14.3.1 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 modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ Version 14; geostatistics and semi-variogram analyses were completed using Isaaks & Co.'s SAGE2001™ software package.

14.3.2 Drill Hole Database

The drill hole database supplied by Midas Gold for mineral resource modeling included 940 drill holes in Excel format. The database consisted of collar locations in UTM 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.

The West End deposit was previously in production as a heap leach operation and many infill drill holes were drilled during the 1980s and 1990s using various methods, as previously described in Section 10. The drill holes were reviewed, and certain drill holes were not considered reliable for use in mineral resource estimation, including rotary and air-track drill holes, and other un-reliable holes flagged by Midas Gold. After removal of selected drill holes and non-bedrock intervals, the final database contained 674 drill holes.

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. Approximately 78% of the assay records have gold fire assays (**AuFA**) and 75% have cyanide soluble gold assays (**AuCN**). Some historic operators selectively used fire assays within the sulfide zones where sulfide mineralization was observed, resulting in a dataset that contains some "spot" AuFA records. This results in an apparent high bias because higher-grade intervals were selected for fire assay. 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. Au_Final, the variable used for estimation of total gold and discussed in the remainder of this section, is shown by drilling campaign in Table 14.10.

Partial or spot assaying for AuCN is prevalent throughout the deposit, especially within the Superior-era drill holes where available AuCN assays do not adequately define the transition from oxide to sulfide gold. This issue was addressed by removing 70 drill holes with incomplete AuCN assays following section-by-section review for completeness and potential impacts to the mineral resource estimate. The final dataset for estimation of cyanide soluble gold is shown in Table 14.10.

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. 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.

Table 14.10: Drill Hole Information used in the West End Mineral Resource Estimate

Company	Au Fire Assay			Au 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.

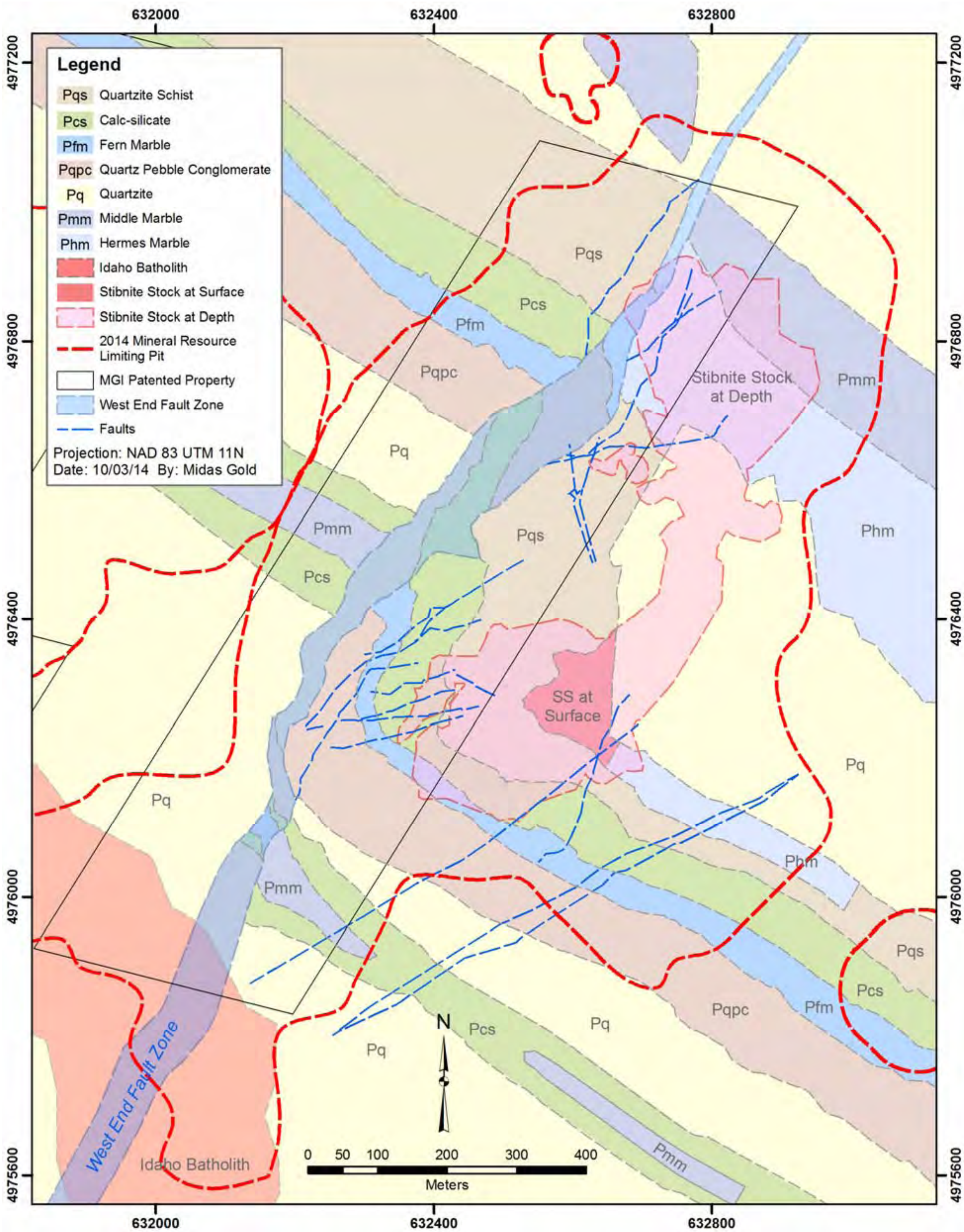
The vast majority of assay lengths are 1.52 m (5 ft) for the historic campaigns and 1.52 m (5 ft) to 2.1 m (7 ft) for the Midas Gold drill holes. The mean sample length is 1.61 m.

14.3.3 Geologic Modeling

The West End deposit occurs in an overturned sequence of steeply dipping Proterozoic to Paleozoic metasediments comprising the Stibnite Roof Pendant. As discussed in Section 7, lithologic units consist of quartzite, quartz-pebble conglomerate, interbedded quartzite and schist, limestones, dolomitic marble, and calc-silicate rocks and range in thickness from 70 – 180 m. The meta-sedimentary rocks are intruded by quartz-monzonite and granitic stocks. Mineralization occurs within and adjacent to fault zones, principally the southeast dipping WEFZ.

The geologic model prepared by Midas Gold consists of eight northeast-dipping lithostratigraphic units which intersect, and are offset across, the WEFZ (Figure 14.6). The WEFZ is modeled as two surfaces representing the hanging wall and footwall of the structural corridor and is up to 80 m wide, dipping 50 to 70 degrees to the southeast with a strike length of over 1.7 km. Additional wireframes representing splay faults and subsidiary structures to the main WEFZ were also provided. The geologic model also includes two intrusive units, the Stibnite Stock of granitic composition which intrudes the metasediments 200 to 300 m east of the WEFZ, and the Idaho Batholith, an intrusion of quartz monzonite composition occurring at the southeast margin of the deposit. Midas Gold provided a topographic surface derived from 1 m gridded LiDAR flown in 2009, a pre-mining topographic surface constructed from historic maps and drill hole collars, and an overburden surface representing the current top of bedrock constructed from drill hole data, and historic pit 'as-builts' representing the extent of historic mining.

Figure 14.6: Geologic Model for West End



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. Gold mineralization is associated with silica alteration occurring as quartz-veinlets, stockworks and zones of silica flooding and replacement. 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.

The oxidation level in the deposit is of moderate and variable depth, with elevated AuCN values occurring at shallow levels, preferentially within certain lithologic units, and locally at deeper elevations between strands of the WEFZ and along splay structures. AuCN mineralization is only sparsely tested below the 1,900 m elevation in widely spaced Midas Gold drill holes.

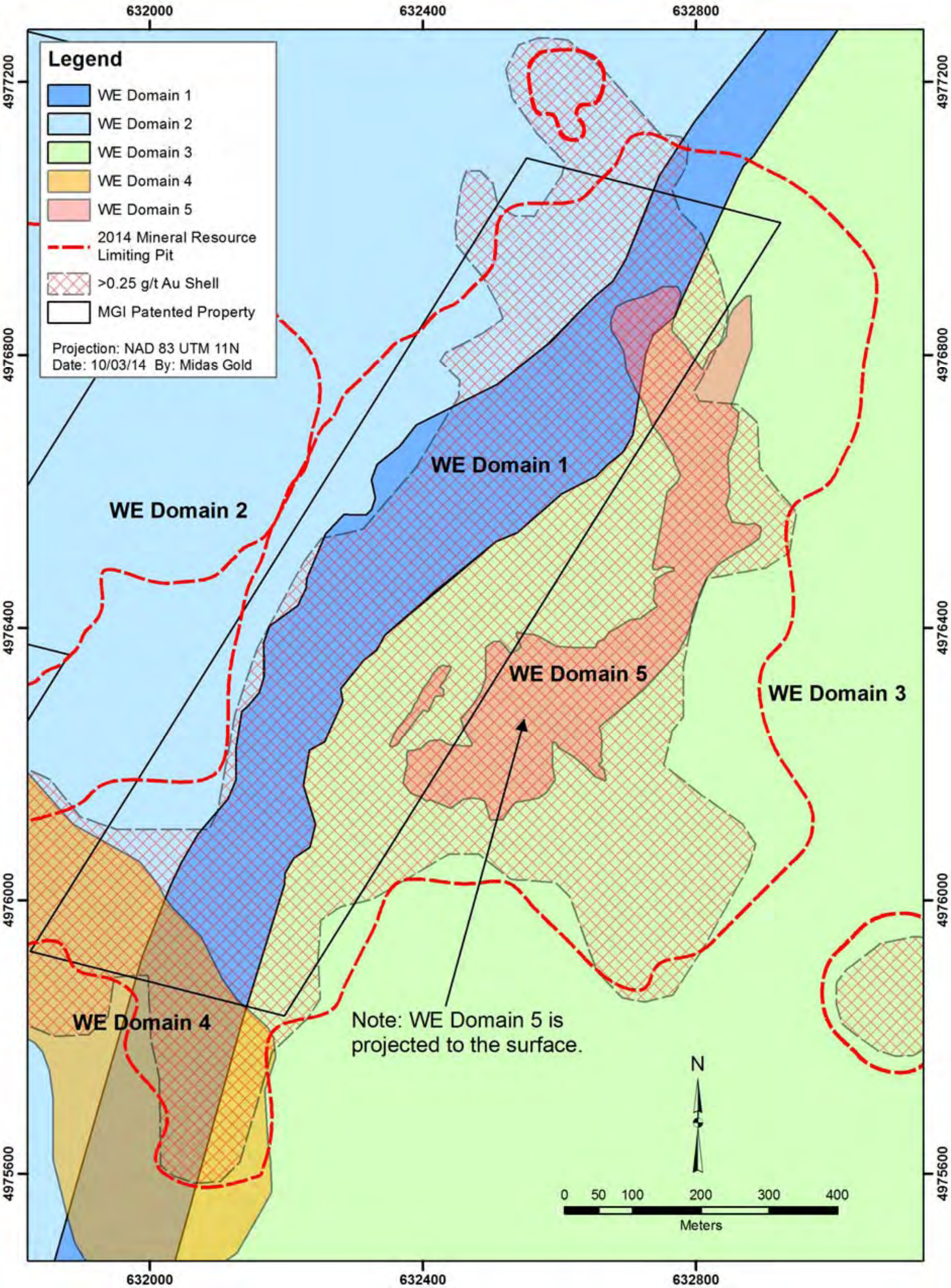
14.3.4 Estimation Domain Modeling

The West End gold estimates utilized the geologic model, a 0.25 g/t Au grade shell and five estimation domains to characterize gold deposition in relation to structural and stratigraphic controls. Boundary treatment during estimation is based on analysis of grade variability across geologic and estimation domain contacts.

A grade shell was constructed based on an indicator estimate using a cutoff grade of 0.25 g/t Au_Final. The shell demonstrates reasonable continuity along strike and vertically. For AuCN, the shell was modified slightly and restricted to elevations above 1,911 m. A grade shell was not developed for silver because mineralization is erratic and generally low grade throughout the deposit.

Estimation domains for Au_Final and AuCN were developed from the geologic model, perceived structural controls on mineralization, grade contouring, graphical plots of assay data and the 0.25 g/t Au grade shells. The WEFZ exerts a strong structural control on gold mineralization but fault contacts are not distinct boundaries; rather gold mineralization extends into favorable stratigraphic units adjacent to the structure. Estimation Domain 1 is the WEFZ expanded 50 m to the east and west, so as to encompass mineralization within and adjacent to the fault zone. Domains 2 and 3 include mostly low-grade material to the west and east of the WEFZ respectively, exclusive of Domains 4 and 5. Domain 4 encompasses generally low-grade gold mineralization hosted primarily within quartz monzonite of the Idaho batholith in the southwest region of the deposit. Domain 5 is entirely within Domain 3 and captures gold mineralization associated with east-northeast striking splay faults within the Stibnite Stock in the eastern region of the deposit. The West End estimation domains are shown on Figure 14.7.

Figure 14.7: Estimation Domains for West End



Stratigraphic controls exert a strong influence on gold mineralization and were investigated within estimation domains through the use of descriptive statistics. In the simplified geologic model, certain lithostratigraphic units encompass multiple rock types (i.e. the quartzite-schist formation) and statistics were prepared for both logged lithology groupings and for samples occurring within modeled geologic units. In general, the distribution of gold within geologic domains is similar to that observed in logged lithology types, with the highest mean gold grades observed within the quartzite-schist formation (quartzite, psammite and schist), the calc-silicate formations (marble, metapelite, calcareous schists) and fault related rocks (breccia and gouge) (Table 14.11). The carbonate units (marble, limestone and dolomite) and clastics (conglomerate and quartzite) are generally lower grade.

Table 14.11: West End Descriptive Statistics by Rock Solid

Statistic	Rock Solid (g/t Au)											
	QZ	Hermes Marble	Upper Quartzite	Quartzite Schist	LCS	FM	QPC	Lower Quartzite	MM	UCS	ID Batholith	Stibnite Stock
Mean	0.518	0.290	0.585	1.114	1.044	0.619	0.645	0.358	0.970	0.439	0.452	0.605
Standard Error	0.031	0.032	0.044	0.019	0.024	0.027	0.017	0.015	0.098	0.039	0.072	0.016
Median	0.170	0.137	0.274	0.4625	0.274	0.137	0.240	0.156	0.309	0.081	0.171	0.343
Standard Deviation	0.909	0.766	0.799	1.754	2.064	1.441	1.158	0.656	1.933	1.031	1.348	0.914
Minimum	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0100	0.0025	0.0050	0.0025
Maximum	8.503	11.52	6.000	22.63	28.22	17.62	16.11	9.394	13.61	12.55	20.91	16.63
Count	877	567	327	8,536	7,587	2,894	4,437	1,832	392	686	347	3,251
CV	1.75	2.64	1.36	1.57	1.97	2.32	1.79	1.83	1.99	2.35	2.98	1.50

Variations in mean grade of gold assays across geologic boundaries were examined using contact plots to determine sub-domains requiring hard-boundary treatment during estimation. In Domain 1, abrupt grade changes were noted between the Hermes Marble and Quartzite-Schist formations and between the Lower Calc-silicate and Lower-quartzite formations. Within Domain 1, the resulting estimation sub-domains separate the Hermes Marble and the Lower Quartzite from other units. AuCN is generally comparable to gold within the Au_Final domains, with the exception of Domain 3 where the Lower Calc-silicate is a sub-domain with 10 m soft boundaries. Boundary conditions of the estimation domains for Au_Final were examined using contact plots and analysis of composited data (discussed below) and indicate both hard and soft boundaries between various estimation domains and lithologic sub-domains which were applied where warranted. For silver, marble units in Domain 1 form a distinctly lower-grade population and were estimated separately from other lithologies. Otherwise, the silver estimation domains are the same as those for gold but do not segregate geologic solids into sub-domains.

14.3.5 Compositing

Gold, AuCN and silver assays were composited downhole on 3 m intervals across geologic and estimation domain boundaries and excluding un-assayed or missing intervals. Composites <1 m were removed from the final data set. The 3 m composite length is an even multiple of the majority of raw assay lengths and represents 50% of the proposed mining bench height and estimation block height.

Table 14.12: West End Raw Composite Statistics by Estimation Domain

Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	LCS/QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Fire Assay Statistics (g/t Au)									
Mean	0.335	0.491	1.179	0.751	0.414	1.046	0.508	0.445	0.493
Standard Error	0.062	0.023	0.017	0.049	0.032	0.041	0.017	0.039	0.013
Median	0.183	0.313	0.644	0.388	0.250	0.337	0.224	0.273	0.293
Standard Deviation	0.847	0.617	1.483	1.007	0.494	2.025	0.898	0.605	0.651
Minimum	0.004	0.003	0.003	0.003	0.014	0.003	0.003	0.017	0.017
Maximum	10.29	5.504	18.39	8.503	3.974	22.03	14.36	5.113	10.16
Count	184	702	7,325	426	235	2,445	2,938	239	2,459
Coefficient of Variation	2.53	1.26	1.26	1.34	1.19	1.93	1.77	1.36	1.32
Statistic	**Domain 1**			**Domain 2**		**Domain 3**		**Domain 4**	**Domain 5**
	HM	**LQ**	**Other**	**All**		**LCS**	**Other**	**ID Batholith**	**Stibnite Stock**
Cyanide Assay Statistics (g/t Au)									
Mean	0.129	0.364	0.661	0.309		0.608	0.344	0.377	0.354
Standard Error	0.014	0.022	0.013	0.020		0.026	0.012	0.040	0.010
Median	0.079	0.238	0.284	0.145		0.179	0.172	0.206	0.209
Standard Deviation	0.176	0.506	1.016	0.493		1.294	0.604	0.533	0.494
Minimum	0.015	0.015	0.015	0.015		0.015	0.015	0.017	0.015
Maximum	1.577	5.143	12.27	4.983		19.46	12.93	4.860	7.948
Count	161	520	5,721	629		2,388	2,730	178	2,410
Coefficient of Variation	1.37	1.39	1.54	1.60		2.13	1.76	1.41	1.39

14.3.6 Evaluation of Outliers

To mitigate estimation risk associated with use of high-grade statistical outliers, capping grades were selected for each estimation domain after declustering and weighting raw 3 m composite data. Capping grade was evaluated using log probability plots and through analysis of contained metal within deciles and centiles, following the Parrish Method (Parrish, 1997). Both methods yielded similar results, except for Domain 1, where the more conservative composite capping grade of 10 g/t Au was selected. AuCN was capped in a manner similar to gold. Silver assays were capped at 10 g/t Ag prior to compositing on 3 m intervals. Table 14.13 shows final capping grades for West End. Low capping grades in domains 2, 4 and 5 are warranted due to low average grade and small relative standard deviation.

Table 14.13: Capping Grades for West End

Statistic	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5
Fire Assay Statistics within Gold Shell (g/t Au)					
Number	8,211	661	5,383	239	2,459
Maximum Value	18.39	8.503	22.03	5.113	10.16
Cap Value	10	3	10	2.3	3
Number Capped	14	24	33	3	27
Mean Uncapped	1.101	0.631	0.753	0.445	0.493
Mean Capped	1.096	0.593	0.731	0.420	0.477
Metal removed (%)	0.46	6.41	2.92	6.05	3.43

Statistic	Domain 1	Domain 2	Domain 3	Domain 4	Domain 5
Cyanide Assay Statistics within Gold Shell (g/t Au)					
Number	6,402	629	5,118	178	2,410
Maximum Value	12.27	4.983	19.46	4.860	7.948
Cap Value	6	4	9	4	3
Number Capped	30	2	12	1	12
Mean Uncapped	0.624	0.309	0.467	0.377	0.354
Mean Capped	0.616	0.307	0.460	0.372	0.345
Metal removed (%)	1.20	0.80	1.50	1.30	2.57

Table 14.14: West End Descriptive Statistics for Capped Composites

Statistic	Domain 1			Domain 2		Domain 3		Domain 4	Domain 5
	HM	LQ	Other	LCS/QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Fire Assay Statistics within Gold Shell (g/t Au)									
Mean	0.333	0.491	1.173	0.695	0.410	1.003	0.505	0.420	0.477
Standard Error	0.061	0.023	0.017	0.038	0.030	0.035	0.016	0.029	0.011
Median	0.183	0.313	0.644	0.388	0.250	0.337	0.224	0.273	0.293
Standard Deviation	0.828	0.617	1.440	0.777	0.467	1.733	0.863	0.456	0.528
Minimum	0.004	0.003	0.003	0.003	0.014	0.003	0.003	0.017	0.017
Maximum	10.00	5.504	10.00	3.000	3.000	10.00	10.00	2.300	3.000
Count	184	702	7,325	426	235	2,445	2,938	239	2,459
Coefficient of Variation	2.49	1.26	1.23	1.12	1.14	1.73	1.71	1.09	1.11
Statistic	**Domain 1**			**Domain 2**		**Domain 3**		**Domain 4**	**Domain 5**
	HM	**LQ**	**Other**	**All**		**LCS**	**Other**	**ID Batholith**	**Stibnite Stock**
Cyanide Assay Statistics within Gold Shell (g/t Au)									
Mean	0.129	0.364	0.653	0.307		0.595	0.342	0.372	0.345
Standard Error	0.014	0.022	0.013	0.019		0.024	0.011	0.037	0.008
Median	0.079	0.238	0.284	0.145		0.179	0.172	0.206	0.209
Standard Deviation	0.176	0.506	0.958	0.472		1.173	0.578	0.495	0.411
Minimum	0.015	0.015	0.015	0.015		0.015	0.015	0.017	0.015
Maximum	1.577	5.143	6.000	4.000		9.000	9.000	4.000	3.000
Count	161	520	5721	629		2388	2730	178	2410
Coefficient of Variation	1.37	1.39	1.47	1.54		1.97	1.69	1.33	1.19

14.3.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipses and establish spatial correlation and sample weighting for the estimates. The nugget effect was derived using down-hole correlograms. Correlograms for capped 3 m composites within the grade shell were developed for Au_Final, AuCN and Ag_Final within estimation Domains 1, 2 and 3. The intrusive rocks in Domains 4 and 5 did not yield reliable correlograms and were estimated using inverse distance weighting. Gold and cyanide soluble gold demonstrate spatial continuity along dominant structural trends and within stratigraphic units. Cyanide soluble gold demonstrates greater vertical continuity than Au_Final. The correlogram parameters are summarized in Table 14.15.

Table 14.15: Correlogram Models for West End

Element	Estimation Domain	Ellipse Axes Azimuth/Plunge			Nugget C0	Sill C1/C2	Modeled Ranges Model 1 / Model 2			Type
		1st	2nd	3rd			1st	2nd	3rd	
Au_Final	1 & 2	254/-47	4/-17	108/-38	0.067	0.601/0.332	16/275	16/82	20/43	Exp
	3	95/-33	45/44	166/27	0.186	0.814	62	33	14	Exp
	4 & 5	Inverse Distance Weighting								
Au_CN	1 & 2	49/-20	142/-6	67/69	0.161	0.839	101	50	35	Exp
	3 - LCS	115/-53	44/14	144/33	0.099	0.901	47	27	14	Exp
	3 - Other	63/-6	175/-75	151/14	0.15	0.85	41	21	10	Exp
	4 & 5	Inverse Distance Weighting								
Ag	1	86/-22	353/-6	69/67	0.234	0.766	86	64	52	Exp
	2 & 3	37/-38	77/44	325/21	0.039	0.961	164	37	14	Exp
	4 & 5	Inverse Distance Weighting								

14.3.8 Block Model Parameters and Grade Estimation

The West End block model comprises 15.24 x 15.24 x 6.096 m blocks (50 x 50 x 20 ft) with coordinates defined in Table 14.16. Blocks were discretized into a 5 x 5 x 2 array of points during estimation.

Table 14.16: Block Model Definition for West End

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
West End	15.24	15.24	6.096	631,727.6	4,975,408.0	1,704.0	97	127	114	0
Note: Block centroid, NAD83 Zone 11N Datum										

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 14.17.

Table 14.17: West End Density Assignment Values

Rock Model Unit	Bulk Density (g/cm^3)	Bulk Density (lbs/ft^3)
Quartzite & Background	2.61	162.9
Quartzite-Schist	2.70	168.6
Lower Calc-Silicate	2.74	171.1
Fern Marble	2.78	173.5
Qtz Pebble Conglomerate	2.63	164.2
Lower Quartzite	2.65	165.4
Middle Marble	2.80	174.8
Upper Calc-Silicate	2.76	172.3
ID Batholith	2.54	158.6
Stibnite Stock	2.61	162.9
Overburden	1.75	109.2

The West End mineral resource estimate was completed for gold, cyanide-soluble gold and silver using the estimation domains and sub-domains discussed previously. The estimate is limited to blocks occurring within the

0.25 g/t Au_Final grade shell below the current LiDAR topographic surface. Estimates for Au_Final, AuCN and Ag_Final in Domains 1, 2 and 3 are derived by ordinary Kriging using the 3 m composite file and correlogram weighting models discussed above. Estimates for Domains 4 and 5 use inverse-distance weighting to various powers. Estimation is performed in two passes; the first pass is limited to a search based on the correlogram ranges at approximately 90% of the sill. The second pass search is expanded to a multiple of the first pass with greater isotropy. Composites occurring above the current bedrock surface from pre-historic mining activity are utilized in the estimate, but material in this region is not included in the mineral resource. Table 14.18 shows estimation criteria for gold and cyanide soluble gold by domain. Silver was estimated in a manner similar to gold but is not detailed in this Report because it is of minor economic significance.

Table 14.18: Summary of Estimation Parameters for West End

Parameter	1			2		3		4	5
	HM	LQ	Other	LCS/ QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Pass 1 – Fire Assay Au									
Pass 1 Method	OK	OK	OK	OK	OK	OK	OK	ID3	ID2
Princ. Axis Az/Plunge(1)	280/-35	280/-35	280/-35	280/-35	280/-35	45/-48	45/-48	0/0	0/0
Int. Axis Az/Plunge	346/30	346/30	346/30	346/30	346/30	93/31	93/31	0/-90	0/-90
Minor Axis Az/Plunge	226/40	226/40	226/40	226/40	226/40	347/-25	347/-25	90/0	90/0
Princ. Axis Srch Dist. (m)	50	50	50	40	40	50	50	40	50
Maj/Int/Minor Axis	1:.67:.55	1:.67:.55	1:.67:.55	1:.67:.55	1:.67:.55	1:.65:.35	1:.65:.35	01:01:01	01:01:01
Search Type	Sector	Sector	Sector	Open	Open	Sector	Sector	Sector	Sector
Comp Restrictions	Hard	Hard	Hard	Hard	Hard	Soft (10m)	Soft (10m)	Hard	Hard
Max Comps/Sector	3	3	3	12	12	3	4	4	3
Min Comps	4	4	4	3	3	4	5	5	4
Min # of Holes	2	2	2	2	2	2	2	2	2
Max Comps/Hole	5	5	5	4	5	5	4	4	4
Pass 2 – Fire Assay Au									
Pass 2 Method	OK	OK	OK	OK	OK	OK	OK	ID3	ID2
Princ. Axis Az/Plunge	280/-35	280/-35	280/-35	280/-35	280/-35	356/2	356/2	0/0	0/0
Int. Axis Az/Plunge	346/30	346/30	346/30	346/30	346/30	79/-75	79/-75	0/-90	0/-90
Minor Axis Az/Plunge	226/40	226/40	226/40	226/40	226/40	267/-15	267/-15	90/0	90/0
Princ. Axis Srch Dist. (m)	150	150	150	150	150	150	150	75	150
Maj/Int/Minor Axis	1:.67:.55	1:.67:.55	1:.67:.55	1:.75:.75	1:.67:.55	1:.7:.4	1:.7:.4	01:01:01	01:01:01
Search Type	Open	Open	Open	Open	Open	Sector	Sector	Sector	Sector
Comp Restrictions	Hard	Hard	Hard	Hard	Hard	Soft (10m)	Soft (10m)	Hard	Hard
Max Comps/Sector	12	12	12	12	12	3	3	4	2
Min Comps	2	2	2	2	2	2	2	2	3
Min # of Holes	1	1	1	1	1	1	1	1	1
Max Comps/Hole	5	5	3	4	4	4	4	4	4

Parameter	1			2		3		4	5
	HM	LQ	Other	LCS/ QZ_SCH	Other	LCS	Other	ID Batholith	Stibnite Stock
Pass 1 – Cyanide Assay Au									
Pass 1 Method	OK	OK	OK	OK		OK	OK	IDS	IDS
Princ. Axis Az/Plunge(1)	49/20	49/20	49/20	49/20		115/53	63/6	0/0	57/-1
Int. Axis Az/Plunge(1)	141/6	141/6	141/6	141/6		145/-33	176/75	90/0	144/73
Minor Axis Az/Plunge(1)	247/69	247/69	247/69	247/69		226/15	332/14	90/90	327/17
Princ. Axis Srch Dist. (m)	65	65	90	65		50	50	65	150
Maj/Int/Minor Axis	1:.5:.35	1:.5:.35	1:.5:.35	1:.5:.35		1:.30:.57	1:.51:.24	01:01:01	1:.64:.36
Sectors	4	4	4	4		4	4	1	4
Domains Permitted	1,3	1,2,3	1,2,3	1,2		1,3	1,3	4	5
Max Comps/Sector	2	2	2	2		3	2	8	2
Max Comps/Sector	2	2	2	2		3	2	8	2
Min Comps	3	3	3	3		2	3	2	3
Min # of Holes	1	1	1	1		1	1	1	1
Max Comps/Hole	4	4	4	4		4	4	4	4

Notes:
(1) Negative plunge is downward.

14.3.9 Block Model Validation

The block model for West End was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was checked by reviewing the block estimate relative to the geologic model, domain boundaries and grade shell. Block model variables were check to ensure that they fall within appropriate ranges. Global bias was assessed by comparing the estimated grade to the nearest neighbor estimate within each estimation domain. Local bias was assessed on swath plots in the X, Y and Z directions on Figure 14.8, Figure 14.9, and Figure 14.10, respectively. Change of support was assessed using a HERCO. The HERCO validation suggests that the Kriged models are under-smoothed with respect to the theoretical grade-tonnage distribution at a cutoff grade of 0.75 g/t Au and classification is restricted to the indicated and inferred classes accordingly. A sensitivity model run to assess the results of capping indicates that capping results in an approximate 2% decrease in grade and 2% decrease in tonnage resulting in a 4% decrease in contained gold. Relative to the 2012 PEA mineral resource estimate, the new model indicates a small increase in indicated mineral resources and a 44% decrease in inferred gold ounces. The overall decrease in inferred gold ounces resulted from conversion of inferred mineral resources to the indicated category. Although new drilling successfully expanded indicated mineral resources through conversion of inferred mineral resources to indicated in some areas, this increase was largely offset by the loss of previously indicated material in other areas due to different treatment of the high-biased selective gold fire assays, where un-assayed total gold intervals were not populated with background CN assay data in 2012 but were in 2014.

Figure 14.8: East-West Gold Swath Plot for West End

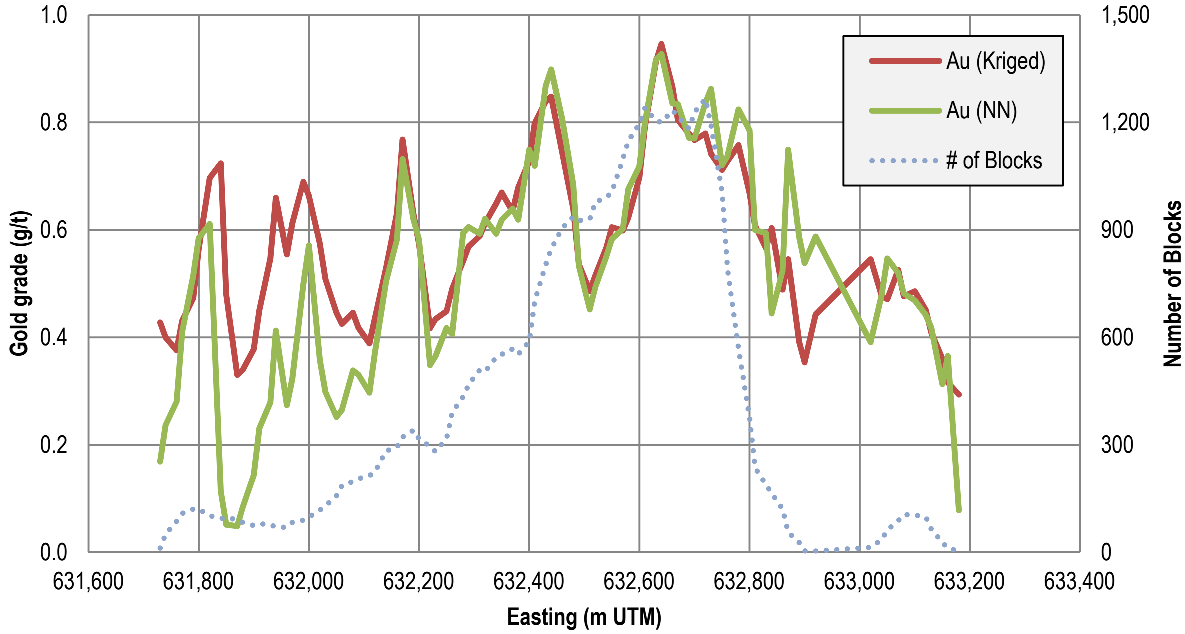


Figure 14.9: North-South Gold Swath Plot for West End

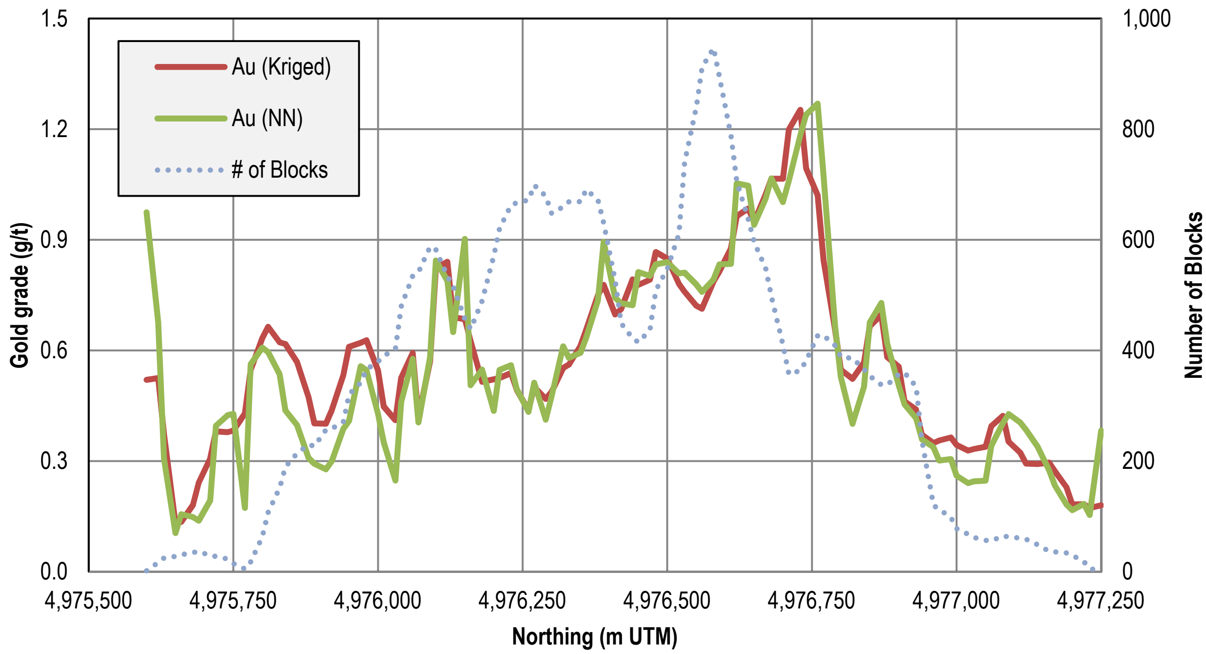
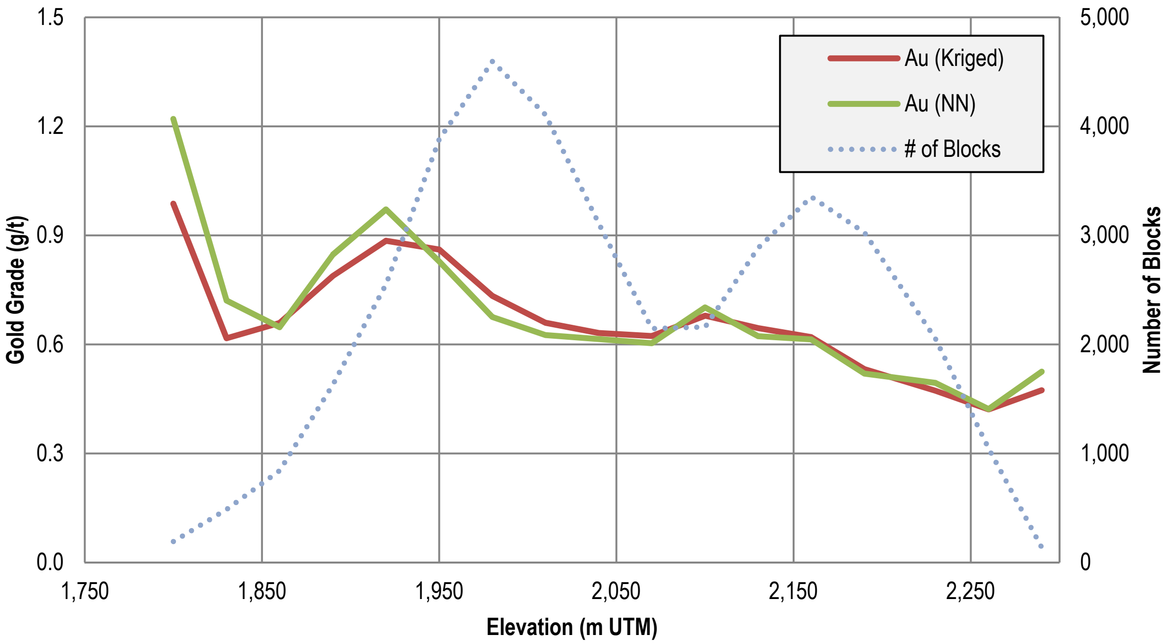


Figure 14.10: Elevation Validation Gold Swath Plot for West End



14.3.10 West End Mineral Resource Classification

Confidence criteria used to guide mineral resource classification include search and composite selection, estimation variance and single-block Kriged results. Blocks eligible for indicated classification are those estimated in the first estimation pass, or those estimated in the second pass by more than 5 composites from two or more drill holes with a Kriging variance <0.45. Distance is not a criteria, but indicated blocks estimated in the second pass generally have an average distance to samples of <50 m and at least one composite within 50 m anisotropic distance. Blocks in Domain 4 are not eligible for indicated classification. Single block Kriged results indicate that the 25 to 70 m drill spacing is sufficient for estimation of quarterly production grade with 85% confidence, suitable for classification of indicated mineral resources. Final classification was applied following manual smoothing of the results on 6 m plan sections to produce a model with reasonably contiguous zones of inferred and indicated blocks.

14.4 YELLOW PINE

14.4.1 Mineral Resource Estimation Procedures

The Yellow Pine mineral resource estimate is based on the validated and verified drill hole database, digitized as-built data of historic workings, interpreted fault structures, sulfide mineralization, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software Geovia GEMS™ Version 6.6; geostatistics and semi-variogram analyses were completed using Snowden Supervisor™ Version 8.2 software.

14.4.2 Drill Hole Database

The drill hole database, supplied by Midas Gold in Excel format, contained collar locations surveyed in UTM grid coordinates, drill hole orientations with downhole surveys, assay intervals with gold, antimony, and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types and specific gravity measurements.

The Yellow Pine deposit was previously in production in the 1930s - 1950s from the so-called glory hole area, while the Homestake area was in production in the 1980s; the entire Yellow Pine area was explored for gold and antimony by numerous operators, up to and including Midas Gold in 2013. The drill hole database contains data for 1,004 separate drill holes representing a mixture of pre-1953 and modern drilling programs. Historical data (i.e. pre-Midas Gold) accounts for approximately 49% of the drill hole database by meterage, as previously described (Section 10). Multiple statistical validations were completed to assess the quality of the historical drill hole data, as discussed in Section 12. A significant number of historic holes were removed from the dataset used for estimation including holes missing critical supporting information, holes with long downhole composited assays, air-track drill holes and all historic pre-1953 drill holes in the northeast portion of the deposit (which is often referred to as the Homestake domain).

For the Yellow Pine deposit, antimony and silver mineral resources were calculated in addition to gold. Table 14.19 shows the number of drill holes and assay intervals utilized in the estimate, which illustrates that the metal values for gold, antimony, and silver were not consistently analyzed for all sample intervals throughout the various historic drilling campaigns. While there are some areas of oxidation within the Yellow Pine deposit, all mineralization is treated as sulfide mineralization with respect to the estimation.

Table 14.19: Drill Hole Information used in the Yellow Pine Mineral Resource Estimate

Company	Gold			Silver			Antimony		
	# Holes	# Samples	Meters	# Holes	# Samples	Meters	# Holes	# Samples	Meters
Barrick	17	2,528	3,909	17	2,528	3,909	17	2,528	3,909
Bradley	109	4,256	6,796	109	4,256	6,796	0	0	0
El Paso	1	60	122	1	60	122	1	60	122
Hecla	67	2,348	3,723	67	2,348	3,723	0	0	0
Midas Gold	226	23,271	43,190	226	23,271	43,190	226	23,271	43,190
Pioneer	1	76	116	1	76	116	1	76	116
Ranchers	145	4,713	7,542	145	4,713	7,542	145	4,713	7,542
Superior	16	393	595	16	393	595	16	393	595
USBM	51	2,828	4,421	51	2,828	4,421	51	2,828	4,421
All	633	40,473	70,414	633	40,473	70,414	457	33,869	59,895

Note: Drill hole information includes un-assayed intervals, and excludes samples in overburden.

The most common assay lengths are approximately 1.5 m long with the majority of assays between 0.8 m and 2.5 m in length. The drill hole database contains 1,762 specific gravity measurements, collected on core samples using a water immersion method and verified with independent, third party laboratory measurements.

14.4.3 Geologic Modeling

The Yellow Pine mineral resource estimate is based on a generalized geologic model consisting of major rock types, major structures, surfaces, historic underground workings and grade shells for gold, antimony and silver (as shown on Figure 14.11). Intrusive rocks types in the geologic model include the primary host rocks, i.e. quartz-monzonite and granite, which are cut by late-stage diabase and latite dikes. As discussed in Section 7, mineralization in the Yellow Pine deposit is structurally controlled and localized by the MCFZ, a generally north to northeast striking, steeply west-northwest dipping fault zone, and 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 and adjacent to the MCFZ, and east of the Hanging Wall Fault (**HWF**). In the geologic model, the MCFZ and HFZ are modeled as structural corridors containing a variety of fault related rock types including breccia, gouge, cataclasite and rubble zones. To the east of the MCFZ are metasediments of the Stibnite roof pendant, which are not sub-divided in the geologic model. 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 Yellow Pine and Homestake pit bottoms, prior to backfilling. Drill data used to construct the top of bedrock surface includes holes drilled from barges through the pit lake by the Rancher's Exploration Company (Ranchers).

14.4.4 Estimation Domain Modeling

The mineral resource estimate for the Yellow Pine deposit utilized grade shells for gold, antimony and silver in five estimation domains. The gold grade shell was constructed manually using a 0.25 g/t grade threshold. Contouring was controlled by grade values and the geologic and structural trends, particularly the MCFZ, HFZ and HWF. Grade shell construction was limited to no more than 60 m beyond any mineralized drill hole intercept. During interpolation the gold shell served as a hard boundary. The grade shells for antimony and silver were constructed using a similar procedure to the gold grade shell using thresholds of 0.1% Sb and 10 g/t Ag. Both antimony and silver shells are located entirely within the gold grade shell and served as hard boundaries for estimation.

The deposit was divided into five estimation domains to segregate regions with different structural controls on gold mineralization, as indicated by oriented core structural measurements, Midas Gold grade contouring, the geologic model, historical underground mapping and historical reports. Estimation domains are shown on Figure 14.12 and subdivide the deposit into the southern, central and northern regions which show progressively more shallow dipping controls on mineralization, and segregate the MCFZ, HFZ and meta-sedimentary units. The boundaries between the estimation domains were treated as soft boundaries during estimation. Descriptive statistics for raw assays within the gold grade shell are shown in Table 14.20.

Figure 14.11: Geologic Model for Yellow Pine

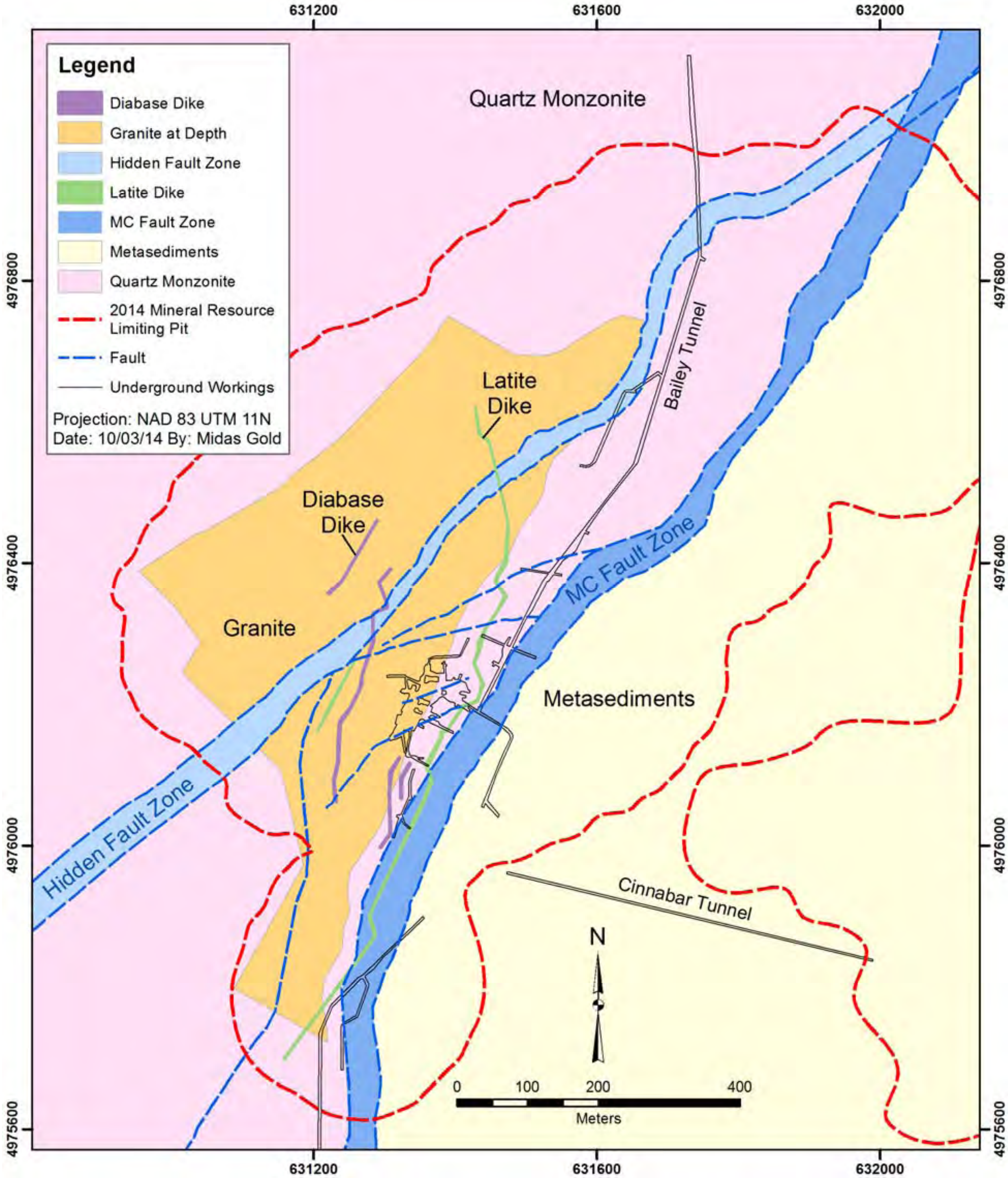


Figure 14.12: Estimation Domains and Grade Shells for Yellow Pine

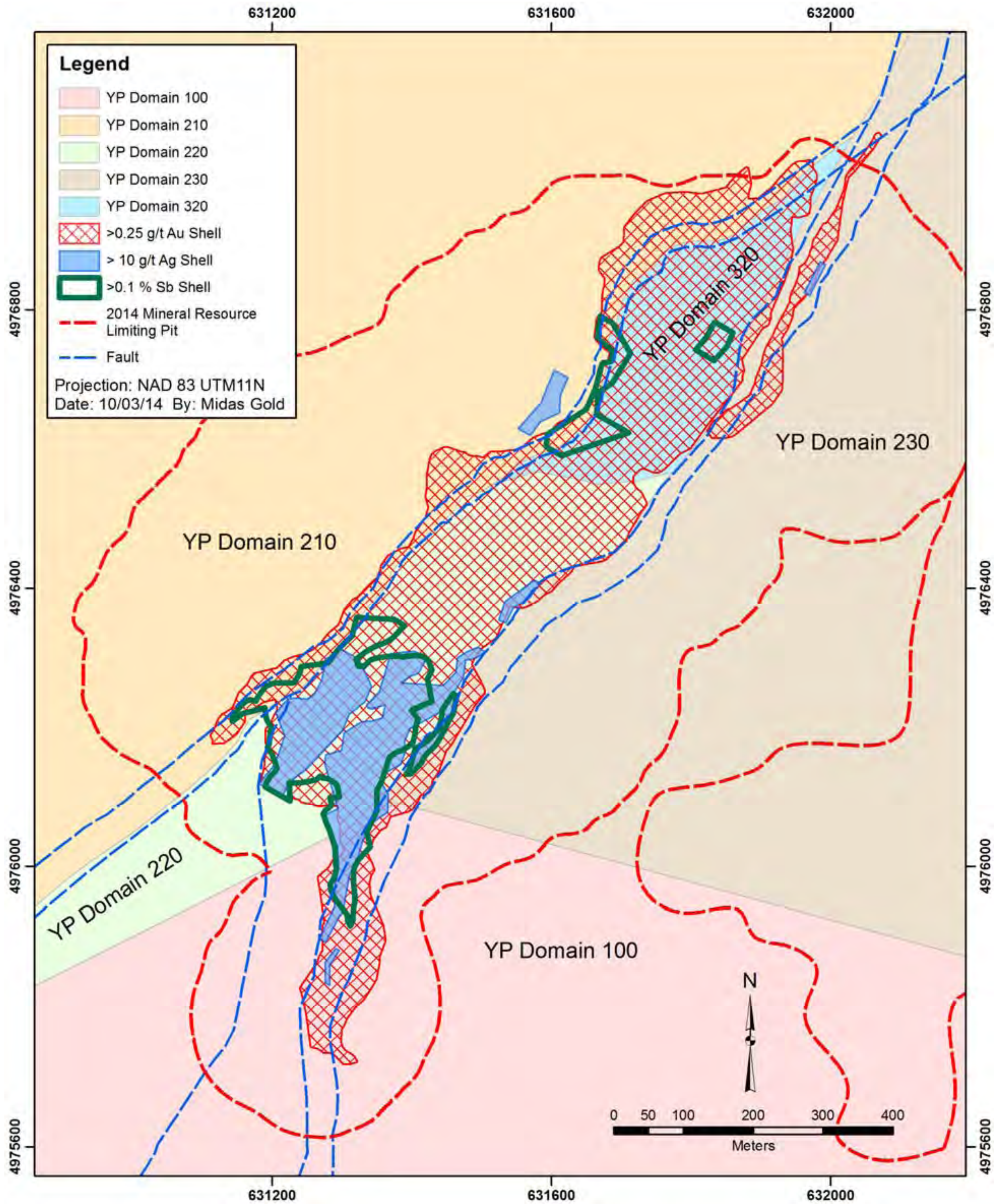


Table 14.20: Descriptive Statistics for Raw Assays for Yellow Pine

Metal	Au Fire Assay (g/t)						
Shell	Inside 0.25 g/t Au						Outside Au
Estimation Domain	All	100	210	220	230	320	All
Mean	1.725	0.976	1.130	2.154	0.872	1.560	0.076
Standard Error	0.014	0.037	0.030	0.019	0.029	0.037	0.002
Median	1.029	0.369	0.489	1.714	0.471	0.686	0.027
Standard Deviation	2.127	1.278	1.674	2.214	1.455	2.353	0.228
Minimum	0.003	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	36.00	8.057	22.29	36.00	28.20	30.62	12.48
Count	23,868	1,208	3,199	13,007	2,498	3,956	16,012
CV	1.23	1.31	1.48	1.03	1.67	1.51	2.99
Metal	Ag (g/t)			Sb (%)			
Shell	10 g/t Ag	Inside Au	Outside Au	0.1 % Sb	Inside Au	Outside Au	
Mean	22.05	2.066	0.414	0.768	0.022	0.003	
Standard Error	1.083	0.038	0.010	0.025	0.001	0.000	
Median	10.97	1.030	0.250	0.290	0.003	0.001	
Standard Deviation	43.01	4.523	1.197	1.454	0.126	0.024	
Minimum	0.017	0.017	0.017	0.001	0.000	0.000	
Maximum	975.0	222.0	111.0	24.10	4.420	1.800	
Count	1,578	14,496	14,198	3,380	12,930	14,075	
CV	1.95	2.19	2.89	1.89	5.77	7.24	
Note: Assay information excludes un-assayed samples and samples within overburden.							

14.4.5 Compositing

Gold, antimony and silver were composited downhole on 3 m intervals within the gold grade shell. Composites at the end of drill holes that were <0.6 m in length were removed from the final data set. Prior to compositing, gold grade was checked relative to sample length and no correlation was found, indicating that capping of outliers can be applied to composites. Composites generated from missing assay data were removed from the data set discussed in the following sections and used for estimation. Descriptive statistics for raw composites are shown in Table 14.21.

Table 14.21: Yellow Pine Raw Composite Statistics by Estimation Domain

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Uncapped Clustered Composites (in g/t Au) Outside 0.25 g/t Gold Shell (AuCode_990)						
Mean	0.073	0.069	0.079	0.059	0.074	0.083
Standard Error	0.002	0.005	0.004	0.002	0.003	0.006
Median	0.030	0.033	0.030	0.023	0.027	0.034
Standard Deviation	0.176	0.132	0.224	0.096	0.144	0.233
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	6.527	1.638	5.314	1.405	1.964	6.527
Count	9,518	590	2,567	2,026	2,915	1,420
Coefficient of Variation	2.41	1.90	2.83	1.63	1.95	2.81

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Uncapped Clustered Composites (in g/t Au) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)						
Mean	1.687	1.001	1.102	2.125	0.860	1.578
Standard Error	0.016	0.046	0.034	0.023	0.032	0.045
Median	1.092	0.477	0.539	1.795	0.509	0.850
Standard Deviation	1.847	1.194	1.431	1.916	1.286	2.025
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	23.48	6.984	12.47	21.13	21.45	23.48
Count	13,087	670	1,796	6,967	1,589	2,065
Coefficient of Variation	1.10	1.19	1.30	0.90	1.49	1.28
Uncapped Clustered Composites (in g/t Ag) Outside 10 g/t Silver Shell (AgCode_1000)						
Mean	0.411	0.456	0.340	0.432	0.421	0.473
Standard Error	0.013	0.042	0.008	0.015	0.037	0.020
Median	0.250	0.250	0.250	0.250	0.250	0.250
Standard Deviation	1.196	0.890	0.358	0.645	1.935	0.688
Minimum	0.080	0.170	0.080	0.170	0.080	0.080
Maximum	89.96	11.27	6.609	7.499	89.96	7.799
Count	8,514	459	2,275	1,848	2,719	1,213
Coefficient of Variation	2.91	1.95	1.05	1.49	4.60	1.46
Uncapped Clustered Composites (in g/t Ag) Inside 10 g/t Silver Shell (AgCode_3000)						
Mean	21.93	22.81	46.51	19.56	42.57	
Standard Error	1.271	3.076	14.33	0.882	17.10	
Median	12.26	13.47	25.23	11.93	12.31	
Standard Deviation	37.08	24.99	79.80	23.52	110.8	
Minimum	0.170	0.382	3.262	0.170	0.170	
Maximum	712.6	146.8	348.4	197.3	712.6	
Count	851	66	31	712	42	0
Coefficient of Variation	1.69	1.10	1.72	1.20	2.60	
Uncapped Clustered Composites (in % Sb) Outside 0.1% Antimony Shell (SbCode_1000)						
Mean	0.019	0.044	0.019	0.025	0.007	0.005
Standard Error	0.001	0.006	0.003	0.002	0.001	0.000
Median	0.003	0.011	0.003	0.003	0.003	0.003
Standard Deviation	0.097	0.094	0.117	0.112	0.023	0.010
Minimum	0.000	0.001	0.000	0.000	0.000	0.000
Maximum	2.558	0.926	2.558	2.148	0.380	0.170
Count	7,249	269	1,285	3,744	1,358	593
Coefficient of Variation	5.00	2.14	6.19	4.59	3.03	1.98
Uncapped Clustered Composites (in % Sb) Inside 0.1% Antimony Shell (SbCode_2000)						
Mean	0.626	0.626	0.884	0.564	1.247	0.543
Standard Error	0.030	0.081	0.145	0.030	0.350	0.097
Median	0.267	0.254	0.376	0.264	0.253	0.349
Standard Deviation	1.099	0.985	1.770	0.912	2.098	0.563
Minimum	0.001	0.003	0.005	0.001	0.017	0.005
Maximum	16.96	5.442	16.96	11.72	8.462	2.310
Count	1,310	147	149	944	36	34
Coefficient of Variation	1.76	1.57	2.00	1.62	1.68	1.04

14.4.6 Evaluation of Outliers

To mitigate estimation risk associated with use of high-grade statistical outliers, capping grades were determined for each estimation domain based on inflection points on log-probability plots of the raw 3 m composites. Capping grades are shown in Table 14.22 as well as percentage of metal removed based on relative change in declustered average grade. Capped composite statistics are shown in Table 14.23. Low capping grades for antimony outside the antimony shell are warranted due to low average grade and small relative standard deviation. In general, the estimation domains and mineralized shells adequately subdivide samples into regions with distinct populations and coefficients of variation acceptably low for geostatistical estimation.

Table 14.22: Capping Grades for 3 m Composites

Statistic	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Au (g/t) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)					
Ndat	670	1796	6967	1589	2064
Maximum Value	6.984	12.465	21.125	21.451	23.484
Cap Value	N/A	10	10	10	10
Number Capped	0	3	44	4	14
Mean Uncapped	0.845	0.944	1.403	0.860	1.301
Mean Capped	0.845	0.943	1.395	0.840	1.288
Lost Metal (%)	0	-0.1	-0.6	-2.3	-1
Ag (g/t) Inside 10 g/t Silver Shell (AgCode_3000)					
Ndat	66	31	712	42	0
Maximum Value	146.8	348.4	197.3	712.6	
Cap Value	80	50	120	100	
Number Capped	1	4	6	3	
Mean Uncapped	22.81	46.51	19.56	42.57	
Mean Capped	21.80	23.99	19.19	26.36	
Lost Metal (%)	-4.4	-48.4	-1.9	-38.1	
Ag (g/t) Outside 10 g/t Silver Shell (AgCode_1000)					
Ndat	382	1,454	3,722	1,360	1,296
Maximum Value	31.02	59.75	70.21	42.45	12.83
Cap Value	20	25	25	15	10
Number Capped	3	5	17	5	5
Mean Uncapped	1.937	1.850	2.478	1.539	1.517
Mean Capped	1.899	1.798	2.404	1.507	1.514
Lost Metal (%)	-1.9	-2.8	-3.0	-2.1	-0.2
Sb % Inside 0.1% Antimony Shell (SbCode_2000)					
Ndat	147	149	944	36	34
Maximum Value	5.442	16.962	11.72	8.462	2.31
Cap Value	N/A	6	9	5	1
Number Capped	0	2	1	4	4
Mean Uncapped	0.626	0.884	0.564	1.247	0.543
Mean Capped	0.626	0.788	0.561	1.087	0.445
Lost Metal (%)	0.0	-10.8	-0.5	-12.8	-18.1
Sb % Outside 0.1% Antimony Shell (SbCode_1000)					
Ndat	269	1,285	3,744	1,358	593
Maximum Value	0.926	2.558	2.148	0.38	0.17
Cap Value	N/A	0.5	0.5	0.3	N/A
Number Capped	0	11	40	2	0
Mean Uncapped	0.044	0.019	0.025	0.007	0.005
Mean Capped	0.044	0.014	0.020	0.007	0.005
Lost Metal (%)	0.0	-25.9	-18.0	-1.4	0.0

Table 14.23: Capped Clustered Composite Statistics for Yellow Pine

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Capped Clustered Composites (in g/t Au) Outside 0.25 g/t Gold Shell (AuCode_990)						
Mean	0.073	0.069	0.079	0.059	0.074	0.083
Standard Error	0.002	0.005	0.004	0.002	0.003	0.006
Median	0.030	0.033	0.030	0.023	0.027	0.034
Standard Deviation	0.176	0.132	0.224	0.096	0.144	0.233
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	6.527	1.638	5.314	1.405	1.964	6.527
Count	9,518	590	2,567	2,026	2,915	1,420
Coefficient of Variation	2.41	1.90	2.83	1.63	1.95	2.81
Capped Clustered Composites (in g/t Au) Inside 0.25 g/t Gold Shell (AuCode_1000, 1100)						
Mean	1.671	1.001	1.098	2.107	0.840	1.558
Standard Error	0.015	0.046	0.033	0.022	0.028	0.042
Median	1.092	0.477	0.539	1.795	0.509	0.850
Standard Deviation	1.751	1.194	1.408	1.812	1.111	1.897
Minimum	0.003	0.003	0.003	0.003	0.003	0.003
Maximum	10.000	6.984	10.000	10.000	10.000	10.000
Count	13,087	670	1,796	6,967	1,589	2,065
Coefficient of Variation	1.05	1.19	1.28	0.86	1.32	1.22
Capped Clustered Composites (in g/t Ag) Outside 10 g/t Silver Shell (AgCode_1000)						
Mean	0.397	0.443	0.340	0.432	0.380	0.473
Standard Error	0.007	0.036	0.008	0.015	0.013	0.020
Median	0.250	0.250	0.250	0.250	0.250	0.250
Standard Deviation	0.607	0.761	0.358	0.645	0.669	0.688
Minimum	0.080	0.170	0.080	0.170	0.080	0.080
Maximum	10.000	7.000	6.610	7.500	10.000	7.800
Count	8,514	459	2,275	1,848	2,719	1,213
Coefficient of Variation	1.53	1.72	1.05	1.49	1.76	1.46
Capped Clustered Composites (in g/t Ag) Inside 10 g/t Silver Shell (AgCode_3000)						
Mean	19.92	21.80	23.99	19.19	26.36	
Standard Error	0.743	2.574	2.718	0.801	4.584	
Median	12.25	13.47	25.23	11.94	12.31	
Standard Deviation	21.67	20.91	15.14	21.38	29.71	
Minimum	0.170	0.380	3.260	0.170	0.170	
Maximum	120.0	80.00	50.00	120.0	100.0	
Count	851	66	31	712	42	0
Coefficient of Variation	1.09	0.96	0.63	1.11	1.13	

Statistic	All Domains	Domain 100	Domain 210	Domain 220	Domain 230	Domain 320
Capped Clustered Composites (in % Sb) Outside 0.1% Antimony Shell (SbCode_1000)						
Mean	0.017	0.043	0.016	0.022	0.006	0.005
Standard Error	0.001	0.006	0.002	0.002	0.001	0.000
Median	0.003	0.010	0.003	0.002	0.003	0.003
Standard Deviation	0.088	0.094	0.079	0.109	0.021	0.010
Minimum	0.000	0.001	0.000	0.000	0.000	0.000
Maximum	2.148	0.926	1.344	2.148	0.380	0.170
Count	7,249	269	1,285	3,744	1,358	593
Coefficient of Variation	5.15	2.18	4.99	5.06	3.48	1.98
Capped Clustered Composites (in % Sb) Inside 0.1% Antimony Shell (SbCode_2000)						
Mean	0.596	0.626	0.762	0.555	1.047	0.406
Standard Error	0.026	0.081	0.088	0.029	0.278	0.049
Median	0.265	0.254	0.376	0.262	0.253	0.349
Standard Deviation	0.943	0.985	1.079	0.883	1.670	0.288
Minimum	0.001	0.003	0.005	0.001	0.001	0.005
Maximum	9.000	5.442	6.000	9.000	5.000	1.000
Count	1,310	147	149	944	36	34
Coefficient of Variation	1.58	1.57	1.42	1.59	1.59	0.71

14.4.7 Statistical Analysis and Spatial Correlation

Exponential and spherical semi-variogram models were generated for gold, antimony and silver to guide the search ellipses and establish spatial correlation and sample weighting for the estimate. The nugget effect was derived using down-hole variograms. Semi-variogram models (Table 14.24) were based on experimental log variograms generated for each estimation domain within the grade shells using Snowden Supervisor™ software. Variography for gold and antimony indicates good spatial correlation along perceived structural trends such as major fault corridors and northeast striking splay structures to the MCFZ, with many directions of maximum continuity oriented sub-parallel to the intersection of northwest dipping and north-south striking faults.

Table 14.24: Semi-Variogram Models for Yellow Pine

Metal	Domain	Gemcom ZXZ Rotations(1)			Nugget C0	Sill C1 and C2	Ranges a1, a2 (m)			Model Structure Type
		Around Z	Around X	Around Z			SM X-ROT	MJ Y-ROT	MN Z-ROT	
Au	100	68	68	-106	0.05	0.17	4	4	14	Exp.
						0.94	69	50	22	Exp.
	210	62	-62	42	0.05	0.21	43	32	15	Spherical
						0.74	70	45	20	Spherical
	220	40	-85	-95	0.1	0.07	26	10	14	Exp.
						0.58	115	60	36	Exp.
	230	-128	75	-112	0.05	0.08	68	40	11	Exp.
						0.94	90	80	25	Exp.
	320	-135	53	-105	0.1	0.62	47	50	24	Exp.
						0.12	90	70	40	Exp.

Metal	Domain	Gemcom ZXZ Rotations[1]			Nugget C0	Sill C1 and C2	Ranges a1, a2 (m)			Model Structure Type
		Around Z	Around X	Around Z			SM X-ROT	MJ Y-ROT	MN Z-ROT	
Ag	100	82	83	-105	0.05	0.31	37	37	18	Exp.
						0.37	100	75	35	Exp.
	210	45	-59	8	0.06	0.37	96	10	12	Exp.
						0.49	120	80	25	Exp.
	220	70	-80	95	0.03	0.21	7	7	9	Exp.
						0.63	115	80	44	Exp.
	230	60	-82	15	0.07	0.45	18	18	9	Exp.
						0.71	105	80	19	Exp.
	320	31	-52	134	0.06	0.51	50	46	19	Exp.
						0.38	95	82	35	Exp.
Sb	100	75	-69	160	0.1	0.1	37	22	16	Exp.
						0.97	75	46	35	Exp.
	210	60	-75	67	0.05	0.26	9	9	9	Exp.
						0.48	95	65	25	Exp.
	220	60	-60	165	0.1	0.32	42	43	11	Exp.
						0.37	90	90	20	Exp.
	230	60	-75	135	0.03	0.35	43	30	12	Exp.
						0.42	80	55	22	Exp.
	320	30	-30	90	0.03	0.07	85	25	8	Spherical
						0.74	101	65	28	Spherical

Notes:
(1) Defined as positive rotation (right hand) of X around Z axis towards Y, rotation around newly created Z and Y axes around X, followed by rotation around newly created Z axis.

14.4.8 Block Model Parameters and Grade Estimation

The block model mineral resource estimate for Yellow Pine was developed with block dimensions of 12.19 x 12.19 x 6.096 m (40 x 40 x 20 ft) with coordinates defined in Table 14.25. Blocks were discretized into a 3 x 3 x 3 array of points during estimation.

Table 14.25: Block Model Definition for Yellow Pine

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Yellow Pine	12.192	12.192	6.096	630,686.096	4,795,346.096	2,295.152	151	161	152	0

Notes: Block centroid, NAD83 Zone 11N Datum

Density was estimated within grade shells in a single pass using inverse distance squared weighting with search ellipse orientations based on the gold variograms models. Un-estimated blocks were assigned the average density for the encompassing estimation domain or lithology solid.

The Yellow Pine mineral resource estimate was completed for gold, antimony and silver using the estimation domains and grade shells discussed above. Within the grade shells, blocks were estimated by ordinary Kriging using the capped 3 m composite file and the semi-variogram models discussed above. Grade shells were treated as hard boundaries and structural domains treated as soft boundaries for sample selection during estimation. Gold and antimony were estimated within and outside the grade shells, but only material within the shells is eligible for indicated classification. Table 14.26 summarizes the estimation parameters for Yellow Pine.

Table 14.26: Summary of Estimation Parameters for Yellow Pine

Au 1000 and 1100 Grade Shell																
Domain		100	100	100	210	210	210	220	220	220	230	230	230	320	320	320
Search Pass		1	2	3	1	2	3	1	2	3	1	2	3	1	2	3
Gemcom ZXZ Rotations	Around Z	68	68	68	62	62	62	40	40	40	-128	-128	-128	-135	-135	-135
	Around X	68	68	68	-62	-62	-62	-85	-85	-85	75	75	75	53	53	53
	Around Z	-106	-106	-106	42	42	42	-95	-95	-95	-112	-112	-112	-105	-105	-105
Search Ellipse Radius	x (m)	60	60	120	60	60	120	60	60	120	60	60	120	60	60	120
	y (m)	45	45	90	45	45	80	45	45	90	45	45	110	45	45	120
	z (m)	20	20	30	15	15	20	25	25	35	15	20	20	25	25	40
High Grade Search Limit	x (m)	45	45	45	45	45	45				45	45	45			
	y (m)	45	45	45	45	45	45				45	45	45			
	z (m)	20	20	20	15	15	15				15	15	15			
	Limit Value	5	5	5	10	10	10				10	10	10			
No of Samples	Min	4	6	4	4	6	4	4	6	4	4	6	4	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
No of Holes Required		1	2	1	1	2	1	1	2	1	1	2	1	1	2	1
Search Type		Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal	Ellipsoidal	Octant	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Ag 3000 Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	82	82	45	45	70	70	60	60	31	31
	Around X	83	83	-59	-59	-80	-80	-82	-82	-52	-52
	Around Z	-105	-105	8	8	95	95	15	15	20	20
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	90	45	80	45	80	45	90	45	100
	z (m)	20	30	15	20	20	40	15	20	20	30
High Grade Search Limit	x (m)					45	45				
	y (m)					45	45				
	z (m)					15	15				
	Limit Value					120.0	120.0				
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Ag Within 1000 Au Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	82	82	45	45	70	70	60	60	31	31
	Around X	83	83	-59	-59	-80	-80	-82	-82	-52	-52
	Around Z	-105	-105	8	8	95	95	15	15	20	20
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	90	45	80	45	80	45	90	45	100
	z (m)	20	30	15	20	20	40	15	20	20	30
High Grade Search Limit	x (m)					45	45				
	y (m)					45	45				
	z (m)					15	15				
	Limit Value					120.0	120.0				
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Sb 2000 Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	75	75	60	60	60	60	60	60	30	30
	Around X	-69	-69	-75	-75	-60	-60	-75	-75	-30	-30
	Around Z	160	160	67	67	165	165	135	135	90	90
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	80	45	80	45	90	45	80	45	80
	z (m)	20	30	15	20	20	30	15	20	20	30
High Grade Search Limit	x (m)			45	45						
	y (m)			45	45						
	z (m)			15	15						
	Limit Value			6	6						
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

Sb Within 1000 Au Grade Shell											
Domain		100	100	210	210	220	220	230	230	320	320
Search Pass		1	2	1	2	1	2	1	2	1	2
Gemcom ZXZ Rotations	Around Z	75	75	60	60	60	60	60	60	30	30
	Around X	-69	-69	-75	-75	-60	-60	-75	-75	-30	-30
	Around Z	160	160	67	67	165	165	135	135	90	90
Search Ellipse Radius	x (m)	60	120	60	120	60	120	60	120	60	120
	y (m)	45	80	45	80	45	90	45	80	45	80
	z (m)	20	30	15	20	20	30	15	20	20	30
High Grade Search Limit	x (m)	25	25	25	25	25	25	25	25	25	25
	y (m)	25	25	25	25	25	25	25	25	25	25
	z (m)	10	10	10	10	10	10	10	10	10	10
	Limit Value	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
No of Samples	Min	6	4	6	4	6	4	6	4	6	4
	Max	16	16	16	16	16	16	16	16	16	16
Max Samples	Per Hole	4	4	4	4	4	4	4	4	4	4
No of Holes Required		2	1	2	1	2	1	2	1	2	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

		SG Ore (Within 1000 Au Grade Shell)					SG Waste (Outside the Au Grade Shell)				
Domain		100	210	220	230	320	100	210	220	230	320
Search Pass		1	2	3	2	3	1	2	3	2	3
Gemcom ZXZ Rotations	Around Z	68	62	40	-128	-135	68	62	40	-128	-135
	Around X	68	-62	-85	75	53	68	-62	-85	75	53
	Around Z	-106	42	-95	-112	-105	-106	42	-95	-112	-105
Search Ellipse Radius	x (m)	120	120	120	120	120	120	120	120	120	120
	y (m)	120	120	120	120	120	120	120	120	120	120
	z (m)	50	50	50	50	50	50	50	50	50	50
No of Samples	Min	2	2	2	2	2	2	2	2	2	2
	Max	5	5	5	5	5	5	5	5	5	5
Max Samples	Per Hole										
No of Holes Required		1	1	1	1	1	1	1	1	1	1
Search Type		Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal	Ellipsoidal
Method		ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2	ID2

The gold estimation strategy was designed to limit the influence of historical underground holes drilled by Bradley, which show an apparent high bias presumably associated with their location in the highest-grade portions of the deposit, as discussed in Section 12. The first pass only estimated blocks within 12 m of Bradley underground drill holes using all composites. In the second pass, search ellipsoid radii were adjusted to represent half the variograms continuity and composite samples from the underground Bradley drill holes were removed from the sample set. The third pass was adjusted as needed to estimate any remaining blocks. Antimony was estimated with two passes of which the first pass search ellipsoid radii represent approximately half of the variogram range of continuity. The second pass was designed to estimate any remaining blocks. Only composite samples from the Midas Gold, Barrick Gold Corporation (**Barrick**), Ranchers and United States Bureau of Mines (**USBM**) drilling campaigns were used to estimate antimony; Bradley samples were excluded due to apparent high bias with respect to antimony grade, even though some of these holes were focused within the highest grades portions of the antimony mineralization.

14.4.9 Block Model Validation

The block model for Yellow Pine was validated by completing a series of graphical inspections, bias checks, sensitivity studies and comparison to prior estimates. Graphically, the model was validated by visually comparing the composites to estimated block grades on plan and section views. Local bias was assessed by comparing the average composite grade against the encompassing block for both gold and antimony and by comparison of the average declustered composite grade and nearest-neighbor estimate to the Kriged estimate on swath plots in the X, Y and Z directions (Figure 14.13, Figure 14.14, and Figure 14.15, respectively). The resultant histograms for gold composites vs. block grades compare well and indicate that the block values are similar to the composite datasets. The histograms for antimony have increased kurtosis indicating a degree of smoothing in the antimony estimate. Model sensitivities were run to assess the impact of historical data on the estimate. Exclusion of the pre-1953 drill hole data results in a 2.4% reduction in average gold grade and an approximate 4% reduction in contained gold at a 0.75 g/t Au cutoff grade, reported within a conceptual pit shell.

Figure 14.13: East-West Validation Gold Swath Plot for Yellow Pine

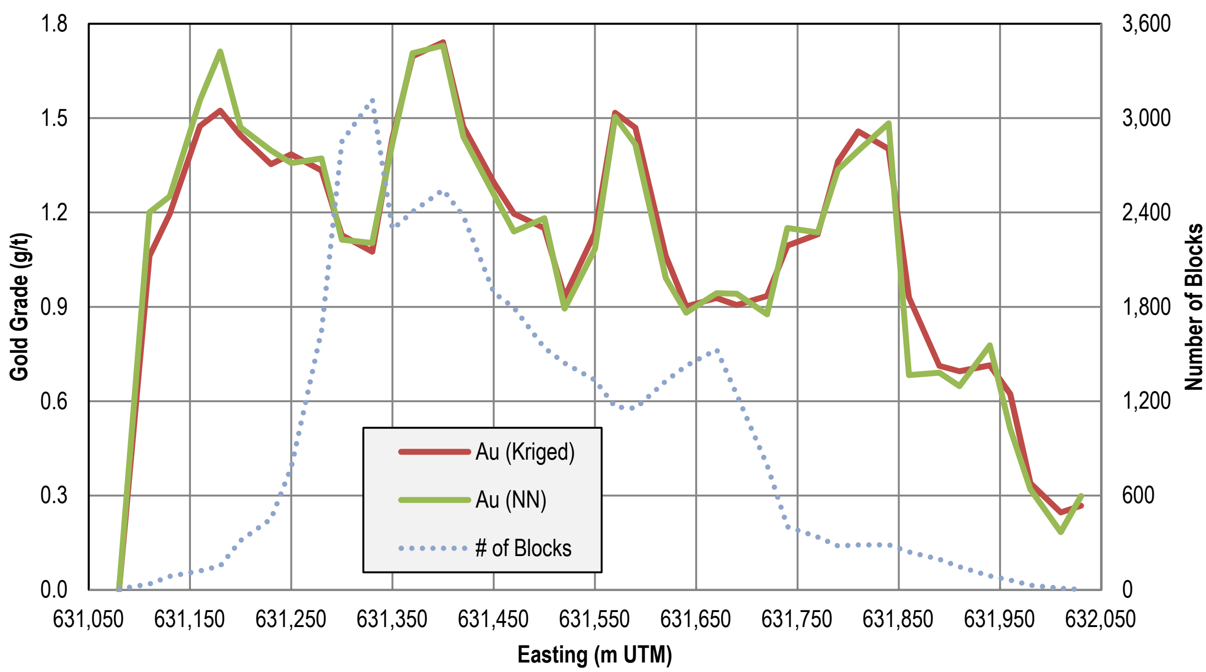


Figure 14.14: North-South Gold Validation Swath Plot for Yellow Pine

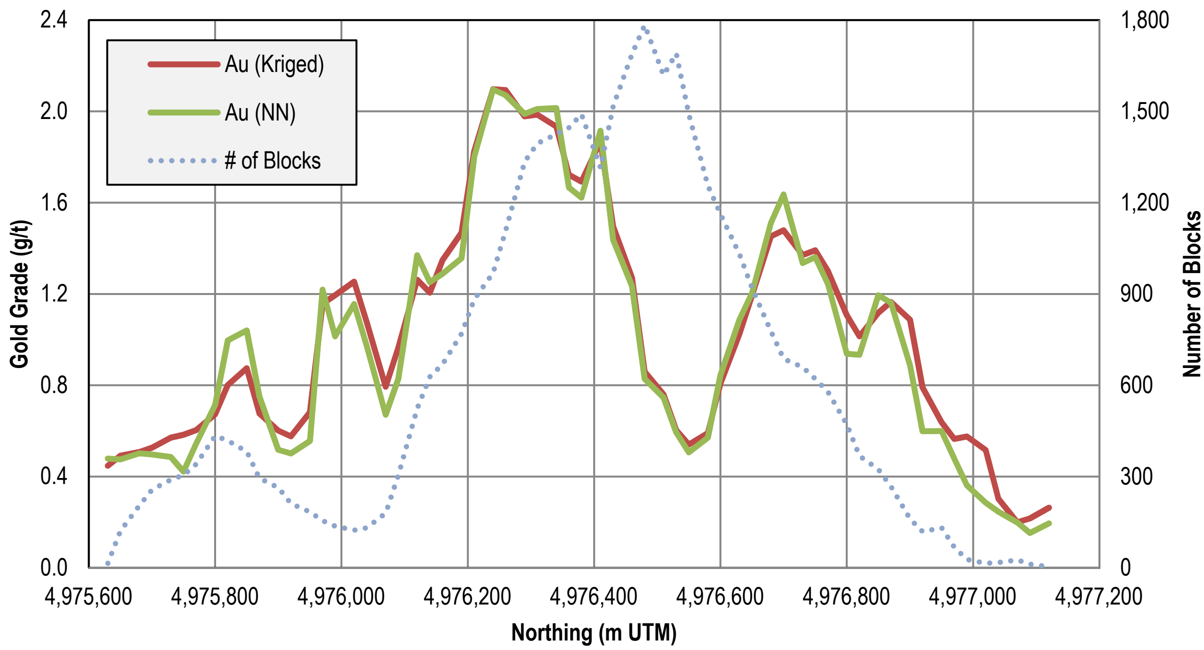
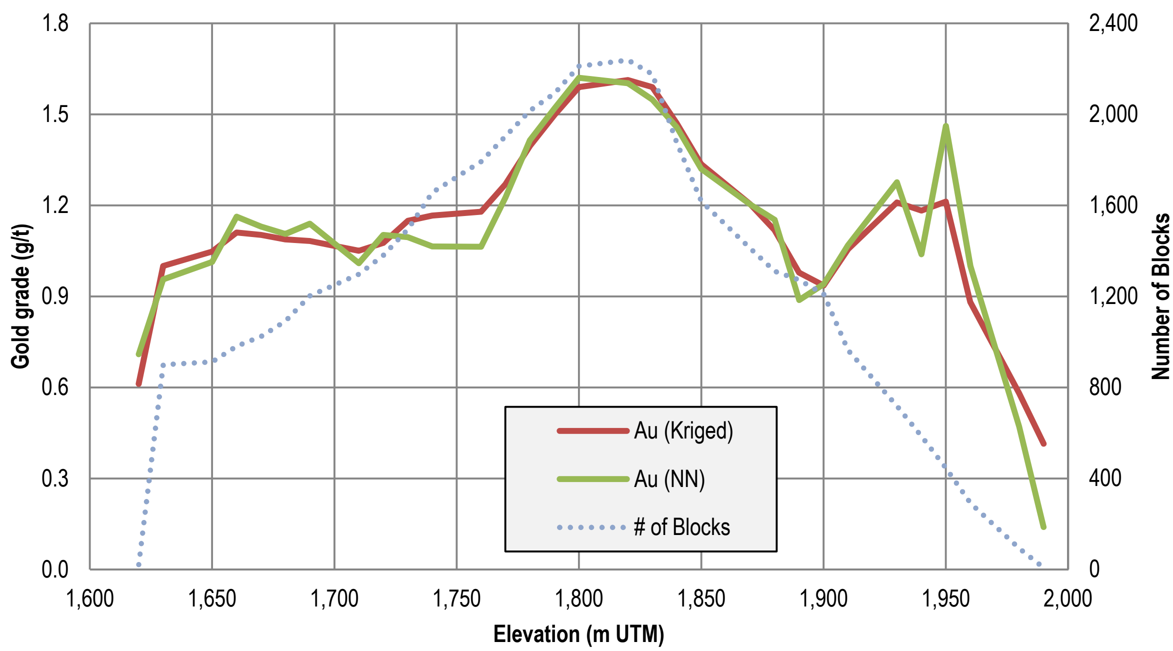


Figure 14.15: Elevation Validation Gold Swath Plot for Yellow Pine



14.4.10 Yellow Pine Mineral Resource Classification

Confidence criteria used to guide mineral resource classification at Yellow Pine included search and composite selection, spatial distribution of samples and well-demonstrated support from “modern” drill data so that grades are not significantly biased by pre-1953 drill holes. This latter requirement ensures that the influence of historic data is partially diminished by the presence of "modern" data in the same area; "modern" data is defined as composites from the post-1980 Midas Gold, Ranchers and Barrick drilling campaigns. Blocks eligible for indicated classification were restricted to those within the gold grade shell and flagged using a separate classification search pass utilizing a 45 x 35 x 25 m search ellipse representing approximately 85% of the modeled variogram sill and requiring at least four samples from two drill holes occurring in at least three octants from "modern" drill data. Final classification was applied following manual smoothing of the results to encompass zones predominantly flagged as indicated. The antimony mineral resource estimates were not classified separately and are instead reported with the gold classification categories.

14.5 HISTORIC TAILINGS

14.5.1 Mineral Resource Estimation Procedures

The Historic Tailings mineral resource estimate is based on the current drill hole database, geologic model of tailings, and LiDAR topographic data. The geologic modeling and estimation of mineral resources was completed using the commercial three-dimensional block modelling and mine planning software packages Geovia GEMS™ 6.6 and Micromine™ version 14; geostatistical analysis was completed using Isaaks & Co.'s SAGE2001™ software package.

14.5.2 Drill Hole Database

The drill hole database, supplied by Midas Gold as an Excel Workbook, contained collar locations surveyed in UTM grid coordinates, assay intervals with gold, antimony, and silver analyses by fire assay and/or cyanide soluble assay, geologic intervals with rock types and in situ density measurements. The database contained data for 73 separate drill holes representing a mixture of historic and modern drilling programs. Some drill holes were not assayed and only used for the establishment the upper and lower boundaries of the tailings and some drill holes did not intercept tailings material. Only Midas Gold drill holes were used in the mineral resource estimate and primarily consist of hollow-stem auger drill holes completed in 2013 with some sonic drill holes completed in 2012. Samples not intersecting tailings material were removed from the data set utilized for estimation.

For the Historic Tailings deposit, antimony and silver mineral resources were calculated in addition to gold. Table 14.27 illustrates that the metal values for gold, antimony, and silver in Midas Gold drill holes were consistently analyzed for all sample intervals throughout the dataset utilized for estimation.

Table 14.27: Drill Hole Data used in the Historic Tailings Mineral Resource Estimate

Element	# Holes	# Assays	Meters
Gold	41	540	339
Antimony	41	540	339
Silver	41	540	339

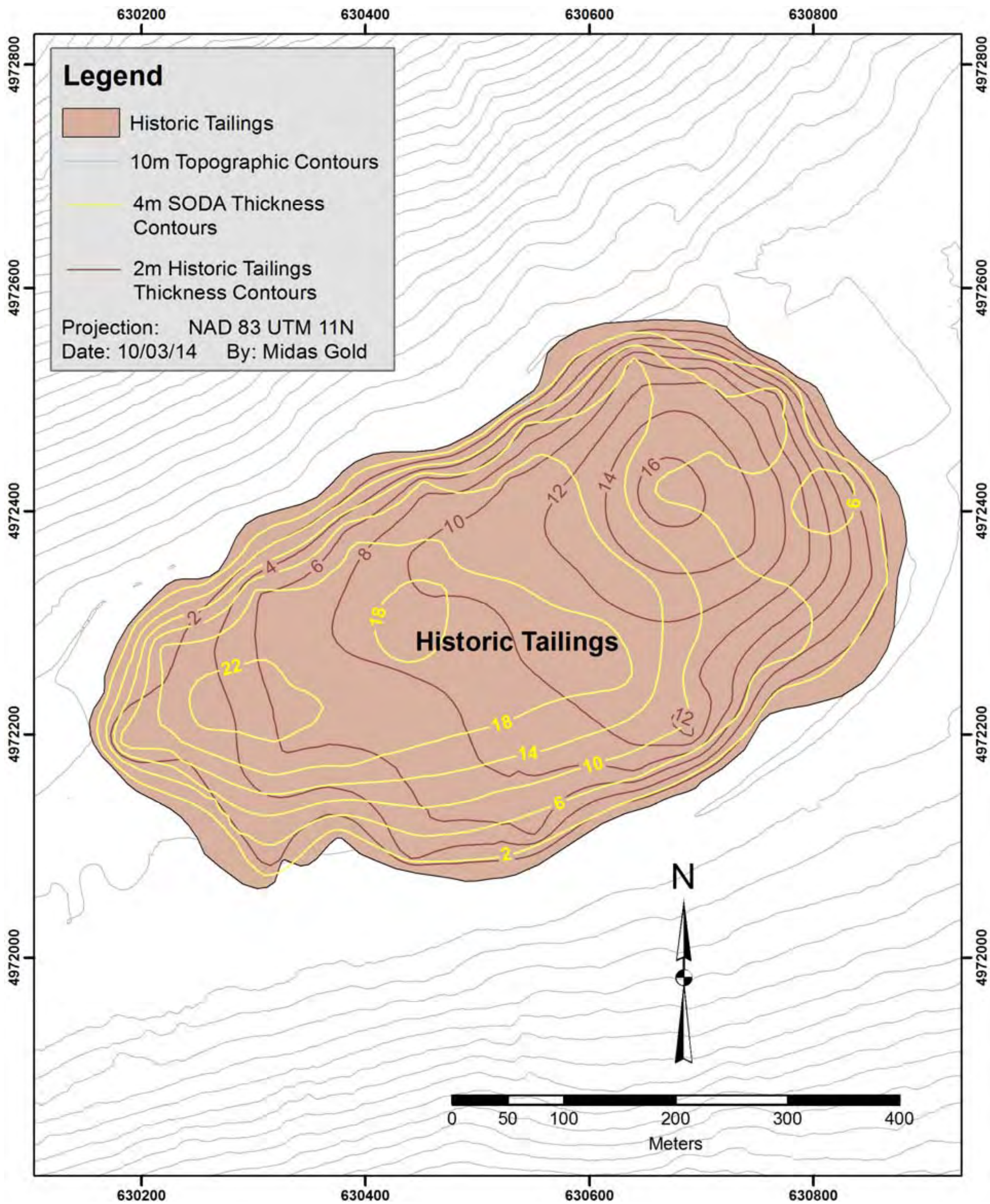
Assays lengths for the auger drill holes were typically 0.61 m (2 ft) which comprises the bulk of the data set. Sonic drill holes present in the deposit were typically assayed on 3.05 m (10 ft) intervals. All drill holes are vertical and the average drill hole spacing is approximately 60 m oriented along a grid rotated to an azimuth of 23 degrees.

14.5.3 Geologic Modeling

The Historic Tailings were hydraulically deposited within the Meadow Creek Valley from the 1920s through the 1950s; the tailings were generated from the Bradley sulfide flotation milling operations. The tailings were later overlain by spent heap leach ore from the 1980s through the 1990s heap leach operations; this area is often referred to as the Spent Ore Disposal Area (SODA). The Historic Tailings deposit is up to 18 m thick, with an average thickness of 6 m; the overlying spent ore material is up to 23 m thick, with an average thickness of 11 m. Historic

Tailings material was wire-framed based on drill hole intercepts, modern LiDAR and orthographic photos, and historic engineering drawings and airborne photos. The total volume of the Historic Tailings wireframe is 1,925,923 m^3.

Figure 14.16: Isopach of the Historic Tailings Deposit



14.5.4 Estimation Domain Modeling

The Historic Tailings solid serves as the only estimation domain utilized in the mineral resource estimate. Descriptive statistics for raw assays within the tailings solid are presented in Table 14.28. The drill holes were drilled on a quasi-regular grid and there is no evidence of data clustering in high- or low-grade areas. Higher-grade material shows northwest trends which shift location vertically, consistent with presumed deposition in tailings beaches during historical operations.

Table 14.28: Raw Assay Statistics for the Historic Tailings

Statistic	Width (m)	Au	Sb	Ag
Count	540	540	540	540
Mean	0.63	1.191	0.173	3.08
Standard Deviation	0.27	0.527	0.117	2.70
Range	2.90	4.636	0.996	41.75
Minimum	0.15	0.054	0.0045	0.25
Lower Quartile	0.61	0.813	0.090	1.80
Median	0.61	1.088	0.169	2.80
Upper Quartile	0.61	1.395	0.237	3.80
Maximum	3.05	4.690	1.000	42.00
Coefficient of Variation	0.42	0.44	0.68	0.88
95% Percentile	0.61	2.211	0.360	5.81
98% Percentile	1.52	2.704	0.440	7.04
99% Percentile	1.83	2.960	0.473	7.50

14.5.5 Compositing

Samples were composited on intervals of 0.61 m and 1.52 m, of which the 0.61 m composites were determined to exhibit a more regular distribution of composite lengths and were selected for estimation. Gold, antimony and silver were composited downhole within the Historic Tailings solid. The composited data yields a histogram with a moderately skewed distribution and very few samples with a grade <0.6 g/t Au.

14.5.6 Evaluation of Outliers

To evaluate potential risk associated with use of high-grade statistical outliers, potential capping grades were assessed using log-probability plots and by analysis of contained metal in deciles and centiles, following the Parrish Method (Parrish, 1997). These results indicate no cap for gold, a cap of 0.6% for antimony and 10 g/t for silver. Descriptive statistics for capped composites are presented in Table 14.29. Note that compositing of the longer sample intervals to 0.61 m yielded more composites than raw assays.

Table 14.29: Historic Tailings Descriptive Statistics for Capped Composites

Statistic	Au (g/t)(1)	Sb_Cap (%)(1)	Ag_Cap (g/t)(1)
Count(1)	568	568	568
Mean	1.183	0.173	2.98
Standard Deviation	0.488	0.104	1.52
Minimum	0.34	0.0045	0.5
Median	1.089	0.167	2.8
Maximum	4.69	0.6	10
Coefficient of Variation	0.412	0.598	0.51
95% Percentile	2.127	0.34	5.8
98% Percentile	2.392	0.414	6.81
99% Percentile	2.701	0.458	7.33

Note:
(1) Units apply to all statistics except "Count".

14.5.7 Statistical Analysis and Spatial Correlation

Correlogram models were developed using the SAGE2001™ software package to guide the search ellipse and establish spatial correlation and sample weighting for the estimate. Correlograms demonstrate low nugget effect with effective ranges close to the drill hole spacing and confirm the northwest anisotropy and stratified nature of the deposit. The correlogram is summarized in Table 14.30.

Table 14.30: Correlogram Models for the Historic Tailings

Metal	Ellipse Axes Azimuth/Plunge(1)			Nugget C0	Sill C1	Ranges a1 (m)			Type
	1st	2nd	3rd			1st	2nd	3rd	
Au	327/2	57/-2	106/87	0.028	0.972	82	50	3	Exp
Sb	336/1	66/-5	76/85	0.02	0.98	133	41	5	Exp
Ag	131/1	41/7	228/83	0.001	0.999	142	63	6	Exp

Note:
(1) Negative plunge is downward.

14.5.8 Block Model Parameters and Grade Estimation

Due to the unconsolidated and stratiform nature of the tailings material, the block model is defined assuming selective mining methods with excellent grade control. Block dimensions are 15.24 x 15.24 x 1.524 m (50 x 50 x 5 ft) with location summarized in Table 14.31. Blocks located partially within the solid were assigned a percent value for reporting purposes.

Table 14.31: Historic Tailings Block Model Definition

Deposit	Dimension (m)			Origin (m)			Number of Blocks			Rotation
	X	Y	Z	X	Y	Z	X	Y	Z	
Historic Tailings	15.24	15.24	1.524	630,007	4,972,007	1,982	68	48	40	0

Notes: Block centroid, NAD83 Zone 11N Datum

The Historic Tailings mineral resource was estimated using ordinary Kriging in a single pass using a search ellipse and sample weighting established by the correlogram models discussed above. Samples were limited to a maximum of 3 per drill hole, increasing the influence of samples from neighboring drill holes. Search ellipse and sample selection parameters are summarized in Table 14.32. Density was estimated from 35 Shelby tube samples of the tailings material; the average dry density of the deposit was calculated to be 1.504 g/cm^3.

Table 14.32: Summary of Estimation Parameters for the Historic Tailings

Description	Au	Sb	Ag
Method	OK	OK	OK
Principal Axis Azimuth / Plunge	325/0	330/0	310/0
Intermediate Axis Azimuth / Plunge	055/0	60/0	40/0
Minor Axis Azimuth / Plunge	0/-90	0/-90	0/-90
Principle Axis Search Distance (m)	175	220	200
Major / Intermediate / Minor Axis	1 / 0.66 / 0.1	1 / 0.5 / 0.1	1 / 0.5 / 0.1
Search Type	Open	Open	Open
Composite Restrictions	Hard	Hard	Hard
Maximum Composites / Sector	N/A	N/A	N/A
Minimum Composites	1	2	3
Minimum # Holes	N/A	N/A	N/A
Maximum Composites / Hole	3	3	3
Maximum Composites	12	12	12

14.5.9 Block Model Validation

The block model for the Historic Tailings was validated by completing a series of graphical inspections, bias checks and reconciliation with historic production records. The block estimates and block percentages were reviewed visually relative to the composite grades and the tailings wireframe. Global bias was assessed by comparison of the Kriged estimate to the nearest neighbor estimate and showed a 3.5% variance. Local bias was assessed by way of a swath plot in the Z direction. Relative to the nearest neighbor estimate, the Kriged estimate displays local low bias on upper level benches, proximal to very high-grade composites in three different drill holes. The results of the Kriged model are generally consistent with estimates of metals reporting to the tailings calculated as the difference between historical mill-feed grade and recovered metal from historical Bradley Mining Company production records.

14.5.10 Historic Tailings Mineral Resource Classification

Confidence criteria used to guide the mineral resource classification includes Kriging variance and anisotropic minimum distance to the nearest composite. Final classification was assigned by digitizing contours around blocks with Kriging variance >0.66 and minimum distance >60 m for the gold estimate, to define areas of inferred classification. The inferred blocks are primarily located on the southern and western margins of the tailings solid where drill data is sparse.

14.6 ECONOMIC CRITERIA AND PIT OPTIMIZATIONS

CIM defines mineral resources as having “reasonable prospects for eventual economic extraction” requiring that mineralization meet certain grade and material volume thresholds sufficient for eventual economic extraction under reasonable production and recovery scenarios at reasonable cutoff grades. Prospects for eventual economic extraction were assessed using an open-pit optimization Lerchs-Grossman algorithm in MineSight® Version 9.00 software. Input parameters were developed from preliminary cost estimates and metallurgical recoveries from preliminary engineering studies, as show in Table 14.33.

Table 14.33: Pit Optimization Parameters by Deposit

Input Parameters	Units	Yellow Pine	Hangar Flats	West End	Historic Tailings	Notes
Mining Cost – Mineral Resource	$/t mined	1.75	1.75	1.75	1.75	*Includes mining G&A*
Mining Cost - Waste	$/t mined	1.90	1.50	1.75	0.50	*Includes mining G&A*
Oxide Processing Cost	$/t mined	N/A	N/A	9.00	N/A	*Excludes G&A costs*
Oxide Au Recovery	%	N/A	N/A	84 * AuCN/ AuFA+8.52	N/A	*Formula based on PFS level metallurgical test results*
Oxide / Sulfide Boundary	CN Au : FA Au	N/A	N/A	0.70	N/A	
Sulfide Processing Cost	$/t milled	16.50	16.50	16.50	16.50	*Excludes G&A costs*
Sulfide Au Recovery	%	93.0	92.0	88.0	80.0	
Dore Transport Cost	$/oz Au	1.15	1.15	1.15	1.15	
Dore Refining Cost	$/oz Au	1.00	1.00	1.00	1.00	
G & A + Rehabilitation Cost	$/t milled	3.50	3.50	3.50	3.50	
Pit Slopes	degrees	48	48	48	N/A	
Au Payability	%	99.5	99.5	99.5	99.5	
Au Selling Price - Initial Case(1)	$/oz	1,400	1,400	1,400	1,400	
Mining Dilution	%	0	0	0	0	
Mining Recovery	%	100	100	100	100	

Note:
(1) See Section 14.7 for comments on effective metal price used; while the mineral resource estimates were estimated using a $1,400/oz gold price, they are reported at higher cutoffs than these parameters generated, which equates to assuming a lower gold price.

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 Historic 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.

14.7 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, 2014). The mineral resources reported in Table 14.34 to Table 14.39, 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 at based on a $1,400/oz gold selling price, which resulted in an open pit sulfide cutoff grade, excluding adjustments, of approximately 0.55 g/t Au and an open pit oxide cutoff grade, excluding adjustments, of approximately 0.35 g/t Au. However, Midas Gold elected to report its base case mineral resource estimate using a 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade which is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfide material and $1,100/oz for oxide material. 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. Sensitivity to cutoff grade is reported in Table 14.40, Figure 14.17, Figure 14.18, Figure 14.19, Figure 14.20, and Figure 14.21.

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 Historic Tailings mineral resource also contains elevated concentrations of antimony. These higher-grade antimony zones are reported separately in Table 14.34 below 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 14.34: Consolidated Mineral Resource Statement for the Stibnite Gold Project

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	21,389	1.60	1,103	4.30	2,960	0.11	54,180
West End	35,974	1.30	1,501	1.35	1,567	0.008	6,563
Yellow Pine	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	***104,506***	***1.63***	***5,464***	***2.65***	***8,904***	***0.07***	***155,169***
Inferred							
Hangar Flats	7,451	1.52	363	4.61	1,105	0.11	18,727
West End	8,546	1.15	317	0.68	187	0.006	1,083
Yellow Pine	9,031	1.31	380	1.50	437	0.03	5,535
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	***25,168***	***1.32***	***1,066***	***2.15***	***1,743***	***0.05***	***25,908***

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 in order 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.

Table 14.35: Antimony Sub-Domains Consolidated Mineral Resource Statement

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated							
Hangar Flats	3,901	2.06	258	7.23	907	0.59	50,729
Yellow Pine	6,080	2.27	443	6.99	1,367	0.58	77,841
Historic Tailings	2,583	1.19	99	2.95	245	0.17	9,648
Total Indicated	***12,564***	***1.98***	***800***	***6.23***	***2,518***	***0.50***	***138,218***
Inferred							
Hangar Flats	1,186	1.94	74	8.05	307	0.68	17,844
Yellow Pine	409	1.36	18	4.86	64	0.50	4,552
Historic Tailings	140	1.23	6	2.88	13	0.18	563
Total Inferred	***1,735***	***1.74***	***97***	***6.88***	***384***	***0.60***	***22,959***

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 0.75 g/t Au cutoff.

Table 14.36: Hangar Flats Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	21,389	1.60	1,103	4.30	2,960	0.11	54,180
Inferred	7,451	1.52	363	4.61	1,105	0.11	18,727

Notes:

(1) Mineral resources are reported in relation to a conceptual pit shell in order 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.*

(2) Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.

Table 14.37: West End Mineral Resource Statement Open Pit Oxide + Sulfide

Classification and Material Type	Cutoff Grade (g/t Au)	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated Oxide	0.45	8,448	0.80	216	1.22	332	0.010	1,769
Indicated Sulfide	0.75	27,526	1.45	1,285	1.40	1,235	0.008	4,794
Total Indicated		***35,974***	***1.30***	***1,501***	***1.35***	***1,567***	***0.008***	***6,563***
Inferred Oxide	0.45	2,057	0.76	50	0.40	27	0.004	168
Inferred Sulfide	0.75	6,489	1.28	267	0.77	161	0.006	916
Total Inferred		***8,546***	***1.15***	***317***	***0.68***	***187***	***0.006***	***1,083***

Notes:

(1) *Mineral resources are reported in relation to a conceptual pit shell in order 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.*

(2) *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 mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au and an open pit oxide cutoff grade of approximately 0.35 g/t Au. The 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade are equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides.*

Table 14.38: Yellow Pine Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	44,559	1.93	2,762	2.89	4,133	0.09	84,777
Inferred	9,031	1.31	380	1.50	437	0.03	5,535

Notes:

(1) *Mineral resources are reported in relation to a conceptual pit shell in order 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.*

(2) *Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.*

Table 14.39: Historic Tailings Mineral Resource Statement Open Pit Sulfide at a 0.75 g/t Au Cutoff

Classification	Tonnage (kt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)	Antimony Grade (%)	Contained Antimony (klbs)
Indicated	2,583	1.19	99	2.95	245	0.17	9,648
Inferred	140	1.23	6	2.88	13	0.18	563

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 full 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. 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.*
(2) *Open pit sulfide mineral resources are reported at a cutoff grade of 0.75 g/t Au. The mineral resources were estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au. The 0.75 g/t Au sulfide cutoff grade is equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz.*

14.8 GRADE SENSITIVITY ANALYSIS

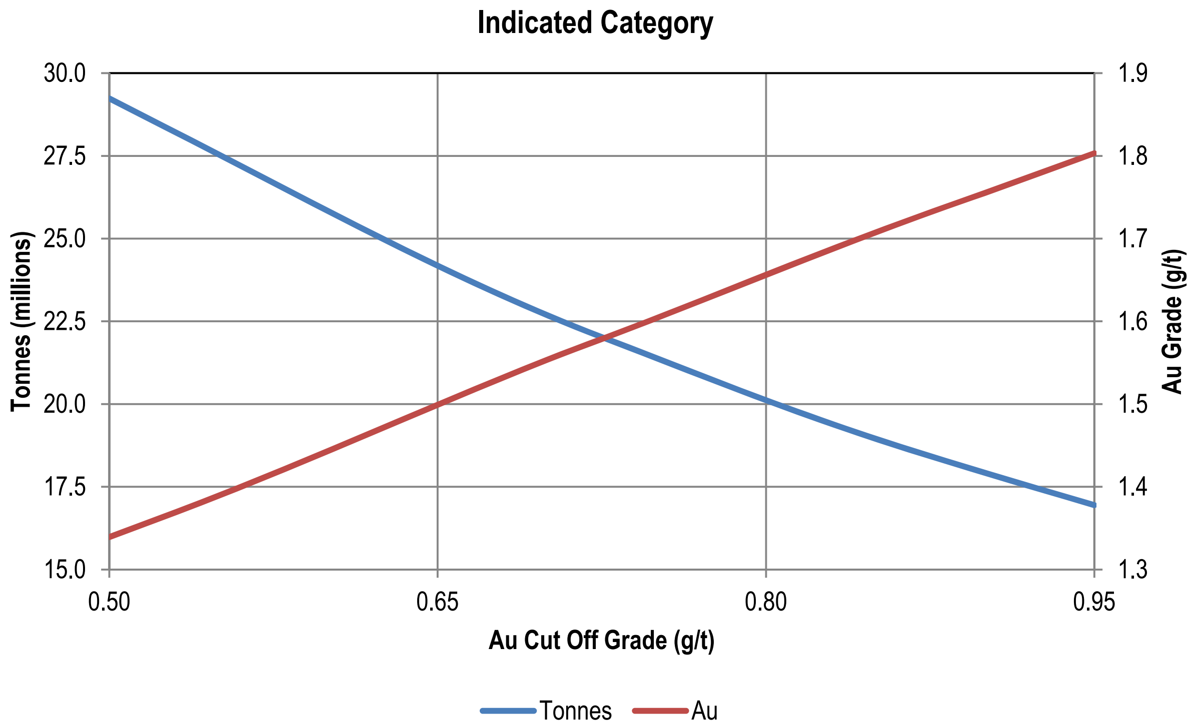
The mineral resources are sensitive to the gold cutoff grade used for reporting. To demonstrate this, the mineral resources are reported at different cutoff grades within the base-case conceptual mineral resource-limiting pit shells (Table 14.40) and are presented graphically on grade-tonnage curves (Figure 14.17, Figure 14.18, Figure 14.19, Figure 14.20, and Figure 14.21). 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 14.40: Combined Sensitivity to Cutoff Grade

Sulfide Cutoff Grade (g/t Au)	Oxide Cutoff Grade (g/t Au)	Yellow Pine (sulfide)		Hangar Flats (sulfide)		West End (oxide + sulfide)		Historic Tailings (sulfide)		Total (oxide + sulfide)	
		Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)	Gold Grade (g/t)	Contained Gold (koz)
Indicated											
0.60	0.30	1.80	2,875	1.44	1,199	1.10	1,717	1.16	102	1.44	5,892
0.65	0.35	1.84	2,838	1.50	1,166	1.16	1,645	1.17	102	1.51	5,750
0.70	0.40	1.89	2,799	1.55	1,133	1.23	1,573	1.17	101	1.57	5,606
0.75	**0.45**	**1.93**	**2,762**	**1.60**	**1,103**	**1.30**	**1,501**	**1.19**	**99**	**1.63**	**5,464**
0.80	0.50	1.97	2,724	1.66	1,071	1.36	1,438	1.21	96	1.68	5,329
0.85	0.55	2.01	2,684	1.71	1,040	1.42	1,375	1.24	92	1.74	5,191
0.90	0.60	2.05	2,643	1.76	1,011	1.48	1,313	1.26	89	1.79	5,056
Inferred											
0.60	0.30	1.16	438	1.35	404	1.00	368	1.21	6	1.16	1,215
0.65	0.35	1.21	421	1.40	391	1.06	351	1.21	6	1.21	1,169
0.70	0.40	1.26	401	1.46	376	1.10	335	1.22	6	1.26	1,117
0.75	**0.45**	**1.31**	**380**	**1.52**	**363**	**1.15**	**317**	**1.23**	**6**	**1.32**	**1,066**
0.80	0.50	1.36	360	1.57	350	1.20	300	1.27	5	1.37	1,016
0.85	0.55	1.41	343	1.63	336	1.23	288	1.30	5	1.42	972
0.90	0.60	1.45	326	1.68	325	1.28	271	1.31	5	1.46	927

Notes:
(1) *Mineral resources are reported in relation to the base-case conceptual pit shell in order 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.*

Figure 14.17: Hangar Flats Sulfide Grade versus Tonnage Curves



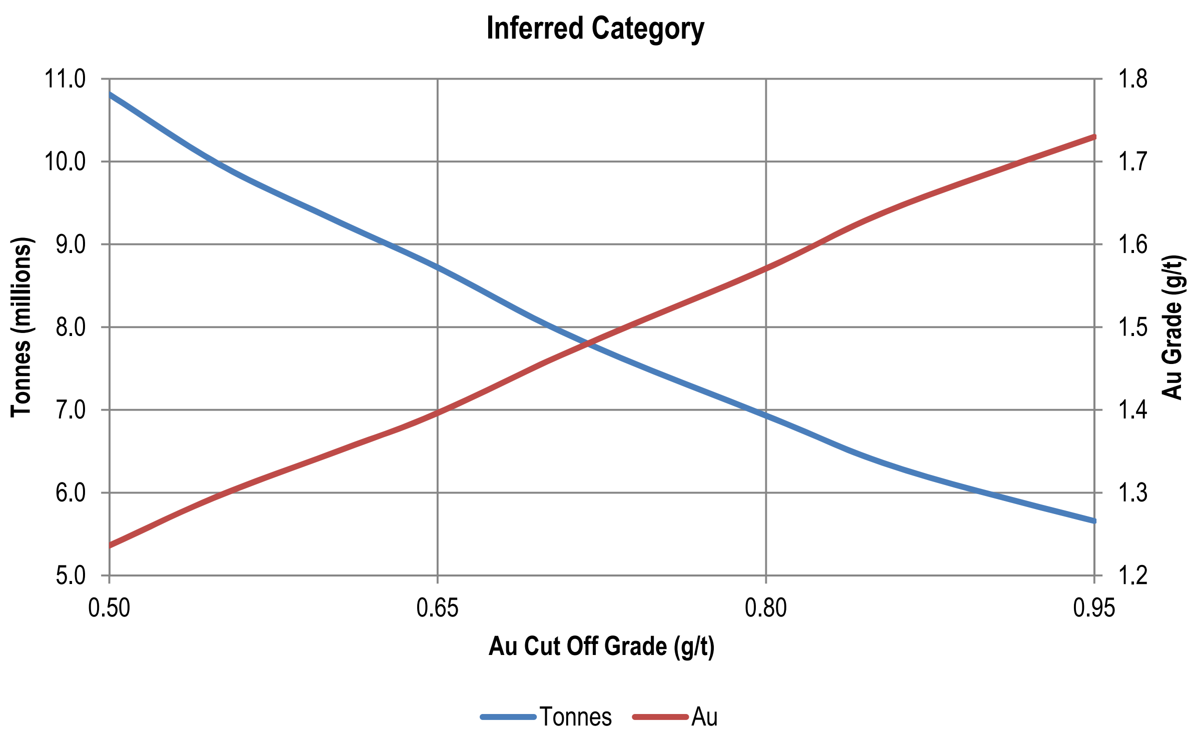
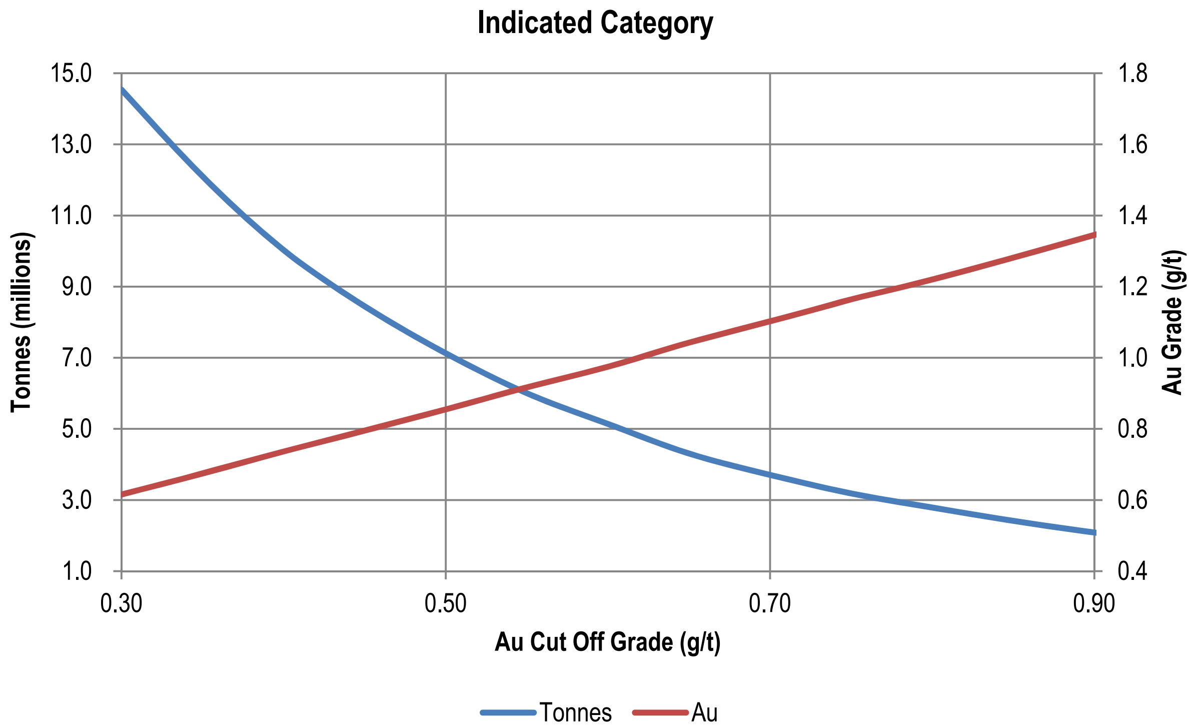


Figure 14.18: West End Oxide Grade versus Tonnage Curves



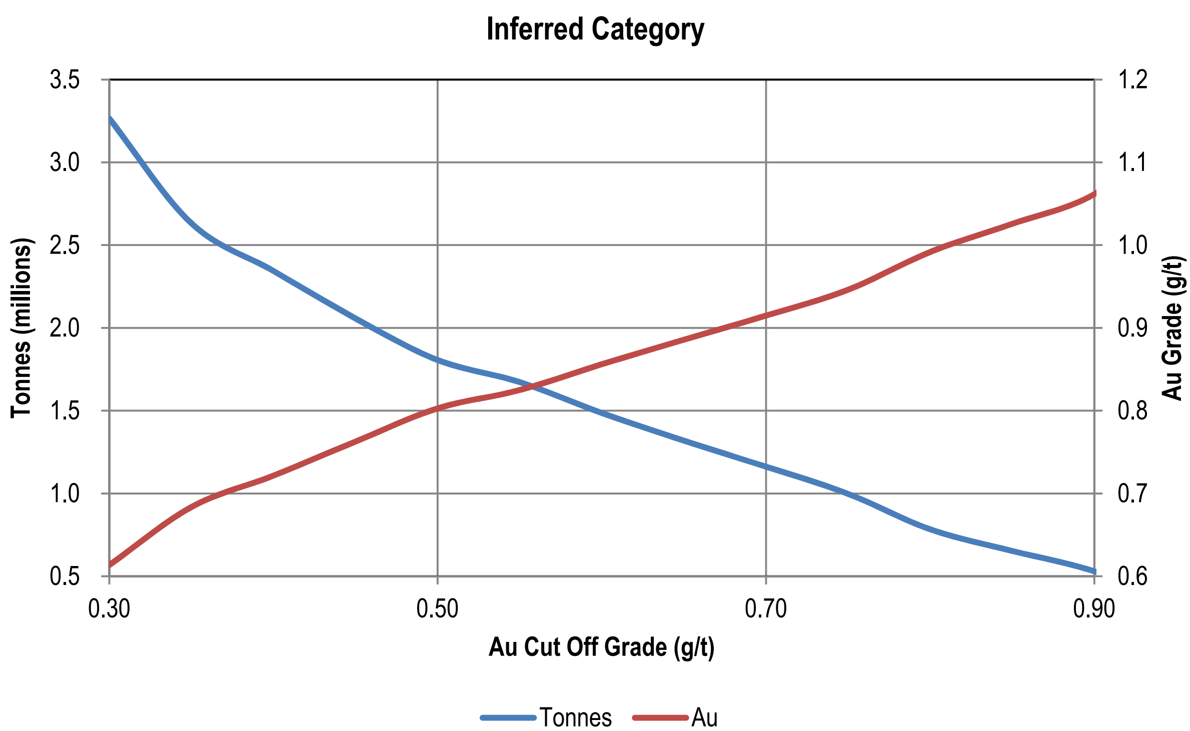
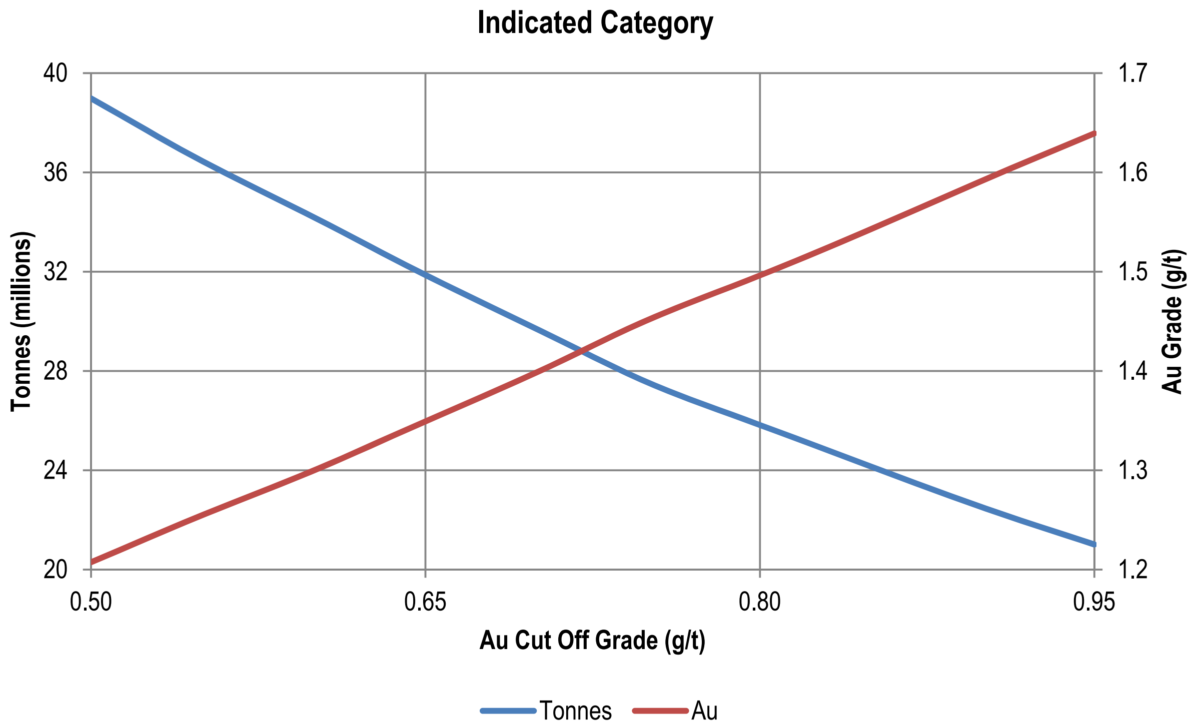


Figure 14.19: West End Sulfide Grade versus Tonnage Curves



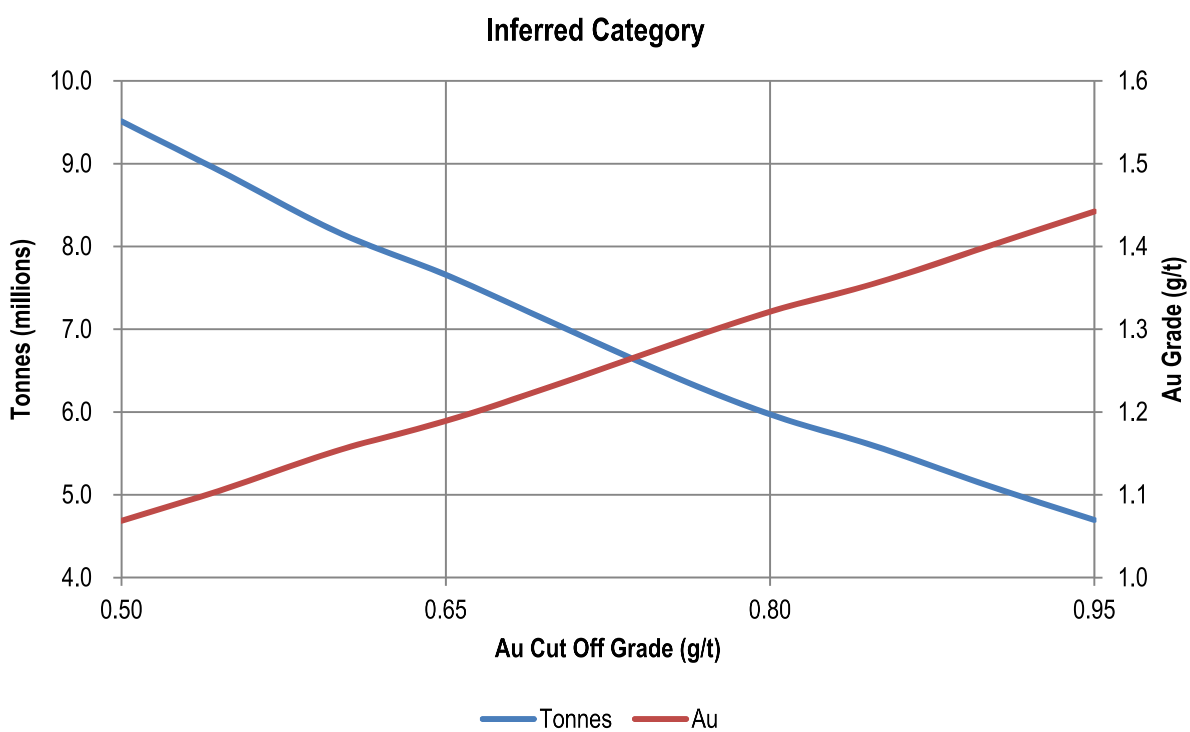
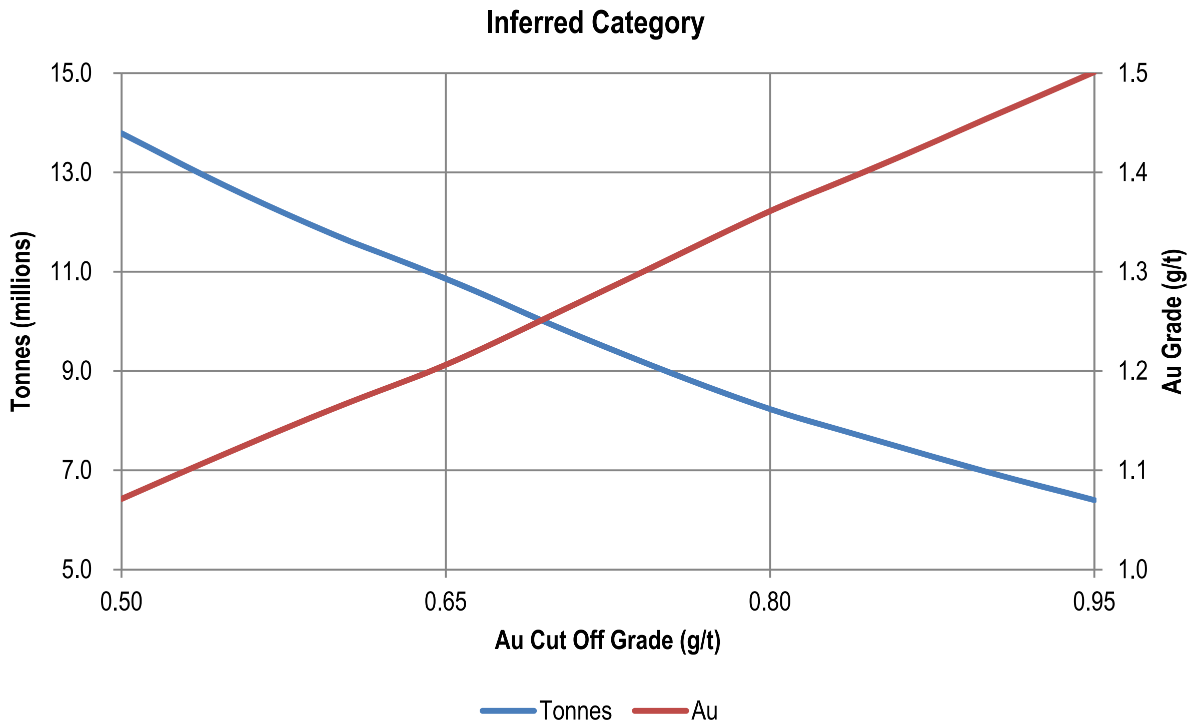


Figure 14.20: Yellow Pine Sulfide Grade versus Tonnage Curves



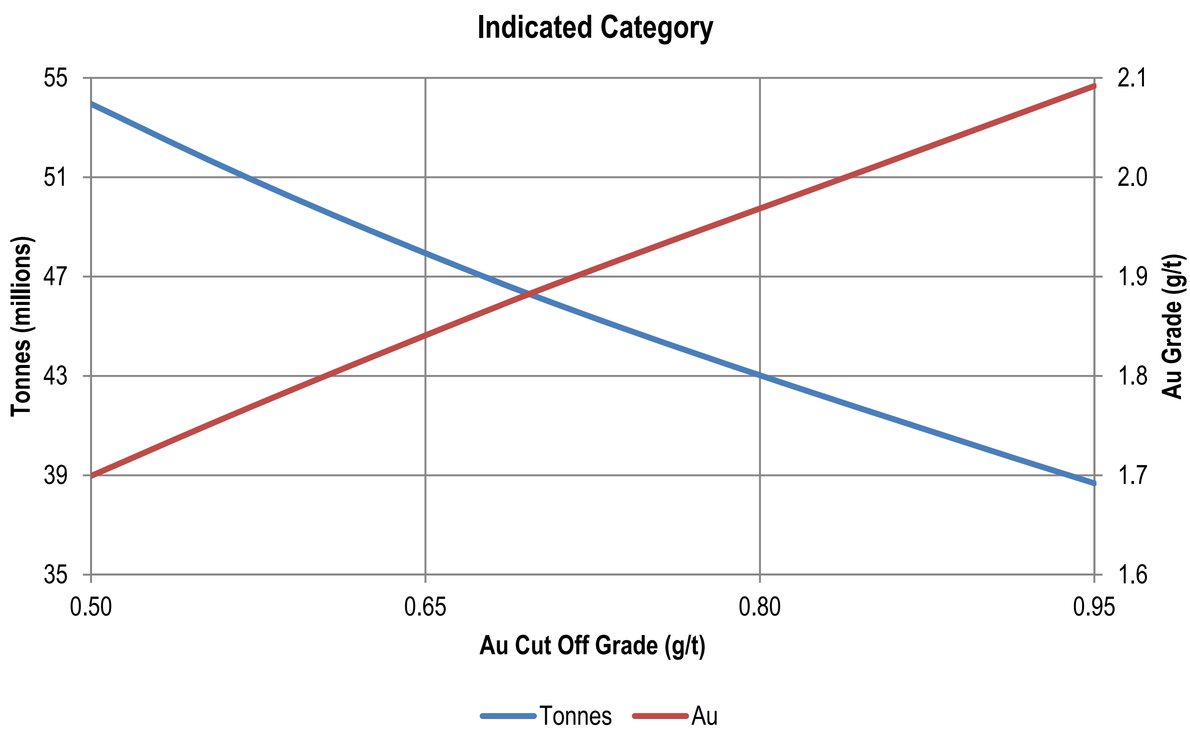
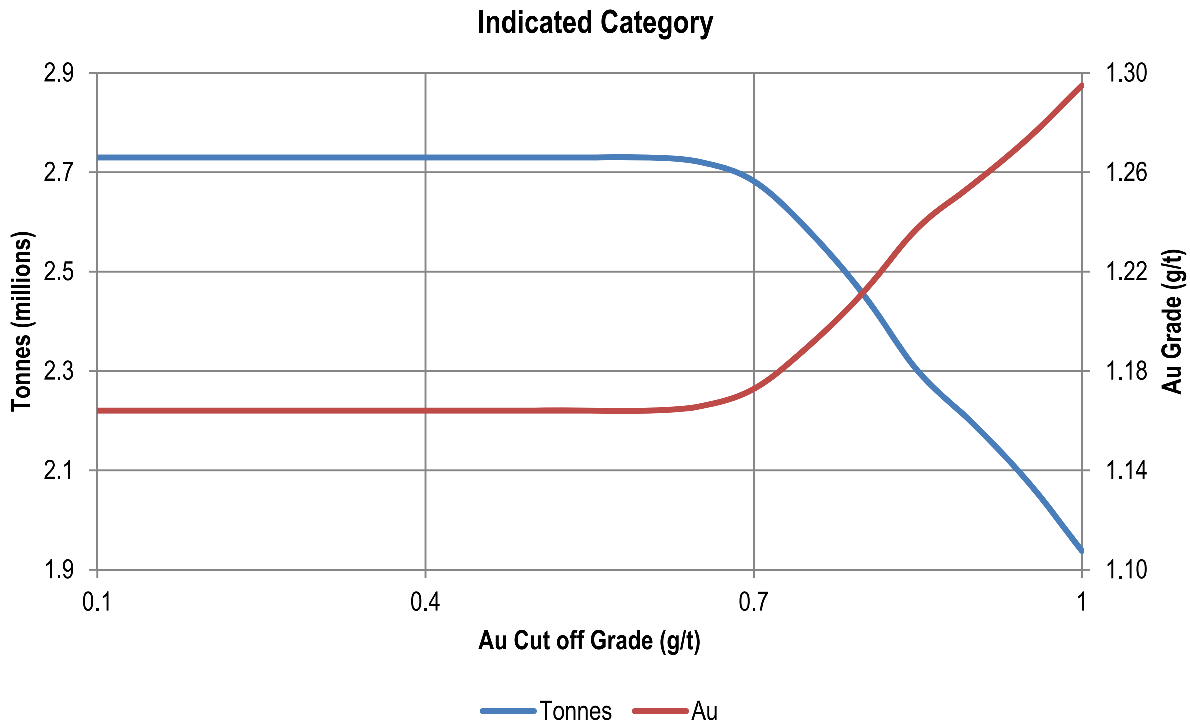
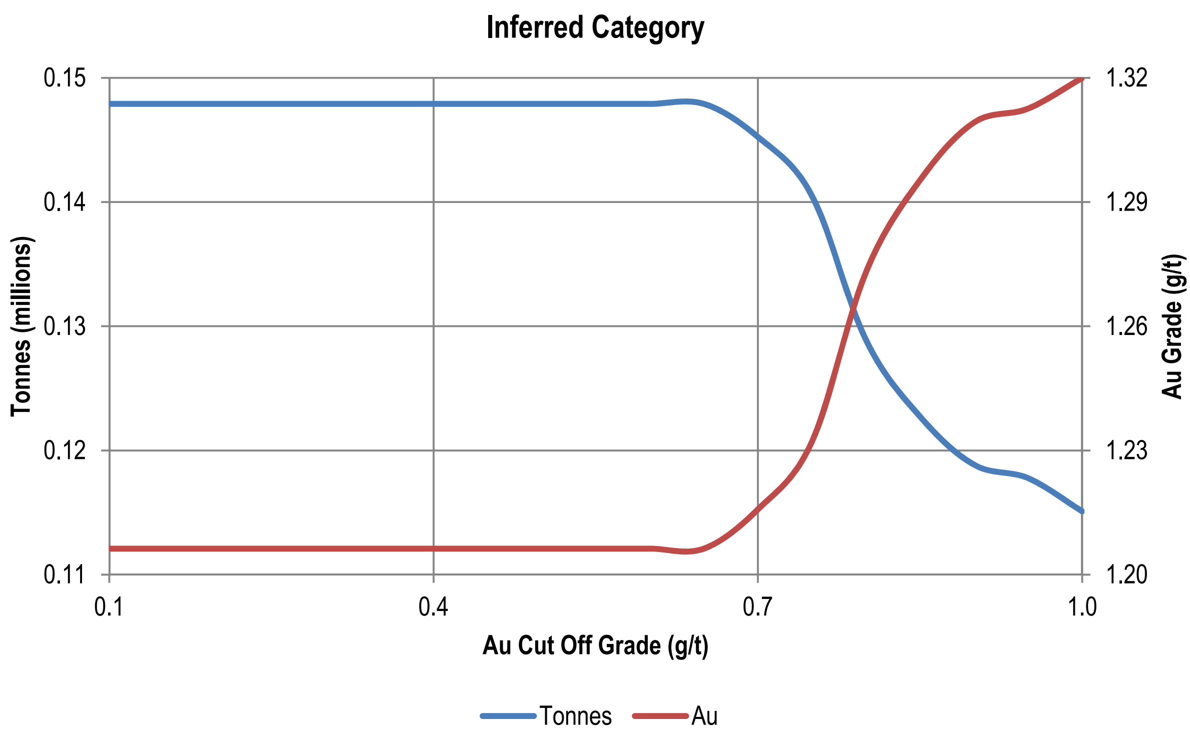


Figure 14.21: Historic Tailings Sulfide Grade versus Tonnage Curves





14.9 COMPARISON OF THE 2012 MINERAL RESOURCE ESTIMATE TO THE CURRENT ESTIMATE

The mineral resource estimates discussed herein incorporate the results of more than 45,000 m of new drilling completed since the cutoff date for the 2012 PEA. The drilling was focused within the Yellow Pine deposit and within a smaller conceptual pit shell at Hangar Flats. Relative to the 2012 PEA, indicated mineral resources for gold increased by 52% in Yellow Pine, 18% at Hangar Flats and remained approximately the same at the West End deposit, where only minimal drilling was completed, as shown in Table 14.41. In addition, the indicated mineral resources for antimony increased by 32% at Yellow Pine and by 22% at Hangar Flats. The West End mineral resource was expanded by drilling, but this increase was largely offset by mineral resource reductions associated with more conservative modeling parameters.

Table 14.41: Percentage Change of the 2012 Mineral Resource Estimate to the Current Estimate

Category	Yellow Pine (sulfide)	Hangar Flats (sulfide)	West End (oxide + sulfide)	Historic Tailings (sulfide)	Total (oxide + sulfide)
Indicated					
Tonnes	65%	24%	6%	100%(5)	34%
Gold Grade	-8%	-5%	-4%	100%(5)	-3%
Silver Grade	325%(6)	228%(6)	100%(4)	100%(5)	407%(6)
Antimony Grade	-18%	-8%	100%(4)	100%(5)	5%
Contained Gold	52%	18%	1%	100%(5)	29%
Contained Silver	604%(6)	308%(6)	100%(4)	100%(5)	579%(6)
Contained Antimony	32%	23%	100%(4)	100%(5)	43%
Inferred					
Tonnes	-72%	-11%	-44%	100%(5)	-55%
Gold Grade	-27%	5%	-7%	100%(5)	-18%
Silver Grade	-1%	5407%(6)	100%(4)	100%(5)	141%
Antimony Grade	-77%	491%	100%(4)	100%(5)	-40%
Contained Gold	-80%	-8%	-48%	100%(5)	-63%
Contained Silver	-72%	4923%(6)	100%(4)	100%(5)	8%
Contained Antimony	-94%	503%	100%(4)	100%(5)	-72%

Notes:

(1) *Mineral resources are reported in relation to conceptual pit shells in order 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.*

(2) *2014 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 mineral resources were initially estimated based on the open pit optimization parameters listed in Table 14.33 and a gold selling price of US$1,400/oz, which resulted in an open pit sulfide cutoff grade of approximately 0.55 g/t Au and an open pit oxide cutoff grade approximately 0.35 g/t Au. The 0.75 g/t Au sulfide cutoff grade and 0.45 g/t Au oxide cutoff grade are equivalent to utilizing the cost assumptions stated in Table 14.33 and a gold selling price of approximately $1,000/oz for sulfides and $1,100/oz for oxides.*

(3) *2012 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 0.42 g/t Au. 2012 mineral resources are reported in relation to the 2012 conceptual pit shells. The cutoff grades and pit shells are based on the open pit optimization parameters discussed in the 2012 PEA (SRK, 2012).*

(4) *Silver and antimony mineral resources at West End were not estimated in 2012 but have been assigned an arbitrary 100% increase.*

(5) *The Historic Tailings mineral resources were not previously estimated but have been assigned an arbitrary 100% increase.*

(6) *The 2012 Yellow Pine and Hangar Flats mineral resource estimates were developed using silver grade shells whereas the 2014 mineral resource estimates were not developed with silver grade shells; the 2014 mineral resource estimation methodology resulted in significant increases in both silver grade and contained silver for the deposits.*

Reductions in the mineral resources occurred primarily where the 2012 PEA estimates allowed extrapolation of grade at depth and around the periphery of the deposits based on the sparse drill data available at the time. Better structural constraints and additional drill hole information incorporated into the updated models now provide for a more conservative estimate, with better controls on gold mineralization. Figure 14.22 through Figure 14.30 depict regions where the new drilling has increased indicated mineral resources within each deposit. The substantial reduction in inferred antimony resources at Yellow Pine relative to the PEA resulted primarily from exclusion of BMC drill holes in the antimony estimate and from use of smaller antimony shells which were constructed using only the higher confidence USBM and post-1973 drill hole data. Numerous BMC underground drill holes located outside of the 2014 antimony shells purportedly intercepted significant antimony mineralization and formed the basis for the majority of inferred antimony resources in 2012. Confirmation drilling completed in 2013 failed to confirm significant antimony mineralization in these areas, suggesting that some antimony mineralization drilled historically from underground may occur in relatively narrow but substantially higher grade zones that would require close-spaced, detailed drilling in order to evaluate the potential for additional antimony mineral resources in these areas.

Figure 14.22: Plan Map of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification

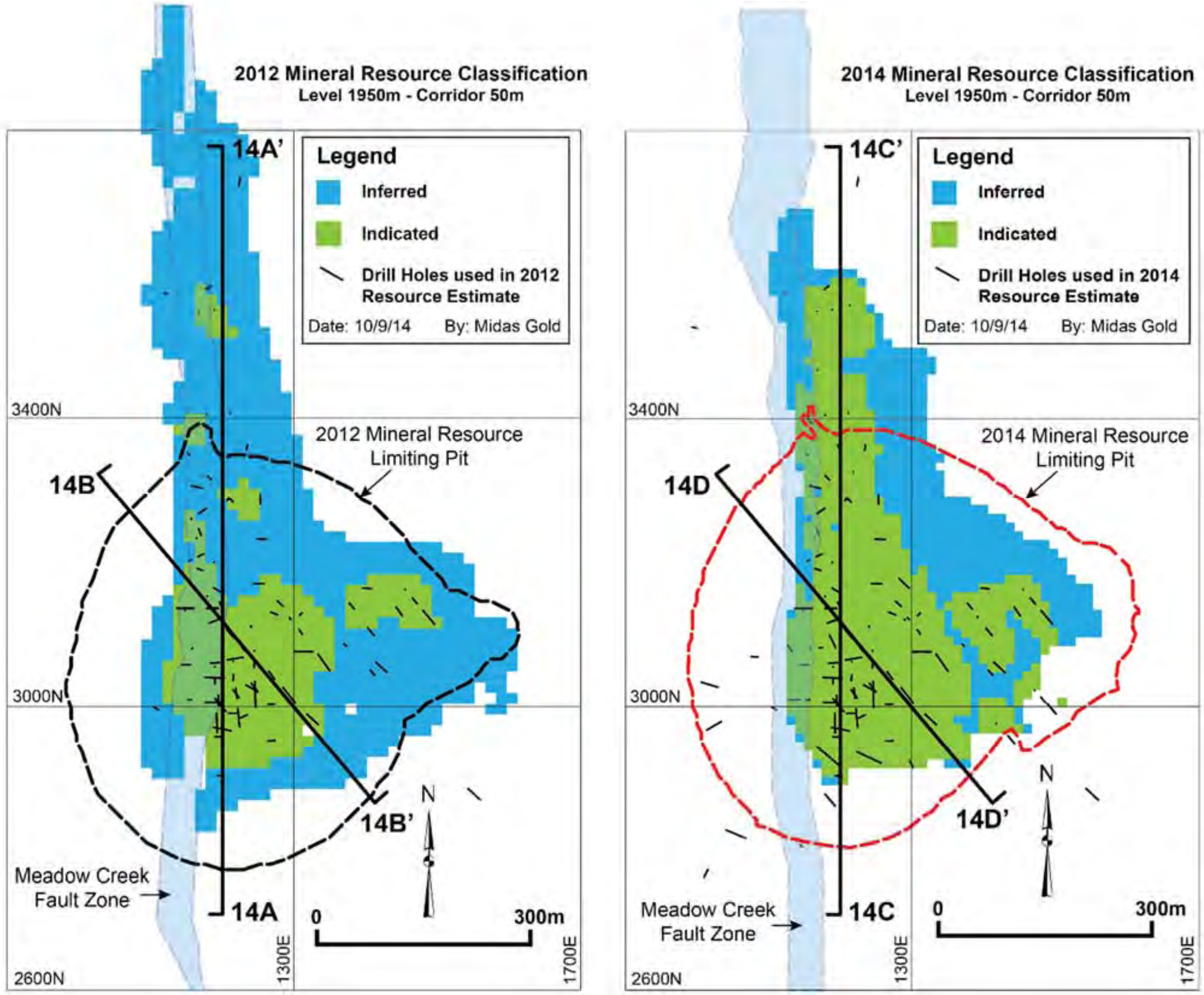


Figure 14.23: Long Section of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification

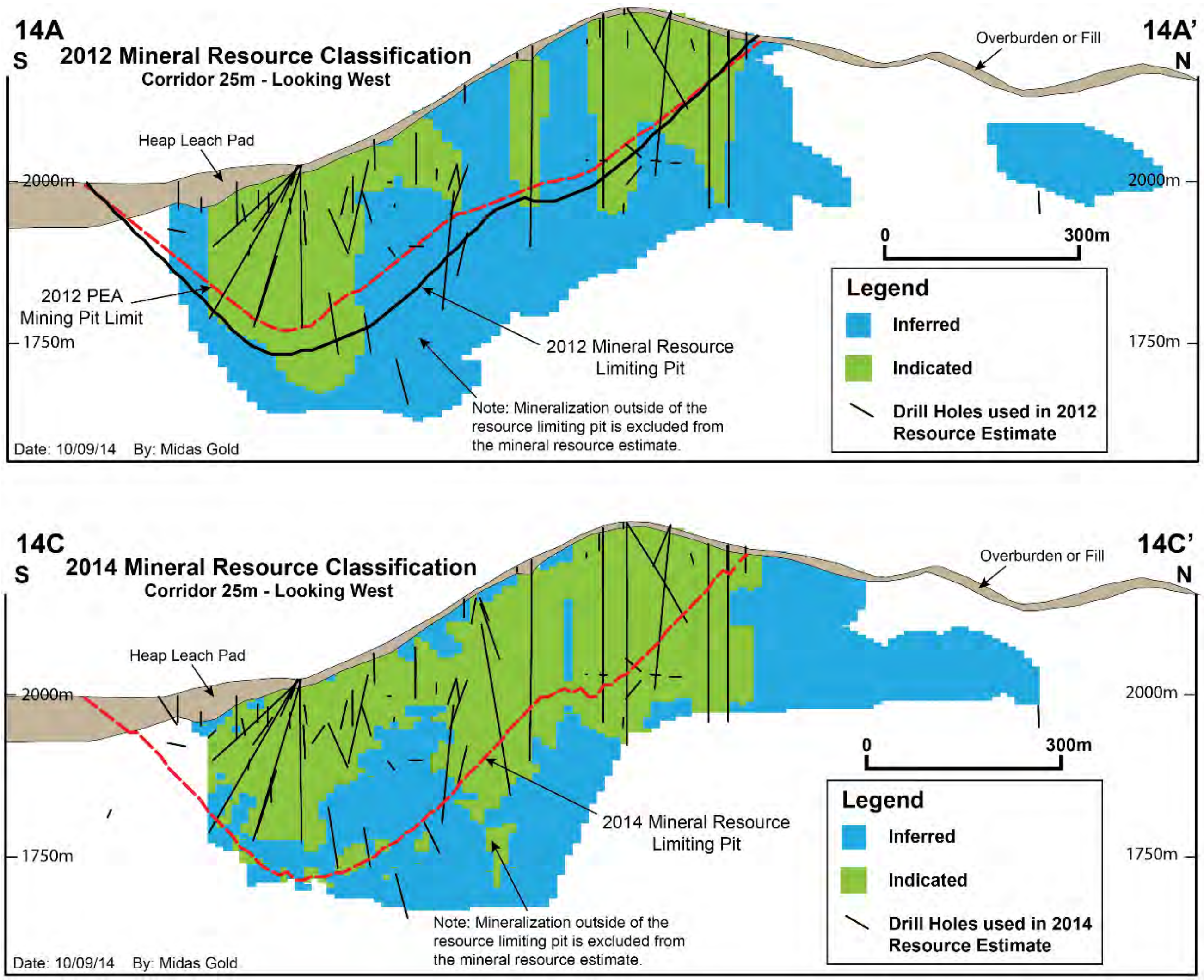
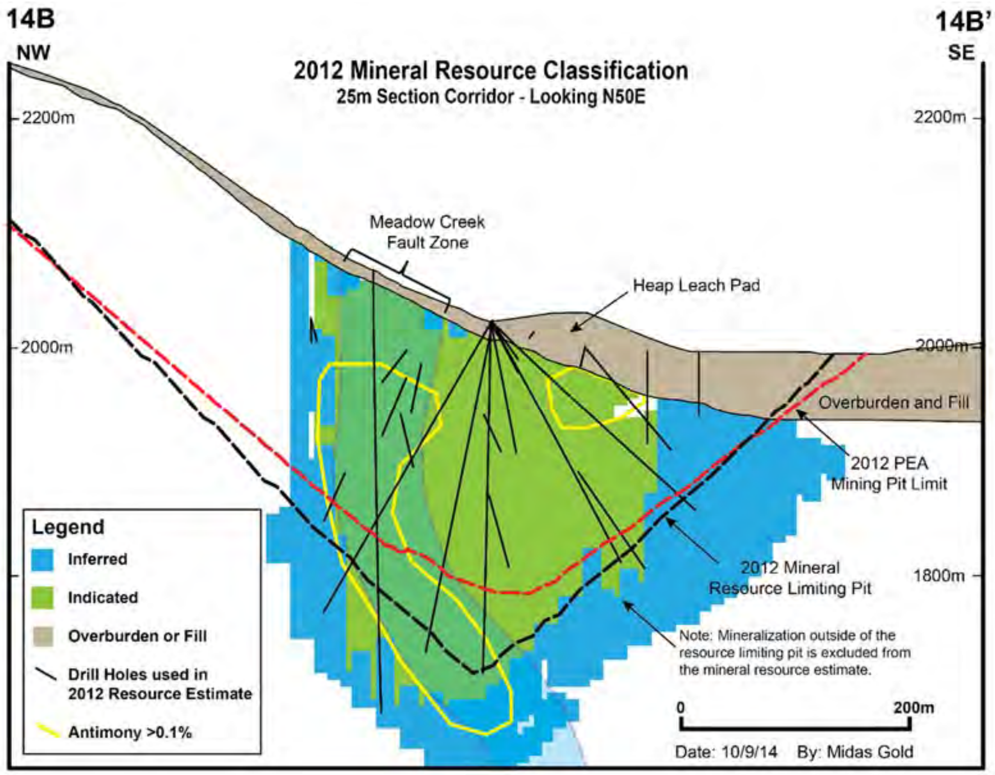


Figure 14.24: Cross Section of Hangar Flats Showing 2012 and 2014 Mineral Resource Classification



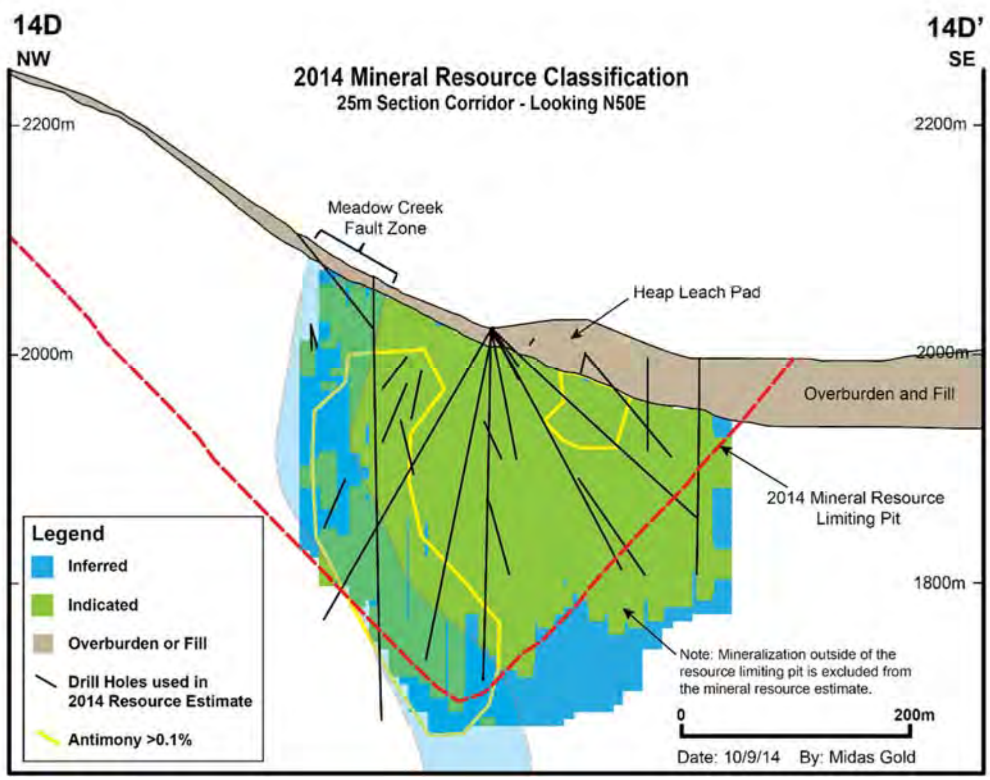


Figure 14.25: Inclined Plan Map of West End Showing 2012 and 2014 Mineral Resource Classification

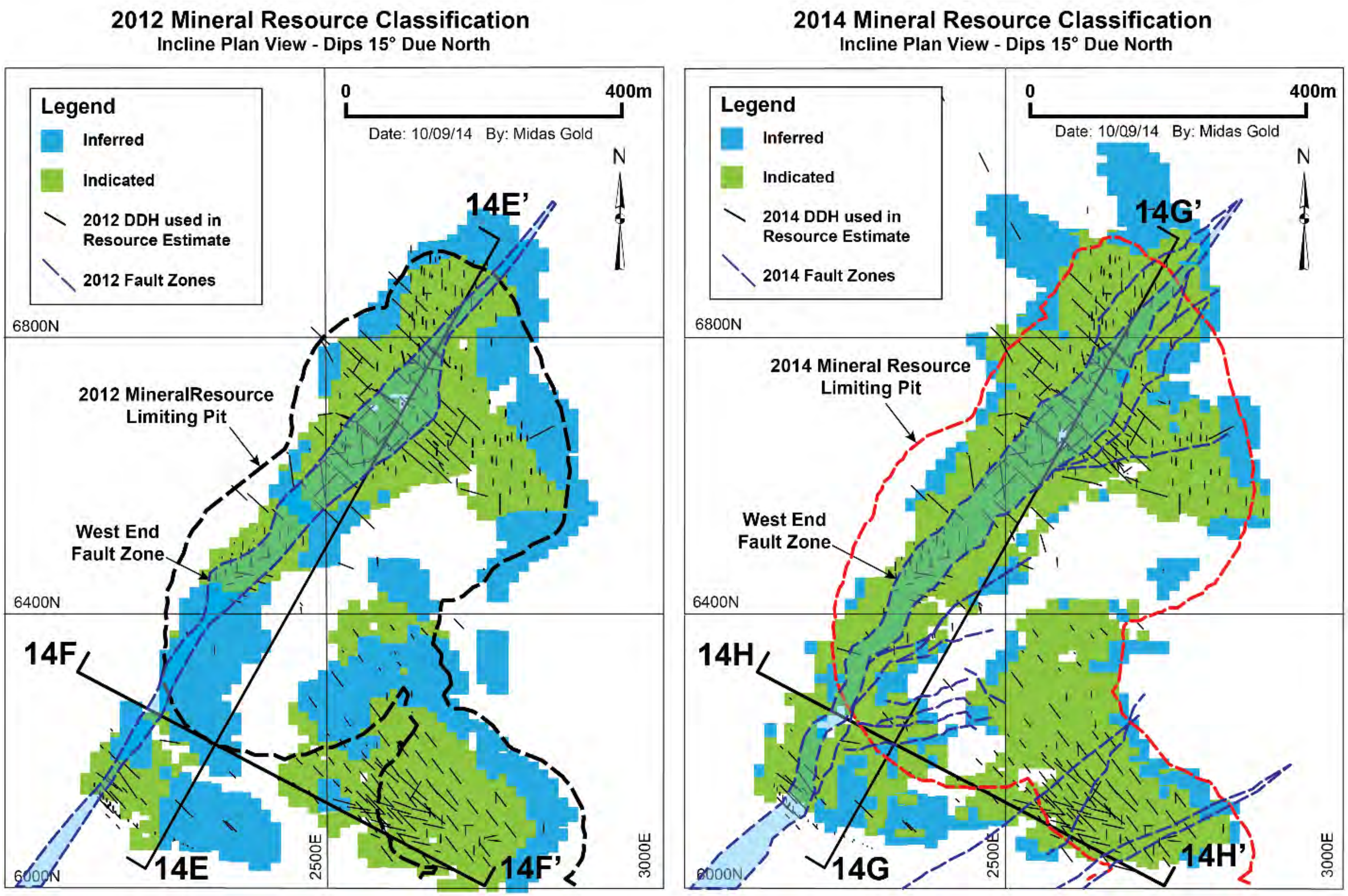
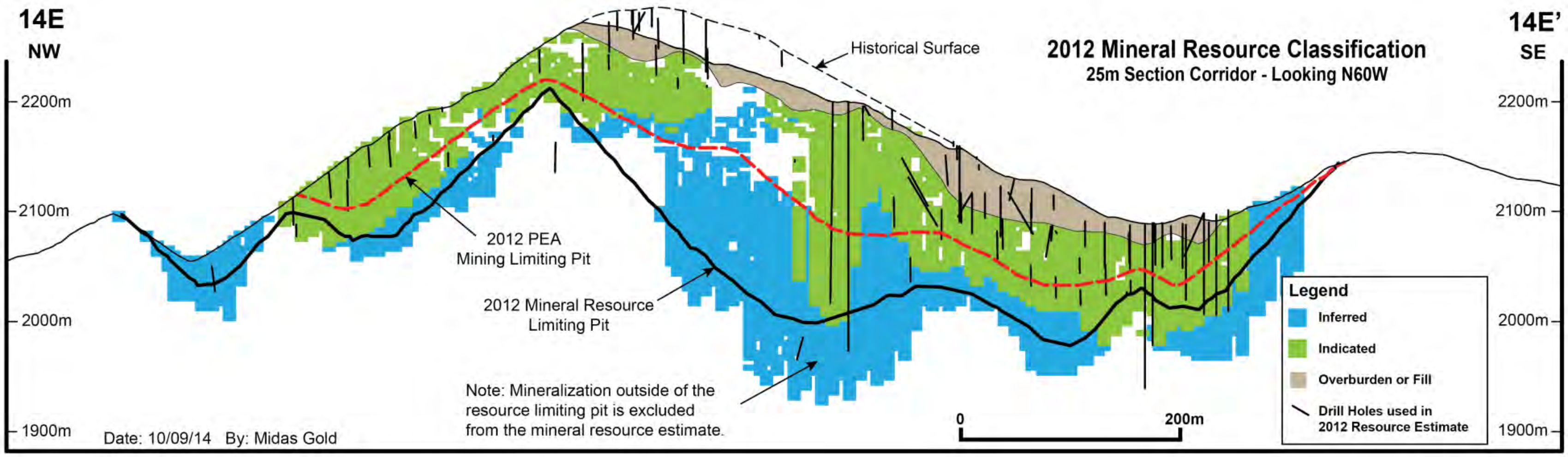


Figure 14.26: Long Section of West End Showing 2012 and 2014 Mineral Resource Classification



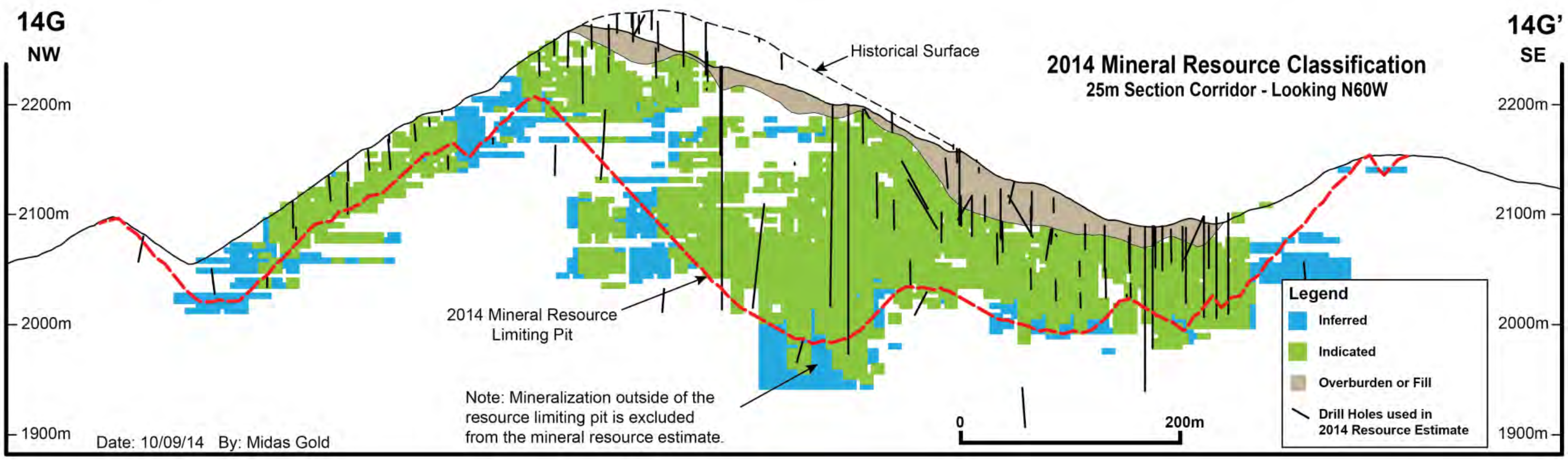
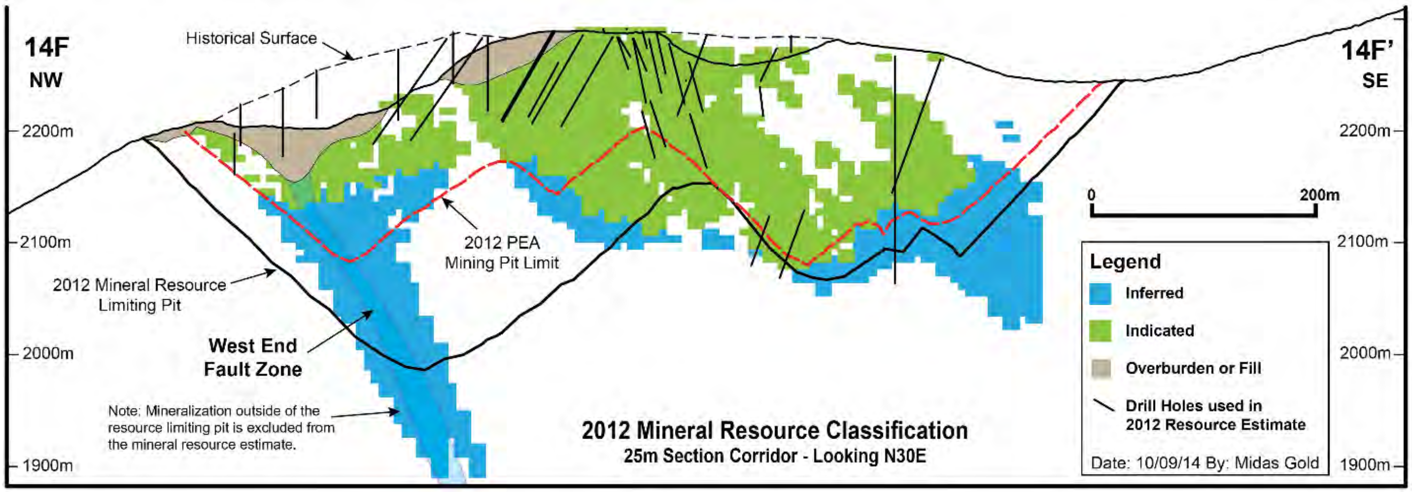


Figure 14.27: Cross Section of West End Showing 2012 and 2014 Mineral Resource Classification



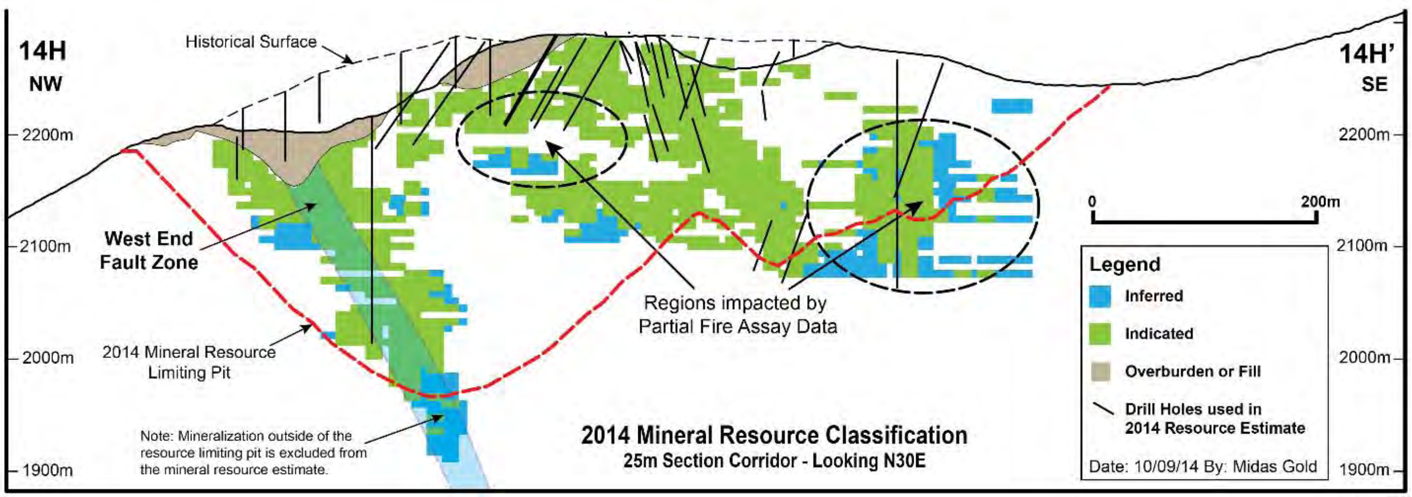
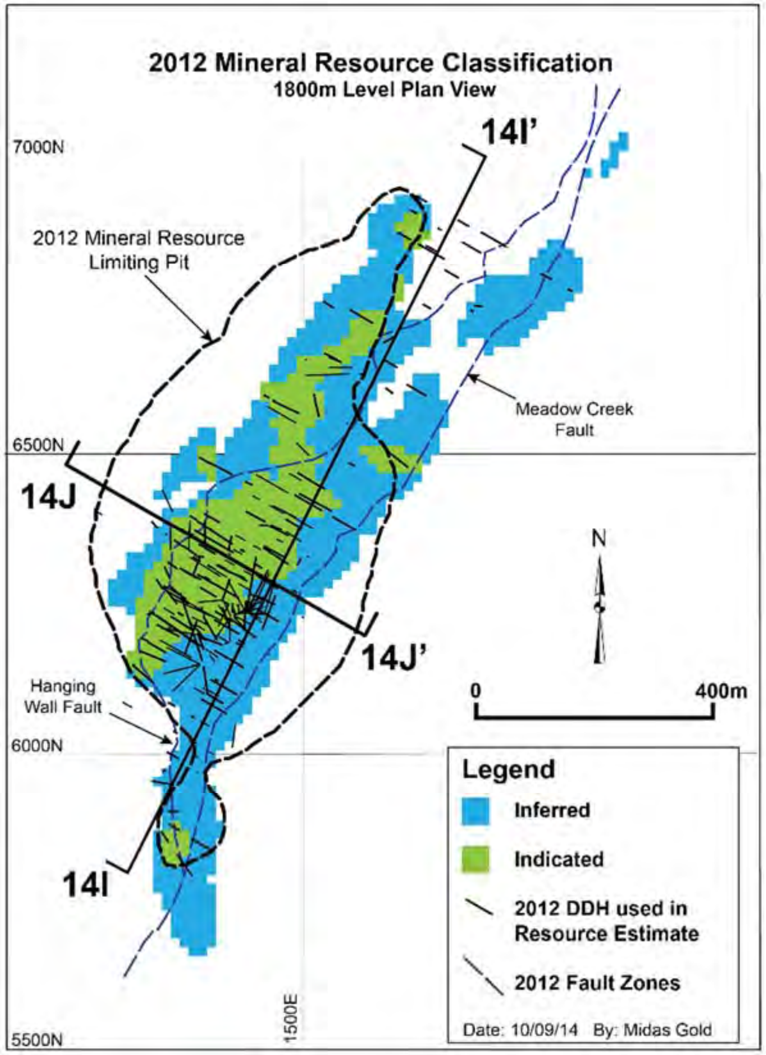


Figure 14.28: Plan Map of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification



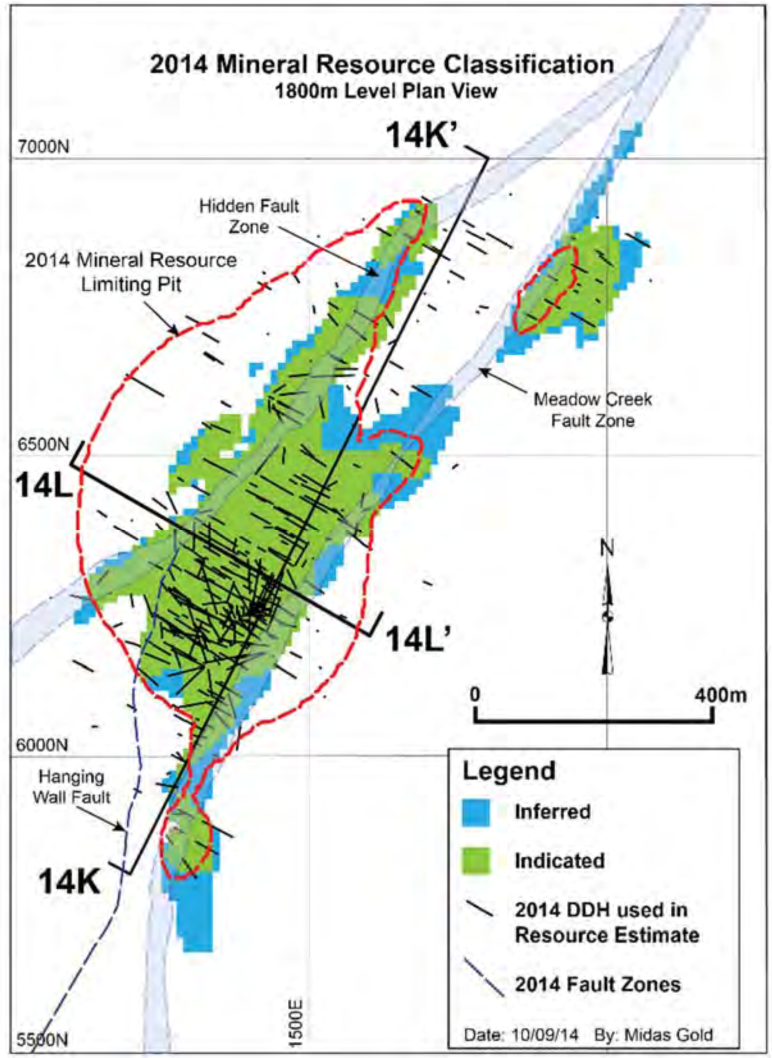


Figure 14.29: Long Section of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification

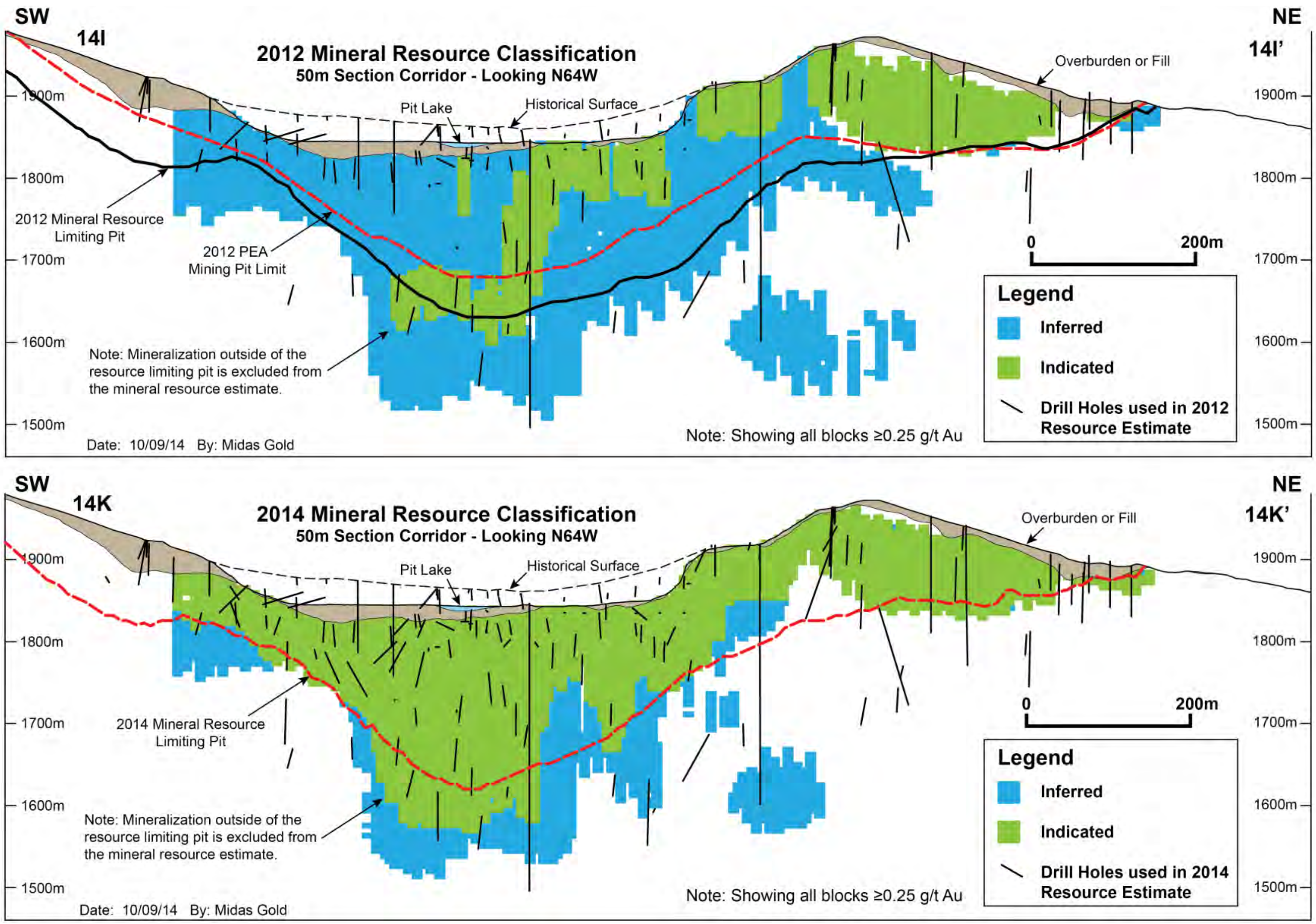
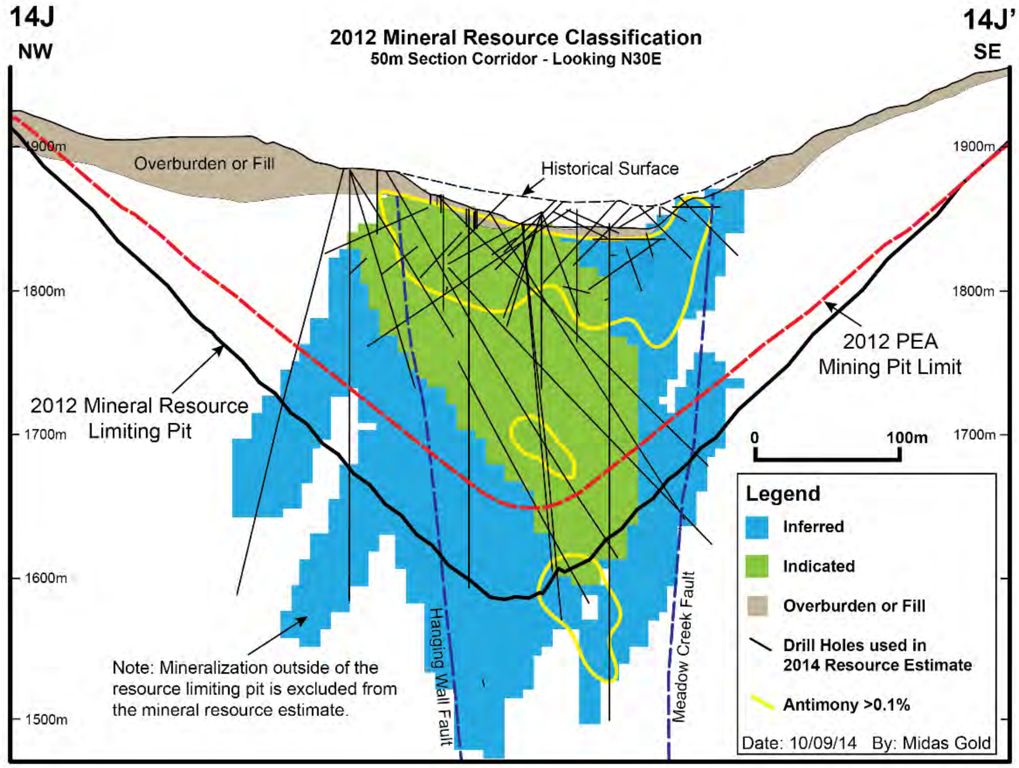
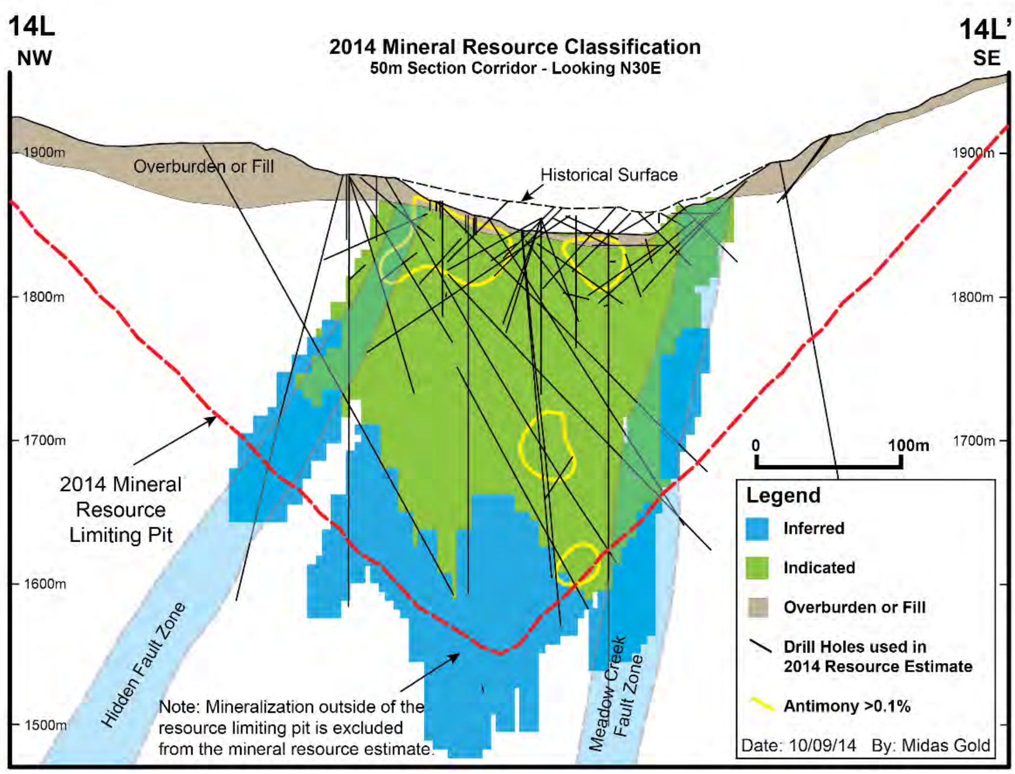


Figure 14.30: Cross Section of Yellow Pine Showing 2012 and 2014 Mineral Resource Classification





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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. The qualified person (**QP**) for the estimation of the Mineral Reserve was John M. Marek, P.E. of Independent Mining Consultants, Inc. The Mineral Reserve estimates reported herein are a reasonable representation of the Mineral Reserves within the Project at the current level of analysis. The Mineral Reserves were estimated in conformity with generally accepted Canadian Institute of Mining and Metallurgy (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines" and are reported in accordance with the Canadian Securities Administrators' NI 43-101. Mr. Marek 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 Mineral Reserve at a higher level of risk than any other North American developing projects.

The Mineral Reserve is the total of all Probable category material (there is no material in the Proven category) that is planned for production. The mine plan that is presented in Section 16 details the production of that Mineral Reserve. No low-grade stockpiles were considered in the mine plan for a lack of an acceptable location; therefore, the Mineral Reserve is established by tabulating the Indicated Mineral Resources that are planned for processing in each year which equates to the Probable Mineral Reserve. The final pit design and internal phase designs that contain the Mineral Reserve were guided by the results of the floating cone algorithm.

15.2 FLOATING CONES

The floating cone algorithm is a tool for phase design guidance. The algorithm applies approximate costs and recoveries along with approximate open pit slope angles to establish theoretical economic breakeven pit wall orientations.

Economic input applied to the cone algorithm is necessarily preliminary as it is one of the first steps in the development of the mine plan. The cone geometries should be considered as approximate as they do not assure access or working room. The important result of the cones is the relative change in geometry between cones of increasing metal prices. Lower metal prices result in smaller pits containing materials with higher margins, which provide guidance to the design of the initial phase designs. The change in pit geometry as metal prices are increased indicates the best directions for the succeeding phase expansions to the ultimate open pit.

Cones were floated for the Yellow Pine (**YP**), Hangar Flats (**HF**) and West End (**WE**) deposits using gold prices ranging from $200 to $1,500 per ounce. The costs and recoveries used for input were based on the PEA results and are provided in Table 15.1 through Table 15.4, inclusively. Net of Process Revenue (**NPR**), defined as Net Smelter Return (**NSR**) less process plant operating expenditures (**OPEX**) and general and administrative costs (**G&A**), was calculated on a block-by-block basis in dollars per ton of ore ($/st ore) to indicate the value of a block.

NPR = NSR – Process Plant OPEX – Site G&A

Designing with NPR was chosen because Stibnite Gold Project Mineral Resources are poly-metallic with separate process streams that have distinct differences in processing costs. It was necessary to account for processing costs in order to determine the appropriate ore type category of a block. Mining costs are not included in the calculation of NPR because the mining cost will be essentially the same for an ore block regardless of the process designation.

Table 15.1 Process Plant Recoveries for Floating Cones

Process	Metal	YP	HF	WE
Oxide Recovery	Gold	80%	80%	CN:FA x 100%
	Silver	80%	80%	80%
Sulfide Flotation Recovery	Gold High Sb	88%	89%	N/A
	Gold Low Sb	93%	92%	$(1.7366 \times CN{:}FA^3 - 2.5676 \times CN{:}FA^2 - 0.0858 \times CN{:}FA + 0.9231) \times 100\%$
	Silver High Sb	68%	68%	N/A
	Silver Low Sb	85%	80%	N/A
Transition Recovery	Gold	N/A	N/A	92%
	Silver	N/A	N/A	80%
POX Recovery	Gold	98%	98%	98%
	Silver	7%	7%	7%
Recovery to Antimony Concentrate	Gold	1.5%	1.5%	1.5%
	Silver	12%	12%	12%
	Antimony	80%	80%	80%
Antimony Con Grades	Antimony	50%	50%	50%
	Moisture	10%	10%	10%

Note: CN:FA is the ratio of cyanide soluble gold / fire assay gold.

Table 15.2 Payables and Transport Costs for Floating Cones

Off Site Costs and Payables	Item	Unit	Value
Payables for Dore	Gold	%	99.0
	Silver	%	95.0
Dore Ref/Transport Cost	Gold	$/paid oz	8.00
	Silver	$/paid oz	0.50
Smelter Payables for Antimony Concentrate	Gold (if Au concentrate grade > 0.292 oz/st)	%	60
	Silver (if Ag concentrate grade > 2.92 oz/st)	%	30
	Antimony	%	65
TC/RC Costs for Antimony Concentrate		$/wet st	141.52
		$/dry st	155.67

Table 15.3 Process Plant and G&A Costs for Floating Cones

Mineral Resource Type	Process Plant and G&A Costs
Oxide	$11.68/st ore
High Antimony Sulfide	$16.93/st ore
Low Antimony Sulfide	$18.43/st ore
Low Antimony Transition	$21.70/st ore

Table 15.4 Other Open Pit Optimization Parameters

Item	Parameter	Value
Royalties	Dore Produced Onsite and Concentrate Shipped Offsite	1.7% Net of Smelter on Gold
Mining Costs	All Pits	$1.73 / st of material
Bench Discounting	Yellow Pine Hangar Flats West End	0.8% / bench 1.1% / bench 1.1% / bench
Open Pit Slope Angles	Variable by open pit sector	Variable based on recommendations from Strata, A Professional Services Corporation (Strata, 2014); flattened to account for haul roads.

Bench discounting was considered in the development of the floating cones as a way to incorporate the time value of money into the pit optimization algorithm. Using the estimated vertical mining rate, an annual discounting rate can be approximated with bench discounting. For example, Yellow Pine had an estimated 12 benches per year mining rate based on a preliminary schedule. Assuming a 10% discount rate per year, each bench is discounted by 0.8% (10% / 12 benches) starting from the pit crest. Since waste rock is stripped ahead of ore to assure ore release, the time value of delayed ore can be compared against the preceding cost of waste stripping. The bench discounting rates are provided in Table 15.4.

Cones were generated by allowing only Indicated Mineral Resource blocks (there are no Measured Mineral Resources) to contribute positive economic value. Confidence classification was based on gold-only estimation. Although the NPR values calculated for each block included value for silver and antimony, if a block was uneconomical based on gold content alone, it was treated as waste. This prevented blocks from contributing positive economics because of their additional silver and antimony content, ensuring that silver and antimony are treated as true byproducts.

Within the floating cones, blocks were categorized by the process that generated the greatest NPR value. If processing a block produced a negative NPR value, the block was considered waste. The block categorization bins by ore deposit used for floating cones are explained below:

Yellow Pine and Hangar Flats blocks were designated as follows:

- High Antimony 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 and cyanide leaching.

or

- Low Antimony Sulfide Ore: Only a gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching.

or

- Waste Rock

West End Blocks were designated as follows:

- Low Antimony Sulfide Ore: A gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching.

or

- Oxide Ore: Gold would be recovered through whole ore cyanide leaching

or

- Transition Ore: A Gold bearing sulfide concentrate would be produced. The sulfide concentrate would be processed onsite through pressure oxidation and cyanide leaching. Additional gold would be recovered through cyanide leaching of the tails.

or

- Waste Rock

15.3 GUIDANCE CONE SELECTION AT YELLOW PINE

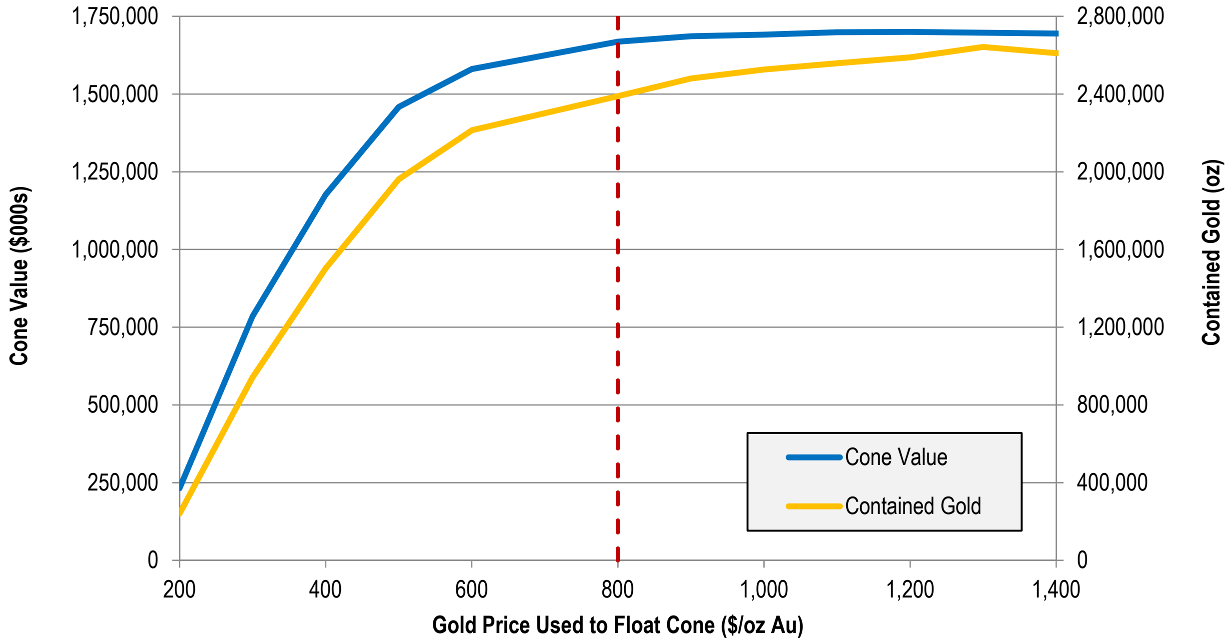
A range of cone geometries were developed for the Yellow Pine deposit by varying the gold price between $200/oz. and $1,500/oz. Costs were held constant in each case and a floating cone pit geometry was established at each assumed metal price. Floating cones establish the pit wall location on a breakeven economic basis. The cones thus derived were then evaluated at a $1,200/oz gold price without changing the size of the cone. The purpose of this work was to see if there was a point of diminishing returns as the cone size increased where little value is added by increasing the pit size. This happens at the $800/oz Au cone; therefore the final pit of Yellow Pine was designed to contain the ore within the $800/oz Au cone. The additional value contained within the pits that were generated at metal prices above $800/oz was incrementally marginal compared to the $800/oz geometry. The benefit of mining a larger pit would become more marginal or even negative once the mine schedule is completed and the value of the pit is evaluated on a discounted basis. The tonnage curves of the cones between $200 and $1,500/oz Au are given in Table 15.5 on and Figure 15.1.

Table 15.5: Yellow Pine Comparison of Increasing Cone Sizes for Constant $1,200/oz Gold Price

Au [1] Price	Mineral Resource (kst)	Net of Process ($/st)	Au (oz/st)	Sb (%)	Ag (oz/st)	Waste Rock (kst)	Total (kst)	Strip Ratio (waste/ore)	Net of Process ($000s)	Mining Cost ($000s)	Value ($000s)	Contained Au (oz)
1,500	49,608	41.66	0.053	0.059	0.043	167,928	217,536	3.39	2,066,669	376,337	1,690,332	2,629,224
1,400	49,247	41.81	0.053	0.059	0.043	161,295	210,542	3.28	2,059,017	364,238	1,694,779	2,610,091
1,300	48,928	41.95	0.054	0.059	0.043	156,242	205,170	3.19	2,052,530	354,944	1,697,586	2,642,112
1,200	47,944	42.34	0.054	0.060	0.043	142,793	190,737	2.98	2,029,949	329,975	1,699,974	2,588,976
1,100	47,383	42.52	0.054	0.061	0.044	135,162	182,545	2.85	2,014,725	315,803	1,698,922	2,558,682
1,000	45,939	43.17	0.055	0.062	0.045	122,965	168,904	2.68	1,983,187	292,204	1,690,983	2,526,645
900	45,090	43.53	0.055	0.063	0.045	114,813	159,903	2.55	1,962,768	276,632	1,686,136	2,479,950
800	43,430	44.17	0.055	0.065	0.046	100,947	144,377	2.32	1,918,303	249,772	1,668,531	2,388,650
600	38,835	45.57	0.057	0.070	0.049	70,637	109,472	1.82	1,769,711	189,387	1,580,324	2,213,595
500	33,817	47.56	0.058	0.077	0.053	52,718	86,535	1.56	1,608,337	149,706	1,458,631	1,961,386
400	24,633	51.39	0.061	0.096	0.064	27,102	51,735	1.10	1,265,890	89,502	1,176,388	1,502,613
300	14,061	59.03	0.067	0.141	0.081	11,861	25,922	0.84	830,021	44,845	785,176	942,087
200	3,256	74.67	0.074	0.331	0.150	3,203	6,459	0.98	243,126	11,174	231,951	240,944

Note:
(1) *The gold price in the first column is the gold price that was used to generate the cone geometry. The remaining columns report the results of geometries being re-evaluated using a $1,200/oz gold price.*

Figure 15.1 Yellow Pine Comparison of Various Cone Sizes at $1200/oz Au



15.4 GUIDANCE CONE SELECTION AT HANGAR FLATS AND WEST END

Hangar Flats and West End were evaluated in the same manner as Yellow Pine. The curves of the increasing cone sizes evaluated at $1,200/oz Au for Hangar Flats and West End are provided on Figure 15.2 and Figure 15.3, respectively.

Figure 15.2 Hangar Flats Comparison of Various Cone Sizes at $1,200/oz Gold

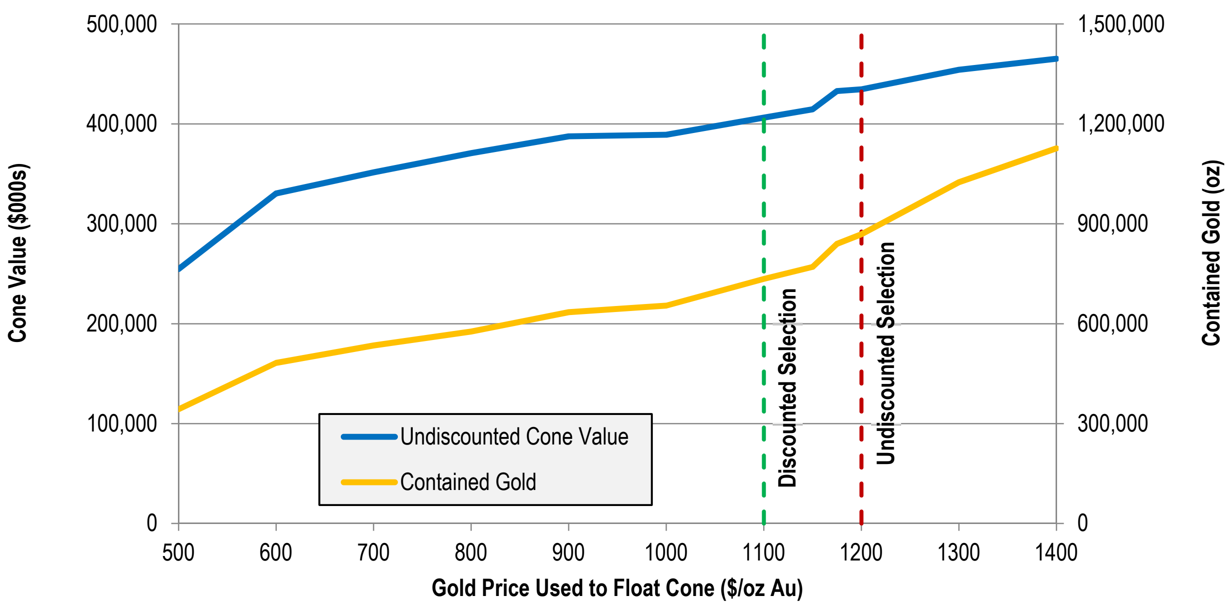
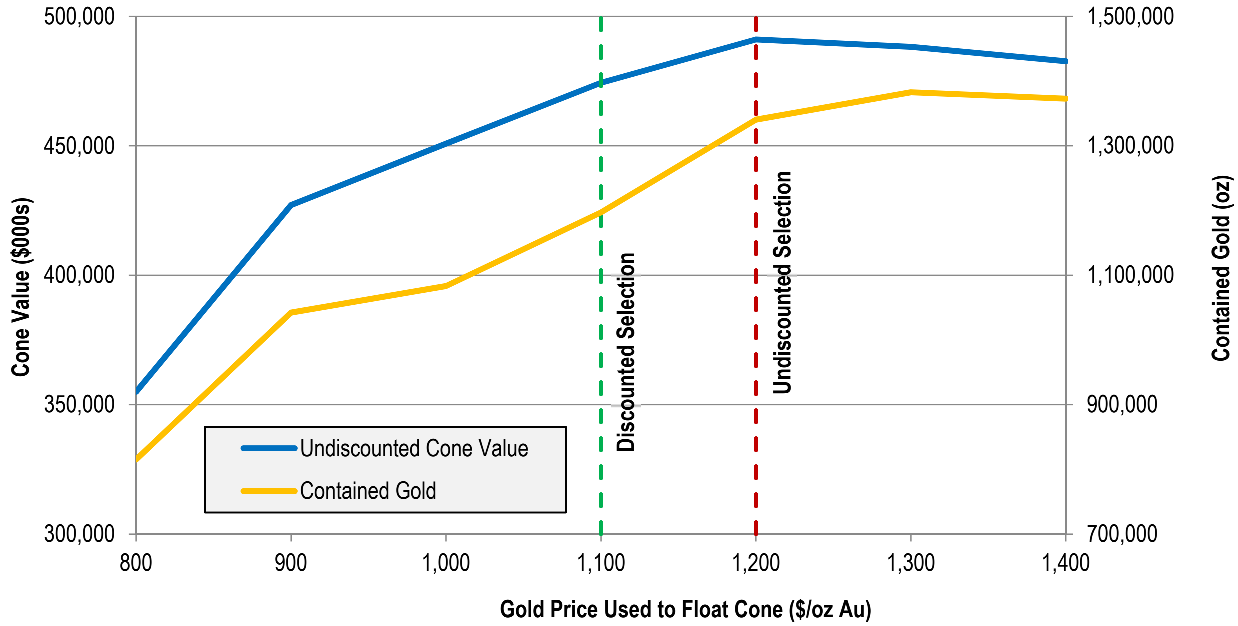


Figure 15.3 West End Comparison of Various Cone Sizes at $1,200/oz Gold



The $1,200/oz Au floating cone appeared to be the appropriate size for both Hangar Flats and West End; however, Hangar Flats and West End come later in mine life than Yellow Pine and additional work was needed to ensure the proper sized floating cone was used for phase design guidance. Significant ore tonnages are not extracted from Hangar Flats until Year 6 and West End until Year 7, but waste rock mining starts several years before. To determine the correct size cone to guide phase design, multiple schedules were developed by scheduling the Yellow Pine phases followed by varying sized Hangar Flats and West End floating cones. Schedule results were then evaluated on a net present value basis. Table 15.6 summarizes the inputs that were used to evaluate the open pit sizing schedule options.

Table 15.6 Hangar Flats and West End Mine Schedule Inputs

Input	Value
Mining Cost	$1.73/st
Approximate capital cost to increase mining capacity	$2.00/st of annual production capacity
Block Value	Net of Process in $/st

For both Hangar Flats and West End, the $1,100/oz cones provided the best present value when evaluated on a time value of money basis and were chosen as the guidance cones for phase design.

15.5 GUIDANCE CONES AND ULTIMATE PIT DESIGNS

Floating cones or other computer generated pits do not consider phase access or bench working room and cannot be used for practical operations. The cones are used only as a guide for the design of operational mining phases. The following items were considered in phase design:

- Slope Angle Constraints: Strata provided inter-ramp and overall slope angle restrictions by sector for the three Mineral Reserve pits. The detailed slope constraints can be found in Section 16. Overall slope angle requirements are respected regardless of inter-ramp angles. While mining occurs on 20-ft benches,

benches are mined to a 40-ft double bench configuration at the pit wall to increase the catch benches to an appropriate width.

- Access: Access to every bench of every phase is incorporated into the pit design. For example, the highest road on the eastern wall of the Yellow Pine pit on Figure 15.5 is incorporated into the pit design to allow access to the West End pit later in the mine life.
- Haul Road Pit Exits: Pit exit locations are chosen to have the haul road exit the pit at the most beneficial location for the haulage of ore to the crusher and waste to the appropriate storage location.
- Realistic Mining Geometries: Computer generated pits have irregular pit walls that will be operationally difficult to mine. Designing pits removes these irregularities and smooths out pit walls.

By incorporating haul roads into the pit design and smoothing irregularities in the pit walls, additional waste rock is often incurred by expanding the pit in the upper benches and some ore that was in the bottom benches of the floating cones is left behind and not mined. For this reason, designed phases often have a higher stripping ratio than optimized cones.

Open pit design criteria are provided in Table 15.7.

Table 15.7 Design Parameters for Mine Pits

Design Parameter	Parameters Value
Haul Road Width Including Ditches and Berms	102 ft
Maximum Haul Road Grade	10%
Bench Height for Mining	20 ft
Face Angle of Benches	64^{o} (Double Benched)
Overall Slope Angles Used	Variable between 39^{o}- 47^{o}
Inter-ramp Slope Angles Used	Variable between 45^{o}- 49^{o}

The design cones for Yellow Pine, Hangar Flats and West End are illustrated on Figure 15.4, Figure 15.6, and Figure 15.8, respectively. The ultimate pits for Yellow Pine, Hangar Flats and West End are shown at the same scale on Figure 15.5, Figure 15.7, and Figure 15.9, respectively, and presented adjacent for comparative purposes.

Figure 15.4 Yellow Pine Cone Floated at $800/oz Gold Price

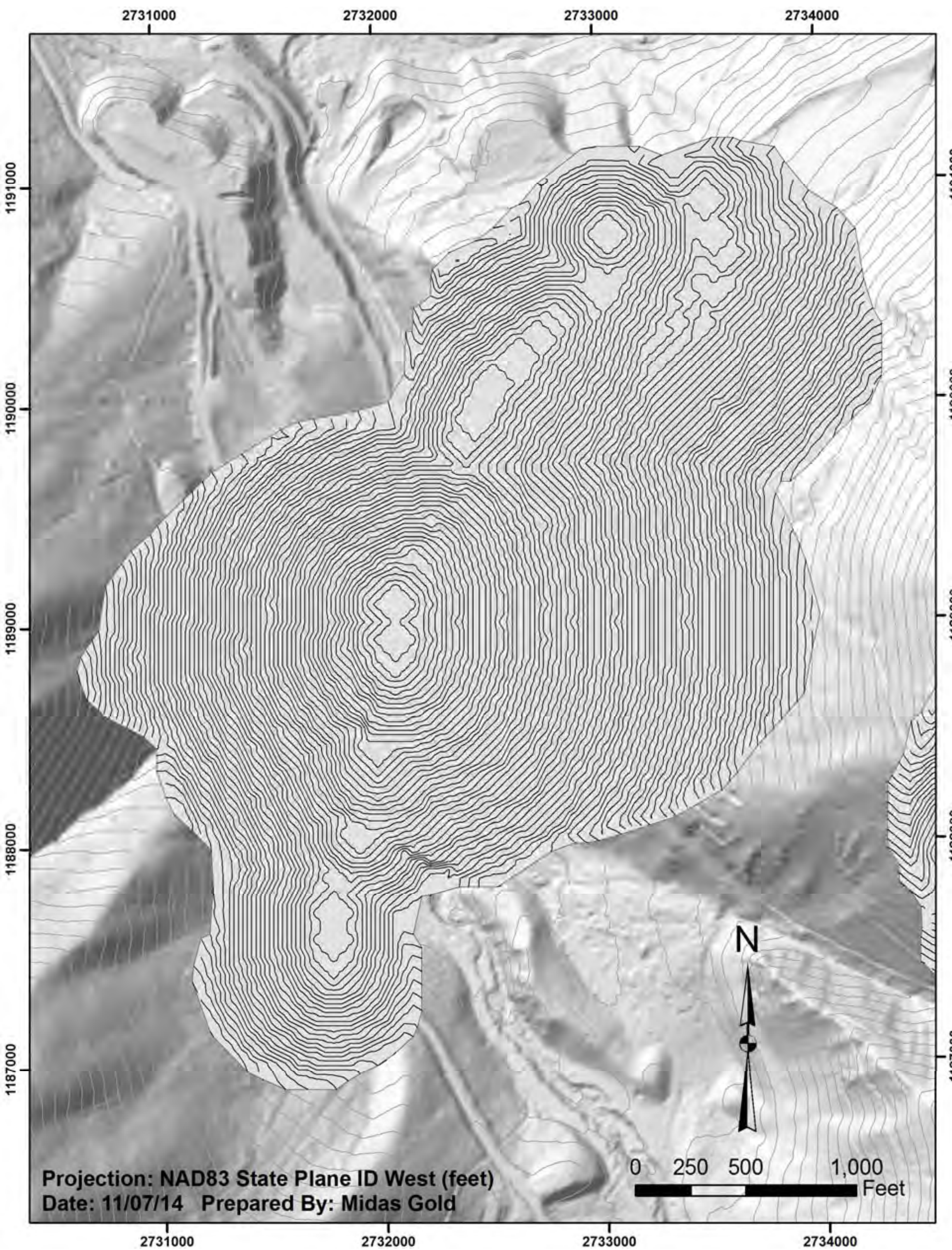


Figure 15.5 Yellow Pine Ultimate Pit

Figure 15.6 Hangar Flats Cone Floated at $1,100/oz Gold Price

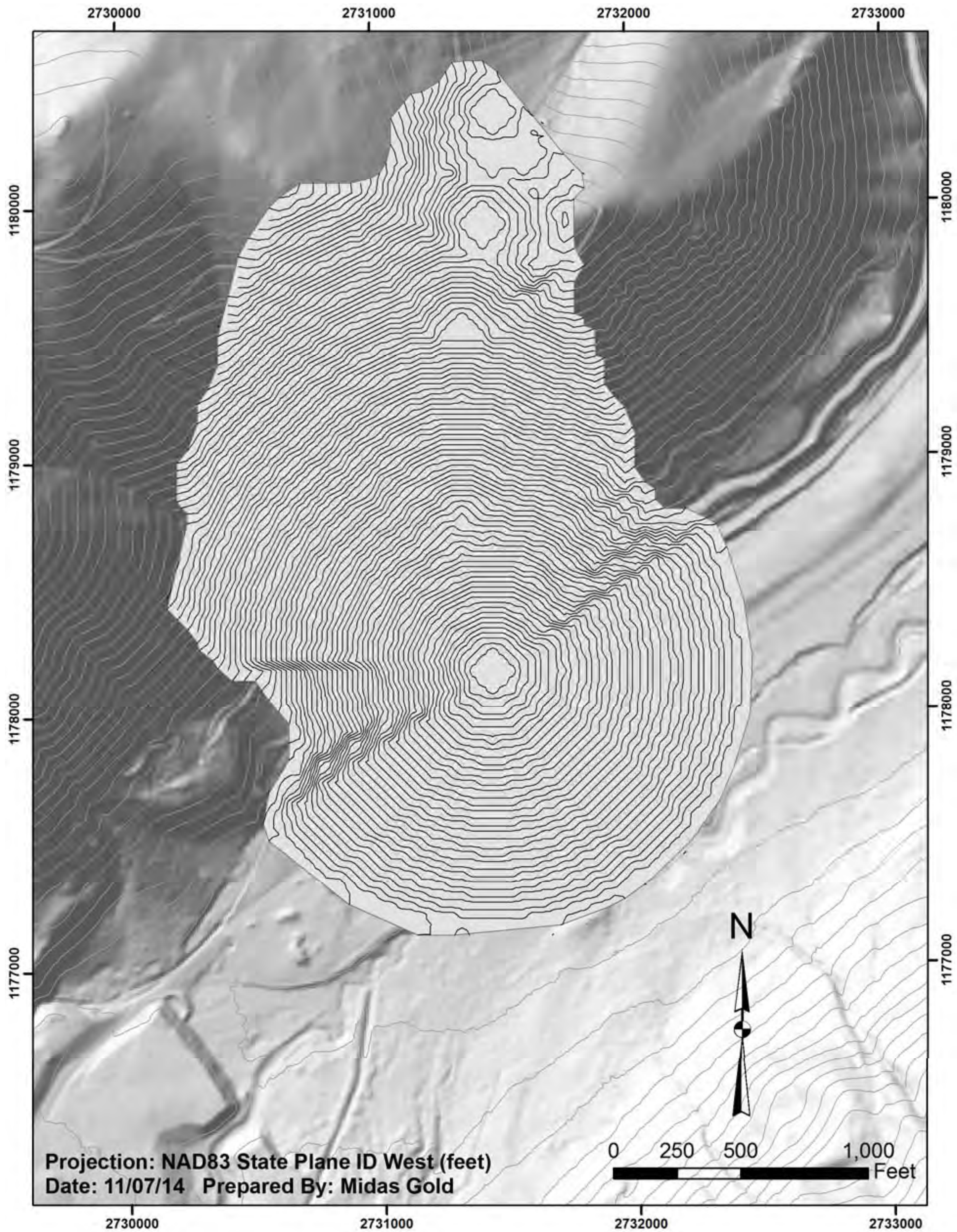


Figure 15.7 Hangar Flats Ultimate Pit

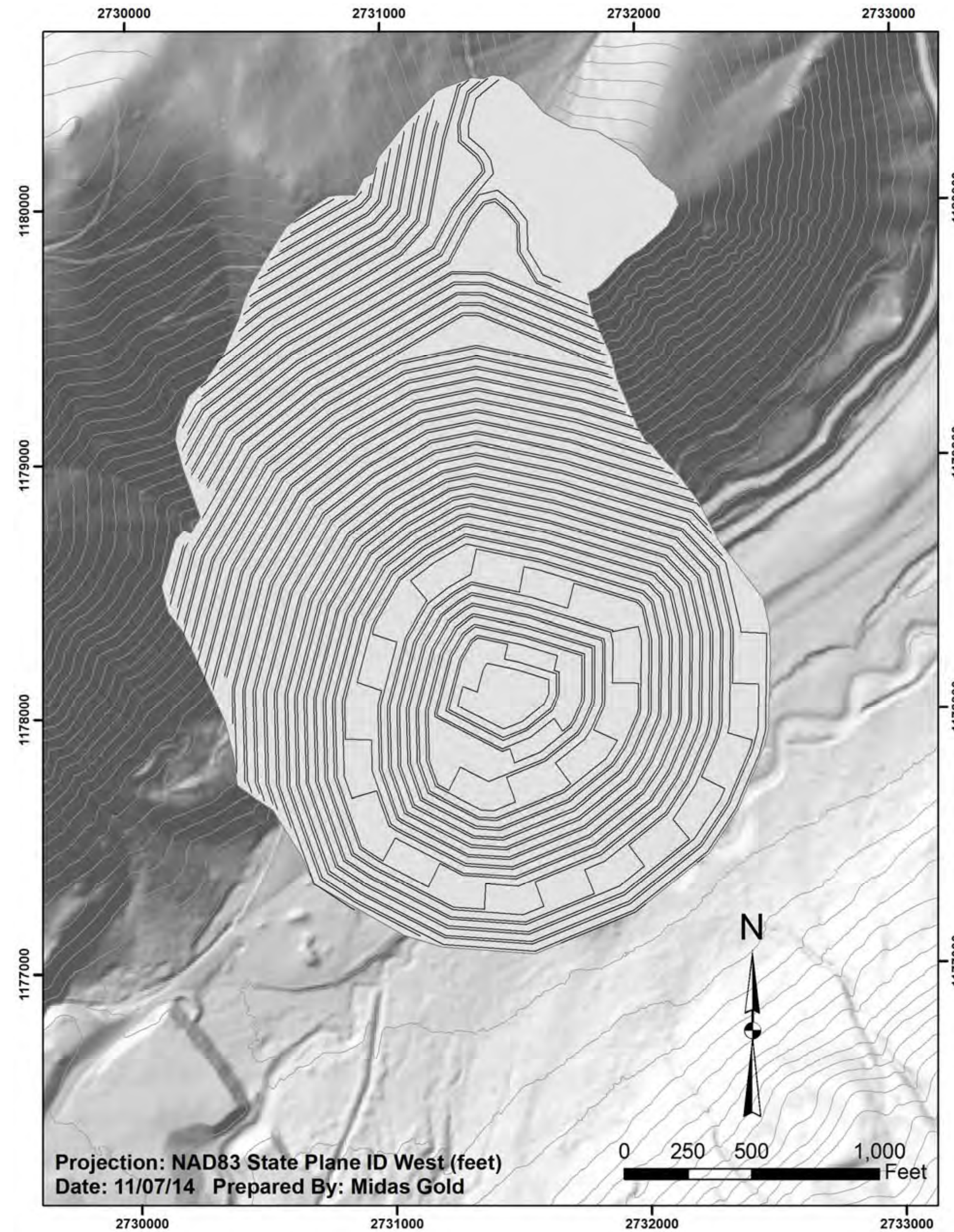


Figure 15.8 West End Cone Floated at $1,100/oz Gold Price

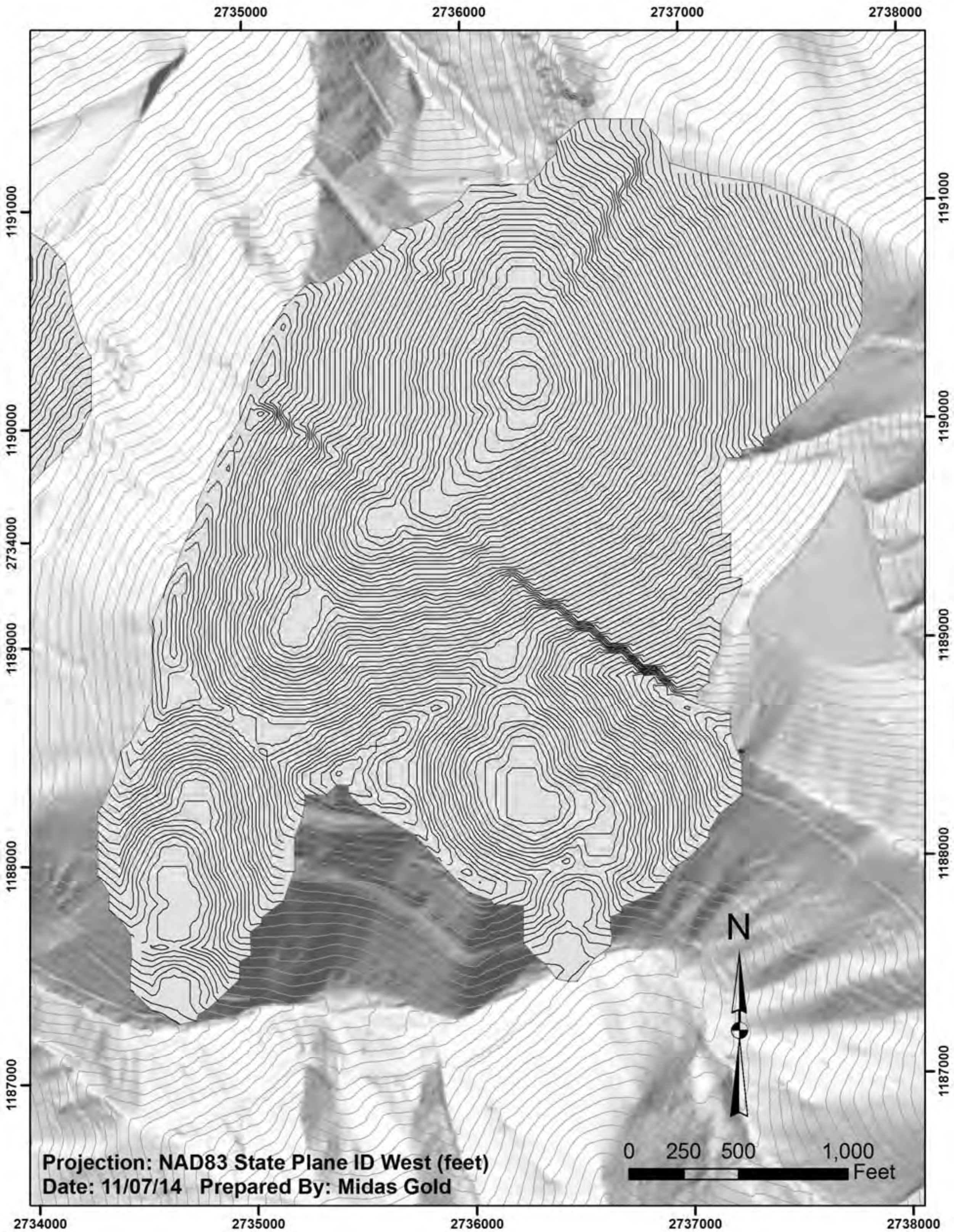


Figure 15.9 West End Ultimate Pit

15.6 HISTORIC TAILINGS

Southwest of the Hangar Flats open pit, and within the planned waste rock storage facility footprint, lies the Historic Tailings impoundment. Metallurgical test results show that the contained gold values in the Historic Tailings produces an economic benefit when fed to the process plant concurrent to primary ores; consequently, the Historic Tailings are planned to be mined and processed through the mill and are included in the Mineral Reserve. The impoundment contains 3,164 kst of Historic Tailings of which 3,001 kst are Indicated Mineral Resources.

15.7 PRE-FEASIBILITY UPDATED COSTS AND RECOVERIES

Final PFS recoveries and processing costs became available towards the end of the PFS work. The block model values were updated with these revised inputs for mine planning purposes. The final PFS inputs used to calculate block NPR values can be found in accompanying sections of this Report. Metal recoveries can be found in Section 13. Transport costs and smelter terms can be found in Section 19. Processing costs used were not final at the time that the Mineral Reserve was defined and those used are provided in Table 15.8; even though the processing costs were revised after the Mineral Reserve was defined, the change in processing costs would not produce a material change in the Mineral Reserve. The final processing costs are incorporated into the Project cash flow included in Section 22. Metal Prices of $1,350/oz. gold, $22.50/oz. silver, and $4.50/lb antimony were used. The logic used to assign process types to each block is similar to the logic that was used for assigning process types for the cone inputs. Updates to processing flow sheets reduced the ore types in West End from three to two: oxide material and mixed sulfide material. The NPR equations are provided in Table 15.9. Ore type is determined by the process that produces the greatest NPR.

IMC conducted a sensitivity check on the impact of incorporating the updated inputs and determined that the existing phase designs are acceptable for use in prefeasibility-level mine planning and no re-design was necessary.

Table 15.8 PFS Processing and G&A Costs by Mineral Resource Type

Mineral Resource Type	Unit Cost
High Antimony Sulfide	$17.00/st
Low Antimony Sulfide	$15.00/st
Oxide	$9.07/st
All Ore Types G&A	$3.40/st

Table 15.9 PFS Logic for Calculating Block Net of Process Values

For Yellow Pine and Hangar Flats Greatest Value of:	High Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) + If Au in Sb Concentrate > 0.146 oz/st: Au Grade x Au loss to Sb Flotation x (Au Smelter Payable – Royalty) x Au Price + If Ag in Sb Concentrate > 8.75 oz/st: Ag Grade x Ag Rec. to Sb Flotation x Ag Smelter Payable x Ag Price + Sb Grade / 100 x Sb Flotation Recovery x Sb Smelter Payable x Sb Price x 2000 - TCRC x Sb Grade x Sb Flotation Recovery / Sb Concentrate Grade - High Antimony Processing Cost
	Low Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) - Low Antimony Processing Cost
	Waste Net of Process	Zero
For West End Greatest Value of:	Low Antimony Mineral Resource Net of Process	Au Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Au Price – Au Refining Charge) x (1 – Royalty) + Ag Grade x (Au Float Rec. + Au Tail Leach Rec.) x POX rec. x % Dore Payable x (Ag Price – Ag Refining Charge) - Low Antimony Processing Cost
	Oxide Mineral Resource Net of Process	Au Grade x % Dore Payable x Oxide Au Recovery x (Au Price – Au Ref Charge) x (1- Royalty) + Ag Grade x % Dore Payable x Oxide Ag Recovery x (Ag Price – Ag Ref Charge) - Oxide Processing Cost
	Waste Net of Process	Zero

Notes:
(1) Au and Ag grades are in oz/st.
(2) % Dore payable is a decimal value.
(3) Royalty is a decimal value.
(4) Au and Ag price is in $/oz.
(5) Sb price is in $/lb.
(6) Au and Ag refining charges are in $/oz.
(7) Processing costs are in $/st.
(8) rec. is recovery in decimal form.

15.8 MINERAL RESERVE ESTIMATE

The designation of Indicated Mineral Resources to the Mineral Reserve category (there are no Measured Mineral Resources) is based on the final PFS process plant metallurgical recoveries, processing costs, and smelter terms. The Mineral Reserve is the sum of the Probable material (there is no Proven material) that is scheduled to be processed in the mine plan that is presented in detail in Section 16. The cutoff grade for material sent to processing ranges from $0.001/st - $8.00/st Net of Process Revenue.

The processing costs used for mine planning ranged from $9.07/st for oxides to $17.00/st for high antimony sulfides with an additional $3.40/st of ore for general and administrative expenses. Therefore, the NSR equivalent of the cutoff grade range is: $12.47/st – $20.40/st Net of Smelter Return. The Mineral Reserves are summarized in Table 15.10 in both imperial and metric units.

Table 15.10: Stibnite Gold Project Probable Mineral Reserves Summary

Deposit	Tonnage	Average Grade			Total Contained Metal		
		Gold	Antimony	Silver	Gold	Antimony	Silver
Imperial Units	**(kst)**	**(oz/st)**	**(%)**	**(oz/st)**	**(koz)**	**(klbs)**	**(koz)**
Yellow Pine	43,985	0.057	0.098	0.090	2,521	86,376	3,973
Hangar Flats	15,430	0.045	0.132	0.086	690	40,757	1,327
West End	35,650	0.035	0.000	0.040	1,265	-	1,410
Historic Tailings	3,001	0.034	0.165	0.084	102	9,903	252
Total Probable Mineral Reserve(1)	**98,066**	**0.047**	**0.070**	**0.071**	**4,579**	**137,037**	**6,962**
Metric Units	**(kt)**	**(g/t)**	**(%)**	**(g/t)**	**(t)**	**(t)**	**(t)**
Yellow Pine	39,903	1.97	0.098	3.10	78.4	39,179	123.6
Hangar Flats	13,998	1.53	0.132	2.95	21.5	18,487	41.3
West End	32,341	1.22	0.000	1.36	39.3	-	43.9
Historic Tailings	2,722	1.17	0.165	2.88	3.2	4,492	7.8
Total Probable Mineral Reserve(1)	**88,964**	**1.60**	**0.070**	**2.43**	**142.4**	**62,159**	**216.5**

Notes:
(1) *Metal prices used for Mineral Reserves: $1350/oz Au, $22.50/oz Ag, $4.50/lb Sb.*
(2) *Block MUST be economical based on gold value only in order to be included as ore in Mineral Reserve.*
(3) *Numbers may not add exactly due to rounding.*

Illustrations of the Yellow Pine, Hangar Flats and West End Mineral Resources and mineralized material that are not part of the reported Mineral Reserves are presented as Figure 15.10, Figure 15.11, and Figure 15.12, respectively.

Figure 15.10: YP Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views

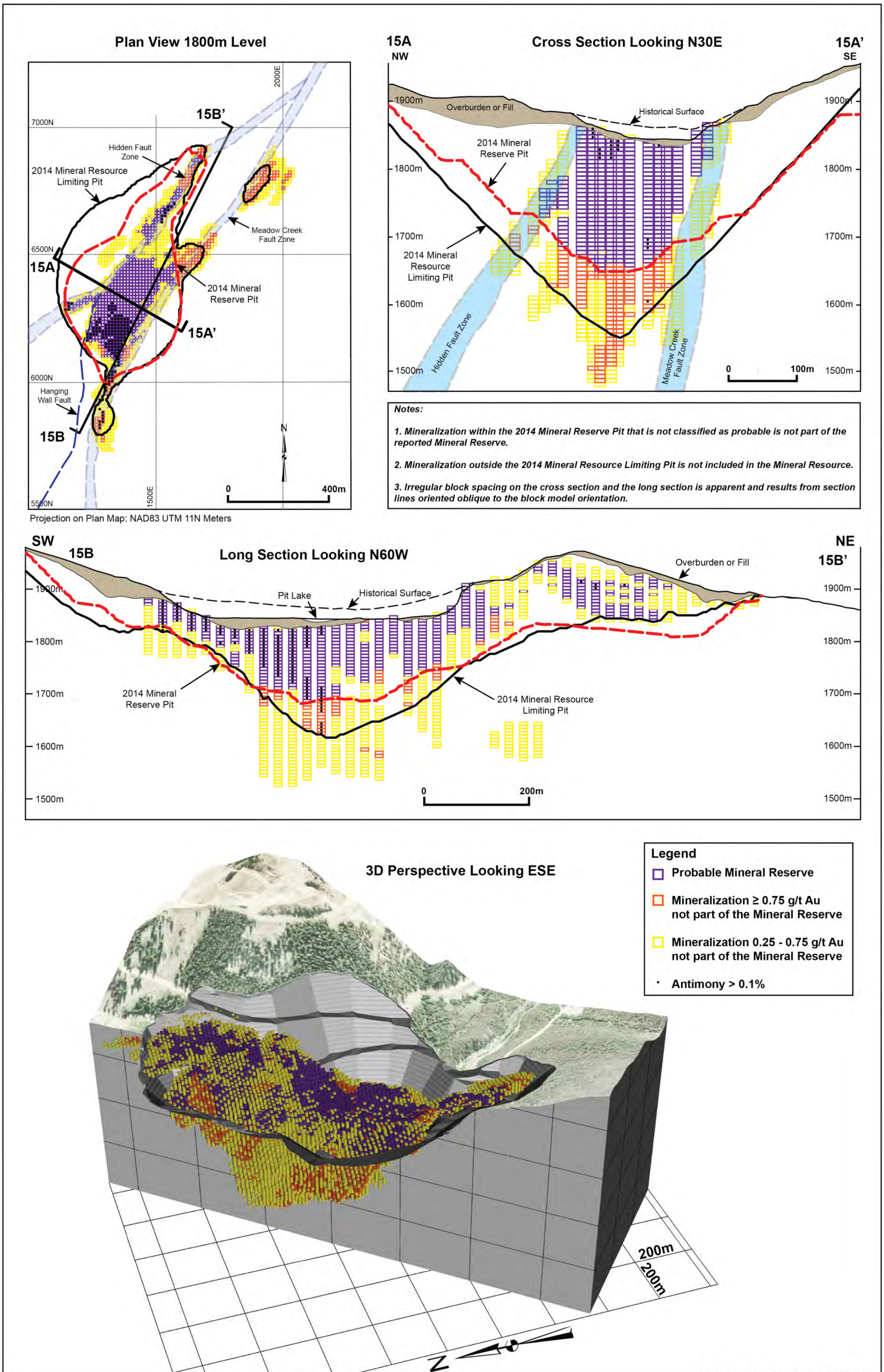


Figure 15.11: HF Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views

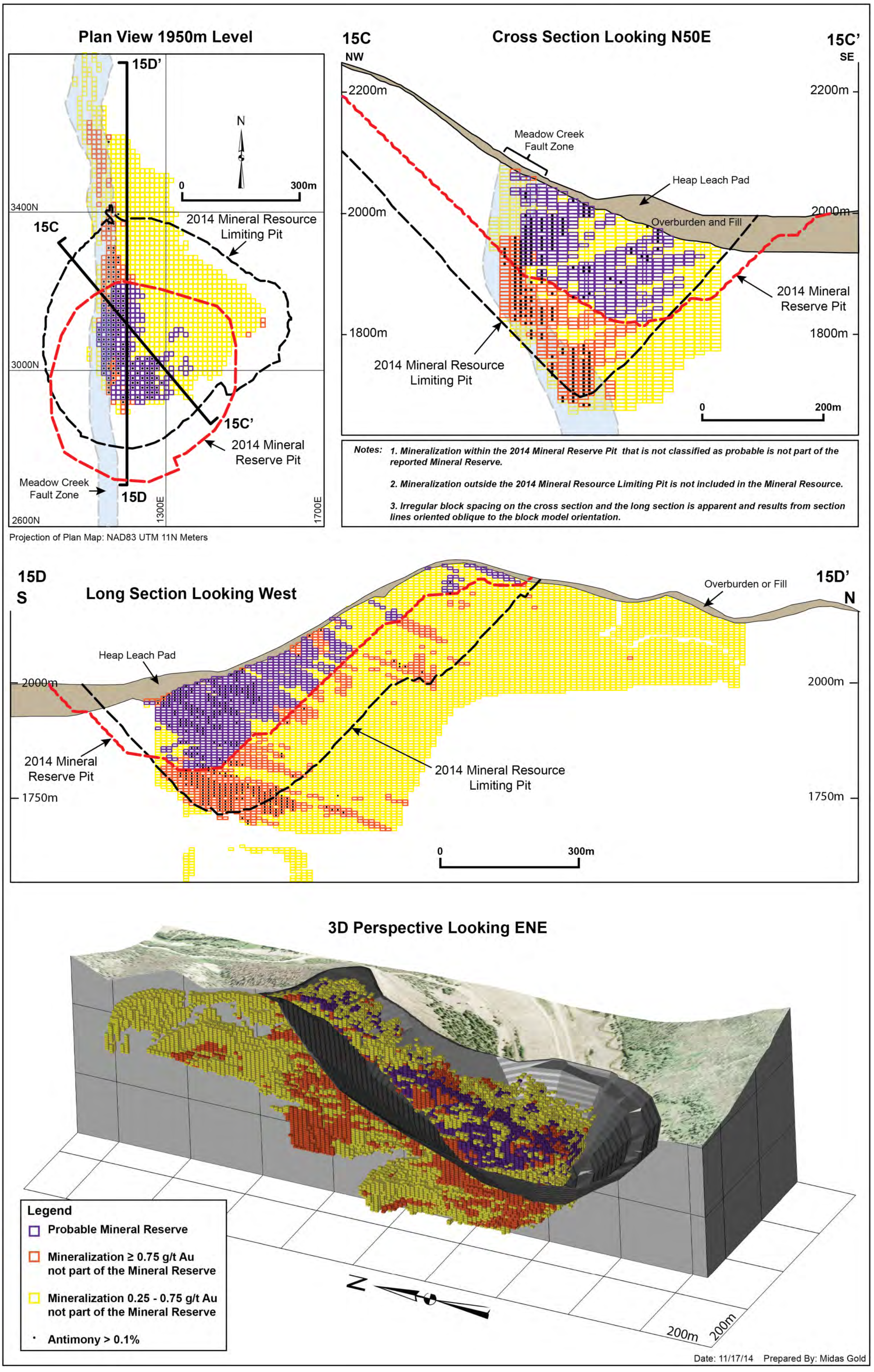


Figure 15.12: WE Mineral Reserves and Mineralized Material in Plan, Section, and 3D Perspective Views