MINE DEVELOPMENT ASSOCIATES
MINE ENGINEERING SERVICES

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PRELIMINARY FEASIBILITY STUDY AND TECHNICAL REPORT FOR THE GRASSY MOUNTAIN GOLD AND SILVER PROJECT, MALHEUR COUNTY, OREGON, USA

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1.0 SUMMARY

Mine Development Associates ("MDA") has prepared this technical report on the Grassy Mountain gold and silver project in Malheur County, Oregon, at the request of Paramount Gold Nevada Corp. ("Paramount"), which is listed on the NYSE American stock exchange (NYSE: PZG). The technical report presents the results of a Preliminary Feasibility Study ("PFS") and includes the first estimate of mineral reserves for the project. Paramount is a reporting issuer in the provinces of Ontario, British Columbia, and Alberta, Canada. Consequently, this report has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

Paramount controls the Grassy Mountain project through its 100% wholly-owned subsidiary Calico Resources USA Corp. ("Calico"), which was formerly a wholly-owned subsidiary of Calico Resources Corp. ("Calico BC"). Paramount acquired Calico BC in July of 2016 by issuing shares in Paramount to the shareholders of Calico BC, and Paramount and Calico BC were subsequently amalgamated.

1.1 Property Description and Ownership

The Grassy Mountain property encompasses approximately 9,300 acres in Malheur County, Oregon about 70 miles west of Boise, Idaho. The geographic center of the property is located at 43.674° N latitude and 117.362° W longitude, and the principal zone of mineralization is located in Section 8 of Township 22 South ("T22S"), Range 44 East ("R44E"), Willamette Meridian. The property consists of 427 unpatented lode claims, nine unpatented mill site claims, six unpatented association placer claims, three patented claims, and two land leases. Annual property holding costs total $107,970.

Calico, a wholly owned subsidiary of Paramount, owns and controls 100% of the mineral tenure of the unpatented mining claims, patented mining claims, Fee lands, and mining leases that comprise the Grassy Mountain property, including all existing exploration and water rights pertaining to the Grassy Mountain project, pursuant to the "Deed and Assignment of Mining Properties" between Seabridge Gold Inc., Seabridge Gold Corporation ("Seabridge") and Calico dated February 05, 2013.

Seabridge retains a 10% Net Profits Interest ("NPI") in the Grassy Mountain project pursuant to the "Deed of Royalties" between Calico and Seabridge dated February 05, 2013. Pursuant to that deed, Seabridge may elect to cause Calico to purchase the 10% NPI for $10M (CAD) within 30 days following the day that Calico has delivered to Seabridge a Feasibility Study on the Grassy Mountain project. A 1.5% royalty on the gross proceeds of the production of minerals from the patented and unpatented claims and a surrounding ½ mile area of interest is held by Sherry & Yates.

The Bishop I and Bishop II Mining Leases, as amended with Bishop et al. ("Bishop") and expiring September 11, 2019, require Annual Minimum royalty payments by Calico, or its assigns, of $30,000 USD (Bishop I) and $3,000 USD (Bishop II). All minimum royalty payments are recoverable against future production royalty payments; records to date indicate that there are accumulated credits of $760,000 and $76,000 that would apply to the Bishop I and Bishop II Leases, respectively. The Bishop I lease includes Fee lands and unpatented placer claims, while the Bishop II lease includes Fee lands. Bishop retains a 6.0% Net Smelter Return ("NSR") royalty based on a gold price above $800 USD per ounce. If ore minerals other than gold are produced, they would be subject to an additional 4.0% NSR royalty. A provision in the Bishop I lease agreement provides for payments to Bishop of $50 for each drill hole on Fee land, $100 for each acre of disturbed Fee land, and $300 for each acre disturbed and lost for Bishop's use.

1.2 Exploration and Mining History

Portions of the Grassy Mountain property were first staked in 1984. After acquiring the property in 1986, Altas Precious Metals ("Atlas") discovered and defined the Grassy Mountain gold-silver deposit, as well as the Crabgrass deposit 1.5 miles to the southwest, through predominantly reverse-circulation rotary ("RC") drilling. Atlas commissioned a 1990 historical feasibility study for an envisioned open-pit heap-leach and milling operation and began to consider underground-mining scenarios, but declining gold prices and the perception of an unfavorable permitting environment discouraged Atlas from developing the project. The property was optioned to Newmont Exploration Ltd ("Newmont") in 1992. Newmont drilled 15 holes in 1994 and completed an in-house economic and mining-method evaluation that was completed in 1995. Newmont determined that the project did not meet corporate objectives and returned the property to Atlas in 1996.

The property was optioned to Tombstone Exploration Company Ltd ("Tombstone") in 1998. Tombstone drilled six holes and returned the property to Atlas in 1998. Seabridge Gold ("Seabridge") completed an acquisition of the Grassy Mountain property from Atlas in 2003. Seabridge did not conduct exploration of the property, and in 2012 Calico Resources Corp. ("Calico") acquired the property. Calico drilled 17 holes before Calico was acquired by Paramount Gold Nevada Corp ("Paramount") in 2016. There has been no historical mineral production from the Grassy Mountain gold-silver deposit.

1.3 Geology and Mineralization

The Grassy Mountain low-sulfidation epithermal hot-spring gold-silver deposit was formed concurrent with fluvio-lacustrine deposition of the Grassy Mountain Formation in the mid-Miocene-age Lake Owyhee volcanic field. Sedimentary units of the Grassy Mountain Formation, which are the host rocks of the deposit, include interbedded conglomerate, sandstone, siltstone, tuffaceous siltstone, and mudstone, as well as several silica sinter deposits. Surface exposures and drilling indicate the host rocks are generally flat-lying to gently arched.

The deposit has extents of 1,900 feet along a N60°E to N70°E axis, as much as 2,700 feet in a northwest-southeast direction, and as much as 1,240 feet vertically. A central higher-grade core with gold grades generally in excess of ~0.03 oz Au/ton coincides with the axis of the Grassy fault, and it is surrounded by a broad envelope of lower-grade mineralization. The central higher-grade core is almost 1,000 feet long along the N60°E to N70°E axis, by 450 feet in width and 450 feet in vertical extent.

Three distinct and overlapping types of gold-silver mineralization are recognized within the central core of the deposit: gold-bearing, chalcedonic quartz ± adularia veins; disseminated mineralization in silicified siltstone and arkose; and gold and silver in bodies of "clay matrix breccia".
Colliform-banded veins tend to carry the highest grades (>0.5 oz Au/ton), in some cases with electrum along the vein margins or within microscopic voids. Some veins carry very little grade or are barren. Vein widths generally range from 1/16 to ~2.0 inches, and vein frequency can average one vein per foot in places, but any individual vein is unlikely to have lateral or vertical extents of significance.

High gold grades are also present in silicified siltstone and arkose with no visible veins. In these cases, gold and silver are inferred to be very finely disseminated in a stratiform manner in the silicified rock. Fine-grained pyrite is commonly disseminated in the silicified siltstone and sandstone where oxidation has not occurred.

Gold-silver mineralized clay matrix breccias are mainly of clast-supported types. Fragments consist of sub-rounded to sub-angular, sand- to boulder-sized clasts of silicified and/or veined arkose and siltstone, with minor amounts of clay and iron-oxide minerals between the clasts. Their true thickness and exact orientations are poorly understood because their margins are commonly irregular-to-gradational, as opposed to planar. The clay matrix breccia mineralization may be more prevalent in the lower portion of the higher-grade core of the deposit, and individual bodies of this material are interpreted to extend at near-vertical angles up and down into the surrounding, low-grade envelope.

1.4 Metallurgical Testing and Mineral Processing

The most recent metallurgical testing was completed in 2017 as part of this PFS. This included tests using nine different composites based on representative lithologies and grade ranges. The results demonstrate that the Grassy Mountain mineralization is free-milling and can be processed with gravity concentration followed by conventional cyanide leaching of the gravity tails. Results from the 2017 test program are consistent with historical testing. A conservative interpretation of the results estimates a gravity recovery of 8.6% of the gold. Carbon-in-leach ("CIL") has been selected for this project for the processing of the gravity tails and is estimated to achieve a gold recovery of 84.9%, for an overall combined gold recovery of 93.5%.

Comminution testing from 2017 showed the samples to be classified as hard. The crusher work index determined from these tests, 21.2 kWh/ton, was used to select primary, secondary, and tertiary crushers. From historical testing, a Bond ball-mill work index of 19.0 kWh/ton (75th percentile value of available data) was used to select the ball mill, along with a feed size of 80% passing 0.39" and the product size of 80% passing 100 mesh.

1.5 Mineral Resource Estimate

The Grassy Mountain gold and silver mineral resources were modeled and estimated by:

- evaluating the drill data statistically;
- separately interpreting gold and silver mineral domains on a set of 070°-looking cross sections spaced at 50-foot intervals, and using these sections to code the drill-hole database;
- rectifying the cross-sectional mineral-domain interpretations on level plans spaced at 10-foot vertical intervals, and using these level plans to code the resource block model;
- analyzing the modeled mineralization spatially and statistically to aid in the establishment of estimation and classification parameters; and
- interpolating grades into the block model, using the coding of the level-plan gold and silver mineral domains to constrain the estimations.

The Grassy Mountain resources have been estimated to reflect potential open-pit extraction and milling, as well as potential underground mining of material lying outside of the resource pit shell. To define the open-pit resources, a pit optimization was run using the parameters summarized in Table 1.1, and a gold-equivalent cutoff grade of 0.012 oz Au/ton was applied to all material withing the pit shell.

Table 1.1 Pit Optimization Parameters

Mining Cost	$	2.00	$/ton
Processing Cost	$	13.00	$/ton processed
Tons per Day		5,000	tons-per-day processed
G&A per Ton	$	2.22	$/ton processed
Au Price	$	1,500	$/oz
Ag Price	$	20	$/oz
Au Recovery		80%	
Ag Recovery		60%	
Au Refining Cost	$	5.00	$/oz produced
Ag Refining Cost	$	0.50	$/oz produced

Underground resources were estimated by applying a a gold-equivalent cutoff grade of 0.060 oz/ton to blocks lying immediately outside of the optimized pit. Table 1.2 lists the parameters used to calculate the underground cutoff grade.

Table 1.2 Parameters Used to Determine Underground Resource Cutoff Grade

Mining Cost	$	50.00	$/ton
Processing Cost	$	25.00	$/ton processed
Tons per Day		5,000	tons-per-day processed
G&A per Ton	$	8.00	$/ton processed
Au Price	$	1,500	$/oz
Ag Price	$	20	$/oz
AuEq Recovery		90%	
Refining Cost	$	5.00	$/oz produced

The gold equivalent grade ("oz AuEq/ton") of each model block was calculated by dividing the silver grade by 100 and adding it to the gold grade. The silver-to-gold equivalency factor of 100 was derived from the metal prices and recoveries in Table 1.1.

The total Grassy Mountain project gold and silver resources, which are dominated by the in-pit resources, are presented in Table 1.3. The resources are inclusive of the project mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1.3 Grassy Mountain Gold and Silver Resources

Classification	Tons	oz Au/ton	oz Au	oz Ag/ton	oz Ag
Measured	17,933,000	0.020	363,000	0.079	1,409,000
Indicated	12,886,000	0.054	695,000	0.146	1,882,000
Measured + Indicated	30,819,000	0.034	1,058,000	0.107	3,291,000
Inferred	1,055,000	0.040	42,000	0.119	125,000

1. *Mineral Resources are comprised of all model blocks at a 0.012 oz AuEq/ton cutoff that lie within an optimized pit, plus blocks at a 0.060 oz AuEq/ton cutoff that lie outside of the optimized pit.*
2. *The mineral resources are inclusive of the project mineral reserves.*
3. *Mineral resources that are not mineral reserves do not have demonstrated economic viability.*
4. *The Effective Date of the Grassy Mountain resource estimate is May 1, 2018.*
5. *Rounding may result in apparent discrepancies between tons, grade, and contained metal content.*

1.6 Mineral Reserves

Modifying factors were applied to the Measured and Indicated mineral resources presented in Table 1.3 to estimate the Proven and Probable mineral reserves for the Grassy Mountain project. The estimated Proven and Probable mineral reserves (Table 1.4) contain 1.72 million tons at an average grade of 0.210 oz Au/ton and 0.30 oz Ag/ton, for 362,00 contained ounces of gold and 516,000 contained ounces of silver. Mineral reserves are included in the estimated Measured and Indicated mineral resources. The Effective Date of the estimated mineral reserves is May 1, 2018.

Table 1.4 Mineral Reserve Statement

Classification	Tons (Million)	Gold Grade oz Au/ton	Silver Grade oz Ag/ton	Contained Metal (oz Au)	Contained Metal (oz Ag)
Proven	0.23	0.191	0.27	43,000	62,000
Probable	1.49	0.214	0.30	319,000	454,000
Proven + Probable	1.72	0.210	0.30	362,000	516,000

1. *Mineral reserves have an Effective Date of May 1, 2018.*
2. *Mineral reserves are reported using the 2014 CIM Definition Standards.*
3. *Mineral reserves are reported inside stope designs assuming drift-and-fill mining methods, and an economic gold cutoff grade of 0.103 oz Au per ton. The economic cutoff grade estimate utilizes a gold price of $1,275/oz, mining costs of $80/ton processed, surface rehandle costs of $0.16/ton processed, process costs of $30/ton processed, general and administrative costs of $11.11/ton processed, and refining costs of $5/oz Au recovered. Metallurgical recovery is 94.5% for gold. Mining recovery is 95% and mining dilution is assumed to be 10.5%. Mineralization that was either not classified or was assigned to Inferred mineral resources was set to waste. A 1.5% NSR royalty is payable. The reserves reference point is the PFS mill crusher.*
4. *Mineral reserves are included in Measured and Indicated resources; tonnage and contained metal have been rounded to reflect the accuracy of the estimate. Apparent discrepancies are due to rounding.*

1.7 Mining Methods

Extraction of the estimated mineral reserves is planned via a proposed underground mine that will be accessed via one decline and a system of internal ramps. Two shafts are planned for ventilation and secondary egress. The planned mining method is drift-and-fill with diesel-powered mining equipment. Cemented rock fill and uncemented rock fill will be used for backfill.

The mine design is based on a production rate of 1,300 to 1,400 tons per day over four days per week, with two shifts per day, to provide sufficient material to feed the 750 tons per day to the mill on a seven day per week basis. The nominal development size is 15 feet wide by 15 feet high for the main decline, 13 feet wide by 13 feet high for horizontal access to production areas, and the production headings will be 20 feet wide x 13 feet high. Ground support was designed to maintain a safe operation.

1.8 Recovery Methods

The Grassy Mountain gold-silver mineralization is considered to be amenable to a combination of gravity concentration and cyanide leaching. A 750 ton per day process plant has been designed to recover and concentrate gold and silver. The plant will be a conventional CIL type and is designed to operate with two shifts per day, 365 days per year, with an overall plant availability of 91.3%. The process plant will produce gold doré bars to be sold to gold refiners.

The plant feed will pass through a jaw crusher as the primary stage and cone crushers for secondary and tertiary size reduction, and then will be ground by a ball mill in a closed circuit with hydro-cyclones. A centrifugal gravity concentrator will collect gravity-recoverable gold from the cyclone underflow and discharge it to an intensive-leach reactor for recovery. The hydro-cyclone overflow with P_{80} of 100 mesh will flow to a CIL recovery circuit via a pre-aeration reactor.

Gold and silver leached in the CIL circuit will be recovered on carbon and eluted in a pressurized Zadra-style elution circuit, then precipitated by electrowinning and smelted in a refining furnace to pour doré bars.

Cyanide in the tailings will be destroyed in an SO_2/air circuit. Detoxified tails will be pumped to a tailings storage facility for final deposition and recovery of decant water. Process water recovered from the decant water will be re-used for grinding and plant utility water.

1.9 Infrastructure

Provisions for infrastructure include: 17 miles of main access road, security fencing, water supply and distribution piping, fuel handling facilities, communications, buildings, explosives storage and handling, borrow source, electrical power supply and distribution, and a tailings storage facility. The general arrangement of the project surface facilities is shown in Figure 1.1.

1.9.1 Tailings Storage Facility

Tailings produced during mineral processing will be conveyed and disposed of in a tailing storage facility ("TSF") west of the mill site. The TSF is fully-lined and provides sufficient storage capacity to contain all tailings produced during the PFS mine production life.

The TSF consists of a dual containment lining system, water recovery, collection, return systems, and storm-water controls. Preliminary design by Golder Assoicates Inc. ("Golder") includes construction of the TSF in stages. Each stage increases the storage capacity of the facility by increasing dam embankment heights and expanding the impoundment basin.

Using data provided to, and additional data collected by, Golder, the TSF is designed to be zero-discharge facility during normal and upset conditions and remain geotechnically stable during the design seismic event.

Figure 1.1 Grassy Mountain Project General Arrangement

1.10 Environmental Studies and Permitting

Calico is in the process of acquiring the necessary local, state, and federal permits for the development of an underground-mining and mill-processing operation at Grassy Mountain. Permitting activities began in 2012 and baseline data collection is ongoing as of the date of this report.

The project will require the following major environmental permits to construct, operate, and close: 1) a Plan of Operations ("Plan") from the BLM; 2) an Oregon Department of Geology and Mineral Industries ("DOGAMI") Consolidated Permit for Mining Operations (Chapter 632, Division 37); 3) an Oregon Department of Environmental Quality ("ODEQ") Chemical Mining Permit (Chapter 632, Division 43); 4) Water rights from the Oregon Department of Water Resources ("ODWR"); 5) an Air Quality Operating Permit ("AQOP") with the ODEQ; and 6) a Conditional Use Permit from Malheur County. The Division 37 Rules (Chapter 632, Division 37, 1991 Oregon Laws (§632-037-0005) provide a well-defined regulatory pathway with definitive permitting requirements and timelines.

An updated Plan of Operations ("Plan") was submitted to the U.S. Bureau of Land Management ("BLM") in 2017. The Plan outlines approximately 265 acres of proposed surface disturbance for the planned underground mine, process plant, waste-rock storage, tailings storage, ore stockpile, water-well sites and distribution system, electrical power substation and distribution system, ancillary facilities, reclamation, and closure. The National Environmental Policy Act ("NEPA") is triggered by the BLM issuance of a completeness letter for the Plan, and BLM has stated that the NEPA review process at Grassy Mountain will be an EIS. Under 2018 Secretarial Order 3355, the EIS must be completed in 365 days and must be less than 150 pages, unless a waiver is obtained. The Draft EIS is prepared for the BLM by a third-party contractor and BLM has chosen HDR Engineering Inc.

Calico has agreed to reimburse DOGAMI and other state agencies for their involvement in processing permit applications for the Grassy Mountain project. An interagency Technical Review Team ("TRT") has been organized to provide interdisciplinary review of technical permitting issues for Oregon's Consolidated Permitting Process. Calico has filed multiple Notices of Intent ("NOI"s), which initiate the state permitting process and baseline data collection and the TRT has accepted the NOIs. In addition, DOGAMI and the TRT have reviewed and approved the *Calico Resources Environmental Baseline Work Plans Grassy Mountain Mine Project*, which was filed May 17, 2017. Calico is working on the baseline studies.

The Oregon Division 37 Consolidated Permit Application is a single application which includes the following: 1) General information, 2) Existing environment-baseline data, 3) Operating plan, 4) Reclamation and closure plan, and 5) Alternative analysis. The key components of the Calico permitting program with the State of Oregon are as follows: A) Environmental baseline studies for all resource categories; B) Meeting all requirements of Division 37 Rules which include, but are not limited to: *i*) preparation of a Consolidated Permit Application; *ii*) obtaining all necessary federal, state, and local permits and authorizations; and *iii*) satisfying any potentially applicable NEPA requirements; and C) implementing a pro-active community involvement and consultation process including: *a*) local hire preference; *b*) local contracting and purchase were practicable; and *c*) mine-worker job training to provide an experienced workforce. After Calico submits the Consolidated Permit Application, the TRT and DOGAMI will conduct a completeness review. Following issuance of a Notice to Proceed, DOGAMI will initiate an Environmental Evaluation ("EE") which includes 1) impact analysis, 2) cumulative impact analysis, and 3) alternatives analysis. The EE is separate from the BLM's NEPA process.

The ODEQ Chemical Mining Permit, Water Rights from ODWR, and the AQOP with ODEQ will be issued based on information submitted in the Consolidated Permit Application. The local permitting process, Conditional Use Permit application, will begin following the completion of this PFS. Other applicable state and federal permits may include, but they are not limited to, the following: 1) Permits to appropriate groundwater or surface water, or to store water in an impoundment; 2) Water Pollution Control Facility; 3) Storm Water Pollution Prevention Plan (EPA); 4) Air Quality Permits; 5) Solid Waste Disposal Permit; 6) Permit for Placing Explosives Hazardous Waste Storage Permit; 8) Land Use Permit; and 9) Any other state permits, if applicable and required under Division 37.

There are no known, ongoing, environmental issues with any of the regulatory agencies. Waste-rock characterization tests have been conducted, with results indicating the waste rock and ore are generally reactive, acid-generating, and have the potential to leach metals. There are no known social or community issues that would have a material impact on the project's ability to extract mineral resources, with identified socioeconomic impacts anticipated to be positive.

A valid exploration permit currently exists with DOGAMI and the BLM. A bond in the amount of \$146,200 is associated with the exploration permit. An existing Notice with the BLM for 2.78 acres of surface disturbance and a monitor well has an associated bond in the amount of \$25,315. An application for "Extension of Time for a Water Right Permit" was filed with the Oregon Water Resources Department. Paramount has until October 1, 2028 to complete the water system and apply water to beneficial use. The company must submit progress reports on October 1 of 2022 and 2027.

1.11 Capital and Operating Costs

Ausenco estimated processing and certain infrastructure costs, Ausenco estimated mining costs, Golder estimated the tailings storage facility costs, and MDA estimated general and administration ("G&A") costs and infrastructure costs. Table 1.5 summarizes the project capital costs. Total initial capital is estimated to be \$109.9 million. Sustaining costs of \$1.1 million have been estimated, which include a refund of cash contribution toward surety bonding. The total life-of-mine ("LOM") capital cost estimate is \$110.9 million. Note that negative capital assumes bounding and working capital is returned to the cash flows as part of sustaining capital.

Table 1.6 shows the estimated operating costs for the LOM. Total operating costs are \$185.8 million. This results in a \$105.63 cost per ton processed or \$528.12 per gold equivalent ounce produced.

Table 1.5 Capital Cost Summary (K USD)

	Initial	Sustaining	Total
Mining Capital	$ 2,928	$ 1,399	$ 4,328
Buildings & Site Infrastructure	$ 12,787	$ -	$ 12,787
Process Capital	$ 25,935	$ -	$ 25,935
Tailings Storage Facility	$ 8,215	$ 5,026	$ 13,241
Plant & Infrastructure Indirect	$ 9,691	$ -	$ 9,691
Off-Site Power and Access	$ 10,328	$ -	$ 10,328
Subtotal Infrastructure & Equipment	***$ 69,885***	***$ 6,426***	***$ 76,311***
Mine Development	$ 7,640	$ 1,799	$ 9,439
Mine Pre-Production	$ 4,598	$ -	$ 4,598
Subtotal Mine Pre-Production	***$ 12,238***	***$ 1,799***	***$ 14,037***
Owner's Capital	$ 7,005	$ (4,142)	$ 2,863
Other Capital	$ 2,092	$ 166	$ 2,259
Working Capital	$ 4,464	$ (4,464)	$ -
Subtotal Other Capital	***$ 13,561***	***$ (8,439)***	***$ 5,122***
Subtotal	***$ 95,684***	***$ (214)***	***$ 95,470***
Contingency	$ 14,195	$ 1,282	$ 15,477
Total Capital	***$ 109,880***	***$ 1,067***	***$110,947***

Table 1.6 Operating Cost Summary (USD)

	Life-of-Mine Cost (K USD)	Cost/ton Processed	Cost per Oz Au *
Mining	$ 114,969	$ 65.37	326.81
Processing	$ 49,332	$ 28.05	140.23
G&A	$ 15,275	$ 8.68	43.42
Reclamation	$ 6,213	$ 3.53	17.66
Total	$ 185,789	$ 105.63	528.12

** Cost per ounce includes silver revenue as credits*

1.12 Economic Analysis

An economic analysis of the project was completed. The metal prices used for the economic analysis include $1,300 per ounce of gold sold and $16.75 per ounce of silver sold. Economic highlights include:

- Average annual gold production of 46,996 ounces per year;
- Average annual silver production of 49,886 ounces per year;
- 7.25-year mine life;
- 341,000 total recovered ounces of gold (340,000 ounces sold);
- 362,000 total recovered ounces of silver (360,000 ounces sold);

- 345,000 gold-equivalent ounces produced[3]
- $528 cash operating cost per ounce of gold[1]
- $853 total cost per ounce of gold[2]
- 27.6% internal rate of return;
- $87,754,000 after-tax NPV (5%);
- $70,621,000 after-tax NPV (8%);
- 60,455,000 after-tax NPV (10%); and
- 2.51-year payback period (from start of production).

[1] *Includes silver revenues as credit*
[2] *Includes silver revenues as credit and all capital*
[3] *Gold equivalent based on ounces of gold and silver produced and gold to silver ratio of $1,300:$16.75*

1.13 Economic Sensitivity Analysis

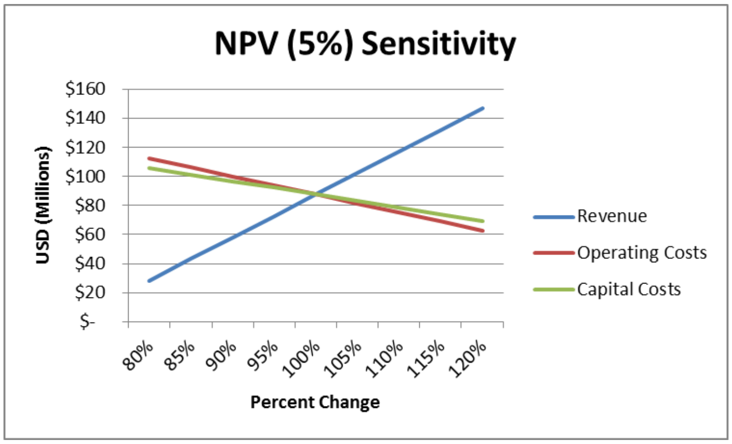
Pre-tax and after-tax cash-flow sensitivities to revenue were evaluated by varying the gold price from $1,200 to $1,500 per ounce in $50.00 increments. The silver price was also modified in these sensitivities based on a constant gold-to-silver price ratio of $1,300 to $16.75 (77.6:1 gold-to-silver price ratio). After-tax metal price sensitivities are shown in Table 1.7.

After-tax sensitivities to changes in revenues, operating costs, and capital costs are shown in Figure 1.2. The Grassy Mountain project is most sensitive to changes in metal prices, but at prices greater than $1,100 per ounce of gold, the project retains a fairly robust return.

Table 1.7 Project Cash-Flow Sensitivity to Gold Price

Au ($/oz Au)	Ag ($/oz Au)	IRR	NPV 5%	NPV 8%	NPV 10%	Payback Start of Prod
$ 1,200	$ 15.46	22.1%	$ 64,871	$ 49,714	$ 40,714	2.91
$ 1,250	$ 16.11	24.9%	$ 76,336	$ 60,200	$ 50,622	2.69
$ 1,300	$ 16.75	27.6%	$ 87,754	$ 70,621	$ 60,455	2.51
$ 1,350	$ 17.39	30.2%	$ 99,132	$ 80,987	$ 70,227	2.35
$ 1,400	$ 18.04	32.8%	$ 110,511	$ 91,354	$ 79,998	2.20
$ 1,450	$ 18.68	35.4%	$ 121,890	$ 101,720	$ 89,770	2.07
$ 1,500	$ 19.33	37.8%	$ 133,243	$ 112,050	$ 99,499	1.97

Figure 1.2 Project Cash-Flow Sensitivities



1.14 Conclusions and Recommendations

The authors believe that the Grassy Mountain project is a project of merit that should be considered for a Feasibility Study. An exploration program that includes drilling of targets with the potential to provide additional mill feed is also warranted. The approximate costs of the recommended work are shown on Table 26.1, which is followed by relevant details.

Table 1.8 Recommended Work Program

Category	Estimated Cost $
Exploration Drilling (19,500ft RC); includes assays, site prep, supplies, down-hole surveys	$ 1,100,000
Surface Exploration (CSAMT, soil sampling, trenching)	$ 150,000
Exploration Geology	$ 50,000
Feasibility Study	**$ 2,000,000**
Total	**$ 3,300,000**

2.0 INTRODUCTION AND TERMS OF REFERENCE

Mine Development Associates ("MDA") has prepared this technical report on the Grassy Mountain gold and silver project, located in Malheur County, Oregon, at the request of Paramount Gold Nevada Corp. ("Paramount"), a company based in Winnemucca, Nevada, USA, and listed on the NYSE American (NYSE: PZG). Paramount controls the Grassy Mountain project through its 100% wholly-owned subsidiary Calico Resources USA Corp. ("Calico"), which was formerly a wholly-owned subsidiary of Calico Resources Corp. ("Calico BC"). Paramount acquired Calico BC in July 2016 by issuing shares in Paramount to the shareholders of Calico BC, and Paramount and Calico BC were subsequently amalgamated. For this report, "Paramount" refers to both Paramount and Calico, unless specifically stated otherwise.

Paramount is a reporting issuer in the provinces of Ontario, British Columbia, and Alberta, Canada. Consequently, this report has been prepared in accordance with the disclosure and reporting requirements set forth in the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101"), Companion Policy 43-101CP, and Form 43-101F1, as well as with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Reserves, Definitions and Guidelines" ("CIM Standards") adopted by the CIM Council on May 10, 2014.

2.1 Project Scope and Terms of Reference

The purpose of this report is to provide a technical summary of the Grassy Mountain gold and silver project in support of a Preliminary Feasibility Study ("PFS") and first estimate of mineral reserves that are based on an updated mineral resource estimate that includes Paramount's 2016 - 2017 drill data. The most recent previous estimate of mineral resources at Grassy Mountain was made in 2015 by Metal Consultants Inc. (Wilson et al., 2015a; 2015b).

The mineral resources that are the subject of this technical report were estimated and classified under the supervision of Michael M. Gustin, Ph.D., C.P.G., and Senior Geologist for MDA. The mineral reserves were estimated and classified by Boris Caro, Aus.I.M.M., an independent mining engineering consultant and associate of Ausenco, a multi-national mining engineering consultancy firm. The mineral resources and reserves reported herein are estimated in accordance with the standards and requirements stipulated in NI 43-101.

The scope of this study included a review of pertinent technical reports and data provided to the authors by Paramount relative to the general setting, geology, project history, exploration activities and results, methodology, quality assurance, interpretations, drilling programs, and metallurgy. This work culminated in the estimation of mineral resources and reserves.

Table 2.1 lists the authors of this report, all of which are qualified persons, as well as the sections of the report for which they are responsible and the date of their most recent site inspection, where applicable.

Table 2.1 Qualified Persons, Dates of Most Recent Site Visits, and Report Responsibilities

Company	Author	Professional Designation	Date of Most Recent Site Visit	QP Responsibilities by Report Section
Mine Development Associates	Tom Dyer	P.E.	8/18/2016	1.9, 1.11, 1.12, 1.13, 16.10, 18, 19, 21, 22, 25.7, 25.9, 25.10, 25.11
	Michael Gustin	C.P.G.	6/1/2017	1.1, 1.2, 1.3, 1.5, 1.10, 1.14, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 20, 23, 24, 25.1, 25.2, 25.3, 25.8, 26.1, 27, 28, 29
Ausenco	David Baldwin	P.Eng.	6/8/2018	18.2.4 thru 18.2.9; 18.3.3; 25.7
	Tommaso Roberto Raponi	P. Eng.	n/a	1.4, 1.8, 13, 17, 25.4, 25.6, 26.2.1, 26.2.2
	Boris Caro	Aus.I.M.M.	n/a	1.6, 1.7, 15, 16 (except 16.10), 21.1.1, 21.2.1, 25.5, 26.2.4
Golder Associates Inc.	Chris MacMahon	P.E.	8/18/2016	1.91,18.4, 25.6, 25.11.1.2, 25.11.2.3, 26.2.3

Mr. Gustin, Mr. Dyer, Mr. Baldwin, Mr. Raponi, Mr. Caro, and Mr. MacMahon are qualified persons under NI 43-101 and have no affiliations with Paramount except that of independent consultant/client relationships.

In addition to the site inspections shown in Table 2.1, Mr. Gustin also accompanied Mr. Dyer and Mr. MacMahon on their August 18, 2016 visit to the project site, which was led by senior technical staff of Paramount. This joint site visit provided a general overview of the Grassy Mountain deposit area, including access to the project, potential surface infrastructure locations, and the site of the proposed portal for the underground mine access to the project. Pertinent geological aspects of the project were also discussed. Mr. Baldwin's site visit on June 8, 2018 was conducted similarly. These site visits also included additional time at Paramount's core storage and field office facilities in Vale, Oregon, which was used to further review technical aspects of the project and inspect representative drill core.

Mr. Gustin completed additional site inspections on November 17, 2016 and June 1, 2017. These site visits included traverses across the Grassy Mountain deposit area, inspection of numerous exposures of altered rock units that host the gold-silver mineralization, and the monitoring of active reverse-circulation and diamond-core drill sites, including the collection and on-site handling of drill samples. Additional days of each of these visits were spent at Paramount's Vale field office inspecting drill core in detail, reviewing all project procedures related to the active drilling programs, and generally communicating with the technical staff about the geology and mineralization of the Grassy Mountain deposit.

Karen Moffitt, an engineer and geotechnical expert, and two associates also participated in the August 18, 2016 site visit. Golder Associates Inc. ("Golder") field and engineering staff, under the direction of

Ms. Moffitt, visited the site several additional times between September 2016 and March 2018 to facilitate the geotechnical field investigations.

Additional site visits were also conducted by MDA Senior Geologist Mr. Paul Tietz, C.P.G., and MDA Senior Associate Geologist Mr. Steven Weiss, Ph.D., C.P.G. Both Mr. Tietz and Mr. Weiss are qualified persons under NI 43-101 and have no affiliations with Paramount except that of independent consultant/client relationship. Mr. Tietz visited the project office and drill core logging facility in Vale, Oregon, for three days in December 2016. Drill core, project data, and logging procedures were reviewed with Paramount's project manager, Mr. Michael McGinnis. Mr. Tietz visited the project office again in January, February, and March 2017 for a total of 18 days, two of which were spent at the Grassy Mountain project reviewing the site geology and surface exposures of hydrothermally altered rocks that host the deposit. During the remainder of this time, Mr. Tietz undertook further review of drill core and assisted Paramount's technical team in the construction of a cross-sectional geological model of the Grassy Mountain gold deposit.

Mr. Weiss visited the project area for 12 days in March 2017. Five days were spent reviewing the geology of the deposit area as well as other areas of hydrothermally altered rocks on the property in detail as part of an effort to develop and evaluate exploration drilling targets. Mr. Weiss spent seven days at the project office and core facility in Vale, Oregon reviewing drill core and cuttings, maps, surface and down-hole geochemical data, and cross-sections for areas adjacent to the Grassy Mountain deposit as well as outlying prospects within the property.

Although Mr. Tietz and Mr. Weiss are not co-authors of this report, their on-site activities contributed to the Data Verification summarized in Section 12.0 of this report.

The authors have relied almost entirely on data and information derived from work done by Paramount and its predecessor operators of the Grassy Mountain project, as well as other sources of information as cited. The authors have reviewed much of the available data, completed multiple site visits, and have made judgments about the general reliability of the underlying data. Where deemed either inadequate or unreliable, the data were either eliminated from use or procedures were modified to account for lack of confidence in that specific information. The authors have made such independent investigations as deemed necessary in the professional judgment of the authors to be able to reasonably present the conclusions discussed herein.

The Effective Date of this technical report is May 21, 2018. The estimated mineral resources have an Effective Date of May 1, 2018. The estimated mineral reserves have an Effective Date of May 1, 2018.

2.2 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

In this report, measurements are generally reported in Imperial units. Where information was originally reported in metric units, MDA has made the conversions as shown below, except in cases where legal, laboratory, or metallurgical measures and results were originally specified or reported in metric units, and their conversion would result in substantive rounding errors or changes to precision.

Currency, units of measure, and conversion factors used in this report include:

Linear Measure

1 centimeter	= 0.3937 inch	
1 meter	= 3.2808 feet	= 1.0936 yard
1 kilometer	= 0.6214 mile	

Area Measure

1 hectare	= 2.471 acres	= 0.0039 square mile

Capacity Measure (liquid)

1 liter	= 0.2642 US gallons

Weight

1 tonne	= 1.1023 short tons	= 2,205 pounds
1 kilogram	= 2.205 pounds	

Currency Unless otherwise indicated, all references to dollars ($) in this report refer to currency of the United States.

Frequently used acronyms and abbreviations

AA	atomic absorption spectrometry
Ag	silver
Amp	amperes
Au	gold
CAD	Canadian dollars
cm	centimeters
core	diamond core-drilling method
ºC	degrees centigrade
d	day
°F	degrees Fahrenheit
ft	foot or feet
ft/d	feet per day
ft^3/min	cubic foot per minute
ft^3/sec	cubic feet per second
gal	gallons (US)
GIS	geographic information system
g/t	grams per tonne
gpm	gallons per minute
ha	hectares
hp	horse power
hr	hour
ICP	inductively coupled plasma analytical method
in	inch or inches

inches wg	air pressure in inches water gauge
kg	kilograms
km	kilometers
kV	kilovolts
kVA	kilovolt-ampere
kW	kilowatts
kWh	kilowatt-hour
l	liter
lab	laboratory
lb	pounds
lb/ft^3	pounds per cubic foot
LOM	life of mine
µm	micron
m	meters
Ma	million years old
mi	mile or miles
min	minutes
mm	millimeters
MV	megavolts
MVA	megavolt amp
MW	megawatts
MWh	megawatt-hour
NSR	net smelter return
oz	ounce
oz Au/ton	ounces of gold per short ton
P_{80}	particle-size distribution of 80% ≤ the nominal dimension
Pcf	pounds per cubic feet
ppm	parts per million
ppb	parts per billion
QA/QC	quality assurance and quality control
RC	reverse-circulation drilling method
ROM	run of mine
RQD	rock-quality designation
sec	seconds
st	Imperial short ton
t	metric tonne or tonnes
T or ton	Imperial short ton (2,000 pounds)
µg	microgram
USD	United States dollars
y	years
yd^3	cubic yard

3.0 RELIANCE ON OTHER EXPERTS

The authors did not conduct any investigations of the environmental, permitting, or social-economic issues associated with the Grassy Mountain project, and the authors are not experts with respect to these issues, or with respect to legal matters, such as the assessment of the legal validity of mining claims, private lands, mineral rights, and property agreements in the United States. The authors have fully relied on Paramount to provide complete information concerning the legal status of Paramount and related companies, as well as current legal title, material terms of all agreements, material environmental and permitting information, and tax matters that pertain to the Grassy Mountain project.

Section 4.0 in its entirety is based on information provided by Paramount. A Mineral Status Report prepared for Paramount by mining attorney Thomas P. Erwin of Erwin, Thompson & Faillers LLP, and dated September 26, 2017, described the property and title aspects of the project. Mr. Richard Delong of EM Strategies, Inc., an environmental consulting firm contracted by Paramount, prepared Sections 4.3 and 4.4, as well as Section 20, on environmental studies, permitting, and social and community impacts. Mr. DeLong has particular expertise in environmental compliance and permitting of mining projects in the western United States.

Mr. Dyer relied on Paramount to provide guidance on the application of taxes in economic analyses (Section 22.0).

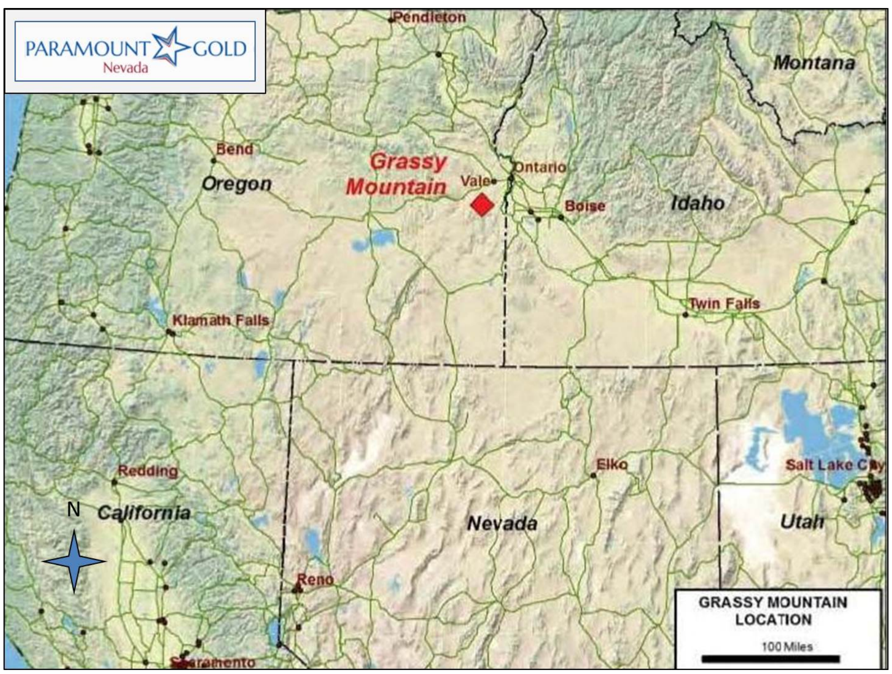
4.0 PROPERTY DESCRIPTION AND LOCATION

Mr. Gustin is not an expert in land, legal, environmental, and permitting matters. This Section 4.0 is based on information provided to MDA by Paramount. Mr. Thomas P. Erwin of Erwin, Thompson & Faillers, LLP prepared a Mineral Status Report for Paramount, dated September 26, 2017. This report described the property and title aspects of the project. MDA presents this information to fulfill reporting requirements of NI 43-101 and Mr. Gustin expresses no opinion regarding the legal or environmental status of the Grassy Mountain project.

4.1 Location

The Grassy Mountain property is situated near the western edge of the Snake River Plain in eastern Oregon, 20 miles south of the town of Vale, Oregon and about 70 miles west of Boise, Idaho (Figure 4.1). The property encompasses approximately 9,300 acres, all located within surveyed townships in Malheur County. The geographic center of the property is located at 43.674° N latitude and 117.362° W longitude, and the principal zone of mineralization is located in Section 8 of Township 22 South ("T22S"), Range 44 East ("R44E"), Willamette Meridian.

Figure 4.1 Location of the Grassy Mountain Project



4.2 Land Area and Mineral Title

The Grassy Mountain property consists of 427 unpatented lode claims, nine unpatented mill site claims, six unpatented association placer claims, three patented claims, and two land leases covering portions of Sections 11 through 15 and 24 of T22S, R43E; portions of Sections 3 through 10 and 16 through 20, T22S, R44E; Sections 31 through 34, T21S, R44E; and Section 36, T21S, R43E, as shown in Figure 4.2. Patented claims were individually surveyed at the time of location. Unpatented claim and Fee land boundaries were established initially by GPS handheld units and in 2011 by onsite survey work. Claim information is summarized in Appendix A.

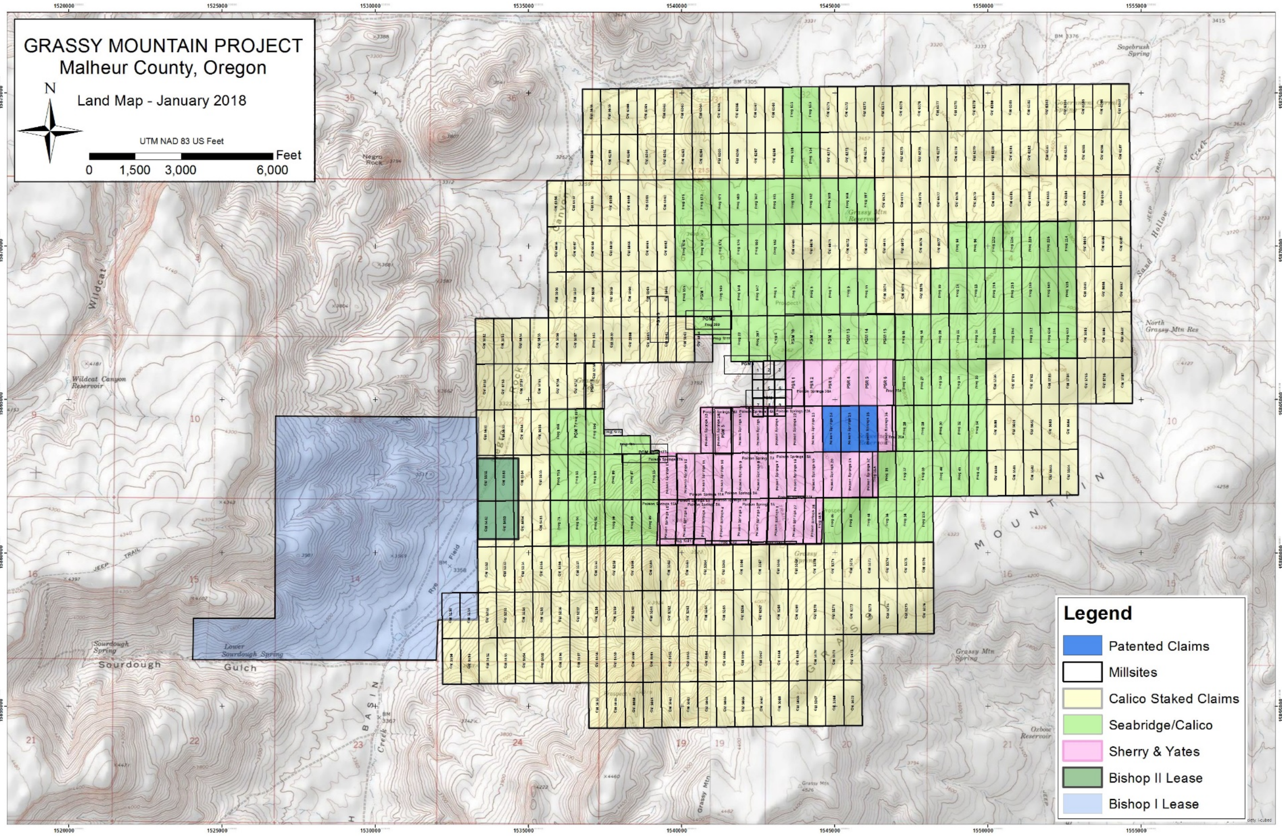
Unpatented claims are subject to annual US Bureau of Land Management ("BLM") fees of $155 per claim. The unpatented annual claim fees have been paid through September 1, 2018. Patented claims are subject to annual property taxes of $102.44 per year. Taxes for 2017/2018 were paid on November 3, 2017.

Calico, a wholly owned subsidiary of Paramount, owns and controls 100% of the mineral tenure of the unpatented mining claims, patented mining claims, Fee lands, and mining leases that comprise the Grassy Mountain property. Calico acquired all right, title and interest in the property, including all existing exploration and water rights pertaining to the Grassy Mountain project, pursuant to the "Deed and Assignment of Mining Properties" between Seabridge Gold Inc., Seabridge Gold Corporation and Calico dated February 05, 2013.

Ownership of unpatented mining claims is in the name of the holder (locator), subject to the paramount title of the United States of America, under the administration of the U.S. Bureau of Land Management ("BLM"). Under the Mining Law of 1872, which governs the location of unpatented mining claims on Federal lands, the locator has the right to explore, develop, and mine minerals on unpatented mining claims without payments of production royalties to the U.S. government, subject to the surface management regulation of the BLM.

Paramount controls 100% of the surface rights to the patented and leased lands that comprise the Grassy Mountain project, with the exception the Bishop II leased lands. The surface rights controlled by Paramount are subject to applicable Federal and State environmental regulations and the agreements outlined in Section 4.3.

Figure 4.2 Grassy Mountain Property Map
(from Paramount Gold Nevada, 2018)



Commented [SW1]: Z:\Paramount\Grassy_Mountain_PFS\Reports\NI43-101\Drafts-from_Paramount\ GM20180525 Claims for PFS.PDF

4.3 Agreements and Encumbrances

Paramount's 100% ownership of the Grassy Mountain project is subject to the underlying agreements and royalties summarized in the following subsections.

4.3.1 Seabridge Gold Corporation ("Seabridge")

Seabridge retains a 10% Net Profits Interest ("NPI") in the Grassy Mountain project pursuant to the "Deed of Royalties" between Calico and Seabridge dated February 05, 2013. Pursuant to the "Deed of Royalties", within 30 days following the day that Calico has delivered to Seabridge a Feasibility Study on the Grassy Mountain project, Seabridge may elect to cause Calico to purchase the 10% NPI for $10M (CAD).

4.3.2 Sherry & Yates Inc. ("Sherry & Yates")

4.3.2.1 2004 Lease and Agreement

On February 14, 2018, Calico exercised their Option to Purchase, whereby Sherry & Yates agreed to sell to Calico all right, title and interest in the three patented and 37 unpatented mining claims. The 2004 Lease and Agreement with Sherry & Yates was terminated.

4.3.2.2 2018 Mining Deed

Sherry & Yates have closed the purchase and sale of the three patented and 37 unpatented mining claims under terms of the 2004 Lease and Agreement. Sherry & Yates retains a 1.5% royalty of the gross proceeds for the production of minerals from the patented and unpatented claims and the surrounding ½ mile area of interest (Sherry & Yates Property). Royalty payments are due 30 days following the end of the calendar quarter in which Calico realizes gross proceeds and the royalty will run with the Sherry & Yates Property. The royalty is not subject to advance-royalty payments made prior to Calico's exercise of their Option to Purchase. The royalty attributed to Sherry & Yates has decreased from 6% to 1.5%.

4.3.3 1989 Bishop I & Bishop II Leases

The Bishop I and Bishop II Mining Leases, as amended, with Bishop et al. ("Bishop"), dated September 11, 1989, include the following terms:

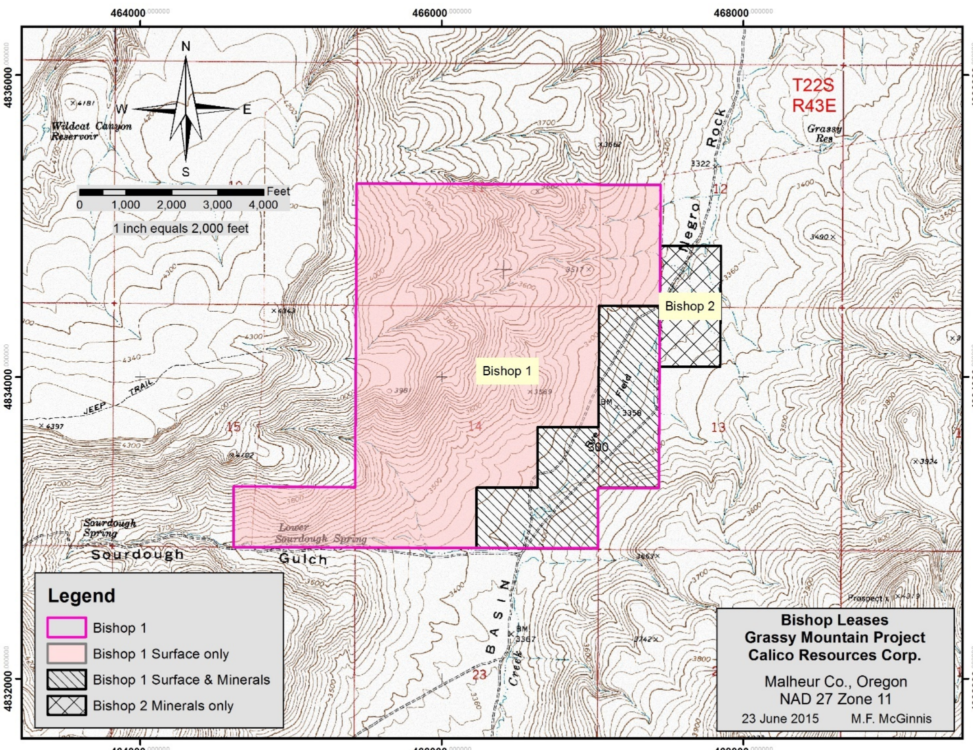
- The Terms shall be 10 years, as amended in 2009, expiring September 11, 2019;
- Annual Minimum royalty payments of $30,000 USD (Bishop I) and $3,000 USD (Bishop II) must be paid by Calico, or its assigns, to keep the Mining Lease and Agreement in good standing. All minimum royalty payments are recoverable against future production royalty payments;
- Bishop retains a 6.0% Net Smelter Return ("NSR") royalty based on a gold price above $800 USD per ounce. If ore minerals other than gold are produced, they would be subject to an additional 4% NSR royalty; and

- A provision in the Bishop I lease agreement provides for payments to be made by the lessee to Bishop as follows: $50 for each drill hole on Fee land; $100 for each acre of disturbed Fee land; and $300 for each acre disturbed and lost for Bishop's use.

Minimum royalty payments made to date indicate that there are accumulated credits of $760,000 and $76,000 that would apply to the Bishop I and Bishop II Leases, respectively.

The Bishop I lease includes Fee lands and unpatented placer claims, while the Bishop II lease includes Fee lands. The surface and mineral rights relating to the Bishop I and II leases are shown in Figure 4.3, and the Bishop I unpatented claims are included in Appendix A.

Figure 4.3 Surface and Mineral Rights of the Bishop I and Bishop II Leases



4.3.4 Total Annual Property Holding Costs

Annual property holding costs total $107,970, as summarized in Table 4.1.

Table 4.1 Grassy Mountain Annual Land Holding Costs

Payment Type	Yearly Cost
BLM Claim Fees – Calico unpatented claims	$ 61,845
BLM Claim Fees – Former Sherry & Yates unpatented claims	$ 4,960
BLM Claim Fees – Bishop placer claims	$ 5,735
Malheur County Claim Recording Fees	$ 2,328
Patented Claims Property Taxes	$ 102
Lease Payments -- Bishop I and II	$ 33,000
Total Annual Cost	**$ 107,970**

4.4 Environmental Liabilities

Except for the exploration surface disturbance and network of wells that will need to be reclaimed, there are no known environmental liabilities associated with the Grassy Mountain project. All exploration drill holes that are not part of the current approved monitor well program have been plugged according to Oregon regulations. Surface disturbance that has not been reclaimed will be used for future development activities and access. The groundwater monitoring wells remain in use for ongoing exploration activities and permit acquisition activities associated with the mine development process. The disturbance is bonded as described below in Section 4.5 of this report.

4.5 Environmental Permitting

There is a valid existing exploration permit with DOGAMI and the BLM. A bond in the amount of $146,200 is associated with the exploration permit. An existing Notice with the BLM for 2.78 acres of surface disturbance and a monitor well has an associated bond in the amount of $25,315. An application for "Extension of Time for a Water Right Permit" was filed with the Oregon Water Resources Department. Paramount has until October 1, 2028 to complete the water system and apply water to beneficial use. The company must submit progress reports on October 1 of 2022 and 2027. The permit allows a maximum pumping rate of 2.0 cubic feet per second (895 gallons per minute).

Permits needed for the type and scope of mining at Grassy Mountain outlined in the PFS of this technical report will involve a number of Federal, State, and local regulatory authorities. The project will require the following major environmental permits to construct, operate, and close: 1) a Plan of Operations from the BLM; 2) a DOGAMI Consolidated Permit for Mining Operations; 3) an Oregon Department of Environmental Quality ("ODEQ") Chemical Mining Permit; 4) Water rights from the Oregon Department of Water Resources; 5) an Air Quality Operating Permit ("AQOP") with the ODEQ; and 6) a Conditional Use Permit from Malheur County. Other applicable State of Oregon and federal permits may include, but are not limited to, the following:

- Fill and Removal Permit(s) (ORS 196.600 and 196.800);

- Permits to appropriate groundwater or surface water, or to store water in an impoundment (ORS 537.130, ORS 537.400, and ORS 540.350);
- Water Pollution Control Facility (ORS 468.740);
- Storm Water Pollution Prevention Plan (EPA);
- Air Quality Permits (ORS 468.310);
- Solid Waste Disposal Permit (ORS 459.205);
- Permit to Clear Right of Way (ORS 477.685);
- Permit for Placing Explosives (ORS 509.140);
- Hazardous Waste Storage Permit (ORS 466.005);
- Land Use Permit (OAR Chapter 632, Division 001); and
- Any other state permits, if applicable and required under Division 37

Further information on environmental studies, permitting, and social and community impacts is discussed in Section 20.0.

4.6 Water Rights

Paramount holds a water right granted by the Oregon Water Resources Department to Calico Resources USA Corp. The water right was issued on April 5, 1990 through State of Oregon Water Rights Application G-11847 and Permit G-10994. Use is limited to not more than 2.0 cubic feet per second (897.6 gallons per minute) measured at the well.

On December 26, 2012, the Oregon Water Resources Department, Water Rights Services Division, granted Final Order Extension of Time for Permit Number G-10994. This extension gave Calico Resources USA Corp. until October 1, 2028 to fully develop and apply water to beneficial use. If the water right has not been developed and proven by the deadline, the State will begin cancellation proceedings.

4.7 Summary Statement

Mr. Gustin is not aware of any significant factors and risks not discussed in this report that may affect access, title, or the right or ability to perform work on the property, although Mr. Gustin is not an expert with respect to such matters.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Access to Property, Physiography and Sufficiency of Surface Rights

Access to the Grassy Mountain project area is provided by Twin Springs Road, a partially maintained unpaved road that originates at US Highway 20 approximately four miles west of the city of Vale, Oregon. The center of the project area may be reached from the Twin Springs Road via 2.5 miles of secondary unpaved roads. Winter and wet weather conditions occasionally limit access to the property, although on-site travel is generally possible year-round.

The project area is located in the plateau region of eastern Oregon. Terrain at the project area is mainly open steppe with mesas, broad valleys, and gently rolling hills to steeper uplands. Elevations range from 3,330 to 4,300 feet above mean sea level. Vegetation consists of sagebrush, weeds, and desert grasses tolerant of semi-arid conditions.

The surface rights as described in Section 4 are sufficient for the mining and exploration activities proposed in this report.

5.2 Climate

The climate can be described as the semi-arid, continental interior type, with average annual precipitation of about 9.25 inches, roughly half of which falls as snow between November and March. Local weather data indicate a mean annual temperature of 52° F, with daily temperatures ranging from an extreme low of -20°F in the winter to extreme highs of 100°F and higher in the summer. Monthly average temperature and precipitation data for Vale, Oregon are shown in Table 5.1.

Table 5.1 Monthly Average Climate Data for Vale, Oregon

Period of Record Monthly Climate Summary

Period of Record : 01/01/1893 to 07/31/2013

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Average Max. Temperature (F)	36.2	43.9	55.5	65.4	74.5	82.8	93.3	90.8	79.7	66.2	48.9	37.8	64.6
Average Min. Temperature (F)	18.7	23.7	29.5	35.1	42.8	49.6	55.6	52.6	43.2	33.7	26.1	20.5	35.9
Average Total Precipitation (in.)	1.17	0.85	0.83	0.72	1.01	0.80	0.26	0.30	0.45	0.68	0.99	1.19	9.25
Average Total SnowFall (in.)	6.2	1.8	0.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1.1	4.2	14.0
Average Snow Depth (in.)	1	1	0	0	0	0	0	0	0	0	0	1	0

Percent of possible observations for period of record.
Max. Temp.: 95.2% Min. Temp.: 95.1% Precipitation: 96% Snowfall: 92.1% Snow Depth: 90%
Check Station Metadata or Metadata graphics for more detail about data completeness.

Western Regional Climate Center, wrcc@dri.edu

It is expected that seasonal road maintenance will be sufficient to provide access to the site for all personnel and any deliveries related to the mine site. Mining and exploration activities can be conducted year-round.

5.3 Local Resources and Infrastructure

As of the Effective Date of this report, several monitoring wells and unpaved access and drilling roads are the only available infrastructure at the Grassy Mountain property. A regional, 500-kV electrical transmission line runs through the southern part of the property, about 2.5 miles south of the proposed mine site. Project power requirements and other infrastructure details are discussed in Section 18.0

Water to support current exploration activity is available from on-site wells. Long-term water needs for mining and processing are forecasted to be approximately. Paramount has already developed capacities of more than 200 gpm from multiple wells near the mill and mine sites. Project water requirements and sources are described in more detail in Section 18.0.

Logistical support is available in Vale, Nyssa, and Ontario, Oregon, all of which are located within 20 miles of the project site. Mining personnel, equipment, fuel, supplies, and engineering and telecommunications services for operations at the Grassy Mountain property are expected to be available from Malheur County, Oregon and the adjacent greater Boise area in neighboring southern Idaho.

6.0 HISTORY

The information summarized in this section of the report has been extracted and modified from Wilson et al. (2015a), which was drawn from Hulse et al. (2012), with additional information derived from multiple sources, as cited. A concise early history of the discovery of the Grassy Mountain deposit and other events to September 1988 was reported by Kelly (1988). Mr. Gustin has reviewed this information and believes this summary accurately depicts the history of the Grassy Mountain project.

Portions of the present Grassy Mountain property were first staked by two independent geologists, Dick Sherry and Skip Yates, in 1984. Atlas Precious Metals ("Atlas") acquired the Grassy Mountain property from Sherry and Yates in 1986. Between 1986 and 1991, Atlas conducted detailed mapping and sampling at the property and drilled a total of 227,397 feet in 400 drill holes. Shallow, apparently stratiform gold mineralization was delineated at the main Grassy Mountain deposit and 1.5 miles to the southwest at the Crabgrass prospect. Atlas identified exploration targets at the Grassy Mountain project based on soil anomalies, conducted further soil and float sampling on several prospects, expanded the original claim block, and collected extensive geologic, mine engineering, civil engineering, and environmental baseline data. The baseline data were compiled to support a 1990 historical feasibility study for an envisioned open-pit heap-leach and milling operation. Atlas then began to consider underground-mining scenarios, but declining gold prices and the perception of an unfavorable permitting environment discouraged Atlas from developing the project, and the property was optioned to Newmont Exploration Ltd ("Newmont") in 1992.

Newmont leased the Grassy Mountain property from Atlas in September 1992 for US$30 million. Newmont geologists mapped the property and completed geochemical sampling. Several ground and airborne geophysical surveys were also conducted. In late 1994, Newmont drilled 15 holes and completed an in-house mineral resource estimate that became the basis for an in-house economic and mining-method evaluation that was completed in 1995. Newmont determined that the project did not meet corporate objectives and returned the property to Atlas in September 1996.

In January 1998, Atlas granted Tombstone Exploration Company Ltd ("Tombstone") the option to purchase 100% of the property. Tombstone executed the option agreement and conducted an exploration program which included six holes for a total of 8,071 feet. Lack of venture capital forced Tombstone to return the property to Atlas in May 1998.

In February 2000, Seabridge Gold ("Seabridge") entered an option agreement with Atlas to acquire a 100% interest in the Grassy Mountain property. Seabridge completed its acquisition of the Grassy Mountain property in April 2003.

Seabridge did not carry out exploration at the Grassy Mountain property and in April of 2011, signed an option agreement granting Calico Resources Corp. the sole and exclusive right and option to earn a 100% interest in the project. The acquisition of the Grassy Mountain property by Calico was completed in 2012. In 2011 and 2012, Calico carried out geologic mapping and sampling, and drilled a total of 13,634 feet in 17 holes. Calico also commissioned a geophysical survey to assist in their exploration efforts at the Grassy Mountain property.

In 2016, Paramount acquired Calico by issuing 7,171,209 common shares to Calico shareholders, whereby Calico stockholders had the right to receive 0.07 of a share of common stock of Paramount for every common share of Calico.

6.1 1986 – 1996 Exploration History

Historical exploration conducted by previous operators includes exploration programs carried out by Atlas, Newmont, Tombstone, and Calico.

6.1.1 Atlas 1986 - 1992

Atlas carried out geologic mapping of the property and recognized soil geochemistry as an important exploration tool at Grassy Mountain. Most Atlas exploration targets were initially identified by claim-corner soil sampling on 600-foot by 1,500-foot spacings. Atlas conducted further soil and float sampling on several anomalies and identified a genetic link between gold mineralization and silicification. Of the 400 drill holes completed by Atlas at the Grassy Mountain property, 196 were RC holes drilled on 75- to 100-foot centers within what became the Grassy Mountain resource area. The remaining holes were drilled at prospects away from the main Grassy Mountain resource area.

In addition to the Grassy Mountain deposit, Atlas delineated another gold prospect called Crabgrass, which is located approximately 1.5 miles southwest of the Grassy Mountain deposit. Atlas drilled 87 RC holes at Crabgrass and defined three separate near-surface mineralized zones.

6.1.2 Newmont 1992 - 1996

Newmont carried out extensive and locally detailed geologic mapping, and both soil and rock-chip sampling. In 1993, Newmont geologists mapped 40 square miles at a scale of 1:6,000 and collected approximately 2,600 soil samples on a 400-foot by 200-foot grid. Newmont began soil sampling on a 400-foot by 200-foot grid, hoping to identify anomalies missed by the Atlas sampling. During 1993 and 1994, Newmont collected more than 400 rock-chip samples and conducted several geophysical surveys. These included a ground-based gravity survey along existing roads, airborne magnetic and radiometric surveys over the entire property, and ground based gradient array (IP/resistivity) surveys over the main deposit and several of the satellite prospects. Ground magnetic surveys were conducted over certain areas. Newmont geologists re-logged the remaining Atlas drill core during this period, and eventually the Atlas RC drill chips as well.

In 1994, Newmont first drilled 11 inclined diamond-core holes designed to intersect and define the geometry of potential high-grade gold zones in the main Grassy Mountain deposit. These were followed with one wedge core hole off of their initial core hole, two holes pre-collared by RC and completed with core, and one additional core hole.

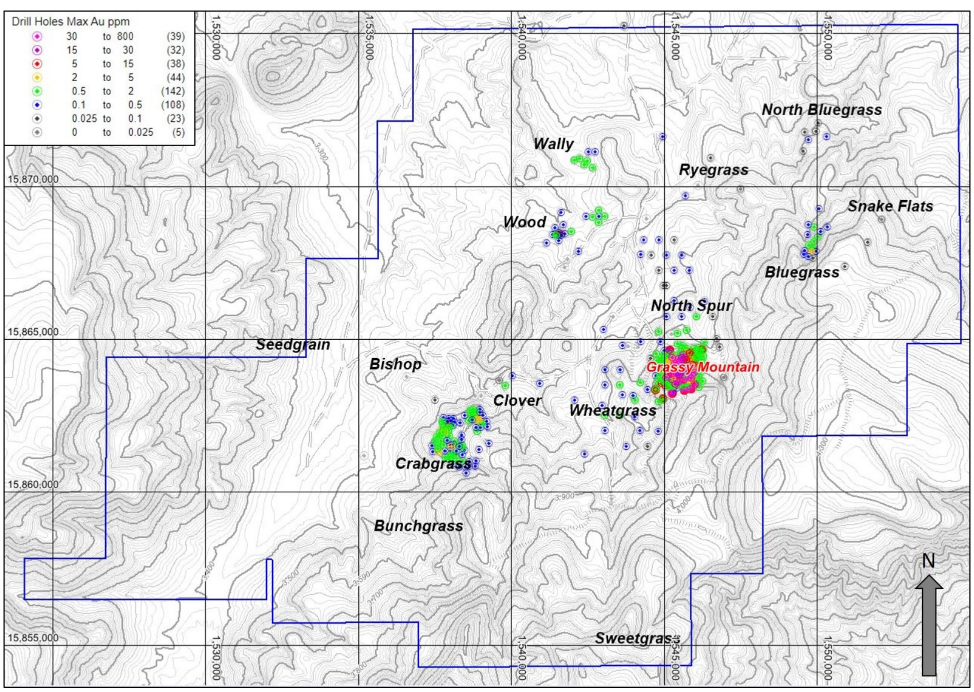
Newmont's 15 holes were all angled and totaled 15,009.5 feet. This drilling defined what Newmont thought could be several gold zones in excess of 0.1 oz Au/ton within an area of the Grassy Mountain deposit measuring approximately 600 feet long by 350 feet wide by 250 feet thick. Mineralization was constrained to the northeast by a single hole which failed to encounter high-grade gold. Newmont considered the western extent of the main high-grade zone effectively closed off after encountering only low-grade gold (0.012 to 0.019 oz Au/ton) and local barren quartz-chalcedony veins. Based on the core drilling and mapping and sampling of surface exposures, Newmont geologists concluded that high gold grades at the Grassy Mountain deposit were controlled by narrow, steeply south-dipping quartz-chalcedony veins and clay matrix breccias that would need to be properly represented during grade modeling and resource estimation.

During 1995 and 1996, Newmont's activities were focused on estimating mineral resources at the main Grassy Mountain deposit. No new exploration work was done during this period.

6.2 1996 Historical Exploration at Outlying Target Areas

By 1996, Atlas and Newmont had identified and named several mineralized and potentially mineralized target areas peripheral to the main Grassy Mountain gold deposit based primarily on rock-chip, float, and soil-sample data. These outlying targets, several of which were drilled to varying degrees, are shown in Figure 6.1 and summarized in the following subsections to provide perspective regarding the historical exploration activities that have been conducted on the property, and to provide context for historical exploration done by Calico in 2011 and 2012.

Figure 6.1 Outlying Target Area Map
(data from Paramount, 2016)



Blue lines are limits of Paramount's claims; UTM NAD83 US Feet, Zone 11 projection; contour interval is 10 feet. 5,000ft grid lines for scale. Dots are drill hole collars through 2012 colored by maximum gold assays.

6.2.1 Wheatgrass

This target area is approximately 1,500 feet southwest of the Grassy Mountain deposit area (Figure 6.1) and was the site of the first drilling on the property. It has been hypothesized that Wheatgrass may be a lateral continuation of mineralization extending from the main Grassy Mountain deposit that is displaced by down-to-the-west faults. A number of RC drill holes have tested this area with some narrow, low-grade intersections being encountered. Most of the historical holes are vertical and widely spaced.

6.2.2 North Spur

North Spur is 2,000 feet to the north-northeast of the main Grassy Mountain deposit (Figure 6.1). Resistant ledges of silicified sandstone indicate hydrothermal fluids flowed through the North Spur area. Three widely spaced vertical RC holes south of the silicified ledges have intervals with grades as high as the 0.015 to 0.058 oz Au/ton range. About 500 feet to the north, a fence of three vertical RC holes is located approximately at the northern margin of the most strongly silicified outcrops. These holes penetrated intervals with generally low grades, but they are sporadically mineralized. Review of RC chips and logs from these holes indicates that gold grades decrease down hole as the sandstone intervals transition to more clay-rich units with depth. All of these holes were drilled vertically and did not adequately test for steeply dipping mineralized structures.

6.2.3 Crabgrass

The three mineralized areas that comprise the Crabgrass prospect (Figure 6.1) appear to be stratiform and contained within the flat-lying to gently east-dipping sandstones above the clay-rich units, but all the historical holes are vertical and RC. Significant low-grade gold mineralization was encountered in numerous holes, which formed the basis for a historical resource estimate, as discussed in Section 6.4.

6.2.4 Bluegrass and North Bluegrass

These targets are located 1.2 miles and 1.6 miles northeast of the Grassy Mountain deposit, respectively (Figure 6.1). Sixteen RC holes were drilled in the area to follow up on rock-chip and float-chip samples with elevated gold contents. The best hole intersected 65 feet averaging 0.035 oz Au/ton beginning at 140 feet down the hole.

6.2.5 Snake Flats

This area is 2.25 miles to the northeast of the Grassy Mountain deposit (Figure 6.1). The target was identified by mapping float of silicified arkose and sinter boulders. A large mercury, arsenic, and antimony soil anomaly extends down-slope for approximately 3,500 feet to the northeast. This is the most aerially extensive surface geochemical anomaly at the project other than at Wheatgrass. Samples from the altered boulders contained up to 0.03 oz Au/ton; the source area for these boulders appears to be somewhere beneath post-mineral basalt in the area. Three RC holes were drilled through about 100 feet of the post-mineral basalt before intersecting unaltered sandstone and siltstone. Additional work is necessary to define a drill target.

6.2.6 Wood

The Wood target is 1.2 miles northwest of the main deposit area (Figure 6.1). Wood was identified by surface rock and soil sampling, followed by surface trenching. Rock-chip samples that were taken from a small outcrop of weakly silicified volcanic rocks returned assays of as much as 0.007 to 0.009 oz Au/ton. Fifteen shallow RC drill holes were completed in the area, with the best intercept being 30 feet averaging 0.073 oz Au/ton beginning at 30 feet down the hole.

6.2.7 Wally

The Wally target is 1.5 miles north-northwest of the Grassy Mountain deposit (Figure 6.1) and has been referred to as the “Big Wally” target in some historical documents. Soil samples in the Wally area defined overlapping arsenic, mercury, antimony, and gold anomalies that straddle a north-northwest-trending fault shown on the district geology map. The best drill hole intercept in this target was 90 feet of 0.025 oz Au/ton, beginning 100 feet down the hole.

6.2.8 Ryegrass

The Ryegrass target is located 1.2 miles north of the Grassy Mountain deposit (Figure 6.1). This area was identified by mapping silicified zones. Follow-up rock-chip sampling of the outcrops returned values of 20 to 25 ppb gold and 900 to 1,000 ppb mercury.

6.2.9 Clover

This target is one mile west of the main deposit (Figure 6.1) and is identified as an area of weakly silicified arkose adjacent to a northeast-trending fault. Rock-chip sampling identified an outcrop containing 25 ppb gold.

6.2.10 Bunchgrass

Bunchgrass is an area of modestly elevated mercury, arsenic, and antimony in soil samples located 0.5 miles south of Crabgrass (Figure 6.1). Wilson et al. (2015a) reported that the target area is approximately 750 feet wide.

6.2.11 Sweetgrass

Sweetgrass is located approximately 1.75 miles southwest of the Grassy Mountain deposit (Figure 6.1). Sampling of a large float boulder of siliceous sinter returned 1,030 ppb Au. Although additional sampling in the area did not return any significant values, more work is warranted to determine the source of this siliceous sinter boulder.

6.3 1998 – 2016 Exploration

6.3.1 Tombstone 1998

Prior to finalizing their agreement with Atlas, Tombstone reviewed data from previous work at the property and commissioned an economic study of alternative development scenarios. Tombstone drilled 10 RC holes, six of which were completed with diamond core ("core") tails, for a total of 8,071 feet. Tombstone relied heavily on Newmont's gradient array surveys to define their drilling targets.

6.3.2 Seabridge 2000 - 2010

Seabridge acquired the property in 2000 and subsequently optioned the property to Calico in early 2011. Paramount represents that Seabridge did not conduct exploration activities at Grassy Mountain.

6.3.3 Calico 2011 - 2016

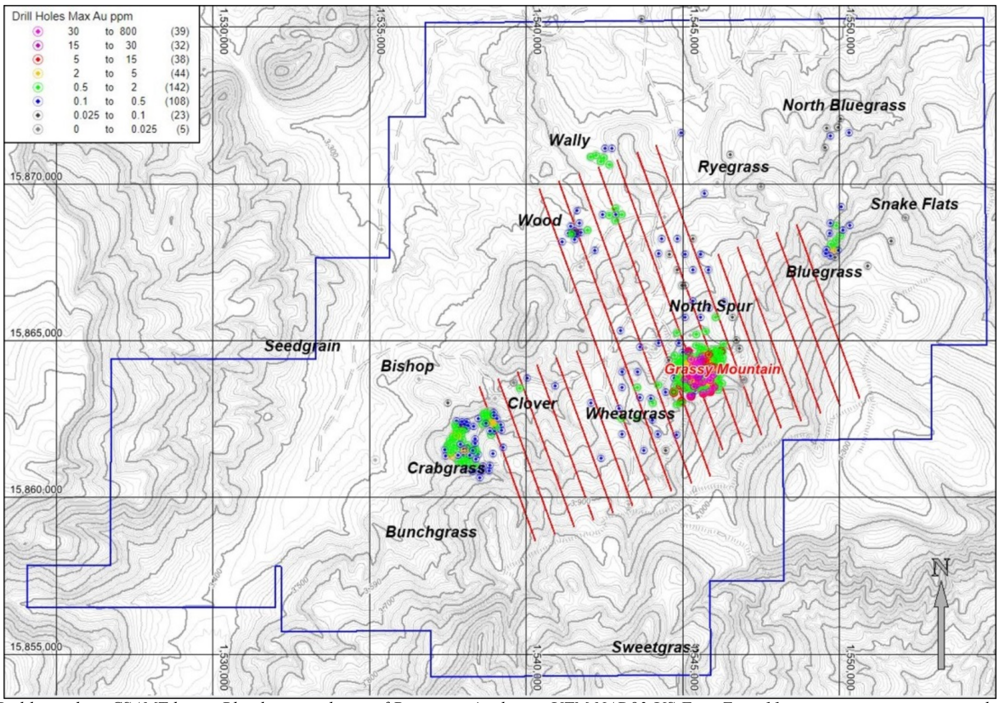
Calico geologists conducted geologic mapping and compiled the Atlas and Newmont geology and surface sample data into property-wide and deposit area maps using conventional GIS procedures. During 2011 and 2012 a total of 13,634 feet were drilled in 14 RC and three core holes. Thirteen of these holes were drilled in the Grassy mountain deposit area and four were drilled in outlying targets.

In 2012, Calico commissioned a 25.1 line-mile controlled-source audio-frequency magnetotelluric ("CSAMT") survey conducted by Zonge Geosciences Inc. ("Zonge"). The survey lines were oriented N20°W (Figure 6.2) and arranged to cross the trend of known mineralization. The CSAMT survey was done under the supervision of consulting geophysicist J.L. Wright, of Wright Geophysics in Spring Creek, Nevada. Mr. Wright documented the survey methods and parameters, analyzed the processed data provided by Zonge, and made geologic and exploration interpretations in a 2012 report to Calico that included 18 inverted resistivity sections and interpretive overlays in PDF format, as well as ArcGIS and MapInfo electronic data files. (Wright, 2012).

An important result of the CSAMT survey was the recognition of a zone of high resistivity that encompassed the main Grassy Mountain gold deposit (Figure 6.3). This was attributed to the zone of extensively silicified rocks that host the main Grassy Mountain gold deposit. The high-resistivity response was visible in sectional and plan views of the resistivity inversion; an example is shown in Figure 6.3.

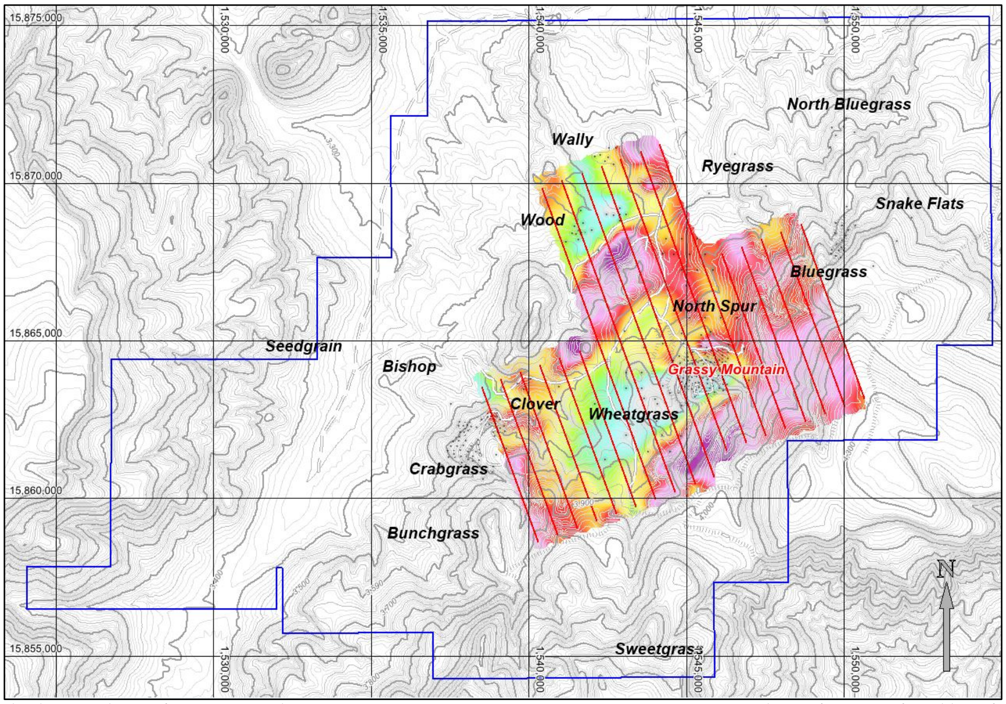
In July 2016, Calico and the Grassy Mountain property were acquired by Paramount. Work carried out by Paramount, the current operator of the project, is summarized in Section 10.2.

Figure 6.2 Map of 2012 CSAMT Lines
(from Wright, 2012)



Red lines show CSAMT lines. Blue lines are limits of Paramount's claims; UTM NAD83 US Feet, Zone 11 projection; contour interval is 10 feet. 5,000ft grid lines for scale. Dots are drill hole collars through 2012 by maximum Au assays, same as in Figure 6.1.

Figure 6.3 CSAMT Inversion: Resistivity at 328 to 656 Feet Below Surface



Blue lines are limits of Paramount's claims; UTM NAD83 US Feet, Zone 11 projection; contour interval is 10 feet. 5,000ft grid lines for scale. Grey dots are drill hole collars through 2012.

6.4 Historical Mineral Resource Estimates

6.4.1 1990 – 1997

A variety of historical resource and reserve estimates for the Grassy Mountain gold deposit were completed on behalf of previous owners and issuers from 1990 through 1997. These historical estimates (Table 6.1 and Table 6.2) were summarized in the 2011 Technical Report prepared by Resource Modeling Inc. (Lechner, 2011), and are described in detail in various internal reports prepared by Atlas, Newmont, and their contractors. In addition, Wilson et al. (2015a) provided a summary of historical estimated resources for the Crabgrass prospect (Table 6.3). All of the estimates presented below in Table 6.1, Table 6.2, and Table 6.3 are relevant only for the historical context of exploration work done during this period and are not to be relied upon. Paramount is not treating these estimates as current mineral resources and Mr. Gustin has not done sufficient work to classify these estimates as current mineral resources. These historical estimates are superseded by the current mineral resources described in Section 14.0.

Table 6.1 1990 – 1997 Historical Open Pit Estimates, Grassy Mountain Deposit
(from Lechner, 2011)

Open Pit Type 'Resources'						
Year	**Source of Estimate**	**Au Cutoff (opt)**	**Tons Above Cutoff**	**Mean Au (opt)**	**Contained Au (oz)**	**Comments**
1990	PAH	0.020	17,200,000	0.061	1,053,100	"Geologic Resource" - global block model tabulation, 1990 Kilborn "feasibility study"
1991	PAH	0.020	15,900,000	0.062	996,000	"Open pit Reserve" - used in 1990 Kilborn "feasibility study"
1993	Newmont	0.010	25,400,000	0.032	803,000	Manual polygonal "Resource"
1993	Newmont	0.020	13,600,000	0.045	617,091	Global recovery "Resource"
1993	Newmont	0.020	14,900,000	0.061	900,010	Global recovery "Resource"
1994	Newmont	0.020	20,300,000	0.039	783,000	"Geologic Resource" - DDH only, conservative vein distribution, normal mean
1994	Newmont	0.020	20,300,000	0.059	1,194,000	"Geologic Resource" - DDH only, optimistic vein distribution, lognormal mean
1994	Newmont	0.020	18,000,000	0.04	721,000	"Open pit Resource" - DDH only, conservative vein distribution, normal mean
1994	Newmont	0.020	18,000,000	0.063	1,126,000	"Open pit Resource" - DDH only, optimistic vein distribution, lognormal mean
1997	PAH	0.020	17,252,000	0.052	899,000	"Measured" and "Indicated" Mineral Resource

Table 6.2 1990 – 1997 Historical Underground Estimates, Grassy Mountain Deposit
(from Lechner, 2011)

Underground Type 'Resources'						
Year	**Source of Estimate**	**Au Cutoff (opt)**	**Tons Above Cutoff**	**Mean Au (opt)**	**Contained Au (oz)**	**Comments**
1990	Atlas	0.500	90,210	1.550	139,765	Manual polygonal underground estimate
1991	Dynatec	0.500	131,632	1.130	148,774	Diluted underground "Reserve"
1993	TWC	0.500	62,943	1.660	104,774	Undiluted underground "Reserve"
1993	PAH	0.100	1,562,000	0.256	414,600	Kilborn "prefeasibility study" for Newmont-diluted "Reserve "
1993	Newmont	0.200	1,400,000	0.156	204,000	"Underground Resource" - DDH only, conservative vein distribution, normal mean
1994	Newmont	0.200	1,400,000	0.350	458,000	"Underground Resource" - DDH only, conservative vein distribution, lognormal mean

Mr. Gustin has not done sufficient work to classify the historical estimates above as current mineral resources or mineral reserves and Paramount is not treating the historical estimates as current mineral resources or mineral reserves. These estimates are relevant only for historical context and are not to be relied upon. These historical estimates use categories of resources that are not in accordance with CIM Standards and Definitions and are superseded by the current mineral resources described in Section 14.0.

Table 6.3 Summary of Historical Estimates for the Crabgrass Deposit
(from Wilson et al. 2015a)

Open Pit Type "Resources"						
Year	Source of Estimate	Au Cutoff (opt)	Tons Above Cutoff	Mean Au (opt)	Contained Au (oz)	Comments
1990	Atlas Interoffice Correspondence	0.010	1,694,832	0.023	38,385	Manual Polygonal "Resource"
1990	Atlas Interoffice Correspondence	0.020	621,583	0.039	24,473	Manual Polygonal "Resource"

Mr. Gustin has not done sufficient work to classify the historical estimates above as current mineral resources and Paramount is not treating the historical estimates as current mineral resources. These estimates are relevant only for historical context and are not to be relied upon. These historical estimates use categories of resources that are not in accordance with CIM Standards and Definitions and are superseded by the current mineral resources described in Section 14.0.

All of the above historical resource and reserve estimates pre-date and are not in accordance with NI 43-101, have not been independently verified by Mr. Gustin and are mentioned here for historical completeness to provide perspective regarding the range of estimates produced using different data, methods, and assumptions. These historical resources and reserves are superseded by the current mineral resources and reserves described in Section 14.0 and Section 15.0 of this report. The mineral resource categories applied to the 1990 through 1997 historical resource estimates are not in accordance with 2014 CIM standards, they are not current, they are considered relevant only for the purposes of historical perspective and completeness, and they are not reliable. Mr. Gustin has not done sufficient work to classify the historical resources as current resources, and Paramount is not treating the historical estimates as current mineral resources.

6.4.2 2007 - 2015

Several historical estimates of mineral resources were completed from 2007 through 2015 by Seabridge (Lechner, 2007) and Calico (Lechner, 2011; Hulse et al., 2012; Brown et al., 2012; Wilson et al., 2015a; 2015b) as summarized in Table 6.4. These historical estimates are relevant only for the purposes of historical perspective and s the historical 2007-2015 estimates as current mineral resources.

6.5 Historical Production

There has been no historical production at the Grassy Mountain project as of the Effective Date of this report.

Table 6.4 Summary of 2007 – 2015 Historical Grassy Mountain Resources

Year	Source	Cutoff (oz Au/ton)	Tons	Grade (oz Au/ton)	Au oz	Resource Class	Comments
2007	1	0.0149	24,084,400	0.042	1,006,000	Ind + Inf	$600/oz gold pit shell
2007	1	0.0998	2,030,457	0.168	342,000	Ind + Inf	$600/oz gold pit shell
2011	2	0.0160	22,464,002	0.044	985,000	Ind + Inf	$600/oz gold pit shell
2012a	3	0.0100	58,648,000	0.022	1,576,900	Ind + Inf	$1,255/oz gold price, open pit
2012a	3	0.0600	3,855,900	0.155	598,700	Ind + Inf	$1,255/oz gold price, underground
2012b	4	0.0120	53,377,000	0.028	1,670,200	M + Ind + Inf	$1,255/oz gold price, open pit
2012b	4	0.0790	2,701,875	0.276	744,300	M + Ind + Inf	$1,255/oz gold price, underground
2015a	5	0.0550	4,289,000	0.142	607,000	M + Ind	$1,300/oz gold price, underground
2015b	6	0.0650	3,245,500	0.155	503,700	M + Ind	$1,300/oz gold price, underground
2015b	6	0.0050	65,668,700	0.018	1,150,500	M + Ind + Inf	$800/oz gold pit shell, excluding underground

M = Measured; Ind = Indicated; and Inf = Inferred

1 Lechner (2007)
2 Lechner (2011)
3 Hulse et al. (2012, March 29)
4 Brown et al. (2012, Nov 29)
5 Wilson et al. (2015, February PEA)
6 Wilson et al. (2015, July Amended PEA)

Mr. Gustin has not done sufficient work to classify the historical estimates above as current mineral resources and Paramount is not treating the historical estimates as current mineral resources. These estimates are relevant only for historical context and are not to be relied upon. It is not known if these historical estimates use categories of resources that are in accordance with CIM Standards and Definitions, and these estimations are superseded by the current mineral resources described in Section 14.0.

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7.0 GEOLOGIC SETTING AND MINERALIZATION

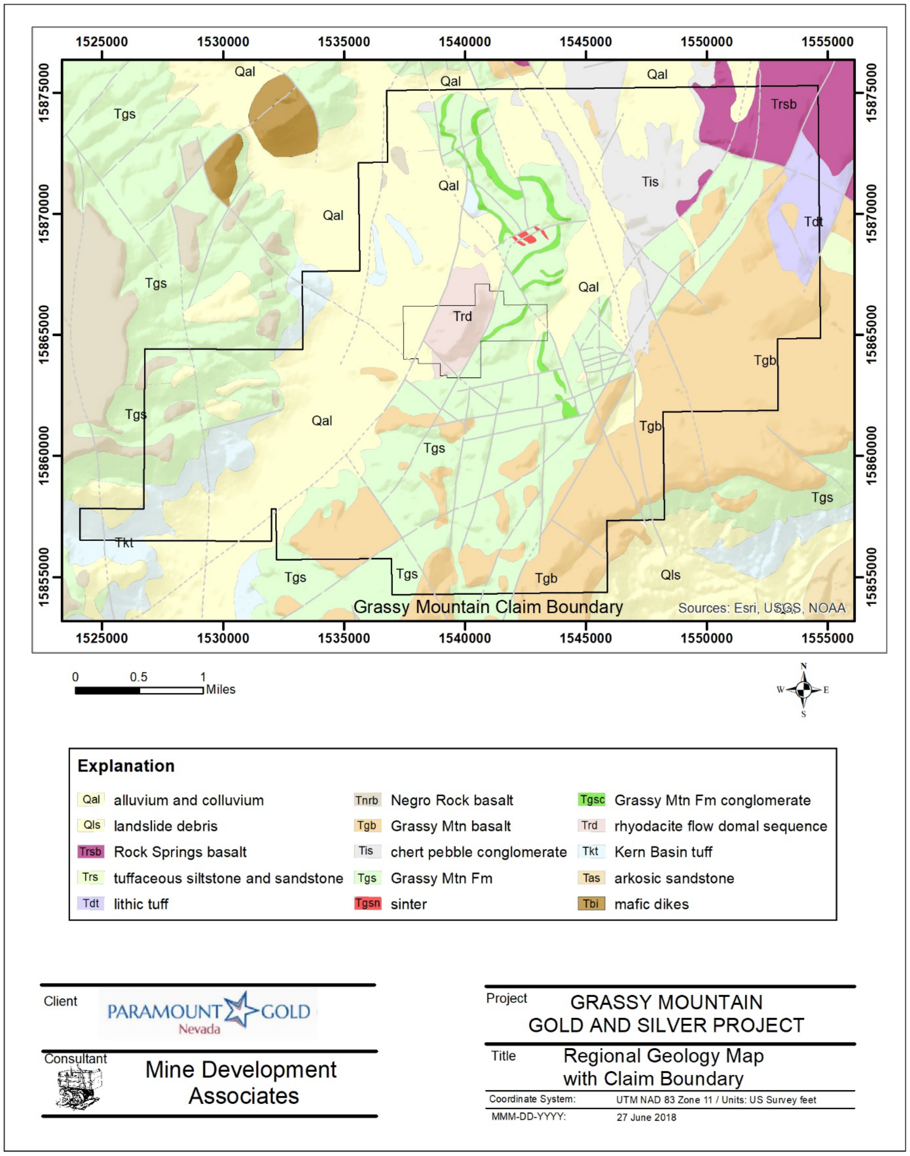
The information presented in this section of the report is derived from multiple sources, as cited. Mr. Gustin has reviewed this information and believes this summary accurately represents the Grassy Mountain project geology and mineralization as it is presently understood.

7.1 Regional Geologic Setting

The Grassy Mountain gold-silver deposit is the largest of 12 recognized epithermal hot-spring precious-metal deposits of the Lake Owyhee volcanic field. The Lake Owyhee volcanic field is located at the intersection of three tectonic provinces: the buried North American cratonic margin, the northern Basin and Range, and the Snake River Plain. During mid-Miocene time, large-volume peralkaline and subalkaline caldera volcanism occurred throughout the region in response to large silicic magma chambers emplaced in the shallow crust (Rytuba and McKee, 1984). The Lake Owyhee volcanic field includes several ash-flow sheets and rhyolite tuff cones that were erupted between 15.5 to 15 Ma (Rytuba and Vander Meulen, 1991). The district geology surrounding the Grassy Mountain gold deposit is shown in Figure 7.1.

At about 15 Ma, subsidence of the Lake Owyhee volcanic field triggered a change in volcanic eruption styles, resulting in small-volume basaltic and rhyolite deposits of limited extent. Volcanism during the middle to late Miocene was characterized by the eruption of small-volume metaluminous high-silica rhyolite domes and flows, as well as small-volume basalt flows and mafic vent complexes in north- and northwest-trending Basin and Range-type fracture zones and ring structures related to resurgent calderas. Regional subsidence involved the development of extensive grabens and facilitated the formation of through-going fluvial systems and large lacustrine basins. Large volumes of fluvial sediments, sourced in part from the exhumed Idaho Batholith to the east and southeast, were deposited contemporaneous with volcanism and hot spring activity during the waning stages of volcanic field development (Cummings, 1991). The resulting regional stratigraphic section is a thick sequence of mid-Miocene volcanic rocks and coeval to Pliocene-age lacustrine, volcaniclastic, and fluvial sedimentary rocks. The oldest units encountered are the flow-on-flow Blackjack and Owyhee Basalts (14.3 to 13.6 Ma). These basalts are overlain by arkosic sandstone, tuffaceous sandstone, and conglomerates of the Deer Butte Formation.

Figure 7.1 Grassy Mountain Regional Geology
(modified from Calico, 2017)



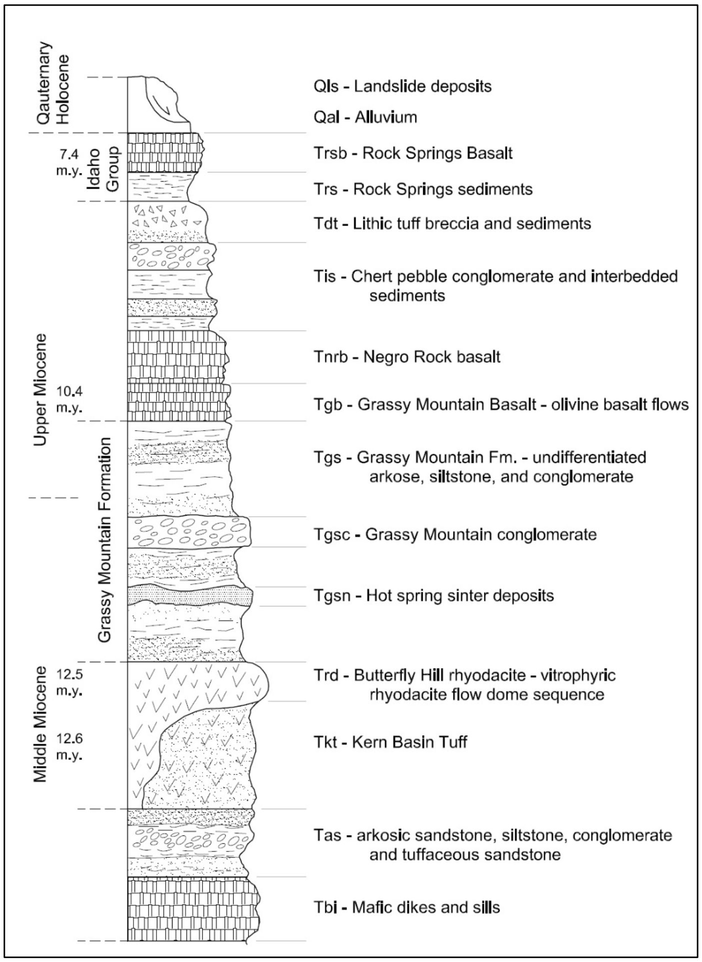
7.2 Local and Property Geology

Bedrock outcrops in the vicinity of the Grassy Mountain property are typically composed of olivine basalt flows and siltstones, sandstones, and conglomerates of the Miocene Grassy Mountain Formation. These rocks are locally covered with relatively thin, unconsolidated alluvial and colluvial deposits. Erosion-resistant basalt flows cap local topographic highs, including Grassy Mountain proper, which is a prominent northeast-elongate ridge that forms a topographic crest about one mile southeast of the Grassy Mountain gold-silver deposit (Figure 7.1). Arkosic sandstones have been encountered at the surface and at depth, but individual beds or sequences have not been correlated across the project area, in part due to lateral sedimentary facies changes and structural offsets. Surface exposures and drill-defined stratigraphy at the Grassy Mountain deposit area reveal complex facies that were produced during the waning stages of volcanism of the Lake Owyhee volcanic field (Lechner, 2011) and development of the coeval Ore-Ida graben.

Figure 7.2 shows the local stratigraphic column in the vicinity of Grassy Mountain. The basal unit is the Kern Basin Tuff, a sequence of pumiceous crystal tuff, which in part displays cross beds and local surge structures, and non-welded to densely welded rhyolite ash-flow tuff. Clast size, thickness of individual ash units, and bedding structures suggest a source in the Grassy Mountain area (Cummings, 1991). The Kern Basin Tuff ranges in thickness from 300 feet on the south bluffs of Grassy Mountain to at least 1,500 feet in a drill hole beneath the Grassy Mountain gold-silver deposit.

A small local flow-dome of approximately 12.5 Ma and known as the Butterfly Hill rhyodacite overlies the Kern Basin Tuff (Figure 7.2). However, in most of the project area the Kern Basin Tuff is overlain by a series of fluvial, lacustrine, and tuffaceous sediments which are assigned to the Miocene Grassy Mountain Formation (Cummings, 1991). These sedimentary units include granitic-clast conglomerate, arkosic sandstone, fine-grained sandstone, siltstone, tuffaceous siltstone, and mudstone (Figure 7.2). The sedimentary units of the Grassy Mountain Formation reportedly range from 300 to over 1,000 feet in thickness, and they comprise the host rocks of the mineral resources at the Grassy Mountain project. Several siliceous "terraces" and silica sinter deposits are interbedded with silicified units of the Grassy Mountain Formation. Terrace construction was apparently episodic and intermittently inundated by fluvial and lacustrine sediments and ash, resulting in an interbedded sequence of siltstone, tuffaceous siltstone, sandstone, conglomerate, and sinter-terrace deposits. Load casts, flame textures, convolute laminations, and other soft-sediment deformation textures are common in both the sinter beds and other sedimentary units (Siems, 1990). The amount and size of the sinter clasts in the sedimentary rocks reflect relative proximity to a terrace. Proximal deposits are angular, inhomogeneous, clast-supported breccias of sandstone, siltstone, and sinter with indistinct clast boundaries in a sulfidic mud-textured matrix.

Figure 7.2 Stratigraphic Column for the Grassy Mountain Area



Note: unit "Tis" with interbedded chert-pebble conglomerate occurs beneath the Grassy Mountain basalt.

According to Lechner (2007), the sedimentary units of the Grassy Mountain Formation are unconformably overlain by 50 to 100 feet of black-chert pebble conglomerate interbedded with unconsolidated siltstone. This unit is recessive, and it is overlain by flows of olivine basalt assigned to the Grassy Mountain basalt, and, in the northwestern part of the property, by the basalt of Negro Rock (Figure 7.2). These mafic lavas are overlain by lacustrine and fluvial siltstone, sandstone, and conglomerate, which are successively overlain by the Rock Springs lacustrine deposits and basalt lavas that together make up the late-Miocene Idaho Group.

7.3 Deposit Area Geology

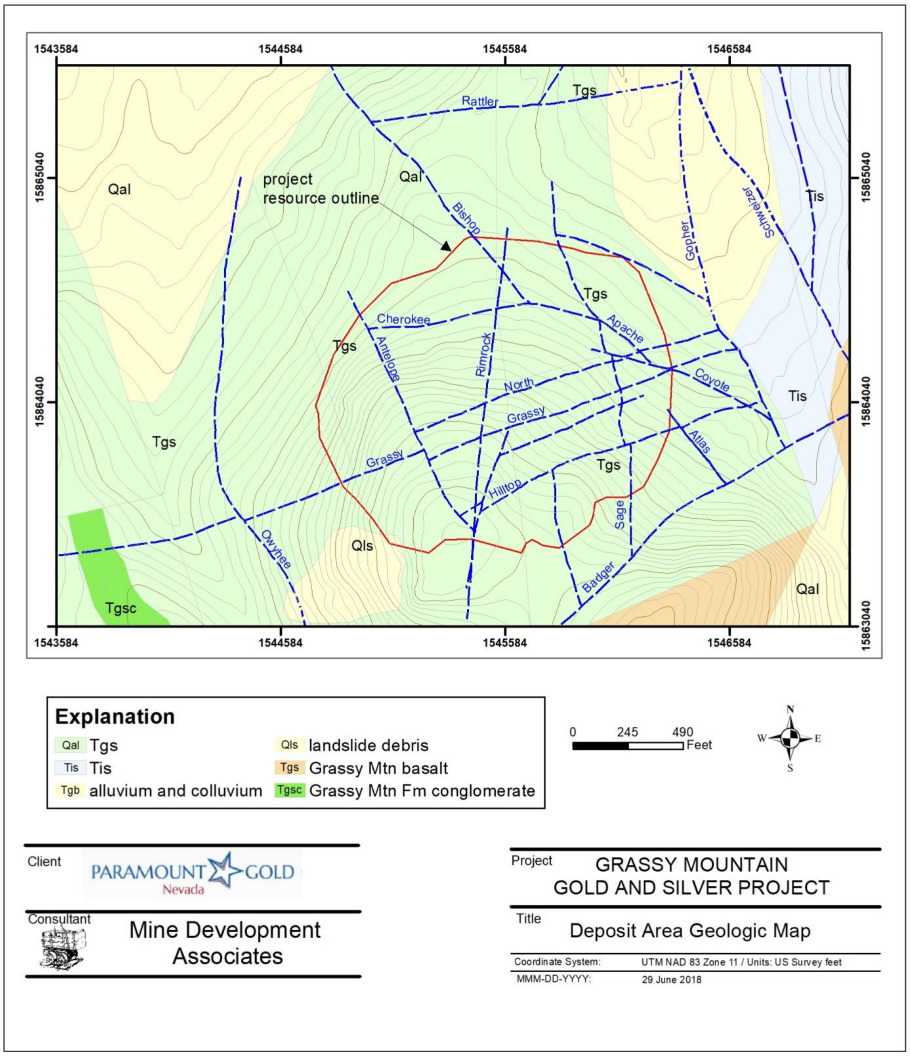
The Grassy Mountain deposit area geology is shown in Figure 7.3. The deposit is centered beneath a prominent, 150-foot high, silicified and iron-stained hilltop that consists of hydrothermally altered arkose and interbedded conglomerate of the Grassy Mountain Formation. Bedding is horizontal at the hilltop and dips at 10° to 25° to the north-northeast on the northern and eastern flanks. The bedding steepens to 30° to 40° on the west side of the hill due to drag folding in the footwall of the N20°W-striking Antelope fault. The southwest slope is covered by landslide debris of silicified arkose.

Several horizons of laminated silica, from a few inches to several feet in thickness, crop out southwest and north of the deposit area and are interbedded within the arkose, siltstone, and conglomerate of the Grassy Mountain Formation. These have been interpreted as beds of silica sinter (Figure 7.2), due in part to the presence of fossil reeds, petrified wood, and other fossil plant debris. Drilling within the Grassy Mountain deposit has penetrated through more numerous and much thicker sinter horizons, indicating the sinter was deposited from hydrothermal fluids venting at the paleo-surface within the accumulating fluvial sedimentary sequence.

Drilling has also shown that in the subsurface of the deposit area the arkosic sandstones and conglomerates are interbedded with numerous intervals of siltstone and mudstone, much of which is thinly laminated. Beds with clay-altered ash to lapilli-sized tephra are common, and there are abundant layers rich in organic carbon ± carbonized plant debris. The laminated siltstone and mudstone intervals reflect a predominantly lacustrine setting that was the site of frequent episodic influxes of fluvial sand- to cobble-sized material.

The gold-silver deposit is situated within a zone of complex extensional block faulting and rotation. Faults at Grassy Mountain are dominated by N30°W to N10°E striking normal faults developed during Basin and Range extension and are inferred to have post-mineral displacement. On the east side of the deposit, these faults are inferred to have down-to-the east movement based on interpreted offsets of a prominent white sinter bed in drill holes, as well as drilled intersections of fault gouge. A set of orthogonal, N70°E-striking high-angle faults of minor displacement are inferred to link the graben faults. One of these, the Grassy fault, has vertical offset of only 10 to 40 feet or less, but it coincides with the axis of the high-grade core of the deposit (see Section 7.4).

Figure 7.3 Deposit Area Geologic Map
(modified from Calico, 2017; blue lines are faults)



7.4 Alteration and Mineralization

Hydrothermal activity and gold mineralization occurred during the accumulation of the Grassy Mountain Formation, and they were coeval with active sedimentation. The water-saturated, unconsolidated sediments therefore required silicic ± potassic alteration to develop sufficient competency to allow for the creation of fractures and structurally-induced open space.

Silicification is the principal hydrothermal alteration type associated with gold-silver mineralization at the Grassy Mountain deposit. It takes the form of silica sinter, pervasive silica flooding, and as cross-cutting chalcedonic veins, veinlets, and stockworks. The silicification is inferred to be largely controlled by hot-spring vents active during accumulation of the Grassy Mountain Formation. The 300-foot deep main sinter is underlain by a zone of strong silicification with silica flooding and chalcedonic quartz veins.

Small amounts of fine-grained pyrite are present in silicified rocks that have not undergone later oxidation. In some parts of the deposit, particularly within arkose and sandy conglomerate units, silicification can be accompanied by potassic alteration in the form of adularia flooding. Orthoclase, present primarily in sand-size grains and in granitic clasts, is unaffected by potassic alteration, and plagioclase is replaced by adularia. The adularia is extremely fine-grained and is identified microscopically or by cobaltinitrite staining. Silicic and potassic alteration zones are surrounded by barren, unaltered, clay-rich (20-40% montmorillonite), tuffaceous siltstone and arkose with minor diagenetic pyrite.

The Grassy Mountain gold-silver deposit is located largely within the silicic and potassic alteration, zones, beginning approximately 200 feet below the surface. The deposit has extents of 1,900 feet along a N60°E to N70°E axis, as much as 2,700 feet in a northwest-southeast direction, and as much as 1,240 feet vertically. The surface expression of the mineralization is indicated by weak to moderately strong silicification and iron-staining, accompanied by scattered, 1/8- to 1.0-inch wide creamy to light-gray chalcedonic veins that filled joints.

The deposit is comprised of a central higher-grade core with gold grades of $\geq$~0.03 oz Au/ton that is surrounded by a broad envelope of lower-grade mineralization. The central higher-grade core is almost 1,000 feet long on the N60°E to N70°E axis, by 450 feet in width and 450 feet in vertical extent, all of which is above the Kern Basin Tuff and below a distinctive sinter unit. Representative cross sections through the deposit are shown in Section 14.7 (see Figure 14.1 through Figure 14.4).

Central Higher-Grade Core Zone

Three distinct and overlapping types of gold-silver mineralization are recognized within the central core of the deposit. These are gold-bearing chalcedonic quartz ± adularia veins, disseminated mineralization in silicified siltstone and arkose, and gold and silver in bodies of clay matrix breccia.

Zones of high-grade mineralization are defined by the presence of chalcedonic quartz ± adularia veins. Mineralized quartz ± adularia vein types include single, banded, colliform, brecciated and calcite-pseudomorphed veins. The colliform veins tend to carry the highest grades (>0.5 oz Au/ton), with visible gold up to 0.02 inches associated with argentite. Veins with relict bladed calcite texture also contain higher gold grades than the banded and single vein types. The gold mostly occurs as electrum along the vein margins or within microscopic voids. Some veins carry very little grade or are barren. At least some of the higher-grade zones of veins are thought to strike approximately N70°E.

Vein widths range from 1/16 to ~2.0 inches. Individually, such narrow veins are unlikely to have lateral or vertical extents of significance, but vein frequency can average one vein per foot in places. Vein swarms have strike lengths of 400 to 700 feet and vertical extents of 100 to 250 feet at elevations of 3,150 to 3,400 feet. Individual veins are too narrow to trace or correlate from hole to hole.

A steep southerly dip (70-85°) of the veins is inferred from vein intersection angles with drill core axes and bedding. Veins are mostly perpendicular to bedding, which generally dips 10-25° NNE within the deposit. Vein intersection angles of 10-25° to the core axis were mostly recorded in core holes GMC-001 to GMC-008 angled at -50° at S20°E, compared with 25° to 50° intersection angles in holes GMC-009 to GMC-011 angled -50° at N20°W. The N70°E strike of the veins is supported by: 1) surface mapping, 2) vein orientation perpendicular to bedding, 3) grade-thickness contouring, and 4) the overall trend in mineralization with grades in excess of ~0.03 oz Au/ton.

The veins cross-cut the silicified sediments and have extremely sharp grade boundaries with the sediments. Vein frequency diminishes abruptly below an elevation of ~3,000 at the west-southwest limit of the higher-grade core to ~3,100 feet at the east-northeastern limit, and very few high-grade veins have been encountered above the higher-grade core of the deposit.

Within the higher-grade core, high gold grades are also present in silicified siltstone and arkose with no visible veins. In these cases, gold and silver are inferred to be very finely disseminated in a stratiform manner in the silicified rock. Fine-grained pyrite is commonly disseminated in the silicified siltstone and sandstone where oxidation has not occurred. Contacts between siltstone and arkose beds seem to be more favorable and carry higher gold grades. In places, beds of tuff and tuffaceous siltstone appear to be particularly favorable host for higher-grade mineralization that lacks associated veins.

The third style of gold-silver mineralization has been referred to by the Newmont and later operators as “clay matrix breccia”, bodies of which may be more prevalent in the lower portion of the higher-grade core of the deposit and are interpreted to extend at near-vertical angles up and down into the surrounding, low-grade envelope. Clay matrix breccias are mainly of clast-supported types and contain sub-rounded to sub-angular, sand- to boulder-sized clasts of silicified and/or veined arkose and siltstone with minor amounts of clay and iron-oxide minerals between the clasts. In drill core, clay matrix breccia intervals are intersected over lengths of as much as several tens of feet, but their true thickness and exact orientations are poorly understood, in part because their margins are commonly irregular-to-gradational and not planar, except where structural fabrics related to fault movement are evident. In some cases, it is difficult to discern where clay matrix breccias end and similar fault-related breccias begin; it is possible the two are in some cases genetically related.

Clay matrix breccias cut, and are therefore paragenetically later than, the silicification and veins. One interpretation is the clay matrix breccias formed by explosive releases of over-pressured water vapor, through faults and fractures during boiling in the waning stages of the hydrothermal activity.

Lower-Grade Envelope

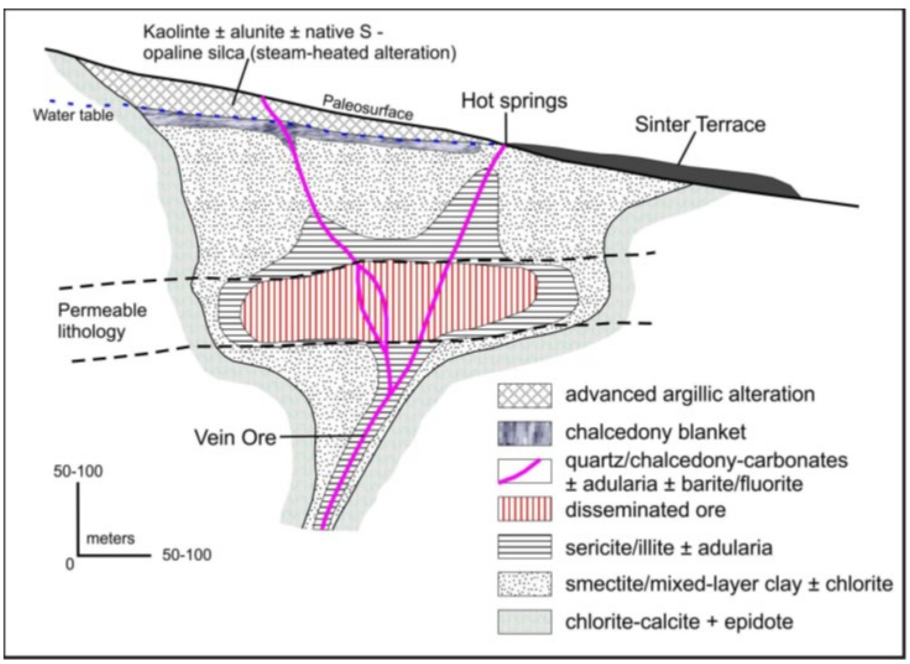
Lower-grade mineralization envelopes the higher-grade core and, further from the core, extends outwards as stratiform, mineralized lenses parallel to bedding (Figure 14.1 and Figure 14.2). There are very few visible chalcedonic veins; the gold and silver are inferred to be disseminated within the silicified arkose and siltstone units. Contacts between arkose, siltstone, and sinter appear to have been preferentially mineralized, and beds of tuff and tuffaceous siltstone also were favorable sites for mineralization. Low-grade mineralization is also present in numerous intervals of silica sinter, but not all sinter intervals are mineralized. Sinter-hosted mineralization may be disseminated, or within fractures where the sinter has been structurally disrupted.

8.0 DEPOSIT TYPES

The geological setting, hydrothermal alteration, styles of gold-silver mineralization, and close spatial and timing association with silica sinter deposition, indicate that Grassy Mountain is an example of the hot-springs subtype of low-sulfidation, epithermal precious-metals deposits. The Grassy Mountain deposit is characterized by stacked sinter terraces that demonstrate hydrothermal fluids vented at the paleosurface concurrent with lacustrine and intermittent fluvial sedimentation. At a depth of 300 feet, the main sinter at Grassy Mountain is underlain by a zone of intense silicification, within which is located the core of the deposit that is the focus of this report.

A conceptual, schematic section (Figure 8.1) shows a low-sulfidation epithermal system and its variable form with increasing depth, and the typical alteration zonation, including the distribution of sinter, a blanket of steam-heated advanced argillic alteration, and water-table silicification (Buchanan, 1981; Sillitoe, 1993). In the case of Grassy Mountain, the broader lower-grade mineralization extends up to and overlaps multiple, stacked deposits of sinter, reflecting near-surface epithermal mineralization as the sedimentary sequence accumulated.

Figure 8.1 Conceptual Hot-Springs Epithermal Deposit Model
(modified from Buchanan, 1981)



9.0 EXPLORATION

Paramount conducted infill, geotechnical, and metallurgical drilling at Grassy Mountain in 2016 and 2017. A total of 22,980 feet were drilled in 30 holes. The drilling focused on the central higher-grade core of the deposit and significantly improved Paramount's knowledge of the continuity and styles of mineralization within this core, while providing samples for geotechnical and metallurgical testing. Details of Paramount's drilling methods and procedures are summarized in Section 10.0. The results of the 2017 drilling are integrated in, and contribute significantly to, the estimation of the Grassy Mountain mineral resources discussed in Section 14.0.

In early 2017, Paramount commissioned an exploration review of the Grassy Mountain project data to evaluate and define exploration drilling opportunities for potential expansion of mill-grade gold resources. This study was focused on the area within the Grassy Mountain claim group controlled by Paramount and was carried out and reported by Mr. Steven Weiss, Senior Associate Geologist for MDA (Weiss, 2017).

Mr. Weiss first compiled and evaluated geological and geophysical maps, soil and rock-chip assay data, and aerial images from files supplied by Paramount. During March 2017, Mr. Weiss reviewed RC drill cuttings and core, drill logs, paper maps, cross sections, and other files at Paramount's office in Vale, Oregon. As part of this review, field traverses were made throughout the property to better understand the geology, rock geophysical response, and effects of hydrothermal alteration within the claim group.

Based on the field traverses, Mr. Weiss noted the high-potassium zones shown by the Newmont airborne radiometric data are likely controlled by abundant potassium-bearing clasts within exposed stratigraphic units of the Grassy Formation, and therefore concluded they are not the result of extensive potassic alteration. District patterns of low total magnetic intensity visible in the Newmont airborne magnetic maps also appear closely related to stratigraphy, as well as regional faults of the Oregon-Idaho graben, rather than major zones of hydrothermal alteration.

Zones of high resistivity defined by the 2012 CSAMT survey correlate in part with the thick volume of silicified rocks that host the Grassy Mountain gold deposit. Drill data and RC chips show the resistivity high that extends southwest from the deposit toward the Crabgrass deposit, and the outlying resistivity high at the Wood area, are not the result of extensive silicification (Weiss, 2017). In these areas, the CSAMT high resistivity response may be from the underlying Kern Basin Tuff ("Tkt") and rhyodacite of Butterfly Hill ("Trd") units.

Four near-mine drill targets were identified at the Grassy Mountain deposit and were recommended for limited expansion drilling (Weiss, 2017). Drilling recommended to test these targets is summarized in Section 26.1. The near-mine targets have significant uncertainties in their locations due to uncertainties in the precise locations, dips, amount of displacement, and timing of the Apache-Coyote and Gopher faults, and the northeast-trending fault in the North Spur, all of which are viewed as potentially mineralized structures. Nevertheless, these targets were considered to be justified by the combination of their proximity to the proposed underground mine and the opportunity to expand the Indicated category of resources, even if only incrementally (Weiss, 2017).

Two separate targets in the outlying Wood prospect were also recognized to have the potential for structurally controlled vein or stockwork mineralization (Weiss, 2017). Drilling recommended for these targets is also summarized in Section 26.1.

Additional surface work was also recommended by Weiss (2017) with the goal of defining further exploration drill targets. This included expansion of the CSAMT coverage to better understand the subsurface at the Crabgrass, Bluegrass, North Bluegrass, Ryegrass and Dennis' Folly areas. The large geochemical anomaly north of Snake Flats was recommended for verification and infill soil sampling and trenching that could help to define one or more new drilling targets. The Dennis' Folly area also was recommended for a modest infill soil-sampling program, the results of which could help define or improve a drilling target there as well (Weiss, 2017).

10.0 DRILLING

Drilling at the Grassy Mountain property is summarized in Table 10.1 and shown in Figure 10.1. The project database includes a total of 264,111 feet drilled by four historical operators, from 1987 through 2012, in 442 holes. Paramount drilled 30 holes and 22,980 feet in 2016 and 2017 to bring the property total to 472 holes and 287,083 feet drilled. Approximately 77% of the footage drilled was at, and adjacent to, the Grassy Mountain deposit area, although nearly 43% of the holes were drilled at outlying prospects (Section 6.2), including the Crabgrass deposit, as well as for water wells. The bulk of the holes at the Grassy Mountain deposit area was drilled entirely by RC, accounting for 77% of the footage drilled. Holes drilled using diamond-core methods account for about 12% of the footage drilled in the deposit area, and holes drilled with RC pre-collars and core tails account for about 11% of the footage drilled. The locations of holes drilled in and near the Grassy Mountain deposit area are shown in Figure 10.2. The reader is referred to Figure 6.1 for a map showing the collar locations of holes drilled to test outlying prospects. The results of drilling at the outlying prospects are summarized in Sections 6.2 and 6.4.

Table 10.1 Grassy Mountain Drilling Summary

Year	Company	# Holes	Hole type	Length (ft)	Area
1987-1991	Atlas	193	RC	154,963	Grassy Mtn
1989-1991	Atlas	5	Core	4,153	Grassy Mtn
1989-1991	Atlas	5	RC & Core	3,502	Grassy Mtn
1987-1991	Atlas	187	RC	62,895	Outlying Prospects
1987-1991	Atlas	10	RC	1,884	Water wells
1992-1996	Newmont	13	Core	13,101	Grassy Mtn
1992-1996	Newmont	2	RC & Core	1,909	Grassy Mtn
1998	Tombstone	4	RC	3,145	Grassy Mtn
1998	Tombstone	6	RC & Core	4,926	Grassy Mtn
2011	Calico	3	Core	2,531	Grassy Mtn
2011-2012	Calico	10	RC	8,518	Grassy Mtn
2012	Calico	4	RC	2,585	Outlying prospects
Historical Total:		***442***		***264,111***	
2016-2017	Paramount	3	RC	1,140	Grassy Mtn
2016-2017	Paramount	3	Core	1,933	Grassy Mtn
2016-2017	Paramount	24	RC & Core	19,907	Grassy Mtn
Paramount Total:		***30***		***22,980***	
All Drilling Total:		***472***		***287,091***	

Within the Grassy Mountain deposit area, approximately 80% of the holes were vertical, or within 3.0° of vertical. Approximately 69% of the core and core-tail holes were inclined at less than -80°. Overall results of drilling within the Grassy Mountain deposit are summarized with representative cross-sections presented in Section 14.0.

At the outlying prospects, where all of the drilling was done with RC methods, approximately 98% of the holes were vertical. The median hole depth was 300 feet outside the deposit area.

Figure 10.1 Location of All Drill Holes within the Claim Area

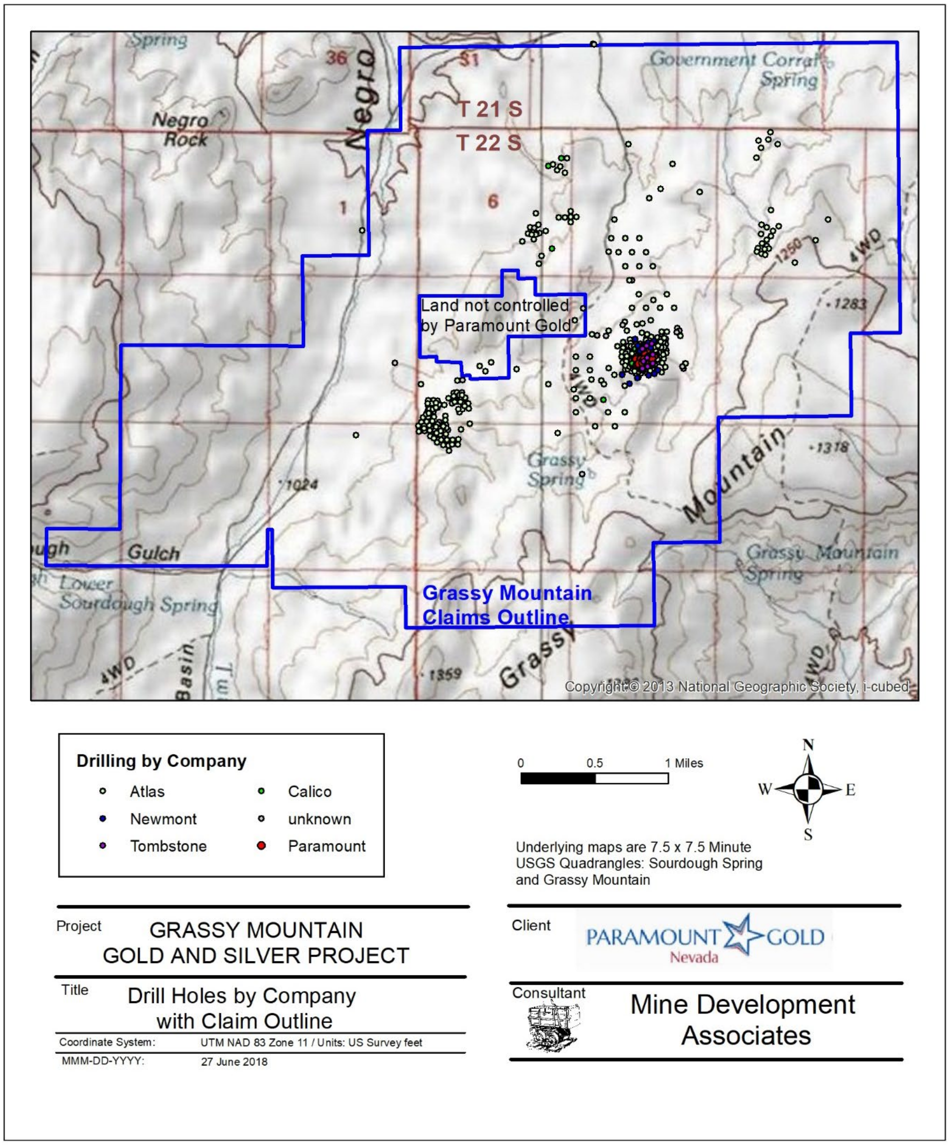
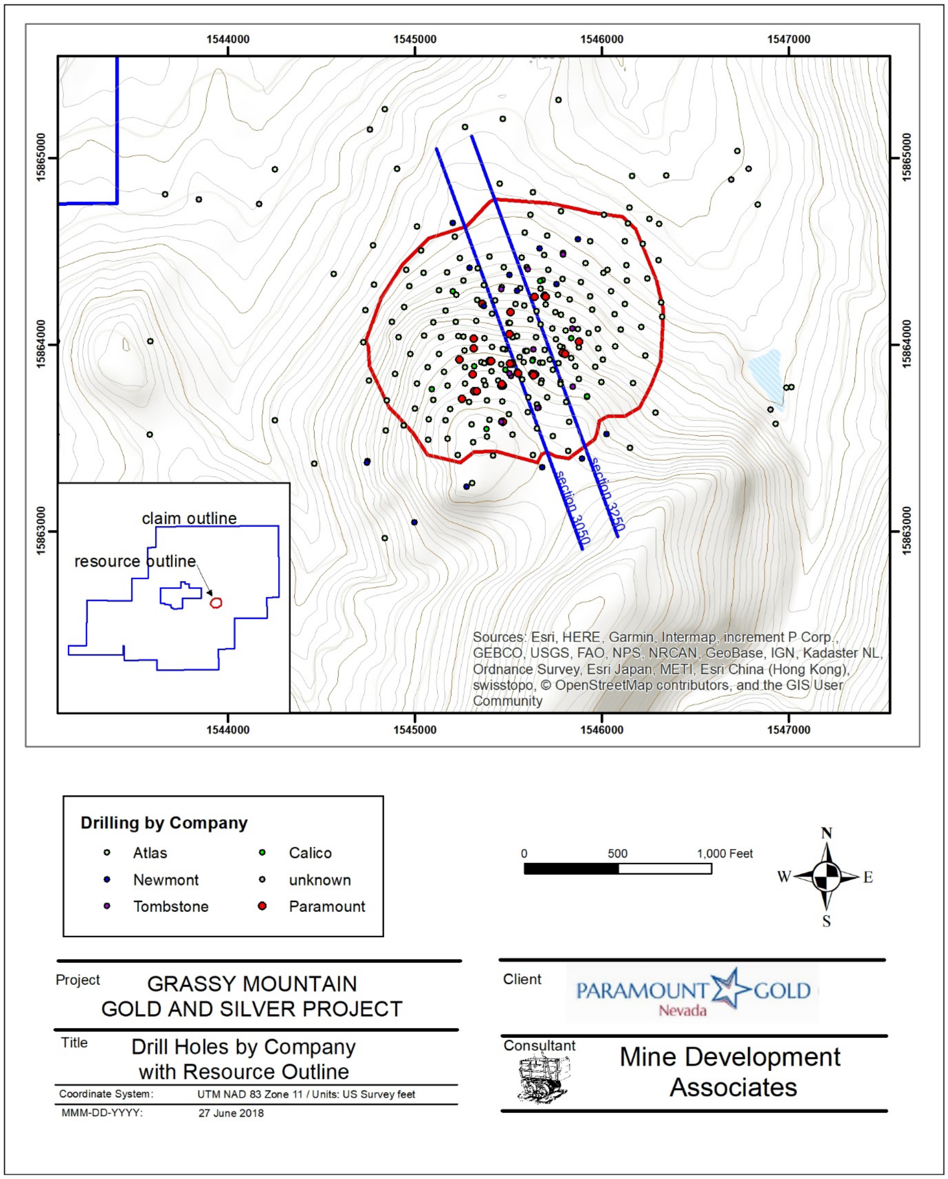


Figure 10.2 Location of Drill Holes in the Grassy Mountain Deposit Area



10.1 Historical Drilling 1987 - 2012

10.1.1 Atlas 1987 - 1992

A small track-mounted rig was mobilized in early 1987 to drill six holes in two target areas. Drill hole 026-004 intercepted 80 feet of mineralization averaging 0.021 opt Au. A follow up drill program consisting of five holes was completed in the spring of 1988. Drill hole 026-009 is considered to be the Grassy Mountain deposit discovery hole, with an intersection of 145 feet of mineralization that averaged 0.075 opt Au. By the end of 1991, Atlas had drilled 227,397 feet in 400 holes. Of the total, 13 holes were drilled for water wells and 187 holes were drilled at outlying prospects.

The Atlas RC holes were drilled by Eklund Drilling Company from Elko, Nevada, using Ingersoll Rand TH-60 and RD-10 truck-mounted drills with a nominal hole diameter of 5¼ inches (Lechner, 2007). The RC cuttings were sampled at five-foot intervals. Twenty-three of the Atlas RC exploration holes were drilled to at least 1,000 feet in depth, and all of these are at the Grassy Mountain deposit area. RC drilling was "almost invariably" done dry, as groundwater was reportedly not encountered above 750-foot depths, with the exception of some local perched water that was intersected along the northern portions of the deposit. Because the deposit is strongly silicified, drilling penetration rates were slow and resulted in excessive bit wear. Drilling in certain areas was completed with some difficulty due to tight hole conditions and caving of rubble zones. In many cases, historical documentation is not sufficient to ascertain with confidence whether a particular hole was drilled dry or wet.

Atlas drilled 10 core holes at Grassy Mountain to confirm high-grade mineralization identified by RC drilling, obtain samples for metallurgical testwork, and to collect geotechnical data. Two confirmation core holes were drilled as NC and NQ angle holes by Longyear, Incorporated. Five core holes drilled specifically to obtain sample material for metallurgical testing were drilled as vertical PQ diameter holes by Boyles Brothers; these holes were pre-collared with RC. Three geotechnical holes were also drilled by Boyles Brothers. Assay records indicate that the confirmation holes were sampled on intervals ranging from 0.5 to 7.5 feet in length, with an average sample length of 4.5 feet. MDA is uncertain whether the core was mechanically split in half or sawed in half for sampling. Whole core from the metallurgical holes was shipped to Hazen Research Inc. for metallurgical testwork, and the geotechnical holes were logged for various geotechnical parameters such as rock quality designation ("RQD"), fracture frequency, etc.

An Atlas geologist was assigned to each drill rig and was responsible for the placement of the rig, drilling and sampling methods, hole depths, and lithologic logging.

The Atlas drilling discovered and completed the initial delineation of the Grassy Mountain deposit. Atlas also discovered and completed all drilling of the Crabgrass deposit.

10.1.2 Newmont 1994

Newmont drilled 14 angled core holes and used a wedge off their first hole to drill a 12th core hole. Two of the last three core holes were pre-collared with RC. This drilling totaled 15,010 feet and was conducted by Longyear Incorporated of Spokane, Washington. All of the holes were drilled with HQ-diameter core, with the exception of six holes in which the HQ core was reduced to NQ-size due to ground conditions. The RC pre-collar portions were sampled over intervals of 5.0 feet. Approximately 90% of the core was sawed in half for sampling, with the remainder mechanically split in half.

Newmont determined that high-grade gold was hosted by steep, southeast-dipping quartz-chalcedony-adularia veins. Vein widths ranged from ¼ inch to two inches and averaged ½ inch. Vein spacing within high-grade zones averaged one vein per foot. The steep southeast dip was inferred from comparison of vein/core intersection angles from southeast-directed holes with those in northwest directed holes. High-grade gold mineralization was inferred to have a relatively sharp base at an elevation of 3,000 to 3,100 feet.

10.1.3 Tombstone 1998

In 1998, Tombstone drilled six core holes with RC pre-collars and four complete RC holes that altogether totaled 8,071 feet of drilling. Dateline Drilling Incorporated from Missoula, Montana performed all of Tombstone's RC drilling. RC samples were collected over 2.5 and 5.0-foot intervals, with both interval lengths sometimes used in the same hole. The RC drilling was conducted wet, as water and mud was used for hole conditioning. The diamond-core drilling was done by Ray Hyne Drilling of Winnemucca, Nevada, while Dateline Drilling completed the RC drilling. Approximately 80% of the core was sawed in half for sampling, with the remainder mechanically split in half.

The Tombstone drilling was concentrated in the higher-grade core of the deposit, with the goal of better defining the higher-grade mineralization. The Tombstone results, however, were judged not to have included the very high-grade (> 2 oz Au/ton) component of the Grassy Mountain mineralization that was encountered in previous Atlas RC and Newmont core holes (French, 1998). French (1998) theorized that the lack of very high-grade intersections might be due to the drilling and related sampling problems encountered during the program. He recommended the use of a more powerful RC rig that would be less susceptible to poor ground conditions and therefore require less hole reaming and conditioning, which would lead to uninterrupted drilling and sample collection.

10.1.4 Calico 2011 - 2012

Calico commenced drilling in August 2011. Three core holes were drilled at the Grassy Mountain deposit using a modified track-mounted LF-90 core drill operated by Marcus and Marcus Drilling Company out of Post Falls, Idaho ("Marcus and Marcus"). HQ (2.5-inch) diameter core was drilled using a triple-tube core recovery barrel. Operating 24 hours per day, a total of 2,530.5 feet of drilling was completed, with average production of 39 feet per day.

A truck-mounted Ingersoll-Rand TH-75 drill operated by Boart Longyear out of South Jordan, Utah began RC drilling at the Grassy Mountain property in October 2011. The drill utilized a cyclone wet splitter for sample collection, with an approximate 40% split retained in the sample bag. Drill cuttings passed through a cyclone and then divided into three streams through the splitter: one for sampling, one for logging and retention for reference, and the excess discarded to the sump. A portion of the sample collected for logging was placed into a plastic chip tray labeled with the hole number and the depth from which the sample was taken. The drill helper collected one sample for each five-foot interval in bags pre-labeled with the sample number under supervision by the site geologist. Each sample bag was sealed at the drill site and remained unopened until it reached the analytical laboratory. After each 20 ft length of drill rod was added to the drill string, the hole was cleaned of material which may have descended while the new section of rod was installed. Calico's 2011 RC samples were partially dried at the drill site prior to shipment for assay. Samples received at the assay laboratory averaged 20 pounds in weight.

The RC drill operated on a single 12-hour daily shift. A Calico geologist was on-site during drilling to monitor the drilling and sample collection, log the drill cuttings, and collect and store a portion of the drill cuttings for future reference. The RC drill rig completed nine holes at the Grassy Mountain deposit area totaling 7,668 feet.

During June of 2012, Calico drilled a total of 3,435 feet in five RC holes. One hole was drilled in the Grassy Mountain deposit area, one was drilled in the Wheatgrass area, one was drilled at the Wood area, and two holes were drilled at the Wally area. Leach Drilling of Dayton, Nevada was contracted for the job using an Ingersoll-Rand DM25/RC track-mounted rig. A cyclone wet splitter was used for sample collection with approximately 40% of the sample retained in the sample bag for analysis. The sampling procedures were the same as those used in 2011. The drill operated on a single 12-hour daily shift. A Calico geologist was on-site during the drilling to monitor the drilling and sample collection, log the drill cuttings, and collect a portion of the drill cuttings for future reference. The drill program was completed on June 28.

The thirteen holes drilled at the Grassy Mountain deposit area increased the drill density within the higher-grade core of the deposit, with the core holes providing much needed additional information regarding the higher-grade mineralization. The hole drilled at Wheatgrass returned results consistent with existing holes in the target area, while the hole drilled at the Wood target was drilled almost 450 feet from the nearest existing hole and returned only very low-grade intersections. The first hole drilled in the Wally area unsuccessfully tested the western extension of previously defined mineralization, while the second hole returned similar results as the existing holes and thereby confirmed the extension of this low-grade mineralization about 200 feet to the north.

10.2 Paramount 2016 - 2017

In support of the updated resource estimation and the PFS reported herein, which evaluates an underground mining operation, Paramount drilled 22,980 feet in a total of 30 holes within the higher-grade core of the Grassy Mountain deposit in 2016 and 2017. The goals of this drilling program included: (*i*) the verification of the historical drill data, particularly the historical RC holes; (*ii*) substantially increasing the quantity of drill core derived from the higher-grade portion of the deposit; (*iii*) obtaining better definition of the controls and extents of the higher-grade mineralization; and (*iv*) obtaining drill core for use in detailed geotechnical logging and metallurgical testing.

Prior core drilling experienced significant problems due to poor ground conditions, particularly in the uppermost portion of the deposit down to the bottom of the upper sinter package. Paramount therefore decided to pre-collar the core holes with RC to depths of approximately 400 feet to 500 feet, which then allowed for core drilling throughout the higher-grade core of the deposit.

Major Drilling America Inc., of Salt Lake City, Utah, was contracted for both the RC and core drilling. RC pre-collars were drilled with a Schramm T450GT track-mounted drill that was operated on a single 12-hour daily shift. A 6½ inch diameter RC bit was used to the planned pre-collar depth. Once the planned depth was reached, 4½ inch steel casing was set for the entire length of the hole and the drill rig was moved to the next RC pre-collar location.

During the RC drilling, small amounts of water were injected down the hole to control dust emissions. RC samples were collected at nominal five-foot intervals via a cyclone rotary splitter and center-discharge tube into 20-inch by 24-inch sample bags that were pre-numbered by Paramount geologists or geotechnicians. Samples typically weighed approximately 15 to 20 pounds for each sample interval. A Major Drilling sampling assistant was on-site during drilling operations to monitor the drilling, perform the sample collection, and collect and store a portion of the drill cuttings in plastic chip trays for future reference and logging. The sampling assistant was trained by a Paramount geologist who was on-site for the first seven RC pre-collars.

Duplicate RC samples were collected at the rate of approximately one per forty regular sample intervals. For duplicate samples, the primary sample was collected from the center discharge tube of the rotary splitter and the duplicate sample was collected from the side discharge tube of the rotary splitter. A "Y-type" splitter was not used at any time for duplicate samples.

Core drilling was accomplished with two track-mounted drills: a Boart Longyear LF-90 drill, and a Boart Longyear LF-230 drill. Both rigs drilled HQ (2.5 inch) diameter core using a triple-tube type core barrel. The drills operated 24 hours per day on two 12-hour shifts, each manned by a two-man crew. A drill foreman was on site as well. A single water truck and driver was able to supply adequate water for the two drills, hauling water from a well approximately one mile north of the drilling area.

Drilling of the first RC pre-collar began in November 2016 and seven RC pre-collars totaling 2,695 feet were completed during the year. Core totaling 3,078 feet was drilled in six holes in 2016. Drilling was suspended from mid-December 2016 through early March 2017. During March, April, and May of 2017, 20 RC pre-collars totaling 8,556 feet were drilled. From March through June of 2017, 8,651 feet of core were drilled in 21 holes. Footages drilled by pre-collar RC and core methods are shown in Table 10.2. Average drill production was 142 feet per 12-hour shift for RC, and 31.1 feet per drill, per 12-hour shift, for core drilling. Three of the RC pre-collars encountered extremely bad ground conditions that led to premature terminations of the holes and precluded the drilling of core in these holes.

All of the goals of Paramount's drilling program were achieved. Beyond obtaining core for detailed geotechnical logging and metallurgical testing, the drill core aided in furthering the understanding of the geology of the deposit, which largely confirmed many of Newmont's conclusions. This in turn formed the base from which the resource model was constructed. Finally, the results of the Paramount drilling program have aided in the verification of the historical data (e.g., see discussion of estimating with and without Paramount drill data in Section 14.9).

Table 10.2 Paramount 2016-2017 RC Pre-Collar vs. Core Lengths

Drill hole	Pre-Collar RC From (ft)	Pre-Collar RC To (ft)	Core From (ft)	Core To (ft)	Total Feet RC	Total Feet Core	Notes
GM16-01	0	380			380	0	Stuck hammer
GM16-02	0	400	400	742	400	342	
GM16-03	0	380	380	785	380	405	
GM16-04			0	744.5	0	744.5	Geotech drill hole
GM16-05	0	360	360	618	360	258	
GM16-06	0	400	400	731	400	331	
GM17-07	0	391	391	850.5	391	459.5	
GM16-08	0	375			375	0	Twisted off rods
GM16-09	0	400	400	795	400	395	
GM17-10	0	400	400	822	400	422	
GM17-11	0	385			385	0	Stuck hammer
GM17-12	0	395	395	689	395	294	Re-drill of GM16-08
GM16-13			0	438.5	0	438.5	Twisted off rods
GM16-14			0	750	0	750	Geotech drill hole
GM17-15	0	320	320	780	320	460	
GM17-16	0	480	480	923	480	443	
GM17-17	0	480	480	929.5	480	449.5	
GM17-18	0	450	450	884.5	450	434.5	
GM17-19	0	450	450	857.5	450	407.5	
GM17-20	0	380	380	856	380	476	
GM17-21	0	460	460	832	460	372	
GM17-22	0	500	500	953.5	500	453.5	
GM17-23	0	400	400	956	400	556	
GM17-24	0	450	450	896	450	446	
GM17-25	0	400	400	887	400	487	
GM17-26	0	520	520	875	520	355	
GM17-27	0	440	440	772	440	332	
GM17-28	0	420	420	862	420	442	
GM17-29	0	440	440	800	440	360	
GM17-30	0	400	400	810	400	410	

10.3 Drill-Hole Collar and Down-Hole Surveys

For the Atlas drilling, collar locations were surveyed by Apex Surveying from Riverton, Wyoming using a total station. Most holes were not surveyed for down-hole direction and deviation, except four RC holes and all of the core holes, which were surveyed using an Eastman down-hole camera (Lechner, 2007).

It is not known with certainty whether Newmont's collar locations were surveyed. Down-hole deviation surveys of the Newmont holes were performed by Scientific Drilling from Elko, Nevada. Newmont handwritten "Drill Hole Summary" sheets indicate that the holes were surveyed using a "gyro" instrument.

For the Tombstone drilling, there are no written records of how the collar locations were surveyed (Lechner, 2007). Surveys of down-hole deviation were reportedly done by Silver State Surveys of Elko, Nevada using a gyroscopic survey tool, but no written records are present in Paramount's archives. No down-hole survey data are available for three of the Tombstone drill holes.

Up until Calico's involvement in the project in 2011, the project coordinates were based on a local grid established by Atlas. All Calico and subsequent drill-hole collar surveys were collected directly in UTM coordinates. See Section 12.1.1 for a discussion on the transformation of historical mine-grid collar locations into UTM coordinates.

During 2011 and 2012, drill collar locations were surveyed by Calico personnel using hand-held Garmin GPS units with a horizontal accuracy on the order of ±10 feet, and later surveyed with a Trimble, survey-grade GPS to ±0.1 feet. Holes were marked in the field with a lath and/or stake.

The 2011 core holes were surveyed for down-hole directional deviation by Marcus and Marcus using a REFLEX EZ-Track survey instrument to obtain multi-shot readings. The 2011 RC holes were surveyed for down-hole deviation by International Directional Services ("IDS") using a Goodrich-Humphrey surface-recording gyroscopic system. Deviation from planned orientations was generally on the order of 3° for core and RC holes, although some of the RC holes deviated by up to 6° in azimuth and 8° in dip.

Down-hole surveys were not performed in the first four of the 2012 RC holes. The final 2012 hole, CAL12R17, was surveyed for down-hole deviation by IDS using a Goodrich-Humphrey surface recording gyroscopic system.

In 2016 and 2017, the Paramount drill-collar locations, as well as many of the historical drill collars in the Grassy Mountain deposit area (see Section 12.1.1), were surveyed by Atlas Land Surveying of Fruitland, Idaho. The owner, Dean J. Coon, is a Registered Professional Land Surveyor (Oregon 65687LS) and was responsible for the field work, data processing, and reporting. All survey work was completed using real-time kinematic ("RTK") surveying techniques with Topcon Hiper V GPS Receivers. In RTK mode, the stated accuracy of the measurements is within 10 mm +/-1 mm for horizontal data and 15 mm +/-1 mm for vertical data. Static data were collected in the field and then submitted to the National Geodetic Service Online Positioning User Service to derive accurate geodetic coordinates tied to the National Spatial Reference System. Using these coordinates, the RTK data were processed through a survey measurement adjustment program, "StarNET", to determine the final coordinates for the located points. These data were projected to the Universal Transverse Mercator grid using the NAD83 datum in units of U.S. Survey feet.

Down-hole deviation surveys of 25 of the 2016 and 2017 Paramount drill holes were performed by IDS of Elko, Nevada using a Goodrich surface-recording gyroscopic system ("SRG"). The SRG is capable of mapping the direction of boreholes and is unaffected by steel pipe or local magnetic-field anomalies. Five of the 2016-2017 drill holes had blockages, such as lost or stuck pipe, casing, or core barrel, that prevented down-hole surveys.

10.4 Sample Quality

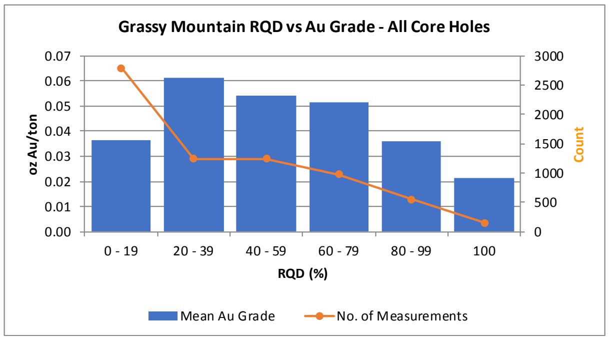
10.4.1 Core Samples

In consideration of the presence of visible gold in the drill core, Newmont decided to evaluate the potential for unrepresentative loss of gold in the splitting of drill core for sampling. During the sampling of their first hole (GMC-001), the minus 10 mesh fines produced during the sawing of drill core into halves were collected for each sample, weighed, and assayed separately (Jory, 1993). Jory reported that the mean of the gold assays of the 171 samples of saw fines collected was 86% higher (0.044 versus 0.024 oz Au/ton) than the associated half-core samples sent to the laboratory. Jory further noted that since the saw fines accounted for less than 0.5% of the total sample weight, sampling of the saw fines was discontinued. However, Newmont did take 38 additional saw-fines samples for hole GMC-001-9, a core wedge off of GMC-001, for which the assay certificate is available. The average of the saw-fines assays is 0.438 oz Au/ton and the mean of the half-core assays is 0.143 oz Au/ton; Newmont did not obtain silver assays for any of their drill samples. The high bias in the saw fines relative to the half-core samples is present at all gold grades, but it increases as the grade increases.

While the unrepresentative loss of gold to the saw fines is not material due to the small amount of these fines relative to half-core samples, these data suggest the potential for the unrepresentative loss of gold to fines that may be generated by other means. One such possibility is in fines that collect in core boxes from broken intervals, which clearly warrant careful collection and splitting along with the sawing of competent pieces of core. Newmont brushed fines out of the core boxes for each sample interval and split the fines into halves, with one half added to the sample bags of sawed core sent to the assay lab and the other half bagged and returned to the core boxes.

Fines can also be lost below the surface during the drilling of core. In an attempt to evaluate this possibility, the relationship between geotechnical data (core recovery and RQD) collected during the logging of the core and gold grades was examined. RQD value is a measure of the degree of natural, in situ fracturing of the mass of rock sampled by the core. A value of 100% represents solid core for the entire interval measured, while 0% applies to a core run in which all pieces of core are less than 10 centimeters (2.5 inches) in length. Figure 10.3 summarizes the relationship between gold grade and RQD for all Grassy Mountain core holes for which RQD data are available.

Figure 10.3 Gold Grade vs. RQD

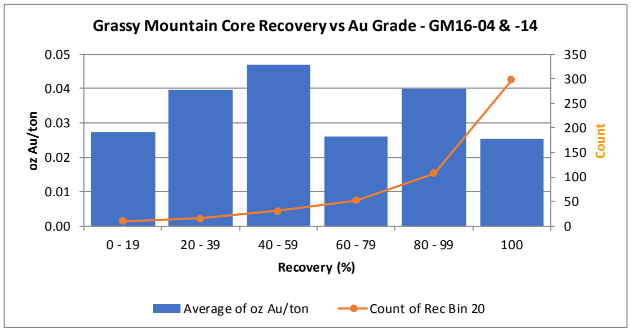


Each blue bar in the graph includes data within a 20% RQD bin, as indicated on the x-axis (RQDs of 100% and greater report to the "100" bin). The heights of the bars are indicative of the average grade of all intervals within the each of the recovery bins, as shown on the y-axis of the left-hand side of the graph. The total number of RQD intervals in each recovery bin is displayed by the orange line, with the scale provided by the y-axis on the right-hand side of the graph.

With the exception of the lowest RQD bin, there is a consistent correlation between RQD and gold grade in which gold grades increase as RQD decreases. This negative correlation is at least in part due to the relationship of higher-grade mineralization with highly fractured zones that yield low RQD values. In some deposits, however, unrepresentative loss of soft, clay-rich, and relatively unmineralized material from the recovered drill core occurs in low RQD zones, which would lead to increased grades in the recovered samples of core. The Grassy Mountain mineralization of all grade ranges is associated with uniformly strong silicification, however, so this mechanism of apparent grade increases is unlikely. As far as the possibility of losing gold related to fines during drilling, the negative correlation between RQD and gold grade does not provide evidence of this, but potential losses cannot be definitively ruled out.

The RQD measurements used in this analysis were extensively reviewed and edited to assure their validity. The bulk of the core-recovery data has not gone through this validation and has many inconsistencies that need to be resolved. Two Paramount holes were validated, and the relationship between recovery and gold grade for these holes is summarized in Figure 10.4.

Figure 10.4 Gold Grade vs. Core Recovery



No clear trend is evident in the data at core recoveries of 60% and greater. Gold grades decrease with decreasing recoveries for core recoveries lower than 60%, but the number of recovery intervals in each bin is relatively low and likely insufficient to support definitive conclusions.

10.4.2 RC Samples

Due to the nature of RC drilling, the possibility of contamination of drill cuttings from intervals higher than the drill bit in the hole is a concern, especially when groundwater is encountered or fluids are added during drilling. Atlas RC holes were reportedly drilled dry unless groundwater was intersected, while Tombstone, Calico, and Paramount RC holes were drilled entirely wet. Comments on geologic logs and other historical documentation suggest that the water table at Grassy Mountain lies near the base of the higher-grade core of the deposit, with 'perched' groundwater noted in a few holes at much higher elevations.

Down-hole contamination can sometimes be detected by careful inspection of the RC drill results in the context of the geology (e.g., anomalous to significant assays returned from samples from post-mineral units), by comparison with adjacent core holes, and by examining down-hole grade patterns.

Cyclic down-hole grade patterns are evident in some of the RC holes at Grassy Mountain. These cycles consist of high gold grades (relative to adjacent samples) every fourth five-foot samples drilled with the same 20-foot drill rod. In a classic case, the first sample of the drill rod will have the highest grade, while the following three samples will gradually decrease in grade. This 'decay' pattern in grade is caused by the accumulation of mineralized material (derived from some level higher in the hole than the drill bit) at the bottom of the hole as the drilling pauses and a new drill rod is added to the drill string. When drilling resumes, the first sample has the greatest amount of contamination, and the successive samples are gradually 'cleaner' as the accumulated contamination is removed and the continuing contamination experienced during the drilling is overwhelmed by the material being drilled. This decay pattern is usually possible to detect only while drilling barren or very weakly mineralized rock. Even in cases where this cyclic gold contamination is of such low grade as to have minimal impact on resource estimation, its presence suggests that similar, and possibly more serious, contamination may have occurred higher in the hole within mineralization, where the contamination can be impossible to recognize.

Atlas did not believe down-hole contamination was a "significant or consistent problem" but did recognize that the bottom of hole 026-034 could be contaminated over a 200-foot interval. During the resource modeling and related detailed review of the project data, MDA identified 21 holes with suspected down-hole contamination of precious-metals values, primarily based on cyclic patterns described above. These intervals are all at the lowermost portions of the holes, and they were either excluded from the mineral domains that constrain the resource estimations or were excluded from use in the resource estimation on the basis of a "no use" code in the assay table.

10.5 Summary Statement

MDA believes that the drilling and sampling procedures provided samples that are representative and of sufficient quality for use in the resource estimations discussed in Section 14.0. Mr. Gustin is unaware of any sampling or recovery factors that have not been addressed that would materially impact the mineral resources discussed in Section 14.0.

Down-hole drilled lengths of the higher-grade gold and silver portions of the deposit, some of which are oriented at high angles, could significantly exaggerate true mineralized thicknesses in cases where steeply-dipping holes intersect the steeply-dipping mineralization. A very high percentage of the Atlas holes were drilled vertically. Possible effects of exaggerated down-hole lengths on the estimation of the current resources was carefully monitored and the model is believed to appropriately represent the higher-grade volumes.

The average down-hole length of the sample intervals used directly in the estimation of the resource gold and silver grades is 4.76 feet, with a minimum length of 0.3 feet and a maximum of 12 feet. The sample lengths are considered appropriate for the Grassy Mountain deposit.

Only four of the 177 Atlas RC holes that directly contribute assay data to the estimation of the project resources were surveyed for down-hole deviation. The four holes that were surveyed deviated from 14 to 35 feet horizontally from the drill collar positions to the distinct lower contact of the higher-grade zone (see Section 14.0), which lies approximately 800 feet below the surface. The average horizontal deviation is 22 feet. This magnitude of deviation is not a major concern considering the resource model block size of 10 x 10 x10 feet.

11.0 SAMPLE PREPARATION, ANALYSIS, AND SECURITY

This section summarizes all information known to Mr. Gustin relating to sample preparation, analysis, and security, as well as quality assurance and quality control procedures, that pertain to the drilling data from the Grassy Mountain deposit. The information has either been compiled under the supervision of Mr. Gustin from historical records as cited, or provided by Mr. Michael McGinnis, the project Manager for Paramount.

11.1 Sample Preparation, Analysis and Security

11.1.1 Atlas 1987 – 1992

The Atlas RC samples were split at the drill site to weigh between 8 and 15 pounds, averaging approximately 12 pounds, and were collected in 10-inch by 17-inch olefin sample bags. An Atlas geologist was stationed at the drill rig and with the samples at all times. Wet RC cuttings were split using a variable wet-cone splitter positioned below the cyclone on the RC rigs. Dry cuttings were split under the cyclone with a Jones splitter. The samples were delivered to a secure storage facility in Vale, Oregon at the end of each shift by Atlas project geologists. The samples were routinely picked up from the Vale storage facility by Chemex Analytical Laboratories ("Chemex") personnel and delivered to their preparation facility located in Boise, Idaho. The samples were dried at 100°C, cone crushed to minus 1/8 inch, and then 300-gram subsamples were taken using a Jones riffle splitter. These subsamples were then reduced to 95% at minus 100 mesh using a ring and puck pulverizer. The coarse reject materials were placed in storage at the Boise facility for possible future use. The 300-gram pulps were shipped by Chemex to their assay facility located in North Vancouver, Canada. Gold and silver were assayed using 30-gram aliquots that were analyzed by fire assay fusion, primarily with an atomic absorption ("AA") finish.

It is not known what type of certification Chemex may have had in 1987-1990, but it was a well-known, commercial assayer and was independent of Atlas.

11.1.2 Newmont 1992 - 1996

Jory (1993) reported that the Newmont core was cut into halves at the Vale field office with vein apices oriented perpendicular to the saw blade. Material too fine to be sawed was carefully swept out of the core boxes for each sample interval, split into halves using a Jones splitter, and recombined with the half-core to be sent for assaying. Newmont core boxes in the possession of Paramount include core fines inside zip-lock plastic sandwich bags, presumably representing the remaining half-split of fines for each sample interval.

Jory (1993) documented that the core samples were picked up by Rocky Mountain Geochemical Corporation ("RMGC") from the Atlas storage facility in Vale, Oregon, and delivered to the RMGC facility that was located in Salt Lake City, Utah, for sample preparation and analysis. A copy of a Newmont report that lacks a title page states that, "Coarse gold (up to 500 microns) problems necessitated careful sample prep procedures for Grassy Mountain core". The following summarized these procedures. The samples were dried at a temperature of 100°C, crushed to minus 10 mesh, split in half with a Jones riffle splitter, and coarse-pulverized to minus 48 mesh. A 200-gram split of the minus 48 mesh material was then ring-pulverized to a nominal, minus 150 mesh particle size, from which a 30-gram aliquot was fire assayed with gravimetric and AA finishes.

Newmont had screen-fire assays completed at RMGC on 20 samples from holes GMC-001 and -002 that had original gold assays in excess of 0.20 oz Au/ton.

There is no documentation regarding the sample security methods Newmont employed during their drilling campaigns.

It is not known what type of certification RMGC may have had in 1992-1996. RMGC was a well-known, independent commercial assayer of that era and was independent of Newmont. While the Newmont check analyses were completed by their in-house laboratory, and therefore were not independent of Newmont, the project database does not incorporate these analyses.

11.1.3 Tombstone 1998

Tombstone RC cuttings were passed through a rotary wet splitter below the cyclone to produce samples weighing 10 to 15 pounds. The splitter was washed before each new sample was taken. A five-gallon bucket placed under the splitter collected the wet samples, the water was partially decanted out of the bucket, and the RC cuttings and remaining fluid were emptied into the sample bag. The bucket was then washed to empty remaining fines into the sample bag as well, and then the sample bags were closed with one-way plastic ties. Tombstone brought the samples to the Vale field office, where they were later picked up by American Assay Laboratory ("AAL") of Sparks, Nevada.

The RC and half-core samples were prepared and analyzed by AAL. The samples were dried at 100°C, crushed to 8 to 10 mesh, and then passed through a Jones riffle splitter to produce a four-pound subsample. These subsamples were pulverized to 90% -150 mesh, blended, and then a 350-gram split was taken. A 30-gram aliquot from the 350-gram split was then analyzed for gold by fire assaying with an AA finish (AAL method FA30). Silver was analyzed by method D210, which included aqua-regia digestion. AAL was independent of Tombstone and remains a well-known commercial laboratory. It is not known what type of certification AAL may have had in 1998.

11.1.4 Calico 2011 – 2012

The 2011 and 2012 drilling samples were transported from the drill sites by Calico personnel to the Calico sample handling and core logging facility located in Vale, Oregon. For drill core, the date, box number, number of boxes transported, and beginning and ending footages of the transported core were recorded on a core handling form.

At the logging facility, Calico personnel measured and recorded core recovery and RQD data. The core was then logged by a Calico geologist who recorded lithological, alteration, mineralization, and structural information, including the angle of intersection of faults with the core, fault lineations, fractures, veins, and bedding. The entire length of core was then prepared for sampling. Sample intervals were based on the geological logs in an effort to separate different lithologies and styles of mineralization and alteration.

Sample length generally did not exceed five feet and, where possible, correlates to the five-foot drilling runs. If any significant veins, veinlets, healed breccias, or other potentially mineralized planar features were present, the geologist marked a line down the length of the core where the core should be sawed or split to ensure a representative sample was taken by the sampler. After logging was completed, sample intervals were marked and assigned a unique sample identification (sample tag), with the sample tag stapled inside of the box at the end of each sample interval. A duplicate sample tag for each interval was placed inside the sample bag, and the sample number was recorded in the sample tag booklet. If contamination or down-hole caving was observed, the interval was flagged and not sampled.

Once the core logging was complete and all of the sample intervals were marked, the core was sprayed with water and photographed. The core boxes were then moved to the sampling station where a technician either split the core with a hydraulic splitter or cut the core in half with a diamond-blade core saw. One half of the core was placed into a cloth sample bag labeled with the sample number. The other half was placed back into the core box for future reference. Core that was intensely broken or very soft was split in half using a small scoop or putty knife and one half was placed in the sample bag. The responsible technician filled out a core cutting/splitting form recording the sample number, the starting and ending footage of the sample interval, the date, and the technician's initials. The sample bags were tied off and stored in the secure core facility until the sample batch was ready to be shipped.

RC samples were typically left at the drill site for two to three days to dry, before being transported by Calico personnel to the Calico storage and core logging facility in Vale. The date and the number of samples transported were recorded on a sample handling form. The samples were arranged in a manner to ensure that all samples, blanks, and standards were accounted for, and were photographed prior to shipment for analysis. RC samples were then air-dried and stored until shipped by commercial freight service to the ALS Minerals ("ALS") laboratory in Reno, NV.

When all of the core and RC samples were prepared for shipment, they were laid out in order (including quality assurance/quality control samples) at the Calico logging facility in Vale. A complete sample inventory was filled out and maintained as an Excel spreadsheet to verify that all samples were accounted for and that bags were not damaged prior to shipment. Drill core sample bags were placed into rice bags, and each rice bag was sealed with a numbered security seal. RC samples were placed into super sacks and each super sack was sealed with a numbered security seal. Only samples from a single drill hole were included in a shipment. A sample submittal form was prepared with the shipment number, security seal numbers, the sample numbers, the type of analyses requested, and a list of samples to be duplicated. A hard copy of the submittal form was included with the sample shipment and an electronic copy was emailed to the lab. A chain of custody form was filled out by the Calico personnel who prepared the shipment. This form included the sample shipment number, the location the samples were shipped from, the total number of containers in the shipment, the security seal numbers, name of the person who prepared the shipment, name of the person who transported the shipment, and the name of the person who received the shipment at the laboratory. When the form was completed at the laboratory by the receiving individual, any damage or discrepancies were noted on the form and the form was sent back to Calico. The driver of each truck was required to sign off on the chain of custody form.

Calico's 2011 and 2012 drilling samples were shipped by a commercial freight service to ALS Minerals ("ALS") in Reno, Nevada. ALS was independent from Calico and maintained an ISO 9001:2008 accreditation for quality management and ISO/IEC17025:2005 accreditation for gold assay methods.

ALS crushed the samples to 75% at <6 millimeters and then split off a 250-gram subsample for pulverization to 85% at <75 microns (200 mesh). Cleaner sand was run through the crusher every five samples or at any color change in the sample noticed by the ALS technicians. Cleaner sand was run through the pulverizer between every sample in the pulverizing step. Pulps were split to separate a 30-gram aliquot for determining gold by fire assay with AA finish (ALS code Au-AA23). A separate five-gram aliquot was used for inductively-coupled plasma atomic-emission spectrometric ("ICP-AES") determination of silver and 32 major, minor, and trace elements following a four-acid digestion (ALS code ME-ICP61). Further aliquots were taken from the same pulp for fire assay with gravimetric finish (ALS code Au-GRA21) if the original gold assay exceeded the 10.0 g Au/tonne upper limit of detection. Samples that assayed greater than 100 g Ag/tonne were reanalyzed using a 10-gram aliquot with a four-acid digestion for silver and an AA finish (ALS code AG-OG62). Samples that assayed greater than 1,500 g Ag/tonne were reanalyzed using a 30-gram fire assay with a gravimetric finish (ALS code Ag-GRA21)

11.1.5 Paramount 2016 - 2017

Samples from Paramount's drilling in 2016 and 2017 were transported by Paramount personnel from the drill sites to the Paramount storage and logging facility in Vale, Oregon. The procedures used by Calico in 2011 and 2012 for sample handling, drying, logging, sample marking, core cutting, and packaging (see Section 11.1.4) were applied by Paramount to the core and RC samples from 2016 and 2017. Competent core was cut into halves with a saw, while highly broken core was split by hand directly from the box using a brush and spoon in an effort to take a representative half-core sample; approximately 10% of the core samples were split by hand. After logging and sampling by Paramount geologists and technicians, core samples were transported by ALS personnel from the project office in Vale, to the ALS sample preparation facility in Reno or Elko, Nevada. Chain of Custody paperwork was completed by Paramount and by ALS. Sample security was maintained at all times by Paramount and ALS. ALS is a commercial assayer independent from Paramount. ALS maintains an ISO 9001:2008 accreditation for quality management and ISO/IEC17025:2005 accreditation for gold assay methods.

ALS crushed the samples to 75% passing a six-millimeter mesh and then split off 250-gram subsamples for pulverization to 85% at -<75 microns (200 mesh). Cleaner sand was run through the crusher every five samples or at any color change in the sample noticed by ALS technicians. Cleaner sand was pulverized between every sample in the pulverizing step. Pulps were split to separate a 30-gram aliquot for determining gold by fire assay with AA finish (ALS code Au-AA23). A separate five-gram aliquot was used for ICP-AES determination of silver and 32 major, minor, and trace elements following a four-acid digestion (ALS code ME-ICP61). Further aliquots were taken from the same pulp for fire assay with gravimetric finish (ALS code Au-GRA21) if the original gold assay exceeded the 10.0 g Au/tonne upper limit of detection. Samples that assayed greater than 100 g Ag/tonne were reanalyzed using a 10-gram aliquot with a four-acid digestion for silver and an AA finish (ALS code AG-OG62). Samples that assayed greater than 1,500 g Ag/tonne were reanalyzed using a 30-gram fire assay with a gravimetric finish (ALS code Ag-GRA21)

11.2 Quality Assurance/Quality Control Procedures

This section summarizes the quality assurance and quality control ("QA/QC") procedures and methods used by the historical and present operators of the Grassy Mountain project. The results of the QA/QC programs are summarized and discussed in Section 12.2.

11.2.1 Atlas QA/QC 1987 - 1992

Atlas employed two primary procedures for QA/QC: (1) random re-sampling of coarse-reject material for samples where the initial assay was in excess of approximately 0.020 oz Au/ton; and (2) analyses of RC rig duplicates of original five-foot samples collected at even 100-foot intervals.

Periodically, Atlas geologists prepared a list of the initial Chemex assays greater than approximately 0.020 oz Au/ton. For every 10th sample from that list, coarse rejects were collected and split into two one-pound subsamples. These coarse-reject subsamples were sent to Cone Geochemical Laboratories ("Cone") in Denver, Colorado and Hunter Mining Labs ("Hunter") in Reno, Nevada. Cone and Hunter were independent of Atlas, but it is not known if these laboratories held certifications at that time. The check samples sent to both laboratories were reportedly prepared using the same procedures. The samples were dried, cone crushed to minus 1/8 inch, and split into 125-gram subsamples that were then ring pulverized to minus 150 mesh. From these pulps, 30-gram aliquots were analyzed by fire assay methods. The duplicate samples that were collected at 200-foot down-hole intervals were sent along with the initial samples to the Chemex facility in Boise, and then to the Chemex assay laboratory in North Vancouver. Hunter assay certificates indicate that their fire assays were finished gravimetrically, while the finish of the Cone assays was not indicated on the available certificate documentation.

The rig duplicates were sent to Chemex along with the original drill samples for preparation and analysis.

11.2.2 Newmont QA/QC 1992 - 1996

Newmont sent 163 check samples to their in-house Newmont Metallurgical Services laboratory in Salt Lake City, Utah for fire assays with AA finishes. The nature of these samples (e.g., pulps, preparation duplicates, or field duplicates) is not known. The original samples were assayed by RMGC.

Text from an original Newmont report or memorandum that lacks the header page describes the testing of drill core from hole GMC-001-9, which was a wedge off of GMC-001. The core was entirely consumed by the testing of three splits that included both halves of the sawed core as two sample sets, as well as samples of the fines derived from sawing of the core that would not normally be sampled.

Newmont requested RMGC to reanalyze 98 samples originally analyzed by RMGC; the nature of these check samples is not known for certain, but evidence suggests they were preparation duplicates.

11.2.3 Tombstone QA/QC 1998

Tombstone sent the following samples to Chemex for check analyses: 14 AAL pulps for pulp-check analyses, 15 two-pound splits of AAL coarse rejects as preparation duplicates, 14 core duplicates, and 15

RC rig duplicates. The RC rig duplicates were originally collected at approximately even 100-foot intervals.

The mesh sizes of the 14 AAL pulps were checked by Chemex prior to analyses. The RC and core duplicates were dried at 100°C and crushed to 65% at less than 10 mesh. These coarse-crush samples, along with the preparation duplicates, were split into 200- to 300-gram subsamples using a Jones riffle splitter, and these subsamples were then ring-pulverized to 95% passing 150 mesh, from which 30-gram aliquots were fire assayed for gold and silver using gravimetric finishes.

In addition to the QA/QC testing described above, Tombstone selected 60 AAL coarse rejects from storage and instructed AAL to coarse pulverize the entire sample to minus 60 mesh. AAL cut the samples into halves with a rotary splitter, sent one set of the halved samples to Chemex for further sample preparation (pulverization to 95% passing 150 mesh) and analysis (30-gram fire assay with AA finish), and completed the same preparation and analysis at AAL using the second set of halved samples. Tombstone referred to these samples as "Assay Prep Checks", while calling the more standard preparation duplicates described in the previous paragraph "Reject Checks".

AAL also routinely completed replicate analyses of AAL original pulps.

11.2.4 Calico QA/QC 2011 - 2012

Calico inserted QA/QC samples every tenth sample in sequence using pre-labeled bags in the same manner as the primary core and RC-chip samples. Drill samples were grouped in batches of 36 samples. Each sample batch contained a field duplicate, a commercially prepared certified reference material ("CRM"), and a blank. The blanks included commercial blank pulps and coarse basalt rock barren of gold (coarse blanks). All four types of control samples were inserted with the drill core; only the CRMs and blank pulps were inserted with the RC samples.

The basalt rock was used to monitor the possibility of contamination potentially introduced during the coarse-crushing and pulverization processes used for drill core. The blank pulps monitored possible contamination that might be introduced after pulverization.

Three commercial CRMs obtained from CDN Resource Laboratories Ltd. ("CDN") were inserted to assess the precision and accuracy of the analyses. These are listed in Table 11.1.

Table 11.1 Grassy Mountain Certified Reference Materials for 2011 - 2012

CRM ID	Certified Value (g Au/tonne)	2 Std. Dev. (g Au/tonne)	Submitted No.
CDN-GS-P3A	0.338	0.022	55
CDN-GS-3J	2.71	0.26	36
CDN-GS-8A	8.25	0.60	21

At the request of Calico, a preparation-duplicate sample was created approximately every 20 samples to assess the homogeneity of the sample material and the overall sample variance. During the 2011 drilling program, 59 sample pulps representing about five percent of the samples from the higher-grade portion of the deposit were also retrieved from ALS and shipped to AAL as check samples.

11.2.5 Paramount QA/QC 2016 - 2017

Paramount compiled an electronic database containing all historical and 2016-2017 drilling information. This database is maintained using SQL software and is housed by an off-site remote server that is controlled by a third-party database expert. All database inquiries and data requests are routed through this third-party expert. All data are controlled by Paramount's designated data manager and this third-party expert in order to prevent any unauthorized changes to the Paramount database. Paramount has established QA/QC protocols for data management, verification, validation, and data screening. These protocols consist of primary and secondary checks on electronic entry of field data, drill-hole data, sample information, assays, and geochemistry. All information is verified and cross checked by Paramount and the third-party database expert to ensure accuracy.

During the 2016 and 2017 drilling program, nine different commercially prepared CRMs obtained from CDN were inserted into the sample sequence for the purpose of QA/QC (Table 11.2). To meet Paramount's QA/QC protocols, the standards needed to assay within three standard deviations of the recommended gold value furnished from CDN. One of the CRMs have certified silver values as well. If any samples assayed outside the three standard deviation limit, the sample previous to and after the failed sample were examined for accuracy and for cohesiveness with the geology and mineralization. Any failures and surrounding samples that were thought out of the ordinary after this examination were re-assayed.

Table 11.2 Grassy Mountain Certified Reference Materials for 2016 – 2017

CRM ID	Certified Value (g Au/tonne)	2 Std. Dev. (g Au/tonne)	Certified Value (g Ag/tonne)	2 Std. Dev. (g Ag/tonne)	No. Submitted
CDN-GS-P3A	0.338	0.022		31	30
CDN-GS-P3C	0.263	0.02			26
CDN-GS-P4F	0.498	0.028			22
CDN-GS-P7E	0.766	0.086			28
CDN-GS-1Q	1.24	0.08	40.7	2.2	32
CDN-GS-3J	2.71	0.26			57
CDN-GS-8A	8.25	0.60			27
CDN-GS-10D	9.50	0.56			12
CDN-ME-1414	0.284	0.026	18.2	1.2	36

A white, marble chip blank sample was variously inserted during 2016 and 2017 for both core and RC samples. If any blank samples assayed above a 0.10 g Au/tonne limit, the sample previous to and after the failed sample were examined for possible contamination sources. Any failures and surrounding samples that were thought out of the ordinary after this examination were re-assayed.

RC rig-duplicate samples were collected at the drill rig as described in Section 10.2.
Paramount also instructed ALS to prepare and analyze preparation duplicates for all holes, while field duplicates were submitted with the original samples for all core holes. Finally, a subset of ALS pulps from RC and core samples were sent to AAL for check assays.

11.3 Summary Statement

Mr. Gustin is satisfied that the procedures and methods used for the sample preparation, analyses, and security of both the historical and Paramount samples are adequate for generating reliable data that is acceptable as used in this report.

12.0 DATA VERIFICATION

12.1 Drill-Hole Data Verification

The current Grassy Mountain drill-hole database, which forms the basis for the Grassy Mountain resource estimation, is comprised of information derived from 485 holes. A total of 282 of these holes were drilled in the general area of the Grassy Mountain resources, including 30 Paramount holes and 252 historical holes.

Paramount originally provided MDA with the project drill-hole database prior to the initiation of the 2016-2017 drilling program. This database was then subjected to the data verification procedures discussed below and corrections were made as appropriate. Following the creation of a verified database, MDA periodically updated this database with the information acquired during Paramount's 2016-2017 drilling program

12.1.1 Collar Data

Atlas established a local grid coordinate system following the discovery of the Grassy Mountain deposit in 1988. This local coordinate system remained in use through to the acquisition of the project by Calico in 2011, following which Calico transformed all relevant project data, including the drill-hole coordinates, into UTM coordinates. The transformation was done by plotting all drill holes on digital topography of the project area in the local coordinates system, projecting these data onto a USGS topographic base map in UTM zone 11 NAD27 coordinates, and rotating and scaling the local-grid data until the contours generated from the Atlas grid matched those from the USGS topographic map contours as closely as possible. The UTM coordinates of each drill hole were then determined. All holes from subsequent drilling programs were surveyed in UTM coordinates.

As part of the 2016-2017 drilling program, all prior drill-hole collars that could be identified in the field were re-surveyed. The collar locations of 82 Atlas holes, six Newmont holes, four Tombstone holes, and nine Calico holes were surveyed. MDA was provided the original digital file produced by the survey contractor, and MDA used this file to compare the new survey locations with those in the existing database. Excluding one hole, in which the location was known to be incorrect in the original project database, the northings from the new survey differed from the database locations by more than 3.0 feet in four holes, with a maximum change of 7.0 feet. The eastings differed by more than 3.0 feet in four holes, with a maximum change of 8.0 feet, and elevations of four holes differed by more than 3.0 feet, with a maximum change of 5.0 feet. These discrepancies were found in a total of eight of the 101 historical holes re-surveyed. The scale of the discrepancies in the hole locations is not considered to be material due to the nature of the Grassy Mountain mineralization and the 10 x 10 x 10-foot block size used to model the resources.

The collars of all holes drilled in 2016-2017 were also surveyed by the contractor. MDA used the original digital survey data for the historical and Paramount drill holes to update the drill-hole locations in the project database.

In addition to the hole locations, the total depths of 47 of the historical holes were checked against historical records. The depth of one hole was found to be off by one foot.

12.1.2 Down-Hole Survey Data

There are 43 historical holes in the Grassy Mountain resource area that have down-hole survey data, and 14 of these were chosen for verification purposes. Excluding three Newmont holes selected, which are discussed below, a total of 168 survey intervals from six Atlas holes, two Tombstone holes, and three Calico holes were checked against historical records. Two azimuth measurements in the database were found to be off by less than one degree, and three inclination errors of less than 1.5 degrees were found. One of the azimuth errors and two of the dip discrepancies occurred in a single hole (Atlas hole 079-001). The project database was corrected to match the historical records. Two survey intervals were also added to the project database as a result of the auditing.

The down-hole survey data for three Newmont holes were also checked. Backup data consisted Newmont handwritten “Drill Hole Summary” sheets. The project database includes more than twice the number of survey intervals than are listed on the summary sheets, and the database azimuths and inclinations have higher precision than those on the summary sheets. The database values are very close to those in the summary sheets, although the values only match exactly when the precision of the two datasets are identical. It appears that the summary sheets are exactly as they are named, which is to say they summarize the down-hole survey data.

There are 209 historical holes within the resource area that lack down-hole survey data in the project database. The drill-collar azimuths and dips for 40 of these holes were checked against historical records and no discrepancies were found.

MDA used digital data derived directly from the down-hole survey instrument to add the 2016-2017 Paramount orientation data to the project database. Down-hole surveys were completed on 25 of the holes drilled by Paramount; down-hole caving precluded surveys for the other five holes.

12.1.3 Assay Data

The original database provided to MDA included a total of 39,124 assay sample intervals from historical holes drilled in the Grassy Mountain resource area. Of these sample intervals, the database assay values for 6,942 of the intervals from 38 Atlas drill holes, two Calico holes, seven Newmont holes, and four holes drilled by Tombstone were checked against historical documents. A total of only five errors in the database gold values were found, including two intervals with assay values from the assay certificates (0.002 and 0.004 oz Au/ton) that had no values in the database, two transcription errors whereby certificate values of 0.001 and 0.002 oz Au/ton were entered into the database as 0.010 and 0.020 oz Au/ton, respectively, and a value of 0 in the database which should have been 0.054 oz Au/ton according to the assay certificate (the 0 value was likely mistakenly transcribed from an adjacent column on the assay certificate). One silver error was found whereby a 0.28 oz Ag/ton value on the certificate was entered in the database as 0.2 oz Ag/ton.

In addition to the errors described above, there were 28 sample intervals with database gold and silver assay values of "0" that had no corresponding assays on the certificates; these intervals presumably had no sample recovery.

All identified errors were corrected in MDA's database, and silver values found for one Atlas hole and three Tombstone holes that were not in the database were added to the database.

MDA received all digital assay certificates relating to Paramount's 2016-2017 drilling program directly from ALS and used these to update MDA's project database.

12.2 Quality Assurance/Quality Control Programs

12.2.1 Atlas 1987 – 1992

Atlas made extensive use of preparation duplicates and field duplicates in an effort to verify their drill-hole gold results. The field duplicates were analyzed by Chemex, the primary assay laboratory used by Atlas, while the preparation duplicates were sent to Cone and Hunter.

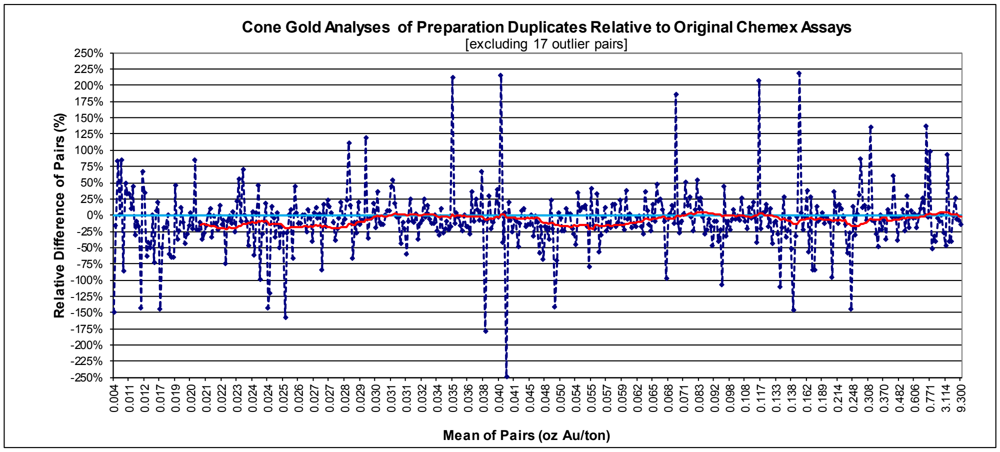
Preparation Duplicates. Preparation duplicates are analyses of pulps derived from secondary splits of the coarsely ground material (coarse rejects) that remain after the primary split is taken for the original assay. Preparation duplicates are therefore used to evaluate the variability introduced by subsampling of the coarsely crushed material. Ideally, preparation duplicates are analyzed by the primary analytical laboratory in order to remove variability introduced by techniques employed by a second lab. In this case, however, Atlas sent the preparation duplicates to two secondary labs.

MDA compiled the data for 458 preparation duplicates derived from coarse rejects of samples from 89 Atlas holes that were analyzed by Cone. The relative-difference graph in Figure 12.1 shows the percentage difference (plotted on the y-axis) of each Cone preparation-duplicate assay relative to its paired primary-sample analysis by Chemex. This relative difference ("RD") is calculated as follows:

$$100\,x\,\frac{(duplicate - original)}{lesser\ of\ (duplicate, original)}$$

The x-axis of the graph plots the means of the gold values of the paired data (the mean of the pairs, or "MOP") in a sequential but non-linear fashion. The red line shows the moving average of the RDs of the pairs, thereby providing a visual guide to trends in the data that aids in the identification of potential bias. Positive RD values indicate that the duplicate-sample analysis is greater than the primary-sample assay. A total of 17 pairs characterized by unrepresentatively high RDs are excluded from Figure 12.1.

Figure 12.1 Cone Analyses of Preparation Duplicates Relative to Original Chemex Gold Assays

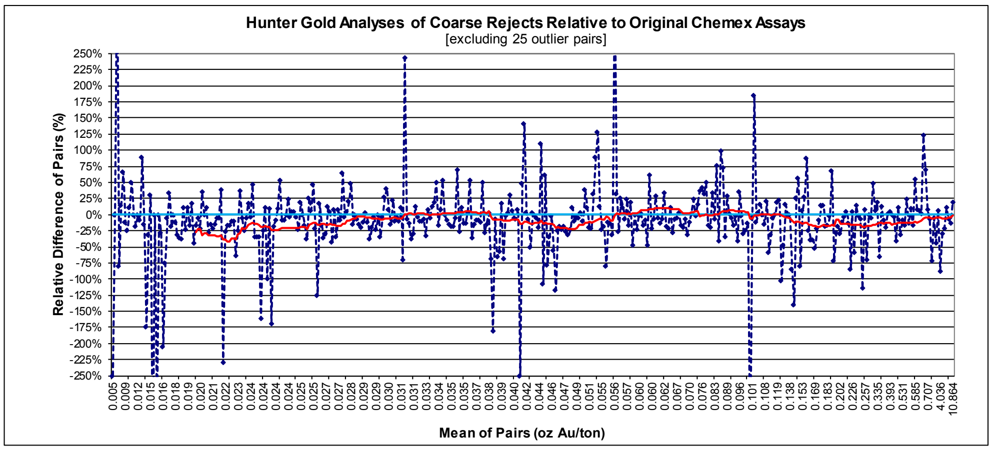


The graph suggests a low bias in the Cone gold results relative to the original Chemex assays over significant portions of the grade range of the data. The mean of Cone analyses (0.226 oz Au/ton) is lower than that of the original results (0.237 oz Au/ton), and the average RD of the pairs is -7% (the average RD can be an approximate measure of the degree of bias, although one must be aware of the statistical effects of pairs with anomalously high RDs). The mean of the absolute value of the RDs ("AVRD") is 29%, which is a measure of the average variability exhibited by the paired data.

Hunter analyzed 428 preparation duplicates from the same original-sample set as analyzed by Cone (Figure 12.2). In this case, 25 extreme outlier pairs are removed for the purposes of this discussion. The mean of the Hunter analyses is lower than the mean of the original Chemex assays (0.208 vs. 0.221 oz Au/ton), and the average of the RDs is -9%. The AVRD is 34%.

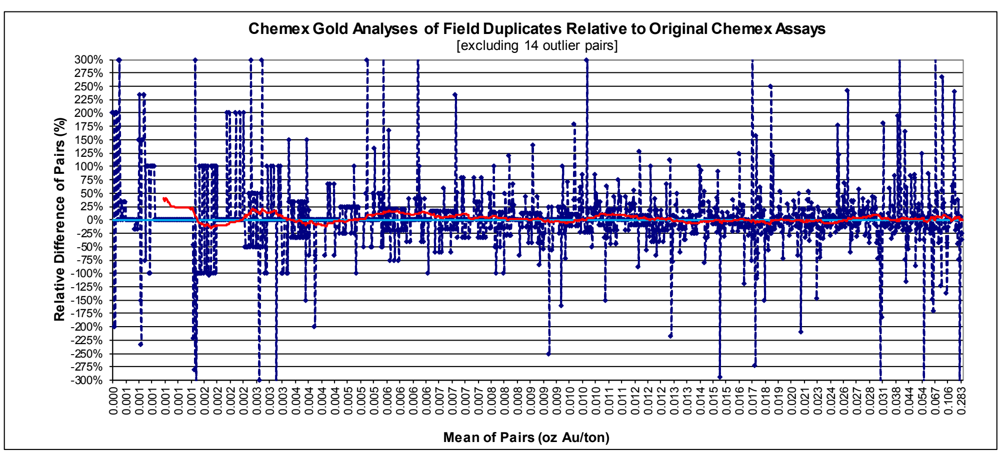
The Hunter and Cone preparation-duplicate data are generally consistent, showing a low bias in the gold results relative to the original Chemex analyses and average variability of approximately 30%. One difference in the duplicate versus original analyses is that the Chemex pulps were prepared to meet a 95% minus 100-mesh particle size, and the Hunter and Cone pulps were pulverized to minus 150 mesh.

Figure 12.2 Hunter Analyses of Preparation Duplicates Relative to Original Chemex Gold Assays



RC Field Duplicates. Field duplicates are secondary splits of drill samples that are mainly used to assess the natural grade variability of the deposit, as well as to evaluate the total subsampling variances attributable to splitting both in the field and in all subsequent subsampling steps in the laboratory. The Atlas field duplicates were collected simultaneously as the original samples at the RC drill sites and sent to Chemex along with the original samples. The results of 1,252 RC duplicates from 165 holes drilled by Atlas were compiled by MDA (Figure 12.3; 38 pairs in which both the original and field-duplicate analyses are less than the detection limit are removed, as are 14 extreme outlier pairs).

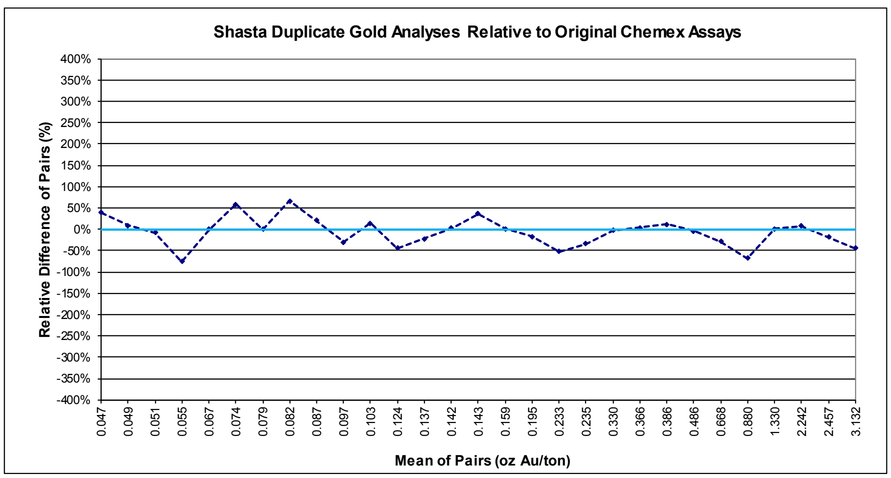
Figure 12.3 Chemex Analyses of RC Field Duplicates Relative to Original Chemex Gold Assays



The field duplicates compare well with the original results, and the means of the datasets are identical (0.016 oz Au/ton). The average of the RD is +4%, while the mean of the AVRD is 35%.

Miscellaneous QA/QC Samples. In addition to the preparation and field duplicates, Atlas sent 32 samples of unknown type (e.g, sample pulps, coarse rejects, or field duplicates) in 1990 from hole 026-034 to Shasta Analytical Geochemistry Laboratory of Redding, California ("Shasta") for 30-gram fire assays. A handwritten note on the paper assay certificate states that these samples consist of a "set of 4th check assays from [this] hole". The Shasta check assays are compared to the original Chemex results in Figure 12.4; one outlier pair and two pairs in which Chemex overlimit assays were not performed are removed from the graph.

Figure 12.4 Shasta Check Analyses Relative to Original Chemex Gold Assays



The paired data compare reasonably well up to a MOP grade of ~0.2 oz Au/ton. At higher grades, the Shasta check assays tend to be lower grade than the Chemex original analyses, although there are far too few pairs to make definitive conclusions. The mean of the Shasta analyses (0.462 oz Au/ton) is significantly lower than the mean of the original Chemex assays (0.533 oz Au/ton), but this difference is largely due to the two highest-grade pairs.

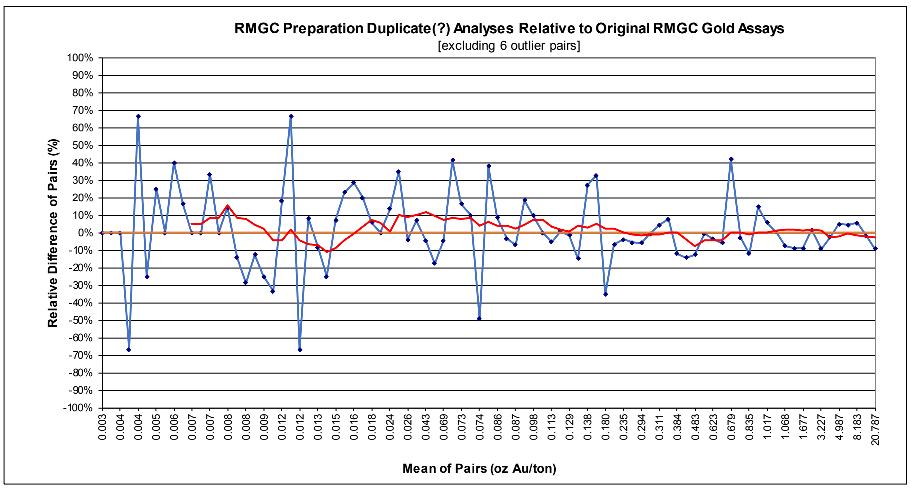
In May 1988, Tombstone sent 12 high-grade Chemex pulps from eight Atlas holes to AAL for check assaying; one of the pulps did not have the 30 grams needed for the one assay-ton (30 gram) gravimetric fire assays. The mean of the 11 check assays (3.835 oz Au/ton) agrees well with the mean of the original Chemex results (3.866 oz Au/ton).

In late 1990, Phelps Dodge Mining Company had four pulps and 27 coarse-reject samples from nine Atlas holes sent to Chemex for assaying. Backup information is not adequate to determine which of the check assays are from pulps versus the coarse rejects. The paired data compare well up to a MOP of approximately 0.14 oz Au/ton; the check assays in the seven pairs at higher grades are on average lower-grade than the original results, but again the quantity of data is insufficient to derive statistically valid conclusions.

12.2.2 Newmont 1992 - 1996

Preparation Duplicates. In 1993, Newmont had RMGC reanalyze 98 samples originally analyzed by RMGC; five of the samples did not have sufficient material to assay. The nature of these check samples is uncertain, but the assay certificate includes "REMARKS" that state, "To report Original Pulp and New Pulp values for Gold fire and Cyanide". This suggests the samples were preparation duplicates. The check results are compared to the originals in Figure 12.5; six outlier pairs are excluded.

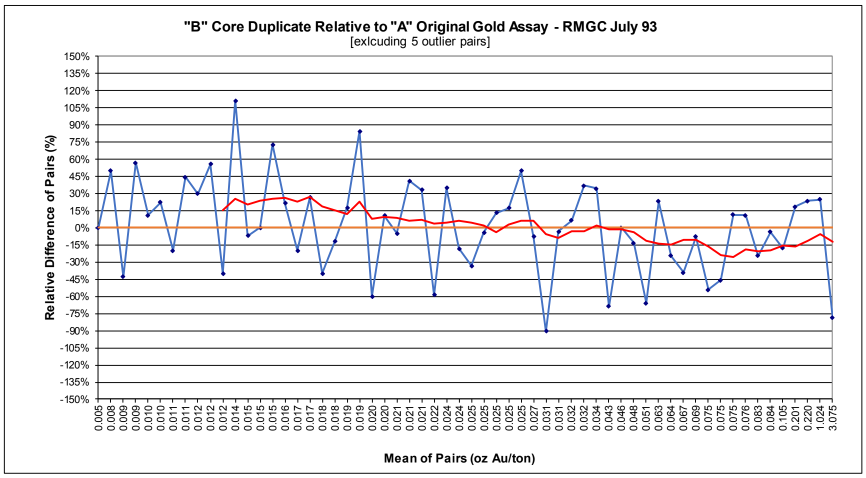
Figure 12.5 RMGC Check Analyses Relative to Original RMGC Gold Assays



The duplicates and originals compare reasonably well, and the mean of the checks (0.903 oz Au/ton) is close to the original (0.923 oz Au/ton). The mean of the RD is +2%, while the mean of the AVRD is 15%.

Core Field Duplicates. Newmont hole GMC-001-9 was wedged off of GMC-001. Newmont submitted both halves of the sawed core from the wedge hole for analyses by RMGC. Newmont's split "A" is presumed to be the original sample in the following analysis and split "B" is therefore considered to be a core duplicate sample. The two sets of 73 core samples were sent to RMGC for sample preparation and fire assaying in July 1993. Figure 12.6 is a RD plot of the data, excluding two pairs that did not have sufficient material to analyze and five outlier pairs.

Figure 12.6 RMGC Core Duplicate "B" Relative to RMGC "A" Gold Assays



The core duplicate values are higher than the originals up to a MOP grade of approximately 0.020 oz Au/ton, then lower than the original at MOP grades of about 0.040 oz Au/ton and higher. The means of the core duplicates and originals are 0.085 and 0.108 oz Au/ton, respectively, but if the highest-grade pair is removed the duplicate mean becomes higher than the original (0.052 and 0.049 oz Au/ton, respectively). The mean of the RD is +2%, while the mean of the AVRD is 30%.

The preparation-duplicate data and core-duplicate data do not identify any significant issues. The two datasets taken together suggest the variability attributable to the splitting of core into halves is approximately 15% (core-duplicate AVRD of 30% minus preparation-duplicate AVRD of 15%).

<u>Miscellaneous QA/QC Samples</u>. In December 1993, Newmont had the "A" and "B" pulps reanalyzed by RMGC. These pulp-check analyses for both datasets yielded results extremely close to the original November 1993 assays, with means of RDs of 0% and 1% for the A and B pulp sets, respectively, and AVRDs of 2% in both cases.

Newmont completed gold fire assays on 163 samples at their in-house metallurgical assay facility in Salt Lake City, Utah as a check on the RMGC results (Jory, 1993). The nature of the check samples (pulps, coarse rejects, or field duplicates) is not known. The mean (0.970 oz Au/ton) and median (0.080 oz Au/ton) of the Newmont checks, as reported by Jory, are both slightly higher than the original RMGC mean (0.942 oz Au/ton) and median (0.078 oz Au/ton).

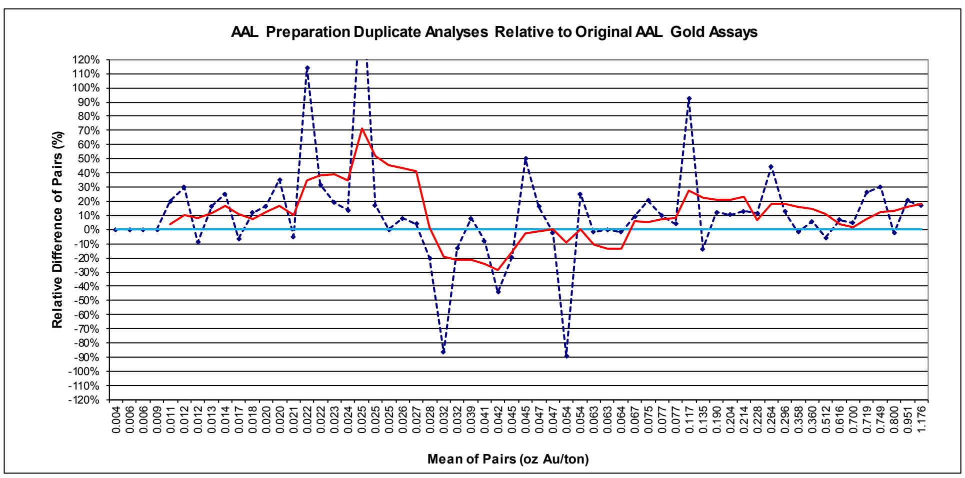
In addition to Newmont's sampling and analytical verification programs discussed above, Tombstone sent nine high-grade samples of Newmont "drill cuttings" from seven holes to AAL for preparation and 30-gram gravimetric fire assays in April 1998. The AAL analyses had a mean of 11.209 oz Au/ton, which compares well with the mean of 11.25 oz Au/ton from RMGC's original assays.

12.2.3 Tombstone 1998

Replicate Analyses. AAL, Tombstone's primary assay lab routinely completed replicate analyses of some of the original assays. Replicate analyses use a second aliquot taken from the primary sample pulp and are typically reported on the same certificate as the original assays. A total of 113 of these analyses were reported by AAL on the same certificates that report the original assays for the 10 holes drilled by Tombstone. The replicate analyses show excellent reproducibility of the original assays, with a mean that is almost identical to the original and an average RD of +1%. The mean of the AVRD is 6%, which is somewhat high for replicate analyses.

Preparation Duplicates. A total of 60 AAL coarse rejects from two holes were further crushed to minus 60 mesh by AAL and split into halves. One half was pulverized and analyzed by AAL and the second set was sent to Chemex to do the same. The results of this modified version of preparation duplicates completed by AAL are shown in Figure 12.7.

Figure 12.7 AAL Preparation Duplicate Analyses Relative to AAL Original Gold Assays



The RD graph shows high biases at low and high grades, while a low bias is evident at MOP grades between approximately 0.025 and 0.06 oz Au/ton. The duplicate mean is higher than that of the original samples (0.175 vs. 0.157 oz Au/ton), and the mean of the RDs is +11%.

A RD graph of the Chemex analyses versus the original AAL results shows a roughly similar form as seen in Figure 12.7, although no bias is present. In this case the duplicate mean (0.159 oz Au/ton) matches the original mean well, and the mean of the RDs is +1%. The means of the AVRD is 20%.

The differences between the AAL and Chemex results is likely more a reflection of insufficient data to adequately evaluate the Tombstone preparation duplicates than some internal differences between the two laboratories.

Miscellaneous QA/QC Samples. Tombstone sent Chemex a set of original AAL pulps for pulp-check analyses, splits of AAL coarse-rejects as preparation duplicates, and some core and RC field duplicates. The mean of 14 pulp-check analyses from three holes (0.523 oz Au/ton) is about 5% higher than that of the original AAL analyses (0.499 oz Au/ton). The mean of 15 Chemex preparation duplicates from six holes is also higher than the AAL mean (0.447 vs. 0.412 oz Au/ton, respectively). A total of 13 core duplicates from four holes yielded a mean (0.119 oz Au/ton) much higher than the original analyses (mean of 0.085 oz Au/ton), but the elimination of one extreme pair (0.414 oz Au/ton for the duplicate vs. 0.080 oz Au/ton for the original) brings the duplicate mean (0.094 oz Au/ton) much closer to the mean of the original samples (0.086 oz Au/ton). The mean of 15 RC duplicates from six holes is again higher than the mean of the original samples (0.055 vs. 0.048 oz Au/ton, respectively).

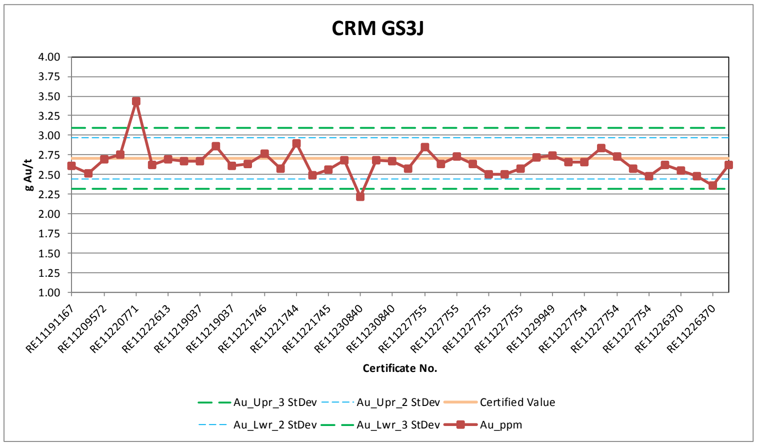
While none of this miscellaneous testwork involves sufficient samples to derive statistically significant conclusions, the check analyses of the various sample sets are consistently higher than the original AAL results.

12.2.4 Calico 2011 – 2012

CRMs. Three sets of CRMs (certified reference materials) were used to evaluate the analytical accuracy and precision of the original ALS analyses of Calico's drill samples. The CRMs were inserted into the original sample stream and analyzed with the drill samples. In the case of normally distributed data, 95% of the CRM analyses are expected to lie within the two standard-deviation limits of the certified value, while only 0.3% of the analyses are expected to lie outside of the three standard-deviation limits. Note, however, that most assay datasets from metal deposits are positively skewed.

Figure 12.8 shows a plot of the ALS analyses of CRM CDN-GS-3J, which has a certified value of 2.71 g Au/t (0.079 oz Au/ton). The x-axis plots the certificate numbers by increasing dates.

Figure 12.8 Chart of ALS Analyses of CRM CDN-GS-3J



Samples outside of the three standard-deviation limits are typically considered to be failures. As it is statistically unlikely that two consecutive analyses of standards would lie between the two and three standard-deviation limits, such samples are also considered to be failures unless further investigations suggest otherwise. All potential failures should trigger investigation, possible laboratory notification of potential problems, and possible reanalyses of all samples included with the failed standard result.

Using the above criteria, two of the ALS analyses of this CRM are three standard-deviation failures. However, the CRM analyses are biased slightly low of the certified value and the low-side failure would not be a failure if the low bias is taken into account.

A similar analysis of the CRM CDN-GS-8, which has a certified value of 8.25 g Au/t (0.241 oz Au/ton) shows no bias and no failures, while CDN-GS-P3A has 12 failures out of the 56 ALS analyses. Although nine of the CDN-GS-P3A failures are on the high side (ALS value > certified value), no bias is evident in the data taken as a whole. CDN-GS-8A has a certified value of 0.338 g Au/t (0.010 oz Au/ton).

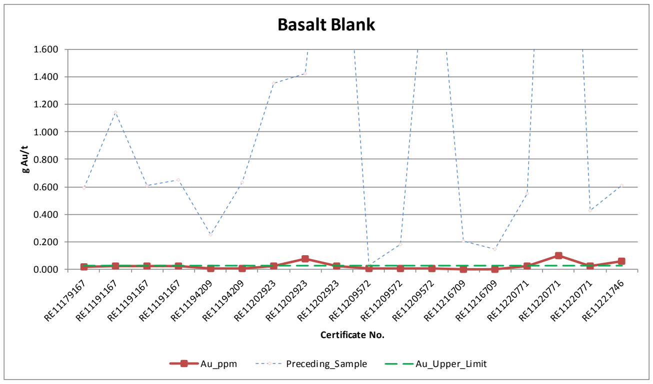
It is not known what actions, if any, were taken in response to the CRM failures.

<u>Coarse Blanks</u>. Coarse blanks are samples of barren material that are used to detect possible contamination in the laboratory, which is most common during sample preparation stages. In order for analyses of blanks to be meaningful, they must be sufficiently coarse to require the same crushing and pulverizing stages as the drill samples. It is also important for a significant number of the blanks to be placed in the sample stream within, or immediately following, a set of mineralized samples, which would be the source of most contamination issues. In practice, this is much easier to accomplish with core samples than RC. Blank results that are greater than five times the lower detection limit of the relevant analyses are typically considered failures that require further investigation and possible re-assaying of associated drill samples. The detection limit of the ALS analyses was 0.005 g Au/t, so blank samples assaying in excess of 0.025 g Au/t (0.0007 oz Au/ton) are considered to be failures.

A total of 18 coarse blanks were analyzed in 2011-2012 by ALS (Figure 12.9). Three of the analyses exceeded the failure threshold, and the highest analysis of the blanks is 0.100 g Au/t (0.003 oz Au/ton). All three of the failures are associated with previous samples that are significantly mineralized. While the blank data provide evidence of cross contamination during ALS sample preparation, the magnitude of this contamination is insignificant.

Analytical Blanks. Analytical blanks are used to monitor possible contamination or calibration problems during the determination of gold concentrations. Calico used a blank commercial pulp supplied by CDN Laboratories (CDN-BL-7) for the QA/QC program. There are 62 ALS analyses of the analytical blank, and five of the analyses exceeded the 0.025 g Au/t (0.0007 oz Au/ton) threshold. The failures range from 0.001, 0.001, 0.003, 0.004, and 0.009 oz Au/ton. It is not common for analytical blanks to generate failures, and the latter three failures are at a level that would warrant investigation and potentially corrective action; it is not known if any actions were taken.

Figure 12.9 Chart of ALS Analyses of Coarse Blanks - Calico



<u>Field Duplicates</u>. Calico collected 40 RC duplicates and 10 core duplicates that were analyzed by the primary laboratory (ALS). The mean of the RC duplicates (0.030 oz Au/ton) is close to the mean of the original assays (0.032 oz Au/ton). Although the average of the RDs is -9%, the removal of two of the higher-grade pairs with anomalously high RDs changes this average to 4%. The mean of the AVRD of the entire dataset is 21%.

The means of the duplicates and original samples are reasonably close (0.043 and 0.040 oz Au/ton, respectively) considering the lack of pairs, but the size of the core-duplicate dataset is too small to derive meaningful conclusions.

<u>Pulp-Checks</u>. Pulp checks are reanalyses of the remaining pulps from the original assays. These reanalyses are typically completed by a second laboratory. A total of 59 ALS original sample pulps from Calico's drilling program were sent to AAL for check assays. Excluding one extreme outlier pair, the mean of the AAL checks compare very well with the mean of the original samples (0.206 versus 0.208 oz Au/ton, respectively), and the average of the RDs is -2%. The mean of the AVRD is 12%, which is relatively high for pulp-check analyses.

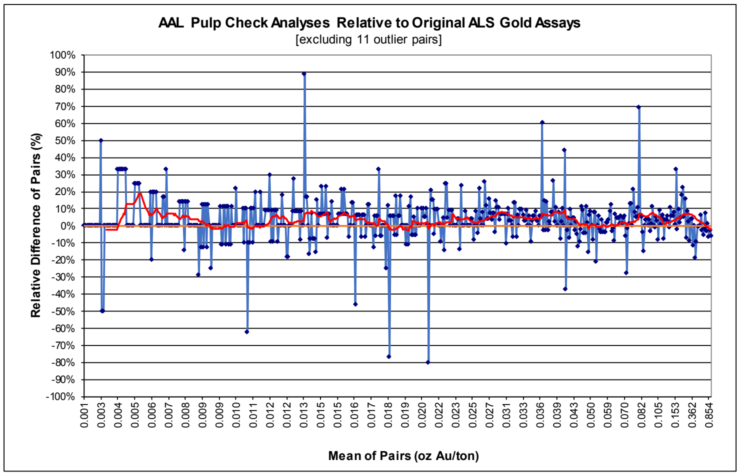
12.2.5 Paramount 2016 - 2017

<u>Certified Reference Materials</u>. Paramount inserted the nine sets of certified CRMs listed in Table 11.1 into the RC and core sample stream. Out of the 270 ALS gold assays of the CRMs, there were a total of nine analyses that exceeded the three standard-deviation limits. Four of these are due to slight high biases in the ALS analyses of GS-P3A and GS-P3C. Of the remaining five cases that can be considered failures, three are from analyses of GS-P4F, although each of these are only slightly above the high-side failure limits.

<u>Pulp Checks</u>. Paramount sent 569 ALS pulps from the 2016-2017 drilling program to AAL for pulp-check analyses (Figure 12.10; 11 outlier pairs are excluded). While the means of the duplicate and original

analyses are identical (0.066 oz Au/ton), the graph provides evidence of a slight high bias in the AAL check assays and the mean of the RDs is +3%. The mean of the AVRD is 8%.

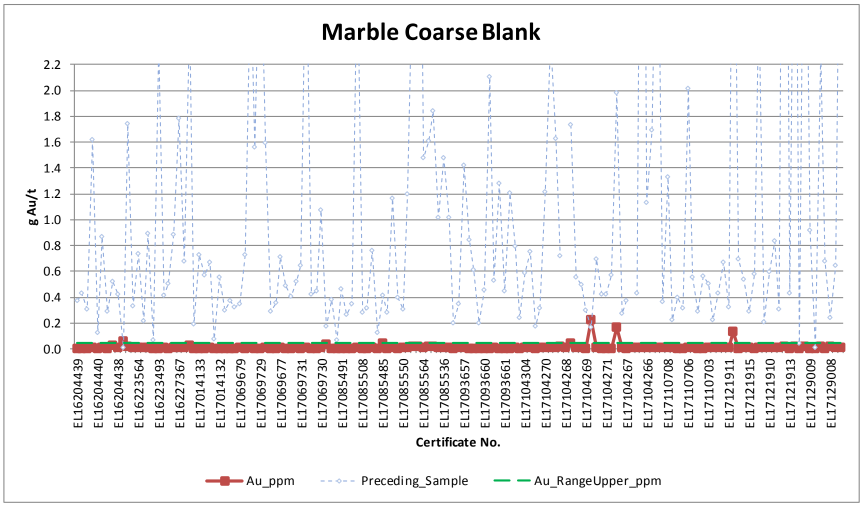
Figure 12.10 AAL Pulp Checks of ALS Original Gold Analyses



A high bias in the AAL results compared to the original ALS assays is apparent in the silver data as well. The mean of the AAL analyses is 4% higher than the ALS mean, the average of the RDs is +6%, and the mean of the AVRD is 10%.

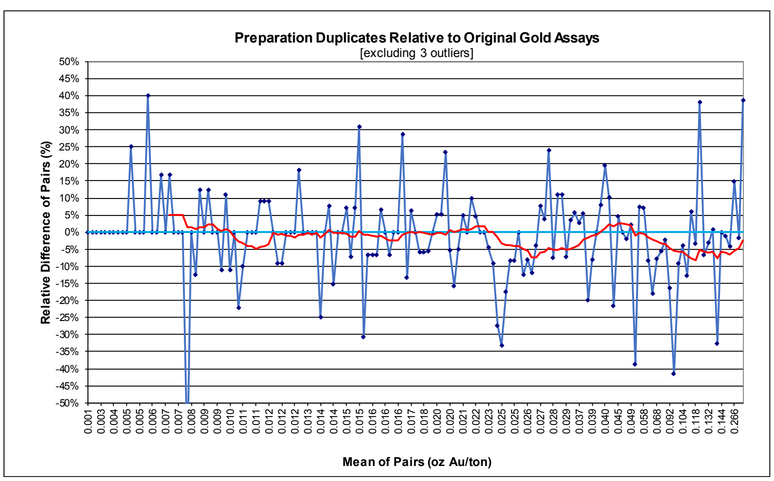
Coarse Blanks. A total of 151 coarse-blanks were analyzed by ALS (Figure 12.11), eight of which exceeded the failure threshold. The failures range from 0.029 to 0.221 g Au/t (0.001 to 0.007 oz Au/ton); three of the blank analyses exceeded 0.1 g Au/t (0.003 oz Au/t). The failures do not correlate well with previous samples that are significantly mineralized, but the data provide the suggestion of cross contamination during ALS sample preparation. The magnitude of this potential contamination in the three highest-grade blank analyses would warrant investigation and, if appropriate, re-assaying of samples that accompany the failures.

Figure 12.11 Chart of ALS Analyses of Coarse Blanks - Paramount



Preparation Duplicates. ALS prepared and analyzed a total of 153 preparation duplicates that were analyzed along with the original samples in 29 of the 30 holes drilled by Paramount (Figure 12.12; three outlier pairs were removed).

Figure 12.12 ALS Gold Analyses Preparation Duplicates - Paramount

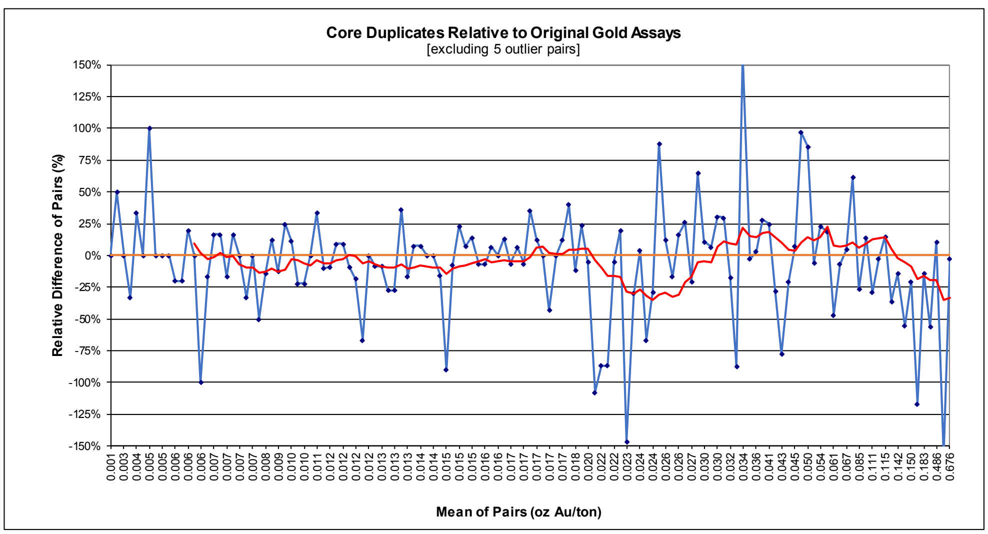


The mean of the gold analyses of the preparation duplicates is very close to the mean of the original assays (0.040 versus 0.039 oz Au/ton, respectively), and the average of the RDs is -1%. The mean of the AVRD is 9%. The silver results are very similar to those of gold, with means of the duplicate and original samples of 0.172 and 0.174 oz Ag/ton, respectively. The mean of the RDs is -1% and the average of the AVRD of 9%.

Core Field Duplicates. Paramount regularly included RC and core field duplicates with the submission of the original core samples to ALS. The core duplicates consisted of half splits of the ½ core remaining, creating ¼-core samples, from all 27 holes drilled at least in part with core. Fines, consisting of pieces of core too small for sawing, were sampled using a scoop and putty knife to obtain an 'eyeball' ½-split (this was identical to the procedure used for the primary ½-core samples). A total of 136 core duplicates and 52 RC duplicates were analyzed by ALS. The two datasets require separate evaluation because the splitting methodologies are completely different.

The ¼-core duplicates are compared to the original results in Figure 12.13; five outlier pairs were removed.

Figure 12.13 Core Duplicates Relative to Original Gold Assays - Paramount

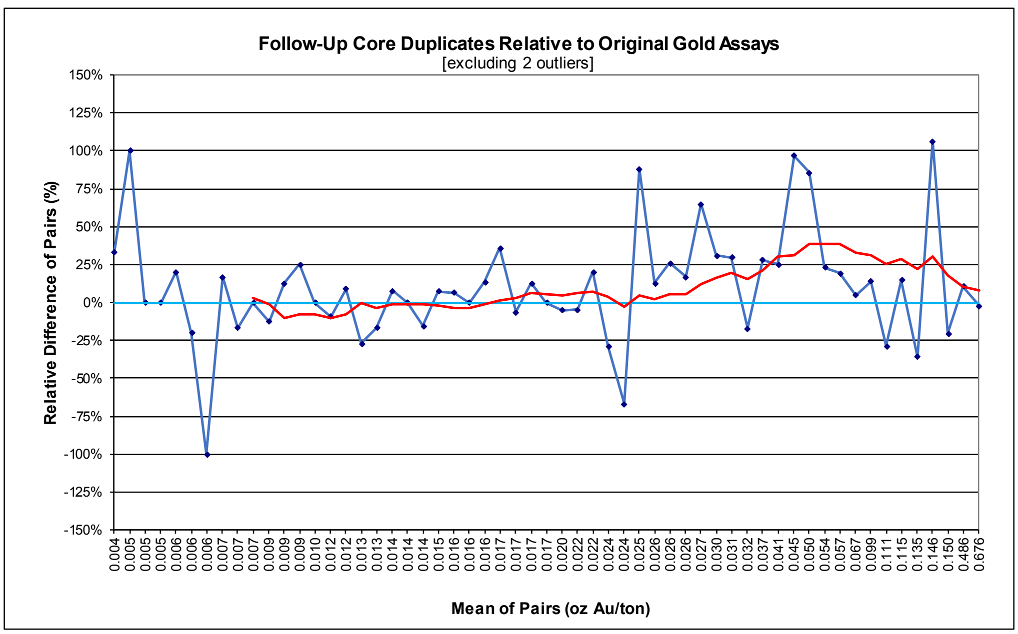


At MOP of up to ~0.02 oz Au/ton, the means of the duplicate and original analyses are identical, although a slight low bias in the duplicate results is evident over much of this grade range. This bias is largely driven by spikes on the graph that are predominantly pairs where the duplicates are lower than the originals. At MOP higher than 0.02 oz Au/ton, variability increases dramatically (AVRD = 40% versus 18% over the lower-grade range) and the duplicate data display both high- and low-bias trends. On

average, the duplicate data are lower grade than the original samples (means of duplicates and originals are 0.078 and 0.093 oz Au/ton, respectively, and the mean of the RDs is -16%).
Excluding seven outlier pairs, the silver results for the core duplicates compare well with the original results, with near identical means and an average RD of -1%. The mean of the silver AVRD is 17%.

The core-duplicate gold results led to the submission of 59 additional core duplicates from 10 of the Paramount holes that include core. In this case, ½-core samples were submitted, and, with the first set of core duplicates and Newmont results regarding fines in mind (see Section 10.4.1), special care was taken to brush out all fines in the core boxes related to each sample interval and include them in the duplicate samples. The gold analyses of this second batch of core duplicates, excluding two outlier pairs, show excellent correspondence with the original ½-core results up to a MOP grade of ~0.02 oz Au/ton (Figure 12.14). At higher grades, the core duplicates are systematically higher grade (duplicate mean is 8% higher than the original mean; average of the RDs is +18%), and as was the case for the first set of core duplicates, variability increases substantially (mean of the AVRD is 33%).

Figure 12.14 Second Set of Paramount Core Duplicates Relative to Original Gold Assays

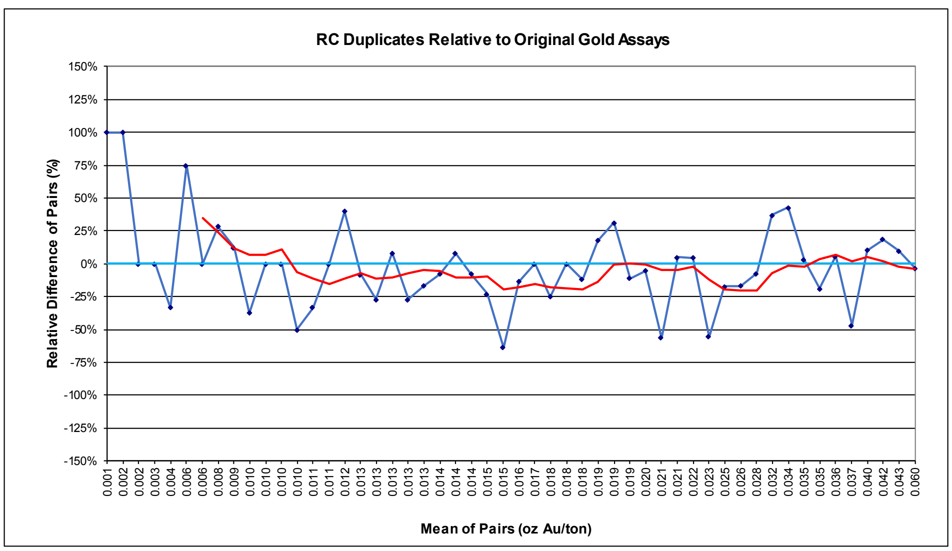


The silver values of the second set of duplicate core samples compare reasonably well with originals. The mean of the duplicates (0.167 oz Ag/ton) is close to the original mean (0.163 oz Ag/ton) considering the relatively small dataset, and the mean of the RDs is +3%. The average of the AVRD is 18%.

It is reasonable to postulate from the core duplicate data that sampling of the core-box fines derived from higher-grade gold samples may have played a significant role in the core-duplicate gold and silver results. Specifically, native gold particles collecting at the bottoms of the boxes in high-grade samples may have been unrepresentatively lost to both the original half-core samples and the first set of ¼-core duplicates. This loss of native gold particles can be attributed to the manual, unsystematic splitting of the core-box fines (fines were sampled with a scoop and putty knife). In contrast, the second set of half-core duplicates likely oversampled gold in the higher-grade samples, as these samples would have incorporated the gold lost from the primary samples (all fines left in the core boxes were brushed into the duplicate sample bags). The possibility of free gold preferentially collecting in fines is supported by the results of Newmont analyses of saw fines (Section 10.4.1). In contrast to gold, silver analyses of both sets of core duplicates compare reasonably well with the original assays.

<u>RC Field Duplicates</u>. A total of 52 RC duplicates are available for 27 of the Paramount drill holes. Most of these holes were completed with core. Figure 12.15 compares the duplicate RC assays to the original results.

Figure 12.15 Paramount RC Duplicates Relative to Original Gold Analyses



The means of the RC duplicates compare well (0.018 versus 0.019 oz Au/ton, respectively), and the mean of the RDs is -1%. There is a suggestion of a low bias in the graph, although this is not well supported due to the low number of pairs. The average of the AVRD is 23%, which is somewhat lower than expected, but could be due to the lack of higher-grade pairs.

The silver analyses of the RC duplicates are systematically lower than the originals, the mean of the duplicates is 0.092 oz Ag/ton while that of the originals is 0.099 oz Ag/ton, and the average of the RDs is

-13%. The cause of this systematic low bias in the silver results is difficult to explain, but perhaps the bias would lessen with more data. The mean of the AVRD is 23%; considering the presence of native gold one would expect the gold variability to be higher than that of silver, which supports the conclusion above of the surprisingly low variability in the RC duplicate gold results.

12.2.6 Discussion of QA/QC Results

The available Atlas QA/QC data of consequence (the preparation and field duplicates) suggest that the original gold assay results may be overstated to some extent. However, the average grade of the duplicate dataset is much higher than the average grade of the Grassy Mountain deposit, and repeat analyses of only the higher-grade portion of a deposit with free gold can yield lower results than original assays. Without further data, it is impossible to know whether there is a high bias in the Atlas results, although a comparison of resources with and without Paramount drill data suggests there are no material issues with the Atlas data (see Section 14.9).

The Newmont QA/QC data do not identify any issues, while it is possible that the Tombstone gold values are slightly understated.

No issues were revealed by the Paramount CRM, blank, and preparation-duplicate data. The core duplicate data suggest that the Paramount gold assays of core, particularly at higher grades, may be understated to some degree. These data also serve to emphasize the importance of careful sampling and splitting of core-box fines.

The variability evidenced by the duplicate data from all operators at Grassy Mountain does not exceed normal bounds, especially considering the presence of visible gold.

12.3 Site Inspections

Mr. Gustin visited the project site on August 18, 2016, November 17, 2016, and June 1, 2018. During these visits, Mr. Gustin reviewed altered and sometimes mineralized outcrops throughout the Grassy Mountain deposit area, as well as other areas within and outside of Paramount's landholdings. Active core and RC drill sites with ongoing sampling and logging were also visited. Each of the three site visits included additional days at the Vale field office inspecting drill core from a number of holes and reviewing all project procedures related to logging, sampling, and data capture.

Paul Tietz, MDA Senior Geologist and a qualified person independent of Paramount, visited the Vale facility for three days in December of 2016 and again in January, February, and March 2017, for a total of 18 additional days. Two of these days were spent at the Grassy Mountain deposit area becoming familiar with the geology of the deposit. The remainder of the time was spent reviewing drill core and project data in detail, as well as assisting Paramount's geological team with cross-sectional geological modeling that was eventually used as a base for resource modeling. These activities involved detailed checking, validation, and in some cases modifications of the Paramount and historical geological data, interpretations, and geological modeling of the Grassy Mountain deposit.

Steve Weiss, an Associate of MDA and a qualified person independent of Paramount, completed a five-day geologic field inspection at the Grassy Mountain project and seven days of on-site drill data and drill-sample review and evaluation in March 2017. This work was carried out interactively with Paramount's project geologists. Time was also spent completing detailed checking, validation, and in some cases modifications of the Paramount and historical geological data and interpretations for the Grassy Mountain deposit and other mineralized areas within the property.

The work of both Mr. Weiss and Mr. Tietz during these site visits contributed to Mr. Gustin's understanding of the project and confidence in the project data.

12.4 Summary Statement

Mr. Gustin experienced no limitations with respect to data verification activities for the Grassy Mountain project. In consideration of the information summarized in this and other sections of the report, Mr. Gustin has verified that the Grassy Mountain project data are acceptable as used in this report, most significantly to support the estimation and classification of mineral resources and reserves.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section has been prepared under the supervision of Mr. Robert Raponi, P. Eng., of Ausenco, Canada. Mr. Raponi believes that the information presented in this section of the report accurately reflects the mineral processing and metallurgical testing results and estimated processing parameters as of the Effective Date of this report. The term "ore" as used in this section refers to mineralized material. When used to describe historical metallurgical tests, the term "ore" has no economic significance.

Studies of recovery and other metallurgical tests were initiated in 1989 by Atlas and were continued intermittently by later operators. The most recent testwork was commissioned by Paramount in 2017.

13.1 Historical Crushing and Grinding Test

In 1989, Hazen Research Inc. of Golden, Colorado ("Hazen") conducted various tests to develop data for a historical feasibility study commissioned by Atlas. Hazen received ten tons of core samples from six Atlas drill holes. Four bulk samples, identified as Zone 1 (low grade), Zone 2 (medium grade), and Zone 3 (high grade), and a composite of Zones 2 and 3 were prepared. Quarter portions of the Zone 3 bulk sample were separated by grade and used to make up composites 1 to 4. A list of the composites and their head grades as shown in Table 13.1.

Table 13.1 Hazen 1989 Composite Summary

Identification	oz/st Au
Zone 1, low-grade, HRI 42683-10	0.02
Zone 2, medium-grade, HRI 42683-20	0.03
Zone 2, high-grade, HRI 42683-30	0.10
Zones 2 + 3, composite	0.06
Composite 1, HRI 43345-1	0.74
Composite 2, HRI 43345-2	0.04
Composite 3, HRI 43345-3	0.25
Composite 4, HRI 43345-4	0.16
Composite 2+3, HRI 43345-2 and -3	0.15

Crushing and grinding tests were conducted on the various 1989 composites as shown in Table 13.2. The data show that the Grassy Mountain mineralization is relatively hard and abrasive.

Table 13.2 Summary of 1989 Crushing and Grinding Tests

Description	Zone 1	Zone 2	Zone 3	Composite	High Grade
MacPherson, grindability Test No.		1	2	3	
Gross Output, lb/hr		12.23	14.06	13.21	
Gross Power, hp-hr/st		19.57	15.62	17.57	
Feed Size, 80% passing, inch		0.848	0.847	0.847	
Product Size, 80% passing, mesh		30	33	30	
Gross Autogenous Work Index, Awi (hp-hr/st)		49.6	36.7	44.1	
Bond Rod Mill Work Index, Rwi (hp-hr/st)		21.9	20.9	21.4	22.1
Bond Ball Mill Work Index, Bwi (hp-hr/st)		25.9	21.5	24.6	
Bond Abrasion Index, Ai (hp-hr/st)	0.9	1.0	0.6	0.9	
Bond Impact Work Index, Wi (hp-hr/st)	5.9	4.4	4.4	4.4	
L.A. Abrasion, % loss at 500 turns	18.6	19.5	20.8	19.6	
% loss at 1,000 turns	35.30				
Compressibility, psi	15,370	14,050	5,700	10,877	

13.2 Other Historical Test Work Reviewed

In 2012, Resource Development Inc. ("RDI") conducted a review of four prior historical metallurgical studies as listed below:

1. Grassy Mountain Metallurgical Studies, Hazen Research Inc., March 14, 1990;
2. Gravity Concentrations Studies on the Grassy Mountain Gold Ore, Hazen Research Inc., July 1991;
3. Grassy Mountain Metallurgical Studies, Golden Sunlight Mines, May 3, 1991; and
4. Grassy Mountain Metallurgical Test Results, Newmont Exploration Inc., December 21, 1993.

Commented [SW3]: QP please provide Author name, report title, year/month, number of pages for the list of References. ditto for reports 2,3 and 4 this list.

13.2.1 Hazen Research Report of March 14, 1990

The samples used for the testwork summarized in the Hazen 1990 report are the same as those subjected to the 1989 Hazen crushing and grinding tests summarized in Section 13.1. The 1990 testwork included gold flotation tests, but these resulted in extractions ranging from about 50% to 70%. They concluded that flotation would not be a viable process option to recover gold from Grassy Mountain mineralized materials.

13.2.1.1 1990 Gravity Tests

Hazen first processed samples on a Wilfley table and the resulting concentrate was cleaned using a Gemeni Table. The gravity gold recovery was in the range of 15.4 to 20.9%.

13.2.1.2 1990 Cyanidation Leach Tests

Batch Agitated-Leach Tests: Batch agitated-leach tests were performed to investigate the effect of pre-aeration, leach time, grind size, pulp density, cyanide concentrations, and carbon addition on gold extraction and reagent consumption. The optimum process conditions were found as follows:

1. Grind: P_{80} of 150 mesh with lime to mill;
2. Pre-aeration: 3.0 hours;
3. Leach and CIP: 24 hours;
4. Pulp density: 45% solids; and
5. NaCN: 500 ppm initially, with degradation to 0.00055 lb.

The 1990 test results for the various composites under the optimum agitated leach conditions are summarized in Table 13.3.

Table 13.3 Summary of 1990 Agitated Leach Results

Ore	Assayed Grade	Gold		Reagents, lb/st	
		Tailings	%	NaCN	Ca(OH)
	Au, oz/st	oz/st	Extraction	Consumed	Added
Zone 1	0.023	0.01	64.5	0.9	0.7
Zone 1		0.01	65.6	0.7	0.7
Zone 2	0.033	0.01	74.6	0.9	1.5
Zone 2		0.01	77.4	0.9	3.1
Zone 3	0.103	0.01	95.3	0.5	3
Zone 3		0.00	96	0.5	4
Zone 2+3	0.057	0.00	94.2	0.5	2.3
Composite 3	0.249	0.01	95.3	0.7	3.5
Composite 3		0.01	97.4	0.6	3.3
Composite 2+3	0.153	0.01	96.6	3.1	3.1
Plant Feed	0.156	0.00	97.1	0.5	2.4

Column and Bottle-Roll Tests: Bottle-roll and column-leach tests were performed at -3/8-inch and -5/8-inch material using Zone 2 and Zones 2+3 samples (Table 13.1). The test results are summarized as follows:

1. Bottle-roll tests extracted 31.8% to 34.8% of the gold for the low-grade sample and 47.3% to 54.7% of the gold from the average-grade sample.
2. The gold extractions in the column tests after 55 days were 44% to 47% for low grade and greater than 60% for the average-grade samples.
3. The gold extraction for both composites increased by ± 3% when the crusher product size was decreased from minus 5/8 inch to 3/8 inch.

Thickening Tests: Thickening tests were performed on a laboratory ball-mill discharge sample containing lime, cyanide, and cyanide-leach tailings. The measured unit area was approximately 2 ft^2/ton/day for approximately 60% solids.

Cyanide Destruction Tests: Cyanide destruction tests were conducted on cyanide-leached slurry using air/SO_2, chlorine, and hydrogen peroxide. Only the chlorine process reached the targeted level of less than 1 ppm weak-acid dissociable cyanide ("CN_{WAD}"), with final slurries containing less than 0.15 ppm CN_{WAD}.

13.2.2 Hazen Research Gravity Concentrations Studies, July 1991

Gravity rougher/cleaner tests were conducted in 1991 using Diester and Gemeni tables at nominal 20-, 48-, and 100-mesh grind samples. The test results indicated the following:

1. The higher-grade feed sample resulted in higher gold extraction, especially in rougher separation;
2. Cleaner concentrates produced from all three composite samples were suitable for direct sale/smelt; and
3. The overall extraction of gold was 20% to 42%.

13.2.3 Golden Sunlight Mines 1991 Report

Golden Sunlight prepared a total of six composites in 1991 from Atlas' drill core samples for this study. The composite head gold assays varied from 0.023 oz Au/ton to 0.140 oz Au/ton. Bond ball-mill work index values of 27.37 and 31.64 kWh/ton were determined. Bottle-roll cyanidation-leach tests were performed with 25% solids and a pH of 11 for 48 hours of leach time. Results are shown in Table 13.4 and Table 13.5.

Table 13.4 Gold Extraction vs. Grind Size at Golden Sunlight

Grind	NaCN	Lime	Head grade	Tails Grade	Recovery
% + 200 mesh	(lb/st)	(lb/st)	(oz/st)	(oz/st)	%
Stage 1					
18.5	0.34	6.64	0.098	0.008	90.4
24.0	0.31	6.76	0.058	0.009	84.1
32.8	0.29	6.77	0.069	0.011	83.8
49.3	0.61	6.19	0.061	0.007	88.2
65.2	0.40	5.92	0.070	0.011	84.3
Stage 2					
15.0	0.46	6.51	0.074	0.008	89.6
22.0	0.37	5.90	0.085	0.009	89.4
34.5	0.46	5.97	0.084	0.007	91.0
59.0	0.67	5.81	0.075	0.009	87.8
70.8	0.42	5.10	0.073	0.014	80.7

Table 13.5 Gold Extraction vs Head Grade at Golden Sunlight

Head Grade	Grind	NaCN	Lime	Tails Grade	Recovery
(oz/st)	% + 200 mesh	(lb/st)	(lb/st)	(oz/st)	%
0.023	25.5	0.56	6.10	0.007	75.0
0.031	20.5	0.53	5.79	0.009	75.3
0.052	24.1	0.39	6.85	0.008	85.8
0.066	23.0	0.41	6.36	0.007	91.3
0.082	26.5	0.49	6.13	0.005	94.3
0.140	22.9	0.59	5.59	0.005	96.2

13.2.4 Newmont, 1993

Newmont prepared five metallurgical composites using core from three drill holes (holes 46-1, 46-2, and 55-2). These composites were based on gold grade and rock type. The composites were high-grade siltstone (GM-1), arkose (GM-2 and GM-4), low-grade siltstone (GM-3), and sinter (GM-5). The gold assays of the composites ranged from 0.0306 oz Au/ton to 0.194 oz Au/ton. Silver and sulfide-sulfur (“$S_{sulfide}$ “) assays varied from 0.1697 oz Ag/ton to 0.4658 oz Ag/ton and 0.04% to 0.21% $S_{sulfide}$.

Column-leach tests were performed on each composite sample at top particle size of minus ¾-inch and/or minus 1/4-inch for 111 days. The column-leach test data are summarized in Table 13.6.

Table 13.6 Summary of Newmont 1993 Column-Leach Results

Description	Size	Residue Assay, ppm		Head Assay, ppm		% Extraction		NaCN	$Ca(OH)_2$
		Au	Ag	Au	Ag	Au	Ag	lb/st	lb/st
GM-1	-1/4"	1.18	7.67	6.54	7.94	81.9	3.3	0.88	4.68
GM-2	-3/4"	1.53	5.08	3.13	5.18	51.0	1.9	0.80	3.48
	-1/4"	0.96	4.95	3.40	5.14	71.7	3.8	0.84	4.44
GM-3	-3/4"	0.56	4.76	1.19	5.17	53.1	7.9	0.84	4.32
	-1/4"	0.63	4.25	1.31	4.34	52.0	2.1	0.92	5.18
GM-4	-1/4"	0.61	4.16	1.40	4.41	56.5	5.7	1.90	7.36
GM-5	-3/4"	0.91	5.86	1.32	5.94	31.1	1.3	0.86	4.08
	-1/4"	0.50	3.85	1.02	3.98	51.1	3.2	0.94	3.34

13.3 Paramount 2017 Metallurgical Testing

The predominant lithologies identified by Ausenco, under the guidance of the Paramount technical team, included arkose, a coarse-grained, highly silicified sandstone, and siltstone (“SLST”), also a silicified rock, but with a finer-grain size than the arkose. Minor lithologies for testwork included mudstone, an even finer-grained siltstone, and clay matrix breccia (“CMB”). In the higher-grade core zone of the deposit, from which the mineral reserves discussed in this report are derived, these host units are intermingled such that it would not be possible to mine any of the lithologies individually. For this reason, a mixed lithology (“ML”) composite was created by Ausenco and the Paramount technical team to best represent the project reserves that are subject to the potential underground-mining operation.

In 2017, Ausenco requested SGS Canada Inc. ("SGS") in Vancouver, Canada to conduct metallurgical testing of samples from the Grassy Mountain deposit. Paramount submitted nine composites from the Grassy Mountain project to SGS. The composites were received as half HQ core from Paramount 2016-2017 drill holes on July 13, 2017. Each composite was composed of various drill sample intervals from multiple holes. The weight and lithology mixtures are shown in Table 13.7. The types of tests carried out by SGS, and the specific composites used in the 2017 testwork, are presented in Table 13.8. The 2017 SGS test results are summarized in the following sub-sections.

Table 13.7 Composite Sample Types, SGS 2017 Testing by Paramount

Sample ID	Weight (lb)
Arkose	150
Mixed Lithology, Drop Weight Test (MLDWT)	150
Mixed Lithology, Low Grade (ML-LG)	286
Mixed Lithology, Average Grade (ML-1)	299
Mixed Lithology, Average Grade (ML-2)	80
Mixed Lithology, High Grade (HG)	199
Silt Stone (SLST)	132
Mudstone	130
Clay Matrix Breccia (CMB)	131

Table 13.8 SGS 2017 Metallurgical Test Matrix

Sample ID	Head Assay	E-GRG	Gravity Separation	Bulk leach on Gravity Tailing	Cyanide Destruction	Carbon Modelling	Rheology	Solid/Liquid Separation
Arkose	x	-	x	x	x	-	x	x
MLDWT	x	-	x	x	x	x	x	x
ML-LG	x	x	x	x	-	-	-	-
ML-1	x	x	x	x	-	-	-	-
ML-2	x	-	x	x	x	-	-	-
HG	x	x	x	x	-	-	-	-
SLST	x	-	x	x	x	-	x	x
Mudstone	x	-	x	x	-	-	-	-
CMB	x	-	x	x	x	-	-	-

13.3.1 2017 Composite Head Assays and Sample Preparation

A head assay for each of the nine composites was determined for gold, silver, sulfur, carbon, and other major, minor, and trace elements by ICP. The head assays of gold, silver, total sulfur, and total carbon are summarized in Table 13.9.

Table 13.9 2017 SGS Composite Head Assays

Sample ID	Au, oz/st				Ag	S	C
	Test 1	Test 2	Test 3	Average	oz/st	%	%
ML-LG	0.053	0.045	0.087	0.062	0.242	0.46	0.04
ML-1	0.133	0.160	0.161	0.151	0.268	0.38	0.05
ML-2	0.164	0.146	0.219	0.177	0.283	0.24	0.08
HG	0.432	0.475	0.347	0.418	0.542	0.22	0.08
SLST	0.114	0.160	0.149	0.141	0.262	0.31	0.04
Mudstone	0.079	0.097	0.078	0.085	0.344	0.34	0.03
CMB	0.161	0.191	0.171	0.174	0.338	0.22	0.04
Arkose	0.062	0.085	0.061	0.069	0.245	0.38	0.03
ML-DWT	0.139	0.066	0.069	0.091	0.315	0.35	0.03

Gold grades ranged from 0.045 to 0.475 oz Au/ton with an average grade of 0.152 oz Au/ton. Silver grades varied from 0.242 to 0.542 oz Ag/ton with an average grade of 0.316 oz Ag/ton. The carbon contents ranged from 0.03% to 0.08% with an average of 0.05%. The sulfur contents were in the range of 0.22% to 0.46% and the average was 0.32%.

Two composites (Arkose and MLDWT) were prepared at SGS for JK drop-weight and other testing. After the JK drop-weight tests were completed, the materials (reject, unused materials, and tested products) were re-combined and stage crushed to -10 mesh for metallurgical testing laboratory as shown in Figure 13.1. The other seven composites were prepared and analyzed by SGS following the procedure summarized in Figure 13.2.

Figure 13.1 Preparation of Two Composites After JK Drop-Weight Tests, 2017

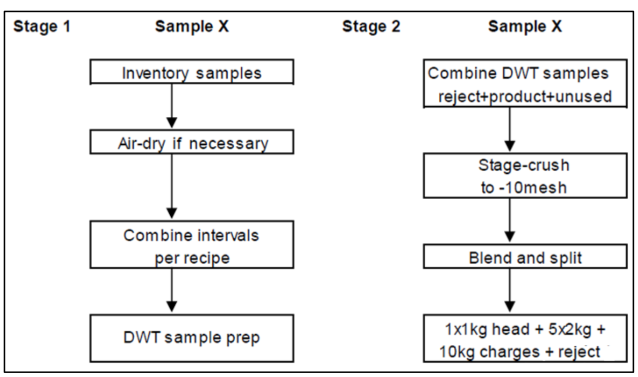
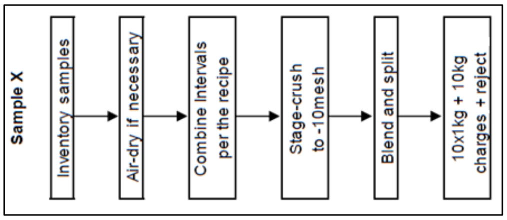


Figure 13.2 Preparation of Seven Composites, 2017



13.3.2 JK Drop-weight Test (DWT)

JK drop-weight tests were conducted on the Arkose and MLDWT samples. The data was interpreted by JKTech Pty Ltd., and the summary of results is presented in Table 13.10. These JK DWT results were used by Ausenco to estimate the crusher work index to be 20.9 kWh/ton.

Table 13.10 Summary of JK DWT Results

Sample ID	S.G.	t_a	A	b	A x b
Arkose	2.56	0.13	100	0.32	32.0
MLDWT	2.51	0.15	99.8	0.30	29.9

The impact-breakage data for these two samples showed they can be classified as hard when compared to other samples in the JKTech database.

13.3.3 Gravity and Leach Tests 2017

Portions of the nine composites sent to SGS in 2017 were split off for gravity and cyanide-leach testing.

13.3.3.1 Extended Gravity Recoverable Gold Test

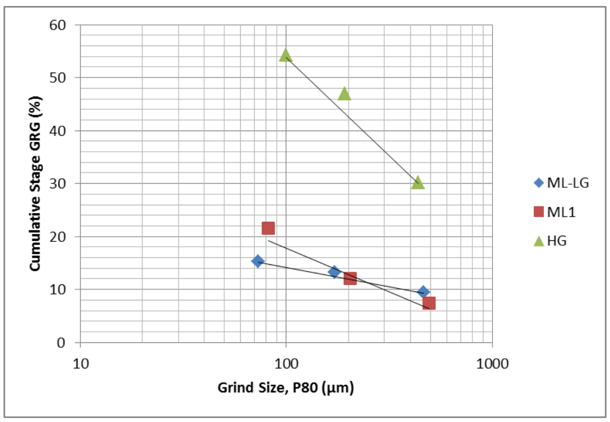
The extended gravity-recoverable gold ("E-GRG") test consists of three sequential liberation and recovery stages using a 44 lb sample. A Knelson concentrator was used after each stage of grinding to concentrate the gravity-recoverable gold. Each gravity-separation concentrate and a subsample of the tailings were analysed for size distribution and assayed for gold and silver. The results were used to construct a metallurgical balance.

Composites ML-LG, ML1, and HG were tested for E-GRG. The test results are summarized in Table 13.11 and Figure 13.3. The total gold and silver extractions for ML-LG were 15.3% and 7.4%, with the composite having a calculated head grade of 0.038 oz Au/ton. The total gold and silver extractions for ML1 were 21.5% and 10.3% with a calculated head grade of 0.117 oz Au/ton. The total gold and silver extractions for HG were 54.2% and 33.0%, respectively. The total gold and silver extractions were higher for the higher head gold-grade samples. HG with considerably higher feed gold grade (calculated 0.356 oz Au/ton) was the best performing sample with substantially higher gold recovery and concentrate grades.

Table 13.11 E-GRG 2017 Test Summary

Sample ID	P80 (mesh)			Mass (%)			Cumulative Au Recovery (%)			Concentrate	Head Grade - Au (oz/st)	
	Stage 1	Stage 2	Stage 3	Stage 1	Stage 2	Stage 3	Stage 1	Stage 2	Stage 3	Au Grade (oz/st)	Direct	Calculated
ML-LG	33	79	200	0.39	0.33	0.25	9.4	13.3	15.3	0.621	0.061	0.038
ML1	32	89	195	0.41	0.33	0.28	7.4	12.1	21.5	2.441	0.152	0.117
HG	34	90	155	0.35	0.33	0.26	30.2	46.9	54.2	20.322	0.417	0.356

Figure 13.3 E-GRG 2017 Results



FLSmidth ("FLS") of Vancouver, Canada was requested to scale up the E-GRG results to estimate full-scale gold extractions with Knelson concentrators at design throughput and grind size. The E-GRG procedure was completed correctly, but some of the size fractions from each test were combined for assay (SGS Vancouver assay procedure). This meant it was not possible to model the data as received. To counter this, FLS adjusted the data based on experience to forecast the missing results. This was done in such a way that the final modeling results may be understated, but they will not be overstated.

The FLS modeling showed that the operational gravity gold extractions to be 5.9% for ML-LG, 5.4% for ML1, and 24% for HG. This assumed treating 30% of circulating load at 100-mesh grind size as shown in Table 13.16. Based on FLS modeling, approximately 9.3% gold recovery is expected from the actual mine-production schedule, assuming a weighted-average mill feed grade of 0.173 oz Au/ton.

Table 13.12 Gravity Circuit Modeling Results by FLS, 2018

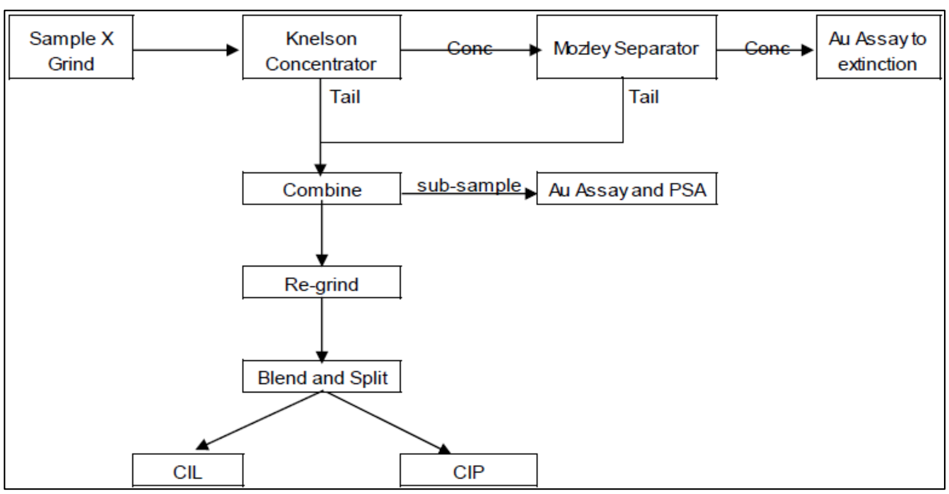
Sample	Feed Grade (oz/st)	Unit Size	% Circulating load treated	Test Work GRG	Corrected Results - 100 mesh GRG	Recovery (%) - from Modelling GRG	Recovery (%) - from Modelling Total Gold	Recovery (%) - Overall Calculated	Approximate lb of Conc	Approximate Conc Grade (oz/st)
LG	0.039	XD20	30	15.3	13.7	43.2	5.9	5.9	1340	2.858
LG	0.039	XD20	50	15.3	13.7	51.3	7.0	7.0	1340	3.383
ML 1	0.116	XD20	30	21.5	14.9	36.2	5.4	5.4	1340	7.641
ML 1	0.116	XD20	50	21.5	14.9	45.3	6.8	6.7	1340	9.566
HG	0.355	XD20	30	54.2	48.4	49.6	24.0	24.0	1340	115.64
HG	0.355	XD20	50	54.2	48.4	57.8	28.0	28.0	1340	121.65

Note: the term "GRG" above is used in place of the term "E-GRG".

13.3.3.2 Knelson-Mozley Gravity Separation at SGS, 2017

Gravity separation of gold was carried out with a Knelson MD-3 concentrator coupled with a Mozley C800 laboratory separator in order to produce enough gravity tailings from low mass recovery/high grade concentrates (to simulate full-scale conditions) for downstream leach tests. A 44-pound split from each of the nine 2017 composites was ground to –10 mesh size and was passed through the Knelson concentrator. The concentrate obtained from each composite was further upgraded by the Mozley separator. The final Mozley concentrate was assayed to extinction to determine the gold contained for mass balancing, and the Knelson and Mozley tailings were then combined for leach testing. Subsamples from the combined tailings were then assayed for gold and screened for particle-size analysis ("PSA"). The remaining combined tailing was ground to a target P_{80} of 150 mesh and split 50/50 by weight for bulk cyanide carbon-in-leach ("CIL") and carbon-in-pulp ("CIP") leach testing. The Knelson-Mozley test procedure is summarized in Figure 13.4.

Figure 13.4 Knelson-Mozley Sample Preparation for CIL and CIP Tests 2017



The Knelson-Mozley gravity separation tests were conducted on all nine composites and the results are summarized in Table 13.13. The gold extractions ranged from 6.1% to 33.9%, with an average of 12.5%. The mass recoveries were in the range of 0.03% to 0.06%. Generally, the gold recovery from high-grade samples was greater than from the low-grade samples.

Table 13.13 Knelson-Mozley Gravity Separation Results 2017

Sample ID	F80	Mozley concentrate		Extraction	Au (oz/st)	
	Mesh	Mass (%)	Au (oz/st)	(%)	Direct	Calculated
HG-G1	36	0.04	309.2	33.9	0.418	0.400
ML1-G1	39	0.05	19.0	7.5	0.151	0.126
ML1-G2	39	0.05	20.1	7.7	0.151	0.131
ML2-G1	34	0.04	32.1	9.2	0.177	0.155
LG-G1	34	0.06	15.5	12.7	0.062	0.075
SLST-G1	33	0.05	37.0	12.8	0.141	0.158
Mudstone-G1	31	0.05	37.3	17.4	0.085	0.098
CMB-G1	34	0.03	90.4	16.2	0.174	0.167
Arkose-G1	26	0.04	10.1	6.1	0.069	0.072
MLDWT-G1	26	0.03	17.6	6.7	0.091	0.070
MLDWT-G2	30	0.04	14.2	6.9	0.091	0.079

13.3.3.3 Bulk Cyanide CIL and CIP Leach Test 2017

Bulk-leach tests were conducted for CIL and CIP modeling to aid the circuit design and to investigate if preg-robbing organic materials are present in the mill feed. Approximately 22 pounds of the Knelson-Mozley gravity tailings from each of the nine 2017 composites were pulverized to 45% solids, pH was set to 10.5 to 11 with lime, dissolved oxygen was maintained at > 6 ppm, 500 ppm of sodium cyanide ("NaCN") was added, and 25 ppm NaCN was maintained through the leaching process with feed grind varying from 160 to 115 mesh.

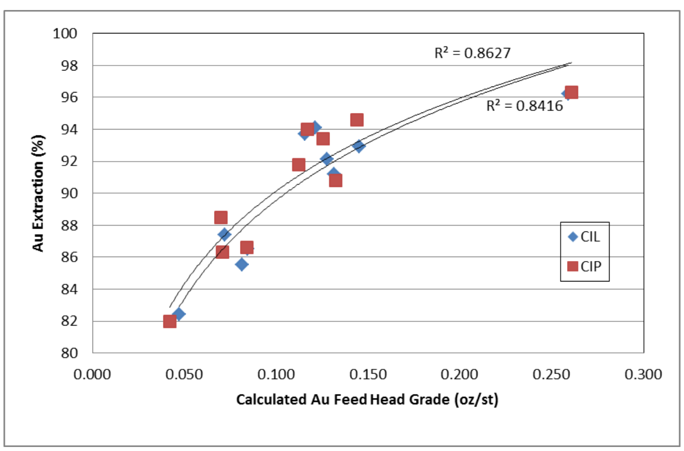
A Hazen report dated March 14, 1990 investigated the effect of pre-aeration, leach time, grind size, pulp density, cyanide level, and carbon addition on gold extraction and reagent consumption. The addition of lime to the grind and pre-aeration for three hours reduced the cyanide consumptions to less than one-pound NaCN/ton of mineralized material tested.

For the 2017 bulk CIL and CIP tests, carbon concentrations were 0.100 lb/gal (12 g/L) and 0.125 lb/gal, respectively. The leach times were 48 hours and 72 hours for the CIL and CIP tests, respectively. Additional CIL and CIP tests with higher cyanide concentration (1,000 ppm added, 500 ppm maintained) were conducted on the gravity tailing from ML1. To determine gold and silver extraction kinetics of CIP tests, solution samples were taken at 2, 6, 24, 48, and 72 hours. At completion, a subsample of slurry was taken and weighed. Selected final pulps were saved to conduct cyanide destruction, rheology, and solid/liquid separation tests. The slurry subsample was filtered and washed well with fresh water, then the subsample was dried and weighed. The pulp density was then calculated and used to estimate the weight of the total residue. The solution samples, carbons, and the residues (subsamples) were assayed for gold and silver. A summary of the results is presented in Table 13.14 and Figure 13.5.

Table 13.14 Bulk Cyanide CIL/CIP Leach Results 2017

Samples	F80	Pulp Density	NaCN concentration		Extraction		Head Assays				Consumption	
							Direct		Calculated			
			Added	Maintained	Au	Ag	Au	Ag	Au	Ag	CaCN	CaO
	mesh	%	ppm	ppm	%	%	oz/st	oz/st	oz/st	oz/st	oz/st	oz/st
ML1-CIL	155	45	800	250	94.1	66.1	0.117	0.252	0.121	0.224	7.3	21.6
ML1-CIP	155	46	800	250	93.4	67	0.117	0.252	0.125	0.230	11.1	22.5
ML1-CIL2	150	43	1000	500	92.1	66.2	0.121	0.248	0.128	0.272	17.2	16.6
ML1-CIP2	150	46	1000	500	91.8	68.9	0.121	0.248	0.112	0.272	19.0	16.0
SLST-CIL	130	46	500	250	93.7	72.9	0.138	0.314	0.115	0.295	8.2	25.1
SLST-CIP	130	43	500	250	94	74.8	0.138	0.314	0.117	0.295	6.4	30.3
ML2-CIL	131	44	500	250	91.2	70.8	0.141	0.325	0.132	0.270	7.9	28.0
ML2-CIP	131	44	500	250	90.8	71.9	0.141	0.325	0.132	0.270	9.0	34.7
LG-CIL	163	43	500	250	82.4	66.1	0.065	0.334	0.047	0.245	9.6	17.2
LG-CIP	163	42	500	250	82	66.1	0.065	0.334	0.042	0.245	8.7	21.3
MUD-CIL	110	42	500	250	85.5	67.9	0.081	0.375	0.081	0.391	7.0	2.9
MUD-CIP	110	44	500	250	86.6	67.9	0.081	0.375	0.084	0.391	9.6	6.1
HG-CIL	155	41	500	250	96.2	84.6	0.265	0.426	0.259	0.455	6.4	26.8
HG-CIP	155	41	500	250	96.3	84.3	0.265	0.426	0.261	0.455	11.4	29.5
CMB-CIL	131	42	500	250	92.9	76.3	0.140	0.354	0.145	0.382	10.5	26.0
CMB-CIP	131	42	500	250	94.6	78.9	0.140	0.354	0.144	0.388	12.0	30.9
Arkose-CIL	132	44	500	250	86.5	59	0.068	0.214	0.084	0.238	8.7	22.7
Arkose-CIP	132	44	500	250	86.3	66.9	0.068	0.214	0.071	0.251	11.4	26.5
MLDWT-CIL	148	41	500	250	87.4	60.8	0.066	0.226	0.072	0.223	11.7	21.3
MLDWT-CIP	148	44	500	250	88.5	67.4	0.066	0.226	0.070	0.250	13.4	21.6

Figure 13.5 Bulk Cyanide CIL/CIP Leach Results 2017



Gold extractions from the CIL tests ranged from 82.4% to 96.2%, with an average of 90.2%, and silver extractions ranged from 59.0% to 84.6%, with an average of 69.1%. From the CIP tests, gold extractions varied from 82.0% to 96.3%, with an average of 90.4%, and silver extractions ranged from 66.1% to 84.3%, with an average of 71.4%. Generally, the CIL and CIP tests gave similar gold and silver extractions.

Average cyanide consumption was 0.64 lb/ton and 0.78 lb/ton for CIL and CIP, respectively. Average lime consumption was 1.44 lb/ton and 1.64 lb/ton for CIL and CIP, respectively.

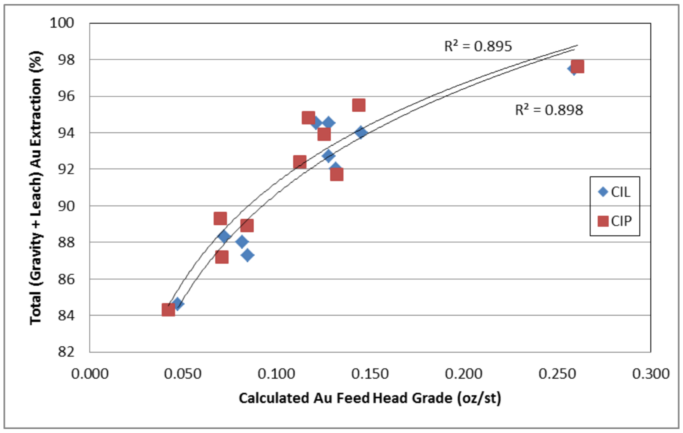
An additional test on gravity tailings from composite ML1, with a higher cyanide concentration, resulted in a slightly lower gold recovery and the same silver recovery.

The combined gold extractions from the gravity separation tests and CIL/CIP tests were calculated and are summarized in Table 13.15 and Figure 13.6. The combined gold extractions from gravity separation and CIL leaching ranged from 84.6% to 97.5% with an average of 91.4%. The overall gold extractions from gravity separation and CIP leaching ranged from 84.3% to 97.6% with an average of 91.5%. Both CIP and CIL tests showed similar extractions over the range of gold grades tested. In other words, no noticeable amount of preg-robbing mineralized material is expected. The choice of leach technology should be made based on capital- and operating-cost evaluations.

Table 13.15 Combined 2017 Gold Recoveries

Samples	Test #	Gold Recovery (%)				
		Individual Process			Combined	
		Gravity	CIL	CIP	Gravity + CIL	Gravity + CIP
ML1	1	7.5	94.1	93.4	94.5	93.9
ML1	2	7.7	92.1	91.8	92.7	92.4
SLST	1	12.8	93.7	94.0	94.5	94.8
ML2	1	9.2	91.2	90.8	92.0	91.7
LG	1	12.7	82.4	82.0	84.6	84.3
MUDSTONE	1	17.4	85.5	86.6	88.0	88.9
HG	1	33.9	96.2	96.3	97.5	97.6
CMB	1	16.2	92.9	94.6	94.0	95.5
ARKOSE	1	6.1	86.5	86.3	87.3	87.2
MLDWT	1	6.7	87.4	88.5	88.3	89.3

Figure 13.6 Combined 2017 Gold Recoveries



13.3.4 Cyanide Destruction Using SO_2/Air Process

Cyanide destruction testwork was conducted as part of the 2017 SGS program. Portions of five of the 2017 Paramount composites were tested. The results indicated that all five tested samples could be detoxified by exposure to a mixture of SO_2 and air ("the SO_2/air process"). The objective was to determine if the SO_2/air process could treat barren leach slurries to achieve residual values of less than 1.0 ppm CN_{WAD}. There is no historical cyanide discharge limit for Oregon; a target of < 1.0 ppm was used for this study.

The CN_{WAD} in the CIP tails was in the range of 101 to 149 ppm. In the steady state of the continuous tests, the pulp density was approximately 40% solids, the test pH was ~8.6, the retention time was from 98 to 145 minutes, the ratios of equivalent SO_2 to CN_{WAD} were in the range of 4.34 to 8.32, the ratios of copper (acts as a catalyst) to CN_{WAD} were in the range of 0.11 to 0.17, and the ratios of lime (pH control) to CN_{WAD} were 8.0 to 20.6. At the end of the tests, the CN_{WAD} in the treated pulps was approximately 0.1 ppm. The continuous tests indicated that it was possible to obtain detoxified product containing less than 1 ppm residual CN_{WAD} by treating the leached pulps by the SO_2/air method. The results are presented in Table 13.16.

Table 13.16 Cyanide SO2/Air Destruction Test Results 2017

Sample ID	conditions				Total Coninuous Test							Ratio of SO2-CN_{WAD}	SO_2 Addition	From Steady State				
	Pulp density	Feed CN_{WAD}	Test pH	Test DO	Total Run Time	Reten tion Time	Discharge							Steady State Time	Retention Time	Ratio of SO_2-CN_{WAD}	Ratio of Cu-CN_{WAD}	Ratio of Lime - CN_{WAD}
							CN_{WAD} Ana Lab	CN_{WAD} Picric Acid	CN_{total}	SCN	CNO							
	%	ppm		ppm	min	min	ppm	ppm	ppm	ppm	ppm	g/g	ppm pulp	min	min	g/g	g/g	g/g
MLDWT-CIP (Batch)	40	101	9.3	9.8	390			0.2				72.6	3,970					
CMB-CIP (Batch)	40	110	8.9	5.2	300			0.9				15.9	3,070					
MLDWT-CIP	40	101	8.7	6.1	655	180	0.1		0.3	8.6	190	12.3	1,010	305	145	8.05	0.12	14.1
ML2-CIP	40	149	8.6	6.1	713	209	0.1		0.3	6.2	150	11.5	1,400	300	141	8.32	0.12	9.4
CMB-CIP	40	110	8.6	6.2	560	194	0.1		0.4	4.1	260	6.8	610	300	127	4.34	0.17	20.6
Arkose-CIP	40	110	8.6	5.6	506	120	0.1		0.1	14.0	260	9.2	830	303	98	7.52	0.14	19.0
SLST-CIP	40	149	8.6	5.8	560	125	0.04		0.3	5.8	110	7.4	900	300	102	5.71	0.11	8.0

13.3.5 Rheology and Solid/Liquid Separation Tests 2017

Solid-liquid separation and rheology tests were conducted on three of the 2017 CIL tailings samples. Tailings samples pH was adjusted to 10.5 using lime slurry. The test process water was prepared by dissolving NaCN into deionized water to the target concentration of 1,000 ppm. The three tailings samples tested had 80% passing sizes of 132, 150, and 135 µm for the Arkose-CIL, MLDWT-CIL, and SLST-CIL samples, respectively.

13.3.5.1 Static Settling

For all samples, preliminary static settling tests were performed using coagulant and flocculant in two-liter (0.528 gallons) graduated cylinders which were fixed with rotating picket-style rakes. Static-settling test results were used to determine the preliminary starting conditions for subsequent dynamic thickening tests. The results are summarized in Table 13.17.

Table 13.17 Preliminary Static Settling Test Results

Sample ID	Coagulant Dosage	Flocculant Dosage	Feed	U/F	Unit Area	Initial Settling Rate	Supernatant Visual	TSS
	ppm	ppm	% w/w	% w/w	ft^2/st/day	ft^3/ft^2/day		ppm
Arkose-CIL Tails	15	35	10	64	0.78	3780	Hazy	21
MLDWT-CIL Tails	20	40	15	66	0.98	2470	Clear	< 10
SLST-CIL Tails	20	20	10	62	1.56	1365	S.C.	36

13.3.5.2 Dynamic Thickening

The dynamic thickening test was conducted on the Arkose-CIL sample tails, MLDWT-CIL sample tails, and SLST-CIL sample tails. Effects of the coagulant (Magnafloc 1687) and flocculant (Magnafloc 10) as well as the thickener unit area were examined. An overall results summary is shown in Table 13.18.

Table 13.18 Dynamic Thickening – Overall Summary

Sample ID	Diluted Thickener	Mag 1687 Dosage	Mag 10	Unit Area	Solids Loading	Net Rise Rate	Underflow Solids	Overflow TSS	Residence Time
	% w/w	ppm	ppm	ft^2/st/day	st/ft^2/hr	ft^3/ft^2/day	% w/w	ppm	hr
Arkose-CIL Tails	10	15,000	35,000	0.49 - 0.98	0.085 - 0.043	558 - 279	47.1 - 58.8	500 - 45	0.47 - 0.95
MLDWT-CIL Tails	15	20,000	40,000	0.59 - 0.98	0.071 - 0.043	277 - 166	52.8 - 57.7	66 - 36	0.57 - 0.95
SLST-CIL Tails	10	20,000	25,000	0.59 - 1.37	0.071 - 0.031	468 - 200	52.0 - 63.6	166 - 71	0.53 - 1.29

13.3.5.3 Thickener Underflow Rheology

Tails from the Arkose-CIL sample thickener underflow was set to a density of from 53.9 to 65.9% solids by weight. A degree of thixotropic response was exhibited by the sample at or above 57.9% solids by weight. Thixotropic response is a "flow–friendly" behavior whereby the resistance to flow decreases during constant shearing. Plug-flow responses were observed at solid densities at or above 64.8% by weight solids. The critical solids density ("CSD") of the Arkose-CIL sample's tails thickener underflow was 61.5% solids by weight with yield stress of 0.0033 psi under unsheared flow condition, and 0.0025 psi under sheared conditions. These parameters were measured after a 3-minute period of constant shearing.

The CSD of the MLDWT-CIL tails thickener underflow sample was 61% solids by weight with a yield stress of 0.0042 psi under un-sheared flow condition and 9 psi under sheared conditions. A degree of thixotropic response was exhibited by the sample above 55.6% solids by weight. Plug-flow responses were observed at the solid densities above 64.6% solids by weight.

The CSD of the SLST-CIL tails thickener underflow sample was 60.5% solids by weight with a yield stress of 0.0029 psi under un-sheared flow condition and 0.00145 psi under sheared conditions. A degree of thixotropic response was exhibited by the sample above 59.2% solids by weight. Plug-flow responses were observed at the solid densities at 66.6% solids by weight. It is recommended to maintain the thickener underflow density to 55% solids by weight or less for using centrifugal pumps.

13.4 Summary and Discussion of Relevant Testwork

Historical and current metallurgical test results demonstrate that Grassy Mountain samples are free milling material that can be processed with conventional cyanide leaching. Results from the 2017 test program fall in line with historical testing completed on the project. Samples demonstrate amenability to gravity concentration with centrifugal-type concentrators (Knelson). Due to some assaying issues with the gravity concentrate samples, a conservative interpretation of the results estimates plant gravity recovery of 8.6% of the gold. Leach testing shows similar recoveries with either CIP or CIL. CIL has been selected for this study with gold recovery of 84.9% on gravity tailings, for an overall, combined gold recovery of 93.5%.

Comminution testing from 2017 showed the samples to be classified as hard. The crusher work index determined from these tests, 21.2 kWh/ton, was used to select primary, secondary, and tertiary crushers. From historical testing, a Bond ball-mill work index of 19.0 kWh/ton (75th percentile value of available data) was used to select the ball mill along with a feed size of 80% passing 0.39" and the product size of 80% passing 100 mesh.

Estimated reagent consumptions from the test results include 0.93 lb per ton processed for NaCN, 8.0 lb per ton processed for lime; and 3.92 lb per ton processed for sodium metabisulphite.

The degree to which historical metallurgical samples are representative of the Grassy Mountain deposit is not known with certainty, but there is no evidence that the historical samples were not representative. It is Mr. Raponi's opinion that the 2017 metallurgical testwork was conducted on samples that are representative of the portion of the deposit that is the subject of this PFS. Mr. Raponi is not aware of any processing factors or deleterious elements that could have a significant effect on potential economic extraction that are not discussed in this report.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource estimation for the Grassy Mountain project was completed in accordance with the guidelines of Canadian National Instrument 43-101 ("NI 43-101"). The modeling and estimation of the mineral resources were completed under the supervision of Michael M. Gustin, a qualified person with respect to mineral resource estimations under NI 43-101. The Effective Date of the resource estimate is May 15, 2018. Mr. Gustin is independent of Paramount by the definitions and criteria set forth in NI 43-101; there is no affiliation between Mr. Gustin and Paramount except that of an independent consultant/client relationship.

The Grassy Mountain project resources are classified in order of increasing geological and quantitative confidence into Inferred, Indicated, and Measured categories in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore NI 43-101. CIM mineral resource definitions are given below, with CIM's explanatory text shown in italics:

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.

The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of Modifying Factors. The phrase 'reasonable prospects for eventual economic extraction' implies a judgment by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. The Qualified Person should consider and clearly state the basis for determining that the material has reasonable prospects for eventual economic extraction. Assumptions should include estimates of cutoff grade and geological continuity at the *selected cut-off, metallurgical recovery, smelter payments, commodity price or product value, mining and processing method and mining, processing and general and administrative costs. The Qualified Person should state if the assessment is based on any direct evidence and testing.*

Interpretation of the word 'eventual' in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage 'eventual economic extraction' as covering time periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods of time.

Inferred Mineral Resource

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An Inferred Mineral Resource is based on limited information and sampling gathered through appropriate sampling techniques from locations such as outcrops, trenches, pits, workings and drill holes. Inferred Mineral Resources must not be included in the economic analysis, production schedules, or estimated mine life in publicly disclosed Pre-Feasibility or Feasibility Studies, or in the Life of Mine plans and cash flow models of developed mines. Inferred Mineral Resources can only be used in economic studies as provided under NI 43-101.

There may be circumstances, where appropriate sampling, testing, and other measurements are sufficient to demonstrate data integrity, geological and grade/quality continuity of a Measured or Indicated Mineral Resource, however, quality assurance and quality control, or other information may not meet all industry norms for the disclosure of an Indicated or Measured Mineral Resource. Under these circumstances, it may be reasonable for the Qualified Person to report an Inferred Mineral Resource if the Qualified Person has taken steps to verify the information meets the requirements of an Inferred Mineral Resource.

Indicated Mineral Resource

An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognize the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Pre-Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade or quality of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability of the deposit. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

14.2 Grassy Mountain Project Data

The Grassy Mountain gold and silver resources were estimated using data generated by Paramount and the historical operators discussed in Section 10.0. These data, which are primarily derived from RC and diamond-core drill holes, as well as current topography and cross-sectional lithologic and structural interpretations, were provided to MDA by Paramount.

14.2.1 Drill-Hole Database

The project drill-hole data are in UTM Zone 11 NAD83 coordinates in US Feet. The database includes information from a total of 485 drill holes, 282 of which were drilled in the Grassy Mountain resource area. The remainder were drilled at various exploration target areas within Paramount's landholdings (see Section 6.2). Of the holes drilled at Grassy Mountain, 256 contribute assay data that are directly used in the estimation of the project resources.

Paramount provided MDA with the project drill-hole database prior to the 2016-2017 drilling program. As discussed in Section 12.1, MDA audited this historical drill data and made corrections to the database as appropriate. MDA then periodically updated the database with the information acquired during Paramount's 2016-2017 drilling program, including gold and silver assay data received directly from the analytical laboratory (ALS).

14.2.2 Topography

As part of Paramount's 2016-2017 work program at Grassy Mountain, a drone aerial survey was conducted over the resource area and detailed topographic data were collected. MDA used the raw data from this survey to create a three-dimensional digital topographic surface for use in the PFS, including the resource modeling.

14.3 Deposit Geology Relevant to Resource Modeling

The Grassy Mountain gold-silver deposit is hosted by arkoses, siltstones, mudstones, and sinters of the Grassy Mountain Formation. As presently drilled, it has extents of 1,900 feet in the strike direction of the higher-grade mineralization (060° to 070°), approximately 2,700 feet perpendicular to the strike, and 1,240 feet in the vertical direction. The deposit is comprised of a central core zone characterized by gold grades in excess of 0.03 oz Au/ton that lies within a broad envelope of lower-grade mineralization. The central core includes the mineralization that is the subject of the economic analysis of the PFS.

The central core zone has extents of almost 1,000 feet along strike, about 450 feet perpendicular to strike, and up to 450 feet in the vertical direction. Subhorizontal and subvertical extensions of the higher-grade central-core mineralization extend outward into the lower-grade envelope, likely due to stratigraphic and structural controls, respectively. The base of the central core is very sharp, marked by a distinct drop in the precious-metal grades, and it is the lower limit of the strong silicification that typifies the entire Grassy Mountain deposit (including the lower-grade envelope).

High-grade mineralization ($\geq$~0.25 oz Au/ton) within the central core zone and its stratigraphic and structural extensions is most frequently associated with thin ($\leq$ 2 inches), often banded, typically steeply-dipping chalcedonic quartz $\pm$ adularia veins/veinlets, although it is important to note that there are examples of high-grade mineralization that have no obvious association with veins, and the presence of veins does not guarantee high grades. The distribution of high-grade mineralization is somewhat erratic, but some systematics to its distribution are evident. For example, the high-grade mineralization is characteristic of the basal portion of the central core, even as continuity remains somewhat limited. In addition, the Grassy fault has also long been hypothesized as playing a pivotal role in the formation of the deposit, and there is evidence of an association of this and other high-angle structural zones with increases in vein density and grades.

Stratigraphic control of mineralization is expressed by lenses of more-or-less concordant mineralization that extend outwards from the margins of both the central core of higher-grade mineralization and its lower-grade envelope. Similar mineralized lenses are associated with the upper portions of the mineralized structural zones as they extend above the central core zone. There are also indications that mineralization within the central core of the deposit may have been influenced by the host stratigraphy as well. While arkose and siltstone are the most common hosts of stratigraphically-controlled mineralization, both sides of the contacts of these interbedded units seem particularly favorable.

MDA believes the Grassy Mountain gold- and silver-bearing hydrothermal fluids were introduced into the Grassy Mountain Formation along a series of 060°- to 070°-striking, steeply-dipping (primarily to the southeast) structural zones, of often minimal displacement, that occur over the full extents of the central core of the deposit. The planar base of this zone and its abrupt change to weakly mineralized and altered rocks below likely reflect the elevation upon which boiling initiated in the ascending hydrothermal fluids and high-grade mineralization was deposited. The unfocussed nature of fluid flow along the many, and sometimes ill-defined, structural zones resulted in the generally erratic deposition of high-grade mineralization throughout the central core zone.

The waning stages of the mineralizing system appear to be manifested by what Newmont named "clay matrix breccias". These breccias are primarily, if not entirely, post-mineral and post-silicification. They are primarily matrix-supported breccias with rotated fragments (some with mineralized quartz veinlets) that range up to boulder-size. Newmont suggested that the breccias formed during, "a period of late-stage boiling along pre-existing conduits as H_2S and CO_2 were expelled from the system" (Jory, 1993). Close inspection of Paramount drill core suggests that the pre-existing conduits are indeed the mineralized structural zones described above. Due to their frequently unconsolidated nature, the clay matrix breccias have geotechnical implications.

Post-mineral faulting has resulted in a slight tilting of the Grassy Mountain deposit and its host stratigraphy to the east.

It is within this context of geology that the gold and silver resource modeling was undertaken.

14.4 Geologic Modeling

Paramount supplied MDA with a set of detailed cross-sectional lithological and structural interpretations that cover most of the extent of the Grassy Mountain deposit. These cross sections were used as the base for MDA's modeling of the gold and silver mineralization.

The structural interpretations were particularly useful to the gold and silver modeling due to the structural controls discussed above. MDA made minor modifications to Paramount's structural interpretations and modeled some additional structures as well.

14.5 Water Table and Oxidation Modeling

Oxidation within the Grassy Mountain deposit is quite variable, making accurate modeling of discreet oxide and/or unoxidized zones impossible. The entire deposit is best characterized as being within a mixed zone (oxidized + partially oxidized + unoxidized), with unoxidized portions typically occurring only on a very local basis.

Hydrologic conditions at the project are discussed in Section 16.3 and Section 16.6.3. Other than potential impacts of down-hole contamination in the RC drill holes (discussed in Section 10.4), the presence or absence of groundwater did not impact the resource modeling.

14.6 Density Modeling

In 1990, Hazen Research, Inc. ("Hazen") completed 314 determinations of bulk density and Atlas completed 61 determinations. The Hazen determinations were done by the water-immersion method on samples of drill core; it is not known if samples with open spaces were coated as part of the testing. The samples were identified by gold grade ranges, but the specific drill intervals tested are not known. The Hazen densities (tonnage factors: ft^3/ton) are summarized in Table 14.1.

Table 14.1 Hazen Research, Inc. Tonnage Factors

Zone	Mean	Median	Min	Max	Count	Grade Range (oz Au/ton)
OZ-1	12.8	12.8	13.7	12.3	63	<0.005
OZ-2	12.8	12.8	14.4	12.3	166	0.003-0.050
OZ-3	13.1	13.0	24.6	11.0	85	0.050-0.750

The Atlas determinations were completed at Atlas' Gold Bar mine in Nevada and are described as being "wet tests" (Steele, 1990). The same internal Atlas memo describes the Hazen method as "wet and dry". It can be inferred from this that the Atlas tests were done using the water-displacement method, but this is uncertain. The drill-core samples tested by Atlas are identified by drill interval and therefore can be spatially located within the deposit.

Newmont completed density testing of 10 samples of drill core (Jory, 1993). Although the test results are not available, Jory stated the results suggest, "a Grassy Mountain tonnage factor closer to 13.3 ft^3/ton".

Paramount requested ALS to complete bulk-density testing on 266 samples of core from Atlas, Calico, and Newmont drilling programs, in addition to 374 determinations on core from Paramount's 2016-2017 drilling program. The determinations were done by the water-immersion method (ALS codes OA-GRA08), coating with paraffin wax when necessary (OA-GRA08A). Two of the sinter determinations are anomalously high (low tonnage factors) and were removed from the dataset.

The density data of Atlas and Paramount were examined collectively and individually by rock types and gold domains modeled as part of the resource estimation. In general, average tonnage factors from the Atlas data for the lithological and grade subgroups are slightly lower (higher density) than those from the Paramount determinations. The combined Atlas and Paramount dataset grouped by modeled gold domain is summarized in Table 14.2. Domain 100 is the low-grade gold domain (~0.006 to ~0.030 oz Au/ton) modeled by MDA and the higher-grade mineralization is within Domain 200 (> ~0.030 oz Au/ton).

Table 14.2 Combined Atlas and Paramount Tonnage Factors

Gold Domain	Mean	Median	Min	Max	Count	Block Model
100	13.3	13.0	21.5	11.6	341	13.5
200	13.0	12.9	14.7	12.4	275	13.5
100+200	13.2	12.9	21.5	11.6	616	n/a
0	14.8	14.5	23.0	11.2	83	14.8

Inclusive of the Hazen tests, the results suggest that the Grassy Mountain mineralization has a consistent density, while unmineralized rocks are distinctly lighter. This is likely a reflection of alteration, as mineralization of all grades is strongly silicified, while unmineralized portions of the host rocks are generally far less silicified, if at all.

The block model tonnage factors shown on Table 14.2 were used for the resource estimation. The model tonnage factors are higher than the measured core to account for unmeasurable open spaces related to the relatively high degree of fracturing that characterizes the Grassy Mountain deposit.

14.7 Gold and Silver Modeling

14.7.1 Mineral Domains

A mineral domain encompasses a volume that ideally is characterized by a single, natural, grade population of a metal or metals that occurs within a specific geologic environment. In order to define the mineral domains at Grassy Mountain, the natural gold and silver populations were first identified on population-distribution graphs that plot the gold- and silver-grade distributions of all of the project drill-hole assays, as well as distribution plots using only analyses from core samples. This analysis led to the identification of three populations for both gold and silver. Ideally, each of these populations can then be correlated with specific geologic characteristics that are captured in the project database, which can be used in conjunction with the grade populations to interpret the bounds of each of the gold and silver mineral domains. In the case of Grassy Mountain, the high-grade population of gold (>~0.25 oz Au/ton) and silver (>~0.4 oz Ag/ton) do not have sufficient continuity for confident modeling of the domains, and therefore these populations were not explicitly modeled. The approximate grade ranges of the lower-grade (domain 100) and higher-grade (domain 200) domains that were modeled for gold and silver are listed in Table 14.3.

Table 14.3 Approximate Grade Ranges of Gold and Silver Domains

Domain	oz Au/ton	oz Ag/ton
100	~0.006 to ~0.03	~0.04 to ~0.15
200	> ~0.03	> ~0.15

The Grassy Mountain gold and silver mineralization was modeled by first interpreting gold and silver mineral-domain polygons individually on a set of vertical, 50-foot spaced, northeast-looking (070°) cross sections that span the extents of the deposit. The mineral domains were interpreted using the gold and silver drill-hole assay data and associated alteration and mineralization codes, as well as sectional lithological and structural interpretations by Paramount. Core photographs were also referred to extensively during the sectional modeling. This information was used to discern the stratigraphic and structural controls of the mineralization discussed in Section 14.3 and to model the domains accordingly. Gold was modeled first, and the sectional gold-domain polygons were then used as guides for defining the silver domains.

The inherent variability of the Grassy Mountain mineralization resulted in the need for including significant quantities of lower-grade mineralization within some volumes of the higher-grade domain (domain 200). As stated above, this variability also precluded confident modeling of the highest-grade population of gold and silver, which therefore was also encompassed within the 200 domains of gold and silver. The highest-grade gold population (>~0.25 oz Au/ton) is perhaps the most readily identifiable grade population in drill core, as it strongly correlated with the presence of thin, often banded, quartz/chalcedony veins/veinlets and/or breccias (and sometimes visible gold) that in certain portions of the higher-grade domain correlate well with highest grades. Taking drill-hole orientations and angles to core axes into account, the high-grade veinlets are most commonly steeply dipping.

The boundary between the lower- and higher-grade domains was largely determined by grade. Although the grade change across this domain boundary is usually sharp, it is locally gradational. The grade change across the sub-horizontal base of the higher-grade domain is usually quite sharp, especially in core holes, and it can be marked by a significant decrease in the intensity of silicification. This basal contact of domain 200 is likely indicative of the elevation of the initiation of boiling in the Grassy Mountain hydrothermal system.

The mineralization captured within the lower-grade domain (domain 100) is much less variable than the higher-grade mineralization. This mineralization is distal from the zone of boiling and related brecciation, and its distribution exhibits strong effects from stratigraphic controls.

The cross-sectional gold and silver mineral-domain envelopes were sliced at 10-foot vertical intervals that match the mid-bench elevations of the block model. The slices were then used to create a new set of mineral-domain polygons for both gold and silver on level plans at 10-foot spacings in order to rectify the domain interpretations to the drill-hole data at the scale of the block model.

Cross-sections showing examples of the gold and silver mineral domains in the central portion of the Grassy Mountain deposit are shown in Figure 14.1 through Figure 14.4.

Figure 14.1 Cross Section 3050 Showing Gold Domains

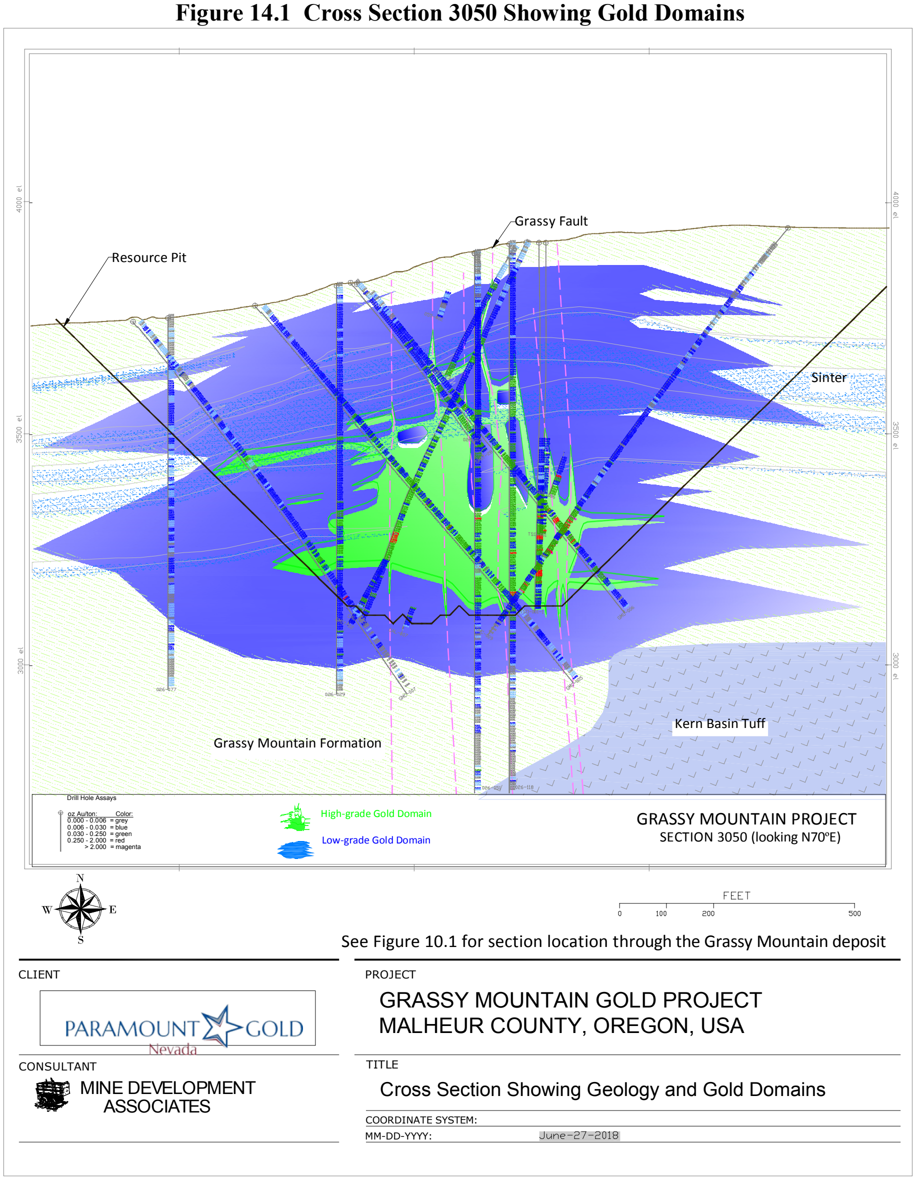
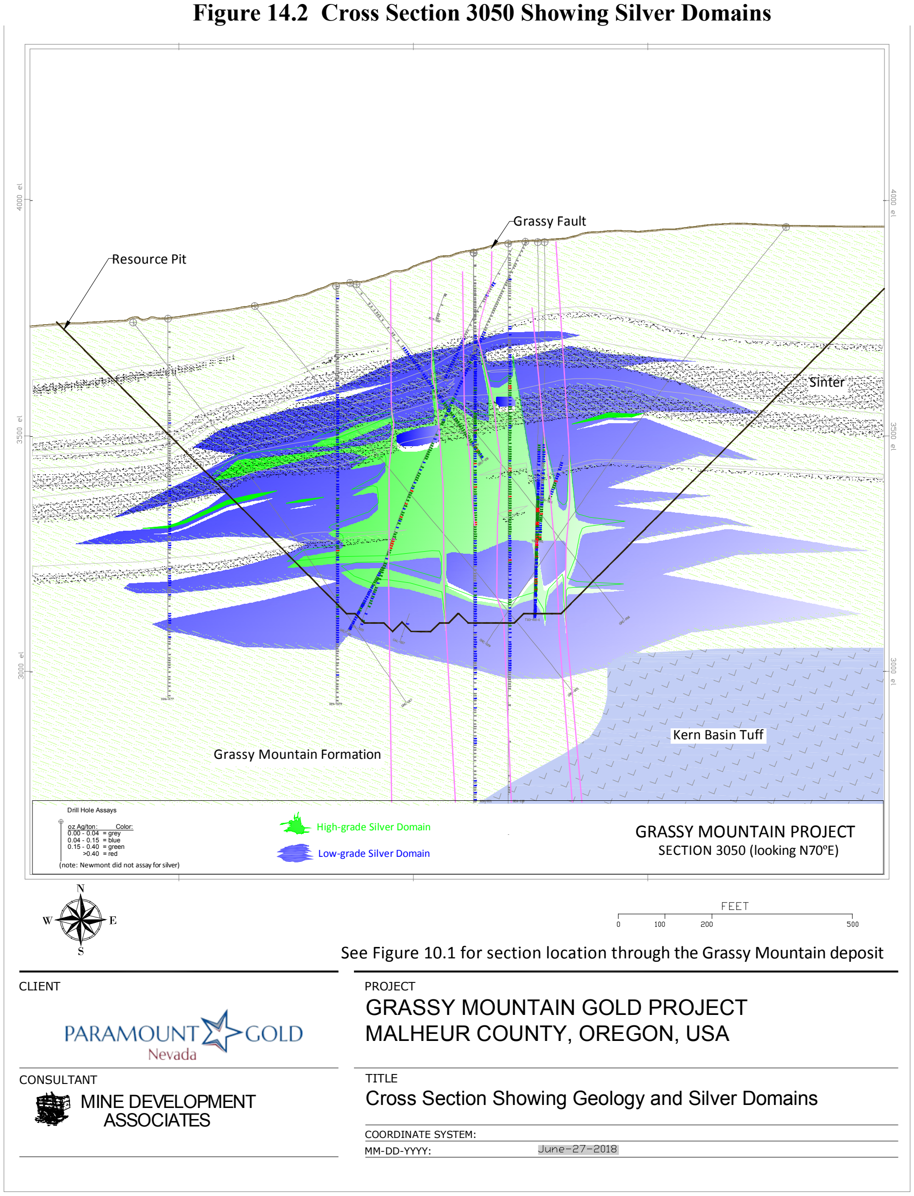


Figure 14.2 Cross Section 3050 Showing Silver Domains



Note: Newmont holes lack silver assays

Figure 14.3 Cross Section 3250 Showing Gold Domains

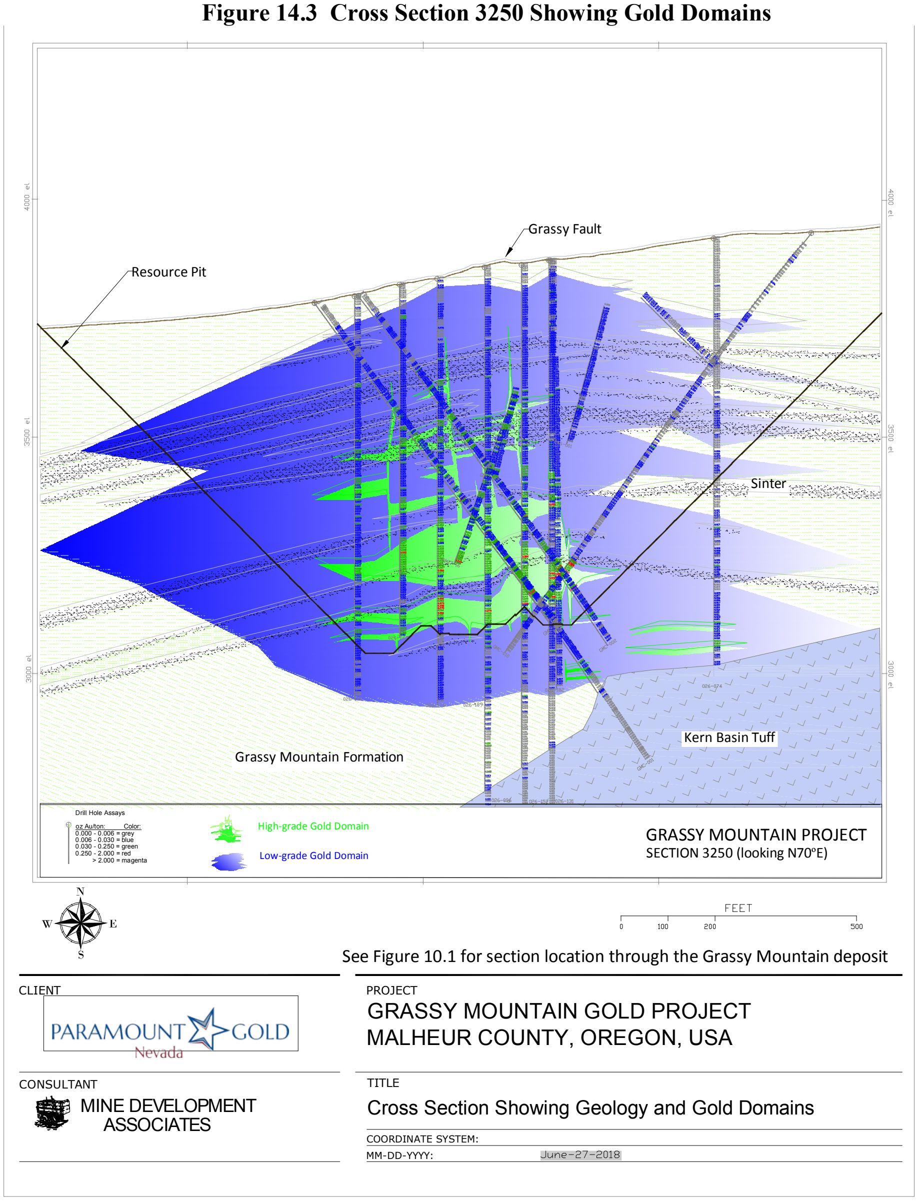
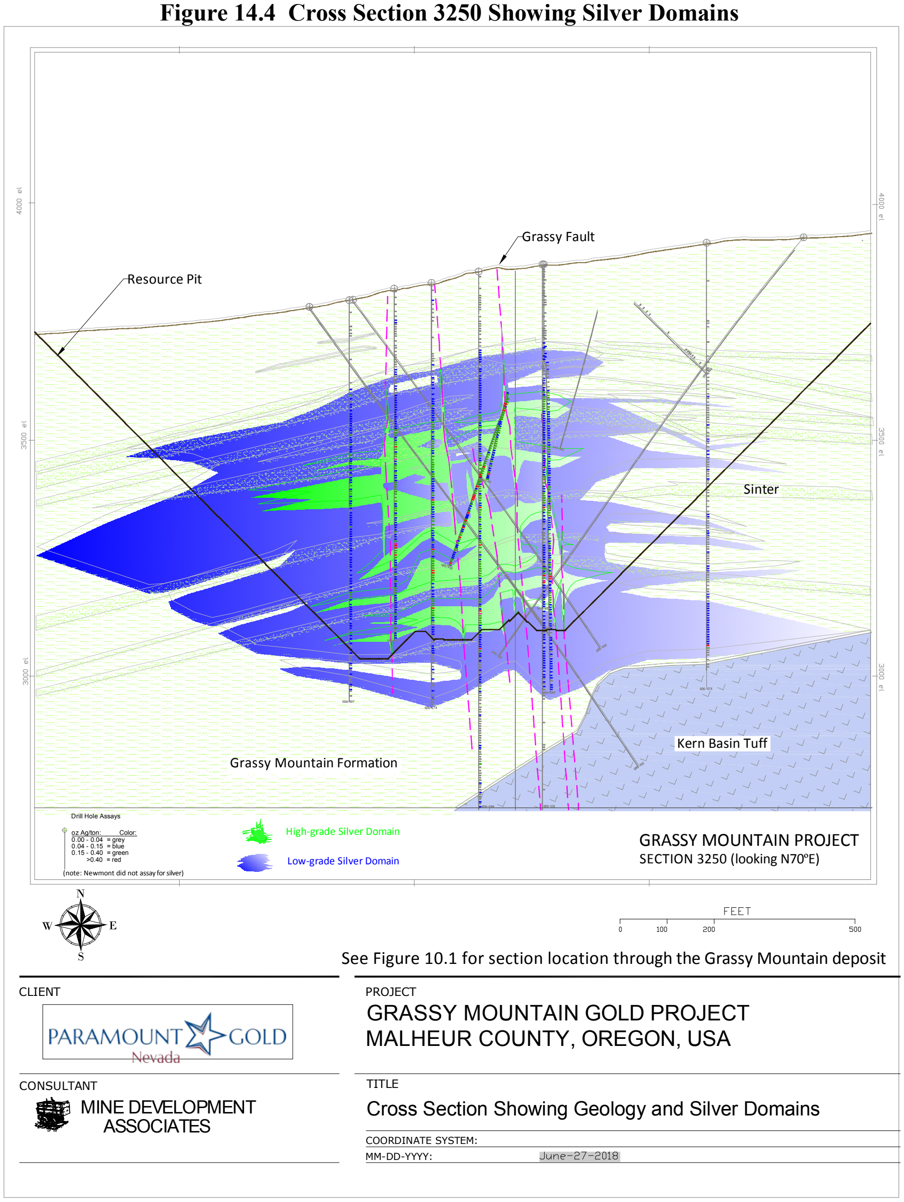


Figure 14.4 Cross Section 3250 Showing Silver Domains



Note: Newmont holes lack silver assays

14.7.2 Assay Coding, Capping, and Compositing

Drill-hole gold and silver assays were coded to the gold and silver mineral domains, respectively, using the cross-sectional polygons. Assay caps were determined by the inspection of population distribution plots of the coded assays, by domain, to identify high-grade outliers that might be appropriate for capping. The plots were also evaluated for the possible presence of multiple grade populations within each of the gold domains. Descriptive statistics of the coded assays by domain and visual reviews of the spatial relationships of the possible outliers, and their potential impacts during grade interpolation, were also considered in the definition of the assay caps (shown in Table 14.4).

Table 14.4 Grassy Mountain Gold and Silver Assay Caps by Domain

Domain	oz Au/ton	Number Capped (% of samples)	oz Ag/ton	Number Capped (% of samples)
0	0.090	8 (<1%)	0.120	12 (<1%)
100	0.300	3 (<1%)	0.600	4 (<1%)
200	10.000	4 (<1%)	7.000	2 (<1%)

Each model block was coded to the volume percentage of each of the two domains for both gold and silver, as discussed below. For model blocks that are not entirely coded to the lower- and higher-grade domains for either or both metals, these outside-domain volumes of the blocks (assigned as "domain 0") were also estimated using assays lying outside of the domains (uncoded, domain 0 assays"). The domain 0 assays used in this dilutionary estimate were also capped as shown in Table 14.4.

In addition to the assay caps, restrictions on the search distances of higher-grade portions of some of the domains were applied during grade interpolations (discussed further below). The use of search restrictions can allow one to minimize the number of samples subjected to capping while properly respecting the highest-grade populations within each domain.

Descriptive statistics of the capped and uncapped coded assays are provided in Table 14.5 and Table 14.6 for gold and silver, respectively.

Table 14.5 Descriptive Statistics of Grassy Mountain Coded Gold Assays

Domain	Assays	Count	Mean (oz Au/ton)	Median (oz Au/ton)	Std. Dev.	CV	Min. (oz Au/ton)	Max. (oz Au/ton)
0	Au	23,361	0.002	0.001	0.007	3.45	0.000	0.732
	Au Cap	23,361	0.002	0.001	0.004	2.15	0.000	0.090
100	Au	24,808	0.013	0.011	0.011	0.82	0.000	0.561
	Au Cap	24,808	0.013	0.011	0.010	0.77	0.000	0.300
200	Au	7,523	0.108	0.044	0.441	4.09	0.000	21.698
	Au Cap	7,523	0.107	0.044	0.405	3.79	0.000	10.000
100+200	*Au*	*32,331*	*0.033*	*0.013*	*0.209*	*6.27*	*0.000*	*21.698*
	Au Cap	*32,331*	*0.033*	*0.013*	*0.193*	*5.81*	*0.000*	*10.000*

Table 14.6 Descriptive Statistics of Grassy Mountain Coded Silver Assays

Domain	Assays	Count	Mean (oz Ag/ton)	Median (oz Ag/ton)	Std. Dev.	CV	Min. (oz Ag/ton)	Max. (oz Ag/ton)
0	Ag	20,921	0.009	0.005	0.011	1.19	0.000	0.496
	Ag Cap	20,921	0.009	0.005	0.010	1.11	0.000	0.120
100	Ag	13,292	0.071	0.064	0.040	0.57	0.003	1.138
	Ag Cap	13,292	0.071	0.064	0.039	0.55	0.003	0.600
200	Ag	6,646	0.262	0.200	0.400	1.52	0.005	18.600
	Ag Cap	6,646	0.260	0.200	0.310	1.19	0.005	7.000
100+200	*Ag*	*19,938*	*0.132*	*0.085*	*0.246*	*1.86*	*0.003*	*18.600*
	Ag Cap	*19,938*	*0.131*	*0.085*	*0.199*	*1.51*	*0.003*	*7.000*

The capped assays were composited at five-foot down-hole intervals respecting the mineral domains. The five-foot composite length is equal to the sample length of RC drill samples, which means that the RC sample data are effectively not composited at all, while core intervals shorter than five feet are composited. This minimal compositing was deliberately chosen as part of an effort to retain some of the inherent variability of the Grassy Mountain mineralization in the resource modeling. Descriptive statistics of Grassy Mountain composites are shown in Table 14.7 and Table 14.8 for gold and silver, respectively.

Table 14.7 Descriptive Statistics of Grassy Mountain Gold Composites

Domain	Count	Mean (oz Au/ton)	Median (oz Au/ton)	Std. Dev.	CV	Min. (oz Au/ton)	Max. (oz Au/ton)
0	23,452	0.00	0.00	0.00	2.15	0.00	0.09
100	24,213	0.01	0.01	0.01	0.74	0.00	0.30
200	6,738	0.11	0.05	0.35	3.30	0.00	9.89
100+200	*30,951*	*0.03*	*0.01*	*0.17*	*5.09*	*0.00*	*9.89*

Table 14.8 Descriptive Statistics of Grassy Mountain Silver Composites

Domain	Count	Mean (oz Ag/ton)	Median (oz Ag/ton)	Std. Dev.	CV	Min. (oz Ag/ton)	Max. (oz Ag/ton)
0	20,910	0.009	0.005	0.010	1.100	0.000	0.120
100	12,985	0.071	0.067	0.038	0.530	0.003	0.600
200	6,137	0.260	0.200	0.295	1.140	0.005	7.000
100+200	*19,122*	*0.131*	*0.085*	*0.191*	*1.460*	*0.003*	*7.000*

14.7.3 Block Model Coding

The level-plan mineral-domain polygons were used to code a three-dimensional block model with a model bearing of 340° that is comprised of 10 x 10 x 10-foot blocks. The volume percent of each mineral domain for both gold and silver is stored within each block (the "partial percentages"). The block model was also coded using the project digital topographic surface described in Section 14.2.2.

The specific-gravity values discussed in Section14.6 were assigned to model blocks, so that blocks coded as having with any partial percentage of gold or silver have a density of 13.5 ft^3/ton and all other blocks are assigned a value of 14.8 ft^3/ton.

14.7.4 Grade Interpolation

The parameters applied to the gold-grade estimations at Grassy Mountain are summarized in Table 14.9. Grade interpolation was completed in three passes using length-weighted composites.

Due to the varying effects of subvertical structural controls and subhorizontal lithological controls, the modeled mineralization has a number of orientations throughout the deposit. The block model was therefore coded to two unique estimation areas (areas 10 and 20). Estimation area 10 encompasses most of the Grassy Mountain deposit and is characterized by shallow dips of the stratigraphic host rocks of up to about -15°. Estimation area 20 is comprised of only the west-southwesternmost portion of the deposit where the dips of the stratigraphic units steepen to approximately -20°. As shown in Table 14.9, the lower-grade gold and silver domains, as well as domain 0, were entirely estimated using search ellipses that reflect these stratigraphic orientations.

The higher-grade gold and silver domains exhibit both subhorizontal (stratigraphic) and high-angle (structural) controls. In order to prioritize the estimation of the highest-grade mineralization, which is most commonly associated with steeply dipping veinlets, the estimation of the higher-grade domain was initiated to reflect high-angle structural control (Table 14.9 - estimation area 10, domain 200, pass 1). The second estimation pass of the higher-grade domain invoked a search ellipse reflective of stratigraphic control while using the same search distance as pass 1 (50 feet). The third and final estimation pass was an isotropic pass, i.e. without either a structural or stratigraphic bias, and was used to estimate domain 200 grades into blocks that were not estimated by the first two passes, which are largely limited to the outer extents of the domain.

Only a very limited portion of the higher-grade gold and silver domains lie in estimation area 20.

Table 14.9 Summary of Grassy Mountain Estimation Parameters

Estimation Pass – Au + Ag Domain	Search Ranges (ft)			Composite Constraints		
	Major	Semi-Major	Minor	Min	Max	Max/Hole
Pass 1 - Doman 0 + 100	100	100	50	2	15	3
Pass 2 - Doman 0 + 100	200	200	100	2	15	3
Pass 3 - Doman 0 + 100	300	300	300	1	15	3
Pass 1 + 2 - Doman 200	50	50	16.7	2	15	3
Pass 3 - Doman 200	100	100	100	1	15	3

Restrictions on Search Ranges

Domain	Grade Threshold	Search Restriction Distance	Estimation Pass
Au 200	>0.30 oz Au/ton	35 feet	2
Au 0	>0.01 oz Au/ton	30 feet	1, 2, 3
Ag 0	>0.04 oz Ag/ton	30 feet	1, 2, 3

Search Ellipse Orientations

Estimation Area	Au + Ag Domains and Controls	Major Bearing	Plunge	Tilt	Estimation Pass
10 [most of the deposit]	Domain 0 + 100 – stratigraphic	0°	0°	-15°	1, 2, 3
	Domain 200 – structural	070°	0°	-85°	1
	Domain 200 – stratigraphic	070°	-10°	0°	2
	Domain 200 – stratigraphic	0°	0°	0°	3
20 [WSW end of the deposit]	Domain 0 + 100 + 200 – stratigraphic	070°	0°	20°	1, 2, 3

Statistical analyses of coded assays and composites, including coefficients of variation and population-distribution plots, indicate that multiple populations of significance were captured in the higher-grade domain (domain 200) of both gold and silver. This recognition of multiple populations within the higher-grade domains, coupled with the results of initial grade-estimation runs that indicated the higher-grade samples were affecting inappropriate volumes in the model, led to the restrictions on the search distances for higher-grade populations within some domains. These restrictions place limits on the maximum distances from a block that the highest-grade composites can be used in the interpolation of gold and silver grade into that block. The final search-restriction parameters were derived from the results of multiple interpolation iterations that employed various search-restriction parameters.

Gold and silver grades were interpolated using inverse-distance to the third power, ordinary-krige, and nearest-neighbor methods. The mineral resources reported herein were estimated by the inverse-distance interpolation, as this method led to results that were judged to more closely approximate the drill data than those obtained by ordinary kriging. The nearest-neighbor estimation was completed as a check on the inverse-distance and krige interpolations.

The estimation passes were performed independently for each of the mineral domains, so that only composites coded to a particular domain were used to estimate grade into blocks coded by that domain.

The estimated grades were coupled with the partial percentages of the mineral domains and the outside-domain volumes to enable the calculation of weight-averaged gold and silver grades for each block. The final resource grades, and their associated resource tonnages, are therefore fully block-diluted using this methodology.

14.7.1 Model Checks

Gold and silver domain volumes coded into the block model were compared to those derived from the cross-sectional and level-plan mineral domains to assure close agreement, and all block-model coding was checked visually. A polygonal estimate using the cross-sectional domain polygons was used as a check on the inverse-distance estimation results, as were the nearest-neighbor and ordinary-krige estimates. No unexpected relationships between the check estimates and the inverse-distance estimate were identified. Various grade-distribution plots of assays, composites, and nearest-neighbor, ordinary-krige, and inverse-distance block grades were evaluated as a check on both the global and local estimation results. Finally, the inverse-distance grades were visually compared to the drill-hole assay data to assure that reasonable results were obtained.

14.8 Grassy Mountain Mineral Resources

The Grassy Mountain deposit has the potential to be mined by open-pit methods. While the mineral reserves discussed in Section 15.0 are estimated on the basis of a proposed underground-mining scenario, these reserves represent only a subset of the entire gold-silver deposit. The Grassy Mountain mineral resources have therefore been estimated to reflect potential open-pit extraction and milling as the primary scenario ("in-pit resources"), with potential underground mining of material lying outside of the pit as a secondary scenario ("underground resources"). The mineral reserves discussed in Section 15.0 are derived from both the in-pit resources and, to a much lesser extent, the underground resources.

To meet the requirement of the in-pit resources having reasonable prospects for eventual economic extraction, a pit optimization was run using the parameters summarized in Table 14.10.

Table 14.10 Pit Optimization Parameters

Mining Cost	$	2.00	$/ton
Processing Cost	$	13.00	$/ton processed
Tons per Day		5,000	tons-per-day processed
G&A per Ton	$	2.22	$/ton processed
Au Price	$	1,500	$/oz
Ag Price	$	20	$/oz
Au Recovery		80%	
Ag Recovery		60%	
Au Refining Cost	$	5.00	$/oz produced
Ag Refining Cost	$	0.50	$/oz produced

The pit shell created by this optimization was used to constrain the in-pit resources, with the added constraint of a gold-equivalent cutoff grade of 0.012 oz/ton applied to all model blocks lying within the optimized pit. The gold equivalent grade ("oz AuEq/ton") of each model block was calculated as follows:

$$\text{oz AuEq/ton} = \text{oz Au/ton} + (\text{oz Ag/ton} \div 100)$$

The silver-to-gold equivalency factor of 100 was derived from the metal prices and recoveries in Table 14.10.

Underground resources were estimated by applying a cutoff of 0.060 oz AuEq/ton to blocks lying immediately outside of the optimized pit. Table 14.11 lists the parameters used to calculate the underground cutoff grade.

Table 14.11 Parameters Used to Determine Underground Resource Cutoff Grade

Mining Cost	$	50.00	$/ton
Processing Cost	$	25.00	$/ton processed
Tons per Day		5,000	tons-per-day processed
G&A per Ton	$	8.00	$/ton processed
Au Price	$	1,500	$/oz
Ag Price	$	20	$/oz
AuEq Recovery		90%	
Refining Cost	$	5.00	$/oz produced

Both the in-pit and underground resources are based on a 5,000 ton per day processing rate, with processing assumed to consist of crushing, milling, and first-stage gravity separation followed by carbon-in-leach recovery. Total project resources, including the in-pit and small amount of underground resources, are presented in Table 14.12. The open-pit and underground portions of the resources are shown in Table 14.13 and Table 14.14, respectively. The resources are inclusive of the mineral reserves defined in Section 15.0. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 14.12 Grassy Mountain Gold and Silver Resources

Classification	Tons	oz Au/ton	oz Au	oz Ag/ton	oz Ag
Measured	17,933,000	0.020	363,000	0.079	1,409,000
Indicated	12,886,000	0.054	695,000	0.146	1,882,000
Measured + Indicated	30,819,000	0.034	1,058,000	0.107	3,291,000
Inferred	1,055,000	0.040	42,000	0.119	125,000

1. Mineral resources are comprised of all model blocks at a 0.012 oz AuEq/ton cutoff that lie within an optimized pit plus blocks at a 0.060 oz AuEq/ton cutoff that lie outside of the optimized pit;
2. oz AuEq/ton (gold equivalent grade) = oz Au/ton + (oz Ag/ton ÷ 100);
3. The mineral resources are inclusive of the mineral reserves reported in Section 15.0;
4. Mineral resources that are not mineral reserves do not have demonstrated economic viability;
5. The Effective Date of the Grassy Mountain resource estimate is May 1, 2018; and
6. Rounding may result in apparent discrepancies between tons, grade, and contained metal content.

Table 14.13 Grassy Mountain Open-Pit Resources

Classification	Tons	oz Au/ton	oz Au	oz Ag/ton	oz Ag
Measured	17,902,000	0.020	360,000	0.078	1,405,000
Indicated	12,826,000	0.054	689,000	0.146	1,875,000
Measured + Indicated	30,728,000	0.034	1,049,000	0.106	3,280,000
Inferred	1,034,000	0.039	41,000	0.119	123,000

Table 14.14 Grassy Mountain Underground Resources

Classification	Tons	oz Au/ton	oz Au	oz Ag/ton	oz Ag
Measured	31,000	0.090	3,000	0.125	4,000
Indicated	60,000	0.095	6,000	0.120	7,000
Measured + Indicated	91,000	0.093	9,000	0.122	11,000
Inferred	21,000	0.075	1,600	0.089	2,000

The Grassy Mountain resources were classified according to the criteria presented in Table 14.15. The parameters of the gold estimation control the resource classification because gold is much more significant than silver from a potential economic standpoint.

Table 14.15 Resource Classification

Class	Criteria	Distance from Nearest Composite
Measured	All estimated blocks with Au Domain 200 coding	≤ 10 feet
	All estimated blocks with Au Domain 100 coding; no Au D200 coding	≤ 50 feet
Indicated	All estimated blocks with Au Domain 200 coding; not classified as Measured	≤ 50 feet
	All estimated blocks with Au Domain 100 coding; no D200 coding; not classified as Measured	≤ 100 feet
Inferred	All other estimated blocks	

The higher-grade gold is highly variable, while the lower-grade mineralization is much more continuous. In consideration of this, two sets of criteria were used in the classification: one set of more restrictive parameters for all blocks coded as having any percentage of gold domain 200 (the higher-grade domain), and another less restrictive set of criteria for all other blocks coded to the lower-grade gold domain (domain 100).

Although the authors are not expert with respect to any of the following aspects of the project, the authors are not aware of any unusual environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors not discussed in this report that could materially affect the potential development of the Grassy Mountain project mineral resources as of the Effective Date of the report.

Figure 14.5 through Figure 14.8 are cross-sections through the central portion of the Grassy Mountain deposit that show estimated block-model gold and silver grades. These figures correspond to the mineral-domain cross-sections presented in Figure 14.1 through Figure 14.4.

Figure 14.5 Cross Section 3050 Showing Block-Model Gold Grades

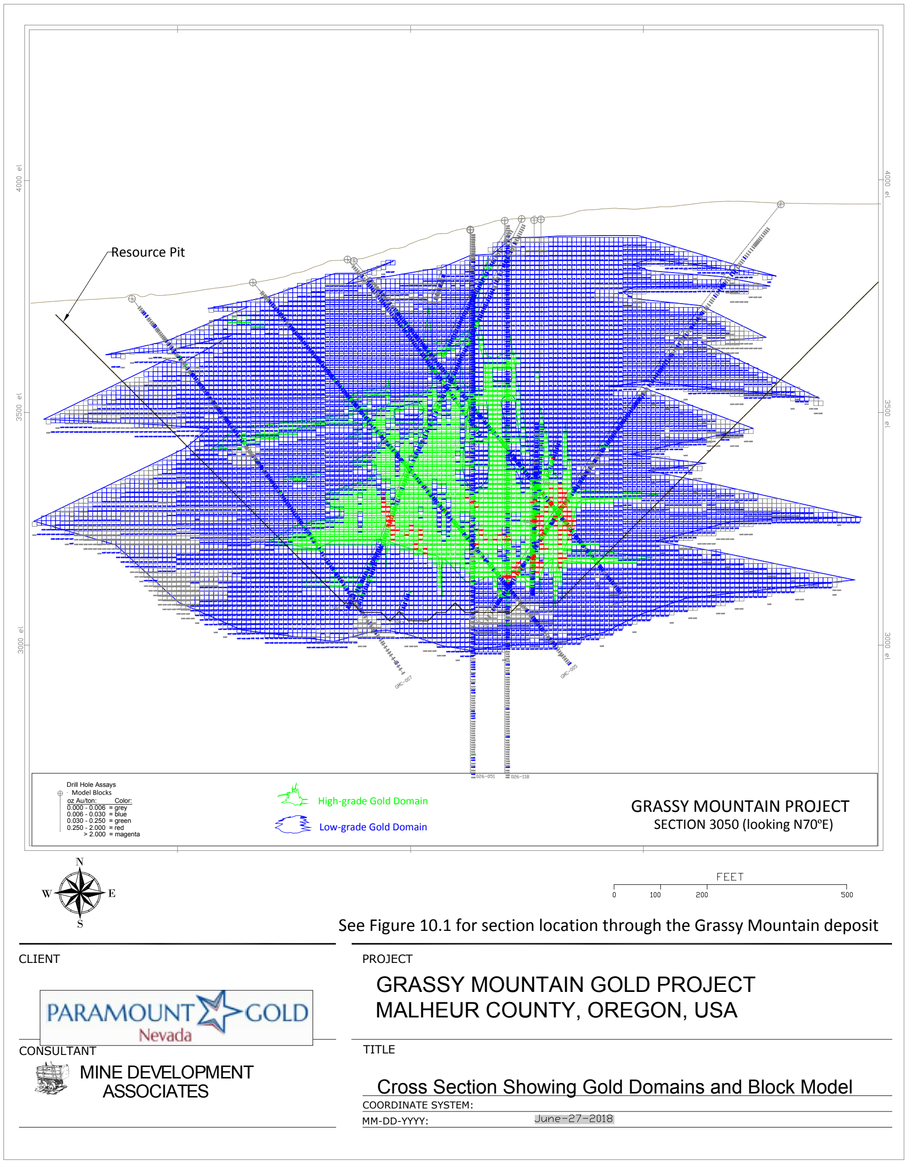


Figure 14.6 Cross Section 3050 Showing Block-Model Silver Grades

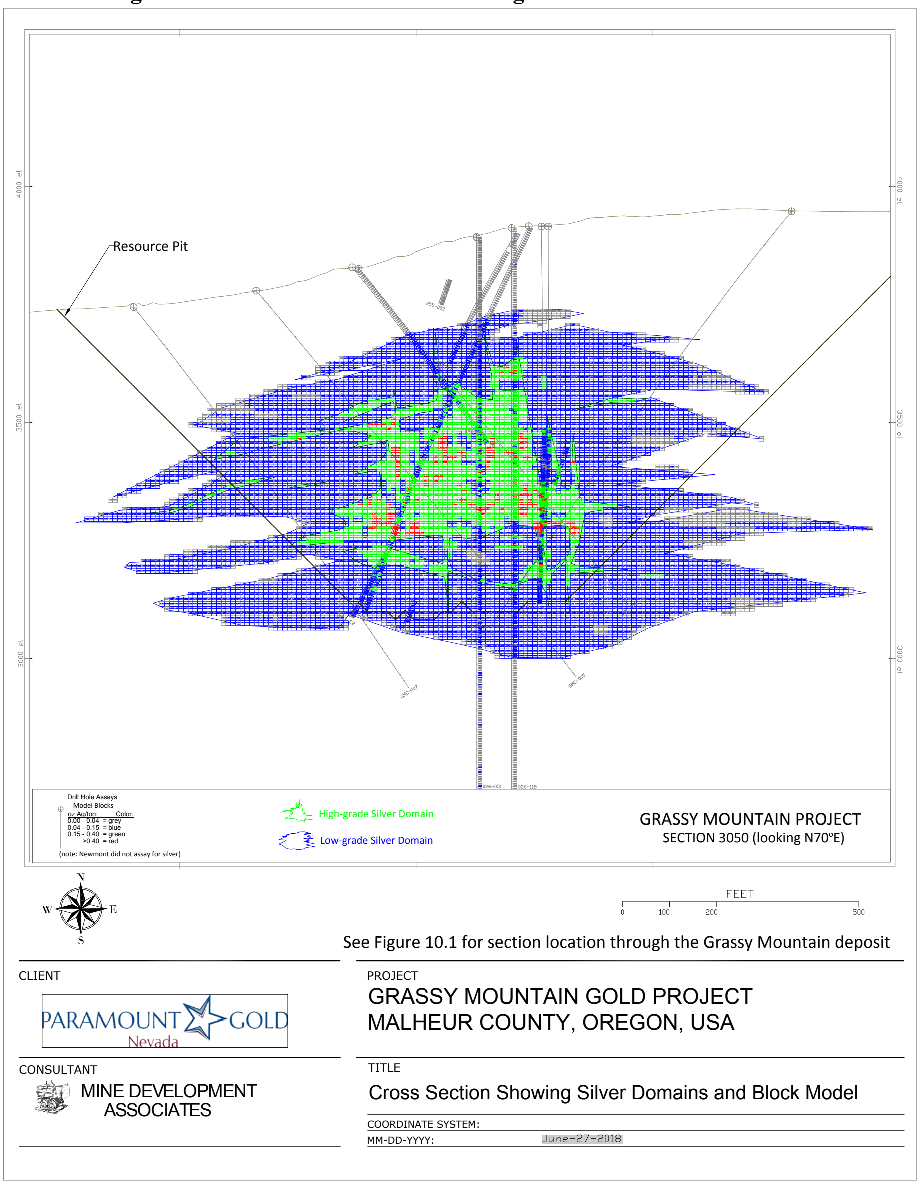


Figure 14.7 Cross Section 3250 Showing Block-Model Gold Grades

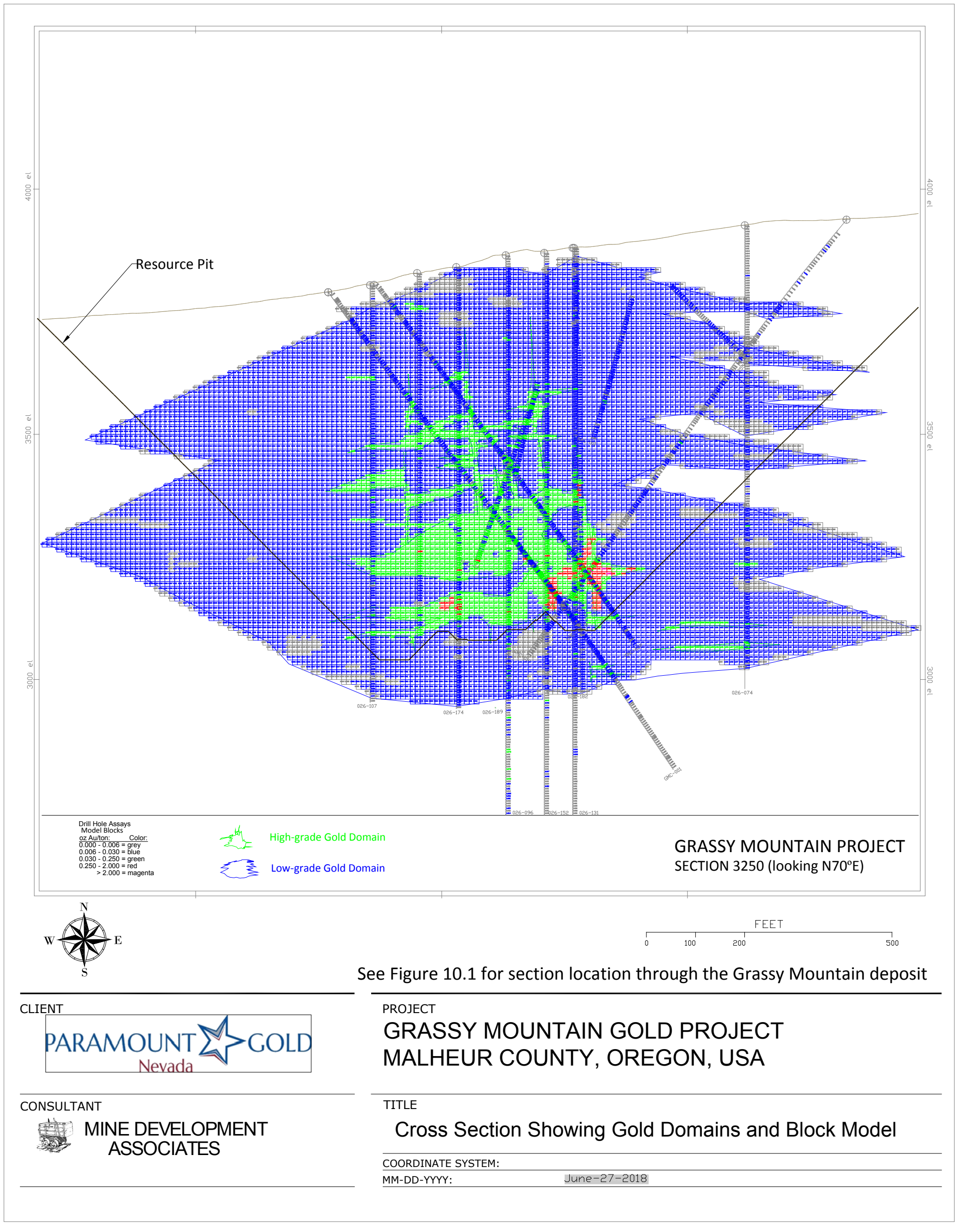
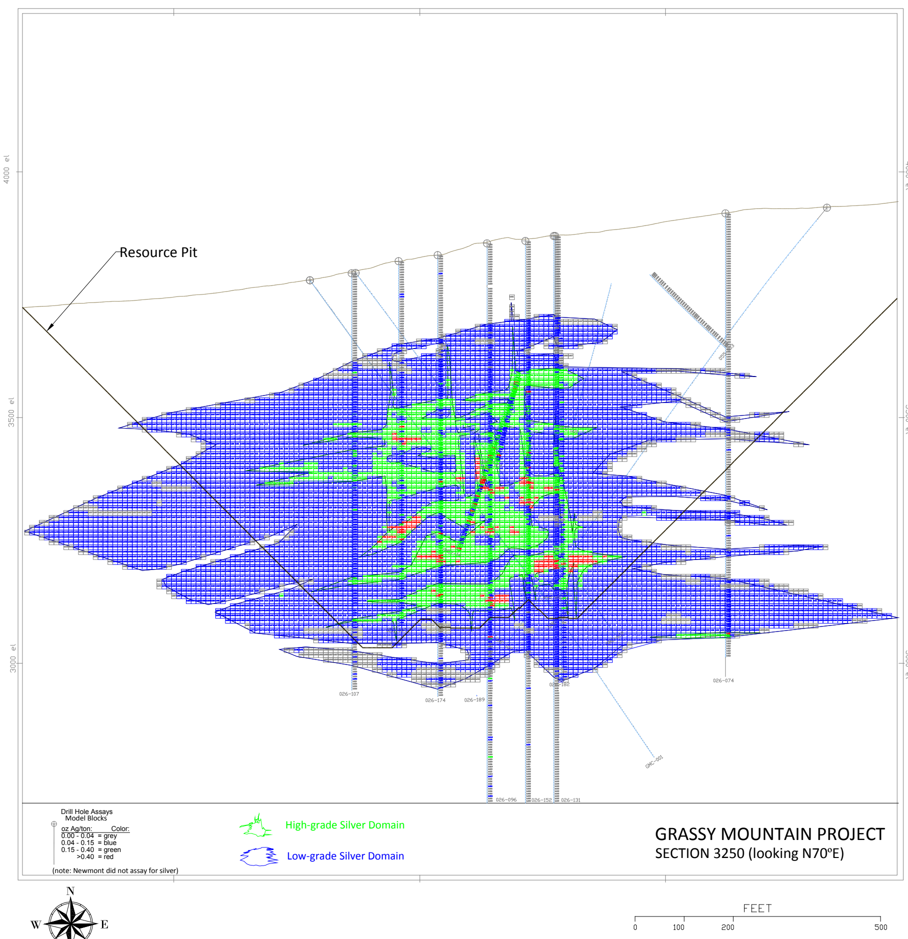


Figure 14.8 Cross Section 3250 Showing Block-Model Silver Grades



See Figure 10.1 for section location through the Grassy Mountain deposit

CLIENT

PARAMOUNT GOLD Nevada

PROJECT

GRASSY MOUNTAIN GOLD PROJECT
MALHEUR COUNTY, OREGON, USA

CONSULTANT

MINE DEVELOPMENT ASSOCIATES

TITLE

Cross Section Showing Silver Domains and Block Model

COORDINATE SYSTEM:

MM-DD-YYYY: June-27-2018

14.9 Comments on the Resource Modeling

A total of 255 holes directly contribute assay data to the estimate of the Grassy Mountain resources. Atlas drilled 180 of these holes, and only four of these were inclined and nine were core or core holes pre-collared by RC. This predominance of vertical RC holes led all subsequent operators to emphasize angled holes and core holes in their drilling programs. There are now 59 core holes, including 27 drilled by Paramount, and 55 angled holes, including 18 by Paramount, that contribute to the resource estimation, almost all of which were drilled into the area of the central higher-grade core of the deposit.

As a check on the impact of the Paramount drilling program, as well as to aid in the verification of the historical drilling data, the Paramount holes were removed from use in a resource estimate that was otherwise identical to that used to estimate the current resources. On a global basis (no cutoff), the exclusion of the Paramount drill data resulted in a decrease in gold ounces of 0.4%. At various cutoffs from 0.005 to 0.090 oz Au/ton, the highest-magnitude change was a decrease in ounces of 0.9%. This constancy in the ounces estimated serves to support the use of the historical drilling data used in the resource estimation. This does not imply that the Paramount drilling had no impact on the current estimate of the project resources; the Paramount core holes greatly increased the quantity of drill core physically available from which to confirm and update the geologic understanding of the deposit, thereby significantly enhancing the confidence in the resource modeling.

The central higher-grade core of the deposit, which would be critical to the economic viability of any potential mining of the deposit, has predominantly been drilled at spacings of about 30 to 50 feet. Even at this relatively tight drill density, the highest-grade mineralization (>~0.2 oz Au/ton) cannot be confidently correlated from hole to hole in many cases. This high-grade population therefore could not be explicitly modeled. Although care was taken to properly represent the distribution of the high-grade population in the resource grade estimation, the locations of these high grades in the resource model likely vary from reality as distances from drill data increase. Closely-spaced drilling will therefore be required in any future underground mine at the Grassy Mountain deposit. This drilling should be undertaken prior to mining of any particular sector of the deposit, with the data used to update the operation's short-term resource model, as well as to create final stope designs for each mining sector.

There is a total of 14,947 sample intervals in the Grassy Mountain drill-hole database that have gold assays but no silver analyses. In most of these cases, entire holes were not assayed for silver. For example, some of the early Atlas holes and none of the Newmont holes were assayed for silver. A total of 4,720 of the sample intervals lacking silver assays lie within the gold domains that form the basis of the resource estimation, while 19,938 sample intervals used in the resource estimation do have silver analyses. The lower quantity of silver analyses is mitigated by the fact that silver would add very little value relative to gold in any potential mining operation.

Structural zones are thought to be one of the principal controls of high-grade mineralization in the central core of the Grassy Mountain deposit. These structural zones are also important from a geotechnical standpoint, as they are characterized by poor rock quality. The geological modeling that supports the current resource estimation includes these fault zones, but additional angled core holes would be useful to better define the extents of the structural zones and thereby aid in refining the geotechnical and high-grade modeling of the deposit.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

Mineral reserves were estimated under the supervision of Mr. Boris Caro and classified in order of increasing confidence into Probable and Proven categories to be in accordance with the "CIM Definition Standards - For Mineral Resources and Mineral Reserves" (2014) and therefore Canadian National Instrument 43-101. Mr. Caro is independent of Paramount and has no affiliations with Paramount except that of independent consultant/client relationship.

CIM mineral reserve definitions are given below, with CIM's explanatory material shown in italics:

Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant Modifying Factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

'Reference point' refers to the mining or process point at which the Qualified Person prepares a Mineral Reserve. For example, most metal deposits disclose mineral reserves with a "mill feed" reference point. In these cases, reserves are reported as mined ore delivered to the plant and do not include reductions attributed to anticipated plant losses.

In contrast, coal reserves have traditionally been reported as tonnes of "clean coal". In this coal example, reserves are reported as a "saleable product" reference point and include reductions for plant yield (recovery). The Qualified Person must clearly state the 'reference point' used in the Mineral Reserve estimate.

Probable Mineral Reserve

A Probable Mineral Reserve is the economically mineable part of an Indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.

Proven Mineral Reserve

A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study. Within the CIM Definition standards the term Proved Mineral Reserve is an equivalent term to a Proven Mineral Reserve.

Modifying Factors

Modifying Factors are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and governmental factors.

The mineral reserves estimated for the Grassy Mountain project are shown in Table 15.1 and are included in the estimated Measured and Indicated mineral resources presented in Table 14.12. The Effective Date of the estimated mineral reserves is May 1, 2018.

Table 15.1 Mineral Reserve Statement

Classification	Tons (Million)	Gold Grade oz Au/ton	Silver Grade oz Ag/ton	Contained Metal (oz Au)	Contained Metal (oz Ag)
Proven	0.23	0.191	0.27	43,000	62,000
Probable	1.49	0.214	0.30	319,000	454,000
Proven + Probable	1.72	0.210	0.30	362,000	516,000

Notes:

5. *Mineral reserves have an Effective Date of May 1, 2018. The Qualified Person for the estimate is Mr. Boris Caro.*
6. *Mineral reserves are reported using the 2014 CIM Definition Standards.*
7. *Mineral reserves are reported inside stope designs assuming drift-and-fill mining methods, and an economic gold cutoff grade of 0.103 oz Au per ton. The economic cutoff grade estimate utilizes a gold price of $1,275/oz, mining costs of $80/ton processed, surface rehandle costs of $0.16/ton processed, process costs of $30/ton processed, general and administrative costs of $11.11/ton processed, and refining costs of $5/oz Au recovered. Metallurgical recovery is 94.5% for gold. Mining recovery is 95% and mining dilution is assumed to be 10.5%. Mineralization that was either not classified or was assigned to Inferred mineral resources was set to waste. A 1.5% NSR royalty is payable. The reserves reference point is the PFS mill crusher.*
8. *Mineral reserves are included in Measured and Indicated resources; tonnage and contained metal have been rounded to reflect the accuracy of the estimate. Apparent discrepancies are due to rounding.*

15.2 Economic Cutoff Grade Calculation

The economic cutoff grade ("COG") used for stope design is based on initial economic parameters shown in Table 15.2. The calculated gold COG is 0.103 oz Au/ton. Silver was not included in the COG calculation due to its relatively small contribution to total economic value. However, revenue for silver is included in the financial model, and therefore silver grade and silver contained metal are reported in the estimated mineral reserves.

Table 15.2 Mineral Reserve Cutoff Grade Input Parameters

Description	Quantity	Units
UG Mining Costs	$ 80.00	$/ton Processed
Surface Rehandle	$ 0.16	$/ton Processed
Process Costs	$ 30.00	$/ton Processed
G&A Costs	$ 11.11	$/ton Processed
Total Operating Costs	$ 121.27	$/ton Processed
Refining Cost	$ 5.00	$/oz Au Recovered
NSR Royalty	1.5	%
Gold Metal Recovery	94.5	%
Gold Selling Price	$ 1,275	$/oz Au
Reserve Cutoff Grade	**0.103**	**oz Au/ton**

The economic COG was used in the stope optimization to identify the Measured and Indicated blocks above GOG available for consideration to be converted to mineral reserves. The mineral resource COG's (Section 14.8) were applied to internal dilution for Measured and Indicated resources that are below the economic COG.

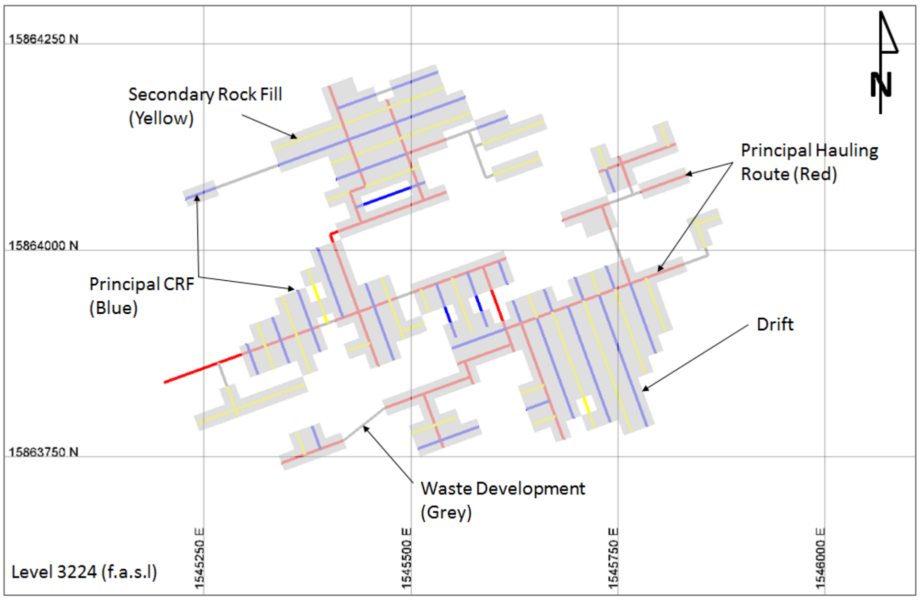
15.3 Mineral Reserve Estimation

The mineral reserves were confined by the design of mineable stope shapes centered on Measured and Indicated blocks with grades greater than the economic COG. For stope optimization, the Stope Optimiser module from Deswik™ software was used. The shapes were developed using 20-foot by 20-foot horizontal and 13-foot high stope-block sizes. Each stope block was queried against the resource block model to determine the tonnages and grades within the stope shapes. Stopes with an average gold grade above the COG were selected to be included in the mine plan and mineral reserves estimate. Some isolated stopes above COG were eliminated from consideration because the development to extract them would cost more than the economic return.

Stope shapes include estimated planned dilution and exclude resource loss where the geometry and grade do not warrant inclusion. The mineral reserve estimate also includes allowances for unplanned dilution (see discussion in Section 15.4).

Development designs were generated concurrently for each stope shape with the purpose of minimizing development in waste. Figure 15.1 shows a typical level design. These designs were done on main levels every 39 feet. The amount of waste development required for the sub-levels (13 feet above and below the main levels) was interpreted based on the required waste development for the main levels.

Figure 15.1 Mine Design, Plan View at Level 3,224



15.4 Dilution

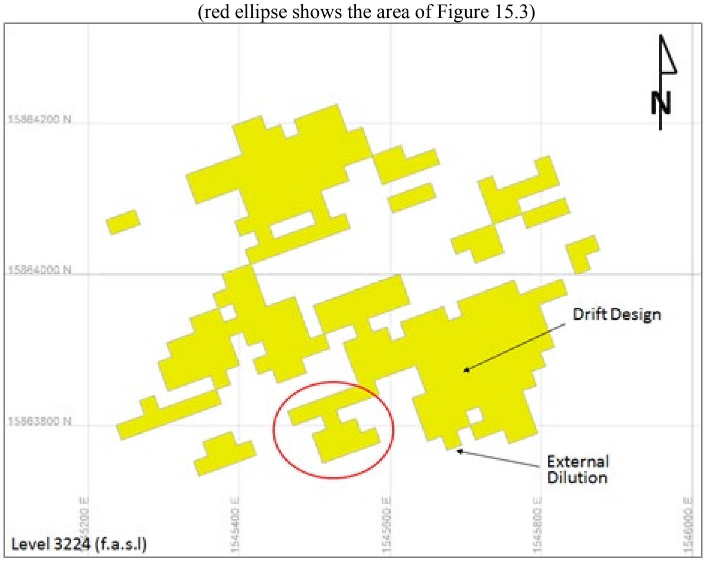
15.4.1 Internal Stope Dilution

Internal dilution is material contained within the mineable stope shapes which is below the economic COG. Material that is classified as Measured or Indicated resources below the economic COG (i.e., the material is above the resource COG) is included with grade. Other material inside of the stopes that is not part of the Measured or Indicated mineral resources was considered to have zero gold and silver grade.

15.4.2 External Stope Dilution

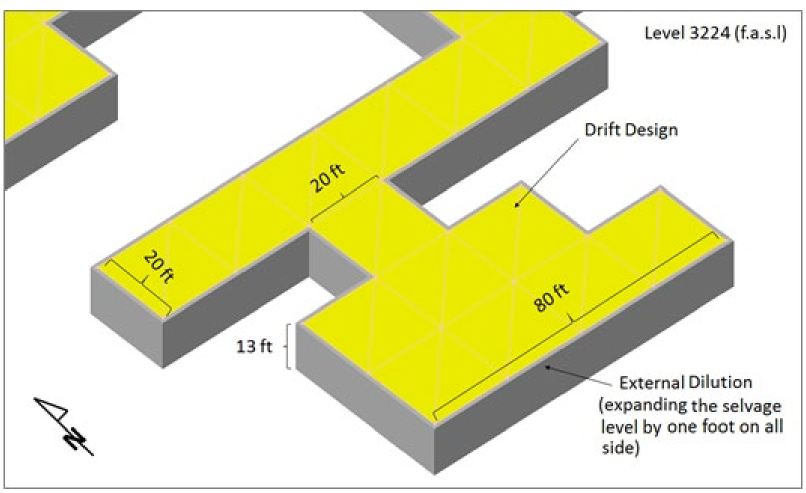
External dilution, corresponding to 6.5%, was estimated by expanding the stope edge by one foot on all sides. The resource model was queried against the expanded volume to determine the appropriate resource grades for silver and gold to be used for this dilution. The external dilution incorporated in the mine plan is 110,000 tons with an average grade of 0.081 oz Au/ton and 0.22 oz Ag/ton. Figure 15.2 shows an example of the planned external dilution on the 3,224 level (approximately 510 feet below the ground surface). The dilution skin is the grey outline surrounding the planned stopes, which are shown in yellow.

Figure 15.2 External Dilution Scheme at Level 3,224
(red ellipse shows the area of Figure 15.3)



The detail in the red ellipse in Figure 15.2 is shown in Figure 15.3. The dilution skin is the grey outline surrounding the planned stopes shown in yellow.

Figure 15.3 External Dilution Detail



15.4.3 Backfill Dilution

Dilution from backfill in stopes will come from the floor when working on top of a prior drift that was backfilled and when mining secondary stopes adjacent to previous stopes already backfilled with cemented rock fill ("CRF"). Mining will be conducted on top of backfill to provide a durable and visible marker horizon to maximize recovery and minimize total costs. Dilution from backfill is estimated to average 4.0% based on similar operations. This dilution is applied to the drifts categorized as secondary in the mining sequence and is in addition to the internal and external dilution estimates. No grades are assumed for the backfill dilution.

15.5 Mining Recovery

Mining recovery is estimated to be 95% based on an assumed ore loss of 5.0%. This is considered appropriate for the high selectivity drift-and-fill mining method selected for the Grassy Mountain deposit (see Section 16.4), and it is based on similar operations in disseminated ore bodies with the same mining method.

15.6 Discussion of Reserves

Mr. Caro is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the mineral reserve estimate.

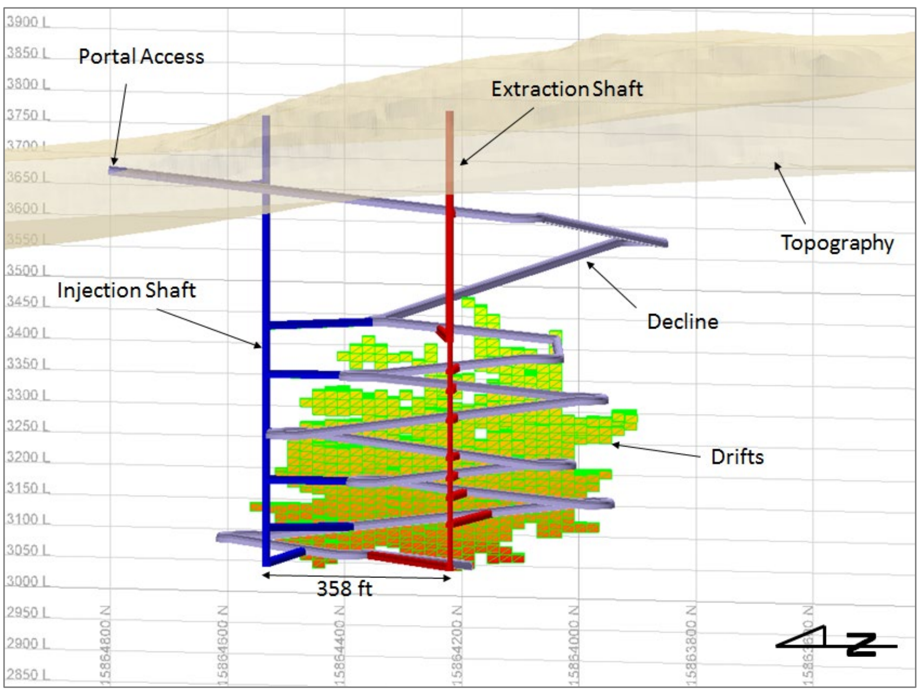
16.0 MINING METHODS

This section was prepared under the supervision of Mr. Boris Caro, an associate of Ausenco. Mr. Caro has reviewed the information used to prepare this section and believes it accurately represents the parameters and procedures used for the proposed PFS mine design.

16.1 Introduction

The Grassy Mountain mine will be accessed via one decline and a system of internal ramps. Two shafts are included in the design to be used for ventilation and secondary egress as shown in the isometric view in Figure 16.1. The planned mining method is drift-and-fill ("D&F"). CRF and rock fill ("RF") will be used for backfill. The planned proportions will be 46% CRF and 54% RF.

Figure 16.1 Isometric View



According with the analysis conducted by Ausenco, the majority of the ground conditions of the Grassy Mountain deposit are classified as being of fair to poor rock quality, and the rock mass rating ("RMR") is typically less than 49. Ground support was designed to maintain a safe operation for these ground conditions.

The mine design was based on a production rate of 1,300 to 1,400 tons per day using four days on and three days off shifts, with two shifts per day, to provide 24-hour coverage during the four operating days during full operation. This will provide sufficient material to feed the 750 tons per day to the mill on a seven day per week basis. The underground production schedule is discussed in Section 16.10 of this report. The nominal development size is 15 feet wide by 15 feet high for the main decline and 13 feet wide by 13 feet high for horizontal access to production areas. Production heading size will be 20 feet wide x 13 feet high. This heading size will allow the miners and associated diesel mining equipment access and flexibility to maximize production from the mine as well as minimize waste haulage from the development headings. The mining cycle involves drilling, blasting, and mucking for the development and production access. The final part of the mining cycle is to backfill the stopes.

The key challenges in attainment of planned production levels and costs are anticipated to be the development of sufficient drift areas, and the interaction between the mining and the backfilling activities. The mine production schedule was created taking into consideration these challenges and two main production sectors were considered for adding more flexibility to the mining operations.

16.2 Geotechnical Analysis and Recommendations

16.2.1 Structural Domains

Rock structures were not assessed because there is no oriented drill-core data currently available for the project. However, observations of the core suggest that there is little systematic structure, except for the very steep features often sub-parallel to core axis that are likely oriented similarly to the interpreted northwest-southeast striking structural set that is associated with mineralization. The remaining structure is typically very small scale, irregular, and generally related to micro-defects within the rock mass.

16.2.2 Golder Geotechnical Appraisal

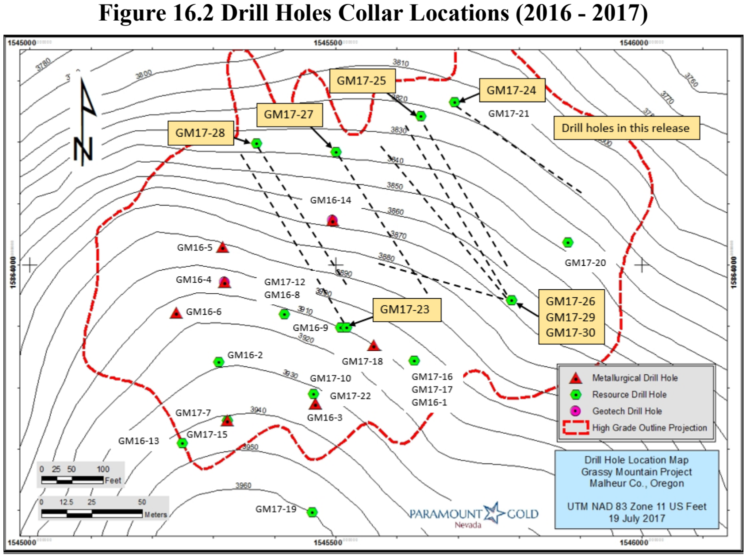
A geotechnical appraisal of the proposed underground mine area was carried out by Golder during 2016-2017 (Golder Associates Inc., 2018). Geotechnical data are available from three different drilling campaigns that were completed prior to the 2016-2017 drill campaign (Table 16.1). Calico, Newmont, and Atlas carried out RQD measurements. Additional geotechnical data from the Newmont and Calico drilling were reviewed, but not used directly in Golder´s 2016-2017 evaluation, due to uncertain reliability and consistency in the data.

Table 16.1 Geotechnical Drill Hole Investigations

Company	Year	Number of Geotechnical Drill Holes	Information Considered	Other Geotechnical Information
Calico	2011	2	RQD	None
Newmont	1992	13	RQD, core photographs in splits	Recovery, fracture, frequency, joint condition rating, hardness, rock strength, underground rock mass ratings (URMR)
Atlas	1986-1992	6	RQD	Recovery, weathering, breakage, hardness, bedding, joints
Paramount	2016-2017	27	RMR	RQD, fractures, ISRM strength rating, weathering index, joint condition rating (JRC)

The 2016-2017 drilling campaign included 27 core holes of HQ3-diameter (2 3/8-inch diameter) drilled using a triple-tube core barrel to maximize core recovery. Two holes were logged in detail for geotechnical characterization by Golder personnel at the drill rig. The other 2016-2017 holes were logged by Paramount personnel according to Golder's instructions and procedures. Figure 16.2 shows the locations of the two Paramount geotechnical hole collar locations within an approximate 0.075 oz Au/ton cutoff boundary.

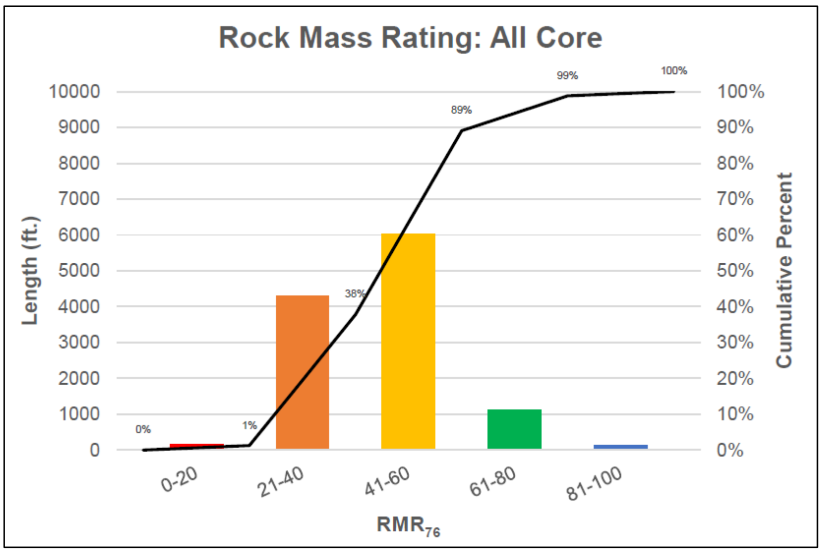
Figure 16.2 Drill Holes Collar Locations (2016 - 2017)



(from Paramount Gold Nevada; 10-foot contour interval.)

Golder utilized the Paramount and Golder geotechnical log data to characterize the orebody and surrounding rock mass based on a calculation of rock mass ratings ("RMR") from the logged data. Figure 16.3 presents the RMR histogram for all core that was geotechnically logged from the 2016-2017 drill campaign. The historical data was not evaluated with the 2016-2017 campaign because the historical logging of RQD data is not comparable with the RMR logging during the 2016-2017 campaign.

Figure 16.3 Golder Rock Mass Rating, All 2016-2017 Core
(from Golder, 2018)



The Golder review of the 2016-2017 drill core indicated the presence of a significant number of zones of broken rock fragments within what Golder termed "a matrix of soil" and referred to as "Soil Matrix Breccia". These zones are more correctly referred to as "clay matrix breccia" as described in detail in Sections 7.4 and 14.3. The clay matrix breccia is readily observed in core in split tubes immediately after drilling, but it is also clearly identifiable after the core has been boxed and somewhat disturbed.

The geological and geotechnical data did not identify any trends or patterns that would allow the delineation of rock quality domains for mine design, with the exception of very poor-quality rock encountered in and around the interpreted subvertical structures. However, very poor-quality rock is not limited to the vicinity of the structures, it is also frequently observed between structures. Therefore, this degree of variability will require a selective mining method that can quickly respond to changing ground conditions.

Golder concluded that, in the absence of spatial patterns in rock quality, three categories of rock quality should be used for PFS level design and cost estimating purposes. Table 16.2 shows the three rock quality categories applied to the design of the Grassy Mountain underground mine workings.

Table 16.2 Rock Quality Categories, Modified from Golder 2017

Rock Quality Category	Description	Approximate Expected Percent of Excavations(a)
Type I	Moderately fractured rock	20%
Type II	Poor quality, highly fractured rock	40%
Type III	Clay matrix breccia and other very poor-quality rock (clay, broken rock and rubble in core boxes)	40% (15% clay matrix breccia, 25% other poor-quality rock)

Note: based on percent encountered within 2016-2017 drill holes.

16.2.3 Ausenco 2017 Geotechnical Work

In 2017, Ausenco's geotechnical group in Santiago, Chile conducted a review of all the available geotechnical information provided by Paramount, including core logs, core photographs, and the work completed by Golder that is summarized above. The main objectives of Ausenco's work were to select a mining method and develop support recommendations for underground openings. While Golder's work was takenen into consideration by Ausenco, Mr. Caro, Ausenco's qualified person, is responsible for the underground mine designs and the geotechnical conclusions and recommendations presented in this PFS.

16.2.3.1 Rock Mass Fabric Domains

The geotechnical holes drilled in the 2016-2017 campaign were not oriented and there was no televiewer information available. Therefore, no rock mass fabric domains have been recognized and they could not be explicitly modeled.

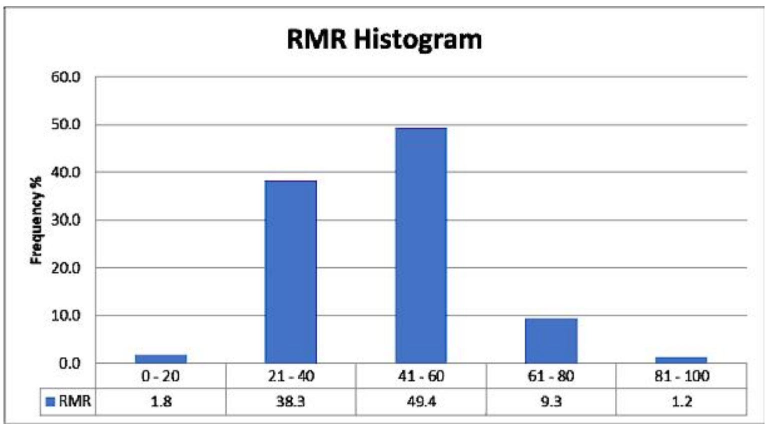
16.2.3.2 Structural Model 3D

For the geotechnical analysis, the orientations of the primary structural zones that are believed to have influenced precious metal distributions in the Grassy Mountain deposit were used to estimate the 3D structural model. For the purposes of this modeling, Ausenco assumed that the major structures are vertical and persist for distances of 100 to 200 feet.

16.2.3.3 Geotechnical Characterization

A statistical analysis was performed by Ausenco on the geotechnical data derived from the core logging of Paramount and Golder. This analysis was performed on each hole as well as the 27 drill holes in aggregate. The RMR results are shown in Figure 16.4.

Figure 16.4 RMR Histogram from 27 Drill Holes



Based on the RMR statistics and Ausenco's interpretation and correlation of the RMR data with the geological database, the Grassy Mountain deposit was assigned to three classes of rocks according to geotechnical quality:

- Class 1: Rocks of Poor geotechnical quality according to RMR; approximately 40% of the deposit.
- Class 2: Rocks of Regular geotechnical quality according to RMR; approximately 50% of the deposit.
- Class 3: Rocks of Good geotechnical quality according to RMR; approximately 10% of the deposit.

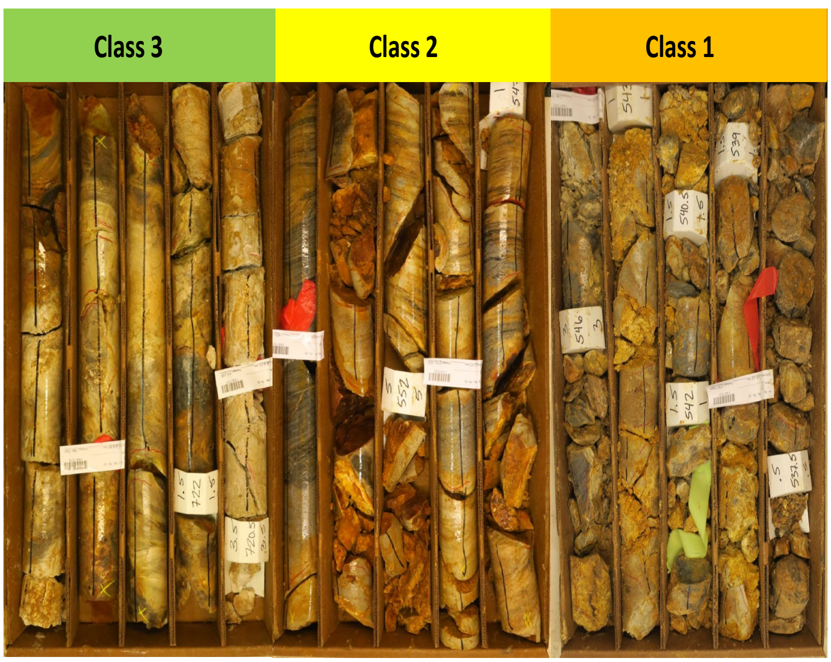
Table 16.3 shows the cumulative frequency values from Figure 16.4 with the classes of rocks assigned by Ausenco.

Table 16.3 Ausenco Rock Quality Categories

Rock Quality (RMR)		Frequency (%)	Rock Class	Deposit (%)
0 - 20	Very Poor	1.8	-	-
20 - 40	Poor	38.3	Class 1	40
40 - 60	Fair	49.4	Class 2	50
60 - 80	Good	9.3	Class 3	10
80 - 100	Very Good	1.2	-	-

The rock qualities of Very Poor and Very Good are not representative of the deposit due to the low frequencies measured, so they were omitted from the three classes of rock assigned. Examples of the three classes are shown in Figure 16.5.

Figure 16.5 Examples of Three Geotechnical Rock Classes



16.2.3.4 Intact Rock Properties Review and Rock Mass Strength

No final results of testing of the intact rock properties or rock mass strength was available for the completion of this PFS, although some testwork was undertaken. This preliminary information was used to check some numerical analyses, but the work will need to be completed as part of additional geotechnical studies required for a Feasibility Study.

16.2.4 Summary of Geotechnical Analysis and Evaluation for Underground Mining

Mr. Caro believes the available geotechnical data are adequate for designing the mine openings associated with the estimation of the Grassy Mountain mineral reserves at a PFS level, but further geotechnical work will be required to complete a Feasibility Study. Risks associated with the current level of geotechnical analysis are discussed in Section 25.11.1, and recommendations for additional work are presented in Section 26.2.4.

While the rock quality is variable, and the deposit is mineable based on the chosen mining method, care will need to be taken during the execution of the mining plan. The selected mining method and underground support recommendations are specified in Sections 16.4 and 16.5.

16.3 Hydrogeology Analysis

This section is based on work completed by SPF Water Engineering of Boise, Idaho ("SPF") as part of the overall PFS. This summary, and the included references, are taken from Clark et al., 2018.

Groundwater flow, expressed as a potentiometric surface, appears to follow topography, from areas of higher to lower surface elevation. The groundwater flow direction is predominantly to the northwest and more or less continuous in the vicinity of the Grassy Mountain deposit.

The horizontal hydraulic gradient is relatively high and somewhat uniform in higher relief areas, reflecting the steep topography, recharge areas, and predominantly low permeability of subsurface deposits. The gradient is lower in areas coinciding with flatter topography and groundwater discharge.

Despite the local variations in groundwater flow and aquifer properties, the apparent, aggregate aquifer system on a more regional scale appears to be relatively consistent. Local discontinuities resulting from fault and/or fracture zones, lithologic facies changes, or some combination of these influences are expressed as local compartmentalization and variations in the groundwater elevation. This concept is supported by groundwater-level monitoring performed on a seasonal basis for a period of several years, suggesting relatively stable trends over time. Aquifer pumping tests also support the general concept of localized zones of higher versus lower permeability. This trend is based on well yields during short-duration pumping (i.e., over a period of a few days) that typically cannot support sustained pumping rates more a few tens of gallons per minute, combined with apparent negative boundary conditions (i.e., associated with lower-permeable deposits and/or faults that limit groundwater flow).

Assignment of discrete aquifers is typically based on several criteria, with the designation being somewhat subjective and relative as a function of scale:

- Physical separation of higher-permeable deposits (aquifers) and lower-permeable deposits (aquitards);
- Hydraulic communication (i.e., similar groundwater elevation trends); and
- Similar lithology, water quality, and geochemical characteristics.

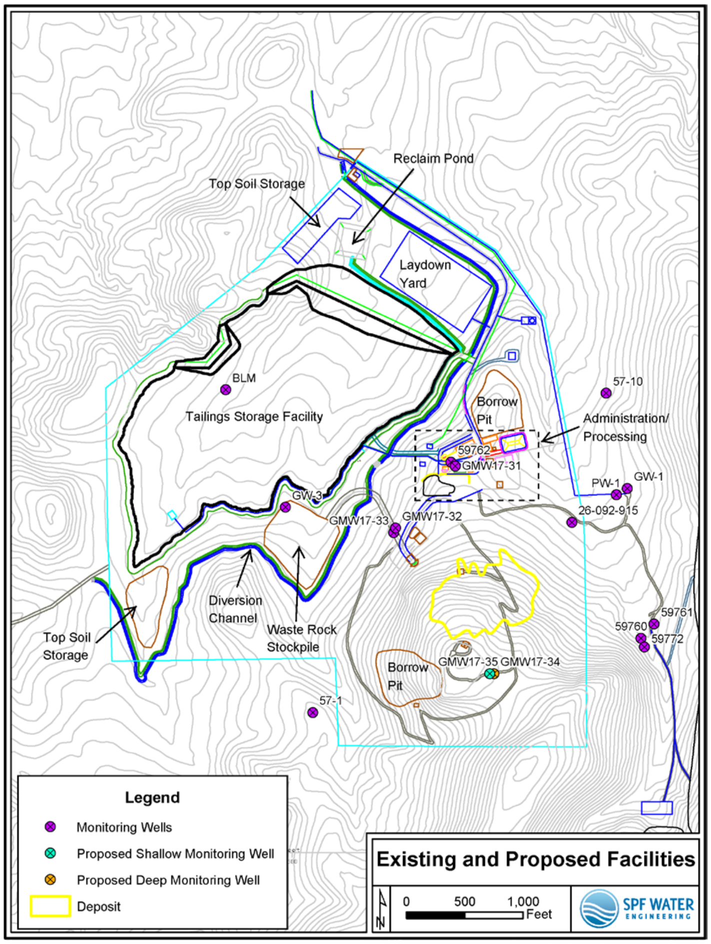
The concept of a single aquifer system has been supported by previous investigations (JMM, 1991; ABC, 1992; SPF, 2016). The overall hydraulic connectivity based on historical and recent water-level and water-quality data has supported a single, heterogeneous, and complex shallow aquifer system. This system contains discrete water-bearing zones that are laterally discontinuous, exhibited by regions with lower permeability (i.e., clay and siltstone, competent bedrock, and silicified deposits) and structural barriers to groundwater flow (SPF, 2016).

The presence of a deeper, regional aquifer has been contemplated based on the measured groundwater elevation over time in well 59762 (approximately 3,100 feet above mean sea level), compared to wells with groundwater elevations of 3,200 feet or higher (SPF, 2016). Well 59762 was completed to approximately 700 feet below ground surface, deeper than other wells installed in the deposit vicinity (Figure 16.6). In terms of elevation, however, other hydraulically down-gradient wells are actually screened over a comparable or lower elevation (i.e., GW-4, PW-4, Prod-1, and GW-6), but exhibit higher groundwater elevations. Since well 59762 was installed, one additional deep well (GMW-17-32) was installed down-gradient from the deposit in 2017, and several deep vibrating wire piezometers ("VWP"s) have been installed directly within the Grassy Mountain deposit (SPF 2017). Those installations have similar, deeper groundwater elevation trends to well 59762 (i.e., 3,000 to 3,100 feet above mean sea level range), supporting the concept of a deeper zone within the aquifer system. The current VWP installations, combined with deep wells, indicate that the potential exists to encounter groundwater within the deposit area both above and below the target dewatering elevation of approximately 3,100 feet, with lower static groundwater elevations found in water-bearing zones at depths of more than approximately 500 feet below ground surface.

The overall direction of groundwater flow and hydraulic gradients are similar for shallow and deeper well completions and are suggestive of a strong topographic influence. The geochemistry of shallow versus deep wells is similar, without a clear distinction based strictly on vertical or depth trends. Therefore, the concept of various zones (shallow and deep hydraulic expression) within the regional aquifer appears to be supported with the available data.

Theoretical groundwater inflow rates into the Grassy Mountain deposit area are on the order of 20 gpm to 100 gpm for sustained pumping, and 250 to 500 gpm for short-duration pumping and reflect the wide span of aquifer parameters and model assumptions utilized for predictive analyses. Actual inflow rates of several tens to a few hundred gpm are anticipated based on median aquifer parameters and model assumptions. However, based on drilling observations within the deposit, and aquifer testing performed outside the deposit area to date, the higher-end range of potential inflow rates associated with higher hydraulic conductivity are unlikely to be encountered during mining activities and, if encountered, the associated high dewatering rates would be anticipated for relatively short durations (i.e., likely on the order of days or weeks). Due to the proposed underground mining approach, the entire groundwater table will not be intercepted at once. Rather, the exposure to groundwater is anticipated to be restricted to subsurface workings that encounter groundwater, if present, such that inflow can be managed or mitigated as the conditions vary.

Figure 16.6 Water Well Locations in Relation to Proposed Infrastructure

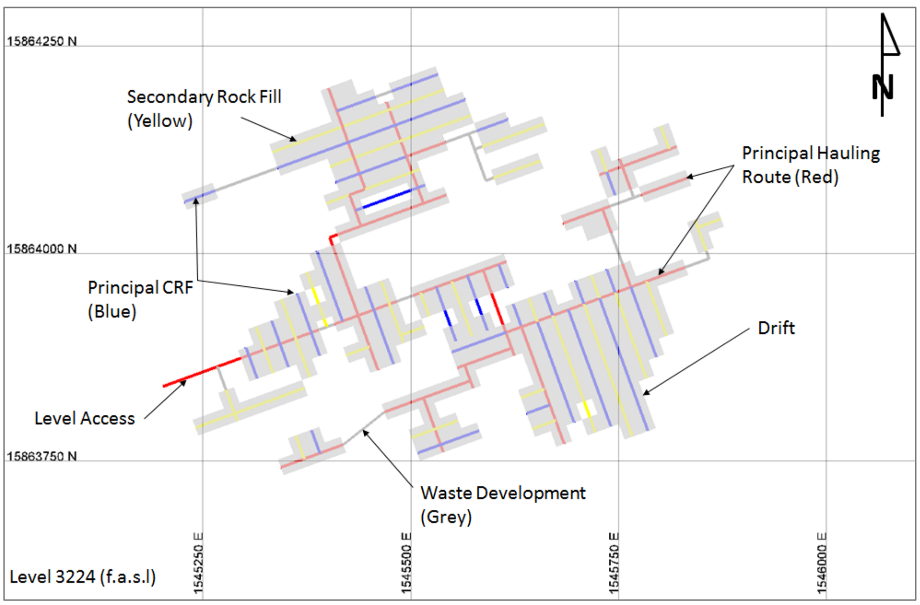


The lower range of inflow rates represents longer-term predicted dewatering as steady-state conditions are approached and reflects lower overall permeability of the aquifer system over a greater area (and likely within the deposit). The higher inflow rates reflect shorter-duration flow rates resulting from dewatering of zones with higher permeability that appear to be laterally discontinuous throughout the area based on borehole drilling and aquifer testing. Based on borehole drilling and aquifer testing performed to date in the vicinity of the orebody, higher permeability areas are thought to more likely be encountered away from the silicified orebody, such as to the north of the deposit and in basin areas. These would be areas characterized by greater amounts of relatively unaltered sedimentary rocks as compared to silicified and/or competent bedrock. Direct testing of aquifer properties within the Grassy Mountain deposit has not been performed to date as verification, but extensive anecdotal evidence from mineral exploration drilling supports the concept of low permeability within the near vicinity of the deposit.

16.4 Mining Methods

The D&F mining method was selected using the methodology proposed by Nicholas (1981), where the geometry and the geotechnical conditions of the deposit are assessed. The D&F method is highly flexible and can achieve high recovery rates in deposits with complex and flat-dipping geometries, as is the case at the Grassy Mountain deposit. Figure 16.7 shows the typical D&F layout proposed for the deposit at the 3224 level.

Figure 16.7 Proposed Drift and Fill Design for the 3224 Level



The maximum D&F dimensions were defined to ensure underground stability, based on the geotechnical conditioned discussed in Section 16.2. These dimensions were estimated using the methodology proposed by Mathews (1981), which considers the hydraulic ratio of the drift and the geology and geotechnical conditions of the deposit.

16.5 Mine Design

The Grassy Mountain orebody will be accessed using a 15 x 15 ft main decline, developed from a portal on surface. The decline will provide the connection to all services. The design intent is to have the decline located as close as possible to the mineralization in order to reduce transportation costs, but sufficiently removed from mining activities to ensure that the decline is geotechnically stable for the planned life-of-mine ("LOM").

A summary of the mine design criteria is shown in Table 16.4.

Table 16.4 Underground Mine Design Criteria

Development Heading Parameters - Horizontal/Incline/Decline	Width (ft)	Height (ft)	Diameter (ft)	Length (ft)	Maximum Gradient (%)
Decline	15	15		varies	12
Level access	13	13		varies	15
Stope	20	13		varies	
Center-line radius of curvature - internal ramps				32.8	
Raise boring			13.12		
Conventional Shaft			19.68		

16.5.1 Access

The main access portal will be located on surface close to the process plant infrastructure. Figure 16.8 shows the location and configuration of the main portal access. Figure 18.1 shows the portal in relation to other facility locations on the mine site.

The main decline ramp will be approximately 100 feet in stand-off distance from the orebody. This distance will allow sufficient space between the decline ramp and the orebody for the excavation of cross levels and access to drift levels. The decline ramp will have dimensions of 15 feet in width by 15 feet in height, and it will be developed with a maximum 12% gradient. This gradient is commonly used in modern underground mines and is within the operating limits of the haul trucks that will be used. Figure 16.9 shows the planned design for the main access portal.

Figure 16.8 Location of Main Portal Access

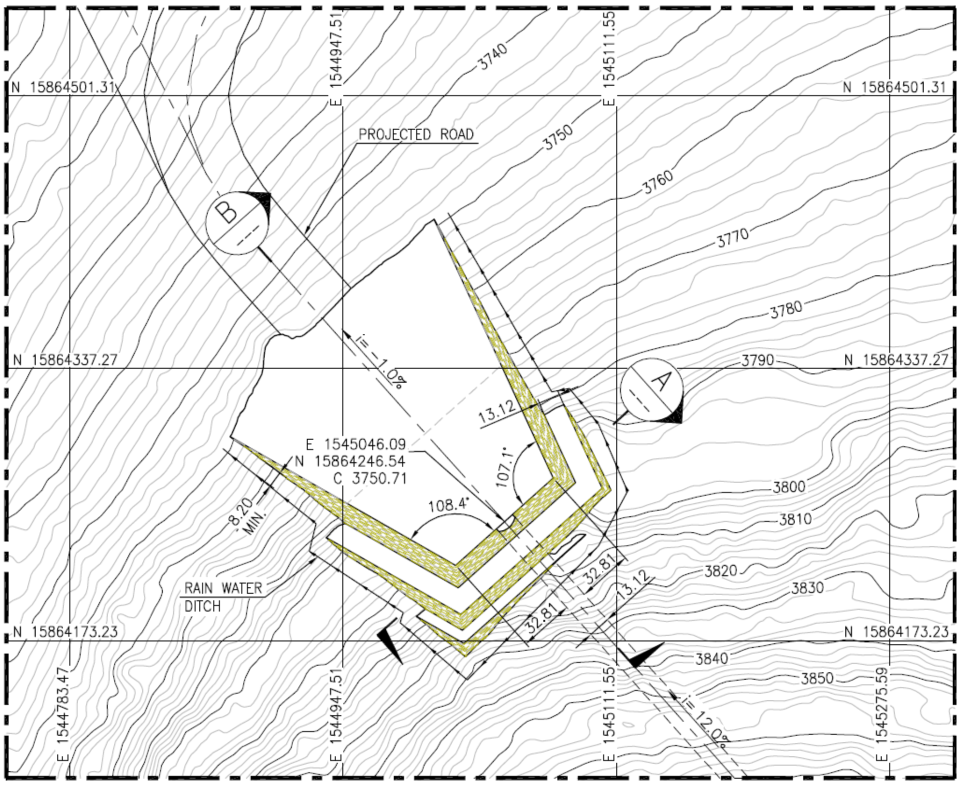
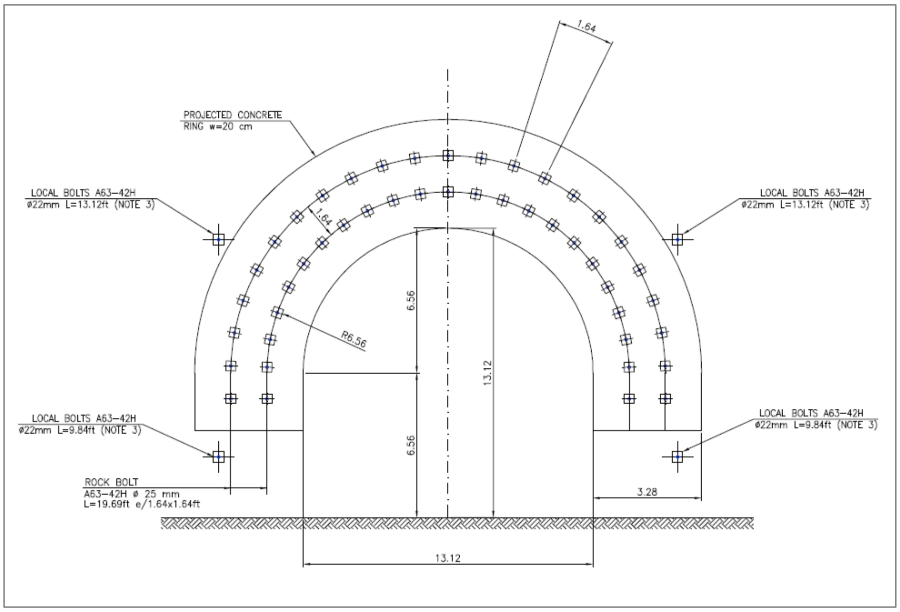


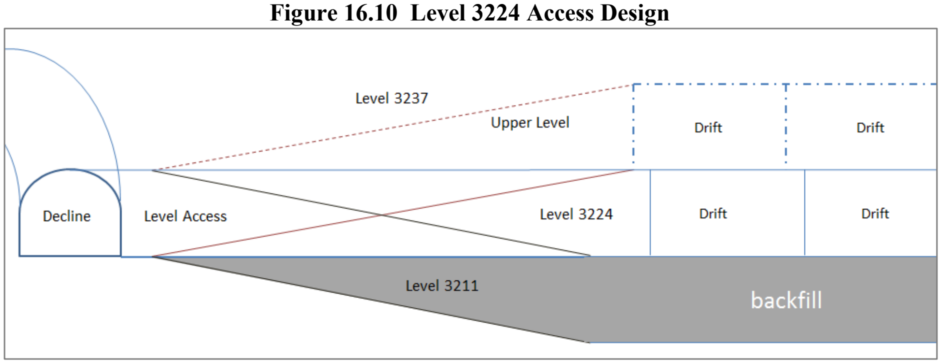
Figure 16.9 Main Portal Design



16.5.2 Level Access Design

Each level will be mined from the decline ramp via a "level access" excavation. This excavation has been designed to be 13 feet wide and 13 feet high to provide clearance for the trucks that will be used to haul material from the levels mined. Each level access will connect with three production levels. Figure 16.10 shows the schematic level access and development for the 3224 level.

Figure 16.10 Level 3224 Access Design



16.6 Mining Services

16.6.1 Mine Ventilation

The ventilation network was designed to comply with U.S ventilation standards for underground mines [Code of Federal Regulations / Title 30. Underground metal and nonmetal mines. Washington, DC: U.S. Government Printing Office, Office of the Federal Register]. Regulatory concentrations for gases are specified by the 1973 American Conference of Industrial Hygienists ("ACGIH") threshold limit values ("TLV"s) [71 Fed. Reg. 3 28924 (2006)]. For diesel particular matter ("DPM"), a permissible exposure limit ("PEL") of 160 µg/m^3 total carbon is specified in the U.S. diesel rule for metal/nonmetal mines [71 Fed. Reg. 28924 (2006)].

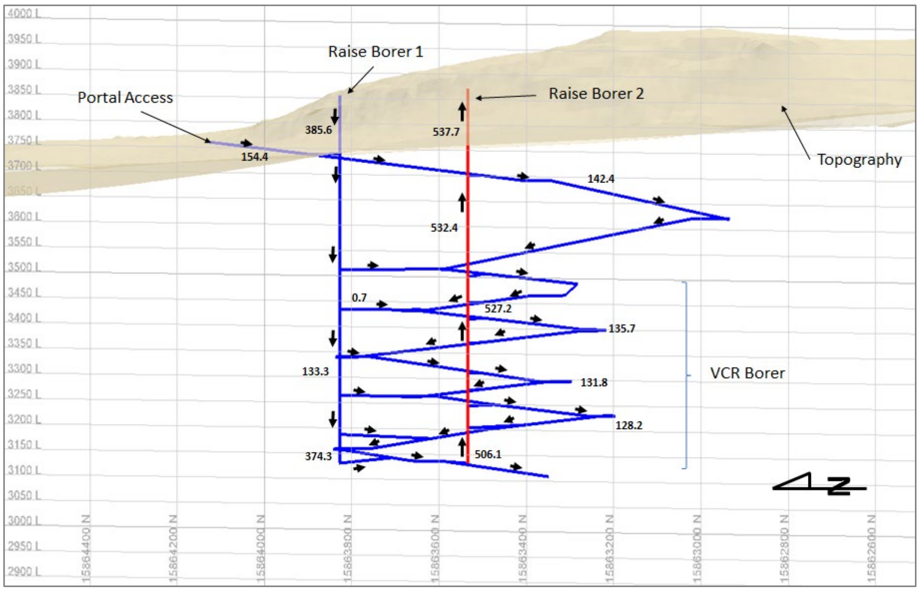
Mine Safety and Health Administration ("MSHA") sets an airflow requirement for the dilution of gas emissions, and an additional airflow requirement for dilution of DPM. These values are published with the list of approved engines on MSHA's internet website. Airflow of 54,000 ft^3/min was selected as a minimum reference for the ventilation design in order to meet the MSHA ventilation standards.

Required airflows were determined at multiple stages during the mine life, using equipment numbers and utilization rates, specific engine types and exhaust output, and the number of personnel expected to be working underground. The designed ventilation system includes the following parameters:

- Required air flow of 540,000 ft^3/min;
- Fan total pressure ("FTP") of 7 inches Hg; and
- Air density of 0.071 lb/ft^3

The planned ventilation will use a push/pull system and will require extraction fans on surface. An extraction vent raise, with dimensions of 19.7 feet x 19.7 feet and 367 feet in length, will connect the lowest level of the mine (3,123-foot level) with the 3,490 level. A raise borer vent raise of 18.58 feet in diameter and 373 feet in length will connect the 3,490 level with the surface (3,863 level), thereby forcing the air flow into the main extraction circuit. Figure 16.11 shows the main components of the proposed ventilation network with inflow and exhaust air-flow directions.

Figure 16.11 Ventilation Network



16.6.2 Underground Water Supply

Water will be needed for underground production drilling and bolting, as well as for diamond drilling. The required LOM water supply has been estimated based on the mine-equipment requirements as summarized in Table 16.5.

Table 16.5 Estimated LOM Water Requirement

Equipment	Maximum	Water Pump (gal/min)	Operational Factors	Water required (gal/min)
Jumbo DD321	2	26.4	70%	37.0
Bolter DS311	1	8.7	70%	6.1
Diamond Drill	1	20	70%	14.0
				gal/min
			Total Required	57.1
			Factor	20%
			Total with factor	68.5

16.6.3 Dewatering

Dewatering is planned to a target elevation of approximately 3,100 feet, from approximately 500 to 700 feet beneath the surface, as per the expected hydrologic conditions (Section 16.3). The active mining area is projected to be about 600 feet by 900 feet in aerial extent.

Dewatering assumptions include:

- Estimated steady-state, bulk-dewatering rates on the order of 20 gpm, with the potential to intercept up to 500 gpm on a short-duration basis (i.e., days to weeks); anticipated based on the PFS-level assessment (Section 16.3).
    - The low-end estimate reflects lower permeability, in the range of 0.003 ft/d, anticipated directly within the deposit area. Due to the expression of individual faults or fault zones, the actual permeability may be more or less. Ausenco is not aware of direct testing of hydraulic conductivity or transmissivity based on aquifer pumping tests within the deposit area to date to confirm this estimate. However, the results of pumping tests performed around the perimeter of the deposit support an aggregate lower hydraulic conductivity within this magnitude due to limited yields and negative boundary conditions.
    - The high-end estimate reflects higher hydraulic conductivity that may be more representative of basin conditions and short-duration inflows into the deposit area that could potentially be intercepted during the mining activities. The anticipated hydraulic conductivity may be on the order of $1x10^{-4}$ cm/s (0.3 ft/d). This condition may arise from contributions from local zones of higher permeability that are effectively dewatered early in the mining process. As the cone of depression or radius of influence extends from the theoretical pumping well(s), the overall aquifer properties are expected to produce less water over time due to overall lower permeability effects.
- The conceptual model for groundwater flow at Grassy Mountain provided the basis for the dewatering estimates. The current model suggests a single aquifer system as a function of scale, supported by the relatively uniform, shallow and deep potentiometric surface and correlation with groundwater elevation and depth. On a local scale, heterogeneity effects are apparent, attributed to local variations in hydraulic properties, facies changes, and/or the occurrence of faults/fault zones.
- Dewatering was simulated by placing theoretical wells along the deposit perimeter and assigning uniform pumping rates to achieve dewatering to the 3,100-foot elevation. Four wells were simulated at five-gpm each, for 20-gpm total pumping requirements under steady-state conditions, resulting in a pumping-level elevation of approximately 2,950 feet to 3,050 feet.
- The dewatering evaluation also examined potential groundwater inflow rates using a combination of steady-state and transient analytical methods.
    - A groundwater flow rate of approximately 20 gpm was predicted in the steady-state analytical model. This value was consistent with the numerical model results based on an assumed, uniform hydraulic conductivity of 0.003 ft/d. The model was sensitive to changes in hydraulic conductivity by half an order in magnitude, with corresponding increases in estimated dewatering by one order in magnitude. For example, assignment of hydraulic

conductivity of 0.3 ft/d resulted in an estimated dewatering rate of approximately 500 gpm. This model assumes steady-state conditions. However, based on drilling observations within the deposit and aquifer testing performed outside the deposit area to date, these higher hydraulic conductivity zones are unlikely to be encountered during mining activities. If encountered, the associated high dewatering rates would be anticipated for relatively short durations, likely on the order of days or weeks.

  - The transient analytical method was used to estimate the predicted dewatering rate of approximately 250 gpm to 600 gpm, assuming a single pumping well scenario, placed at the center of the deposit. The theoretical drawdown effects at the perimeter of the deposit were evaluated after one year of continuous pumping to produce 600 feet of drawdown (assuming an initial groundwater elevation of 3,700 feet for up-gradient conditions and an assumed dewatering elevation of 3,100 feet). The higher flow rate range is consistent with anticipated short-duration inflow amounts over the span of days to weeks.

- The dewatering estimates reflect inherent uncertainty, both in the available datasets and necessary simplifying assumptions for representing a complex system. However, these results are considered appropriate for PFS-level mine planning.
- The construction of special stations for dewatering is a planned part of the mine development. These stations will have dimensions of 32.8 feet in length by 13 feet in height, by 13 feet in width, with a slope of 12%.

16.6.4 Electrical Distribution

An underground 480 V transformer will be placed near the entrance to the portal at the start of mining. This will supply power to electrical equipment used to develop the main decline and to portable fans. A main power line will be installed along the rib of the decline to carry 1.4 kV when development has advanced far enough that carrying power at 480 volts becomes too inefficient. This line will be connected to a transformer that will be moved underground. Line power will also be extended to the locations of the two ventilation shafts to supply power to the ventilation fans.

Upon completion of the decline to the 3224 level, and commencement of mine production activities, a second underground transformer will be purchased for use in the lower areas of the mine.

16.6.5 Mine Communications

Inside the mine, a leaky-feeder VHF radio system will be used as the primary means of communication. The system will allow for communications between the underground mine and surface operations.

16.6.6 Refuge Station

Two emergency refuge stations are considered to be necessary in case of fire or rockfalls that would block access and prevent full evacuation of personnel. These refuges will allow the staff to remain safe in the underground mine for 48 hours. The refuges are mobile, each can accommodate up to 20 people within the protected chamber, and they will be arranged so that they are always no more than 650 feet from the areas where the mine operation personnel are located. Figure 16.12 shows an example of a refuge station.

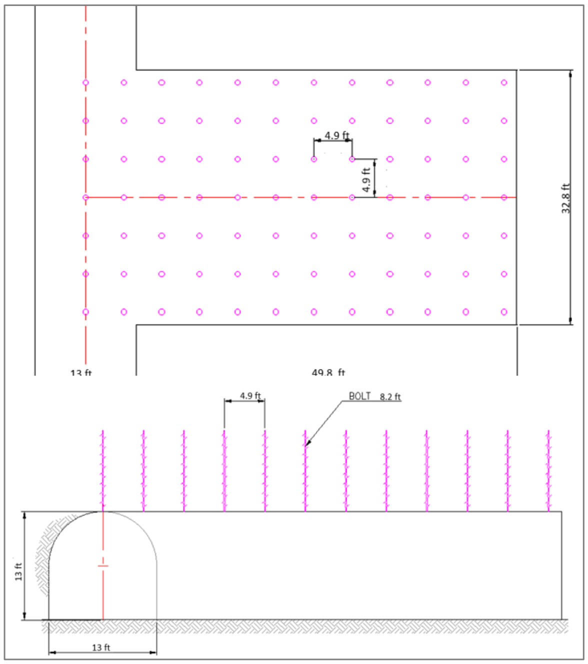
Figure 16.12 Typical Mobile Refuge Station

16.6.7 Maintenance Facilities

Stations will be developed for maintenance of underground equipment without the equipment having to return to surface. Two maintenance stations will be constructed during the LOM measuring 32.8 feet wide, 49.8 feet long, and 13 feet high and will be equipped with tools appropriate for minor repairs and maintenance only.

The underground maintenance stations will be located close to the decline ramp on the 3510 and 3237 levels. Figure 16.13 shows the planned configuration of the maintenance stations.

Figure 16.13 Underground Maintenance Station



16.7 Underground Mining Operations

16.7.1 Drilling

Production and development drilling will be done using electric-hydraulic development-drill jumbos. Twin-boom jumbos are planned for large-dimension development rounds. Bolting machines will provide for ground support installation.

Drilling productivities were built up from first principles and vary by heading dimensions. The critical time path to cycle and advance a heading is the jumbo drilling time. To meet production requirements, the nominal 20 feet wide by 13 feet high production headings will require a double-boom jumbo drilling 54 holes that are each 1¾ inch diameter and 12 feet deep. Each boom is expected to have a penetration rate of 3.8 ft/min and will require 180 minutes to complete a single cycle using the two booms.

16.7.2 Blasting

Local contractors will perform blasting services. Emulsion will be used for most production blasting and development rounds. Boosters, primers, detonators, detonation cord, and other ancillary blasting supplies will also be required. Bulk explosives will be stored in a secure powder magazine on surface in accordance with current applicable explosives regulations.

Blasting will occur at designated times using a centralized blasting system. Where ventilation allows, multi-blasting of isolated high-priority development headings is anticipated.

Once the jumbo drill has completed the drilling cycle, the emulsion blasting agent will be loaded into the holes with the respective nonel blasting cap and booster. The timing of the round with the nonel caps is extremely important as it is critical to pulling the maximum amount of distance per round.

For decline development with a 15-foot by 15-foot profile, an estimated 591 pounds (268 kg) of emulsion is required for each round. The powder factor will be 3.90 lbs/ton (1.95 kg/t) assuming 164 tons of material movement per round. For level access development, an estimated 507 pounds (230 kg) of emulsion will be required for each round. A powder factor of 3.86 lbs/ton (1.93 kg/t) is estimated for level-access development assuming 129 tons of material will be moved per round. For drift production development with a 20-foot by 20-foot, an estimated 617 pounds (280 kg) of emulsion will be required for each round. The powder factor will be 3.14 lbs/ton (1.57 kg/t) assuming 194 tons of ore will be moved per round.

16.7.3 Ground Support

Ground support will be installed with specifications based on the geotechnical analysis discussed in Section 16.2. The support analysis was carried out using empirical techniques based on recommendations from Barton et al. (1974) and Barton (2002). The empirical design used the support abacus approach, which relates the rock mass quality (Q) to an equivalent dimension (De). The Q value was obtained from a range of values that define each rock class, derived from Gonzalez de Vallejo (2004) as follows:

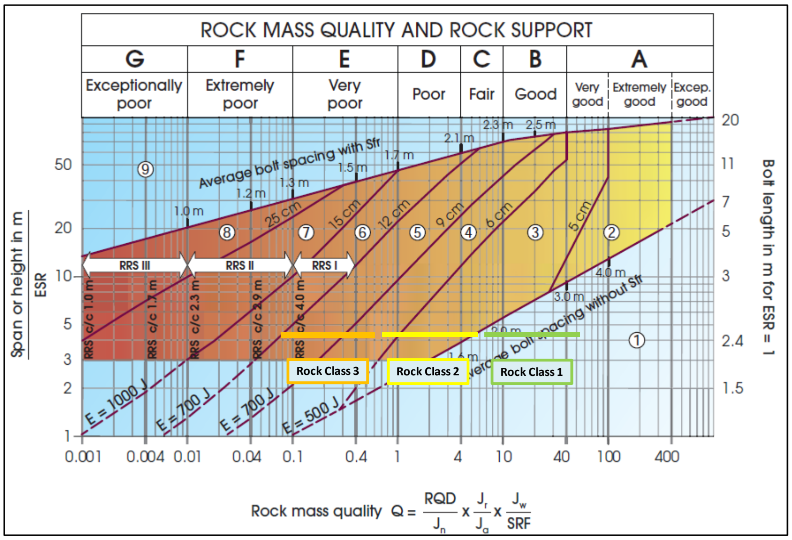
$$RMR = 9 * Ln\ (Q) + 44$$

These values use the RMR and Q values summarized in Table 16.6 that are based on the three rock classes shown in **Figure 16.14**.

Table 16.6 Interpreted Relationship between RMR and Q

Rock Class	$RMR_{B'89}$		Q Barton '74	
	Min	Max	Min	Max
Class 1	21	40	0.08	0.64
Class 2	41	60	0.72	5.92
Class 3	61	80	6.61	54.60

Figure 16.14 Rock Mass Quality and Rock Support



The De value is obtained by dividing the size of the excavation by the excavation support ratio ("ESR"), which is relative to the intended use of the excavation and the required factor of safety ("FOS"), and based on the following empirical relationship:

$$De = \frac{Excavation\ Span, diameter\ or\ height\ (ft)}{Excavation\ Support\ Ratio\ ESR}$$

An ESR = 1.0 was selected as a conservative parameter for the *De* calculation.

Dimensions for the decline, level access, and drifts were calculated using the following equation:

$$De = \frac{15\ (ft)}{1.0} = 15$$

- Decline: 15 x 15 feet;
- Level Access: 13 x 13 feet; and
- Drifts: 20 x 13 feet.

The decline ramp 15-foot by 15-foot dimension was used as maximum excavation span, which is considered for permanent infrastructure. The other infrastructure is considered to be temporary. The recommended support is shown in Table 16.7 using the information summarized in **Figure 16.14**.

Table 16.7 Support Recommended (based on Barton, 2002)

Rock Class	Q	Value	Support Pattern (ft)			Shotcrete (ft)
Class 1	0.08	Min	4.3	x	4.3	0.2 – 0.3
	0.64	Max	5.2	x	5.2	
Class 2	0.72	Min	5.6	x	5.6	0.16 – 0.2
	5.92	Max	6.9	x	6.9	
Class 3	6,61	Min	5.9	x	5.9	Occasional
	54,60	Max	10.8	x	10.8	

The support pattern and shotcrete thickness for the decline ramp should be assigned according to the rock class indicated in Table 16.7. The geotechnical stability analysis also indicates that the support pattern and shotcrete thickness for the level access and drifts will be associated to rock class 2 and rock class 1, respectively.

The length of the bolts ("Lb") as proposed by Barton (1980) was estimated using the following empirical relationship:

$$Lb = 2 + 0.15 * \frac{D}{ESR}$$

where the D is the maximum span dimension for each access and ESR = 1.0.

Table 16.8 shows the rock bolt length calculated for the decline ramp, level access, and drift sections.

Table 16.8 Estimated Rock Bolt Lengths

Infrastructure	Section (ft)	Bolts Length (ft)
Decline Ramp	15 x 15	8.8
Level Access	13 x 13	8.5
Drift	20 x 13	8.5

The installation of the advance support as a function of the distance to the excavation front has been evaluated using the empirical relationship as follows:

$$Max\ Span\ Unsupported = 2 * ESR * Q\text{^}0.4$$

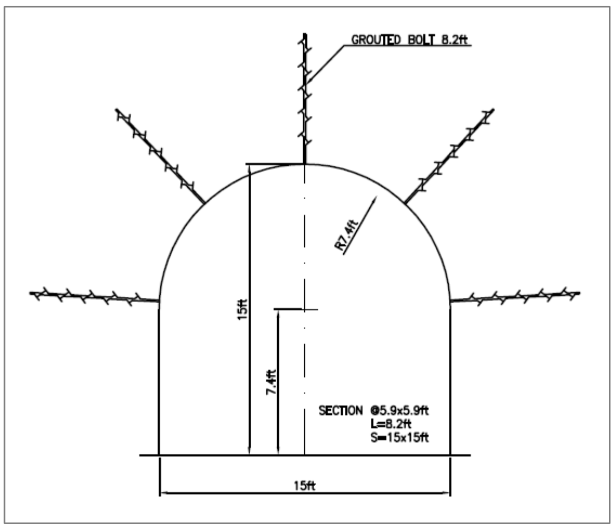
Table 16.9 shows the maximum distance to the excavation front by rock class.

Table 16.9 Maximum Unsupported Span

Rock Class	Q	Range	Maximum Span (Unsupported) (ft)
Class 1	0.08	Min	2.3
	0.64	Max	5.6
Class 2	0.72	Min	5.9
	5.92	Max	13.5
Class 3	6.61	Min	14.0
	54.60	Max	32.5

An example of ground support design for the main decline in Grassy Mountain is shown in a cross-section view in **Figure 16.15**.

Figure 16.15 Cross-Section of Support for Decline (Rock Class 3)

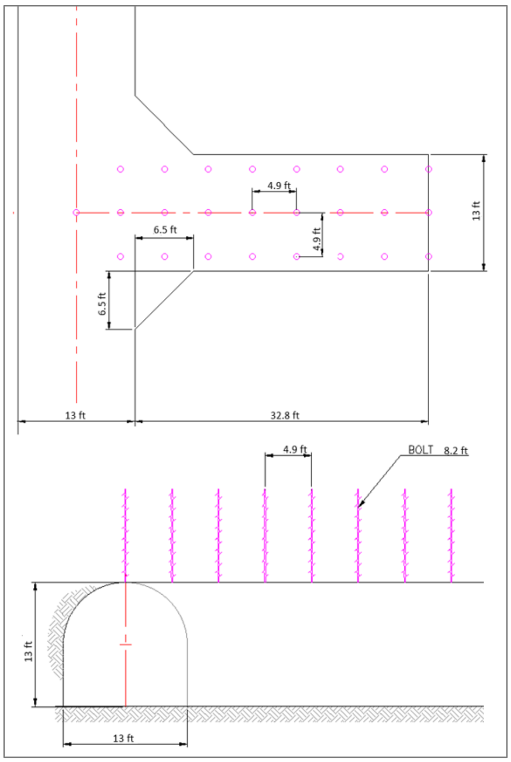


16.7.4 Mucking

Load-haul-dump vehicles ("LHD"s) with a nominal 5.2 yd^3 bucket capacity will be used for primary ramp development and excavation of level accesses, drifts, and footwall drives. Backfill placement will also be done using the 5.2 yd^3 LHDs.

The LHDs will be used to load the underground mining trucks using the same main access for each level. Material will then be hauled by the trucks up the main ramp to the surface and dumped at the ore stockpile or waste dump as appropriate. A front-end loader will feed the ore from the stockpile into the primary crusher. Where necessary, a small area of the ramp's back will be excavated to make sufficient room for loading operations as shown in Figure 16.16.

Figure 16.16 Typical Loading Bay

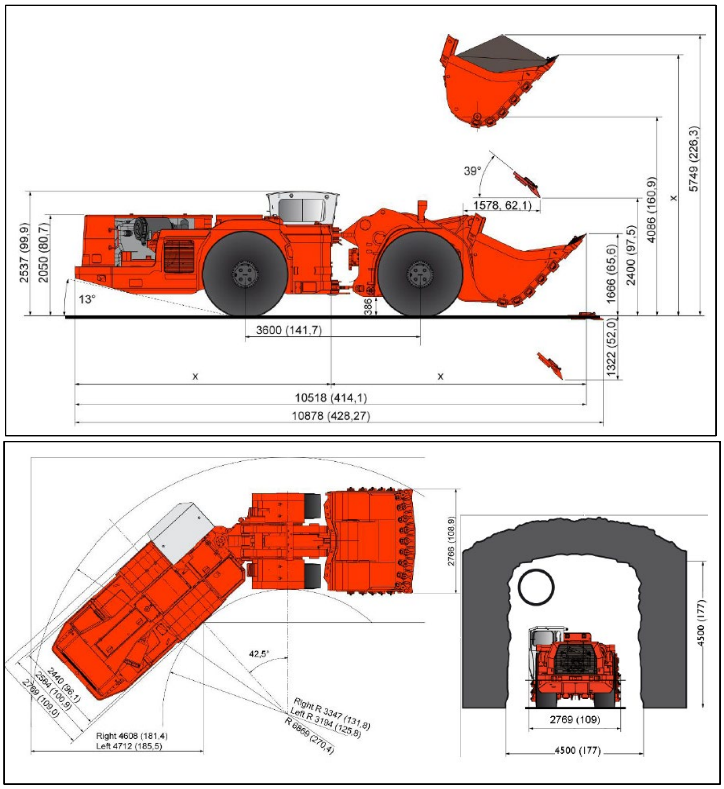


16.7.5 Hauling

The planned haulage will use conventional low-profile underground-mining trucks of 22.9 yd^3 capacity (Figure 16.17). For all mining levels, trucks loaded underground will transport the ore and waste directly to surface. Once unloaded on the surface, the truck capacities will be used to transport backfill materials on their return trips into the mine.

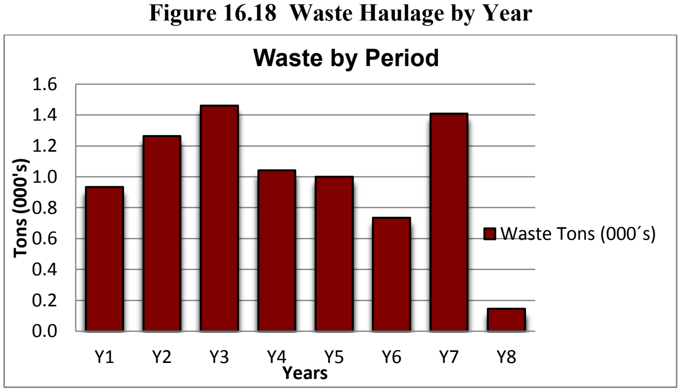
Ore that is hauled to surface as part of mine scheduling will be placed in the crusher stockpile. Waste rock hauled to surface will be deposited at a waste-rock facility that will be located about 1,640 feet from the main portal. The tonnage of waste hauled to surface over the LOM is summarized in **Figure 16.18**. This waste will be fully utilized over the mine life as cemented rock-fill material, reducing the total amount of borrow material required over the mine life.

Figure 16.17 Underground Mine Truck Examples



Note: dimensions given in millimeters with inches in parentheses; commas represent decimal points.

Figure 16.18 Waste Haulage by Year



16.7.6 Backfill

The backfill method has been selected according to the geological and geotechnical conditions of the deposit, as well as the selected D&F mining method. The main objectives of the backfill are to provide stability to the drifts and to control dilution associated with the ore extraction. CRF and RF will be used in the primary and secondary drifts, respectively.

To the extent possible, the waste rock from underground operations will be used for CRF and rock from the borrow pit will be used for RF. A plant to produce the CRF will be built as part of the project infrastructure (see Section 18.0).

16.7.6.1 Cemented Rock Fill

CRF will be used to backfill primary drifts allowing for reasonable recovery of secondary drifts. The CRF will have the following properties:

- Cement: 5.0%;
- Water / Cement (ratio): 0.8 to 1.2;
- Waste Rock: 70% – 98% (rock with good geotechnical rating); and
- Granulometry size: -6 inches.

It is assumed that the cement will properly encapsulate any potentially acid-generating material. Thus, the mine waste will be used as available. This will reduce the mine waste storage to zero over the LOM. When mine waste is not available, rock from the borrow pit on the east side of the project will be utilized for CRF.

The CRF plant will be located near the portal. Haul trucks will be used to haul the CRF down the decline and into the locations to be used. LHDs will have special "jamming plates" attached to the bucket so that the CRF can be jammed as tight as possible. Control of the CRF slump properties will be an important factor to its successful use. The CRF will need to be thin enough for trucks to handle in the transporting and dumping of the material, but stiff enough to allow the LHDs to pack the material into position. The slump properties will be adjusted based on locations and experience.

It is assumed that the curing time for the CRF will be approximately 28 days. Following curing, the secondary drift can be filled with RF using LHDs. Recommendation for additional work is discussed in Section 26.2.4.

16.7.6.2 Rock Fill

The RF will be used in the secondary drifts according to the design and mine plan. It will act as an unconfined filling adjacent to the primary drifts which will have been previously filled with CRF. Basalt material from the borrow pit on the east side of the project will be used. For the purposes of the PFS, this basaltic rock is assumed to be free of sulfides and therefore not acid generating.

RF material will be hauled at the run-of-mine (“ROM”) size. The transport and disposal of RF into the drifts will use mine trucks that will place the material at unloading points inside the mine, where it will subsequently be loaded and transported to the drifts using LHDs. The LHDs will push the material into place as tight as possible using the loader bucket.

16.8 Mine Equipment

Mine operations will be based on the usage of mobile mining equipment suitable for underground mines. The estimate of the fleet size was based on equipment running-time requirements to achieve the mine production plan. The estimate of the running time for the mine equipment was conducted through the usage of mine-operating factors. Mine-operating factors are summarized in Table 16.10.

Table 16.10 Mine Equipment Operating Factors

Underground Mining Equipment	Availability (%)	Utilization (%)	Operational Efficiency (%)
Drilling Development Jumbo (Jumbo DD21-40)	85	70	70
Bolter (Sandvik DS311)	85	70	70
LHD 5.2 yd^3 (LH410)	85	70	80
Front-end Loader (JCB 456ZX)	85	70	75
Low Profile Truck (AD30)	85	70	75
Telehandler (JCB 540-170)	85	80	80
Bulldozer (Cat D6T)	70	70	80
Motor Grader (Paus PG5HA)	70	70	80
Fuel Truck	75	70	80
Service Truck	75	70	70
Diamond Drilling (Hydracore Gopher)	70	70	70

During the first year of usage, the utilization is restricted to the range between 50% and 60% for all the equipment due to the limited mining fronts. The productivities of development drill and LHD are 222 ft/hr and 114 ton/hr, respectively. The truck productivity varies from 15 to 153 ton/hr depending the haul distance. The shortest haul distance is only 1,640 ft and the longest is more than 12,000 ft.

The maximum permanent underground mine equipment required for the LOM are summarized in Table 16.11.

Table 16.11 Mine Equipment Requirements

Underground Mining Equipment	Quantity
Drilling Development Jumbo (Jumbo DD21-40)	2
Bolter (Sandvik DS311)	1
LHD 5.2 yd^3 (LH410)	4
Front-end Loader (JCB 456ZX)	1
Low Profile Truck (AD30)	3
Emulsion Loader	1
Telehandler (JCB 540-170)	2
Bulldozer (Cat D6T)	1
Motor Grader (Paus PG5HA)	1
Fuel Truck	1
Service Truck	1
Diamond Drilling (Hydracore Gopher)	1

Commented [MG4]: BC: This equipment is from a contractor, therefore, no hours or cost is reported. It will be part of a full services package. I am happy to mention here but no extra information is provided in any other section

During very limited peak times, the maximum requirement of Low Profile Trucks will be five, however, only three trucks are considered as permanent fleet and additional trucks will be provided by a local contractor spordically. The total hours, average hours and maximum hours per month are displayed in Table 16.12.

Table 16.12 Mine Equipment Hours

Underground Mining Equipment	Total Operating Hours	Average Hour per Month	Maximum Hours per Month
Drilling Development Jumbo (Jumbo DD21-40)	27,936	274	406
Bolter (Sandvik DS311)	17,677	173	259
LHD 5.2 yd^3 (LH410)	26,189	257	369
Front-end Loader (JCB 456ZX)	14,022	163	185
Low Profile Truck (AD30)	68,159	667	1,165
Telehandler (JCB 540-170)	45,977	451	686
Bulldozer (Cat D6T)	21,470	210	230
Motor Grader (Paus PG5HA)	21,538	211	233
Fuel Truck	22,715	223	227
Service Truck	15,584	153	162
Diamond Drilling (Hydracore Gopher)	21,429	210	227

Some relevant quantities used for the mine fleet and mining operating cost estimates are provided in Table 16.13.

Table 16.13 Relevant Mining Quantities

Underground Mining Equipment	Unit	Total	Average per Month
Horizontal Drilling	ft	22,341,670	221,205
Emulsion	lb	6,507,246	64,428
Diesel	Gallons	1,968,798	19,493
Power	kWh	34,413,307	340,726

16.9 Mine Personnel

Personnel requirements for the LOM are summarized in Table 16.14. The table includes staff for mine management, operation, maintenance and technical services.

The peak mine personnel required will be 63 workers. The shift system for administrative personnel is planned to be 5 days on and 2 days off, at 10 hours per day. Production-related mining personnel (operators, fitters, electricians, and assistants) will work a shift system of 4 days on and 3 days off in two teams. Each team will provide 12 hours per day coverage so that the mine can operate for a 24 hour per day, 4 days per week. Some personnel may work additional overtime through weekends for care-and-maintenance requirements, as needed.

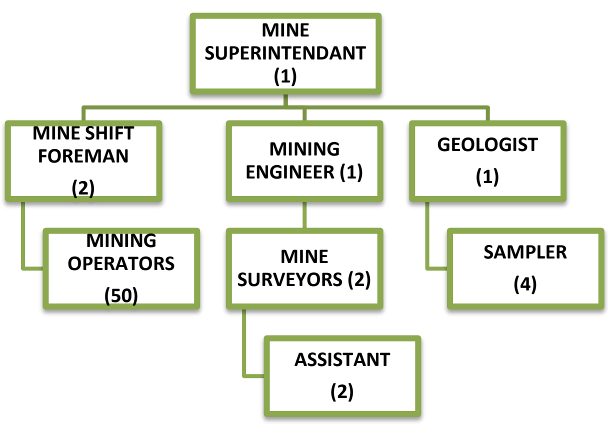
The operating calendar is based on 360 operating days per year.

Table 16.14 Mine Manpower

Role	Role per Shift	Shift	Schedule (days on/off)	N° of Employees
Mine Superintendent	1	1	5/2	1
Planning Engineer	1	2	5/2	1
Geologist	1	2	4/3	1
Mine Shift Foreman	1	2	4/3	2
Surveyor	1	2	5/2	2
Assistant Surveyor	1	2	5/2	2
Sampler	2	2	5/2	4
Mine Operations Manpower				
Maintenance	4/5	2	4/3	11
Drilling	2	2	4/3	4
Bolter	1	2	4/3	2
LHD	4	2	4/3	8
Front end Loader	1	2	4/3	2
Truck	3	2	4/3	6
Ancillary Equipment	4	2	4/3	8
Diamond Drilling	1	2	4/3	2
Assistance	3/4	2	4/3	7
Total Mining Labor	**32**			**63**

The planned mine organization chart is shown in Figure 16.19.

Figure 16.19 Mine Organization Chart



16.10 Mine Production Schedule

MDA used the Proven and Probable mineral reserves defined by Ausenco to create a mine production schedule using MineSched™ (version 9.1), which allows for the scheduling of both underground development and production. The primary inputs used to develop the schedule include:

- The resource block model with defined material types;
- Development centerlines drawn in the direction of mining;
- Solids representing the stopes or production areas to be mined;
- Locations defining stockpiles, processing plant, and waste dumps;
- Material movement definition;
- Mining sequence among developments and production areas;
- Development and production rates by location; and
- Definition of the periods to be used.

The naming convention for material types considered either ore or waste. Ore was assigned to four categories based on grade: high-grade ("HG"), medium-grade ("MG"), low-grade ("LG"), and sub-grade. Sub-grade is material that is below the mining economic COG, but above the resource COG. The basic assumption is that a stope that is economic to be mined will be processed in its entirety. Thus, if internal waste in an economic stope is classified as Measured or Indicated mineral resources, these resources will be converted to Proven or Probable mineral reserves, respectively, and will contribute to the revenue stream.

Waste is comprised of: (*i*) material classified as Measured or Indicated mineral resources that is below both the mining COG and the resource COG; or (*ii*) material classified as Inferred mineral resources. Waste is considered to be internal dilution within a stope, which would be mined and sent to the process plant. All waste material is considered to have zero grade and therefore does not contribute to the revenue steam.

The development centerlines were provided by Ausenco. Some adjustments were made to the centerlines for proper linkage and mining direction. The level development provided by Ausenco included development in ore, development in waste, and centerlines for the stopes on the main levels. The centerlines were not used in MineSched™ because the production was represented using the stope solids.

The mining solids were provided by Ausenco and were used as provided to define mining locations. Other locations included stockpiles, the mill, and a single waste dump. Three stockpiles were used for LG, MG, and HG material so that higher-grade material can be fed to the mill prior to lower-grade material. All mill material was scheduled to report to the stockpile before being fed into the mill.

Waste development in each sublevel was estimated using the ratio of waste development footage per ore ton, calculated from the main level. Material movement allowed for all of the waste to be sent directly to the waste dump, which includes development tonnages mined. Material mined from the stopes will be routed to the stockpile and then rehandled into the plant.

The mining sequence was defined to make sure that there was sufficient underground development completed for a level prior to mining stopes on that level. Ausenco designed the stope solids using "stope blocks" with dimensions of 20 feet by 20 feet, by 13 feet in height. An advance rate of 15 feet per day was assumed, which would yield 290 tons per day in a single cut. It was anticipated that two stopes could be mined during the day on some levels where sufficient stoping areas would be available. Based on the number of headings, a maximum production of 290 tons per day would be possible with a single heading, or 580 tons per day for two headings on a level.

The PFS contemplates mining of primary and secondary stopes. This will require completion of the primary stope to allow placement and curing of the CRF before the secondary stope can be mined. Ausenco specified that there should be a 28-day delay between primary and secondary stopes to allow for curing time. Detailing the sequence between primary and secondary stopes will be completed as part of short-term mine planning. MDA reviewed each main level to determine a production rate based on the sequence of primary and secondary stopes. This was done by assigning a sequence number for each stope block and then reviewing the difference in the sequence number between the primary and secondary stopes.

The difference between the primary and secondary stopes, together with the production rate, defined a maximum productivity that could be accomplished for secondary stoping based on the delay for the primary stopes to be back-filled. MDA determined the maximum tons per day for each main level (Table 16.15) and these values were also used for the sublevels below and above the main levels.

Table 16.15 Maximum Productivity Estimate

	Level	Max Number of Headings	Maximum Tons per Day Secondary	Headings	Used
Lower Levels	3068	1	200	290	200
	3081	1	200	290	200
	3094	1	275	290	275
	3107	2	400	580	400
	3120	2	500	580	500
	3133	2	690	580	580
	3146	2	850	580	580
	3159	2	900	580	580
	3172	2	900	580	580
	3185	2	900	580	580
	3198	2	900	580	580
	3211	2	1000	580	580
Upper Levels	3224	2	1100	580	580
	3237	2	1200	580	580
	3250	2	1300	580	580
	3263	2	1500	580	580
	3276	2	1500	580	580
	3289	2	1500	580	580
	3302	2	1200	580	580
	3315	2	1000	580	580
	3328	2	900	580	580
	3341	2	800	580	580
	3354	2	700	580	580
	3367	2	500	580	500
	3380	2	400	580	400
	3393	2	275	580	275
	3406	2	275	580	275
	3419	2	200	580	200
	3432	2	200	580	200
	3445	2	200	580	200
	3458	1	200	290	200
	3471	1	200	290	200
	3484	1	200	290	200
	3497	1	200	290	200
	3510	1	200	290	200
	3523	1	200	290	200

The final PFS production schedule was calculated in MineSched and then summarized in Excel. Ore loss and dilution were applied using Excel spreadsheets. Waste development rates were smoothed out in Excel by Ausenco. Table 16.16, Table 16.17, and Table 16.18 show summaries for the planned mine production, material sent to the mill, and the stockpile balance, respectively. Table 16.19 shows the proposed development schedule.

Table 16.16 Mine Production Summary

	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Mined Ore Above Economic	K Tons	-	135	182	189	178	163	169	63	11	-	1,089
COG	oz Au/ton	-	0.296	0.385	0.278	0.259	0.298	0.322	0.254	0.395	-	0.304
	K ozs Au	-	40	70	53	46	48	54	16	4	-	332
	oz Ag/ton	-	0.298	0.328	0.331	0.312	0.359	0.422	0.506	0.335	-	0.352
	K ozs Ag	-	40	60	63	55	58	71	32	4	-	383
Subgrade Ore	K Tons	-	76	90	83	95	110	102	45	7	-	609
	oz Au/ton	-	0.062	0.064	0.067	0.065	0.066	0.068	0.070	0.049	-	0.066
	K ozs Au	-	5	6	6	6	7	7	3	0	-	40
	oz Ag/ton	-	0.193	0.213	0.218	0.198	0.236	0.252	0.265	0.227	-	0.224
	K ozs Ag	-	15	19	18	19	26	26	12	2	-	136
Internal Waste	K Tons	-	1	1	2	1	1	1	1	0	-	8
Level Access Mined as Ore	K Tons	0	1	-	-	-	-	-	-	-	-	1
	oz Au/ton	0.149	0.115	-	-	-	-	-	-	-	-	0.125
	K ozs Au	0	0	-	-	-	-	-	-	-	-	0
	oz Ag/ton	0.141	0.221	-	-	-	-	-	-	-	-	0.199
	K ozs Ag	0	0	-	-	-	-	-	-	-	-	0
Total Mined to Stockpile	K Tons	0	214	274	274	275	274	271	109	18	-	1,708
	oz Au/ton	0.149	0.210	0.277	0.213	0.190	0.203	0.226	0.175	0.256	-	0.218
	K ozs Au	0	45	76	58	52	56	61	19	5	-	372
	oz Ag/ton	0.141	0.259	0.288	0.295	0.271	0.308	0.357	0.401	0.290	-	0.304
	K ozs Ag	0	55	79	81	74	84	97	44	5	-	519
Backfill Dilution	K Tons	0	4	5	5	5	5	5	2	0	-	31
Total w/ Ore Loss & Dilution	Tons	0	220	282	282	283	282	279	113	18	-	1,759
	oz Au/ton	0.142	0.199	0.261	0.201	0.181	0.192	0.213	0.166	0.241	-	0.206
	ozs Au	0	44	74	57	51	54	59	19	4	-	362
	oz Ag/ton	0.143	0.252	0.279	0.285	0.263	0.297	0.343	0.383	0.281	-	0.294
	ozs Ag	0	55	79	80	74	84	96	43	5	-	517

Table 16.17 Material Sent to the Mill

	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Internal Waste	K Tons	1	1	1	1	1	1	1	0	-	8
Sub-Grade Material	K Tons	74	83	76	88	103	98	87	7	-	616
	oz Au/ton	0.063	0.065	0.068	0.066	0.067	0.067	0.070	0.051	-	0.066
	K ozs Au	5	5	5	6	7	7	6	0	-	41
	oz Ag/ton	0.196	0.209	0.219	0.200	0.223	0.248	0.257	0.226	-	0.223
	K ozs Ag	14	17	17	18	23	24	22	2	-	137
Low-Grade Material	K Tons	94	114	130	130	111	121	45	5	-	750
	oz Au/ton	0.154	0.153	0.159	0.156	0.156	0.152	0.146	0.147	-	0.154
	K ozs Au	14	17	21	20	17	18	7	1	-	116
	oz Ag/ton	0.251	0.256	0.290	0.272	0.305	0.335	0.425	0.358	-	0.295
	K ozs Ag	24	29	38	35	34	41	19	2	-	221
Medium-Grade Material	K Tons	25	37	40	30	32	25	13	2	-	203
	oz Au/ton	0.315	0.318	0.310	0.301	0.297	0.304	0.319	0.321	-	0.309
	K ozs Au	8	12	12	9	10	7	4	1	-	63
	oz Ag/ton	0.294	0.327	0.341	0.334	0.294	0.395	0.683	0.372	-	0.352
	K ozs Ag	7	12	13	10	9	10	9	1	-	71
High-Grade Material	K Tons	19	34	22	20	21	25	6	3	-	150
	oz Au/ton	0.857	1.136	0.822	0.762	0.948	1.070	0.833	0.789	-	0.947
	K ozs Au	17	38	18	15	20	26	5	3	-	142
	oz Ag/ton	0.490	0.532	0.497	0.480	0.675	0.783	0.558	0.249	-	0.571
	K ozs Ag	10	18	11	10	14	19	3	1	-	86
Backfill Dilution	K Tons	4	5	5	5	5	5	3	0	-	31
Total to Plant	K Tons	217	274	274	275	274	274	154	18	-	1,759
	oz Au/ton	0.201	0.266	0.205	0.184	0.196	0.215	0.139	0.241	-	0.206
	K ozs Au	44	73	56	51	54	59	21	4	-	362
	oz Ag/ton	0.253	0.280	0.287	0.265	0.295	0.343	0.345	0.280	-	0.293
	K ozs Ag	55	77	79	73	81	94	53	5	-	516
Plant Throughput	TPD	594	750	750	750	750	750	423	50	-	

Table 16.18 Stockpile Balance

	Units	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
Added	K Tons	220	282	282	283	282	279	113	18	-
	oz Au/ton	-	-	-	-	-	-	-	-	-
	K ozs Au	44	74	57	51	54	59	19	4	-
	oz Ag/ton	-	-	-	-	-	-	-	-	-
	K ozs Ag	55	79	80	74	84	96	43	5	-
Removed	K Tons	217	274	274	275	274	274	154	18	-
	oz Au/ton	-	-	-	-	-	-	-	-	-
	K ozs Au	44	73	56	51	54	59	21	4	-
	oz Ag/ton	-	-	-	-	-	-	-	-	-
	K ozs Ag	55	77	79	73	81	94	53	5	-
Balance	K Tons	3	12	20	28	36	42	-	-	-
	oz Au/ton	0.067	0.071	0.073	0.072	0.071	0.075	-	-	-
	K ozs Au	0	1	1	2	3	3	-	-	-
	oz Ag/ton	2.752	3.382	3.184	3.120	3.693	3.546	-	-	-
	K ozs Ag	1	3	5	6	10	11	-	-	-

Table 16.19 Development Schedule

Development Type	Units	Pre-Prod -2	Pre-Prod -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Main Decline	K Feet	1.3	4.4	1.5	-	-	-	-	-	-	-	-	7.3
Vent Drift	K Feet	0.1	1.1	1.5	-	-	-	-	-	-	-	-	2.6
Level Access	K Feet	-	-	0.5	0.4	0.2	0.4	0.4	0.4	0.5	0.5	-	3.2
Level Development Waste	K Feet	-	-	2.1	1.5	1.0	1.4	2.1	3.0	2.1	0.3	-	13.4
Level Development Ore	K Feet	-	0.1	11.9	0.3	-	-	-	-	-	-	-	12.2
Vent Shaft	K Feet	-	1.0	0.5	-	-	-	-	-	-	-	-	1.5
Total Development	K Feet	1.4	6.6	18.0	2.2	1.1	1.7	2.4	3.3	2.7	0.8	-	40.2

Figure 16.20 and Figure 16.21 show the proposed yearly production schedule in terms of tons and gold and silver ounces for the LOM.

Figure 16.20 Mine Production Schedule

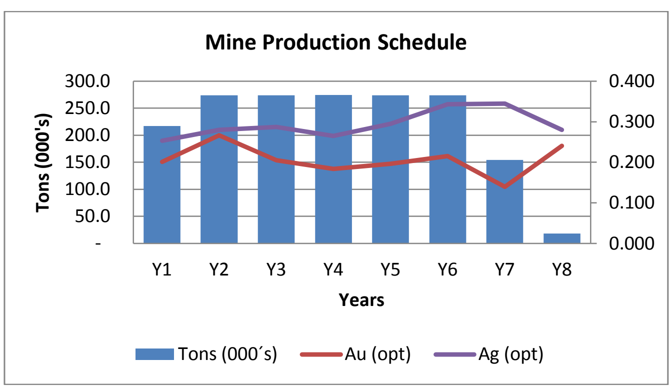
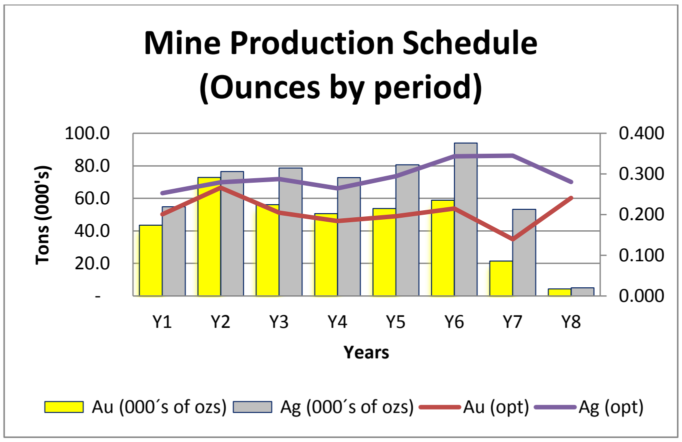


Figure 16.21 Mine Production Schedule (Ounces by Period)



17.0 RECOVERY METHODS

This section has been prepared under the supervision of Mr. Robert Raponi, P. Eng., of Ausenco. Mr. Raponi believes that the information presented in this section of the report accurately reflects the mineral recovery and proposed processing parameters as of the Effective Date of this report.

17.1 Mineral Processing Overview and Flowsheet Development

Based on the information and test results summarized in Section 13.0, the Grassy Mountain gold-silver mineralization is considered to be amenable to the proposed recovery process that will involve a combination of gravity concentration and cyanide leaching. A nominal process plant treatment rate of 750 short tons per day has been designed to recover and concentrate gold and silver. The plant will be of the conventional CIL type and is designed to operate with two shifts per day, 365 days per year, with an overall plant availability of 91.3%. The process plant will produce gold doré bars to be sold to gold refiners.

The plant feed will be hauled from the underground mine to a mobile crushing facility that includes a jaw crusher as the primary stage and cone crushers for secondary and tertiary size reduction. The crushed ore will be ground by a ball mill in a closed circuit with hydro-cyclones. A centrifugal gravity concentrator will collect gravity-recoverable gold ("GRG") from the cyclone underflow and discharge it to an intensive-leach reactor ("ILR") for recovery. The hydro-cyclone overflow with P_{80} of 100 mesh will flow to a CIL recovery circuit via a pre-aeration reactor.

Gold and silver leached in the CIL circuit will be recovered on carbon and eluted in a pressurized Zadra-style elution circuit and then precipitated by electrowinning in the gold room. The gold-silver precipitate will then be mixed with fluxes and smelted in a refining furnace to pour doré bars. Carbon will be re-activated in a carbon regeneration kiln before being returned to the CIL circuit.

Leached tailings will be detoxified in an SO_2/air cyanide destruction circuit. Detoxified tails will be pumped to a tailings storage facility ("TSF") for final deposition and recovery of decant water. Process water recovered from the decant water will be re-used for grinding and plant utility water.

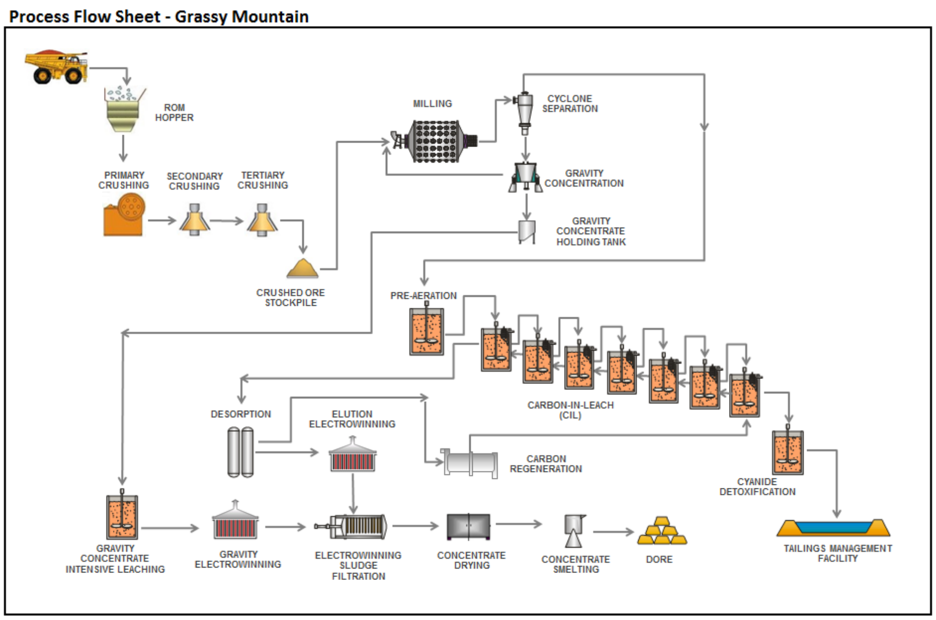
17.1.1 Flowsheet Development

The process flowsheet was developed based on the historical comminution data and 2017 SGS laboratory testwork results as outlined in Section 13.0. The flowsheet considers that the process plant will consist of the following unit operations:

- Crushing and stockpile;
- Grinding and classification;
- Gravity concentration with concentrate intensive leaching;
- CIL leaching;
- Carbon management;
- Gold room; and
- Detoxification and tails deposition.

The simplified process flowsheet is shown in Figure 17.1. Note that the term "tailings management facility" in Figure 17.1 is equivalent to the term "TSF" that is used elsewhere in this report.

Figure 17.1 Grassy Mountain Process Flow Sheet



17.1.2 Major Process Design Criteria

The principal process-design criteria for the Grassy Mountain project are outlined in Table 17.1.

Table 17.1 Grassy Mountain Process Design Criteria Summary

Description	Units	Value
Ore Throughput	tons/y	273,750
Design Grade - Au	oz/ton	0.24
Design Grade - Ag	oz/ton	0.34
Operating Schedule		
Crusher Availability	%	70
Plant Availability	%	91.3
Throughput, Daily - average	tons/day	750
Plant capacity, Hourly	tons/hour	31
Crushing (Three Stage)		
Primary Crusher	type	Single Toggle Jaw Crusher
Secondary/Tertiary Crusher	type	Cone Crusher
Fine Ore Stockpile Residence Time - Live	day	5
Grinding		
Circuit Type		3-Stage Crush, Ball mill
Bond Ball Mill Work Index	kWh/ton	19.0
Ball Mill Power	hp	1,072
Feed Particle Size, F_{80}	inch	0.394
Product Particle Size, P_{80}	U.S. mesh	100
Gravity Concentration		
Overall Gravity Gold Recovery	%	10
Carbon-In-Leach		
Total Leach Time	hour	24
Number of Tanks	#	1 pre-aeration + 7 leach / adsorption
Cyanide Addition	lb/ton	0.82
Lime Addition	lb/ ton	2
Carbon Concentration	lb/ft^3	1.56
Carbon Loading (Au + Ag)	oz/ton	175
Desorption/Electrowinning/Refining		
Elution method	-	Pressure Zadra
Carbon batch size	ton	3.3
Elution CIL strips per week	#	7
Gravity leach solution electrowinning batches per week	#	7
Cyanide Destruction		
Method	-	SO_2 Air
Residence time	hour	2
CN_{WAD} target	ppm	<0.1
Sodium Metabisulphite Addition	lb/ton	4

17.2 Process Plant Description

The proposed process plant layout is shown in Figure 17.2.

17.2.1 Crushing and Stockpile

The crushing facility will be mobile and have a two-stage crushing circuit that will process the underground ore at a nominal processing rate of 42 tons per hour. The major equipment and facilities at the ore-receiving and crushing areas include:

- Ore-surge bin;
- Stationary ore-bin grizzly;
- Vibrating grizzly feeder;
- 35-inch x 26-inch primary jaw crusher;
- 5-foot x 14-foot double-deck vibrating screen;
- 120-hp secondary cone crusher;
- Covered fine-ore stockpile; and
- Stockpile-reclaim hopper and belt feeder.

The ore will be trucked from the underground and dumped directly into the ore-surge bin, or it will be stockpiled on the stockpile storage pad which can be reclaimed by a front-end loader for continuous feed. Particles larger than the stationary ore-bin grizzly can be removed by the front-end loader for individual breakage using a mobile rock breaker.

The ore from the bin will be withdrawn by the grizzly feeder where the coarse oversize will report directly into a single-toggle jaw crusher. The feed material will be crushed, and the product will discharge from the crusher onto the coarse-ore conveyor that also receives the vibrating-grizzly undersize ore. The combined crushed ore and the grizzly undersize will be transferred to the double-deck secondary screen. The top deck and bottom deck oversize fractions from the secondary screen will discharge directly into the secondary cone crusher. The bottom-deck undersize from the secondary screen will be collected by the screen-discharge conveyor, which will deliver feed material to the fine-ore stockpile conveyor.

This conveyor is expected to be fitted with a weightometer to monitor crushing-plant throughput and assist with operational and metallurgical accounting. The fine ore reporting to the stockpile-feed conveyor will be transferred to the fine-ore storage area. The fine-ore storage consists of a conical-crushed ore stockpile with 24 hours of live capacity.

Figure 17.2 Process Plant Layout

Fine ore from the stockpile will be reclaimed by the process plant front-end loader at a combined nominal rate of 29 tons per hour, discharging ore to the fine-ore stockpile reclaim hopper. The hopper has a live ore mass of 29 tons, providing the ball-milling circuit with one-hour worth of feed. Fine ore is to be withdrawn from the hopper by means of the reclaim belt feeder, which discharges onto the ball-mill-feed conveyor. The fine ore from the ball-mill-feed conveyor will discharge directly into the ball mill. The ball-mill-feed conveyor will be equipped with a weightometer to provide feed rate data for feed-rate control to the grinding circuit.

17.2.2 Grinding and Classification

The primary grinding circuit will consist of a ball mill in a closed circuit with classifying cyclones. The circuit will be equipped with a single centrifugal gravity concentrator to recover GRG. Approximately 33% by weight of the cyclone underflow will be fed to the gravity concentrator. The proposed grinding circulation load for the closed-circuit ball mill is 350% of new feed.

The primary grinding circuit is designed for a product size P_{80} of 100 mesh. The major equipment in the primary grinding circuit will include:

- One 11-foot diameter (inside shell) by 17-foot effective grinding length (“EGL”) single-pinion ball mill driven by a single 1,072 hp fixed-speed wound-rotor drive motor (“WRIM”); and
- One cyclone cluster, consisting of two 15-inch diameter cyclones (1 operating, 1 standby).

As required, steel balls will be added into the ball mill using a ball bucket and kibble system to maintain grinding efficiency.

Ore addition to the ball mill is to be supplemented with process water to achieve a slurry density of approximately 72% solids (by weight). The ball-mill discharge will flow through a trommel screen, which will remove any trash or broken mill balls and discharge them into a tote. The latter will be removed either by hand or by the process plant bobcat. The trommel-screen underflow will discharge into the cyclone-feed pump box, where it will be diluted with process water to 61% solids (by weight) and pumped by one of the two cyclone-feed pumps up to the cyclone distribution manifold. One of the two cyclones will remove an overflow stream of 42% solids (by weight) comprised of product-sized particles, while the cyclone-underflow fraction of 70% solids (by weight) will report to the cyclone-cluster underflow weir box.

The cyclone overflow will flow via gravity to the trash screen, which diverts trash or fibrous materials into a trash bin. The trash screen underflow then will flow via gravity to the pre-aeration tank.

Maintenance activities in the grinding and classification area will be serviced by the mill area crane with a capacity of 33 tons, and the grinding area hoist (3.3 tons), which will be used for ball-mill charging duties and minor lifts. Spillages in the grinding and classification area will be pumped by the grinding area sump pump into the cyclone-feed pump box.

17.2.3 Gravity Concentration and Intensive Leaching

The grinding circuit will be equipped with a centrifugal gravity concentrator to recover coarse gold before it goes to the leach circuit. A gravity recovery circuit has been included due to the presence of free gold in the ore. If left in the leach feed, coarse gold particles with a low surface area to volume ratio, may not dissolve completely before leaving the leach circuit, causing reduced gold recovery. While testwork showed that gravity recoveries reach 20% for high-grade feed samples, the expected LOM average is about 10%. This is near to the point at which the need for a gravity circuit becomes questionable. Issues with the GRG testwork caused uncertainty about gravity recoveries and resulted in some conservatism. Further GRG testwork may ultimately show that the gravity circuit could be possibly be eliminated in favor of whole ore leaching.

A manual splitter box will deliver approximately 33% of the hydro-cyclone underflow to the centrifugal gravity concentrator via a scalping screen which will remove particles larger than 10 mesh. The centrifugal concentrator will function as an automated batch process. Feed will be accepted from the cyclone underflow for a specified time, paused to allow the concentrate built up in the bowl to be flushed, and finally accepted again as the cycle begins anew. Gravity concentrator tails will flow back to the ball mill during the feeding portion of the cycle. Cycle times are 45 minutes each and about 46 lb of concentrate is expected from each cycle of the selected unit.

Flushing water will rinse the gravity concentrate to a gravity concentrate hopper located beneath the concentrator. Water will be continually decanted as the GRG concentrate accumulates in the hopper.

Access to the gravity concentrator hopper storage area will be restricted to authorized personnel only. Once per day the concentrate accumulated in the gravity concentrate hopper will be pumped to the ILR where it is dosed with sodium hydroxide and high levels of NaCN. The ILR then will process the batch for about 20 hours to ensure all gold particles have been dissolved. It then decants the pregnant leach solution from the solids, aided by flocculant, and pumps it to the gravity electrolyte tank near the gold room.

17.2.4 Leaching

A pre-aeration tank was included ahead of the leach circuit, as testwork showed this reduced consumption of cyanide and lime in leaching by passivating the sulfides in the ore. The high level of dissolved oxygen (“DO_2”) in the tank oxidizes the surface of sulfides in the ore, preventing them from reacting with cyanide and lime in the leach circuit and thereby reducing consumption of those reagents.

A CIL circuit was selected due to the small throughput required. Laboratory-scale leach kinetics modeling indicated that a carbon-in-pulp (“CIP”) process produced marginally better performance than CIL, but significantly higher carbon-processing requirements more than doubled the size of the carbon elution and regeneration circuit. This disparity in leach performance shrinks as plant throughput increases, until a break-even point is reached at higher feed rates. A trade-off study comparing the economics of CIL and CIP leach circuits at this throughput level would be recommended for the next phase of engineering to confirm this process selection.

Testwork determined that the gold is leached to completion with a residence time of 24 hours. Carbon-management modeling determined that seven leach stages are required, the optimum carbon concentration is 1.56 lb/ft^3, and the advance rate to minimize solution losses is 3.3 tons per day.

The pre-aeration tank will mix the cyclone overflow with lime; low-pressure air will be pumped in, raising the DO_2 to about 10 ppm. Passivated slurry will overflow the pre-aeration tank to the first CIL tank, where it will be adjusted with lime to about pH 10 and cyanide will be added.

Leached gold and silver will be adsorbed onto granular carbon, which is present in all tanks in a CIL-type leach circuit. Barren carbon will be added to the last CIL tank and will travel up through the circuit in the opposite direction from the slurry flow. Carbon will advance once per day with carbon transfer pumps, which pump carbon-laden slurry to the next tank in the train. Carbon will be retained in the tanks after the transfer with inter-stage screens, which have mesh baskets sized to allow slurry to pass through but not the loaded carbon.

Leached tails will overflow the last tank and are to be pumped to the carbon safety screen, which collects carbon that would otherwise be lost to the tailings in the event of a hole in one of the inter-stage screens. Loaded carbon is to be pumped from the first leach tank to the elution circuit via a loaded-carbon screen, which will separate the carbon from slurry and send the slurry back to the leach circuit.

17.2.5 Carbon Management

Acid Wash: Loaded carbon from the leach circuit is to be loaded into an acid-wash column, where it will be submerged in a 3% hydrochloric-acid solution in order to dissolve lime scale that would otherwise interfere with the elution process. After soaking for 30 minutes, the acid will be drained, and several bed volumes of raw water are to be circulated through the column to rinse and neutralize the acid from the carbon. After rinsing, the carbon will be pumped to the elution column via carbon-transfer water.

Carbon Elution: A pressure Zadra circuit was selected for elution of gold and silver from carbon due to the small carbon processing requirements of the CIL circuit and unknown water quality from the raw water wells. A pressure Zadra circuit is less complicated than comparable alternatives, and is less sensitive to poor water quality, which makes it a better choice in this instance.

Strip solution (eluate) will be made up in the strip-solution tank using raw water dosed with 2% sodium hydroxide and 0.2% cyanide to form an electrolyte for the electrowinning process. This solution will be circulated through the elution column via an eluate heater, which heats the solution, the carbon, and the column to 275°F. The elution system will be pressurized to keep the solution from flashing to steam in the heater or elution column.

A recovery heat exchanger will transfer heat from the hot pregnant solution exiting the column to the incoming solution before passing through the solution heater. This will reduce the energy required to maintain the solution temperature and cool the pregnant solution before it enters the electrowinning cell. Once system temperature is reached, the hot pregnant eluate solution will be directed to the electrowinning cell, where the metals will be plated onto the cathodes. Solution continues to circulate through the elution column and electrowinning cell until all metals are eluted from the carbon and deposited in the electrowinning cell.

Carbon Regeneration: Once the elution cycle is complete, the barren carbon will be transferred to the regeneration-kiln feed hopper where it is metered into the regeneration kiln. The kiln will heat the carbon to 1,380°F, which regenerates it by burning off foulants that would otherwise reduce the carbon's ability to adsorb metals in the leach circuit.

Carbon leaving the kiln will fall into a water-filled quench tank, which will keep it from continuing to burn. Some burning will occur, and the fines generated from this must be removed before the carbon is returned into the leach circuit. The quench tank will also serve as a holding tank for carbon that is ready for use in the leach circuit. Regenerated barren carbon will be pumped from the quench tank to the first leach tank via a barren-carbon sizing screen, which will remove the fines and dewater the carbon.

Carbon Pre-Attrition: Bags of new carbon are to be processed in a pre-attrition tank before being sent to the leach circuit. This process will break off the corners of the angular, coconut-shell-based carbon particles and separates them so they do not end up in the leach circuit. The fine particles are too small to be retained by the leach tank inter-stage screens and would therefore pass through to the leach tails. Without this step, the sharp corners of the loaded carbon particles would be broken off in the leach circuit and these carbon fines would flow to tails carrying the gold they had adsorbed and causing gold losses.

The new carbon is to be charged to the pre-attrition tank via a bag breaker where water is to be added to produce an effective solids density of about 50% carbon. A high-intensity agitator will stir the carbon slurry vigorously to break off the sharp corners, reducing the angular particles to a more spherical shape. Once complete, the new carbon will be pumped to the regeneration-kiln-quench hopper where it will be held to be added to the leach circuit via the carbon-sizing screen, which will separate the fines from the coarse carbon.

Carbon Transport Water: All carbon movements in the elution and regeneration circuits are to be accomplished using carbon-transport water. A transport-water tank and pump are planned to supply transport water to carbon movement demands as needed. As an example, when moving carbon from the acid-wash column to the elution column, the carbon will be drained into an eductor with transport water passing through it in sufficient quantity and velocity to carry it to the next destination at an effective solids density of about 20%. As the carbon arrives at the elution column, strainers in the column-discharge ports allow the transport water to exit while the column retains the carbon.

Transport water will pick up fines when moving carbon from one place to another due to both the previous process and the attrition associated with the carbon movement itself. Once the movement is complete, strained or decanted transport water will report to a small carbon-fines clarifier where flocculant will settle the fines and the overflow water recharges the transport-water tank. Process water can be added as necessary to maintain the level in this tank to account for leaks and spillage. The carbon fines are to be removed from the clarifier underflow periodically and shipped to the refinery to recover contained precious metals.

17.2.6 Gold Room

The gold room will house the electrowinning cells, smelting furnace, and associated support equipment within a security envelope that will limit access to authorized gold-room personnel only. Access in and out is to be controlled by security personnel at both a man door and a vehicle-access roll-up door for the armoured car. The armoured-car door is to be enclosed by a fence with an automated gate controlled by security personnel. The exception to this will be an emergency exit door which will set off alarms when opened from the inside.

Once per week, both the CIL and gravity-electrowinning cells will be opened and the sludge cleaned out manually with a high-pressure spray gun. Sludge from the clean-up will flow by gravity to the electrowinning-sludge-filter feed tank and into manually operated pressure canister filters to be dewatered. Dewatered sludge is to be collected in trays and placed in the drying oven overnight. No evidence was found of mercury-bearing minerals, so no mercury retort has been included in the design.

Dried sludge will be removed from the oven the next day and combined with fluxes in a flux mixer before being charged into the smelting furnace. The dried sludge and flux mixture must be completely dry to avoid explosions when adding it to the furnace on top of an already melted charge. Once all the mixture has been added to the furnace, and sufficient time is allowed for the melt to become fully liquid, the slag will be poured off into a conical slag pot. The liquid metal can then be poured into a mold cascade to allow multiple bars to be poured at once. Cooled doré bars will be cleaned and stamped, and the bars will then be placed in the gold-room safe to await shipment to the refinery via armoured car.

Dust collection will be provided in the gold room for flux mixing. Mist scrubbers are planned for the electrowinning cells and smelting-furnace off gasses and extraction fans.

17.2.7 Cyanide Detoxification and Tailings Deposition

A cyanide-destruction circuit has been included in the design to comply with the tailings-discharge permit requirements. Testwork shows that SO_2/air process was the most effective detoxification method and the only one to reduce weak-acid dissociable (“WAD”) cyanide levels to 0.1 ppm.

The CIL tailings will be pumped to the 2-stage agitated cyanide-detoxification tanks, where lime will be added to buffer pH, copper sulfate will be added as a reaction catalyst, and sodium meta-bisulfite (“SMBS”) will be added as a source of SO_2. The tanks are sized to provide two hours of residence time for the reaction to proceed to completion.

Detoxified slurry will overflow the second detoxification tank to the final tailings pump box where it will be pumped to the TSF by the final tailings pumps. At the TSF, the tailings will be deposited using spigotting manifolds positioned along the rim of the impoundment to create low-angle deposition beaches. The position of the spigotting manifolds will be moved periodically to produce an even beach head and push decant water towards the decant-water pool. A pontoon-mounted decant-return water pump will be provided to pump decant water back to the process-water tank for re-use in the plant.

17.3 Reagents

The reagents will be prepared and stored in separate self-contained areas within the process plant and delivered by individual metering pumps or centrifugal pumps to the required addition points. Acidic and basic reagents are to be stored and mixed in physically separated areas to ensure no exposure of cyanide to acidic chemicals, which would generate hydrogen-cyanide gas.

Estimated reagent consumptions are estimated as follows:

- Sodium cyanide - 0.93 lb per ton processed
- Lime - 8.0 lb per ton processed
- Sodium metabisulphite - 3.92 lb per ton processed

Hydrated Lime: Preparation of the hydrated lime will require:

- A bulk-storage silo;
- A mixing tank;
- Dosing pumps feeding a ring main; and
- Automatically-controlled dosing points from the ring main.

Hydrated lime will be used in leaching and detoxification for pH control. The hydrated lime will be delivered to site by bulk tanker and blown into a bulk-storage silo. When the mixing-tank level is low, hydrated lime will be added to the tank via a rotary valve and screw feeder. Process water will be added at the same time to maintain the mixture strength of 20%, forming a milk-of-lime suspension.

Milk-of-lime will be distributed to the various dosing points using a ring main that provides constant flow to various destinations. Dosing will be accomplished with drop lines off the ring main with automated on-off valves that open when pH is low and close when the operator specified target is reached.

Sodium Cyanide: Preparation of NaCN will require:

- A bulk handling system;
- Mixing and holding tanks; and
- Dosing pumps.

NaCN will be used in leaching as a lixiviant and in elution as a carbon-stripping aid. Cyanide will be delivered to site in 1-ton bulk bags contained within wooden boxes and stored in a separate area of the plant from the other chemicals. The NaCN will be dosed from the cyanide-storage tank to dosing points via dedicated positive-displacement metering pumps. The discharge piping is to be arranged such that the low-use pumps can be used as back-up spares for the leach-dosing pump.

When the cyanide-storage tank level is low, a cyanide-mix batch is started by removing a cyanide bulk bag from its box and dropping it onto a bag breaker, which discharges cyanide into the mix tank. The mix tank will have been previously filled with sufficient raw water, and buffered with sodium hydroxide to pH 12, to produce a cyanide-mixture strength of 28%. Once mixing is complete and there is sufficient room in the holding tank, the mixed cyanide solution will be pumped to the cyanide-storage tank by a NaCN transfer pump.

Sodium Hydroxide: Preparation of sodium hydroxide will require dosing pumps. Sodium hydroxide is to be delivered to site in 275-gallon totes at about 50% solution strength. It will be dosed to its various demands at full strength using dedicated positive-displacement metering pumps. Sodium hydroxide will be used for pH control in cyanide mixing and in the ILR. It will also be used as an electrolyte in carbon elution/electrowinning.

Sodium Meta-bisulfite ("SMBS"): Preparation of SMBS will require:

- A bulk-handling system;
- Mixing and holding tanks; and
- Dosing pumps.

SMBS will be used as a source of SO_2 for cyanide destruction with the SO_2/air process. It will be delivered to site in 1-ton bulk bags and will be stored in the reagents-storage building.

The SMBS will be held in the SMBS storage tank after it is mixed. When the storage-tank level is low, a SMBS mix will be started by dropping a bulk bag of SMBS onto a bag breaker, which discharges SMBS into the mix tank. The mix tank will have been previously filled with sufficient process water to produce a mixture strength of 20%. Once mixing is complete, and if there is sufficient room in the holding tank, the mixed SMBS solution will be pumped to the SMBS holding tank by a SMBS-transfer pump.

From the storage tank, SMBS will be dosed to the detoxification circuit via dedicated positive-displacement metering pumps for each stage. A third pump is provided as an installed spare for the detoxification-dosing pumps.

Copper Sulphate: Preparation of copper sulfate will require:

- A bulk-handling system;
- A combined mixing/storage tank; and
- Dosing pumps.

Copper sulfate will be used as a catalyst in cyanide destruction by the SO_2/air process. It will be delivered to site in 1.0-ton bulk bags and will be stored in the reagents storage and handling building. Copper sulfate will be mixed and stored in a combined mixing/storage tank laid out such that the mixing tank will be directly above the storage tank and mixed solution will flow by gravity into the storage tank.
When the storage-tank level is low, copper sulfate will be added to the mixing tank by dropping a bulk bag onto a bag breaker, which discharges copper sulfate into the mix tank. The mix tank will have been previously filled with sufficient process water to produce a mixture strength of 15%. Once mixing is complete, and there is sufficient room in the holding tank, the mixed copper-sulfate solution will be

transferred by gravity to the holding tank. From there, copper sulfate will be dosed from the storage tank to the detoxification circuit via duty/standby positive-displacement metering pumps.

Hydrochloric Acid: Preparation of hydrochloric acid will require a mixing tank and dosing pumps. Hydrochloric acid will be used to remove lime scale from loaded carbon in the acid-wash column of the elution circuit. It will be delivered to site in 275-gallon totes at about 32% solution strength and will be stored in the reagents storage and handling building.

Hydrochloric acid will be mixed with raw water in a dilute-acid mixing tank to 3% strength and pumped to the acid-wash column with a dedicated dilute-acid transfer pump.

17.4 Process Plant Air Services

Low-Pressure Air: Three low-pressure air blowers will supply air to the pre-aeration, leach, and detoxification tanks. The installed blowers will be multiple-stage, centrifugal-type blowers and will be used with a "blow-off" arrangement to adapt to fluctuations in air demand.

Plant & Instrument Air: Two rotary-screw plant air compressors will provide high-pressure compressed air, operating in lead-lag mode, to meet the demand for plant and instrument-air requirements. Wet plant air will be stored in the plant-air receivers to account for variations in demand prior to being distributed throughout the plant. Instrument air will be dried in the instrument-air dryer before distribution throughout the plant.

17.5 Process Water Services

Raw Water: Raw water will be pumped from borehole wells to a 200,000-gallon raw-water storage tank located in the water-services area for feed to the plant. Raw water is to be supplied to the plant by two raw-water pumps in a duty/standby configuration. Raw water in the raw-water storage tank will be used to supply the process-water tank, gland water, reagent mixing, and fire-protection requirements.

Potable Water: Potable water will be sourced from the raw-water tank and treated in the potable-water treatment skid. The treated water will be stored in the potable-water storage tank for use by two potable-water pumps in a duty/standby configuration.

Gland Water: Gland water will be supplied from the raw-water tank and distributed to the plant by two gland-seal water pumps in a duty/standby configuration.

Process Water: Process water will be comprised mainly of TSF reclaim water. Process water is to be stored in the process-water storage tank and distributed by the two process-water pumps, in a duty/standby configuration.

17.6 Quality Control Assay and Metallurgical Laboratory

The Grassy Mountain process plant will be equipped with automatic samplers to collect shift and routine samples for aqua-regia digestion, atomic-absorption analyses, and fire assays. The samples to be analyzed will include head, intermediate products, tailings, and doré. The data obtained will be used for product quality control and routine process optimization.

The metallurgical laboratory will perform metallurgical tests for quality control and process-flowsheet optimization. The laboratory will include equipment such as laboratory crushers, ball mill, sieve screens, bottle rollers, leach reactors, balances, DO_2 meters, and pH meters.

17.7 Projected Energy Requirements

Process-related power consumption is estimated to be 53.3 kWh per ton processed.

18.0 PROJECT INFRASTRUCTURE

18.1 Project General Arrangement

The project general arrangement, which was developed by MDA with assistance from Ausenco and Golder, is shown in Figure 18.1.

18.2 Site Facilities

This summary of the proposed site facilities was prepared by MDA and Ausenco.

18.2.1 Access Roads

The main access road to Grassy Mountain will utilize an existing BLM road to the site. This road is approximately 17 miles long and will need to be upgraded to include some straightening and widening in portions. An engineering firm in Ontario, Oregon completed preliminary designs for the road alignment and provided general road profiles. Approximately 50 culverts of 18 inches in diameter will be required to allow drainage under the roadway. Once the road is built to subgrade level, approximately 120,000 tons of ¾-inch gravel will be used to surface the road to a depth of six inches.

Costs for the road construction were estimated by a local contractor that would utilize a portable crusher to provide the gravel to be used. An aggregate source located on private property has been identified, although other public sources may also be available. The cost of upgrades to the main access road is estimated to be $3.302 million and is included in the initial capital estimate.

18.2.2 Security and Fencing

Security fencing will be installed around the entire mine site, including the borrow source area. The total length of the perimeter fence is estimated to be 22,350 feet. There will be a main gate where the main access road enters the site, and a second gate will be placed at the southern end of the property. The main gate will include a parking area and a guard shack. The southern access gate is anticipated to remain locked with access only allowed as needed.

18.2.3 Water Supply and Distribution

Water supply is anticipated to come from two sources: 1) SPR 01 well between the plant and the borrow source; and 2) additional wells drilled near SPR 02 about three miles north of the proposed mine site. SPR 01 is expected to be low producing and will primarily be used as a backup well.

Two wells will be drilled in the area of SPR 02, and water from these wells will be pumped along the main access road through a pipeline to the mine site. The majority of water will come from the SPR 02 area.

Storage tanks will be placed at both the SPR 01 and SPR 02 locations to allow for temporary storage as needed.

Figure 18.1 Grassy Mountain Project General Arrangement

18.2.4 Fire Protection

Water for fire protection will be distributed from the fire water tank located at the base of Grassy Mountain via a network of piping and will be maintained under a constant pressure with a jockey pump. The piping will be looped and sectionalized to minimize loss of fire protection during maintenance. Where located outside buildings, fire water piping will be buried below the ground surface to eliminate the potential of pipes freezing.

Yard hydrants will be limited to the fuel storage tank area. Wall hydrants will be used in lieu of yard hydrants, and these will be located on the outside walls of the buildings in cabinets that will be heated during winter months.

Fire protection within buildings will include standpipe systems, sprinkler systems, and portable fire extinguishers. Standpipe systems will be provided in all structures that exceed 46 feet in height, as well as where required by building code, local authorities, or the insurance underwriter.

Sprinklers will be provided at the following locations or to protect the following items:

- Truck workshop;
- Assay laboratory;
- Over hydraulic or lube packs that contain more than 120 gallons of fluid;
- Lube-storage rooms;
- Any conveyor belts that are within tunnels or other enclosed spaces which would be hazardous to fight fires manually;
- Transformers; and
- Warehouse.

18.2.5 Fuel Supply, Storage, and Distribution

A single double-walled steel tank will be used for diesel storage. There will be one 8,200-gallon tank for mobile mining equipment. The fuel will be used by both underground and surface mobile equipment. The surface equipment will primarily be fuelled at a fuel island near the storage tanks. The underground-mining equipment includes a fuel truck that will be used to fuel underground equipment as required. This fuel truck may be used to fuel surface equipment as needed.

The locations of the fuel tanks and fuel island are shown in Figure 18.1.

A small portable tank will be maintained for unleaded gasoline as required for light vehicles and other small equipment (e.g., portable pumps). This tank will be stored in a location away from fire hazards and will be placed within a lined berm area as required by local regulations. Light vehicles that return off-site overnight will be fuelled in other locations, thereby reducing the storage requirements for gasoline on site.

18.2.6 Compressed Air Supply

High-pressure compressed air will be provided by two duty screw compressors, one standby screw compressor, and a duty-plant air receiver. There will be two high-pressure air uses: instrument air and plant air. The instrument air will be dried and then stored in a dedicated air receiver. The plant air will be fed straight from the plant air receiver without a drying step.

Low-pressure air for pre-aeration tank air requirements will be provided by two duty and one standby centrifugal blowers.

18.2.7 Communications

On-site communications will comprise inter-connected mobile and fixed systems, including a land-line telephone network, portable two-way radios, and internet. Access for internet and corporate network connection will be made via satellite connections.

Underground communication with the surface will be via a leaky-feeder system as described in Section 16.6.5.

18.2.8 Transportation

Main transportation of personnel and supplies will be via the main access road. No provisions have been made at this time for the transport of employees, as they will be required to drive out or car pool at their own expense.

18.2.9 Buildings

A total of nine buildings are planned to be constructed at the site to support mining, processing, and administrative activities. The locations of these buildings are shown in Figure 18.1.

Administration Building

The administration building will be a double-width Atco trailer of approximately 3,600 ft^2. It will contain the mine general manager's office, as well as accounting and human resources offices.

Plant Office and Changehouse

The plant office building and changehouse will constructed as a single-level modular wood-frame building of approximately 2,900 ft^2. It will contain the plant offices and change rooms for the process plant staff and labor force. These facilities will be complete with showers, basins, toilets, lockers, and overhead laundry baskets.

Plant Maintenance and Warehouse

The process-plant maintenance and warehouse building will be a pre-engineered steel-frame and metal-clad building of approximately 1,075 ft^2. This building will be used to perform maintenance for process equipment, as well as for the storage of equipment spare parts.

Mine Office

The mine office and changehouse will be constructed as a single-level modular wood-frame building of approximately 4,300 ft^2. This building will include Engineering and Geology offices as well as mine-operations offices. The building will also have showers, basins, toilets, lockers, and overhead laundry baskets. The building will also include first-aid facilities, along with safety-training areas to be used for site-wide training.

Truck Workshop and Warehouse

The truck workshop and warehouse building will be a pre-engineered steel-frame and metal-clad building with an area of 7,100 ft^2 and will be positioned adjacent to the mine-office building. This area will be divided into two sections, one for warehousing spare parts and tool storage and the other for a maintenance workshop. An overhead crane will be included in this building, above the maintenance workshop.

Vehicle Wash-Bay Facility

The vehicle wash-bay facility will be an open-air, 50- by 50-foot concrete slab with a fluid-collection sump and will be located adjacent to the truck workshop and warehouse. Wash water will be collected in the sump where settling will occur prior to the water being recirculated back to the wash system. An oil-water separation system will be included in the facility to recover hydrocarbons prior to re-use of the wash water. The recovered hydrocarbons will be collected and shipped offsite for disposal in accordance with applicable environmental regulations.

Laboratory

The laboratory will be constructed as a single-level modular wood-frame building of approximately 1,850 ft^2 situated adjacent to the process building. The laboratory building will house all laboratory equipment for assaying, metallurgical, and environmental requirements. Dust-collection equipment will be located external to the laboratory building.

Gold Room

The gold room will be a pre-engineered steel-frame and metal-clad building of approximately 1,850 ft^2. This building will be used to pour doré gold, which will be shipped off site for refining.

Reagent Storage and Handling

Reagents will be stored and handled in a pre-engineered, open-air building consisting of a roof and top-panel walls. This building will have an area of approximately 3,500 ft^2 and will be located near the process-plant building.

18.2.10 Explosives Storage and Handling

Explosives-storage facilities will be constructed at the southwest side of the Grassy Mountain project (Figure 18.1). This location uses the hill as a natural barrier between the explosives-storage facility and other infrastructure. The storage facilities will consist of leased powder magazines as per vendor quotation. Dirt berms will be places around the magazines for additional security.

Explosives will be delivered to site by vendors using the main access. Explosives will be delivered to the working face using stainless-steel totes on flatbed trucks.

18.2.11 Borrow Source

A borrow pit will be located on the east side of the property where there are basalts that are believed to be suitable for both construction and mine-backfill material, and a small borrow pit north of the processing area is planned for additional construction material. The borrow mining would be done by a contractor, and some of the material may be crushed for use as RF and CRF as needed. A small contractor laydown-yard is also planned near the main borrow source area.

18.3 Electrical Power Supply and Distribution

MDA estimated the costs for electrical-power distribution to site based on vendor quotations. MDA also estimated the costs for electrical distribution for the underground-mining operations, while Ausenco estimated the cost of power distribution for the remainder of the site, including the mill.

MDA performed a trade-off study to determine the most economical option to supply power to the site: line power brought in to supply electricity from the grid, or power generation using natural-gas-fired generators. The analysis determined that the long-term cost of power generation on site would be approximately $0.221 per kWh compared to a cost of about $0.065 per kWh for line power. The total capital cost of installing line power is slightly over $7 million. Power-generation equipment was assumed to be leased and was included in the unit cost for power. Based on MDA's evaluation, the installation of line power is the preferred option and will have a payback period of approximately 2.2 years.

For the PFS, it was assumed that power supply would initially be from diesel power generators located on site. This would be used for slightly over a year during construction and initial mining of the decline. During the construction period a new power line would be constructed along the main access road to site.

18.3.1 On-Site Power Generation

Quotations for portable power generation, including the leasing of generation equipment, were obtained from vendors. Once construction of the primary power lines has been completed, the generators would remain on site for backup in case of power outages. Power generation is estimated based on monthly rates and fuel, as the rate per kWh will vary depending on power consumption. The cost of on-site generation is estimated to be around $0.53 per kWh when mining starts, and it reduces to about $0.48 per kWh during the main development of the mine.

18.3.2 Line Power

A quotation was obtained for the construction of line power to deliver approximately 5.3 MW of power to site, including a 23-mile distribution circuit, a new 69/34.5 kV to 14 MV transformer, and a new 34.5-kV 167-amp regulator. The power line would be constructed from the Hope Substation near Vale, Oregon to the mine site along the main access road. The line-power operating cost is estimated at $0.0632 per kWh.

18.3.3 Site Power Distribution

The plant power distribution from the powerhouse will be via overhead powerlines. The distribution voltage to the local electrical rooms will be 4.16 kV. There will be a combination control-room and motor-control-center room. This room will be pre-fabricated and loaded with electrical equipment prior to delivery to site. The power distribution from the electrical rooms will be 480 V.

The total connected load for the process plant is expected to be 4.9MW, with an average power draw of 3.3 MW.

18.3.4 Underground Mine Power Distribution

At the start of mining an underground 480 V transformer will be placed near the entrance to the portal. This will supply power to electrical equipment used to develop the main decline and portable fans. Once development has advanced far enough that carrying power at 480 V becomes too inefficient, a main power line will be installed along the rib of the decline to carry 4.16 kV and connected to the transformer which will be moved underground.

Upon completion of the decline to the 3224 level, and the initiation of production-mining activities, a second underground transformer will be purchased for use in the lower areas of the mine.

Line power will also be carried up the hill to the location of the two ventilation shafts to supply power to the ventilation fans.

18.4 Tailings Storage Facility

Golder caution to readers: In this Item, all descriptions and estimates related to the locations and designs of the tailings storage facility are forward-looking information. There are many material factors that could cause actual results to differ materially from the designs, forecasts, or projections set out in this Item. Some of the material factors include differences from the assumptions regarding the following: facility locations, permitting, production rate, processing methods, water recovery and usage, and construction methods. The material factors or assumptions that were applied in drawing the conclusions, forecasts and projections set forth in this Item are summarized in this report. Any significant differences from these factors or assumptions will have material impacts on the locations and designs of facilities as set forth in this report.

Golder completed a trade-off study for two TSF locations to provide Paramount with sufficient information to select a preferred location (Conway et al. 2016). Option 1 is located in the valley east of the mine portal, and it was the preferred TSF location presented in the Amended Preliminary Economic Assessment (PEA) for the project, prepared by Metal Mining Consultants Inc. on July 9, 2015. Option 2 is located in the broad valley west of the mine portal. Conceptual designs were completed for each option using the same design criteria and key design components. The advantages and disadvantages for each option were presented and discussed in the detail in the trade-off study. Based on a review of the existing topography beneath the facility, the elevation of the facility, the construction material requirements, and the assumed risks associated with each option, Option 2 was selected as the preferred alternative. All discussions of the TSF within this PFS report are in reference to Option 2.

The proposed TSF will cover approximately 110 acres and will be located in the broad valley immediately west of the Grassy Mountain mine portal and process facilities (Figure 18.1, prepared by MDA). The TSF will fill the valley and require embankments on the north and west sides to impound the tailings. The main embankments will cross the natural drainage on the north side of the TSF, and small secondary embankments will be constructed across saddles along the western ridge.

The embankments will be constructed in stages with soil and/or rock materials generated from on-site borrow sources using downstream construction methods. The embankments will have a maximum overall upstream slope of 3H:1V, with a downstream slope of 2.5H:1V. The TSF will be a 100% geomembrane-lined facility with a continuous, composite engineered lining system extending across the impoundment basin and the upstream slope of each stage.

The embankments are designed to be geotechnically stable during the design seismic event. For this preliminary design, Golder performed a site-specific seismic and faulting hazard assessment to estimate peak ground motions resulting from various seismic events. The design seismic event will vary based on the dam hazard classification required by the regulatory agencies during the consolidated permitting process. For this preliminary design, Golder utilized a design earthquake with a return period of 2,475 years for closure conditions. This exceeds the requirements for a Low Hazard Dam Classification.

The composite lining system within the impoundment basin will consist of (from bottom to top) a six-inch to 12-inch thick, prepared subgrade, a 300-mil thick, enhanced geosynthetic clay liner ("GCL"), 80-mil HDPE geomembrane liner, an 18-inch thick drainage layer, and a six-inch thick filter layer. An underdrain collection system consisting of perforated piping will be located within the drainage layer to promote drainage of the tailings. The upstream slope of the embankments will utilize the same composite lining system, but without the overlying piping, drainage and filter layers.

A reclaim pond, located north of the TSF, will capture all process solution collected in the underdrain collection system. The lining system for the reclaim pond will consist of (from bottom to top): a prepared-in-place subgrade, 60-mil HDPE secondary geomembrane liner, HDPE geonet, and 80-mil HDPE geomembrane primary liner. The geonet located between the two geomembranes will serve as the leakage collection and recovery system ("LCRS").

The supernatant pool will be maintained away from the embankments on the eastern side of the facility by controlled deposition of tailings from spigots installed around the perimeter of the facility. Water separating from the tailings solids after deposition will be managed with two independent return-water systems. One will manage flows collected in the Reclaim Pond from the underdrain collection systems and the other will manage water collected in the supernatant pool. The supernatant pool will be managed with a pump installed either on the eastern edge of the facility or on a floating barge within the pool. Water from both systems will be returned to the mill for use in the process circuit. At all times, process fluid pipelines will be located above secondary containment that consists of either geomembrane liners or concrete containment structures.

The TSF has been designed as a zero-discharge facility capable of storing the 500-year, 24-hour storm event and an allowance for wave run-up due to wind action. Permanent and temporary stormwater diversions will collect and divert a majority of the stormwater runoff around the facility to a natural drainage on the north side of the TSF.

At an average deposition rate of 680 dry short tons per day, an assumed settled density of 70 pounds per cubic foot ("pcf") and a total capacity of 3.2 million tons, the facility will have an approximate design life of 12.9 years. However, for the PFS mine production, only 1.76 million tons are planned to be delivered to the TSF. Therefore, only Stages 1, 2, and a portion of 3 will be required for the 7.25-year PFS mine life. Details of the PFS-level TSF design are presented in MacMahon et al. (2018).

18.5 Mining-Related Facilities

18.5.1 Cemented Rock Fill Plant

As discussed in Section 16.7.6.1, about 46% of the stopes mined will be backfilled with CRF. A CRF-MS07-1BN4 Eagle 7000 Rockfill plant was quoted by Simem Underground Solutions. The capital costs include:

- $836,807 for the basic CRF plant; and
- $119,500 for optional upgrades, including a winterization package and aggregate bins.

18.5.2 Waste-Rock Management Facility

The mined waste rock will ultimately be used as CRF material. This PFS assumes that the cement will lock in and neutralize potential acid generation when backfilled into the stopes; further work will be necessary to confirm this assumption.

During operation, a stockpile of waste rock will be managed on the surface to be used as CRF as needed. Due to the potential sulfides in the waste rock material, the temporary waste rock management facility ("WRMF") is assumed to be a lined facility. The composite lining system will consist of (from bottom to top) a six-inch to 12-inch thick, prepared subgrade, a 300-mil thick enhanced GCL, 80-mil HDPE geomembrane liner, and an 18-inch thick drainage layer. A collection system consisting of perforated piping will be installed within the drainage layer to collect any water coming in contact with the waste rock. The location of the WRMF, adjacent to the TSF, will allow the lining system to tie into the TSF lining system to provide continuous containment (see Figure 18.1). The WRMF collection pipe will gravity drain towards the TSF where it will be installed within the TSF drainage layer and ultimately outlet at the Reclaim Pond. The WRMF collection pipe will remain isolated from the TSF underdrain collection system so the water can be handled separately, if necessary.

All of the material from the WRMF will be used through the life of the mine. So, in final reclamation, the WRMF will have been removed.

18.5.3 Borrow Pit

The borrow pit will be on the east side of the project (Figure 18.1). Basalt material will be mined from the borrow pit for use in construction, backfill, and reclamation. The construction use will include ROM material for fill and TSF-embankment construction, as required. Backfill uses include both BF and CRF material for backfilling of underground stopes. The reclamation use will include capping material where required.

Borrow material will be generated using contract mining. Material for CRF will be crushed to minus 6-inches. During initial construction where more material is needed, the borrow mining will utilize larger equipment, while smaller equipment will be used during production when the amount of material required is reduced.

MDA estimated the initial ROM material cost to be $1.89 per ton. With the smaller equipment the cost will increase to $3.90/ton, with an estimated cost of $6.69/ton for crushed material sent to the CRF plant. This estimate was done using first principle costs and a 30% increase to assume the contractor. The contractor is expected to be on site about 6 to 9 months out of the year. Mobilization costs of $50,000 per year were included in the unit cost estimate.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Metal Pricing

No market studies have been undertaken for this PFS.

Gold-silver doré will be the commercial products from the Grassy Mountain operation. Gold-silver doré from mining operations is readily sold on the global market to commercial smelters and refineries, and it is reasonable to assume that doré from the Grassy Mountain project will also be salable.

To determine appropriate metal prices to be used for economic analysis and cutoff grades, MDA has considered spot prices in the months prior to the Effective Date of this report and reviewed current metal prices used in recent NI 43-101 technical reports. The metal prices used for the economic evaluation in this PFS have been chosen based on consensus prices as described by CIM guidelines in italics below, along with spot-price trends.

Consensus Prices
The use of consensus prices obtained by collating the prices used by peers or as provided by industry observers, such as analysts for example, may be used in some cases. This methodology has the advantage of providing prices that are acceptable to a wide body of industry professionals (peers). The disadvantage is that sometimes these predictions can be consistently wrong for reasons beyond the QP's control. These prices are generally acceptable for most common commodities, major industrial minerals, and some minor minerals."

Metal prices used in several recently published technical reports that estimate reserves ranged from about $1,250 to $1,300 per ounce of gold, with the most current reports using $1,300 per ounce.

Table 19.1 shows monthly gold prices compiled from Kitco.com. This shows that three-year and 12-month rolling-average gold prices have been on a fairly steady rise within the past year. The monthly gold prices have stayed above $1,300 per ounce for the last four months. Based on gold prices recently used for technical reports, and the current trend in gold prices, MDA has used $1,300 gold prices for the Grassy Mountain economic model.

Table 19.1 Kitco Monthly Gold Prices (USD/oz Au – May 2017 to April 2018)

Month / Yr	Average	High	Low	3-Yr Avg	1-Yr Avg
May-17	$1,245.00	$1,266.20	$1,220.40	$1,216.56	$1,258.63
Jun-17	$1,260.26	$1,293.50	$1,242.20	$1,216.04	$1,257.28
Jul-17	$1,236.22	$1,267.55	$1,211.05	$1,213.96	$1,248.86
Aug-17	$1,282.32	$1,318.65	$1,257.70	$1,213.58	$1,243.96
Sep-17	$1,314.98	$1,346.25	$1,282.55	$1,215.70	$1,243.04
Oct-17	$1,279.51	$1,303.30	$1,261.80	$1,217.28	$1,244.11
Nov-17	$1,282.28	$1,294.90	$1,267.20	$1,220.22	$1,247.97
Dec-17	$1,261.05	$1,291.00	$1,240.90	$1,221.86	$1,257.11
Jan-18	$1,331.67	$1,354.95	$1,311.00	$1,224.07	$1,268.70
Feb-18	$1,331.52	$1,352.45	$1,314.10	$1,226.97	$1,276.79
Mar-18	$1,324.66	$1,352.40	$1,307.75	$1,231.03	$1,284.59
Apr-18	$1,334.74	$1,351.45	$1,313.20	$1,234.83	$1,290.35

Monthly average silver prices are presented in Table 19.2. The table shows that the three-year and 12-month rolling-average silver price has been on a relatively steady decrease. The 12-month average at the end of April 2018 was $16.77 per ounce. Based on the silver price trend, MDA used a $16.75 per ounce silver price for the Grassy Mountain economic model.

Table 19.2 Kitco Monthly Silver Prices (USD/oz Ag – May 2017 to April 2018)

Month / Yr	Average	High	Low	3-Yr Avg	1-Yr Avg
May-17	$ 16.76	$17.31	$16.22	$ 16.92	$ 17.89
Jun-17	$ 16.96	$17.60	$16.47	$ 16.84	$ 17.87
Jul-17	$ 16.14	$16.79	$15.22	$ 16.71	$ 17.55
Aug-17	$ 16.91	$17.60	$16.13	$ 16.63	$ 17.33
Sep-17	$ 17.45	$18.21	$16.82	$ 16.60	$ 17.17
Oct-17	$ 16.94	$17.41	$16.58	$ 16.59	$ 17.11
Nov-17	$ 17.01	$17.15	$16.57	$ 16.62	$ 17.07
Dec-17	$ 16.16	$16.87	$15.71	$ 16.62	$ 17.05
Jan-18	$ 17.17	$17.52	$16.98	$ 16.62	$ 17.08
Feb-18	$ 16.66	$17.19	$16.35	$ 16.61	$ 16.98
Mar-18	$ 16.47	$16.65	$16.25	$ 16.62	$ 16.89
Apr-18	$ 16.61	$17.20	$16.28	$ 16.63	$ 16.77

19.2 Contracts

Paramount's land and royalty obligations are summarized in Section 4.3. There are no other contractual obligations attributed to the project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

EM Strategies, Inc., a permit-acquisition strategy and government-relations consulting firm, provided the following information on environmental considerations, permitting, and social or community impacts.

As of the Effective Date of this report, Paramount's wholly-owned subsidiary, Calico Resources USA Corp. ("Calico"), is in the process of acquiring the necessary local, state, and federal permits for the development of an underground-mining and mill-processing operation in southeastern Oregon.

20.1 Introduction

The permitting activities by Calico for the Grassy Mountain mine began in 2012 with engagement with the State of Oregon and the collection of baseline data. The baseline-data collection is ongoing. In addition, Calico submitted an updated Plan of Operations ("Plan") to the BLM in September 2017. The BLM has determined that additional information is necessary in order to find that the Plan is complete (see Section 20.3.2). The Plan outlines approximately 265 acres of proposed surface disturbance as summarized in Table 20.1.

Table 20.1 Proposed Surface Disturbance for the Grassy Mountain Mine

Component	Public Acres	Private Acres	Total Acres
Portal Area	0.0	3.3	3.3
Waste Rock Storage Area	7.5	0.0	7.5
Tailings Storage Facility (TSF)	101.2	0.0	101.2
Process/Administration Area[1]	15.0	2.5	17.5
Laydown/Yard Areas	17.2	0.0	17.2
Roads	13.5	1.8	15.3
Water Tank and Road	0.0	0.5	0.5
Water Wells and Water Pipeline[2]	11.8	0.8	12.6
Fence[3]	15.4	0.0	15.4
Borrow Areas	44.7	3.0	47.7
Diversion Ditches and Sediment Basins	8.6	0.0	8.6
Growth Media Stockpiles	8.3	0.0	8.3
Landfill	0.7	0.0	0.7
Exploration[4]	5	5	10
Total	**248.9**	**16.9**	**265.8**

[1]This includes the mill, refining plant, administrative building, parking lot, security building, mining contractor yard, reagent storage, assay laboratory, and substation.
[2]Includes the water supply pipeline at 16,164 feet with a 30-foot construction disturbance width and well locations each at 0.25 acres.
[3]Includes the perimeter fence at 22,358 feet with a 30-foot construction disturbance width.
[4]The actual location of the exploration activities within the Project Area is currently unknown and is assumed to be on all public lands.

The project includes the following activities and facilities:

- One underground mine;
- One waste-rock storage area;
- One carbon-in-leach processing plant;
- Three borrow-pit areas;
- One TSF;
- Ore stockpile;
- One reclaim pond;
- A water-supply well field and pipeline, associated water-delivery pipelines, and power;
- A power substation and distribution system;
- Access and haul roads;
- Ancillary facilities that include haul, secondary, and exploration roads; truck workshop, warehouse, storm-water diversions, sediment-control basins, reagent and fuel storage, storage and laydown yards, explosives magazines, freshwater storage, monitoring wells, meteorological station, administration/security building, borrow areas, growth-media stockpiles, landfill, and solid and hazardous-waste management facilities; and
- Reclamation and closure, including the potential development of an E Cell for the TSF.

20.2 Permitting History

Permitting activities for the Grassy Mountain project have spanned 30 years. During the late 1980s, Atlas collected a wealth of geologic, mine engineering, civil engineering, and environmental baseline data to support an historical feasibility study that was completed in 1990. During 2012 to 2016, Calico began the permitting process for an underground-mining operation at Grassy Mountain. Since the acquisition of Calico by Paramount in 2016, the permitting process has continued with the Oregon Department of Geology and Mineral Industries ("DOGAMI"), Malheur County, and the BLM.

20.3 Project Permits

The project will require the following major environmental permits to construct, operate, and close: 1) a Plan from the BLM; 2) a DOGAMI Consolidated Permit for Mining Operations; 3) an Oregon Department of Environmental Quality ("ODEQ") Chemical Mining Permit; 4) water rights from the Oregon Department of Water Resources ("ODWR"); 5) an Air Quality Operating Permit ("AQOP") with the ODEQ; and 6) a Special Use Permit from Malheur County.

20.3.1 State of Oregon Permit Processing

Calico entered into a Memorandum of Understanding for Cost Recovery ("MOU") with the DOGAMI on November 3, 2014. The MOU provides a mechanism whereby Calico, as the project proponent, agrees to reimburse DOGAMI and other primary state agencies for their involvement in processing permit applications for the Grassy Mountain project. In addition, DOGAMI has hired a consulting firm to provide expertise that is not available from the staff of the various agencies that are involved with the permitting process. The key components of the Calico permitting program with the State of Oregon are as follows:

- Environmental baseline studies for all resource categories described in Oregon's Chapter 632, Division 37 Chemical Process Mining Rules;
- Meeting all requirements of Division 37 Rules that include, but are not limited to: 1) preparation of a Consolidated Permit Application; 2) obtaining all necessary federal, state, and local permits and authorizations; and 3) satisfying any potentially applicable NEPA requirements; and
- Implementing a proactive community involvement and consultation process including: 1) local-hire preference; 2) local contracting and purchasing where practicable; and 3) mine-worker job training to provide an experienced workforce.

A key authorization permit that will be needed is the permit for Chemical Processing Mining, as required under Chapter 632, Division 37, 1991 Oregon Laws (§632-037-0005). The Consolidated Permit also requires approval by ODEQ under Division 43, Chemical Mining Rules (OAR 430-043-000), which address other environmental stipulations. "Chemical Process Mining" means a mining and processing operation for metal-bearing ores that uses chemicals to dissolve metals from ore. The Calico processing facility will employ cyanide in the metallurgical process. The Division 37 Rules provide a well-defined regulatory pathway with definitive permitting requirements and timelines.

Calico has filed multiple Notices of Intent ("NOI"s) under Division 37, which initiate the state permitting process and baseline-data collection. The reason for the multiple NOIs is that the scope of the operation, as well as the configuration of the Project Area, have changed through the project history. Each change requires the submittal of a new NOI and a re-initiation of the permitting process. In addition, the initial NOI filing was done to initiate the agency Division 37 permit process and provide for public notice that the project is proceeding into the permitting phase. As part of initiating the public notification, an interagency Technical Review Team ("TRT") has been organized to provide interdisciplinary review of technical permitting issues for Oregon's Consolidated Permitting Process. This TRT has convened numerous times and accepted the NOIs.

In addition, DOGAMI administrators and the TRT have reviewed and approved the *Calico Resources Environmental Baseline Work Plans Grassy Mountain Mine Project,* which was filed on May 17, 2017. In July 2017, a "Notice of Prospective Applicant's Readiness to Collect Baseline Data" was issued to Calico by DOGAMI. The environmental baseline program is currently being implemented by Calico, and this program is expected to be completed by the fourth quarter of 2018. Baseline studies of air quality, grazing, recreation, and visual resources have already been completed and accepted by the TRT. This information is supplemental to an earlier database developed by Atlas and Newmont.

With the TRT approval of the work plans, Calico is now authorized to prepare the Division 37 Consolidated Permit Application for the Grassy Mountain Gold mine. However, the application cannot be submitted until all the baseline study reports have been accepted by the TRT. The application preparation is being initiated concurrent with completion of the baseline studies. This single application, as required under Oregon Laws, will include the following elements:

- General information;

- Existing environmental baseline data;
- Operating plan;
- Reclamation and closure plan; and
- Alternatives analysis.

Upon completion of the Consolidated Permit Application, a completeness review will be conducted by the TRT, and a Notice to Proceed with the preparation of draft permits will be issued by DOGAMI. This notice will also involve a directive by DOGAMI to use the third-party contractor to prepare an Environmental Evaluation ("EE"), which is to be issued at least 60 days prior to the issuance of any draft permits. This EE is not a federal NEPA requirement, it is a State of Oregon requirement that includes: 1) impact analysis; 2) cumulative-impact analysis, and 3) alternatives analysis (OAR 632-037-0085).

Concurrent with this assessment, DOGAMI will also use the contractor to prepare a Socioeconomic Analysis. This analysis will identify major and reasonably foreseeable socioeconomic impacts on individuals and communities located in the vicinity of the proposed mine. In particular, the analysis will describe impacts on population, economics, infrastructure, and fiscal structure (OAR 632-037-0090).

This process for permit review and approval will also involve a consolidated public hearing on all draft permits, as well as the draft operating permit. Other applicable state and federal permits may include, but are not limited to the following:

- Permits to appropriate groundwater or surface water, or to store water in an impoundment (ORS 537.130, ORS 537.400, and ORS 540.350);
- Water Pollution Control Facility (ORS 468B.050);
- Storm Water Pollution Prevention Plan (EPA);
- Air Quality Permits (ORS 468A.040);
- Solid Waste Disposal Permit (ORS 459.205);
- Permit for Placing Explosives (ORS 509.140);
- Hazardous Waste Storage Permit (OAR 340-102-0010);
- Land Use Permit (OAR Chapter 632, Division 001); and
- Any other State of Oregon permits, if applicable and required under Division 37

The State of Oregon has retained a project manager to oversee the permitting program and lead the review team. A Project Coordinating Committee ("PCC") was also formed for the purpose of sharing information, further coordinating the federal, state, and local permitting requirements, optimizing communication, facilitating the regulatory process, and avoiding duplicative effort. The PCC has convened formally and conducted a series of public meetings in the cities of Ontario and Bend, Oregon. These meetings were attended by agencies, public officials, project supporters, and non-governmental organizations ("NGO"s).

Division 37 mandates DOGAMI to manage and facilitate the regulatory permitting process. It requires that a series of public meetings are held, to be coordinated by DOGAMI or its contractor. The PCC is charged with gathering comments from the public regarding the specifics of the project. DOGAMI acts as the facilitating state agency and state clearinghouse for the mine-permitting process. It is the applicant's responsibility to secure other needed state permits, such as air-pollution control, storm-water pollution prevention plan, and land-use permits as may be required. However, the Division 37 process is designed to promote a consolidated permitting pathway.

DOGAMI coordinates with the other agencies to avoid duplication on the part of the applicants and related agency requests. DOGAMI is also responsible for reviewing mine-operating plans and issuing reclamation permits. It establishes reclamation-bond amounts for the project, working closely with Calico. As part of DOGAMI's permitting process, it also requires the preparation of detailed environmental baseline-data collection work plans that direct the inventorying of the various existing natural and human resources that may be impacted by the project. These include air quality, surface and groundwater quality and hydrology, vegetation, fisheries, wildlife, socioeconomic, historical/cultural, and other resource categories.

The basic information for a Division 37 Consolidated Permit Application involves:

- Determining existing environmental baseline conditions;
- Providing an operating plan (mine plan and reclamation/closure plan);
- Providing an alternatives analysis;
- Providing an environmental evaluation;
- Providing a socio-economic impact analysis;
- Developing a plan to minimize pollution and erosion;
- Protecting fish and wildlife during operations and closure (fish and wildlife standards);
- Providing a water balance;
- Establishing financial-assurance requirements; and
- Inclusion of all other state, federal, and local permit applications required under Division 37.

DOGAMI officials have indicated that the Division 37 timeline for this requirement can be expected to be about one year from the date that a "complete application" (as deemed complete by DOGAMI) is submitted for the regulatory process to be concluded, and a permit issued.

20.3.2 BLM Plan of Operations and Federal Processing

At this time, it is not contemplated that the Grassy Mountain project will require either a federal National Pollutant Discharge Elimination System Permit ("NPDES") from the U.S. Environmental Protection Agency ("EPA") or U.S. Army Corps of Engineers 404 Dredge and Fill Permit. The Grassy Mountain project does not involve a discharge to Waters of the U.S., nor does it involve construction in wetlands or placement of dredge tailings or fill material into waters of the U.S. However, the project will require a Plan of Operation approval from the BLM.

The Plan Application is submitted to the BLM for any surface disturbance in excess of five acres. The Plan Application describes the operational procedures for the construction, operation, and closure of the project. As required by the BLM, the Plan Application includes a waste-rock management plan, quality-assurance plan, a storm-water plan, a spill-prevention plan, reclamation plan, a monitoring plan, and an interim-management plan. In addition, a reclamation report with a Reclamation Cost Estimate ("RCE") for the closure of the project is required. The content of the Plan Application is based on the mine-plan design and the data gathered as part of the environmental baseline studies. The Plan Application includes all mine and processing design information and mining methods. The BLM determines the completeness of the Plan Application, and a completeness letter is submitted to the proponent. The RCE is reviewed and the bond is determined prior to the BLM issuing a decision record on the Plan Application.

Submittal of the Plan Application took place in September 2017. The BLM has requested additional details, which are expected to be provided to the BLM by the fourth quarter of 2018. However, several key baseline reports still need to be completed for inclusion in the Plan Application. These reports have yet to be, or are just being, reviewed by the relevant agencies. The BLM will likely need to complete their review of the baseline reports in the Plan Application and approve the final version of these reports.

20.3.3 National Environmental Policy Act ("NEPA")

The NEPA process is triggered by a federal action, and, as is the case with the Grassy Mountain project, the BLM issuance of a completeness letter for the Plan will be the trigger for the federal action. The NEPA review process is completed by either an Environmental Assessment ("EA") or an Environmental Impact Statement ("EIS"). The BLM has stated that the NEPA review process for this project will be an EIS.

The EIS process is conducted in accordance with NEPA regulations (40 CFR 1500 et. seq.), BLM guidelines for implementing the NEPA in BLM Handbook H-1790-1 (updated January 2008), and BLM Washington Office Bulletin 94-310. The intent of the EIS is to assess the direct, indirect, residual, and cumulative effects of the project and to determine the significance of those effects. Scoping is conducted by the BLM and includes a determination of the environmental resources to be analyzed in the EIS, as well as the degree of analysis for each environmental resource. The scope of the cumulative analysis is also addressed during the scoping process. Following scoping and baseline information collection, the Draft EIS is prepared for the BLM by a third-party contractor. When the BLM determines the Draft EIS is complete, it would then be submitted to the public for review. Comments received from the public would be incorporated into a Final EIS, which would in turn be reviewed by the BLM and the public prior to the BLM issuing a record of decision ("ROD"). Under an EIS there can be significant impacts. The project proponent pays for the third-party contractor to prepare the EIS, and also pays recovery costs to the BLM for any work on the project by BLM specialists.

The BLM is requiring the preparation of an EIS to comply with the NEPA for the Grassy Mountain project. Under the 2018 Secretarial Order 3355, the EIS must be completed in 365 days (from the NOI publication in the Federal Register to the signing of the ROD) and must be less than 150 pages in length, unless a Department of Interior wavier is obtained, which then allows for 300 pages.

20.4 Environmental Study Results and Known Issues

As previously discussed, the deposit and property have been known for over 30 years. However, there have been long periods of non-operation. There are no known, ongoing environmental issues with any of the regulatory agencies. Calico has been conducting baseline-data collection for six years for environmental studies required to support the Plan Application and the state and federal permitting process. Results indicate limited biological and cultural issues, air-quality impacts appear to be within State of Oregon standards, traffic and noise issues are present but at low levels, and socioeconomic impacts are positive.

20.5 Waste Disposal, Monitoring, Water Management

Waste-rock characterization tests have been conducted. Results indicate that the waste rock and mineralized rock are generally reactive, acid generating, and have the potential to leach metals. As a result, waste-rock and tailings management are expected to be key issues in the permitting of the mining operation.

20.6 Social and Community Issues

Social and community impacts have been, and are being, considered and evaluated for the various Plan amendments performed for the project in accordance with the NEPA and other federal laws, as well as the State of Oregon Socioeconomic Analysis. Potentially affected Native American tribes, tribal organizations, and/or individuals are consulted during the preparation of all Plan amendments to advise on the proposed projects that may have an effect on cultural sites, resources, and traditional activities.

The most recent planning by Malheur County, Oregon, is consulted during the preparation of Plan amendments. Potential community impacts to existing population and demographics, income, employment, economy, public finance, housing, community facilities, and community services are evaluated for potential impacts as part of the State of Oregon and the NEPA process.

There are no known social or community issues that would have a material impact on the project's ability to extract mineral resources. Identified socioeconomic issues (employment, payroll, services and supply purchases, and state and local tax payments) are anticipated to be positive.

20.7 Mine Closure

A closure plan has not yet been completed and there are no current estimates for the reclamation bond. Closure costs are estimated in Section 21.3.1. The anticipated closure scenario would include plugging the mine portal, while for the tailings closure the approach would consist of fluid management through evaporation, covering the tailings with an operational layer of waste rock or approved closure material, a synthetic liner and growth media, and then revegetation. The process of managing the solutions from the tailings drain-down would require multiple years. Residual tailings drainage would likely be managed with evaporation ponds/cells. The waste-rock dump would be moved to the tailings facility, although no waste rock is envisioned to exist at the end of mining operations in this PFS. Other facilities would be regraded, covered with growth media, and revegetated. The closure scenario for the tailings would likely result in conditions that require long-term management of the evaporation ponds/cells and associated ancillary facilities at the site, which will require a financial instrument to cover those cost into the future.

20.8 Environmental and Permitting Risks and Opportunities

As with almost all mining projects, there are inherent risks and opportunities related to the final outcome of the project. Most of the risks related to environmental and permitting are based on the uncertainty of the permitting program and the timing to obtain all necessary permits and authorizations. Other environmental and permitting risks can involve new regulations, tightening of standards for air or water quality, and legal challenges.

Subsequent high-level engineering studies and environmental baseline studies are required to further define these risks and identify opportunities, as will be conducted at the feasibility level. To facilitate project permitting and development for the PFS and permitting programs, and to design a sustainable project and reduce environmental risks, Calico has adopted the following environmental principles for the project:

- Protect local surface and groundwater quality and quantity by applying best mining practices (“BMP”s) and water treatment, as necessary;
- Confirm the presence of potential threatened and endangered or sensitive amphibians, wildlife, or plant species at the site;
- Effectively manage all related mine waste, including lining the tailings storage facility, waste-rock underground as backfill, and segregation and selective handling of waste rock as necessary;
- Reduce the carbon footprint for the project by processing the gold concentrate on site;
- Conduct environmental monitoring to ensure compliance with all applicable state, federal, and local laws, regulations, and ordinances;
- Transport all fuel to the mining operation according to accepted transport and spill prevention and response procedures developed specifically for the project;
- Integrate pro-active wildlife habitat mitigation and enhancement proposals with an environmentally responsible reclamation and closure plan;
- Provide adequate financial assurance for implementing an effective reclamation and closure plan to ensure long-term protection and rehabilitation of the mine site; and
- Implement a responsible community and statewide public affairs program to further open communications, maximize local job opportunities and involvement, and meet environmental justice requirements for the project.

Collectively, these objectives or environmental principles will guide project development. They will also serve to reduce risk and enhance related project opportunities.

21.0 CAPITAL AND OPERATING COSTS

MDA has compiled the capital and operating costs for Section 21. Contributions to this section have been made by Ausenco for mining and processing and Golder for the TSF. Contributions to infrastructure costs have also been made by Ausenco.

Table 21.1 summarizes the project capital costs. Total initial capital is estimated to be $109.9 million. Sustaining costs of $1.1 million have been estimated, which includes a refund of the cash contribution toward surety bonding. The total LOM capital cost estimate is $110.9 million. Note that negative quantities of capital refer to bonding and working capital that is returned to the cash flows as part of the sustaining capital.

Table 21.1 Capital Cost Summary (K USD)

	Initial	Sustaining	Total
Mining Capital	$ 2,928	$ 1,399	$ 4,328
Buildings & Site Infrastructure	$ 12,787	$ -	$ 12,787
Process Capital	$ 25,935	$ -	$ 25,935
Tailings Storage Facility	$ 8,215	$ 5,026	$ 13,241
Plant & Infrastructure Indirect	$ 9,691	$ -	$ 9,691
Off-Site Power and Access	$ 10,328	$ -	$ 10,328
Subtotal Infrastructure & Equipment	***$ 69,885***	***$ 6,426***	***$ 76,311***
Mine Development	$ 7,640	$ 1,799	$ 9,439
Mine Pre-Production	$ 4,598	$ -	$ 4,598
Subtotal Mine Pre-Production	***$ 12,238***	***$ 1,799***	***$ 14,037***
Owner's Capital	$ 7,005	$ (4,142)	$ 2,863
Other Capital	$ 2,092	$ 166	$ 2,259
Working Capital	$ 4,464	$ (4,464)	$ -
Subtotal Other Capital	***$ 13,561***	***$ (8,439)***	***$ 5,122***
Subtotal	***$ 95,684***	***$ (214)***	***$ 95,470***
Contingency	$ 14,195	$ 1,282	$ 15,477
Total Capital	***$ 109,880***	***$ 1,067***	***$110,947***

Table 21.2 shows the estimated operating costs for the LOM. Total operating costs are $185.8 million. This results in a $105.63 cost per ton of ore processed or $528.12 per gold-equivalent ounce produced.

Table 21.2 Operating Cost Summary (USD)

	Life-of-Mine Cost (K USD)	Cost/ton Processed	Cost per Oz Au *
Mining	$ 114,969	$ 65.37	326.81
Processing	$ 48,456	$ 27.55	137.74
Rehandle	$ 875	$ 0.50	2.49
G&A	$ 15,275	$ 8.68	43.42
Reclamation	$ 6,213	$ 3.53	17.66
Total	$ 185,789	$ 105.63	528.12

** Cost per ounce includes silver revenue as credits*

21.1 Capital Costs

The following subsections describe the capital-cost estimates for mining, process, tailings, and site and owner capital.

21.1.1 Mining Capital

The mining capital estimate has an accuracy range of -20% and +35%. The basis of the mining capital cost estimate is as follows:

- Capital items were estimated through the preparation of a list containing individual capital items to be acquired and prices obtained from quotes or benchmark information from Ausenco´s database.
- Mine-development quantities obtained from the mine production schedule and unit costs obtained from cost model developed by Ausenco.
- Pre-production cost for administration and services needed during pre-production period. This cost was estimated from first principles for labor, consumables, and equipment-running cost through the cost model developed by Ausenco.

Mining capital costs are summarized in Table 21.3.

Table 21.3: Mining Capital Costs

	Initial KUSD	Sustaining KUSD	Total KUSD
Mining capital	2,928	1,399	4,328
Mine development	7,640	1,799	9,439
Pre-production	4,598		4,598
Total capital	**15,166**	**3,198**	**18,365**

Mining capital includes infrastructure, mine development, and minor items. Infrastructure includes the mine portal, ventilation items, dewatering items, power supply and power reticulation items, explosive magazine, and communications. Minor items include small vehicles for transportation of personnel and material, surveying equipment, tools, mobile air compressors, underground lamps, computers and software, and other minor items. Mine development includes the main decline ramp and vertical developments.

Pre-production involves labor, services, and administration costs incurred during the pre-production period. This period covers the commencement of work at the mine site up to the first ore fed to the process plant.

Due to the relative low usage of the mining equipment fleet, as indicated by the total operating hours shown in Table 16.12, there is no sustaining capital allocated for the mine equipment units. This is considered a risk by Mr. Caro and should be examined in more detail as part of a Feasibility Study.

21.1.2 Process Capital

The process plant design incorporates a two-stage crushing circuit, a grinding circuit, a CIL circuit, a standard ADR gold-recovery circuit, cyanide detoxification, and tailings filtration. Wet tailings will be pumped to the lined TSF. The process capital cost estimate was prepared based on new 2018 budget quotations for major mechanical equipment, platework, reagents, and certain ancillary structures. These PFS estimates have been developed from bulk take-offs on detailed earthworks, concrete, structural and internal steel, and major pipelines.

Electrical and instrumentation equipment lists were prepared based on recent projects with similar processes and plant size. Factors have been applied to cover in-plant electrical distribution, instrumentation, piping, and allowances for minor mechanical equipment and minor platework. Estimates for air, water, and fire-protection utilities were based on Ausenco's database of pricing. Capital costs for the ore crushing and process plant components required to support plant operations are shown below in Table 21.4.

The process capital cost shown in Table 21.1 includes an estimated $565,000 for tailings management based on material take-offs for tails-discharge pipelines and decant-water return pipelines.

Table 21.4 Direct Processing Capital Costs

Process Component	Total Cost (USD Million)
Crushing	2.7
Grinding	3.9
Gravity Separation	1.0
Carbon-In-Leach	4.5
Intensive Cyanidation	0.7
ADR Plant	6.2
Cyanide Detox	0.4
Gold Room	0.3
Tailings Thickening	0.2
Reagents	1.9
Plant Building	0.8
Plant Water Systems	1.9
Plant Air Systems	0.8
Process Control System	0.6
Total Direct Costs	**25.9**

Commented [MG5]:

21.1.2.1 Water Storage and Treatment

Water storage and treatment requirements were estimated and include the following tanks for erection:

- 1 – 30-foot diameter by 42-foot-tall raw-water carbon-steel tank;
- 1 – 20-foot diameter by 20-foot-tall process-water carbon-steel tank;
- 1 – 13-foot diameter by 13-foot-tall potable-water carbon-steel tank; and
- 1 – 7.5-foot diameter by 8.0-foot-tall water-treatment plant (gray water).

21.1.2.2 Power Distribution

A 4.16 kV overhead powerline was included to feed the process plant, ancillary buildings, and tailings area. The powerline will run a total of 5,740 feet, which branches off the main 15 kV powerline from the mine site. The main power-supply equipment includes the following:

- 1 – 14,400 V to 4160-V 6-MVA transformer (main feed);
- 2 – 4,160 V to 480-V 2.0-MVA transformers (process plant area);
- 1 – 4,160 V to 480-V 0.75-MVA transformer (crushing area).

21.1.2.3 Fuel Storage and Distribution

Fuel storage and distribution requirements were estimated and include the following equipment:

- Fuel tank, pump, and station for mining equipment; and
- Fuel tank, pump, and station for mobile equipment.

21.1.2.4 Ancillary Buildings

Ancillary buildings costs are summarized in Table 21.5 and include the following:

- Process Plant
  - Gold recovery (pre-engineered building);
  - Electrical room (pre-engineered building);
  - Reagents (roof only);
  - Air services (roof only);
- Administration, plant dry, and offices (extra wide Atco trailers);
- Plant maintenance and warehouse (sprung building);
- Truck shop (pre-engineered building);
- Mine dry and offices (extra wide Atco trailers); and
- Assay laboratory and equipment (modular building).

Table 21.5 Process Buildings Direct Costs

Direct Costs	Total Cost (USD Million)
Process Plant	0.7
Admin, Plant Dry & Offices	0.6
Plant Maintenance & Warehouse	0.2
Mine Dry & Offices	0.6
Truck Shop	1.3
Assay Lab	1.5
Total Process Buildings Costs	**5.1**

Estimates of structural steel quantities were prepared by Ausenco based upon calculations derived from general-arrangement drawings and sketches. Site buildings were based upon general-arrangement drawings and the site plan. Supply and install rates were included based upon a cost-per-square-footage from Ausenco's historical data. Overhead cranes are part of the mechanical equipment list, and concrete and internal support steel are part of the engineer's material take-offs. In addition to the building structures, the estimated costs include the supply of the buildings' electrical equipment, fittings, and furnishings. The cost to supply power and water services to the buildings was included with the electrical and water supply and distribution costs.

Construction and permanent camps are not required as labor forces will reside in nearby cities.

21.1.2.5 Mobile Equipment

The mobile fleet required to support plant operations is shown in Table 21.6.

Table 21.6 Process Mobile Equipment Capital

Description	Total Cost (USD Million)
Direct Mobile Fleet Costs	0.4
Grader (Road Maintenance)	0.2
15 Passenger Crew Vans	0.1
Ambulance/Rescue Vehicle	0.2
Pipe Fusing Machine - HDPE Pipe 4" To 16"	0.1
Forklift	0.1
4WD Twin Cab Utility - 1/2 Ton vehicles	0.1
4WD Twin Cab Utility - 3/4 Ton vehicles	0.4
Total Direct Costs	**1.6**

21.1.2.6 EPCM Services

EPCM services costs cover such items as engineering and procurement services, construction-management services (site based), project office facilities, IT, staff transfer expenses, secondary consultants, field inspection and expediting, corporate overhead, and fees. EPCM services were calculated to be $5.1 million, which equates to 13.5% of the direct costs. An inclusion of 8.0% of this contingency amount was added for general expenses, as well as accommodation in Vale, Ontario and Boise, Idaho for the EPCM team and vendor and commissioning consultants during construction. This equates to $0.4 million.

21.1.2.7 Contingency

Contingency is a cost element to accommodate unknown items that cannot be properly defined at the current stage of the project, but which are expected to occur within the defined scope. The project contingency is meant to cover the normal inadequacies that are inherent in the estimate due to the dynamic nature of project engineering and construction. It is assumed that the contingency will be spent. The contingency allowance specifically excludes costs arising from scope changes, budget held as management reserve by Paramount, and all other items that are excluded from the capital-cost estimate.

The overall recommended contingency was calculated to be $14.2 million, which equates to 15.6% of total installed costs (total direct costs, indirect costs, EPCM, and owner's costs). Contingency attributed to individual contributors to the capital-cost estimate is as follows:

- Ausenco (process and on-site infrastructure) - 20%;
- Mining - 15%;
- Process plant and on-site infrastructure - 20%;
- Tailings management - 15%;
- Off-site infrastructure - 15-50%; and
- Other - 15%.

The amount of risk is assessed with due consideration of the preliminary level of design work, the manner in which pricing was derived, and the preliminary nature of the plan for project implementation.

21.1.2.8 Exclusions

The following items are not considered in the Class 4 PFS estimate:

- Senior finance charges;
- Residual value of temporary equipment and facilities;
- Residual value of any redundant equipment;
- Cost of any downtime;
- Cost of any isolation and de-isolation of plant and equipment;
- Environmental approvals;
- This study or any further project studies;
- Force majeure issues;
- Scope changes;
- Special incentives (schedule, safety, or others);
- Allowance for loss of productivity and/or disruption due to religious, union, social, and/or cultural activities;
- Financial analysis;
- Management reserve (project contingency);
- Owner's escalation costs;
- Owner's foreign-exchange exposure;
- On-site batch plant;
- Meal allowances; and

- Travel time

21.1.3 Tailings Storage Facility Capital

Golder caution to readers: In this Item, all estimates and descriptions related to capital and operating cost estimates are forward-looking information. There are many material factors that could cause actual results to differ materially from the estimates, forecasts or projections set out in this item. Some of the material factors include differences from the assumptions regarding the following: *mining and processing methods, labor and overhead costs, locations and designs of facilities, cost escalation and inflation, production and productivity assumptions, and project schedule.* The material factors or assumptions that were applied in drawing the conclusions, forecasts and projections set forth in this Item are summarized in this report. Any significant differences from these factors or assumptions will have material impacts on the estimates, forecasts and projections set forth in this report.

The TSF design and construction costs were estimated by Golder for the PFS. The first stage provides the basic infrastructure to be able to operate the TSF, including underdrains, channels, and underdrain reclaim pond. The following three stages provide incremental additional capacity through dam and impoundment expansions. The four total stages of the design were estimated to provide the following incremental tailings storage capacity:

- Stage 1 = 0.9 million tons (Years 1 to 2);
- Stage 2 = 0.7 million tons (Years 2 to 3);
- Stage 3 = 1.0 million tons; (Years 3 to 6);
- Stage 4 = 0.6 million tons (not required for PFS mine production); and
- Total = 3.2 million tons (1.76 million required for PFS mine production).

Golder estimated costs for four stages of TSF development. However, the PFS includes a requirement to store only 1.76 Mt of tailings. Through Stage 2, the TSF only provide storage capacity for approximately 1.6 million tons. Stage 3 as designed provides an additional 1.0 million tons of tailings storage. Therefore, the cost estimates for the PFS include Stages 1, 2, and a portion of Stage 3 required to contain the 1.76 million tons of tailings estimated from the PFS mine production. Thus, the stage 3 capital cost was factored down by MDA by the amount of remaining capacity required for the PFS mine production.

MDA used Golder's construction cost estimate to develop the yearly capital estimates considering the timing required for construction of the TSF expansions. The cash flow model assumes that Stages 1, 2, and 3 will be constructed in years 1, 3 and 5, respectively. Table 21.7 shows the capital cost applied for the TSF in the cash-flow model.

Table 21.7 TSF Capital Cost by Year
(prepared by MDA)

	Units	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5 *	Yr 6	Total
Site Preparation	K USD	$ 409			$ 347		$ 79		$ 835
Tailings Impoundment Earthworks	K USD	$ 2,764			$ 1,888		$ 429		$ 5,081
Impoundment Geosynthetics	K USD	$ 2,638			$ 1,367		$ 310		$ 4,315
Underdrain Pipe System	K USD	$ 83			$ 66		$ 15		$ 164
Underdrain Collection Pond & Channel	K USD	$ 211			$ -		$ -		$ 211
Roads & Stormwater Improvements	K USD	$ 470			$ 105		$ 24		$ 599
Waste Rock Dump	K USD	$ 648			$ -		$ -		$ 648
Design & Construction Engineering	K USD	$ 992			$ 323		$ 73		$ 1,388
Subtotal Estimated Capital Cost	K USD	$ 8,215	$ -	$ -	$ 4,097	$ -	$ 930	$ -	$ 13,241
15% Contingency	K USD	$ 1,232	$ -	$ -	$ 614	$ -	$ 139	$ -	$ 1,986
Total Estimated Capital Cost	K USD	$ 9,447	$ -	$ -	$ 4,711	$ -	$ 1,069	$ -	$ 15,228

**Yr 5 costs are based on a linear reduction of Golder's Stage 3 incremental capital cost estimate. Reduction performed by MDA to reduce the Stage 3 incremental storage capacity from 1.0 million tons to 179,000 tons.*
*** Table 21-7 as presented was developed by MDA for Stages 1, 2, and part of 3 using the Golder's complete capital cost estimate for Stages 1 through 4.*

21.1.4 Site and Owner Capital

Site and owner's capital includes categories for buildings, site infrastructure and administration, power distribution, access road, environmental costs, administration light vehicles, safety and security, and owner's costs. The total initial cost estimate for these items is $30.6 million, with total capital, including sustaining capital, estimated at $26.6 million. Note that the negative sustaining capital is due to the return of bonding capital as described below.

Table 21.8 Site and Owner Capital

	Initial	Sustaining	Total
Building & Site Infrastructure	$ 12,787	$ -	$ 12,787
Power Distribution	$ 7,026	$ -	$ 7,026
Access Road	$ 3,302	$ -	$ 3,302
Environmental	$ 200	$ -	$ 200
Administration Light Vehicles	$ 166	$ 166	$ 333
Site and Administration	$ 45	$ -	$ 45
Safety & Security	$ 92	$ -	$ 92
Owner's Capital	$ 7,005	$ (4,142)	$ 2,863
Total Site and Owner Capital	$ 30,623	$ (3,975)	$ 26,647

Buildings and site infrastructure were estimated by Ausenco. This includes $2.375 million for site development and earthworks, $3.389 million for power distribution, $6.062 million for buildings, $0.705 million for fuel storage and distribution, and $0.256 million for information technology and communications.

Power distribution costs have been based on quotations from vendors and include approximately 23 miles of powerline, transformers, and voltage regulators. Engineering, freight, and installation costs of leased power-generation equipment to be used for initial construction and later backup power are also included.

The access road costs were estimated based on a contractor quotation for 17 miles of access road improvements. Environmental capital costs include monitor-well installation and perimeter fencing.

Owner's capital includes bonding capital and G&A costs incurred during construction. Bonding is assumed to be issued using a surety bond in which the company would be required to provide two-thirds of the reclamation costs ($4.142 million) as collateral. An operating cost is added in the amount of 2.0% of the bond amount per year, compounded monthly. After two years of proven production, the collateral would be returned to Paramount, which accounts for the negative amount shown in Table 21.8.

21.2 Operating Costs

The following subsections describe the operating cost estimates for mining, process, tailings, general and administration, and reclamation.

21.2.1 Mine Operating Costs

The total mining cost per ton of processed ore over the LOM is estimated at $65.37 per ton processed.

Mining costs were estimated on a detailed basis that includes personnel salaries, main supplies items, equipment running needs based on the monthly schedule, cost of support services, and mine administration cost. The mining costs are listed in Table 21.9. The cost of surface ore rehandling was estimated by the mining team and allocated as a separate item in Table 21.2.

Table 21.9 Mining Operating Costs

Cost Summary	Operating Expense KUS$	Unit Cost US$/Ton
Horizontal development	5,993	3.41
Ore production	39,622	22.53
Backfill material	8,909	5.07
Mine Administration	12,403	7.05
Main equipment leasing	9,130	5.19
Manpower	22,691	12.9
Mining services	16,221	9.22
Mining operating cost	**114,969**	**65.37**

Labor costs were developed based on operation, maintenance, services and technical-support personnel, and the unit cost per employee. Table 21.10 shows the list of personnel and salaries for the mine area.

Table 21.10 Mine Personnel Salaries

Role	Role per Shift	Shift	Schedule (days on/off)	N° of employees	Base Wage ($/y)	Cost per employee ($/y/employee)	Cost per year ($/y/role)
Mine Superintendent	1	1	5/2	1	125,000	$168,750	$168,750
Planning Engineer	1	2	5/2	2	90,000	$121,500	$243,000
Geologist	1	2	5/2	2	90,000	$121,500	$243,000
Mine Shift Foreman	1	2	4/3	2	95,000	$128,250	$256,500
Surveyor	1	2	4/3	2	44,533	$60,120	$120,239
Assistant Surveyor	1	2	4/3	2	38,000	$51,300	$102,600
Sampler	2	2	4/3	4	38,000	$51,300	$205,200
Mine Operation Personnel							
Maintenance	5	2	4/3	10	56,846	$76,742	$767,421
Horizontal Drill	2	2	4/3	4	61,776	$83,398	$333,590
Bolter	1	2	4/3	2	61,776	$83,398	$166,795
LHD	4	2	4/3	6	61,776	$83,398	$667,181
Front end Loader	1	2	7/7	4	61,776	$83,398	$166,795
Truck	3	2	4/3	6	61,776	$83,398	$500,386
Support Equipment	4	2	4/3	8	43,368	$58,547	$468,374
Diamond Drill	1	2	4/3	2	43,368	$58,547	$117,094
Support Services	3	2	4/3	6	20,000	$27,000	$162,000
Total Mining Labor	**32**			**63**			**$4,688,925**

Equipment operating cost was estimated from first principles taking into consideration main consumables, maintenance, and labor cost.

The average mine equipment operating cost is shown in Table 21.11.

Table 21.11 Mine Equipment Operating Cost

Underground Mining Equipment	$/hour
Drilling Development Jumbo (Jumbo DD21-40)	188
Bolter (Sandvik DS311)	158
LHD 5.2 yd^3 (LH410)	200
Front-end Loader (JCB 456ZX)	163
Low Profile Truck (AD30)	139
Telehandler (JCB 540-170)	46
Bulldozer (Cat D6T)	72
Motor Grader (Paus PG5HA)	67
Fuel Truck	49
Service Truck	46
Diamond Drilling (Hydracore Gopher)	66

The mine equipment operating cost is considered high in unit terms due to the four-day work week.

Consumable unit prices are summarized in Table 21.12.

Table 21.12 Mine Consumable Unit Price

Consumable	Unit	Price
Diesel	$/gal	2.27
Lubricant	$/gal	6.92
Grease	$/lb	0.98
Power	$/kWh	0.065
Horizontal drill bit	$/unit	65
Horizontal drill bar	$/unit	210
Emulsion	$/lb	0.57
Booster	$/unit	2.00
Nonel	$/unit	6.75
Detonator	$/unit	2.80
Bolt 8”	$/unit	27.0
Bolt 10”	$/unit	31.0
Mesh	$/ft^2	0.51
Shotcrete	$/ft^2	0.85
CRF	$/ton	20.68
RF	$/ton	3.90

Mine development cost was also estimated from first principles taking into considerations the cost of operating mobile equipment, labor and consumables, and the equipment productivity.

Mine services cost includes ventilation, dewatering, power consumption excluding drilling machines, operating cost for support vehicles, and other minor items.

Mine administration cost includes administration and technical services personnel, training, travel, safety clothing and personal protective equipment, administration consumables, and other minor items.

21.2.2 Process Operating Costs

The process operating cost was estimated for the Grassy Mountain based on the design process plant feed rate of 750 tons per day. These operating costs excluded mining, G&A, and surface services. The process plant operating costs were estimated at $7.542 million per year, or $27.55 per ton processed, and the surface ore-rehandling cost is estimated at $0.50 per ton processed.

The processing plant throughput is designed to operate at 750 tons per day or 273,750 tons per year. The mine plan ramps up in year 1 and ramps down in year 10 and 11 with a LOM average of approximately

226,000 tons processed per year. Total throughput was estimated to be 2,486,000 short tons over the 11-year mine life. All operating costs were estimated based on the design mill feed rate with -20% and +30% accuracy.

The process costs were divided into five categories as presented in Table 21.13, and detailed in Table 21.14.

Table 21.13 Process Operating Cost Categories

Category	Item Description		Scope Details
1	**Fixed Cost**	General Maintenance	Equipment spare parts and maintenance.
2		Labor	Work force required by process plant.
3	**Variable Costs**	Power	Electricity consumed by equipment, rental generator, transformer and lighting, process plant ancillary facilities.
4		Reagents & Operating Consumables	Reagents, fuel, ball mill grinding media.
5		Maintenance Consumables	Major wear parts for mills such as liner, crusher screen deck. Cathode mesh for EW cells.

Table 21.14 Processing Plant Operating Cost Summary

Item	K$/year	% of Total	$/ton Mill Feed
Fixed Costs			
Labor	3,315	44.0%	12.11
General Maintenance	459	6.1%	1.68
Sub-total (Fixed Costs)	**3,774**	**50.1%**	**13.79**
Variable Costs			
Power	1,487	19.7%	5.43
Reagents & Operating Consumables	1,800	23.9%	6.58
Maintenance Consumables	479	6.4%	1.75
Sub-total (Variable Costs)	**3,767**	**49.9%**	**13.76**
Total Cost	**7,541**	**100%**	**27.55**

21.2.2.1 Process Labor

Plant operations labor costs were estimated to be $12.11/ ton milled. The estimated labor force for plant operations and plant maintenance was estimated at 35 persons in total. Annual salaries and wages were supplied by MDA. The estimate is based upon providing a labor force to support continuous operations at 24 hours per day, 365 days per year.

Due to the process plant being relatively small, some operations roles have been combined to minimize the number of operators and supervisors. It was assumed that the same organizational structure will be retained for the LOM.

21.2.2.2 Process General Maintenance

Annual maintenance supplies costs were estimated at an average of 5% of major capital equipment costs. The total general maintenance operating costs are $459,000 per year, or $1.68/ton processed.

21.2.2.3 Process Electrical Power

The power supply cost of $1.487 million per year, or $5.43 per ton processed, is based on an average use of 18,300 MWh per year without including the rental backup generator and any spare or standby equipment. The unit cost of electricity was based on fixed rates of $0.0632 per kWh, as provided by Idaho Power.

21.2.2.4 Reagents & Operating Consumables

The reagent consumption rates were estimated from metallurgical testwork as summarized in Section 17.3. For anti-scalant and refining flux, the requirements were estimated based on similar projects completed by Ausenco. Annual reagents and operating consumables (fuel) are $1.800 million per year, or $6.82 per ton processed; freight cost of $155 per ton were added to all reagent costs.

21.2.2.5 Process Maintenance Consumables

Maintenance consumables costs include crusher liners and screens, grinding media, and gold-room cathode mesh. Consumables costs were estimated based on the similar projects completed by Ausenco. The crusher and ball-mill liners and media consumption rates were bench marked against Ausenco standards. The maintenance consumables cost is $479,000 per year, or $1.75 per ton processed.

21.3 General and Administration Costs

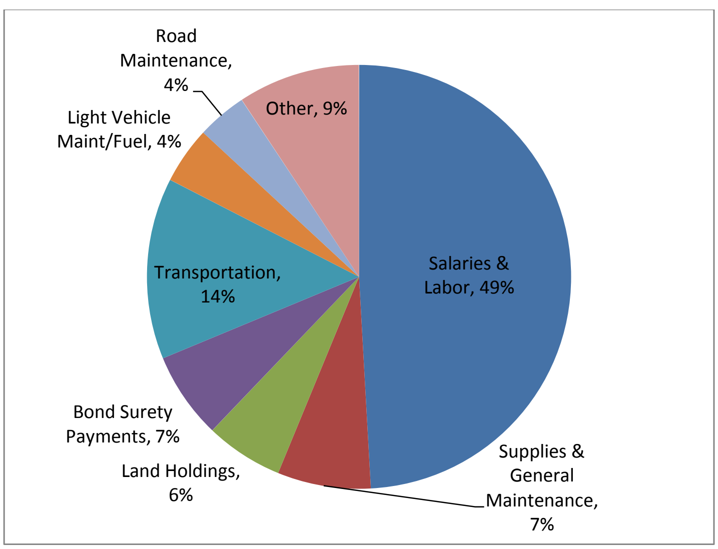
Table 21.15 shows the annual G&A costs, which also includes the pre-production period. Note that the pre-production amount has been capitalized in the cash-flow analysis, so the total G&A operating cost applied is $15,275,000 for the LOM.

Figure 21.1 shows the G&A operating cost breakdown by category.

Table 21.15 Annual G&A Costs

Personnel Costs	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Admin Salaried Personnel	K USD	$ 294	$ 235	$ 235	$ 235	$ 235	$ 235	$ 235	$ 235	$ 59	$ -	$ 1,998
Admin Hourly Personnel	K USD	$ 236	$ 195	$ 195	$ 195	$ 195	$ 195	$ 195	$ 195	$ 49	$ -	$ 1,650
Safety & Security Salaried Personnel	K USD	$ 125	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 100	$ 25	$ -	$ 850
Safety & Security Hourly Personnel	K USD	$ 391	$ 337	$ 337	$ 337	$ 337	$ 337	$ 337	$ 337	$ 84	$ -	$ 2,836
Environmental Salaried Personnel	K USD	$ 76	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 60	$ 15	$ -	$ 514
Recruitment Costs	K USD	$ 142	$ 113	$ 113	$ 113	$ 113	$ 113	$ 113	$ 113	$ 28	$ -	$ 963
Total Personnel Costs	K USD	$ 1,263	$ 1,041	$ 1,041	$ 1,041	$ 1,041	$ 1,041	$ 1,041	$ 1,041	$ 260	$ -	$ 8,810
General G&A Costs												
Supplies & General Maintenance	K USD	$ 188	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 150	$ 38	$ -	$ 1,275
Land Holdings	K USD	$ 156	$ 125	$ 125	$ 125	$ 125	$ 125	$ 125	$ 125	$ 31	$ -	$ 1,063
Off Site Overhead	K USD	$ 23	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 5	$ -	$ 153
Legal, Audits, Consulting	K USD	$ 15	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 3	$ -	$ 102
Computers, IT, Internet, Software, Hardware	K USD	$ 15	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 3	$ -	$ 102
Environmental, Montoring Wells, Reporting	K USD	$ 15	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 3	$ -	$ 102
Bond Surety Payments	K USD	$ 155	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 124	$ 31	$ 1,180
Donations, Dues, PR	K USD	$ 60	$ 48	$ 48	$ 48	$ 48	$ 48	$ 48	$ 48	$ 12	$ -	$ 408
Fees, Licenses, Misc Taxes, Insurance	K USD	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Travel, Lodging, Meals, Entertainment	K USD	$ 23	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 5	$ -	$ 153
Employee Transportation	K USD	$ 365	$ 292	$ 292	$ 292	$ 292	$ 292	$ 292	$ 292	$ 73	$ -	$ 2,482
Telephones, Computers, Cell Phones	K USD	$ 23	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 18	$ 5	$ -	$ 153
Light Vehicle Maintenance, Fuel	K USD	$ 115	$ 91	$ 91	$ 91	$ 92	$ 91	$ 91	$ 91	$ 23	$ -	$ 778
Small Tools, Janitorial, Safety Supplies	K USD	$ 15	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 3	$ -	$ 102
Equipment Rentals	K USD	$ 30	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 24	$ 6	$ -	$ 204
Access Road Maintenance	K USD	$ 100	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 80	$ 20	$ -	$ 680
Office Power	K USD	$ 103	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 12	$ 3	$ -	$ 192
Total General G&A Costs	K USD	$ 1,400	$ 1,049	$ 1,049	$ 1,049	$ 1,049	$ 1,049	$ 1,049	$ 1,049	$ 356	$ 31	$ 9,129
Total G&A	K USD	$ 2,663	$ 2,090	$ 2,090	$ 2,090	$ 2,090	$ 2,090	$ 2,090	$ 2,090	$ 616	$ 31	$ 17,938

Figure 21.1 G&A Operating Cost Breakdown



21.3.1 Reclamation Costs

Golder caution to readers: In this Item, all estimates and descriptions related to capital and operating cost estimates are forward-looking information. There are many material factors that could cause actual results to differ materially from the estimates, forecasts or projections set out in this item. Some of the material factors include differences from the assumptions regarding the following: *mining and processing methods, labor and overhead costs, locations and designs of facilities, cost escalation and inflation, production and productivity assumptions, and project schedule*. The material factors or assumptions that were applied in drawing the conclusions, forecasts and projections set forth in this Item are summarized in this report. Any significant differences from these factors or assumptions will have material impacts on the estimates, forecasts and projections set forth in this report.

The majority of the reclamation cost will be for the TSF, the cost of which was estimated by Golder. The reclamation cost was estimated by TSF stage and was used based on the reserves tonnage to be processed. MDA proportioned the reclamation cost as required by the Golder estimate to account for 1.76 million tons of tailings stored in the TSF. The total tailings reclamation cost is estimated to be $3.982 million. MDA added 20% of this total for other reclamation to be done at site, and then increased the combined estimate by 30% to reflect federal cost structures that would be required for bonding. This total bonding amount was used as the estimate for reclamation and was spread over 12 months after the end of production.

22.0 ECONOMIC ANALYSIS

MDA completed an economic analysis based on the cash flow developed from the production schedule and capital and operating costs previously discussed. The metal prices used for the evaluation include $1,300 per ounce of gold sold and $16.75 per ounce of silver sold. Economic highlights include:

- Average annual gold production of 47,000 ounces per year;
- Average annual silver production of 50,000 ounces per year;
- 7.25-year mine life;
- 341,000 total recovered ounces of gold (340,000 ounces sold);
- 362,000 total recovered ounces of silver (360,000 ounces sold);
- 345,000 gold-equivalent ounces produced[3];
- $528 cash operating cost per ounce of gold[1];
- $853 total cost per ounce of gold[2];
- 27.6% internal rate of return ("IRR");
- $87,754,000 after-tax Net Present Value ("NPV") at 5% discount rate;
- $70,621,000 after-tax NPV at 8% discount rate;
- 60,455,000 after-tax NPV at 10% discount rate; and
- 2.51-year payback period (from start of production).

[1] *Includes silver revenues as credit.*
[2] *Includes silver revenues as credit and all capital.*
[3] *Gold equivalent is based on ounces of gold and silver produced and gold to silver ratio of $1,300:$16.75.*

22.1 Economic Parameters and Assumptions

The economic analysis has been based on metal prices of $1,300 per ounce for gold and $16.75 per ounce for silver, capital and operating costs, royalties, application of depreciation, and tax rates. The remaining assumptions used in the analysis come from the mining and processing production schedules. Capital and operating costs have been discussed previously in Section 21.0. The metal prices were selected according to the criteria described in Section 19.1.

22.1.1 Royalties

The Seabridge NPI royalty was not applied. This is based on Seabridge's intention to convert the NPI/Put Option into common shares at the appropriate time (most likely at the completion of a Feasibility Study).

A gross proceeds royalty of 1.5% was applied under the Sherry & Yates agreement as discussed in Section 4.3.2. This royalty has been applied by calculating the total recovered ounces of gold and silver, multiplied by the metal prices and payable percentage, and then subtracting transportation and refining costs.

22.1.2 Taxes

Taxable income was calculated based on the net pre-tax cash flow, less depreciation and losses carried forward. Depreciation was calculated by depreciating the initial capital of $109,880 over a three-year period as allowed by the Tax Cuts and Jobs Act of 2017. Losses carried forward include $27 million for Paramount's Sleeper mine operations, $7 million for Calico losses incurred through the merger, and other losses of $15 million that have occurred since the Paramount-Calico merger.

Federal income taxes were applied to the taxable income at a rate of 21%.

22.1.3 Project Physical Values

The PFS physical values include quantities of development, mined and processed material, and produced metals that provide the basis for the cash-flow analysis. These values were derived from the mining and processing schedules previously discussed in the Mining Methods section. They were reformatted into the cash-flow sheet as shown in Table 22.1.

Table 22.1 Grassy Mountain Project Physical Quantities

Mine Development	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Total
Capital Development - Main Decline	K ft	5,748	1,525	-	-	-	-	-	-	-	-	7,273
Capital Development - Vent Drift	K ft	1,122	1,513	-	-	-	-	-	-	-	-	2,634
Level Access & Development	K ft	90	14,470	2,200	1,134	1,732	2,409	3,313	2,672	785	-	28,805
Vent Shafts	K ft	996	487	-	-	-	-	-	-	-	-	1,482
Total Development Distances	K ft	7,956	17,995	2,200	1,134	1,732	2,409	3,313	2,672	785	-	40,195
Mine Production												
Total Mined to Stockpile	Tons	0	220	282	282	283	282	279	113	18	-	1,759
Excluding Ore Loss & Including Dilution	oz Au/ton	0.142	0.199	0.261	0.201	0.181	0.192	0.213	0.166	0.241	-	0.206
	ozs Au	0	44	74	57	51	54	59	19	4	-	362
	oz Ag/ton	0.14	0.25	0.28	0.29	0.26	0.30	0.34	0.38	0.28	-	0.29
	ozs Ag	0	55	79	80	74	84	96	43	5	-	517
Processing												
Mine to Plant	K Tons	-	217	274	274	275	274	274	154	18	-	1,759
	oz Au/ton	-	0.201	0.266	0.205	0.184	0.196	0.215	0.139	0.241	-	0.206
	K ozs Au	-	44	73	56	51	54	59	21	4	-	362
	oz Ag/ton	-	0.25	0.28	0.29	0.26	0.29	0.34	0.34	0.28	-	0.29
	K ozs Ag	-	55	77	79	73	81	94	53	5	-	516
	Recovery- Au	0%	94%	95%	94%	94%	94%	94%	92%	95%	0%	94%
	K oz Au Rec	-	41	69	53	47	50	56	20	4	-	341
	Recovery - Ag	0%	66%	69%	70%	67%	70%	74%	74%	69%	0%	70%
	K oz Ag Rec	-	36	53	55	49	57	69	39	4	-	362
Payable Gold	K oz Au Rec	-	41	69	53	47	50	56	20	4	-	340
Payable Silver	K oz Ag Rec	-	36	52	54	49	56	69	39	3	-	360
Recoverable Au Equivalent Ounces	Rec AuEq K Ozs	-	41	70	53	48	51	56	20	4	-	345

22.1.4 Other Economic Assumptions

MDA used multiple discount rates for calculating NPV, including 5.0%, 8.0%, and 10.0%. The economic model was completed in Microsoft® Excel using basic Excel functions and formulas to calculate the NPV and IRR. Sensitivity tables were developed using Excel data table analysis.

22.2 Preliminary Feasibility Cash Flow

The PFS cash flow is presented in Table 22.2 and is based on the economic parameters and assumptions previously discussed. The after-tax NPV at 5.0% discount rate is $87,754,000, with an after-tax IRR of 27.6%.

Table 22.2 Grassy Mountain Project Cash Flow

Revenues	Units	Pre-Prod	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Total
Gold Price	$/oz Au	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	$ 1,300	
Silver Price	$/oz Ag	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	$ 16.75	
Gross Revenue - Au	K USD	-	53,244	90,219	68,618	61,506	65,544	72,227	25,757	5,379	-	-	442,494
Gross Revenue - Ag	K USD	-	606	879	912	817	944	1,156	656	58	-	-	6,028
Refining Cost - Au	K USD	-	205	347	264	237	252	278	99	21	-	-	1,702
Refining Cost - Ag	K USD	-	18	26	27	24	28	34	20	2	-	-	180
NPR Royalty (1.5%)	K USD	-	804	1,361	1,039	931	993	1,096	394	81	-	-	6,700
Net Revenue	K USD	-	52,822	89,364	68,201	61,131	65,215	71,974	25,900	5,334	-	-	439,940
Operating Costs													
Expensed Mine Development	K USD	-	1,025	747	445	676	934	1,273	985	282	-	-	6,368
UG Mining Costs	K USD	-	16,494	17,641	17,529	15,233	15,027	15,063	9,503	2,111	-	-	108,602
Surface Rehandle	K USD	-	108	136	136	137	136	136	77	9	-	-	875
Process Costs	K USD	-	5,973	7,542	7,542	7,562	7,542	7,542	4,254	500	-	-	48,456
G&A Costs	K USD	-	2,090	2,090	2,090	2,090	2,090	2,090	2,090	616	31	-	15,275
Reclamation Costs	K USD	-	-	-	-	-	-	-	-	4,660	1,553	-	6,213
Total Operating Costs	K USD	-	25,691	28,156	27,742	25,699	25,729	26,104	16,908	8,177	1,584	-	185,789
Cash Cost per Ton Processed	USD	$ -	$ 118	$ 103	$ 101	$ 94	$ 94	$ 95	$ 110	$ 451	$ -	$ -	$ 106
Cash Cost per Oz Au Sold	USD	$ -	$ 627	$ 406	$ 526	$ 543	$ 510	$ 470	$ 853	$ 1,976	$ -	$ -	$ 546
Cash Cost per Oz AuEq Sold	USD	$ -	$ 620	$ 402	$ 519	$ 536	$ 503	$ 462	$ 832	$ 1,955	$ -	$ -	$ 538
Net Operating Cash Flow	K USD	-	27,132	61,207	40,460	35,432	39,486	45,870	8,992	(2,843)	(1,584)	-	254,151
Capital Costs													
Mine Development	K USD	7,640	1,799	-	-	-	-	-	-	-	-	-	9,439
Mine Pre-production	K USD	4,598	-	-	-	-	-	-	-	-	-	-	4,598
Mining Capital	K USD	2,928	1,150	-	-	125	-	-	125	-	-	-	4,328
Buildings & Site Infrastructure	K USD	12,787	-	-	-	-	-	-	-	-	-	-	12,787
Process Capital	K USD	25,935	-	-	-	-	-	-	-	-	-	-	25,935
TSF	K USD	8,215	-	-	4,097	-	930	-	-	-	-	-	13,241
Plant & Infrastucture Indirects	K USD	9,691	-	-	-	-	-	-	-	-	-	-	9,691
Off-Site Power and Access	K USD	10,328	-	-	-	-	-	-	-	-	-	-	10,328
Owners Capital	K USD	7,005	-	-	(4,142)	-	-	-	-	-	-	-	2,863
Other Capital	K USD	2,092	-	-	-	-	166	-	-	-	-	-	2,259
Sub-Total	K USD	91,221	2,949	-	(45)	125	1,096	-	125	-	-	-	95,470
Working Capital	K USD	4,464	(4,464)	-	-	-	-	-	-	-	-	-	-
Contingency	K USD	14,195	-	-	1,024	-	257	-	-	-	-	-	15,477
Total Capital	K USD	109,880	(1,515)	-	979	125	1,353	-	125	-	-	-	110,947
Pre-Tax Total Cost per Ton Processed	USD	$ -	$ 112	$ 103	$ 105	$ 94	$ 99	$ 95	$ 110	$ 451	$ -	$ -	$ 169
Pre-Tax Total Cost per Oz Au Sold	USD	$ -	$ 590	$ 406	$ 544	$ 546	$ 537	$ 470	$ 860	$ 1,976	$ -	$ -	$ 872
Pre-Tax Total Cost per Oz AuEq Sold	USD	$ -	$ 584	$ 402	$ 537	$ 539	$ 530	$ 462	$ 838	$ 1,955	$ -	$ -	$ 860
Pre-Tax Cash Flow	K USD	(109,880)	28,646	61,207	39,481	35,307	38,133	45,870	8,867	(2,843)	(1,584)	-	143,205
Cumulative Pre-Tax Cash Flow	K USD	(121,415)	(81,233)	(20,026)	19,455	54,762	92,895	138,765	147,632	144,789	143,205	-	518,827
Net Operating Cash Flow NPV (5%)	K USD	103,535	-	-	-	-	-	-	-	-	-	-	103,535
Depreciation													
Starting Capital Pool Capital	K USD	109,880	109,880	81,233	20,026	-	-	-	-	-	-	-	321,018
Depreciation	K USD	-	28,646	61,207	20,026	-	-	-	-	-	-	-	109,880
Remaining Capital Pool	K USD	109,880	81,233	20,026	-	-	-	-	-	-	-	-	211,139
Loss Carried Forward (Sleeper)	K USD	27,000	-	-	-	-	-	-	-	-	-	-	27,000
Losses Carried Forward (Calico)	K USD	7,000	-	-	-	-	-	-	-	-	-	-	7,000
Losses Carried Forward (Calico Post 2017)	K USD	15,000	-	-	-	-	-	-	-	-	-	-	15,000
Starting Loss Carried Forward	K USD	49,000	49,000	49,000	49,000	29,545	-	-	-	-	-	-	225,545
Loss Applied	K USD	-	-	-	19,455	29,545	-	-	-	-	-	-	49,000
Remaining Loss Carried Forward	K USD	49,000	49,000	49,000	29,545	-	-	-	-	-	-	-	176,545
Total Depreciation and Applied Loss	K USD	-	28,646	61,207	39,481	29,545	-	-	-	-	-	-	158,880
Net Taxable Income	K USD	-	-	-	-	5,762	38,133	45,870	8,867	-	-	-	98,632
Taxes													
Federal Income Tax	K USD	-	-	-	-	1,210	8,008	9,633	1,862	-	-	-	20,713
Net Cash Flow	K USD	(109,880)	28,646	61,207	39,481	34,097	30,125	36,237	7,005	(2,843)	(1,584)	-	122,492
Net Cash Flow (combined Pre-prod)	K USD	(109,880)	28,646	61,207	39,481	34,097	30,125	36,237	7,005	(2,843)	(1,584)	-	122,492
Cumulative Cash flow	K USD	(109,880)	(81,233)	(20,026)	19,455	53,552	83,677	119,914	126,919	124,076	122,492	-	438,947
Break Even Years (from start of prod)	Yrs	1.00	1.00	1.00	0.51	-	-	-	-	-	-	-	3.51

Internal Rate of Return	%	27.6%
Net Present Value (5%)	5%	$ 87,754
Net Present Value (8%)	8%	$ 70,621
Net Present Value (10%)	10%	$ 60,455
Payback From Start of Project	Yrs	3.51
Payback From Start of Production	Yrs	2.51

22.3 Sensitivity Analysis

Pre-tax and after-tax cash-flow sensitivities to revenue were evaluated by varying the gold price from $1,200 to $1,500 per ounce in $50.00 increments. The silver price was also modified in these sensitivities based on a constant gold-to-silver price ratio of $1,300 to $16.75 (77.6:1 gold-to-silver price ratio). After-tax metal price sensitivities are shown in Table 22.3.

Operating and capital cost sensitivities were evaluated from +/- 20% of the values in 10% increments. Results from changes to operating costs are shown in Table 22.4 and results from changes to capital costs are shown in Table 22.5.

After-tax sensitivities to changes in revenues, operating costs, and capital costs are shown in Figure 22.1.

Table 22.3 After-Tax Cash-Flow Sensitivity to Gold Price

Au ($/oz Au)	Ag ($/oz Au)	IRR	NPV 5%	NPV 8%	NPV 10%	Payback Start of Prod
$ 1,100	$ 14.17	16.3%	$ 41,877	$ 28,655	$ 20,801	3.50
$ 1,150	$ 14.82	19.3%	$ 53,406	$ 39,227	$ 30,806	3.17
$ 1,200	$ 15.46	22.1%	$ 64,871	$ 49,714	$ 40,714	2.91
$ 1,250	$ 16.11	24.9%	$ 76,336	$ 60,200	$ 50,622	2.69
$ 1,300	$ 16.75	27.6%	$ 87,754	$ 70,621	$ 60,455	2.51
$ 1,350	$ 17.39	30.2%	$ 99,132	$ 80,987	$ 70,227	2.35
$ 1,400	$ 18.04	32.8%	$ 110,511	$ 91,354	$ 79,998	2.20
$ 1,450	$ 18.68	35.4%	$ 121,890	$ 101,720	$ 89,770	2.07
$ 1,500	$ 19.33	37.8%	$ 133,243	$ 112,050	$ 99,499	1.97

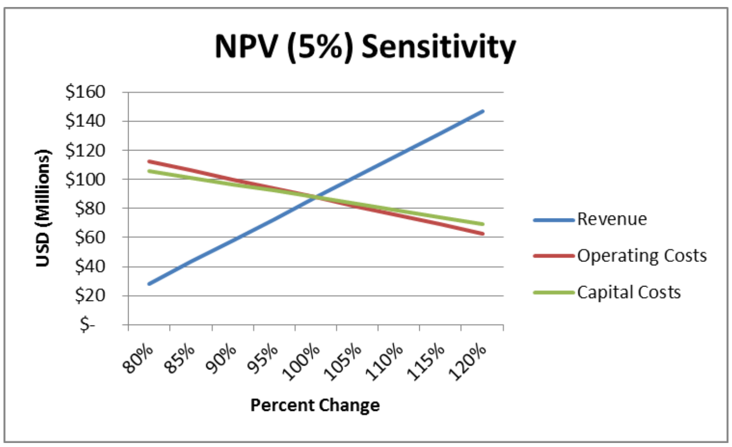
Table 22.4 After-Tax Cash-Flow Sensitivity to Operating Costs

% of Base	IRR	NPV 5%	NPV 8%	NPV 10%	Payback Start of Prod
80%	33%	$ 112,541	$ 93,043	$ 81,502	2.21
85%	32%	$ 106,345	$ 87,437	$ 76,240	2.27
90%	30%	$ 100,148	$ 81,832	$ 70,978	2.35
95%	29%	$ 93,951	$ 76,226	$ 65,717	2.42
100%	28%	$ 87,754	$ 70,621	$ 60,455	2.51
105%	26%	$ 81,557	$ 65,015	$ 55,193	2.60
110%	25%	$ 75,319	$ 59,353	$ 49,867	2.69
115%	23%	$ 69,078	$ 53,686	$ 44,535	2.80
120%	22%	$ 62,837	$ 48,019	$ 39,204	2.91

Table 22.5 After Tax Cash-Flow Sensitivity to Capital Costs

% of Base	IRR	NPV 5%	NPV 8%	NPV 10%	Payback Start of Prod
80%	38%	$106,030	$ 89,268	$ 79,322	1.97
85%	35%	$101,461	$ 84,606	$ 74,606	2.10
90%	32%	$ 96,892	$ 79,944	$ 69,889	2.23
95%	30%	$ 92,323	$ 75,282	$ 65,172	2.37
100%	28%	$ 87,754	$ 70,621	$ 60,455	2.51
105%	26%	$ 83,185	$ 65,959	$ 55,738	2.65
110%	24%	$ 78,573	$ 61,238	$ 50,954	2.78
115%	22%	$ 73,959	$ 56,514	$ 46,166	2.92
120%	20%	$ 69,345	$ 51,790	$ 41,378	3.07

Figure 22.1 After-Tax Cash-Flow Sensitivity



As with most precious metal deposits, Grassy Mountain is most sensitive to changes in metal prices as shown by the slope of the line in Figure 22.1. At prices greater than $1,100 per ounce of gold, the project retains a fairly robust return.

The sensitivity to operating and capital costs are similar in Figure 22.1.

23.0 ADJACENT PROPERTIES

The authors have no information to report on adjacent properties.

24.0 OTHER RELEVANT DATA AND INFORMATION

The authors are not aware of any additional information necessary to make this technical report understandable and not misleading.

25.0 INTERPRETATIONS AND CONCLUSIONS

The following summarizes the interpretations and conclusions of the qualified persons associated with this report.

25.1 Adequacy of the Data Used in Estimating the Project Mineral Resources

Mr. Gustin has reviewed the Grassy Mountain project data, including information relevant to the project history, geology, and mineralization, and verified the drill-hole data used in the resource estimation. Mr. Gustin has visited the project site on multiple occasions, as have other MDA qualified persons. Based on this work, it is Mr. Gustin's opinion that the project data are adequate for the modeling and estimation of the current Measured, Indicated, and Inferred gold and silver resources discussed in this report.

25.2 Geology, Mineralization, and Mineral Resources

The Grassy Mountain gold-silver deposit is characterized by low-sulfidation epithermal mineralization hosted within a package of interbedded arkose, siltstone, mudstone, and siliceous sinter. The deposit is comprised of a central higher-grade core with gold grades of >~0.03 oz Au/ton that is surrounded by a broad envelope of lower-grade mineralization. The central core zone, which includes all of the current mineral reserves, exhibits subhorizontal stratigraphic controls as well as subvertical structural controls, while the lower-grade envelope is dominated by stratigraphically controlled, disseminated mineralization.

The highest-grade mineralization (>~0.2 oz Au/ton) in the central core of the deposit lacks the continuity required for explicit modeling. While efforts were taken to appropriately represent the volume of the highest-grade mineralization in the resource modeling, the exact locations of this mineralization become less certain as distances from the drill data increase. Closely-spaced infill drilling will therefore be required in advance of potential underground mining of any particular sector of the deposit.

25.3 Exploration Potential

Within and adjacent to the Grassy Mountain deposit there is potential to expand the higher-grade portion of the deposit with near-mine exploration drilling. The Apache-Coyote and Gopher faults, as well as the northeast-trending fault in the North Spur, are viewed as potentially mineralized structures that could localize gold grades sufficient for the anticipated mining and milling scenario. These near-mine drill targets are considered to offer potential for incremental expansion of the high-grade core of the deposit and have uncertainties in their precise locations and dips.

Historical exploration has identified a number of prospects for possible hot-springs style, epithermal gold-silver mineralization within the property, outboard from the Grassy Mountain deposit. Exploration drilling has potential to discover structurally controlled vein or stockwork gold-silver mineralization at two targets in the Wood prospect, about one-mile northwest of the Grassy Mountain mine site. Recommended drilling is proposed in Section 26.1.

Potential for new drilling targets may be developed at the outlying Crabgrass, Bluegrass, North Bluegrass, Ryegrass, northern Snake Flats, and Dennis' Folly areas. Recommended surface work to develop targets in these areas is discussed in Section 26.1 .

25.4 Metallurgy and Processing

Metallurgical work demonstrates that Grassy Mountain gold-silver samples tested are free-milling and can be processed with gravity concentration by centrifugal-type Knelson concentrators followed by conventional cyanide leaching. Results from the 2017 test program fall in line with historical testing completed on the project. Due to some assaying issues with the gravity-concentrate samples, a conservative interpretation of the results estimates gravity recovery of 8.6% of the gold. While leach testing shows similar recoveries with either CIP or CIL, CIL was selected for this study with gold recovery of 84.9% on gravity tailings, for an overall, combined gold recovery of 93.5%.

25.5 Mineral Reserves, Mining Methods, and Mine Planning

Proven and Probable reserves, effective May 1, 2018, consist of a total of 1.72 million tons with an average gold grade of 0.210 oz Au/ton and an average silver grade of 0.30 oz Ag/ton, for 362,000 contained ounces of gold and 516,000 contained ounces of silver. Underground drift-and-fill mining was selected to achieve a mine production rate of 1,300 to 1,400 tons/day with two shifts per day working four days per week. This will be sufficient to provide a throughput of 750 tons/day, seven days per week, to the planned mill.

The drift-and-fill minng method was chosen to achieve the mine production at an appropriate mining selectivity. The majority of drifts required to access the ore at the mining levels can be developed through mineralized areas, which will minimize development in waste.

Backfill material will consist of CRF and RF, with CRF to be used in the primary drifts and RF to be used in the secondary drifts. Geotechnical studies show that underground support will be required. The bolt pattern, bolt length, shotcrete thickness and mesh requirements were estimated considering the rock quality and the dimensions of the mine-design elements.

Production schedules were made from which the personnel and equipment requirements were determined. Due to the location of the mine, it is reasonable to assume that qualified personnel and equipment may be reasonably attainable for the operation.

25.6 Tailings Storage Facility

Based on the data and other information described in this report, Golder provided a preliminary design for a TSF sufficient to contain the tailings generated from the PFS mine production. At this stage of the project, there appears to be reasonable certainty that the location of the TSF as chosen for this PFS will be utilized as planned and that no significant design changes are likely to be required, provided that additional geotechnical drilling does not identify issues that are not expected as of the Effective Date of this report.

25.7 Infrastructure

Infrastructure has been specified for the project as appropriate for a prefeasibility-level study. The infrastructure provides the necessary power, water, buildings, and other support services required to operate the project.

A trade-off study was completed to compare the economics of providing power via generation on site versus by line power. The best economics are obtained through the use of line power, although initial construction and mining will require generators prior to the construction and commissioning of line power.

25.8 Environmental Permitting

The process for acquiring the necessary local, state, and federal permits for the development of an underground mining and mill processing operation at Grassy Mountain began in 2012 and is on-going. There are no known environmental issues with any of the regulatory agencies, but a significant amount of the permitting process remains to be accomplished.

A valid exploration permit currently exists with DOGAMI and the BLM, and a bond in the amount of $146,200 is associated with the exploration permit. An existing Notice with the BLM for 2.78 acres of surface disturbance and a monitor well has an associated bond in the amount of $25,315.

25.9 Capital and Operating Costs

Total initial capital is estimated to be $109.9 million, sustaining costs of $1.1 million have been estimated, and the total LOM capital-cost estimate is $110.9 million. This includes about $69.9 million for mine, infrastructure, and equipment, $12.2 million for pre-production capital, owners capital of $7 million, and $14.2 million for contingency.

Total LOM operating costs are estimated at $185.8 million. This results in a $105.63 cost per ton processed, or $528.12 per gold-equivalent ounce produced.

The estimated costs provide the basis for the economic analysis and are considered appropriate for a prefeasibility-level study.

25.10 Economic Analysis

The economic analysis was done considering the revenues, costs, and tax application for Paramount. Revenues were based on prices of $1,300 per ounce of gold and $16.75 per ounce of silver, as well as gold and silver recoveries provided by Ausenco. Costs were applied based on estimates for capital and operating costs provided by MDA, Ausenco, and Golder. Tax considerations were applied based on input from Paramount.

The resulting economic analysis shows an IRR of 27.6% and a 5% NPV of $87.75 million USD. This demonstrates that the Grassy Mountain project is a project of merit. The project is most sensitive to changes in metal prices. At prices greater than $1,100 per ounce of gold the project retains a fairly robust return.

25.11 Risks and Opportunities

25.11.1 Risks

25.11.1.1 Mining

- The ability to achieve the mining capital and operating costs are considered to be one of the most significant mining risks. This risk is related to the efficiencies used for mining parameters such as mining-cycle durations, equipment productivities, and consumable ratios. Lower efficiencies of these parameters would increase the estimate of both mining capital and operating costs.
- Adverse rock properties can cause longer mining cycles, poorer equipment efficiencies, and higher material and labor consumption ratios.
- The mine-design stability analysis and ground-support definitions remain preliminary in nature. This may be a significant risk if changes in these parametersare needed as new information becomes available. The most important risk is the increase in cost due to additional ground support requirements, or changes in the mine design, mine sequence or backfill definitions.
- The hydrogeology study was carried out by SPF during 2017. According to the study, SPF estimated the water table to be located near the lower levels of the mine, but the dewatering requirements remain somewhat speculative, which could impact the profitability of the operation.
- While water wells developed in the area have shown to produce sufficient flow, it is uncertain how they will react under long-term pumping. Stable water sources of sufficient capacities will be critical to the successful operation of the mine.

25.11.1.2 Tailings Storage Facility

- During preliminary design, Golder identified potentially weak soils within the TSF dam foundation that will require additional investigation, testing, and analyses. Future findings may require changes to the TSF dam materials or geometry to increase geotechnical stability (ie. downstream buttress and/or foundation overexcavation), which could result in an increase to capital construction costs.

- Consolidated permitting for the TSF is on-going and may require changes to the seismic design criteria based on dam hazard classification, as well as the containment system, each of which could result in some capital cost impact to the project. Cost impacts may be an increase or reduction depending on the yet-to-be determined regulatory requirements.

25.11.2 Opportunities

25.11.2.1 Mining

- The selected mining method could be further optimized to achieve better selectivity through a reduction in the mining width currently projected. This may cut down on internal dilution.

25.11.2.2 Infrastructure

- Borrow-pit mining assumes the use of a contractor with a 30% profit margin. Savings may be gained through competitive bidding for the mining contract or if no contractor was used.
- Private sources for gravel construction along the access route may be obtainable. This could lead to more simplified permitting for the development of these sources, and it could potentially reduce costs of the gravel for the access-road construction.

25.11.2.3 Tailings Storage Facility

- A potential cost saving opportunity is the possibility that a higher settled dry tailings density than the 70 pcf assumed for sizing of the TSF could be attained, which would then lead to a smaller dam with lower capital costs. Preliminary testing indicates a higher density may be achieved, and this could be proven through further bench-scale testing of the tailings and updating of the depositional model.
- An additional opportunity for cost savings is through refinement of TSF dam zoning, staging, and sequencing to reduce capital costs and/or defer some sustained capital costs.

25.11.2.4 Capital and Operating Costs

- Capital cost reductions may be possible through the use of used buildings and equipment.

26.0 RECOMMENDATIONS

The authors believe that the Grassy Mountain project is a project of merit that should be considered for a Feasibility Study. In addition, an exploration program that includes drilling of targets with the potential to provide additional mill feed is also warranted. The approximate costs of the recommended work are shown on Table 26.1, followed by a description of the exploration program and a summary of workd needed to complete a Feasibility Study.

Table 26.1 Recommended Work Program

Category	Estimated Cost $
Exploration Drilling (19,500ft RC); includes assays, site prep, supplies, down-hole surveys	$ 1,100,000
Surface Exploration (CSAMT, soil sampling, trenching)	$ 150,000
Exploration Geology	$ 50,000
Feasibility Study	**$ 2,000,000**
Total	**$ 3,300,000**

26.1 Recommended Exploration

Drilling: A total of 13 inclined holes, for 12,795 feet of drilling, are recommended for near-mine expansion drilling of four targets. This drilling is summarized in Table 26.2. Two separate targets in the outlying Wood prospect are also recommended for drilling to test for the presence of structurally controlled vein or stockwork mineralization. A total of seven inclined holes for 6,650 feet of drilling are recommended for the Wood and East Wood targets as summarized in Table 26.2. Positive results in any of the proposed holes could form the basis for further drilling up and down dip, as well as along strike.

Surface Exploration: Expansion of the CSAMT coverage to the Crabgrass, Bluegrass, North Bluegrass, Ryegrass, and Dennis' Folly areas is recommended to identify silicified zones and structural controls that, if present, could be considered for exploration drilling. Verification and infill soil sampling and trenching at the large geochemical anomaly north of Snake Flats, and in the Dennis' Folly area, are also recommended to define one or more new drilling targets.

Table 26.2 2017 Proposed Exploration Drilling

Target	Prop_ID	Priority	Azim	Dip	TD_Ft	X_NAD83ft	Y_NAD83ft
Near Mine							
West	EXP17-14	1	340	-50	900	1,545,186	15,863,575
West	EXP17-15	2	355	-50	900	1,545,244	15,863,578
Apache-Coyote	EXP17-18	3	270	-60	1,000	1,546,552	15,864,220
Apache-Coyote	EXP17-17	4	270	-57	975	1,546,587	15,864,126
Apache-Coyote	EXP17-16	5	280	-60	1,000	1,546,552	15,864,220
East	EXP17-13	6	325	-70	1,100	1,546,487	15,864,261
East	EXP17-12	7	350	-70	1,100	1,546,553	15,864,277
North Spur	EXP17-01	8	90	-50	1,000	1,545,419	15,865,652
North Spur	EXP17-02	9	90	-50	820	1,545,656	15,865,649
North Spur	EXP17-03	10	90	-50	1,000	1,545,153	15,865,658
North Spur	EXP17-04	11	90	-50	1,000	1,545,142	15,865,081
North Spur	EXP17-05	12	90	-50	1,000	1,545,416	15,865,074
North Spur	EXP17-06	13	90	-50	1,000	1,545,690	15,865,067
Near Mine Holes:	*13*			*Feet:*	*12,795*		
Peripheral							
East Wood	EXP17-08	14	270	-50	1,000	1,542,907	15,869,139
East Wood	EXP17-09	15	270	-50	1,000	1,542,656	15,869,145
Wood	EXP17-19	16	340	-60	650	1,541,591	15,868,446
Wood	EXP17-20	17	300	-60	1,000	1,541,470	15,868,444
East Wood	EXP17-11	18	270	-50	1,000	1,543,016	15,869,427
East Wood	EXP17-10	19	270	-50	1,000	1,542,349	15,869,151
East Wood	EXP17-07	20	270	-50	1,000	1,543,141	15,869,134
Peripheral Holes:	*7*			*Feet:*	*6,650*		
				All Recommended Total:	**19,445**		

26.2 Recommended Feasibility Study Elements

Additional studies should be conducted prior to, or as part of, a Feasibility Study that will increase confidence in the project reserves. This should include metallurgical testwork, additional process-plant design, additional TSF design, and optimization of reserves and mine plans.

26.2.1 Metallurgical Test Work

Recommended testwork for a feasibility-level study should be based on additional samples that are representative of the deposit. Such composites should undergo:

- Comminution testing appropriate for the crushing and grinding circuits selected, including three-stage crushing that requires Bond crusher and ball-mill work indices;

- Additional E-GRG tests with larger samples is recommended to improve confidence in the estimated gravity recoveries discussed in Section 13.3.3.1;
- Leach tests on gravity tailings are recommended to optimize the selected grind, leach-time, and reagent conditions. Pre-aeration is included currently, but additional testing is needed to evaluate the costs and benefits;
- Oxygen-uptake tests should be performed to optimize sizing of the oxygen or low-pressure air plant;
- Cyanide detoxification tests are recommended at the actual discharge conditions for CN_{WAD} to define the required process-design criteria.
- Agitator and carbon suspension tests at the design CIL slurry density should be performed to ensure operability with no leach feed thickener; and
- Mineralogy and gold-deportment studies are recommended to further optimize the process design.

Variability samples should also be selected to represent the end members of each lithological unit. Testing of these samples is recommended to evaluate variation in grade, hardness, and mineralogical characteristics. Testing should include:

- Comminution tests including Bond crusher, ball mill, rod mill, and abrasion indices; and
- Leach tests after gravity concentration at the selected optimum conditions.

26.2.2 Process Plant Design

A trade-off study comparing the economics of CIL and CIP leach circuits at the PFS throughput rate is recommended to confirm this process selection.

A pressure Zadra circuit was selected for elution of gold and silver from carbon due to the small carbon processing requirements of the CIL circuit and the unknown quality of the water-quality from the PFS water sources. These assumptions require confirmation.

26.2.3 Tailings Strorage Facility Design

For the PFS, Golder made key assumptions regarding the location and design criteria of the TSF. To better define the risk and cost of the TSF, Golder recommends the following as the design advances to the Feasibility stage:

- Confirm that the settled tailings density is 70 pcf. If additional testing on tailings material indicates that the settled density is significantly different from 70 pcf, the layout and staging of the TSF should be revised;
- Due to the variability of subsurface materials and presence of low-strength materials in the TSF foundations, additional geotechnical investigation and geotechnical laboratory testing is recommended to delineate and evaluate the variations and discontinuities in the subsurface materials beneath the TSF. This program will assist in refining the understanding of how variations and discontinuities in the subsurface clay properties may affect stability of the embankment. If

low-strength materials are determined to be more prevalent than assumed in this design, a buttress may be required along the downstream toe of the embankments to satisfy the stability requirements, increasing overall construction costs;

- Update the seismic-hazard assessment with the data obtained from the geotechnical investigations recommended above. If the site subsurface soils conditions vary by more than 20% from those assumed in the PFS, the design ground motions for the project site should be re-evaluated;
- These recommendations are for the current TSF location. For any other location, additional geotechnical investigation will be required to evaluate the potential for the presence of low-strength materials and swelling of subsurface clays. If either low-strength materials or clays with high swell potential are determined to affect the stability and design of the embankment, additional engineering and construction costs may be incurred;
- Proposed construction borrow areas should be investigated to confirm that materials of sufficient quantity and quality are available for TSF construction. If additional borrow areas are required that are more distant, construction costs of the TSF will increase as compared to the costs estimated in the report; and
- The TSF embankment sections should be optimized with zoning to utilize the available on-site borrow material. The use of native soils that do not require drilling and blasting may decrease overall construction costs.

26.2.4 Reserves and Underground Mining

Additional optimization of the mine design, mine production, and mine stability should be undertaken within a Feasibility Study. This should include:

- A sensitivity study of variations in quantities and location of the material above the resource and reserve cut-off grades is recommended to facilitate the identification of areas for increasing the estimate of mineral reserves;
- Optimize the mine design based on detailed review of each mining level. This will assist in optimizing the project economics by running the following analysis:
    - Identify priority areas for in-fill drilling, with the goal of adding high-grade material into the mineral resources, which would then provide the potential of increasing the mineral reserves as well.
    - Assess sub-economical areas that, with further work, could be added to the mineral reserves.
    - Update the estimate of economic value contained by each mine level, confirming the mining method, mining sequence, and mineral reserve estimate; and
- Further analyze the underground equipment types and sizes to identify possible increases to mining and cost efficiencies.

The mining method should also be assessed in the Feasibility Study from a geotechnical perspective, considering mine stability, ground support definitions, mining sequence, mine production, and mining costs. The goals and specific aspects of this assessment should include:

- Improve the understanding of rock-mass characterization and rock-strength properties, it is recommended to drill four geotechnical core holes. Two of these drill holes should penetrate the area of the decline ramp, and the other two should cross the zone of the production drifts. Improvements in the rock-mass characterization would allow better stability analysis and definition of ground-support requirements;
- Complete structural mapping of the recommended geotechnical core holes using an Acoustic/Optical Televiewer survey. This information should then be used to construct a structural 3D-model of the deposit;
- Complete laboratory testing of core samples from the proposed geotechnical core holes to improve the understanding of the rock-mass strength and rock properties;
- Assess controlled mine-blasting techniques to attempt to optimize the overbreak. Excessive overbreak could lead to over-dilution along with infrastructure damage, especially to the drifts and stopes backfilled with CRF;
- Further examine underground seismic hazards. This work could be conducted in concert with TSF seismic studies;
- Complete in-situ stress measurements using acoustic emission methods at depths of 100 feet, 200 feet, and 300 feet for numerical 3D-modeling;
- CRF laboratory testing should be completed to assess the optimal rock size and blending requirements for the required strengths. The long-term stability of CRF material should include an analysis of the long-term properties, including stability and potential for acid generation. The backfill methodology, sequence, and rates utilized by the mine-production schedule should also be confirmed when new CRF quality information becomes available; and
- The basalt material obtained from the borrow-pit area should undergo additional testing to determine rock properties, including the potential for acid generation and RF strength.

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28.0 DATE AND SIGNATURE PAGE

Effective Date of report: May 21, 2018

The data on which the contained resource estimates are based was current as of the Effective Date.

Completion Date of report: July 9, 2018

"Michael M. Gustin"
Michael M. Gustin, PhD, C.P.G.

Date Signed:
July 9, 2018

"Thomas L. Dyer"
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July 9, 2018

"David Baldwin"
David Baldwin, P.Eng.

Date Signed:
July 9, 2018

29.0 CERTIFICATE OF QUALIFIED PERSONS

MICHAEL M. GUSTIN, C.P.G.

I, Michael M. Gustin, C.P.G., do hereby certify that I am currently employed as Senior Geologist by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Geology from Northeastern University in 1979 and a Doctor of Philosophy degree in Economic Geology from the University of Arizona in 1990. I have worked as a geologist in the mining industry for more than 30 years. I am a Licensed Professional Geologist in the state of Utah (#5541396-2250), a Licensed Geologist in the state of Washington (# 2297), a Registered Member of the Society of Mining Engineers (#4037854RM), and a Certified Professional Geologist of the American Institute of Professional Geologists (#CPG-11462).
2. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously explored, drilled, evaluated and modeled similar volcanic-hosted epithermal gold-silver deposits in the western US and Mexico. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I most recently visited the Grassy Mountain project site on June 1, 2017.
4. I am responsible for Sections 1.1, 1.2, 1.3, 1.5, 1.10, 1.14, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11, 12, 14, 20, 23, 24, 25.1, 25.2, 25.3, 25.8, 26.1, 27, 28, and 29 of this report titled, "*Preliminary Feasibility Study and Technical Report for the Grassy Mountain Gold and Silver Project, Malheur County, Oregon, USA*", with an effective date of May 21, 2018 (the "Technical Report").
5. I had no involvement in the Grassy Mountain property prior to the work completed for Paramount Gold Nevada Corp. I am independent of Paramount Gold Nevada Corp. and all of its subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
6. As of the Effective Date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible for not misleading.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 9th day of July 2018.

"Michael M. Gustin"
Michael M. Gustin

CERTIFICATE OF QUALIFIED PERSON

THOMAS L. DYER, P.E.

I, Thomas Dyer, P. E., do hereby certify that I am currently employed as Senior Engineer by Mine Development Associates, Inc., 210 South Rock Blvd., Reno, Nevada 89502 and:

1. I graduated with a Bachelor of Science degree in Mine Engineering from South Dakota School of Mines & Technology in 1996. I have worked as a Mining Engineer for 23 years since graduation. During my Engineering career I have held various positions of increasing responsibility at operating mines performing life of mine planning and cost estimates. During the last 12 years I have been engaged in consulting on various lead, zinc, gold, silver, copper, and limestone deposits both for underground and open pit operations. This consulting work has primarily consisted of providing production schedules, mine cost estimates, and cash-flow analysis.

2. I am registered as a Professional Engineer – Mining in the State of Nevada (# 15729). I am also a Registered Member of SME (# 4029995RM) in good standing.

3. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

4. I am one of the authors of the Technical Report titled *"Preliminary Feasibility Study and Technical Report for the Grassy Mountain Gold and Silver Project, Malheur County, Oregon, USA"* dated effective May 21, 2018 (the "Technical Report"). I am responsible for the preparation of sections 1.9, 1.11, 1.12, 1.13, 16.10, 18, 19, 21, 22, 25.7, 25.9, 25.10, and 25.11 of this report. I have visited the property on August 18, 2016 to review future infrastructure and road requirements.

5. I have had no prior involvement with the Grassy Mountain Gold Project.

6. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

7. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 9th day of July 2018

"Thomas L. Dyer"
Thomas L. Dyer

CERTIFICATE OF QUALIFIED PERSON
BORIS G. CARO, MSC, AUS.I.M.M.

I, Boris G. Caro, P.E., do hereby certify that I am an independent technical advisor contracted by Ausenco Chile Limitada to serve as Qualified Person and:

1. I graduated with a Bacherlor of Engineering and Mining Engineering degree from Universidad de Santiago de Chile in 1999 and a Master of Science Degree in Mineral Economics from Curtin University of Technology in 2006. I have worked as a mining engineer and study manager in the mining industry for 20 years. I am a Member of Australasian Institute of Mining and Metallurgy (Membership Number 305462) and a Registered Member of the Chilean Mining Commission (Registered Member Number 0229).
2. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101"). I have previously studied and evaluated or developed and mined Mineral Reserves in similar underground selective mining method with volcanic-hosted epithermal gold-silver deposits in West Africa, Peru, Mexico, Chile and Colombia. I certify that by reason of my education, affiliation with certified professional associations, and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
3. I have not visited the Grassy Mountain project site.
4. I am responsible for Sections 1.6, 1.7, 15, 16 (except 16.10), 21.1.1, 21.2.1, 25.5, and 26.2.4 of this report titled, "Preliminary Feasibility Study and Technical Report for the Grassy Mountain Gold and Silver Project, Malheur County, Oregon, USA", with an Effective Date of May 21, 2018 (the "Technical Report").
5. I have had no prior involvement with the Grassy Mountain property or project that is the subject of the Technical Report, and I am independent of Paramount Gold Nevada Corp., and all of their subsidiaries, as defined in Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
6. As of the Effective Date of this Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make those parts of this Technical Report for which I am responsible for not misleading.
7. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated this 9th day of July 2018

"Boris G. Caro"
Boris G. Caro

CERTIFICATE OF QUALIFIED PERSON
DAVID BALDWIN, P. ENG.

I, David Baldwin, Study Manager at Ausenco, do hereby certify that:

1. I am a mechanical engineer and study manager at Ausenco Engineering Canada, 855 Homer Street, Vancouver BC, Canada.
2. I am a graduate of the University of Victoria with a Bachelor's in engineering degree in Mechanical Engineering. I am also a graduate of Queen's University in Kingston, Ontario, with a Masters of Business Administration.
3. I am a Registered Member (#153727) in good standing of Engineers and Geoscientists of British Columbia (EGBC).
4. I have worked in the Mineral Processing Industry for a total of over 5 years during, and after my attending the University of Victoria.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience and affiliation with a professional association, I meet the requirements of a Qualified Person as defined in National Instrument 43-101.
6. I am responsible for the preparation of Sections 18.2.4, 18.2.5, 18.2.6, 18.2.7, 18.2.8, 18.2.9; 18.3.3; and 25.7 of the technical report titled "Preliminary Feasibility Study and Technical Report for the Grassy Mountain Gold and Silver Project, Oregon, USA" with an Effective Date of May 21, 2018 (the "Technical Report"). I visited the property on June 18, 2018.
7. I have no prior involvement with the project.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated July 9, 2018

"David Baldwin"
David Baldwin

CERTIFICATE OF QUALIFIED PERSON

Tommaso Roberto Raponi, P. Eng.

To Accompany the Report entitled, "Preliminary Feasibility Study and Technical Report for the Grassy Mountain Gold and Silver Project, Malheur County, Oregon, USA" prepared for Paramount Gold Nevada Corp. effective date May 21, 2018 and dated July 9, 2018.

I, *Tommaso Roberto Raponi, P. Eng.,* do hereby certify:

1. I am a Senior Mineral Processing Specialist at Ausenco Engineering Canada Inc., 11 King St West, Suite 1550, Toronto, ON, M5H 4C7.
2. I hold a Bachelor's degree in Geological Engineering from University of Toronto, Toronto, Ontario, Canada.
3. I am registered as Professional Engineers in Ontario and British Columbia. I have worked for more than *34* years in the mining industry in various positions continuously since my graduation from university. I have worked as an independent consultant since 2016;
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited site.
6. I am responsible for Sections 1.4, 1.8, 13, 17, 25.4, 25.6, 26.2.1, and 26.2.2 of the Technical Report.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
9. I have read NI 43-101 and Form 43- 101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 9th day of July 2018

"Original document signed and sealed"
Tommaso Roberto Raponi, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

APPENDIX A

Summary of Grassy Mountain Claim Information

~~Calico Resources USA Corp. Owned Unpatented Claims~~					
~~Claim Name/Number~~	~~BLM ORMC Number~~	~~Location Date~~	~~Claim Name/Number~~	~~BLM ORMC Number~~	~~Location Date~~
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Calico Resources USA Corp. Owned Unpatented Lode Claims					
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GM 5060	168000	9/15/2011	GM 5853	168126	9/24/2011
GM 5061	168001	9/15/2011	GM 5854	168127	9/24/2011

Commented [SW6]: CW, pls help, need this row to repeat as header on successive pages

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GM 5063	168003	9/17/2011	GM 5856	168129	9/24/2011
GM 5064	168004	9/17/2011	GM 5857	168130	9/24/2011
GM 5065	168005	9/17/2011	GM 5858	168131	9/24/2011
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GM 5067	168007	9/17/2011	GM 5860	168133	9/24/2011
GM 5068	168008	9/17/2011	GM 5861	168134	9/24/2011
GM 5069	168009	9/17/2011	GM 5862	168135	9/24/2011
GM 5070	168010	9/17/2011	GM 5863	168136	9/24/2011
GM 5071	168011	9/17/2011	GM 5864	168137	9/24/2011
GM 5072	168012	9/17/2011	GM 5885	168138	9/22/2011
GM 5150	168013	9/15/2011	GM 5886	168139	9/22/2011
GM 5151	168014	9/15/2011	GM 5887	168140	9/22/2011
GM 5152	168015	9/15/2011	GM 5956	168141	9/24/2011
GM 5153	168016	9/15/2011	GM 5957	168142	9/24/2011
GM 5154	168017	9/15/2011	GM 5958	168143	9/24/2011
GM 5155	168018	9/15/2011	GM 5959	168144	9/24/2011
GM 5156	168019	9/15/2011	GM 5960	168145	9/24/2011
GM 5157	168020	9/15/2011	GM 5961	168146	9/24/2011
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GM 5170	168033	9/17/2011	GM 6061	168159	9/24/2011
GM 5171	168034	9/17/2011	GM 6062	168160	9/24/2011
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GM 5252	168038	9/15/2011	GM 6072	168164	9/21/2011
GM 5253	168039	9/15/2011	GM 6073	168165	9/21/2011
GM 5254	168040	9/15/2011	GM 6074	168166	9/21/2011
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GM 5267	168053	9/23/2011	GM 6162	168179	9/24/2011
GM 5268	168054	9/23/2011	GM 6174	168180	9/21/2011
GM 5269	168055	9/23/2011	GM 6175	168181	9/21/2011
GM 5270	168056	9/23/2011	GM 6176	168182	9/21/2011
GM 5271	168057	9/23/2011	GM 6177	168183	9/21/2011
GM 5272	168058	9/23/2011	GM 6178	168184	9/21/2011
GM 5273	168059	9/23/2011	GM 6179	168185	9/21/2011
GM 5274	168060	9/23/2011	GM 6180	168186	9/21/2011
GM 5275	168061	9/23/2011	GM 6181	168187	9/21/2011
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GM 5368	168079	9/23/2011	GM 6271	168205	9/20/2011
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GM 5370	168081	9/23/2011	GM 6273	168207	9/20/2011
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GM 5655	168104	9/18/2011	GM 6366	168230	9/21/2011
GM 5680	168105	9/22/2011	GM 6367	168231	9/21/2011
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GM 5755	168113	9/18/2011	GM 6377	168239	9/20/2011
GM 5756	168114	9/25/2011	GM 6378	168240	9/20/2011
GM 5757	168115	9/25/2011	GM 6379	168241	9/20/2011
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GM 5780	168117	9/22/2011	GM 6381	168243	9/22/2011
GM 5781	168118	9/22/2011	GM 6382	168244	9/22/2011
GM 5782	168119	9/22/2011	GM 6383	168245	9/22/2011
GM 5783	168120	9/22/2011	GM 6384	168246	9/22/2011
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Frog #2	104798	5/6/1988	Frog #170	104963	5/19/1988
Frog #5	104801	5/6/1988	Frog #171	104964	5/19/1988
Frog #7	104803	5/6/1988	Frog #172	104965	5/19/1988
Frog #9	104805	5/6/1988	Frog #173	104966	5/19/1988
Frog #11	104807	5/6/1988	Frog #174	104967	5/19/1988
Frog #16	104812	5/6/1988	Frog #175	104968	5/19/1988
Frog #18	104814	5/6/1988	Frog #176	104969	5/19/1988
Frog #19	104815	5/6/1988	Frog #195	104988	5/22/1988
Frog #20	104816	5/6/1988	Frog #196	104989	5/22/1988
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Frog #28	104824	5/7/1988	Frog #212	105005	5/27/1988

Frog #29	104825	5/7/1988	Frog #213	105006	5/27/1988
Frog #30	104826	5/7/1988	Frog #214	105007	5/27/1988
Frog #31	104827	5/7/1988	Frog #215	105008	5/27/1988
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Frog #34	104830	5/7/1988	Frog #226	105019	5/26/1988
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Frog #36	104832	5/7/1988	Frog #230	105023	5/26/1988
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Frog #38	104834	5/7/1988	Frog #252	105913	7/21/1988
Frog #39	104835	5/7/1988	Frog #649	107597	8/17/1988
Frog #40	104836	5/7/1988	Frog #650	107598	8/17/1988
Frog #41	104837	5/7/1988	Frog #651	107599	8/17/1988
Frog #42	104838	5/7/1988	Frog #652	107600	8/17/1988
Frog #46	104839	5/7/1988	Frog #755	107703	8/23/1988
Frog #47	104840	5/7/1988	Frog #756	107704	8/23/1988
Frog #48	104841	5/7/1988	Frog #10A	108086	9/28/1988
Frog #85	104878	5/8/1988	Frog #25A	108087	9/27/1988
Frog #86	104879	5/8/1988	Frog #26A	108088	9/27/1988
Frog #87	104880	5/8/1988	Frog #35A	108089	9/27/1988
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Frog #89	104882	5/8/1988	Frog #46B	108091	9/27/1988
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Frog #91	104884	5/8/1988	Frog #3	126210	10/29/1989
Frog #92	104885	5/8/1988	Frog #1274	126212	10/27/1989
Frog #93	104886	5/8/1988	Frog #1275	126213	10/27/1989
Frog #94	104887	5/8/1988	Frog #1277	126215	10/27/1989
Frog #96	104889	5/17/1988	Poison Springs 1A	146318	7/19/1993
Frog #98	104891	5/17/1988	Poison Springs 3A	146319	7/19/1993
Frog #107	104900	5/20/1988	Poison Springs 5A	146320	7/20/1993
Frog #108	104901	5/20/1988	Poison Springs 6A	146321	7/20/1993
Frog #109	104902	5/20/1988	Poison Springs 7A	146322	7/18/1993
Frog #110	104903	5/20/1988	Poison Springs 8A	146323	7/18/1993
Frog #111	104904	5/20/1988	Poison Springs 9A	146324	7/19/1993
Frog #112	104905	5/19/1988	Poison Springs 11A	146325	7/19/1993

Frog #113	104906	5/19/1988	Poison Springs 14A	146326	7/18/1993
Frog #133	104926	5/20/1988	Poison Springs 16A	127904	1/28/1990
Frog #134	104927	5/20/1988	Poison Springs 17A	127905	1/28/1990
Frog #135	104928	5/20/1988	Poison Springs 18A	146327	7/18/1993
Frog #136	104929	5/20/1988	Poison Springs 22A	146328	7/18/1993
Frog #147	104940	5/22/1988	Poison Springs 26A	146329	7/18/1993
Frog #148	104941	5/22/1988	Poison Springs 27A	146330	7/19/1993
Frog #149	104942	5/22/1988	Poison Springs 38A	146331	7/18/1993
Frog #150	104943	5/22/1988	PGM 10	174057	3/30/2017
Frog #167	104960	5/19/1988	PGM 11	174058	3/29/2017
Frog #168	104961	5/19/1988	PGM 12	174059	3/29/2017
PGM 1	174048	3/29/2017	PGM 13	174060	3/29/2017
PGM 2	174049	3/29/2017	PGM 14	174061	3/29/2017
PGM 3	174050	3/31/2017	PGM 15	174062	3/29/2017
PGM 4	174051	3/30/2017	PSR 1	174063	3/30/2017
PGM 5	174052	3/30/2017	PSR 2	174064	3/29/2017
PGM 6	174053	3/31/2017	PSR 3	174065	3/29/2017
PGM 7	174054	3/31/2017	PSR 4	174066	3/29/2017
PGM 8	174055	3/31/2017	PSR 5	174067	3/29/2017
PGM 9	174056	3/31/2017	PSR 6	174068	3/29/2017
Poison Springs #1	74965	5/1/1984	Poison Springs #16	74980	5/2/1984
Poison Springs #2	74966	5/1/1984	Poison Springs #17	74981	5/2/1984
Poison Springs #3	74967	5/1/1984	Poison Springs #18	74982	5/3/1984
Poison Springs #4	74968	5/1/1984	Poison Springs #19	74983	5/3/1984
Poison Springs #5	74969	5/1/1984	Poison Springs #20	74984	5/3/1984
Poison Springs #6	74970	5/1/1984	Poison Springs #21	74985	5/3/1984
Poison Springs #7	74971	5/1/1984	Poison Springs #22	74986	5/3/1984
Poison Springs #8	74972	5/1/1984	Poison Springs #23	74987	5/3/1984
Poison Springs #9	74973	5/1/1984	Poison Springs #26	74990	5/25/1984
Poison Springs #10	74974	5/1/1984	Poison Springs #27	74991	5/24/1984
Poison Springs #11	74975	5/1/1984	Poison Springs #28	74992	5/24/1984
Poison Springs #12	74976	5/1/1984	Poison Springs #32	74996	5/25/1984
Poison Springs #13	74977	5/2/1984	Poison Springs #36	82455	4/5/1985
Poison Springs #14	74978	5/2/1984	Poison Springs #37	82456	4/5/1985

Poison Springs #15	74979	5/2/1984			

Calico Resources USA Corp. Owned Unpatented Millsite Claims					
Claim Name/Number	BLM ORMC Number	Location Date	Claim Name/Number	BLM ORMC Number	Location Date
Don #1	108077	9/28/1988	Don #6	108082	9/28/1988
Don #2	108078	9/28/1988	Don #7	108083	9/28/1988
Don #3	108079	9/28/1988	Don #8	108084	9/28/1988
Don #4	108080	9/28/1988	Don #9	108085	9/28/1988
Don #5	108081	9/28/1988			

Calico Resources USA Corp. Owned Patented Claims
Claim Name/Number
Poison Springs 24
Poison Springs 25
Poison Springs 35

Bishop I Association Placer Claims		
T22S, R43E, Willamette Meridian		
Claim Name	**BLM Serial No.**	**Section**
Bishop 1	ORMC 116169	W1/2SW1/4 Section 1
Bishop 2	ORMC 116170	S1/2SW1/4 Section 11 and N1/2NW1/4 Section 14
Bishop 3	ORMC 116171	SE1/4 Section 11
Bishop 4	ORMC 116172	NE1/4 Section 14
Bishop 5	ORMC 116173	SE1/4 Section 14
Bishop #5	ORMC 125516	NW1/4SE1/4 Section 14