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Castle Mountain Project

Technical Report on the Castle Mountain Project Feasibility Study

San Bernardino County, California, USA

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DATE AND SIGNATURES PAGE

The effective date of this report is February 26, 2021. The issue date of this report is March 17, 2021. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons.

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CASTLE MOUNTAIN PROJECT
TECHNICAL REPORT ON THE CASTLE MOUNTAIN PROJECT FEASIBILITY STUDY

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1 SUMMARY

1.1 INTRODUCTION

Equinox Gold Corp. (Equinox, or the Company) retained independent consultants to prepare a Feasibility Study Technical Report (the Report, or the Study) for the Castle Mountain Project (the Project) in the state of California, USA. Equinox, through its indirect wholly owned subsidiary NewCastle Gold Ltd (NewCastle), has 100% of the right, title, and beneficial interest in and to Castle Mountain Venture (CMV) which owns the Castle Mountain Mine (or the Property).

The Castle Mountain Project is being developed in two stages, Phase 1, and Phase 2. The Phase 1 project, completed in 2020 and currently operating, consists of a double-lined run of mine (ROM) heap leach facility to treat 14,000 short tons of ore per day (ton/d). The Phase 2 project will consist of the expansion that is described within this Report.

Key aspects included in the Study for Phase 2 are further advancement on metallurgical testwork, updates to the mineral resource and mineral reserve estimates and pit design, updated mine schedule, expanded heap leach operations, the addition of a mill to process higher-grade ore, a filtered tailings facility, infrastructure development, cost estimates, and a financial model.

This Study includes supporting engineering and design to provide a feasibility level of accuracy in the project estimates and includes detailed assessments of resources and reserves, metallurgy, mining, processing, environmental and other relevant considerations that demonstrate the viability of the expansion of the project. Mineral resource and reserve estimates disclosed within this Report supersede all previous estimates for the Castle Mountain Project. The following companies contributed to the study:

- M3 Engineering & Technology (M3), Arizona, USA – Mineral processing, metallurgical testing, recovery methods, process infrastructure, economic evaluation and analysis, environmental and permitting studies.
- Equity Exploration Consultants Ltd. (Equity), Vancouver, British Columbia, Canada – Mineral resources, geology.
- Nilsson Mine Services Ltd. (NMS), Vancouver, British Columbia, Canada – Mineral reserves, pit designs, waste designs, mining costs.
- Geo-Logic Associates (GLA), California, USA – Heap Leach pad design and water balance.
- The MINES Group Inc. (MINES), Nevada, USA – Filtered tailings facility design and meteoric analysis.

This executive summary highlights the work on the feasibility study between 2019 and 2021 and outlines the planned transition from current operations (Phase 1) to the proposed Phase 2 expansion of the Project.

1.2 PROPERTY LOCATION AND INFRASTRUCTURE

The Project is in the historic Hart Mining District, at the southern end of the Castle Mountains, San Bernardino County, California, located 60 miles (100 km) directly south of Las Vegas, Nevada. The Project is in a high desert area near the Mojave National Preserve and Castle Mountains National Monument.

Year-round road access is available from the city of Las Vegas, Nevada approximately 70 miles (113 km) by road north of the Project. The road access is paved highway from Las Vegas to Walking Box Ranch Road, and then by an 18 mile (29 km) unpaved two-lane road to the Project area. This existing access road is well maintained and of good quality for necessary vehicular access as required for construction and operation of the Project.

Existing site infrastructure includes:

- Administration and modular mine offices,
- Main haul road connecting the backfilled JSLA pit with the ROM heap leach pad,
- Phase 1 processing plant which includes solution handling pumps, solution storage tanks a carbon column plant and cyanide unloading and storage area,
- A 24 M gal (90 ML) lined event pond,
- A diesel power generation plant, and
- An assay and metallurgical laboratory.

Many of these currently operating facilities will continue to operate into Phase 2. A location and access map of the project is presented in Figure 1-1.

Figure 1-1: Site Location and Access Map

1.3 PROPERTY OWNERSHIP

The Property includes eight patented claims and 1,226 unpatented lode, placer and mill site claims which are registered under the Castle Mountain Venture and Viceroy Gold Corporation, which are wholly owned subsidiaries of Equinox. Many of the claims overlap and as such the total area of the individual claims is not representative of the overall total area covered.

Table 1-1: Summary of Land Tenure by Type at the Castle Mountain Mine

Type	Claims	Area (acres)	Area (hectares)
Patented lode	8	1,301	526
Unpatented lode	449	8,980	3,634
Unpatented mill site	723	3,598	1,456
Unpatented placer	54	3,639	1,473
Total	**1,234**		

Equinox acquired NewCastle on December 22, 2017 and NewCastle became a wholly owned subsidiary of Equinox. The transaction was a three-way merger between Trek Mining Inc, NewCastle Gold, and Anfield Gold Corp. with the resulting company renamed to Equinox Gold Corp. NewCastle has 100% of the right, title and beneficial interest in and to Castle Mountain Venture (CMV) which owns the Castle Mountain Mine.

Throughout this summary, NewCastle (or CMV) are used when referring to the owner/operator of the Castle Mountain Mine. Equinox's ownership and control of NewCastle and CMV are implicit whenever they are mentioned. Where necessary for clarity, NewCastle and Equinox are explicitly named.

Equinox has full legal access to the Project with respect to surface and mineral rights. All claim maintenance payments to the United States Bureau of Land Management (BLM) and property tax payments to San Bernardino County are in good standing. There are no known dates of expiration to mining claims pertinent to the Project.

The Project is subject to several royalties which are payable to different parties. The Franco-Nevada royalty applies to all ounces from the Project, and the other royalties are area specific. Royalties payable include:

- 2.65% Franco-Nevada royalty applied to all ounces
- 5.00% Conservation royalty
- 2.00% American Standard royalty
- 5.00% Huntington Tile royalty

There are no known environmental liabilities on the Project.

1.4 CLIMATE AND PHYSIOGRAPHY

Castle Mountain experiences a desert climate with hot summers and cool winters, with temperatures attenuated by altitude and aridity relative to the surrounding valleys. Average daily lows and highs for the project site range from 28°F to 52°F (-2°C to 11°C) in the winter and 66°F to 93°F (19°C to 34°C) in the summer while annual average precipitation is just over 9 inches (in) (230 mm).

The Project is in the eastern Mojave Desert which transitions to the Basin and Range region to the north and the Colorado Desert to the south. The Castle Mountains are a relatively small range extending north-northeast from the northern end of Lanfair Valley in California into Piute Valley in Nevada. The Project is located near the southern end of the Castle Mountain range with elevations at the Project site ranging from approximately 4,100 ft to 5,100 ft (1,250 m to 1,555 m).

1.5 HISTORY

The Hart Mining District covers the southern end of the Castle Mountains. Several hundred old prospects, pits, trenches, waste rock dumps and underground workings extend over an approximate two square miles (5.2 km^2) area overlapping the Project area. In 1907, three underground gold mines were brought into production at Oro Belle, Big Chief and Jumbo, and by 1911, the mined veins were exhausted.

A resurgence in exploration activity commenced in 1968 until the early 2000's with a variety of operators. Viceroy Gold Corporation (Viceroy) together with MK Gold Corporation completed a feasibility study and commenced gold production at Castle Mountain in 1991. By 1996, the Jumbo South and Leslie Ann (JSLA) deposits were considered exhausted. JSLA was subsequently backfilled with waste rock from the Jumbo and Oro Belle pits. Mining from the Jumbo pit ceased in 2001 due to localized pit-wall stability issues resulting in the deepest bench mined approximately 200 ft (61 m) above the final depth of planned pit design. Mining from the Oro Belle and Hart Tunnel deposits ceased in 2001 due to low gold prices. Heap leaching continued until 2004, primarily in a rinsing operation to recover residual gold values and reduce the cyanide levels in the heap. Reclamation began in 2001 and by 2012 all criteria for successful reclamation had been met.

A total of 1.24 Moz was recovered from 36.2 Mton (32.8 t) processed at an average grade of 0.043 oz/ton (1.47 g/t) with a combined average recovery of 80% from milled and heap leached ore between 1991 and 2004.

Minimal exploration activity occurred between 2005 and 2011. NewCastle (then Castle Mountain Mining Company Limited) acquired the Project in 2012.

1.6 GEOLOGICAL SETTING

The Castle Mountain gold deposit is located in the Hart Mining District. Proterozoic metamorphic and plutonic rocks form the basement of the Castle Mountains; these are overlain by pre-volcanic sediments, and Miocene sedimentary and volcanic rocks.

The oldest known unit in the stratigraphic package is metamorphic Proterozoic basement rocks comprised of a massive sequence of biotite schist, biotite gneiss and meta-granite. Locally overlying the basement rocks is a Proterozoic sedimentary sequence of conglomerate with lesser sandstone. The regionally extensive Peach Springs Tuff unconformably overlies the Proterozoic units.

The Miocene-age Castle Mountains Volcanic Sequence (CMVS) includes all volcanic units above the Peach Springs Tuff and below the Piute Range volcanic rocks. The CMVS was emplaced during three intrusive-extrusive episodes between around 18.8 and 13.5 million years ago.

The CMVS is defined by the Jacks Well Formation characterized by epiclastic and volcanic rocks with minor mudstone, the Linder Peak rhyolitic volcanic and volcaniclastic rocks, and the Hart Peak rhyolite and late dacite dikes. Linder Peak is represented by a complex suite of volcanics and volcaniclastics including flow-domes, and clastic tuffs comprised of monolithic breccia, polylithic breccia, and ashfall tuffs.

Castle Mountain Project is classified as a low-sulfidation epithermal gold deposit. CMVS rocks are the primary host of epithermal gold mineralization at the Project. Structure and associated rock porosity-permeability characteristics are the first-order control on the distribution of gold. Silica alteration and iron oxide minerals generally occur with gold mineralization. Gold and electrum are the dominant gold-bearing minerals identified from gold deportment studies.

1.7 EXPLORATION AND DRILLING

Exploration by NewCastle includes an airborne LiDAR survey, geophysical surveys including Transient Electromagnetic (TEM) and gravity, detailed mapping and surface grab and chip sampling. The deposit area exposures were mapped in detail and combined with a comprehensive geochemical and petrographic study of the rock types to evaluate the structural and stratigraphic setting. NewCastle exploration work was streamlined to create a framework for logging and relogging that was integrated into a refined geologic model including lithology, oxidation, structure, and alteration models for this study.

Grid-controlled rock sampling was conducted over seven prospective areas to expand on the rock and soil sampling completed by Viceroy. Future exploration should follow up on geochemical anomalies and mineralized trends on East Ridge, East Flats and Egg Hill, Northwest Rim and Benson.

Drilling on the Project is summarized by the material type intersected, the in-situ hard rock or the backfill and waste dump materials, respectively. Purpose designed drill holes have been completed to support the Feasibility Study, including drilling for samples for metallurgical testing, infrastructure condemnation, geotechnical study, and potential water sources.

Diamond, reverse circulation (RC) and conventional rotary (rotary), drilling methods have been used within the hard rock with a total of 1,557,140 ft (474,597 m) within 2,111 holes. The legacy drilling completed by Viceroy was completed entirely within hard rock material using rotary, RC and diamond drilling methods for a total of 1,184,180 ft (360,920 m) within 1,772 drill holes. NewCastle has completed an additional 372,960 ft (113,677 m) of hard rock drilling in 339 drill holes at the Project, primarily using angled RC and diamond core drilling to improve the geological understanding of the deposits.

The JSLA backfill and waste dumps have been drilled exclusively by NewCastle in 1,685 reverse air blast (RAB) and RC holes with a total footage of 370,212 (112,835 m).

Blastholes were used to monitor production during historical Viceroy operations. The blasthole samples cover the benches in the Jumbo and Oro Belle pits and a small portion of the benches in JSLA.

1.8 SAMPLING AND VERIFICATION

Samples from the Viceroy and NewCastle exploration drilling have been utilized in preparing the Mineral Resource Estimate. Core and RC sample intervals are a nominal 5 ft (1.5 m) length but range from 2 ft to 7 ft (0.6 - 2.1 m) in length.

Core and chip samples from diamond, RC, and RAB holes were transported to the secure on-site logging facility where they were processed and prepared for shipment by NewCastle. NewCastle maintained a Quality Assurance/Quality Control (QA/QC) sampling program, including insertion and review of coarse blanks, certified reference materials (CRM), and duplicates. Samples were shipped directly to the independent laboratory for preparation and analyses.

NewCastle drill hole samples were prepared and assayed by ALS Global (ALS) or Bureau Veritas (BV), formerly Inspectorate, at their facility in Reno or Elko, Nevada. Check assays were completed at American Assay Laboratories in Sparks, Nevada. All the laboratories are International Standards Organization (ISO) accredited operations which are independent of Equinox.

Gold was assayed by 1.06 oz (30 g) fire assay with atomic absorption spectroscopy finish (AAS). Gold assays returning greater than 0.2917 oz/ton (10.00 g/t) gold were re-assayed by fire assay with a gravimetric finish and gold assays returning greater than 0.006 oz/ton (0.2 g/t) gold were analyzed for gold cyanide solubility.

Viceroy drill hole samples were collected at 5 ft (1.5 m) intervals over the entire length of each drill hole. Routine pulp duplicate analyses were performed at the primary lab. The QA/QC practices implemented by Viceroy do not have current records; however, check assay samples submitted to umpire commercial labs and the Castle Mountain Mine lab (that was in operation at the time that Viceroy operated the mine) did not indicate systematic bias or accuracy issues with the original assays from the primary labs (Temkin, 2012).

Legend and Rocky Mountain Geochemical (RMG) in Reno, Nevada were the primary laboratories used by Viceroy. Both laboratories were independent of Viceroy; however, neither was accredited.

Viceroy drill hole samples were analyzed for gold and silver by fire assay on a one-assay ton (29.166 g) subsample followed by AAS finish, with samples returning gold values greater than 0.100 oz/ton (3.43 g/t) being re-assayed by fire assay on a one-assay ton subsample with a gravimetric finish.

NewCastle collected 647 bulk density measurements which have been converted to tonnage factors and coded to the lithological model.

NewCastle operations followed a standard operating procedure for processing, data collection, and sampling of the drill holes. All samples had adequate security and tracking measures employed during preparation and transport. Records of the drilling and samples were retained at the Property and at the Vancouver office.

The data used in the resource models and resource estimation was reviewed for critical errors and to evaluate the quality of the data. Location data for the collars and downhole survey measurements were checked for gross errors and coordinate conversion accuracy. The assay data was checked for ranking accuracy and the QA/QC results were evaluated statistically and plotted for visual evaluation. Given the high proportion of Viceroy samples within the hard rock database, the results were reviewed in twin drill hole analysis, sample pair analysis and an evaluation for downhole contamination. The bulk density measurements were verified against the lab certificates. It is the QP’s opinion that the sample preparation, security, and analytical procedures are adequate. The results of the data verification demonstrate the data is adequate for use in Mineral Resource estimation and preparation of Mineral Reserves.

1.9 MINERAL PROCESSING AND METALLURGICAL TESTING

Significant metallurgical testwork has been performed on Castle Mountain samples from 2015-2020. As the plan is to process lower grade ROM ore on a leach pad and higher grade ore using conventional milling with Carbon-in-Leach (CIL), there was a need to carry out a wide extent of testing for each process route and on a wide variety of samples. Data from this work along with historical production data has formed the basis for the project process design criteria.

Testwork performed in 2020 has supplemented extensive test programs previously conducted in 2015 and 2018. Drill core samples were used, and the focus was on expanding the metallurgical understanding of the material to be processed through increased spatial and lithological representation within the mineral resource. The key testwork carried out included:

- Column leach tests on heap leach grade ore using the same parameters as in prior testing to verify and supplement the results,
- Column load permeability tests, and
- Gravity concentration followed by leaching of the gravity tails and whole ore leaching of higher-grade mill feed samples.

Additional test programs conducted in 2020 to support the feasibility study include:

- Mineralogical analysis and gold deportment study,
- Materials handling and comminution tests,
- Carbon loading and oxygen uptake tests,
- Cyanide detoxification tests,
- Thickening, tailing filtration and slurry rheology tests,
- Filtered tailings geotechnical stability analysis, and
- Testwork to determine the potential amenability to ore sorting.

Castle Mountain ore in general can be characterized as friable but moderate to relatively hard based on the testwork considered. Based on the testwork, bond ball work indices ranged from 12.3 to 18.0 kWh/ton (13.6 to 19.8 kWh/t). A weighted average of 15.2 kWh/ton (16.7 kWh/t) based on lithology was selected for the design of the grinding circuit. The Axb results from seven SMC tests ranged from 38.1 to 56.1 while the 80th percentile was 43.0.

The arithmetic average gold recovery from all column leach tests was 80%, while the weighted gold recovery based on ounces per lithology type was 82%. The historical production data from 1992 to 2004 was over 76% recovery specifically for the heap leach ore. Considering lab and historical operating data combined with the plan to leach ROM size ore, the permeability, and effective leaching of the side slopes, the expected LOM heap leach gold recovery is expected to be 67% during the LOM operation and 74% after final rinsing.

For mill grade ores processed through the mill with gravity concentration and a leach/CIL circuit with 30 hours of retention time, an overall gold recovery of 94% is expected.

1.10 MINERAL RESOURCE ESTIMATES

Equity completed a Mineral Resource estimate update for Equinox's Castle Mountain Project, inclusive of both Phase 1 and Phase 2 resources and includes both the JSLA pit backfill material and in-situ hard rock mineral resources.

The Mineral Resources presented herein conform with the most recent CIM Definition Standards (CIM, 2014), have been prepared according to CIM Best Practice Guidelines (CIM, 2019), and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101 Standards of Disclosure for Mineral Projects (BCSC, 2016).

Equity is satisfied that the resource estimates and classification of resources reported herein represent a reasonable estimate of the gold contained in the Castle Mountain Project. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resources will be converted into mineral reserves. Inferred resources have a greater amount of uncertainty as to their existence and whether they can be mined legally or economically. It is reasonably expected that the majority of Inferred resources could be upgraded to Indicated (or Measured) with continued exploration.

The CIM Definition Standards on Mineral Resources and Reserves (CIM, 2014) state that:

"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."

In order to sufficiently test the reasonable prospects for eventual economic extraction by an open pit, pit shells were generated using the variable slope Lerchs Grossmann algorithm in Hexagon's MinePlan® software. The results of the pit optimization partially form the basis of the Mineral Resource Statement and are used to constrain the Mineral Resource with respect to the CIM Definition Standards. Pit optimization does not constitute an attempt to estimate reserves. A summary of the Measured, Indicated and Inferred Resources exclusive of Reserves are summarized in Table 1-2 and Table 1-3 for imperial and metric units, respectively.

Areas of uncertainty that may materially impact the Mineral Resource estimate include commodity price assumptions, metal recovery assumptions, mining and process cost assumptions, pit slope angles and applied top cut values. In the opinion of the QP there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors which would materially affect the Mineral Resource estimate.

Table 1-2: Castle Mountain Open Pit Resources Exclusive of Reserves (Imperial units)

Classification	Au Cut-off	Tons	Au	Contained Au
	(oz/ton)	(kton)	(oz/ton)	(koz)
Measured	0.005	861	0.020	17
Indicated	0.005	80,975	0.018	1,453
Measured and Indicated	0.005	81,836	0.018	1,470
Inferred	0.005	77,048	0.018	1,422

Notes:
1. Mineral Resources are reported exclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500/oz gold.
3. Open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold and are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarized in Table 14-19.
4. The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Any discrepancies in the totals are due to rounding.
7. Mineral resources from the Castle Mountain Project presented herein have an effective date of June 30, 2020.

Table 1-3: Castle Mountain Open Pit Resources Exclusive of Reserves (Metric units)

Classification	Au Cut-off	Tonnes	Au	Contained Au
	(g/t)	(kt)	(g/t)	(koz)
Measured	0.17	781	0.68	17
Indicated	0.17	73,452	0.62	1,453
Measured and Indicated	0.17	74,233	0.62	1,470
Inferred	0.17	69,890	0.63	1,422

Notes:
1. Mineral Resources are reported exclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500 /oz gold.
3. Open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold and are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarised in Table 14-19.
4. The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Any discrepancies in the totals are due to rounding.
7. Mineral resources from Castle Mountain Project presented herein have an effective date of June 30, 2020.

1.11 MINERAL RESERVE ESTIMATES

The Proven and Probable Mineral Reserves at the Castle Mountain Project have been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves. The Project Mineral Reserves are based on the conversion of the Measured and Indicated Resources within the Feasibility Study mine plan, with open pit phase designs guided by Lerchs-Grossmann optimized pit shells.

The Mineral Reserve estimate for the Castle Mountain Project, effective June 30, 2020 is summarized in Table 1-4. The Mineral Reserves have been reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold.

Table 1-4: Mineral Reserve Statement

Imperial	Tons (kton)	Gold Grade (oz/ton)	Gold (koz)
Proven	93,600	0.016	1,498
Probable	190,690	0.014	2,670
Total Proven and Probable	**284,290**	**0.015**	**4,168**
Metric	**Tonnes (kt)**	**Gold Grade (g/t)**	**Gold (koz)**
Proven	84,910	0.55	1,498
Probable	172,990	0.48	2,670
Total Proven and Probable	**257,900**	**0.51**	**4,168**

Notes:
1. The Mineral Reserve estimate with an effective date of June 30, 2020 is based upon the Mineral Resource estimate prepared for Equinox Castle Mountain Venture by Trevor Rabb P.Geo, and described in Section 14, with an effective date of June 30, 2020.
2. The Mineral Reserve was estimated by Nilsson Mine Services Ltd. with supervision by John Nilsson P.Eng. who is a Qualified Person as defined under NI 43 - 101.
3. Mineral Reserves are reported within the ultimate reserve pit design with overall economics developed for $1350/oz gold with appropriate royalties applied.
4. Mineral Reserves are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold.
5. The mining costs average $1.78/ton ($1.96/t) mined, processing costs are $1.33/ton ($1.47/t) for ROM and $12.62/ton ($13.91/t) for milling. G&A was $0.72/ton ($0.79/t) ore processed.
6. The average process recovery was 73.9% for ROM and 94.5% for milling.
7. Ore tons are reported in thousands of short tons (kton) and ounces for Imperial.
8. Mineral Resource is exclusive of Mineral Reserves.

1.12 MINING

Mining will be an open pit operation using conventional diesel-powered truck and shovel mining equipment. The current Phase 1 operation consists of a 14,000 ton/d (12,700 t/d) ROM operation with a focus on mining backfilled material that was placed in the JSLA pit from the previous mining operation 20 years ago. The Phase 2 expansion will increase production to 53,500 ton/d and extract hard rock material from open pits which will be drilled, blasted, and loaded to mine trucks using hydraulic shovels and wheel loaders. Phase 2 mine production is split with 50,000 ton/d (45,400 t/d) to the heap leach and 3,500 ton/d (3,200 t/d) to the mill.

The Phase 2 mine plan includes 14 years of operation expanding the overall life of mine (LOM) to 19 years and delivering 266.6 Mton (241.9 Mt) of ROM heap leach ore with an average diluted grade of 0.012 oz/ton (0.40 g/t) gold to the leaching operation. The mill will commence operation one year later and will process 17.7 Mton (16.1 Mt) of ore with an average diluted grade of 0.067 oz/ton (2.28 g/t) gold. In some years, a small portion of ROM ore will be crushed and re-directed to the mill when availability permits.

Five pit areas are considered in the reserve statement with pits at JSLA (3 phases), Jumbo, Oro Belle, East Ridge (2 phases) and South Domes (2 phases). There is a total of nine phases of open pit mining starting with JSLA backfill and moving north, and then to South Domes to complete the operation. The material movement by mining phases, ROM leach material and milling ore processed are shown in Figure 1-2 to Figure 1-4 below. The mining sequence of the phases allows for backfilling of waste as the pit reaches final limits.

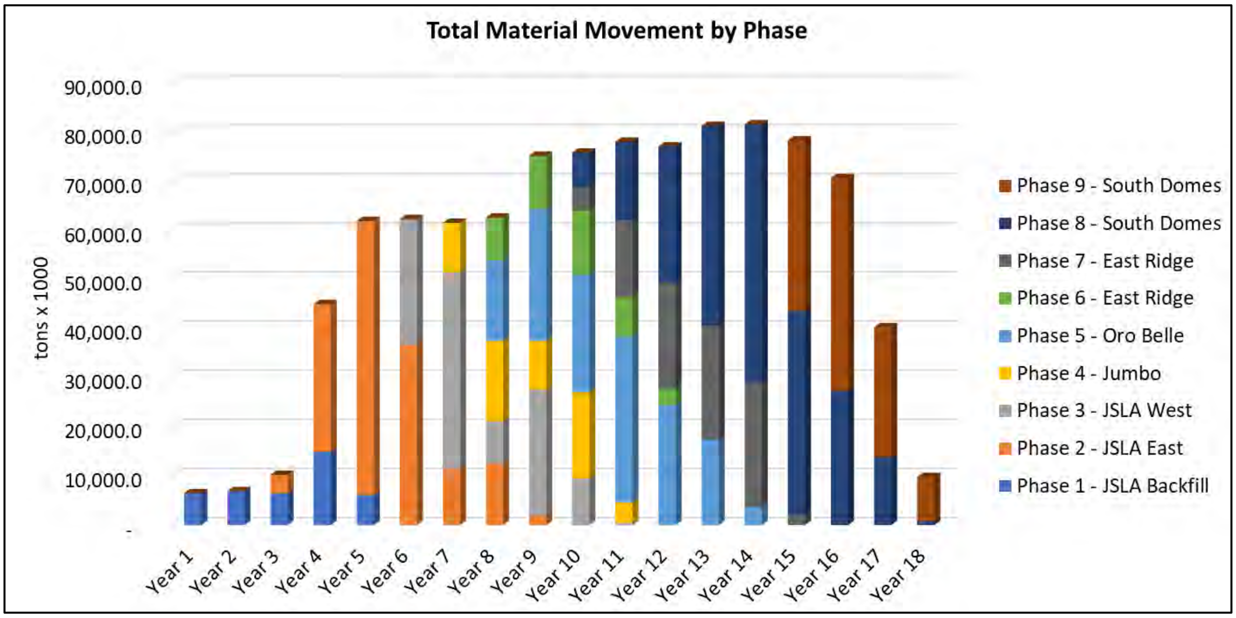


Figure 1-2: Material Movement by Phase

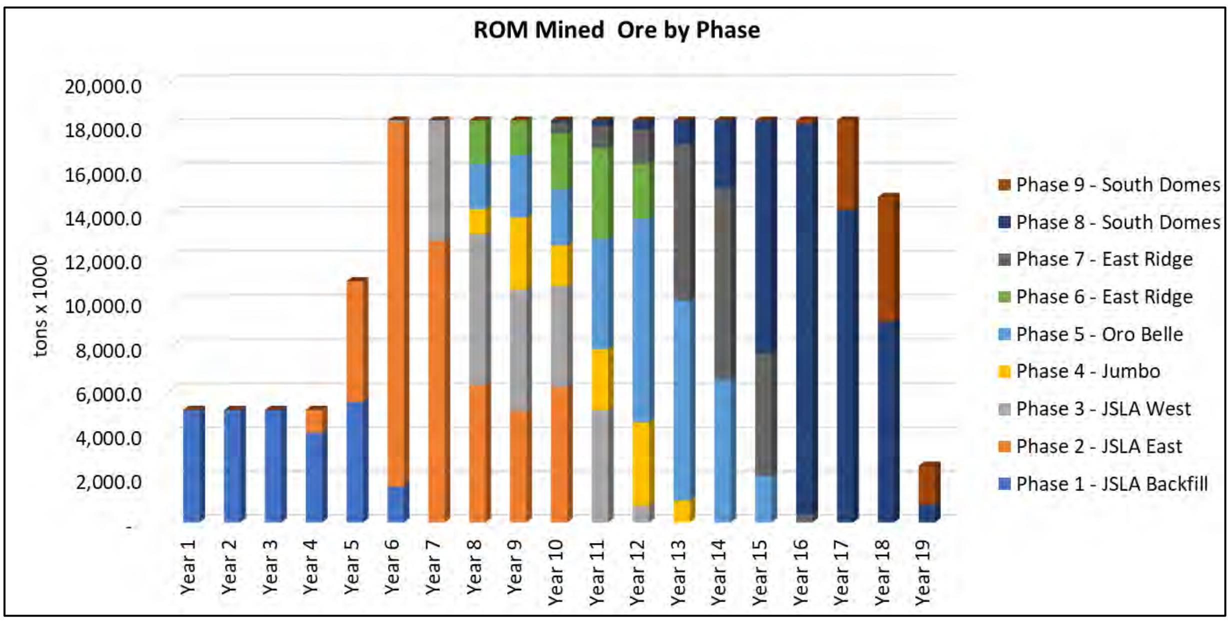


Figure 1-3: ROM Ore by Phase

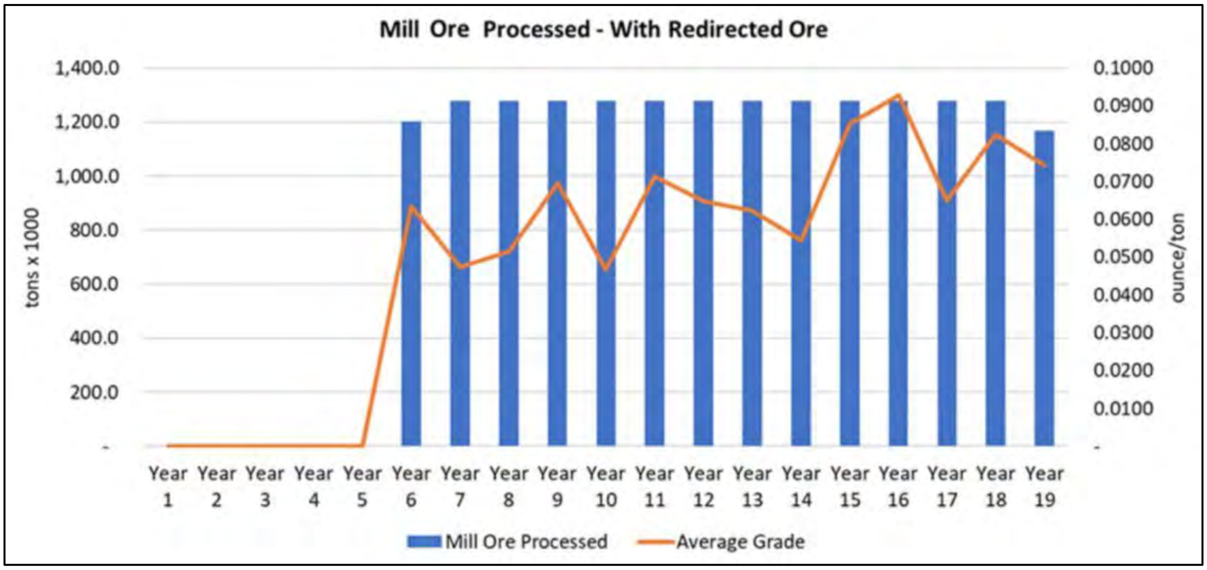


Figure 1-4: Mill Ore Processed

The mine plan incorporates the following elements:

- Staggered mining equipment deliveries in Year 4 and Year 5,
- Ramp up of overall mining rate to 60 Mton/y (54 Mt/y) through to Year 8 then expand gradually to 80 Mton/y (73 Mt/y) through to Year 16 when production begins to drop through Year 19,
- Overall sequence of development in the JSLA, Jumbo, Oro Belle and East Ridge area is clockwise development to final pit limits in each area to allow for an orderly sequence of backfilling waste as pits are completed,
- Sequence at South Domes is an initial southwest pit with an expansion to the northeast, and
- The resource block model was developed on 20 ft (6.1 m) benches. The mine design was developed using the 20 ft bench height with triple benching to 60 ft between design catch benches or berms. Operations are planned for a 30 ft (9.1 m) bench height. Sinking rates in the schedule were limited to 300 ft/y (91 m/y) or the equivalent of 10 benches/year. Drills, loading units and support equipment appropriate for mining a 30 ft bench height have been selected for the mine plan and associated cost estimates.

Phase 1 mining is being conducted by contract mining services. Mine supervision and technical management are handled by the CMV mining team while all other mining functions are the contractor's responsibility. A transition to operator owned mining services or fleet will start prior to Year 5 in parallel with Phase 2 mining. Full Phase 2 mining production coincides with the start of the fully expanded processing facilities, estimated to be in Year 6.

Table 1-5 shows the transition in production of ore and waste over this period. Figure 1-5 illustrates the anticipated project transition from Phase 1 to Phase 2 start up for mining operations.

Table 1-5: Phase 1 to Phase 2 Transition Plan

Operational Phase	LOM Production Year	Phase 2 Expansion	Ore Production (kton)	Waste Production (kton)	Total Production (kton)
Phase 1	1	-5	5,200	1,260	6,460
Phase 1	2	-4	5,160	1,730	6,890
Phase 1	3	-3	5,140	1,960	7,100
Phase 1 + Phase 2 Pre-strip	4	-2	5,140	13,100	18,240
Phase 1 to 2 Ramp-up	5	-1	11,150	42,630	53,780
Phase 2	6	1	19,300	43,830	63,130
Phase 2	7-19	2-14	233,200	597,400	830,600
Subtotal Phase 2			**252,500**	**641,230**	**893,730**
Total Phase 1 + Phase 2			**284,300**	**701,920**	**986,220**

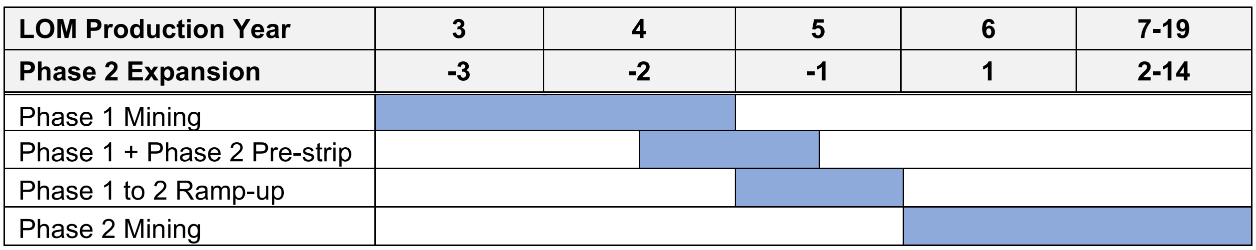


Figure 1-5: Phase 1 to Phase 2 Timeline

The total in-pit waste is 701.9 Mton (636.8 Mt) which is to be placed in the various waste rock management facilities and within open pits once final pit limits are reached. The waste includes 15.0 Mton (13.6 Mt) of Inferred Mineral Resources within the ultimate reserve pit limits which presents an opportunity for future resource classification conversion. The overall strip ratio is 2.47:1. Final waste dump slopes are 2H:1V or 26.5°. There is a northwest waste dump and southeast waste dump designed within the Mine Property boundary.

The mining equipment will operate on 30 ft (9.1 m) high benches with double benching in waste, up to 60 ft (18.2 m) high. Berms will be left on alternate benches in hard rock. Wall slope design recommendations have been implemented for inter-ramp slopes with variable berm widths and bench face angles. Inter-ramp slope angles which vary from 48 to 52° are determined by geological domains, with modified slope angles within structural domains of 40 to 46°. Bench face angles vary from 60 to 79° depending on the domain and host lithology.

Equipment sizing for ramps and working benches is based on the use of 250 ton rigid frame trucks. Haulage and in-pit access roads will be double lane access and have 100 ft (30m) width, which is three times the equipment width plus berm and ditch. The maximum ramp gradients are 10% in-pit but can be constructed to 8% to maximize productivity. Working benches were designed for 115 to 130 ft (35 to 40 m) minimum on pushbacks, although some push-backs do work in a retreat manner to facilitate access.

The initial mining fleet requirement for the Phase 2 expansion that will be purchased in the first three years is summarized in Table 1-6.

Table 1-6: Mining Equipment Summary

Equipment	Details	Total
Production Blasthole Drill	8 7/8"	2
Wall Control Drill	4 1/2' - 9"	2
Hydraulic Shovel	2996 hp 44.5 yd^3	2
Wheel Loader	1739 hp 28 yd^3	2
Haul Truck	2650 hp 250 ton	17
Track Dozer	600 hp	5
Wheel Dozer	620 hp	2
Grader	290 hp 16 ft	3
Water Truck	1450 hp 32,000 gal	2
Wheel Loader	541 hp 10 yd^3	1
Haul Truck	825 hp 61 ton	3
Excavator	524 hp 6 yd^3	1
Tire Manipulator	Large Tire	1
Vibratory Compactor	130 hp 7.5 ft	1
Backhoe	105 hp 1.3 yd^3	1
Articulated Truck	450 hp 40 ton	1
Fuel and Lube Truck	100 ton 8,000 gal	1
Tractor and Low Bed	160 ton	1
Flatbed Hiab Truck	10 ton	1
Rough Terrain Forklift	33 ton	1
Shop Forklift	18 ton	1

Alluvium, backfill, and waste dump material will be free-digging. Hard rock will require drilling and blasting. Ore grade control will utilize rotary blast holes drilled across a full bench height of 30 ft (9.1 m). Blastholes will be grid drilled to facilitate breakage and will be loaded with ammonium nitrate and emulsion explosives. The blastholes will be sampled to provide analytical results for grade control and mine planning. Drilling will be in advance of the mined benches to allow proper short-term planning.

Heap leach ROM ore is being hauled to the existing Phase 1 leach pad. In Phase 2 of the LOM plan, ROM will be hauled to a new, adjacent Phase 2 leach pad that will be developed progressing from South to North, then towards the West. Mill feed will be placed in a stockpile adjacent to the primary crusher and re-handled by wheel loaders to feed the crusher.

1.13 RECOVERY METHODS

The current operation consists of a 14,000 ton/d (12,700 t/d) run of mine (ROM) heap leach operation with gold recovery in carbon columns. The planned expansion for Phase 2 will include a 50,000 ton/d (45,350 t/d) ROM heap leach and a new 3,500 ton/d (3,175 t/d) crushing, milling and leach/CIL plant for recovering gold and silver from mill grade ore.

For Phase 2, the heap leach pad will be designed to process 18.2 million short tons (Mton) (16.5 Mt) annually at an average life of mine (LOM) grade of 0.012 oz/ton (0.54 g/t), while the mill will be designed to process approximately 1.3 Mton (1.2 Mt) annually at an average LOM grade of 0.068 oz/ton (2.28 g/t). Phase 2 expansion will extend operations to approximately 19 years with an additional estimated three years of heap rinsing as part of reclamation where gold will continue to be leached and recovered.

ROM heap leach ore will be loaded into haul trucks and stacked in 25 ft (8 m) lifts on the heap leach pad to be leached with a dilute cyanide solution using a drip irrigation system for 80 days. After percolating through the ore, the pregnant gold and silver bearing solution will flow by gravity to a pregnant solution tank where it is pumped to a 12,000 gpm (750 L/s) carbon-in-column (CIC) circuit to recover the precious metal from solution. The carbon adsorption circuit will consist of two trains of five cascading carbon columns.

ROM mill ore will be loaded into haul trucks and dumped on the ROM storage pad for recovery by a front-end loader and feed to a two-stage crushing plant intended to reduce ore to 80% passing ½ in prior to feeding a single ball mill. The ball mill will be a 16.5 ft x 21 ft long (5 m x 6.4 m) mill equipped with a single 3,300 hp (2,460 kW) wound rotor induction motor with a VFD. The mill will process a nominal throughput of 162 ton/h (fresh feed), producing a final product P_{80} of 150 µm. A batch gravity concentrator will treat a portion of the grinding circuit circulating load to recover any gravity recoverable gold with the concentrate being processed in a batch intensive leach reactor (ILR).

Cyclone overflow will flow by gravity to a 68 ft (21 m) diameter high-rate pre-leach thickener which will thicken the slurry to 45-50% solids. Thickened slurry will be pumped to a hybrid leach/CIL circuit using a series of seven agitated tanks (30 hours retention time) using cyanide solution in the presence of activated carbon to extract the gold. The thickener overflow will flow by gravity to the non-cyanide solution tank to be used as make-up water in the grinding circuit.

The carbon handling circuit is designed to handle carbon from both the heap leach CIC circuit and the mill-CIL circuit in separate batch processes. Loaded carbon at an average of approximately 15 tons/day (13.6 t/d) will be washed with hydrochloric acid and stripped under pressure. An indirect propane-fired rotary carbon regeneration kiln will treat up to 18 tons (16 t) of carbon per day, equivalent to 100% regeneration of stripped carbon.

The resulting pregnant solution from the carbon handling and ILR circuits will undergo electrowinning (EW) in four cells operating in parallel and the recovered precious metal sludge will be dried in a retort furnace to recover any mercury present. The dried sludge will be refined in an induction furnace to produce gold and silver doré. Doré bars will be the final product and will be stored in a vault within a secure area prior to shipment.

Leached slurry from the leach/CIL circuit will report to a cyanide recovery thickener to recycle as much water and cyanide as possible back to the process. Flocculant will be added to aid in settling solids to produce a thickened product at approximately 60% solids, which will be treated in an SO_2/Oxygen cyanide destruction process.

The final tailings will be pressure filtered in two of three tailing filters (1 unit on standby). The filter cake at approximately 18% moisture will discharge to a stockpile to be reclaimed by front end loader and loaded into articulated trucks for haulage to the filtered tailings facility.

The major process equipment for gold recovery is summarized in Table 1-7.

Table 1-7: Major Process Equipment

Item	Quantity	Description	Power
Barren Solution Pump	4	12 in Vertical Turbine	1,000 hp
Pregnant Solution Pump	4	8 in x 10 in Horizontal Centrifugal	150 hp
CIC Column	10	18 ft. diam. 5 per train	-
Primary Crusher	1	49 in x 37 in Jaw	175 hp
Secondary Crusher	1	60 in Standard Cone	500 hp
Ball Mill	1	16.5 ft diameter x 21 ft F/F	3,300 hp
Gravity Concentrator	1	Centrifugal Bowl, 48 in bowl diam.	60 hp
Pre-leach Thickener	1	68 ft diam. High rate	-
Leach/CIL Tanks	7	37 ft diam. x 40 ft height; Agitated	100 hp
Cyanide Recovery Thickener	1	68 ft diam. High rate	-
Filter Feed Pump	2	10 in x 8 in Horizontal Centrifugal	350 hp
Tailing Filter	3	8.2 ft x 8.2 ft Pressure filter, 64 chambers, 16 min cycle	100 hp
Acid Wash Vessels	2	FRP construction	6 ton capacity
Strip Vessels	2	Pressure vessel; Stainless steel construction	6 ton capacity
Carbon Regeneration Kiln	1	5 ft diam. x 50 ft long, Horizontal Propane-Fired Indirect	1,500 lb/h 18 ton/day
Mercury Retort	1	3 ft^3 Electric	30 kW
Electrowinning Cells	4	Sludging, 2000 amps @ 6 volts	-
Smelting Furnace	1	Induction Furnace	450 kW

Reagents used within the plant will be mixed on-site and distributed via reagent handling systems. These reagents include:

- Lime (CaO)
- Sodium cyanide ($NaCN$)
- Hydrochloric acid (HCl)
- Caustic soda ($NaOH$)
- Sodium metabisulfite (SMBS)
- Copper sulfate ($CuSO_4$)
- Flocculant
- Antiscalant

Process water needs for the recovery plant will fluctuate seasonally. Make-up water for the heap leach will change with the amount of evaporation and precipitation each month. Net evaporative losses will range from 150 gpm to 700 gpm (10 L/s to 45 L/s), averaging approximately 400 gpm (25 L/s) annually, while ROM ore on the leach pad will need to be saturated with moisture at an average of 10% and this results in an average consumption of approximately 670 gpm (42 L/s).

Additional water is required for the mill process and will be largely made up with recycled water. The Project will mitigate water consumption by use of low evaporation buried drip emitters, limiting the amount of water retained in ponds with larger evaporative losses, use of binders and dust collectors that limit water needs for dust suppression and by using extensive water recycling in the process.

The Phase 2 expanded Project is anticipated to produce 3,203,000 oz gold over the course of the mine life and rinsing of the heap leach pad.

Figure 1-6 shows the expanded Phase 2 process plant layout.

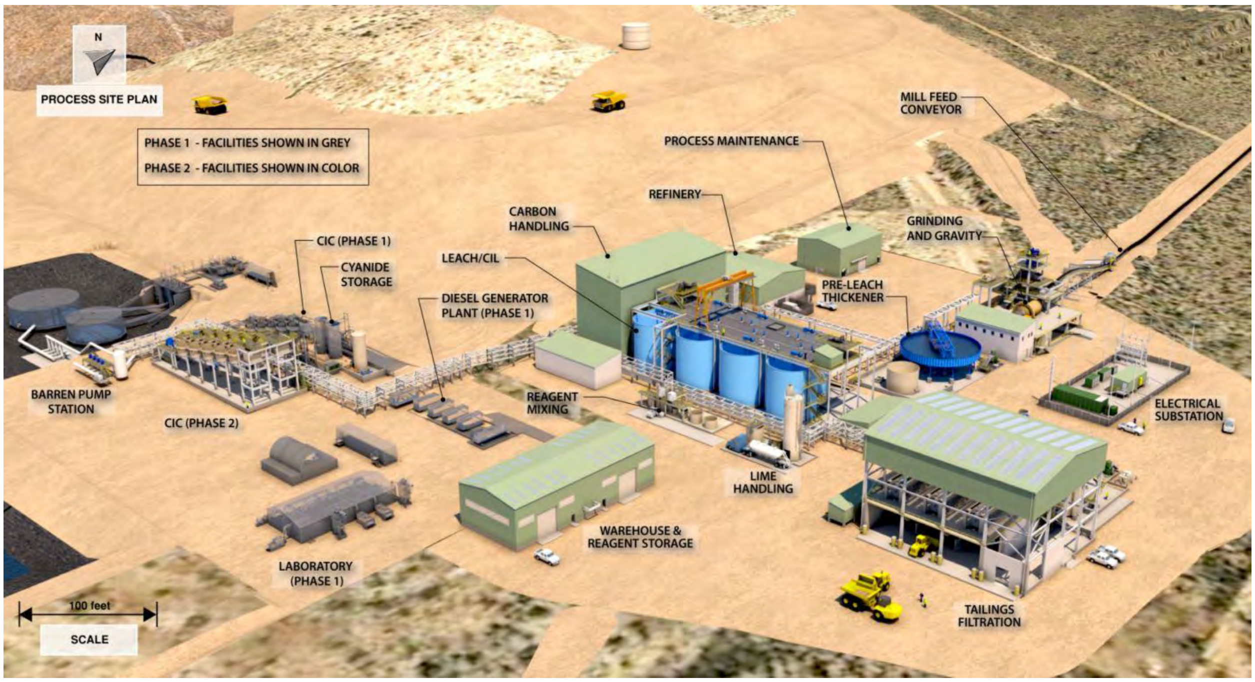


Figure 1-6: Process Plant 3D View Looking Northwest

1.14 PROJECT INFRASTRUCTURE

The Phase 2 expansion will continue to utilize existing facilities including the recently built Phase 1 facilities to the greatest extent possible. Phase 2 infrastructure will increase in size to meet the expanded project parameters and include new site improvements to support the operation of the required new process plant and mining facilities. The project supporting infrastructure will include:

- Site access, on site and service road access (most currently in operation)
- Mining haul roads (currently in operation and to be expanded)
- Truck service shop, fueling station, tire change pad and wash facility
- ROM ore stockpile area
- Water supply and distribution systems
- Surface water management infrastructure
- Lined filtered tailings facility
- Topsoil reserve areas
- Process plant maintenance building
- Reagents storage and warehousing building
- Security gatehouse including medical triage area and evacuation helipad
- Communications system and plantwide process control

Figure 1-7 shows the expanded Phase 2 site plan.

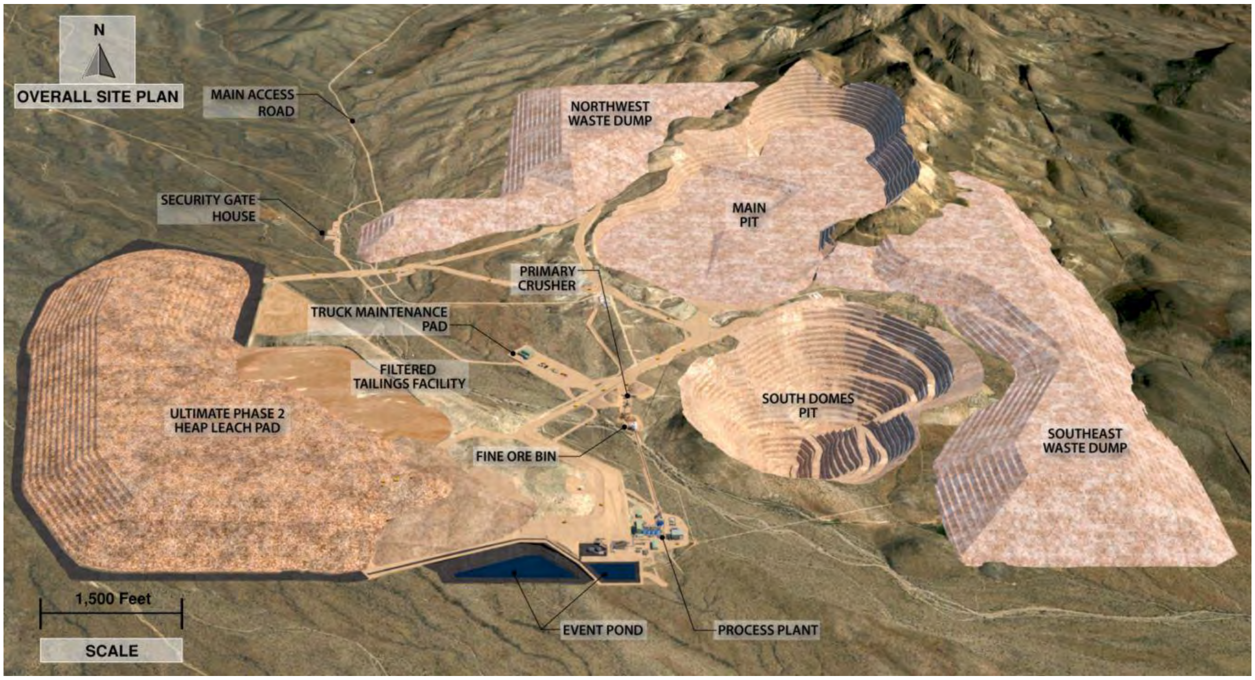


Figure 1-7: Overall Project Site Plan

The Castle Mountain mine will be a net zero discharge facility with regards to water with the main water loss occurring via evaporation from the surface of the heap leach pad and filtered tailings facility. Water is also used in saturating the heap leach pad and dust control mitigation for roads and site development, as necessary. The Project site-wide water balance indicates an expected make-up water demand to range from approximately 1,150 gpm to 1,900 gpm (72 L/s to 120 L/s) depending on the season. In addition to the water use mitigation measures mentioned above, further water demand reduction will be attained through greater use of onsite dust suppressants, strategic seasonal construction planning during wetter months, and optimizing the heap leach make-up water requirements through efficiency improvements.

Water supply at site currently includes three historical wells providing approximately 150 gpm (10 L/s) total and connected via existing underground pipelines to an existing 300,000 gal (1.1 ML) water tank, as well as two production wells, W-01 and W-02, with pumps installed in 2019 at the start of Phase 1 project. These production wells are located at the edge of the JSLA pit (W-01) and in the area of what will become the South Domes pit (W-02). These are bedrock wells which produce approximately 400 gpm (25 L/s) total and are connected to a recently constructed 300,000 gal (1.1 ML) raw water tank.

Additional water for the Phase 2 expansion is expected to be extracted from new wells. Recent water exploration has shown very good potential for both water near site and in a neighboring water basin. It is anticipated that once developed, wells in both areas will be able to produce between 500 and 1,000 gpm (32 and 64 lpm) of water each. The project expansion development includes the addition of new wells, and well pumps in both locations as well as an overland pipeline and booster pumps to meet the make-up water demands.

Electrical power requirements for Phase 2 are approximately 10 MW and this is to be provided by a connection to grid power which will be routed to site via a new transmission line from an existing Nevada Energy (NVE) sub-station near Searchlight, NV, similar to that previously used at the site and along the same right of way. Additional options including solar power have been investigated and could be developed as part of the project construction.

Filtered tailings from the mill will be produced at a moisture content of 19% to 22% by dry weight basis (16-18% wet basis) and will be delivered using 40 ton articulated dump trucks to a lined facility. Stacking of filtered tailings is considered best available technology for handling and placing this type of material.

The tailings will be spread by dozer atop the reclaimed former Viceroy heap leach pad. Development of the filtered tailings facility will occur in four stages to allow for both the placement of appropriate volumes of material to match production and the rinsing of heap leach side slopes which will be directly abutted to the final filtered tailings facility footprint. The heap leach and filtered tailings will form a co-deposited and integrated facility. Rinsing is required to allow for recovery of residual gold ounces within the heap as well as to reduce cyanide levels to compliant levels within the placed heap leach material prior to final reclamation.

By placing filtered tailings abutted to the new heap leach facility and on top of the historic leach pad, the area of disturbance on the site will be minimized. This will increase the long-term stability on the western edge of the facility and allow integrated management of solution between the tailings and heap leach facility, allowing for further recycle of cyanide.

1.15 ENVIRONMENTAL PERMITTING AND REQUIREMENTS

The mine operations encompass both public and private land, accordingly, the County of San Bernardino (County) and the United States Bureau of Land Management (BLM) have served as co-leading agencies for implementing environmental review. The 1990 Environmental Impact Statement / Environmental Impact Report (1990 EIS/EIR), the 1998 Castle Mountain Mine Expansion Project Environmental Impact Statement / Environmental Impact Report (1998 EIS/EIR), and the 2020 BLM Environmental Assessment cumulatively provided authorization for current mine operations.

The County approved minor revisions to the Mine and Reclamation Plan and issued a revised Mining Conditional Use Permit (CUP) and Reclamation Plan 90M-013 which expires December 31, 2035. The 2020 BLM NEPA analysis resulted in the BLM issuing a Decision Record and Finding of No Significant Impact (FONSI) and approved the revised Mine and Reclamation Plan on February 27, 2020.

The Phase 2 mine expansion is expected to require a new or updated environmental review (likely in the format of an EIS/EIR) as well as several new state and federal permits and amendments. The federal lead agency, the BLM, and the California state lead agency, the County, will cooperate to prepare a single environmental review document. Federal, state, county, and local agency officials will review and comment on the analysis provided through the environmental review process.

There will be public review and comment periods initiated by a Notice of Intent. Once the co-lead agencies complete their assessment and publish a Final EIS/EIR, then subsequently, each of the lead agencies prepare their respective approvals. The BLM issues a Record of Decision (ROD) and associated project stipulations to satisfy project specific mitigation measures adopted by the agency to lessen project impacts. The County will ultimately vote to certify the EIR and approve (or deny) the CUP for the Project and associated conditions of approval, which like the BLM, provide mitigation to lessen project impacts.

Once lead agency operating permits have been granted, CMV can apply to remaining local, state, and federal agencies who issue further discretionary and non-discretionary permits.

1.16 CAPITAL AND OPERATING COST

Total initial capital cost is estimated at $389 million excluding the mining equipment fleet which is estimated at $121 million and expected to be leased to own over five years, or a total of $510 million considering the fleet purchased upfront. Capital costs are summarized in Table 1-8 along with the estimated sustaining capital needs of the Phase 2 project and closure costs. Sustaining capital costs for the project are primarily accounting for mining and additional stages of the heap leach pad and filtered tailings facility development. Total sustaining capital costs during production until closure are $147 million. Closure costs totaling $22 million are included separately for the end of mine life. Estimates are expressed in US dollars ($), Q4 2020 with no escalation.

Direct costs as well as all indirect costs and appropriate contingencies for all facilities have been included within the estimate and define the full projected cost to bring the Phase 2 expansion into production as defined by this report.

Initial mining capital costs are based on conversion to an Equinox owned mining fleet from the contract-based fleet being utilized for current operations, necessary parts, and spares for the fleet, as well as slope monitoring equipment and mine development and pre-stripping. A major part of the mining equipment fleet could be leased which results in a reduction of $121 million of initial capital. Leasing the mining equipment adds to the operating cost; however, the net impact is an improvement to the Internal Rate of Return (IRR).

The Project execution strategy is based on an engineering, procurement, and construction management (EPCM) implementation approach. Contingency has been estimated through an analysis of the level of detail in estimating each specific discipline and overall is included at 12.5% on plant and infrastructure items. The contingency has not been applied to mining or working capital as is typically the case. The accuracy of the estimate is defined as -10% to +15%.

Table 1-8: Phase 2 Capital Cost Summary

Item	Initial ($M)	Sustaining ($M)	Total ($M)
Mine Mobile Equipment[1]	154	70	224
Mine Development	41	11	52
Mine Total	**195**	**81**	**276**
General Siteworks	11	-	11
Heap Leach and Solution Handling	38	56	94
Process Plant	62	-	62
Tailings Filtration and Storage	16	1	17
Infrastructure	41	-	41
Freight	8	-	8
Direct Plant and Infrastructure Total	**176**	**57**	**233**
EPCM, Vendor Support and Other Indirects	51	-	51
Transmission Line	15	-	15
Owner's Cost, Working Cap and Taxes	40	-	40
Sub-total Plant and Infrastructure	**282**	**-**	**-**
Contingency	33	9	42
Total CAPEX	**510**	**147**	**657**
Less Leased Mining Equipment	***(121)***	**-**	***(121)***
Total CAPEX (with Leased Mining Equipment)	**389**	**-**	**536**

Note 1: Mining equipment includes all applicable sales tax.

M3 has assembled the capital cost estimate with consulting input from GLA, The MINES Group and NMS as well as Equinox. Utility Transmission Line costs are based on the 2018 PFS estimate from Nevada Energy.

Direct operating costs have been estimated for mining, processing and general and administrative (G&A) costs. Mining costs were developed by NMS, processing, and infrastructure costs by M3 and G&A by Equinox. Total operating costs for the expanded Phase 2 project are $9.32/ton ($10.28/t) of ore processed as described in Table 1-9. Mining equipment purchase costs are all considered capital costs and excluded from operating costs. Table 1-10 shows the estimated cash cost over the course of the expanded Phase 2 project for production of 3,187,000 ounces of payable gold.

Table 1-9: Operating Cost Phase 2 Summary

Description	Unit Cost ($/ton mined)
Mining	1.75
Description	**Unit Cost $/ton ore**
Mining	6.20
Processing (Total)	2.45
G&A	0.65
Sub-Total	**9.30**
Refining and Transportation	0.02
Total	**9.32**

Table 1-10: Phase 2 Cash Cost Estimate

Item	Total Cost ($M)	Unit Cost ($/oz)
Mining	1,567	492
Processing – Heap Leach	365	115
Processing – Mill/CIL	255	80
G&A	164	51
Operating Cost	**2,351**	**738**
Royalties	214	67
Refining and Transportation	5	2
Adjusted Operating Cost	**2,570**	**806**
Sustaining Capital	147	46
Salvage Value	(3)	(1)
Reclamation and Closure	22	7
All in Sustaining Cost (AISC)	**2,736**	**858**

1.17 ECONOMIC ANALYSIS

The economic analysis was completed primarily utilizing a discounted cash flow model. Currency is provided in US dollars, unless otherwise noted. Table 1-11 summarizes the spend plan for initial capital. Table 1-12 summarizes the resulting project economics at a gold price of $1,500/oz.

The Phase 2 project, from an economic analysis perspective, begins with detailed engineering activities and procurement of major equipment in preparation for construction which is expected to start 2.5 years ahead of start-up. The period of project execution resulting in significant capital spend with construction activities will begin approximately 2 years prior to full operations starting. Figure 1-8 illustrates the relationship Phase 2 project related activities through project start up.

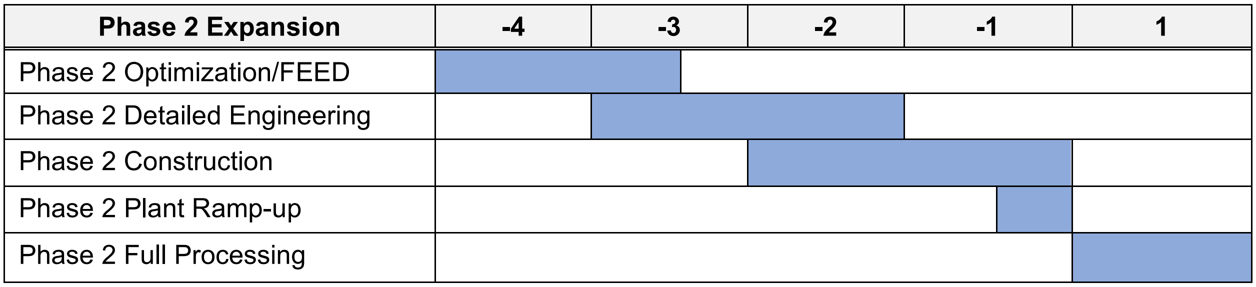


Figure 1-8: Phase 2 Schedule

Table 1-11: Phase 2 Initial Capital Spend Plan

Phase 2 Year	Ore Production (kton/y)	Mining Initial Capital Spend ($M)	Plant Initial Capital Spend ($M)	Working Capital	Total ($M)
Pre-Prod Year -3	5,150	-	28	-	**28**
Pre-Prod Year -2	5,150	109	204	-	**313**
Pre-Prod Year -1	11,150	62	68	14	**144**
Phase 2 Prod Year 1	19,300	24	-	1	**25**
Expanded Operations	19,500	Sustaining	Sustaining	N/A	-

The Phase 2 project cash flow is estimated to be $1,280 million over the 17-year operating life, 14 years of mining with an additional 3 years of rinsing. The Project after-tax NPV at a discount rate of 5% is estimated to be $639 million. The after-tax cash flow results in a 5.3-year payback period after start-up of commercial operation with an after-tax IRR of 17.5%. With leasing the mining fleet, the after-tax NPV remains at $639 million while the after-tax IRR improves to 18.3%, and the payback period is 5.4 years.

Table 1-12: Financial Summary

Category	Units	Value	
Production Summary			
Phase 2 Ore material mined	Mton	894	
Phase 2 Ore tons processed	Mton	253	
Phase 2 Life (Processing)	y	14	
Phase 2 Life (Processing + Rinsing)	y	17	
Heap Leach Ore	Mton	235	
Head grade	oz/ton	0.0119	
Recovery	%	74	
Recovered Gold	koz	2,095	
Mill Ore	Mton	18	
Head grade	oz/ton	0.0665	
Recovery	%	94	
Recovered Gold	koz	1,108	
Total Recovered Gold	koz	3,203	
Total Payable Gold	koz	3,187	
Capital Costs			
Phase 2 Initial Capital	$M	510	
Sustaining Capital	$M	147	
Operating Costs			
Mining	$/ton mined	$1.75	
Mining	$/ton processed	$6.20	
Processing	$/ton processed	$2.45	
G&A	$/ton processed	$0.65	
Refining and Transportation	$/ton processed	$0.02	
Total Operating Cost	$/ton processed	$9.32	
Total Production Cost	$/ton processed	$806	
All-In Sustaining Cost	$/oz Au	$858	
Economic Indicators			
		Without Leasing	**With Leasing**
Internal Rate of Return (IRR), Pre-tax	%	18.9	19.7
Internal Rate of Return (IRR), After-tax	%	17.5	18.3
Undiscounted Cashflow, Pre-tax	$M	1,550	1,539
Undiscounted Cashflow, After-tax	$M	1,280	1,268
Net Present Value (NPV) @5%, Pre-tax	$M	784	784
Net Present Value (NPV) @5%, After-tax	$M	639	639
Payback Period (Based on After-tax)	y	5.3	5.4

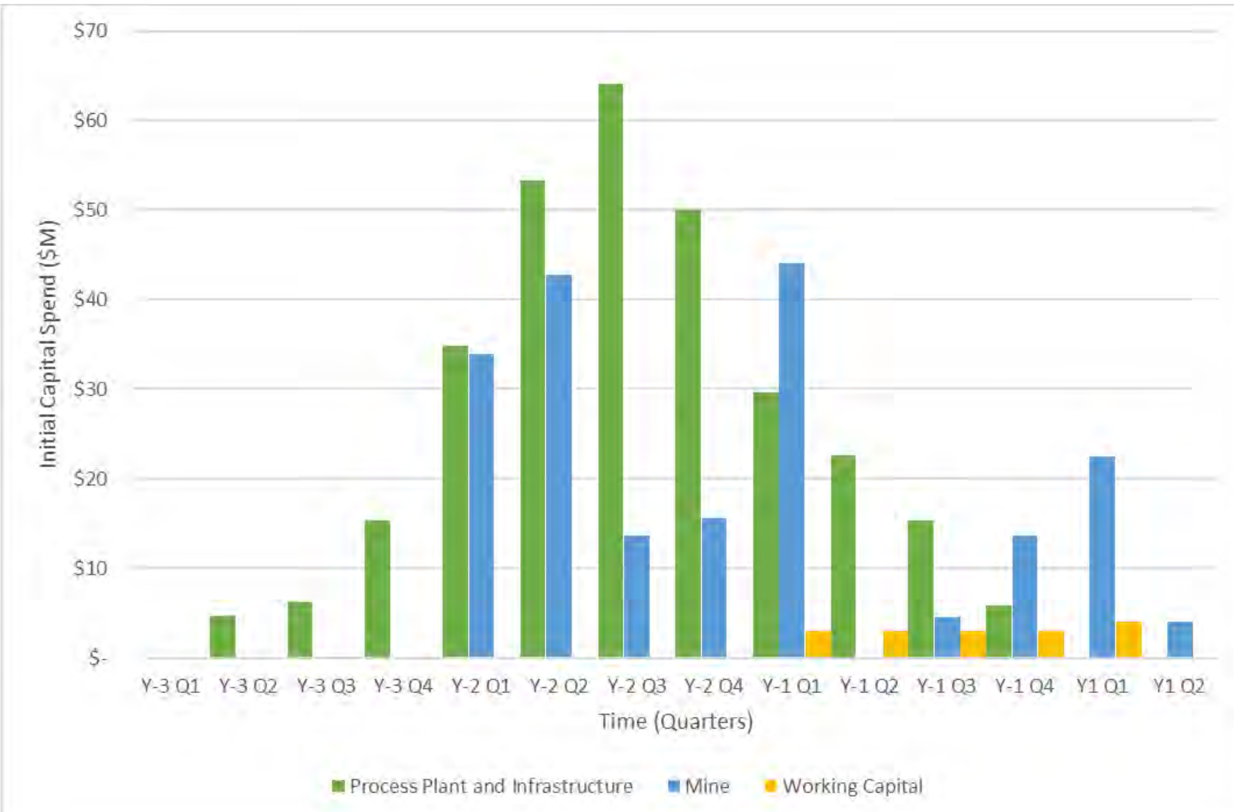


Figure 1-9: Initial Capital Spend Plan (Quarterly)

Figure 1-10 shows the results of the economic sensitivity analysis. The Project is most sensitive to overall operating cost and gold price, and least sensitive to cyanide usage.

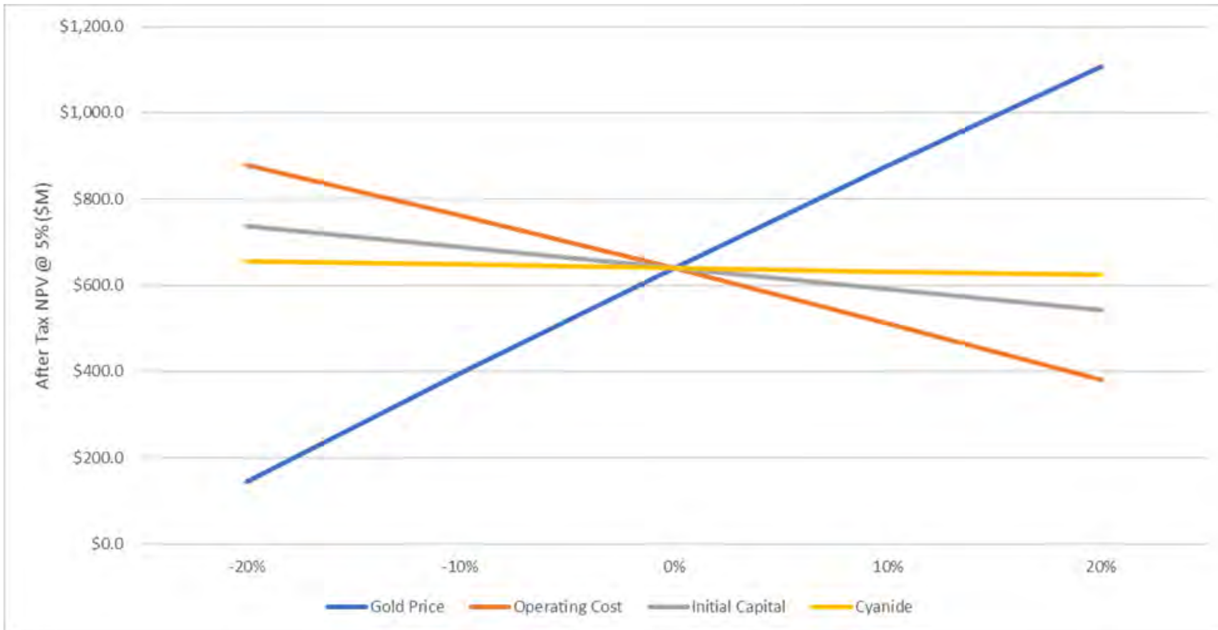


Figure 1-10: Sensitivity Chart

Table 1-13 summarizes the financial model.

Table 1-13: Financial Model Summary

Mining Operations	$/ton mined	$/ton ore	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Total Material Mined (kton)			893,764	-	-	-	63,138	63,186	63,968	73,161	73,781	72,812	73,196	78,450	78,615	78,358	79,877	61,258	28,686	5,278	-	-
Total Ore (ROM & Mill)			252,910	-	-	-	19,453	19,527	19,528	19,528	19,343	19,506	19,528	19,528	19,294	19,527	19,528	18,825	16,052	3,745	-	-
Gold Grade (oz/ton)			0.0157	-	-	-	0.0140	0.0136	0.0138	0.0144	0.0124	0.0144	0.0161	0.0144	0.0145	0.0168	0.0188	0.0152	0.0238	0.0353	-	-
Contained Gold (koz)			3,982	-	-	-	273	265	270	282	240	281	315	282	279	327	366	287	382	132	-	-
Total Recovered Gold (koz)			3,203	-	-	-	203	194	199	213	177	213	233	210	206	249	278	215	285	177	86	65
Payable Metal																						
Payable Gold (koz)			3,187	-	-	-	202	193	198	212	176	212	232	209	205	248	276	214	283	176	86	64
Revenues																						
Net Revenues			$4,776,182	$0	$0	$0	$303,366	$289,475	$296,511	$317,346	$263,788	$317,518	$348,113	$313,672	$307,250	$371,178	$414,051	$320,208	$424,307	$263,961	$128,822	$96,617
Operating Cost	**$/ton mined**	**$/ton ore**																				
Mine	$1.75	$6.20	$1,566,941	$0	$0	$0	$100,003	$118,476	$120,933	$128,005	$130,418	$123,540	$121,970	$134,403	$137,146	$124,005	$128,523	$115,557	$69,164	$14,797	$0	$0
Process Plant - Heap Leach		$1.44	$365,189	$0	$0	$0	$27,259	$27,689	$27,562	$26,984	$27,303	$26,884	$27,401	$27,248	$27,241	$26,843	$26,919	$26,357	$23,179	$8,144	$4,672	$3,504
Process Plant - CIL		$1.01	$254,575	$0	$0	$0	$17,622	$17,793	$17,932	$18,520	$17,988	$18,615	$18,135	$18,299	$18,032	$18,725	$18,660	$18,381	$18,210	$17,664	$0	$0
G&A		$0.65	$164,102	$0	$0	$0	$11,444	$11,452	$11,452	$11,452	$11,433	$11,450	$11,452	$11,452	$11,428	$11,452	$11,452	$11,379	$11,092	$9,819	$3,368	$2,526
Total Operating Cost (Minus Refining)		$9.30	$2,350,808	$0	$0	$0	$156,328	$175,410	$177,879	$184,961	$187,142	$180,488	$178,958	$191,402	$193,848	$181,025	$185,553	$171,674	$121,645	$50,423	$8,040	$6,030
Cash Flow																						
Capital Expenditures																						
Initial Capital																						
Mine			$195,522	$0	$108,978	$62,397	$24,146	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant			$299,546	$28,126	$203,512	$67,909	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital																						
Mine			$80,872	$0	$0	$0	$1,846	$8,131	$13,570	$16,469	$3,342	$650	$7,351	$12,831	$15,473	$236	$662	$236	$75	$0	$0	$0
Process Plant			$66,555	$0	$0	$0	$3,404	$0	$1,295	$26,965	$0	$0	$22,249	$0	$0	$12,641	$0	$0	$0	$0	$0	$0
Total Capital Expenditures			$642,495	$28,126	$312,490	$130,306	$29,396	$8,131	$14,865	$43,434	$3,342	$650	$29,601	$12,831	$15,473	$12,877	$662	$236	$75	$0	$0	$0
Cash Flow before Taxes			$1,549,799	-$28,126	-$312,490	-$130,306	$106,685	$97,812	$93,487	$78,846	$65,860	$126,286	$130,006	$101,645	$88,419	$159,377	$199,090	$121,290	$261,629	$192,037	$110,385	$88,361
Cumulative Cash Flow before Taxes				-$28,126	-$340,616	-$470,922	-$364,237	-$266,425	-$172,938	-$94,091	-$28,231	$98,055	$228,061	$329,707	$418,125	$577,503	$776,593	$897,883	$1,159,512	$1,351,548	$1,461,933	$1,550,294
Taxes			$270,196	$0	$0	$0	$0	$1,602	$1,887	$2,282	$917	$2,927	$11,773	$12,529	$11,791	$27,983	$35,183	$18,578	$54,481	$43,247	$25,573	$19,443
Cash Flow after Taxes			$1,279,603	-$28,126	-$312,490	-$130,306	$106,685	$96,210	$91,600	$76,565	$64,943	$123,359	$118,233	$89,116	$76,628	$131,394	$163,907	$102,712	$207,148	$148,790	$84,812	$68,918
Cumulative Cash Flow after Taxes				-$28,126	-$340,616	-$470,922	-$364,237	-$268,027	-$176,427	-$99,862	-$34,919	$88,440	$206,674	$295,790	$372,418	$503,812	$667,719	$770,431	$977,579	$1,126,369	$1,211,181	$1,280,099

1.18 CONCLUSIONS

Based on evaluation of the data available and collected from the study conducted, the Qualified Persons (QPs) have drawn the following conclusions:

1.18.1 General

The level of investigation for all elements of this study, as confirmed by all Technical Report QPs, is consistent and typical of a feasibility level study.

As of the effective date of this Technical Report, Equinox holds a 100% interest in the Castle Mountain Project.

1.18.2 Geology and Mineral Resource Estimate

The deposits within the Castle Mountain Project are part of a low sulfidation epithermal system characterized by gold mineralization commonly occurring with silica alteration and iron oxide minerals. Mineralization is controlled by first order porosity and permeability of the lithological units that form the host volcanic complex and structural zones which can provide the conduit for hydrothermal fluids that carry mineralization.

The Project has a combined Measured and Indicated mineral resources exclusive of mineral reserves that are amenable to open pit mining that total 82 Mton (74 Mt) at 0.018 oz/ton (0.62 g/t) gold for 1,470 koz contained gold. These mineral resources occur dominantly within the oxide portion of the ore body and include portions of transition and sulfide ore. The Measured mineral resources exclusive of mineral reserves that are amenable to open pit mining total 861,000 tons (781,000 t) at 0.020 oz/ton (0.68 g/t) gold for 17 koz contained gold. Indicated mineral resources exclusive of mineral reserves that are amenable to open pit mining that total 81 Mton (73 Mt) at 0.018 oz/ton (0.62 g/t) gold for 1,453 koz contained gold.

Contributions to the changes to the current mineral resources are predominantly due to the differences between criterion used for Mineral Resource classification which relies dominantly on drillhole spacing.

1.18.3 Mining and Mineral Reserve Estimate

The current LOM plan was developed based upon Measured and Indicated Mineral Resources only and includes Proven and Probable Mineral Reserves of 284.3 Mton (257.9 Mt) with a grade of 0.015 oz/ton (0.514 g/t) at a cut-off grade of 0.005 oz/ton (0.17 g/t) Au. The total waste mined will be 701.9 Mton (636.8 Mt). The strip ratio average is 2.47:1 waste:ore tons.

Mining for the project expansion will be as an Owner-operated conventional diesel-powered truck and shovel operation. Current operations are focused on mining of previously backfilled material in the JSLA pit while expanded operations will focus on expansion of current pits and new pit development.

There is considerable potential to expand the Mineral Resources and Mineral Reserves as there are significant gold anomalies from the grab samples at East Ridge, East Flats and Egg Hill greater than 0.0292 oz/ton (1.00 g/t) gold. In addition, higher gold prices could make it economic to expand the Mineral Reserve pits shown in this feasibility study and ultimately connect the JSLA and South Domes pits.

1.18.4 Mineral Processing and Metallurgical Testing

Significant metallurgical testwork from 2015 to 2020 has been completed on both heap leach and mill material to establish the design criteria for the Phase 2 expansion. The testwork program completed provided the necessary data to define a process flowsheet and engineering parameters. This allowed for design and cost estimation of a conventional process plant for the feasibility study as well as defining the metal recoveries and operating consumables. Recovery methods proposed for the expansion consist of proven and well-known technologies in the industry.

After evaluating the column leach tests by feed size, ore zone and lithology, the arithmetic average gold recovery was 81%, 80% and 80% respectively. A weighted gold recovery based on ounces per lithology was calculated as 82%. The average gold recovery based on laboratory column testwork for Castle Mountain low grade ore was 80%.

To estimate the Castle Mountain gold recovery for the production heap leach from the lab data, operating and environmental conditions were considered. This includes ROM particle size distribution, permeability, effective leaching of the side slopes, etc. The ROM material for the Castle Mountain Project is predicted to have an F_{80} of 152 mm to 203 mm. When considering the ore size and other data, a lab to field deduction of 6% was applied to the average lab recovery of 80% for an expected LOM heap leach gold recovery of 74% after solution application is stopped. To account for the typical time impact in recovering gold from a large leach facility at closure, the expected gold recovery during LOM operations is considered to be 67% with a final recovery of 74% attained only after extracting residual gold values and reducing cyanide levels in the heap. This is expected to span a period of approximately three years after mining has ceased.

For this Feasibility Study, gravity followed by gravity tail leach in a CIL circuit was selected for the process plant based on economics. An overall gold recovery of 94% is expected from mill grade ores processed through the mill after 24-hour hybrid leach/CIL retention time. The plant has been sized conservatively with 30 hours retention time in the leach/CIL tanks.

1.18.5 Project Infrastructure

The existing infrastructure including current operations has been integrated into the proposed designs to support the expansion operations to the greatest extent possible. Further Phase 2 infrastructure developments include a security gatehouse, expanded site fencing, on site access roads, truck fleet service facilities, and dedicated process storage and warehousing facilities.

A site wide water balance was developed considering multiple scenarios based on historic and current climatic conditions at site and in the surrounding areas. Resulting make-up water demands were evaluated and used to inform water supply requirements. Recently completed groundwater studies on and off site indicate a high likelihood that sufficient capacity for fresh water can be attained through a combination of on-site and off-site sources.

Mill tailings were tested and analyzed, and it was concluded they are amenable to filtered stacking. To limit new land use on the property it was determined that adequate capacity is available on the historic Viceroy heap leach pad and adjacent to the expanded heap leach facility to serve the LOM requirement. This results in an optimized footprint while minimizing new disturbance on the property.

Several options for power supply were studied in detail with each being determined to provide viable alternatives for power supply to the expanded operations. The selected basis for the feasibility study is a connection to the grid via a new powerline serviced by NV Energy and SCE. Other alternatives including supplementing with renewable power, specifically solar, have been investigated and show potential opportunity.

1.18.6 Environmental Studies and Permitting

The Castle Mountain Mine is located on both public and private land, and historically has been environmentally permitted by co-lead agencies, the County of San Bernardino at the state level, and the United States Bureau of Land Management (BLM) at the federal level. The current operation was issued a revised Mining CUP by the County in August 2019 while the BLM issued a Decision Record and FONSI in February 2020 approving the revised Mine and Reclamation Plan. These key permits along with others provide the authorization for current mine operations at the project site.

Significant resource monitoring and environmental analyses have been conducted and continue to be completed on site to assure compliance and environmental stewardship of the project site.

The Mine has all permits required to conduct mining for Phase 1 and is permitted to operate within the Mine Permit boundary which has an area of 3,910 acres (1,583 ha). The Phase 2 Project will operate within the Mine Property boundary; however, modification to approved mine and reclamation plan elements, including increased mining and water extraction rates will require updates to existing permits.

Mine expansion as considered in this feasibility study is expected to require a new or updated environmental review (likely in the format of an EIS/EIR) as well as several new state and federal permits and amendments. Future amendments to the mine and reclamation plan to account for mine expansion are also expected and will include facility decommissioning, land recontouring, and revegetation.

1.18.7 Cost Estimates

Detailed capital and operating cost estimates have been developed including consideration for all direct and indirect costs associated with execution of the expansion project and required supporting infrastructure as well as sustaining costs, and reclamation and closure costs.

The cost estimate is based on preliminary engineering including 250 feasibility level design drawings covering all engineering disciplines, design criteria and detailed material take-offs. Mechanical and electrical equipment pricing were obtained for all major equipment (>$50,000) including 89% (or 140 items) through budgetary quotes. Bulk material pricing was estimated from recent projects budget quotes, local contractor budgetary review and actual pricing from Phase 1 construction. Mining equipment costs are based on quotes from major suppliers. Labor rates have been estimated using a weighted average of prevailing non-union shop wages from a published source (Davis-Bacon; 50%) and actual labor rates on site for Phase 1 construction (50%).

Project operating cost is based on a detailed build up of staffing requirements, reagent and fuel consumptions, mining activities, maintenance, and power demand. All major consumables have been specifically quoted with delivery to site.

The costs reflect a joint effort conducted by M3, NMS and specialist sub-consultants to adequately define project cost to a -10% to +15% accuracy level.

Analysis of the resulting economic parameters shows the expansion project to be economically viable.

1.19 RECOMMENDATIONS

The Castle Mountain Phase 2 expansion is a feasible project with good economics and should be progressed to the next stage which includes permitting along with optimization and front-end engineering ahead of detailed engineering.

2 INTRODUCTION

2.1 ISSUER AND PURPOSE

Equinox retained independent consultants to prepare a Feasibility Study Technical Report for the Castle Mountain Project in the state of California, USA. Equinox, through its indirect wholly owned subsidiary NewCastle, has 100% of the right, title and beneficial interest in and to Castle Mountain Venture which owns the Property.

The study includes updates to the mineral resource and mineral reserve estimates. Key aspects included in the study are further advancement on metallurgical testwork, pit design, updated mine schedule, proposed expanded heap leach operations, the addition of a high-grade processing mill including filtered tailings facility, infrastructure development, costs, and financial model. The findings and conclusions are based on information available at the time of preparation and data supplied by other consultants as indicated. The effective date of this Report is 26 February 2021, which represents the date of information used in the Report. The effective date of the mineral resource estimate for the Project is 30 June 2020, which represents the date of exploration information used for mineral resource estimation. The effective date of the mineral reserve estimate for the Project is 30 June 2020. There has been no material change to the information between the effective date and the signature date of the Report.

2.2 SOURCES OF INFORMATION AND QUALIFIED PERSONS

2.2.1 Qualified Persons

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in NI 43-101 and are members in good standing of appropriate professional institutions. QP certificates of authors are provided in Appendix A. The QPs are responsible for the specific sections as shown in Table 2-1.

Table 2-1: List of Qualified Persons, Inspections and Responsibilities

Qualified Person	Company	Certification	Date of Site Visit	Section Responsibilities
Gabriel Secrest	M3	P.E.	November 19, 2019	Sections 1.1, 1.2, 1.3, 1.4, 1.14, 1.15, 1.16, 1.17, 1.18.1, 1.18.5, 1.18.6, 1.18.7, 1.19, 2, 3, 4, 5, 13.8.5, 17.3.1, 18 (except for 18.5.3), 19, 20, 21 (except for 21.1.7, 21.1.9 and 21.2.2), 22, 23, 24, 25.1.1, 25.1.5, 25.1.6, 25.1.7, 25.2, 25.3.1, 25.3.3, 26.1, 26.5, 26.7, and 27.
Laurie M. Tahija	M3	P.E.	November 19, 2020	Sections 1.9, 1.13, 1.18.4, 13, 17 (except for 17.3.1), 25.1.4, and 26.6.
Eleanor Black	Equity Exploration	P. Geo	August 13-15, 2019	Sections 1.5, 1.6, 1.7, 1.8, 6, 7, 8, 9, 10, 11, 12 (except for 12.4), 25.3.2, and 26.2.
Trevor Rabb	Equity Exploration	P. Geo	August 13-15, 2019	Sections 1.10, 1.18.2, 12.4, 14, 25.1.2, and 26.3.
John Nilsson	Nilsson Mine Services Ltd.	P.Eng.	N/A	Sections 1.11, 1.12, 1.18.3, 15, 16, 21.1.7, 21.1.9, 21.2.2, 25.1.3, and 26.4.
R. Douglas Bartlett	Clear Creek Associates	P. Geo	April 28, 2020	Section 18.5.3.

2.2.2 Notable Personal Inspection Details

Gabriel Secrest (M3) completed a site visit of the Castle Mountain property on November 19th, 2019, accompanied by Equinox representatives Mr. Kevin Scott and Mr. Travis O'Farrell as well as several M3 discipline lead engineers. During his site visit, Mr. Secrest visited the construction progress of the Phase 1 facilities including the heap leach pad as well as existing locations and planned locations for site infrastructure. These locations included current and potential future raw water well site locations, the existing access road, potential routing for permanent plant power through Walking Box Ranch, JSLA backfill pit, the historical heap facility and planned locations for the high-grade mill, process plant and refinery.

Eleanor Black and Trevor Rabb (Equity) completed a site visit of the Castle Mountain property on August 14th, 2019 and the Henderson, Nevada project office on August 15, 2019. The site visit was led by Equinox representatives Scott Heffernan and Owen Nicholls, and included a review of the Project geology, logging and storage areas, and a general property overview. The QPs were able to verify the logging and sampling procedures and to inspect several diamond drill holes to ensure sample intervals and logged data were accurately reflected in the Project's database. Legacy hard copy data including original drill hole logs and assay certificates were checked and the geological model criteria were presented at the Henderson, Nevada office.

2.2.3 Other Sources of Information

Qualified Persons utilized information from references listed in Section 27 of this report.

2.3 UNITS OF MEASURE

This report uses U.S. Customary Units expressed in short tons (2,000 lbs, abbreviated in this report as "ton" to differentiate from metric tons "t"), feet, and gallons consistent with U.S. Standards. For added reference, the International System of Units (SI) expressions are included throughout the report within parenthesis when appropriate.

Monetary units are expressed in United States Dollars ($) unless otherwise specified.

2.4 TERMS OF REFERENCE

Table 2-2 shows the units of measure used in this study, and other terms and abbreviations are shown in Table 2-3. Table 2-4 shows the conversions for common units.

Table 2-2: Units of Measure

Unit	Abbreviation	Unit	Abbreviation
Above mean sea level	amsl	Day	d
Billion Years	Ga	Days per week	d/w, dpw
Canadian Dollars	C$	Days per year (annum)	d/y(a), dpy(a)
Centimeter	cm	Degree	°
Centimeters per second	cm/sec	Degrees Celsius	°C
Cubic Feet	ft^3	Degrees Fahrenheit	°F
Cubic Feet per second	cfs	Dollars per short ton	$/ton
Cubic feet per minute	cfm	Dollars per ounce	$/oz
Cubic Meters	m^3	Feet	ft
Cubic Meters per second	cms	Feet above sea level	fasl
Cubic Yards	yd^3	Gallons	gal

Unit	Abbreviation
Gallons per minute	gpm
Gallons per minute per square foot	gpm/ft^2
Gram	g
Grams per metric ton	g/t
Greater than	>
Hectare	ha
Hertz (frequency)	Hz
Horsepower	hp
Hour	h
Hours per day	h/d, hpd
Hours per week	h/w, hpw
Hours per year	h/y(a), hpy(a)
Inch	in, “
Kilo (thousand)	k
Kilogram	kg
Kilometer	km
Kilovolt	kV
Kilowatt	kW
Kilowatt hour per short ton	kWh/ton
Kilowatt per short ton	kW/ton
Kilowatt-hour	kWh
Less than	<
Linear foot	LF
Liter	l or L
Liters per hour per square meter	L/h/m^2
Liters per minute	lpm
Liters per second	L/s
Megawatt	MW
Meter	m
Meters above sea level	masl
Micrometer (micron)	µm
Milligram	mg
Milligrams per liter	mg/L
Milliliter	mL
Millimeter	mm
Million	M
Million dollars	$M
Million gallons	M gal
Million ounces	Moz
Million liters per year	Ml/y
Million metric tonnes	Mt, Mtonnes

Unit	Abbreviation
Million metric tonnes per year	Mt/y
Million short tons	Mton
Million short tons per year	Mton/y
Million Years	Ma
Minute (time)	min
Month	mo
Ounce	oz
Ounces per ton	oz/t, (metric) oz/ton (short)
Parts per billion	ppb
Parts per million	ppm
Pascal (Newtons per square meter)	Pa (N/m^2)
Percent	%
Phase (Electrical)	ph
Pound	lb, lbs
Pounds per Square Inch	psi
Pounds per Square Inch gauge	psig
Pounds per cubic foot	pcf
Pounds per ton	lbs/ton
Qualified Person	QP
Specific gravity	SG
Square Feet	SF, ft^2
Square kilometer	km^2
Square mile	mi^2
Standard cubic feet per hour	ft^3/h, SCFH
Thousand troy ounces	koz
Thousands of metric tons	kt
Thousands of short tons	kton
Ton (short ton)	ton
Tonne (metric ton)	t
Tons per day	t/d (metric), mtpd (metric), ton/d (short), tpd (short)
Tons per month	tpm, (short) ton/mon (short)
Volt	V
Volume Percent	vol %
Weight Percent	wt %
Year (annum)	y (a)

Table 2-3: Non-Unit Abbreviations Used in this Report

Unit	Abbreviation
Acid rock drainage	ARD
Adsorption-Desorption-Recovery	ADR
ALS Global	ALS
American Assay Laboratories Inc.	AAL
American Society for Testing and Materials	ASTM

Unit	Abbreviation
Ammonium Nitrate / Fuel Oil	AN/FO
Antimony	Sb
Arsenic	As
Atomic Absorption	AA
Atomic Absorption Spectroscopy	AAS
Authorities to Construct	ATCs
Bed Volume	BV
Bismuth	Bi
Bond abrasion index	Ai
Bond crusher impact	CWi
Bottle Roll Test	BRT
Bureau of Land Management	BLM
Bureau Veritas	BV
California Environmental Quality Act	CEQA
California Regional Water Quality Control Board, Colorado River Basin Region	RWQCB
California Register of Historic Resources	CRHR
Call and Nicholas, Inc.	CNI
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Carbon adsorption rate test	CAR
Carbon adsorption capacity test	CAC
Carbon in Column	CIC
Carbon in Leach	CIL
Castle Mountain Mine	CMM, or the Property
Castle Mountain Project	The Project
Castle Mountain Venture	CMV
Castle Mountains Volcanic Sequence	CMVS
Caustic (or caustic soda / Sodium Hydroxide)	NaOH
Certified Reference Materials	CRM
Closed side setting	CSS
Coefficient of determination	R^2
Coefficient of variance	CV
Comma Separated Values	CSV
Conditional Use Permit	CUP
Conventional Rotary Drill	CR, or rotary
Copper	Cu
Copper sulfate	$CuSO_4$
County of San Bernardino	County
Cyanide soluble gold	AuCN
Diamond Drill	DD
Digital elevation model	DEM
Drop Weight test	DWT
Eighty percent passing	P_{80}
Electrowinning	EW
Engineering, Procurement and Construction Management	EPCM
Environmental Impact Study and Environmental Impact Review	EIS/EIR
Equinox Gold Corp.	Equinox, or the Company
Equity Exploration Consultants Ltd.	Equity

Unit	Abbreviation
East Ridge	ER
Feasibility Study Technical Report	The Report, or the Study
Federal National Environmental Policy Act	NEPA
Fiber-reinforced plastic	FRP
Front-end loader	FEL
Geo-Logic Associates	GLA
Global Positioning System	GPS
Gold	Au
Gresham Savage Nolan & Tilden, PC	Gresham|Savage
Gresham, Savage, Nolan & Tilden LLP	Gresham Savage
High-density polyethylene	HDPE
High-sulfidation epithermal	HSE
Human machine interface	HMI
Hydrochloric acid	HCl
Induction	I
Inductively coupled optical emission spectrometry	ICP-OES
Intensive Leach Reactor	ILR
Internal Rate of Return	IRR
International Directional Services LLC	IDS
International Standards Organization	ISO
International System of Units	SI
Inter-Ramp Slope Angles	ISA
Inverse Distance Cubed	ID3
Inverse Distance Squared	ID2
Jumbo	JB
Jumbo South-Leslie Ann (Pit)	JSLA
Kappes, Cassiday and Associates	KCA
Laser	L
Lead	Pb
Lerchs-Grossman	LG
Life of Mine	LOM
Light Detection and Ranging	LiDAR
Lime	CaO
Linear low-density polyethylene	LLDPE
Liquid Natural Gas	LNG
Low-sulfidation epithermal	LSE
M3 Engineering & Technology Corporation	M3
McClelland Laboratories Inc.	MLI
Mercury	Hg
Milk of Lime	MOL
Mine Technical Services Ltd.	MTS
Mineral and Exploration Geochemistry	MEG
Mojave Desert Air Basin	MDAB
Mojave Desert Air Quality Management District	MDAQMD
Molybdenum	Mo
Motor control centers	MCC
National Historic Preservation Act	NHPA
National Pollutant Discharge Elimination System	NPDES
Nearest Neighbor	NN
Net Present Value	NPV

Unit	Abbreviation
Net Smelter Return	NSR
Network Intrusion Detection System	NIDS
New Source Review	NRS
NewCastle Gold Ltd	NewCastle
Nilsson Mine Services Ltd.	NMS
North American Datum	NAD
Ordinary Kriging	OK
Oro Belle	OB
Oxygen Uptake Rate	OUR
Pebble Quicklime	CaO
Permits to Operate	PTOs
Pre-Feasibility Study	PFS
Proterozoic	Pc
Proterozoic Basement	Pc Basement
Proterozoic sedimentary rocks	Pc Seds
Qualified Person	QP
Quality Assurance/Quality Control	QA/QC
Record of Decision	ROD
Reduction to Major Axis	RMA
Relative standard deviation	RSD
Reverse Air-Blast (Drilling)	RAB
Reverse Circulation	RC
Rock Quality Designation	RQD
Rocky Mountain Geochemical	RMG
Run of Mine	ROM
Scanning Electron Microscope equipped with an Energy Dispersive Spectrometer	SEM-EDS
Security Event Management	SEM
Semi-autogenous grinding	SAG
Silver	Ag
Sodium Cyanide	NaCN
Sodium metabisulfite	SMBS
South Dome	SD
Spill Prevention Control and Countermeasure Plan	SPCC
Steinert Combined Sensor	KSS FLI XT
Sulfur impregnated carbon	SIC
Surface Mining and Reclamation Act	SMARA
Surface recording gyro	SRG
Tellurium	Te
Thallium	Tl
The MINES Group Inc.	MINES
Three dimensional	3D
Transient Electromagnetic (survey)	TEM
Tungsten	W
Two dimensional	2D
Unified Threat Management	UTM
United States Bureau of Land Management	BLM
Universal Transverse Mercator	UTM
US Fish and Wildlife Service	USFWS
Variable frequency drives	VFD
Viceroy Gold Corporation	Viceroy

Unit	Abbreviation
Volcanoclastic	Vx
Volume weighted average price	VWAP
Weak Acid Dissociable	WAD
Weak acid dissociable cyanide	CN_{WAD}
Work Breakdown Structure	WBS
X-ray transmission sensors	XRT
Zinc	Zn

Table 2-4: Conversions for Common Units

Metric Unit	Imperial Measure
1 hectare	2.47 acres
1 metre	3.28 feet
1 kilometre	0.62 miles
1 gram	0.032 ounces (troy)
1 tonne	1.102 tons (short)
1 gram/tonne	0.029 ounces (troy)/ton (short)
1 tonne	2,204.62 pounds
Imperial Measure	**Metric Unit**
1 acre	0.4047 hectares
1 foot	0.3048 metres
1 mile	1.609 kilometres
1 ounce (troy)	31.1 grams
1 ton (short)	0.907 tonnes
1 ounce (troy)/ton (short)	34.28 grams/tonne
1 pound	0.00045 tonnes

3 RELIANCE ON OTHER EXPERTS

In preparing the report, M3 has relied upon contributions from a range of technical and engineering consultants as well as Equinox. M3 has reviewed the work of the other contributors and finds that this work has been performed to normal and acceptable industry and professional standards and can be relied upon for purposes of this report.

M3 is not an expert in legal and land tenure matters and has relied upon verification of land title and tenure as documented in an updated title report prepared by Gresham Savage Nolan & Tilden Corporation dated March 10th, 2020. This report is relied upon to determine land and claim ownership as it relates to the Castle Mountain Project.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Project is located in the historic Hart Mining District, at the southern end of the Castle Mountains, San Bernardino County, California, approximately 70 mi (113 km) south of Las Vegas, Nevada by road (Figure 4-1). The Project is located in a high desert area near the Mojave National Preserve and Castle Mountains National Monument. The Project is located at latitude 35° 16' North and longitude 115° 06' West.

The Project survey control is based on a local coordinate system which is utilizes NAD 83 UTM zone 11 in imperial units (feet). The conversion from meters to feet is converted by dividing the UTM meter units by 0.3048, with four significant figures.

Figure 4-1: Location Map – Castle Mountain Project

4.2 PROPERTY AND TITLE IN SAN BERNARDINO, CALIFORNIA

4.2.1 Mineral Rights

The Project includes 1,301 acres of patented lode claims, 8,980 acres of unpatented lode claims, 3,598 acres of unpatented mill site claims, and 3,639 acres of placer claims (Table 4-1, Figure 4-2). The claims are registered under the Castle Mountain Venture and Viceroy Gold Corporation. Many of the claims overlap and as such the total area of the individual claims is not representative of the overall total area covered. The effective area for operation is covered in Section 4.5.

Table 4-1: Summary of Land Tenure by Type at Castle Mountain Mine, San Bernardino County, California

Type	Claims	Area (acres)	Area (hectares)
Patented Lode	8	1,301	526
Unpatented Lode	449	8,980	3,634
Unpatented Mill Site	723	3,598	1,456
Unpatented Placer	54	3,639	1,473
Total	**1,234**		

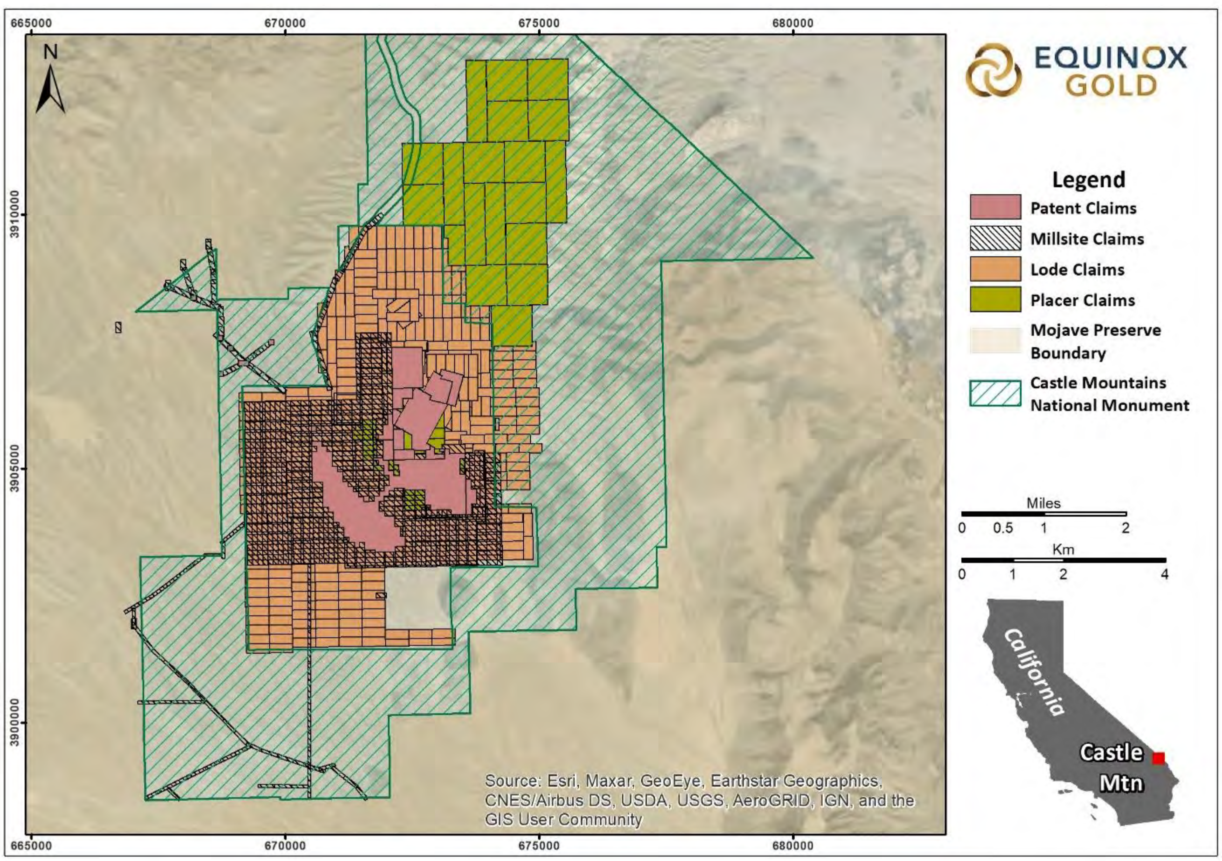


Figure 4-2: Tenure Map for the Castle Mountain Mine

Patented and unpatented claims are located in Townships, Ranges and Sections shown in Table 4-2.

Table 4-2: Township & Range in Castle Mountain

Township & Range, all San Bernardino Base & Meridian	Sections
T12N R18E	23
R13N R17E	13
T14N R17E	1, 9, 11-14, 17, 18, 22-27, 30, 32, 34-36

4.2.2 Acquisition of Castle Mountain Venture

Equinox (previously Trek Mining Inc.) acquired NewCastle Gold Ltd. (NewCastle) on December 22, 2017 and NewCastle became a wholly-owned subsidiary of Equinox. Subject to certain obligations, NewCastle has 100% of the right, title, and beneficial interest in and to Castle Mountain Venture (CMV) which owns the Castle Mountain Mine. NewCastle acquired its interest through its acquisition of Telegraph Gold Inc. (Telegraph), an Ontario corporation, on April 23, 2013. This followed Telegraph's acquisition of CMV on September 6, 2012.

NewCastle, formerly known as Castle Mountain Mining Company Limited, was incorporated under the Business Corporations Act (Ontario) on December 16, 2009 and commenced activities as a capital pool company on January 29, 2010 under the name of Foxpoint Capital Corp.

On April 25, 2013, NewCastle completed its acquisition of Telegraph by the way of an amalgamation of Telegraph with a subsidiary of NewCastle. At the time of the transaction, NewCastle changed its name from Foxpoint Capital Corp to Castle Mountain Mining Company Limited with a registered head office located in Toronto, Ontario, Canada.

At the time of Telegraph's purchase of CMV, it was 75% owned by Viceroy and 25% owned by MK Resources LLC (MKR). Viceroy was a wholly owned subsidiary of Sprott Resource Lending Corporation (Sprott) and MKR was a subsidiary of Leucadia National Corporation (Leucadia). Telegraph acquired both interests through concurrent transactions that each closed on September 6, 2012. MKR's 25% interest was acquired for $2,000,000, which was paid in cash. Telegraph acquired the shares of Viceroy and therefore the remaining 75% interest in CMV from Sprott.

4.3 CALIFORNIAN LAND RIGHTS

4.3.1 Annual Claim Maintenance Payments to BLM

NewCastle is required to pay an annual federal claim maintenance fee to the United States Bureau of Land Management (BLM) for each unpatented lode, mill site and placer mining claim in the amount of $165. These payments are due on September 1st of each year. Payments to the BLM are in good standing for 2020-2021.

4.3.2 Annual Property Tax Payments to San Bernardino Country, CA

Property taxes are payable to San Bernardino County (the County) for the 24 tax parcels (patented & unpatented lode claims, mill site claims and placer claims) that comprise the Project.

Payments are due on a semi-annual basis and have been remitted to the County. Total payments of $96,978.03 have been made by NewCastle for the 2020-2021 tax year.

4.3.3 Title Report

On March 10, 2020, Gresham, Savage, Nolan & Tilden LLP (Gresham Savage) of San Bernardino, California, prepared an updated title report (Gresham Savage, 2020), which related only to changes that had occurred since the date of the previous title report prepared by Gresham Savage Nolan & Tilden, PC (Gresham|Savage) on July 13, 2017, which was an update to previous title report prepared by Gresham|Savage on November 1, 2016 (the “November 2016 Update”), which was an update to the Supplemental Title Report/Update prepared by Gresham|Savage on May 9, 2016 (the “May 2016 Update”), which was an update to the Supplemental Title Report/Update prepared by Gresham|Savage on August 7, 2012 (the “2012 Update”), which was an update of the Supplemental Title Report/Update prepared by Gresham|Savage on February 25, 2004 (the “2004 Update”), which in turn was an update to the Title Opinion dated September 18, 1991, prepared by Harris, Trimmer & Thompson, Reno, Nevada (the "Harris Opinion," and collectively with the each of the other referenced Updates, the “Prior Reports”).

4.4 ROYALTIES

A number of net smelter return (NSR) royalty agreements are in place on the Project as shown in Table 4-3 and illustrated in Figure 4-3. Royalty outline data were provided by Equinox.

Table 4-3: Distribution of Outstanding Net Smelter Royalties

Claim/Patent	%	Owner
Turtle Back	5	Conservation Fund
Milma	5	Conservation Fund
Golden Clay	5	Huntington Tile
All Claims	2.65	Franco-Nevada
Pacific Clay	2	American Standard

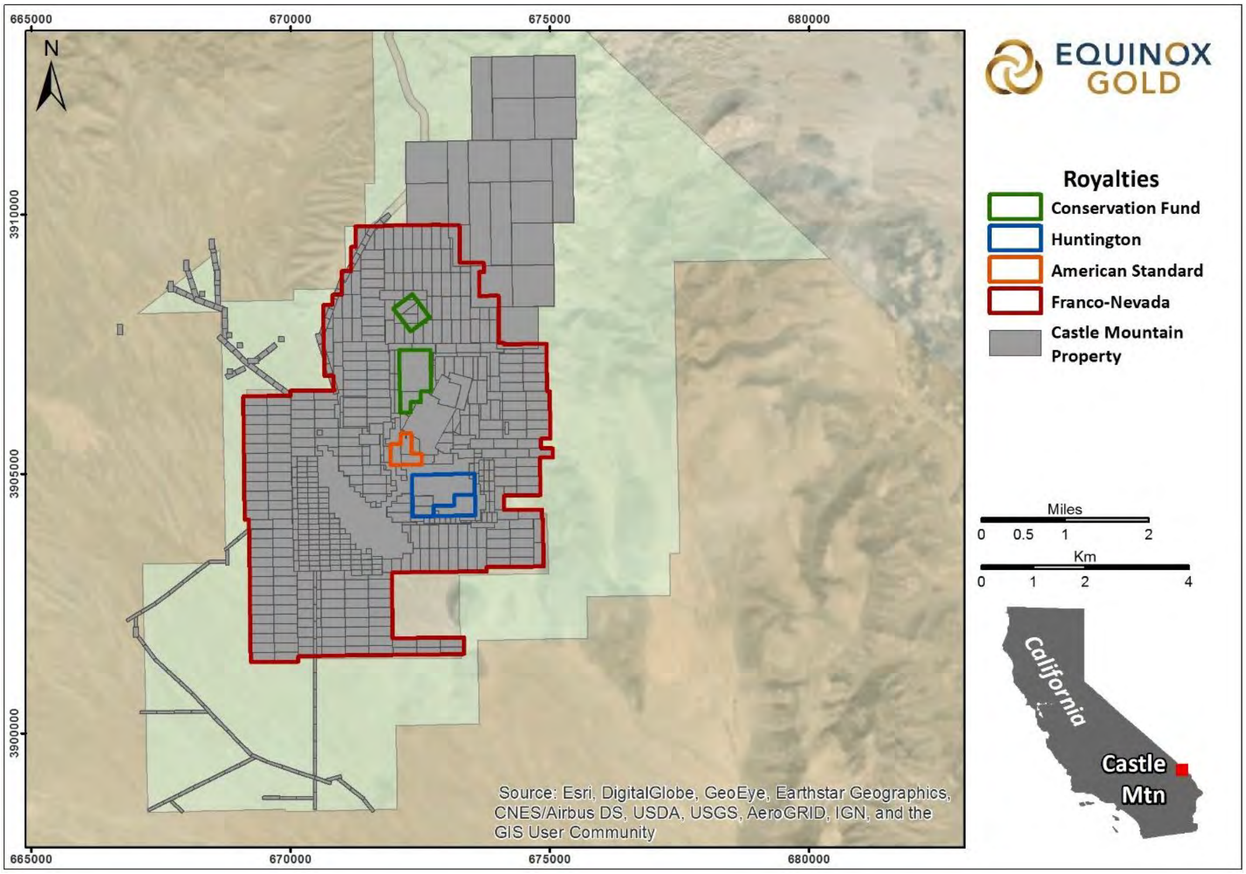


Figure 4-3: Map of Net Smelter Royalties

On April 11, 2016, NewCastle announced that it had arranged a royalty consolidation and private placement financing with Franco-Nevada Corporation (Franco-Nevada) for gross proceeds of \$3.4 million of which \$2,236,364 was ascribed to the royalty consolidation.

On June 16, 2016, NewCastle closed the royalty consolidation transaction, whereby NewCastle and Franco-Nevada agreed to create, in return for a cash payment of \$2,236,364, a new 2.65% net smelter royalty covering all minerals produced from the Project. The new royalty overrides the five separate pre-existing royalties held by Franco-Nevada and covers the existing Project and extends ten miles from the boundary of the Project. The new royalty does not require any advanced minimum royalty payments.

4.5 PERMITTING

The Mine is currently active and retains all permits required to conduct mining for Phase 1; this includes compliance with all applicable federal, state, and local regulations and ordinances.

The Mine is permitted to operate within the Mine Property boundary which has an area of 3,910 acres (1,583 ha) (Figure 4-4), of which 1,375 acres are currently approved for mining and related activity. The Phase 2 Project will similarly operate within the existing permitted Mine Property Permit boundary; however, modification to approved mine and reclamation plan elements, including increased mining and water extraction rates will require updates to existing permits. The additional water is to be obtained from near site and off-site production water wells.

The project lead agencies are U.S. Bureau of Land Management for federal (project) lands, and the County of San Bernardino for private (project) lands. Both agencies are required to assess, analyze, and mitigate (to the extent possible) environmental impacts related to Phase 2 mine expansion plans. These required environmental assessments of the effects of mine expansion are required through the California Environmental Quality Act (CEQA) and the Federal National Environmental Policy Act (NEPA).

The County and BLM consult with each other to ensure that the Project meets applicable State laws, regulations, and local ordinances, as well as Federal laws, regulations, and agency specific policy, prior to approving development. The current lead agency permits (County and BLM) are shown below:

- San Bernardino County, Land Use Services Department: Conditional Use Permit. SAMR/88-003/DN585-1145N; Reclamation Plan 90M-013. The operational term has been extended from 2025 to 2035.

- Bureau of Land Management: Record of Decision, Castle Mountain Mine Expansion Project San Bernardino County, California Environmental Impact Statement No. DES 97-10 State Clearinghouse No. 95081031, March 13, 1998.

- Bureau of Land Management: Decision Record, Modification of Castle Mountain Mine Plan of Operations, DOI-BLM-CA-D090-2020-0002-EA, February 27, 2020. The operational term has been extended from 2020 to 2035.

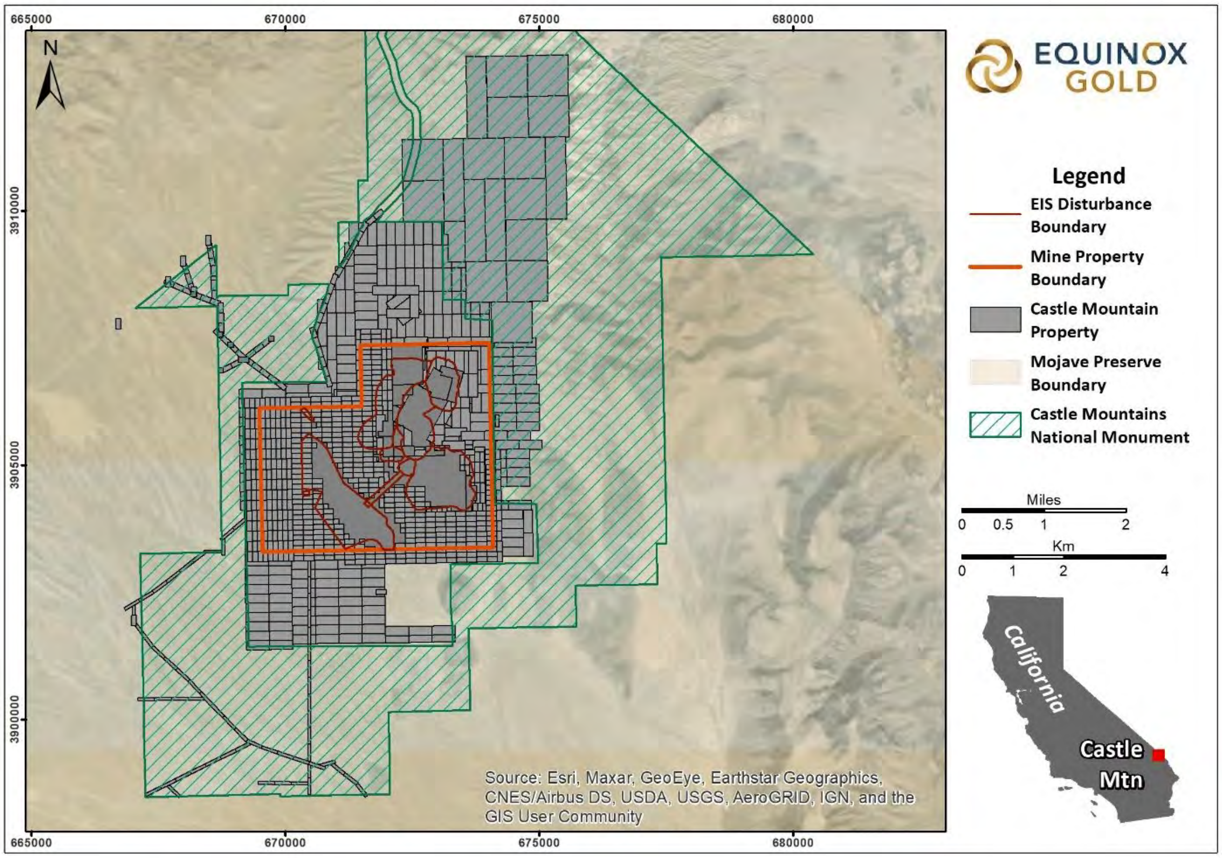


Figure 4-4: Map of Permitted Work Area, Castle Mountain Project

A comprehensive description of the existing permits and the permitting process is provided in Section 20.

4.6 ENVIRONMENTAL LIABILITIES

Aside from standard mine closure and reclamation, there are no known environmental liabilities on the Project.

4.7 LAND SUMMARY & SIGNIFICANT RISK FACTORS

All unpatented and patented claims are in good standing with respect to fees, taxes and levies for the 2020-2021 year. Equinox asserts that it has full legal access to the Project with respect to surface and mineral rights.

The QP is not aware of any significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Project. There is a risk that all the required permit modifications may not be approved for the Phase 2 operation. NewCastle has prepared applications for the permit modification and feels that the regulatory process is transparent and reasonable.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

Castle Mountain is being redeveloped in two stages, described as Phase 1 and Phase 2. The Phase 1 operation consists of a run of mine (ROM) heap leach facility to treat 14,000 short tons of ore per day (ton/d). The Phase 2 Castle Mountain Project will expand the current operation to allow a heap leach processing rate of 50,000 ton/d and a mill-CIL plant to process 3,500 ton/d.

The Project is accessible year-round by road. Las Vegas, Nevada, the closet major urban center, is located approximately 70 mi (113 km) north of the project area by road. Interstate US-95 is taken south from Las Vegas for one hour to Nevada State Route 164, which intersects the unpaved Walking Box Ranch Road approximately 5 mi west of Searchlight, Nevada. An 18 mi drive along the Walking Box Ranch Road provides access to the project area (see Figure 5-1). This existing access road is well maintained and of good quality for necessary traffic access as required for construction and operation of the project.

Figure 5-1: Property Access Map, Castle Mountain Project

5.2 CLIMATE

Castle Mountain experiences a desert climate with hot summers and cool winters, with temperatures attenuated by altitude and aridity relative to the surrounding valleys.

Average daily lows and highs for the project site range from 28°F (-2°C) to 52 °F (11°C) in the winter and 66°F (19°C) to 93°F (34°C) in the summer months, respectively. Most precipitation is received during the summer months via thunderstorm activity. Precipitation as snow during the winter months is common but ground accumulation is minimal and rarely lasts longer than a couple of days. Average annual total precipitation is just over 9 in.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

Site infrastructure supports current Phase 1 operations and includes the aforementioned access road from Nevada State Route 164, modular administration and mine offices, the main haul corridor connecting mining of the backfilled JSLA pit with the Phase 1 run of mine lined heap leach pad and collection channels, the Phase 1 processing plant which includes solution handling pumps and storage tanks, a 23.5 M gal lined event pond, a gold carbon-in-column plant and cyanide unloading and storage area, a diesel driven power generation plant and laboratory. Infrastructure from the previous Viceroy operations has been largely removed; however, three small wells and a 250,000 gal water holding tank continue to be utilized to support Phase 1 operations. The majority of Phase 1 make up water is sourced from two newer wells on site which feed to a 300,000 gal water holding tank enabling gravity water supply to the surrounding facilities.

An aerial representation of current Phase 1 infrastructure and historic heap leach pad and mine areas is presented in Figure 5-2.

Figure 5-2: Site Infrastructure Map, Castle Mountain Mine (July 1st, 2020 Representation)

Electrical power supply for the property has historically come from the Searchlight electrical substation via a high voltage (69 kV) powerline which followed a path parallel to Nevada State Route 164 and turned south along the project access road. Reclamation of former operations required removal of power poles and the installed power line from the project substation to a termination point near Walking Box Ranch. Power poles between the Searchlight substation and this point of termination remain largely intact for potential reutilization. The surrounding area also boasts a significant number of solar power generating stations on both the California and Nevada sides of the local border presenting the potential optionality for the use of renewable energy. Project phase 1 operations uses on-site power generation via diesel fired power generators. This main power plant, located adjacent to the carbon column plant, includes four (4) Tier 4F, 455 kW Diesel Generator Sets. In addition, there are another six (6) smaller generators at areas around the site ranging from 36 kW to 240 kW. Proximity to the Las Vegas/Henderson metropolitan area provides good access to several fuel distribution sources for both diesel and natural gas. The Phase 1 process plant area can be seen in Figure 5-3.

Figure 5-3: Castle Mountain Phase 1 Process Plant Aerial View

The southwestern United States is a major mining district with significant experienced mining labor and resources. These local resources have been utilized for start-up of phase 1 operations and will be used to a further extent to provide the necessary manpower for the Phase 2 expansion.

5.4 PHYSIOGRAPHY

The Project is located in the eastern Mojave subregion which gradually blends with the Basin and Range region to the north and the Colorado Desert to the south. Elevations are higher in the eastern Mojave compared to the other Mojave subregions, but differences in elevations are not as pronounced as those seen in Basin and Range terrains (Michaelsen, 2013).

The Castle Mountains are a relatively small range extending north-northeast from the northern end of Lanfair Valley in California into Piute Valley in Nevada. The range is about ten miles (16.1 km) in length and 2-3 miles (3.2 to 4.8 km) in width trending across the northern end of the Piute Range near the California-Nevada state borderline. The Project is located near the southernmost extent of the Castle Mountain range at an elevation of about 4,500 fasl (1,370 masl), and elevations at the Project site range from about 4,100 fasl (1,250 masl) to 5,100 fasl (1,554.5 masl).

Vegetation in the project area is typical of the Mojave Desert, with several cactus species (cholla and barrel) interspersed with woodlands of Joshua trees, blackbrush scrub, creosote bush scrub and desert grasslands (Figure 5-4).

Figure 5-4: Site Typical Flora; Joshua Tree and Scrub Brush in the Foreground

6 HISTORY

This section details exploration work conducted prior to Equinox. Recent exploration work carried out by Equinox is detailed in Section 9. Section 10 contains historical exploration drilling details.

6.1 HISTORICAL MINING

The Hart Mining District covers the southern end of the Castle Mountains. Several hundred old prospects, pits, trenches, waste rock dumps and underground workings extend over an approximate two square miles (5.2 km^2) area. In 1907, three underground gold mines were brought into production at Oro Belle, Big Chief and Jumbo. Mining activity decreased from 1910 to 1911 as the mineralized veins of interest were exhausted. The Big Chief Mine was reopened as the Valley View Mine and operated from 1932 to 1944 utilizing an old shaft. No production records are available for these historical operations.

In the 1920s, development began for the quarrying of kaolinite clay alteration zones associated with, but peripheral to the main gold deposit. Clay production was reported to have exceeded 200,000 tons (181,437 t) by 1957.

6.2 HISTORICAL GOLD EXPLORATION

Exploration in the Hart Mining District began in 1968 and carried through to the early 2000’s. The following section is summarized from Temkin (2012) and the synopsis of the work carried out is presented in Table 6-1. Exploration target areas mentioned in this section are shown on Figure 6-1.

6.2.1 Vanderbilt Gold Corporation, 1968 – 1980

Systematic exploration in the Hart Mining District began in 1968 by Vanderbilt Gold. Their program was focused on sampling of historic mine dumps and underground workings. Encouraging results led to the acquisition of the Oro Belle patented claims in 1979 and staking of the Southern Belle 1-9 lode claims in 1980. Exploration drilling that was completed in 1980 included a 28-hole conventional rotary drilling program (DH 1-28) that indicated the presence of broadly disseminated low-grade gold mineralization in the vicinity of the Oro Belle shaft. The average depth of this drilling was about 150 ft (46 m).

6.2.2 Vanderbilt Gold Corporation – B&B Mining, 1981 – 1984

In 1981 B&B Mining acquired the Mountain Top lode claims, adjacent to the Vanderbilt holdings who drilled four conventional rotary drill holes (MT 1-4) on the southeast side of Egg Hill nearby the historic Green and Gold Mine. No significant gold mineralization was identified in these holes. In 1983, B&B Mining and Vanderbilt Gold signed a joint venture agreement to conduct exploration within and around the Hart Mining District.

Detailed geologic mapping was conducted during that year. Between 1983 and 1984, Vanderbilt operated the project and completed 64 conventional rotary drill holes (OBR 1-61) within the Oro Belle, Jumbo and Hart Tunnel areas, and an additional four conventional circulation holes (VVR 1-4) in the historic Valley View Mine area. The average depth of these 65 holes was approximately 250 ft (76 m). In late 1984 B&B Mining became the majority partner and operator on the Project as Vanderbilt Gold's interest eventually was reduced to less than 10% as Vanderbilt ceased involvement in the project.

6.2.3 Freeport Mineral Ventures, 1980 – 1985

In late 1980 and early 1981, Freeport Mineral Ventures (Freeport) staked 352 "MYO" lode claims in the southern Castle Mountains. During that time, they conducted regional-scale geologic mapping as well as grid-style rock chip sampling and geochemistry. Between 1982 and 1984, a total of 26 vertical conventional rotary drill holes were completed to an average depth of 450 ft (137 m). This work was conducted in the Northwest Rim area, a couple miles (3 km) north-northwest of the future JSLA open pit. Freeport failed to locate significant gold mineralization during this effort and after failing to reach agreements with other claim holders in better target areas, proceeded to drop their claims and terminate activities.

6.2.4 B & B Mining / Viceroy Gold, 1984 – 1985

In early 1984, B&B Mining amalgamated with Viceroy Petroleum Ltd. to form Viceroy Resource Corporation. B&B Mining was later renamed Viceroy Gold Corporation (Viceroy) and became the U.S. subsidiary to Viceroy Resource.

6.2.5 Hemlo / Noranda Exploration, 1987 – 1990

Noranda began mineral exploration in the Castle Mountains in 1987, including regional-scale geologic mapping and rock chip sampling. Their activities included geophysical survey work in the form of several lines of IP and biogeochemistry and microbial studies over Jumbo South, Lesley Ann and Northwest Rim. This work led to the drilling of RC holes HV 1 - 20, 23 - 28 and 30, in the Northwest Rim area in 1987 and 1988. Subsequently, RC holes HV 31 - 43, in 1988, and RC holes LH 4 – 8, were drilled west of Razorback Butte in 1990. No significant gold mineralization was identified during this work and consequently, Noranda/Hemlo dropped their interest in the Project in 1990.

6.2.6 Viceroy Gold, 1985 – 1991

Exploration conducted by Viceroy Gold included detailed geologic mapping, stream sediment sampling, grid-controlled rock sampling programs, geophysical surveys and biogeochemical sampling.

In the spring of 1984 Viceroy Gold gained a majority interest and became operator on the Project. In mid-1985, Viceroy Gold commenced drilling. The "OBR" collar designation, for Oro Belle rotary holes, was dropped from the hole numbers at this point and current records show future holes were designated simply by numbers only. Rotary holes 62 to 79 were completed by the end of 1985, with 16 holes at Jumbo and two holes at Oro Belle. The average depth of these holes was approximately 300 ft (91 m).

In 1986, Viceroy increased the scope of activities to accelerate its ownership of the Project. The program included extensive drilling to advance the understanding of the controls on mineralization. Drilling focused on testing known targets to greater depths with holes averaging 500 ft (152 m). Rotary holes 80 to 195 were drilled in eight separate areas and resulted in the identification of significant gold mineralization in the Jumbo South and Lesley Ann areas. The discovery of the Jumbo South deposit, (hole #91), encountered 175 ft (56 m) of 0.036 oz/ton Au (1.23 g/t) and 135 ft (41 m) of 0.153 oz/ton Au (5.24 g/t) with average grades greater than 0.015 oz/ton Au (0.514 g/t) continuing to a depth of 725 ft (221 m). The discovery of the Lesley Ann deposit (with hole #150) encountered 180 ft (55 m) of 0.076 oz/ton Au (2.60 g/t), which was completely blind and covered by as much as 250 ft (78 m) of post-mineral alluvium.

By the fall of 1986 Viceroy Gold had acquired 100% in the Project and initiated a feasibility study to determine the economics of producing from the Oro Belle, Jumbo, Jumbo South and Lesley Ann deposits. With confirmation of a positive feasibility study for production from these three deposits, Viceroy sought major financing with Hemlo Gold in 1987, providing for the development of the Project. Under this agreement Viceroy retained 100% interest in the central area while Hemlo acquired a 50% exploration share in the remainder of the Castle Mountains. The permitting process for the mine was initiated in 1987, and approval of the mine was granted by the United States Bureau of Land Management in 1990.

6.2.7 Viceroy Gold/MK Gold, 1991 – 2001

Early in 1991, MK Gold purchased a 25% interest in the Castle Mountain Project and became the contract mining operator. Later in 1991 construction began for mining the Jumbo South and Lesley Ann deposits as one open pit.

Minimal exploration activity beside reclamation of all previous exploration and development disturbance occurred between 2005 and 2011.

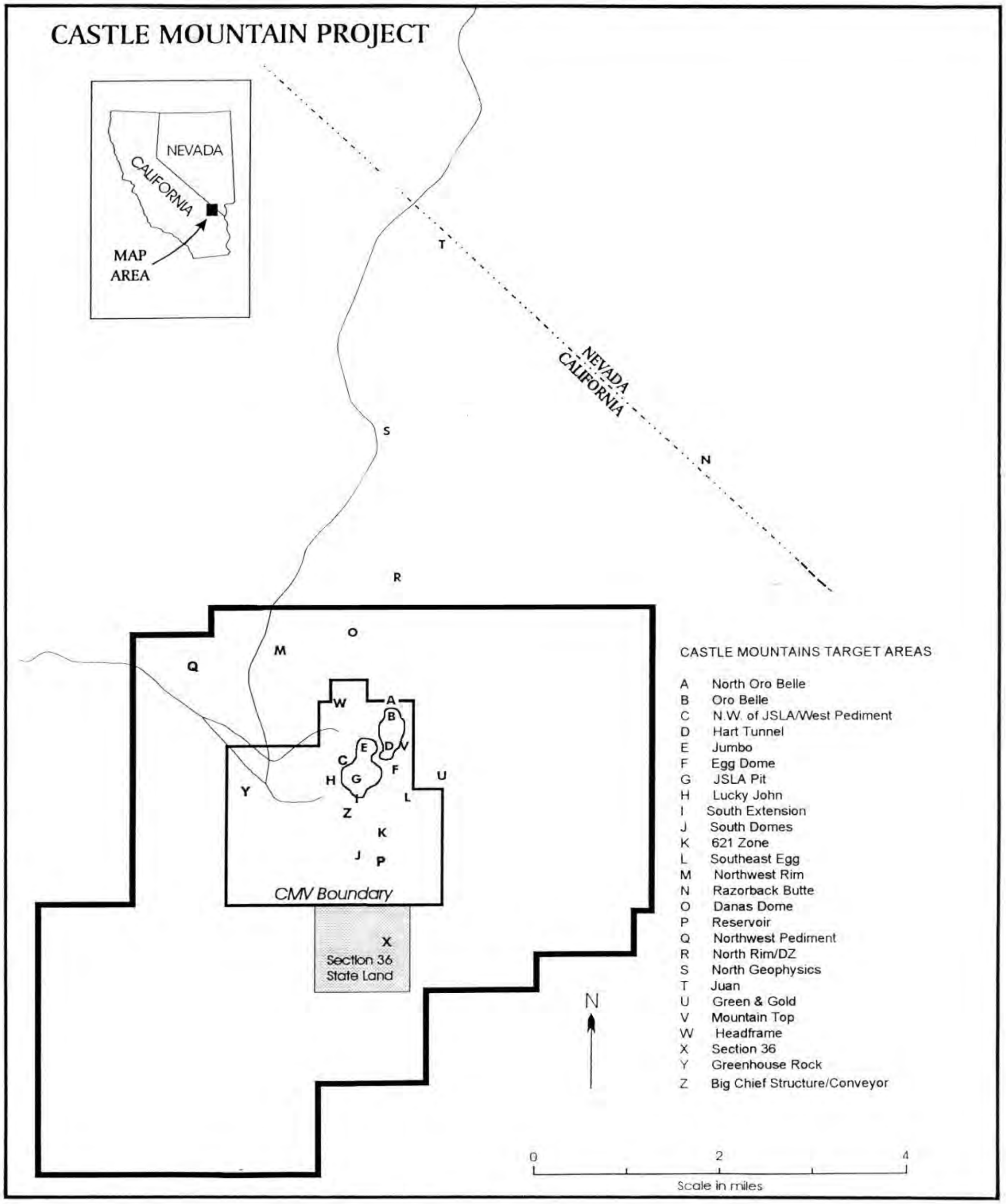
NewCastle (then Castle Mountain Mining Company Limited) acquired the Project in 2012. In December 2017, a three-way merger was completed between Trek Mining, NewCastle Gold, and Anfield Gold Corp., with the resulting company renamed to Equinox Gold Corp.

Table 6-1: Summary of Legacy Exploration Activity

Year	Company	Prospects	Summary of Exploration Activity
1907-1967	Unknown	Oro Belle, Big Chief, Jumbo	Underground mining operations
1968	VanderBilt Gold Corporation	Unknown	Sampled historical mine dumps and underground workings
1979		Oro Belle	Acquired Oro Belle patents
1980		Southern Belle	Staked 9 lode claims at southern belle, completed 28 conventional rotary (CR) holes and 1,980 ton bulk sample for vat-leach testing
1980-1981	Freeport Ventures	Regional	Staked 352 "MYO" lode claims, conducted regional-scale geologic mapping and grip rock chip and geochemical sampling
1981	B&B Mining and Vanderbilt Gold Corporation	Regional	Acquired lode claims adjoining the Vanderbilt land holdings, completed 4 CR holes
1982-1984	Freeport Ventures	Unknown	Drilled 26 CR holes, claims allowed to lapse; exploration ceased
1983	B&B Mining and Vanderbilt Gold Corporation	Unknown	Drilled 159 CR drill holes, completed surface geologic mapping, rock chip sampling, soil mercury survey, rehabilitation of underground working, underground mapping and rock chip sampling, surface magnetometer and very low frequency EM geophysical surveys
1984		Regional	B&B amalgamated with Viceroy Petroleum Ltd. To form Viceroy Gold Corporation. In late 1984, Viceroy became the majority partner and operator as Vanderbilt's interest reduced below 10% and ceased involvement in the JVA

Year	Company	Prospects	Summary of Exploration Activity
1984	Viceroy Gold	Regional	18 CR drill holes, geologic mapping, stream sediment and soil sampling, rock chip, channel and panel sampling, IP and magnetics surveys, biogeochemical surveys
1986		Oro Belle, Jumbo South, Leslie Ann	116 CR drill holes, geological mapping, Viceroy
1987-1989		Unknown	FS study for open pit heap leach mine, condemnation drilling with 420 CR and RC holes, discovery of South Extension, Jumbo, Hart Tunnel, South Dome. Additional microscopy, petrology, geochemistry
1987-1990	Hemlo Gold/Noranda Exploration	Regional (Outside of Viceroy-owned claims)	RC drilling, mapping and rock sampling, IP, biogeochemical and microbial surveys. Noranda dropped interests in 1990
1990	Viceroy Gold	Multiple targets (8)	16 RC holes, discovery of gold in North Oro Belle area
1991	Viceroy Gold and MK Gold Company	Multiple targets	MK Gold purchased a 25% interest in the mine and became contract mining operator, commercial production begins. Drilling resumes on Oro Belle, Jumbo, discovery of Lucky John and deep 621 Zone
1992-1993		Multiple targets (15)	Exploration, development and condemnation drilling with 263 RC holes at Jumbo, North Oro Belle, South Domes, discovery of Southeast Egg zone
1993-1994		Multiple targets (19)	252 RC holes mostly concentrated on Hart Tunnel, Oro Belle, Egg Dome, Lucky John, South Extension
1994-1995		Hart Tunnel, Oro Belle, Mountain Top	RC drilling, gold intersections at Hart Tunnel, Oro Belle, Mountain Top
1996		Multiple Targets	Intermountain Mine Services (IMS) contracted to design underground exploration program to further define mineralization from surface drilling, formulate plan to develop and mine mineralized material

Source: Temkin, 2012.



Source: Temkin, 2012. Note: Land boundaries shown on this figure are historic and do not reflect the Project's current boundaries.

Figure 6-1: Map of Legacy Target Areas (dated 2002)

6.3 PAST PRODUCTION

Viceroy Gold Corporation/MK Gold Corporation commenced gold production on the Project in 1991 (Figure 6-2). The JSLA open pit from the 1990 feasibility was exhausted in 1996. The Jumbo pit ceased production in 2001 due to local wall stability issues which left the deepest bench mined approximately 200 ft above the planned bottom mining elevation. Mining on the Oro Belle and Hart Tunnel deposits ceased later in 2001. Heap leaching continued until 2004.

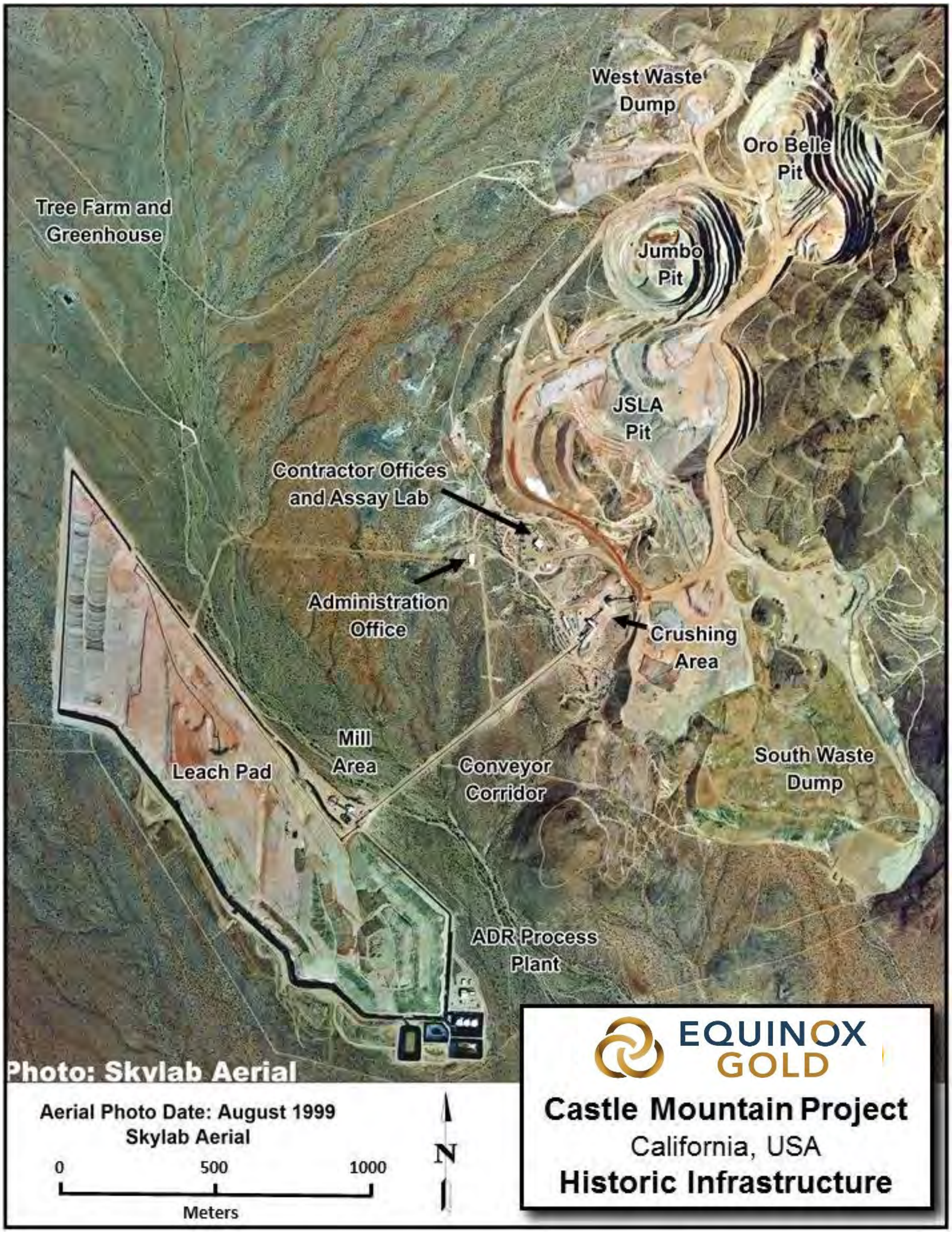
The mineral processing included two circuits:

- A conventional heap leach circuit where ore was crushed in three stages with the minus ⅜ in. (9.5 mm) product of the tertiary crushing delivered to the leach pad via conveyor system.
- A modified milling circuit to treat high-grade ore greater than 0.100 oz/ton (>3.43 g/t) where feed was ground to 100 mesh (149 µm) and treated with cyanide solution while still in the ball mill. Later in the mine life, a supplemental gravity circuit was added. Mill tailings were then agglomerated and conveyed to the heap leach pad where they were treated in the same manner as the heap leach feed.

Since the residence time in the ball mill was significantly less than the 24 hours required to achieve full cyanide dissolution, the initial gold recoveries were in the range of 33% to 40%. When the gravity circuit was added, gold recoveries exceeded 50%. The agglomerated tailings were estimated to achieve 91.3% gold recovery with an overall recovery for mill feed material of about 95% of the gold.

The leach cycle extended 43 months after pad loading ceased and resulted in the production of approximately 116,120 oz of gold which represents 12% of the total leach production.

Total gold production from all deposits was more than 1.24 Moz with an approximate silver production of 400,000 oz (Table 6-2). The annual production for the combined mill and leach ore the is presented in Table 6-3.



Source: Scott et al., 2018

Figure 6-2: Historic Viceroy Mining and Processing Infrastructure

Table 6-2: Viceroy Past Production From 1991 to 2004

Activity	Tonnage (000 ton)	Grade (oz/ton Au)	Contained Ounces Au (000s)	Recovered Ounces Au (000s)	Recovery (%)
Ore Mined	37,683	0.040	1,520		
Waste Mined	102,260	n/a			
Total Mined	**139,943**				
Ore Milled	1,967	0.144	283	269[1]	95.0[2]
Ore Leached	34,226	0.037	1,267	974[3]	76.9[4]
Total Processed	**36,193**	**0.043**	**1,550**	**1,243**	**80.2**

Notes:
1. A total of 269,000 oz Au comprises 120,000 oz Au recovered from mill circuit and 149,000 oz Au recovered from agglomerated tailings placed on the heap leach pad.
2. Recovery calculated as 269,000 oz Au recovered from 283,000 oz Au.
3. A total of 974,000 oz Au comprises 1,123,000 oz Au minus 149,000 oz Au recovered from agglomerated tailings sent to the heap leach circuit.
4. Recovery calculated as 974,000 oz Au recovered from 1,267,000 oz Au.
5. Totals may not sum due to rounding.

Table 6-3: Viceroy Annual Production

Item	Units	Total	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004
Total, Leach Plus Mill															
Total to Pad	000s ton	36,193	2581	3658	4083	4204	4120	4103	3891	4123	4120	1308	-	-	-
Grade	oz/ton	0.043	0.051	0.058	0.056	0.05	0.037	0.038	0.027	0.034	0.04	0.037	-	-	-
Contained, Annual	000s oz	1,550	131.6	211.5	230	211.3	151.6	155.9	105.1	140.2	164.8	48.4	-	-	-
Recovered, Annual	000s oz	1,243	78	133.2	170.3	156.9	122.2	122.4	89.1	95	118.7	77.7 [1]	56.7	14.8	8.2

Notes:

1. Includes 36,487 oz Au recovered from clean up and closure activities.
2. Source: Pressacco, 2013.

6.4 EQUINOX MINERAL RESOURCE ESTIMATES

Equinox has completed numerous technical reports that include Mineral Resource Estimates completed in accordance with NI 43-101 (Wakefield and Tschabrun, 2017; Gray et al. 2016; Cox at al., 2014 and Pressacco, 2013). These Mineral Resource Estimates are no longer current and have been superseded by the Mineral Resource Estimate summarized in Section 14 of this Report.

6.4.1 Pre-Feasibility Mineral Resource Statement

The Pre-Feasibility (PFS) Mineral Resource Estimate with an effective date of March 29, 2018 was produced by Don Tschabrun, SME RM, Mine Technical Services Ltd. (Scott et al., 2018). This Mineral Resource Estimate is no longer current and has been superseded by the Mineral Resource Estimate summarized in Section 14 of this report.

The PFS Mineral Resource estimate utilized an inverse distance weighting method bounded by multiple grade shells and geologically interpreted domains. A resource classification was developed based on estimation pass. The PFS Mineral Resource Estimate is presented in Table 6-4 (imperial units) and Table 6-5 (metric units) at a gold cut-off grade of 0.005 oz/ton (0.17 g/tonne) for hard rock resources and 0.004 oz/ton (0.14 g/tonne) and contained within a Lerchs-Grossman (LG) shell based on a gold price of $1,400/oz.

Based on the assumptions provided in Table 6-6, the ROM breakeven gold cut-off grade was calculated to be 0.005 oz/ton (0.17 g/t).

Table 6-4: Pre-Feasibility Study Mineral Resource Estimate Inclusive of Reserves (Imperial Units)

	Measured			Indicated		
Cut-off (Au oz/ton)	Tons (million)	Gold Grade (oz/ton)	Gold Oz (million)	Tons (million)	Gold Grade (oz/ton)	Gold Oz (million)
Hardrock (0.005)	177.1	0.0169	2.99	71.7	0.0161	1.15
Backfill (0.004)	0.0	0.0000	0.00	18.0	0.0101	0.18
Total (0.005)	177.1	0.0169	2.99	89.7	0.0149	1.34
Hardrock (0.035)	13.4	0.0777	1.04	5.3	0.0765	0.40
	Measured + Indicated			Inferred		
Cut-off (Au oz/ton)	Tons (million)	Gold Grade (oz/ton)	Gold Oz (million)	Tons (million)	Gold Grade (oz/ton)	Gold Oz (million)
Hardrock (0.005)	248.8	0.0167	4.15	167.2	0.0121	2.02
Backfill (0.004)	18.0	0.0101	0.18	21.7	0.0081	0.18
Total (0.005)	266.8	0.0162	4.33	188.9	0.0116	2.20
Hardrock (0.035)	18.6	0.0774	1.44	5.8	0.0826	0.48

Notes:

1. The Mineral Resource is from the Castle Mountain Pre-Feasibility Study with an effective date of March 29, 2018 and is no longer current.
2. The Qualified Person for the estimate is Don Tschabrun, SME RM.

3. Mineral Resources are inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
5. The Mineral Resource is based on a gold cut-off grade of 0.005 oz/ton.
6. The Mineral Resource is contained within an LG shell limit using a $1,400 gold price as well as cost and recovery parameters presented in Table 6-6.
7. The source for this information is Scott et al., 2018.

Table 6-5: 2018 Pre-Feasibility Study Mineral Resource Estimate Inclusive of Reserves (Metric Units)

	Measured			Indicated		
Cut-off (Au g/t)	Tonnes (million)	Gold Grade (g/t)	Gold Oz (million)	Tonnes (million)	Gold Grade (g/t)	Gold Oz (million)
Hardrock (0.17)	160.6	0.579	2.99	65.1	0.552	1.15
Backfill (0.14)	0.0	0.000	0.00	16.3	0.346	0.18
Total (0.17)	160.6	0.579	2.99	81.4	0.511	1.34
Hardrock (1.20)	12.1	2.664	1.04	4.8	2.623	0.40
	Measured + Indicated			Inferred		
Cut-off (Au g/t)	Tonnes (million)	Gold Grade (g/t)	Gold Oz (million)	Tonnes (million)	Gold Grade (g/t)	Gold Oz (million)
Hardrock (0.17)	225.7	0.572	4.15	151.7	0.415	2.02
Backfill (0.14)	16.3	0.346	0.18	19.7	0.278	0.18
Total (0.17)	242.0	0.556	4.33	171.4	0.399	2.20
Hardrock (1.20)	16.9	2.652	1.44	5.2	2.832	0.48

Notes:

1. The Mineral Resource is from the Castle Mountain Pre-Feasibility Study with an effective date of March 29, 2018 and is no longer current.
2. The Qualified Person for the estimate is Don Tschabrun, SME RM.
3. Mineral Resources are inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Numbers in the table have been rounded to reflect the accuracy of the estimate and may not sum due to rounding.
5. The Mineral Resource is based on a gold cut-off grade of 0.17 g/t.
6. The Mineral Resource is contained within an LG shell limit using a $1,400 gold price as well as cost and recovery parameters presented in Table 6-6.
7. The source for this information is Scott et al., 2018.

Table 6-6: 2018 Pre-Feasibility Study Lerchs-Grossman Parameters

Parameter	Unit	Value
Gold Price	$/oz	1,400
Marketing Cost	$/oz	5.00
Royalty – Jumbo, JSLA, Oro Belle	%	2.65
Royalty – South Domes	%	7.65
Mining Cost: ROM	$/ton	2.08
Mining Waste: Rock	$/ton	1.43
Mining Waste: Backfill & Overburden	$/ton	1.19
Process Cost: ROM	$/ton	1.54
Process Recovery: ROM	%	72.0
General & Administrative	$/ton	1.00
Mining Dilution	%	0.0
Pit Slope	°	48

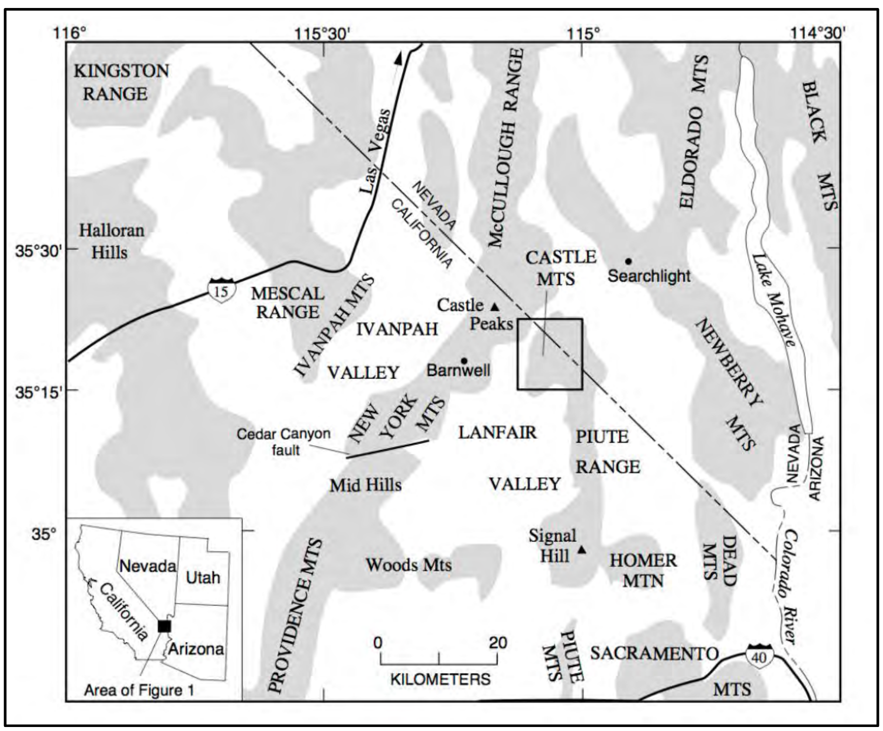
Note: All dollars are $ and tons are dry tons. (Source: Scott et al., 2018)

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGICAL SETTING

The Castle Mountain epithermal gold deposit is in the Hart Mining District (Tucker and Sampson, 1943; Wright, 1953; Hewett, 1956; Linder, 1989), at an elevation of approximately 4,500 ft (1,372 m) in the southern portion of the Castle Mountains Range, eastern San Bernardino County, California.

The Castle Mountains Range is in the eastern Mojave Desert within the southern Basin and Range Province (Theodore, 2007). The Castle Mountains comprise a small range of Miocene volcanic rocks at the northern end of the Lanfair Valley in eastern California and extend north into Nevada (Figure 7-1). Tectonically, the Castle Mountains are located along the northwestern margin of the Colorado River extensional corridor, a regional tectonic feature (Howard et al., 1994). Extension in the Colorado River corridor occurred in the mid-Tertiary and is characterized by detachment faulting in Nevada, California and Arizona that extends from south of the Las Vegas-Lake Mead shear zone to north of the San Andreas fault (Ausburn, 1991, 1995). Miocene volcanism and the structural trends of Miocene volcanic rocks and faults in the region are related to continental extension. While no regional low-angle normal (detachment) faults outcrop in the Castle Mountains (or in the Piute Range to the south), detachment faults are exposed to the west and east, in the Kingston Range and Black Mountains, respectively (Howard et al., 1994). These field relations suggest the Castle Mountains Range occur along the western margin of detachment faulting (Ausburn, 1991). Structures in the Castle Mountains are temporally and spatially consistent with crustal extension in the region but lack significant extension like that within the Colorado River Extensional Corridor to the east (Linder, 1989; Capps and Moore, 1991, 1997; Nielson et al., 1999; Spencer, 1985).



(Source: Neilson et al., 1999)

Figure 7-1: Location Map of Castle Mountains in SE California and SW Nevada

The southern portion of the Northern Colorado River Extensional Corridor experienced early Miocene calc-alkaline volcanism accompanied by weak to moderate north-south extension between ca. 20–16 Ma due to the collapse of the topographically elevated Kingman arch to the north (Faulds et al., 2001). By 15.5 Ma, east-west extension intensified and was accommodated on major easterly dipping normal faults and are represented by regionally extensive detachment faulting in the Castle Mountains area. Throughout the Northern Colorado River Extensional Corridor, the onset of east-west extension marked the transition from deposition of intermediate volcanic rocks to more bimodal to felsic dominated composition. The thickness of pre- and syn-extensional volcanic units indicate the bulk of the volcanic pile was deposited prior to, or immediately after the onset of east-west extension. Felsic volcanism spanned the entire duration of extension leading to accumulations of thick, complexly faulted, felsic volcanic piles in half-grabens formed during ongoing east-west extension.

Capps and Moore (1997) proposed multiple episodes of deformation in the Castle Mountains including Proterozoic deformation, Mesozoic deformation, and Miocene dilation. This deformational regime is associated with growth faults and hypabyssal dike emplacement and cryptic Miocene northwest-striking faulting. Proterozoic deformation is manifested by well-developed foliation commonly dipping to the northeast in basement rocks. Other structural events are manifested primarily as north-northeast striking normal faults with variable dips ranging from low to sub-vertical. Capps and Moore (1997) make note of a cryptic northwest fabric that crosscuts the prominent northeast structures.

7.2 LOCAL GEOLOGY

The Castle Mountain Range was mapped by R. C. Capps from 1987 through 1994. In 1997, a map and discussion of the geology was published by the Nevada Bureau of Mines and Geology, titled “Castle Mountains Geology and Gold Mineralization, San Bernardino County, California and Clark County, Nevada” (Capps and Moore, 1997). Unless noted, the following discussion of the regional geology of the Castle Mountains is derived from Capps and Moore (1997). An idealized, schematic stratigraphic section of the Castle Mountain geology is presented in Figure 7-2 and is discussed in detail below.

Proterozoic basement is exposed along the northeastern flank of the Castle Mountains and has been intersected in drilling below the northern Oro Belle Pit; to the west of the Jumbo Pit, and underneath shallow alluvium near the northern portion of the leach pad. Biotite schist, biotite gneiss and metamorphosed granite comprise the Proterozoic basement that is dated at approximately 1650 Ma (Wooden and Miller, 1990). Only narrow zones of hydrothermal alteration and weak gold mineralization have been encountered in basement rocks.

Locally overlying the metamorphic basement rocks is a poorly sorted, clast supported conglomerate with local well-bedded sandstone up to 181 ft (55 m) thick. Clasts are completely composed of rounded to sub-angular Proterozoic metamorphic rocks, ranging from pebble to cobble size. These Proterozoic sedimentary rocks are referred to as Pc. Seds and have been intersected by drill holes, notably in the northern Oro Belle pit area and below the JSLA pit.

Unconformably overlying the Pc. Seds, is the Miocene Peach Springs Tuff (Young and Brennan, 1974; Glazner et al., 1986; Nielson et al., 1990; Buesch, 1992), dated at around 18.8 Ma (Ferguson et al., 2013). The Peach Spring Tuff is a felsic welded ignimbrite tuff that is regionally extensive throughout the southwestern USA.

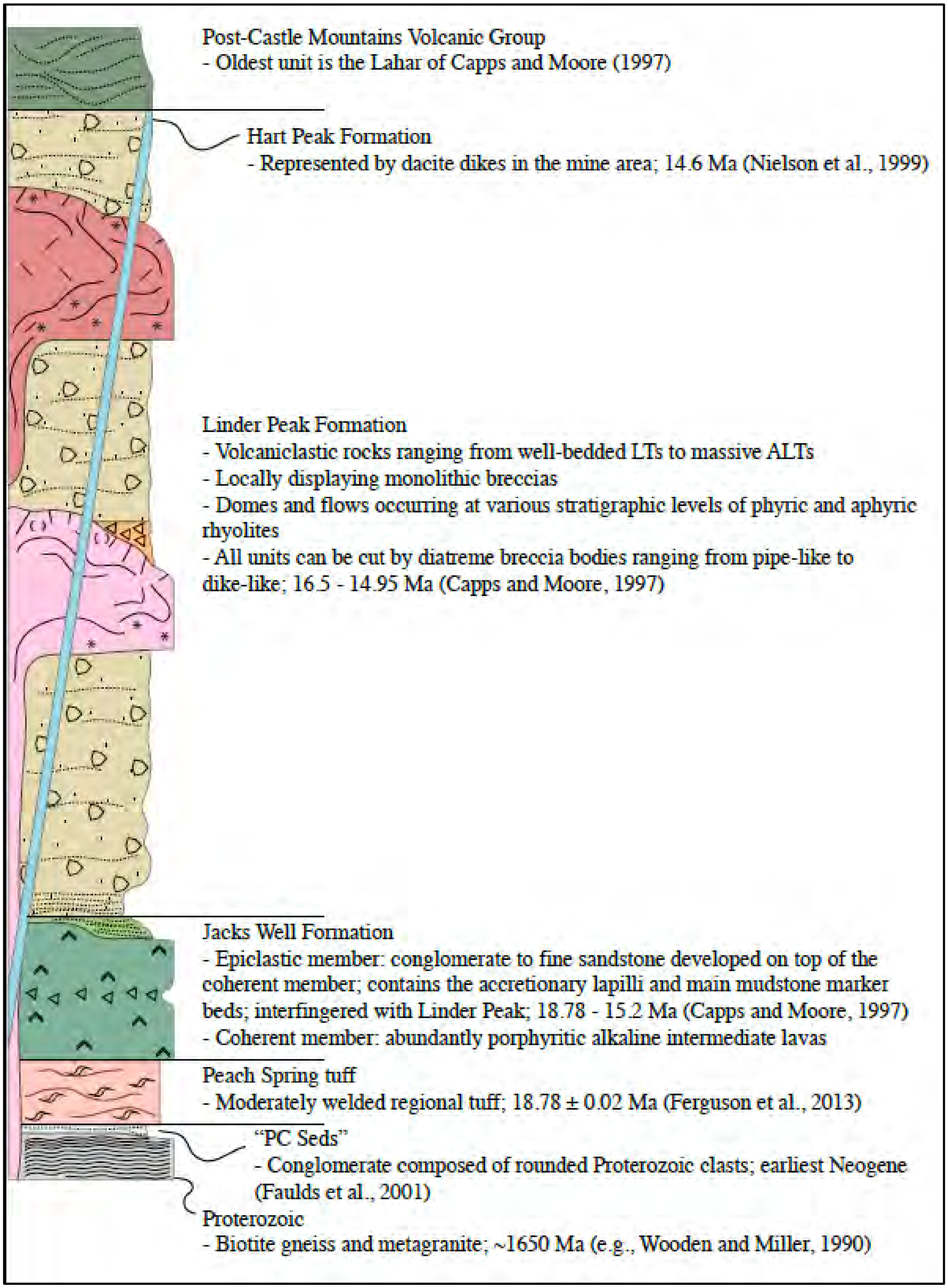
Overlying the Peach Springs Tuff are three formations that comprise the Miocene Castle Mountains Volcanic Sequence (CMVS), the Jacks Well formation (15.20 ±0.03 Ma), Linder Peak (14.90 – 16.5 Ma) and Hart Peak (13.8 – 16.3 Ma) formations. The CMVS consists primarily of rhyolitic domes, flows, and felsic tuff, and lesser andesitic, latitic, and basaltic lava emplaced during three main igneous episodes between around 18.8 and 13.5 Ma. The CMVS is the host sequence of the epithermal gold mineralization at the Project.

The Jacks Well formation is a mixed sequence of andesitic to basaltic lavas and associated volcaniclastic rocks. Common rock types include trachyandesite to basaltic-andesite flows, minor rhyolite ash-flow tuff locally displaying accretionary lapilli textures, and locally abundant debris flow and epiclastic deposits.

The Linder Peak formation overlays the Jacks Well formation and comprises abundant pyroclastic-surge tuff, and volcaniclastic rocks that include porphyritic and aphyric rhyolite that collectively form flow-dome complexes.

The Hart Peak formation is comprised of porphyritic rhyolite flows, plugs, welded ash-flow tuff; pyroclastic-surge tuff; rhyolite dikes and volcaniclastic rocks. The Heart Peak formation and intrusive equivalents post-date the Jacks Well and Linder Peak formation and was deposited after gold deposition at Castle Mountain.

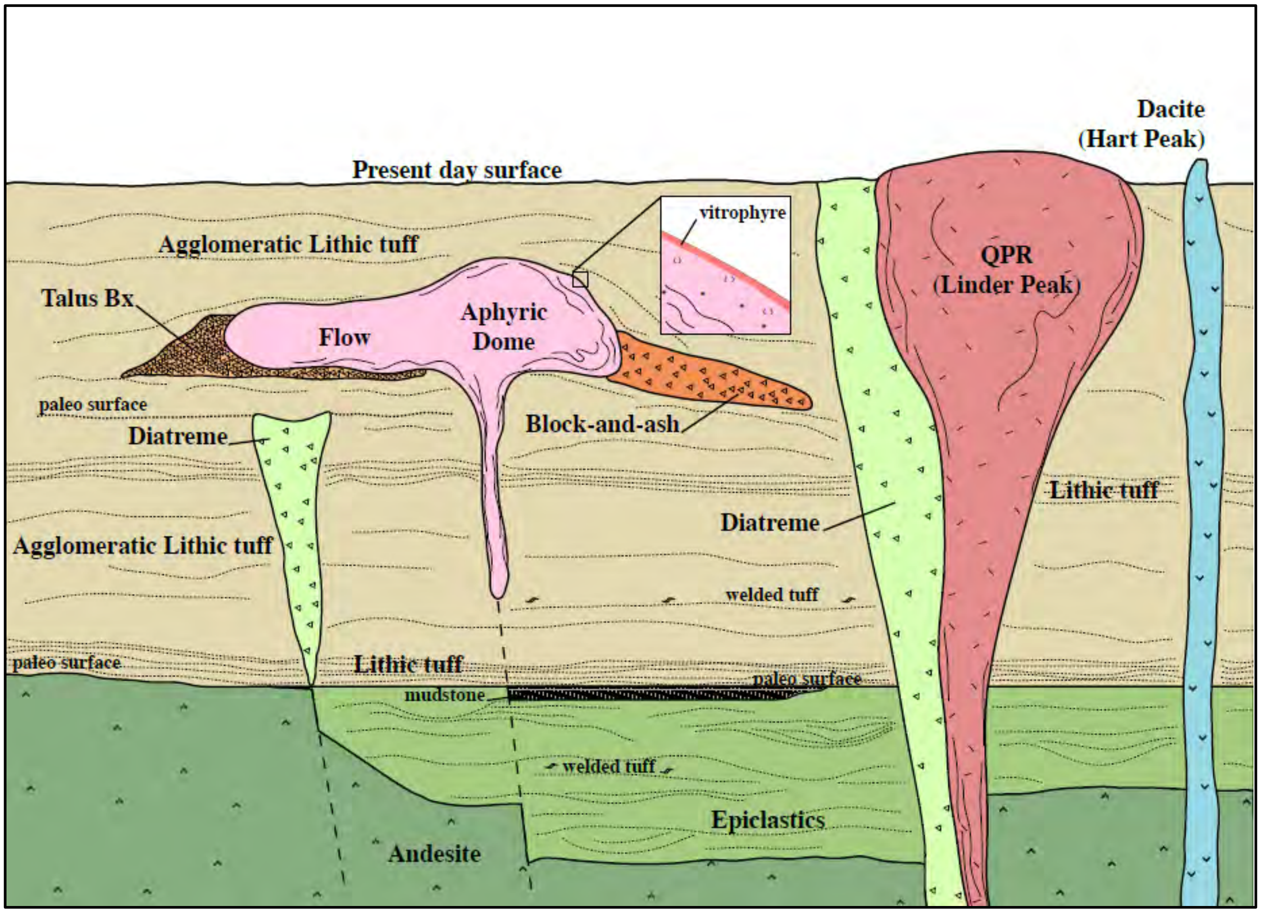
The Piute formation overlies the Hart Peak formation and is characterised by post mineralization volcanic flows and lahars of intermediate to mafic composition including trachyandesite to trachybasalt and associated volcaniclastic and epiclastic rocks aged 13-14 Ma (Nielson and Nakata, 1993; Capps and Moore, 1997).



(Source: modified from Tharalson, 2017)

Figure 7-2: Schematic Stratigraphic Section for the Castle Mountain Property

The Castle Mountain stratigraphy is dissected by north-south and northeast-southwest trending extensional faults interpreted to be both syn- and post-volcanic. Continuous extension and normal faulting focused protracted volcanism and hydrothermal activity. The CMVS is cut by Hart Peak formation dacite dikes that exploited pre-existing structures during emplacement. This is supported by their orientation and extent of mineralization on/near their margins. A schematic reconstruction of the principal lithostratigraphic elements at Castle Mountain Project is presented as Figure 7-3 (Tharalson, 2017).



(Modified from Tharalson, 2017)

Figure 7-3: Schematic Reconstruction of the Principal Lithologic and Stratigraphic Elements – Derived from Detailed Mapping and Logging at the Castle Mountain Project (Not to Scale)

7.3 PROJECT GEOLOGY

Mapping, drilling and extensive relogging campaigns carried out by Equinox, have been synthesized into a three-dimensional geological model to characterize the Project geology, structure, alteration, and mineralization. Equinox geologists have built on the work of Capps and Moore (1997) and Tharalson (2017) to model the Project geology. The geological mapping interpretation which incorporates the lithogeochemical and mapping studies reported by Barrett (2016a, b), Monecke (2017) and Nicholls et al. (2017) is presented in Figure 7-4. This interpretation forms the framework for the geological model (Figure 7-5).

7.3.1 Lithology

Table 7-1 presents a summary of the lithologies modelled at the Project. The Proterozoic Basement (Pc. Basement) and individual rock types that form the Jacks Well formation. are not volumetrically significant and the unit description within Section 7.2 adequately summarizes those rock types (Table 7-1 codes 23 through 31). Most of these units form flat lying stratigraphy that is locally offset by major faults. These units, the CMVS and overlying post mineralization cover units are described with photographic examples in Table 7-2.

Table 7-1 Castle Mountain Lithology

Lithology	Lithology Model Code	Model Code
Alluvium	02-Alluvium	2
Debris Flow	03-DebrisFlow	3
Dacite	05-Dacite	5
Hart Peak Rhyolite	05-HartRhy Migos	5
Rhyolite Breccia	07-RhyBx	7
Porphyrytic Rhyolite	09-RhyPorphyritic	9
Aphyric Rhyolite	11-RhyAphyric	11
Volcaniclastic Diatreme	14-VxDiatreme	14
Volcaniclastic	16-Vx	16
Mudstone	22-Mudstone	22
Epiclastics	23-Epiclastics	23
Andesite	27-Andesite	27
Peach Springs Tuff	29-PeachSpring	29
Proterozoic Sediments	30-PcSediments	30
Proterozoic Basement	31-PcBasement	31

Rhyolite rock types of the Linder Peak formation dominate surface exposures and comprise most of the drilling. Rhyolites are the most volumetrically abundant rock type at the Project. Rhyolite rocks occur as a complex package of flow-domes, and clastic tuffs comprised of monolithic breccia, polylithic breccia, and ashfall tuffs. Individual rhyolite flow-domes are identified using contact relationships with geometries that are vertically continuous and laterally restricted, by subtle compositional and textural variations, and in some cases vitrophyre or volcanic glass which mark the rapid cooling of molten rhyolite. Sub-types include quartz porphyritic, quartz-feldspar porphyritic, and aphyric rhyolites with an array of textures ranging from flow-banded, massive, vitrophyric/perlitic, spherulitic, and vesicular (Table 7-2 codes 07, 09, and 11). Geochemically, there is little to no chemical distinction between individual rhyolite bodies found in the Castle Mountain Project area which indicate a single magma source (Barrett, 2016a).

Volcaniclastic facies are the second most abundant rock types found on the Project. In contrast to the coherent rhyolite facies, volcaniclastic facies have a property-wide stratigraphy which provides some predictability and an overall better understanding of the depositional mechanism. Initial work established a massive matrix- to clast-supported polymictic breccia and an ash-dominated lithic and lapilli tuff (Monecke, 2016). Mapping has refined the basic stratigraphy to include a block-and-ash flow tuff unit, finer-grained stratified lithic and lapilli tuffs (formerly termed

LT), as well as both stratified and massive polymictic agglomerate tuffs (formerly termed ALT) (Table 7-2 code 16). On a property-scale, stratigraphy generally coarsens upward, with the exception of the basal block-and-ash flow tuff. The contacts appear to be generally conformable, and in some cases gradational.

Volcaniclastic diatreme consist of diatreme breccia facies and are exposed through the central portion of East Ridge as well as in the JSLA, Oro Belle and Jumbo pits. Diatreme facies have a limited surface expression due to being vertically continuous and having a laterally restricted geometry. Diatreme composition and textures closely resemble agglomerate tuffs leading to inconsistent identification at surface. Diatreme facies are polymictic, matrix supported breccia with a massive clay-rich rock flour matrix. Clast abundance ranges from 20-70%, with >90% rhyolite, 5% to 10% andesite, <1% gneiss and rare epiclastic mudstone clasts. Diatremes locally contain pumice fragments, but strong alteration typically obscures much of the original fine textures. Clasts range from <2 mm to over 2 m. The matrix is clay-altered ash and lesser very fine glass. There is no internal structure and clast orientations are random (Table 7-2 code 14).

Individual diatreme bodies range from <1 m to >100 m wide with smaller diatremes having well-defined contacts, whereas larger diatremes can have either sharp well-defined contacts or more erratic, hard to define contacts. At depth, diatreme bodies thin and pinch out into numerous narrower diatreme bodies and faults/fractures.

7.3.2 Trachydacite (Dacite) Dikes

Felsic rocks are crosscut by a series of unaltered, locally columnar-jointed intermediate dikes termed trachydacite to trachyandesite (Tiha) by Capps and Moore (1997), although historically and colloquially they are referred to as dacite dikes on the Project. Geochemically and mineralogically, these are trachydacite since they are high-potassium and contain approximately 20% quartz (Barrett, 2016a, b). Dacite dikes are biotite-feldspar phyric and have sharp, typically chilled margins where they contact the host rocks (Table 7-2 code 05). The dacite is interpreted to post-date the mineralizing event and are ascribed to the Hart Peak formation.

At least three distinct dacite dikes occur on the Property. The most prevalent dacite dike trends north-south and is exposed on the east side of East Ridge. A second major dike is recognized in the Oro Belle pit which trends approximately 045° (Table 7-2 code 03). A third less prominent dike trends 020° and is discontinuous along the top of East Ridge for approximately 984 ft (300 m).

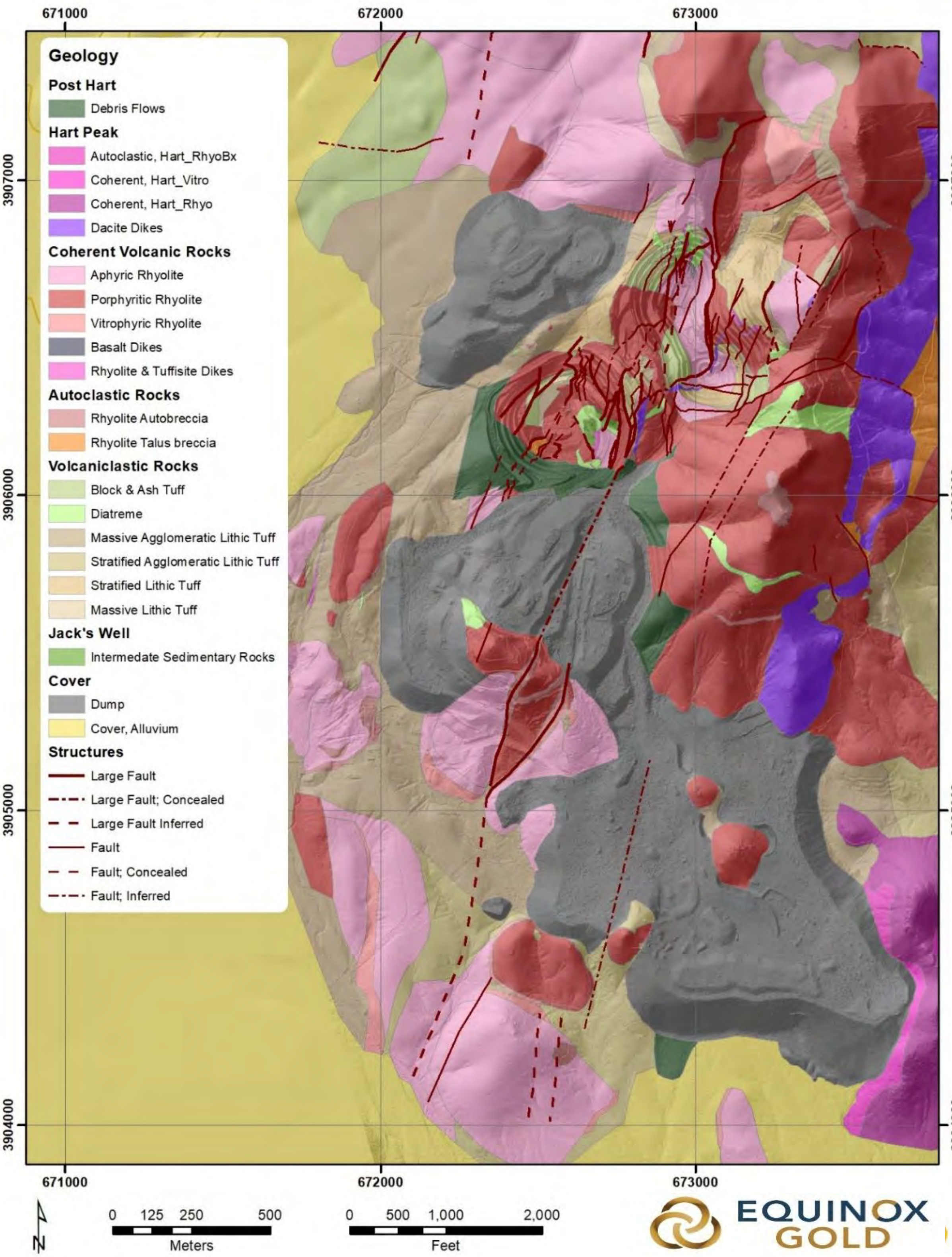


Figure 7-4: Castle Mountain Project Geology

Table 7-2: Photograph Examples of Castle Mountain Rock Types

Code	Modelled Lithology (Model code) *Stratigraphic Unit* Description	Photograph
03	Debris Flow (03) *Piute Range* Heterolithic, coarse sandstone to pebble conglomerate.	
05	Dacite (05) *Hart Peak Formation* Dacite dyke (right side) cross-cutting Linder Peak formation rhyolites in the Oro Belle pit.	

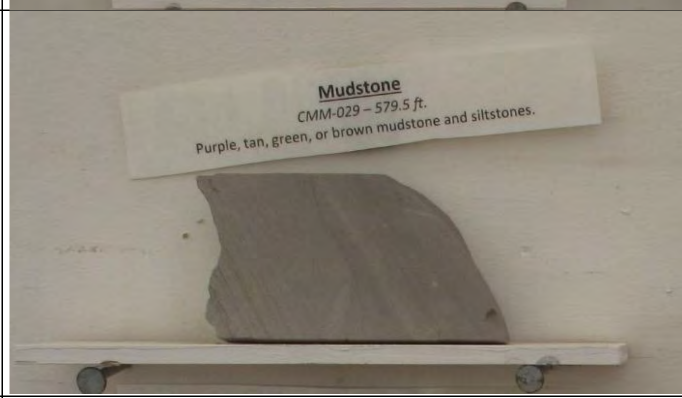
Code	Modelled Lithology (Model code) *Stratigraphic Unit* Description	Photograph
07	Rhyolite Breccia (07) *Linder Peak Formation* Angular cm-scale rhyolite clasts (monomictic), varies from matrix to clast supported with silica infill.	
09	Porphyritic Rhyolite (09) *Linder Peak Formation* Weakly flow-banded quartz phyric rhyolites.	

Code	Modelled Lithology (Model code) *Stratigraphic Unit* Description	Photograph
11	Aphyric Rhyolite (11) *Linder Peak Formation* Monomict rhyolite breccia consisting of spherulitic quartz phyric rhyolite fragments in rock flour matrix.	
11	Aphyric Rhyolite (11) *Linder Peak Formation* Aphyric flow-banded rhyolites in core.	

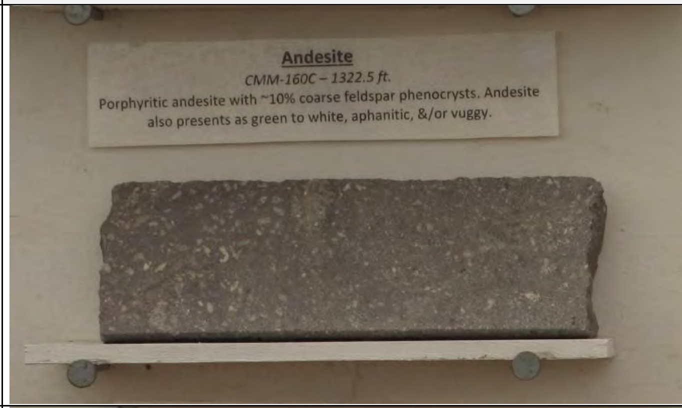
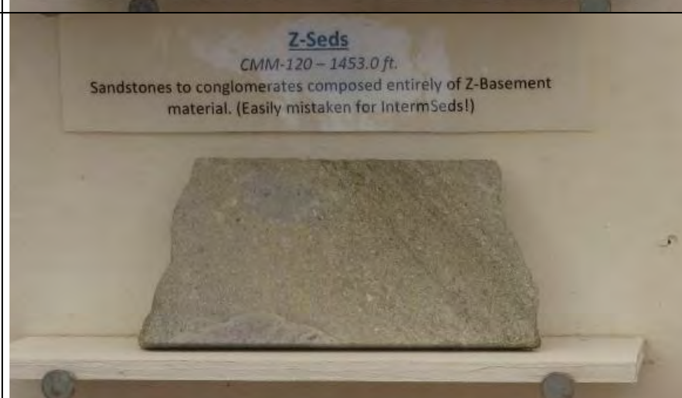
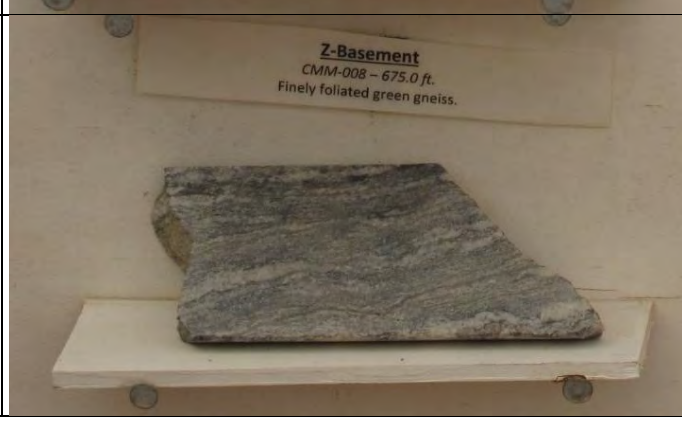
Code	Modelled Lithology (Model code) *Stratigraphic Unit* Description	Photograph
14	Volcaniclastic Diatreme (14) *Linder Peak Formation* Diatreme with 70 cm clast of massive quartz phyric rhyolite and a 10 cm clast of andesite. The matrix contains numerous other clasts ranging from 3-6 cm in size.	
16	Volcaniclastic (16) *Linder Peak Formation* Bedded lithic tuff with stratified ash matrix and fine-grained lithic clasts <1 mm.	

<table>
<tr><th>Code</th><th>Modelled Lithology (Model code)
Stratigraphic Unit
Description</th><th>Photograph</th></tr>
<tr><td>16</td><td>Volcaniclastic (16)
Linder Peak Formation

Finely bedded lithic tuff.</td><td>
</td></tr>
<tr><td>22</td><td>Mudstone (22)
Jack Well Formation

Mudstone</td><td>
</td></tr>
<tr><td>23</td><td>Epiclastics (23)
Jack Well Formation

Sandstone with graded bedding.</td><td>
</td></tr>
</table>

Code	Modelled Lithology (Model code) *Stratigraphic Unit* Description	Photograph
24	Andesite (24) *Jack Well Formation* Porphyritic andesite with 10% feldspar crystals.	
29	Peach Springs Tuff (29) Welded ignimibrite felsic tuff with fiamme.	
30	Pc. Sediments or Pc. Seds (30) Sandstone	
31	Pc. Basement (31) Biotite Gneiss, Meta-granite Foliated gneiss.	

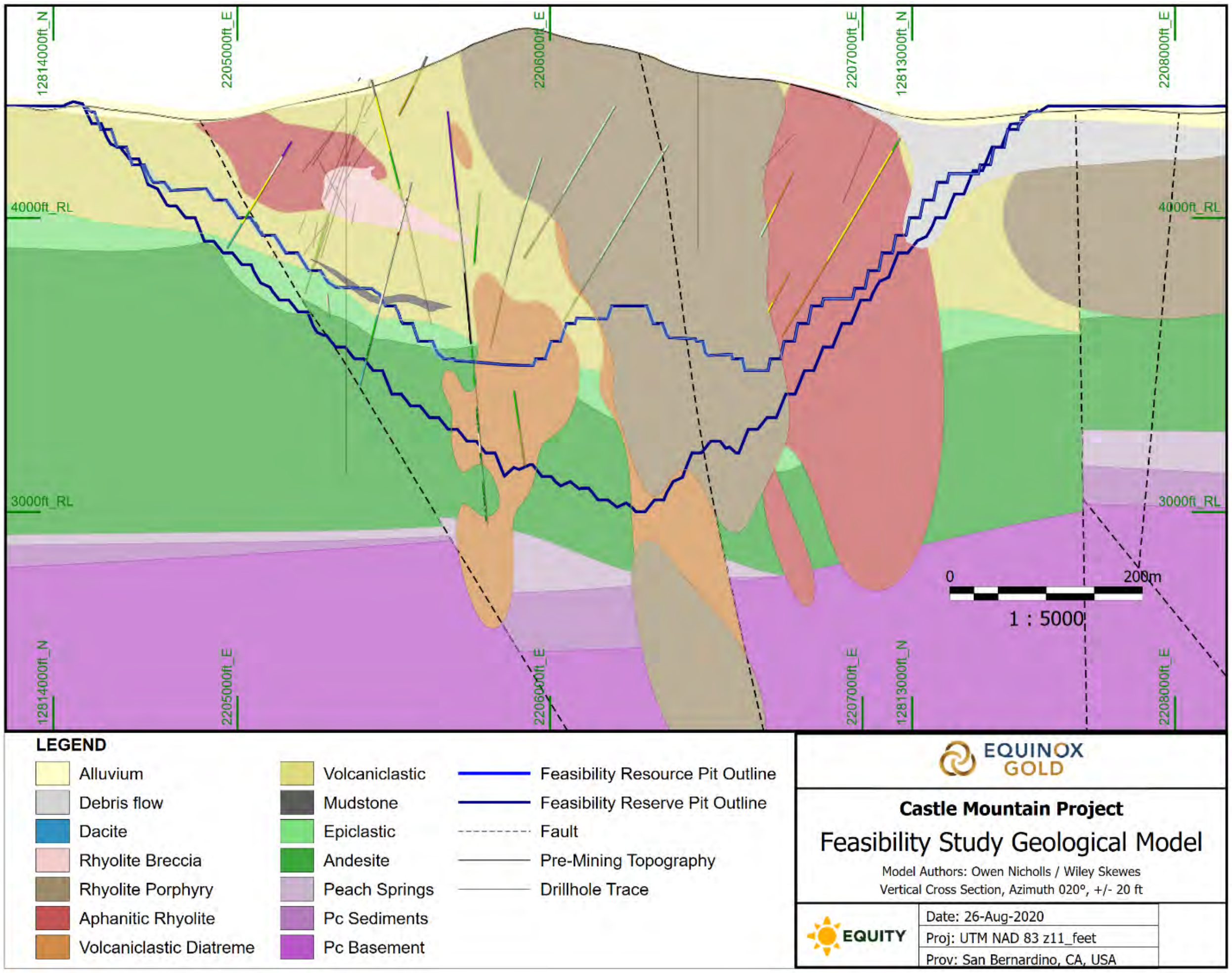


Figure 7-5: Cross Section of the Castle Mountain Project Geological Model

7.3.3 Structure

There are two dominant orientations of faults on the Project. The youngest generation of faults trend northeast-southwest and offsets earlier east-west trending faults. The moderately to steeply southeast dipping, northeast striking faults are the dominant structures. There are conjugate sets of northwest dipping faults. These commonly express “y” patterns in the field due to the southeast dipping set truncating and locally offsetting the northwest dipping sets. These are normal faults with a minor dextral strike-slip component. These faults have been mapped on surface and have also been modelled in 3D from drilling data and are named Maverick, Goose, Ice Man, Ripley, Dillon, McLane, Predator.

Capps and Moore (1997) recognized east-west faults which they interpreted to be pre- to syn-Linder Peak since they consistently cut Jack Wells rocks and locally extend into the Linder Peak rocks. The limited exposure of east-west faults mapped by Equinox geologists indicate these faults are truncated by the north-northeast trending faults. The cross-cutting relationships and limited exposure of east-west faults suggest these faults predate the more dominant north-northeast trending faults. Additional support for the early timing of east-west faults is the ubiquity of north trending faults through all lithofacies in Linder Peak, whereas the east-west faults have only been observed cutting aphyric rhyolite flow-domes and inferred to cut the surrounding volcaniclastic rocks. Furthermore, the geometry of many of the mapped diatremes appear to have an east-west orientation suggesting the east-west faults could have acted as initial pathways for diatreme emplacement.

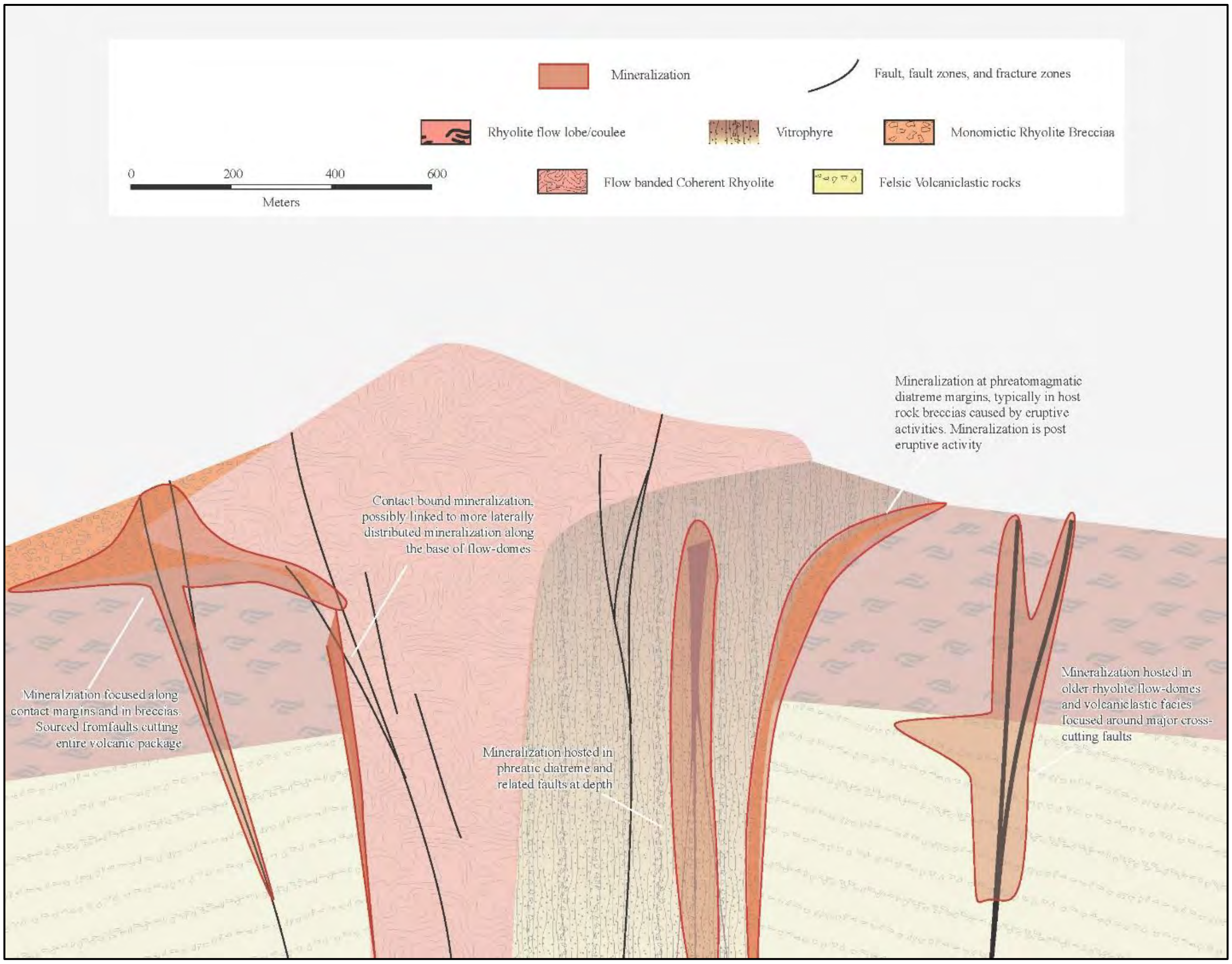
7.3.4 Mineralization

Structure and associated rock porosity-permeability characteristics are the first-order control on the distribution of gold. Flow-dome breccia margins, phreatic diatremes, fault cataclasite and fractures focused provided conduits for hydrothermal fluids and contain the highest gold grades. Unfractured coherent flow-dome facies, clay altered volcaniclastic facies, and clay altered phreatomagmatic diatremes with low or variable permeability are weakly mineralized due to lower fluid interaction. Lower permeability units that are mineralized have been cut by structures such faults, fractures, or phreatic diatremes that promoted hydrothermal fluid interactions.

Gold is focused along structures and margins of facies contacts. It is believed that sub-vertical structures acted as pathways for magma and are responsible for the emplacement of the felsic volcanic package. These same structures also acted as conduits for gold-bearing hydrothermal fluids. Intersections of the steep structures with more permeable volcanic rocks created an environment for enhanced gold precipitation from hydrothermal fluids, possibly due to processes of boiling and interaction with meteoric water.

Lithologic controls are dependent on the host rock texture. Tuff beds, auto-breccias, and hydrothermal breccias have permeable fragmental textures. Brittle rhyolite flows and intrusive equivalent rocks exhibit intense fracturing and are characterized by cooling joints, vesicular zones, spherulitic vugs, and flow foliations. Gold occurs within secondary silica in all these features. Major fault and fracture systems and intersections of fracture systems provided structural controls for mineralization. In the deposit area, north-northeast-striking, mineralized fracture zones are exposed in outcrop.

The morphology of mineralization follows two patterns. Firstly, gold is enriched along steep to vertical brecciated contacts of flow-domes and phreatomagmatic diatremes. Secondly, gold occurs in broad tabular zones that correlates with the general orientation bedding. The lateral extent of the mineralized bodies centered around fault zones are dictated by the intensity and extent of fracturing and faulting, in addition to the paleo-porosity of the host rocks (Figure 7-6).



(Source: Nicholls et al., 2017)

Figure 7-6: Schematic Model for Zone of Gold Mineralization Found on the Castle Mountain Property

Some faults and fracture zones are not gold-bearing since the structural regimes through the Project were active both pre- and post-mineralization. Gold seems to have precipitated during a single phase within a larger and longer-lived structural and hydrothermal event.

Silicification is commonly associated with gold occurring as pervasive silica flooding and quartz veining. Quartz veins can be vitric and “gel-like” or opaque white-gray opal. Vitric quartz veins typically occur in clusters as sheeted veins or stockwork in zones ranging from 3 to 35 ft (1 to 10 m) wide. Amorphous quartz occurs as discontinuous irregular veins and as open space filling quartz. The strongest silica alteration associated with gold is found along brecciated coherent rhyolite margins; this results in mosaic breccias where angular rhyolite clasts are within a hydrothermal-related silica matrix.

Gold on the Project occurs in oxidized fractures, faults, discontinuous veins, and breccia matrix. Gold mineralization correlates best with the deep red, red-brown and brown iron oxide that can range in color from pink to red-brown. The iron oxide intensity and appearance are commonly controlled by the volcanic facies occurring as discontinuous, fracture-controlled textures in coherent rhyolite facies, as matrix replacement in rhyolite breccias, wispy selvages and clast haloes in volcaniclastic rocks, and pervasive or matrix selective in diatremes. These iron oxide textures can be cut by fracture and vein filling iron oxide that ranges in color from brown-tan to red.

Visible gold is rarely observed in hand specimen and core. In petrographic samples collected near JSLA, visible gold is associated with iron oxide and silica and proximal to illite and adularia alteration (Cline, 2016). Gold deportment studies from Oro Belle by Chudy and Lane (2020) indicate that mineralization is roughly 79% native gold, 17% electrum and 4% silver minerals by frequency of grain count. Quartz may be intergrown with iron oxides/hydroxides, most commonly as hematite, which have formed as oxidation products of former sulfide minerals. There is a low abundance of sulfides observed on the Project. The most common sulfide mineral is pyrite, and varies from nil to 1%, which occurs within clasts and matrix.

8 DEPOSIT TYPES

Castle Mountain is classified as a low-sulfidation epithermal gold deposit (Scott et al., 2018), a sub-type of the epithermal class of gold and silver deposits (Sillitoe and Hedenquist, 2003). Epithermal gold-silver systems are driven by magmatic activity and form high-level vein, stockwork, disseminated and/or replacement deposits that may be mined by open-pit and/or underground methods. Some deposits also contain substantial resources of Ag, Pb, Zn, Cu and/or Hg.

Epithermal deposits are generally of relatively recent (Cenozoic) age although there are also examples stretching back to 3.46 Ga (John et al., 2018). Bias towards young deposits likely reflects the generally poor preservation of these high-level deposits in tectonically unstable regions (John et al., 2018).

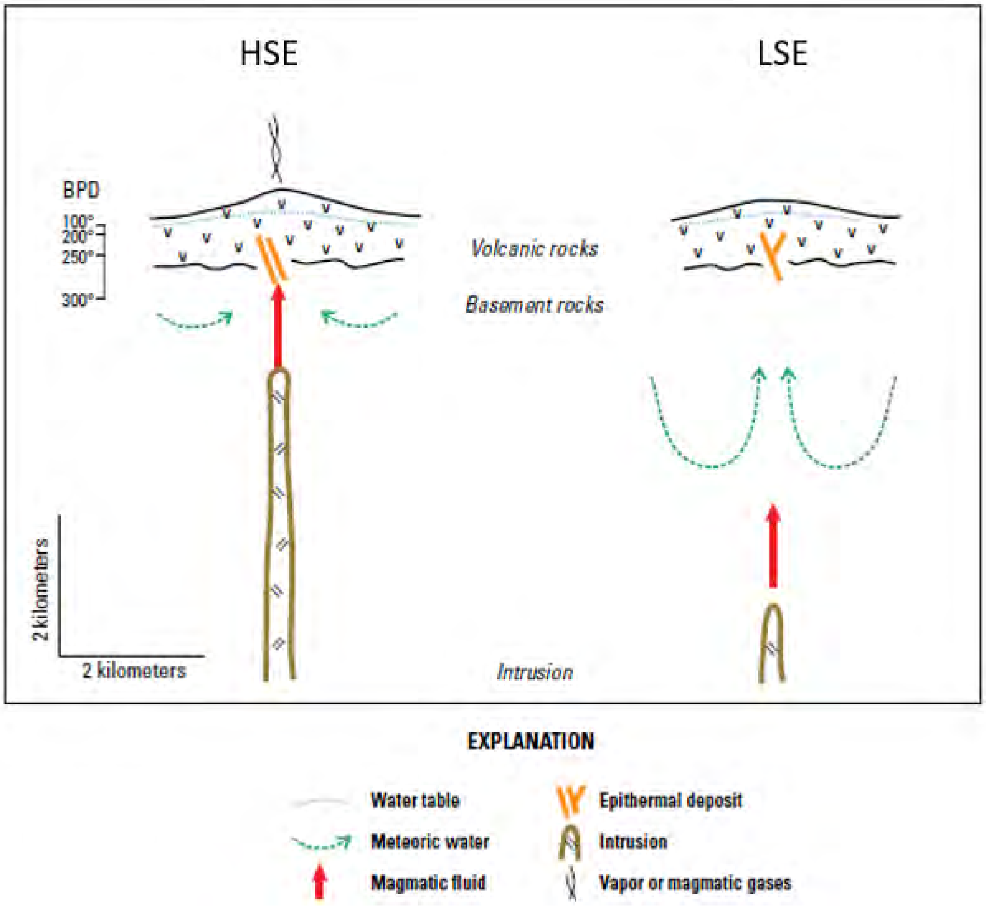
Most epithermal deposits are genetically related to hydrothermal systems formed from magmatic ± meteoric fluids (Sillitoe and Hedenquist, 2003). Igneous rocks parental to magmatic fluids range from calc-alkalic to alkalic andesite-dacite and bimodal basalt-rhyolite suites.

Mineralization occurs at depths of 50 to 1500 m below the paleowater table and at temperatures of 150 to 300ºC (John et al., 2018). Sulfur fugacity of the ore fluid is used to subdivide epithermal deposits into high- (HSE), intermediate- and low- (LSE) sulfidation types (Sillitoe and Hedenquist, 2003). HSE and LSE are endmembers with intermediate-sulfidation systems sharing features of both. The HSE to LSE spectrum is manifested in mineralogical changes, an increasing input of meteoric fluid (Figure 8-1) and increasing bimodal rhyolite-basalt magmatism (Sillitoe and Hedenquist, 2003; White and Hedenquist, 1995).

Epithermal deposits occur in mobile belts where they form in localized regions of neutral to extensional stress. LSE deposits favor regions under extensional stress such as intra-, near- and back-arc rifts as well as post-collisional rifts (Sillitoe and Hedenquist, 2003). Some of these rift settings also favor development of bimodal basalt and rhyolite suites. HSE deposits favor stress-neutral to mildly extensional regions with calc-alkalic magmatism.

The primary ore mineralogy in LSE systems consists of electrum (Au-Ag alloy with >20 wt% Ag) as well as silver-bearing sulfide, selenide and sulfosalt minerals, whereas HSE systems consist mostly of gold, Au-Ag alloy (<10 wt% Ag) and Cu-As sulfate (enargite) (John et al., 2018). Overall sulfide abundances are significantly higher in HSE deposits (5-90 vol%) than LSE (<1-20 vol%).

Epithermal deposits are typically formed by multiple pulses of hydrothermal activity, typically marked by cross-cutting structural features and localized to pervasive host rock alteration. Structural features that host mineralization include massive veins, vein swarms, vein stockworks, sheeted veins, hydrothermal breccia, residual vuggy silica, diatreme and fault intersections (Sillitoe, 1993; John et al., 2018). LSE deposits are more likely to form mineralized veins with classic comb and crustiform texture. HSE mineralization, on the other hand, occurs mostly in breccia, diatreme and replacement deposits (John et al., 2018). High-grade shoots may develop along structural intersections, as do lower-grade stockwork and/or breccia ores. Figure 8-1 is a schematic cross-section showing the difference between (left) high sulfidation epithermal (HSE) deposits, formed mostly from magmatic-hydrothermal fluids, and (right) low sulfidation epithermal (LSE) deposits formed by geothermal systems.



(Source: John et al., 2018; Simmons et al., 2005)

Figure 8-1: High Sulfidation Epithermal (HSE) Deposits vs. Low Sulfidation Epithermal (LSE) Deposits

Alteration assemblages are typically zoned laterally and vertically around ore bodies and relative to the paleowater table (John et al., 2018). Alteration in LSE systems is more restricted than HSE, with the idealized sequence comprising an outward gradation from a quartz-chalcedony ± adularia core through argillic and then propylitic assemblages (Simmons et al., 2005). HSE systems, on the other hand, comprise more pervasive quartz + alunite assemblages that grade outward through advanced argillic, argillic and propylitic assemblages (John et al., 2018). Alteration above paleowater table is marked by steam-heated acid and silica sinter assemblages, with argillic and advanced argillic assemblages typically spanning across the paleowater table.

The characteristics of epithermal deposits are amenable to a number of exploration methods. In areas of sufficient outcrop, geological mapping can be used to define the extent and high-flux regions of the hydrothermal system. Geochemical sampling, including rocks and soils, can rely on a large suite of pathfinder elements (Au, Ag, Cu, Zn, Pb, As, Sb, Bi, Se, Te, Tl, Mo, W, Sn, Hg) to vector into the more prospective parts of the system or identify systems under cover. Induced polarization surveys can be used to define sulfide-bearing rocks as well as resistive quartz-dominant alteration zones.

9 EXPLORATION

This section details exploration activity executed by Equinox (through its subsidiary NewCastle) from 2012 to the present. Details of exploration work conducted prior to Equinox are included in Section 6, History, and detail of the exploration drilling is provided in Section 10, Drilling.

9.1 SURVEYING

Heritage Surveying of Las Vegas, Nevada conducted an airborne LiDAR survey during 2012 to produce a detailed Digital Elevation Model (DEM) and capture the extent of previous mining. The results of the survey produced a 4 ft (1.2 m) contour map that covered 5,675 acres (2,300 ha).

Compass Tools of Denver, Colorado completed a high-resolution, drone- and fixed-wing based aerial photogrammetry survey during March 2017 to provide an updated topographic surface of the Project area. This survey was completed to increase resolution of the existing open pits and provide detailed locations of surface disturbance following the completion of the drill programs.

PhotoSat of Vancouver, British Columbia, Canada completed a follow up satellite survey and acquired detailed satellite imagery of the project area during December 2018. This survey produced a DEM with a 3 ft (0.9 m) contour interval and orthophotos of the project area.

9.2 MAPPING

Initial exploration in spring 2014 included detailed geologic mapping of the well exposed portions of the deposit areas and an evaluation of the deposit structure and stratigraphy. Further structural geology studies completed in spring 2015 focused on historic open pits and the relogging of diamond drill core to develop a new geological model. During the 2016 field season, a comprehensive mapping, sampling, petrographic and lithogeochemical program was initiated by NewCastle in an attempt to develop a deposit scale geological framework to assist future mapping and core logging. This study generated a detailed 1:2,000 scale geological map of the Project and assessed controls on gold mineralization as discussed in Section 7.3 (Nicholls et al., 2017).

9.3 GEOCHEMICAL SAMPLING

In tandem with the geological mapping, Equinox has taken 1,458 select and grid-controlled grab samples representing a suite of lithologies, structures and alteration. Selective sampling was undertaken in the Jumbo and Oro Belle pits as well as other areas with outcrop on the Project. Grid controlled sampling was completed over several prospective areas including East Ridge, East Flats, Egg Hill, Benson, North Jumbo, and Northwest Rim (Figure 9-1).

Several of these areas presented exploration potential with anomalous to significant gold from the grab samples. East Ridge, East Flats and Egg Hill were characterized by grab samples greater than 0.0292 oz/ton (1.00 g/t) gold. The target areas are located to the east of the Oro Belle pit and have not been drill tested. The Benson target hosts analogous rock types to Castle Mountain Project and twelve samples returned assays greater than 0.0292 oz/ton (1.00 g/t) gold. Northwest Rim was characterized as a 4.7 mi (7.5 km) long northeast striking area of quartz-carbonate veins with rock grab samples greater than 0.0146 oz/ton (0.50 g/t) gold.

All samples were assayed for gold by fire assay and multi-element by Inductively Coupled Plasma (ICP) analysis using a four-acid digest. In addition, 138 samples were analysed for whole rock using a lithogeochemical package. Barrett (2016a) carried out a lithogeochemical and petrographic study to differentiate the volcanic, intrusive and volcaniclastic units observed during the mapping and to quantify the degree of hydrothermal alteration. This study and a follow-up investigation of South Domes drill core indicated that that enrichment in antimony, arsenic and bismuth occurs outboard from gold mineralization, but generally at shallower levels (Barrett, 2016b). The associated enrichment in antimony, arsenic and bismuth at relatively shallow levels may be used as an exploration vector pointing to zones of gold enrichment at deeper levels and/or laterally and may potentially be used as an effective tool to predict the occurrence of bonanza grade gold-bearing veins at depth.

Channel sampling was conducted within the East Ridge and Egg Hill target areas. A total of 4,401 ft (1.25 km) of sample was cut using a gas-powered diamond saw. 220 channel samples were taken at 20 ft (6.1 m) intervals. The channel samples were treated in the same manner as drill hole data (i.e. noting a starting point, the total channel length and orientations capturing the trace of the channel).

Rock and channel samples were submitted to ALS Global of Reno or Elko, Nevada for gold assay on a 30 g aliquot by fire assay with an atomic absorption spectroscopy (AAS) finish. Prior to analysis, the sample was weighed, dried (110°C), crushed to greater than 70% passing 2 mm, then split into a 0.250 kg fraction that was pulverized to >85% passing a 75-micron mesh. Whole rock was determined by fusion followed by ICP-AES for major oxides and a lithium-borate fusion followed by an acid digest and ICP-MS finish for trace and rare earth elements.

Equinox has not completed any soil sampling. Viceroy completed extensive soil sampling programs which covered the entire property and predate the anthropogenic disturbance from the open pit operations. A total of 5,383 samples were collected and have results for gold, arsenic, silver mercury and antimony. Soil samples were collected at a nominal 400 ft (122 m) grid spacing.

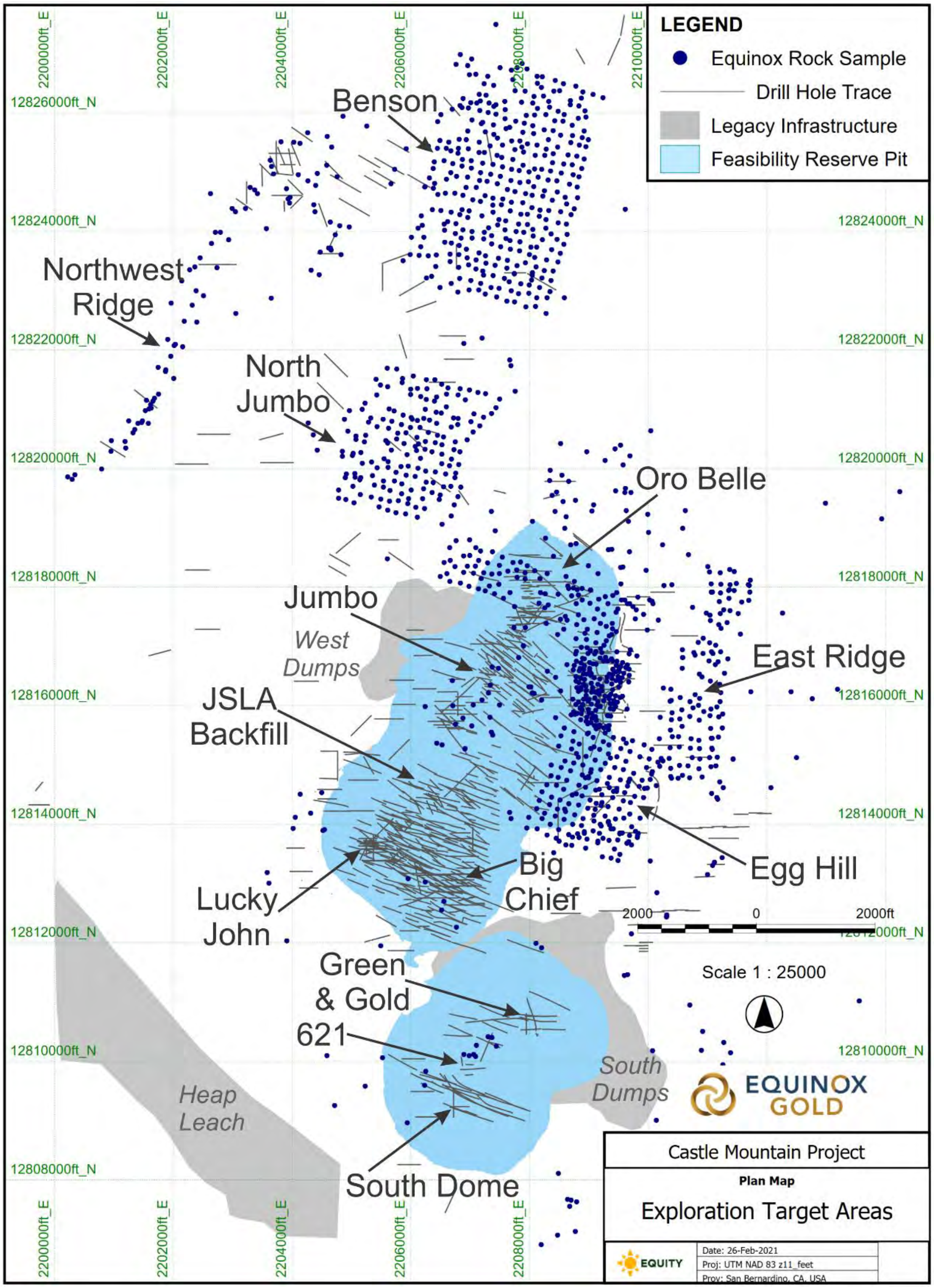


Figure 9-1: Map of Castle Mountain Exploration Targets

9.4 GEOPHYSICS

Equinox completed a Transient Electromagnetic survey (TEM) using instruments from Zonge International of Tucson, Arizona from February 2 to February 9, 2015. The survey design included soundings at 50 locations to evaluate alluvial material and assist in defining the groundwater level southwest of the mine. The TEM survey appears to show a strong resistivity contrast between the highly resistive alluvial material and bedrock that exhibits lower resistivity. Correlating with historic water wells, several inversion sections indicated gradual thickening of alluvial material to the west and a possible geophysical marker defining the top of the water table.

A ground-based gravity survey was completed to assist in determining the depth of alluvial cover. The Phase I survey was conducted from December 19 to December 30, 2014 and totaled 615 stations over a 984 x 984 ft (300 x 300 m) survey grid (Magee, 2014; Wright, 2015a). Within the eastern part of the survey area (closest to the mine), the gravity response is strongly influenced by bedrock geology and coincides where alluvial cover is thinnest. Correlating with drill hole geology, gravity highs strongly suggested the presence of near surface of the Proterozoic basement. A prominent northeast trending gravity low correlated to increasing rhyolite thickness and conformed to the modeled rhyolite-andesite contacts.

A Phase II gravity survey was performed over the deposit area from June 14 to June 16, 2015 to increase the geophysical resolution of the volcanic stratigraphy and mineralization. The survey totaled 779 stations on a 328 x 328 ft (100 x 100 m) grid (Wright, 2015b). The survey interpretation suggested a steeply dipping normal fault, with a west-side down offset, on the eastern side of the survey grid.

10 DRILLING

Extensive drilling has been completed on the Castle Mountain Project by Viceroy and Equinox (Table 10-1, Figure 10-1). Diamond, RC, and rotary drilling methods have been used within the hard rock portions of the deposit with a total of footage of 1,557,140 (474,597 m) within 2,111 holes (Figure 10-2). Recent drilling completed by Equinox accounts for total footage of 372,960 (113,677 m) within 339 drill holes. The drilling executed by Equinox represents 24% of the total hard rock footage drilled on the Project. The Equinox hard rock drilling represents dominantly angled drill holes and diamond drill core.

To support the Feasibility Study additional holes were drilled in 2019 and 2020 for geotechnical and condemnation purposes. Four HQ geotechnical diamond drill core holes were drilled for a total of 4,242 ft (1,293 m). To test for mineralization in infrastructure sites, 26,995 ft (8,228 m) was drilled in 31 RC condemnation holes. Portions of the condemnation drilling were within the waste dumps, but the majority of the holes are within the hard rock.

Equinox drilled 1,685 holes within the JSLA backfill and waste dumps with a total footage of 370,212 (112,835 m) (Table 10-2, Figure 10-3). The backfilled JSLA pit was drilled in 2017 and 2018 by RC and RAB to support the backfill resource estimate. Equinox hard rock drill holes that transect the backfilled JSLA pit were drilled as RC pre-collars followed by diamond drilling within the hard rock. The drill holes within the backfilled JSLA pit are summarised in Table 10-2. All drilling of the backfilled JSLA pit was executed by Equinox.

The legacy drilling completed by Viceroy is included herein, as it is material information to the hard rock portion of the resource estimate. Viceroy drilling was completed entirely within hard rock using conventional rotary, RC and diamond drilling methods for a total footage of 1,184,180 (360,920 m) within 1,712 drill holes.

Table 10-1: Summary of Castle Mountain Hard Rock Drilling by Operator & Drill Type

Operator/Hole Type	Hole Count	Total Footage (ft)	Total Meterage (m)
Equinox	**339**	**372,960**	**113,677**
Core	121	134,929	41,126
RC	195	210,237	64,080
RC-Core	23	27,794	8,471
Viceroy	**1,772**	**1,184,180**	**360,920**
Core	66	44,339	13,514
RC	1,227	881,031	268,525
Rotary	479	258,810	78,881
Total	**2,111**	**1,557,140**	**474,597**

Table 10-2: Summary of Castle Mountain Backfill Drilling by Drill Type

Operator/Hole Type	Hole Count	Total Footage	Total Meterage
Equinox	**1,685**	**370,212**	**112,835**
RAB	1,265	26,842	8,181
RC	278	212,937	64,900
RC-Precollars	29	340	104
Diamond Core	113	130,093	39,650

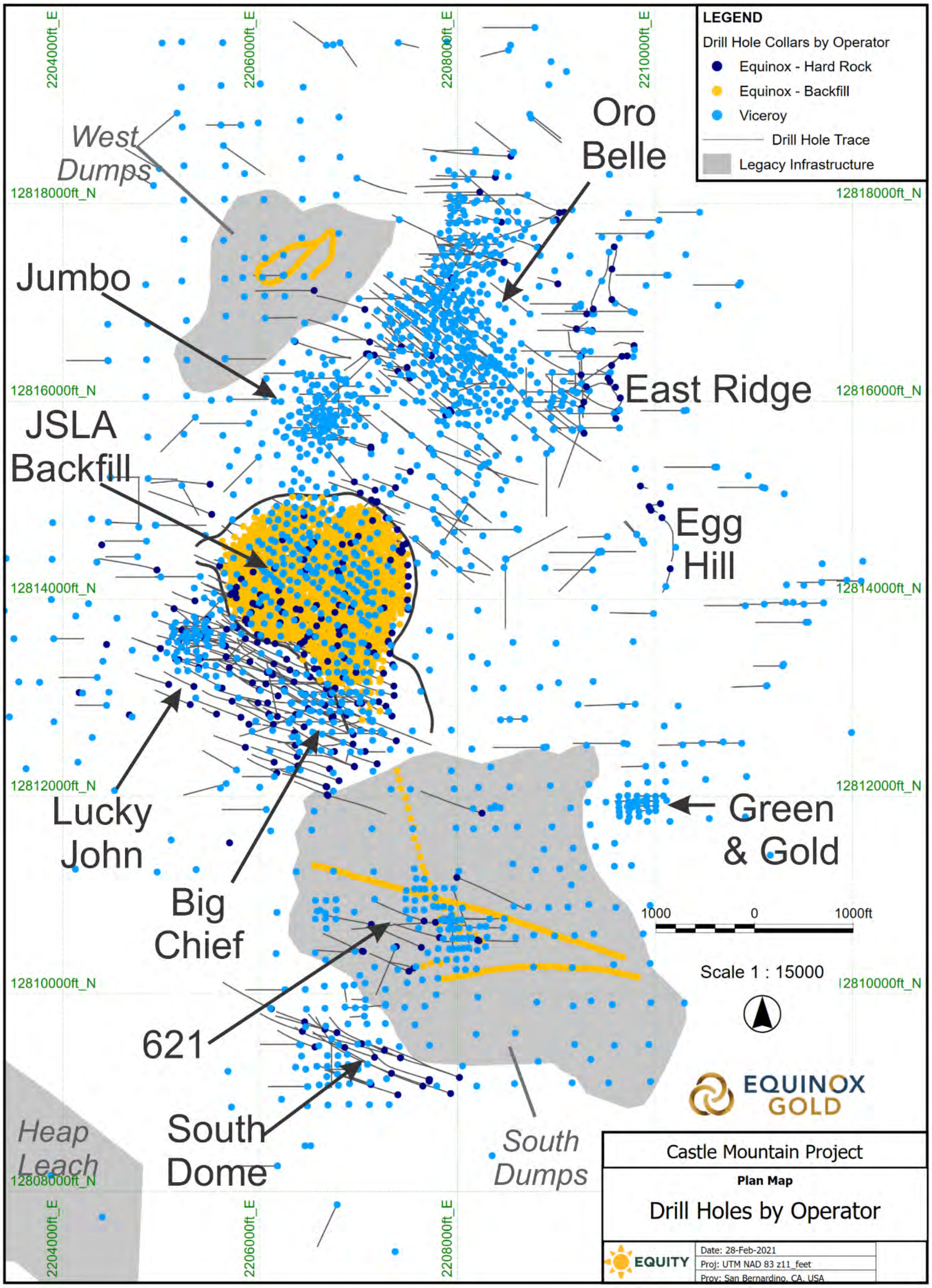


Figure 10-1: Map of Drilling by Operator

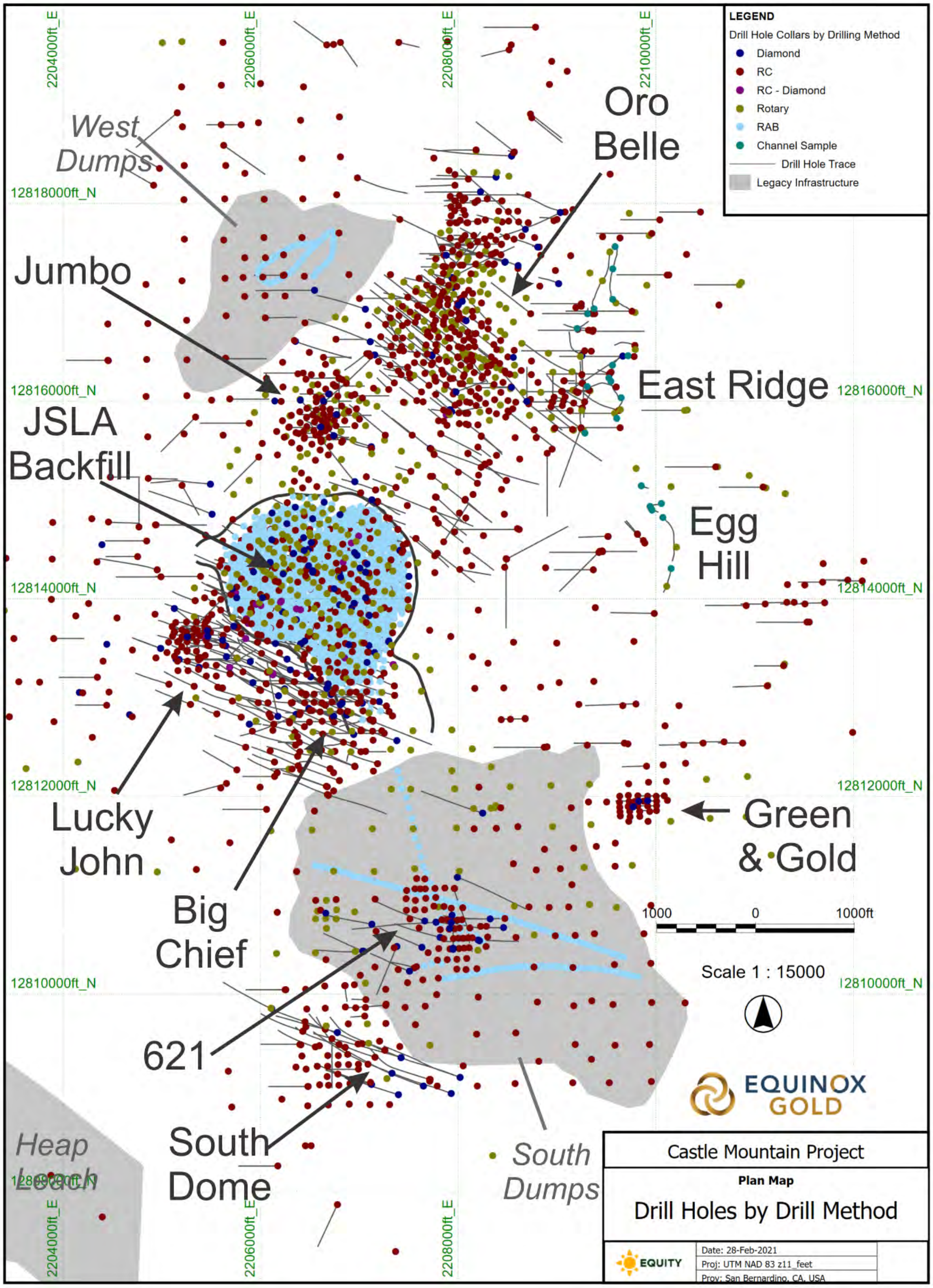


Figure 10-2: Map of Drilling by Drill Hole Type

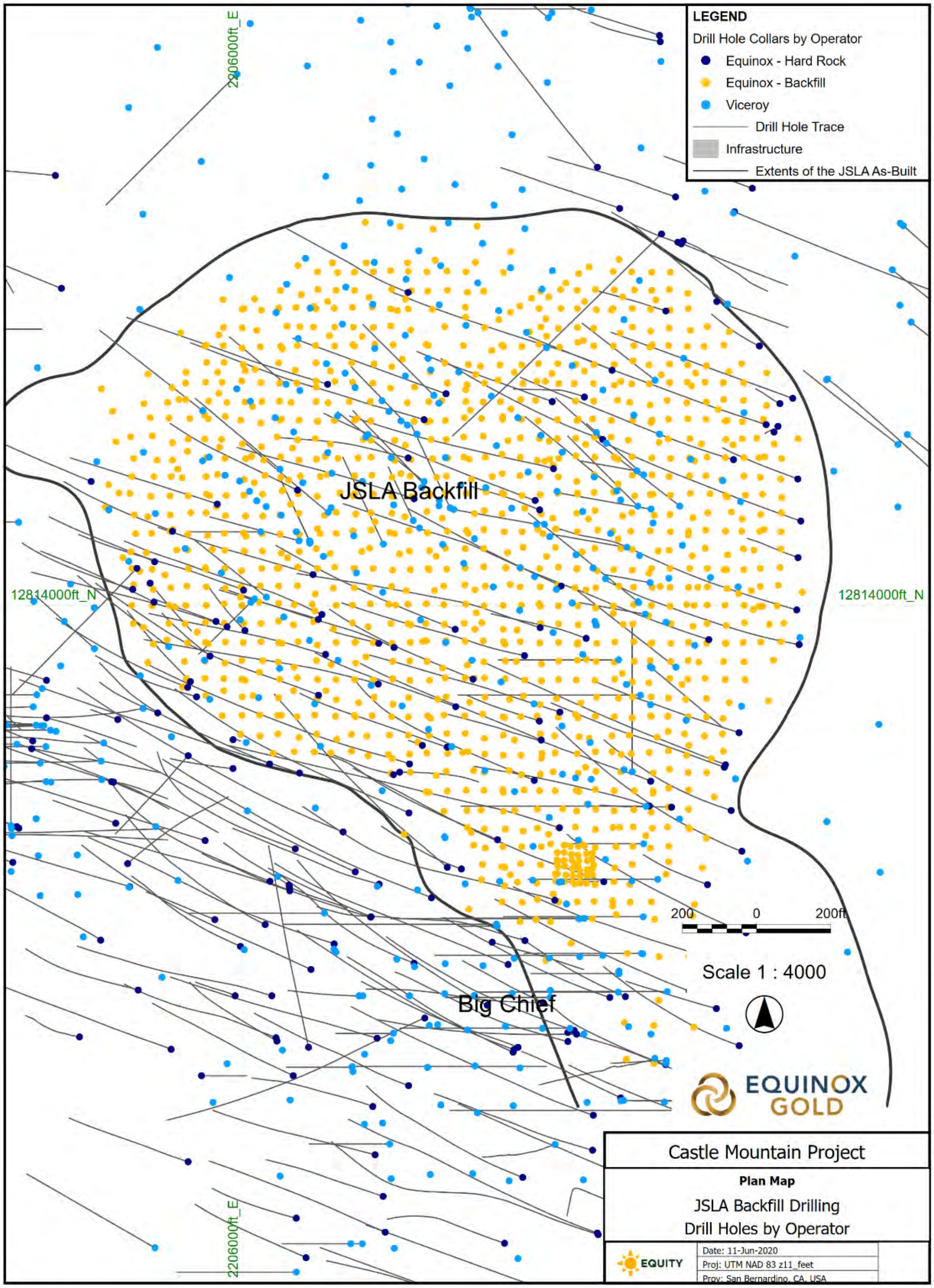


Figure 10-3: Map of JSLA Backfill Drilling by Operator

10.1 VICEROY LEGACY HARD ROCK DRILLING

Between 1984 and 1993, Viceroy completed a total of 1,712 drill holes on the Castle Mountain Project which accounts for drilling footage totaling 1,184,180 (360,920 m). The legacy drilling on the project included a mix of vertical rotary drilling and dominantly vertical RC and diamond drilling. Few inclined holes were drilled.

Blastholes were used to monitor production during historical Viceroy operations (1990 to 1996). Legacy samples digitized by Equinox geologists cover all benches in the Jumbo and Oro Belle pits and a small portion of the benches in JSLA. Production blasthole data has not been verified by the QP and only the de-surveyed bench sample and result are available for review.

The Viceroy drilling data was compiled from digital records and verified by Equinox staff using hardcopy records that are currently stored at the Project's office in Henderson, Nevada. Verification included spot checking and rectification of inconsistent records. Viceroy drill hole collar location data were originally recorded using Viceroy's Castle Mountain Mine Grid projection. All original Viceroy drill hole collar coordinates have been re-projected to UTM NAD 83 UTM zone 11 meters and then converted from meters to feet to match the current methods that Equinox uses to survey drill hole collar locations.

10.2 EQUINOX HARD ROCK DRILLING

10.2.1 Hard Rock Exploration Drilling

Drilling by Equinox has been completed using primarily angled drill holes to delineate targets, advance the geological models and increase the confidence of resource classification. Drilling executed by Equinox was oriented and drilled at lower angles in an attempt to intersect the structures associated with, and controlling, the deposit geometry as well as the local lithologic and stratigraphic controls of economic mineralization (Gray et al., 2016).

10.2.1.1 Equinox Phase 1.1-1.3 Drilling; 2013-2015

In March 2013, Equinox initiated a three-phased exploration and resource definition drilling program on the Project. A total of 70,570 ft (21,510 m) of RC and diamond core drilling was completed over 77 drill holes.

Phase 1.1 comprised 18,092 ft (5,514 m) of diamond core drilling and 6,785 ft (2,068 m) of RC drilling over 30 drill holes. The Phase 1.1 drill program was designed to follow-up on historical drill holes within and proximal to previously mined pits as well as test mineralization in selected exploration targets. The primary goal was to verify and validate the historical drill hole database and to collect data to be used for an initial resource estimate.

Previous exploration focused on northeast-trending structures and margins of rhyolite domes for the principal control on the mineralization. Early in the 2013 drill program this concept was tested but the results highlighted that the intersection of the north-south and northeast-trending structures as a significant control for gold mineralization. Later drill testing was modified to test the intersection of structures.

Phase 1.2 commenced in 2014 and included 33,254 ft (10,136 m) in 41 drill holes. Drilling targeted areas between and under the existing open pits. Mineralization encountered in the Lucky John target area demonstrated exploration potential for strike and depth extensions to known zones and near surface mineralization occurring adjacent to and on trend of the existing mineral resources. Broad intervals of gold mineralization were encountered around the south end of the existing JSLA pit. This phase included four PQ diameter (85.1 mm) diamond holes drilled specifically for metallurgical column tests.

Phase 1.3 program commenced in early 2015 and included 6,549 ft (1,996 m) of RC and 5,889 ft (1,795 m) of HQ core in 10 deep holes to further follow up on the drilling that returned high-grade intervals within the Lucky John target area.

10.2.1.2 Equinox Phase 2.1 2016 Drilling

The Equinox Phase 2.1 RC and diamond drilling program was undertaken from June to October 2016 and consisted of 46 exploration and infill resource drill holes. Total footage for RC and diamond drilling amounted to 39,470 ft (12,030 m) and 26,744 ft (8,151 m), respectively. One 1,000 ft (305 m) hydrogeological test hole was drilled. Phase 2.1 drilling totaled 65,423 ft (19,941 m).

The program targeted the southern part of the current resource area, in the Big Chief and South Domes areas. These targets were historically sparsely drilled and occur adjacent to previously mined areas. They were considered to have good potential for near-term mineral resource expansion as well as possible strike extensions of the Lucky John high-grade gold mineralization encountered in 2014 and 2015 (Gray et al., 2016).

10.2.1.3 Equinox Phase 2.2 2016-2017 Drilling

The Equinox Phase 2.2 drilling began in late October 2016 with the objective of expansion and infill of the existing published measured and indicated resource estimate as defined by the 2015 Technical Report and Updated Mineral Resource Estimate (Gray et al., 2016). Additional drill holes were added to the program to support ongoing metallurgical testwork, to source clay for the leach pad liner, and to test for additional water sources for mine development.

A total of 148 RC and diamond core drill holes were completed comprising 136 resource expansion and infill drill holes, four water well test holes, four PQ metallurgical test holes, and four PQ geotechnical holes to test for clays amenable to use as a leach pad liner. The four geotechnical holes were each completed to a depth of 150 ft (45.7 m) for a total of 600 ft (183 m). The holes were drilled immediately to the west of Big Chief Hill in an area that has historically been excavated for sources of clay on the periphery of strongly altered rhyolite flows or domes.

The total footage drilled was 169,944 ft (51,799 m) including 160,341 ft (48,872 m) of resource and infill drilling; 5,620 ft (1,713 m) of water well test drilling; 3,383 ft (1,031 m) of PQ metallurgical drilling; and 600 ft (183 m) of clay test hole drilling.

10.2.1.4 Equinox Phase 2.3 2017-2018 Drilling

In late 2017, Equinox completed a drilling program aimed at infill drilling at South Domes and exploration drilling in several other areas on the Project. A total of 29,447 ft (8,978 m) in 31 RC and diamond core holes was drilled.

10.2.2 Metallurgical Drilling

Eight PQ metallurgical specific holes were drilled for a total of 7,306 ft (2,227 m) (Figure 10-4). Four holes were drilled for metallurgical column testing for an aggregate 3,923 ft (1,195.6 m), with two holes under the Jumbo pit, one hole into the Lucky John zone, and one hole into the southern margin of JSLA in 2014. In 2018, three holes twinned the diamond drill holes CMM-033, CMM-161 and CMM-212 all within JSLA backfill. A fourth metallurgical hole was pre-collared but the JSLA dump backfill proved too deep to successfully complete the drill hole and was abandoned at 440 ft (134 m). Additional metallurgical tests have utilized sample material from the exploration drilling.

10.2.3 Geotechnical Drill Holes

Four HQ geotechnical diamond drill holes were drilled for a total of 4,242 ft (1,293 m) in November and December 2019, to support a slope stability study to update recommended inter-ramp slope angles and bench design parameters. All four holes were oriented to pierce the Feasibility Reserve pit shell (Figure 10-4).

10.2.4 Condemnation Drill Holes

In April of 2020, an RC condemnation drill program totalling 26,995 ft (8,228 m) was undertaken to determine if there was untested economic potential within areas designated for mine facilities and within the waste dumps. Assay results confirmed there is little to no significant economic potential under the proposed infrastructure sites (Figure 10-4). Several holes were collared on historic dumps and intersected gold mineralization, including up to 55 ft (17 m) of 0.0272 oz/ton (0.93 g/t) in CMM-294 highlighting the potential for mineralization within the legacy waste dumps.

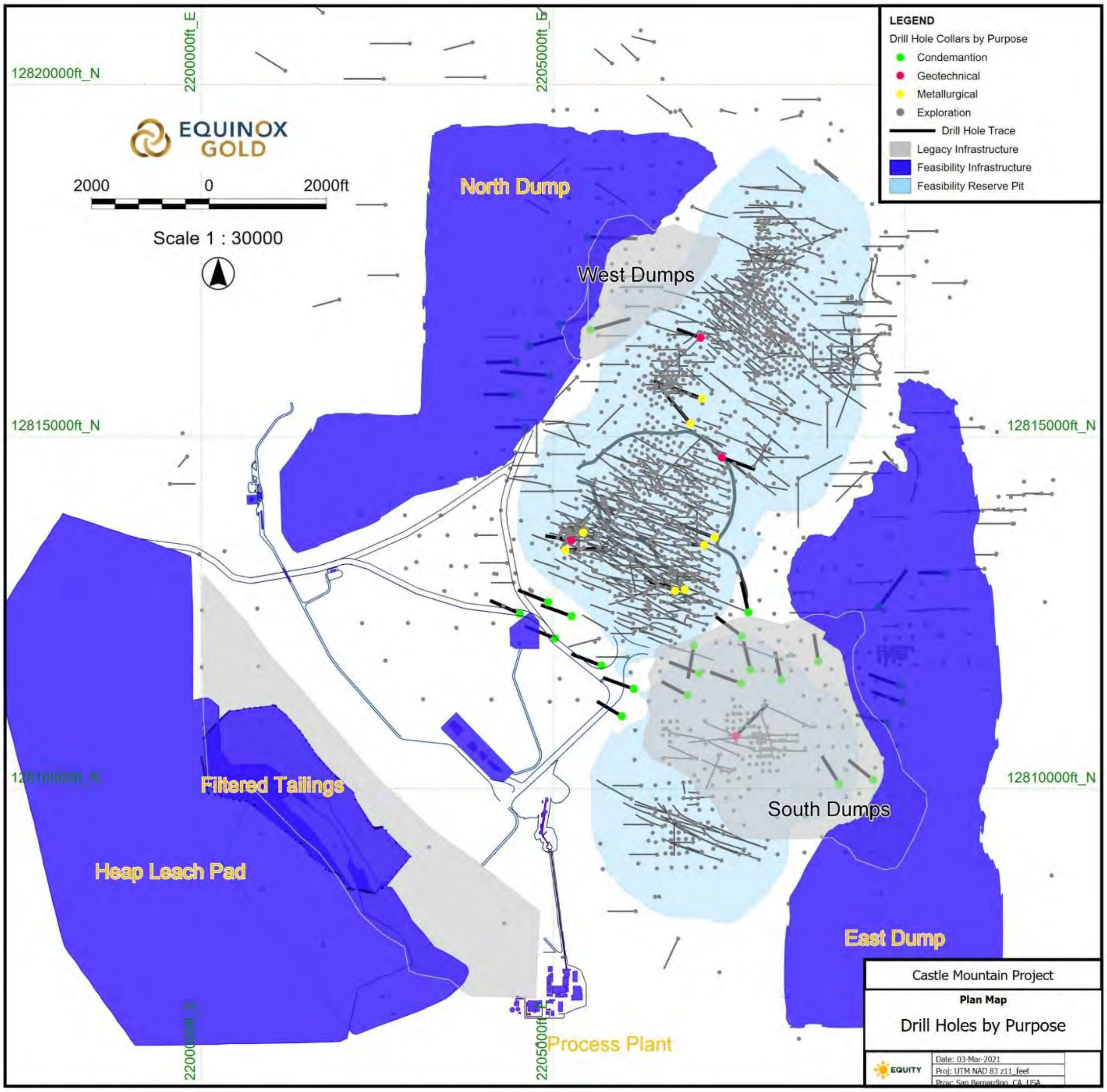


Figure 10-4: Map of Drilling by Purpose

10.3 EQUINOX BACKFILL & WASTE DUMP DRILLING

10.3.1 Equinox 2017 RAB Drilling

In January and February 2017, a RAB drill program was carried out across the JSLA backfill and south waste dumps. The goals of the program were for condemnation purposes and to test for low-grade mineralization. A total of 7,002 ft (2,134 m) from 273 RAB drill holes were drilled, including 242 drill holes within the backfilled JSLA pit were drilled with 100 ft (30 m) spacing and 31 drill holes completed in lines across the south waste dumps. RAB drill holes were drilled to depths between 18 ft (5.5 m) and 30 ft (9.1 m) depending on the stability of the dump material and the amount of material recovered for sampling.

10.3.2 Equinox 2018 RC Drilling

A 52-hole RC program totalling 9,510 ft (2,899 m) within the backfilled JSLA pit was completed over a two-week period in January 2018. Drilling was completed using a Shramm T450 RC drill rig. Termination depth for the drill holes was selected at the 4,300 fasl (1,311 m) level, yielding an average hole depth of approximately 182 ft (55.5 m). In the fall of 2018, 66 additional RC holes were completed for a total of 23,445 ft (7,146 m) with an average hole depth of 355 ft (108 m).

10.3.3 Equinox 2018 RAB Drilling

A RAB drill program consisting of 617 holes totalling 12,340 ft (3,761 m) was drilled in February 2018. The drill program was designed to test the top 20 ft (6.1 m) of the JSLA pit backfill material on a 50 ft (15.2 m) grid-spacing. An infill grid consisting of 32 holes was drilled on 20 ft grid-spacing to evaluate grade control drilling practices. The infill grid was centered on RC hole RC18-1-2.

The RAB program was extended to include portions of the north and south waste dumps, with 46 holes drilled on the north dump as an initial test of ROM material. An additional 107 samples were collected across two lines at the southern end of the south dump, following up on the 2017 RAB program. The extended RAB drilling program was completed in March 2018 for a total of 375 drill holes in 7,500 ft (2,286 m).

10.4 EQUINOX DRILLING METHODS & PROCEDURES

Exploration drilling at the Castle Mountain Project site is managed by trained Equinox personnel using established Project specific procedures. The operations are supported from the site facilities that includes offices, a secured laydown area including a covered, open-air, logging facility and a core storage facility.

The exploration manager and project geologists are responsible for ensuring logging geologists and technicians are aware of, and follow the logging, sampling, and sample shipment procedures. From 2012 to 2017, all logging, sampling, and shipment data were recorded on Excel templates. Starting in 2017, this data was entered directly into a tablet computer using MX Deposit logging software, which also serves as the database and QA/QC monitoring platform. All drill depths and drill runs were recorded in feet.

10.4.1 Drilling Methodology

10.4.1.1 Diamond Core Drilling

Diamond drilling was completed using conventional PQ (3.35 in/85.1 mm) and HQ (2.5 in/63.5 mm) tooling. Where necessary to complete the drill hole to target depth, holes were reduced to NQ (1.875 in/47.6 mm). Diamond tipped face-discharge drill bits were used to increase productivity and recovery. In late January 2017, the tooling was converted to HQ3 drill bits and triple tube tooling using inner core tube splits to further improve recovery and preserve core condition. Metallurgical holes were drilled with conventional PQ tooling using face-discharge diamond core drill bits.

10.4.1.2 Reverse Circulation Drilling

RC drilling was conducted conventional RC air-hammer and tricone drilling. Two drill rigs used center return, face sampling air hammer RC drilling in dry to minimal-water conditions but switched to conventional RC or tricone conditions when water inundated the air-hammer. For deeper holes, a Shramm 685 drill rig using Symmetrix casing and center return, face sampling air hammers was utilized. The Symmetrix RC casing system was used in drill holes where greater than 400 ft (122 m) of backfill was anticipated; the deepest cased hole utilized up to 800 ft (244 m) of Symmetrix casing within the JSLA pit.

10.4.1.3 Reverse Air-Blast Drilling

The RAB drill programs utilized an Atlas-Copco D-65 blast hole drill rig. The RAB drilling used a downhole air-blast button or tricone bit to drill a hole through unconsolidated, dry material. The RAB drill collects the sample direct from the top of the drill hole outside the drill string and then directs the chips to a cyclone where the sample is recovered and bagged. Material with a high clay content used the tricone while more rocky material required the use of the button bit. Drilling foam was added to lubricate the bit and suppress dust. The JSLA RAB drill holes were drilled on an approximate 50 ft by 50 ft grid.

10.4.2 Drilling Procedures

The procedures implemented by Equinox are detailed in Sections 10.4.2.1 through 10.4.2.8 and summarized in Figure 10-5 (Equinox, 2018). All downhole measurements and logging were recorded in feet.

Source: (Equinox, 2018)

Figure 10-5: Equinox Drill Handling and Sampling Workflows

10.4.2.1 Transportation Procedure

Equinox personnel are responsible for transporting the diamond drill core or RC/RAB samples from the drill site to the logging facility. Core or chips samples are secured in pickup trucks and transported along project roads to the logging facility. The drill chips and core are unloaded, organized by hole depth, and examined for drilling errors or irregularities.

10.4.2.2 Location Procedure

Drill collar locations are initially located using hand-held Garmin or Trimble GPS receivers. Fore- and back-sights for drilling azimuth were located using hand-held Brunton compass employing a magnetic declination correction of 11.51º east. An azimuth orientation line was sprayed on the ground with fluorescent orange paint prior to arrival of the drill rig. Inclination was checked by either hand-held inclinometer or by Brunton compass inclinometer.

When drilling is complete, the collars are marked with a cement monument and a labelled stake. Periodically throughout the program, Mineral Exploration Services of Reno, Nevada collected survey quality differential GPS locations for drill collars using a Trimble R2 dual frequency GPS with a horizontal accuracy of 0.79 in (2 cm). All hole locations were collected in UTM NAD83 zone 11 meters and converted to feet.

10.4.2.3 Downhole Surveys

Downhole surveys for the drilling were provided by International Directional Services LLC (IDS) of Chandler, Arizona. All downhole surveys including core and RC holes were conducted using a surface recording gyro (SRG); collected at 50 ft (15.3 m) intervals inside the drill string. The SRG corrects for the magnetic declination (11.51º east) at the time of data collection. Survey measurements were provided on paper and as digital files. No down hole surveys were performed on the RAB drilling due to the shallow depths and vertical drilling orientation.

10.4.2.4 Core Photos

All cores were photographed prior to cutting using a high definition digital camera. All core photographs are labelled with the hole ID number, box number and from-to depth. The digital archive of photographs is maintained at the site office with a back-up stored at the Henderson, Nevada and Vancouver, BC, Canada offices.

10.4.2.5 Geotechnical Logging

Geotechnical logging was performed on all core drilled, including footage drilled, core recovery, RQD, fracture frequency and joint condition. Each category value was determined for each core tube pulled, or block to block. Aluminum tags labeled with the beginning and ending footage of the box and hole number are stapled into the core box to denote sample intervals.

10.4.2.6 Geologic Logging

Geologic logging was carried out on all core and all RC or RAB chips. Reference chip trays are collected, washed, and geologically logged using a binocular microscope. Logging was carried out either at the logging facility or at the Henderson office. Geologic data was digitally recorded into a logging template developed specifically by Equinox geologists for the Project. Principal data fields collected included lithology, mineralogy (iron oxide, manganese oxide, pyrite and gold), alteration (silica, clay, and chlorite). Structural including fractures, faults, veins, and contacts are collected for core.

10.4.2.7 Core, Pulp and Reject Sample Storage

All Equinox drill core is stored at the core storage facilities on the Project site. The pulps and coarse rejects returned by the laboratory are stored in shipping containers at the storage facility for reference and future testwork.

10.4.2.8 Data Adequacy

It is the QP's opinion that the drilling procedures are adequate to support mineral resource estimation. There are no drilling or sampling factors that could materially impact the accuracy and reliability of the results.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The sample preparation, analysis, and quality assurance/quality control (QA/QC) has been reviewed for samples assayed for gold that are used in preparing the Mineral Resource Estimate presented in Section 14. Gold assays from cyanide leach have been routinely assayed and have been reviewed for their use to support the weathering domains. Silver and other elements were analyzed in various drill campaigns; however; assaying methods used are not considered representative.

11.1 VICEROY HARD ROCK SAMPLING METHODS

The following description of Viceroy sampling was modified from the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018) and Temkin (2012). The QA/QC data has not been digitally compiled and was not reviewed by the QP.

11.1.1 Sampling and Security

All legacy drill core sampling was collected systematically on 5 ft (1.5 m) intervals over the entire length of the drill hole. The core was sawn into equal halves utilizing a standard lapidary blade, then measured and marked into 5 ft (1.5 m) intervals. Individual samples were prepared utilizing one-half of the sawn core. Samples were prepared and submitted in batches for shipment to the laboratory.

RC and rotary samples were collected at 5 ft (1.5 m) intervals over the entire length of each drill hole. All dry drill cuttings were split through a Gilson splitter, retaining a sample size of approximately 15 lbs (7 kg). A single sample was collected for each interval, which was sent to the commercial lab for analysis. Wet drill cuttings were split through a revolving wet splitter that was continuously adjusted to collect approximately 15 lbs (7 kg) of material. Individual samples were collected in 5 gal (19 L) buckets lined with an oversized sample bag into which flocculent was added. Samples were left to settle for 20 to 30 minutes, and then were decanted before being secured and laid out to dry. Reference samples were collected in plastic chip containers for logging purposes.

All drill samples were either retrieved directly from the Project by the primary laboratory, Legend of Reno, Nevada, or were shipped directly to Legend via a contract shipping company.

11.1.2 Analyses

Sample preparation at Legend consisted of crushing the entire sample to -10 mesh, splitting out a 200 g (7.05 oz) subsample, and grinding the subsample to greater than 80% passing -150 mesh. Gold and silver were determined by fire assay on a one-assay ton (29.166 g) subsample followed by atomic absorption spectroscopy finish (AAS). Assay values were reported in oz/ton units, and the lower detection limits for gold and silver were 0.001 oz/ton (0.034 g/t) and 0.050 oz/ton (1.714 g/t), respectively. Assays returning gold values greater than 0.100 oz/ton (3.428 g/t) were re-assayed by fire assay on a one-assay ton subsample with a gravimetric finish.

A subset of Viceroy drill hole samples was assayed by Rocky Mountain Geochemical (RMG) in Sparks, Nevada. The sample preparation procedure used by RMG is unknown. Gold and silver were determined by fire assay followed by AAS or gravimetric finish. Assay values were reported in oz/ton units and the lower detection limits for gold and silver were 0.005 and 0.100 oz/ton (0.017 and 3.428 g/t), respectively.

Legend and RMG were not certified nor accredited ISO 17025 laboratories for analysis by fire assay with AAS finish at the time they were performing analytical services for the Project.

At the culmination of mining activities, all samples including core, rotary cuttings, rejects, and pulps were destroyed during the reclamation activities.

11.1.3 Quality Assurance and Quality Control

Automatic intra-laboratory pulp duplicates were performed for any sample that either contained visible gold or had an original assay value of greater than 0.100 oz/ton (3.42 g/t) gold. The standard check assay procedure consisted of three steps. The first was to analyse the original pulp with a full one-assay ton fire assay with a gravimetric finish. The second step was to produce a coarse reject duplicate and perform a full one-assay ton fire assay with a gravimetric finish. The final analysis was for a metallic screen fire assay. This data is not available and is not presented in further detail.

The QA/QC for Viceroy data is not available, and this section was taken from Temkin (2012), which summarized the practices undertaken by Viceroy. Routine duplicate analyses were performed on the rotary, RC and diamond drill holes utilizing the same pulp as that used for the initial analyses. The duplicate analyses were conducted on every tenth sample for approximately 60% of the drill sample population, and every twentieth sample, for approximately 30% of the samples. The remaining approximate 10% of the drill samples had duplicate analyses performed at intervals of every fifth sample or every fifteenth sample.

Assay precision from the pulp duplicates was variable with gold grade, but generally acceptable. Approximately 80% of low-grade < 0.010 oz/ton gold (0.343 g/t) samples reported precision of ±10%. Medium-grade (0.010 to 0.100 oz/ton / 0.343 to 3.428 g/t Au) samples reported precision of ±17% and 90% of the high-grade (> 0.100 oz/ton / 3.428 g/t Au) samples reported precision of ±25%.

Check assay samples submitted to other commercial labs and the Castle Mountain Mine lab did not indicate any systematic bias or accuracy issues with Legend's original assays (Temkin, 2012).

11.1.4 Production Blasthole Samples

There are no details of the Viceroy sampling procedure or the QA/QC. The data was not used in the Mineral Resource Estimate within Section 14, except to cross validate the mined-out portions of the model against the blasthole production data.

11.2 EQUINOX SAMPLING

This section summarizes the Project specific standard operating procedures outlined by Equinox (Equinox, 2018).

11.2.1 Sampling and Security

Once the core has been washed, checked for drilling errors and prepared with meter marks by technicians, Equinox geologists determined sample intervals based on observed changes in lithology, alteration and mineralization features. Sample intervals are nominally 5 ft (1.5 m) in length, but range between 2 ft (0.6 m) and 7 ft (2.1 m). Metal sample tags are stapled on to the left side of the core and folded over onto the corresponding sample break on the core. Tyvek sample tags are filled out in a sample book including sample intervals and QA/QC insertions. One part of the sample tag placed into the box and the tag book is retained for reference if required. The sample intervals and inserted QA/QC samples are recorded in a digital sample cut sheet and used for drill hole sample intervals within the Project's drill hole database and laboratory sample submittal.

For core drilling executed between 2013 and 2016, the core was marked for sampling as described above and sent to the ALS Global (ALS) in Reno or Elko, Nevada where it was sawn in half and bagged for sample preparation and analysis. The remaining half of the sawn core was returned to the site for storage in the core storage facility. Starting in 2017, the drill core was sawn in half, sampled, and bagged by Equinox personnel using an electric saw at the logging facility at the Project site.

RC drill cuttings are collected in labelled sample bags by the drill contractor on continuous 5 ft (1.5 m) intervals from a rotary cyclone splitter. The rotary splitter is set for 50:50 splits to collect half of the material which is bagged, labelled and submitted to the laboratory as the original sample. The remaining portion is discarded except when a field duplicate sample is collected. RC sample weights typically range from 4.4 to 22 lb (2 to 10 kg) and average about 13 lb (6 kg). At the end of a sample run, the sample bag opening is secured and laid out on plastic ground liner to facilitate drying of the sample. After approximately three to seven days, Equinox personnel collect the samples from the field and transport them to the secured laydown yard.

The RAB drill collects the sample directly from the top of the drill hole outside the drill string, and then directs the chips to a cyclone where the sample is recovered and subsequently laid out on a clean tarp to air dry. The material is quartered, and original samples are generated from material in opposite quadrants (one and three). Field duplicates, when inserted, are prepared using material from adjacent quadrants (two and four). Each sample was collected on 18-foot and 30-foot intervals in the 2017 campaign, and each sample was collected on 20-foot intervals in the 2018 campaign. Following collection at the drill, bagged samples are stored in the laydown yard where they are organized to be shipped to the laboratory. Samples from the bags are sufficiently mixed to get a representative sample for the drilling interval.

The individual sample bags are placed in large bins with lids. The laboratory collects palletized core and sample shipment bins directly from the logging facility. Shipments are secured to the commercial laboratory service truck and transported directly to the laboratory facility in Reno or Elko, Nevada. Each shipment is accompanied with a physical sample list and laboratory requisition form that is also submitted to the laboratory via email.

11.2.2 Analyses

Drill samples were assayed at either the ALS in Reno or Elko, Nevada, or the Bureau Veritas (BV), formerly known as Inspectorate, in Sparks or Reno, Nevada. ALS laboratories in the USA have International Standards Organization (ISO) 17025:2005 and ISO 9001:2008 accreditation. BV has ISO 9001 and ISO/IEC 17025 accreditation since 1996. In 2017, the BV accreditation was covered under ISO ISO/IEC 17025:2017 and RG-MINERAL:2017 regulations. Condemnation sampling and umpire pulp check assays were completed at American Assay Laboratories Inc. (AAL) in Sparks, Nevada which is accredited under ISO 17025:2005.

11.2.2.1 ALS

Samples submitted to ALS are prepared by drying and then crushing to 70% passing 0.08 in (2 mm). An 8.82 oz (250 g) sub-sample is taken from the crushed material and pulverized to 85% passing 200 mesh (75 µm) (PREP-31). A 30 g aliquot of pulverized material (pulp) is then assayed for gold and silver by conventional fire assay methods followed by AAS analysis (Au-AA23). Gold assays returning greater than 0.292 oz/ton gold (10 g/t) are re-assayed by fire assay and gravimetric finish on a separate 30 g aliquot (Au-GR21). Starting in 2017, gold assays returning greater than 0.006 oz/ton (0.2 g/t) gold are analyzed for gold cyanide solubility by mixing a 30 g aliquot of pulp with dilute cyanide solution and agitating for one hour and finishing by AAS (Au-AA13).

11.2.2.2 Bureau Veritas

Samples submitted to the BV facility are dried and crushed to 70% passing 0.08 in (2 mm). An 8.82 oz (250 g) sub-sample is taken from the crushed material and pulverized to 85% passing 200 mesh (75 µm) (PRP70). A 1.06 oz (30 g) aliquot of pulverized material (pulp) is then assayed for gold by conventional fire assay methods followed by AAS analysis (FA430). Gold assays returning greater than 0.292 oz/ton (10 g/t) gold are re-assayed by fire assay and gravimetric finish on a separate 30 g aliquot. Gold assays returning greater than 0.006 oz/ton (0.2 g/t) gold were analyzed for gold cyanide solubility by mixing a 30 g aliquot of pulp with dilute cyanide solution and agitating for one hour and finishing by AAS (CN403).

11.2.2.3 American Assay

The condemnation RC chip samples were submitted to AAL for the primary analysis by fire assay of a 50 g aliquot with inductively coupled optical emission spectrometry (ICP-OES) finish (FA-PB50-ICP) and a multi-element geochemical suite of 53 elements by aqua regia digestion and analysis by inductively coupled mass spectrometry (ICP-5AM48). Samples with values over 0.0044 oz/ton (0.15 g/t) gold were analyzed by mixing a 50 g aliquot with hot cyanide solution and analyzing with an ICP-OES finish (CN50). Check assays were completed on pulps using conventional fire assay methods on a 50 g aliquot with gold analysis by ICP-OES. Gold assays returning greater than 0.292 oz/ton (10 g/t) gold were reanalyzed by fire assay with gravimetric finish.

11.2.3 Quality Assurance and Quality Control

Equinox has conducted QA/QC monitoring of gold assays on its drill programs by inserting blanks, certified reference materials (CRM), RC field duplicates, pulp duplicates, and umpire pulp duplicates (Table 11-1). CRMs, blanks and duplicates were inserted into the sample stream consistently; however, insertion rates varied between campaigns. A selection of samples from mineralized intervals were submitted to an umpire laboratory for check assay at the completion of each drill campaign.

Table 11-1: Summary of QA/QC Samples

Sampling Program	Number of Samples	Field Blanks	Certified Reference Materials	Field Duplicate	Pulp Duplicate	Pulp Duplicate External	Total QA/QC Samples
Bedrock	62,606	1,431	2,869	1,985	49	447	6,781
Backfill	7,666						
Total	70,272						

Coarse blanks were sourced locally from coarse crushed construction rock to monitor the sample preparation. Blanks were inserted at a 2% frequency within Equinox drilling. Equinox used ten CRMs sourced from Mineral and Exploration Geochemistry (MEG) of Reno, Nevada to monitor the accuracy of a range of expected gold values. Table 11-2 summarizes the CRM samples submitted by Equinox which were inserted at a 4% frequency rate.

Table 11-2: Summary of CRM Samples

	CRM Expected Values			Original CRM Insertion			
CRM	Expected Average Gold (oz/ton)	Expected Average Gold (g/t)	Standard Deviation	Number of Samples	Average Gold (oz/ton)	Average Gold (g/t)	Average Z-Score
Total				**2,869**			
Au.10.03	0.0018	0.06	0.006	678	0.0018	0.06	0.002
Au.12.11	0.0438	1.50	0.081	468	0.0437	1.497	-0.032
Au.12.32	0.0181	0.62	0.017	682	0.0181	0.622	0.14
Au.17.05	0.0015	0.05	0.004	19	0.0016	0.054	1.066
NBM-2a	0.0002	0.007	0.002	20	0.0002	0.006	-0.425
NBM-4a	0.0022	0.075	0.007	23	0.0024	0.081	0.858
NBM-5b	0.0481	1.65	0.225	66	0.0493	1.689	0.172
S105006X	0.1313	4.50	0.099	647	0.1312	4.497	-0.028
S107007X	0.0438	1.50	0.068	234	0.0456	1.563	0.92
S107009X	0.1371	4.70	0.194	32	0.1438	4.93	1.184

The QA/QC analyses for gold are reviewed on a batch by batch basis by the Equinox geologists. A series of protocols are followed to define QA/QC failures and determine the type of follow up action required. If a control sample result falls outside acceptable limits, the assay laboratory is instructed to re-assay the batch of samples or a selection of samples around the QA/QC sample. If the re-analysis passes the criteria, then it replaces the results from the original certificate within the Project database.

Field duplicates were collected with a 3% frequency rate for all Equinox exploration samples to monitor the sample variability. RC field duplicates were collected from the second half of the sample created by the cyclone rotary splitter at the drill rig. RAB field duplicates were collected from the two unsampled quadrants adjacent to the quadrants used for the original sample.

The RC condemnation drilling had a total sample submission of 6,634 original and QA/QC samples including 418 blanks, 276 CRMs, and 540 field duplicates. The insertion rates are within industry standard and respectively are 8%, 5% and 10%. All the CRM performed with ±3 standard deviation criteria except one which was re-analyzed with surrounding samples making the effective failure rate nil. The blanks consisted of 361 coarse crush and 57 certified silica blanks which all performed below warning level. Analysis of field duplicates was done by calculating the coefficient of variance (CV) for each sample, which averages 18% for all sample pairs. The QA/QC results from this sampling are adequate; however, the results were not available for inclusion within the Mineral Resource in Section 14 and as such are not included the summary of sampling in Table 11-1 and Table 11-2.

11.3 BULK DENSITY

Bulk density samples were measured for 647 samples from multiple drill holes to provide measurements from a variety of rock types (Figure 11-1). Samples were selected at the logging facility by Equinox geologists. 4-6 in (10-15 cm) long core samples were selected, marked with markers and tags and submitted in the core box to ALS for wax immersion bulk density measurements. Measurements were conducted on split HQ or PQ size core using the water immersion method after coating with paraffin wax (ALS method OA-GRA08a). The bulk density measurements were converted to imperial units as a tonnage factor.

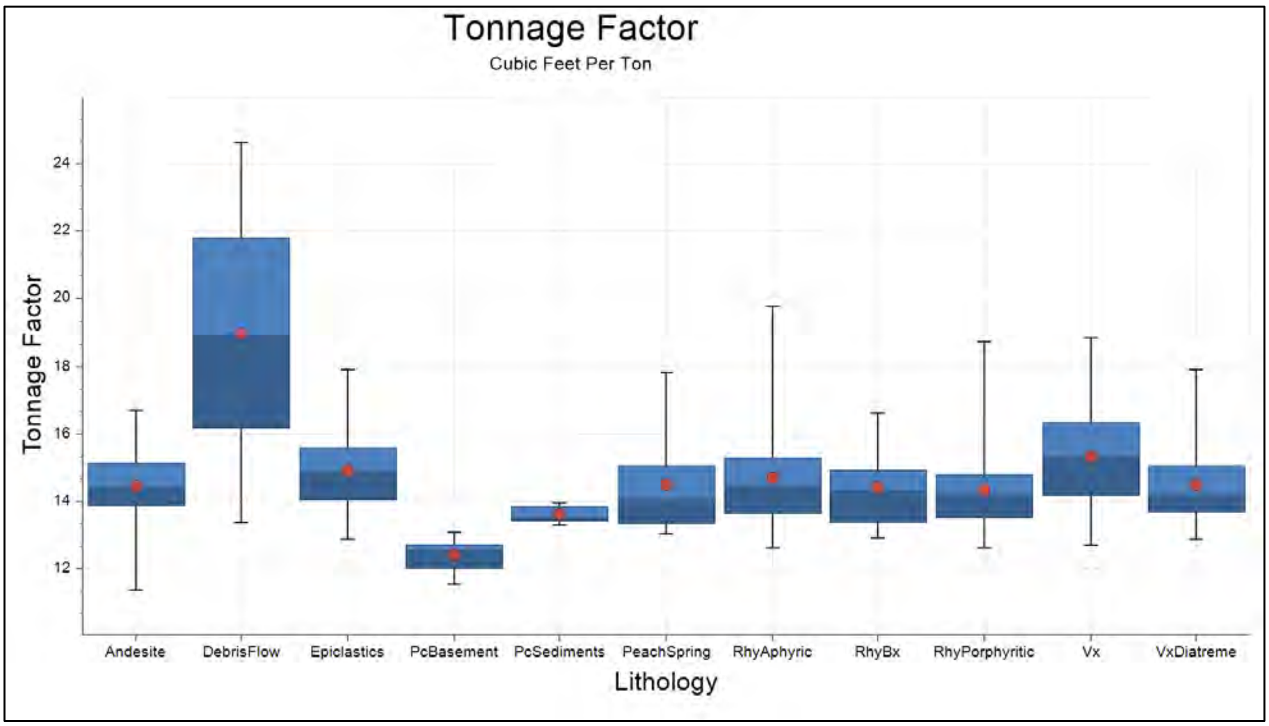


Figure 11-1: Box and Whisker Plot of the Bulk Density by Modelled Lithology for the Castle Mountain Project

11.4 DATABASES

Beginning in 2018, the Project's drilling data was stored in an MX Deposit database hosted on a cloud platform with secure socket layer encryption. Prior to 2018, the data was managed in an Access database. Two databases were provided by Equinox as individual CSV files. All drill hole data received was provided in imperial units, with the exception of drill hole collar locations which were in metric units (metres). Collar locations were all converted from metres to feet in a manner that is consistent with the Castle Mountain local grid which is NAD83 UTM zone 11 North feet (imperial units). The values are converted from meters to feet by dividing the meter values by 0.3048, with four significant figures. Original Viceroy drilling incorporated into the drill hole database had original coordinates recorded in Viceroy's Castle Mountain imperial mine grid. The Viceroy drill hole collar coordinates were subsequently converted from the Castle Mountain Mine grid to metric UTM NAD83 zone 11 projection and then to imperial units. Independent hard rock backfill and production blasthole datasets were provided on delivery dates shown in Table 11-3.

Table 11-3: Resource Database Delivery

Resource Dataset	Delivery (Month/Day/Year)	Database	Export Type
Hard Rock	06/12/2019	MX Deposit	comma separated values files
Backfill	10/24/2019	MX Deposit	comma separated values files
Production Blasthole	08/20/2020	Excel file	comma separated values files

In the assay database compiled by Equinox, the assay results are each ranked by the preferred assay method of most to least representative using a python script. The methods are prioritized as follows: gravimetric finish fire assay followed by AAS finish fire assay. In cases where there is an original result and pulp repeat result, the repeat result is used in the preferred ranking.

11.5 DATA ADEQUACY

It is the QP's opinion that the sample preparation, security, and analytical procedures are adequate to support mineral resource estimation.

12 DATA VERIFICATION

12.1 RESOURCE DATA VERIFICATION

Two drill databases were received from Equinox representing hard rock and backfill drilling. Collectively the two databases are named the “Feasibility databases” in this chapter. The Feasibility databases were received as a series of CSV files. Each CSV file was imported into Micromine™ 3D software for validation and use. Spatial 2D and 3D files were provided in DXF format and as ArcGIS geodatabases. The PFS database was provided for validation and comparison purposes.

The following steps have been taken to verify the databases, including:

- Validation of the drill hole database using Micromine™ software drill database validation tools:
  - Follow up action to rectify sample/geological intervals beyond end of hole
  - Confirming translation of historical grid coordinates to current local grid coordinates
- Independently reviewing and plotting QA/QC results
- Verifying of bulk density determinations
- Comparing twinned drill hole results
- Selecting and comparing pulp check analyses

12.2 DRILL HOLE LOCATION DATA

The Feasibility databases contained information recording collar coordinates and downhole survey information. All drill hole coordinate data is located within the extents of the tenures described in Section 4.2.1.

12.2.1 Drill Hole Collar Location Verification

The surveyed collar locations for the Feasibility databases correlate well with the digital elevation model from the 2018 LiDAR survey, except where Viceroy holes in the JSLA pit have collar elevations representing the pre-mining topography.

The legacy Viceroy collar coordinate spread sheet was used to confirm that coordinate system translation was accurately completed. A point file was created using the legacy Viceroy mine grid surveyed drill hole locations which were re-projected by QGIS software to NAD 83 UTM zone 11 feet using the custom projections. These locations were compared with the locations from the Feasibility databases and found to be identical.

12.2.2 Drill Hole Downhole Survey Verification

Downhole survey data was reviewed for spurious readings determined by discrepancies greater than one degree per 30 ft for azimuth readings. Some discrepancies that were identified between the PFS database and the supplied datasets are explained by ongoing work undertaken by Equinox since 2018 to address and improve database integrity.

The database included 1,746 holes with no downhole surveys. The majority of the unsurveyed holes were drilled vertically where the deviation has a minimal effect on the location of the drill hole. There are 429 unsurveyed angled drill holes that have been projected straight from the collar orientation which represent 20% of the hard rock holes and 17% of total footage. A deviation model was created for the angled RC drilling to test if a correction was required. Of the 47 surveyed angled holes, 41 steepen down hole at an average rate of -0.015°/ft. As such, the deviation model was applied to unsurveyed holes and implicit models were run using the deviation model data. Resulting changes did not affect unmined portions of the hard rock model. The unsurveyed drill hole data was therefore deemed acceptable.

12.3 RESOURCE ASSAY RESULT VERIFICATION

The following checks were completed for all drill holes:

- Sample intervals exceeding the total hole depth.
- Verification of the ranking of fire assay methods against the result in the hard rock gold "Au-FA&GRAV PPM" column and backfill the "Au ppm final" column, which are used in the resource estimate.
- Compilation and charting of QA/QC data to review assay result accuracy and reproducibility.
- Umpire pulp duplicate sample selection and comparison.
- Verifying conversion factors used to convert legacy assay results originally reported in ounces per short ton to grams per tonne.

No critical deficiencies were identified from these checks and several minor errors were corrected within the Feasibility databases. Core recovery for the Project averaged 90%. There is no relationship between core recovery and gold grade.

12.3.1 Performance of the Equinox Hard Rock & Backfill QA/QC Samples

The Equinox QA/QC program results for the hard rock and backfill feasibility databases were evaluated together. A Z-score for Certified Reference Materials (CRM) was calculated, and a "failure" criterion was defined as any CRM analysis returning a Z-score of >3 or <-3. Several failures were identified due to mishandling errors where the CRM types were likely swapped at the time of insertion. Table 12-1 provides CRM performance summaries with and without mishandling errors.

A total of 2,869 CRM samples, representing an insertion rate of 4%, were included within the Equinox sample stream. The insertion rate and expected CRM grades are appropriate for the deposit type and Project stage. A first pass calculation of Z-scores showed a failure rate of 3.6% (N = 102). Omitting samples with obvious mishandling or labelling errors resulted in reduction of the overall failure rate to 1.6% (Table 12-1). Plotting of the CRM results showed a degree of systematic bias depending on the CRM; however, the average values were acceptable overall (Figure 12-1). The "Au" series CRM showed a weak low bias whereas the "S" series CRM showed a weak to moderate positive bias depending on the specific CRM.

Table 12-1: CRM Performance for the Equinox Samples

CRM	Expected Average Gold (g/t)	Standard Deviation	Original CRM Insertion: Number of Samples	Original CRM Insertion: Average Gold (g/t)	Original CRM Insertion: Average Z-Score	Mishandled CRM Removed: Number of Samples	Mishandled CRM Removed: Average Gold (g/t)	Mishandled CRM Removed: Average Z-Score	Number of Failures
Total			**2,869**			**2,814**			**47**
Au.10.03	0.06	0.006	678	0.06	0.002	674	0.057	-0.542	0
Au.12.11	1.5	0.081	468	1.497	-0.032	465	1.494	-0.073	3
Au.12.32	0.62	0.017	682	0.622	0.14	664	0.611	-0.512	24
Au.17.05	0.05	0.004	19	0.054	1.066	19	0.054	1.066	2
NBM-2a	0.007	0.002	20	0.006	-0.425	20	0.006	-0.425	0
NBM-4a	0.075	0.007	23	0.081	0.858	23	0.081	0.858	0
NBM-5b	1.65	0.225	66	1.689	0.172	66	1.689	0.172	0
S105006X	4.5	0.099	647	4.497	-0.028	620	4.534	0.346	15
S107007X	1.5	0.068	234	1.563	0.92	231	1.577	1.131	3
S107009X	4.7	0.194	32	4.93	1.184	32	4.93	1.184	0

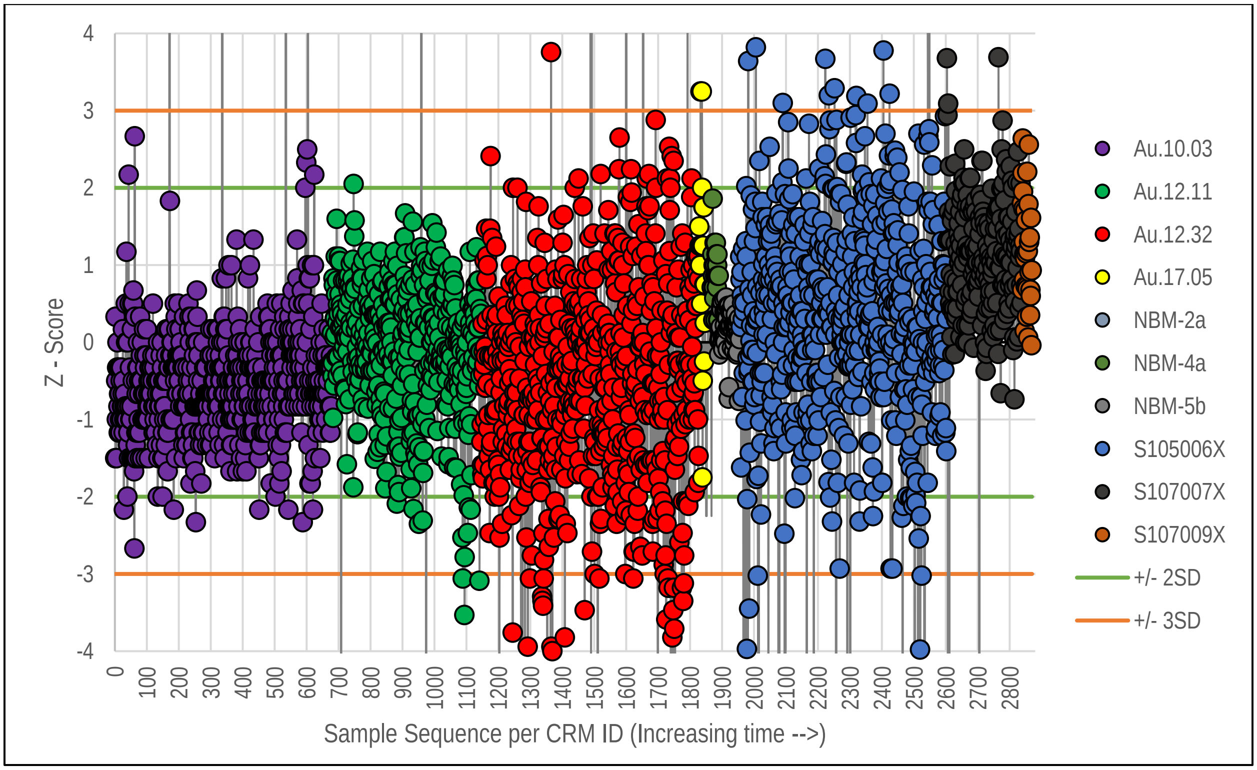


Figure 12-1: Shewart Chart of CRM Performance by CRM type

Blanks were reviewed for carry-over exceeding ten times the detection limit, equal to 0.0015 oz/ton (0.05 g/t) gold. There were 1,432 blank samples which represented a 2% insertion rate for Equinox drilling. While the expected values for the blank material were appropriate, the insertion rate was low compared to current industry best practices. Ten samples (<1%) exceeded a threshold of ten times the detection limit that was used to indicate contamination (Figure 12-2). No significant carryover contamination was observed in the blank results.

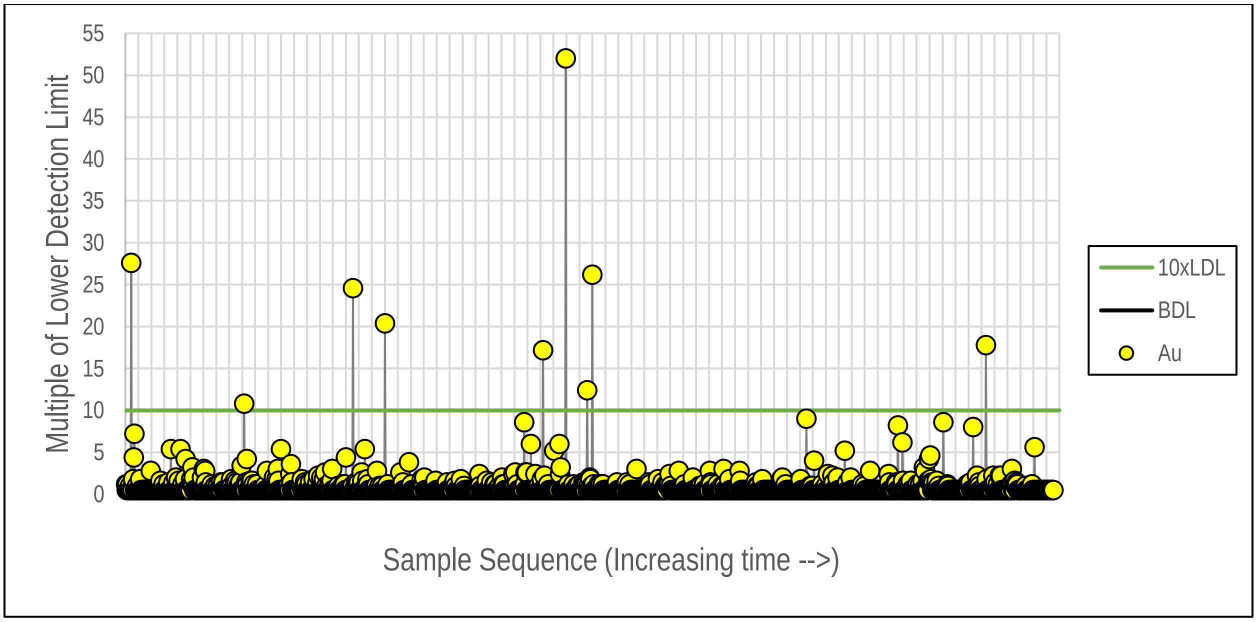


Figure 12-2: Blank Performance for the Project

The Project's quality control data included 2,481 duplicate sample analyses, including 1,972 field (RC/RAB) and 492 pulp duplicate samples (Table 12-2). Duplicates were paired using the duplicate's parent sample ID which was recorded with the duplicate sample. The pulp duplicate population was small and consisted primarily of laboratory pulp duplicates (N=49) that had not been reported due to insufficient population size and umpire laboratory duplicates (N=443). The duplicates were implemented at a 3% insertion rate for field duplicates and a 0.6% insertion rate for the umpire duplicates within Equinox drill programs. The field duplicate insertion rate was adequate for the Project stage. The intra-laboratory and umpire pulp duplicate insertion rate was low relative to the sample population compared to industry standard best practices.

The relative standard deviation (RSD) and coefficient of variation (CV) were calculated for each pair. The RSD was 15% and 12% for field and pulp duplicates, respectively which are within acceptable range. The field duplicates had an acceptable level of variance and subtle bias towards the duplicate results as shown in the Reduction to Major Axis (RMA) plot in Figure 12-3.

Table 12-2: Duplicate Values and Statistics

Duplicate Type	Number of Samples	Primary Sample - Average Gold (g/t)	Duplicate Sample - Average Gold (g/t)	CV (%)	RSD (%)
Field (RC)	1972	0.184	0.174	33	15
Pulp (Umpire)	443	0.626	0.607	30	12

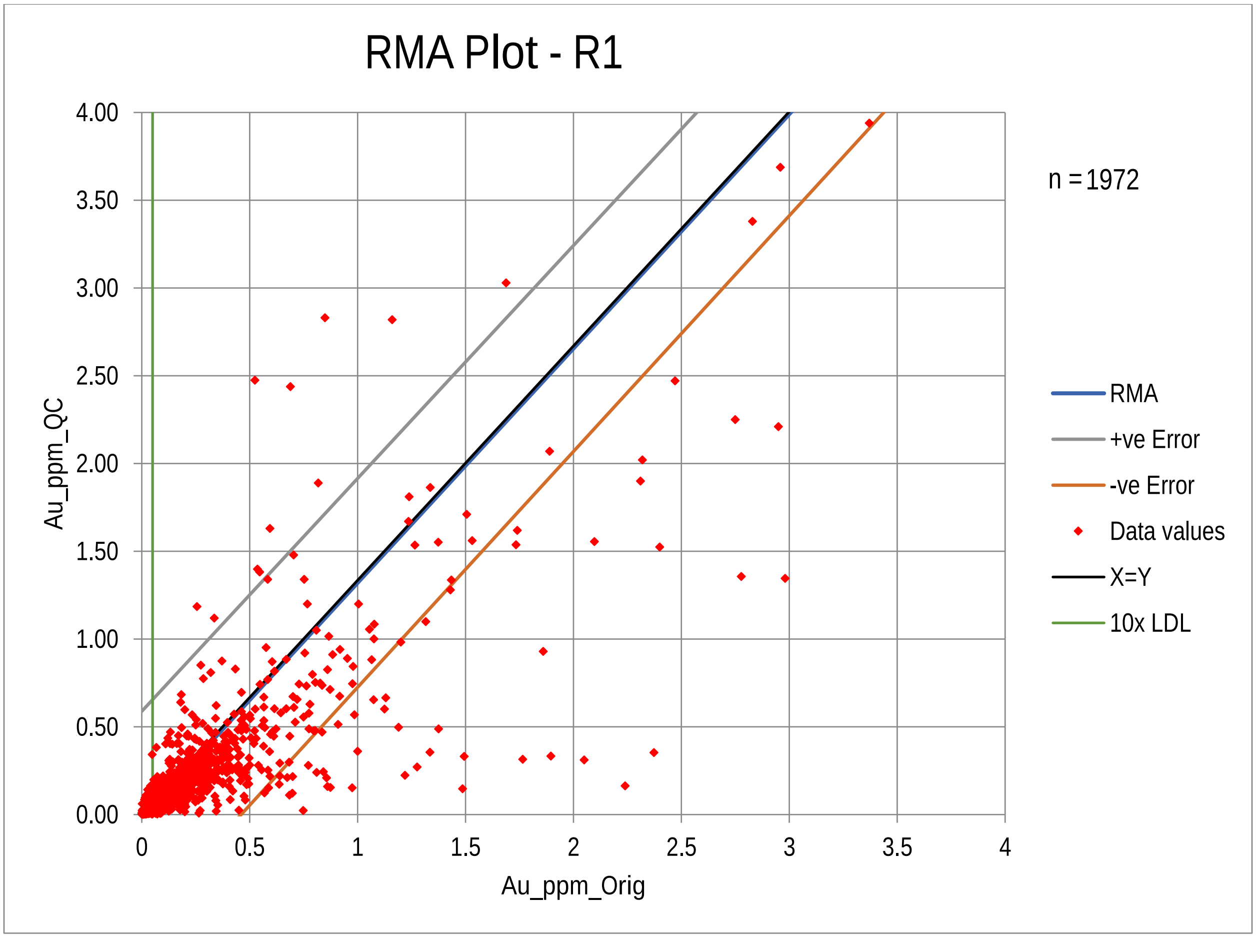


Figure 12-3: RMA Plot of the RC and RAB Field Duplicates

A total of 302 pulp duplicates were selected by Equinox to be analyzed at the umpire laboratory. An additional 141 pulps were selected from holes where the original assays were informing medium and high-grade resource domains. The umpire pulps were pulled from the storage facility and submitted to AAL for fire assay as described in Section 11.2.2.3. Pulp checks are used to quantify the reproducibility of results and to determine if there is any laboratory specific bias. All 443 umpire pulp duplicates are presented in Figure 12-4 which show the RMA plot for the original and umpire results. The comparison demonstrates that the results are reproducible and have a 3% bias toward the original results.

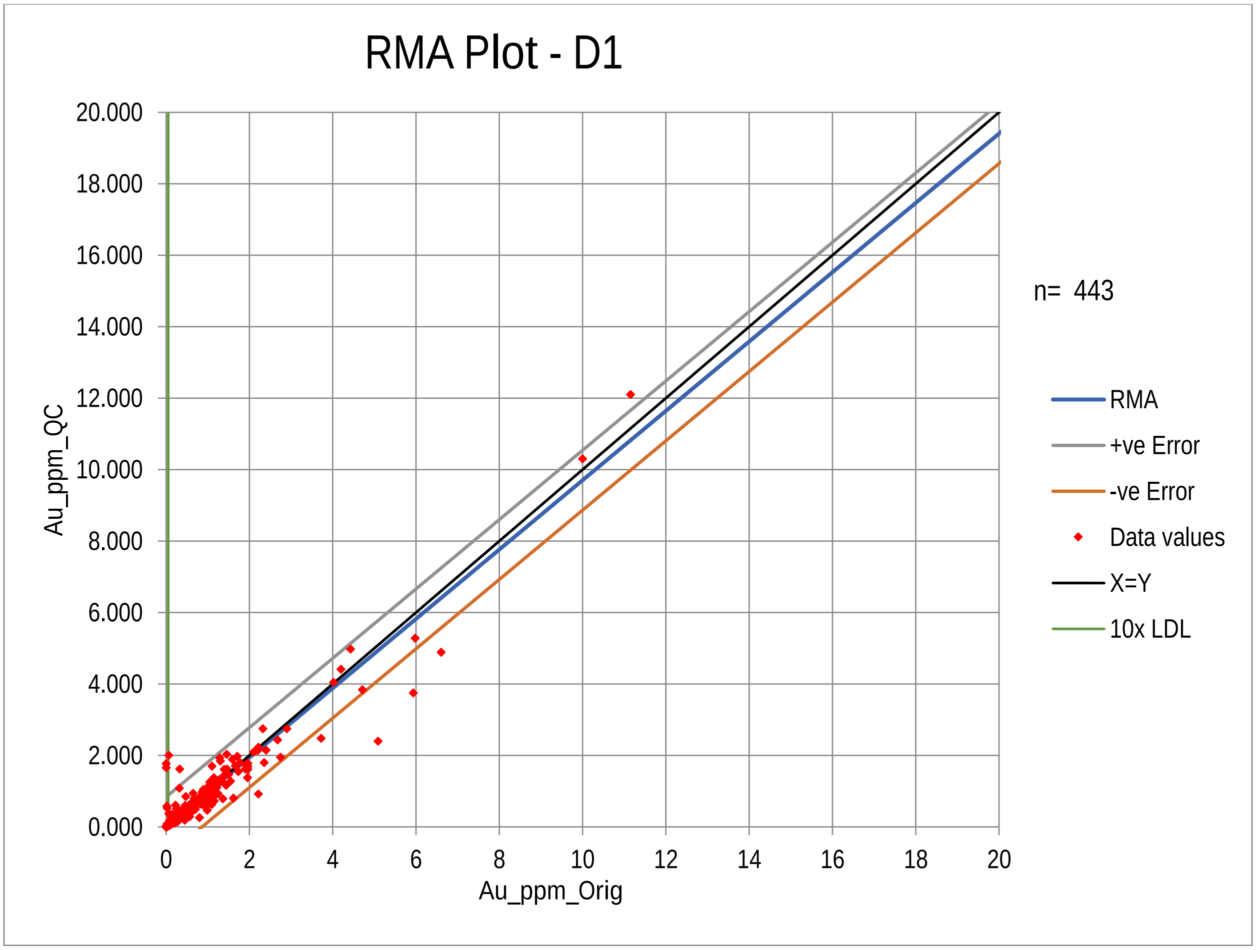


Figure 12-4: RMA Scatter Plot of Umpire Pulp Duplicates

The sample populations were sufficient to demonstrate that the quality of the assay results is acceptable. CRM results demonstrated that the gold assay values were reasonably accurate, and the failure rate was acceptable after mishandling errors were excluded. Results from the insertion of blanks showed that there were no systematic issues with carryover during sample preparation. The results from the duplicate analysis demonstrated reasonable reproducibility with no consistent, material grade bias between the original and umpire assay laboratories.

12.4 VICEROY SAMPLE RESULTS

The Viceroy QA/QC data was not available and thus could not reviewed. Additionally, the pulps and core samples were destroyed during reclamation efforts and as such no further verification is possible. With no other data available for the Viceroy assay verification within the Feasibility databases, the author used a variety of statistical and spatial comparisons to confirm that the results were digitized consistently using the PFS database (Scott et al., 2018) and legacy spreadsheets. Hardcopy Viceroy assay certificates were observed during the site visit and have been verified by the QP and checked in detail within previous technical reports (Cox et al., 2014 and Pressacco, 2013). The Viceroy assay data represented 80% of the sample population and was concentrated in the mined-out portions of Oro Belle, Jumbo and JSLA. To assess the degree of risk to the Mineral Resource estimation due to reliance on Viceroy samples, and the lack of associated legacy QA/QC data a series of checks were completed, including:

- Twin Hole Analysis,
- Assessment of downhole contamination, and
- Sample pair analysis.

12.4.1 Twin Hole Analysis

There are ten twin drill hole pairs that were identified using the criteria that the holes were oriented in the same orientation and had collars spaced no farther than 30 ft (9.1 m) apart. The holes were all drilled by Viceroy using rotary, RC or diamond methods to confirm an original rotary or RC hole. Only one twin pair representing a diamond hole drilled by Equinox and an RC hole drilled by Viceroy was present. A summary of the twin drill holes is provided in Table 12-3.

Table 12-3: Comparison of Twin Drilling

Hole ID	Type	Operator	Twin Hole ID	Twin Type	Twin Operator	Average Pair Distance (ft)	Average Pair Distance (m)	Quality of Twin	Comment
V0162	Rotary	Viceroy	V0001	Rotary	Viceroy	19.5	5.9	reasonable	
V0286	Rotary	Viceroy	V0047A	RC	Viceroy	2	0.6	reasonable	
V0023	Rotary	Viceroy	V0165	Rotary	Viceroy	10	3	reasonable	
V0170	Rotary	Viceroy	V0176	Rotary	Viceroy	20	6.1	reasonable	
V2093	RC	Viceroy	V0610	RC	Viceroy	23	7	reasonable	
CMM-030C	DDH	Equinox	V1135	RC	Viceroy	24.6	7.5	reasonable	
V2146	RC	Viceroy	V2145	RC	Viceroy	10	3	Very poor	Reject V2145
V0511	RC	Viceroy	VDDH02	DDH	Viceroy	10.5	3.2	Poor	Reject V0511
V0156	Rotary	Viceroy	VDDH10	DDH	Viceroy	22	6.7	Poor	Reject V0156
V0718	RC	Viceroy	VDDH35	DDH	Viceroy	9	2.7	Poor	Reject V0718

The twin pairs were compared for logged lithology, individual assays, and 20 ft (6.1 m) bench composite comparisons downhole. Overall, the logged lithological data was similar. The bench composites were compared downhole and statistically. The average RSD for the 20 ft bench (6.1 m) composites was 43% which compares reasonably for twin drilling. Rotary and RC twins or RC-RC twins had reasonable average comparisons. Diamond twins to original RC or rotary drilling generally had a poor comparison, indicating that there was possible downhole contamination during drilling. The four diamond twins were retained within the dataset and the original rotary or RC holes were excluded from the Feasibility database to prevent conflicting information during the estimation; the excluded holes were V2145, V0511, V0156, and V0718. Examples of the downhole comparison plots are shown in Figure 12-5 where independent drill holes are represented by blue and red lines, and sample pair distance is represented by the black line.

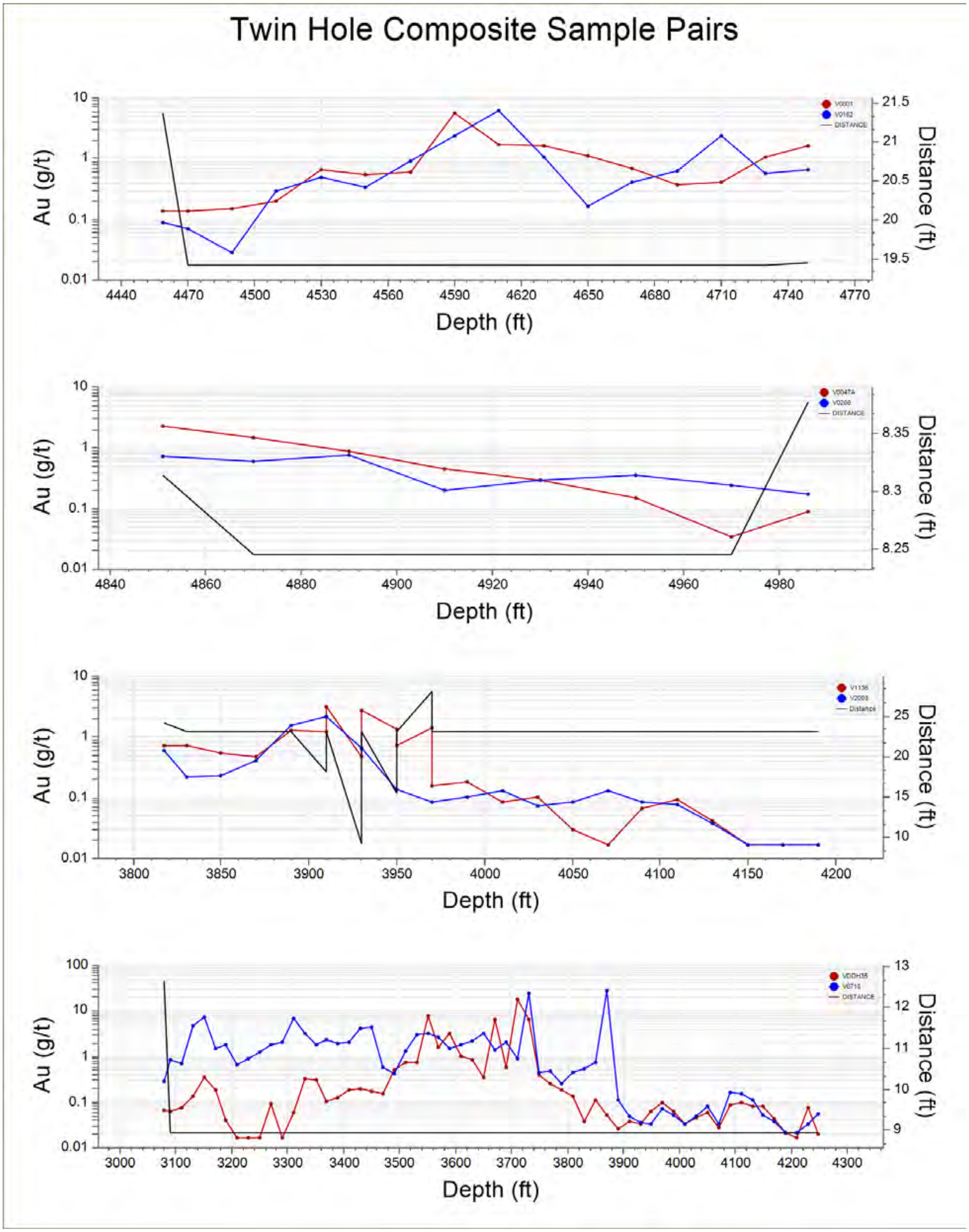


Figure 12-5: Downhole Comparison of 20 ft Bench Composites

12.4.2 Downhole Contamination of RC Samples

To determine the potential for downhole contamination of RC samples, five drill hole pairs representing a diamond drill hole and an RC drill hole were examined. Most of the drill holes included in this analysis occurred within the mined-out portions of the deposit. Figure 12-6 to Figure 12-8 show a comparison of the downhole grade with each hole represented by blue and red lines, and distance between samples represented by a black line. Where there is no line indicating sample pair distance, samples were within 10 ft (3.05 m).

The results of the analysis showed some potential for downhole contamination on a bench scale, in addition to high grade values likely to occur more frequently in the RC drill holes compared to diamond drill holes. This corroborated the DDH-RC sample pair data found in Section 12.4.3.

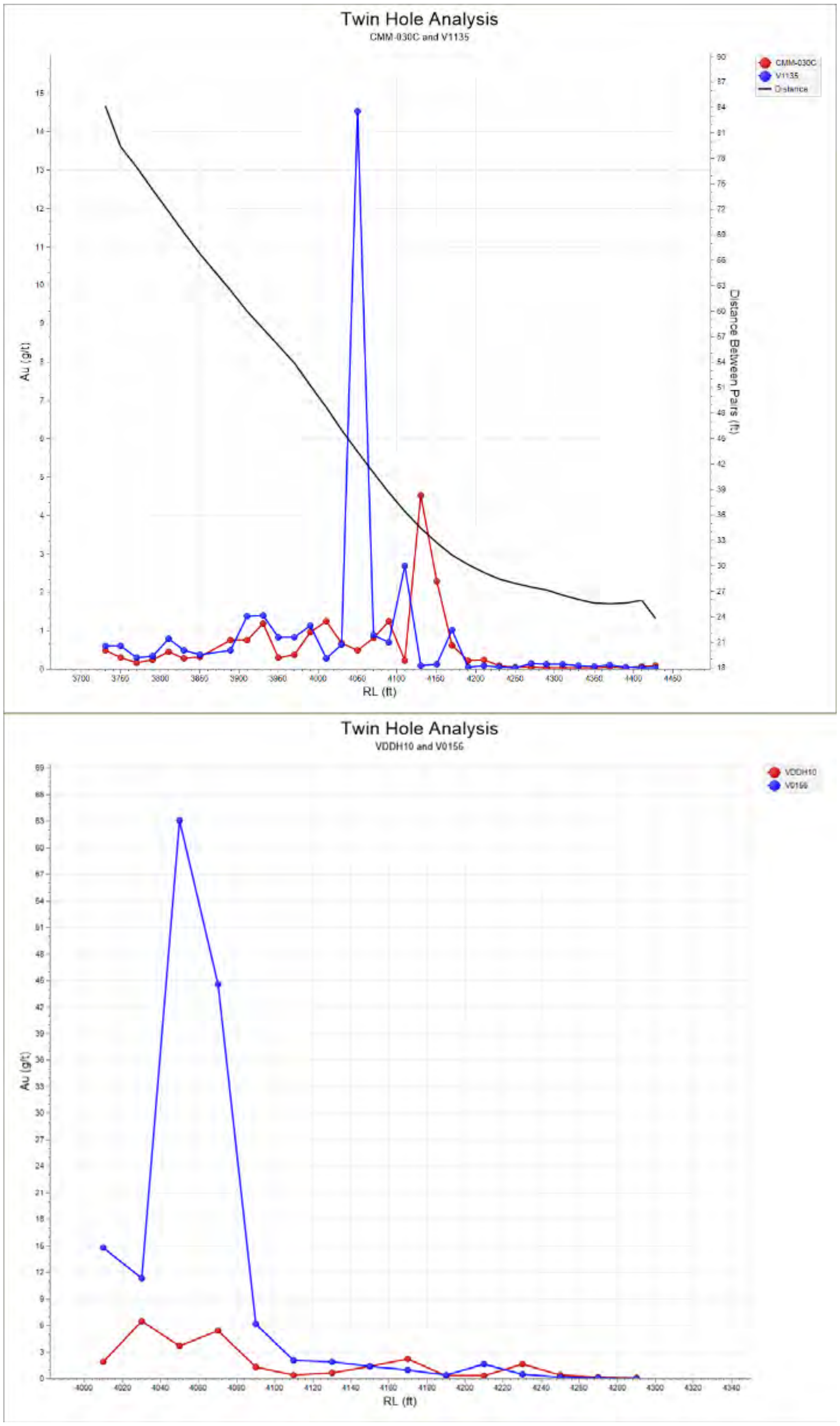


Figure 12-6: Twin Hole Analysis Showing Diamond Drill Holes in Red & RC Holes in Blue (CMM-030C and V1135, VDDH10 and V0156)

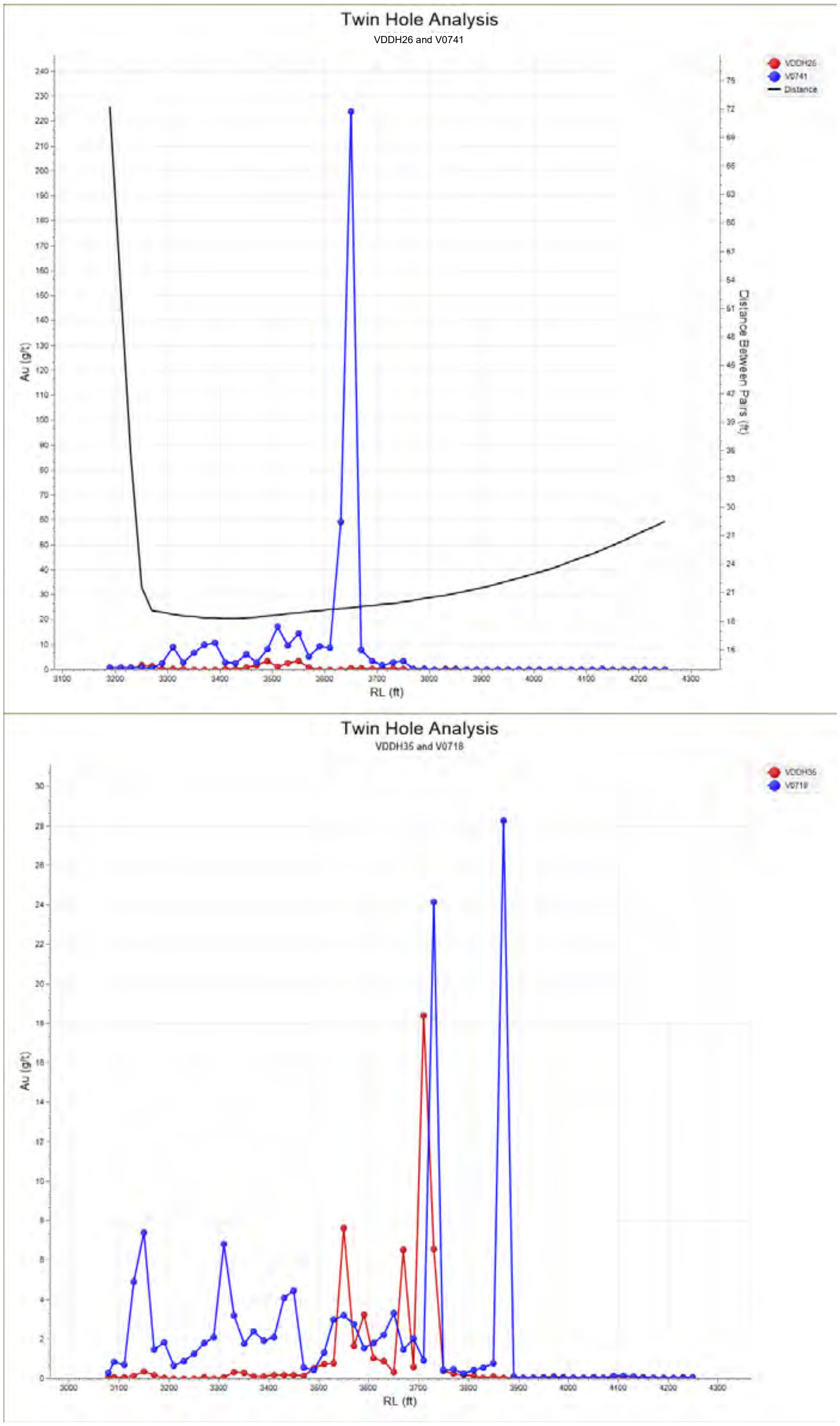


Figure 12-7: Twin Hole Analysis Showing Diamond Drill Holes in Red & RC Holes in Blue (VDDH26 and V0741, VDDH35 and V0718)

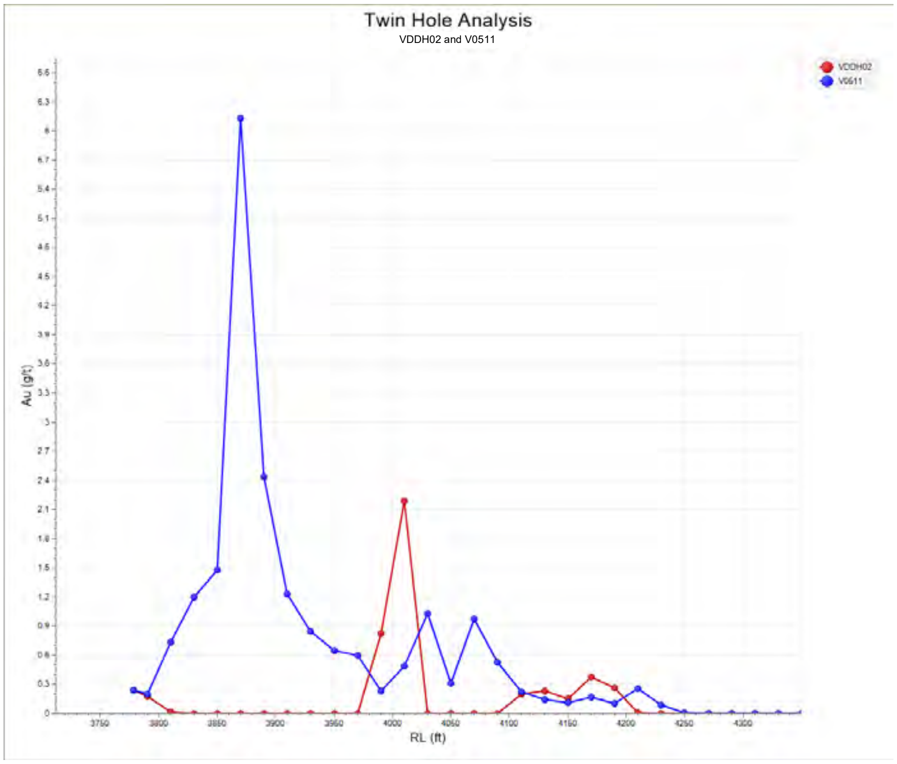


Figure 12-8: Twin Hole Analysis Showing Diamond Drill Holes in Red & RC Holes in Blue (VDDH02 and V0511)

12.4.3 Composite Pair Analysis

Bench composite sample pairs were generated based on operator and hole type to assist in determining the adequacy of legacy data. There are 138 20 ft (6.1 m) bench composites samples pairs. It should be emphasized that drilling direction was considerably different between holes drilled by Viceroy and those drilled by Equinox. Figure 12-9 shows a summary of drill hole orientations by operator. Holes drilled by Viceroy were dominantly vertical whereas holes drilled by Equinox (indicated by CMM in Figure 12-9) were inclined at an average of -65º.

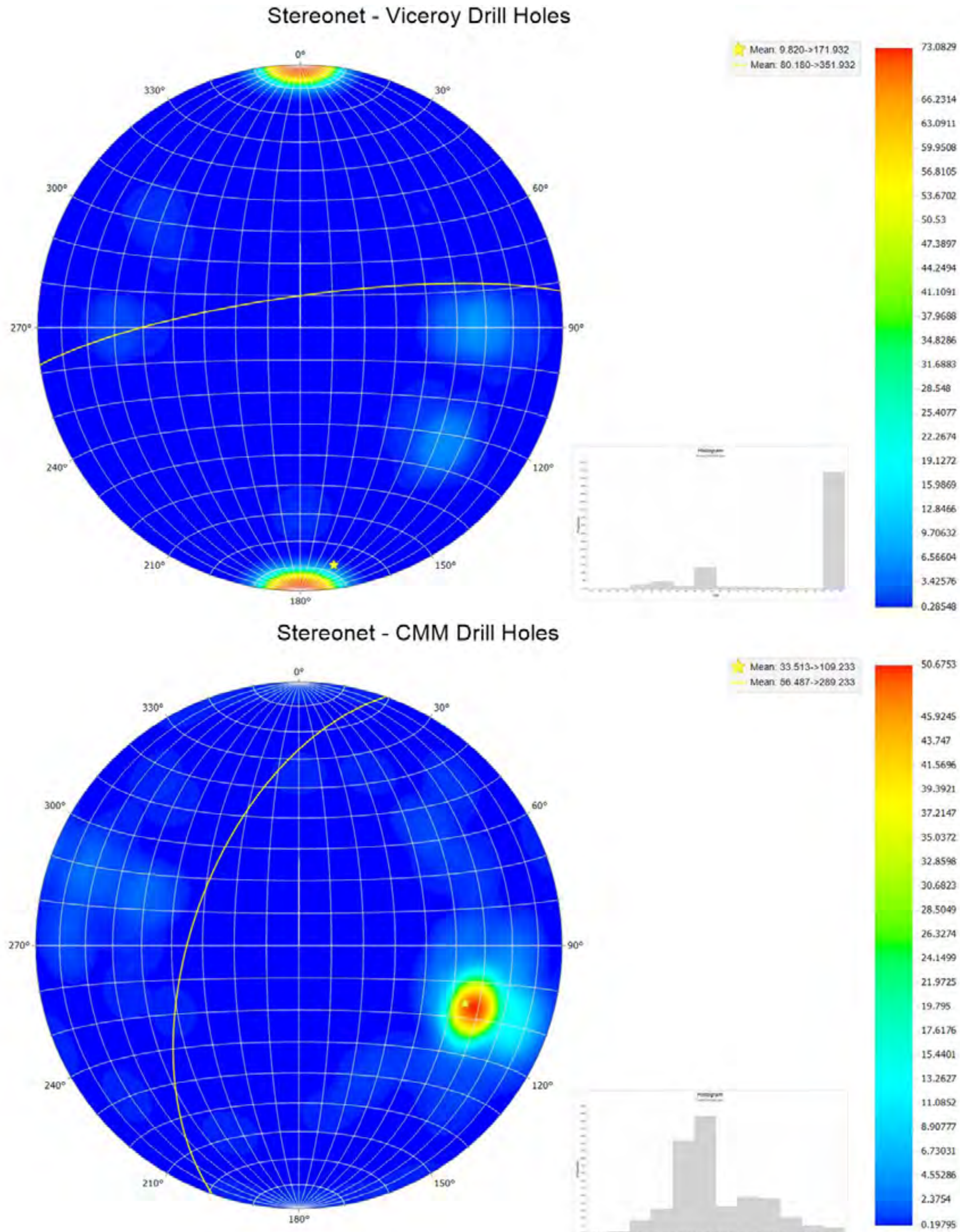


Figure 12-9: Comparison of Drill Hole Orientations by Operator

Composite sample pairs representing Equinox and Viceroy were generated for samples within 10 ft (3.05 m) irrespective of hole type (Figure 12-10). Sample pair data showed poor correlation of sample pairs, with some bias towards CMM holes greater than 0.0438 oz/ton (1.50 g/t) gold.

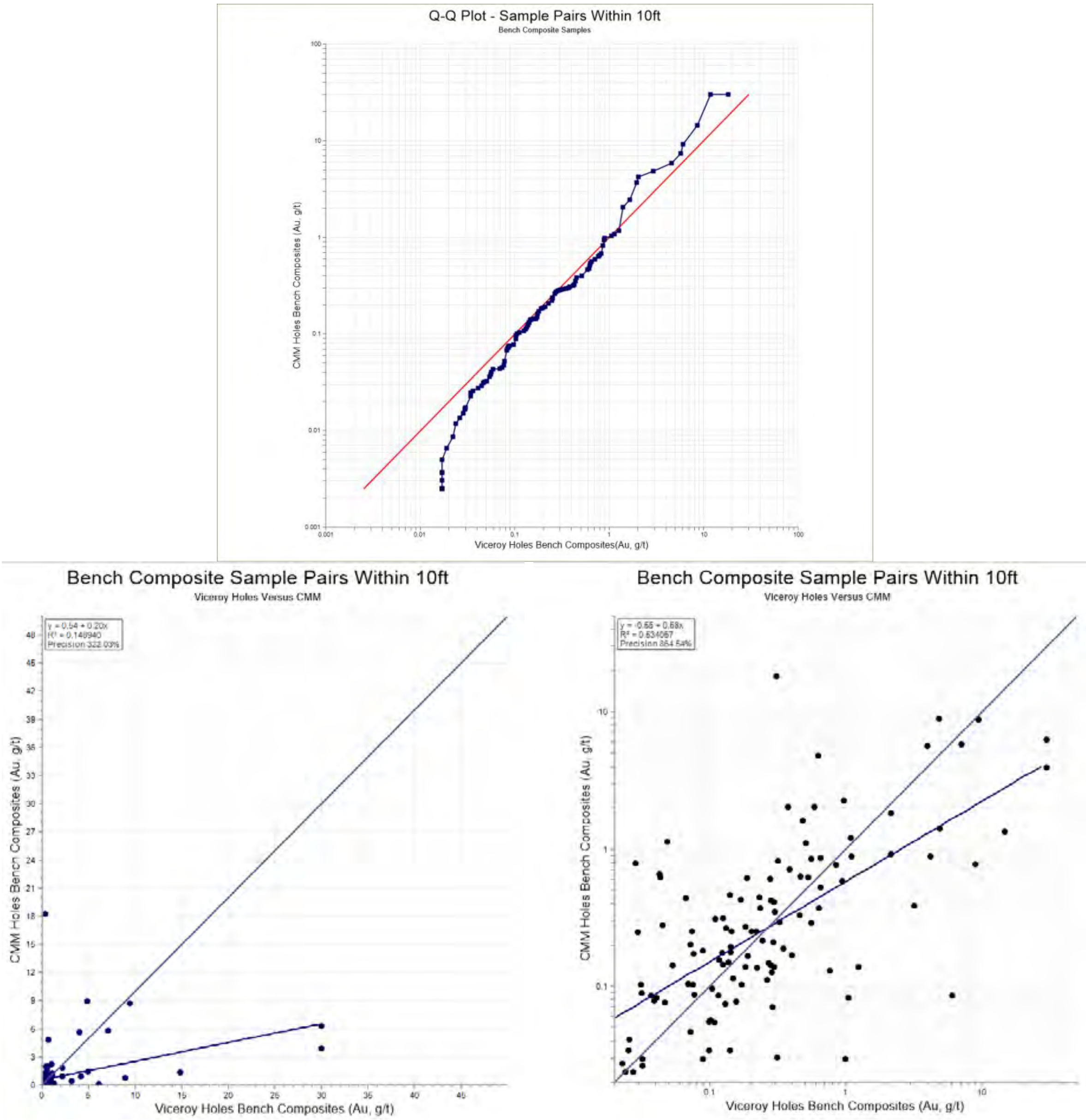


Figure 12-10: Q-Q Plot, Summary Statistics, and Scatterplots of Equinox and Viceroy Sample Pairs Within 10 ft (3.05 m)

Composite sample pairs were also generated comparing RC and diamond drill holes irrespective of operator (Figure 12-11). There was poor correlation between RC and diamond drill holes in addition to systematic bias observed within samples representing RC holes over the entire range of grades.

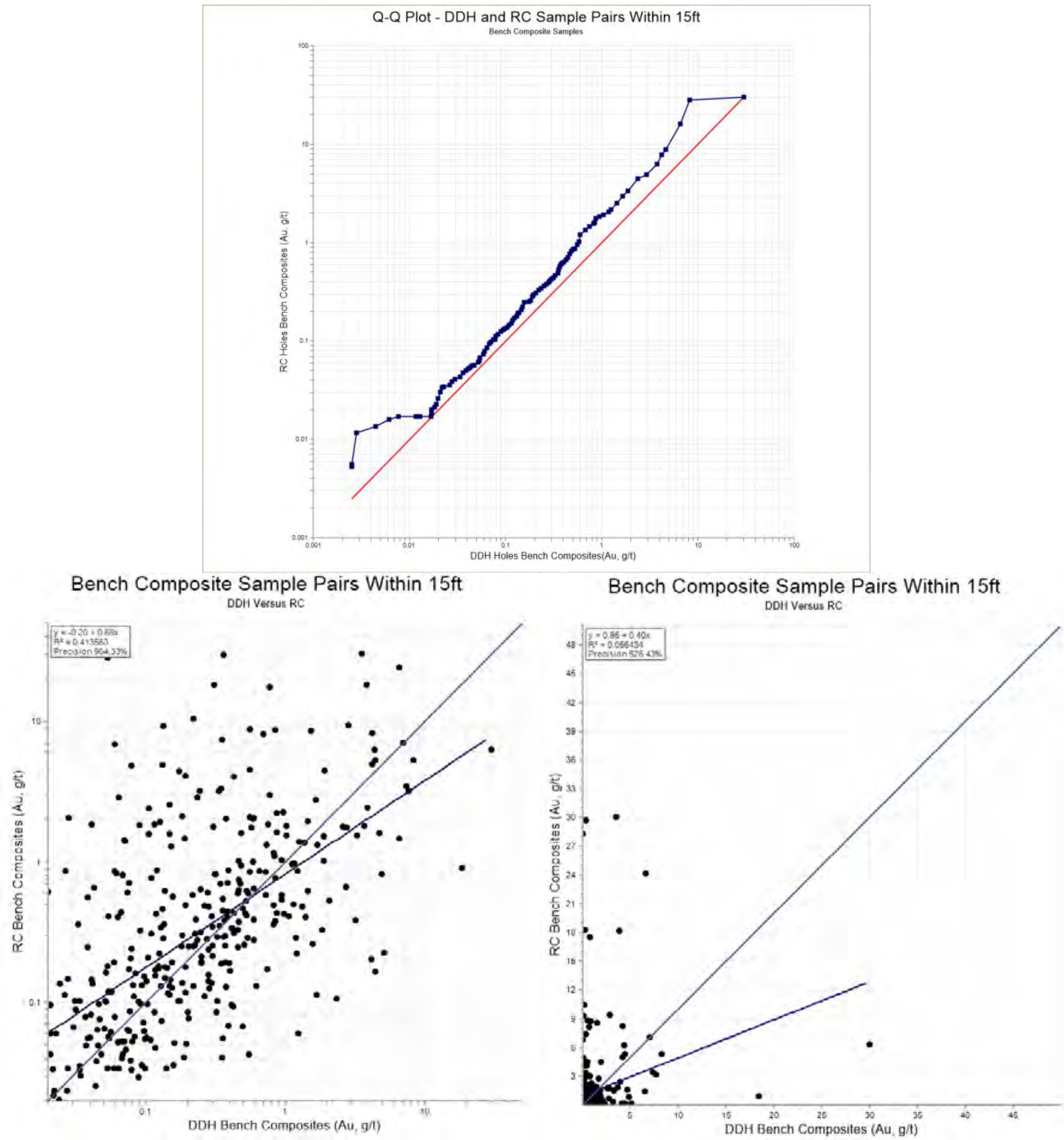


Figure 12-11: Q-Q plot, Summary Statistics, and Scatterplots of RC and Diamond Drill Hole Sample Pairs within 15 ft (4.6 m)

Lastly, composite sample pairs were generated comparing RC and diamond drill holes drilled by Equinox (Figure 12-12) and Viceroy (Figure 12-13). In both comparisons, there existed a bias towards samples representing RC holes.

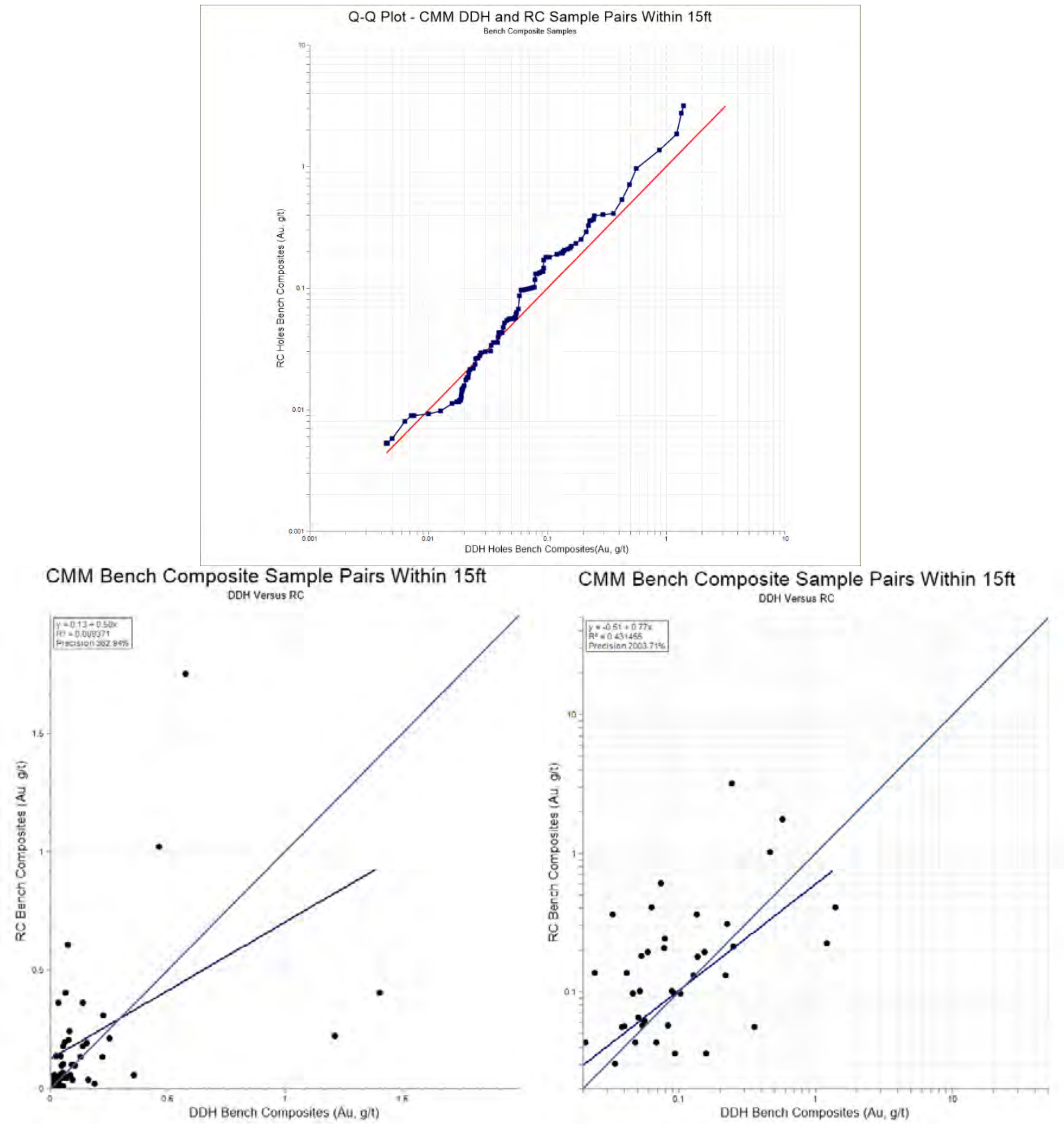


Figure 12-12: Q-Q Plot, Summary Statistics, and Scatterplots of RC and Diamond Drill Hole Sample Pairs Within 15 ft (4.6 m) Drilled by Equinox

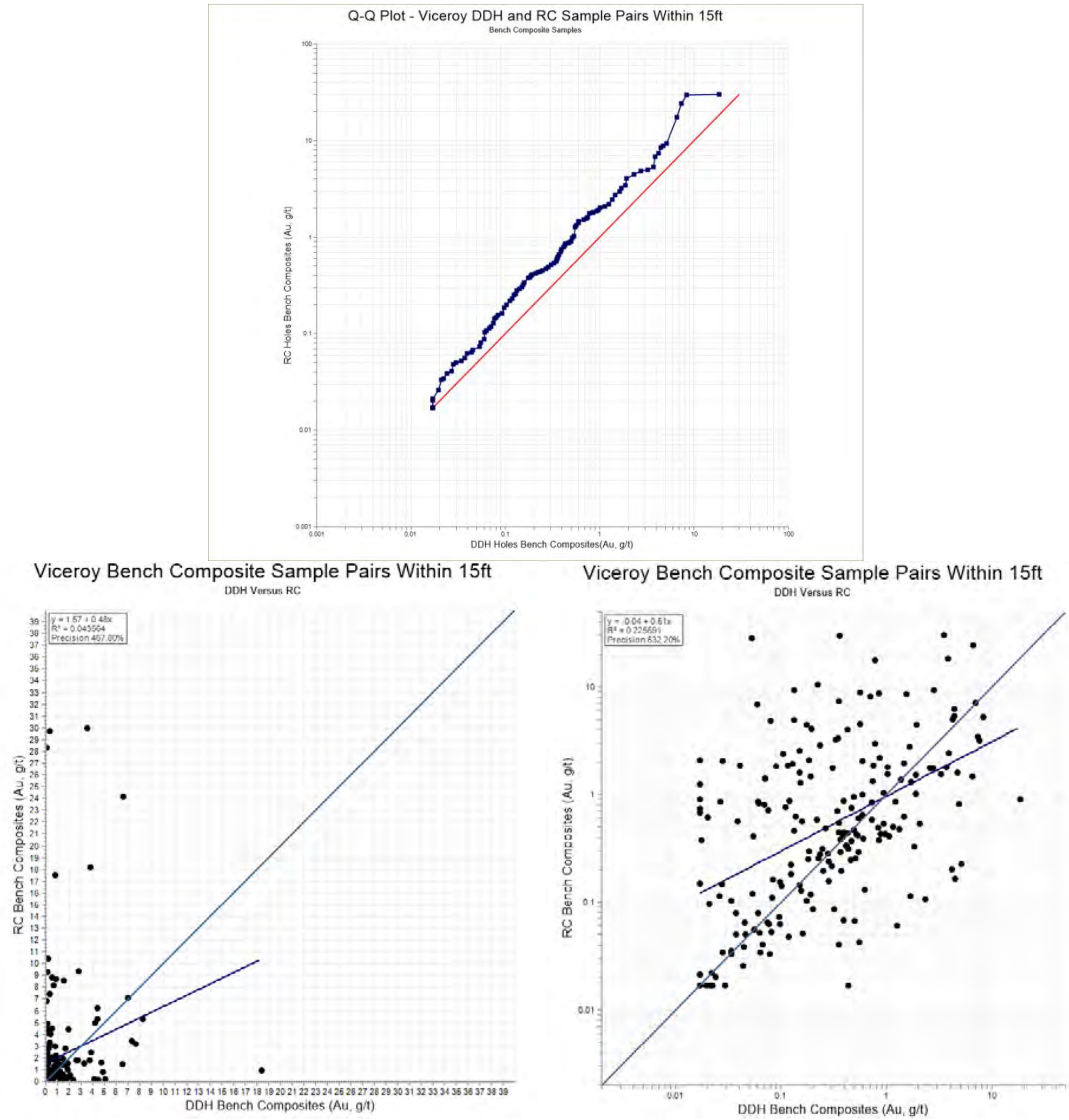


Figure 12-13: Q-Q Plot, Summary Statistics, and Scatterplots of RC and Diamond Drill Hole Sample Pairs within 15 ft (4.6 m) Drilled by Viceroy

In summary, RC drill holes tended to show some bias compared to diamond drill holes, irrespective of operator. Sample pair analysis between RC and diamond drill holes showed poor reproducibility due to different sampling methods and sample support. Holes drilled by Viceroy showed some bias up to 0.0583 oz/ton (2.00 g/t) gold, which was likely impacted due to drilling direction (Equinox holes were inclined, whereas Viceroy holes were vertical). Overall, the apparent issues in the Viceroy sample data (i.e. some bias in sample assay values from RC holes) were also apparent in the Equinox data and could be partially explained by the drill hole orientations.

12.5 BULK DENSITY

The laboratory certificates containing the 647 results for bulk density were reviewed (Table 12-4). Wax coated half core samples were observed in their core boxes during the site visit by the QP. The bulk densities were converted to tonnage factors. The tonnage factor for the dumps and JSLA backfill, 18.8 ft^3/ton (1.73 g/cm^3), was derived from work completed for the PFS which corresponds with an average bulk density of 14.43 ft^3/ton (2.24 g/cm^3) with a swell factor of 30% (Scott et al, 2018). Lithologies without bulk density measurements were assigned from comparable lithologies.

Table 12-4: Summary of the Bulk Density Data for the Project

Model Code	Lithology	Sample Count	Average Tonnage Factor (ft^3 per short ton)	Average Bulk Density (g/cm^3)
1	Backfill and Waste Dumps ([1])	0	18.8	1.73
2	Alluvium	0	16.8	1.9
3	Debris Flow	2	17.3	1.85
5	Dacite	0	14.3	2.24
5	Hart Peak Rhyolite	0	14.2	2.25
7	Rhyolite Breccia	9	14.4	2.24
9	Porphyritic Rhyolite	158	14.3	2.25
11	Aphyric Rhyolite	112	14.7	2.2
14	Volcaniclastic Diatreme	79	14.5	2.22
16	Volcaniclastic	146	15.3	2.11
22	Mudstone	1	16.1	1.99
23	Epiclastics	33	14.9	2.16
27	Andesite	82	14.4	2.23
29	Peach Springs Tuff	9	14.5	2.23
30	Proterozoic (Pc) Sediments	7	13.6	2.36
31	Proterozoic (Pc) Basement	9	12.4	2.59
	Total	**647**		
	Average - Hard Rock		**14.43**	**2.24**

Note: 1. The density of the dumps and backfill is derived from the average bulk density of the hard rock material 14.43 ft^3/ton (2.24 g/cm^3) with a swell factor of 30%.

12.6 DATA ADEQUACY

It is the opinion of the QP that the drill hole collar locations, downhole surveys and assay data supplied by Equinox are of adequate quality for use in Mineral Resource and Mineral Reserve estimates. Viceroy sample data comprise the majority of the sample population and do not have accompanying QA/QC data however investigations indicate that the Viceroy results are reasonably comparable to the Equinox assay results.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Significant metallurgical testwork has been performed on Castle Mountain samples from 2015 to 2020. A summary of all known metallurgical testwork for Castle Mountain was presented in the 2018 KCA PFS (Scott et al., 2018) and is presented again in this section of the report. Reports were issued with all results at the completion of all programs. Further testwork completed in 2020 in support of the FS is discussed in this section.

The plan is to process lower grade run of mine (ROM) ore on a leach pad and to process higher grade ore using conventional milling with Carbon-in-Leach (CIL). Testwork performed in 2020 has been completed using previous drill core samples with a focus on expanding the metallurgical understanding of the material to be processed through increased spatial and lithological representation within the mineral resource. The test objectives were to:

- Perform column leach tests on heap leach grade ore using the same parameters as in prior testing (i.e., size, cyanide concentration, application rate, etc.) and verify if the results were similar,
- Carry out additional testwork on heap leach grade ore including load permeability and mineralogy to support the feasibility study,
- Perform testwork on the mill grade ore using both gravity concentration followed by leaching of the gravity tails and whole ore leaching to select the process based on the expected recovery,
- Carry out further testwork on the mill grade ore to support the feasibility study design including materials handling tests, comminution (crushing, ball, and abrasion work indices as well as JK drop weight and SMC tests), carbon loading, detoxification of cyanide, oxygen uptake, thickening, tailing filtration, slurry rheology and filtered tailings geotechnical stability,
- Variability testing to verify optimum grind size for mill grade ore, and
- Testwork to determine the potential amenability to ore sorting.

The test programs were completed by various labs with the main work carried out by McClelland Laboratories Inc. (MLI), an independent commercial metallurgical lab located in Sparks, Nevada.

The 2015 and 2018 testwork analyzed the process using larger ROM material taken from JSLA backfill and has shown results similar to crushed material. M3 has utilized all available information to develop the Castle Mountain process flowsheet and provide expected plant performance results and recoveries. Data in sections and tables may be presented in metric or imperial units as provided in the original laboratory test reports. Where appropriate, both sets of units have been listed.

13.2 SUMMARY OF TESTWORK

Notable historical testwork includes activities performed prior to startup of the mine in 1992, as well as extensive testing and data recorded during the 12 years of production. More recent test programs occurred in 2014-2015 by MLI when the property was owned by NewCastle, in 2017-2018 by MLI when Equinox acquired the property, and then in 2019-2020 by MLI.

M3 also reviewed the Metallurgy and Processing sections (Sections 13 and 17) of the following Castle Mountain technical reports:

- Technical Reports prepared by RPA in 2013 and 2014,
- The Technical Report prepared by Advantage Geoservices in 2016, and
- The NI 43-101 Technical Report prepared by Kappes Cassidy & Associates in 2018 (Scott et al., 2018).

Reports containing all results were issued at the close of each metallurgical and physical testing program and are shown in Table 13-1 and Table 13-2. These reports, especially testwork completed since 2015, were used for developing the design criteria of the Castle Mountain feasibility.

Table 13-1: Metallurgical and Physical Testwork – Cyanide and Leach Testing

Date	Lab	Samples	BRTs	Sizes, mesh	Time, hrs	Column Tests	Sizes, in	Time, days	Reference
Cyanide / Leach Testing									
Pre-Historic Production									
February-87	Bateman	Jumbo South Bulk	3	100, 150, 200	72	5	3, 1½, 1, 3/4, 3/8	33	Bateman, 1987a
November-87	Bateman	87-7, 87-8, 87-9, 87-6A,6B,6C	6	100	24	15	2½, 1½, 3/4, 3/8	40 to 67	Bateman, 1987b
January-88	Bateman	Jumbo South DDH-3	3	100	24	6	3/4, 3/8	58 to 63	Bateman, 1988a
1988	Bateman	Leslie Ann DDH-1, DDH-2, DDH-8, DDH-10, DDH-11	7	100	24	19	2½, 1½, 3/4, 3/8, 1/4	67 to 118	Shoemaker 1988
September-88	Bateman	Leslie Ann DDH-10	---	---	---	10	3/8, ¼	69 to 105	Bateman, 1988b
July-89	McClelland	DDH-8, DDH-13, DDH-12, DDH-15, DDH-16, DDH-17, DDH-19, DDH-20	---	---	---	3	3/4	78	MLI, 1989
October-89	McClelland	DDH-18, DDH-3M, DDH-3U	1	100	72	4	3/8	68 +10 rinse	MLI, 1989b
Historic Production									
January-93	McClelland	HL Residue	6	100	24	---	---	---	MLI, 1993
March-95	McClelland	1994 Crusher Composites A: Jul-Sept, B: Apr-June, C: Jan-May	---	---	---	9	75%-3/8, 90%-3/8, 80%-¼	69	MLI, 1995a
May-95	McClelland	RC Cuttings - South Extension	4	As Received 100 mesh	120	---	---	---	MLI, 1995b
July-95	McClelland	Bulk Primary Crusher Product	---	---	---	1	6	85	MLI 1995
May-96	McClelland	141 South Ext DDH-56, DDH-57, DDH-58, DDH-59	3	80% -¼ in	240	2	¼	66 to 71	MLI 1996
Initial New Development Testing									
February-15	McClelland	Jumbo (CMM-012, 013), JSLA (CMM-014, 017) - All DDH-PQ	52	10 (1.7mm)		33	3/8(21), 3/4(6), 2(6)	75-168	MLI, 2015a
October-15	McClelland	ROM (Oro Belle South, JSLA Backfill)	2	10 (1.7mm)	96	4	ROM (2), 3/8 (2)	157-164	MLI, 2015b
PFS Testing									
June-17	RDI	ROM (JSLA Backfill)	15	6", 2", 3/8"	72	5	ROM	42	RDI, 2017

Date	Lab	Samples	BRTs	Sizes, mesh	Time, hrs	Column Tests	Sizes, in	Time, days	Reference
May-18	McClelland	ROM (JSLA Backfill)	---	---	---	4	ROM	130-140	MLI, 2018b
May-18	McClelland	JSLA, LG Master Composite	---	---	---	2	2, 3/8	120-130	MLI, 2018a
May-18	McClelland	Variability BRT Testing (S Domes, Oro Belle, JSLA Master, Andesite)	20	50 mm (1), 9.5 mm (19)	480 (20 days)	---	---	---	MLI, 2018a
May-18	McClelland	Variability Gravity / Leach Tests (S Domes, Oro Belle, JSLA HG Master, Andesite)	12	100	96	---	---	---	MLI, 2018a
May-18	McClelland	Gravity/Leach Tests, Variable Grind (JSLA HG Master)	4	48, 65, 100, 150	96	---	---	---	MLI, 2018a
August-18	KCA	Gravity / CIL Test, High Grade Comp.	4	100	variable	---	---	---	KCA, 2018c
FS Testing									
June-2020	McClelland	Low-grade - South Domes, Oro Belle, East Ridge, JSLA	12	10 (1.7mm)	96	21	1 1/4”, 3/8”	1440 (60 days)	MLI, 2020a
June-2020	McClelland	Mill grade ore - South Domes, Oro Belle, East Ridge, JSLA, variable grind	30	200, 100, 48	96	---	---	---	MLI, 2020a

Table 13-2: Metallurgical and Physical Testwork – Other Testing

Date	Lab	Tests	Description	Reference
Other Testing				
Historic Production				
May-98	Glasgow	Compacted Permeability	Production samples	Glasgow, 1998
Initial New Development Testing				
February-17	McClelland	Comminution	Crusher Index (CWi) and Bond Abrasion (Ai) for Four PQ core samples	MLI, 2015a
PFS				
May-18	Pocock	Gravity Sedimentation, Vacuum and Pressure Filtration (JSLA HG Master)		MLI, 2018a
July-18	KCA	Compacted Permeability (JSLA LG/HG Master, Pulp Agglomerated)	High-Grade/Low-Grade blend of 1:4, at varying cement dosage	KCA, 2018a
July-18	KCA	Compacted Permeability (JSLA ROM w/ Mill Tails)	ROM / Mill Tails Varying Blends for Tailings Disposal Investigations	KCA, 2018b
FS				
2020	MLI/FLS	Comminution	Crusher Index (CWi), Bond Abrasion (Ai), Bond Abrasion (Ai), JKDropweight, SMC, Bond ball mill work index (BWi)	FLSmidth, 2020
2020	MLI/Pocock	Solid/Liquid Separation	Thickener (static and dynamic), Filtration (pressure and vacuum)	MLI, 2020a
2020	MLI/Pocock	Rheology		Pocock, 2020
2020	Cyanco	Detox	Caro’s acid and SO_2/Air (SMBS)	Cyanco, 2020
2020	Steinert	Ore Sorting		Steinert, 2020
2020	PMC	Mineralogy	In-situ Gold Deportment Study of Column Leach Head and Residue	PMC, 2020
2020	MLI/Pocock	CIC and CIL	Sulfur Speciation, Carbon Adsorption Rate, Carbon Adsorption Capacity, Oxygen Update Rate	MLI, 2020b Addendum
2020	Call & Nicholas	Compacted Permeability	Simulated stack height of 400 ft.	CNI, 2020

13.3 METALLURGICAL TESTWORK SAMPLING ORIGIN AND COMPOSITES

Core samples from 2013-2018 exploration drilling were used for metallurgical testing. Intervals were selected from these samples to provide better representation of areas and lithology of the Castle Mountain deposit that had previously not been tested as extensively. Intervals from the half HQ core were used to make up 24 composites (12 heap leach and 12 mill-CIL) for metallurgical testing in 2020. Table 13-3 shows the composites used for testing on heap leach grade ore. In addition, a single blended master composite was generated from 3 of the heap leach composites. Table 13-4 shows the composites used for testing on mill grade ore.

One higher grade sample was tested as heap leach grade ore as a check on the improvement in recovery, and one lower grade sample was tested as mill grade ore to determine the impact of dilution.

Table 13-3: Heap Leach Grade Ore Composites

Composite	Zone	Hole ID	From	To	Length	Lithology	Avg Calc Head g/t Au
4505-001-LG	South Domes	CMM-122C	687.25	1237.5	550.25	Vx	1.39
4505-002-LG	South Domes	CMM-255C	80	193.5	113.5	Rhy-Aph	0.34
			257	333	76		
4505-003-LG	South Domes	CMM-242C	597	700	103	Diatreme	0.37
4505-007-LG	JSLA	CMM-229C	1042	1164	122	Andesite	0.35
			1213	1259.8	46.8		
			1282	1373	91		
4505-010-LG	Jumbo	CMM-273C	316	422	106	Vx	0.24
4505-015-LG	East Ridge	CMM-010C	250	325	75	Rhy-Aph	0.15
			430	670	240		
4505-017-LG	Oro Belle	CMM-119C	12.616	348	335.4	Vx	0.21
4505-018-LG	Oro Belle	CMM-119C	352	624	272	Rhy-Aph	0.44
4505-019-LG	Oro Belle	CMM-119C	1030	1125	116.8	Andesite	0.42
4505-020-LG	Oro Belle	CMM-120C	20	282	262	Vx	0.17
4505-021-LG	Oro Belle	CMM-120C	454	602	148	Rhy-Aph	0.28
		CMM-276C	0	129	129		
4505-025-LG	Oro Belle	CMM-120C	1267	1419	152	Epiclastic	0.19
4505 LG MC	Blend	Blend	-	-	-	See Note	0.82

Note: Blend consists of 24% South Domes (Vx), 19% East Ridge (Rhy-Aph), 57% Oro Belle (Vx)

Heap leach composite head assay results show that head grades determined by direct assay agreed reasonably closely (as can be seen by the trend line with an R^2 of 0.91) with the calculated head grades from the metallurgical tests, as shown in Figure 13-1.

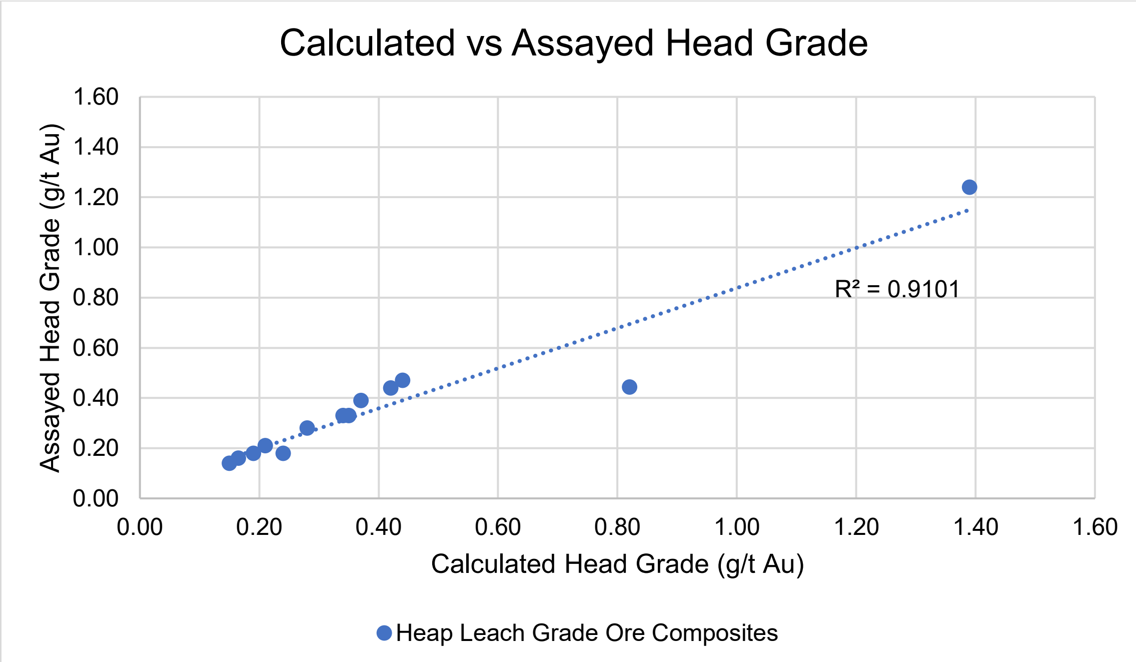


Figure 13-1: 2020 Heap Leach Grade Ore Composites: Calculated vs Assayed Head Grade

Table 13-4: Mill Grade Ore Composites

Composite	Zone	Hole ID	From	To	Length	Lithology	Avg Calc Head g/t Au
4505-004-HG	South Domes	CMM-248C	644	714.5	70.5	Vx	0.82
		CMM-252C	565	655.5	90.5		
4505-005-HG	South Domes	CMM-016C	717	895	178	Rhy-Aph	0.79
4505-006-HG	South Domes	CMM-250C	804.5	897	92.5	Vx	1.36
4505-008-HG	JSLA	CMM-070C	1440	1540	100	Diatreme	0.94
4505-009-HG	JSLA	CMM-033C	472	799	327	Rhy-Bx	0.68
4505-011-HG	Jumbo	CMM-021C	5	24	19	Rhy-Aph	0.28
4505-012-HG	Jumbo	CMM-283C	885	1038	153	Andesite	1.0
4505-013-HG	Jumbo	CMM-281C	753.6	786	32.4	Rhy-Porph	0.65
4505-022-HG	Oro Belle	CMM-119C	432.5	469	37.0	Rhy-Aph	1.04
			553	573	20		
4505-023-HG	Oro Belle	CMM-120C	433.5	454	20.5	Diatreme	1.12
4505-024-HG	Oro Belle	CMM-036C	603	661	58	Andesite	1.73
4505-026-HG	Oro Belle	CMM-036C	354	586	232	Vx	0.78

The arithmetic mean of the 12 mill grade samples in 2020 was 0.92 g/t which is lower than the LOM average mill grade of 2.3 g/t, but note that the mean from 5 mill grade samples in 2015 was 15.0 g/t and mean from 14 mill grade samples in 2015 was 3.47 g/t, so the entire range of mill grade ores has been tested in the various campaigns.

Mill grade ore composite head assay results show that head grades determined by direct assay agreed reasonably closely (as can be seen by the trend line with an R^2 of 0.92) with the calculated head grades from the metallurgical tests, as shown in Figure 13-2.

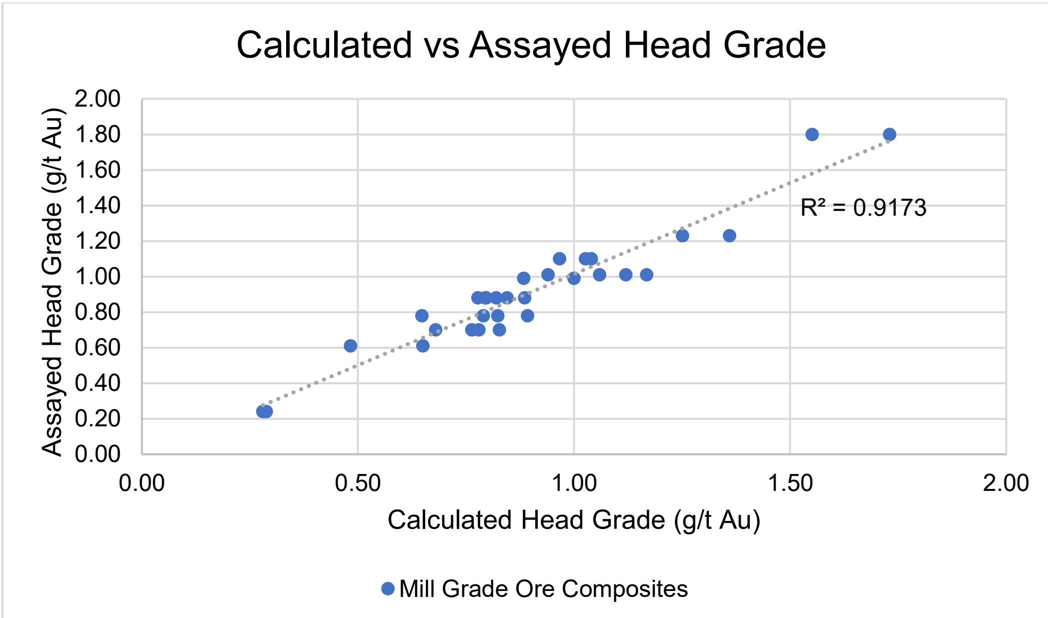


Figure 13-2: 2020 Mill Grade Ore Composites: Calculated vs Assayed Head Grade

The locations of the individual samples used to make up these composites are shown in plain view in Figure 13-3. Figure 13-4 to Figure 13-8 show cross-sections of the sampling effort. The ore bodies are composed of five main ore zones, namely JSLA, Jumbo, Oro Belle, East Ridge and South Domes. By the end of the mine life the northern four zones are combined and referred to as main pit.

Overall, samples used for testing completed for the Castle Mountain Project appear to be representative of the ore that will be processed.

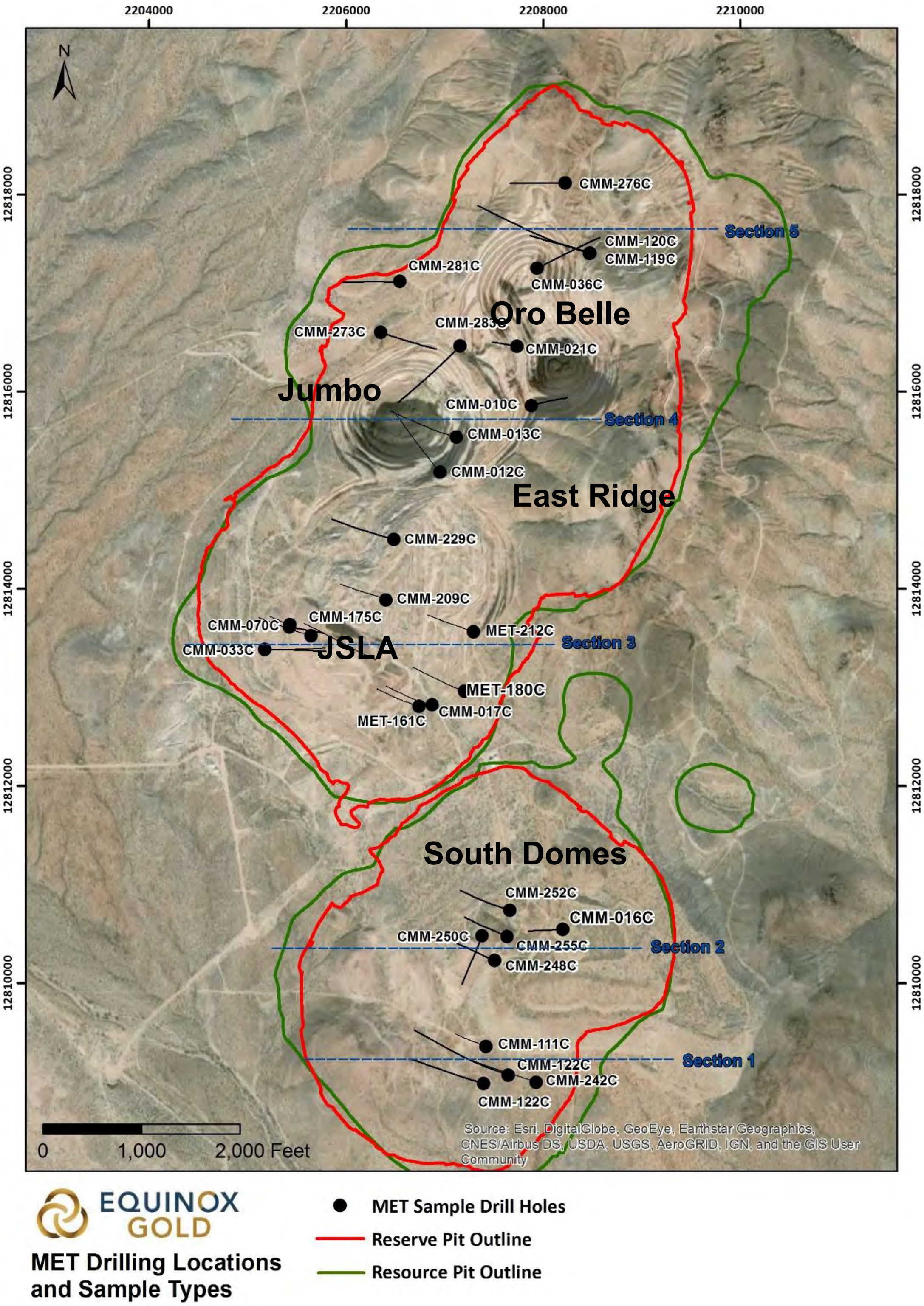


Figure 13-3: Metallurgical Sample Locations

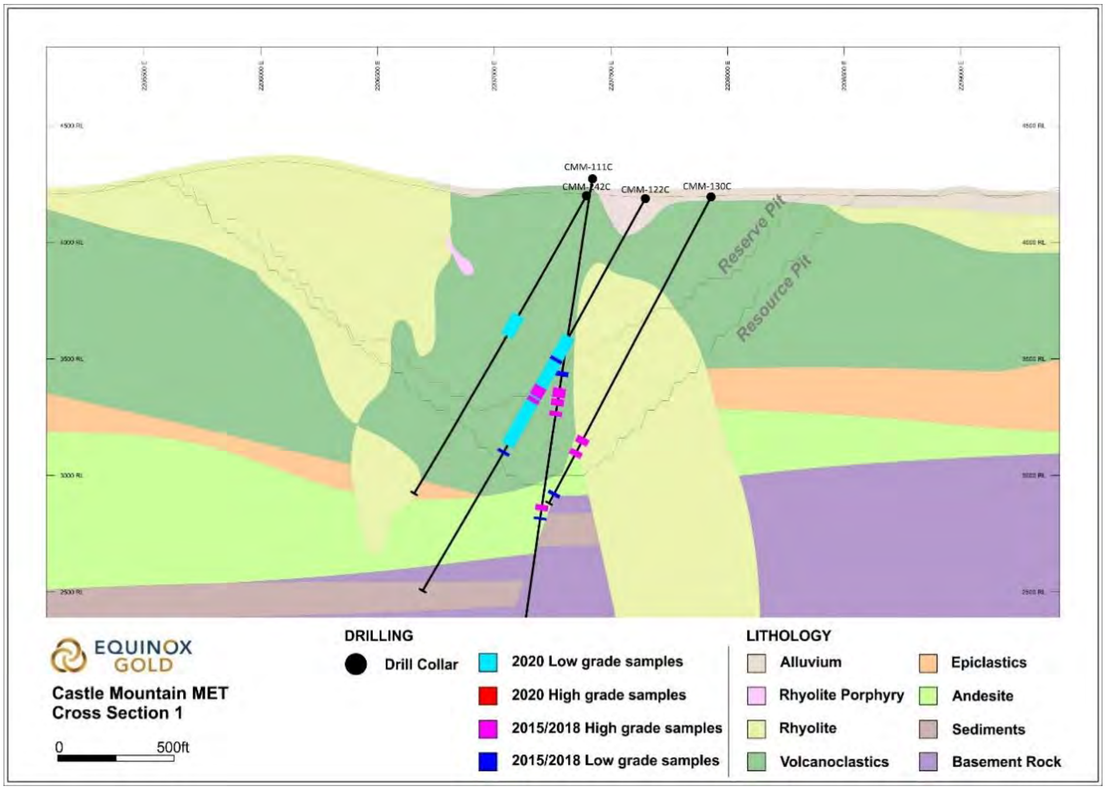


Figure 13-4: Castle Mountain Metallurgy Cross Section 1

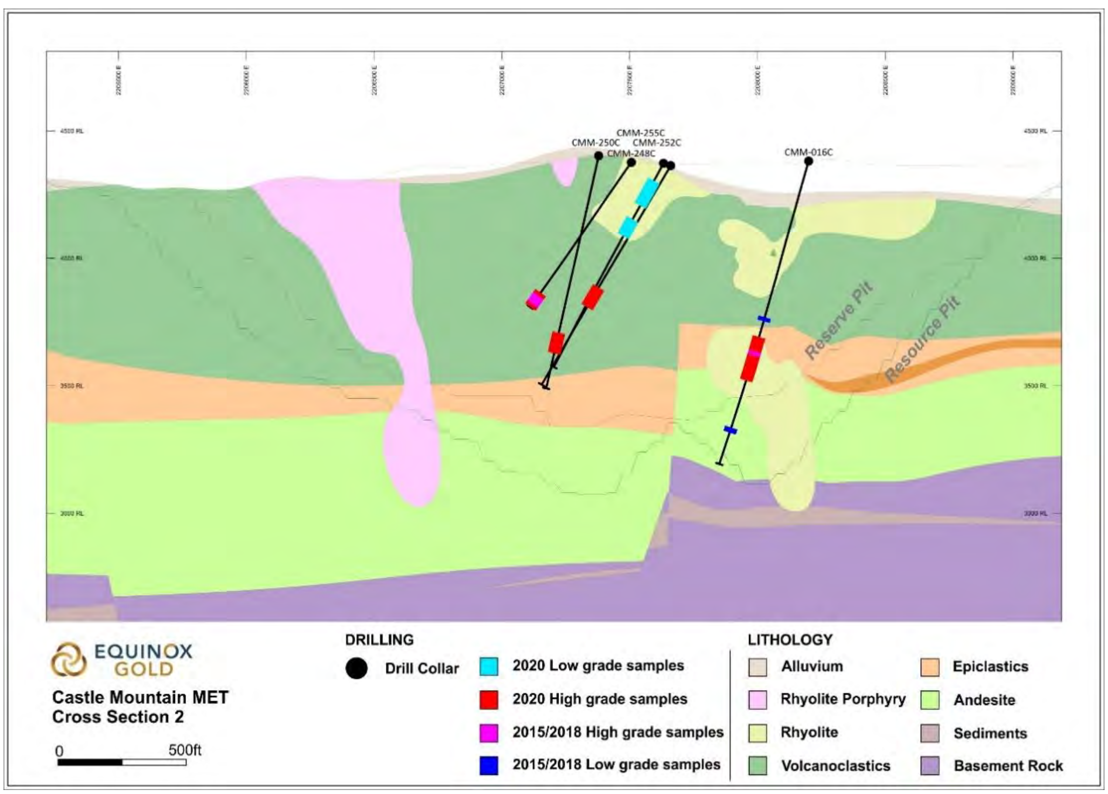


Figure 13-5: Castle Mountain Metallurgy Cross Section 2

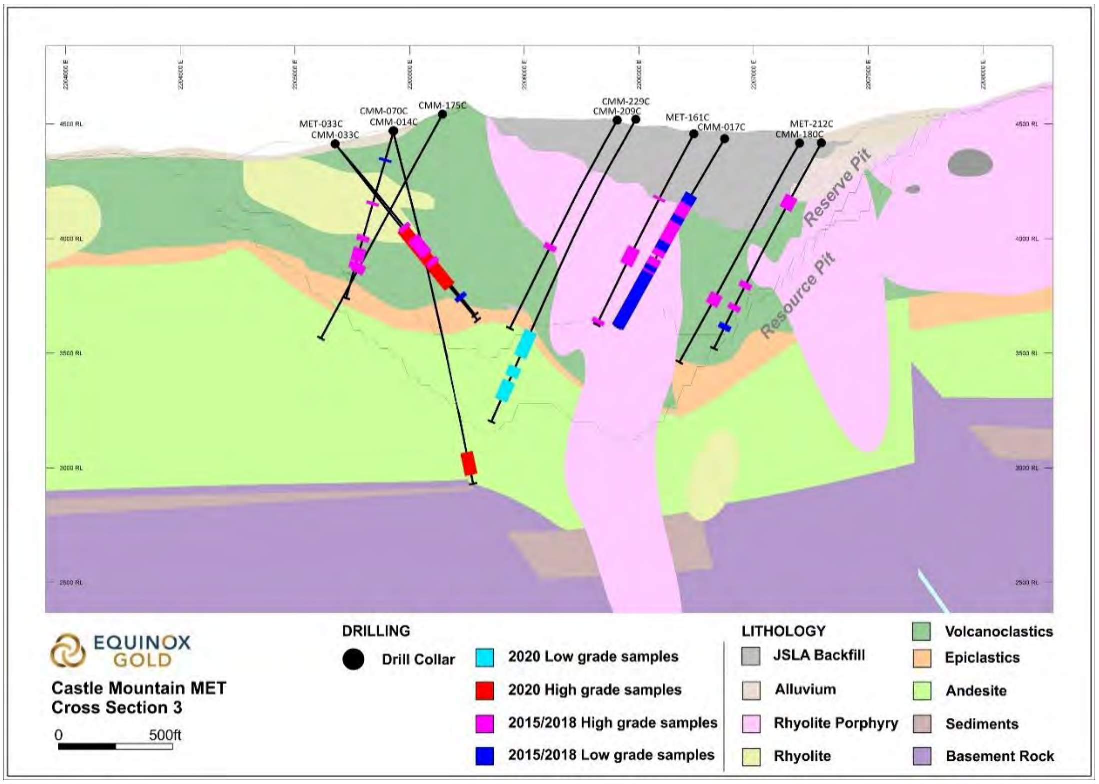


Figure 13-6: Castle Mountain Metallurgy Cross Section 3

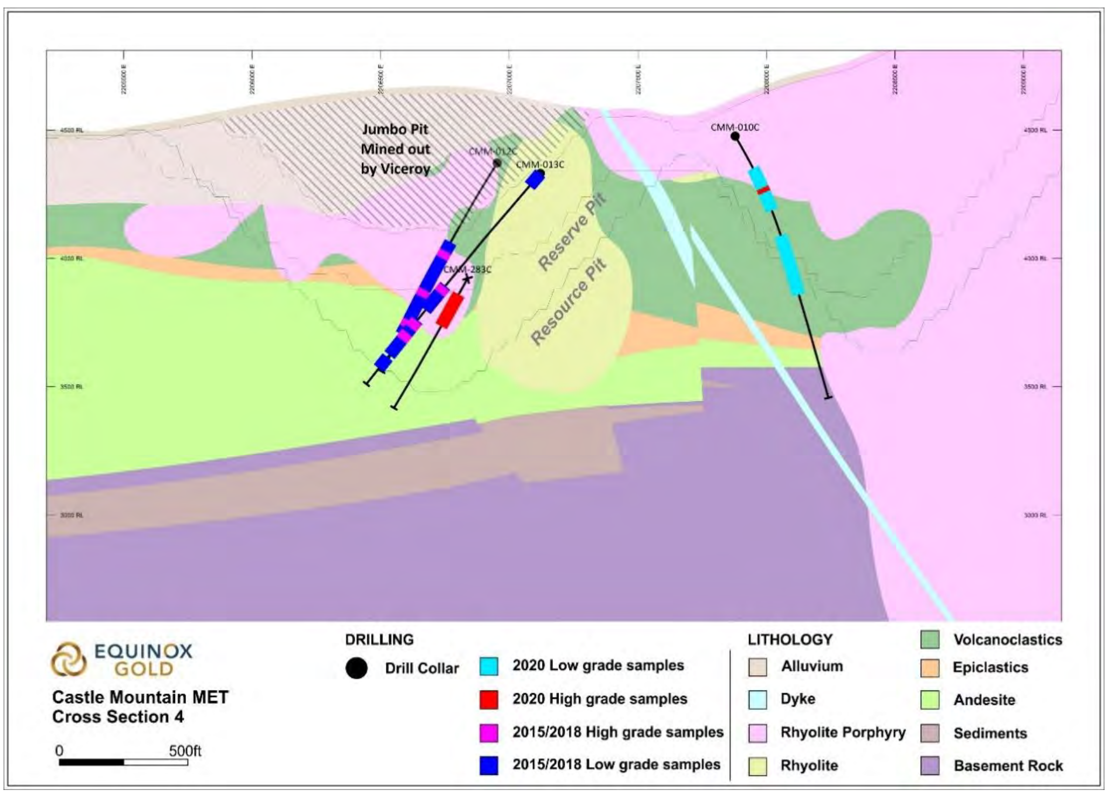


Figure 13-7: Castle Mountain Metallurgy Cross Section 4

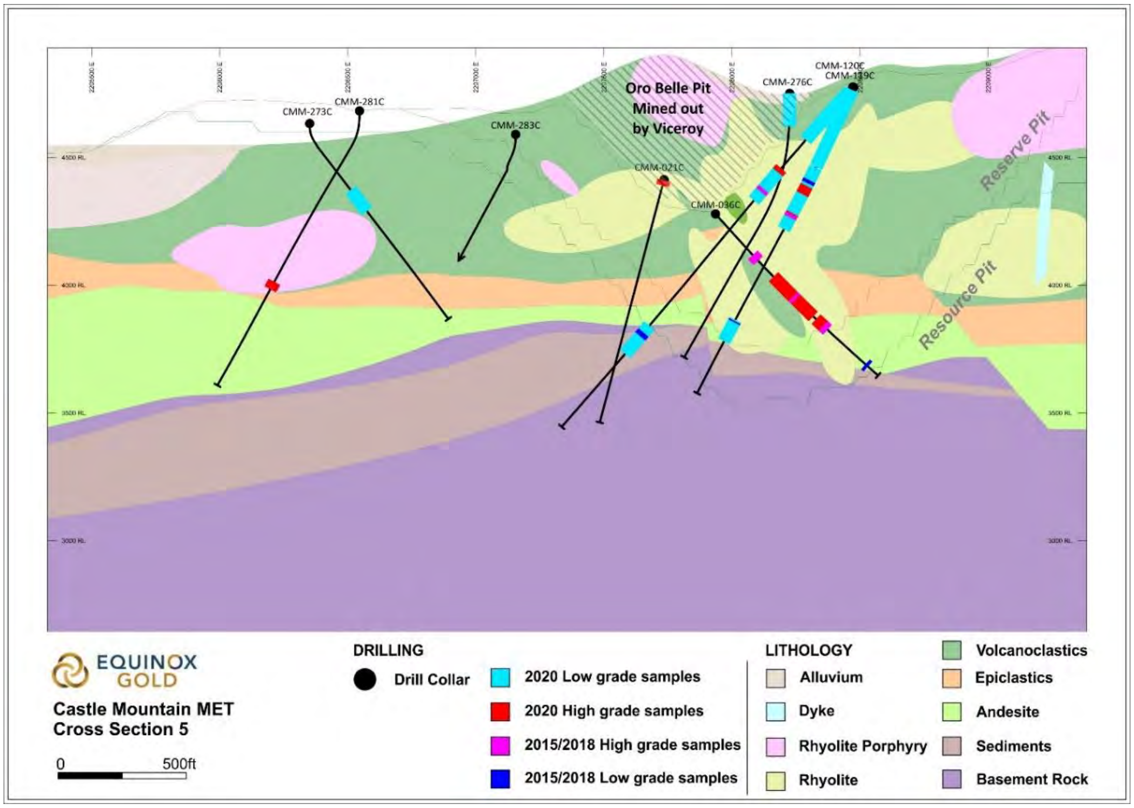


Figure 13-8: Castle Mountain Metallurgy Cross Section 5

13.4 SAMPLE ELEMENT ANALYSIS

ICP scans and mercury analyses were completed for all heap leach and mill grade ore composites.

ICP readings show some elevated concentrations for some elements (e.g. As, Cu, Ni, Pb, and Zn), that may be detrimental to carbon adsorption depending on solubility and ratio to concentration of precious metal in solution. These values are primarily associated with Andesite samples which are near the bottom of the pits and represent a small fraction of the overall orebody.

Sulfur content for the composites ranged from 0.01% to 1.64% with the average being 0.14%. Sulfur content in the heap leach composites was less than 0.75% which may impact recovery and/or reagent consumption. Sulfur content in the mill grade composites was generally low with the exception of one composite being 1.64%. ICP analysis for the heap leach and mill composites are listed in Table 13-5 and Table 13-6.

Table 13-5: Multi-Element Analysis for Heap Leach Grade Ore Composites

		4505-001-LG South Domes	4505-002-LG South Domes	4505-003-LG South Domes	4505-007-LG JSLA	4505-010-LG Jumbo	4505-015-LG East Ridge	4505-017-LG Oro Belle	4505-018-LG Oro Belle	4505-019-LG Oro Belle	4505-020-LG Oro Belle	4505-021-LG Oro Belle	4505-025-LG Oro Belle
Analysis	**Unit**	**Vx**	**Rhy-Aph**	**Diatreme**	**Andesite**	**Vx**	**Rhy-Aph**	**Vx**	**Rhy-Aph**	**Andesite**	**Vx**	**Rhy-Aph**	**Epiclastic**
Ag	mg/kg	0.92	3.24	0.92	1.74	0.45	0.53	1.59	1.30	1.20	1.14	0.94	1.05
Al	%	5.40	4.68	5.04	7.54	5.59	5.50	5.47	5.62	7.55	5.50	5.83	6.93
As	mg/kg	42.7	11.9	33.5	180.5	55.3	54.7	125.5	115.0	72.2	122.0	77.0	85.0
Ba	mg/kg	230	20	230	1,980	90	90	230	100	1,540	170	90	1,380
Be	mg/kg	3.96	2.93	4.21	3.01	3.33	3.27	6.23	3.47	2.48	5.07	3.86	2.83
Bi	mg/kg	0.10	0.08	0.09	0.16	0.08	0.13	0.10	0.17	0.14	0.09	0.08	0.12
Ca	%	0.23	0.22	0.17	1.20	0.10	0.10	0.16	0.16	0.79	0.13	0.14	0.68
Cd	mg/kg	0.04	0.05	0.04	0.10	0.06	0.07	0.15	0.05	0.13	0.16	0.03	0.06
Ce	mg/kg	60.3	33.7	55.0	265	56.4	53.1	65.6	50.3	198.5	58.1	50.5	166.0
Co	mg/kg	1.9	0.4	1.4	21.0	0.2	0.2	17.7	1.2	19.2	2.4	0.8	10.2
Cr	mg/kg	100	115	106	76	69	72	80	97	43	106	101	98
Cs	mg/kg	10.45	3.81	12.40	4.05	4.66	4.33	9.57	4.80	3.23	7.53	4.67	3.72
Cu	mg/kg	11.0	6.5	7.2	110.0	4.5	7.3	14.7	8.7	79.7	8.3	6.0	45.3
Fe	%	0.95	0.48	0.82	4.22	0.52	0.51	0.90	0.70	2.48	0.82	0.64	2.81
Ga	mg/kg	13.25	10.95	12.25	23.5	14.95	14.65	15.85	13.40	19.15	15.90	14.60	17.95
Ge	mg/kg	0.19	0.15	0.19	0.37	0.17	0.12	0.14	0.12	0.24	0.13	0.14	0.24
Hf	mg/kg	3.5	2.9	3.3	9.2	3.3	3.2	3.5	3.1	6.3	3.4	3.2	5.8
In	mg/kg	0.029	0.024	0.025	0.056	0.027	0.031	0.023	0.028	0.037	0.028	0.029	0.039
K	%	4.70	4.40	4.43	3.97	4.24	4.54	4.79	4.62	5.38	5.09	4.96	5.93
La	mg/kg	31.4	16.7	28.4	138.0	29.8	26.9	32.4	23.5	100.0	28.0	24.8	85.7
Li	mg/kg	58.3	122.5	81.3	56.8	97.5	90.5	81.6	61.0	25.7	89.6	53.6	45.7
Mg	%	0.14	0.03	0.13	0.83	0.22	0.21	0.21	0.16	0.35	0.13	0.10	0.78
Mn	mg/kg	111	94	335	467	162	150	559	161	481	274	142	261
Mo	mg/kg	5.53	13.25	4.70	5.70	9.84	9.59	9.00	12.75	3.35	8.60	25.0	8.23
Na	%	1.25	0.17	1.07	0.93	0.32	0.31	0.40	0.93	1.02	0.41	1.40	0.63
Nb	mg/kg	21.5	19.4	21.0	34.7	23.8	23.2	22.1	22.9	31.8	21.8	23.6	23.0
Ni	mg/kg	6.6	4.3	6.2	44.8	1.5	1.5	12.9	4.0	31.5	4.8	2.9	26.1
P	mg/kg	310	20	240	3,830	50	30	310	180	2,350	210	230	1,680
Pb	mg/kg	18.6	13.8	19.3	32.7	19.3	18.0	19.6	19.5	32.0	18.0	18.9	24.7
Rb	mg/kg	311	231	231	136.0	212	239	287	286	208	270	288	238
Re	mg/kg	<0.002	<0.002	<0.002	0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	0.002	0.008
S	%	0.01	0.01	0.01	0.41	0.01	0.01	0.08	0.02	0.01	0.05	0.02	0.71
Sb	mg/kg	13.20	29.0	9.13	8.59	12.25	12.20	19.75	14.00	8.39	23.9	21.3	12.50
Sc	mg/kg	1.9	0.9	1.5	9.3	1.1	1.0	2.2	1.5	4.6	1.8	1.5	7.2
Se	mg/kg	<1	1	1	1	1	<1	<1	<1	1	<1	1	1
Sn	mg/kg	2.5	2.8	2.1	2.1	2.6	2.5	3.8	2.9	1.6	2.4	2.8	1.7
Sr	mg/kg	179.5	77.4	131.0	802	138.5	137.0	143.5	127.0	565	155.0	140.0	488
Ta	mg/kg	1.46	1.40	1.39	1.57	1.73	1.67	1.49	1.67	1.52	1.53	1.71	1.14
Te	mg/kg	<0.05	<0.05	<0.05	0.16	<0.05	0.05	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05
Th	mg/kg	22.3	18.60	20.9	34.2	23.3	22.8	21.9	22.1	27.0	21.5	22.8	25.4
Ti	%	0.090	0.035	0.073	0.599	0.050	0.049	0.105	0.062	0.392	0.086	0.060	0.388
Tl	mg/kg	1.42	1.61	1.38	1.49	1.31	1.28	1.56	1.50	2.26	1.77	1.57	1.99
U	mg/kg	3.4	2.6	3.0	13.1	2.9	2.7	5.1	3.5	7.2	4.3	5.0	8.5
V	mg/kg	24	47	20	120	6	6	17	10	72	16	8	84
W	mg/kg	5.7	1.9	4.5	11.8	1.1	1.1	3.5	3.0	7.6	2.9	2.5	5.3
Y	mg/kg	21.8	19.4	21.6	22.5	23.2	22.2	23.3	21.0	17.3	22.5	25.1	18.7
Zn	mg/kg	38	14	38	96	23	25	49	25	84	33	24	62
Zr	mg/kg	109.0	71.3	99.1	416	84.9	83.5	102.5	82.0	307	99.2	83.5	258

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Table 13-6: Multi-Element Analysis for Mill Grade Ore Composites

Analysis	Unit	4505-004-HG South Domes Vx	4505-005-HG South Domes Rhy-Aph	4505-006-HG South Domes Vx	4505-008-HG JSLA Diatreme	4505-009-HG JSLA Rhy-Bx	4505-011-HG Jumbo Rhy-Aph	4505-012-HG Jumbo Andesite	4505-013-HG Jumbo Rhy-Porph	4505-022-HG Oro Belle Rhy-Aph	4505-023-HG Oro Belle Diatreme	4505-024-HG Oro Belle Andesite	4505-026-HG Oro Belle Vx
Ag	mg/kg	1.50	1.05	1.01	1.84	1.30	2.01	6.06	1.97	1.80	1.14	1.82	1.20
Al	%	5.39	5.17	5.89	6.98	4.86	6.13	8.28	5.47	5.75	5.71	6.92	5.77
As	mg/kg	44.0	29.3	31.6	127.0	64.1	40.6	116.5	41.7	91.0	51.2	183.5	86.9
Ba	mg/kg	260	20	150	1,330	190	20	1,880	230	60	360	1,380	120
Be	mg/kg	4.14	4.74	4.95	2.18	3.42	4.75	5.40	3.39	3.53	2.36	2.65	5.58
Bi	mg/kg	0.10	0.02	0.12	0.11	0.14	0.10	0.13	0.08	0.08	0.08	0.15	0.10
Ca	%	0.16	0.07	0.16	0.77	0.06	0.16	0.99	0.14	0.15	0.21	0.60	0.17
Cd	mg/kg	0.09	0.02	0.03	0.08	0.09	0.09	0.10	0.09	0.06	0.03	0.10	0.14
Ce	mg/kg	62.6	40.0	58.5	190.5	43.6	48.4	222	65.9	46.7	66.4	173.5	62.6
Co	mg/kg	3.5	0.4	0.8	14.1	0.7	1.3	9.2	1.2	0.7	2.8	11.6	1.0
Cr	mg/kg	100	108	74	101	136	59	23	82	103	116	95	64
Cs	mg/kg	4.06	6.18	4.38	2.25	3.36	8.45	8.51	6.56	4.50	3.55	2.79	4.86
Cu	mg/kg	9.8	3.0	5.2	63.4	6.5	3.8	110.5	9.5	6.5	12.0	51.9	5.7
Fe	%	0.89	0.50	0.89	3.11	0.74	0.55	2.97	0.66	0.64	1.08	3.07	0.76
Ga	mg/kg	12.80	11.95	15.25	15.10	10.55	17.75	21.0	14.50	14.25	12.10	19.05	14.55
Ge	mg/kg	0.14	0.13	0.14	0.24	0.11	0.12	0.25	0.13	0.12	0.14	0.23	0.14
Hf	mg/kg	3.8	3.0	3.8	6.1	3.0	3.5	9.5	3.5	3.1	3.7	5.5	3.6
In	mg/kg	0.031	0.023	0.027	0.039	0.037	0.031	0.041	0.025	0.031	0.027	0.043	0.033
K	%	4.73	5.24	4.58	4.66	5.43	4.00	5.11	4.21	5.07	5.37	5.46	4.17
La	mg/kg	29.8	17.9	27.1	94.8	19.5	21.6	107.0	33.8	21.1	32.7	88.1	30.4
Li	mg/kg	85.2	56.7	44.2	34.6	104.5	76.7	22.4	52.9	89.4	42.7	32.2	46.7
Mg	%	0.11	0.05	0.17	0.69	0.04	0.56	0.41	0.20	0.14	0.16	0.36	0.22
Mn	mg/kg	480	71	175	319	98	282	403	198	139	175	290	183
Mo	mg/kg	4.51	3.53	3.41	5.41	8.53	32.3	3.30	5.63	17.70	5.66	6.81	4.60
Na	%	1.28	1.08	1.04	0.83	0.28	0.28	1.42	0.55	0.71	0.55	1.03	1.39
Nb	mg/kg	21.5	21.8	24.5	26.3	19.8	25.7	44.7	22.4	23.5	21.9	22.4	24.6
Ni	mg/kg	6.5	2.4	4.3	34.3	3.9	1.6	9.9	4.8	3.1	9.1	36.0	5.2
P	mg/kg	260	20	120	2,380	50	30	3,120	150	170	480	1,950	200
Pb	mg/kg	18.9	14.7	17.4	25.2	15.9	26.5	41.1	15.4	19.7	17.6	25.9	18.5
Rb	mg/kg	303	258	256	122.5	237	336	133.0	206	307	254	267	239
Re	mg/kg	<0.002	<0.002	<0.002	0.002	<0.002	<0.002	<0.002	<0.002	0.002	<0.002	<0.002	<0.002
S	%	0.01	<0.01	0.01	1.64	0.02	0.01	0.02	<0.01	0.03	0.02	0.01	0.05
Sb	mg/kg	10.60	14.30	7.98	6.97	23.5	8.56	12.50	7.70	14.15	9.20	11.35	8.73
Sc	mg/kg	1.7	1.0	1.5	6.5	1.0	1.4	3.2	1.4	1.5	2.6	8.0	1.9
Se	mg/kg	1	1	<1	2	1	<1	1	1	1	1	1	1
Sn	mg/kg	2.1	2.2	2.5	1.4	2.6	3.3	1.5	2.2	3.1	2.0	1.3	3.6
Sr	mg/kg	210	50.5	115.5	560	131.5	103.5	1,280	168.0	105.0	266	447	105.5
Ta	mg/kg	1.50	1.54	1.71	1.29	1.42	2.02	2.00	1.67	1.77	1.49	1.10	1.81
Te	mg/kg	<0.05	<0.05	<0.05	0.71	<0.05	<0.05	<0.05	<0.05	<0.05	<0.05	0.07	<0.05
Th	mg/kg	22.2	20.8	23.5	29.4	19.85	26.3	28.7	23.4	23.3	21.9	22.3	24.2
Ti	%	0.090	0.037	0.070	0.436	0.040	0.049	0.454	0.070	0.057	0.120	0.430	0.081
Tl	mg/kg	1.60	1.26	1.20	1.47	1.78	1.59	1.23	1.10	1.72	1.64	2.06	1.20
U	mg/kg	4.2	2.9	3.2	5.9	2.9	3.0	9.0	3.3	3.8	5.9	8.5	3.8
V	mg/kg	45	7	26	72	17	6	91	19	9	33	85	10
W	mg/kg	6.6	3.4	5.3	8.3	5.1	1.8	10.7	8.7	3.8	5.7	18.0	2.4
Y	mg/kg	25.6	22.3	24.2	16.9	21.2	27.2	17.0	24.3	22.5	20.4	17.6	22.6
Zn	mg/kg	33	17	37	69	17	31	114	36	23	28	62	51
Zr	mg/kg	115.0	73.8	103.0	319	75.1	81.9	470	92.8	79.6	115.5	246	97.7

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Low amounts of mercury have been observed in the composites sampled. On average, 2.3% of the mercury in the heap leach composites was extracted and loaded on the carbon. Mercury levels for both heap leach and mill grade ore composites are shown in Table 13-7 and Table 13-8 respectively. Mercury levels are minor and are not expected to be an issue.

Table 13-7: Mercury Analysis Results – Heap Leach Grade Ore Composites

Composite	Ore Zone	Lithology	g Hg/t ore
4505-001-LG	South Domes	Vx	0.422
4505-002-LG	South Domes	Rhy-Aph	1.04
4505-003-LG	South Domes	Diatreme	0.457
4505-007-LG	Main Pit	Andesite	1.30
4505-010-LG	Jumbo	Vx	0.055
4505-015-LG	East Ridge	Rhy-Aph	0.359
4505-017-LG	Oro Belle	Vx	0.668
4505-018-LG	Oro Belle	Rhy-Aph	0.642
4505-019-LG	Oro Belle	Andesite	1.46

Table 13-8: Mercury Analysis Results – Mill Grade Ore Composites

Composite	Ore Zone	Lithology	g Hg/t ore
4505-004-HG	South Domes	Vx	1.12
4505-005-HG	South Domes	Rhy-Aph	0.247
4505-006-HG	South Domes	Vx	0.537
4505-008-HG	JSLA	Diatreme	1.56
4505-009-HG	JSLA	Rhy-Bx	1.20
4505-011-HG	Jumbo	Rhy-Aph	0.239
4505-012-HG	Jumbo	Andesite	0.662
4505-013-HG	Jumbo	Rhy-Porph	0.488
4505-022-HG	Oro Belle	Rhy-Aph	0.841
4505-023-HG	Oro Belle	Diatreme	0.870
4505-024-HG	Oro Belle	Andesite	2.11
4505-026-HG	Oro Belle	Vx	0.312

13.5 HEAP LEACH TESTING

13.5.1 2015 ROM and Crushed Ore Column Testing

The 2015 column leach tests were performed in large columns (4 ft diameter by 20 ft tall) with ROM samples of -300 mm with 80% passing 50 mm as well as smaller columns at nominal 80% passing 9.5 mm. The samples were JSLA backfill material and in-situ ore from Oro Belle.

The ROM column leach tests had a gold recovery of 86% and 71% for Oro Belle and JSLA respectively (avg 78%) after leaching, rinsing, and draining for 174 days. The comparable column leach tests done on the same sample in parallel with the ROM tests at a crushed size of 9.5 mm had similar ultimate gold recoveries of 90% and 72%, as seen in Table 13-9. Gold extraction was progressing at a slow but significant rate from both ROM tests when leaching was terminated suggesting gold recovery would be the same given sufficiently long leach time, as shown in Figure 13-9. Notably, the higher-grade sample from JSLA was above crossover grade and is considered material that should be mill feed. This sample had lower recoveries in both tests and the tails assays had spotty residual gold which may indicate presence of unleached coarse gold particles.

Table 13-9: Summary of Metallurgical Results, ROM and Crushed Ore Column Tests, 2015

Sample	Feed Size	Leach Time, days	Test Type	Au Rec., %	gAu/mt ore Ext'd.	Tail	Calc'd. Head	Avg. Head	Ag Rec., %	gAg/mt ore Ext'd.	Tail	Calc'd. Head	Avg. Head	Reagent Req., kg/mt ore NaCN Cons.	Lime Added
Oro Belle	ROM	160	CLT	85.8	1.03	0.17	1.20	1.11	11.8	0.2	1.5	1.7	1.6	0.27	0.9
Oro Belle	80%-9.5mm	157	CLT	89.6	1.20	0.14	1.34	1.11	11.1	0.2	1.6	1.8	1.6	1.51	1.1
Oro Belle	80%-1.7mm	4	BRT	84.8	1.34	0.24	1.58	1.11	17.6	0.3	1.4	1.7	1.6	<0.07	1.4
JSLA	ROM	164	CLT	71.0	1.49	0.61	2.10	2.06	22.2	0.6	2.1	2.7	2.3	0.28	0.8
JSLA	80%-9.5mm	157	CLT	72.0	1.21	0.47	1.68	2.06	20.8	0.5	1.9	2.4	2.3	1.71	1.0
JSLA	80%-1.7mm	4	BRT	69.7	2.28	0.99	3.27	2.06	34.5	1.0	1.9	2.9	2.3	<0.07	1.2

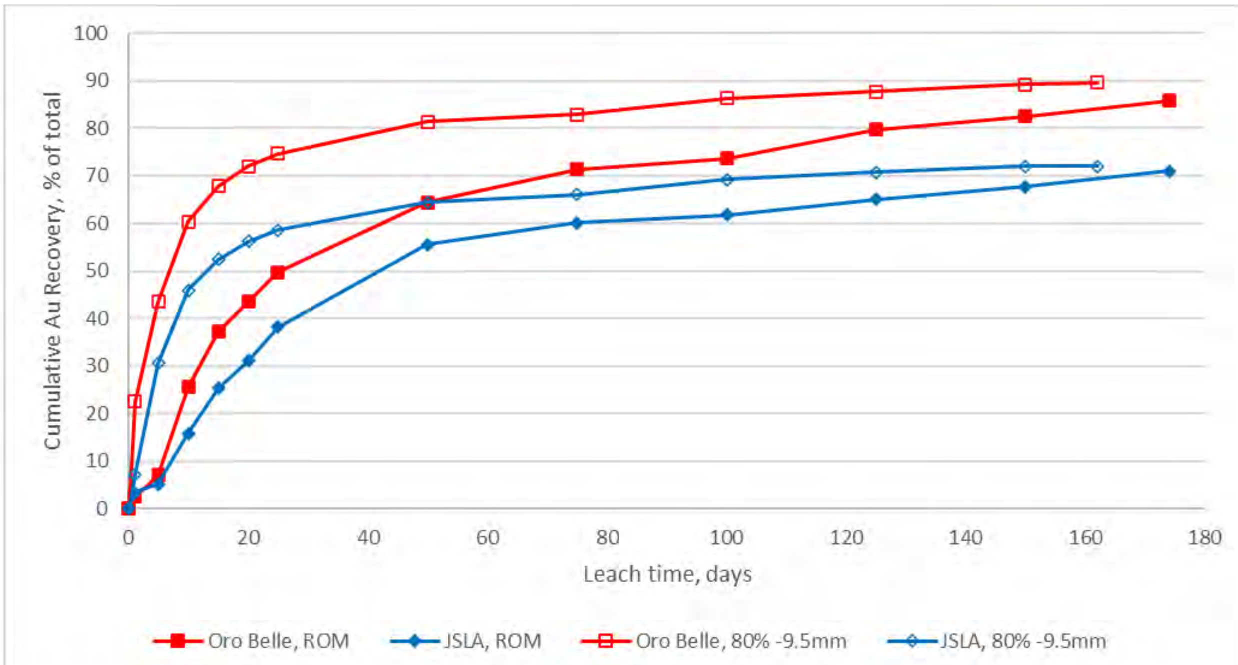


Figure 13-9: ROM and Crushed Sample Leach Curves, 2015

13.5.2 2018 ROM Column Leach and Crushed Ore Bottle Roll Testing

The 2018 ROM column leach tests were again performed in large columns with JSLA backfill ROM samples of -300 mm with 80% passing 50 mm as well as bottle roll tests at nominal 80% passing 50 mm and 9.5 mm. Column tests on crushed ore were not conducted in parallel in this campaign.

The ROM column leach test gold recovery ranged from 66% to 85% with an average gold recovery of 77% after leaching, rinsing, and draining for 173 days. The two tests with lower recovery were lower grade samples at 0.17 g/t and 0.32 g/t. See the summary metallurgical results in Table 13-10 and leach curves in Figure 13-10.

These tests are helpful to predict ROM heap leach recovery, but it is difficult to use these tests as a comparison of ROM recovery to crushed ore recovery. The four ROM column results averaged 77% recovery. The bottle roll tests did not correlate well to the ROM columns. The corresponding bottle roll tests at -50 mm and -9.5 mm averaged 63% and 72%. The results indicate the bottle roll tests at -9.5 mm may underestimate ROM heap leach recoveries by up to 13% and on average by 5%.

It is recommended to conduct additional ROM testing on in-situ ore paired with 9.5 mm, 25 mm, and 50 mm columns to validate optimal size for heap leach.

Table 13-10: Summary Metallurgical Results, ROM, and Crushed Ore Column Tests, 2018

Sample	Test	Feed Size	Test No.	Leach Time, Days	Au Rec., %	gAu/mt ore Ext'd.	Tail	Calc'd. Head	Avg. Head	Reagent Requirements, kg/mt ore NaCN Cons.	Lime Added
RAB 17-083	Column	ROM (-460mm)	173	P-4	84.6	0.33	0.06	0.39	0.41	0.22	1.3
RAB 17-083	Bucket*	ROM (-460mm)	141	V-4	89.6	0.43	0.05	0.48	0.41	2.04	1.3
RAB 17-083	Bucket*	-150mm	141	V-8	87.5	0.63	0.09	0.72	0.41	1.67	1.3
RAB 17-083	BRT	100%-50mm	4	CY-4	72.2	0.26	0.10	0.36	0.41	0.07	1.2
RAB 17-083	BRT	80%-9.5mm	4	CY-8R	71.4	0.30	0.12	0.42	0.41	<0.07	1.5
RAB 17-098	Column	ROM (-460mm)	173	P-1	65.5	0.19	0.10	0.29	0.32	0.32	1.4
RAB 17-098	Bucket*	ROM (-460mm)	141	V-1	33.3	0.29	0.58	0.87	0.32	1.98	1.4
RAB 17-098	Bucket*	-150mm	141	V-5	60.5	0.23	0.15	0.38	0.32	1.34	1.4
RAB 17-098	BRT	100%-50mm	4	CY-1	55.0	0.11	0.09	0.20	0.32	<0.07	1.1
RAB 17-098	BRT	80%-9.5mm	4	CY-5	63.2	0.12	0.07	0.19	0.32	<0.07	1.7
RAB 17-123	Column	ROM (-460mm)	173	P-3	85.1	0.40	0.07	0.47	0.52	0.27	1.8
RAB 17-123	Bucket*	ROM (-460mm)	141	V-3	86.5	0.45	0.07	0.52	0.52	1.81	1.8
RAB 17-123	Bucket*	-150mm	141	V-7	86.5	0.64	0.10	0.74	0.52	1.78	1.8
RAB 17-123	BRT	100%-50mm	4	CY-3	63.9	0.39	0.22	0.61	0.52	0.15	1.9
RAB 17-123	BRT	80%-9.5mm	4	CY-7	75.4	0.43	0.14	0.57	0.52	<0.07	2.2
RAB 17-146	Column	ROM (-460mm)	173	P-2	71.4	0.15	0.06	0.21	0.17	0.23	1.1
RAB 17-146	Bucket*	ROM (-460mm)	141	V-2	65.4	0.17	0.09	0.26	0.17	1.47	1.1
RAB 17-146	Bucket*	-150mm	141	V-6	80.8	0.21	0.05	0.26	0.17	1.29	1.1
RAB 17-146	BRT	100%-50mm	4	CY-2	61.5	0.08	0.05	0.13	0.17	<0.07	0.8
RAB 17-146	BRT	80%-9.5mm	4	CY-6	76.9	0.10	0.03	0.13	0.17	<0.07	1.4

* Flooded static bucket/barrel leach test.
Note: Column leach cyanide consumption calculated based on balance through the end of the leach cycle (before rinsing).

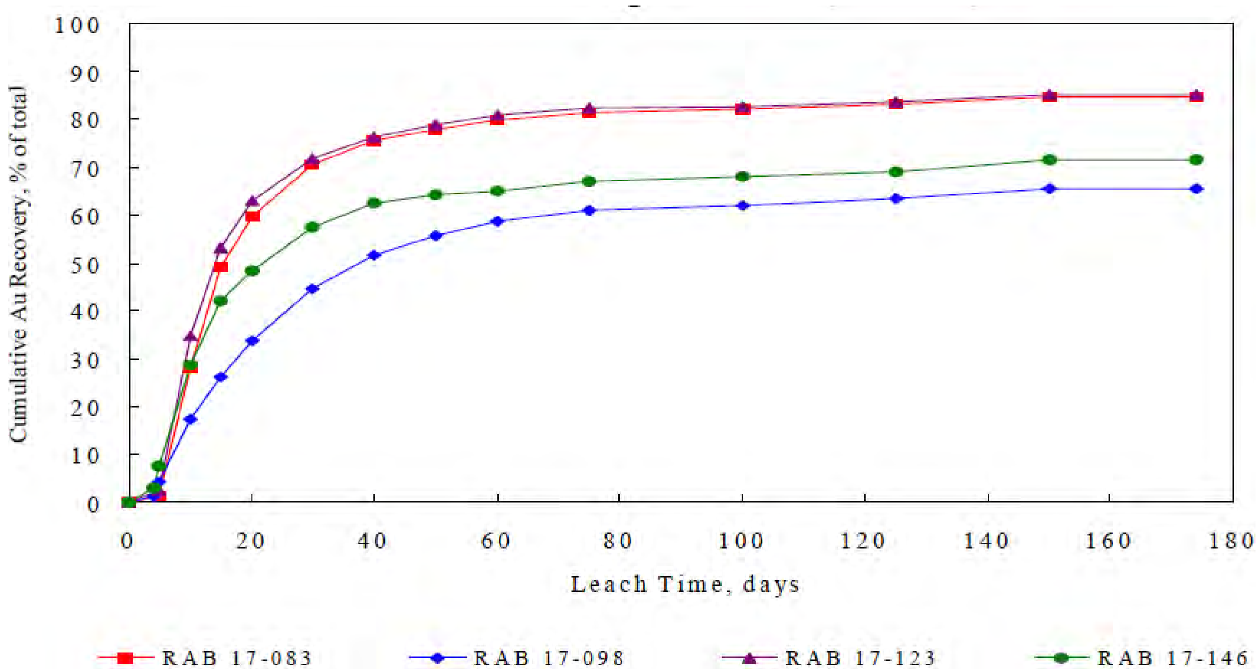


Figure 13-10: JSLA ROM Column Test Leach Curves, 2018

13.5.3 2020 Testing

The test program was performed by MLI and included bottle roll tests and column tests on 12 composites. The 96-hour bottle roll tests were completed using 80% minus 1.7 mm material to determine lime requirements for column leach tests and check direct head assay results against bottle roll test calculated head grades.

Comparative column leach tests were conducted on all composites and on one additional heap leach master composite. The blend of the heap leach master composite consisted of 24% South Domes, 19% East Ridge, and 57% Oro Belle zones.

Tests were planned on 50 mm feeds and 9.5 mm feeds. Composites were made-up from half HQ drill core that was crushed slightly finer than 100% minus 50 mm to ensure fresh, broken surfaces for leaching. Where sufficient material was available, the composites (8) were subjected to column leach testing at an approximately 80% passing 25 mm to determine sensitivity to crush size and more closely approximate ROM. Tests were also completed on material with a particle size of 80% passing 9.5 mm to provide direct comparison to previous testing and commercial production from the Castle Mountain Project. More tests were performed using 9.5 mm material as they require less material.

Table 13-11 shows the number of tests done by ore zone.

Table 13-11: Summary of 2020 Heap Leach Material Testing

Ore Zone	Bottle Roll Tests	Column Leach Tests	
	1.7 mm	9.5 mm	25 mm
JSLA	1	1	1
South Domes	3	3	2
East Ridge	1	1	1
Jumbo	1	1	-
Oro Belle	6	6	4
Heap Leach Master Composite	-	-	1
Total	**12**	**12**	**9**

13.5.3.1 Heap Leach 2020 Bottle Roll Test Results

Table 13-12 is a summary of the bottle roll test results for the heap leach grade ore composites. Detailed results can be found in MLI Report Job No. 4505, June 2020. The Epiclastic lithology had the lowest recovery and Vx had the highest recovery.

Table 13-12: Summary of Bottle Roll Tests on Low-Grade Composites

Composite	Ore Zone	Lithology	Gold Rec %	Gold Calc Head g /t Au	NaCN kg/t ore	Lime kg/t ore
4505-001-LG	South Domes	Vx	92.9	0.85	<0.07	1.6
4505-002-LG	South Domes	Rhy-Aph	90.0	0.30	<0.07	0.9
4505-003-LG	South Domes	VxDiatreme	88.2	0.34	<0.07	1.4
4505-007-LG	JSLA	Andesite	73.5	0.34	0.18	1.5
4505-010-LG	Jumbo	Vx	94.1	<0.17	0.09	1.3
4505-015-LG	East Ridge	Rhy-Aph	71.4	0.14	<0.07	1.0
4505-017-LG	Oro Belle	Vx	70.0	0.20	<0.07	1.5
4505-018-LG	Oro Belle	Rhy-Aph	80.8	0.52	<0.07	1.4
4505-019-LG	Oro Belle	Andesite	91.5	0.47	<0.07	2.4
4505-020-LG	Oro Belle	Vx	52.6	0.19	<0.07	1.2
4505-021-LG	Oro Belle	Rhy-Aph	76.7	0.30	<0.07	1.0
4505-025-LG	Oro Belle	Epiclastic	40.0	0.20	0.16	0.8

Calculated head grade versus gold recovery was plotted. There is a trend with higher head grade generally resulting in higher recovery as shown in Figure 13-11; however, it is not possible to curve fit this data. Recoveries varied for samples with different head grades and for samples with similar head grades.

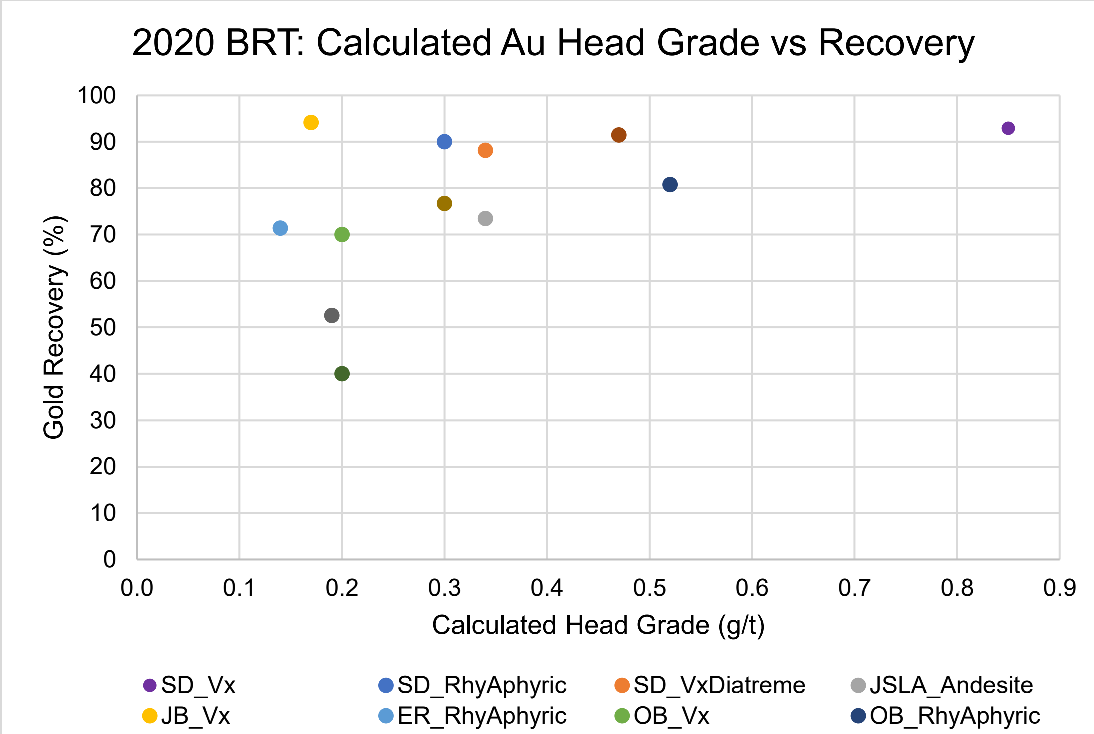


Figure 13-11: 2020 Bottle Roll Tests – Calculated Gold Head Grade vs Recovery

The average gold and silver recovery from these tests were 77% and 22% respectively. The Oro Belle deposit had the most variation compared to the other ore zones with gold recoveries ranging from 40% to 92%. Table 13-13 lists the average bottle roll test results by ore zone.

Table 13-13: Average Bottle Roll Test Results by Ore Zone

Ore Zone	Gold Recovery (%)	Silver Recovery (%)
JSLA	74	33
South Domes (SD)	90	17
East Ridge (ER)	71	20
Jumbo (JB)	94	20
Oro Belle (OB)	69	23

Figure 13-12 shows the gold recovery for each ore zone by lithology. Gold recovery for South Domes was consistent across the lithologies tested. Recovery for Oro Belle did vary based on lithology. Recovery for Vx varied for the Oro Belle but was consistent for South Domes and Jumbo. Recovery for the two tests done on Vx from the Oro Belle varied from 53% to 70%. The composites for these two tests were from different drill holes taken from similar depths. Results of the Vx tests for South Domes and Jumbo had a gold recovery of 93% and 94%, respectively. Epiclastic had the lowest recovery. Figure 13-13 shows the variation in recovery by lithology from the 2020 test. The dash line represents the arithmetic average from all 12 tests, being 77%.

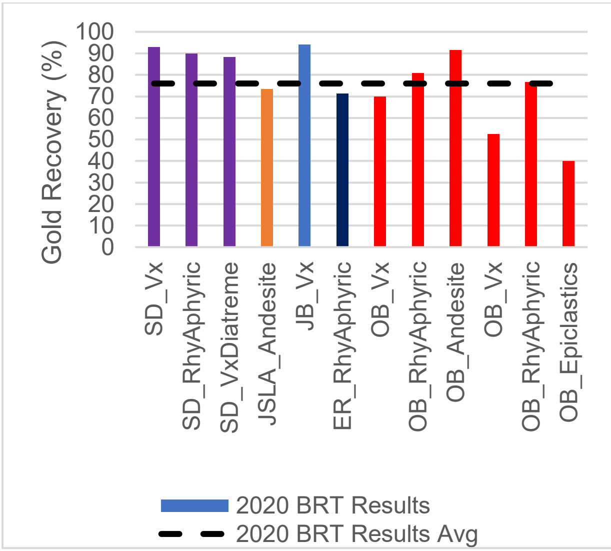


Figure 13-12: Gold Recovery for Each Ore Zone by Lithology

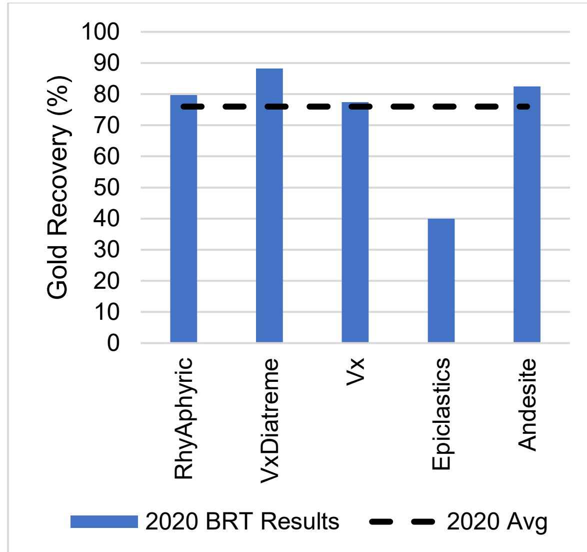


Figure 13-13: Arithmetic Average Gold Recoveries by Lithology

All bottle roll tests were done at 40% solids by weight with a targeted cyanide concentration of 0.1% NaCN (1 g NaCN/L). Lime was added to maintain the pH between 10 and 11. The average consumption for cyanide was 0.18 lb/ton ore (0.09 kg/t ore). Lime consumption was an average of 2.7 lb/ton ore (1.3 kg/t ore). There was no consistent relationship between reagent consumption and head grade or recovery.

13.5.3.2 Heap Leach 2020 Column Leach Test Results

The summary of column leach tests is shown below in Table 13-14 and includes 12 tests at a nominal 9.5 mm size and eight tests at a nominal 25 mm size.

Table 13-14: Summary of Column Tests on Heap Leach Composites

Composite	Ore Zone	Lithology	Size mm	Gold Rec %	Gold Calc Head g/t Au	Gold Extracted g/t Au	Gold in Tails g/t Au	NaCN kg/t ore	Lime kg/t ore
4505-001-LG	South Domes	Vx	24	87.0	1.46	1.27	0.19	1.51	1.3
			9.5	96.2	1.32	1.27	0.05	2.66	1.3
4505-002-LG	South Domes	Rhy-Aph	25	85.3	0.34	0.29	0.05	0.72	0.9
			9.5	88.2	0.34	0.30	0.04	0.84	0.9
4505-003-LG	South Domes	VxDiatreme	9.5	89.2	0.37	0.33	0.04	1.27	1.1
4505-007-LG	JSLA	Andesite	25	64.9	0.33	0.24	0.13	1.01	1.2
			9.5	69.7	0.33	0.23	0.10	1.14	1.2
4505-010-LG	Jumbo	Vx	9.5	87.5	0.24	0.21	0.03	0.89	1.0
4505-015-LG	East Ridge	Rhy-Aph	22	66.7	0.15	0.10	0.05	0.74	1.0
			9.5	73.3	0.15	0.11	0.04	1.13	1.0
4505-017-LG	Oro Belle	Vx	22	71.4	0.21	0.15	0.06	0.95	1.2
			9.5	66.7	0.21	0.14	0.07	1.20	1.2
4505-018-LG	Oro Belle	Rhy-Aph	24	83.7	0.47	0.36	0.07	0.76	1.1
			9.5	88.9	0.47	0.40	0.05	1.24	1.1
4505-019-LG	Oro Belle	Andesite	9.5	95.2	0.44	0.40	0.02	0.94	1.9
4505-020-LG	Oro Belle	Vx	23	37.5	0.16	0.06	0.10	0.83	1.0
			9.5	41.2	0.17	0.07	0.10	1.16	1.0
4505-021-LG	Oro Belle	Rhy-Aph	24	78.6	0.28	0.22	0.06	0.82	1.0
			9.5	82.1	0.28	0.23	0.05	1.10	1.0
4505-025-LG	Oro Belle	Epiclastic	9.5	36.8	0.19	0.07	0.12	0.96	0.8

The average gold recovery for the 9.5 mm and 25 mm column tests was 77% and 74% respectively. Recovery from the 9.5 mm columns ranged from 37% to 96% and from 38% to 87% for the 25 mm columns. Gold recovery between sizes for each of the zones tested seems to indicate that 25 mm columns had higher recovery than 9.5 mm columns, but the tail assays show that the residual gold is within 0.01 g/t for 5 of the column pairs which is within the repeatability of the assay measurement and these differences are not significant.

There were two notable samples with poor recovery, one Oro Belle Vx and one Oro Belle epiclastic, and these matched closely with the BRT results. These were two of the lowest grade samples tested, both <0.2 g/t. The remainder of the samples had recoveries of 65% or above and averaged 81% recovery. 4505-001-LG and 4505-020-LG samples were selected for further mineralogical analysis and gold deportment study to explain the difference in recovery between Vx in South Domes and Vx in Oro Belle and is discussed in Section 13.8.6.

The results have been grouped by ore zone and lithology to examine specific trends. Table 13-15 and Table 13-16 list the results by size and ore zone. The column leach test results on the heap leach master composite at 25 mm had a gold recovery of 76.5%.

Table 13-15: 2020 Column Leach Test Results by Size and Ore Zone

No. of Tests	Ore Zone	Nominal Size mm	Avg Gold Recovery %
1	JSLA	9.5	70
1	JSLA	25	65
1	Jumbo	9.5	88
6	Oro Belle	9.5	68
4	Oro Belle	25	68
1	East Ridge	9.5	73.3
1	East Ridge	25	66.7
3	South Domes	9.5	91
2	South Domes	25	86
1	Heap Leach Master Composite	25	76.5

Gold recovery by lithology ranged from 37% to 95% with an average of 75%. The lithology with the lowest recovery was Epiclastic which had a calculated head grade below the criteria specified above for evaluation. Table 13-16 lists the results by lithology.

Table 13-16: 2020 Column Leach Test Results Summary

No. of Tests	Lithology	Gold Recovery (%)
8	RhyAphyric	84
1	VxDiatreme	89
7	Vx	70
1	Epiclastic	37
3	Andesite	77

Cyanide concentration for the column leach tests was maintained at a concentration of 0.1% (1,000 ppm) and the average consumption was 2.0 lb/ton ore (1.0 kg/t ore). Lime addition to maintain the pH between 10 – 11 was an average of 2.24 lb/ton ore (1.3 kg/t ore). No correlation is apparent between sodium cyanide consumption and calculated head grade or gold recovery.

13.5.3.3 ICP Analysis on Pregnant Solution

Multi-element analyses using ICP were done on the pregnant solution for the heap leach (column tests) and mill (gravity plus tails leach) composite tests. Two heap leach composites out of 12 had higher levels of copper than the other composites. These two samples were from the JSLA and Oro Belle ore zones with copper levels of 66 ppm and 62 ppm, respectively.

One mill grade ore composite out of the 12 had a higher level of copper. Analysis shows this composite with a copper level of 31 ppm while the rest of the samples had levels below 4 ppm. Copper can be concerning because at high levels it may increase cyanide consumption, decrease gold recovery, or impact carbon loading/movement. However, these levels are low enough in all cases to cause minimal concern.

13.5.3.4 Physical Ore Characteristics

Saturation and retained moisture of the heap leach grade ore samples are shown in Table 13-17. For the 9.5 mm crushed material, saturation moisture ranged from 12% to 25% and the moisture retained ranged from 9% to 19%. The Oro Belle andesite sample had the highest saturation (25%) out of all the samples. The respective sample at a coarser size had a saturation of 18%. As expected, the coarser material had both a lower saturation moisture and a lower retained moisture. The saturation moisture for the 25 mm material ranged from 8.7% to 13.7% averaging 11.4% while the retained moisture ranged from 6.4% to 10.8%, averaging 8.9%. The heap leach master composite saturation moisture was 15.2% while the retained moisture was 12.3%. The ROM samples tested in 2018 had an average saturation moisture of 12.5% and an average retained (drain down) moisture of 9.7%.

Table 13-17: Heap Leach Grade Ore Sample Saturation and Retained Moisture

Composite	Ore Zone	Lithology	Feed Size	Moisture, wt. %		-200 mesh content, %
			mm	To Saturate	Retained	
4505-001-LG	South Domes	Vx	25	8.8	6.8	1.9
4505-007-LG	JSLA	Andesite	25	8.7	6.4	3.2
4505-002-LG	South Domes	Rhy-Aph	25	10.0	8.2	2.4
4505-015-LG	East Ridge	Rhy-Aph	25	13.0	10.2	3.1
4505-001-LG	South Domes	Vx	9.5	14.6	11.7	5.6
4505-002-LG	South Domes	Rhy-Aph	9.5	11.9	8.9	6.2
4505-003-LG	South Domes	Diatreme	9.5	16.9	12.8	7.3
4505-007-LG	JSLA	Andesite	9.5	15.0	11.5	8.5
4505-010-LG	Jumbo	Vx	9.5	24.3	18.5	17.3
4505-015-LG	East Ridge	Rhy-Aph	9.5	17.3	13.1	6.6
4505-021-LG	Oro Belle	Rhy-Aph	25	11.3	9.0	3.1
4505-017-LG	Oro Belle	Vx	9.5	13.7	10.8	9.9
4505-018-LG	Oro Belle	Rhy-Aph	9.5	13.0	9.8	5.4
4505-020-LG	Oro Belle	Vx	9.5	12.9	10.0	6.8
4505-017-LG	Oro Belle	Vx	9.5	22.3	17.5	12
4505-018-LG	Oro Belle	Rhy-Aph	9.5	21.1	16.4	9.9
4505-019-LG	Oro Belle	Andesite	9.5	24.5	18.2	12.7
4505-020-LG	Oro Belle	Vx	9.5	20.0	15.8	10.4
4505-021-LG	Oro Belle	Rhy-Aph	9.5	18.8	14.7	9
4505-025-LG	Oro Belle	Epiclastic	9.5	16.2	12.5	7.2
4505 LG MC	Blend	Heap Leach Master Composite	25	15.2	12.3	7.9

13.5.3.5 Fines and Saturation

Tails screen analysis was done on residue from each of the column leach tests. The quantity of fines (material finer than 200 mesh) was compared to the saturation by sample tested, lithology and ore zone. As expected, the saturation moisture results correlated closely with the fines content of the samples tested. Samples from the Jumbo ore zone had the most fines and the highest average level of saturation, followed by Oro Belle. The saturation for Jumbo and Oro Belle was on average 24% and 17%, respectively. Figure 13-14 shows the average saturation and fines for each ore zone. No ponding was observed from any of the column tests.

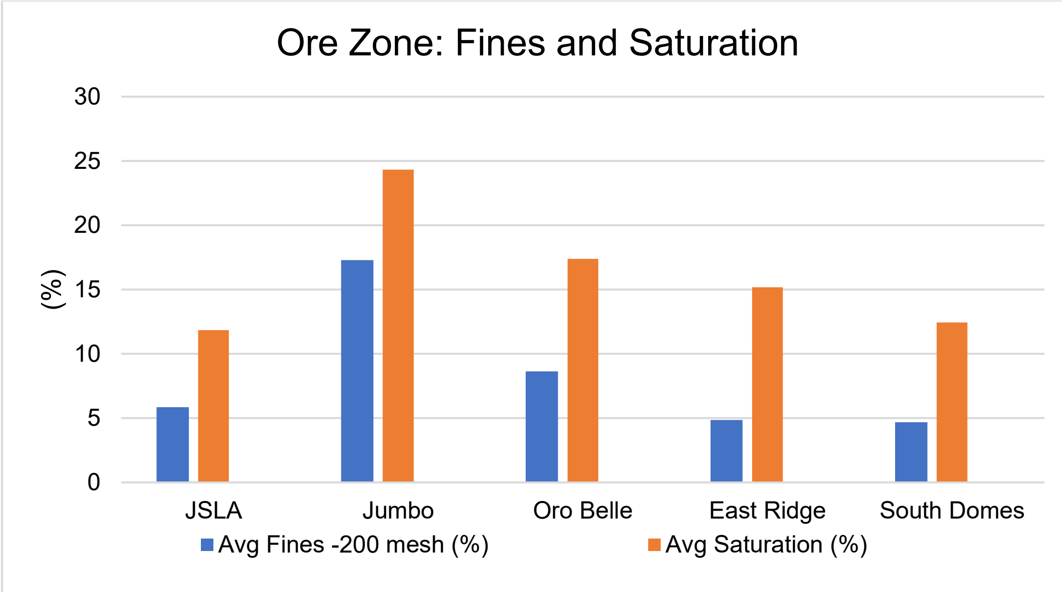


Figure 13-14: Ore Zone: Fines and Saturation

The lithology with the most fines and highest saturation was Vx followed by Andesite. Figure 13-15 lists the average saturation and fines for each lithology.

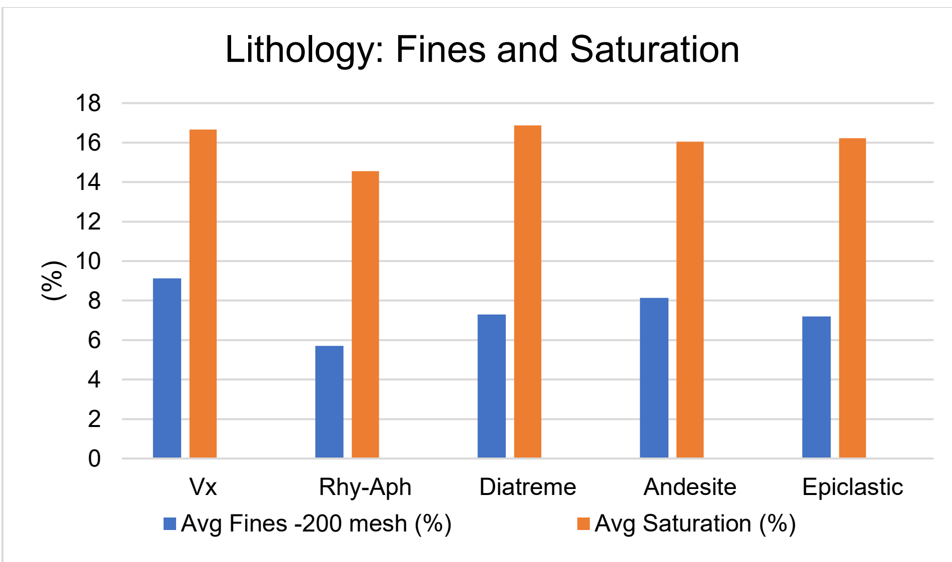


Figure 13-15: Lithology: Fines and Saturation

Based on the crushed samples tested, Jumbo ore zone with the Vx and Diatreme lithology is estimated to have a high quantity of fines and saturation requirement. The water balance may be impacted when the material with high fines and saturation is placed on the pad.

13.5.3.6 Tail Screen Assays by Size Fraction

Tail screen analysis was done on the leach residue for all column tests. The average fines in the 200 mesh (75 µm) was 7% with the respective gold distribution averaging 3%. Overall, the fraction of gold decreased with the decreasing particle size indicating most of the gold was associated with a coarser size or that the fines fraction leached preferentially.

13.5.3.7 Cyanide Consumption

Cyanide consumption for the column tests used to estimate the gold recovery range from 0.5 to 4.8 lb/ton ore (0.25 to 2.4 kg/t ore) with the average being 2.0 lb/ton ore (1.0 kg/t ore). Samples with a head grade below 0.02 oz/ton (0.684 g/t ore) had a consumption less than 3 lb/ton (1.5 kg/t) and samples above a head grade of 0.02 oz/ton (0.684 g/t ore) had a consumption greater than 3 lb/ton ore (1.5 kg/t ore). Lithology had no effect on cyanide consumption as seen in Figure 13-16.

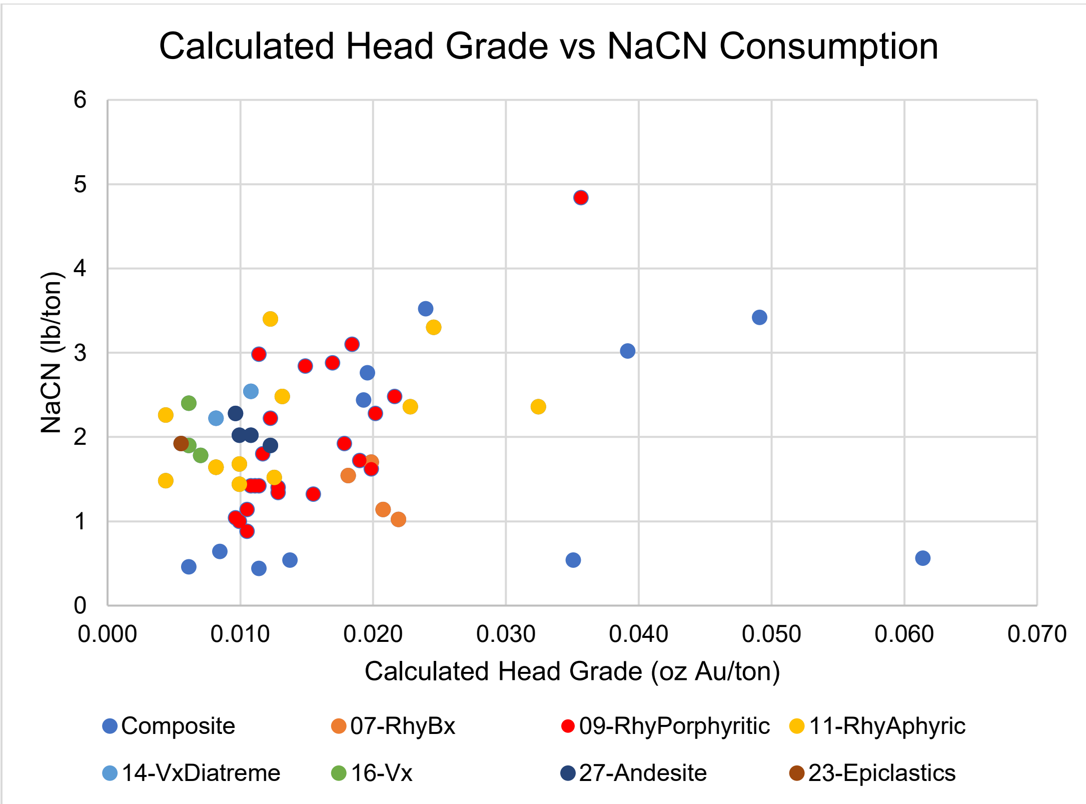


Figure 13-16: NaCN Consumption by Gold Head Grade

The average cyanide consumption weighted by the respective LOM tons in each ore zone yields a consumption of 1.9 lb/ton ore (0.95 kg/t ore). Field cyanide consumptions are expected to be lower than those measured in the lab. A typical factor used in industry of 30% was applied to obtain a cyanide consumption of 0.6 lb/ton ore (0.3 kg/t ore). This also benchmarks well against similar operations.

13.5.4 Heap Leach Ore Gold Recovery

The ROM heap leach recovery estimate for the feasibility study is based upon the testwork from 2015 through 2020. All heap leach grade column tests were evaluated by feed size, ore zone and lithology.

In order to strictly evaluate heap leach recovery, only samples with a calculated head grade between 0.006 – 0.036 oz/ton (0.2 – 1.25 g/t) were included, unless noted otherwise, and used to estimate a recovery.

After applying the above criteria, 53 tests fell within the heap leach grade criteria. Three tests with a head grade greater than 0.036 oz/ton and three tests with a head grade below 0.006 oz/ton were included, for a total of 59 tests that were used to determine gold recovery. The three tests above the cut-off grade were included since they were done concurrently with ROM column tests. The three tests below the cut-off grade were included since the they represented samples from an ore zone (East Ridge) or lithology (epiclastic) not previously tested. A total of 59 tests were evaluated and used to determine the gold recovery.

13.5.4.1 Average Recovery by Feed Size

Column leach test results were evaluated by feed size. The results indicated slight variation in gold recovery between the different feed sizes as shown in Table 13-18. The arithmetic average gold recovery of the column leach tests that met the calculated head grade was 81.5%.

Table 13-18: Average Gold Recovery Summary

Nominal Feed Size (mm)	Avg Gold Recovery (%)	No. of Tests
9.5	83	32
19	80	7
25	78	7
50	80	7
Arithmetic Avg Recovery	**81.5**	**53**

13.5.4.2 Average Recovery by Ore Zone

The test results were also reviewed by ore zone. Results for the East Ridge ore zone are included even though samples had a calculated head grade below 0.006 oz/ton.

Column leach test results indicated no significant difference between gold recovery and size. The average gold recovery by size for each ore zone is shown in Figure 13-17. The gold recovery for the Jumbo ore zone was consistent for all column tests (approximately 85%). The ROM column leach test done on Oro Belle had a gold recovery of 86%. The Oro Belle column tests performed at a finer size had a slightly lower gold recovery than the ROM test. The tests done on the JSLA backfill material is shown for comparison. The JSLA backfill material is previously mined material from the JSLA ore deposit which is not part of Phase 2.

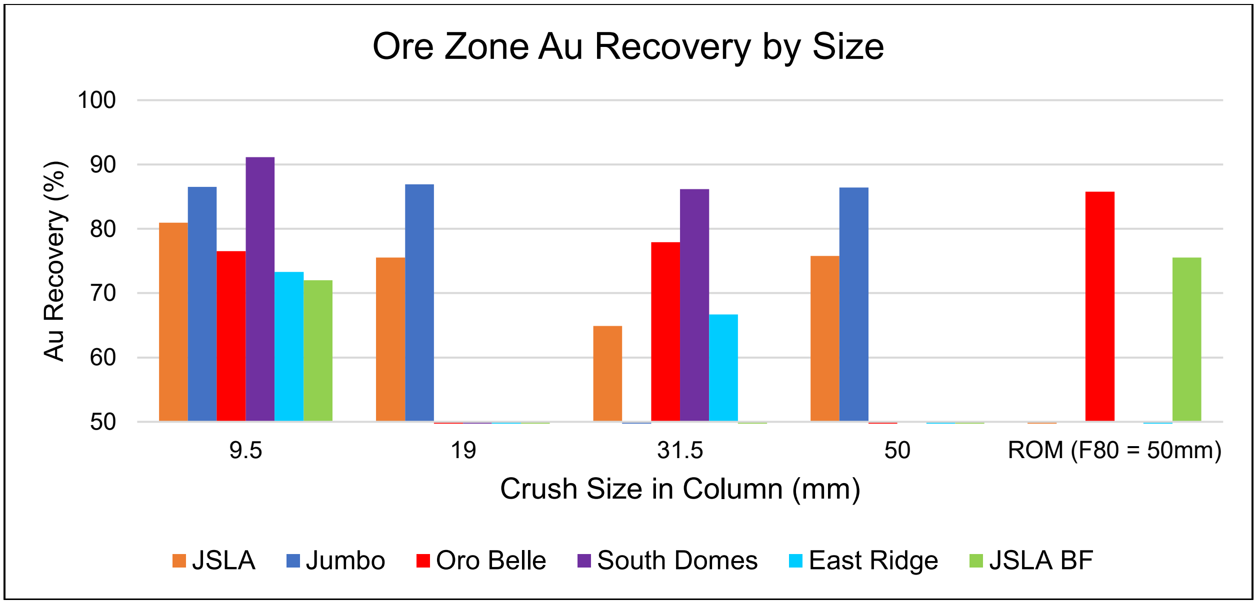


Figure 13-17: Ore Zone Gold Recovery by Size

13.5.4.3 Weighted Average Recovery by Lithology

The column leach test results by lithology are plotted in a box and whisker plot as shown in Figure 13-18. As seen, lithologies RhyAphyric and Andesite had the greatest gold recovery distribution.

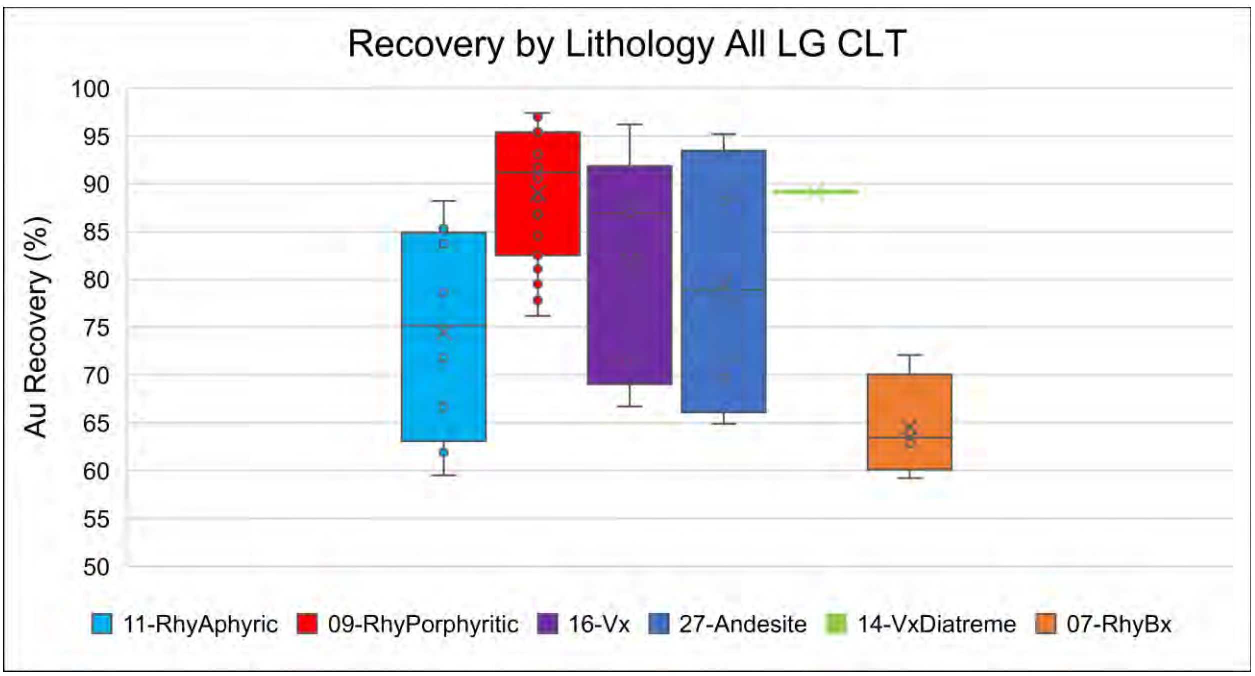


Figure 13-18: Recovery by Lithology All LG CLT

The average gold recovery for the lithologies ranged from 37% to 89%. Lithologies RhyAphyric, RhyPorphyritic and Vx which account for 86% of the ounces, had a gold recovery of 77%, 89% and 75%, respectively. Table 13-19 lists the percent of tons and ounces for each lithology and the average gold recovery from the column leach tests that met the calculated head assay criteria.

Table 13-19: Average Gold Recovery by Lithology

Lithology	Heap Leach Tons %	Heap Leach Ounces %	No. of Tests	Avg Gold Rec %
11-RhyAphyric	26.6	25.1	12	77
09-RhyPorphyritic	32.3	36.6	22	89
16-Vx	29.6	24.5	3	75
27-Andesite	2.1	1.8	4	80
14-VxDiatreme	4.8	6.9	2	89
23-Epiclastic	3.6	4.2	1	37
07-RhyBx	0.7	0.7	4	65
Total / Weighted Average	**99.7**	**99.8**	**48**	**80**

A weighted gold recovery, using the ounces associated with each lithology, was calculated to be 80%.

13.5.4.4 Calculated Head Grade by Gold Recovery

Gold recovery relationship to head grade is shown in Figure 13-19 for all 48 tests. While there was no clear relationship observed from this data, lower grade samples (<0.008 oz/ton) typically have lower than average recovery, while higher grade samples (>0.025 oz/ton) have higher than average recovery.

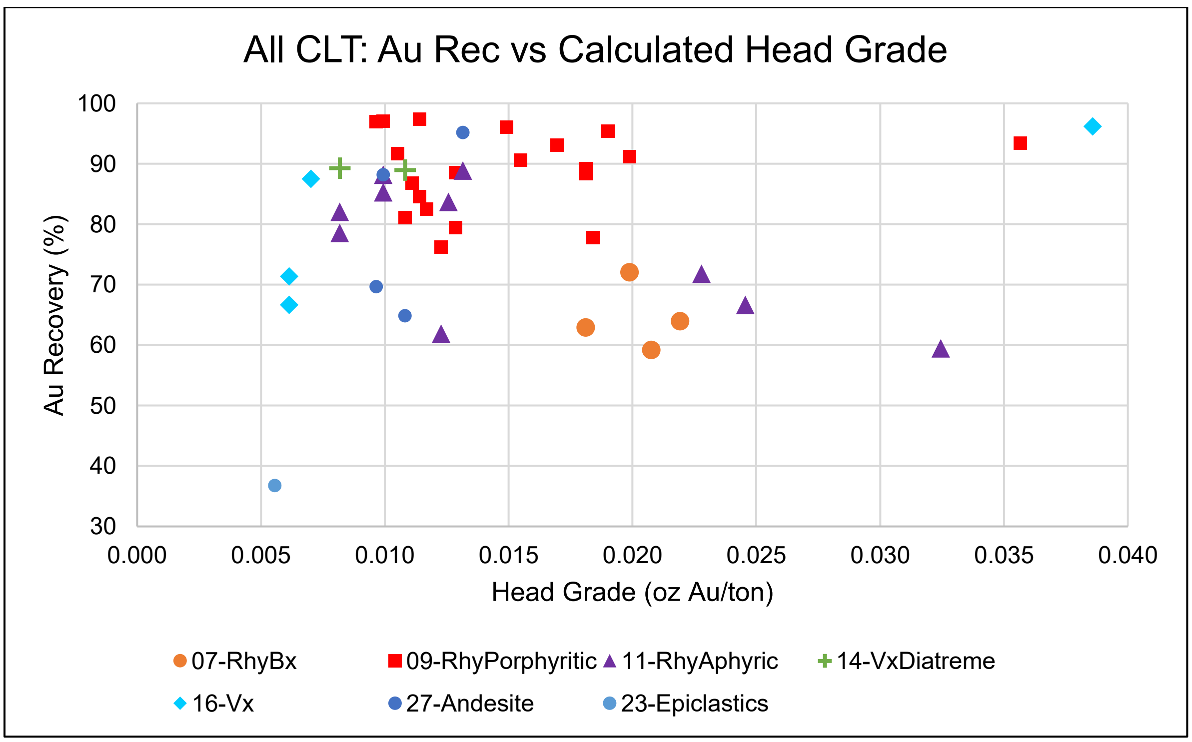


Figure 13-19: Calculated Head Grade by Gold Recovery

13.6 MILL GRADE ORE TESTING

13.6.1 2015 and 2018 Testing

In 2015, gravity plus gravity tail cyanidation testwork and whole ore leach tests were completed on six composites. All tests were done at 75 µm and showed gold recovery greater than 95%. The gravity recoverable gold varied from 18% to 87%, with a combined recovery for all samples above 95%. All whole ore leach tests had a recovery above 95%.

For the 2015 testwork, cyanide consumption for the gravity tail cyanidation tests averaged 1.4 lb/ton ore (0.70 kg/t ore). Lime consumption ranged from 2.8 to 5.4 lb/ton ore (1.40 to 2.70 kg/t) ore with the average being 4.7 lb/ton ore (2.35 kg/t ore). Cyanide consumption for the whole ore leach tests ranged from 0.30 to 0.90 lb/ton ore (0.15 to 0.45 kg/t ore) with the average being 0.45 lb/ton ore (0.22 kg/t ore). Lime consumption ranged from 3 to 5 lb/ton ore (1.50 to 2.50 kg/t ore with the average being 4 lb/ton ore (2 kg/t ore).

In 2018, gravity plus gravity tail cyanidation testwork and whole ore leach tests were completed on nine composites. All tests were done at 150 µm. The recovery to gravity varied from 16% to 46%, with a combined recovery above 95% for all samples. A tenth high grade sample was used for grind size optimization test with gravity, cyanidation, and comminution testing. The grind sizes evaluated were 80% minus 300, 212, 150 and 106 µm. Gold recoveries were all high and slightly improved with finer size. Cyanide consumption was less than 0.20 lb/ton ore (0.10 kg/t ore) for all tests. To further evaluate gravity plus tails leach versus whole ore leach additional testing was on three RC drill composites. Tests were done at 75 µm for 72 hours. The recoveries for all the tests after 72 hours were greater than 95%.

For the 2018 testwork, cyanide consumption for the gravity tail cyanidation tests ranged from 0.20 to 0.24 lb/ton ore (0.10 to 0.12 kg/t ore) with the average being 0.20 lb/ton ore (0.10 kg/t ore). Lime consumption ranged from 1.6 to 6.4 lb/ton ore (0.80 kg/t ore to 3.20 kg/t ore) with the average being 3.28 lb/ton ore (1.64 kg/t ore). Cyanide consumption for the whole ore leach tests ranged from 0.30 to 0.90 lb/ton ore (0.15 to 0.45 kg/t ore) with the average being 0.44 lb/ton ore (0.22 kg/t ore). Lime consumption ranged from 3 lb/ton ore (1.50 to 2.50 kg/t ore) with the average being 4 lb/ton ore (2.0 kg/t ore).

In 2018, CIL and gravity CIL tests were done on a single composite. All tests were done at 150 µm. Different times were evaluated to determine the retention time for CIL. For direct CIL testing, the gold recovery was 92% after 96 hours. For gravity and tail CIL testing, the combined recovery was 93% for 48 hours.

Test results from 2015 and 2018 are summarized along with 2020 results and are provided in Section 13.6.3.

13.6.1.1 Comminution

In 2015, FLS performed Bond abrasion index (Ai) and Bond crusher impact (CWi) testing on seven samples of rock types most commonly associated with mineralization at Castle Mountain. Samples were prepared to meet all testing standards and all work was conducted according to industry best practices. The results from these tests show the CWi and the Ai classify as Very Soft to Medium, as shown in Table 13-20. Results suggest relatively low required crusher energy and wear part replacements.

Table 13-20: Abrasion Index and Crusher Impact Test Results (2015 results)

Lithology	No. of Samples Tested	ID	Relative Density	Bond Abrasion Index (Ai)		Crusher Work Index (CWi)		
				grams	Classification	kW-hr/ton	kW-hr/t	Classification
Ash Tuff	20	3878-204	2.11	0.0115	Very Soft	9.70	10.7	Very Soft/Soft
Conglomerate Multi-Lithic	20	3878-203	2.13	0.2165	Medium	13.6	15.0	Soft/Medium
Rhyolite	20	3878-201	2.30	0.2978	Medium	13.90	15.30	Soft/Medium
Rhyolite-Breccia	20	3878-202	2.19	0.1602	Soft	13.9	15.40	Soft/Medium

13.6.1.2 Thickening

In October 2017, thickening testwork was conducted on leached residue from a JSLA mill grade ore sample and results were discussed in the PFS for the Castle Mountain Project (Scott et al., 2018).

13.6.2 2020 Testing

Twelve mill grade ore composites were selected and used for testing to further evaluate whole ore leach versus gravity concentration tests with agitated leaching of the gravity tails and confirm design parameters. The test program for mill grade ore includes:

- Head Assays
- Comminution Testing – Ai, RWi, BWi, SMC, JK Dropweight Test
- Grind Size/Recovery
- Gravity Recovery/Leach Tests
- Whole Ore Leach Tests
- Cyanide Detox Testing
- Solid and liquid separation
- Geotech testing

The calculated gold head grades of the composites ranged from 0.019 to 0.045 oz/ton (0.65 to 1.55 g/t ore). The expected gold grade to the mill is greater than 0.037 oz/ton (1.26 g/t ore), and the average LOM grade is 0.067 oz/ton (2.3 g/t ore).

13.6.2.1 Comminution

In 2020, FLS performed a series of comminution testwork which included seven (7) SMC tests® and one (1) JK Drop Weight test (DWT) (FLSmidth, 2020). On a full JK Drop-Weight test, ore is grouped into several different size fractions and tested at increasing energy levels. This test yields an illustration of the size-by-size rock strength for the ore being tested. Furthermore, a relationship is established between breakage and impact energy, which is described by the JK parameters A and b. The SMC Test uses a single size fraction at increasing energy levels. The size-by-size rock strength characteristics of an ore, that are determined by a full drop-weight test, are utilized to estimate the JK rock breakage parameters A and b.

FLSmidth received six (6) different samples for SMC testing in 2019. One (1) additional sample was received for one (1) additional SMC test® and one (1) JKTech Drop Weight test. The samples were prepared according to standard procedures and tested in accordance with the standard practices for these tests. Table 13-21 provides a summary of the tests which were conducted on the seven (7) samples.

In the case of the 4505-004-HG, 4505-005-HG, 4505-006-HG, 4505-009-HG, 4505-012-HG,4505-023-HG and 4505-CC-1 samples from Castle Mountain Project, the A and b estimates are based on the results of full drop-weight testing on similar ore (4505-CC-1). The full drop-weight test results were used to calibrate the DWi versus A and b correlations.

Table 13-21: SMC and Drop Weight Test Results

Description	Lithology	Zone	ID	DWi	DWi	M_{ia}	M_{ih}	M_{ic}	A	b	Axb	sg	ta	SCSE
	% tons			kWh/m³	%	kWh/t	kWh/t	kWh/t						kWh/t
Vx	49.2	South Domes	4505-004-HG	5.31	32	19.1	13.5	7.0	64.9	0.67	43.48	2.28	0.49	9.5
Rhy-Aph	24.3	South Domes	4505-005-HG	5.33	33	19.2	13.6	7.0	64.3	0.67	43.08	2.28	0.49	9.5
Vx	49.2	South Domes	4505-006-HG	4.66	25	17.5	12	6.2	67.9	0.76	51.60	2.24	0.56	9.2
Rhy-Bx	1.5	JSLA	4505-009-HG	6.15	43	20.9	15.2	7.9	68.1	0.56	38.14	2.35	0.42	9.9
Andesite	8	Jumbo	4505-012-HG	4.17	20	15.4	10.4	5.4	63.7	0.88	56.06	2.33	0.62	8.5
Diatreme	1	Oro Belle	4505-023-HG	5.47	34	19.1	13.5	7.0	**64.1**	**0.67**	**42.95**	2.34	0.48	9.4
-	-	-	4505-CC-1	4.45	23	16.0	10.9	5.6	64.1	0.84	53.84	2.37	0.59	8.6

Bolded values (80th percentile) were used in the JKSimMet software to determine power requirements for a SAG mill option.

In 2020, MLI performed crusher work index (CWi) testing on whole HQ core (4505-CC-1). Results to date are shown in Table 13-22 and indicate the interval was very soft.

Table 13-22: Crusher Work Index Test Result

Work Index, kWh/ton:	9.43
Work Index, kWh/ton:	10.40
Crusher Work Index Classification	Very Soft

In 2020, MLI also performed Bond abrasion index (Ai), Bond rod mill work index (RWi), and Bond ball mill work index (BWi) testing on six mill grade ore composites. Bond ball mill work index was performed by MLI on one JSLA master composite sample in 2015 and is shown for comparison. Results to date are shown in Table 13-23.

Table 13-23: Abrasion Index, Bond Rod & Ball Mill Work Index Test Results

Description	Lithology	Zone	ID	Abrasion Index (Ai)		Bond Ball Work Index (BWi)			Rod Mill Work Index (RWi)		
	% tons			grams	Classification	kW-hr/ton	kW-hr/t	Classification	kW-hr/ton	kW-hr/t	Classification
Vx	49.2	South Domes	4505-004-HG	0.3663	Abrasive	14.20	15.65	Medium	15.71	17.32	Hard
Rhy-Aph	24.3	South Domes	4505-005-HG	0.3651	Abrasive	14.13	15.58	Medium	11.67	12.87	Medium
Vx	49.2	South Domes	4505-006-HG	0.2192	Abrasive	13.78	15.20	Medium	12.40	13.67	Medium
Rhy-Bx	1.5	JSLA	4505-009-HG	0.7134	Very Abrasive	17.61	19.41	Hard	15.48	17.06	Hard
Andesite	8	Jumbo	4505-012-HG	0.1205	Moderately Abrasive	12.33	13.60	Medium	11.55	12.73	Medium
Diatreme	1	Oro Belle	4505-023-HG	0.4232	Very Abrasive	16.19	17.84	Hard	17.27	19.04	Hard
Vx; Rhy-Aph	49.2; 24.3	-	JSLA MC 4210-067	-	-	18.03	19.87	Hard	-	-	-

Test results based on abrasion index and crusher work index classify the rock as soft but abrasive. Based on the testwork, an abrasion index of 0.264 g was selected to be used in the design. Rod and ball mill work index results indicate the material is relatively hard. Based on the testwork, a rod mill work index of 14.0 kWh/ton and bond ball work index of 15.2 kWh/ton was selected for the design of the grinding circuit. These selected indices are a weighted average of the lithologies tested by the estimated percentage of tons they account for. Since JSLA ore is scheduled first in the mine production, high wear and energy consumption may be observed.

13.6.2.2 Whole Ore Leach and Gravity Tails Leach Tests

The summary of whole ore leach (WOL) and gravity tails leach (Grav/CN) tests are shown in Table 13-24 and include 12 test pairs at the following conditions:

- P_{80} of 150µm (100 mesh)
- Leached at the following conditions:
  - 96 hour leach time
  - 40% solids (w/w)
  - pH 10.5-11.0 maintained with hydrated lime
  - 0.5 g NaCN/L maintained

Note for Grav/CN tests the ground ore was first processed through a Knelson concentrator in a single pass before leaching the gravity tails.

Table 13-24: Summary of WOL and Grav/CN Tests on Mill Composites

Composite	Ore Zone	Lithology	Test Type	Gold Rec %	Gold Rec	Gold Rec	Gold Calc Head g/t Au	NaCN kg/t ore	Lime kg/t ore
4505-004-HG	South Domes	Vx	WOL	N/A	N/A	98.9	0.82	0.07	1.0
			Grav./CN	16.1	81.5	97.6	0.84	0.07	0.8
4505-005-HG	South Domes	Rhy-Aph	WOL	N/A	N/A	98.7	0.79	0.07	0.8
			Grav./CN	19.8	78.7	98.5	0.75	0.07	0.8
4505-006-HG	South Domes	Vx	WOL	N/A	N/A	98.5	1.36	0.07	1.5
			Grav./CN	26.5	71.1	97.6	1.25	0.09	1.3
4505-008-HG	JSLA	Diatreme	WOL	N/A	N/A	85.1	0.94	0.19	1.4
			Grav./CN	28.1	63.3	91.4	1.17	0.27	1.3
4505-009-HG	JSLA	Rhy-Bx	WOL	N/A	N/A	94.1	0.68	0.07	0.8
			Grav./CN	8.3	86.5	94.8	0.76	0.07	0.7
4505-011-HG	Jumbo	Rhy-Aph	WOL	N/A	N/A	89.3	0.28	0.07	1.9
			Grav./CN	6.1	86.9	93.0	0.29	0.10	2.2
4505-012-HG	Jumbo	Andesite	WOL	N/A	N/A	96.0	1.00	0.09	1.8
			Grav./CN	11.7	84.9	96.6	0.88	0.15	1.4
4505-013-HG	Jumbo	Rhy-Porph	WOL	N/A	N/A	96.9	0.65	0.07	1.0
			Grav./CN	19.2	76.7	95.9	0.48	0.07	1.3
4505-022-HG	Oro Belle	Rhy-Aph	WOL	N/A	N/A	96.2	1.04	0.09	1.3
			Grav./CN	19	75.5	94.5	1.00	0.08	1.6
4505-023-HG	Oro Belle	Diatreme	WOL	N/A	N/A	92.9	1.12	0.07	1.2
			Grav./CN	17.9	74.6	92.5	1.06	0.10	1.7
4505-024-HG	Oro Belle	Andesite	WOL	N/A	N/A	94.8	1.73	0.07	2.1
			Grav./CN	18.8	76	94.8	1.55	0.14	2.7
4505-026-HG	Oro Belle	Vx	WOL	N/A	N/A	93.6	0.78	0.07	1.7
			Grav./CN	10.5	83.5	94.0	0.83	0.15	2.1

Gravity gold recovery ranged from 6% to 28%. The recovery on the gravity tail leach varied from 63% to 87% for a combined average recovery of 95% after 96 hours.

The 2020 testwork showed good amenability to gravity concentration and gravity tail leaching for all mill grade ore samples.

Cyanide and lime consumption for the gravity tail cyanidation tests were similar to the consumptions from the 2015 and 2018 testwork. Cyanide consumption ranged from 0.14 to 0.54 lb/ton ore (0.07 to 0.27 kg/t ore) with an average of 0.20 lb/ton ore (0.10 kg/t ore). Lime consumption ranged from 1.2 to 5.4 lb/ton ore (0.60 to 2.70 kg/t ore) with the average being 2.82 lb/ton ore (1.41 kg/t ore).

ICP analysis was done on pregnant solution on gravity tail leach tests. Analyses show arsenic concentration at less than 0.2 mg/L, copper concentration ranging from 0.4 mg/L to 3.6 mg/L with one sample being at 30.8 mg/L and zinc ranging from 0.2 mg/L to 1 mg/L.

Overall gold recovery ranged from 89% to 99% with the average recovery being 95%.

13.6.2.3 Grind Size vs Recovery

In 2020, a grind recovery series was run on four composites: 4505-004-HG, 4505-005-HG, South Domes Master Composite and Main Pit Master Composite. These results are presented in Figure 13-20 alongside the grind recovery series conducted in 2018. Four tests were done at both 150 µm and 300 µm with better recovery in all cases at 150 µm. Grinding to 106 µm resulted in lower recovery in two of the three tests and grinding further down to 74 µm did not improve total recovery in either of the two samples. Based on these test results, a 150 µm (100 mesh) grind size appears to be the optimum size for the mill grade ore.

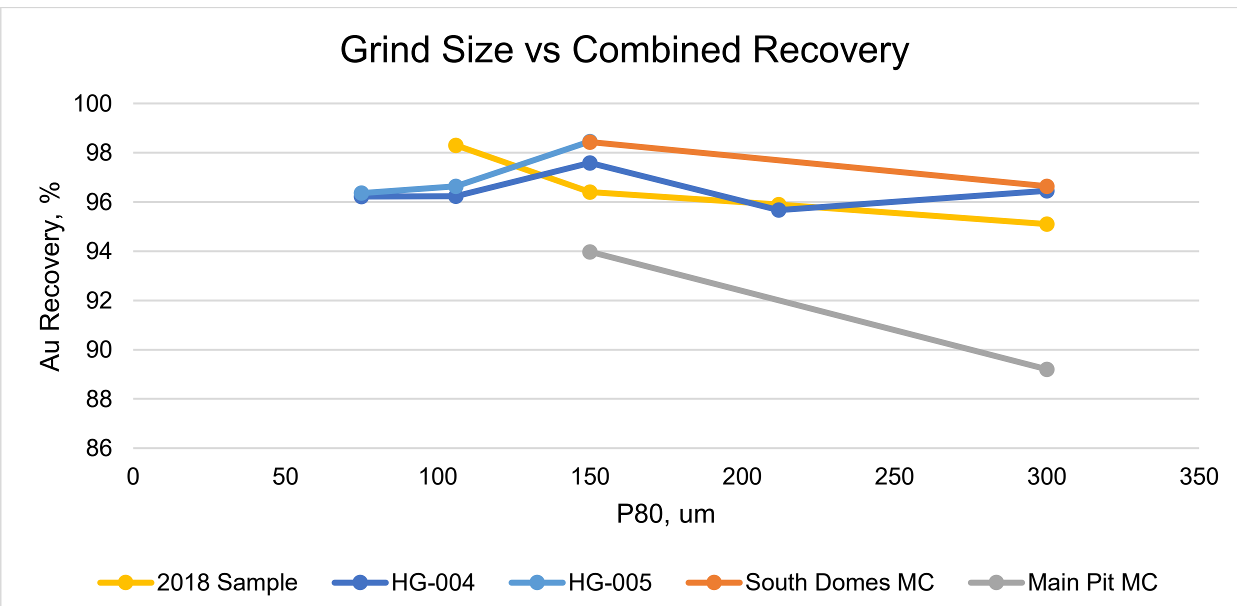


Figure 13-20: Grind Size vs Combined Recovery

13.6.3 Mill Grade Ore Gold Recovery

The results for the 2020 testwork, which is representative of the majority of lithologies within the mine and consisted of 12 samples ground to a P_{80} of 150 µm, indicated faster leach kinetics for gravity plus tails leach in comparison to whole ore leach without gravity. Gravity plus tails leach at 24 hours shows a similar recovery to that of whole ore leach after 96 hours. Results for the 2020 combined gravity and tails leach indicated an average of 94.7% recovery at 24 hours. Whole ore leach had an average recovery of 92.5% at 48 hours. Individual whole ore leach results at 48 hours range from 87% to 96% compared to the gravity plus tails leach tests at 24 hours which range from 91% to 98%. Table 13-25 shows resulting average recoveries for incremental tested leach times. Figure 13-21 below shows the resulting leach curves from 2020 testwork showing average recovery.

Table 13-25: 2020 Testwork – Average Recovery vs Time

Leach Time	Whole Ore Leach Recovery (%)	Gravity + Tails Leach Recovery (%)
2	40.5	59.6
4	57.2	76.8
8	62.6*	87.6
24	84.4	94.7
48	92.5	94.2
72	93.6	95.4
96	94.6	95.4

* Value interpolated; 8-hour reading was not taken for Whole Ore Leach (2020 Testwork)

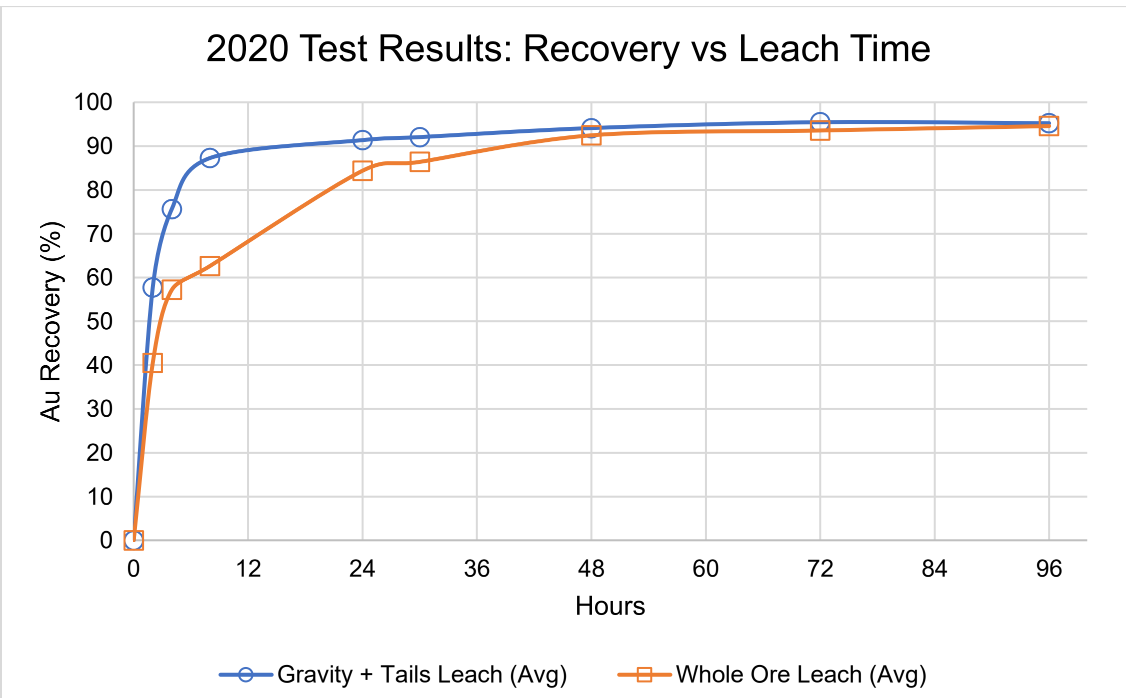


Figure 13-21: 2020 Test Results – Recovery vs Leach Time

Cyanide consumption for the whole ore leach tests ranged from 0.14 to 0.40 lb/ton ore (0.07 to 0.20 kg/t ore) with the average being 0.16 lb/ton ore (0.08 kg/t ore). Lime consumption ranged from 1.6 to 4.2 lb/ton ore (0.80 to 2.10 kg/t ore) with an average of 2.8 lb/ton ore (1.4 kg/t ore).

Gravity plus gravity tail cyanidation and whole ore leach testwork completed in 2015, 2018, and 2020 was evaluated. Test programs were executed primarily at a P_{80} of 75 and 150 µm with an overall leach time of 72 and 96 hours. Review of the resulting leach curves and recoveries from each of these test programs were then used as basis for establishing criteria with which to evaluate use of a gravity circuit within the Castle Mountain flow sheet.

Gold recovery for whole ore leach and gravity plus tail leach was universally high independent of grade, in all tests evaluated as can be seen in Figure 13-22.

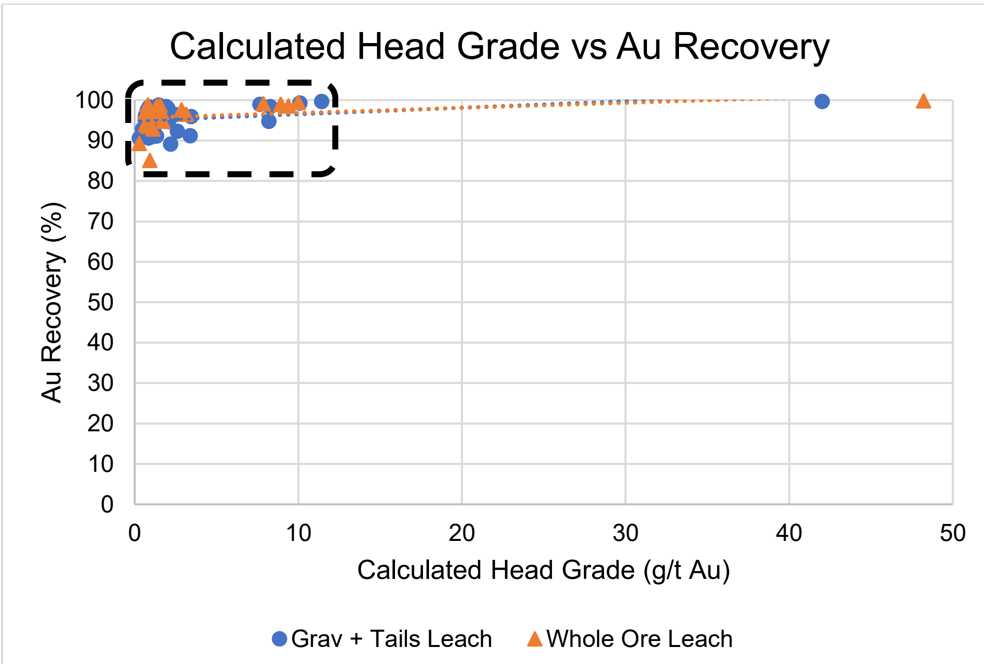


Figure 13-22: Calculated Head Grade vs Gold Recovery

Figure 13-23 is the same data plotted in Figure 13-22 with adjusted x-axis scale since almost all test results fell within this range.

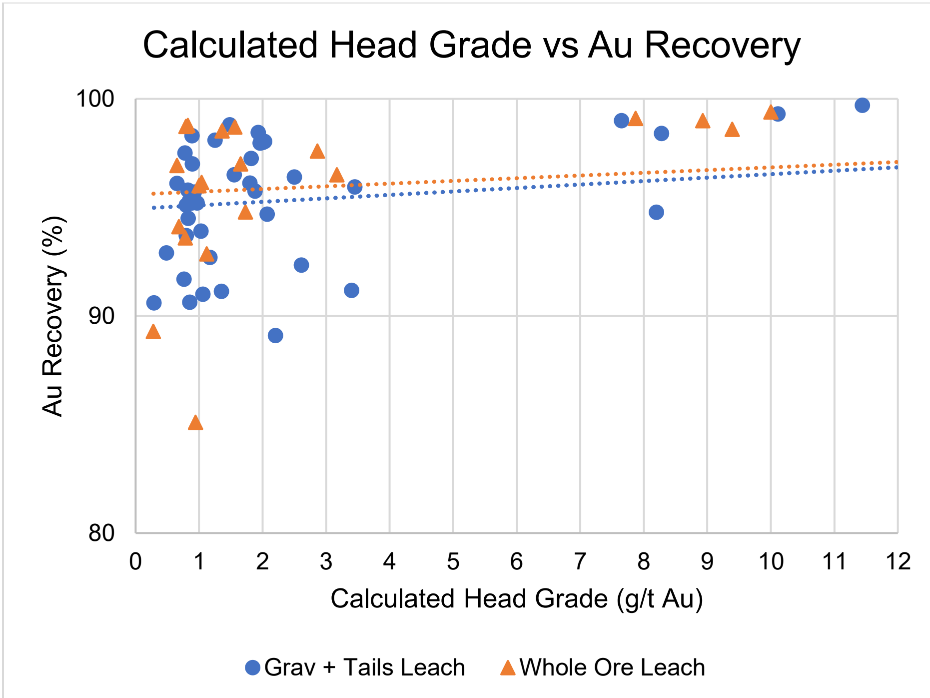


Figure 13-23: Calculated Head Grade vs Gold Recovery (Adjusted Scale)

Results of leaching testwork at 150 µm from 2015, 2018 and 2020 were plotted for whole ore leach and gravity plus tails leach to show the range in recovery by lithology. Figure 13-24 shows that resulting ranges for recovery are generally similar between whole ore leach and gravity plus tails leach.

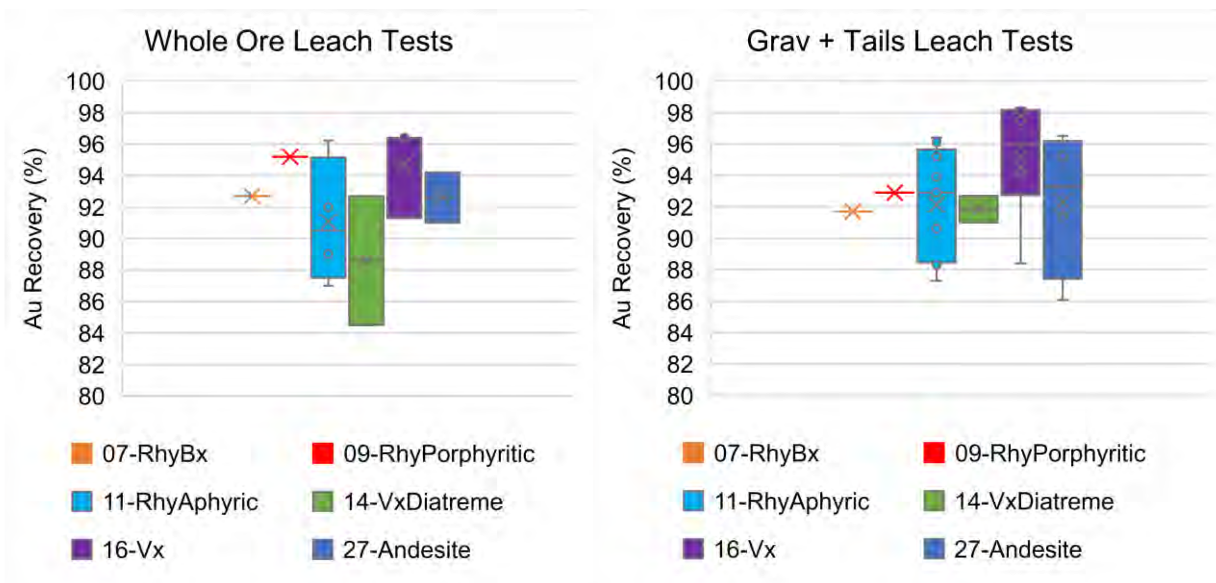


Figure 13-24: % Recovery Range by Lithology, 150 µm (2015-2020) Combined Results

An average of all the tests was taken for each lithology. These values were then weighted based on the percentage of ounces that these lithologies carry in the project mine plan. The resulting average recoveries by lithology and weighted by percent ounces are shown below in Table 13-26.

Table 13-26: Weighted Average Recoveries by Lithology – 150 µm (2015-2020)

Lithology	Whole Ore Leach Recovery (%) – 24 Hours	Whole Ore Leach Recovery (%) – 48 Hours	Gravity + Tails Leach Recovery (%) – 24 Hours	Gravity + Tails Leach Recovery (%) – 48 Hours	% Ounces
Rhy-Bx	86.7	92.7	91.7	90.8	1.2
Rhy-Porph	89.7	95.2	92.9	91.2	17.7
Rhy-Aph	81.5	91.1	92.1	93.3	22.1
Diatreme	81.9	88.6	91.9	90.8	2.3
Vx	86.1	94.7	95.2	96.1	42.4
Andesite	86.1	92.6	92.3	92.5	8.4
Weighted Average	**85.6**	**93.6**	**93.6**	**94.0**	**94.0**

Whole ore leach recovery was incomplete after 24 hours having achieved only 86% and required 48 hours to reach recoveries of nearly 94%. Gravity plus gravity tails leach nearly reached ultimate recovery of 94% after 24 hours and was the same as whole ore leach after 48 hours and only had marginal recovery when the gravity tails leach continued to 48 hours.

All whole ore leach and gravity plus tails leach tests were evaluated by ore zone. Average gold recovery varied slightly between whole ore leach tests and gravity plus tails leach as shown in Table 13-27.

Table 13-27: Weighted Average Recoveries by Ore Zone – 2015, 2018, 2020 Testwork

Ore Zone	Whole Ore Leach Recovery (%) – 48 Hours	Gravity + Tails Leach Recovery (%) – 24 Hours	% Ounces
JSLA	93.1	94.9	33.9
Oro Belle	92.8	93.2	13.9
South Domes	94.9	94.3	26.2
Jumbo	92.1	92.9	9
Weighted Average	**93.5**	**94.2**	**83**

For this Feasibility Study, gravity followed by gravity tail leach in a hybrid leach/CIL circuit was selected for the process based on economics. An overall gold recovery of 94% is expected from mill grade ores processed through the mill after 24-hour CIL retention time.

13.6.3.1 Cyanide Consumption

Cyanide consumption is estimated from gravity tails leach testwork at 150 µm in 2018 and 2020 on ore with grades exceeding 0.036 oz/ton (1.25 g/t). This includes 13 gravity tails leach tests and 3 gravity tails CIL tests. Cyanide consumption for the tests used to estimate the gold recovery range from 0.2 to 1.6 lb/ton ore (0.1 to 0.8 kg/t ore) with the average being 0.4 lb/ton ore (0.2 kg/t ore). Based on this, cyanide consumption for mill grade ore is estimated to be 0.4 lb/ton ore (0.2 kg/t ore) over the LOM.

13.7 MILL GRADE ORE MASTER COMPOSITE 2020 TESTING

Two master composites were made representing Main Pit and South Domes, respectively. Both samples were tested through the entire milling process: crushing, grinding, gravity, pre-leach thickening, CIL, detox, tailings thickening, filtering and tailing stacking. Emphasis was placed on testing detox, thickening, filtering and tailings stacking.

13.7.1 Sample Prep and Grinding

Two master composites, Main Pit HG MC and South Domes HG MC, were produced by MLI as per the composite make-up presented in Table 13-28 and Table 13-29. These samples were crushed and ground to a P_{80} of 150 µm (100 mesh).

Table 13-28: Main Pit Mill Grade Ore Master Composite Make-up

Composite	Lithology	Hole ID	Make-up Proportion
HG-011	Rhy-Aph	CMM-021C	15%
HG-012	Andesite	CMM-283C	5%
HG-013	Rhy-Porph	CMM-281C	30%
HG-022	Rhy-Aph	CMM-119C	15%
HG-024	Andesite	CMM-036C	5%
HG-026	Vx	CMM-120C	30%

Table 13-29: South Domes Mill Grade Ore Master Composite Make-up

Composite	Lithology	Hole ID	Make-up Proportion
HG-004	Vx	CMM-248C	25%
		CMM-252C	
HG-005	Rhy-Aph	CMM-016C	50%
HG-006	Vx	CMM-250C	25%

13.7.2 Bulk CIL for Tailings Tests

The two master composites were split and four bulk CIL tests were completed to generate tailing at two different residual free cyanide concentrations for the subsequent detox tests. Parameters used for each bulk CIL test are shown below, cyanide conditions are presented in Table 13-30, and CIL recovery results are presented in Table 13-31.

- Approximately 11 kg of ore, P_{80} of 150 µm (100 mesh)
- 50% solids (w/w)
- pH 11 maintained with hydrated lime
- 48 hours leach time
- Au/Ag extraction method CIL
- Carbon concentration 10 g/L

Table 13-30: CIL Initial and Residual Cyanide Concentration

Composite	Initial Cyanide Concentration g NaCN/kg ore	Residual Cyanide Target g NaCN/kg ore (ppm CN_{WAD})[1]
Main Pit HG MC	500	250 (125)
Main Pit HG MC	300	150 (75)
South Domes HG MC	500	250 (125)
South Domes HG MC	300	150 (75)

Note 1: at 50% solids, g NaCN/kg ore is approximately 1/2 ppm CN_{WAD} in solution.

For each composite, recoveries were essentially the same (within 0.8%) at both initial NaCN concentrations. South Domes composite gold recovery averaged 98.4%. Main Pit composite gold recovery averaged 94.0%.

Bulk tests were done at higher density (50% vs 40%) than other high-grade leach tests, shorter leach time (48 hours vs 96 hours), and at lower cyanide concentrations (500 and 300 ppm vs 1,000 ppm).

The weighted average recovery from 1 kg whole ore cyanidation tests of the three constituent South Domes composites was 98.7% compared to 98 and >98.8 for the bulk tests. The weighted average whole ore recovery from the six constituent Main Pit composites was 95.1% compared to 94.2% and 93.8% for the bulk tests.

Based on this, relative to the tests on individual composites discussed in Section 13.6.2, it appears that increasing the slurry density (40% to 50%) reducing the leach time (96 to 48 hours) and decreasing the NaCN concentration (1 g/L to 0.5 or 0.3 g/L) did not impact recovery.

Silver recoveries were comparable to the individual 1 kg tests, although most of the loaded carbon silver grades were below detection limits, so these results are not definitive.

Calculated gold head grades are reasonably close to the predicted head grades.

Table 13-31: Bulk Bottle Roll Tests (CIL) Recoveries

Composite	g/t Au Calculated Head	Initial NaCN Conc., g/L	Au Recovery, %	g/t Au		Ag Recovery, %	g/t Ag		Reagent Requirements kg/t ore	
				Extracted	Tail		Extracted	Tail	NaCN Cons.	Lime Added
Main Pit HG MC	0.69	0.50	94.2	0.65	0.04	<38.9	<0.7	1.1	0.24	1.4
Main Pit HG MC	0.80	0.30	93.8	0.75	0.05	<38.9	<0.7	1.1	0.09	1.8
South Domes HG MC	1.02	0.50	98.0	1.00	0.02	<50.0	<0.7	0.7	<0.07	0.9
South Domes HG MC	<0.85	0.30	>98.8	0.84	<0.01	40.0	0.4	0.6	0.09	1.3

13.7.3 Cyanide Detox Testing

Caro's Acid and SO_2/air cyanide destruction tests were performed on two Castle Mountain slurry samples to identify detoxification operating parameters, final residual cyanide values, and estimated reagent requirements.

Each of the samples were tested using Caro's Acid and SO_2/air. Target free cyanide levels for the tests were weak acid dissociable cyanide (CN_{WAD}) levels of 25 and 1 ppm. Tests on Main Pit samples were run at 55% solids density while tests on South Domes samples were run at 60% and 62% solids density. Test procedures are detailed in the Cyanco test report (Cyanco, 2020).

Based on the results of the testwork program both methods of cyanide detox will work on the Castle Mountain Main Pit and South Domes slurry tested. Both methods tested were successful in reducing the CN_{WAD} levels to the desired targets of below 25 and 1 ppm. Test results indicated that a higher pulp density for both the Main Pit ore and the South Domes ore did not allow for proper mixing of the reagents. In addition, a two-hour retention time and sparging of pure (93%) oxygen is recommended for the SO_2/air method.

Required treatment limits should be confirmed and further testwork would be recommended to optimize conditions (reagents/air) in conjunction with optimization of the CIL circuit.

13.7.4 Thickening

Three solid-liquid separation programs were conducted at Pocock Industrial Inc. for MLI on the Castle Mountain mineralization. The first took place in October 2017. The second program was conducted in March and April 2020 on master composites from Main Pit and South Domes ore zones. This included two CIL feed samples (Main Pit and South Domes HG MCs) and two detoxed CIL residue samples (Main Pit and South Domes HG MCs). In June, ten additional tests were done on composites from JSLA, South Domes, Oro Belle and Jumbo ore zones to further investigate potential thickening, filtration, and viscosity issues.

Thickening tests (static and dynamic) and filtration tests (vacuum and pressure) were completed. The P_{80} of the samples of the Main Pit ore were 122 µm for the CIL feed sample and 134 µm for the detoxed residue sample. The P_{80} of the samples of the South Domes ore were 147 µm for the CIL feed sample and 144 µm for the detoxed residue sample. The specific gravity of the samples was back calculated using solids concentration and pulp densities and ranged from 2.65 to 2.84. Lime was used to adjust the pH of the CIL feed samples to 11.5.

The following contains a summary of the recommendations from the 2020 testing program, (Pocock, 2020).

- A single flocculant could not be used to provide acceptable results for both the CIL feed samples and the detoxed residue samples. The flocculant selected for the best performance was an anionic polyacrylamide flocculant (SNF AN910SH) for the CIL feed samples while a cationic product (SNF FO 4190 SSH) was selected for the detoxed residue samples. Flocculant (anionic) dose requirements for thickening on the CIL feed samples ranged from 21 to 39 g/t and were delivered at a solution concentration of 0.1 g/L. Flocculant (cationic) dose requirements for thickening on the detoxed residue samples ranged from 50 to 73 g/t and were delivered at a solution concentration of 0.2 g/L.
- Results of dynamic or high-rate thickening tests indicated that the Main Pit CIL feed sample was difficult to thicken and the feed solids concentration required was low at 12.3% while the South Domes CIL feed sample thickened well at a feed solids concentration of 19.2% The pH of both CIL feed samples was raised to 11.5 to reduce the amount of flocculant used and to improve the clarity of the overflow.

- Results of dynamic or high-rate thickening tests indicated that the Main Pit detoxed residue sample thickened well at a feed solids concentration of 15% while the South Domes detoxed residue sample thickened well at a feed solids concentration of 17.8%.
- The predicted operating density range for a high rate thickener on Main Pit or South Domes CIL feed with a grind size of approximately 80% passing 100 mesh was 50% to 62%. The recommended maximum operating density for Main Pit ore was 52% and for South Domes ore is 62%.
- Based on the testwork, a net loading rate range for high rate thickener sizing for Main Pit CIL feed is between 2.74 and 3.91 m^3/m^2h and the range for South Domes CIL feed is 2.21 to 3.11 m^3/m^2h.
- Recommended thickener feed solids concentrations varied between the two ore bodies tested. Main Pit material (CIL feed and detoxed CIL residue) performed acceptably at feed solids concentrations between 10 and 15 percent. South Domes material performed well at feed solids concentrations up to 15 to 20 percent.
- Since one grinding thickener will be installed in the plant to treat CIL feed, a thickener feed solids concentration of 12% and a net loading rate of 3.43 m^3/m^2h is recommended to be used for sizing. Optimum flocculation efficiency and thickener performance was achieved when flocculant solution concentration was maintained at 0.01 to 0.02%.
- A second thickener will be installed in the plant to treat CIL residue (cyanide recovery), and it is recommended that the same parameters be used to size the thickener.

13.7.5 Filtration

A brief summary of the recommendations from the 2020 testing program is as follows, (Pocock, 2020).

- Vacuum filtration tests and pressure filtration tests were performed on Main Pit and South Domes detoxed CIL residue underflow generated from the thickening tests described in Section 13.7.4. All tests were conducted on material that was at a pH of 8.5.
- Vacuum filtration tests were done using an applied vacuum level of 20” Hg. Tests were done with and without filter aid (flocculant). Filter cake moistures varied from 23 to 33 percent with a production rate range of 37.6 $kg/m^2 \cdot hr$ to 724.7 $kg/m^2 \cdot hr$. Both Main Pit tests and one South Domes test failed to produce filter cakes at moisture contents low enough for good discharge and stacking. Vacuum filtration is not recommended.
- Pressure filtration tests were done using air blow only and light membrane squeeze during air blow followed by a full pressure membrane squeeze. A driving force of 80 pounds per square inch gauge (psig) was used for all fill and air blow operations. When squeeze was applied during air blow, 100 psig squeeze was applied until the last 30 seconds of air blow when the pressure was increased to 232 psig for the remainder of the cycle.
- With filter feed solids of 50% for Main Pit ore and 59% for South Domes ore, membrane squeeze reduced the estimated moisture of the filter cake from 19.9% to 18.1% for the Main Pit sample and from 17.5% to 16.4% for the South Domes sample. At these moistures, the filter cakes produced from pressure filtration testing of the leach residue material were easily dischargeable from the testing apparatus and generated a stackable and conveyable cake. Cycle times ranged from 12 minutes for the South Domes material to 20.5 minutes for the Main Pit material.

13.8 OTHER TESTWORK AND ANALYSIS

13.8.1 Blast & Fragmentation Analysis

Blast and fragmentation analysis were done for ore and waste material by AMC Consultants (Rogers, 2020). The lithology units modeled make up 95% of the mined ore and include rhyolite (45%), volcaniclastic (32%), andesite (13%) and epiclastic (5%). A summary of the fragmentation of the ore material for each lithology is listed in Table 13-32.

Table 13-32: Ore Material Fragmentation Summary for Each Lithology

Fragmentation	Rhyolite	Volcaniclastic	Andesite	Epiclastic
Top size (in)	17	14	14	13
F_{80} (in)	8	6	6	6
F_{50} (in)	4	3	3	3

Note: Data source from AMC Consultants (Rogers, 2020)

From these results, a top size of 17 in with an F_{80} of 8 in was considered for the ROM for Phase 2. The size distribution of the four lithologies are shown in Figure 13-25. The screen analysis from the ROM tests done in 2018 were plotted for comparison. This analysis suggests the fragmentation particle size distribution may be somewhat coarser than the ROM samples from the Oro Belle and JSLA backfill that were used in the tests.

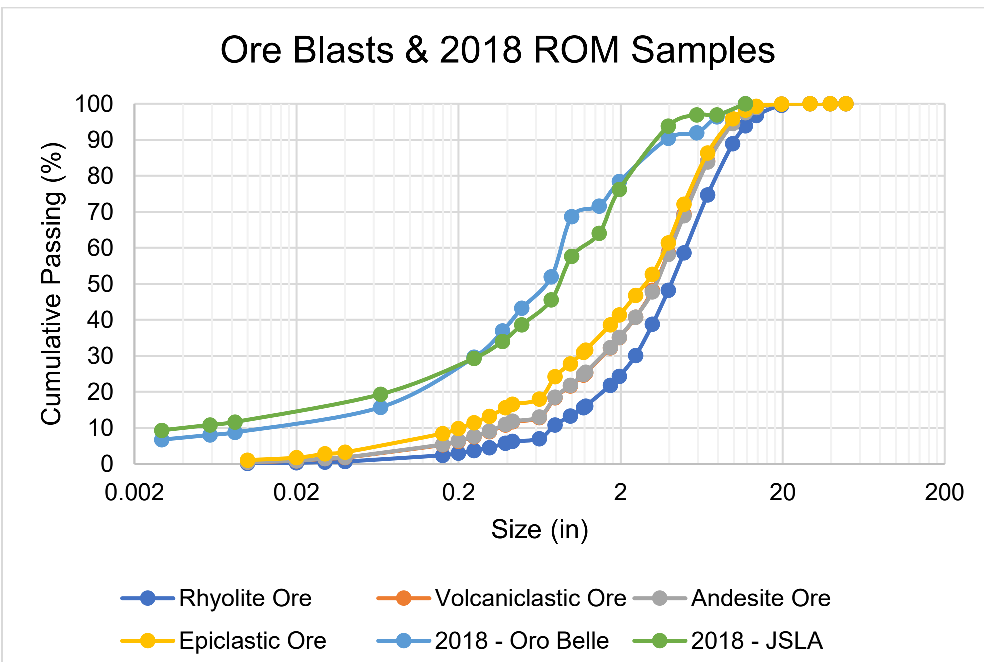


Figure 13-25: Ore Size Distribution

13.8.2 Sulfur Speciation

Sulfur speciation by pyrolysis were done to determine the sulfur content in the composites tested.

Two out of the 12 heap leach composites had higher sulfur than the other samples. Results for 4505-007-LG and 4505-025-LG had a sulfur content of 0.40% and 0.67% respectively and had a low gold recovery. The remaining composites had a lower sulfur content (below 0.8%). Head analyses of the two composites with high sulfur content also indicated higher levels of base metals (e.g. copper, nickel, and zinc). Table 13-33 is a summary of the sulfur content and gold recovery.

Table 13-33: Heap Leach Composites Sulfur Content

Ore Zone	Lithology	Composite	Hole No.	%S (Total)	%S (Sulfate)	%S (Sulfide)	Gold Rec %	NaCN kg/t ore
South Domes	Vx	4505-001-LG	CMM-122C	<0.01	<0.01	<0.01	96.2	2.66
South Domes	Rhy-Aph	4505-002-LG	CMM-255C	<0.01	<0.01	<0.01	88.2	0.84
South Domes	Diatreme	4505-003-LG	CMM-242C	<0.01	<0.01	<0.01	89.2	1.27
JSLA	Andesite	4505-007-LG	CMM-229C	0.4	0.19	0.21	69.7	1.14
Jumbo	Vx	4505-010-LG	CMM-273C	<0.01	<0.01	<0.01	87.5	0.89
East Ridge	Rhy-Aph	4505-015-LG	CMM-010C	<0.01	<0.01	<0.01	73.3	1.13
Oro Belle	Vx	4505-017-LG	CMM-119C	0.07	0.07	<0.01	66.7	1.2
Oro Belle	Rhy-Aph	4505-018-LG	CMM-119C	0.02	0.02	<0.01	88.9	1.24
Oro Belle	Andesite	4505-019-LG	CMM-119C	<0.01	<0.01	<0.01	95.2	0.94
Oro Belle	Vx	4505-020-LG	CMM-120C	0.05	0.05	<0.01	41.2	1.16
Oro Belle	Rhy-Aph	4505-021-LG	CMM-120C CMM-276C	0.02	0.02	<0.01	82.1	1.1
Oro Belle	Epiclastic	4505-025-LG	CMM-120C	0.67	0.13	0.53	36.8	0.96

One out of the 12 mill composites had a higher level of sulfur while the rest of the composites had an average sulfur content of 0.01%. The high sulfur content composite, 4505-008-HG, had the lowest gold recovery and highest cyanide consumption. Head analyses on the composite with higher sulfur also had higher levels of base metals. Table 13-34 is a summary of the sulfur content, gold recovery and reagent consumption.

Table 13-34: Mill Composites Sulfur Content

Ore Zone	Lithology	Composite	Hole No.	%S (Total)	%S (Sulfate)	%S (Sulfide)	Gold Rec %	NaCN kg/t ore
South Domes	Vx	4505-004-HG	CMM-248C CMM-252C	<0.01	<0.01	<0.01	97.6	0.07
South Domes	Rhy-Aph	4505-005-HG	CMM-016C	<0.01	<0.01	<0.01	98.5	0.07
South Domes	Vx	4505-006-HG	CMM-250C	<0.01	<0.01	<0.01	97.6	0.09
JSLA	Diatreme	4505-008-HG	CMM-070C	1.63	0.23	1.4	91.4	0.27
JSLA	Rhy-Bx	4505-009-HG	CMM-033C	0.01	0.01	<0.01	94.8	0.07
Jumbo	Rhy-Aph	4505-011-HG	CMM-021C	<0.01	<0.01	<0.01	93	0.07
Jumbo	Andesite	4505-012-HG	CMM-283C	0.01	0.01	<0.01	96.6	0.15
Jumbo	Rhy-Porph	4505-013-HG	CMM-281C	<0.01	<0.01	<0.01	95.9	0.07
Oro Belle	Rhy-Aph	4505-022-HG	CMM-119C	0.02	0.02	<0.01	94.5	0.08
Oro Belle	Diatreme	4505-023-HG	CMM-036C	0.02	0.02	<0.01	92.5	0.1
Oro Belle	Andesite	4505-024-HG	CMM-036C	<0.01	<0.01	<0.01	94.8	0.14
Oro Belle	Vx	4505-026-HG	CMM-120C	0.04	0.04	<0.01	94	0.15

The composites with higher sulfur were from the deepest part of the respective ore body as shown in Figure 13-6 and Figure 13-8. Based on the results from tests of the composites above, gold recovery and/or cyanide consumption may be impacted when material with high sulfur is processed. Higher sulfur content was only detected in more minor lithology units of the mine plan, namely andesite, epiclastic and diatreme containing 4%, 4% and 5% of total ounces, respectively. Further, high sulfur was present in only one of four andesite samples tested and only one of three diatreme samples tested (MLI, 2020b).

13.8.3 Carbon Adsorption Rate and Capacity

A batch carbon adsorption rate test (CAR) and adsorption carbon capacity (CAC) test were completed on one mill master composite made up of the following composite samples: 4505-004-HG (60%), 4505-022-HG (30%) and 4505-024-HG (10%), based upon relative lithological representation in the ore body.

A CAR test was conducted to determine the Fleming K and n values for the CIL system. These constants allow simulation of the CIL circuit and calculation of the expected carbon and solution profiles. Results from the CAC Test are shown in Table 13-35 (MLI, 2020b).

Table 13-35: Carbon Adsorption Capacity Test Results

	Carbon Dose, g/L Pulp		Solution Analysis Barren Grade, ppm		Carbon Loading g/t	
Test	Target	Actual	Au	Ag	Au	Ag
CAC-1	0.1	0.1	0.55	0.46	1737	755
CAC-2	0.5	0.5	0.17	0.24	895	481
CAC-3	1.0	1.1	0.04	0.11	520	360
CAC-4	2.0	2.1	0.00	0.03	284	180
CAC-5	5.0	5.1	0.00	0.01	113	70
CAC-6	10.0	9.8	0.00	0.00	59	44

The CAC test to measure the equilibrium gold capacity (K value) was done at carbon concentrations of 0.10, 0.50, 1, 2, 5 and 10 g/L.

The gold in solution versus the carbon loading were plotted and aligned well as in seen in Figure 13-26.

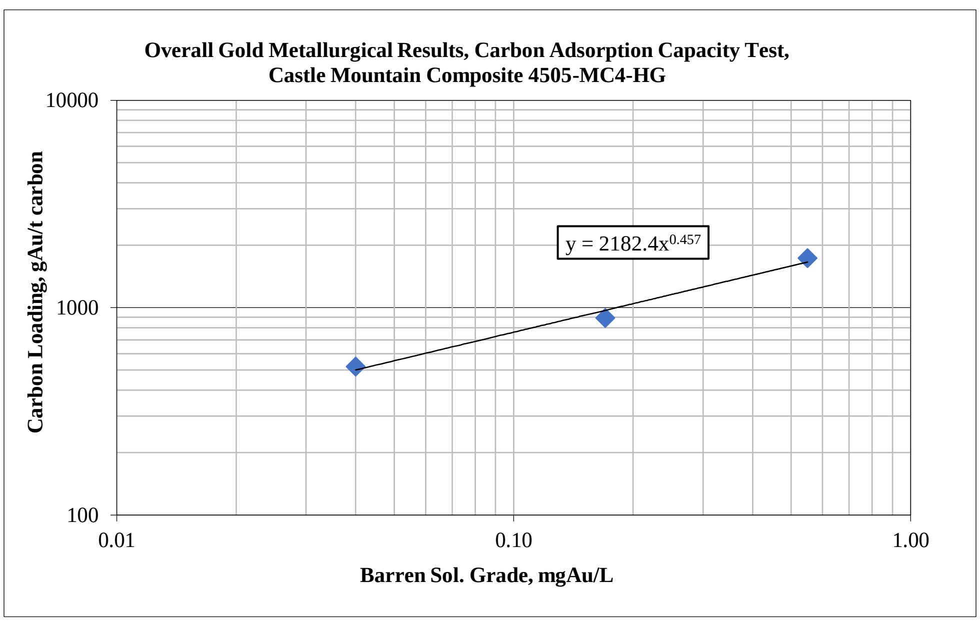


Figure 13-26: Overall Gold Metallurgical Results, Carbon Adsorption Capacity Test

Freundlich isotherm coefficients calculated from the data are shown in Table 13-36.

Table 13-36: Freundlich Isotherm Coefficients

	K	n
Au	2182	0.457
Ag	1227	0.593

The carbon loading (g Au/t C) from the capacity test is 1,737 g Au/t C (50 oz/ton) based on solution assays. Using the isotherm coefficients generated from the testwork, a loading of 1,661 g Au/t C (48 oz/ton) is expected using the following equation:

$$Au_{carbon} = K[Au]^n$$

Preliminary modelling of the adsorption circuit using 'typical' isotherm coefficients indicated a carbon loading of 1,660 g Au/t C (48 oz/ton).

Loaded carbon values typically fall in the range of 1,500 to 4,000 g/t. The results from the test completed show that the results are within industry norms.

A carbon adsorption rate test was conducted, and results are shown in Table 13-37.

Table 13-37: Carbon Adsorption Rate Test, 10 g Carbon/L Pulp

Contact Time, hrs	Solution Analysis Barren Grade, ppm		Calculated Carbon Loading g/t		Assayed Carbon Loading g/t	
	Au	Ag	Au	Ag	Au	Ag
0.33	0.28	0.27	36.4	21	-----	-----
0.75	0.14	0.14	46.4	30	-----	-----
1.50	0.03	0.06	54.2	36	-----	-----
3.00	0.00	0.02	56.2	39	-----	-----
6.00	0.00	0.00	56.2	40	-----	-----
15.00	0.00	0.00	56.2	40	-----	-----
24.00	0.00	0.00	56.2	40	57.2	<50

Kinetics for the adsorption rate tests were fast as shown in Figure 13-27. After three hours, carbon loading remained constant and soluble losses were zero indicating a carbon concentration of 10 g/l at the specified parameters above being suitable for adsorption. Higher rate of adsorption raises stage efficiency and reduces loss of soluble gold.

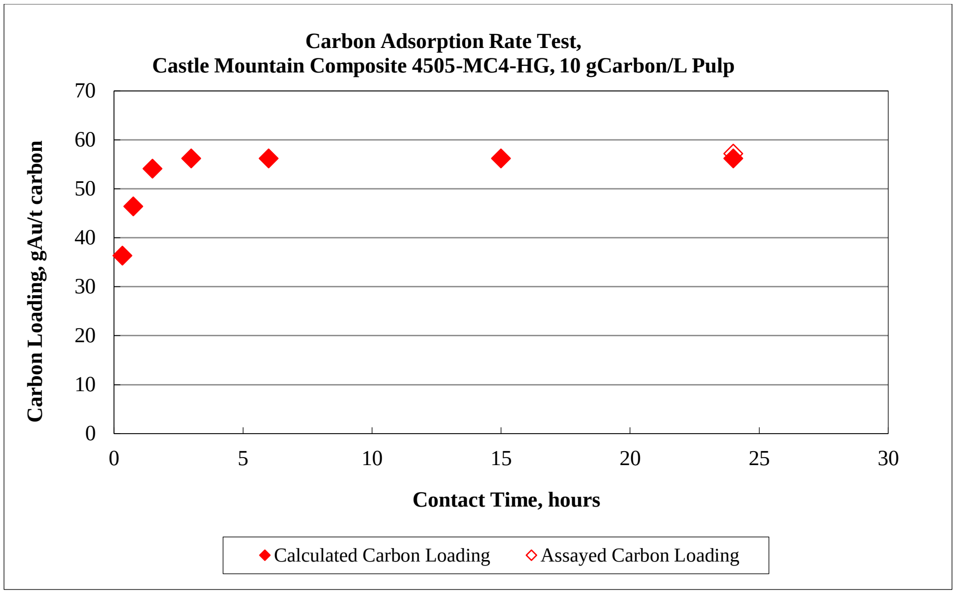


Figure 13-27: Carbon Adsorption Rate Test (4505-MC4-HG)

13.8.4 Oxygen Uptake Rate

A standard procedure was used for an oxygen uptake rate (OUR) test on one mill feed sample. The series of oxygen decay measurements was repeated at time intervals indicated in Table 13-38.

Table 13-38: Oxygen Uptake Results Summary

Time	0 Hr	1 Hr	2 Hr	3 Hr	4 Hr	5 Hr	6 Hr	24 Hr	36 Hr	48 Hr
Slope - mg/L/min	0.0363	0.38	0.33	0.33	0.07	0.65	0.65	1.31	0.45	0.37

The data indicates that the oxygen uptake demand of the Castle Mountain ore is >0.3 mg/l/min for the first three hours and after four hours increases to >0.6 mg/L/min. Higher oxygen demand later in the test is unusual and was likely caused by an error in the test procedure. Regardless, the oxygen consumption is within the range commonly supplied by air sparging so oxygen injection in the CIL circuit is not required.

An OUR test was completed on the same single master composite sample as the CAR and CAC tests, 4505-004-HG (60%), 4505-022-HG (30%) and 4505-024-HG (10%). Compared to air injection, oxygen in the leaching step may increase the rate of gold dissolution, however the same ultimate recovery is achieved. Figure 13-28 shows the result of the two tests. Note that the tests were whole ore leach and not gravity tails leach, recall in Figure 13-21 the kinetics of leaching gravity tails are much faster than whole ore leach, so the kinetics benefit of oxygen is reduced, further supporting that oxygen injection in the CIL circuit is not required (MLI, 2020b).

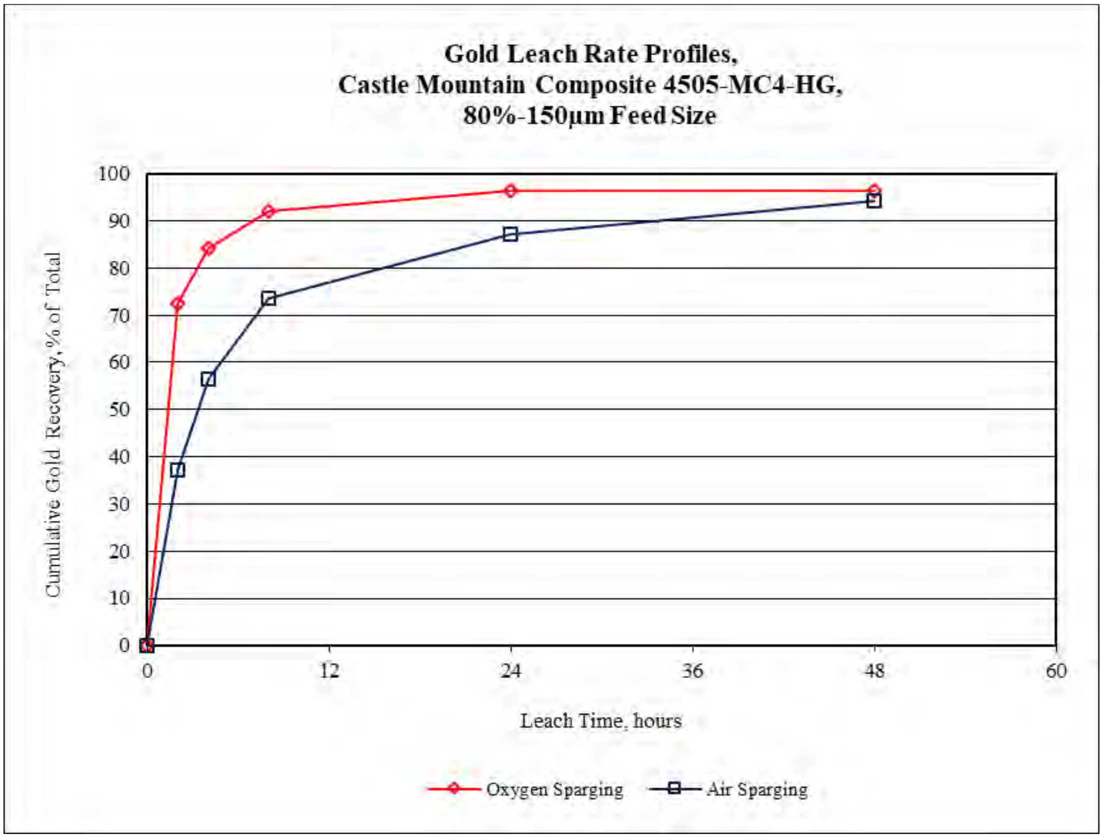


Figure 13-28: Gold Leach Rate Profiles, Oxygen Uptake Rate Test

13.8.5 Compacted Permeability Testing

In 2015, seven heap leach column residues were tested for Load vs Hydraulic Conductivity Testing up to an applied pressure of a simulated stack height of 250 ft (76 m). Four samples tested were from the Jumbo ore zone and three from JSLA. Five of the seven column residues performed well with permeabilities ranging from 10^{-1} to 10^{-3} cm/sec. One of the seven column residues showed poor permeability (10^{-4} cm/sec) from a 50 ft (15 m) simulated stack height onwards. Another residue showed poor to moderately poor permeability (10^{-5} cm/sec) from 75 ft (23m) simulated stack height onwards. Both residues with poor permeability were from the Jumbo ore zone and the respective lithology was Rhy-Porph (Gray, 2016).

In 2020, further Load vs Hydraulic Conductivity Testing was conducted up to a simulated stack height of 400 ft (122 m). Two new column residue samples from South Domes and East Ridge ore zones were tested at Call and Nicholas in Tucson. The South Domes sample showed good permeability results (10^{-2} cm/sec) like the five from 2015 and the East Ridge sample had lower permeability (10^{-3} cm/sec) but still significantly higher than two of the samples from 2015. Both samples tested had relatively consistent permeability from a simulated stack height of 250 ft to 400 ft indicating there in a very low risk in permeability issues from stacking up to 300 ft. The ore does compact under load and experiences some breakage, but the test results show that the effect on hydraulic conductivity will not meaningfully affect the heap leach (Call & Nicholas, 2020).

13.8.6 Mineralogy

In the 2020 column testing, gold recovery for Vx samples varied significantly between the South Domes and Oro Belle at 96% and 41% for LG-001 and LG-020 samples, respectively. Detailed mineralogy work was undertaken to help determine the reason for this difference. It should also be noted that the average grade of these two samples was significantly different with LG-001 averaging 0.041 oz/ton (1.39 g/t) and LG-020 averaging 0.005 oz/ton (0.17 g/t) and as such these samples are outside of the normal heap leach grade ore range 0.006 to 0.036 oz/ton (0.20 to 1.25 g/t), so the analysis of 2020 column leach samples in Section 13.5.4 excludes these two samples. The assay results from the split sample products analysed in the mineralogy work confirmed this large difference in head grade, and this partially explains the difference seen in recovery.

A Scanning Electron Microscope equipped with an Energy Dispersive Spectrometer (SEM-EDS) was used to examine several polished sections for four samples: a head a tails sample from Oro Belle, and a head and tails sample from South Domes. This work is reported by Chudy and Lane (PMC, 2020).

For the Oro Belle sample, gold-bearing grains are predominantly associated with red and red-brown breccia (97%) as well as vein-quartz. The breccias consist of variable but high amounts of quartz (47-57 wt.%) and potassic feldspar (35-47 wt.%). Minor phases include micas (muscovite/sericite, illite, biotite, chlorite, clays), pyriboles and iron and titanium oxides. The red and red-brown breccia particles were observed to have a high degree of porosity and vugs at 15-18%.

For the South Domes sample, gold-bearing grains are predominantly associated with porphyry with a very minor amount contained in polymictic breccia particles. The porphyritic rock consists of quartz (40-45 wt.%), potassic feldspar (29-36 wt.%) and sodic feldspar (14-16 wt%). Minor phases include Fe-Ti oxides, sulfides, and carbonates, with a strong decrease in abundance in the tails compared to the heads. The porphyry particles were observed to be more porous (10%) than the polymictic breccia (<5%).

Gold deportment in the head and tail samples is provided in Table 13-39. For both the Oro Belle and South Domes samples, native gold was preferentially leached and had lower frequency count in the tails sample, significantly more so for South Domes which was consistent with the relatively higher metallurgical recovery observed in the testwork. The analysis showed a higher amount of leaching of gold particles containing 80%+ gold compared to either electrum or predominantly silver particles.

Table 13-39: Frequency Count of Gold Occurrences

	Oro Belle		South Domes	
	Head (%)	Tails (%)	Head (%)	Tails (%)
Native Gold (>80% Au)	79	37	56	2
Electrum (50 – 80% Au)	17	57	44	83
Gold Bearing Silver (<50% Au)	4	6	0	15
Total	100	100	100	100
Total Gold Grain Count	**80**	**49**	**143**	**46**

The gold grain frequency distribution in head samples for Oro Belle and South Domes is shown in Figure 13-29. The gold grain size is largely less than 32 µm, with greater than 50% of the grains less than 8 µm. A similar distribution is observed in the tailings samples in Figure 13-30. The total amount of gold grain observations from head sample to tails sample decreased more for the South Domes sample than the Oro Belle sample, consistent with the relatively higher metallurgical recovery observed in the testwork for the South Domes sample.

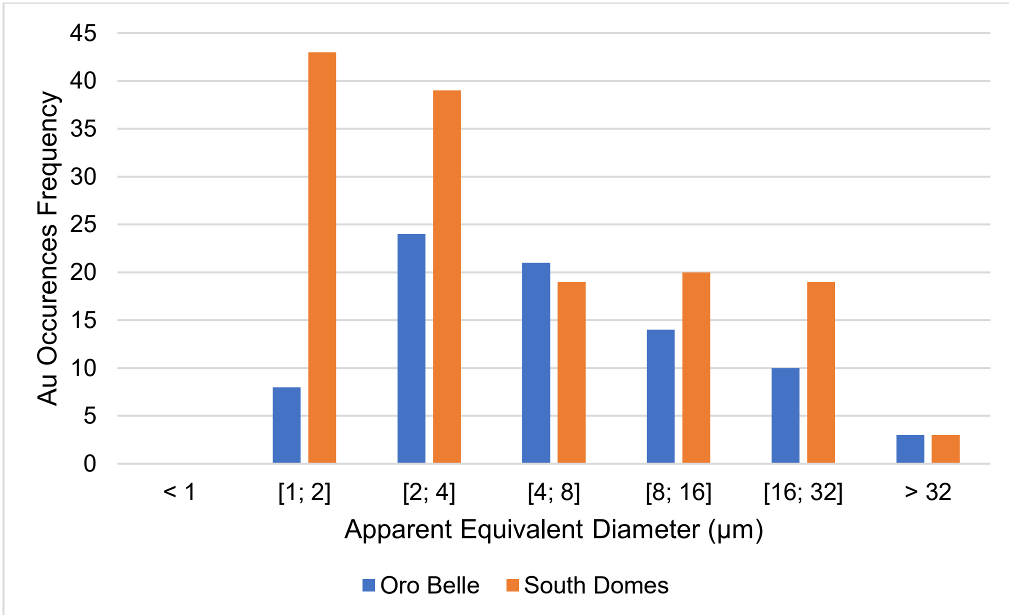


Figure 13-29: Gold Grain Size Frequency Distribution in Head Samples

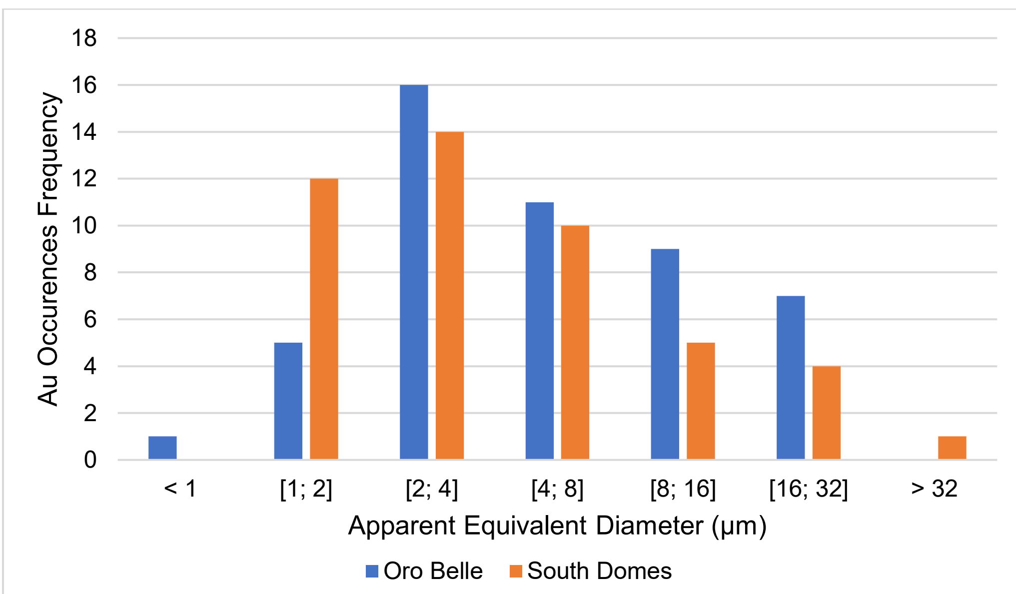


Figure 13-30: Gold Grain Size Frequency Distribution in Tails Samples

A significant difference in the amount of locked gold was found between the Oro Belle head sample and the South Domes head sample. As can be seen in Figure 13-31, approximately 76% of the gold was locked in the Oro Belle head sample. The majority of this is locked with quartz and/or Fe-oxides or feldspar, while 24% is exposed, predominantly attached to quartz in vugs.

The tailings sample as expected shows a large percentage of the gold (94%) is locked, largely in quartz and to a lesser extent in feldspar.

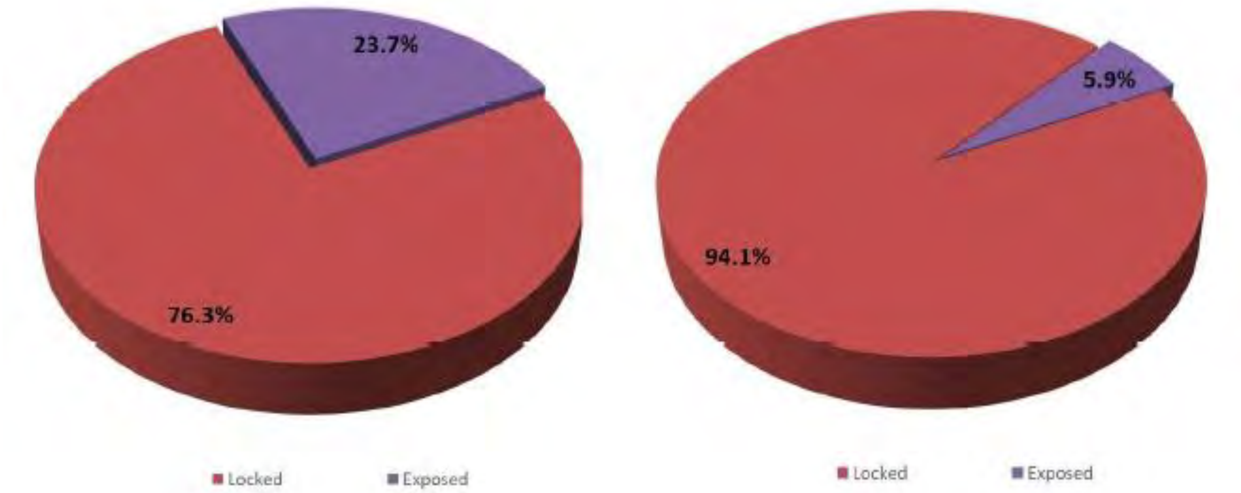


Figure 13-31: Gold Grain Exposure by Weight in Oro Belle Head and Tails Sample

In comparison, only approximately 36% of the gold in the South Domes head sample is locked, largely in feldspar and/or quartz, while 55% is found on grain boundaries and 8% is exposed. In the tails sample, a greater percentage of the gold is observed to be locked. Approximately 94% of the gold in Oro Belle Tails is locked predominantly in quartz and feldspars with only 6% exposed in vugs and pores. For South Domes, 53% of the gold is locked predominantly in quartz and 0% is exposed, but 47% of the gold is not observed.

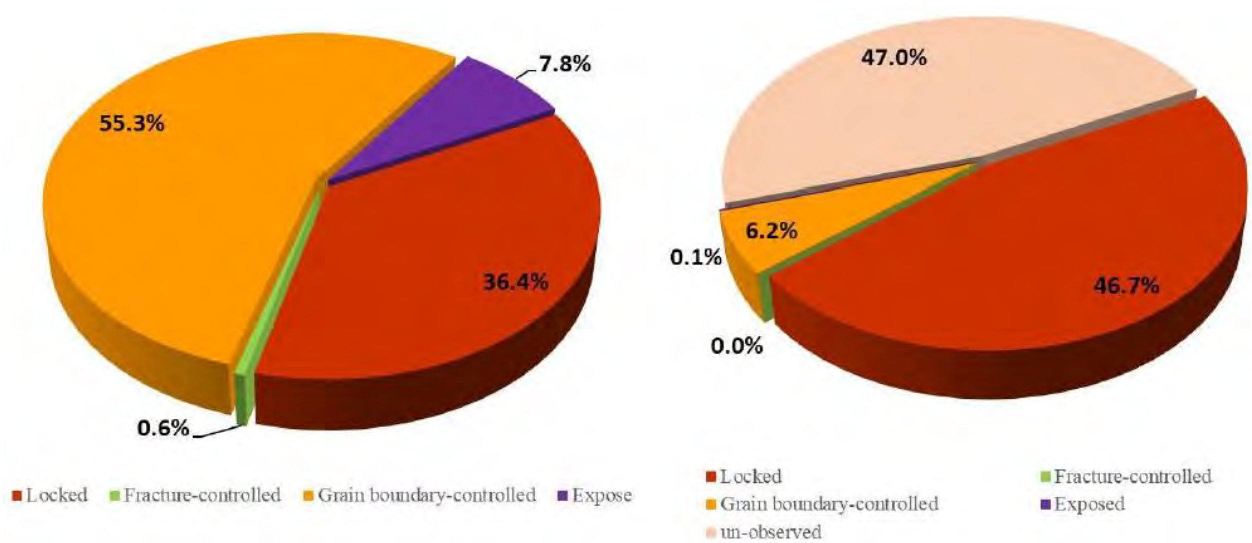


Figure 13-32: Gold Grain Exposure by Product in South Domes Head and Tails Sample

Unobserved gold is calculated based upon assays indicating the presence of gold, yet not observing the expected quantity in the polished sections. The reason for a lower number of observations is likely due to a nugget effect in the assay, or very fine locked gold particles and/or gold grains which were simply not exposed in the 5 polished sections scanned. For the purposes here, it can be assumed that unobserved is equivalent to locked.

The reason for better metallurgical recovery of the South Domes sample could be the lower proportion of gold locked in quartz.

While the gold in the Oro Belle Vx head sample is found to be 97% hosted in one distinguished unit, red/red-brown breccia, the gold in the Oro Belle Vx tails sample is contained 78% in the red/red-brown breccia while the white breccia and vein quartz contain 9% each, indicating preferential leaching of gold from the red/red-brown breccia particles. This agrees with the observed degree of porosity, which is significantly higher than in other units and which increase the amenability to cyanide solutions.

In the South Domes head sample, nearly 100% of the gold was found to be hosted in one distinguished unit, porphyry, while in the tails the porphyry contained only 52% of the gold, with the remaining 48% hosted in polymictic breccia. This could be explained again by the observed degree of porosity, as porphyry is slightly more porous. Figure 13-33 shows a SEM-BSE image of a gold-bearing polymictic breccia. This figure provides a good example of particles that are a main contributor to an apparent nugget effect in the South Domes head sample.

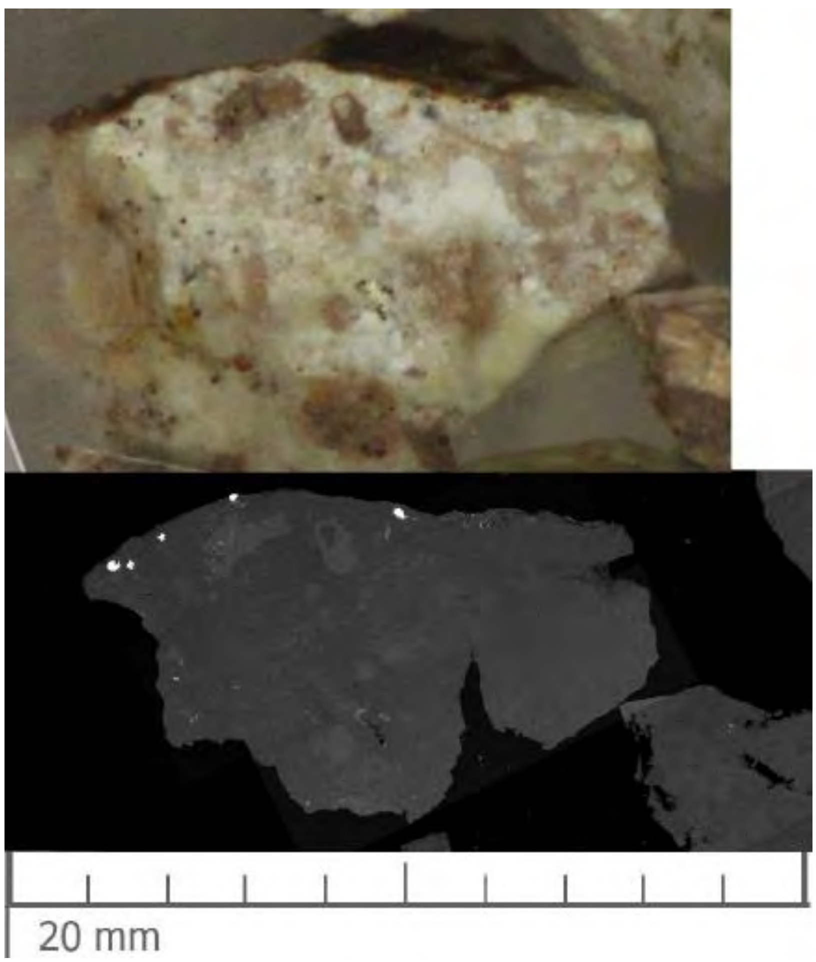


Figure 13-33: Photograph and SEM Image of Gold-Bearing 'Porphyry' Particle. South Domes Head +4.5mm Fraction

13.8.7 Viscosity

Viscosity tests were conducted at Pocock Industrial Inc. on the Main Pit and South Domes master composites in May 2020 (Pocock, 2020).

Ten additional tests were done on composites from JSLA, South Domes, Oro Belle and Jumbo ore zones to further investigate potential thickening, filtration, and viscosity issues. These tests were completed in June 2020.

Viscosity tests were performed using a Fann Model 35A instrument on samples of underflow generated from the thickening tests described in Section 13.7.4 to establish a general set of data to design thickening and underflow pumping equipment.

In general, all samples have a relatively large percentage of fine material, i.e. smaller than 25 µm, ranging from 27% to 45%. The effect of the fines is to cause the fluid parameters of the slurry to move away from the Newtonian model of viscosity. Newtonian fluids have a shear stress to shear rate ratio that is linear starting at zero. The slope of the line on a graph is the viscosity of the fluid. The slurries tested have a non-linear relationship and a definite yield stress that must be overcome before the fluid motion starts. This is described as a Bingham Plastic Fluid. The viscosity in this case is not constant until the flow reaches the relative straight part of the curve. The slope of the straight section is called the “coefficient of rigidity” by Pocock or sometimes called “plastic viscosity” by different test labs.

Underflow pulps with yield stress values in excess of 30 N/m^2 (Pascals) measured on pre-sheared pulps are normally beyond the capabilities of conventional thickening and pumping systems. Paste type thickeners and pumps would be required.

Plastic viscosity values also must be considered. In general, slurries with a coefficient of rigidity above 0.050 Pa-sec (50 centipoise) become a problem with longer pipelines because of the increase in pipe friction loss. The increased viscosity must be considered in the design of any agitators that will be handling the slurry.

The range of operation for underflow from the cyanide recovery thickener is planned to be 50% to 60% solids. Figure 13-34 indicates that only four of the fourteen samples tested can reach 60% solids before reaching a yield stress of 30 Pa and one of the samples Grinding Thickener Feed 4505-011 (Jumbo Rhy-Aph) cannot reach 50% before reaching a yield stress of 30 Pa. This ore type will be problematic if not blended to reduce the percentage of fines.

Based on the test data obtained, it appears that nearly all the material samples can be operated in the desired range of 50% to 60% solids.

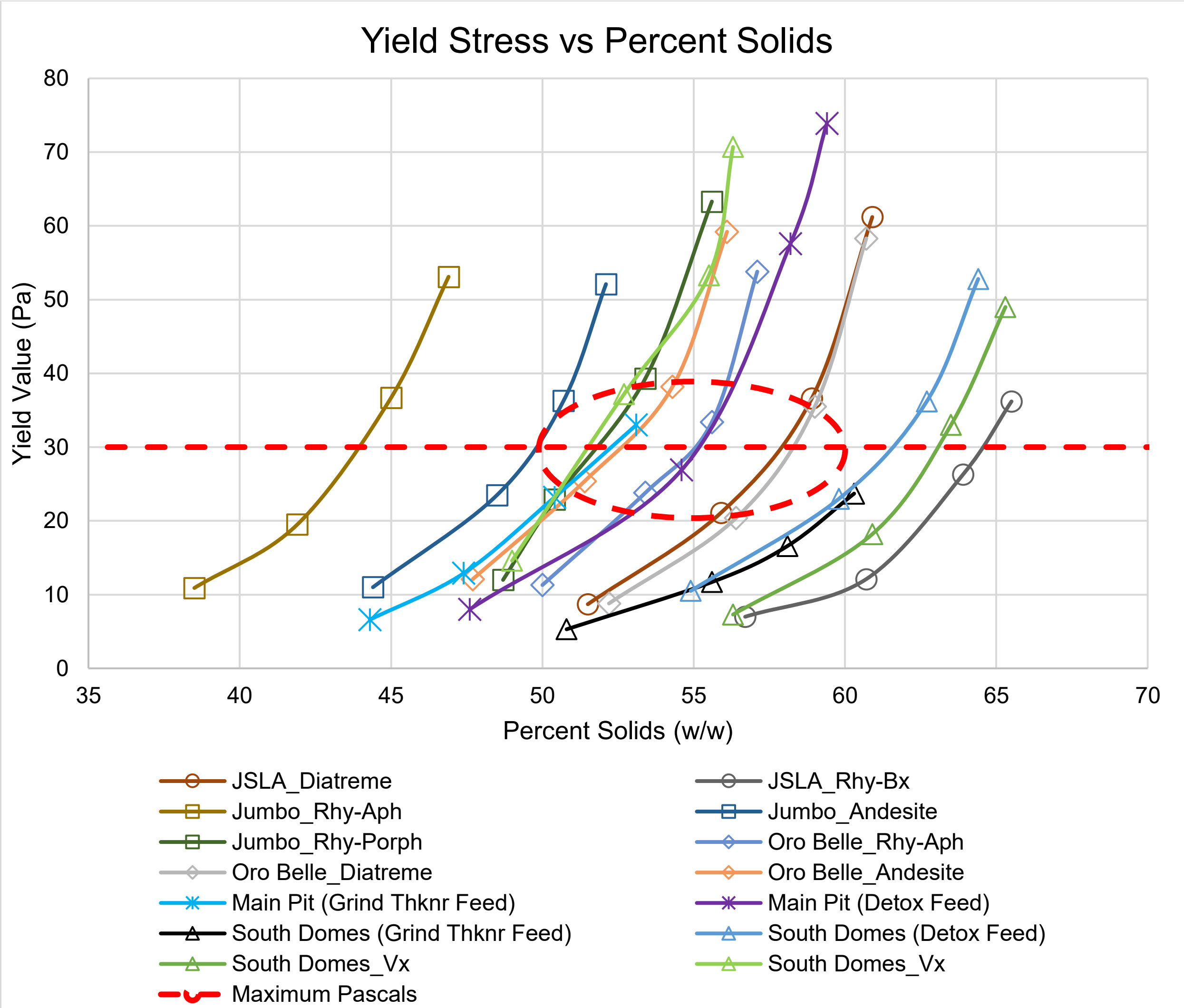


Figure 13-34: Yield Stress vs Percent Solids

13.8.8 Ore Sorting Tests

Bulk ore sorting is a pre-concentration technology currently used at many mining operations across the world. Sorting involves scanning individual rock particles on a conveyor using various types of sensor technologies. The readings for each individual rock are then analyzed by high-speed software (algorithms) to identify rocks with metal values (or other characteristics) above and below a pre-determined setpoint. At the end of the conveyor belt, focused high-pressure air jets or mechanical levers then separate the designated mill grade ore rock pieces for processing and reject heap leach grade and waste pieces. The amenability to ore sorting depends on the material characteristics of a deposit.

At Castle Mountain, the potential advantage of ore sorting is that heap leach material in the range of the mill grade ore could be upgraded, or the mid/low grade mill feed ore could be split from the mill grade ore and be sent to the heap leach pad, potentially lowering operating costs.

The objective of the testwork was to determine if Castle Mountain samples could be upgraded or if lower-grade ore could be “rejected” and be sent to a heap leach for treatment using sensor-based sorting. The ore sorting testwork was carried out by Steinert US at their test facility located in Walton, KY in January 2020.

The sorting testwork requires sample material that represents all rock types that may be mined on site. Samples for testing included approximately 400 kg of test material selected by Equinox personnel from material that was available from ongoing testwork. The samples for testing included three bulk samples and seven grab samples. The material has been sized to +12 mm to -50 mm.

A full-size Steinert Combined Sensor (KSS FLI XT) was used for the testing. This sorter includes the following sensors:

1. Dual-energy X-ray transmission sensors (XRT),
2. Color camera (F),
3. 3-D Laser (L), and
4. Induction (I).

Based on preliminary work, it was determined that XRT and color would be the sensors used to test the Castle Mountain material.

The XRT scans conducted on these samples showed marginal separation. This is most likely due to the relatively similar densities of the gold-bearing material and the barren waste rock.

Color scans conducted on the samples showed little upgrade. Metal recovery appeared tied to mass pull.

Precious metal values in the Castle Mountain mine ore (samples tested) appear to be highly disseminated and are likely not amenable to bulk ore sorting.

No additional ore sorting testing is recommended at this time (Steinert, 2020).

13.9 PRODUCTION DATA

Production data has been used to validate recent post-production testwork, although the life of mine grade during production was higher than current life of mine grades.

The Castle Mountain mine was in production from 1992 until 2004. Ore was stacked on the leach pad for nine years from 1992 to 2001. Processing and recovering gold continued for an additional three years until 2004 as part of rinsing and cleanup during closure.

The initial process plant commissioned in 1992 included heap leaching of ore that was crushed to 100% - 9.5 mm in a three-stage crushing circuit. The plant incorporated a milling circuit in 1993 that agglomerated low grade heap leach ore with partially leached mill tailings. The upgraded system followed this process:

1. Mill-grade ore was ground with cyanide to 80% passing 100 mesh (150 μm) using a ball mill.
2. Ball mill product (cyclone overflow) was directed to a thickener for solid/liquid separation.

3. Thickener overflow (pregnant solution) was sent to the carbon circuit (ADR) for gold recovery. Mill tails (thickener underflow) was agglomerated using a 1:10 ratio with the crushed heap leach ore and sent to the heap leach pad for additional leaching.

The gold extraction by leaching in the mill averaged 35% prior to being agglomerated with the crushed ore for the heap leach.

In 1995, a gravity gold recovery circuit was added to the mill, with annual gravity recoveries ranging from 13% to 22%. The gravity/cyanided tailings were then agglomerated with the crushed ore as before. After Castle Mountain added the gravity circuit, the combined recovery in the mill (gravity plus leaching) increased to approximately 45-50%.

Total gold recovery of high-grade ore from the mill from gravity concentration, milling in cyanide solution, and heap leaching of the agglomerated slurry was estimated by Castle Mountain to be 95%. See Table 13-40 for a summary of the recoveries obtained during production throughout the various stages.

Table 13-40: Recoveries as Estimated by Castle Mountain During Production for High-Grade Ore

Gold Extraction Recovery Source	Recovery
Mill	
1993 leaching in the milling circuit (prior to heap leaching on the pad)	35%
1995 gravity gold recovery circuit	13-22%
Combined mill with gravity and leaching	**45-50%**
Heap Leach	
End of 2001 heap leach ore recovery (prior to rinsing)	71%
Life of mine heap leach ore recovery (with 3 years of rinsing)	77%
Total Life of Mine	**95%**

Production records show that, over the life of the Mine, 36.9 Mton of ore with a gold grade of 0.043 oz/ton (1.47 g/t) containing approximately 1.55 Moz gold were loaded onto the heap leach pads.

The 36.9 Mton comprise approximately 34.2 Mton with a gold grade of 0.037 oz/ton (1.27 g/t) which was fed directly to the heap leach circuit and 1.967 Mton at 0.144 oz/ton (4.9 g/t) that was placed on the leach pad from agglomerated tailings from the milling circuit.

Over the life of the mine, approximately 80% of the stacked gold was recovered (total leach-grade and mill-grade combined recovery). Approximately 12% of this total production of gold was recovered in the 43 months following the end of mining. A summary of the ore production and metallurgical recoveries by year is shown in Table 13-41.

Table 13-41: Castle Mountain Mine Production Data

Item	Units	Total	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004
Total, Leach Plus Mill															
Grade	oz/ton	0.043	0.051	0.058	0.056	0.050	0.037	0.038	0.027	0.034	0.040	0.037	-	-	-
Annual Recovery	%	-	59.3	63.0	74.0	74.3	80.6	78.5	84.8	67.8	72.0	89.9	-	-	-
Cumulative Recovery	%	80.2	59.3	61.6	66.6	68.6	70.6		71.7	72.9	72.3	72.3	75.0	78.7	79.7
Mill Ore															
Grade	oz/ton	0.144	-	0.183	0.210	0.162	0.113	0.138	0.108	0.138	0.108	0.113	-	-	-
Average Grind/Gravity Recovery	% of mill feed		-	35.9	33.5	37.9	42.3	49.7	45.0	50.5	48.7	54.9	-	-	-
Average Leach Recovery	% of mill tailings		-	92.2		91.9	91.3	90.1	90.9	89.9	90.3	88.9	-	-	-
Mill Recovery	% of mill feed	95	0.0	95	95	95	95	95	95	95	95	95	0.0	0.0	0.0
Leach Ore															
Grade	oz/ton	0.037	0.051	0.055	0.044	0.042	0.031	0.027	0.026	0.032	0.034	0.031	-	-	-
Cumulative Recovery	%	76.9		59.3	59.9	61.9	63.5	65.4	65.9	67.7	67.4	67.2	70.6	75.1	76.2

Source: Castle Mountain Mining Company Limited – Castle Mountain Project Technical Report NI 43-101, May 30, 2014, page 13-6

13.10 RECOVERY SUMMARY

13.10.1 Overall Heap Leach Recovery

After evaluating the column leach tests by feed size, ore zone, and lithology, the arithmetic average gold recovery was 81%, 80% and 80% respectively. A weighted gold recovery based on ounces per lithology was calculated as 82%. The average lab recovery for Castle Mountain low grade ore was 80%.

To estimate the Castle Mountain gold recovery for the production heap leach from the lab data, operating and environmental conditions were considered. This includes ROM particle size distribution, permeability, effective leaching of the side slopes, etc. The ROM material for the Castle Mountain Project is predicted to have an F_{80} of 152 mm to 203 mm. When considering the ore size and other data, a lab to field deduction of 6% was applied to the average lab recovery of 80% for an expected LOM heap leach gold recovery of 74% after solution application is stopped. To account for the typical time impact in recovering gold from a large leach facility at closure, the expected gold recovery during LOM operations is considered to be 67% with a final recovery of 74% attained only after extracting residual gold values and reducing cyanide levels in the heap. This is expected approximately three years after mining has ceased.

13.10.2 Overall Mill Grade Ore Recovery

For this Feasibility Study, gravity followed by gravity tail leach in a CIL circuit was selected for the process based on economics. An overall gold recovery of 94% is expected from mill grade ores processed through the mill after 24-hour hybrid leach/CIL retention time. The plant has been sized conservatively with 30 hours retention time in the leach/CIL tanks.

14 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

Equity Exploration Consultants Ltd. (Equity) completed a mineral resource estimate update for Equinox's Castle Mountain Project, inclusive of the JSLA pit backfill material and in-situ hard rock mineral resources. This Mineral Resource Statement supersedes the previous statement completed by Mine Technical Services Ltd. (MTS) with an effective date of March 29, 2018 that is summarized in the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018).

The Mineral Resources presented conform with the most recent CIM Definition Standards (CIM, 2014), have been prepared according to CIM Best Practice Guidelines (CIM, 2019), and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101 Standards of Disclosure for Mineral Projects (BCSC, 2016). The resource estimate was completed by Trevor Rabb, P.Geo., of Equity. Mr. Rabb is a qualified person independent of Equinox and its wholly owned subsidiaries in accordance with NI 43-101 guidelines, meeting requirements of education, project experience, and affiliation to a recognized professional association. The Mineral Resource Statement for the Castle Mountain Project is presented in Table 14-20 of Section 14.13.

This section summarizes the methodology, data and validation techniques used by Equity in estimating the mineral resources for the Castle Mountain Project.

14.2 METHODOLOGY

The Mineral Resource estimate detailed in this report was prepared using Leapfrog v5.0 and Micromine 2020 software, for both the JSLA backfill and hard rock resource models. Both software packages were used for modelling, geostatistical and estimation purposes. Geostatistical evaluation and estimation were completed in metric units and converted into imperial units (troy ounces per short ton and short tons). The geologic hard rock models and backfill model were both generated using Leapfrog and estimation was completed using Micromine. The main steps of the methodology used were as follows:

- Site visit for audit of exploration program practices and review of local geology.
- Review of 2018 MTS Mineral Resource estimate and updated geological models.
- Database review and audit.
- Modification of geological model.
- Sample compositing.
- Capping study on primary and composited samples.
- Spatial statistics on primary and composited samples.
- Variography of composited samples and historic production blasthole samples for each resource area.
- Gold grade estimation using composited samples.
- Validation of gold grade estimates.
- Classification of Mineral Resources in accordance with CIM definitions (CIM, 2014).
- Constraining of estimated blocks using an optimized pit.
- Comparison of Mineral Resource estimate compared to historical blasthole sample data and historical production records.
- Converting results in metric units to imperial units.

The main differences between the 2020 and 2018 resource models were:

- Classification was informed using drill hole spacing.
- Estimation of the hard rock Mineral Resources using grade shell domains, including a low, medium and high-grade domain with thresholds respectively of 0.0050 oz/ton, 0.0146 oz/ton and 0.0321 oz/ton (0.17 g/t, 0.50 g/t Au and 1.10 g/t Au).
- Unmineralized rock types were excluded from grade shell domains.
- Estimation of the JSLA backfill domains using solids generated from monthly as-built mine site survey records.
- Bench compositing completed on 20 ft (6.10 m) bench heights using the same bench intervals as the block model.
- Depletion of hard rock Mineral Resources to account for current topography and monthly Viceroy as-built surveys.
- The use of blended grades and density based on partial block partial percentages where there are overlaps between the backfill and hard rock models.
- The use of anisotropic searches based on lithological domains that exert strong controls on mineralization.

14.3 DRILL HOLE DATABASE

Equinox provided separate drill hole databases for the hard rock and backfill mining areas. A more thorough explanation of the drill hole data used is provided in Sections 10 through 12 of this report. A summary of the drill hole and surface data used for estimation is shown in Table 14-1. Due to some RC pre-collars drilled for main hole diamond drilling, the total footage may not sum.

Table 14-1: Summary of Drill Hole and Surface Data Used for Resource Estimate

Material Type	Hole Type	Hole Count	Total (ft)	Total (m)
Backfill	CORE	113	130,093	39,652
	RAB	1265	26,842	8,181
	RC	278	212,937	64,903
	RC Pre-collars	23		
Hard Rock	DDH	206	202,819.5	61,819
	RC	1391	1,064,273	324,390
	Rotary	479	258,810	78,885
	Channel	20	4,118.3	1,255

The condemnation drilling described in Section 10.2.5 and 11.2.3 was excluded in the preparation of the Mineral Resource estimate as the data was unavailable at the time of evaluation. Condemnation drill holes occur outside of the mineral resource pits (see Figure 10-4). It is therefore the opinion of the QP that the exclusion of the condemnation drilling from the preparation of the Mineral Resource are immaterial to the Mineral Resources disclosed in Table 14-20 and Table 14-21.

14.4 GEOLOGICAL MODELLING

Equinox provided Equity with four geologic models supporting the hard rock resource:

1. A lithology model,
2. An alteration model that includes clay and silicification,
3. An oxidation model that includes oxide, transition and fresh rock domains, and
4. A fault model.

Backfill and depletion models were generated for the backfilled JSLA pit and mined out portions of Castle Mountain Project, respectively. All models were generated in Leapfrog v5.0 software using a pre-mining topographic surface.

14.4.1 Topography

Three topographic surfaces were generated, representing:

1. Current topography,
2. Pre-mining topography, and
3. End of mine prior to backfilling.

The topographic surfaces were based on the Viceroy as-built pit surveys, 2017 Compass Tools LiDAR survey, 2018 Lanfair survey and 2018 PhotoSat survey.

Pre-mining topography was based on Viceroy as-built surveys that predate mining activity. Portions of the pre-mining topographic surface were informed by the 2017 and 2018 LiDAR surveys, specifically where no mining or activity had altered the topography.

The topographic surface representing the end of mine, prior to backfilling of the JSLA open pit, was generated using the 2017 and 2018 LiDAR surveys and end of mining Viceroy as-built surveys for the entire JSLA pit and northern portions of the Jumbo pit where sloughing and pit-wall failures have occurred.

14.4.2 Lithology Model

The lithology model was created from published geological mapping, drill logs from drilling completed by Equinox and interpretation of historic Viceroy drill hole logs. The lithology model interacted with a fault model (see Section 14.4.7). Rock types representing barren or weak to non-mineralized lithologies including post-mineralization units and poor host rock lithologies were grouped together and excluded from grade shell domains (see Section 14.4.6). The main lithologies are summarized in Table 14-2 and shown in Figure 14-1.

Table 14-2: Castle Mountain Lithology Codes

Lithology	Lithology Model Code	Model Code	Grouping
Alluvium	02-Alluvium	2	Barren, post mineralization
Debris Flow	03-DebrisFlow	3	Barren, post mineralization
Dacite	05-Dacite	5	Barren, post mineralization
Hart Peak Rhyolite	05-HartRhy Migos	5	Barren, post mineralization
Rhyolite Breccia	07-RhyBx	7	Mineralized
Porphyrytic Rhyolite	09-RhyPorphyritic	9	Weak to non-mineralized
Aphyric Rhyolite	11-RhyAphyric	11	Weak to non-mineralized
Volcanoclastic Diatreme	14-VxDiatreme	14	Mineralized
Volcanoclastic	16-Vx	16	Mineralized
Mudstone	22-Mudstone	22	Mineralized
Epiclastics	23-Epiclastics	23	Mineralized
Andesite	27-Andesite	27	Mineralized
Peach Springs Tuff	29-PeachSpring	29	Mineralized
Proterozoic Sediments	30-PcSediments	30	Mineralized
Proterozoic Basement	31-PcBasement	31	Mineralized

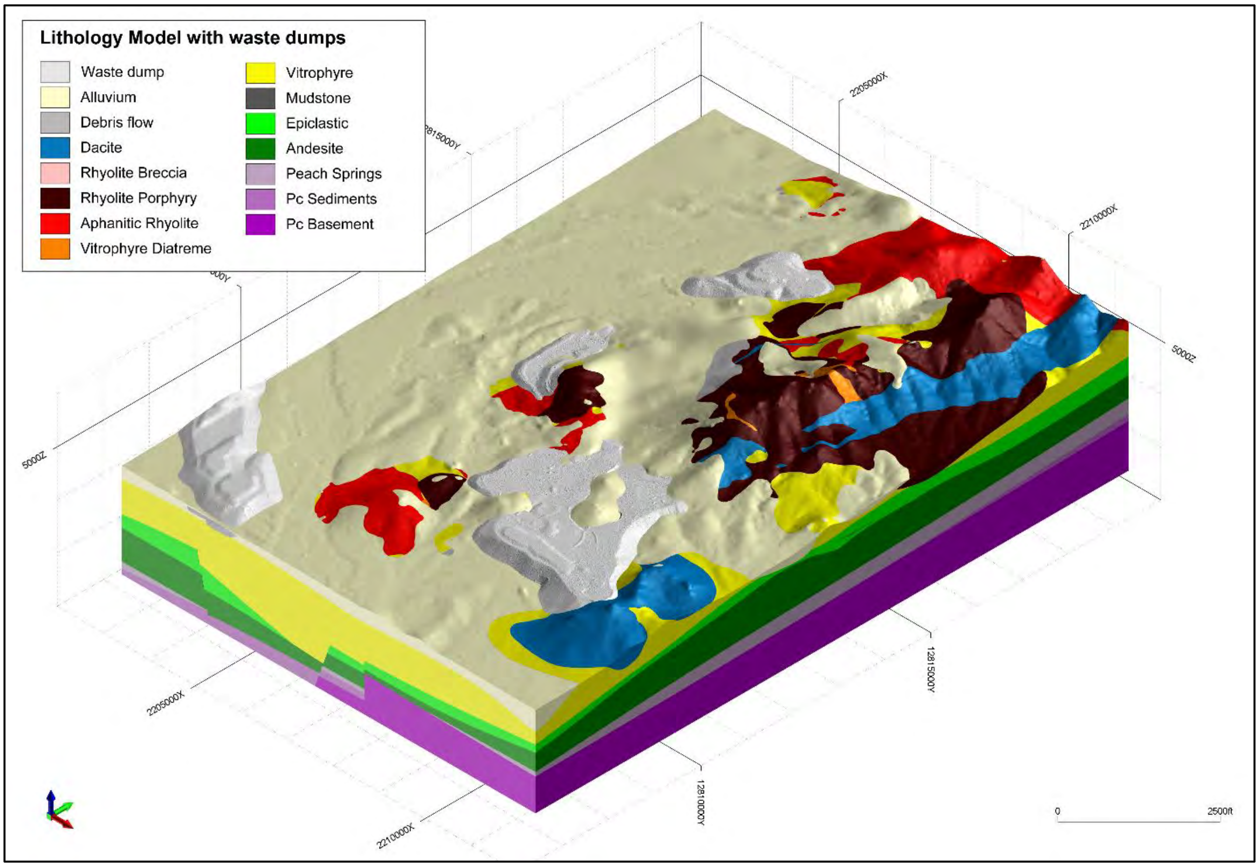
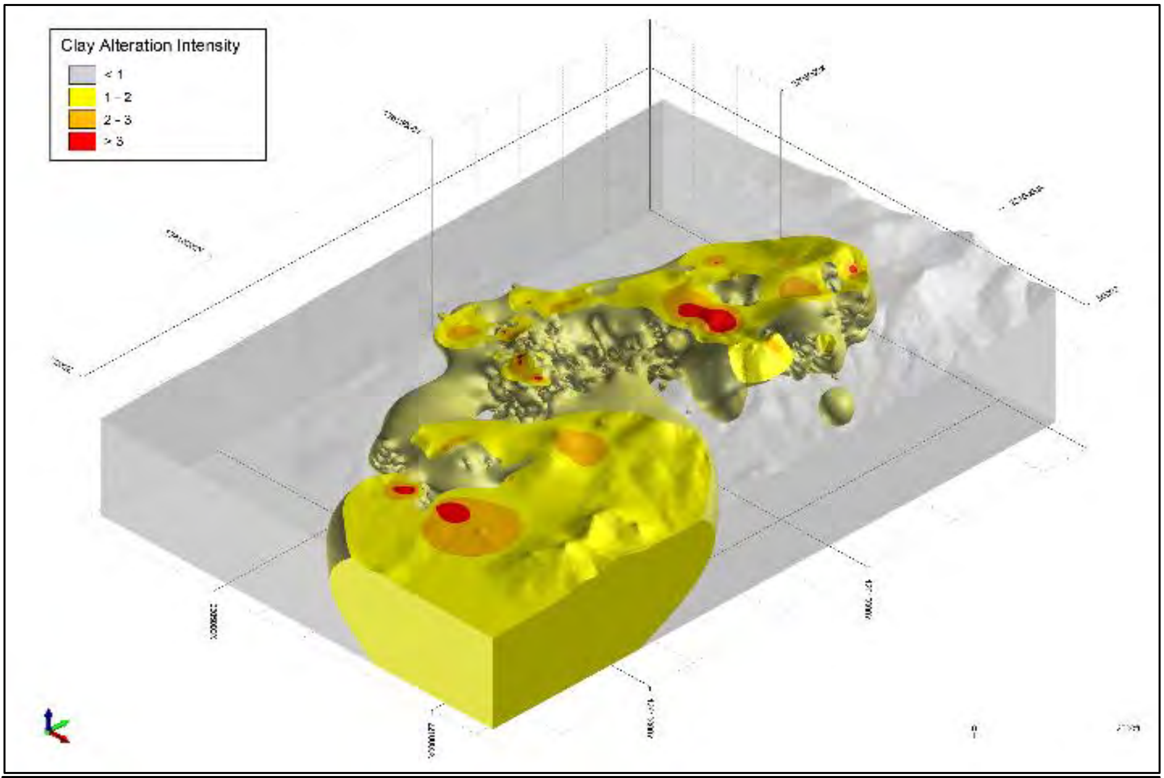


Figure 14-1: Castle Mountain Lithology Model (View Towards Northwest)

14.4.3 Alteration Model

The alteration model for the Castle Mountain Project includes alteration intensity wireframes for silicification and clay alteration. The alteration intensity models were generated by imputing numeric values to the alteration textures: on fractures = 0, patchy = 1, and pervasive = 2. For historic holes without logged alteration texture, a value of 1 was imputed. The numerical textural rankings were added to the numerical intensity rankings, ranked 0 to 4 in order of increasing intensity, for a final alteration intensity index. Wireframes representing the final alteration index were generated using Leapfrog radial basis function (RBF) numerical modelling features to generate wireframes corresponding to alteration indexes of < 1, 1 – 2, 2 – 3 and > 3 representing trace, weak, moderate, and strong alteration. The alteration model was generated for use as an exploration vectoring tool and does not have direct application to determining geotechnical or geometallurgical parameters. Alteration intensity models for clay and silicification are shown in Figure 14-2.



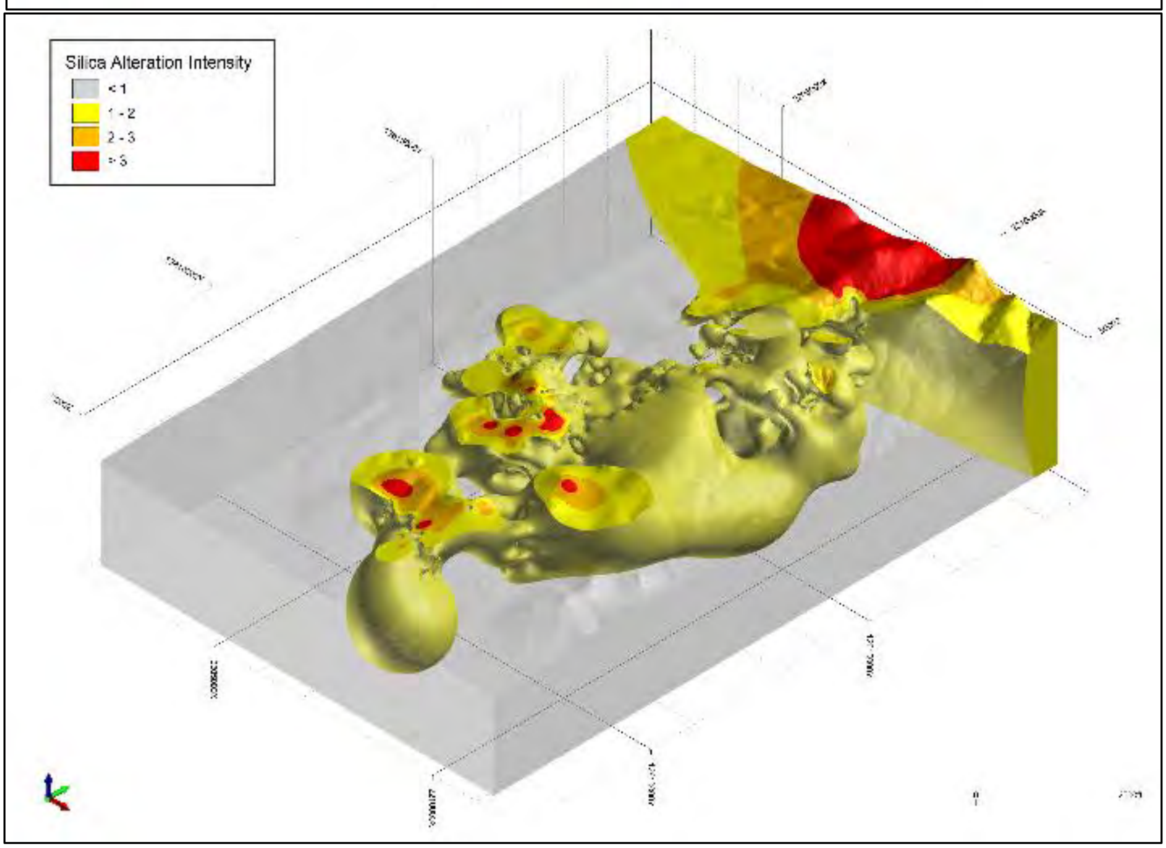


Figure 14-2: Castle Mountain Alteration Model for Clay and Silica (View to Northwest)

14.4.4 Oxidation Model

The oxidation model for Castle Mountain was generated based on numerically ranked visual abundance of iron oxide (0 to 4), and visible pyrite abundance (0 to 5) using traditional field methods applied to core and chip logging. If the ranked pyrite abundance exceeded that of iron oxide, the interval was assigned to sulfide. If the ranked abundance of pyrite and iron oxide were equal or within one, the interval would be assigned to transition. The assignment to transition also considered intervals uphole and downhole. Intervals lacking data were excluded from the model. Lithologies excluded from the oxide model include debris flow and alluvium as both units are young and post-date mineralization.

Average thickness of the modelled oxide domain is 1,000 ft (305 m), with the thickness increasing locally up to 2,000 ft (610 m) around faults. Thickness of the modelled transition domain is variable and is not well developed over the area. Average thickness of the transition zone where present is 180 ft.

The models generated using the above method were compared to sample intervals where cyanide extractable gold assay (AuCN) values and fire assay gold values both occur. Due to the limited coverage of AuCN values, only the oxidation models of the South Domes and Six 21 areas could be validated using AuCN and gold fire assay geochemistry. Within these areas, the transition and sulfide zones were found to have AuCN to gold fire assay ratios less than 0.6, whereas oxide zones typically had AuCN to gold fire assay ratios greater than 0.6, and average 0.8. Oxidation models are shown in Figure 14-3.

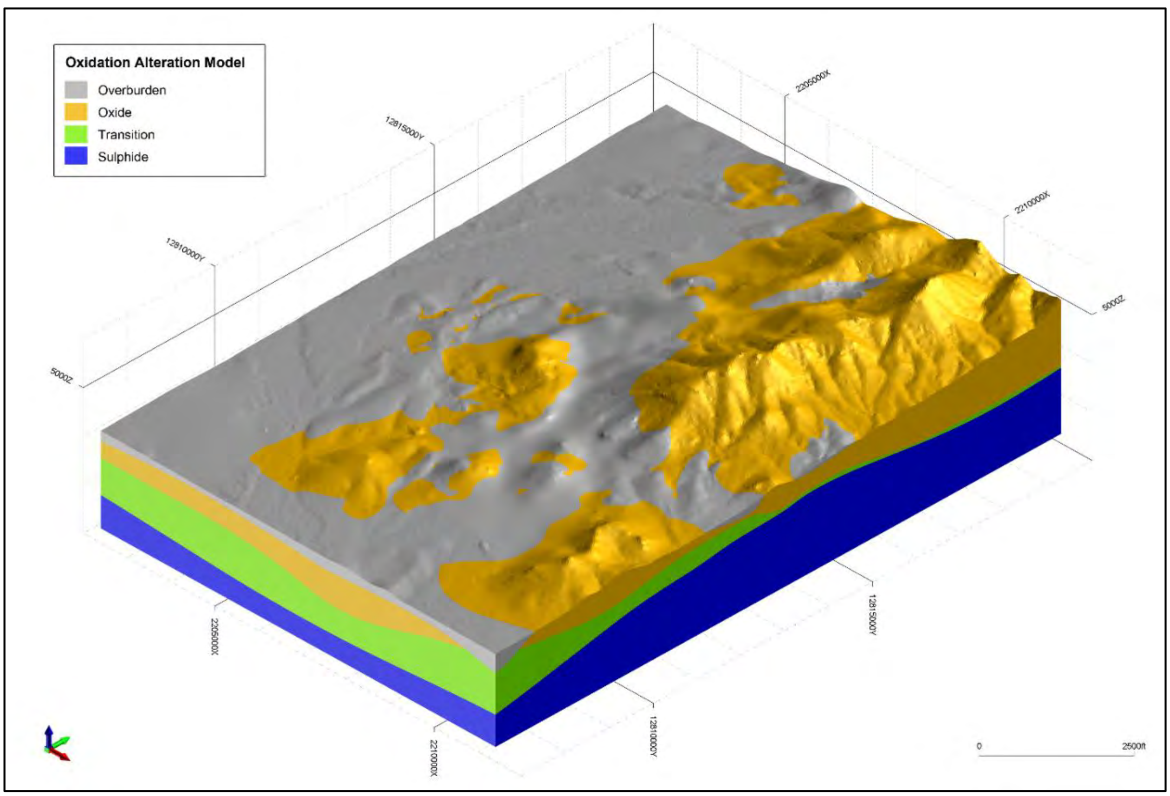


Figure 14-3: Castle Mountain Oxidation Model (View to Northwest)

14.4.5 Fault Models

Five major displacement faults and four minor displacement faults were modelled throughout the deposit. The major faults introduced up to 300 ft (91 m) of vertical offset between of lithologies and highlight mineralized structures. The faults present within the resource model are summarized in Table 14-3 and shown in Figure 14-4.

Table 14-3: Summary of Modelled Faults

Fault Name	Dip	Dip Direction	Vertical Offset (ft)	Sense	Displacement
Dillon	71	283	200 to 250	Normal Fault, East side down	Major
Maverick East	51	113	200 to 300	Normal Fault, West side down	Major
Maverick West	78	112	0 to 120	Normal Fault, West side down	Major
McLane	60	128	100 to 300	Normal Fault, West side down	Major
Predator	81	103	75 to 250	Normal Fault, East side down	Major
Ripley	81	121	-	-	Minor
Gruber	88	340	-	-	Minor
Ice Man	87	158	-	-	Minor
Connor	85	305	-	-	Minor

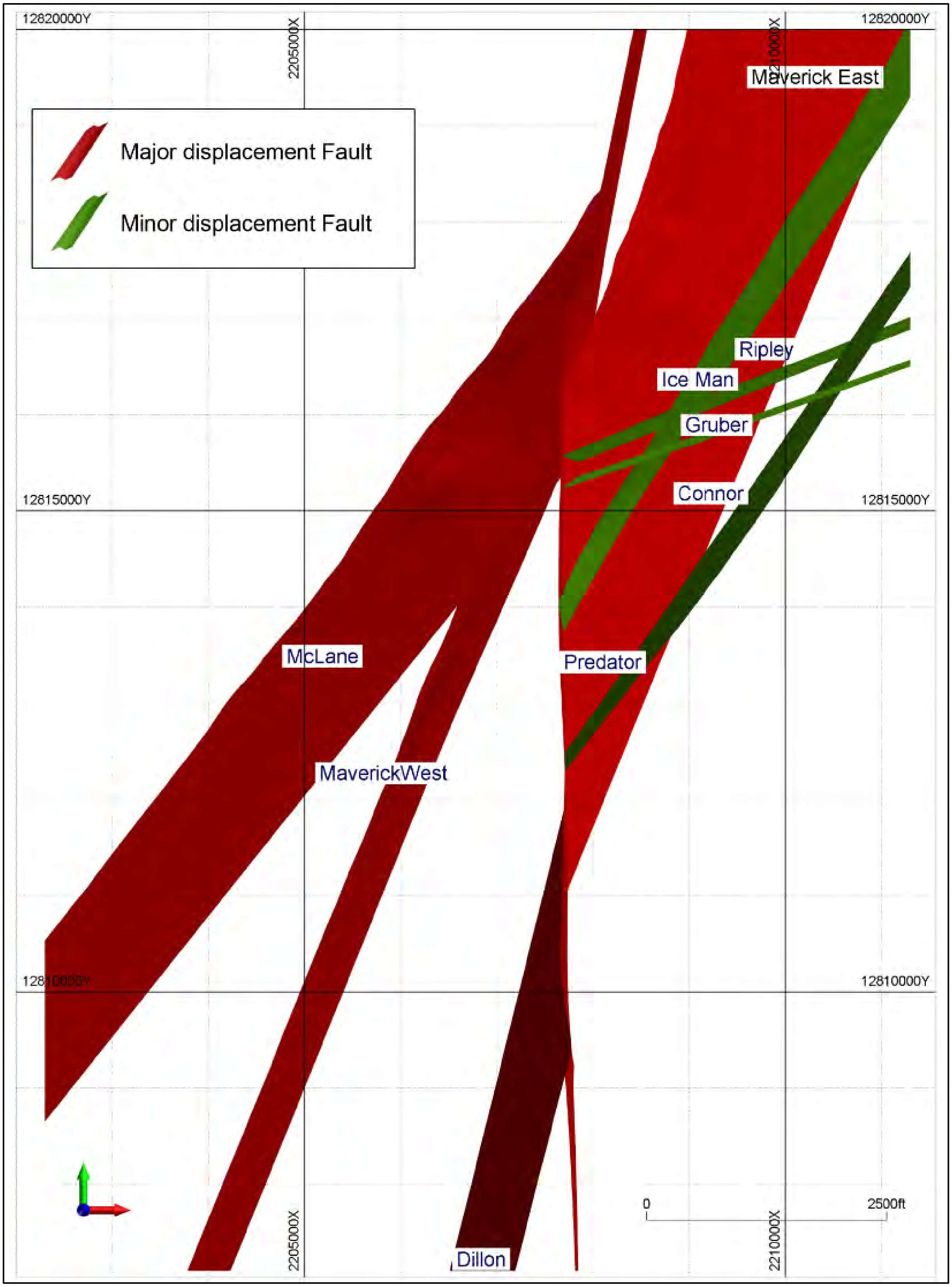


Figure 14-4: Plan View Showing Modelled Faults

14.4.6 Grade Shell Domains

Four grade shells were developed to use as estimation domains. The grade shells were generated in Leapfrog v5.0 software using bench height composite samples at three different grade thresholds 0.0050 oz/ton, 0.0146 oz/ton and 0.0321 oz/ton (0.17 g/t, 0.50 g/t, and 1.10 g/t) gold. The grade shells were generated with trends that represent the various controls on gold mineralization including the mineralizing fault network, the orientations of main lithological contacts and alteration wireframes representing elevated silicification. Final grade shells used for estimation excluded those portions of the grade shell consisting of barren or weak to non-mineralized lithologies including post-mineralization units and poor host rock lithologies. The grade shell domains are summarized in Table 14-4 and shown in Figure 14-5 to Figure 14-8.

Table 14-4: Summary of Domains

Domain	Domain Code	Grade Threshold (Au, oz/ton)	Grade Threshold (Au, g/t)	Number of Composite Samples	Average Grade (Au, oz/ton)	Average Grade (Au, g/t)
Barren or Weak to non-mineralized Lithologies	100	0.0015	0.05	3149	0.0015	0.05
Internal Waste	101	0.0015	0.05	11	0.0018	0.06
Low Grade	102	0.0050	0.17	10,891	0.0055	0.19
Medium Grade	103	0.0146	0.50	17,959	0.0134	0.46
High Grade	104	0.0321	1.10	4,638	0.0627	2.15

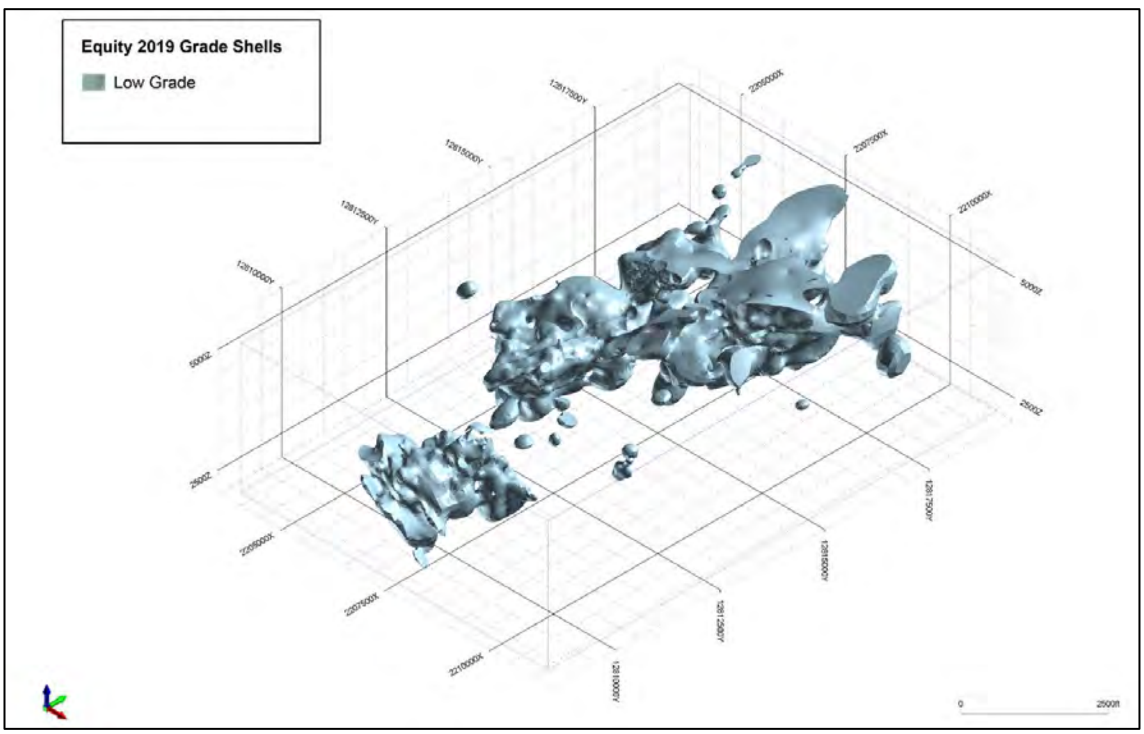


Figure 14-5: Low Grade Gold Domains (View to Northwest)

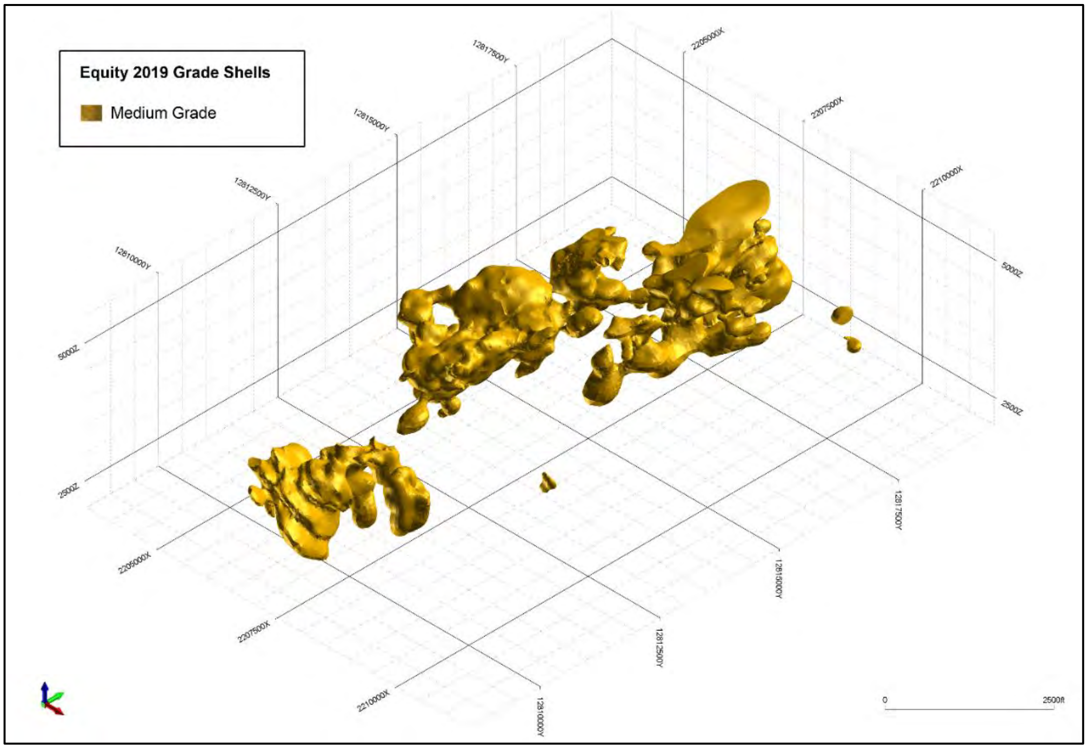


Figure 14-6: Medium Grade Gold Domains (View to Northwest)

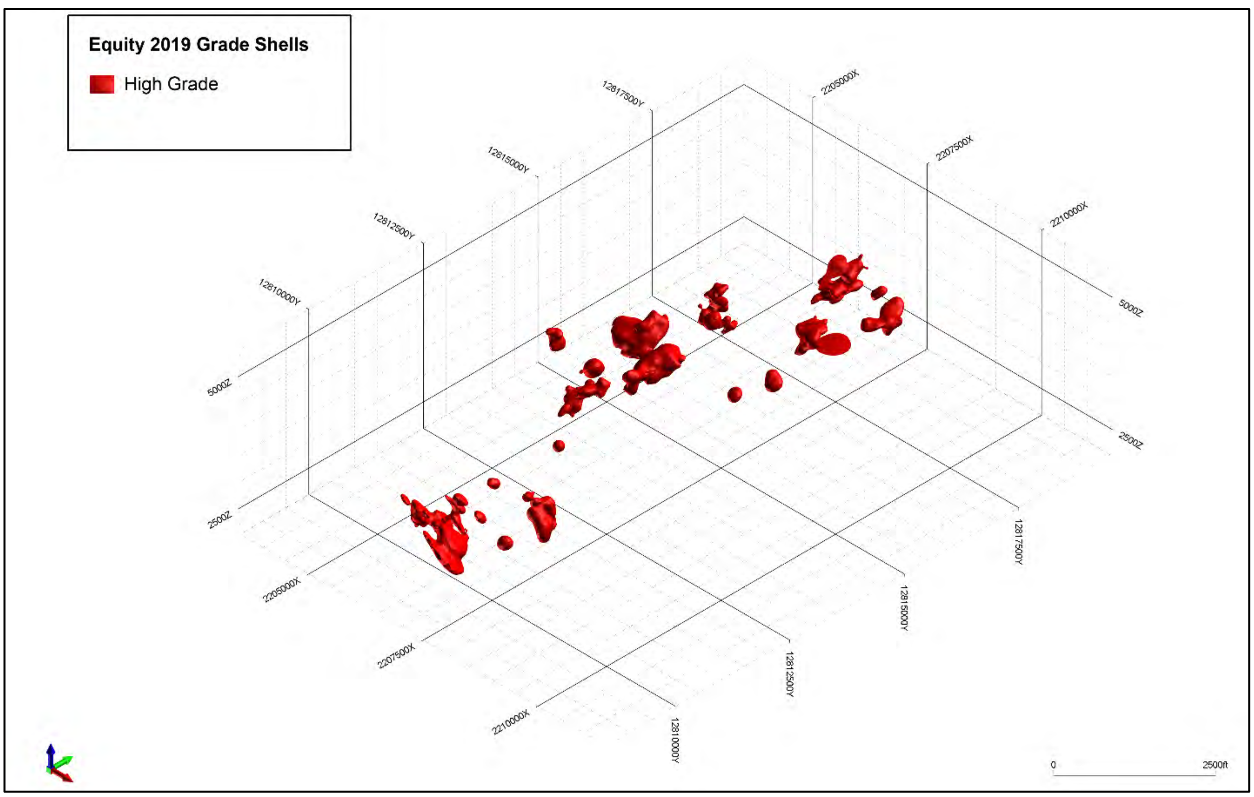


Figure 14-7: High Grade Gold Domains (View to Northwest)

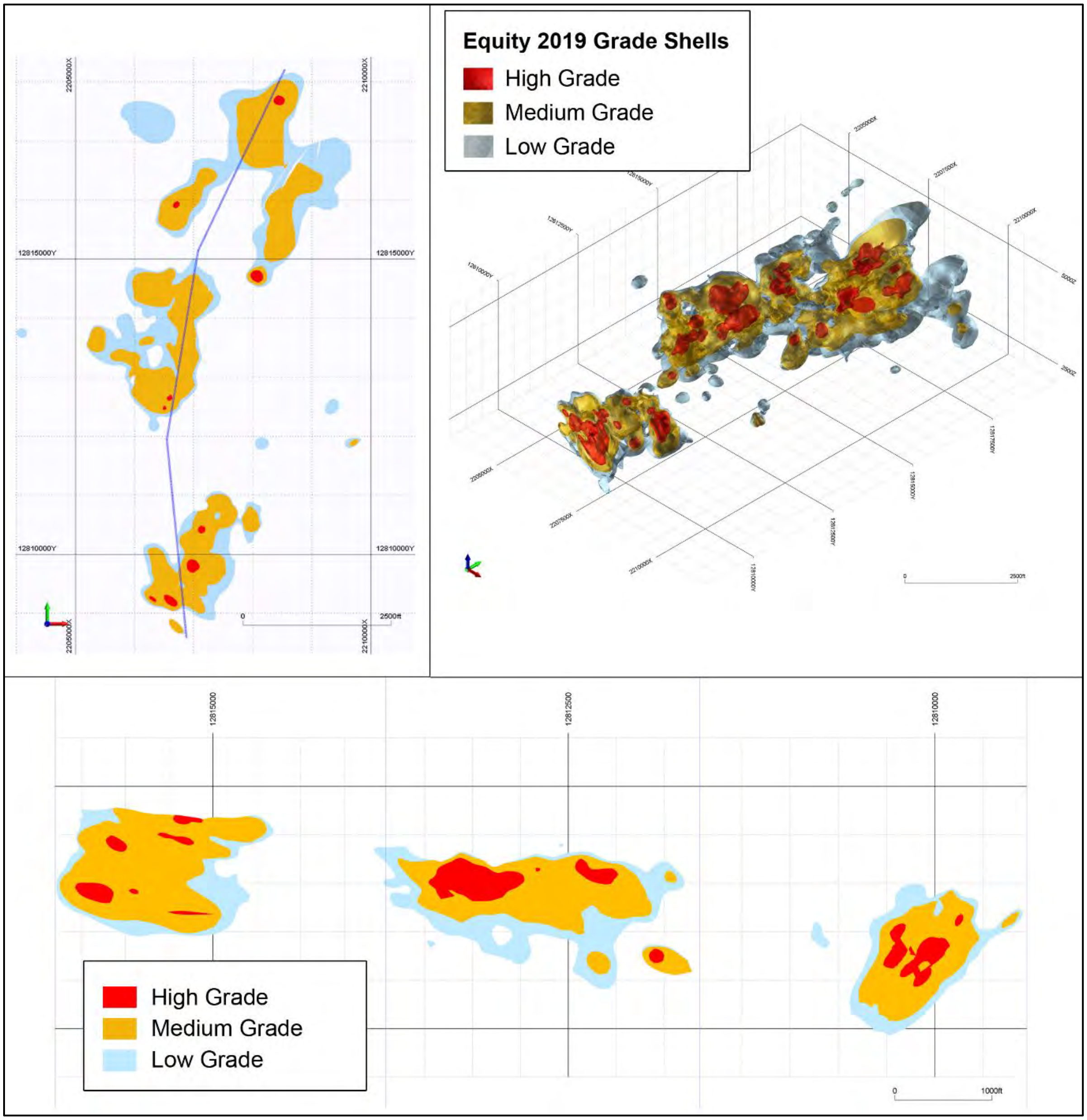


Figure 14-8: Composite Plan, Perspective & Long Section Views of Nested Grade Shell Domains

14.4.7 JSLA Backfill Model

The JSLA backfill model was developed independently of hard rock models and is based on the placement of backfill material into the JSLA pit. Temporal control for the backfill periods and final mined shape of the JSLA pit was developed using 43 monthly as-built surveys originally generated by Viceroy during active mining. The monthly as-built surveys were grouped into seven volumes based on backfill sequence and similarities between gold grades encountered in the backfill drilling. A summary of the monthly volumes and grouped backfill domains are summarized in Table 14-5 and shown in Figure 14-9.

Table 14-5: Summary of Backfill Stage Domains

Backfill Stage	Corresponding Monthly Units (year-month)	Volume (ft^3)	Volume (m^3)
1	96-10, 96-11, 96-12, 97-01, 97-02, 97-03, 97-04	140,510,000	3,978,800
2	97-05, 97-06, 97-07, 97-08, 97-09, 97-10, 97-11, 98-01, 98-02	227,930,000	6,454,300
3	98-03, 98-04, 98-05, 98-06, 98-07, 98-08, 98-09, 98-10, 98-11, 98-12, 99-01, 99-02, 99-03	246,630,000	6,983,800
4	99-04, 99-05, 99-06, 99-07, 99-08, 99-09, 99-10, 99-11, 99-12, 00-01, 00-02	232,400,000	6,580,800
5	00-03, 00-04, 00-05, 00-06, 00-08	22,753,000	644,290
6	00-07, 00-09, 00-10, 00-11, 00-12, 01-01, 01-02	31,411,000	889,460
7	01-03, 01-04	6,180,900	175,020

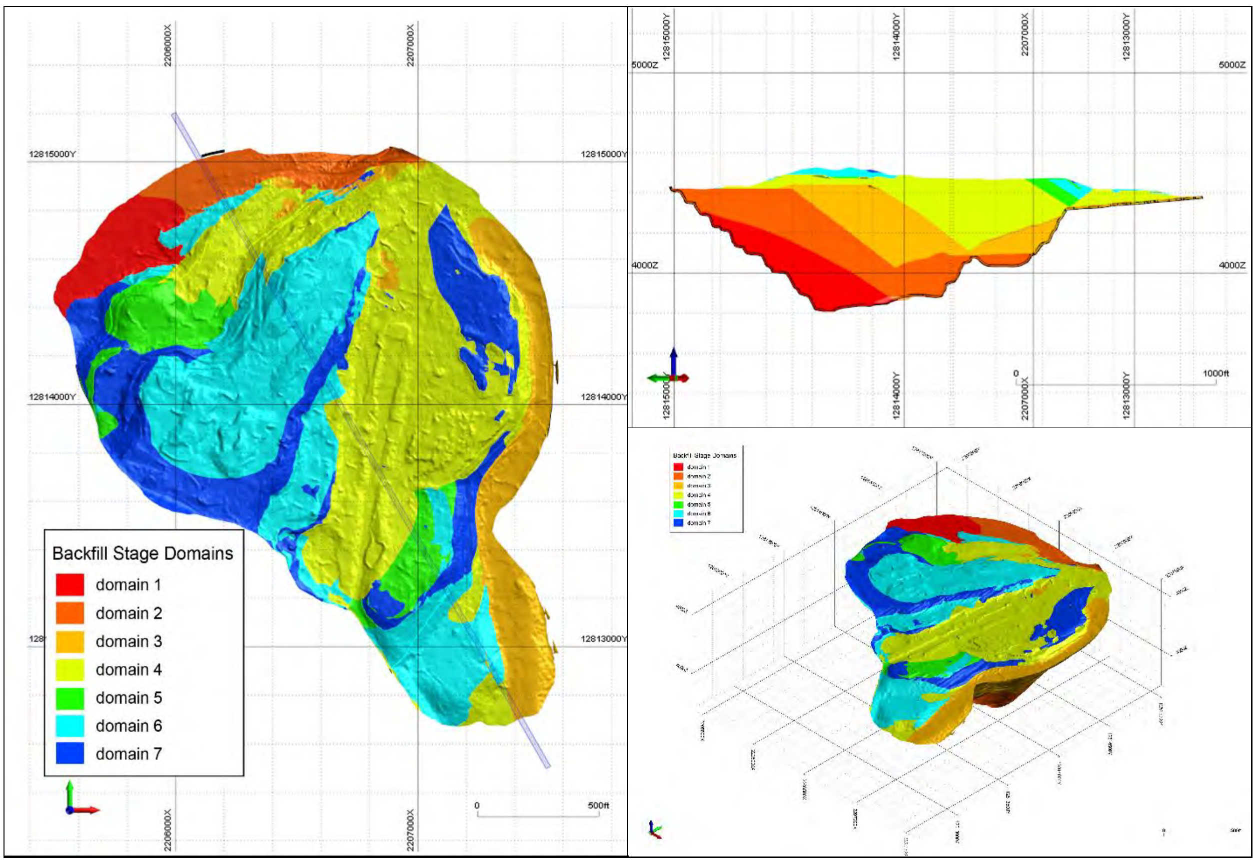


Figure 14-9: Composite Plan and Perspective View (Looking Northwest) of the JSLA Backfill Stage Domains

14.5 COMPOSITING AND CAPPING

Composites were generated for drill holes within the backfill and hard rock resource areas, and for surface channel samples within the hard rock resource areas. Compositing was completed separately for the drill holes and channel samples. The sampling of drill holes at Castle Mountain Project was predominantly done at five-foot intervals. To reduce variability and match the mining selectivity at Castle Mountain Project, samples were composited to twenty-foot bench height intervals. Holes with missing intervals were composited to the respective bench height interval, with gold values length-weighted using the original sample lengths contained within the bench interval. This was done to avoid inserting zero grades for missing samples and unnecessarily diluting the model. Composite samples that were less than 2.5 ft (0.8 m) or samples with less than 10% coverage within the bench intervals were discarded. Channel samples were composited to 30 ft (9.14 m) lengths along the channel sample trace. Residual composite channel samples with lengths of less than 10 ft (3.05 m) were backstitched and comprise the final interval.

14.5.1 Hard Rock Model

Original samples were capped at 4.3750 oz/ton (150 g/t) gold prior to compositing to limit the influence of extreme outliers during the compositing process. In total, 18 samples were capped to 4.3750 oz/ton (150 g/t) gold. Due to the historic nature of some of the Viceroy assay data, there is significant benching of the data at the historic analytical detection limit of 0.0010 oz/ton (0.005 g/t) and increasing in 0.0010 oz/ton increments corresponding to gold values of 0.0010 oz/ton, 0.0020 oz/ton, 0.0030 oz/ton, 0.0040 oz/ton and 0.0050 oz/ton (0.034 g/t, 0.069 g/t, 0.103 g/t, 0.136 g/t and 0.170 g/t). To assist in determining if applying half the detection limit to historic Viceroy samples is appropriate for bottom cutting, statistical analysis of recent drilling was examined, which have lower detection limits of 0.00015 oz/ton (0.005 g/t) gold. Statistical analysis was completed on new drilling for values between 0.00015 oz/ton and 0.00100 oz/ton (0.005 and 0.034 g/t) gold, and then from 0.00007 oz/ton to 0.00100 oz/ton (0.0025 to 0.034 g/t) gold. In both circumstances, the distribution of these values is uniform, and exhibit mean values close to 0.0005 oz/ton (0.017 g/t) suggesting that a bottom cut value of 0.0005 oz/ton (0.017 g/t) for historic samples represents an average of expected values for samples below detection limit. Therefore, for historic Viceroy sample data, values reporting less than detection limit 0.001 oz/ton (0.034 g/t) gold were replaced with values representing half detection limit 0.0005 oz/ton (0.017 g/t) gold.

Gold grade capping of composited samples was performed globally for all domains. Final gold grade capping values were chosen based primarily on a top cut analysis utilizing probability plots, and covariance versus capping values (top cut values) for all three grade shell domains (Figure 14-10 to Figure 14-13). The treatment of domain boundaries and spatial continuity of high grade was considered for the final capping value, resulting in a final capping value of 0.8750 oz/ton (30 g/t) gold for all domains. This capping value is consistent with historical capping levels employed by Viceroy during operation of Castle Mountain Mine. A summary of the comparison between raw assays and capped composited samples is shown in Table 14-6.

The resource model is sensitive to capping methodology and values. Capping prior to compositing was investigated to assess the resource model's sensitivity to the capping and compositing strategy. Capping on raw assays was completed using top cut values appropriate to each grade shell domain. The results indicate that the estimated gold grades and metal content are comparable and are within 5% between the two methods of capping raw assays and capping composite samples.

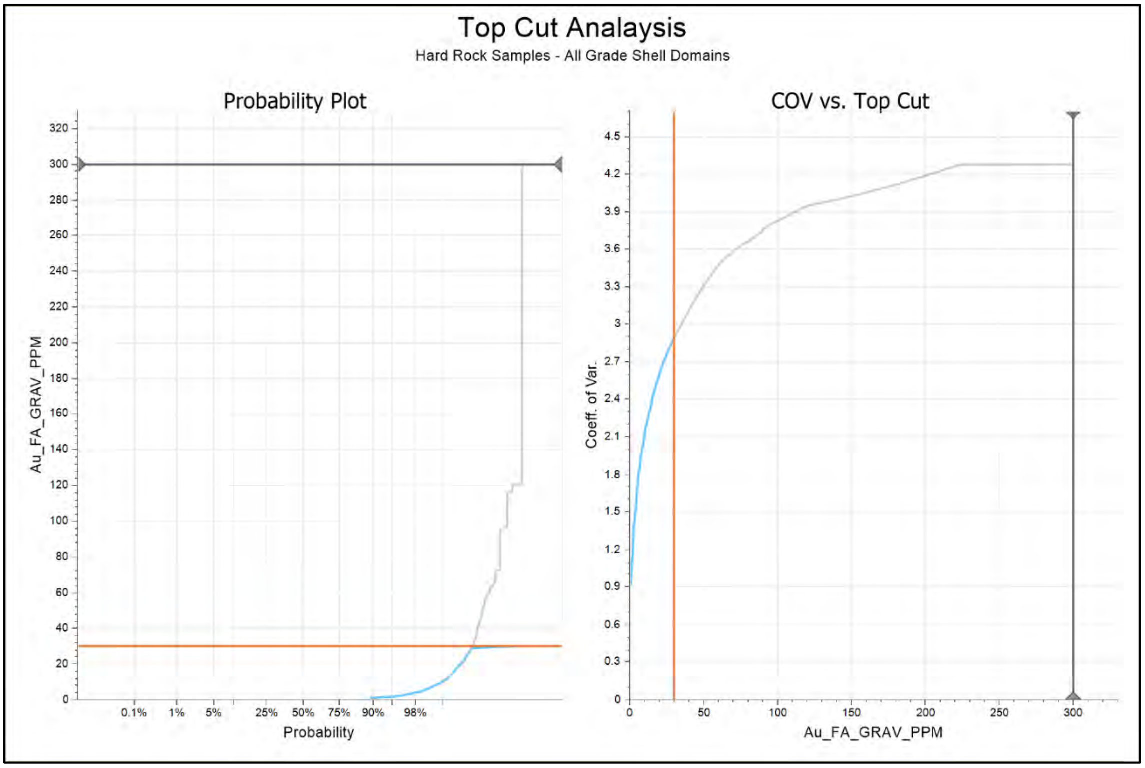


Figure 14-10: Top Cut Analysis of all Composite Samples

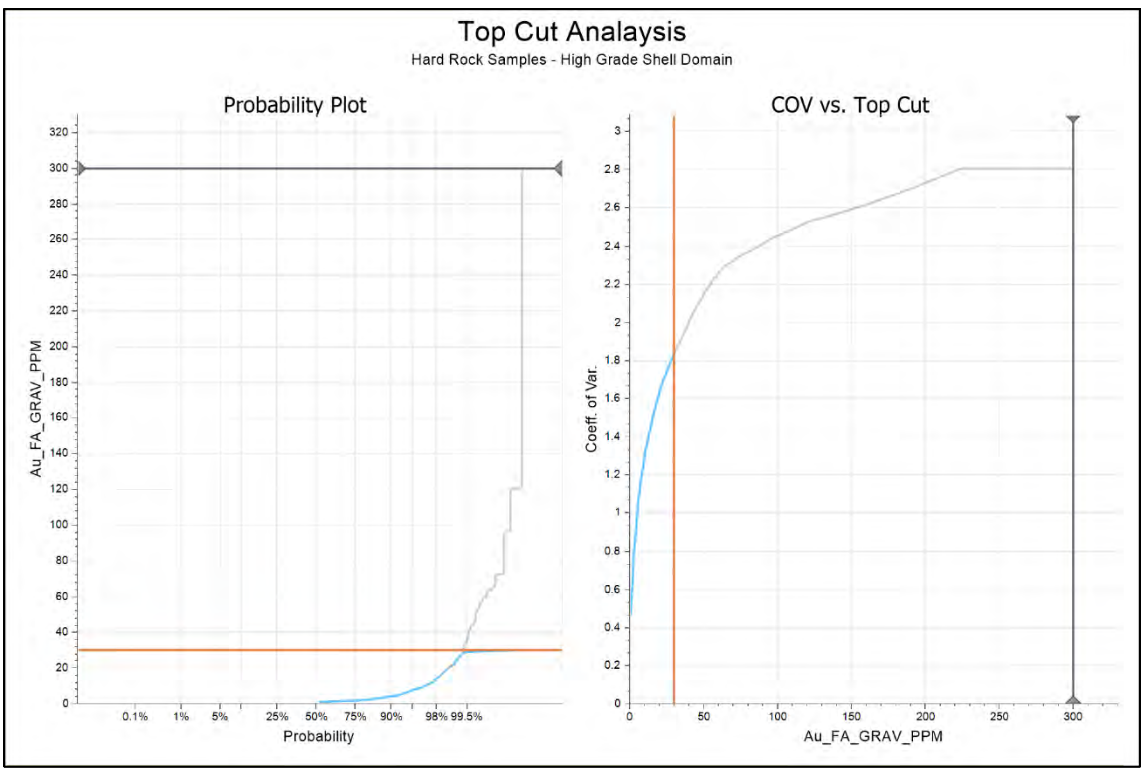


Figure 14-11: Top Cut Analysis of Composite Samples Within the High Grade Domain

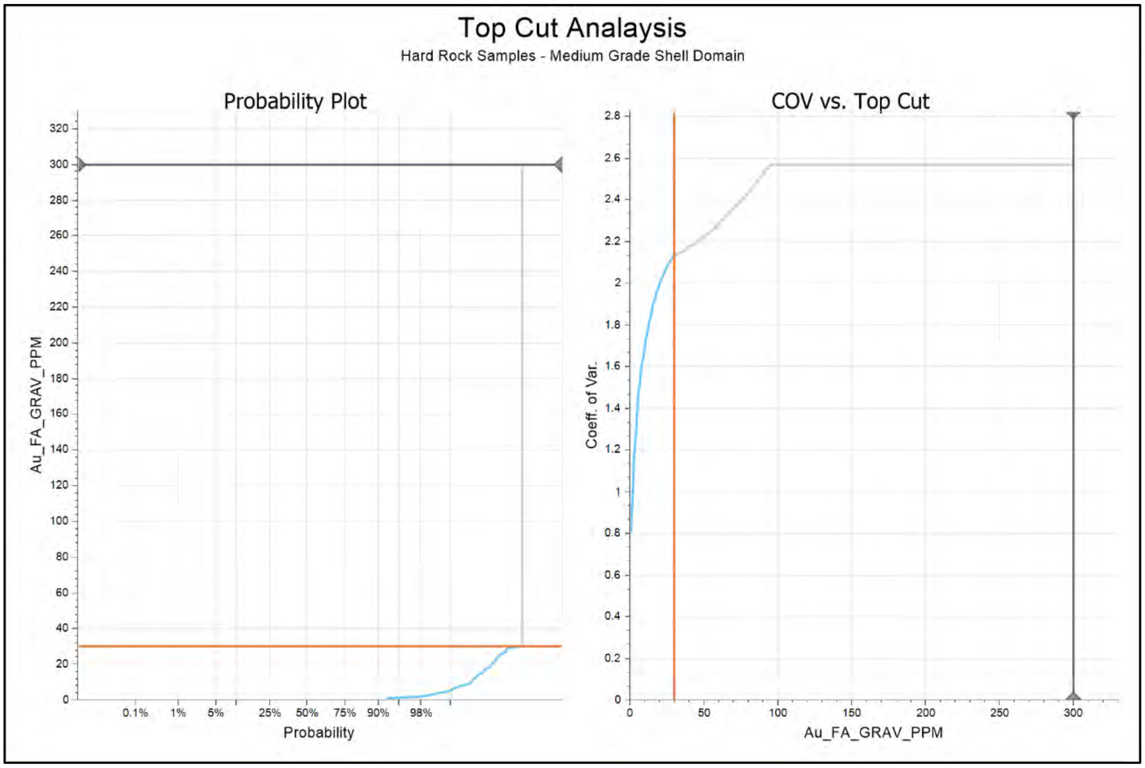


Figure 14-12: Top Cut Analysis for Composite Samples Within the Medium Grade Domain

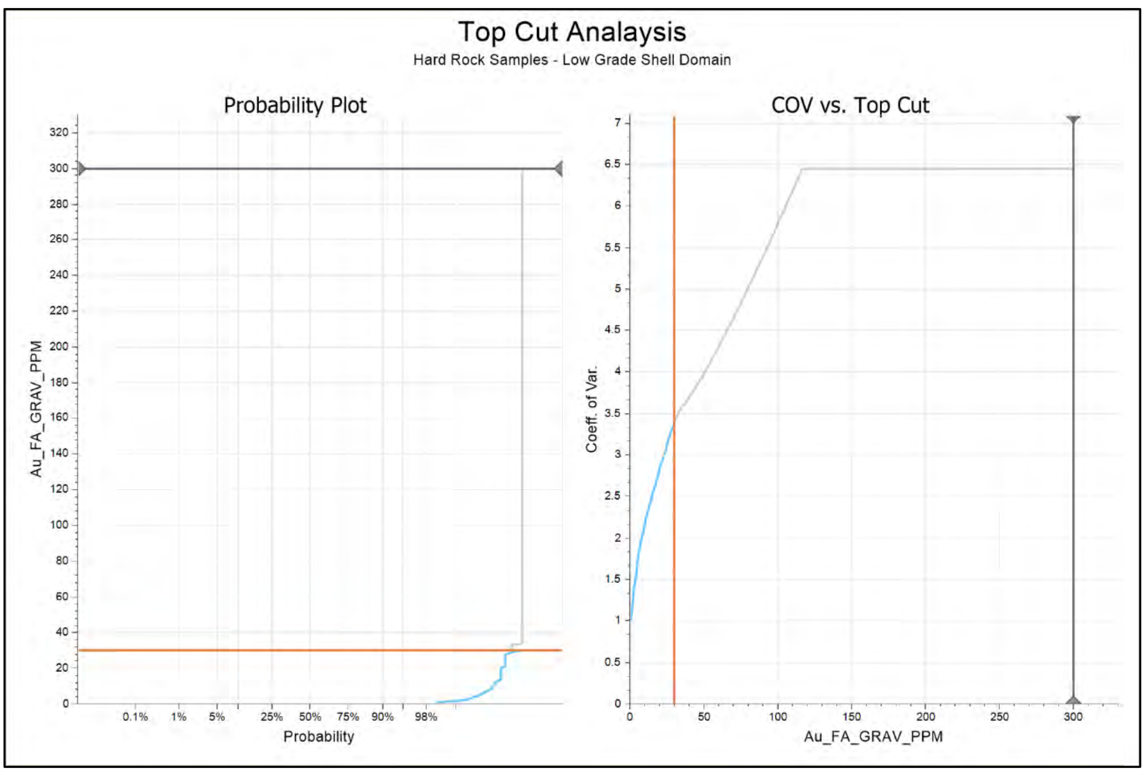


Figure 14-13: Top Cut Analysis for Composite Samples Within the Low Grade Domain

Table 14-6: Summary of Original and Composited Sample Capping Statistics

Domain	Original Samples			Composited Samples		
	Sample Count	Length Weighted Average		Sample Count	Average Grade	
		Au, oz/ton	Au, g/t		Au, oz/ton	Au, g/t
Non-Mineralized	12,511	0.0016	0.06	3,149	0.0016	0.05
Internal Waste	43	0.0016	0.06	11	0.0017	0.06
Low Grade	42,127	0.0056	0.19	10,891	0.0054	0.18
Medium Grade	74,796	0.0147	0.50	17,959	0.0133	0.46
High Grade	18,088	0.0602	2.06	4,638	0.0578	1.98
Outside	132,147	0.0016	0.06	31,440	0.0015	0.05

14.5.2 Backfill Model

Individual samples were capped at 0.2917 oz/ton (10 g/t) gold prior to compositing to limit the influence of outliers during the compositing process. One single sample was capped 0.2917 oz/ton gold prior to compositing. A summary of the sample statistics by JSLA backfill stage are provided in Table 14-7.

Table 14-7: Summary of Original and Composited Backfill Samples

JSLA Backfill Stage	Original Samples			Composited Samples		
	Sample Count	Length Weighted Average		Sample Count	Average Grade	
		Au, oz/ton	Au, g/t		Au, oz/ton	Au, g/t
1	668	0.0076	0.26	377	0.0079	0.27
2	1,188	0.0043	0.15	568	0.0043	0.15
3	1,826	0.0086	0.29	849	0.0076	0.26
4	2,864	0.0088	0.30	1,732	0.0091	0.31
5	309	0.0167	0.57	256	0.0186	0.64
6	558	0.0109	0.37	623	0.0114	0.39
7	111	0.0141	0.48	142	0.0139	0.48

14.6 CONTACT ANALYSIS

In an attempt to determine the treatment of domain boundaries during the grade estimation strategy, contact analysis was performed for each grade shell domain and oxidation domain. Contact plots representing grade profiles within and away from the grade shell domains for low, medium and high-grade domains are shown in Figure 14-14, whereas oxidation domains are shown in Figure 14-15. Contact plots show histograms of the number of samples in blue, and average grades in red. The final treatment of the domain boundaries is summarized in Table 14-8. High-, medium- and low-grade shell domains show slight increase in grade towards the contact of the high-grade shells. Gold grades also decline into the transition zones and decrease further into the sulfide domains.

Within the model there are some areas where transitional domains are absent and oxide domains overly sulfide zones. Most of these occurrences correspond with a lack of deeper drilling that may have encountered transition or sulfide mineralization. It is anticipated that additional drilling and assaying will generate a more robust oxidation model.

Table 14-8: Summary of Treatment of Domain Contacts

Contact	Treatment in Resource Model
Non Mineralized Rock Types	Hard Boundary
High and Medium grade	Soft Boundary
Medium and Low Grade	Soft Boundary
High and Low Grade	Hard Boundary
Low Grade and Internal Waste	Hard Boundary
Oxide - Transition	Soft Boundary
Transition - Sulfide	Soft Boundary

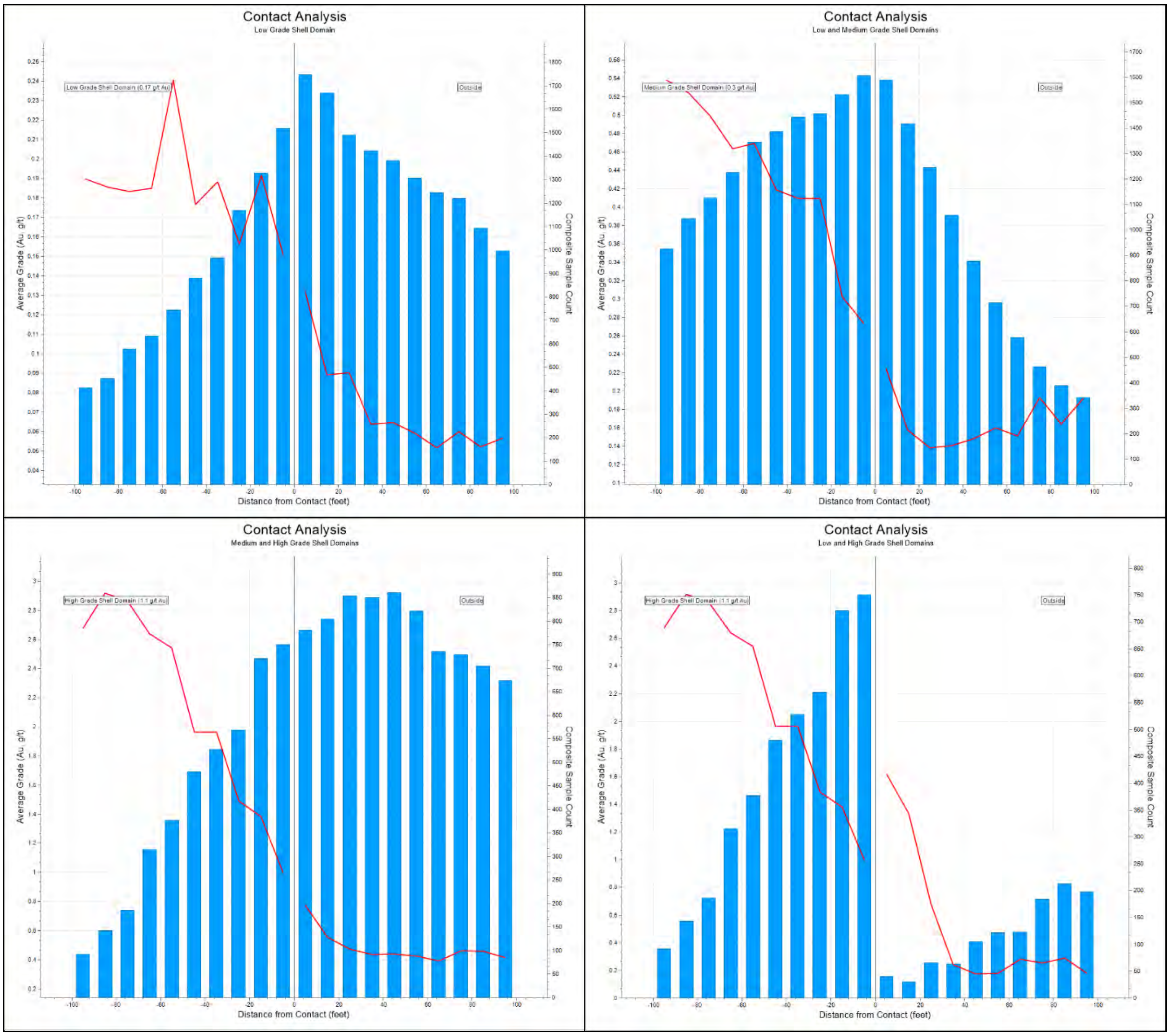


Figure 14-14: Grade Domain Boundary Analysis

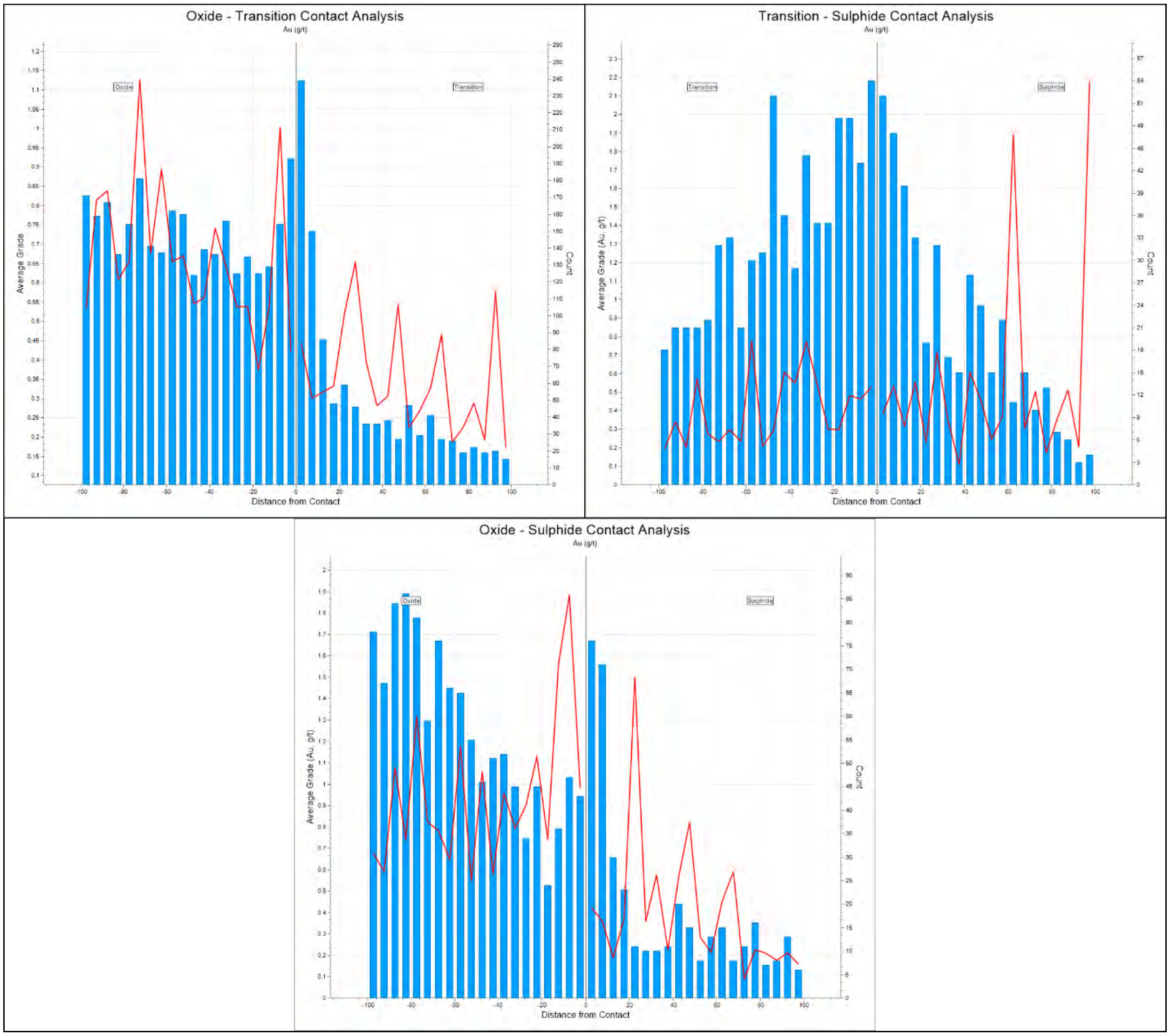
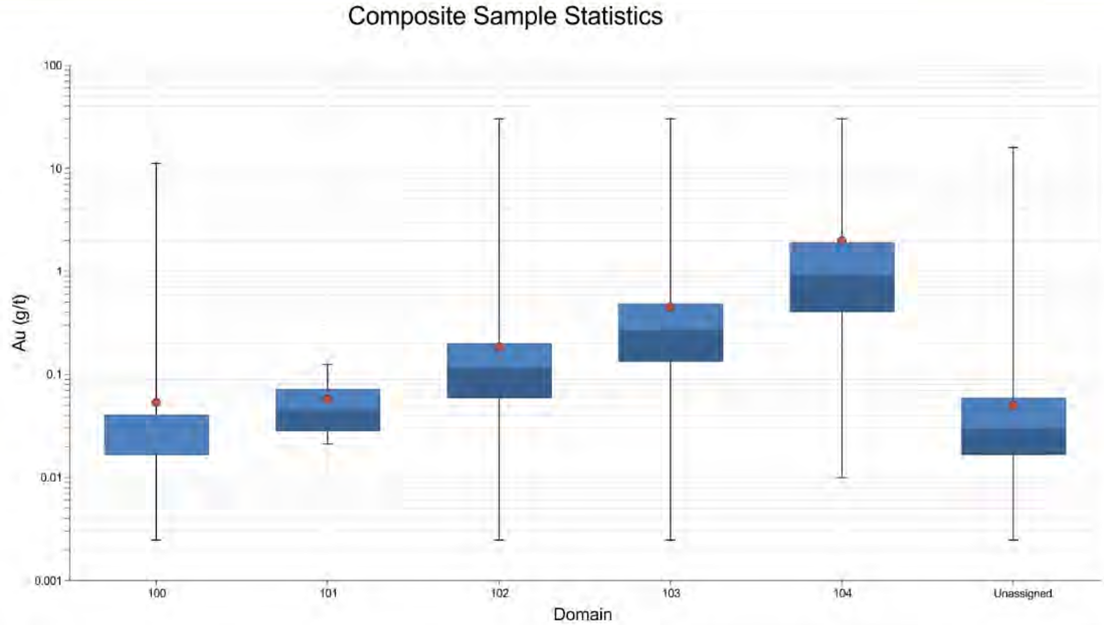


Figure 14-15: Geochemical Weathering Boundary Analysis

For the backfill portion of the resource, all backfill stage domains were treated as hard boundaries as these domains represent depositional continuity during backfilling of the JSLA pit.

14.7 STATISTICAL ANALYSIS

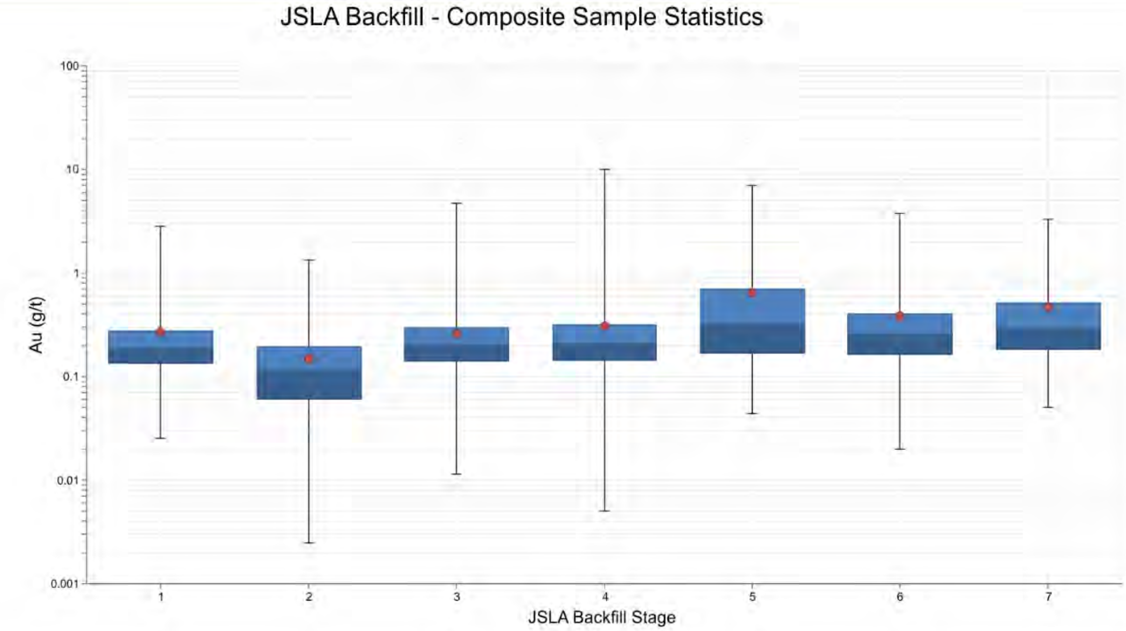
A summary of the composite sample statistics is presented in Figure 14-16 by corresponding grade domain codes for all domains used in the hard rock, JSLA backfill and waste dump resources are summarized in Figure 14-17.



	100	101	102	103	104	Unassigned
No of Points	3149	11	10891	17959	4638	31440
Minimum	0.0025	0.02	0.0025	0.0025	0.01	0.0025
Maximum	11.29	0.13	30	30	30	15.90
Mean	0.05	0.06	0.18	0.46	1.98	0.05
Variance	0.05	0.001	0.40	0.94	13.12	0.01
Std Dev	0.23	0.04	0.63	0.97	3.62	0.12
COV	4.20	0.61	3.43	2.13	1.83	2.43
Quartile 1	0.02	0.03	0.06	0.14	0.41	0.02
Median	0.02	0.04	0.12	0.27	0.92	0.03
Quartile 3	0.04	0.07	0.20	0.49	1.91	0.06

Number of records	68128
Number of keys	6

Figure 14-16: Box and Whisker Plot of Composite Samples by Grade Domains



	1	2	3	4	5	6	7
No of Points	377	568	849	1732	256	623	142
Minimum	0.03	0.0025	0.01	0.01	0.04	0.02	0.05
Maximum	2.80	1.33	4.70	10	6.97	3.77	3.32
Mean	0.27	0.15	0.29	0.31	0.64	0.39	0.48
Variance	0.10	0.02	0.08	0.21	0.83	0.20	0.28
Std Dev	0.31	0.14	0.28	0.46	0.91	0.44	0.53
COV	1.13	0.96	1.10	1.45	1.43	1.13	1.11
Quartile 1	0.14	0.06	0.14	0.15	0.17	0.17	0.19
Median	0.19	0.12	0.20	0.21	0.33	0.26	0.30
Quartile 3	0.28	0.19	0.29	0.31	0.70	0.40	0.51

Number of records	4547
Number of keys	7

Figure 14-17: Box and Whisker Plot of JSLA Backfill Composite Samples by Stage Backfill Domains

14.8 VARIOGRAPHY

Variogram modelling was completed for four resource areas to optimize interpolation parameters and to assess the viability of Ordinary Kriging for use as an interpolator. Directional variograms were calculated and modelled using exploration drilling data for four main resource areas: Oro Belle, Jumbo, JSLA – Big Chief, and South Domes – Six 21 (see Figure 14-22). With the exception of South Domes – Six 21, variograms were also calculated and modelled using production blasthole to assist in validating the variogram models generated using drill hole data. A summary of the modelled variogram ranges and directions for drill hole data and production blasthole data are included in Table 14-9 and Table 14-10, respectively. Modelled variograms for production blasthole data from the mined out areas are shown in Figure 14-18 to Figure 14-20, and for exploration data from South Domes – Six 21 shown in Figure 14-21.

Table 14-9: Summary of Variogram Parameters for Area Based Variogram Modelling

Area	Rotation Angles (°)			Variogram Model Parameters					
	Z	X	Y	Nugget	Structure	CC	Major (ft)	Semi-Major (ft)	Minor (ft)
Oro Belle	24	0	6	0	1	0.75	30	20	30
					2	0.25	100	110	150
Jumbo	210	25	0	0	1	1	40	60	30
JSLA - Big Chief	310	0	0	0	1	1	30	45	45
South Domes - Six 21	245	0	10	0	1	1	60	120	40

Table 14-10: Summary of Variogram Parameters Based On Blasthole Sampling for Mined Deposits

Area	Rotation Angles (°)			Variogram Model Parameters					
	Z	X	Y	Nugget	Structure	CC	Major (ft)	Semi-Major (ft)	Minor (ft)
Oro Belle	24	0	6	0	1	0.1	20	15	20
					2	0.75	60	60	60
					3	0.15	110	110	80
Jumbo	210	25	0	0	1	0.85	40	40	30
					2	0.15	60	60	90
JSLA	310	0	0	0	1	0.83	20	20	60
					2	0.17	160	120	60

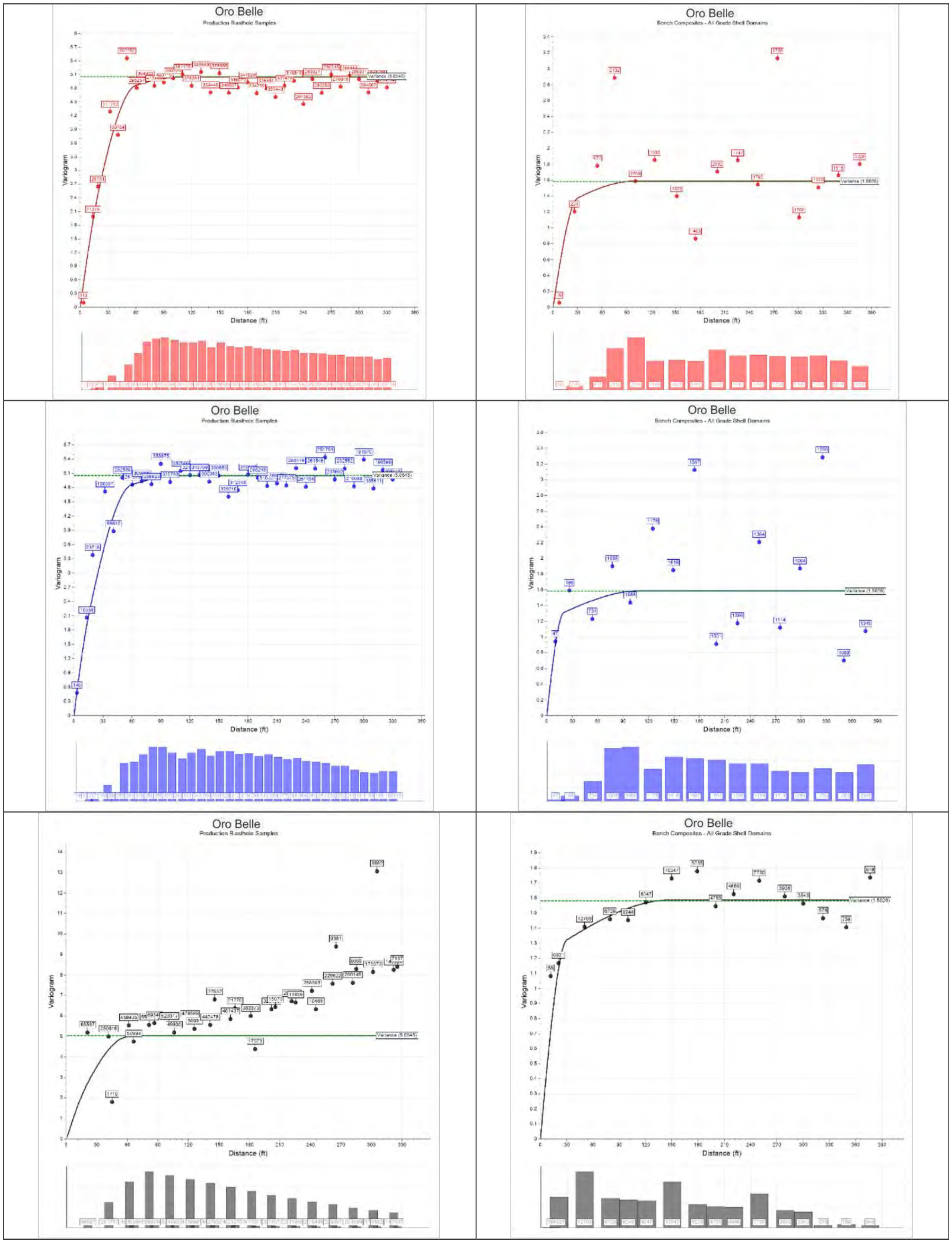


Figure 14-18: Oro Belle Variogram Modelling for Production Blasthole Samples (left) and Bench Composite Samples (right)

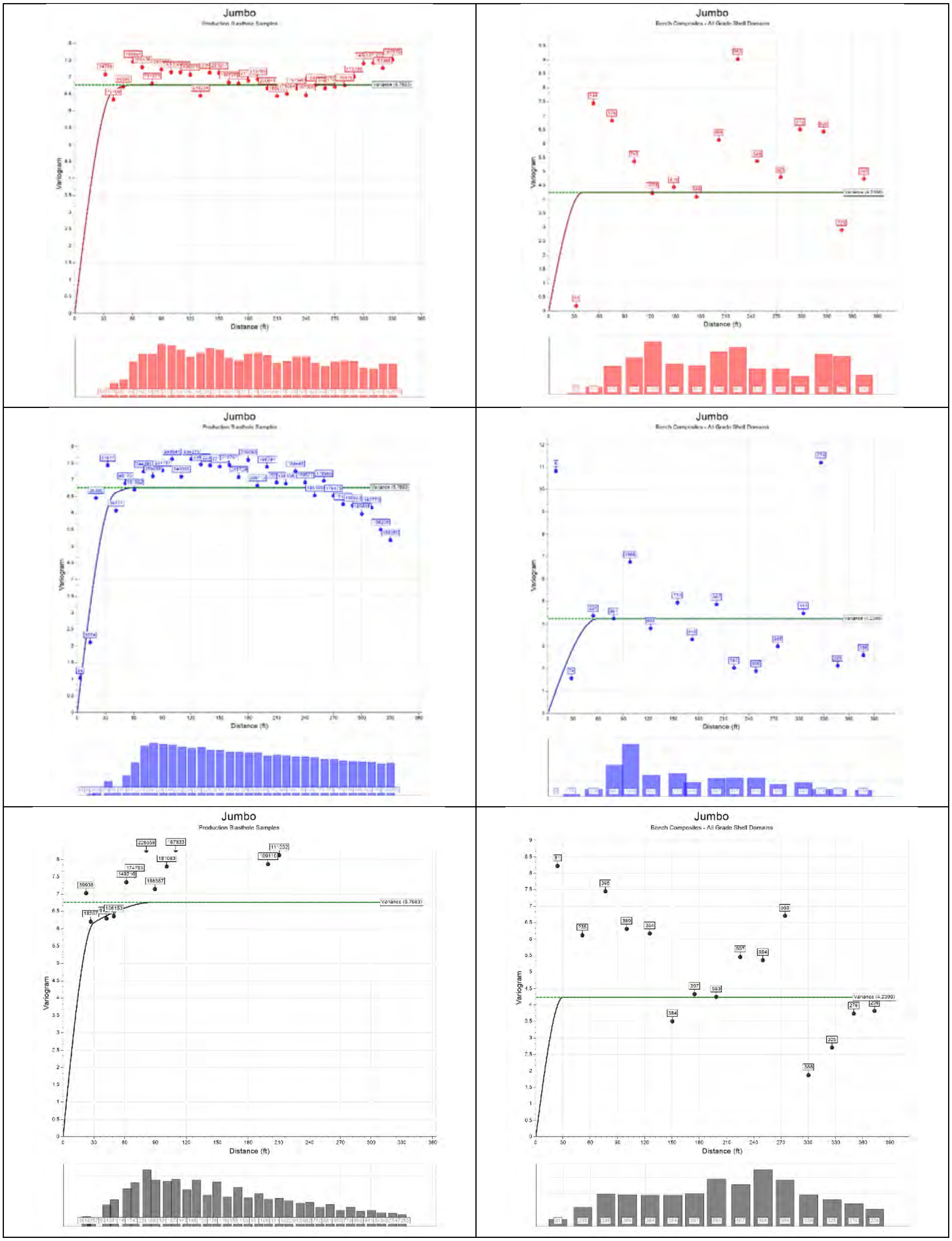


Figure 14-19: Jumbo Variogram Modelling for Production Blasthole Samples (left) and Bench Composite Samples (right)

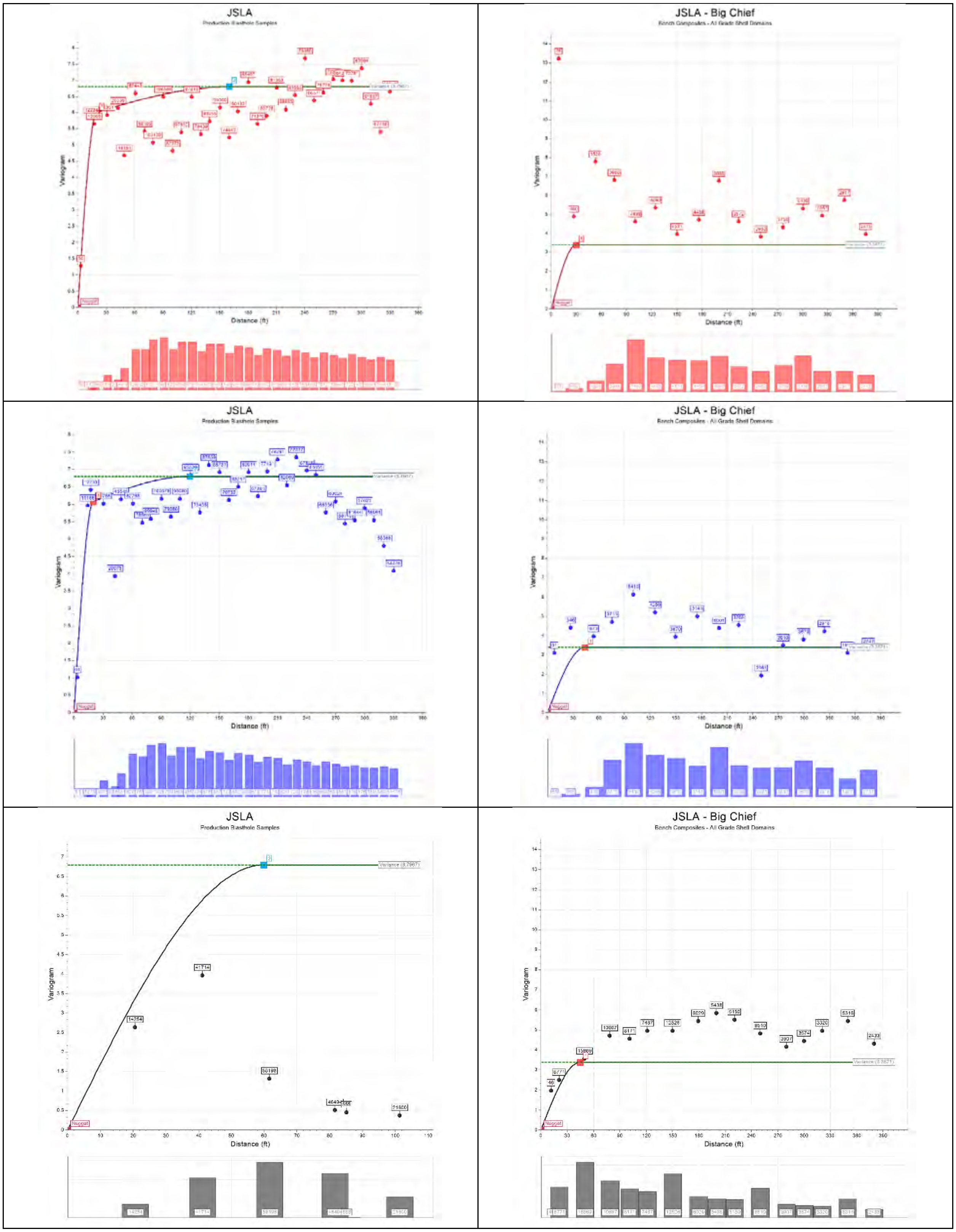


Figure 14-20: JSLA Variogram Modelling for Production Blasthole Samples (left) and Bench Composite Samples (right)

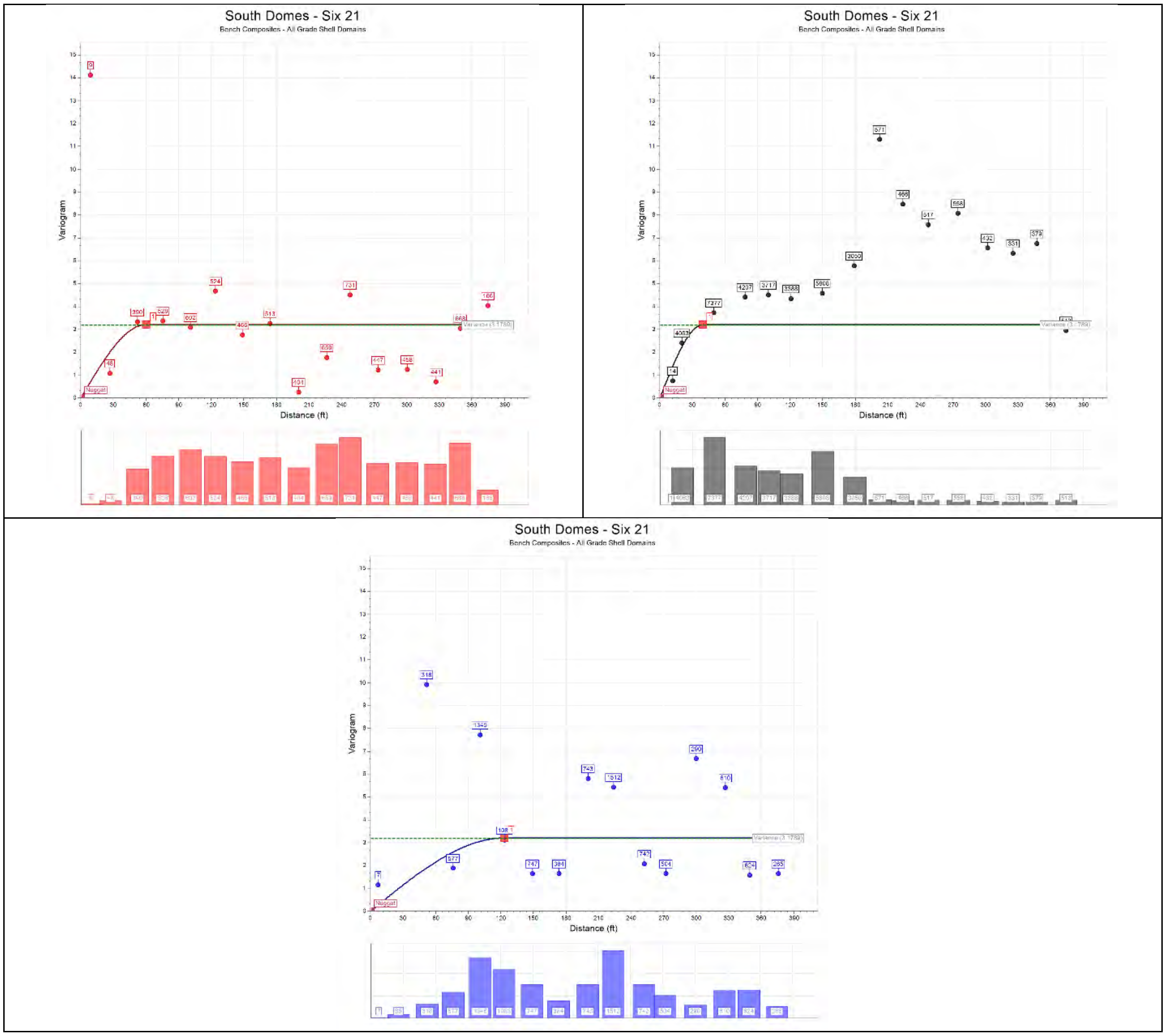


Figure 14-21: South Domes Variogram Modelling

Variogram ranges generated from production blasthole data indicate good spatial continuity up to 60 ft (18 m) on average, and generally decay towards the maximum ranges that vary from 90 to 160 ft (27 to 49 m) depending on the resource area. In general, variograms produced using composited drill hole samples produce less stable variograms and do not reproduce the production data. Both datasets support that good spatial continuity is achieved from 40 to 60 ft (12 to 18 m) for most resource areas. The area within and around Jumbo exhibits shorter ranges up to a maximum range of 60 ft (18 m) for drill hole data and 90 ft (27 m) for blasthole data.

14.9 BULK DENSITY

Bulk density was collected using the paraffin wax immersion method on 647 core samples. The bulk density measurements were converted to tonnage factors as discussed in Section 12.5. Average bulk density values were assigned to the block model according to the lithological model domain. For lithologies with inadequate sample coverage, average bulk density values of alike rock types were assigned. The bulk density values for each lithology type are summarized in Table 14-11.

Table 14-11: Summary of Bulk Density and Tonnage Factor Values by Lithology

Model Code	Lithology	Sample Count	Average Tonnage Factor (ft^3 per short ton)	Average SG (g/cm^3)
	Backfill and Waste Dumps	0	18.8	1.7
2	Alluvium	0	16.8	1.90
3	Debris Flow	2	17.3	1.85
5	Dacite	0	14.3	2.24
5	Hart Peak Rhyolite	0	14.2	2.25
7	Rhyolite Breccia	9	14.4	2.24
9	Porphyrytic Rhyolite	158	14.3	2.25
11	Aphyric Rhyolite	112	14.7	2.20
14	Volcanoclastic Diatreme	79	14.5	2.22
16	Volcanoclastic	146	15.3	2.11
22	Mudstone	1	16.1	1.99
23	Epiclastics	33	14.9	2.16
27	Andesite	82	14.4	2.23
29	Peach Springs Tuff	9	14.5	2.23
30	Proterozoic (Pc) Sediments	7	13.6	2.36
31	Proterozoic (Pc) Basement	9	12.4	2.59

14.10 BLOCK MODEL AND GRADE ESTIMATION METHODOLOGY

The block model definitions for the Castle Mountain resource model are defined in Figure 14-12.

Table 14-12: Castle Mountain Block Model Definitions

Block Model Information			
	X	Y	Z
Block Size (ft)	30	30	20
Number of Blocks	300	430	140
Minimum Centre	2202315	12807115	2610
Maximum Centre	2211285	12819985	5390
Origin	2202300	12807100	2600
Rotation	0	0	0

Block sizes were determined from the historical mining methods, and preliminary mine design used in the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018).

The methodology for resource estimation for the Castle Mountain hard rock resource model was based on the following:

- For drill holes without missing intervals, samples were composited to 20 ft (6.1 m) bench intervals using the methodology presented herein. For drill holes with missing or unsampled intervals, length weighted average grades composited to the bench interval were used. Samples with less than 10% bench coverage or samples less than 2.5 ft (0.8 m) were discarded.
- Global capping of composite samples to 0.8750 oz/ton (30 g/t) gold.
- Grade shell development based on bench composite data.
- Anisotropic search parameters were informed by surfaces generated from the lithology model where there was visual gold grade continuity corresponding to lithological contacts.
- Blocks were estimated within three grade shell domains representing high-, medium- and low-grade, with hard boundaries between the high- and low-grade shell domains.
- Only blocks estimated within an area representing drill hole spacing less than 300 ft (91 m) were considered as a Resource.
- ID2 was used to estimate block grades within the non-mineralized and waste rock domains.
- ID3 was used to estimate block grades within the low-, medium- and high-grade shell domains.
- ID3 was used to estimate the backfill and waste dump domains.
- Average bulk density ("SG") values for each lithology were assigned to the block model.

Search distances were based on variogram modelling of drill hole and production composite samples. The search distances used for the resource estimate considered drill hole spacing and grade shell dimensions. Interpolation parameters are summarized in Table 14-13.

Table 14-13: Summary of Interpolation Parameters for Hard Rock, JSLA Backfill and Waste Dumps

Hard Rock						
Pass	Search Radius (ft)			Number of Samples		
	Major	Semi Major	Minor	Minimum	Maximum	Max per hole
1	100	75	50	4	12	3
2	200	130	100	3	12	2
3	500	300	250	2	12	-

JSLA Backfill						
Pass	Search Radius (ft)			Number of Samples		
	Major	Semi Major	Minor	Minimum	Maximum Per Octant	Max per hole
1	125	125	125	1	2	3

Waste Dumps						
Pass	Search Radius (ft)			Number of Samples		
	Major	Semi Major	Minor	Minimum	Maximum Per Octant	Max per hole
1	75	75	50	2	10	1
2	125	125	50	2	10	1

Search orientations were based on the lithology model where localized grade continuity corresponds to lithology contacts. Where there were no observable trends in grade continuity, an isotropic search orientation was used. Mean search directions by lithology are summarized in Table 14-14. The lithology model was given specific names based on target area and an identifier to aid in modelling, which was assigned a unique numeric domain code based on the lithology model code.

Table 14-14: Mean Search Directions by Lithology

Domain Code	Lithology	Mean Dip (˚)	Mean Dip Direction (˚)
	Waste Rock and Non-Mineralized Rock Types		
502	05-HartRhy Migos	6	4
201	02-Alluvium	4.5	207
301	03-DebrisFlow	7	232
501	05-Dacite	88	92
912	09-RhyPorph Six21-NWA	88	92
1102	11-RhyAph BigChief-EastFlow	6	66
1103	11-RhyAph BigChief-West	30	42
1104	11-RhyAph BigChief-WestFlow	7	259
1118	11-RhyAph SD-FlashFlow	8	90
1117	11-RhyAph Six21-DMXFlow	1	315
1119	11-RhyAph Six21-NoName	25	40
1120	11-RhyAph WestFlow	7	259
	Mineralized Rock Types		
701	07-RhyBx-Auto LuckyJohn	29	92
901	09-RhyPorph BigChief	80	94
902	09-RhyPorph ER-Dre	84	220
904	09-RhyPorph JSLA-Kendrick	82	108
906	09-RhyPorph Jumbo	90	332
907	09-RhyPorph JumboFlow	9	328
909	09-RhyPorph OroBelleFlow	20	100
910	09-RhyPorph SD-Snoop	70	89
911	09-RhyPorph SD-Tupac	86	80
1101	11-RhyAph BigChief-East	43	315
1105	11-RhyAph ER-JayZ	83	56
1107	11-RhyAph LuckyJohn	27	125
1109	11-RhyAph OB-50centFlow	3	46
1112	11-RhyAph SD-Flash	48	27
1114	11-RhyAph SD-Mathers	60	92
1115	11-RhyAph SD-Quest	80	85
1116	11-RhyAph Six21-Biggie	60	60

Domain Code	Lithology	Mean Dip (˚)	Mean Dip Direction (˚)
1117	11-RhyAph Six21-DMX	50	244
1120	11-RhyAph West Flow	3	141
1404	14-VxDiatreme LuckyJohn	89	220
1601	16-Vx	5	183
2201	23-Epiclastics	5	196
2701	27-Andesite	5	188
2901	29-Peach Spring	4	218
3001	30-Pc. Sediments	4	220
3101	31-Pc. Basement	5	232
	Rock Types Not Included in Anisotropy Model		
1110	11-RhyAph OB-Outkast	0	0
1111	11-RhyAph OB-OutkastFlows	0	0
1402	14-VxDiatreme EastRidge	0	0
1405	14-VxDiatreme OroBelle	0	0
1406	14-VxDiatreme SouthDomes	0	0
1110	Waste Dumps and Backfill	0	0

14.11 MODEL VALIDATION

The resource models were validated by completing a series of swath and cross validation plots. Depleted resource areas were compared to estimates generated with blasthole samples and historic production records. A comparison of estimators was also completed.

14.11.1 Swath Plots

Swath plots were generated for six main resource areas (Figure 14-22). Swath plots were generated on 60 ft (18 m) swath indexes for east-west and north-south swath directions. Vertical swath plots were generated using a 40 ft (12.2 m) interval. Swath plots are shown in Figure 14-23 to Figure 14-27. Blue lines correspond to the block estimate, red lines are the nearest neighbor estimate, black lines represent composite samples, and blue histograms represent the number of composite samples. Block estimates show good correlation with composite samples and nearest neighbor with grade trends honored. As most historical holes drilled by Viceroy are subvertical, there is some bias due to drilling orientation in easting and northing swath plots.

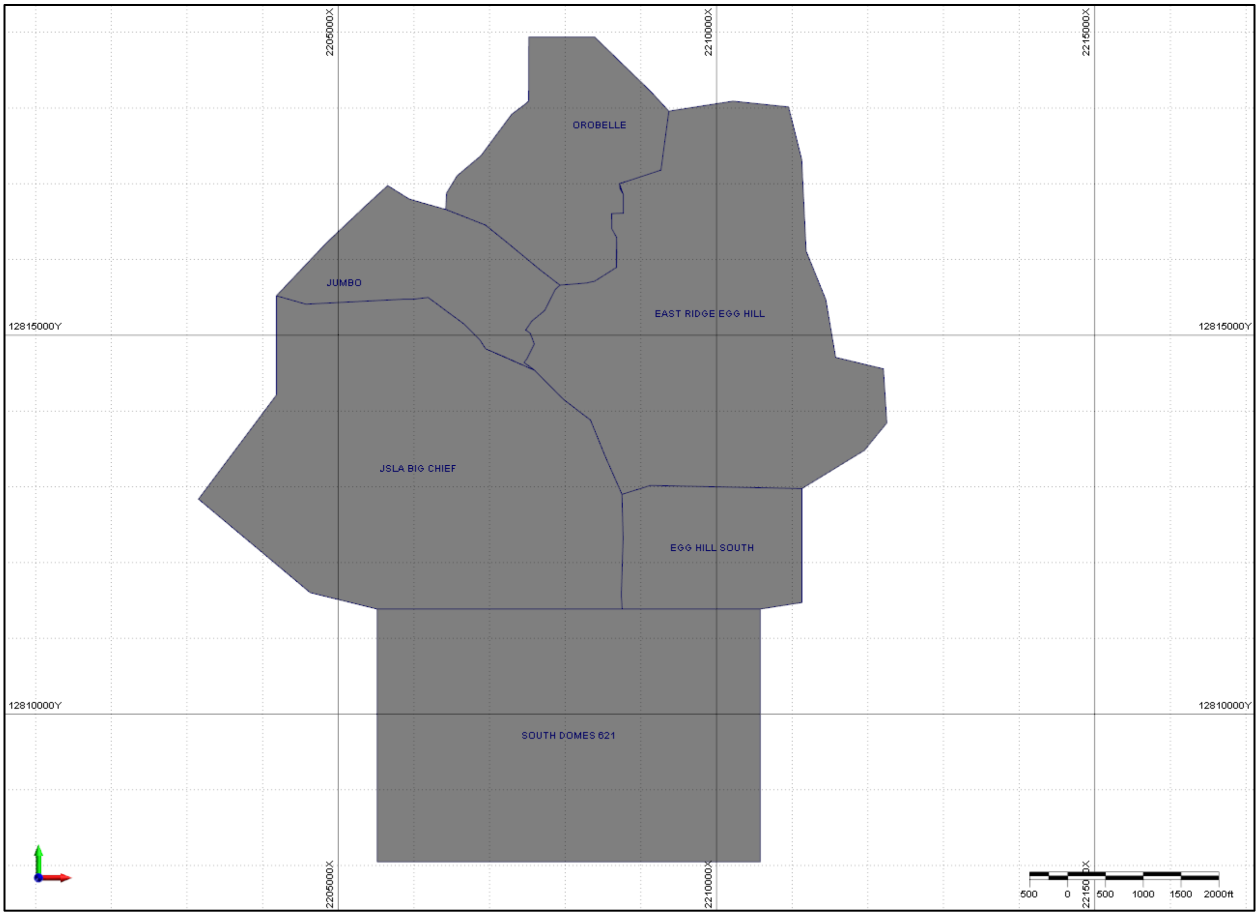


Figure 14-22: Resource Areas Used for Validation

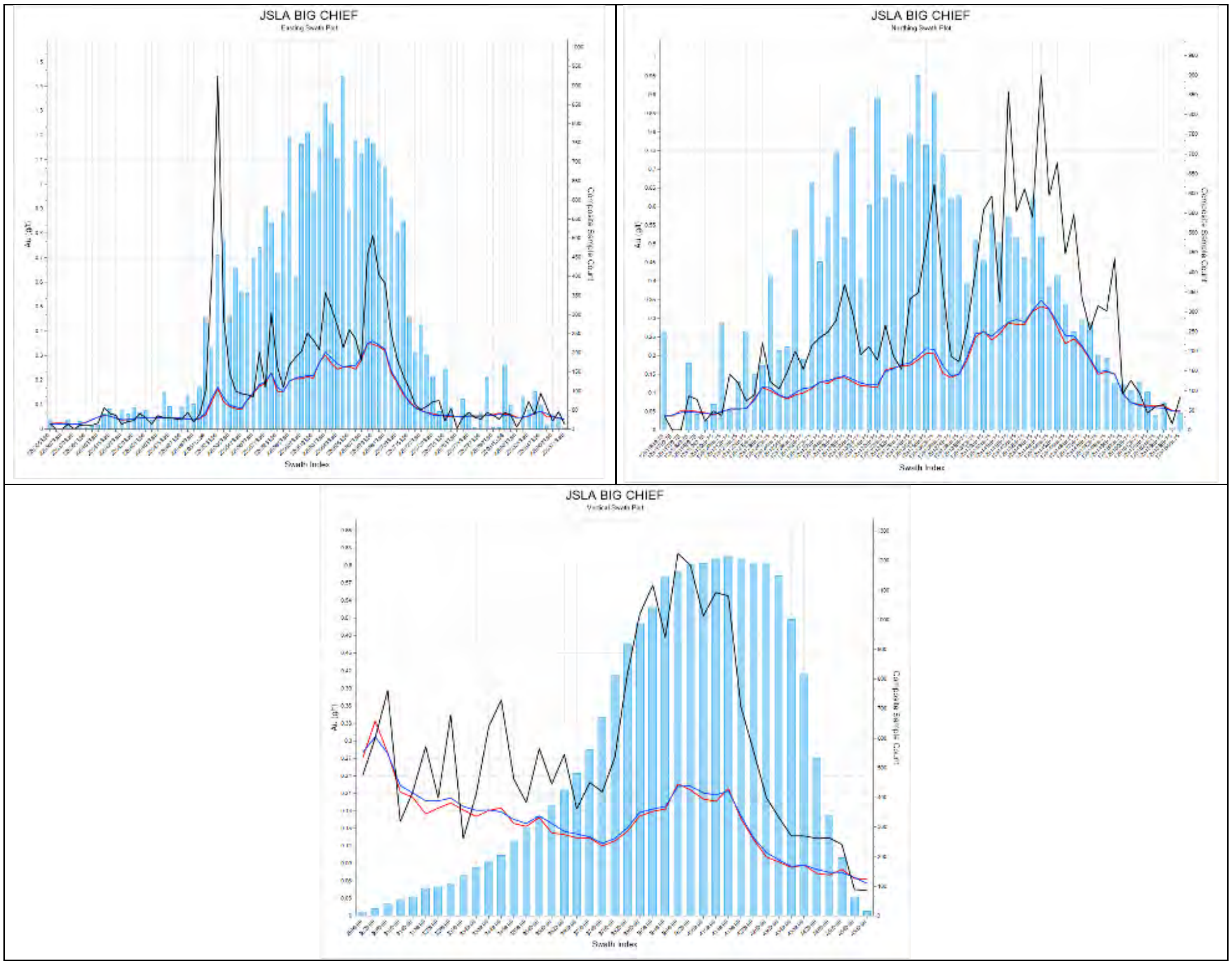


Figure 14-23: Swath Plots for JSLA-Big Chief Resource Areas

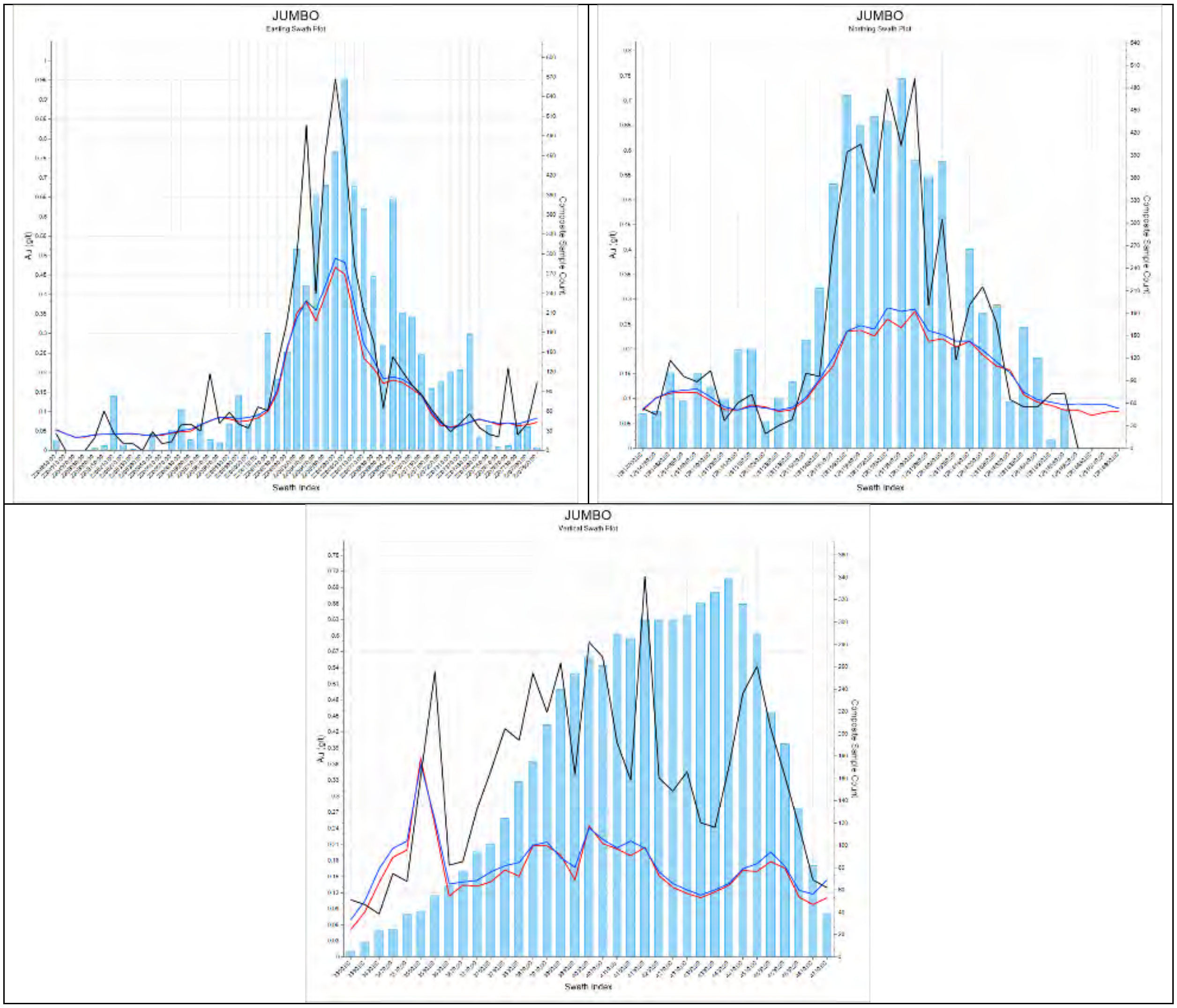


Figure 14-24: Swath Plots for Jumbo Resource Area

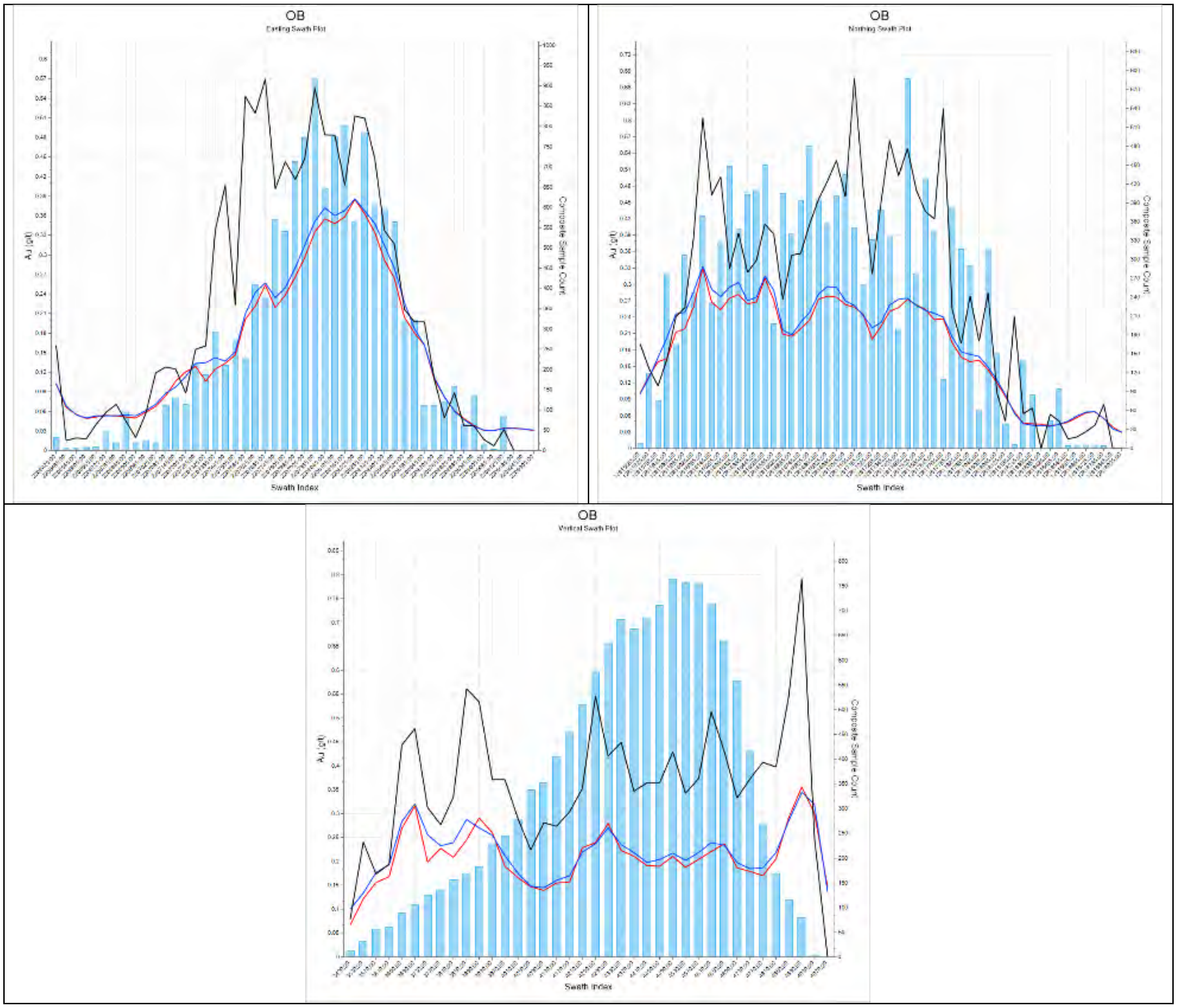


Figure 14-25: Swath Plots for Oro Belle Resource Area

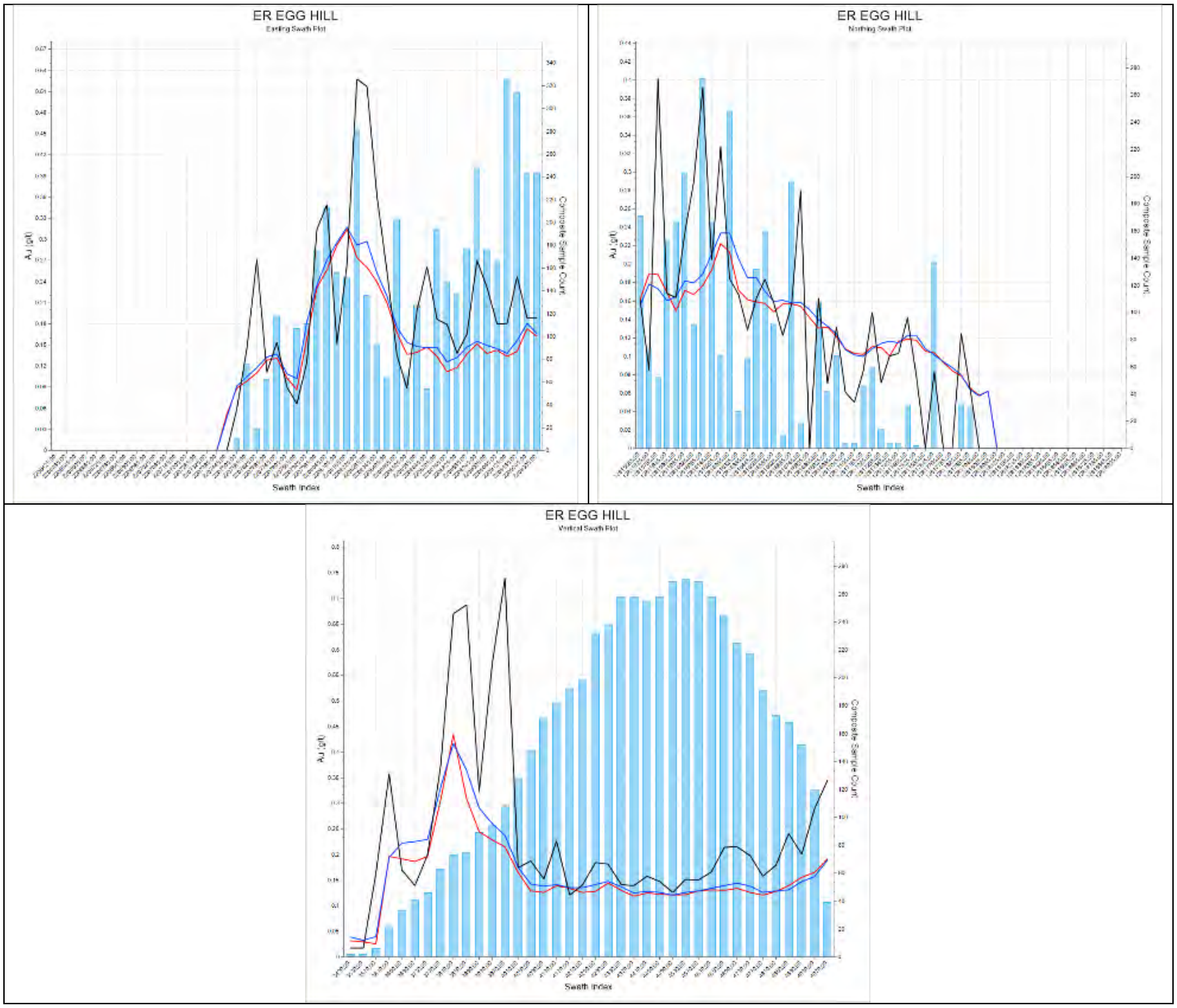


Figure 14-26: Swath Plots for East Ridge – Egg Hill Resource Areas

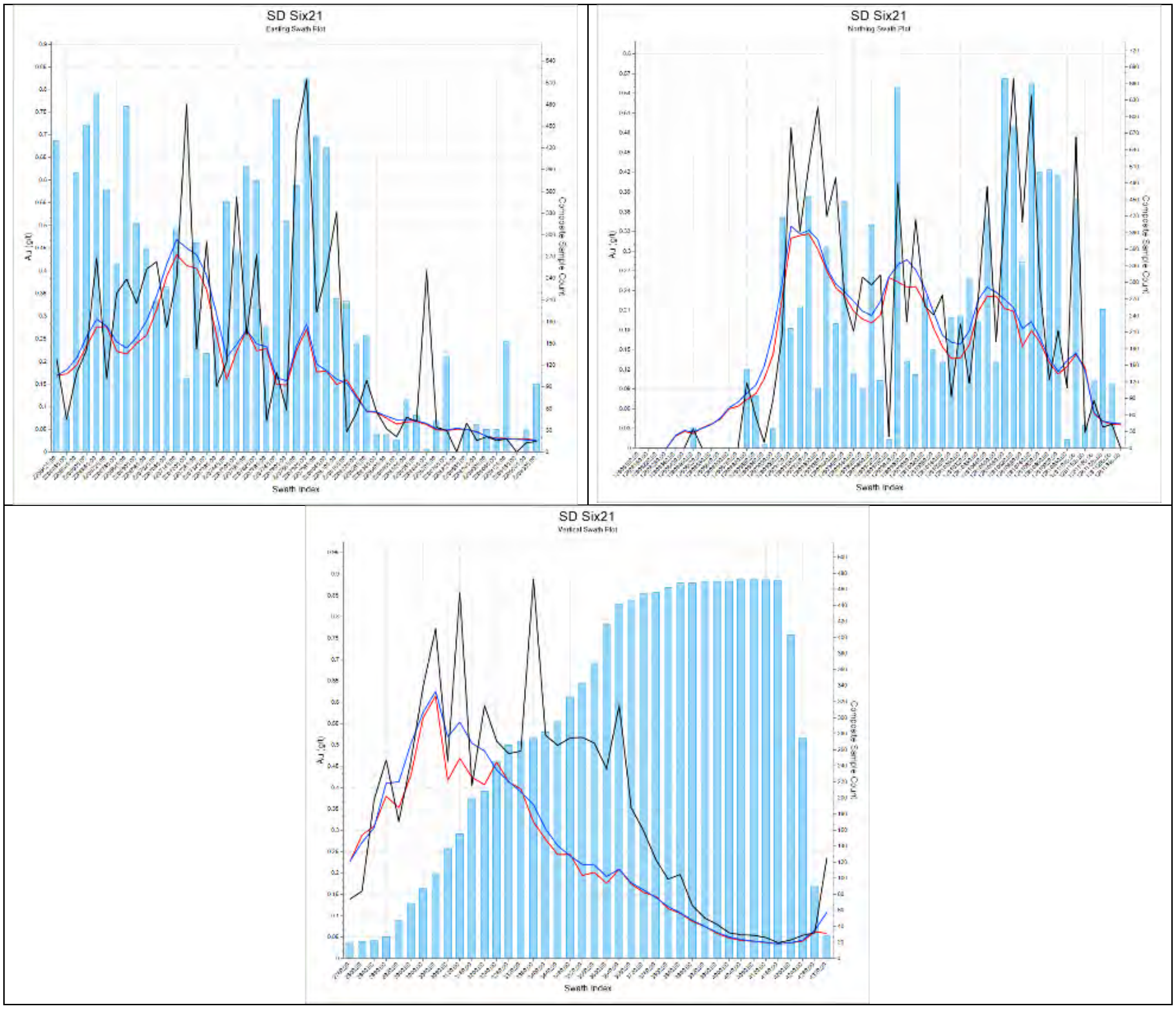


Figure 14-27: Swath Plots for South Domes – Six 21 Resource Areas

14.11.2 Cross Validation

Cross validation plots were generated for block estimates versus composite sample grades. Overall, the block estimates and composite samples show good correlation. Measured classification shows a tendency to have lower correlation to block estimates due to clustering of high-grade composite samples that have nugget effect. Cross validation plots are shown in Figure 14-28 to Figure 14-31.

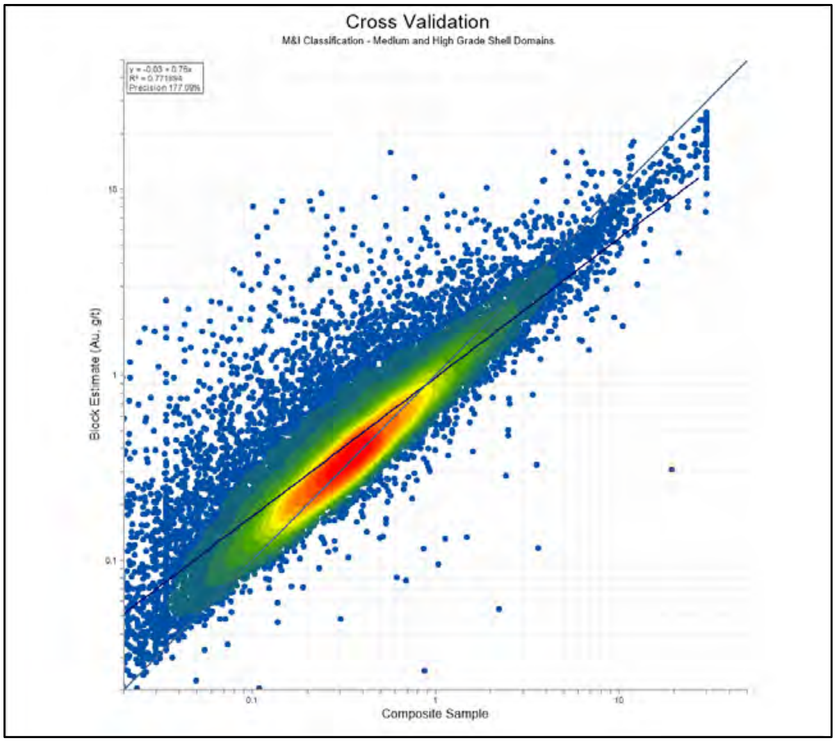


Figure 14-28: Cross Validation of Composite Samples Against Block Estimates for Medium- and High-Grade Domains Within Measured and Indicated Classified Blocks

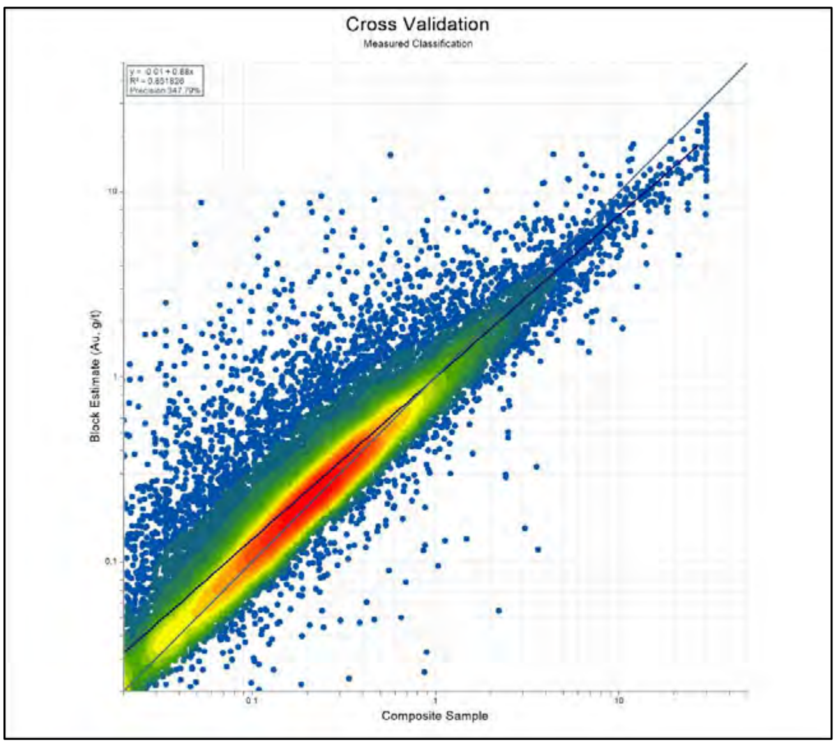


Figure 14-29: Cross Validation of Composite Samples Against Block Estimates for Medium- and High-Grade Domains within Measured Classified Blocks

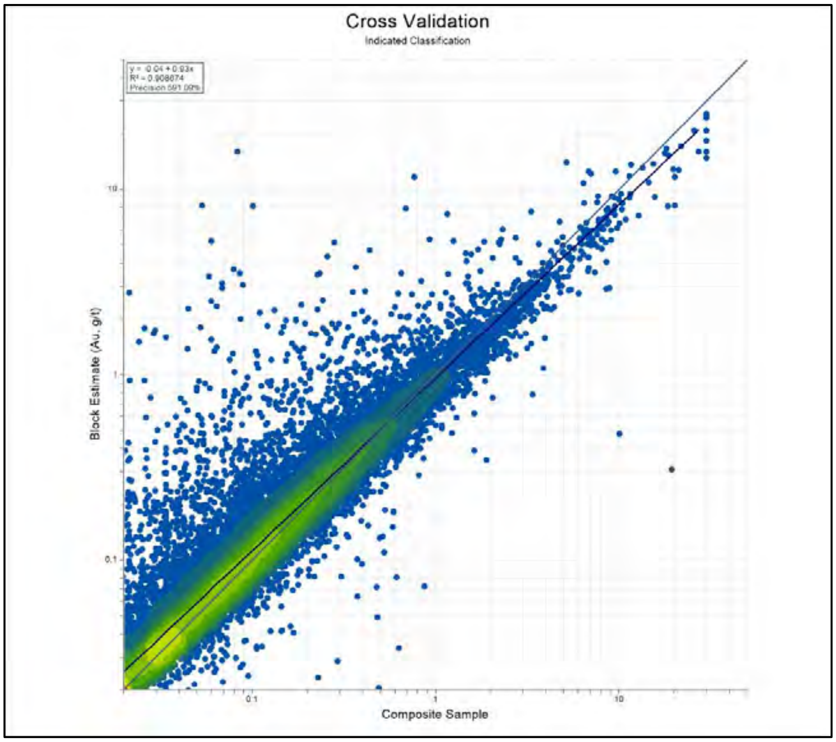


Figure 14-30: Cross Validation of Composite Samples Against Block Estimates for Medium- and High-Grade Domains Within Indicated Classified Blocks

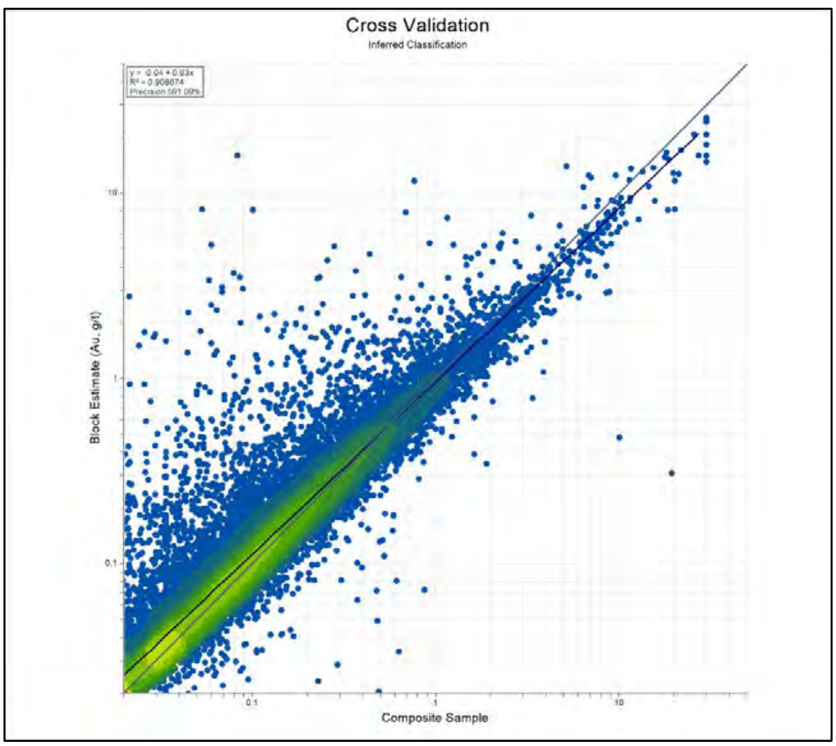


Figure 14-31: Cross Validation of Composite Samples Against Block Estimates for Medium- and High-Grade Domains Within Inferred Classified Blocks

14.11.3 Comparisons to Historical Production and Production Drilling

The resource estimate was compared to historical production and blasthole production drilling (Table 14-15).

Table 14-15: Comparison of Current Resource Model to Historical Production and Production Blasthole Model Estimates

	Tonnes	Tons	Grade (Au, g/t)	Grade (Au, oz/ton)	Contained Gold (oz)
Grade Control Model - 30 g/t Au cap, no cut-off	97,108,383	107,053,669	0.51	0.015	1,581,175
Current Resource Model, no cut-off	97,107,305	107,052,481	0.48	0.014	1,509,551
Percent Difference	**0.0%**		**-5%**	**-5%**	**-5%**
	Tonnes	**Tons**	**Grade (Au, g/t)**	**Grade (Au, oz/ton)**	**Contained Gold (oz)**
Grade Control Model - 30 g/t Au cap, 0.5 g/t Au cut-off	26,371,617	29,072,447	1.30	0.038	1,103,623
Current Resource Model, 0.5 g/t Au cut-off	23,767,545	26,201,681	1.41	0.041	1,079,165
Percent Difference	**-10%**		**8%**	**8%**	**-2%**
	Tonnes	**Tons**	**Grade (Au, g/t)**	**Grade (Au, oz/ton)**	**Contained Gold (oz)**
Historic Production (Ore Milled from 1991 to 2004)	32,831,000	36,193,000	1.47	0.043	1,550,000
Historic Production (Ore Mined from 1991 to 2004)	34,182,000	37,683,000	1.37	0.040	1,520,000
Current Depleted Resource Model (0.5 g/t Au cut-off)	34,357,586	37,876,293	1.56	0.046	1,727,845
Percent Difference to Historic Ore Milled	**5%**	**5%**	**6%**	**6%**	**11%**
Percent Difference to Historic Ore Mined	**1%**	**1%**	**14%**	**14%**	**14%**
	Tonnes	**Tons**			
Total Mined	126,954,190	139,956,113			
Total Depleted within Resource Model	119,300,607	131,518,694			
Percent Difference	**-6%**	**-6%**			

In general, the current resource model reports fewer tons at slightly higher gold grades compared to the historic grade control model. Total contained gold between the current resource model compared to the grade control model and historical milled production is within 5%. Disparities between the total historical tonnage mined and the depleted resource may be attributed in part to the accuracy of the pre-mining topography, as-built surveys of the mined-out areas that were subsequently backfilled, and density values used for the current resource model.

14.11.4 Comparisons to Other Estimation Techniques

The hard rock model was estimated using several different estimation techniques including Inverse Distance Squared (ID2), Inverse Distance Cubed (ID3), Ordinary Kriging (OK) and Nearest Neighbor (NN). All estimation methods, except for NN, were found to generate comparable results. Estimates generated using OK reported greater tonnes at a slightly lower grade when compared to the estimates generated using inverse distance methods. A comparison of the estimation techniques is shown in Table 14-16.

Table 14-16: Summary of Estimation Technique Comparison

Interpolant	Blocks Estimated >0.005 oz/ton (>0.17 g/t) Au	Average grade (Au, oz/ton)	Average grade (Au, g/t)
ID^3	273,554	0.0163	0.56
ID^2	274,230	0.0163	0.56
OK	280,653	0.016	0.55
NN	218,897	0.0195	0.67

14.12 MINERAL RESOURCE CLASSIFICATION

Block model quantities and grade estimates were classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). Mineral resource classification is subjective in nature and guided by the data used in preparing the estimate. Classification of Castle Mountain Project resources has considered geological continuity, data spacing, data type, data source, data quality, and geostatistical evaluation of these data.

The statistical criteria used for measured and indicated mineral resources is that the annual ore production should be known to at least ±15% with 90% confidence. To determine adequate hole spacing to fulfill the statistical criteria used for resource classification, blasthole data was decimated using 10 ft (3 m) spacings ranging from 10 ft up to 300 ft (91 m) within volumes representing monthly production volumes, that cumulatively represent an annual production volume. The results of the decimation showed that estimates can be reproduced reliably and within 10% of undecimated data using drill hole spacings of up to 90 ft (27 m) (Figure 14-32). Between 90 ft and 190 ft (58 m), estimates can be reproduced to within 10% to 12% but with lower confidence. With drill hole spacings exceeding 190 ft, the resulting estimates using decimated data could not be reconciled to within 15%. Therefore 190 ft was chosen as the drill hole spacing threshold for Indicated. When drill hole spacing exceeds 300 ft, potential mineralized zones are no longer detectable; therefore, 300 ft was chosen as an upper threshold for Inferred classification.

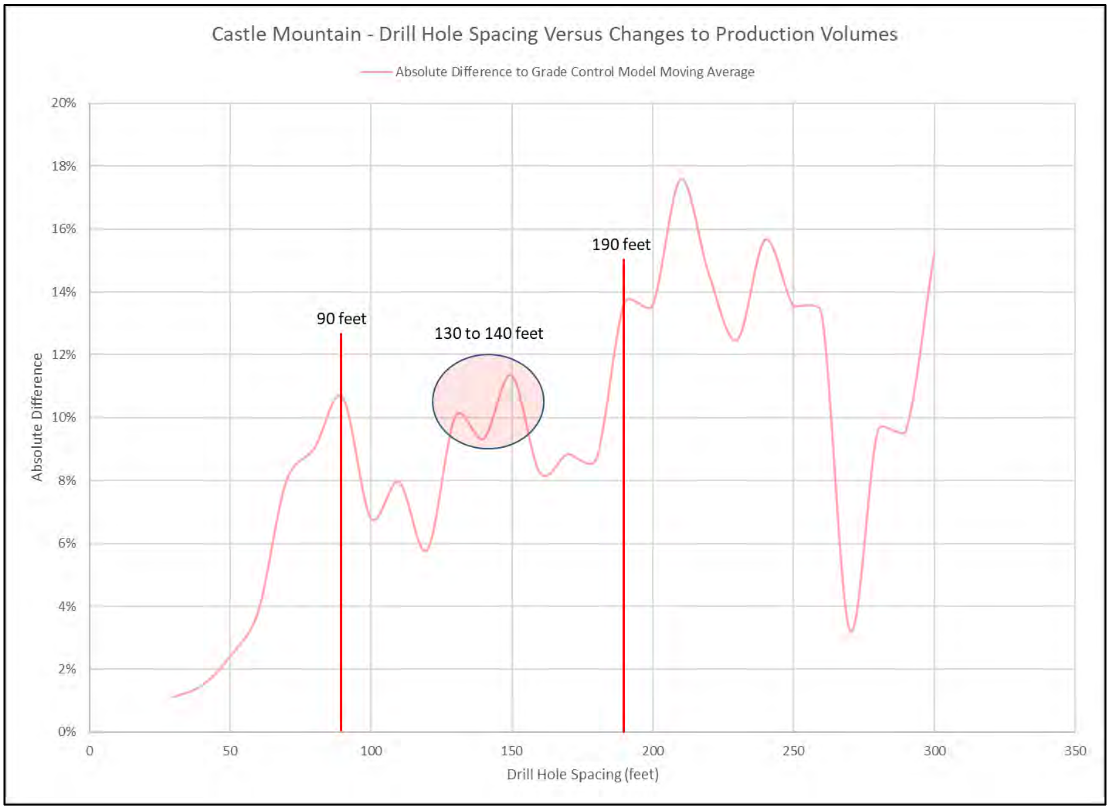


Figure 14-32: Absolute Difference to Production Volumes as a Function of Drill Hole Spacing

A summary of the hard rock resource classification is provided in Table 14-17. The JSLA backfill resource model was classified using the criteria summarized in Table 14-18.

Table 14-17: Summary of Hard Rock Resource Classification

Hard Rock Resource Model Classification						
Classification	Drill Hole Spacing (ft)			Depth Below Surface (ft)	Estimation Pass	Grade Domain
	Average	Minimum	Maximum			
Measured	85	60	100	< 900	1 or 2	Within Low Grade Domain
Indicated	145	100	190	-	-	-
Inferred	245	190	300	-	-	-

Table 14-18: Summary of JSLA Backfill Resource Classification

JSLA Backfill Resource Model Classification				
Classification	Drill Hole Spacing (ft)			Depth Below Surface (ft)
	Average	Minimum	Maximum	
Measured	50	10	80	< 30
Indicated	125	80	160	-
Inferred	180	160	250	-

14.13 MINERAL RESOURCE STATEMENT

The CIM Definition Standards on Mineral Resources and Reserves (CIM, 2014) state that:

"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."

In order to sufficiently test the reasonable prospects for eventual economic extraction by an open pit, pit shells were generated using the variable slope Lerchs Grossmann algorithm in Hexagon's MinePlan® software. The results of the pit optimization partially form the basis of the mineral resource statement and are used to constrain the mineral resource with respect to the CIM Definition Standards. Pit optimization does not constitute an attempt to estimate reserves. The open pit optimization parameters are summarized in Table 14-19.

Table 14-19: Summary of Pit Optimization Parameters

Parameter	Unit	Amount	Additional Comments
Gold Price	$/oz	1,500	
Marketing Cost	$/oz	1.51	
Royalty – Jumbo, JSLA, Oro Belle	%	2.65	
Royalty – South Domes	%	7.65	Cumulative royalty
Mining Cost: ROM	$/ton	1.52	Incremental cost of $0.02/ton per bench from the pit entrance at the 4500 ft bench elevation
Mining Waste: Rock	$/ton	1.27	
Mining Waste: Backfill & Overburden	$/ton	2.30	
Process Cost: ROM	$/ton	2.70	
Process Cost Milling	$/ton	13.74	Cross over cut-off grade (COG) for all rock types have been applied using lithology - process recovery matrix
Process Recovery: ROM	%	72.4	Variable for each rock type, from 65 to 80%
Process Recovery: Mill	%	94.0	Variable for each rock type, from 92 to 96%
General & Administrative	$/ton	0.72	
Mining Dilution	%	3.0	Dilution calculated using adjacent block contact points below COG
Pit Slope	º	48	

A summary of the Measured, Indicated and Inferred Resources exclusive of Reserves are summarized in Table 14-20 and Table 14-21. Mineral resources and depicted in cross section in Figure 14-33 and plan view in Figure 14-34.

Table 14-20: Castle Mountain Open Pit Resources Exclusive of Reserves (imperial units)

Classification	Au Cut-off	Tons	Au	Contained Au
	(oz/ton)	(kton)	(oz/ton)	(koz)
Measured	0.005	861	0.020	17
Indicated	0.005	80,967	0.018	1,453
Measured and Indicated	0.005	81,828	0.018	1,470
Inferred	0.005	77,040	0.018	1,422

Notes:

1. Mineral Resources are reported exclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500/oz gold.
3. Open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold and are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarized in Table 14-19.
4. The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Any discrepancies in the totals are due to rounding.
7. Mineral resources from Castle Mountain Project presented herein have an effective date of June 30, 2020.

Table 14-21: Castle Mountain Open Pit Resources Exclusive of Reserves (metric units)

Classification	Au Cut-off	Tonnes	Au	Contained Au
	(g/t)	(kt)	(g/t)	(koz)
Measured	0.17	781	0.68	17
Indicated	0.17	73,452	0.62	1,453
Measured and Indicated	0.17	74,233	0.62	1,470
Inferred	0.17	69,890	0.63	1,422

Notes:

1. Mineral Resources are reported exclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500/oz gold.
3. Open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold and are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarized in Table 14-19.
4. The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Any discrepancies in the totals are due to rounding.
7. Mineral resources from Castle Mountain Project presented herein have an effective date of June 30, 2020.

Table 14-22: Castle Mountain Open Pit Resources Inclusive of Reserves (imperial units)

Type	Au COG	Measured			Indicated			Measured and Indicated			Inferred		
		Tons	Au	Contained	Tons	Au	Contained	Tons	Au	Contained	Tons	Au	Contained
	(oz/ton)	(kton)	(oz/ton)	koz Au	(kton)	(oz/ton)	koz Au	(kton)	(oz/ton)	koz Au	(kton)	(oz/ton)	koz Au
Oxide	0.005	90,864	0.017	1,515	233,467	0.016	3,696	324,330	0.016	5,211	66,935	0.017	1,127
Trans, Sul.	0.005	1,690	0.021	35	18,501	0.021	380	20,191	0.021	415	21,494	0.020	425
JSLA Backfill	0.004	4,478	0.012	54	30,305	0.008	239	34,784	0.008	293	3,524	0.008	28
Waste Dumps	0.004	-	-	-	-	-	-	-	-	-	3,145	0.009	29
Combined	-	**97,032**	**0.017**	**1,604**	**282,273**	**0.015**	**4,315**	**379,305**	**0.016**	**5,919**	**95,097**	**0.017**	**1,608**

Notes:

1. Mineral Resources are reported inclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500/oz gold.
3. Hard rock open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold whereas JSLA Backfill and Waste Dump Mineral Resources are reported using a cut-off grade of 0.0040 oz/ton (0.14 g/t) gold. Resources are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarized in Table 14-19.
4. The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
5. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
6. Any discrepancies in the totals are due to rounding.
7. Mineral resources from Castle Mountain Project presented herein have an effective date of June 30, 2020.

Table 14-23 Castle Mountain Open Pit Resources Inclusive of Reserves (metric units)

Type	Au COG	Measured			Indicated			Measured and Indicated			Inferred		
		Tonnes	Au	Contained	Tonnes	Au	Contained	Tonnes	Au	Contained	Tonnes	Au	Contained
	(g/t)	(kt)	(g/t)	koz Au	(kt)	(g/t)	koz Au	(kt)	(g/t)	oz Au	(kt)	(g/t)	koz Au
Oxide	0.17	82,430	0.57	1,515	211,797	0.54	3,696	294,228	0.55	5,211	60,722	0.58	1,127
Trans, Sul.	0.17	1,533	0.72	35	16,784	0.70	380	18,317	0.70	415	19,499	0.68	425
JSLA Backfill	0.14	4,063	0.41	54	27,493	0.27	239	31,555	0.29	293	3,197	0.27	28
Waste Dumps	0.14	-	-	-	-	-	-	-	-	-	2,853	0.32	29
Combined	-	**88,026**	**0.57**	**1,604**	**256,074**	**0.52**	**4,315**	**344,099**	**0.54**	**5,919**	**86,271**	**0.58**	**1,608**

Notes:

1. Mineral Resources are reported inclusive of reserves.
2. Mineral Resources are reported using gold price of $1,500 /oz gold.
3. Hard rock open pit Mineral Resources are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold whereas JSLA Backfill and Waste Dump Mineral Resources are reported using a cut-off grade of 0.0040 oz/ton (0.14 g/t) gold. Resources are constrained using an optimized pit generated using Lerchs Grossmann pit optimization algorithm with parameters summarized in Table 14-19.
   The Mineral Resource statement has been prepared by Trevor Rabb, P.Geo. (Equity) who is a Qualified Person as defined by NI 43-101.
4. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
5. Any discrepancies in the totals are due to rounding.
6. Mineral resources from Castle Mountain Project presented herein have an effective date of June 30, 2020.

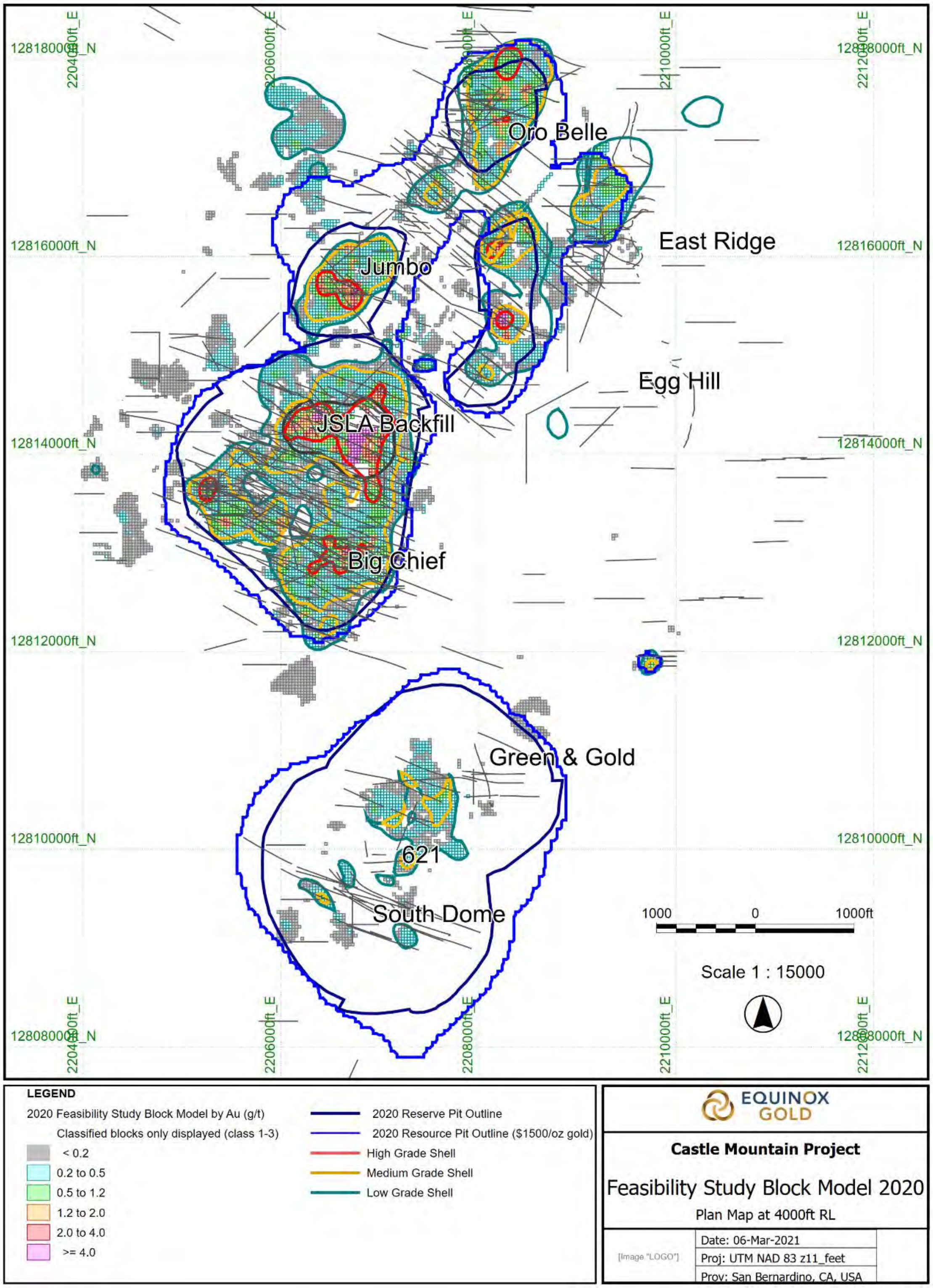


Figure 14-33: Plan Map at 4,000 ft RL Showing Block Model Colored by Estimated Gold Grade (g/t) and Showing Resource and Reserve Pits

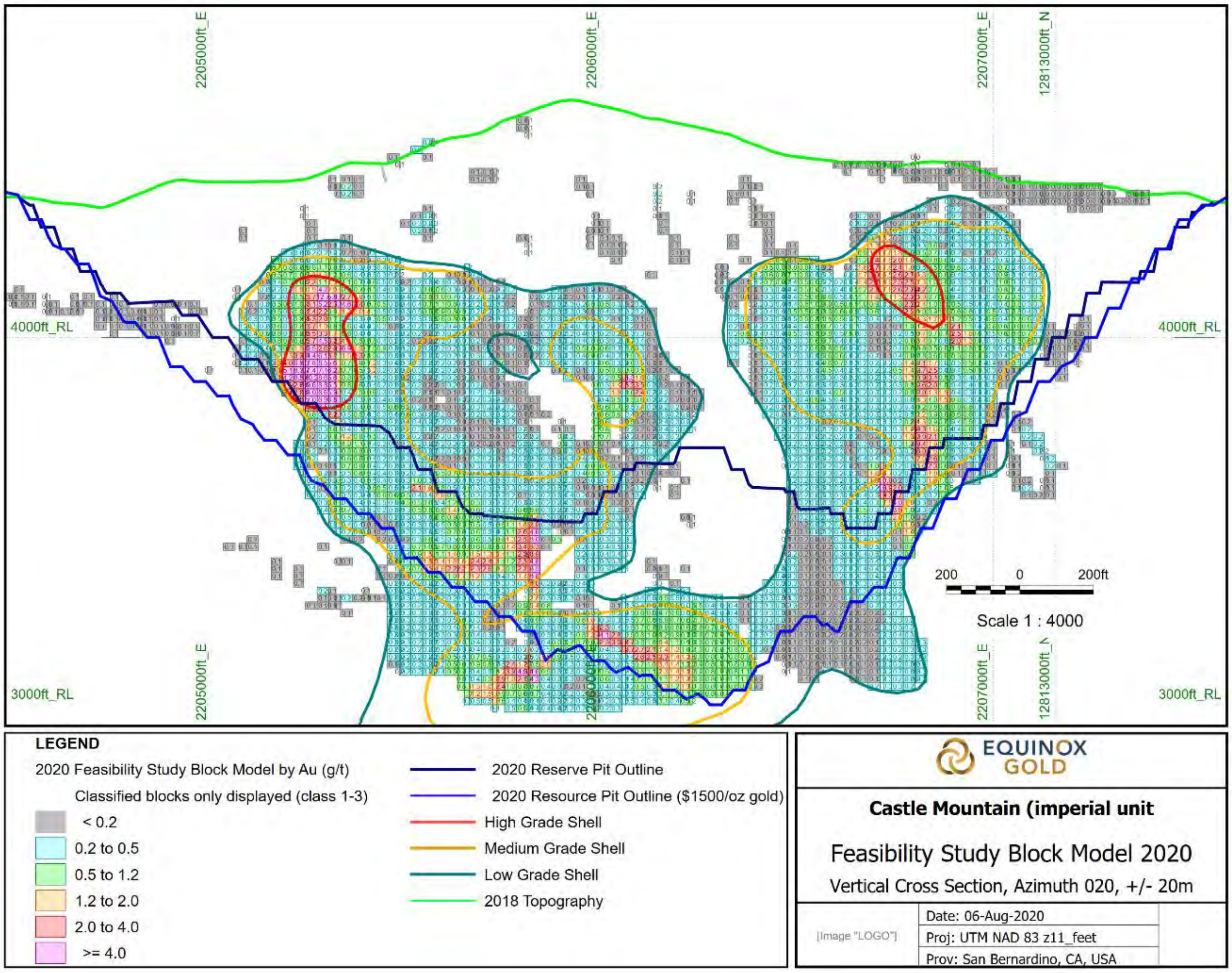


Figure 14-34: Cross Section Oriented at 020˚ Looking Northeast at Big Chief Area Showing Block Model Colored by Estimated Gold Grade (g/t) and Showing Resource and Reserve Pits

14.14 COMPARISON WITH 2018 RESOURCE ESTIMATE

There are several key differences between the two resource models. The differences can be summarized as follows:

- The 2020 Mineral Resource Estimate was generated using Micromine 2020.
- Geological model and estimation domains were generated using Leapfrog v5.0.
- Bench composites that contained missing sample intervals were assigned grades using a down the hole, length weighted average. Intervals less than 2.5 ft (0.8 m) or with less than 10% bench coverage were discarded.
- Classification informed by drill hole spacing.
- Locally varying anisotropy was incorporated from the lithology model where mineralization exhibits lithological controls.
- Non mineralized and post mineral rock types represented in the lithology model are excluded from the grade shell domains.
- Grade shell domain shapes honor anisotropies that control mineralization.

A summary comparing the resource pit constrained hard rock mineral estimates inclusive of reserves from March 29, 2018 and October 24, 2019 is presented in Table 14-24 and Table 14-25 for imperial and metric units, respectively. The data presented in Table 14-24 and Table 14-25 is provided for comparison purposes. JSLA backfill is excluded. The reader is referred to Section 14.13 for the Project's current Mineral Resource statement. Contributions to the changes to the mineral resources are predominantly due to the differences between criterion used for Mineral Resource classification.

Table 14-24: Comparison of Hard Rock Mineral Resources to March 29, 2018 Mineral Resource Estimate Inclusive of Reserves (imperial units)

Resource	Pre-Feasibility (March 29, 2018)			Feasibility (October 24, 2019)			Tonnage	Grade	Contained Au
Zones	2018 Resource Pit (PFS)			March 2020 Resource Pit (FS)					
Au (g/t) Cut-off	> 0.005 oz/ton OP Hardrock			> 0.005 oz/ton OP Hardrock					
Classification	Tons	Au	Contained Au	Tons	Au	Contained Au			
	(Mton)	(oz/ton)	(Moz)	(Mton)	(oz/ton)	(Moz)	% Diff.	Diff.	% Diff
Measured	177.1	0.0169	2.99	92.6	0.017	1.55	-48%	0.00	-48%
Indicated	71.7	0.0161	1.15	252.0	0.016	4.08	251%	0.00	254%
M&I	248.8	0.0167	4.15	344.5	0.016	5.63	39%	0.00	36%
Inferred	167.2	0.0121	2.02	88.4	0.018	1.55	-47%	0.01	-23%

Table 14-25: Comparison of Hard Rock Mineral Resources to March 29, 2018 Mineral Resource Estimate Inclusive of Reserves (Metric units)

Resource	Pre-Feasibility (March 29, 2018)			Feasibility (October 24, 2019)			Tonnage	Grade	Contained Au
Zones	2018 Resource Pit (PFS)			March 2020 Resource Pit (FS)					
Au (g/t) Cut-off	> 0.17 g/t OP			> 0.17 g/t OP					
Classification	Tonnes	Au	Contained Au	Tonnes	Au	Contained Au			
	(Mt)	(g/t)	(Moz)	(Mt)	(g/t)	(Moz)	% Diff.	Diff.	% Diff
Measured	160.6	0.57	2.99	84.0	0.57	1.55	-48%	0	-48%
Indicated	65.1	0.55	1.15	228.6	0.55	4.08	251%	0	254%
M&I	225.7	0.57	4.15	312.5	0.56	5.63	39%	0	36%
Inferred	151.7	0.41	2.02	80.2	0.60	1.55	-47%	0.01	-23%

14.15 FACTORS THAT MAY AFFECT THE MINERAL RESOURCE

Areas of uncertainty that may materially impact the mineral resource estimate include:

- Commodity price assumptions,
- Metal recovery assumptions,
- Mining and process cost assumptions,
- Pit slope angles, and
- Applied top cut values.

In the opinion of the QP, there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors which would materially affect the Mineral Resource estimate.

15 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

The Mineral Reserves for the Castle Mountain Project are based on the conversion of the Measured and Indicated Resources within the ultimate reserve open pit design. Measured Resources are converted to Proven Reserves and Indicated Resources are converted to Probable Reserves.

This section describes the open pit optimization process including key assumptions and economic considerations that lead to pit limit selection and the reporting of Mineral Reserves used for mine planning and scheduling in Section 16.

15.2 OPEN PIT OPTIMIZATION

15.2.1 General

The Project pit optimization and design has been carried out using Hexagon Mining's MinePlan 3D V15.60-1 software for large scale open pit mining. A series of unsmoothed pit shells were created using a Lerchs Grossmann algorithm with revenue factors declining from unity. Optimization input parameters are summarized in Table 15-1. The pit shells were used as a basis for selecting an ultimate reserve pit and to develop detailed pit phase designs to be used in production scheduling. Pit designs were updated for open pits at JSLA, Jumbo, Oro Belle, East Ridge and South Domes.

Table 15-1: Castle Mountain Open Pit Optimization Parameters

Parameter	Unit	Amount	Additional Comments
Gold Price	$/oz	1,350	
Payable	%	99.95	
Marketing Cost	$/oz	1.51	
Royalty – Franco Nevada	%	2.65	Covers entire project including Jumbo, JSLA, Oro Belle
Royalty – American Standard	%	4.65	Cumulative royalty west of JSLA
Royalty – Huntington Tile	%	7.65	Cumulative royalty over South Domes
Royalty – Conservation	%	7.65	Cumulative royalty west of Oro Belle
Mining Cost: ROM	$/ton	1.47	Incremental cost of $0.02/ton per bench from the pit entrance at the 4,500 ft bench elevation
Mining Waste: Backfill & Overburden	$/ton	1.22	
Process Cost: ROM	$/ton	1.33	
Process Cost Milling	$/ton	12.62	Cross over cut-off grade for all rock types have been applied using lithology - process recovery matrix
Process Recovery: ROM	%	73.9	Variable for each rock type, from 65 to 80%
Process Recovery: Mill	%	94.5	Variable for each rock type, from 92 to 96%
General & Administrative	$/ton	0.72	
Mining Dilution	%	3	Grade reduction
Pit Slope	°	48-52	Structural area 40-46°
Haulage	$/ton/bench	0.02	Added for each 20 ft (6.1 m) bench below an entrance bench at 4500 ft elevation.
Sustaining Capital	$/ton	0.05	

15.2.2 Economic Parameters Applied to Mine Design

15.2.2.1 Metal Prices

The Mineral Resource estimate is based on a commodity price of $1,500/oz for gold. The Mineral Reserves estimate is based on a commodity price of $1,350/oz for gold. The metal prices used for pit optimizations were set by Equinox.

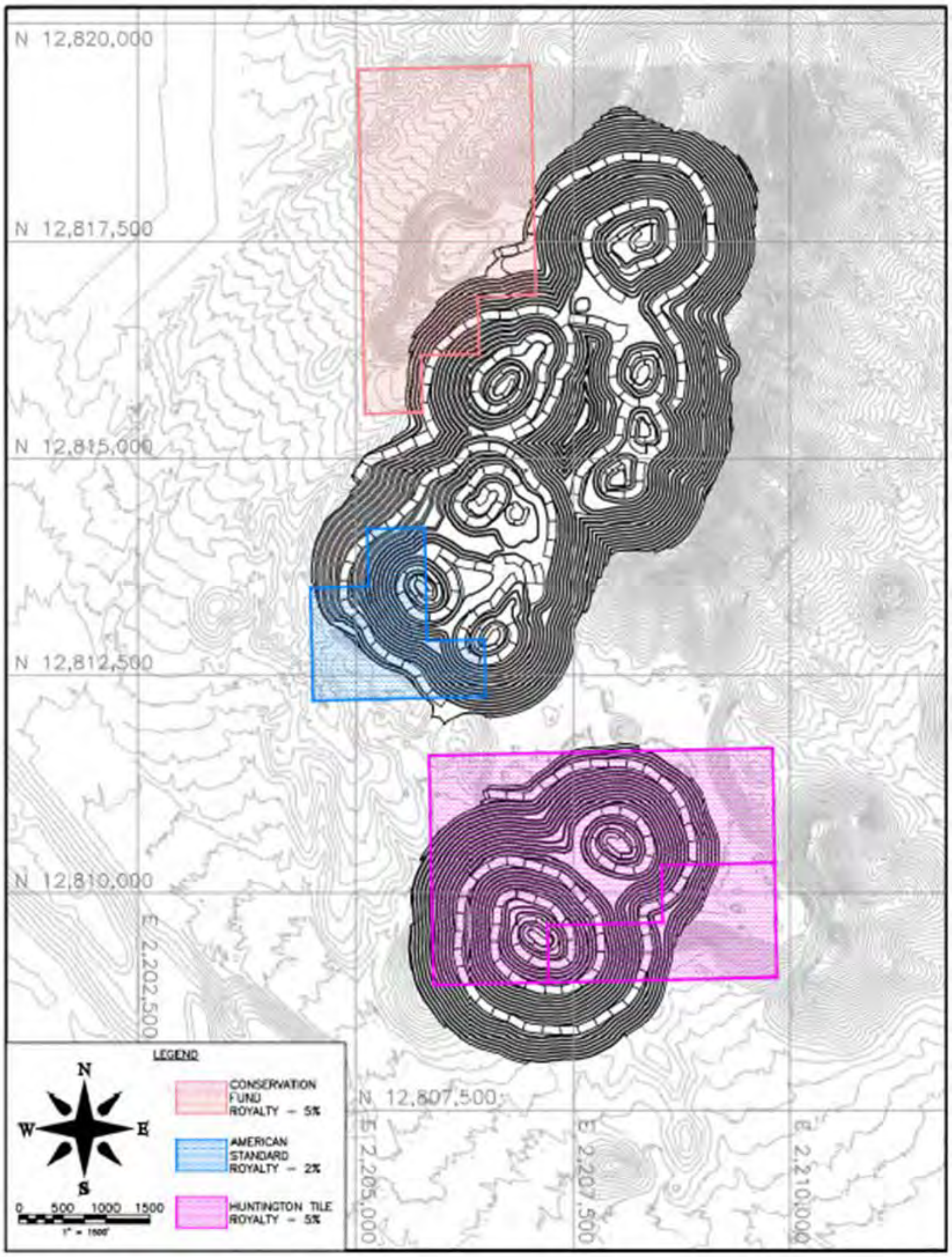
15.2.2.2 Gold Sales, Costs and Royalties

Gold will be shipped offsite as doré. The basis for pit optimization was the potential net revenue per ton calculated for each block in the resource estimate. Metal price and offsite costs for transportation and refining were used in the resource value determination. Gold payable was set at 99.95% and gold sales cost was estimated to be $1.51/oz.

The Project is subject to royalties payable to different parties:

- 2.65% Franco Nevada royalty applied to all ounces,
- 5.00% Conservation royalty,
- 2.00% American Standard royalty, and
- 5.00% Huntington Tile royalty.

The location of the various royalties applied are shown in Figure 15-1.



Source: NMS, 2020

Figure 15-1: Royalty Locations

15.2.2.3 Onsite Operating Costs and Increments

The onsite operating costs used for pit limit analysis were based upon the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018) and include general and administration (G&A), processing and mining (Table 15-1). The G&A costs were estimated to be \$0.72/ton. The processing cost for Run of Mine (ROM) heap leach was estimated to be \$1.33/ton. The cost for milling processing was estimated to be \$12.62/ton. Preliminary operating costs for mining were estimated to be \$1.47/ton mined for mineral resources and \$1.22/ton for waste. An incremental haulage cost of $0.02/ton/bench was added for each 20 ft (6.1 m) bench below an entrance bench at 4500 ft (1372 m) elevation.

15.2.2.4 Sustaining Capital Consideration

Sustaining capital consideration has been made for leach pad, tailings storage and mining equipment. These are ongoing costs related directly to heap leaching and tailings placement. Mining equipment is consumed in direct relationship to mined quantities. Based upon estimates from the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018), an allowance of $0.05/ton was made for equipment sustaining capital and $0.40/ton for leach pad and tailings storage sustaining capital.

15.2.3 Metallurgical Parameters

15.2.3.1 Process Selection

Run of Mine (ROM) heap leaching was selected as the processing option for the majority of the material at the Project. ROM material will be placed on the heap leach pad at an average rate of 50,000 ton/d. Higher grade material will be processed via mill processing at proposed throughput of 3,500 ton/d.

15.2.3.2 Process Recovery

Metallurgical recovery estimates have been provided based upon the Preliminary Feasibility Study for the Castle Mountain Project (Scott et al., 2018) and ongoing test-work. Recoveries for heap leach and mill processing, by lithology type are shown in Table 15-2.

Table 15-2: Metallurgical Recovery Assumptions

Model Code	Lithology Description	Heap Leach (%)	Mill (%)	Proportion of PFS Reserve Blocks (%)
2	Alluvium	72.4	94.0	0.4
3	Debris Flow	72.4	94.0	0.4
5	Dacite	72.4	94.0	0.4
7	Rhyolite Breccia	65.0	94.0	0.6
9	Porphyrytic Rhyolite	80.0	94.0	24.9
11	Aphyric Rhyolite	70.0	93.0	25.9
14	Volcaniclastic Diatreme	85.0	95.0	4.4
16	Volcaniclastic (Vx)	65.0	96.0	23.3
22	Mudstone	72.4	94.0	0.1
23	Epiclastics	72.4	94.0	4.3
27	Andesite	72.4	92.0	12.2
29	Peach Springs Tuff	72.4	94.0	0.7
30	Proterozoic Sediments	72.4	94.0	1.8
31	Proterozoic Basement	72.4	94.0	0.7

15.2.3.3 Process Destination

Metallurgical recoveries for each process option are variable by lithology, as described above. This variability results in different marginal and cross-over cut-off grades for each lithology. Results for the Aphyric Rhyolite lithology unit (model code 11) are shown in Figure 15-2. Calculated waste cut-off grades and process crossover points are shown in Figure 15-3 and Figure 15-4. These figures demonstrate the variability of possible cut-off grades by lithology.

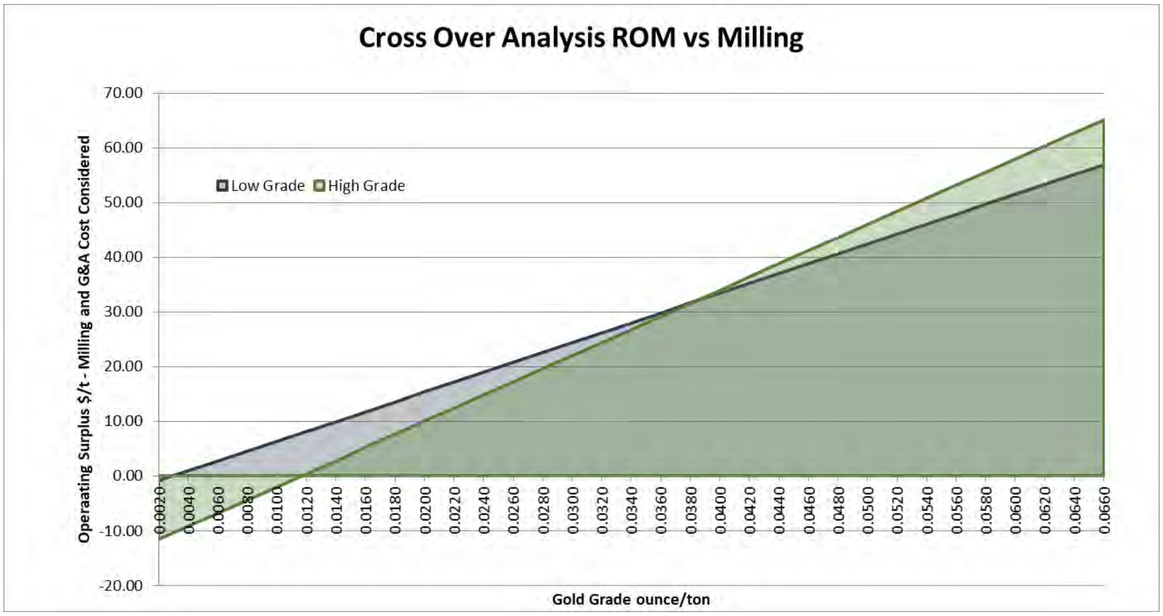


Figure 15-2: Aphyric Rhyolite Cross-over Analysis Heap Leach vs Milling

Figure 15-3: Waste Cut-off Grade

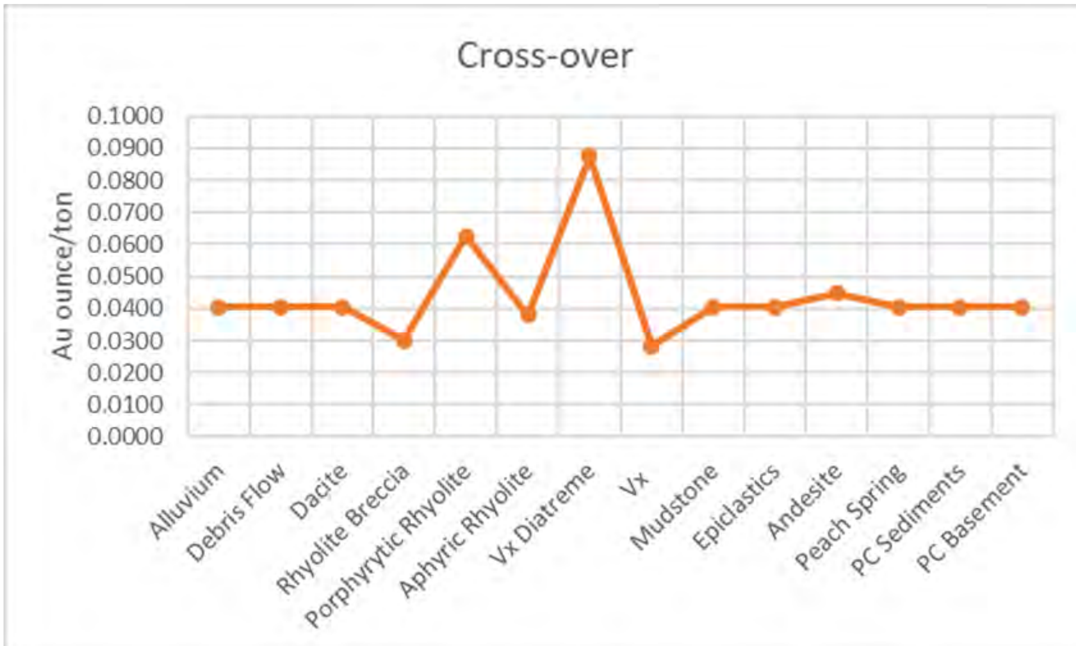


Figure 15-4: Cross-over from Heap Leach to Mill Cut-off

For the purposes of pit optimization, reporting of Mineral Reserves and scheduling, a higher than calculated heap cut-off grade of 0.005 oz/ton (0.17 g/t) has been applied in all mine planning work. Mill feed cut-off grade was based upon the crossover value estimated using the operating surplus net of processing and general and administration cost for the block net smelter return (NSR) values.

15.2.4 Block Model

15.2.4.1 General

A combined resource block model for the JSLA pit backfill material and in-situ hard rock mineral resources was developed by Equity and is described in Section 14. The resource block model is an ore percent model where the grade, tonnage factors and resource classification are pro-rated by the material type. The block model, topography surfaces and lithology wireframes were imported into MinePlan 3D mine planning software. The block model limits and block dimensions are shown in Table 15-3. Pro-rated values for grade, tonnage factors and resource classification were used for mine planning.

Table 15-3: Block Model Limits

Model Limits Fresh Rock and Backfill Combined				
	Units	**Minimum**	**Maximum**	**Length**
Limits X	ft	2,202,300	2,211,300	9,000
Limits Y	ft	12,807,100	12,820,000	12,900
Limits Z	ft	2,600	5,400	2,800
Block X	ft	30.0		
Block Y	ft	30.0		
Block Z	ft	20.0		
Blocks X	blocks	300		
Blocks Y	blocks	430	Blocks	
Blocks Z	blocks	140	18,060,000	

15.2.4.2 Resource Classification

Resource Class

In accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014), the Measured and Indicated resources have been used to define the pit limits and for reporting of Mineral Reserves for scheduling.

Mining Dilution and Recovery

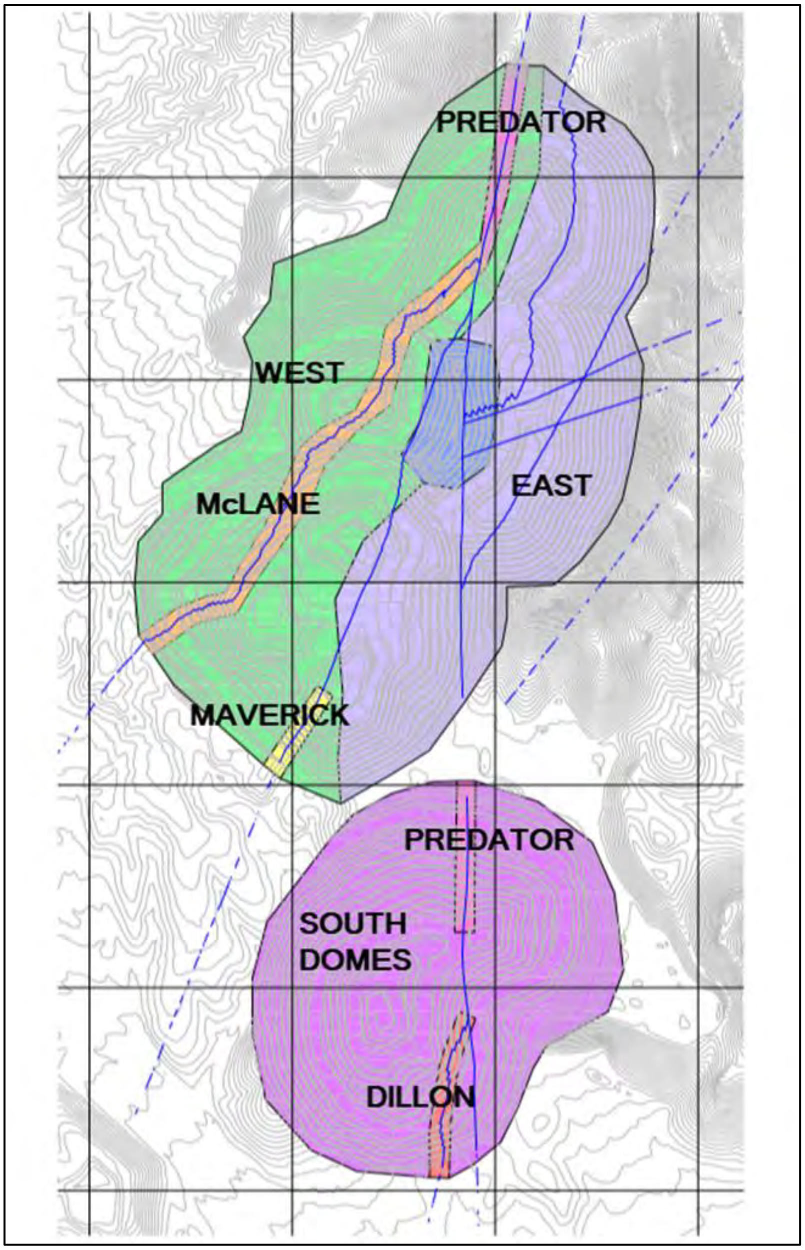
Internal dilution is incorporated in the resource model by virtue of the compositing and interpolation method used to obtain the block grades. Initial resource evaluation resulted in the selection of 0.005 oz/ton (0.17 g/t) cut-off grade for long range planning. A waste contact block analysis was undertaken to evaluate potential external dilution. Mining dilution is expected to result in 3% grade reduction from the resource estimate.

15.2.4.3 Wall Slope Geotechnical

Geotechnical wall slope recommendations have been provided by Call and Nicholas, Inc. (CNI) (Call and Nicholas, 2020). This report provided updated slope recommendations including inter-ramp slope angles and bench design parameters based on laboratory testing, geotechnical drilling information and an updated geotechnical model.

Overall analytical stability analyses were conducted on key areas of preliminary pit designs with updated rock-mass strengths derived from the laboratory testing applied to zones of similar geology. CNI developed a rock quality block model using the rock quality designation parameter (RQD).

Slope recommendations implemented in the mine plan include slope angle classified by “Geology Specific” and “Over-riding Structural Controls” domains (Figure 15-5). Geology Specific domains were identified in the South Domes, West and East domains. Structural domains were coded for the Predator, McLane, Maverick and Dillon fault zones to specified widths based on alteration. Additional domains were added for Alluvium, Backfill and Waste Dumps.



Source: Call and Nicholas, 2020.

Figure 15-5: Slope Design Domains CNI

Inter-ramp slope angle (ISA) recommendations are shown for geology specific and overriding structural control domains in Table 15-4 and Table 15-5, respectively. The geology specific domains are defined in Section 7.

Table 15-4: Geology Specific Inter-Ramp Slope Angles

Domain	Inter-Ramp Slope Angles (°)			
	Rhyolite	Volcaniclastic	Epiclastic	Andesite
South Domes	51	50	50	48
West	51	50	50	48
East	52	52	52	52

Table 15-5: Overriding Structural Controls

Structural Domain	Inter-Ramp Slope Angles (°)
Dillon Fault	40° ISA ±100 ft perpendicular to the fault
McLane Fault	40° ISA ±200 ft perpendicular to the fault
Maverick West Fault	46° ISA ±200 ft perpendicular to the fault
Predator Fault	46° ISA ±100 ft perpendicular to the fault
Island	Removal or operational remediation

Bench design parameters by domain are presented in Table 15-6, including bench-face angle (BFA). A total of 13 slope sectors were defined using a combination geotechnical domains and lithology type, which have been applied in the pit optimization and final designs (Table 15-7 and Figure 15-6).

Table 15-6: Bench Design Parameters by Domain

Domain	ISA (°)	Geology	Design BFA (°)	Design Bench Height (ft)	Design Bench Width (ft)
East	52	All	79	60	35
West South Domes	51	Rhyolite	72	60	29
	50	Volcaniclastic Epiclastic	72	60	31
	48	Andesite	70	60	32
Dillon Fault McLane Fault	40	All	60	60	37
Maverick West Fault Predator Fault	46	All	65	60	30

Table 15-7: Slope Code Summary

Code	Slope Sector Description	OSA (°) MSEP	ISA (°)	Design BFA (°)	Design Bench Height (ft)	Design Bench Width (ft)	Model Code	Fault Zone SLOP1	CNI Sector SLOP2
1	East All	50	52	79	60	35	All		1
2	West and South Domes Default	48	50	72	60	31			
3	West Rhyolite	49	51	72	60	29	7,9,11		2
4	West Volcanics & Epiclastic	48	50	72	60	31	14,16,23		2
5	West Andesite	46	48	70	60	32	27		2

Code	Slope Sector Description	OSA (°) MSEP	ISA (°)	Design BFA (°)	Design Bench Height (ft)	Design Bench Width (ft)	Model Code	Fault Zone SLOP1	CNI Sector SLOP2
6	South Domes Rhyolite	49	51	72	60	29	7,9,11		3
7	South Domes Volcanics & Epiclastics	48	50	72	60	31	14,16,23		3
8	South Domes Andesite	46	48	70	60	32	27		3
9	Dillon and McLane Faults	40	40	60	60	37		1, 2	
10	Maverick West and Predator Faults	46	46	65	60	30		3,4	
11	Alluvium	28	28	36	20	10	2		
12	Backfill	28	28	36	20	10			
13	Waste Dumps	28	28	36	20	10			

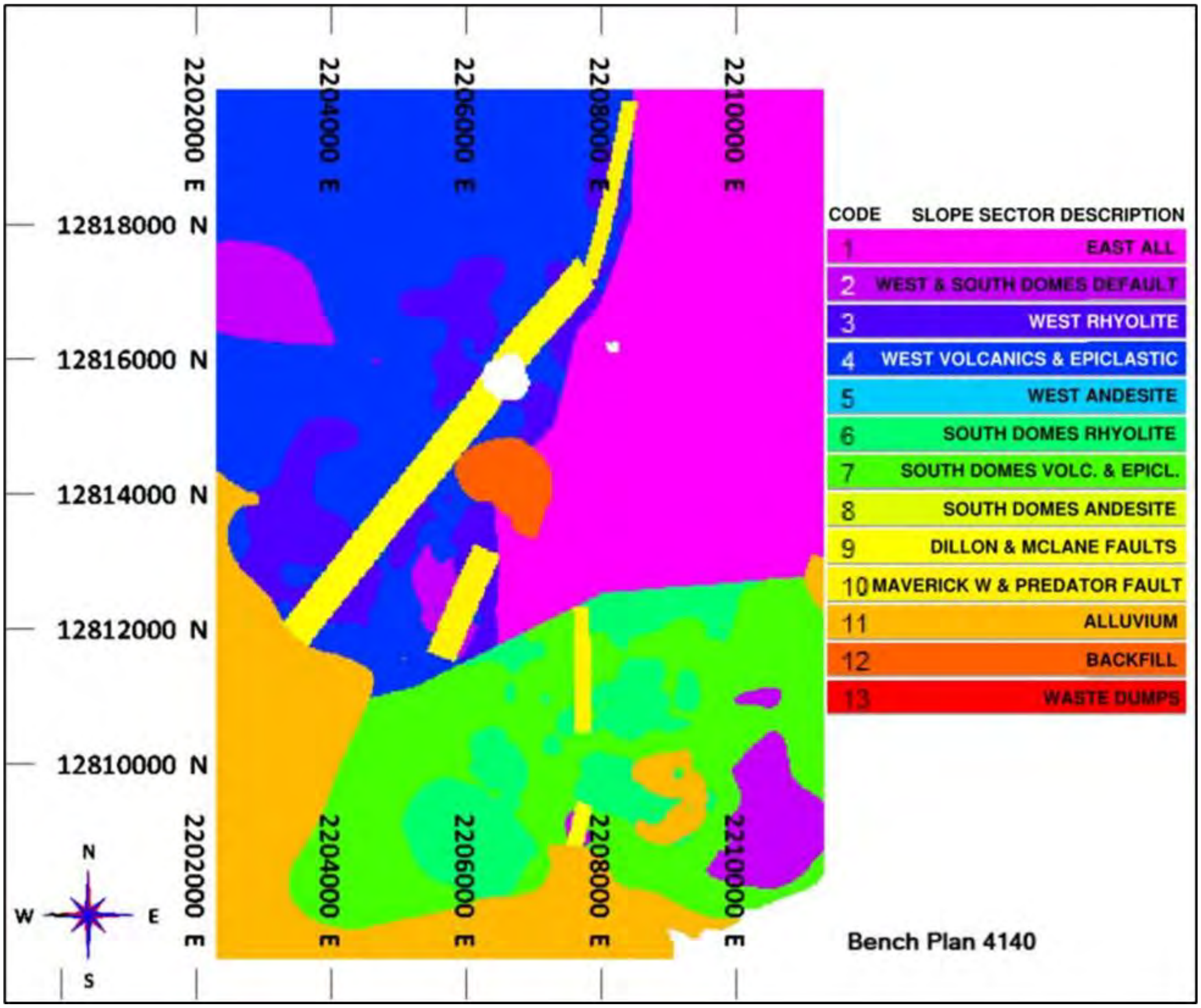


Figure 15-6: Slope Sector Bench Plan

A typical cross section of the lithology is shown in Figure 15-7.

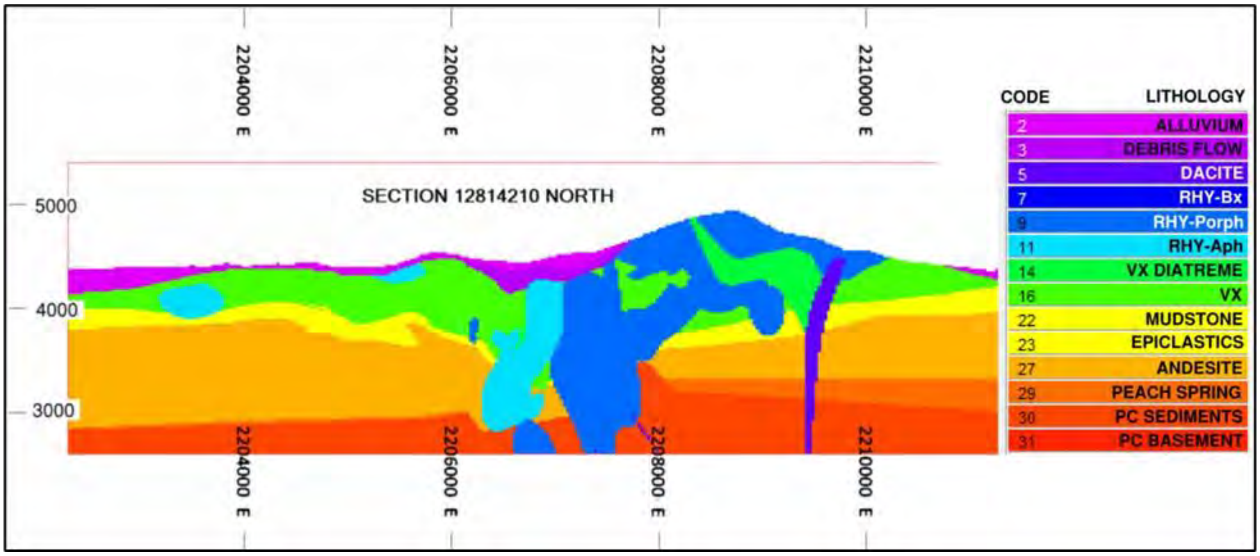


Figure 15-7: Lithology Section

15.2.5 Pit Limit Analysis

15.2.5.1 Pit Limits

Unsmoothed pit limits were developed using a MinePlan® variable slope Lerchs Grossmann algorithm. The preliminary net mine gate revenue and operating costs were used to estimate the value of each regular block in the model. A series of 30 nested pit limits were defined using revenue factors between 0.10 and 1.00.

Table 15-8 summarizes ROM, mill feed and waste for the 30 nested pit shells.

Table 15-8: Lerchs Grossmann Pit Shell Summary

			HEAP LEACH ROCK – IN-SITU			HEAP LEACH BACKFILL			HEAP LEACH TOTAL			MILL				
SHELL	Revenue Factor	Gold Value $/oz	RUN OF MINE kton	Diluted grade oz/ton	ROM NSR $/ton	RUN OF MINE kton	Diluted grade oz/ton	ROM NSR $/ton	RUN OF MINE kton	Diluted grade oz/ton	ROM NSR $/ton	RUN OF MINE kton	Diluted grade oz/ton	MILL NSR $/ton	WASTE TOTAL kton	TOTAL kton
1	0.100	$135	21,493	0.0121	12.05	8,332	0.0109	10.38	29,825	0.0118	11.58	1,280	0.0834	103.20	10,575	41,680
2	0.131	$177	27,552	0.0121	11.97	12,252	0.0100	9.54	39,804	0.0115	11.22	1,931	0.0895	111.06	19,357	61,092
3	0.162	$219	31,988	0.0119	11.81	14,834	0.0097	9.21	46,822	0.0112	10.99	2,250	0.0879	109.03	25,193	74,265
4	0.193	$261	44,322	0.0122	11.92	17,314	0.0094	8.93	61,636	0.0114	11.08	3,171	0.0804	99.49	44,056	108,863
5	0.224	$303	77,987	0.0123	12.01	20,873	0.0091	8.65	98,860	0.0116	11.30	5,234	0.0846	104.37	106,899	210,993
6	0.255	$344	90,017	0.0123	11.96	22,099	0.0090	8.58	112,116	0.0116	11.29	5,866	0.0814	100.45	125,616	243,598
7	0.286	$386	110,113	0.0120	11.75	23,031	0.0090	8.55	133,144	0.0115	11.20	6,866	0.0827	102.28	168,712	308,722
8	0.317	$428	120,246	0.0120	11.69	24,792	0.0089	8.43	145,038	0.0115	11.13	7,446	0.0808	99.93	188,077	340,561
9	0.348	$470	150,602	0.0119	11.54	30,580	0.0086	8.17	181,182	0.0113	10.97	8,920	0.0809	100.19	252,603	442,705
10	0.379	$512	219,419	0.0121	11.50	30,590	0.0086	8.17	250,009	0.0117	11.09	15,208	0.0733	89.48	425,940	691,157
11	0.410	$554	230,658	0.0121	11.48	30,603	0.0086	8.17	261,261	0.0117	11.09	15,700	0.0737	89.94	452,567	729,528
12	0.441	$596	239,170	0.0121	11.44	30,603	0.0086	8.17	269,773	0.0117	11.07	16,430	0.0734	89.50	477,363	763,566
13	0.472	$638	242,514	0.0121	11.44	30,603	0.0086	8.17	273,117	0.0117	11.08	16,657	0.0735	89.56	488,561	778,335
14	0.503	$680	246,941	0.0121	11.45	30,603	0.0086	8.17	277,544	0.0117	11.09	17,257	0.0736	89.52	509,737	804,538
15	0.534	$722	258,842	0.0120	11.40	30,603	0.0086	8.17	289,445	0.0116	11.05	18,437	0.0737	89.54	555,079	862,961
16	0.566	$763	262,297	0.0120	11.38	30,603	0.0086	8.17	292,900	0.0116	11.04	18,676	0.0736	89.41	565,165	876,741
17	0.597	$805	272,402	0.0121	11.45	30,603	0.0086	8.17	303,005	0.0117	11.12	20,756	0.0745	90.04	645,223	968,984
18	0.628	$847	273,929	0.0121	11.45	30,603	0.0086	8.17	304,532	0.0117	11.12	20,812	0.0745	89.98	650,159	975,503
19	0.659	$889	275,101	0.0121	11.46	30,603	0.0086	8.17	305,704	0.0117	11.13	20,926	0.0745	89.98	655,896	982,526
20	0.690	$931	276,896	0.0121	11.45	30,603	0.0086	8.17	307,499	0.0118	11.13	21,024	0.0745	90.01	663,836	992,359
21	0.721	$973	280,271	0.0121	11.46	30,603	0.0086	8.17	310,874	0.0118	11.13	21,290	0.0746	90.01	679,013	1,011,177
22	0.752	$1,015	281,588	0.0121	11.46	30,603	0.0086	8.17	312,191	0.0118	11.13	21,314	0.0745	89.97	682,688	1,016,193
23	0.783	$1,057	283,293	0.0121	11.45	30,603	0.0086	8.17	313,896	0.0118	11.13	21,426	0.0744	89.87	689,967	1,025,289
24	0.814	$1,099	285,231	0.0121	11.44	30,603	0.0086	8.17	315,834	0.0118	11.12	21,480	0.0744	89.79	695,982	1,033,296
25	0.845	$1,141	287,130	0.0121	11.44	30,603	0.0086	8.17	317,733	0.0118	11.12	21,572	0.0743	89.72	704,427	1,043,732
26	0.876	$1,182	287,639	0.0121	11.43	30,603	0.0086	8.17	318,242	0.0118	11.12	21,593	0.0743	89.70	706,175	1,046,010
27	0.907	$1,224	289,083	0.0121	11.43	30,603	0.0086	8.17	319,686	0.0118	11.12	21,641	0.0743	89.65	712,628	1,053,955
28	0.938	$1,266	290,772	0.0121	11.43	30,603	0.0086	8.17	321,375	0.0118	11.12	21,688	0.0742	89.62	720,353	1,063,416
29	0.969	$1,308	291,568	0.0121	11.42	30,603	0.0086	8.17	322,171	0.0118	11.11	21,693	0.0742	89.61	722,712	1,066,576
30	1.000	$1,350	292,278	0.0121	11.42	30,603	0.0086	8.17	322,881	0.0118	11.11	21,716	0.0742	89.59	725,539	1,070,136

The net operating surplus based upon pit optimization input and before capital expenditures is shown graphically in Figure 15-8. Also shown are net surplus at 5% and 10% discount rates.

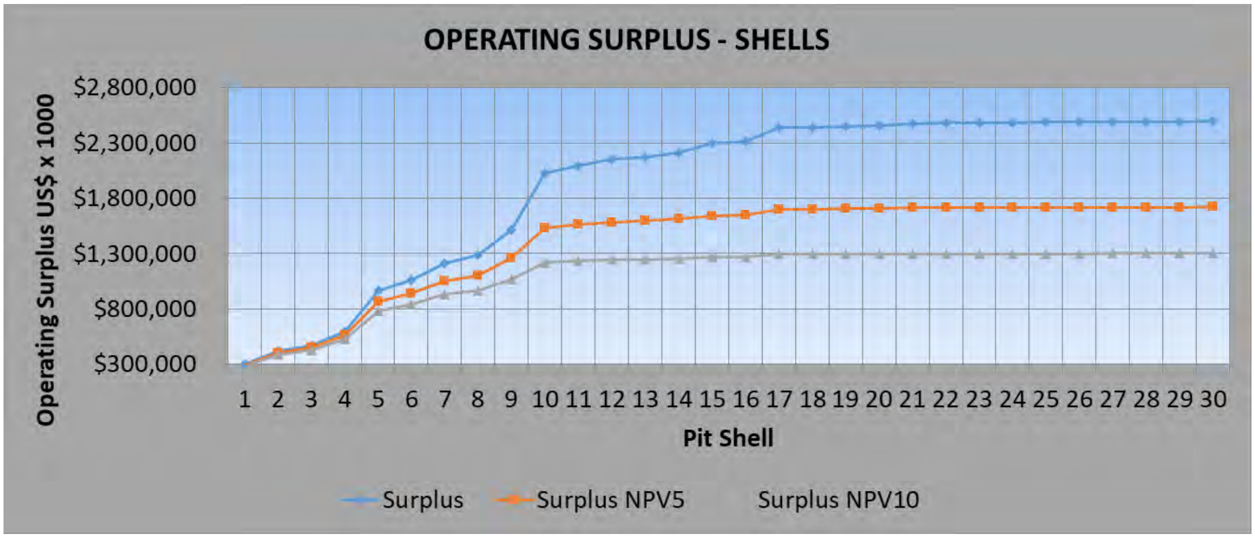


Figure 15-8: Operating Surplus – Pit Shells

The cumulative material for each pit shell is shown in Figure 15-9.

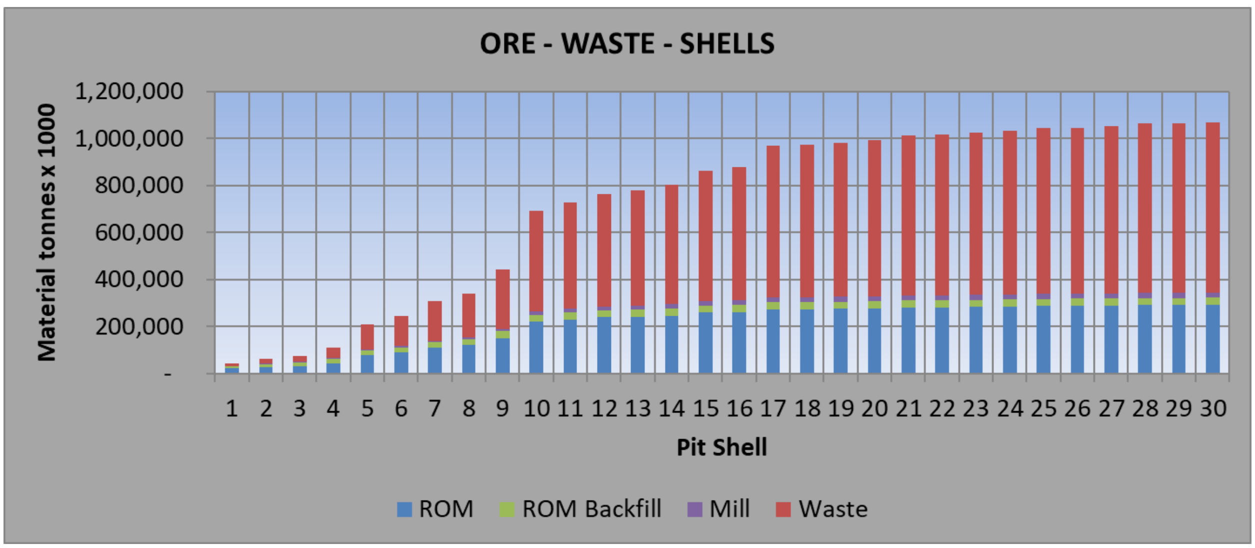


Figure 15-9: Ore and Waste – Pit Shells

The contained gold for each pit shell is shown in Figure 15-10.

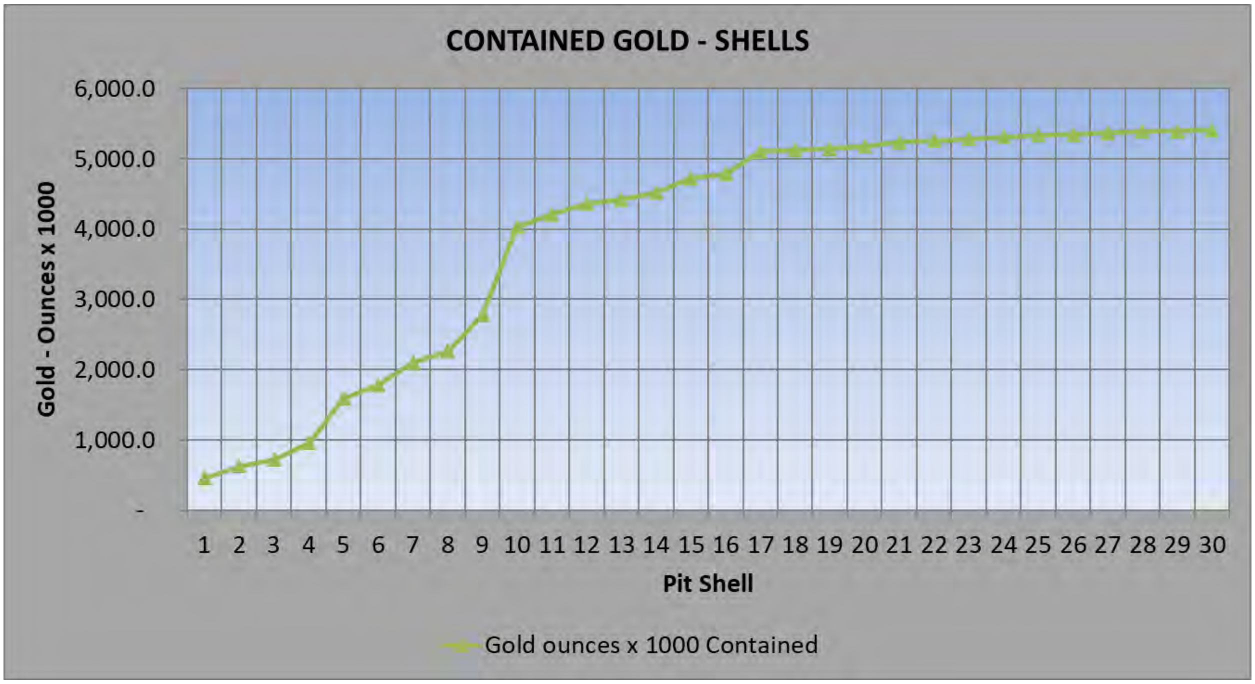


Figure 15-10: Contained Gold – Pit Shells

The cumulative proportion of discounted value and potential ore tonnage is shown together in Figure 15-11.

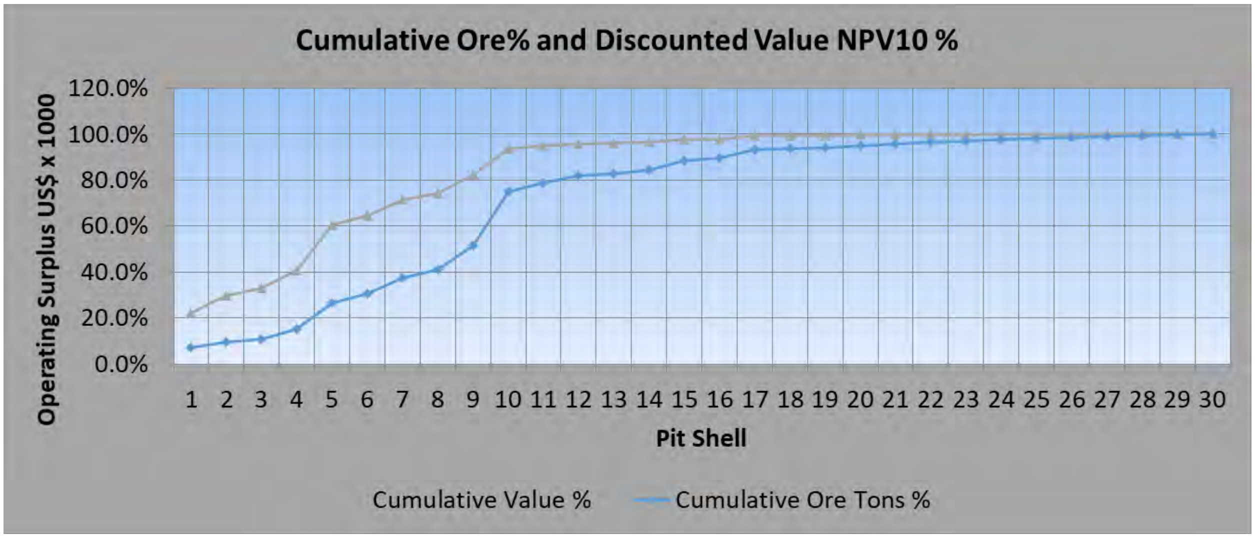
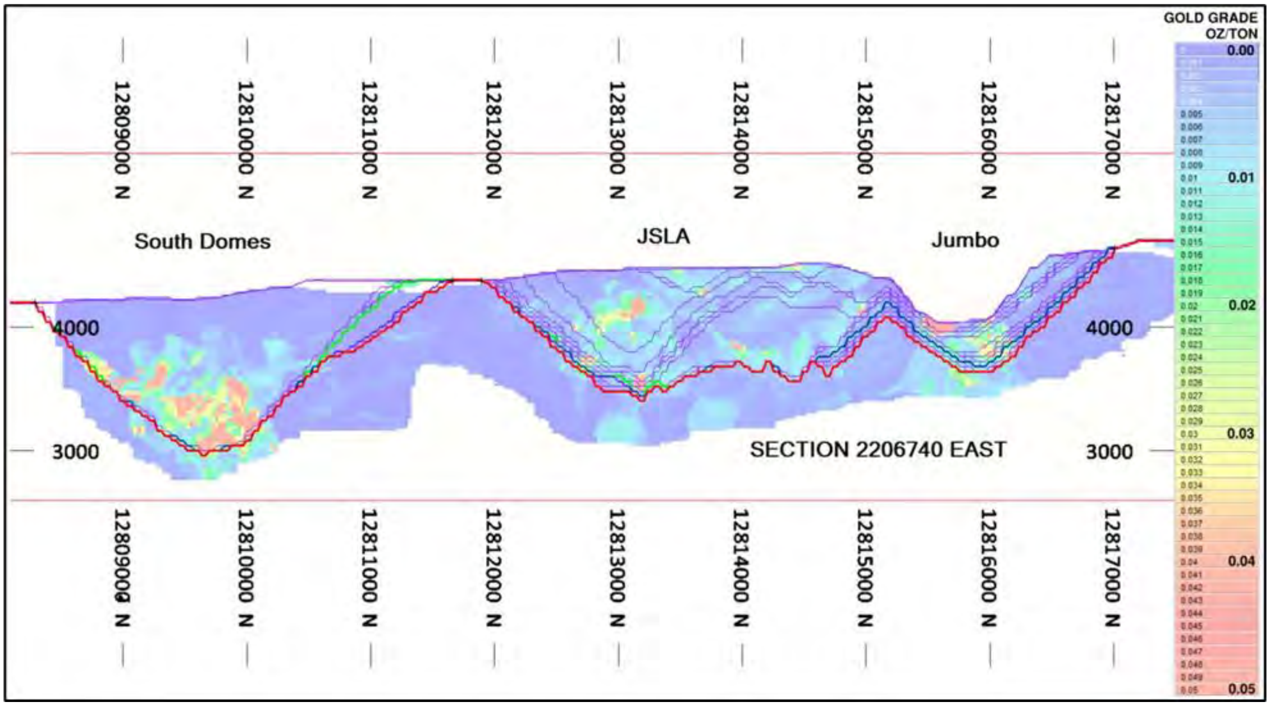


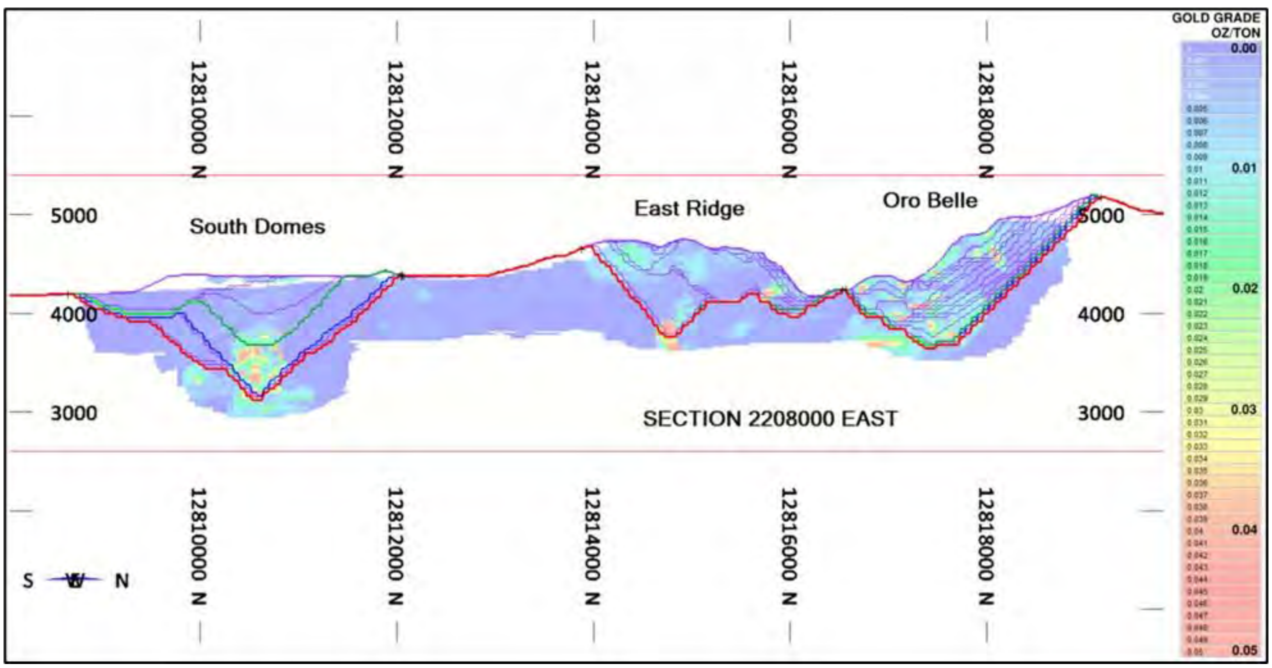
Figure 15-11: Cumulative Resource and Discounted Value – Pit Shells

The nested pit limits were used to guide pit design and are shown on the sections and plan below (Figure 15-12 to Figure 15-14).



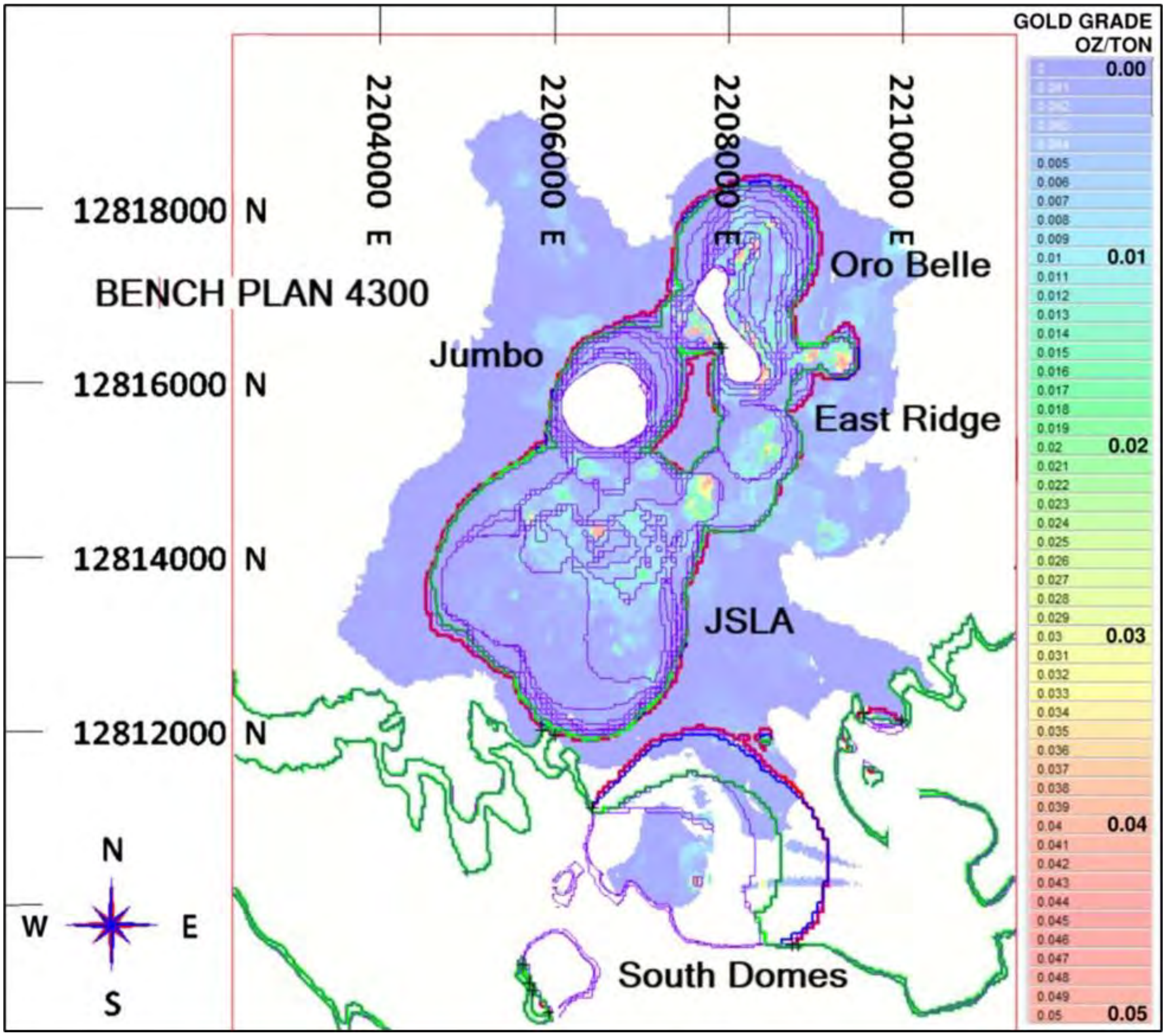
Source: NMS, 2020

Figure 15-12: Section 2206740 East Lerchs Grossmann Pit Limits



Source: NMS, 2020

Figure 15-13: Section East Lerchs Grossmann Pit Limits

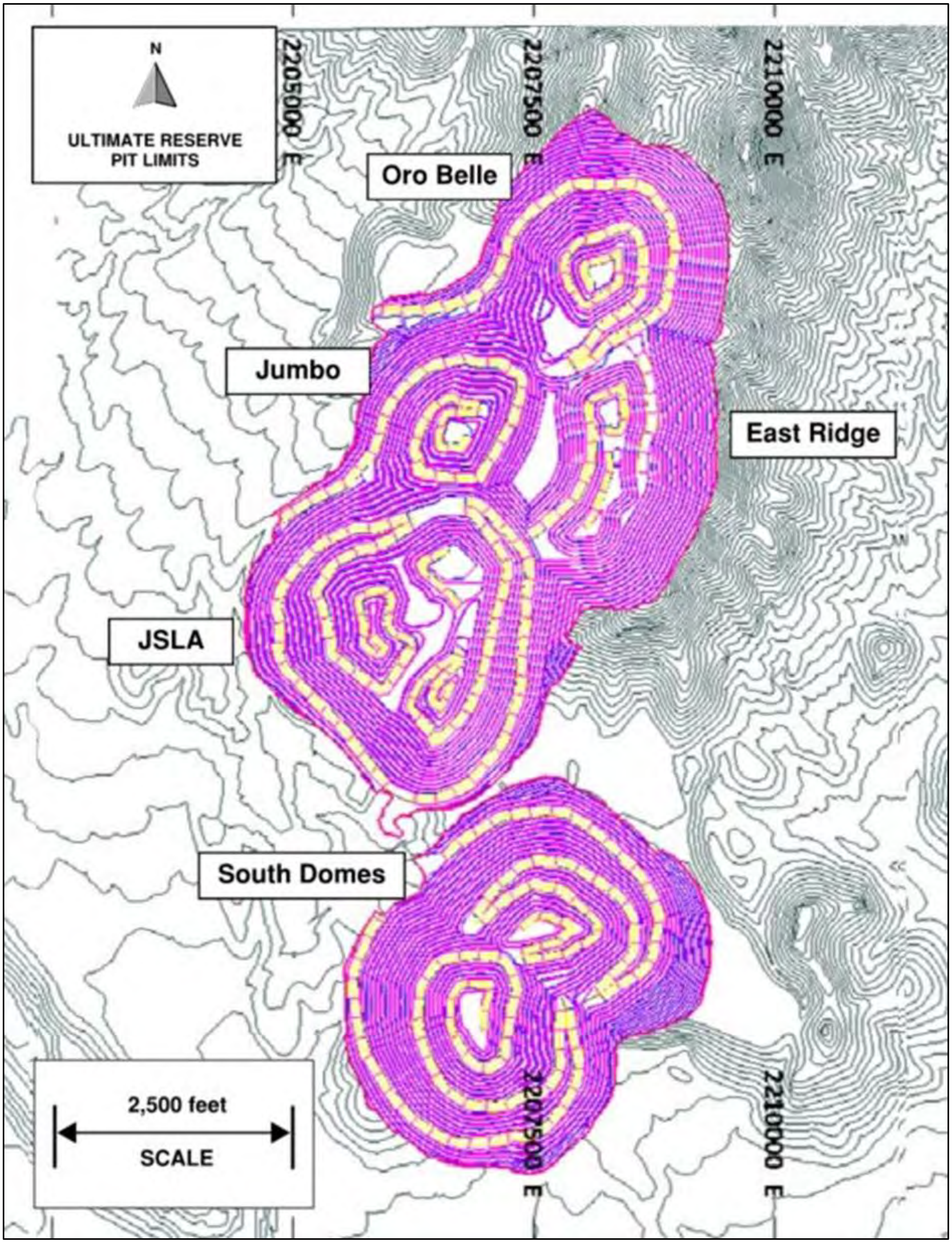


Source: NMS, 2020

Figure 15-14: Bench Plan 3385 Lerchs Grossmann Pit Limits

The design basis for the initial pit designs was 50,000 ton/d ROM to the leach pad and 3,500 ton/d to the mill. Preliminary schedules for this throughput indicated that the 99.3% of the net present value could be attained over 17.7 years with the 17th pit shell developed with a revenue factor of 0.60. This pit was selected as a guide for a detailed design.

A detailed description of the pit design is provided in Section 16. The open pit has been designed to be developed in nine phases. The ultimate reserve pit configuration is shown in Figure 15-15.



Source: NMS, 2020

Figure 15-15: Ultimate Reserve Pit Limits

15.3 MINERAL RESERVE SUMMARY

The Castle Mountain Mineral Reserves are summarized in Table 15-9. The Mineral Reserves have been reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold.

Table 15-9: Mineral Reserve Summary

Imperial	Tons (kton)	Gold Grade (oz/ton)	Gold (koz)
Proven	93,600	0.016	1,498
Probable	190,690	0.014	2,670
Subtotal	**284,290**	**0.015**	**4,168**
Metric	**Tonnes (kt)**	**Gold Grade (g/t)**	**Gold (koz)**
Proven	84,910	0.55	1,498
Probable	172,990	0.48	2,670
Subtotal	**257,900**	**0.51**	**4,168**

Notes:
1. The Mineral Reserve estimate with an effective date of June 30, 2020 is based upon the Mineral Resource estimate prepared for Equinox by Trevor Rabb P.Geo, and described in Section 14, with an effective date of June 30, 2020.
2. The Mineral Reserve was estimated by Nilsson Mine Services Ltd. with supervision by John Nilsson P.Eng. who is a Qualified Person as defined under NI 43 ‐ 101.
3. Mineral Reserves are reported within the ultimate reserve pit design with overall economics developed for $1350/oz gold with appropriate royalties applied.
4. Mineral Reserves are reported using a cut-off grade of 0.005 oz/ton (0.17 g/t) gold.
5. The mining costs average $1.78/ton ($1.96/t) mined, processing costs are $1.33/ton ($1.47/t) for ROM and $12.62/ton ($13.91/t) for milling. G&A was $0.72/ton ($0.79/t) ore processed.
6. The average process recovery was 73.9% for ROM and 94.5% for milling.
7. Ore tons are reported in thousands of short tons (kton) and ounces.
8. Mineral Resource is exclusive of Mineral Reserves.

The Mineral Reserve estimate has been based upon economic parameters, geotechnical design criteria and metallurgical recovery assumptions detailed within this section. Changes in these assumptions will impact the Mineral Reserve estimate. In general, increases in operating costs, reductions in revenue assumptions or reductions in metallurgical recovery may result in increased cut-off grades, reductions in in-pit resources and increasing strip ratios. Reductions in operating costs, increases in revenue assumptions, or increases in metallurgical recovery may result in reduced cut-off grades and increases in in-pit resources. Environmental permits are required for the Phase 2 mine plan to proceed. These permits have a material influence on the reserve statement.

16 MINING METHODS

16.1 INTRODUCTION

Mine planning, equipment selection and cost estimation for the Castle Mountain Project was undertaken by Nilsson Mine Services Ltd. (NMS). This section of the report summarizes the design process, the mine schedule, the mine operations plan, operating and capital cost estimates for the open pit.

16.2 SUMMARY

16.2.1 Project Description

The current Phase 1 project configuration consists of a 14,000 ton/d (12,700 t/d) run of mine (ROM) heap leach operation with a planned expansion in Phase 2 to 50,000 ton/d (45,400 t/d) ROM and 3,500 ton/d (3,200 t/d) milling. The mine will be a conventional diesel powered truck and shovel operation. The Mineral Reserve is estimated to be 284.3 Mton (257.9 Mt) with an average grade of 0.015 oz/ton (0.514 g/t) gold reported at a 0.005 oz/ton (0.17 g/t) gold cut-off grade. The Mineral Reserves will be mined by open pit methods in nine phases of open pit development and expansion. The overall strip ratio is 2.47:1. The total in-pit waste is 701.9 Mton (636.8 Mt). The overall mine life including Phase 1 and Phase 2 is 19 years. Phase 2 mining depends on several things including permits and is expected to commence two years prior to full production. The expanded heap leach and mill are anticipated to reach full production by the start of Year 6, designated as Year 1 for Phase 2.

Phase 1 mining will be focused on mining backfilled material in the JSLA pit. In Phase 2, the ROM, mill feed and waste will be drilled by diesel powered drills and blasted using ammonium nitrate and fuel oil or with emulsion as required in wet conditions. ROM, mill feed and waste will be loaded into 250 ton mine trucks by 44.5 yd^3 diesel hydraulic shovels and 32.0 yd^3 wheel loaders. Waste will be placed in designated disposal sites adjacent to the pit in the early years and backfilled to mined out pits later in the mine life. Heap leach ROM ore will initially be hauled to the existing Phase 1 leach pad. In Phase 2 of the Life of Mine (LOM) plan, ROM will be hauled to a new Phase 2 leach pad that will be developed southwest of the mine progressing from south to north. Mill feed will be placed in a stockpile adjacent to the primary crusher and re-handled by wheel loaders.

16.2.2 Resources and Mineable Reserves

The resource block model for the Project was developed using conventional block modelling techniques. A combined ore percent resource block model for the JSLA pit backfill material and in-situ hard rock mineral resources is described in Section 14. Measured and Indicated Resources have been used to report Mineral Reserves and respectively develop the mine plan.

Mineral Reserves are summarized by phase in Table 16-1. These Mineral Reserves have been summarized within the ultimate reserve pit design used in this study.

Table 16-1: Mineral Reserves by Phase (Imperial Units)

	ROM ROCK		ROM BACKFILL		ROM TOTAL		MILL		TOTAL			
Pit Design Phase	**Ore Tons (kton)**	**Diluted Gold Grade (oz/ton)**	**Ore Tons (kton)**	**Diluted Gold Grade (oz/ton)**	**Ore Tons (kton)**	**Diluted Gold Grade (oz/ton)**	**Ore Tons (kton)**	**Diluted Gold Grade (oz/ton)**	**Ore Tons (kton)**	**Diluted Gold Grade (oz/ton)**	**Waste Tons (kton)**	**Total Tons (kton)**
Phase 1 - JSLA Backfill	-	-	26,524	0.0088	26,524	0.0088	193	0.0567	26,717	0.0091	14,166	40,883
Phase 2 - JSLA East	50,088	0.012	3,130	0.0085	53,218	0.0114	3,334	0.0481	56,552	0.0136	95,817	152,369
Phase 3 - JSLA West	28,401	0.011	-	-	28,401	0.0107	2,973	0.0735	31,374	0.0167	78,638	110,012
Phase 4 - Jumbo	13,837	0.013	-	-	13,837	0.0127	1,112	0.0886	14,949	0.0183	43,206	58,155
Phase 5 - Oro Belle	39,323	0.012	-	-	39,323	0.0121	2,783	0.0559	42,106	0.0150	104,402	146,508
Phase 6 - East Ridge	12,667	0.010	-	-	12,667	0.0097	365	0.0394	13,032	0.0105	31,128	44,160
Phase 7 - East Ridge	24,818	0.011	-	-	24,818	0.0107	2,072	0.0852	26,890	0.0165	66,150	93,040
Phase 8 - South Domes	57,313	0.014	-	-	57,313	0.0136	2,313	0.0767	59,626	0.0160	167,800	227,426
Phase 9 - South Domes	11,636	0.015	-	-	11,636	0.0147	1,415	0.0835	13,051	0.0222	100,618	113,669
Total	**238,086**	**0.012**	**29,654**	**0.0087**	**267,740**	**0.0117**	**16,560**	**0.0683**	**284,300**	**0.0150**	**701,925**	**986,225**

16.2.3 Open Pit Mine Plan

The mine production forecast is summarized in Table 16-2. The overall mine production has been scheduled by bench and development phase on an annual basis. A cut-off of 0.005 oz/ton (0.17 g/t) was applied for determining waste. In general, mill feed cut-off was based upon maximum net of process value for each metallurgical ore type. In some years when mill feed quantities are low, redirection of ROM material was implemented to reach mill capacity. Redirected ore totals 1.14 Mton (1.03 Mt) over the life of mine. The mine plan will depend on several activities including permitting and incorporates the following assumptions:

- Phase 1 – initiated in month 1 with contractor mining of backfilled ROM ore from the JSLA open pit to Phase 1 leach pad.
- Phase 2 – initiated in month 36 with JSLA pioneering and access road construction to commence using contractors in month 37. Contractor to develop JSLA East pit to 4,540 ft elevation.
- Staggered mining equipment deliveries month 42 to month 48. Mining rate begins to ramp up as equipment is delivered.
- Ramp up overall mining rate to 73 Mton/y (66 Mt/y) through to Year 9 then expand gradually to 80 Mton/y (73 Mt/y) through to Year 16 after which production begins to drop through Year 19.
- Overall sequence of development in the JSLA, Jumbo, Oro Belle and East Ridge area is clockwise development to final to pit limits in each area to allow for an orderly sequence of backfilling waste as pits are completed.
- Sequence at South Domes is an initial southwest pit with an expansion to the northeast.
- The resource block model was developed on 20 ft (6 m) benches. The mine design was developed using the 20 ft bench height with triple benching to 60 ft (18 m) between design catch benches or berms. Operations are planned for a 30 ft bench height. Sinking rates in the schedule were limited to 300 ft/y or the equivalent of 10 benches/year. Drills, loading units and support equipment appropriate for mining a 30 ft (9 m) bench height have been selected for the mine plan and associated cost estimates.

Table 16-2: Annual Mine Production Schedule

MINING SCHEDULE

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
ROM Ore	kton	5,110.0	5,110.0	5,110.0	5,110.0	10,950.4	18,249.6	18,250.0	18,250.0	18,250.0	18,250.0	18,250.0	18,250.0	18,250.0	18,249.7	18,250.0	18,250.0	18,250.0	14,774.6	2,576.0	**267,740.3**
Gold Grade	oz/ton	0.011	0.009	0.009	0.008	0.009	0.011	0.011	0.011	0.011	0.010	0.010	0.013	0.011	0.012	0.012	0.014	0.013	0.019	0.018	**0.012**
Gold Recoverable	oz/ton	0.008	0.007	0.006	0.006	0.006	0.008	0.008	0.008	0.008	0.007	0.008	0.009	0.008	0.009	0.009	0.011	0.010	0.014	0.013	**0.009**
Mill Ore	kton	93.0	50.0	17.5	29.4	201.5	1,057.9	1,259.4	1,479.9	1,217.0	723.0	1,255.7	1,661.6	1,021.7	915.8	1,364.3	1,273.2	492.5	1,922.4	524.0	**16,560.0**
Gold Grade	oz/ton	0.060	0.055	0.044	0.058	0.056	0.064	0.047	0.052	0.070	0.046	0.072	0.065	0.062	0.056	0.085	0.093	0.093	0.082	0.064	**0.068**
Gold Recoverable	oz/ton	0.056	0.052	0.041	0.055	0.054	0.061	0.045	0.049	0.067	0.044	0.068	0.061	0.058	0.053	0.081	0.088	0.088	0.077	0.060	**0.064**
Waste Rock	kton	-	-	-	10,569.0	33,797.0	32,532.3	39,789.6	42,238.0	52,325.5	47,627.4	50,560.7	50,604.4	57,660.1	59,449.1	48,776.5	52,783.0	42,515.5	11,988.7	2,178.0	**635,395.0**
Waste Alluvium	kton	-	-	-	354.0	1,101.0	2,196.0	75.0	271.5	291.5	426.0	893.0	2,137.6	1,518.4	-	2,807.2	3,821.8	-	-	-	**15,893.0**
Waste Backfill	kton	1,260.3	1,730.2	1,960.4	2,083.3	7,406.5	6,645.0	3,811.7	1,384.6	669.0	25.0	-	-	-	-	-	-	-	-	-	**26,976.0**
Waste Dump	kton	-	-	-	91.0	327.0	2,457.0	-	344.0	408.3	6,729.7	1,853.0	542.0	-	-	7,160.1	3,748.9	-	-	-	**23,661.0**
Total Waste	kton	1,260.3	1,730.2	1,960.4	13,097.3	42,631.6	43,830.3	43,676.4	44,238.1	53,694.3	54,808.1	53,306.7	53,284.0	59,178.6	59,449.1	58,743.8	60,353.7	42,515.5	11,988.7	2,178.0	**701,925.0**
Total Material	kton	6,463.3	6,890.2	7,088.0	18,236.7	53,783.4	63,137.8	63,185.7	63,968.0	73,161.4	73,781.2	72,812.4	73,195.5	78,450.3	78,614.6	78,358.1	79,876.9	61,258.1	28,685.7	5,278.0	**986,225.3**

PROCESSING SCHEDULE

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
ROM Ore	kton	5,110.0	5,110.0	5,110.0	5,110.0	10,950.4	18,249.6	18,250.0	18,250.0	18,250.0	18,065.6	18,228.2	18,250.0	18,250.0	18,016.3	18,250.0	18,250.0	17,547.5	14,774.6	2,576.0	**266,598.3**
Gold Grade	oz/ton	0.011	0.009	0.009	0.008	0.009	0.011	0.011	0.011	0.011	0.010	0.010	0.013	0.011	0.012	0.012	0.014	0.012	0.019	0.018	**0.012**
Gold Recoverable	oz/ton	0.008	0.007	0.006	0.006	0.006	0.008	0.008	0.008	0.008	0.007	0.008	0.009	0.008	0.009	0.009	0.011	0.010	0.014	0.013	**0.009**
Recovery	%	72.2%	72.1%	72.2%	73.6%	70.5%	71.5%	69.0%	69.7%	71.1%	72.7%	72.1%	70.5%	74.2%	78.3%	75.2%	78.1%	85.8%	76.9%	73.3%	74.1%
Recoverable Gold	oz	38,885	34,735	31,407	30,786	66,487	140,300	141,195	142,345	137,122	131,165	136,798	163,484	150,091	164,350	164,096	193,583	174,878	212,649	33,305	**2,287,660**

	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
MILLING																					
Mill Ore	kton	-	-	-	-	-	1,203.0	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,277.5	1,169.0	**17,702.0**
Gold Grade	oz/ton	-	-	-	-	-	0.063	0.047	0.052	0.070	0.047	0.071	0.065	0.062	0.054	0.085	0.093	0.065	0.082	0.074	**0.067**
Gold Recoverable	oz/ton	-	-	-	-	-	0.060	0.045	0.049	0.066	0.044	0.067	0.061	0.059	0.051	0.081	0.088	0.061	0.077	0.070	**0.063**
Recovery	%	0.0%	0.0%	0.0%	0.0%	0.0%	94.9%	94.5%	95.1%	95.6%	94.6%	93.9%	94.0%	94.3%	94.3%	95.0%	94.6%	94.4%	93.8%	93.6%	94.5%
Recoverable Gold	oz	-	-	-	-	-	72,455	57,214	62,614	84,890	56,345	85,655	77,921	74,980	65,536	103,625	112,194	78,398	98,808	81,340	**1,111,977**

16.2.4 Waste Rock Storage and Stockpile Plan

The total waste to be mined from the open pit is 701.9 Mton (636.8 Mt). Waste has been reported as waste rock that will be drilled and blasted and as alluvium, backfill and waste dump material that will be free-digging. Over the life of mine, waste will be placed in the East Dump, the Northwest Dump and backfilled to the JSLA, Jumbo, Oro Belle and East Ridge open pits.

16.3 OPEN PIT DESIGN

16.3.1 General

This section of the report describes the basis for the reserve open pit design including the design parameters, design summary, Mineral Reserves and waste material storage method.

The reserve open pit design has been based upon the following key considerations:

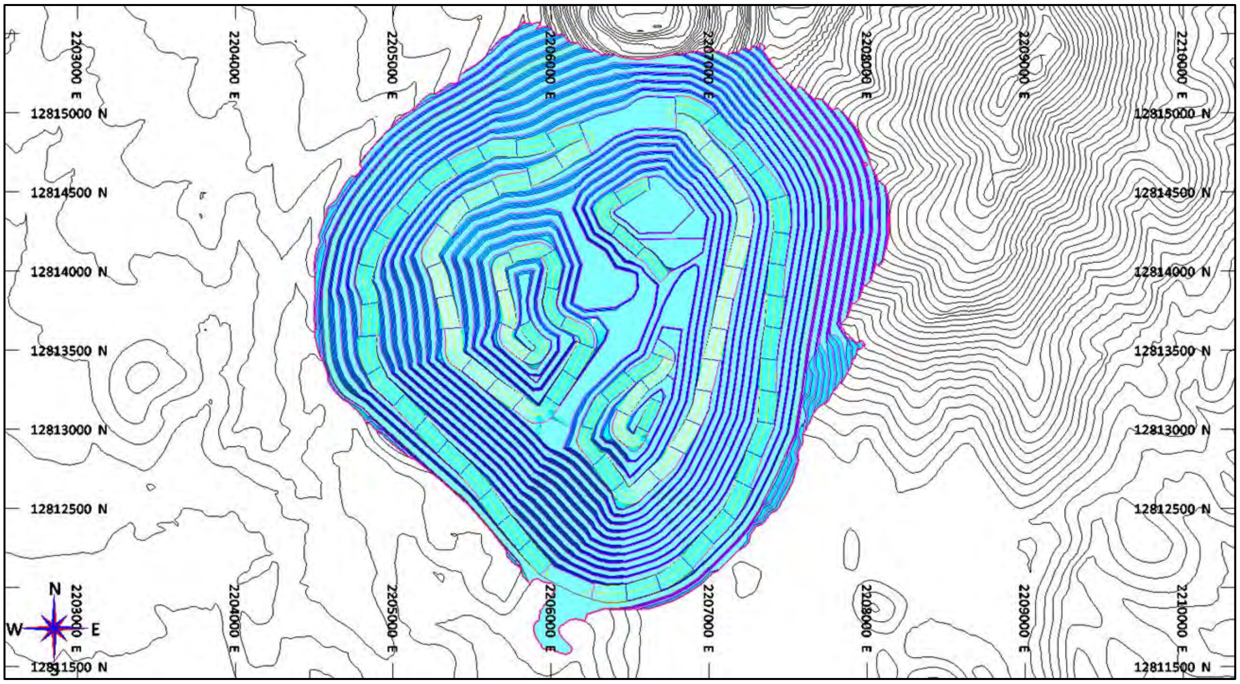
- Overall and inter-ramp slope recommendations provided by Call & Nicholas, Inc (Call & Nicholas, 2020), as discussed in 15.2.4.3.
- Waste dump final slopes of 2H:1V or 26.5°.
- Operating constraints of the equipment selected for mining:
    - Minimum mining width defined by double side loading of trucks with allowance for an access ramp.
    - Bench height achievable and within the safe operating reach of the primary loading units.
    - Minimum haulage road operating width and maximum effective grade within the operating limitations of the primary haulage units.
- Logical and efficient scheduling of material movement from multiple phases of pit expansion to the ROM pad, stockpiles and to final waste material placement sites.
- Minimum footprint for disturbance of the surrounding area.

16.3.2 Design Summary

The mining equipment will operate on a 30 ft (9m) high bench. Berms will be left on alternate benches in hard rock. CNI wall slope design recommendations have been implemented for inter-ramp slopes with variable berm widths and bench face angles applied as described in Section 15.2.4.3 (Call & Nicholas, 2020).

16.3.2.1 JSLA

The JSLA Phase 1 pit is based upon the Pre-feasibility Study plan to mine backfill at a rate of 14,000 ton/d (Scott et al., 2018). The design reflects use of the existing final ramp in hard rock from the backfilled pit. The Phase 2 design, shown in Figure 16-1, developed in this study has a double access ramp system that allows the pit to be split for scheduling of two phases. These ramps will exit the pit on the south side at 4,300 ft elevation. The pit bottoms will be at 3,480 ft elevation on the east and west sides. The pit will be 3,700 ft across in the east-west direction and 3,750 ft in the north-south direction. The overall wall height will be 1,200 ft on the east side.

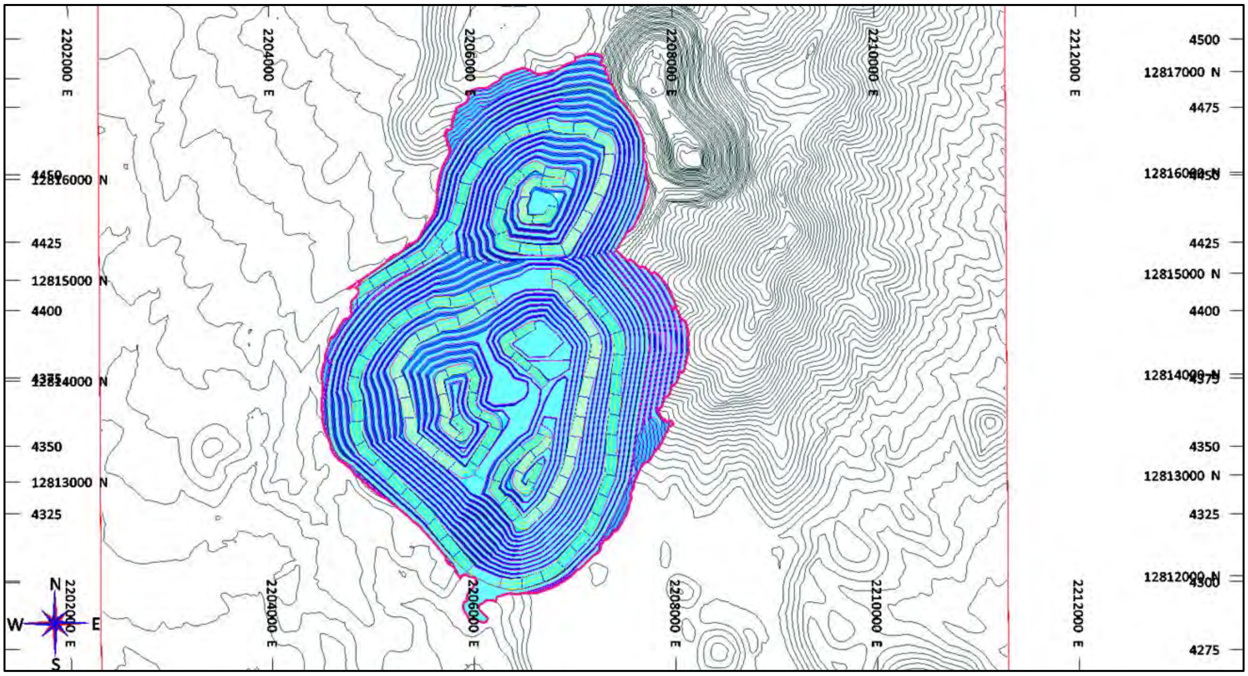


Source: NMS, 2020

Figure 16-1: JSLA Pit Design

16.3.2.2 Jumbo

The Jumbo pit design is shown in Figure 16-2. It will be located immediately north of the JSLA pit and intersect the JSLA design and the existing Oro Belle design to the north. The access ramp to the Jumbo pit will be along the north wall of the JSLA pit and the exit will be at 4,400 ft elevation. The bottom of the pit will be 3,760 bench. Overall wall height will be 940 ft. The pit will measure 2,000 ft in the east-west direction and 1,900 ft in the north-south direction.

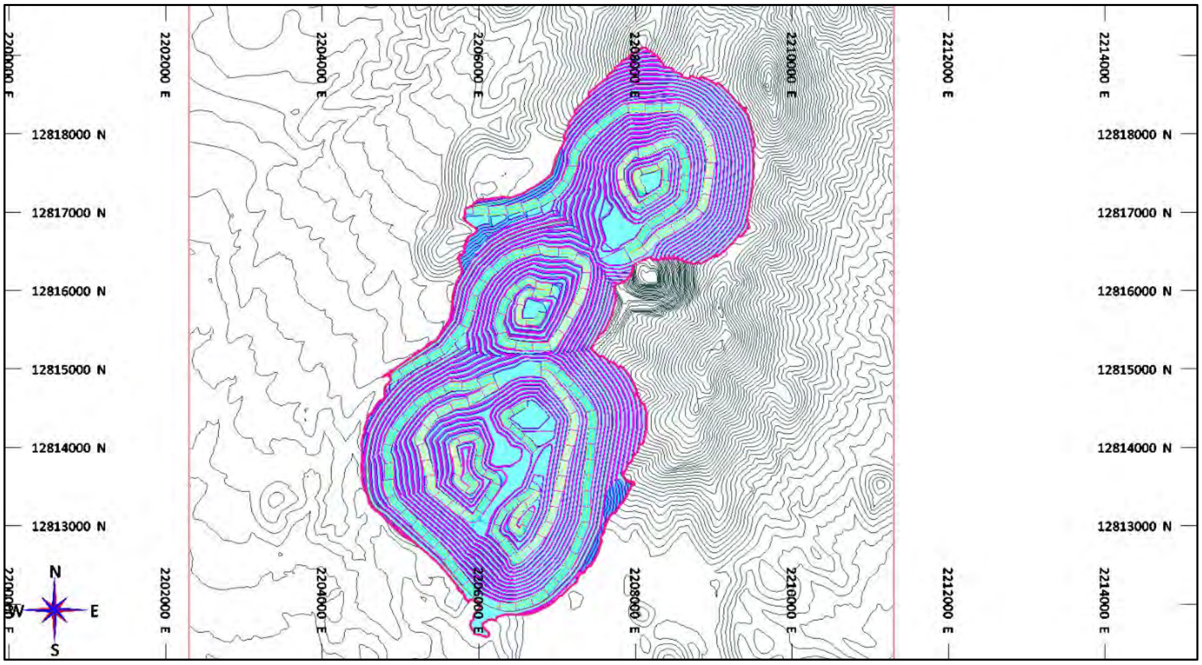


Source: NMS, 2020

Figure 16-2: Jumbo Pit Design

16.3.2.3 Oro Belle

The Oro Belle design, shown in Figure 16-3, is immediately north of Jumbo. The access ramp will break out 4,660 ft elevation on an existing dump. The north wall will break through the ridge crest at 5,180 ft elevation and the east wall will break through at 5,020 ft. The maximum wall height will be 1,420 ft on the north side. The pit will measure 2,580 ft in the east-west direction and 3,050 ft in the north-south direction. The narrow conical pit bottom resulted in some ore losses at depth due to ramp access issues. There may be an opportunity to regain some of those losses in future design iterations.



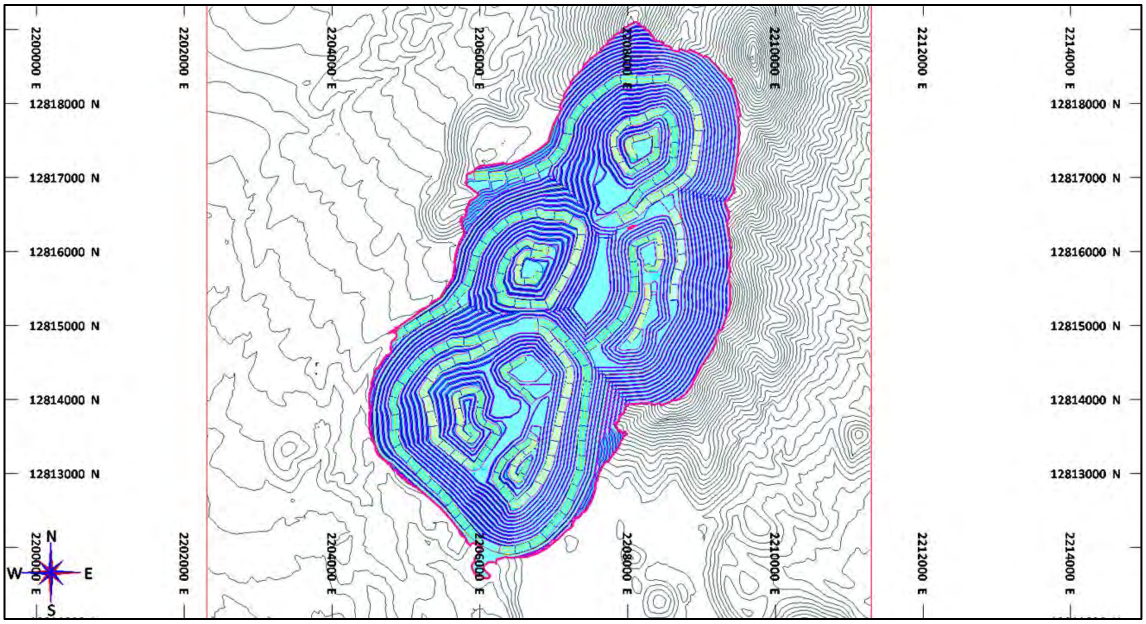
Source: NMS, 2020

Figure 16-3: Oro Belle Design

16.3.2.4 East Ridge

The East Ridge pit was designed to be developed as two phases. Access to the first internal phase will be from the east side of the ridge on an external road up from the surface of the East Dump. This road will provide access to the crest of ridge and the upper benches of East Ridge and Oro Belle. A road will be left in the east wall of the first phase to allow access to the upper benches of the second phase. Extraction of material from the lower benches of the East Ridge pit will be via the Oro Belle ramp and then finally out through the JSLA ramp connected by backfill.

The crest of the East pit will be at 4,920 ft elevation. The overall wall height when complete will be 1,100 ft. The pit will be 1,600 ft wide in the east-west direction and 2,900 ft in the north-south direction. The final configuration of East Ridge is shown in Figure 16-4.

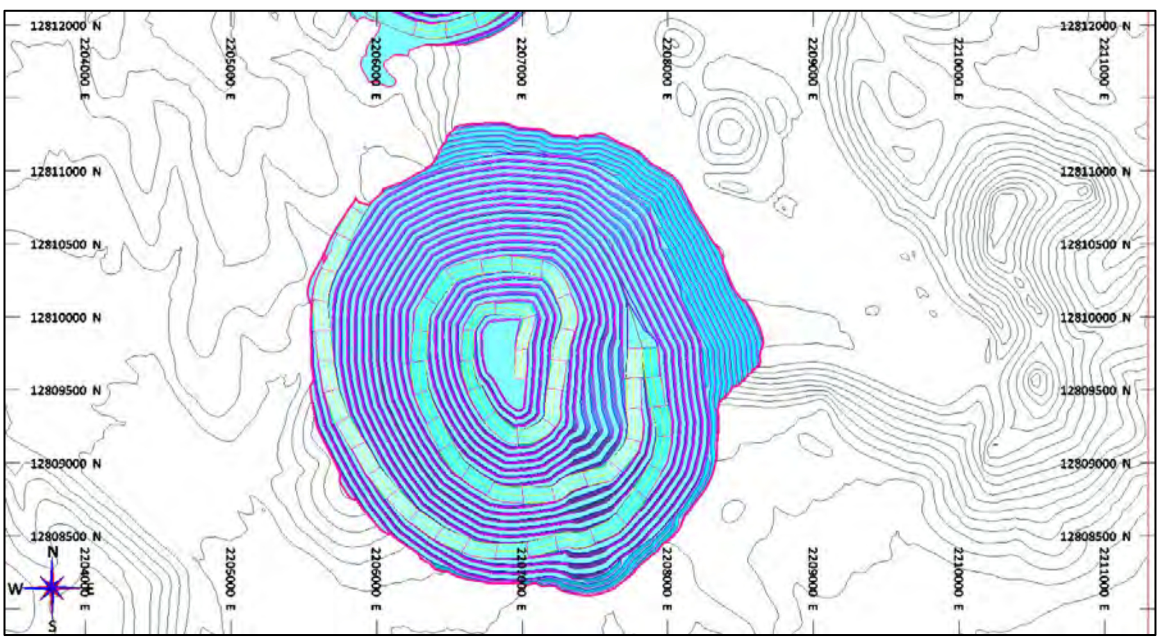


Source: NMS, 2020

Figure 16-4: East Ridge Pit

16.3.2.5 South Domes

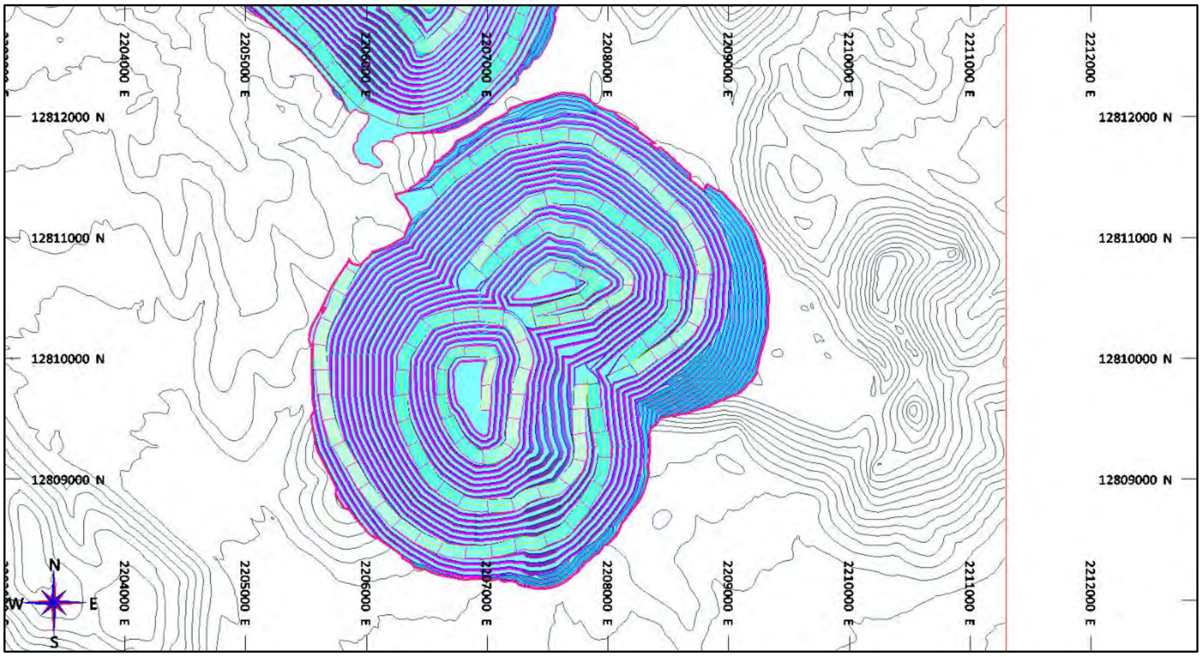
The South Domes Phase 1 pit is shown in Figure 16-5. The pit will be 1,260 ft deep and measure 3,100 ft across in the east-west direction and 3,150 ft in the north-south direction. The north and northeast walls will be developed through up to 220 ft of dump material placed during previous mining. The main ramp will be developed in a counter-clockwise direction with a switchback on the east wall at the intersection with the Phase 2 Pit.



Source: NMS, 2020

Figure 16-5: South Domes Phase 1

South Domes Phase 2 is an expansion of the Phase 1 pit to the northeast. It will be developed with a counter-clockwise ramp that switches back three times on the way to the bottom. The east wall will be developed in 180 ft of waste dump at the surface. Final wall height will be 1,080 ft. The Phase 2 South Domes pit expansion is shown in Figure 16-6.

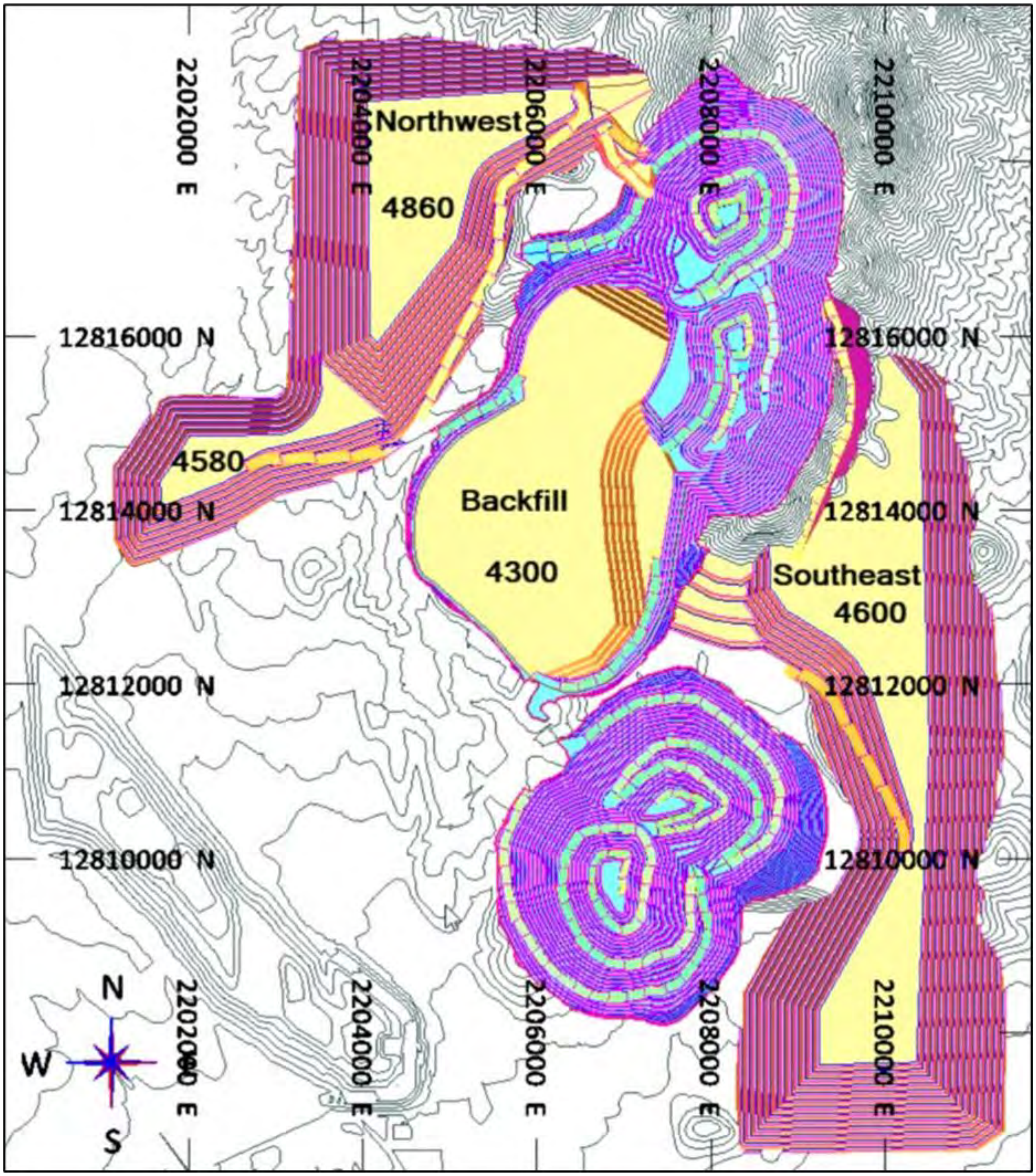


Source: NMS, 2020

Figure 16-6: South Domes Phase 2

16.3.3 Waste and Low-Grade Storage

Conceptual designs for waste dumps and waste backfill plans have been developed to accommodate volumes scheduled over the life of mine. These designs and possible final elevations are shown in Figure 16-7 below. The lift height between berms will be variable over time. The face slope during construction of the dumps was assumed to be 38° and overall final reclamation slopes will be 26.5°. The waste dump configuration with backfill completed to the JSLA and Jumbo pits is shown in Figure 16-7.



Source: NMS, 2020

Figure 16-7: Waste Dump Configuration

The general sequence for dump development and destination quantities used for haulage productivity estimation are shown in Figure 16-8.

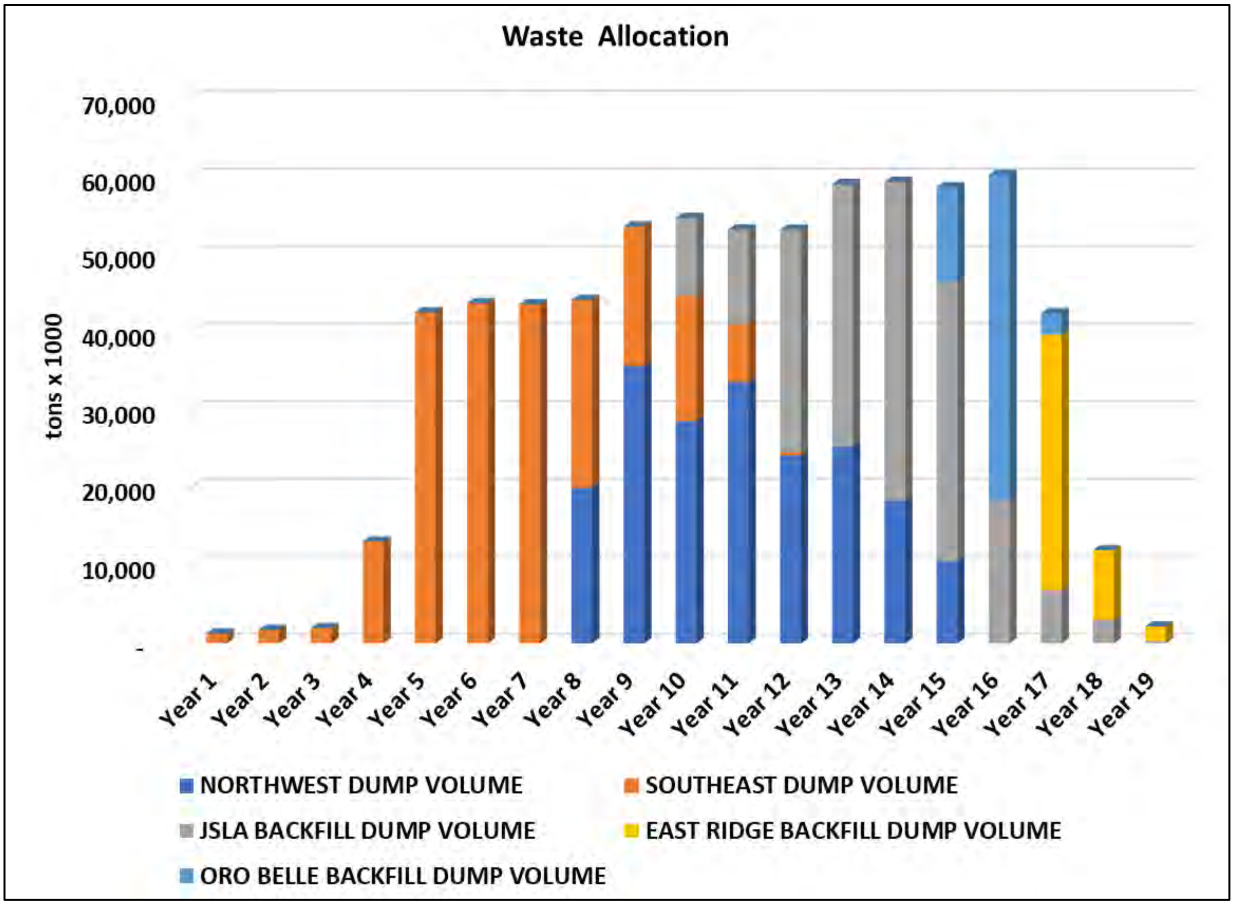


Figure 16-8: Waste Dump Material Allocation

16.3.4 Haulage Roads

Surface haulage roads will connect the pit ramps to the ROM leach pads, high grade stockpile and waste dumps. When possible, these roads will be constructed using waste rock. As in the pit, surface haulage roads will have a running surface three times the width of the largest haulage truck with allowance for ditches and berms. Roads will have a maximum grade of 10% but may be constructed to 8% to improve haulage cycle times and reduce truck component wear.

A typical haulage road cross section is shown in Figure 16-9. The trucks proposed in this mine plan are 250 ton class units with an overall width of 24 ft and 4 in (24’4”). The road allowance in the pits is 100 ft to accommodate two-way traffic, ditches and berm. This allowance has been increased on the surface to 120 ft.

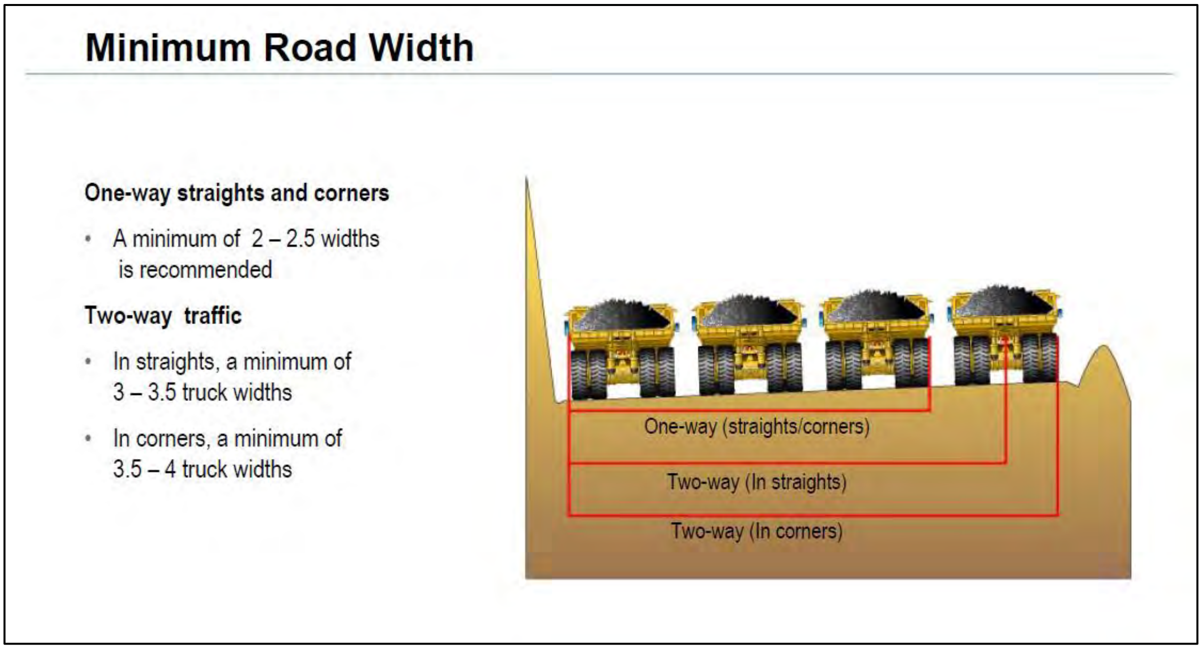


Figure 16-9: Haulage Road Width Guideline

Roads will be dressed with 12 in of coarse crush, 4 in, followed by 10 in of fine crush,1 in, road dressing. External roads on surface will be built with an 18 in layer of clean sand sub-base.

16.4 MINE PRODUCTION SCHEDULE

16.4.1 Summary

The open pit mine development plan consists of nine pit development phases expanding to two large open pits. A waste dump will be located along the east property boundary and a second dump will be expanded to the northwest property boundary. A high-grade mill feed stockpile will be located adjacent to the mill primary crusher.

The mine will operate as a conventional diesel-powered truck shovel operation. The typical production cycle will be drilling, blasting, grade control, loading and hauling. Primary loading units will be hydraulic shovels and wheel loaders with support equipment providing development access, road maintenance and equipment servicing capability.

The mine will operate for 19 years delivering 266.5 Mton (241.8 Mt) of ROM ore with an average grade of 0.012 oz/ton (0.41 g/t) gold to the leaching operation. The mill will commence operation after Year 5 and will process 17.8 Mton (16.1 Mt) of ore with an average grade of 0.066 oz/ton (2.26 g/t) gold. The total effective waste mined will be 701.9 Mton (636.8 Mt). The effective overall strip ratio will be 2.47:1.

16.4.2 Cut-off Grade Selection

As described in Section 15, pit limit analyses have been carried out using a Lerchs Grossmann algorithm to define a series of nested pit shells that can indicate maximum mining limits and potential high value starter pit areas. These limits are based upon maximizing gross operating surplus for a given set of input parameters. Application of cut-off grades for determination of waste and ore for ROM processing is complicated by the variability of recovery for the various lithology/metallurgical ore types. Calculated cut-off grades for recovery of processing and general & administration costs vary from 0.0025 to 0.0040 oz/ton (0.086 to 0.137 g/t) gold. Typical accuracy of mine lab assaying and low ore value led Equinox management to take a decision to apply a higher cut-off grade of 0.005 oz/ton (0.17 g/t) gold and this has been used for all reporting and scheduling of ore in the mine plan.

The cut-off grades for milling have been established using a two-step process. Net of processing ore values for ROM and milling were estimated for each block in the resource model using recoverable grades, revenue assumptions and onsite costs exclusive of mining. The highest value was used to assign a processing destination. A preliminary production schedule was then developed for ROM at 50,000 ton/d (45,400 t/d) with a 3,500 ton/d (3,200 t/d) target for milling. In some years there was a surplus of mill feed and this was stockpiled. When there was a shortage of mill feed in a given year, stockpiled material was recovered and processed. If there was a shortage after stockpile recovery then higher value ROM material was re-directed to mill feed until the design capacity was achieved.

16.4.3 Pit Sequencing

The pit development phases are shown as three-dimensional solids in the perspective view Figure 16-10. The pit solids have been intersected with a precedence order reflecting the clockwise sequence of phase development and expansion. A typical bench plan with the development phase limits and the block model NSR values are shown in Figure 16-11.

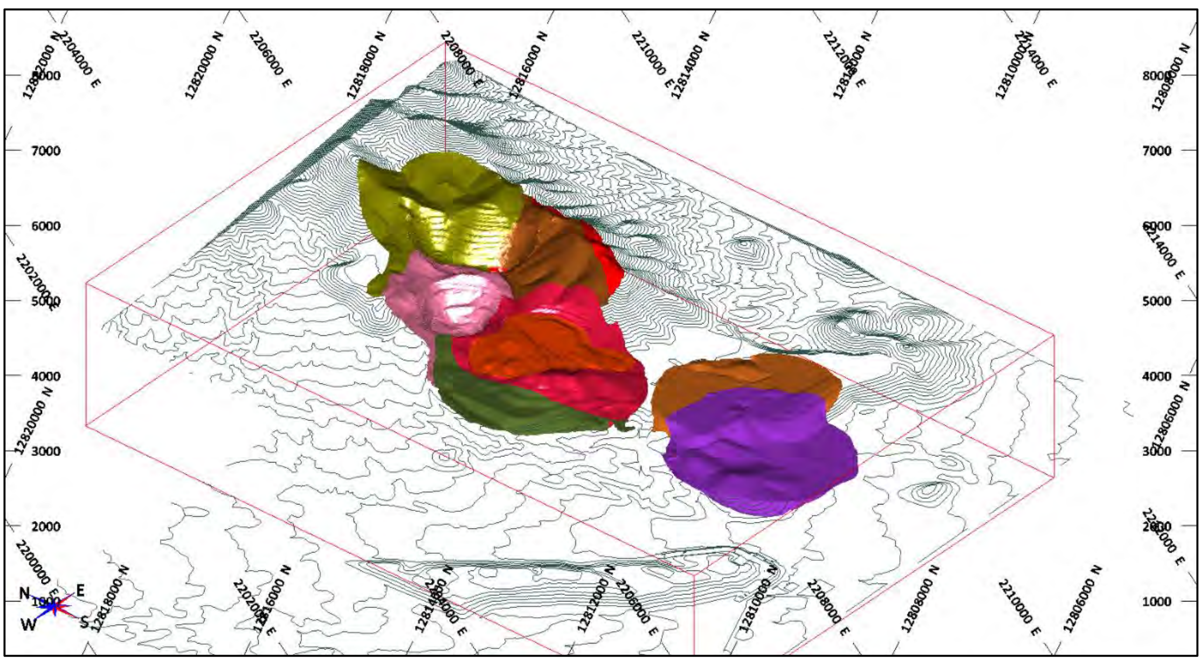


Figure 16-10: Pit Phase Solids Perspective View

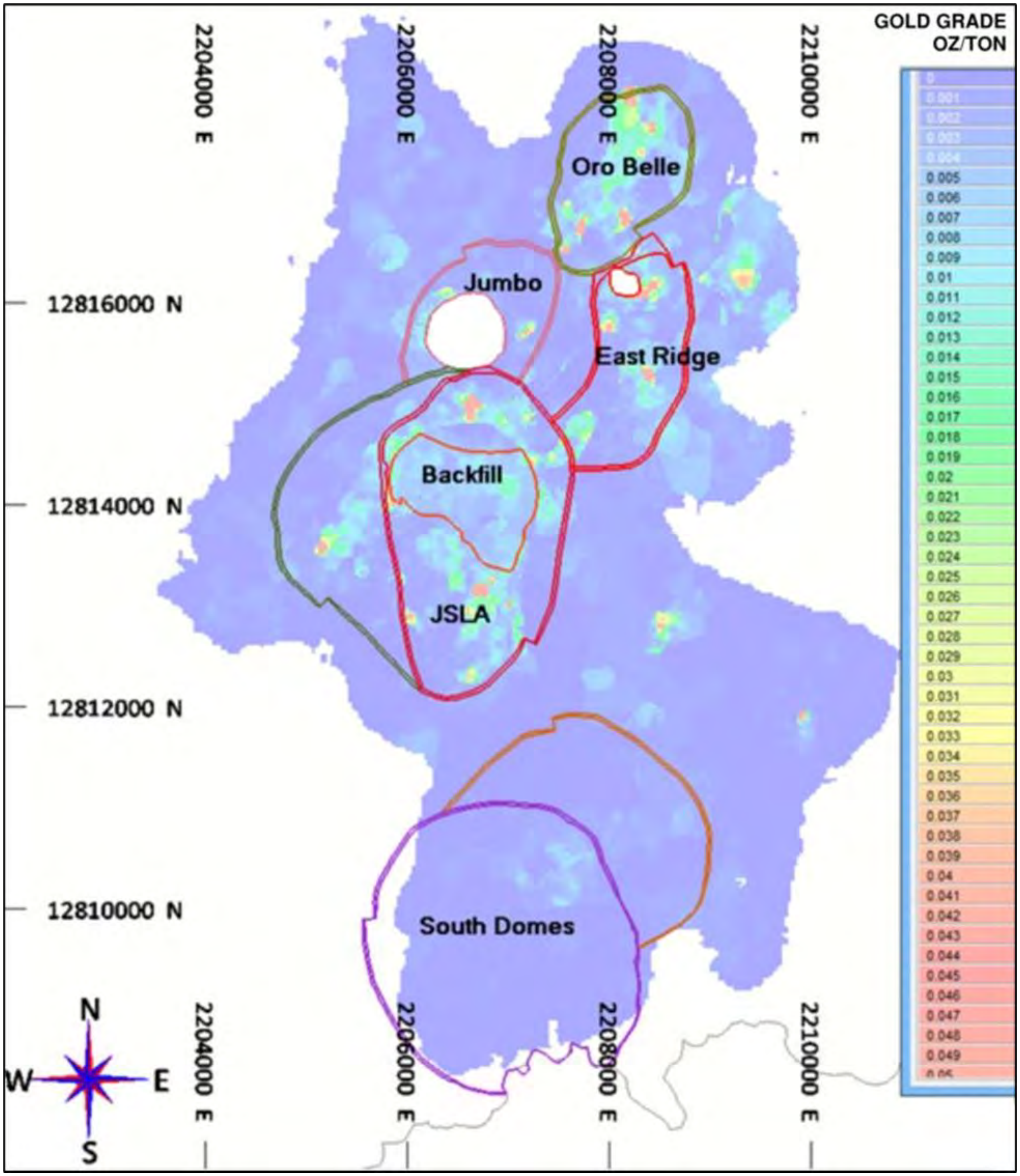


Figure 16-11: Bench Plan 4180 Phases and Gold Grade

16.4.4 Production Schedule Summary

The mine production schedule is summarized in Table 16-2. Figure 16-12 to Figure 16-16 present additional detail of material sources and destinations over time in the production schedule.

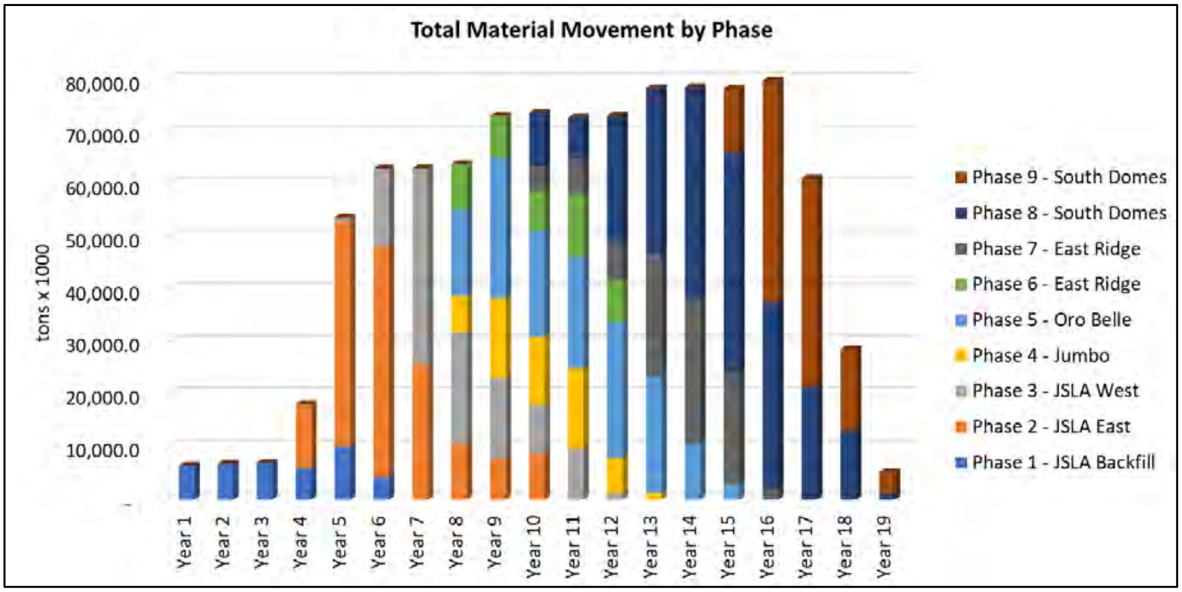


Figure 16-12: Material Movement by Phase

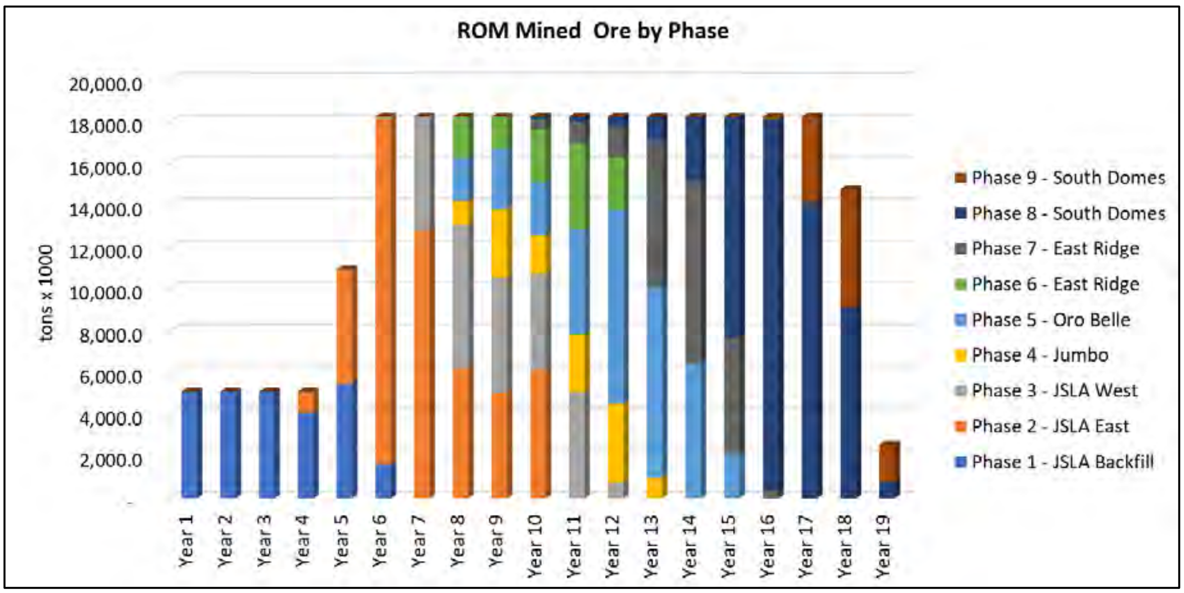


Figure 16-13: ROM Ore by Phase

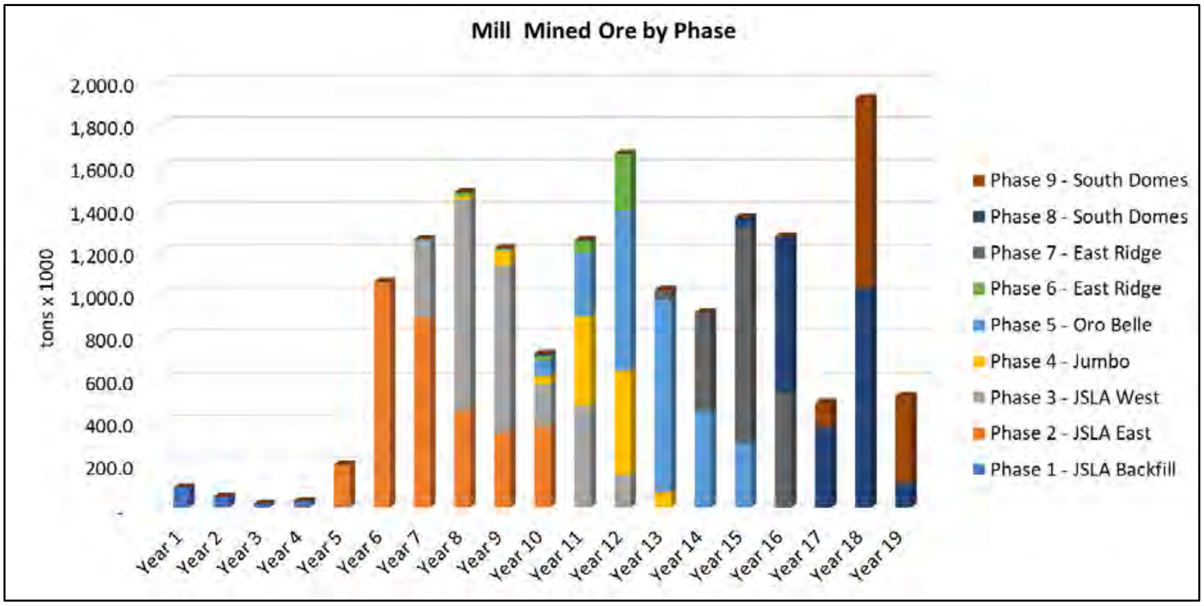


Figure 16-14: Mill Ore by Phase

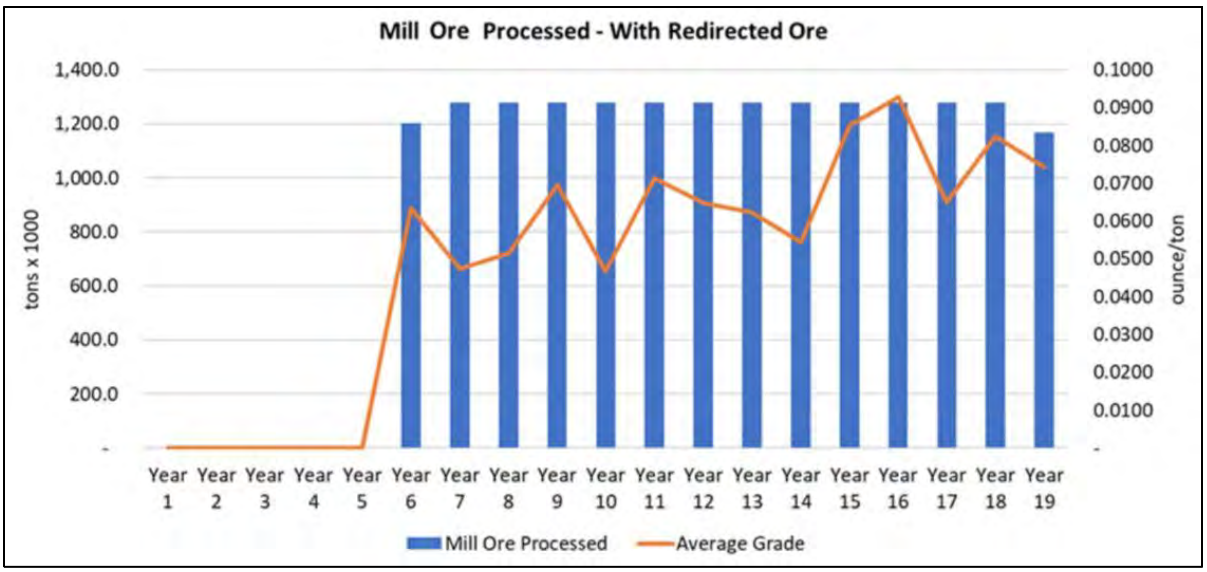


Figure 16-15: Mill Ore Processed

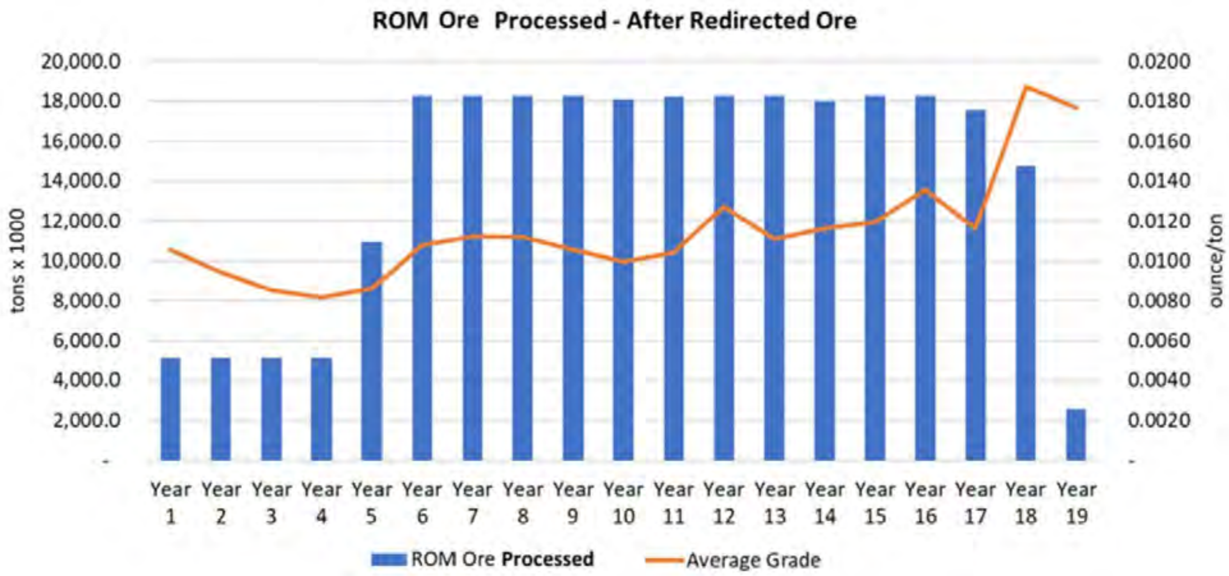


Figure 16-16: ROM Ore Processed

16.4.5 Mine Development Pre-Production and Year 4

The Phase 1 mine plan continues with contractor mining of JSLA open pit backfill material that is economic to process by ROM leaching. It is proposed to initiate hard rock mining of the JSLA East Pit with contractors in month 37 of the plan. Open pit development to mid Year 4 is shown in Figure 16-17.

Mining in the JSLA Backfill pit will take place to 4260 bench. Active benches in the JSLA East pit include 4760 to 4540 mined by contractors and 4520 bench through 4440 bench with the owners fleet. Total material moved in Year 4 totals 18.3 Mton, including 5.1 Mton ROM ore. Open pit development to the end of Year 4 is shown in Figure 16-18.

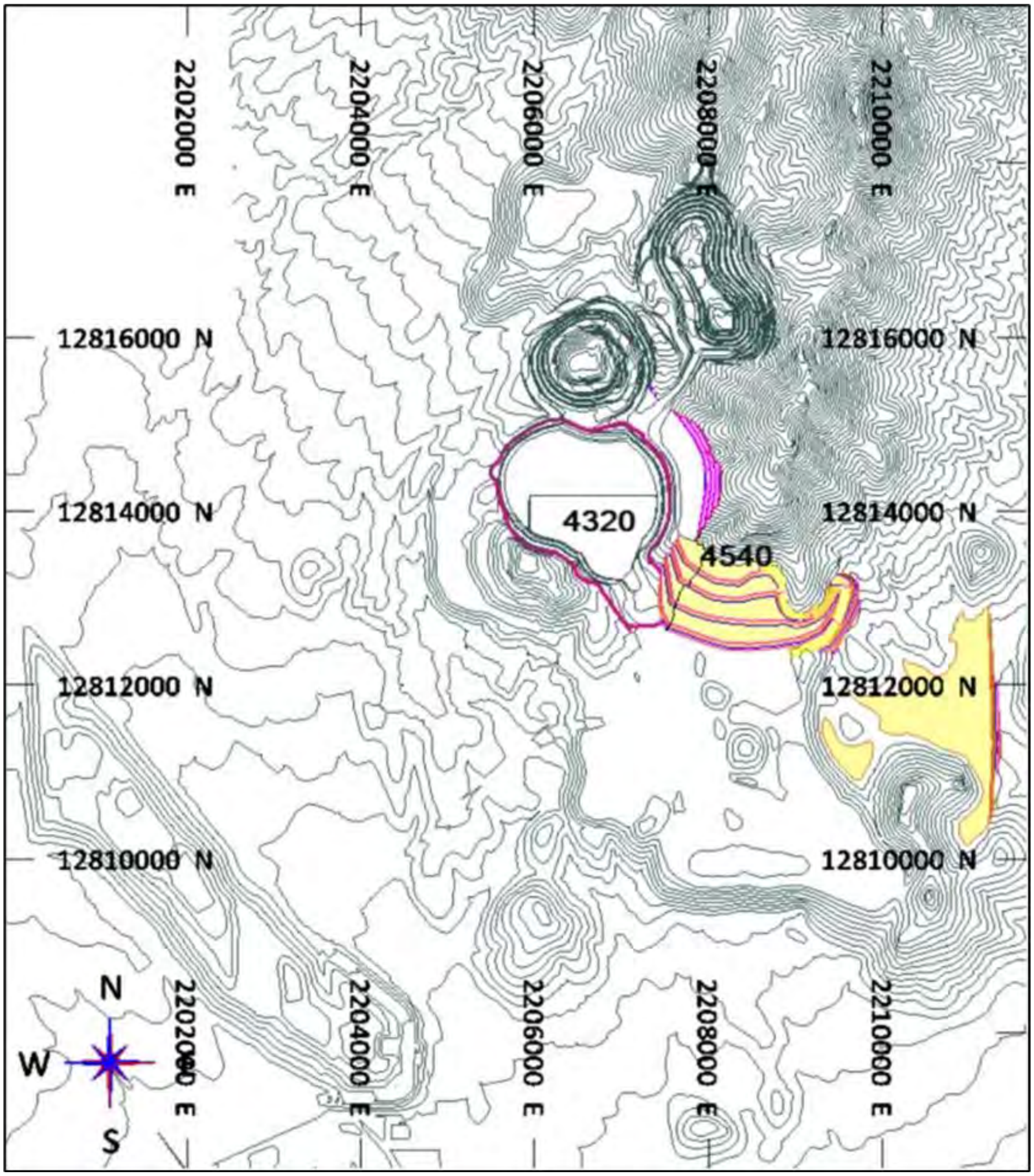


Figure 16-17: Mine Development Mid-Year 4 - Contractor

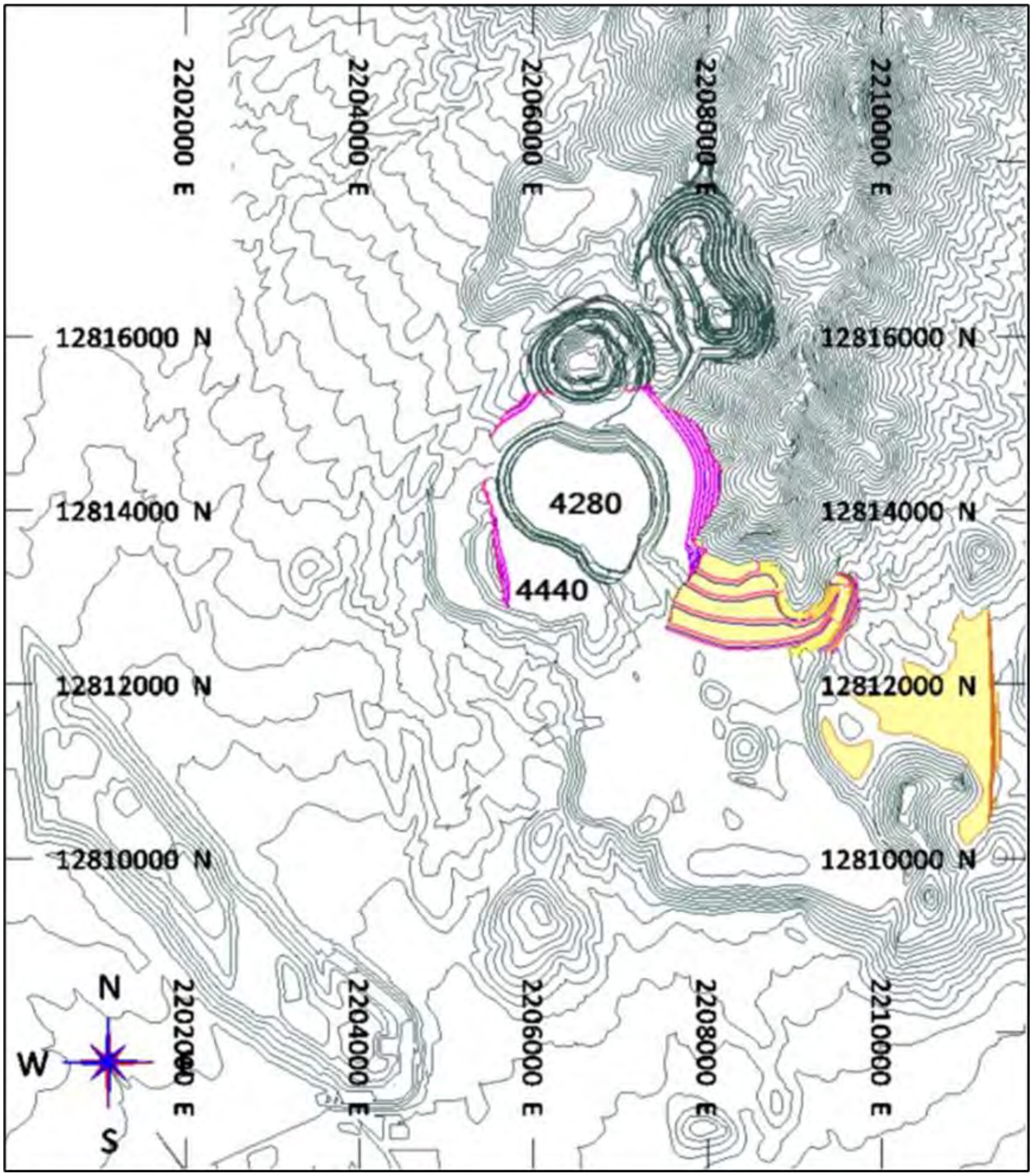


Figure 16-18: Mine Development – End of Period Year 4

16.4.6 Mine Development Year 5

Mining continues in the JSLA Backfill Pit which is completed to bench 4140 during Year 5 (Figure 16-19). Overall production for Year 5 is 53.8 Mton. A total of 10.95 Mton ROM leach ore is placed on the leach pad.

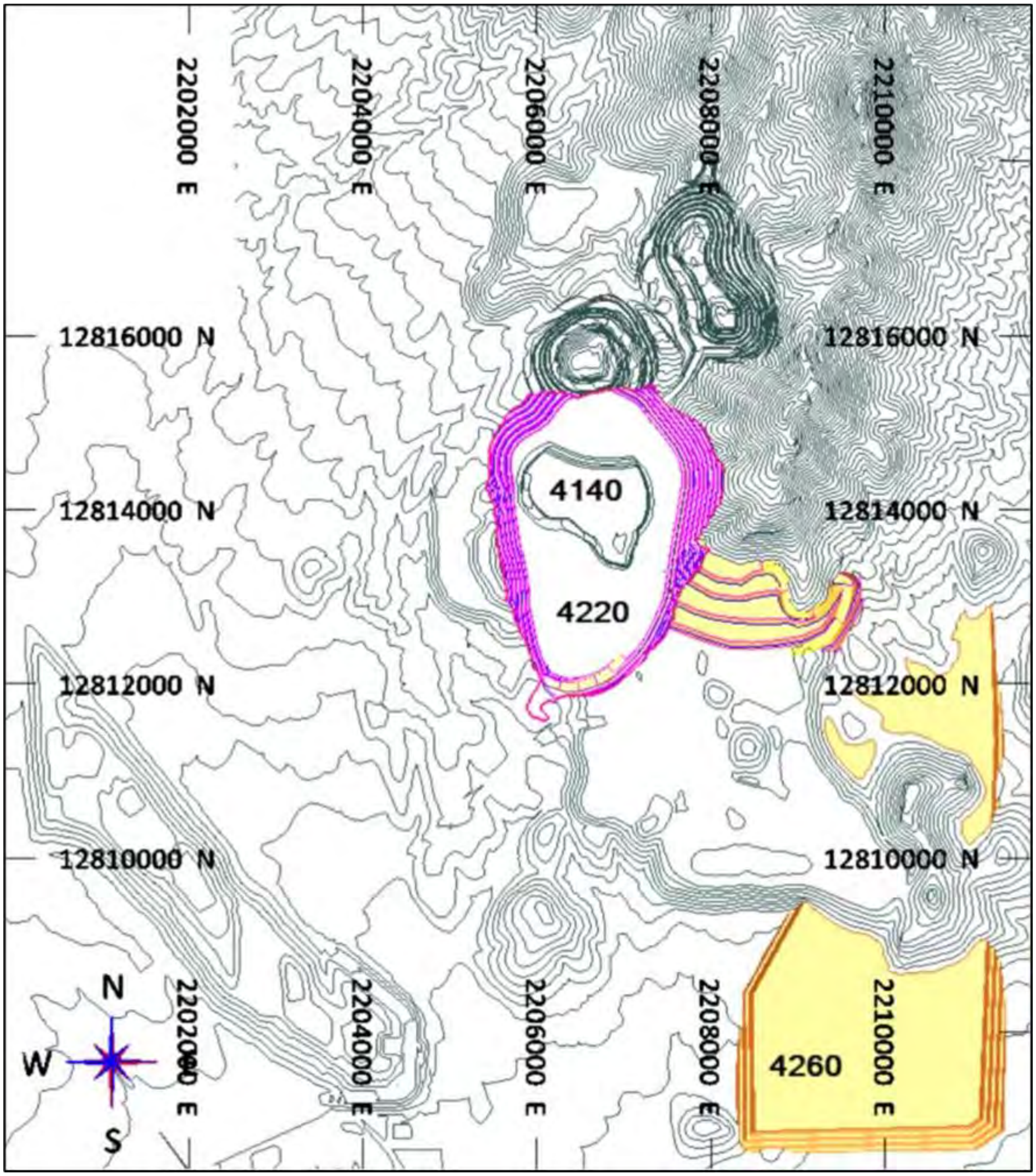


Figure 16-19: Mine Development Year 5

16.4.7 Mine Development Year 6 to Year 8

During Years 6 to 8, JSLA East Pit is mined to 3860 bench (Figure 16-20). In Year, 6 full production is reached for ROM ore production at 50,000 ton/d. JSLA West Pit developed to 3980 bench. Jumbo Pit pioneering commences in Year 8 and by Year 8 has reached 4520 bench. Oro Belle and East Ridge Pits are pioneered as well in Year 8 to the 4800 Bench. Total annual mine production averages 63.4 Mton during these years.

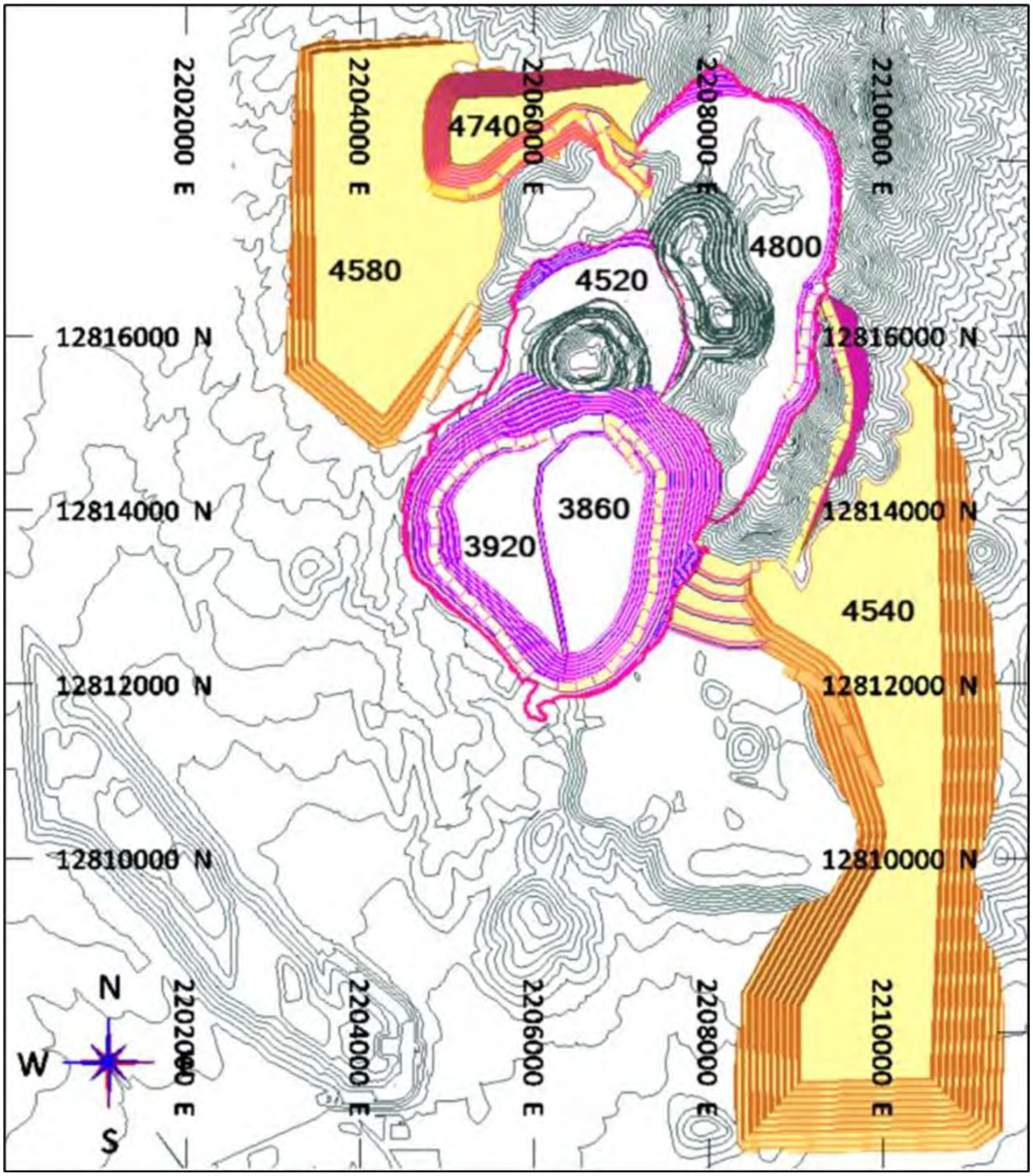


Figure 16-20: Mine Development – Year 8

16.4.8 Mine Development Year 9 to Year 12

The JSLA Pit is completed in Year 12 (Figure 16-21). Oro Belle Pit reaches 4260 bench in Year 12. East Ridge Phase 6 Pit is completed in Year 12 and East Ridge Phase 7 Pit is down to 4360. The South Domes Phase 1 Pit is pioneered in Year 10 and 11 and by Year 12 is down to 4180 bench. The average annual mining rate during this time is 73.2 Mton. The Northwest and Southeast dumps are filled to capacity during this time interval and backfilling commences in JSLA Pit.

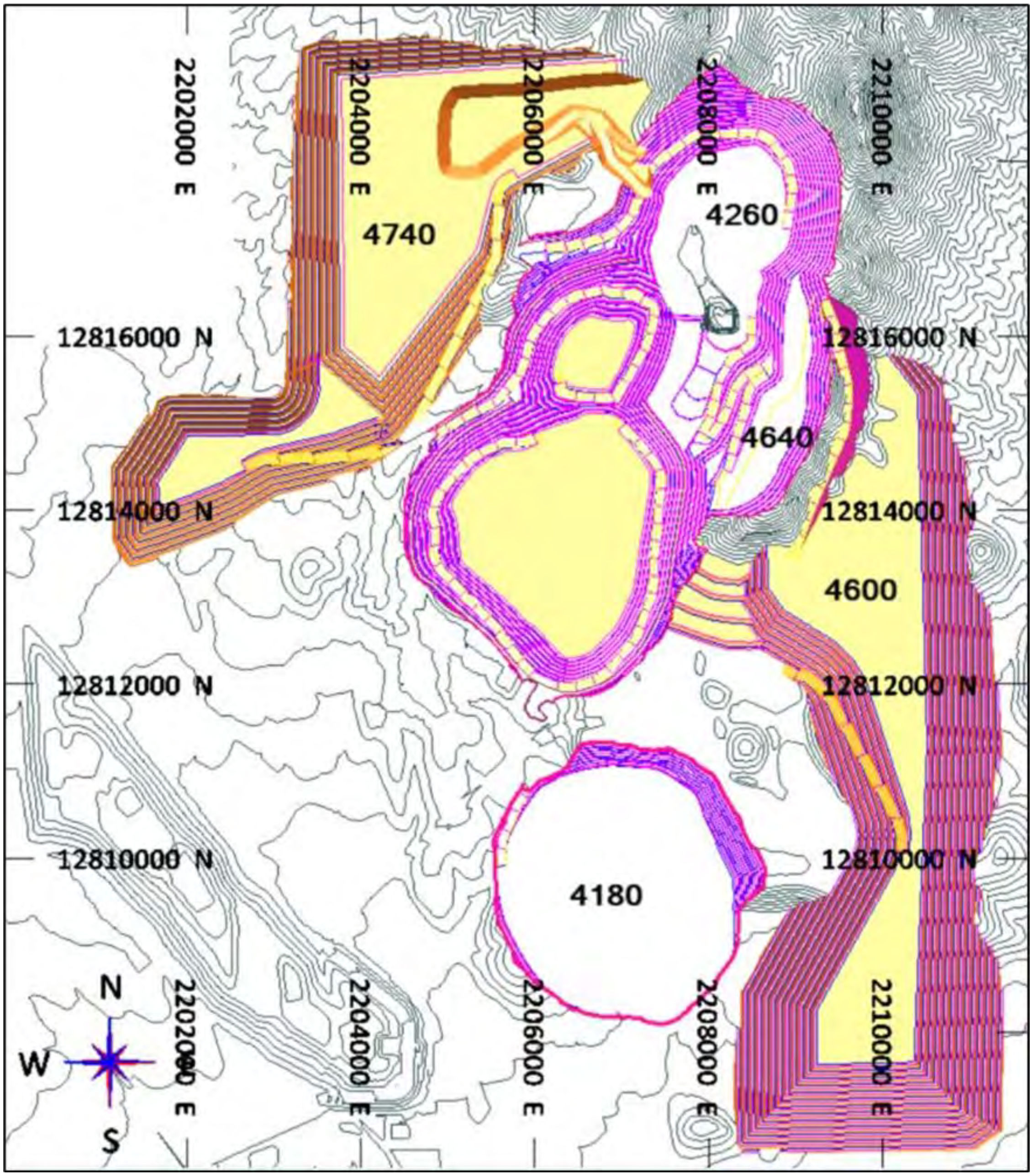


Figure 16-21: Mine Development Year 12

16.4.9 Mine Development Year 13 to Year 16

The Oro Belle Pit is completed in Year 15 and East Ridge Phase 7 Pit is completed in Year 16 (Figure 16-22). Pioneering of South Domes Phase 2 commences in Year 15 and by the end of the year Phase 1 is down to 4060 and Phase 2 is down to 3620. Waste backfilling continues in the JSLA and Jumbo Pits.

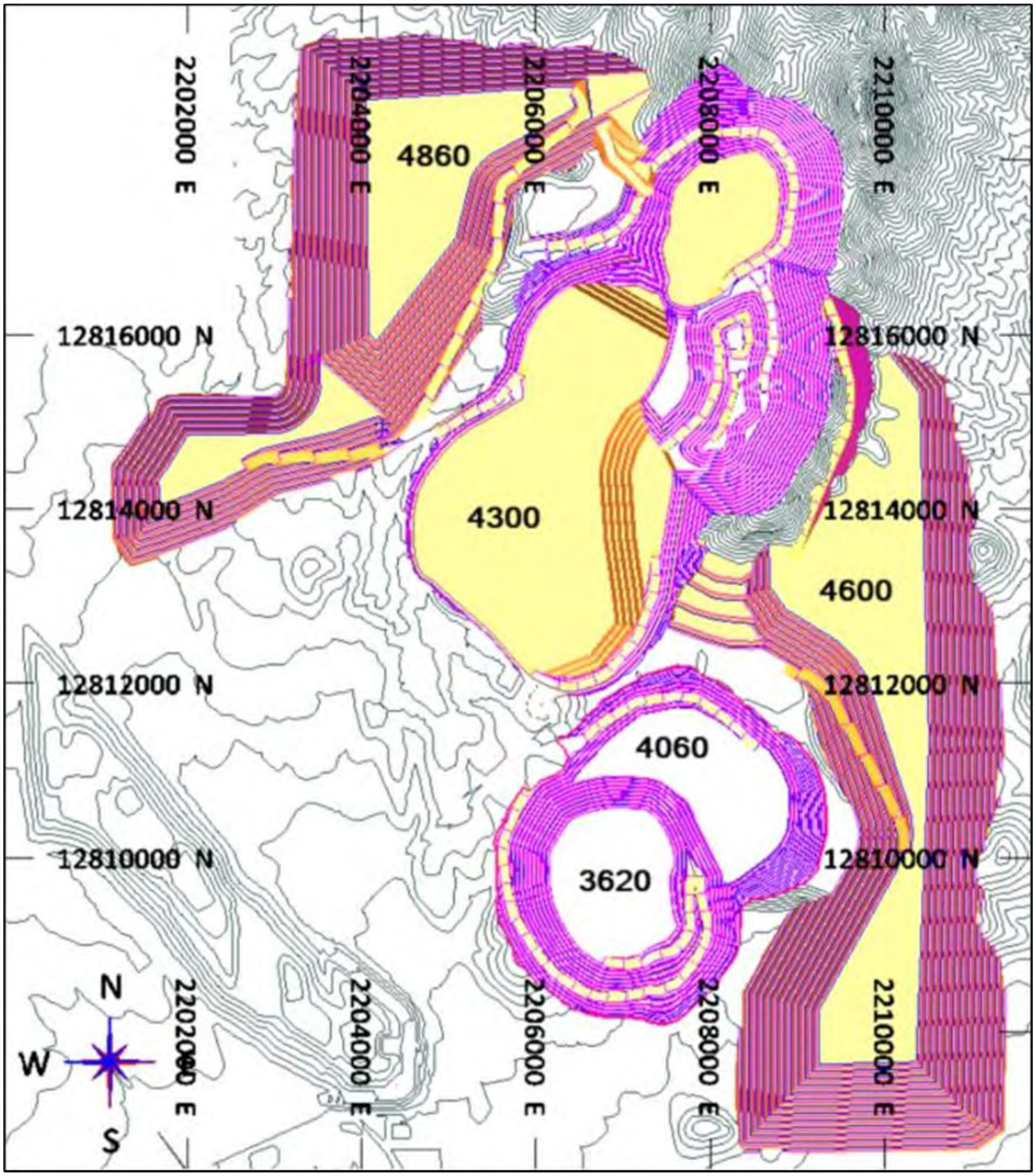


Figure 16-22: Mine Development Year 16

16.4.10 Mine Development Year 17 to Year 19

South Domes Phase 1 is completed during the first half of Year 19 and South Domes Phase 2 is completed later in the year. Overall mining rates may become limited by sinking rate limits of 300 ft/y in Year 18 and Year 19 (Figure 16-23).

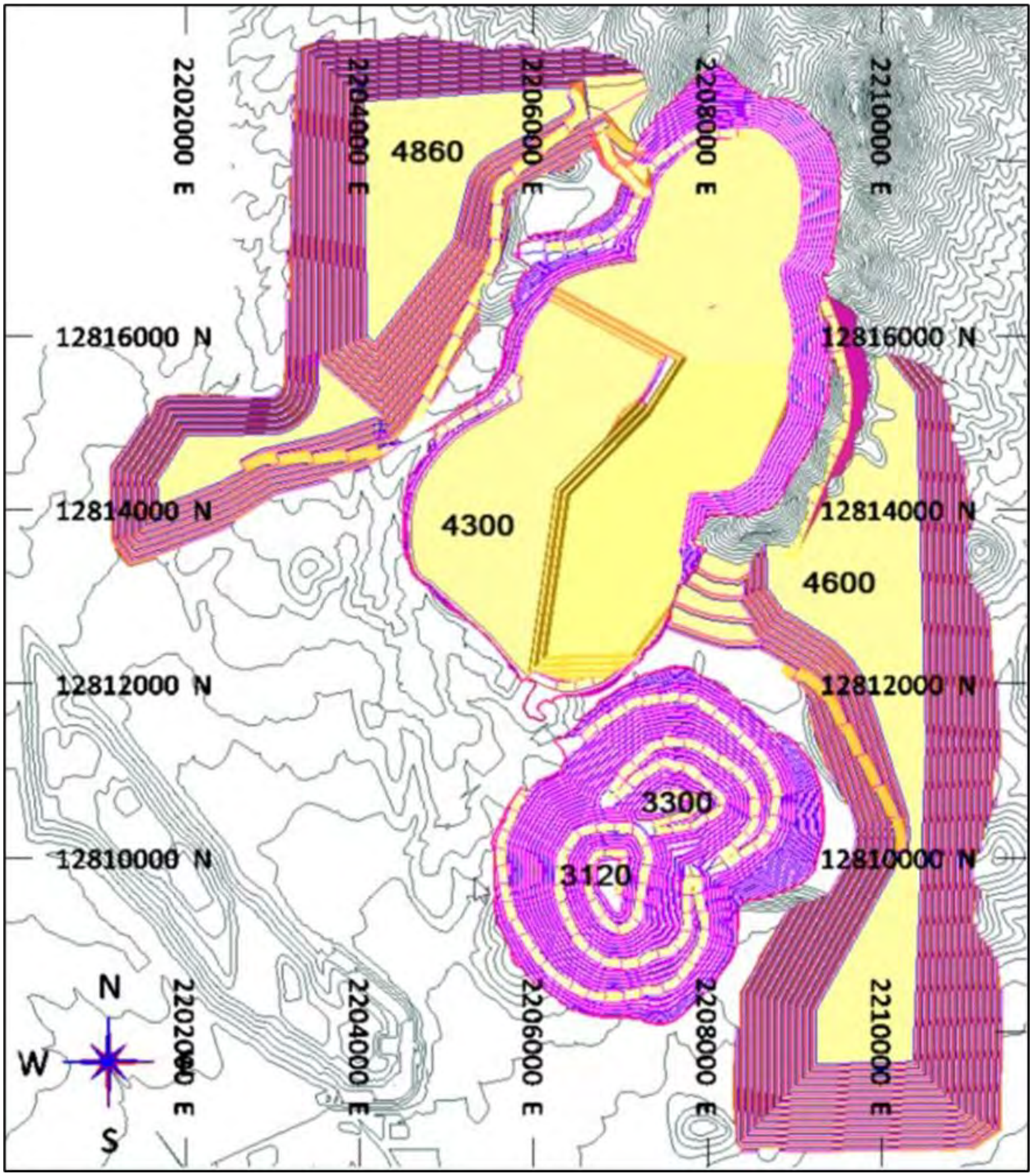


Figure 16-23: Mine Development Year 18

16.4.11 Engineering and Grade Control

The mine engineering group will be responsible for short-, medium- and long-range planning as well as day to day grade control functions and maintenance and monitoring of the dispatch system which will control the movement of trucks, shovels and drills.

Geotechnical engineers will monitor slopes and collect information on structure, material characteristics, hydrology and waste characterization on an ongoing basis as the mine is developed in order to improve the mine design criteria and ensure operational efficiency. They will also be involved in optimizing the blasting procedures to minimize wall damage while maximizing fragmentation.

Geologists and grade control technicians will be responsible for blasthole sampling, assaying and grade control. Grade control will be focussed on gold grade for the purposes of identifying ore and waste boundaries and separation between ROM and mill feed. Lithological characterization will also be important for metallurgical recovery predictions and ongoing column test-work.

Planning engineers will update mine plans at the short-, intermediate- and long-range scale as required. Stockpiling and milling cut-off grades will be adjusted over time in response to changing economic conditions and updated databases. Engineers will collect and evaluate mine operations equipment productivity and cost data to optimize ongoing operations.

Modern mine planning and geostatistical packages will be required for use in the engineering department to integrate the information coming from ongoing exploration, long range models, short range production plans and operational production data.

16.5 MINE EQUIPMENT

16.5.1 Summary

Mine equipment has been selected given the following considerations:

- The topographical challenges of the site including high vertical relief and steep slopes.
- The simultaneous distribution of multiple operating faces at several locations determined by the long-range plan.
- The necessity to minimize unit operating costs by using large scale mining equipment.
- Use of well proven equipment technology and coordination of operating machines using advanced systems.
- Use of equipment assembled with modular components in order to minimize onsite maintenance allowing maintenance personnel to focus on servicing and component replacement.

The mine will operate a primary fleet of 44.5 yd^3 diesel hydraulic shovels, 8 7/8” rotary drills and 250 ton end dump trucks.

The annual equipment requirements for the mine by year are summarized in Table 16-3.

Table 16-3: Production Fleet Requirement by Phase 2 Year

Year	Details	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Production Blasthole Drill	8 7/8"	2	2	2	3	3	3	3	3	3	4	4	3	3	3	2	2
Wall Control Drill	4 1/2' - 9"	1	2	2	2	3	3	3	3	3	3	2	2	2	2	1	
Hydraulic Shovel	2996 hp 44.5 yd^3	1	2	2	2	2	3	3	3	3	3	3	3	3	2	1	1
Wheel Loader	1739 hp 28 yd^3	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Haul Truck	2650 hp 250 ton	5	12	17	18	20	20	20	20	20	21	24	24	24	22	14	14
Track Dozer	600 hp	3	5	5	5	5	5	5	5	5	5	5	5	5	4	2	2
Wheel Dozer	620 hp	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Grader	290 hp 16 ft	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Water Truck	1450 hp 32,000 gal	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Wheel Loader	541 hp 10 yd^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck	825 hp 61 ton	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2
Excavator	524 hp 6 yd^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Year	Details	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Tire Manipulator	Large Tire	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Vibratory Compactor	130 hp 7.5 ft	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe	105 hp 1.3 yd^3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Articulated Truck	450 hp 40 ton	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Fuel and Lube Truck	100 ton 8,000 gal	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tractor and Low Bed	160 ton	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Flatbed Hiab Truck	10 ton	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Rough Terrain Forklift	33 ton	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shop Forklift	18 ton	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mechanics Truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Welding Truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Staff Pickup Trucks	1 ton Pickup	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10
Pit Services Pickup	1 ton Pickup	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Staff Pickup Trucks	1 ton Crewcab	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Pit Services Pickup	1 ton Crewcab	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Shovel Crew Flat Deck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shovel Crew Hiab	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Crew Hiab	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Tower	8 Kw	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Hydraulic Hammer	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Rescue Vehicle	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.5.2 Major Mine Equipment Operating Parameters

The mine will operate 24 hours per day 365 days per year. Shift employees will work 12-hour shifts on a 28-day cycle.

In general, it is expected that major equipment will have an effective operating time of 50 minutes per hour. Equipment is expected to have 85% to 90% availability initially, declining with age.

Detailed equipment productivity calculations have been made on an annual basis for drills, shovels, and trucks. Support equipment operating time has been factored on an annual basis according to material movement.

16.5.3 Drilling & Blasting

16.5.3.1 Drilling

The primary blasthole drills will be diesel powered machines capable of drilling 8 7/8" holes single pass for a 30 ft (9.1 m) bench height including subgrade. They will be used for production hole drilling in ore and waste and can be configured for buffer row drilling on wall control patterns if required. Specialized wall control drills will be provided for the majority of the buffer rows and pre-shear holes. These drills can also be used for drilling sub-horizontal drain holes for wall slope depressurization.

The fleet will initially consist of two production units and one wall control drill. Production drill additions will be made in Year 7 and 11, wall control drill additions will be made in Year 4 and Year 7 with the fleet peaking at seven drills. Wall control drill replacements have been scheduled for two units in Year 11 and Year 12. Blasthole drilling requirements have been estimated on an annual basis according to the production schedule and wall control drilling requirements for pre-shear and trim blasting.

16.5.3.2 Production Blasting

Ore and waste will be mined on a 30 ft bench. Ore will be drilled using a burden and spacing of 20.0 x 23.0 ft. Waste will be drilled on a 23.0 ft x 26.5 ft pattern. Subgrade drilling will be 3.0 to 4.0 ft to allow even breakage to the design bench elevation. Blasthole cuttings will be sampled and assayed for grade control.

The wall control blasting will consist of two rows of 6 ½ in trim holes with a burden and spacing of 16.0 x 18.4 ft. These will be drilled with the wall control or production drills depending on equipment distribution through the pit development phases. The sub-grade drilling depth will be reduced in areas of final berm locations. Pre-shear holes, 4 ½ inches in diameter will be drilled at a 5 ft spacing on final walls.

Blasting will be carried out with a combination of ammonium nitrate and emulsion explosives. An overall blend of 75% ammonium nitrate/fuel oil (AN/FO) and 25% emulsion has been assumed. The overall production blasting agent consumption is expected to be 0.36 lbs/ton of material.

Blastholes will be single primed and initiated using non-electric methods. An explosive supply contractor will deliver bulk explosives to the borehole. The mine Drill & Blast Engineer will supervise the contractor.

16.5.3.3 Explosives Storage

The contractor will provide and deliver bulk explosives to the borehole where primers will be installed and tied in, crushed rock stemming will be placed in the hole collars and the contractor will initiate the blast. Explosives storage on site will consist of a magazine for packaged explosives and primers and a cap magazine.

16.5.4 Loading

At peak production, the loading fleet will consist of three 44.5 yd^3 diesel hydraulic shovels and two 32 yd^3 wheel loaders. The wheel loaders and two of the shovels will be required in Year 4 as deliveries permit, the contractor is phased out in the JSLA Backfill pit and pioneering development is completed on the JSLA East pit is completed. A third shovel will be required in Year 9 as the overall production rate increases from 75 to 81.0 Mton/y. Typical material distribution weighting between loading units will vary between 65% - 70% to shovels and the balance to the wheel loaders.

The loading equipment will operate two 12-hour shifts per day. Operating efficiencies of 83% are anticipated for the loading fleet. Annual average equipment availability is expected to be 90% when operations begin in Year 4 declining to 80% as equipment ages.

The productivity calculations assume good digging conditions, four to five pass loading of the trucks with an overall cycle time of 3.70 minutes for shovels and 5.2 minutes for wheel loaders.

16.5.5 Haulage

The haulage trucks proposed for Castle Mountain are 250 ton capacity rigid frame end dump trucks. The initial fleet requirement in Year 4 is 10 trucks. Additional trucks are added to the fleet over time until the fleet size peaks in Year 13 at 24 trucks. The truck fleet size and expected annual operating hours are shown in Figure 16-24.

The haulage trucks will operate two 12-hour shifts per day. Operating efficiencies of 83% are anticipated for the truck fleet. Equipment availability is expected to be over 90% when operations start declining as equipment ages. Truck fleet utilization of availability will typically be over 95%.

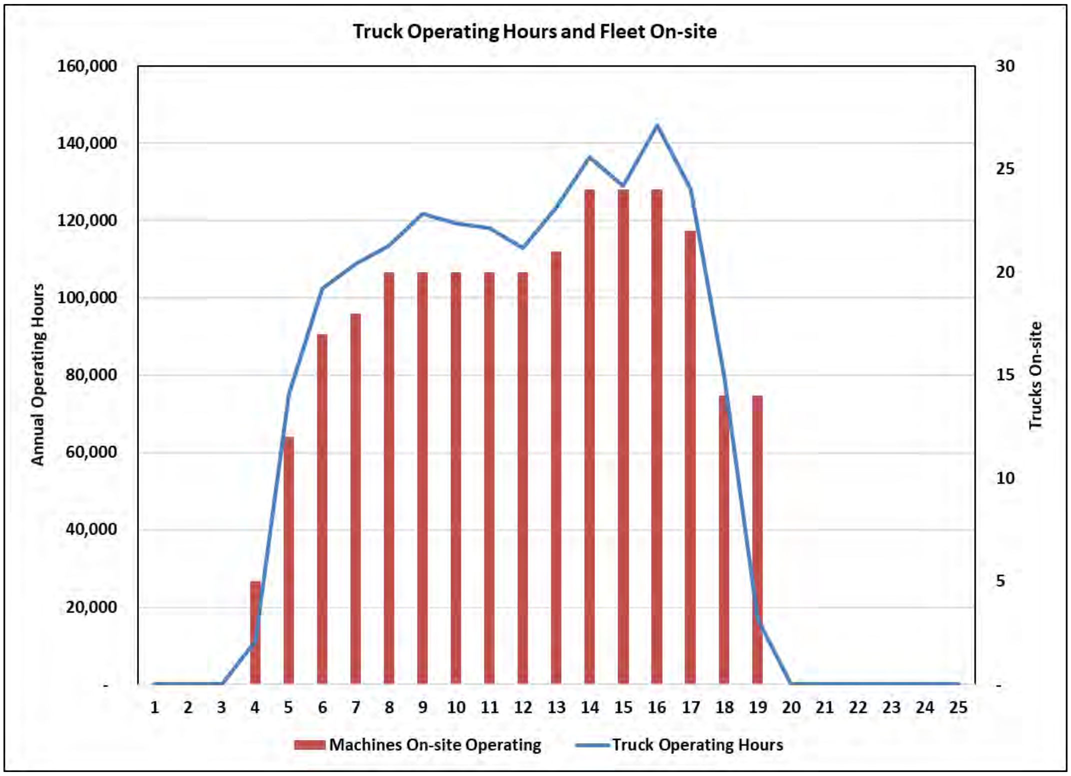


Figure 16-24: Truck Operating Hours and Fleet On-site

The cycle times for ore and waste were estimated for each bench and material destination for each year of production and were based on haul profile distances and road grades. These were then used to estimate haul truck productivities and overall fleet requirements.

A total of 12 haulage trucks will be delivered by the end of Year 4 with additions made during most years to Year 14 when the fleet peaks at 24 units.

16.5.6 Mine Support Equipment

The mining support equipment includes five track dozers, two-wheel dozers, three graders, two water trucks. Miscellaneous ancillary equipment is also required to service, maintain the major equipment and support ongoing open pit operations.

Track dozers will operate on ROM heap leach pad dozing and ripping, on active benches pushing back break and performing heavy dozer operations around operating shovels. In the open pit they will also build roads, prepare sinking cut faces, clean berms, scale walls and rip hard toes. On waste dumps the track dozers will maintain positive grades on the bench surfaces near the crest and provide safe berms for truck dumping.

Road graders and rubber tire dozers will maintain road, dump and bench surfaces to provide level running surfaces. Water trucks will be used in the road maintenance program to provide dust control and safe winter running conditions.

A complement of ancillary equipment as listed in Table 16-3 will also be available to perform service functions including fuelling, provide work area lighting, excavation capability for wall scaling, clean-up and ditching etc. as required to ensure a safe self sufficient mine operation.

Pick-up trucks and crew-cabs will be required for transportation of supervisors, technical staff and maintenance personnel.

Explosives will be delivered to the blasthole. The contractor will provide support equipment to pump wet holes, deliver blasting accessories and stem holes. The bulk delivery truck and storage facilities will be provided by the explosives contractor.

16.5.7 Manpower

The supervision and technical positions in the mine are summarized in Table 16-4.

Table 16-4: Mine Salaried Positions

Mine Supervision	
Mine Manager	1
Mine Superintendent	1
Mine Foreman	4
Drill & Blast Foreman	2
Mine Training Coordinator	2
Mine Shifter & Dispatcher	4
Mine Clerk	1
Subtotal	**15**
Mine Maintenance	
Maintenance Superintendent	1
Electrical Foreman	1
Maintenance Foreman	4
Maintenance Planner	2
Maintenance Clerk	1
Subtotal	**9**
Engineering & Geology	
Chief Mine Engineer	1
Senior Mine Engineer	1
Drill & Blast Engineer	1
Geotechnical Engineer	1
Senior Surveyor	2
Surveying Technician	2
Senior Mine Geologist	1
Mine Geologist	2
Environmental Coordinator	1
Grade Control Technical	4
Subtotal	**16**
Total	**40**

The mine will work 12-hour shifts with four crews. Maintenance will be undertaken with a combination of employees and equipment supplier technicians as required for specialized service functions. Hourly employees are summarized in Table 16-5.

Table 16-5: Mine Staff and Hourly Employees

Project Year	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14
Salaried Employees																
Mine Supervision	15	15	14	14	14	14	14	14	14	14	14	14	14	14	14	14
Mine Maintenance	8	9	9	9	9	9	9	9	9	9	9	9	9	9	4	4
Engineering & Geology	13	16	16	16	16	16	16	16	16	16	16	16	16	16	9	9
Subtotal	**36**	**40**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**39**	**27**	**27**
Hourly Employees																
Operations	58	119	138	143	148	158	156	156	150	156	167	157	166	148	93	81
Maintenance	39	65	65	65	71	76	76	76	76	78	78	78	77	60	42	20
Subtotal	**97**	**184**	**203**	**208**	**219**	**234**	**232**	**232**	**226**	**234**	**245**	**235**	**243**	**208**	**135**	**101**
Total	**133**	**224**	**242**	**247**	**258**	**273**	**271**	**271**	**265**	**273**	**284**	**274**	**282**	**247**	**162**	**128**

16.6 MINE ANCILLARY FACILITIES

16.6.1 Service Complex

A mine service complex will be located with service bays, warehouse, wash bays and fuel storage facilities.

16.6.2 Mine Electrical Power

Mine electric power distribution will be required for dewatering wells and limited lighting.

16.6.3 Mine Dispatch

A mine dispatch system will be installed to maximize equipment utilization productivity. The dispatch base station and central control will be located in the service complex. Units will be mounted on drills, shovels, loaders and trucks. The conceptual configuration would include GPS ground reference stations and repeaters. The system would interface with the mine planning system and survey system on drills.

16.6.4 Mine Water Management

Water infiltration into the pits is expected once the mine expands below the water table. Water will be removed by a mobile centrifugal dewatering pump and will be used as general make-up water at site offsetting the amount of raw water pumped from the wells.

16.6.5 Fuel Storage

The estimated fuel requirement, in Phase 2, for the mine equipment is shown in Figure 16-25. The annual quantity requirement varies from approximately 5.5 million gallons/ year in Year 5 to 9.7 million gal/year in Year 16 of the mine plan. Additional fuel is also required for preparation of AN/FO for blasting when conditions are dry. Approximately 6% fuel oil by weight is added to the ammonium nitrate prills. Fuel storage will be in two 20,000 gal tanks providing approximately two days storage. Proximity to Las Vegas allows for frequent delivery.

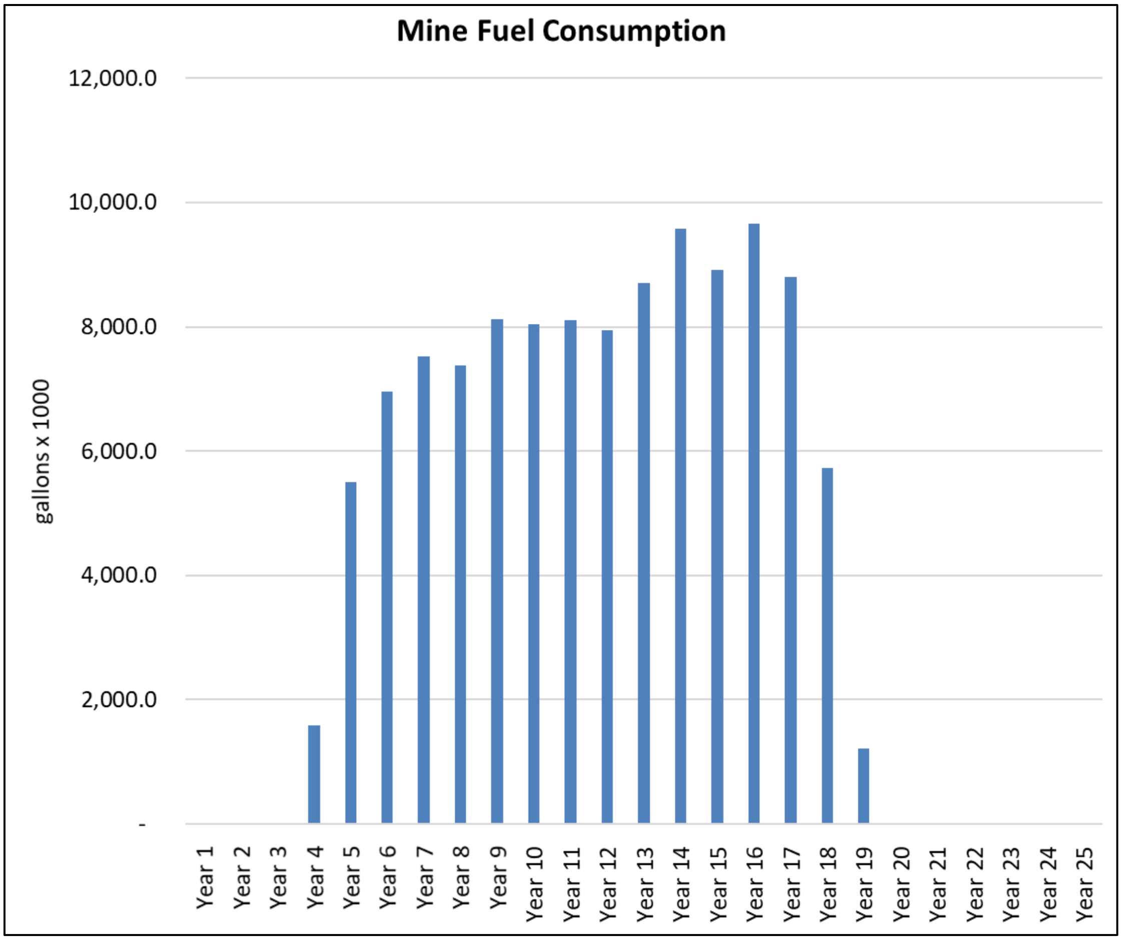


Figure 16-25: Fuel Consumption

16.7 MINE OPERATING COST ESTIMATE

16.7.1 Summary

The forecast operating costs for the open pit are divided into the following areas:

- General Mine Expense
- Drilling
- Blasting
- Loading
- Hauling
- Support Equipment – Roads & Dumps
- Contract Services

Operating costs are further subdivided within these cost centers as follows:

- Salaries & Wages
- Fuel & Power
- Consumables & Maintenance Parts and Outside Services

The Phase 1 and Phase 2 mine plan operating costs over the LOM are summarized in Table 16-6. Average unit costs were estimated to be $1.78/ton. The annual mine operating cost for Phase 2 starting in Year 6 is provided in Table 21-18.

Table 16-6: Summary of LOM Unit Operating Costs

Category	Units	Cost	Percentage
General Mine Expense	$/ton	$0.10	5.7%
Drilling	$/ton	$0.12	6.8%
Blasting	$/ton	$0.16	9.0%
Loading	$/ton	$0.28	15.9%
Hauling	$/ton	$0.72	40.7%
Roads & Dumps	$/ton	$0.29	16.1%
Contract Services	$/ton	$0.10	5.8%
Total	**$/ton**	**$1.78**	**100.0%**
Total	**$/ton processed**	**$6.17**	

16.7.2 General

Salaries and wages used in the mine operating costs have been set by Equinox to reflect their current base costs, burdens and shift premiums in the area.

The mine will operate diesel powered equipment. Diesel price was set at $2.75/gal for ongoing operations. Major equipment lubrication consumption was based upon current costs experienced at a nearby Equinox operation and supplier provided consumption rates.

Equipment operating costs for major equipment was obtained from equipment suppliers including Caterpillar, Komatsu, Sandvik and Epiroc. Components rebuild and replacement schedules were used to calculate annual operating costs based upon cumulative machine hours.

General mine expense includes all salaries for mine operations & maintenance management and geology & engineering. Allowances have been made for software licenses for dispatch and mine planning systems, supplies, communications, training and outside consulting services in support of operations.

Drilling costs represent 7% of the overall mining cost and will average $0.12/ton over the life of mine.

Blasting costs represent 9% of the overall mining cost and average $0.16/ton. The blasting cost estimate has been based upon the scheduled material movement schedule, expected powder factors for ore and waste, drill patterns and explosives supply component cost estimates by W.A. Murphy Inc. Equinox will provide diesel fuel onsite as required for AN/FO.

Loading costs represent 16% of the overall mining costs and will average $0.28/ton over the life of mine. These costs reflect a combination of wheel loaders and hydraulic shovels.

Haulage costs are the largest cost center and represent 41% of the total mining cost at $0.72/ton. Haulage costs are variable with time and changes in pit phases, depth, and heap leach pad height.

Initial haulage costs are $0.47/ton trending higher as the open pit and heap leach pad increase in size to $1.24/ton by Year 18 of the overall mine plan.

Roads, dumps and support costs represent 16% of the total mining cost and average $0.29/ton. This cost center includes operation of all support equipment for road, open pit, waste dump and heap leach pad maintenance.

The contract services cost center averages $0.10/ton and includes clearing & grubbing of open pits and waste dump areas, some road development and supply of magnesium chloride for dust control.

16.8 MINE CAPITAL COSTS

16.8.1 Capital Cost Estimate Summary

16.8.1.1 Pre-production Development

The capital cost estimate for mine pre-stripping is based upon a detailed mine development plan for material movement. Open pit development at JSLA will be undertaken by contractors building access to the upper benches on the east side and pioneering to 4540 bench where the owner's fleet can be phased into production. The pioneering work is planned for Year 3 and into Year 4 while the owner's fleet is being ordered, delivered, erected and commissioned. Phase 1 contractor rates were used as base cost with adjustment for drilling and blasting. The cost for pre-stripping is estimated at $36.8 million.

Clearing and grubbing of the southeast dump and JSLA pit area and stockpiling of topsoil is estimated at $2.1 million.

16.8.1.2 Mobile Equipment

The mine equipment fleet capital cost estimate is summarized in Table 16-7. Delivery, erection, commissioning and sales taxes are included in the equipment costs shown in the table. Tires are included in the equipment prices.

Mine equipment costs have for the most part been estimated based upon new equipment. Some used equipment has been included such as water trucks, fuel truck, low bed, tire manipulator and other low intensity use machines. A moderate percentage of used track dozers and haulage trucks purchased initially would speed up availability of equipment on site in the first year and also space out the rebuild requirements as the equipment ages.

Life of mine capital costs for Phase 2 are shown in Table 16-8.

Table 16-7: Mine Equipment Capital (Ongoing Additions and Replacements)

Typical Unit		Equipment Requirement		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	TOTALS
			Unit Cost	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($ x 1000)	($x1000)
Production Blasthole Drill	8 7/8"	Epiroc Pit Viper 235	$2,369,850	$4,740	$0	$0	$2,370	$0	$0	$0	$0	$0	$2,370	$0	**$9,479**
Wall Control Drill	4 1/2' - 9"	Epiroc - SmartROC D65	$966,787	$967	$967	$0	$0	$967	$0	$0	$0	$967	$967	$0	**$4,834**
Hydraulic Shovel	2996 hp 44.5 yd^3	Caterpillar 6060 FS	$10,680,651	$10,681	$10,681	$0	$0	$0	$10,681	$0	$0	$0	$0	$0	**$32,042**
Wheel Loader	1739 hp 28 yd^3	Caterpillar 994K	$6,083,223	$6,083	$6,083	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$12,166**
Haul Truck	2650 hp 250 ton	Caterpillar 793F	$4,576,122	$16,123	$32,033	$22,881	$4,576	$9,152	$0	$0	$0	$0	$2,887	$8,660	**$96,312**
Track Dozer	600 hp	Caterpillar D10T2	$1,549,344	$3,772	$3,099	$0	$0	$0	$0	$0	$0	$0	$3,099	$3,099	**$13,068**
Wheel Dozer	620 hp	Caterpillar 844K	$1,920,038	$3,840	$0	$0	$0	$0	$0	$0	$0	$3,840	$0	$0	**$7,680**
Grader	290 hp 16 ft	Caterpillar 16MG	$1,074,810	$2,150	$1,075	$0	$0	$0	$0	$0	$0	$2,150	$1,075	$0	**$6,449**
Water Truck	1450 hp 32,000 gal	Caterpillar 785G WTR MEGA	$1,700,996	$1,701	$1,701	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$3,402**
Wheel Loader	541 hp 10 yd^3	Caterpillar 988KXE	$946,602	$947	$0	$0	$0	$0	$0	$0	$0	$0	$0	$947	**$1,893**
Haul Truck	825 hp 61 ton	Caterpillar 773G	$944,054	$2,832	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$2,832**
Excavator	524 hp 6 yd^3	Caterpillar 390D L	$1,073,065	$1,073	$0	$0	$0	$0	$0	$0	$0	$0	$1,073	$0	**$2,146**
Tire Manipulator	Large Tire	Caterpillar 988K CWS TM30P	$763,996	$764	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$764**
Vibratory Compactor	130 hp 7.5 ft	Caterpillar CS64	$148,973	$149	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$149**
Backhoe	105 hp 1.3 yd^3	Caterpillar 440	$152,055	$152	$0	$0	$0	$0	$0	$0	$0	$0	$0	$152	**$304**
Articulated Truck	450 hp 40 ton	Caterpillar 740GC	$623,833	$624	$0	$0	$0	$0	$0	$0	$0	$0	$0	$624	**$1,248**
Fuel and Lube Truck	100 ton 8,000 gal	Caterpillar 777 Mega System	$1,230,179	$1,230	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$1,230**
Tractor and Low Bed	160 ton		$1,078,098	$1,078	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$1,078**
Flatbed Hiab Truck	10 ton		$183,175	$183	$0	$0	$0	$0	$183	$0	$0	$0	$0	$183	**$550**
Rough Terrain Crane	80 tonne	Linkbelt Rough Terrain RTC 8090	$944,968	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Rough Terrain Forklift	30 tonne	Hyster 700	$808,125	$808	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$808**
Shop Forklift	16 tonne	Taylor-360M Forklift	$377,125	$377	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$377**
Mechanics Truck		8100 National	$334,025	$334	$0	$0	$0	$0	$0	$0	$334	$0	$0	$0	**$668**
Welding Truck			$80,813	$81	$0	$0	$0	$0	$0	$0	$81	$0	$0	$0	**$162**
Staff Pickup Trucks	1 ton Pickup		$53,875	$539	$0	$0	$0	$0	$539	$0	$0	$0	$0	$539	**$1,616**
Pit Services Pickup	1 ton Pickup		$53,875	$323	$0	$0	$0	$0	$323	$0	$0	$0	$0	$323	**$970**
Staff Pickup Trucks	1 ton Crewcab		$64,650	$194	$0	$0	$194	$0	$0	$194	$0	$0	$194	$0	**$776**
Pit Services Pickup	1 ton Crewcab		$64,650	$194	$0	$0	$194	$0	$0	$194	$0	$0	$194	$0	**$776**
Pit Services Bus	10 Passenger		$43,100	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Shovel Crew Flat Deck			$161,625	$162	$0	$0	$0	$0	$162	$0	$0	$0	$0	$162	**$485**
Shovel Crew Hiab			$242,438	$242	$0	$0	$0	$0	$242	$0	$0	$0	$0	$242	**$727**
Surface Crew Hiab			$242,438	$242	$0	$0	$0	$0	$242	$0	$0	$0	$0	$242	**$727**
Surface Crew Stinger			$269,375	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$0**
Lighting Tower	8 kW	Amida	$26,507	$159	$0	$0	$0	$159	$0	$0	$0	$159	$0	$0	**$477**
Hydraulic Hammer		TB-XC Hydraulic Breaker	$199,338	$199	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$199**
Mine Rescue Vehicle			$80,813	$81	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	**$81**
Engineering Hardware & Software		Mine Planning & Geology	Variable	$797	$75	$75	$75	$75	$75	$75	$75	$75	$75	$75	**$1,547**
Mine Dispatch			Variable	$3,162	$667	$108	$135	$156	$119	$0	$0	$0	$135	$65	**$4,546**
		Total		**$66,983**	**$56,380**	**$23,063**	**$7,544**	**$10,509**	**$12,566**	**$463**	**$490**	**$7,191**	**$12,068**	**$15,313**	**$212,589**

Table 16-8: Mine Capital Cost Summary

		Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Total
Mobile Equipment	$ x 1000	$66,982.9	$56,380.1	$23,063.4	$7,543.7	$10,509.4	$12,565.9	$462.9	$489.8	$7,190.5	$12,067.7	$15,312.9	$75.0	$75.0	$75.0	$75.0	**$212,869.1**
Spares & Inventory	$ x 1000	$2,679.3	$2,255.2	$922.5													**$5,857.1**
Slope Monitoring Equipment	$ x 1000	$447.6	$1,301.7	$160.6	$587.3	$160.6	$160.6	$587.3	$160.6	$160.6	$762.9	$160.6	$160.6	$587.3	$160.6	$0.0	**$5,558.5**
Training Simulators	$ x 1000																
Development & Pre-Stripping	$ x 1000	$38,868.6	$2,460.3	$1,845.8	$0.0	$2,899.7	$3,742.9	$2,291.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	**$52,109.2**
Total	$ x 1000	**$108,978.4**	**$62,397.3**	**$25,992.3**	**$8,131.0**	**$13,569.7**	**$16,469.4**	**$3,342.1**	**$650.4**	**$7,351.1**	**$12,830.6**	**$15,473.5**	**$235.6**	**$662.3**	**$235.6**	**$75.0**	**$276,393.9**
Cumulative Total	$ x 1000	**$108,978.4**	**$171,375.7**	**$197,368.0**	**$205,499.0**	**$219,068.7**	**$235,538.1**	**$238,880.2**	**$239,530.6**	**$246,881.7**	**$259,712.3**	**$275,185.8**	**$275,421.4**	**$276,083.7**	**$276,319.3**	**$276,393.9**	

16.9 OPPORTUNITIES

- There are approximately 15 Mton of Inferred Mineral Resources within the ultimate reserve pit limits that are currently classified as waste. This material may represent a future opportunity.
- Modifications in some of the current pit designs or sequencing of phases may result in recovery of additional resources that were excluded from the Mineral Reserves due to ramp locations and access requirements in general.

16.10 RECOMMENDATIONS

- Mine designs were developed for the 20 ft bench height of resource model provided for mine planning. Mill-CIL feed is an important part of the overall plan for the Project and dilution on higher bench heights should be monitored going forward to determine if proper selectivity can be achieved in practice on larger benches.
- Wall slopes should also be monitored carefully, particularly in the areas where fault zones intersect haulage roads and where points occur at pit phase overlap areas.
- Waste backfill to completed pits may be further optimized to reduce haulage profiles during peak production periods.
- Alternative long-range planning schedules may reduce the number of operating phases in some years and improve equipment distribution.

17 RECOVERY METHODS

17.1 INTRODUCTION AND GENERAL DESCRIPTION

Current operations consist of a 14,000 short ton of ore per day (ton/d) run of mine (ROM) heap leach operation. The planned expansion for Phase 2 will include a 50,000 ton/d (45,400 t/d) ROM heap leach and a new 3,500 ton/d (3,200 t/d) crushing, milling and Carbon-in-Leach (CIL) plant for recovering gold and silver from mill grade ore. This section of the Report describes the facilities that will be required for the Phase 2 expansion. The design for the processing facilities is based on metallurgical testwork and analysis described in Section 13 of this Report.

For Phase 2, the heap leach pad will be designed to process 18.2 Mton (16.5 Mt) annually at an average life of mine grade of 0.012 oz/ton (0.41 g/t), while the mill will be designed to process 1.3 Mton (1.2 Mt) annually at an average LOM grade of 0.068 oz/ton (2.32 g/t). When considering both Phase 1 and Phase 2, operations will extend to approximately 19 years with an additional estimated three years of heap rinsing as part of reclamation where gold will continue to be leached and recovered.

The existing heap leach pad will be expanded to allow processing of the lower grade ROM ore at a rate of 50,000 ton/d (45,400 t/d) with a Carbon-in-Column (CIC) circuit to recover the gold and silver from the leach solution. The higher-grade ore will be processed in a 3,500 ton/d (3,200 t/d) crushing and milling circuit with gravity recovery, followed by a leach/CIL circuit for recovery of gold and silver from the gravity tailing.

A carbon handling circuit including acid wash, desorption (stripping), and carbon regeneration will be added to process carbon from both the CIC circuit and the leach/CIL circuit. The gold and silver recovered from the desorption circuit will be processed through EW cells to produce a sludge. The EW sludge will be processed using a retort oven for drying and mercury recovery, and then refined in a melting furnace to produce gold and silver doré bars.

The plant will be a conventional crushing and milling facility with a hybrid leach/CIL gold recovery circuit, cyanide detoxification and tailings filtration. There are no new or novel processing steps. Results from the test programs were used to develop the corresponding process design criteria, mechanical equipment list, flowsheets, and operating costs. Where appropriate, the design parameters used are supported by past production results. The plant design will include flexibility for treatment of all ore types as per testwork at design throughput.

The crushing circuit will operate at an availability of 75%, resulting in a nominal hourly throughput of 194 ton/h (176 t/h). The remainder of the plant will operate 24 hours per day, 365 days per year at an availability of 90%, resulting in a nominal hourly throughput of 162 ton/h (147 t/h). The carbon plant will be sized to process 18 tons (16 t) of loaded carbon daily (12 tons from the heap leach CIC circuit daily and 6 tons from the mill leach/CIL circuit every other day).

The estimated yearly process production summary is shown in Section 22 and was based on a detailed analysis of recovery from the many samples tested as described in detail in Section 13. Ore grades and relative amounts of mill ore and heap leach ore vary throughout the mine life. Delay time due to stacking and holdup in inventory for ounces recovered from the heap leach are considered on a month to month basis in early stages, but then are expected to reach a steady state for the remainder of the mine life until rinsing when the production of gold will diminish. Typical annual gold production for Phase 2 and total LOM gold production for the heap leach and mill are listed in Table 17-1.

Table 17-1: Phase 2 and LOM Gold Production

Ore	Avg. Gold Rec (%)	Avg. Annual Phase 2 Gold Production (Ounces)	Total Gold Production from Rinsing (Ounces)	Total Gold Production (Ounces)
Heap Leach Ph. 1 Years 1-5	-	-	-	189,000
Heap Leach Ph. 2 Years 6-19 (Production) Years 20-22 (Rinsing)	67*	142,000	236,000	2,095,000
Mill Years 6-19	94	80,000	N/A	1,108,000
Total (Phase 2)	-	222,000	-	**3,203,000**
Total (LOM)				**3,392,000**

* Gold Recovery increases to 74% once final rinsing is conducted.

The major process unit operations are illustrated in a simplified process flowsheet in Figure 17-1. Plant layout for both the heap leach CIC and mill are shown in Figure 17-2. A summary of the key process design criteria is provided in Table 17-2 and Table 17-3.

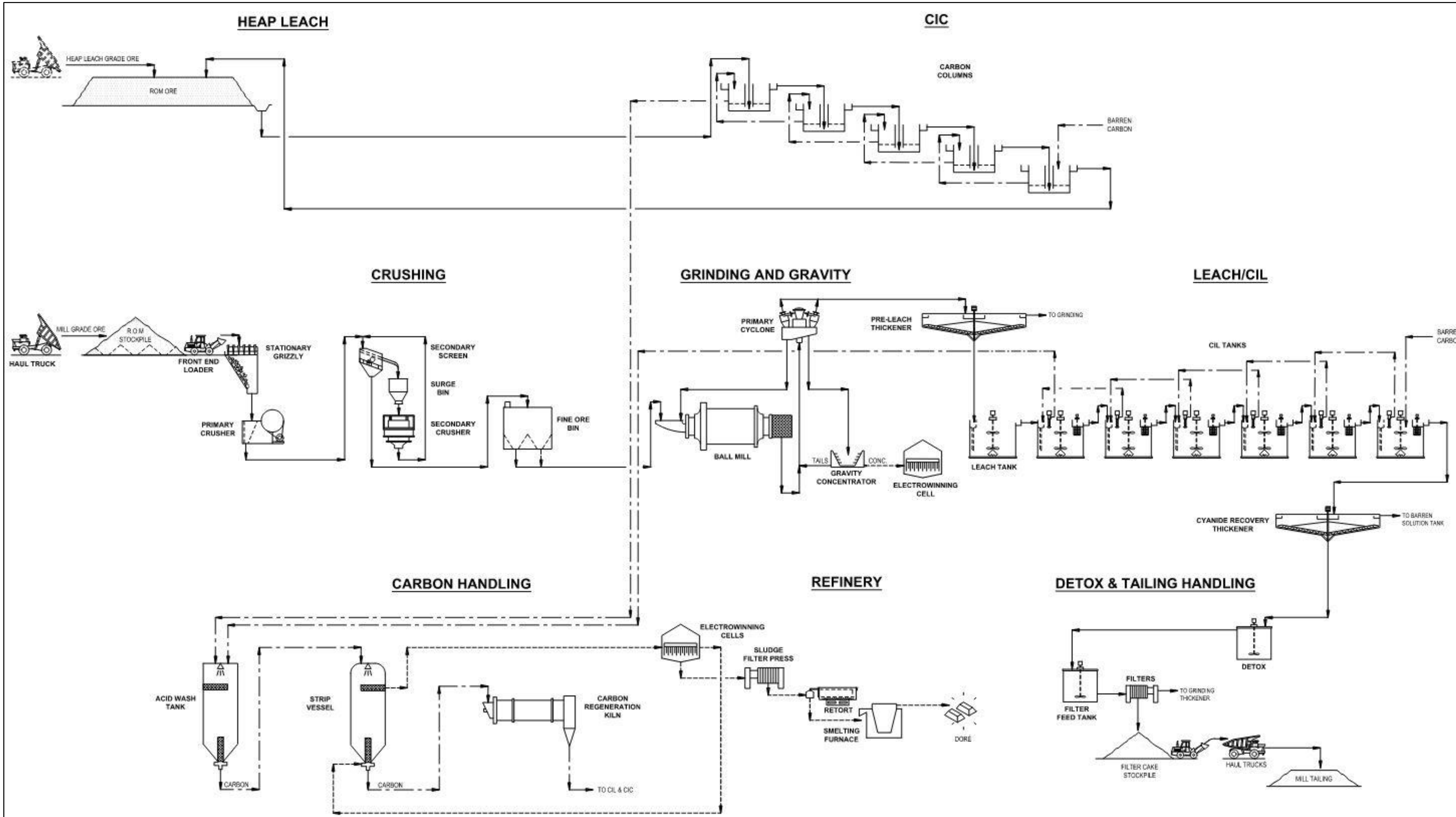


Figure 17-1: Overall Process Flowsheet

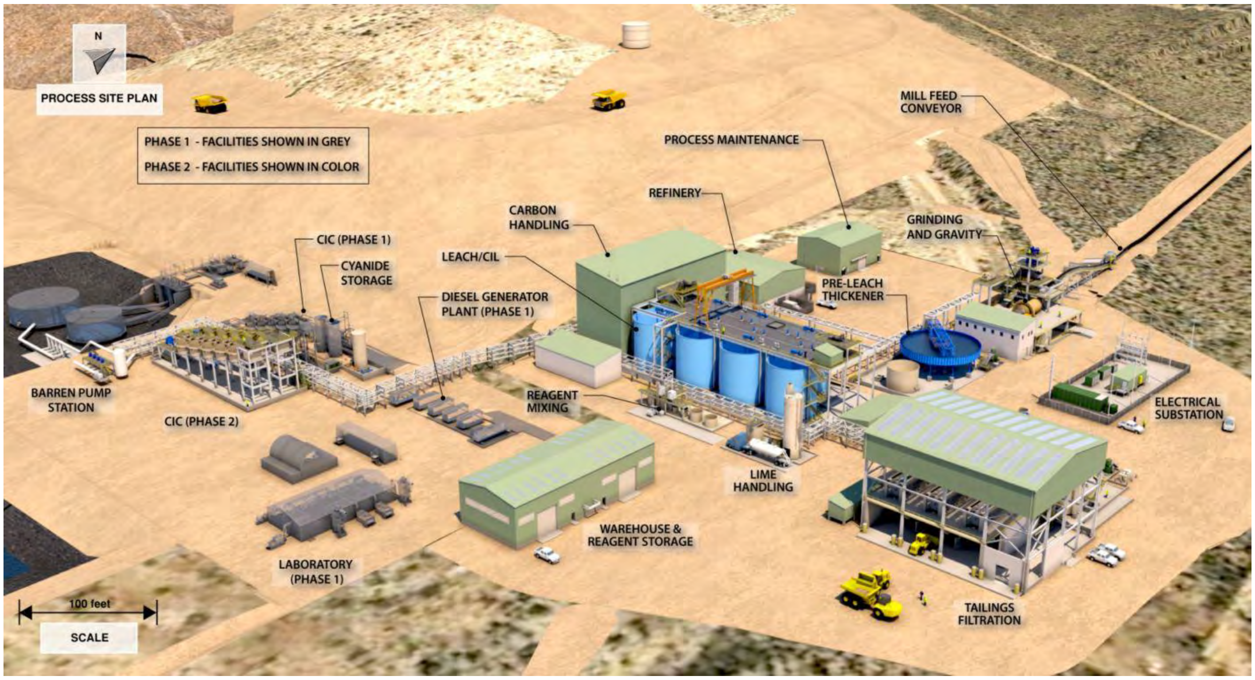


Figure 17-2: Process Plant Facility

Table 17-2: Major Process Design Criteria Summary – Heap Leach

Parameter	Unit	Value
Daily Processing Rate - Heap Leach	ton/d	50,000
Operating Days per Year	days	365
LOM Heap Feed Grade	oz Au/ton	0.012
ROM Feed Size, F_{100}	in	17
ROM Feed Size, F_{80}	in	7
ROM Feed Size, F_{50}	in	4
Lift Height	ft	25
Ultimate Stack Height	ft	300
Primary Leach Cycle	days	80
Saturation Moisture	%	10
Drain Down Moisture	%	9
Application Rate	gpm/ft^2	0.004
Barren Solution Flow	gpm	12,800
Barren Solution Cyanide Concentration	ppm	500
Pregnant Solution Flow	gpm	12,000
CIC Columns, Specific Flow Rate	gpm/ft^2	22.5
Carbon Loading	oz Au+Ag/t	100

Table 17-3: Major Process Design Criteria Summary – Mill

Parameter	Unit	Value
Daily Processing Rate - Mill	ton/d	3,500
Operating Days per Year	days	365
Availability, Crusher	%	75
Availability, Mill	%	90
LOM Mill Feed Grade	oz Au/ton	0.068
Specific Gravity	-	2.3
Crusher Feed Size, F_{100}	in	17
Crusher work Index	kWh/ton	12.1
ROM Stockpile Capacity	ton	100,000
Fine Ore Bin Capacity (live)	ton	3,500
Ball Mill Feed Size, F_{100}	in	0.5
Ball Mill Product Size, P_{80}	µm	150
Ball Mill Circulating Load	%	300
Bond Ball Mill work Index	kWh/ton	15.2
CIL Cyanide Concentration	NaCN/L	500
CIL Slurry Feed Density	%	45
CIL Retention Time	h	30
Carbon Loading	oz Au+Ag/t	80
Detox Feed, CN_{WAD}	ppm	125
Detox Discharge, CN_{WAD}	ppm	25
Detox Retention Time	h	2
Tailing Filter Feed Slurry Density	%	60
Final Tailings Cake Moisture (Wet Weight Basis)	%	18
Filter Cycle Time	min	20.5

17.2 MAJOR EQUIPMENT AND FACILITIES

The major equipment is summarized by each main process area in Table 17-4 and Table 17-5. Carbon from both the heap leach and the mill is handled in a common facility which is summarized in Table 17-6. Equipment sizing considers a range of operational parameters based on rheological properties of slurry anticipated where applicable.

Table 17-4: Major Process Equipment – Heap Leach

Item	Quantity	Description	Power
Barren Solution Pump	4	12 in Vertical Turbine	1,000 hp
Pregnant Solution Pump	4	8 in x 10 in Horizontal Centrifugal	150 hp
CIC Column	10	18 ft. diam. 5 per train	-

Table 17-5: Major Process Equipment – Mill

Equipment	Quantity	Description	Power
Primary Crusher	1	49 in x 37 in Jaw	175 hp
Secondary Crusher	1	60 in Standard Cone	500 hp
Ball Mill	1	16.5 ft diameter x 21 ft F/F	3,300 hp
Gravity Concentrator	1	Centrifugal Bowl, 48 in bowl diam.	60 hp
Pre-leach Thickener	1	68 ft diam. High rate	-
Leach/CIL Tanks	7	37 ft diam. x 40 ft height; Agitated	100 hp
Cyanide Recovery Thickener	1	68 ft diam. High rate	-
Filter Feed Pump	2	10 in x 8 in Horizontal Centrifugal	350 hp
Tailing Filter	3	8.2 ft x 8.2 ft (2.5 m x 2.5 m) Pressure filter, 64 chambers, 16 min cycle	100 hp

Table 17-6: Major Process Equipment – Carbon Handling and Refining

Equipment	Quantity	Description	Key Criteria
Acid Wash Vessels	2	FRP construction	6 ton capacity
Strip Vessels	2	Pressure vessel; Stainless steel construction	6 ton capacity
Carbon Regeneration Kiln	1	5 ft diam. x 50 ft long, Horizontal Propane-Fired Indirect	1,500 lb/h 18 ton/day
Mercury Retort	1	3 ft^3 Electric	30 kW
Electrowinning Cells	4	Sludging, 2000 amps @ 6 volts	-
Smelting Furnace	1	Induction Furnace	450 kW

17.3 PROCESS DESCRIPTION

17.3.1 Heap Leaching

17.3.1.1 Stacking and Leach Pad Operation

ROM ore will be delivered to the leach pad by the mining fleet.

Pebble Quicklime (CaO) will be added for pH control of the process from two 100 ton silos. The lime will be metered into a clamshell and dumped into the loaded trucks which will then deliver the ore to the active stacking area. Lime will be added in proportion to the tonnage of ore being hauled with an estimated consumption of 2.35 lbs/ton based on metallurgical testwork.

The ore haul trucks will operate on top of the lift being constructed. A ramp, or ramps, will be constructed to reach the top of each current lift. The trucks will direct-dump the ore on the current lift and a dozer will push the ore over the edge of the lift to form the expanding heap. The stacked ore will be deep-shank cross-ripped with the dozer prior to leaching. Ore will be stacked in 25 ft high lifts to a height of approximately 300 ft above adjacent native grades.

Prior to stacking a new lift over the top of an old one, the top of the old lift will be cross ripped to break up any cemented/compacted sections and to redistribute any fines that may have been stratified by the irrigation solution or rainfall.

17.3.1.2 Leach Pad Design

Several leach pad base grading options were evaluated to optimize the native terrain by minimizing earthworks cuts and fills while maintaining a solution collection system that maintains one foot or less of solution head over the primary liner as required by current site regulations. The native terrain within the property limits and the Phase 2 leach pad expansion area flows to the southeast towards Phase 1 and the historic leach pad.

Solution within the Phase 1 leach pad is collected via a series of 4 inch perforated corrugated HDPE pipes at close spacing (approximately 15 ft apart) that flow towards 18 inch perforated corrugated HDPE pipes as the main headers. Main solution collection piping headers are oversized to operate at a design flow of approximately 50% of the pipe's capacity to allow for adequate protection against pipe deflection and clogging. These 18 inch headers can drain an area of approximately 600,000 ft^2 or less at the proposed irrigation rate of approximately 0.004 gal/min/ft^2. These sub-cells then drain to the main perimeter solution collection channel which flows to the process plant. The channel has a robust design to operate under load of a multi-lift pad over the project duration. Smaller 4 inch collection header lengths have been minimized to provide less of an opportunity for pipe collapse and failure.

Phase 2 design considered both use of a single pad to be adjoined with the current Phase 1 pad as well as construction of a separate pad. Both options would use a solution collection channel running from North to South which minimizes the length of solution collection piping required to report solution to the main header. The single pad option, which is the basis for the feasibility study, requires ore to be backfilled over the top of the new solution collection channel while the separate pad option provided for an open solution collection channel without backfill. A single pad decreases the project footprint and disturbance area, improves logistics and hauling distance.

In order to mitigate concern over the backfilling of the solution collection channel, the solution collection channel will have a more robust pipe specification as well as a channel corridor which will be backfilled with a layer of competent rock on top of a porous waste rock in order to be free draining. Pipe junctions will be encased in concrete or other high strength materials to achieve proper fortification and assure solution flow. Figure 17-3 illustrates the basic solution pipeline corridor design concept.

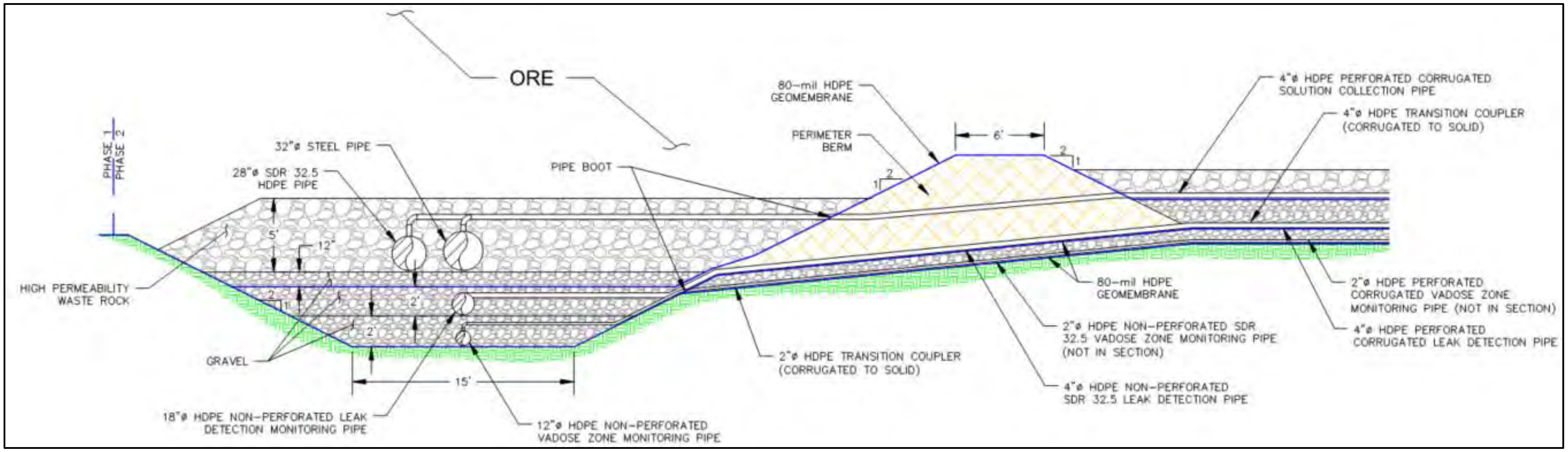


Figure 17-3: Heap Leach Pad Solution Corridor

The leach pad has been designed with a lining system in accordance with California Code of Regulations Title 27 requirements and International Cyanide Code recommendations which meets or exceeds the North American standards and practices for lining systems, piping systems and process ponds. These standards and the proposed leach pad are intended to minimize the potential for facility operations to impact soils, surface water, and groundwater in and around the site.

The ROM leach pad will be placed both on native alluvial soils and on the side slope of the existing closed historic heap leach pad. The new leach pad will be a multiple-lift, single-use type pad. Similar to the current Phase 1 design, the Phase 2 heap leach pad extension will be lined using a double-liner system consisting of prepared subgrade, a vadose zone monitoring system, a secondary 80 mil LLDPE geomembrane liner, a two-foot layer of gravel for leak detection, a primary 80 mil LLDPE geomembrane liner, and a two-foot thick layer of gravel and perforated HDPE piping network for liner protection and solution collection.

The prepared subgrade will consist of compacted 3/8" fine ore which was previously crushed, leached and rinsed on the historic heap leach pad. A minimum two-foot thick overliner of spent crushed ore also from the historic leach pad will be spread over the primary LLDPE liner to protect the liner from puncture and to provide a permeable blanket drainage layer for heap leach irrigation solutions. A puncture test was conducted for Phase 1 using the existing leached and rinsed ore with an 80 mil LLDPE liner and assumed a stacked ore height of approximately 400 ft above the liner. The puncture test showed little to no indentations with no punctures nor adverse stretching of the liner. Phase 2 heap leach design intends to stack to an elevation of 300 ft. Figure 17-4 illustrates the heap leach pad layers and liner.

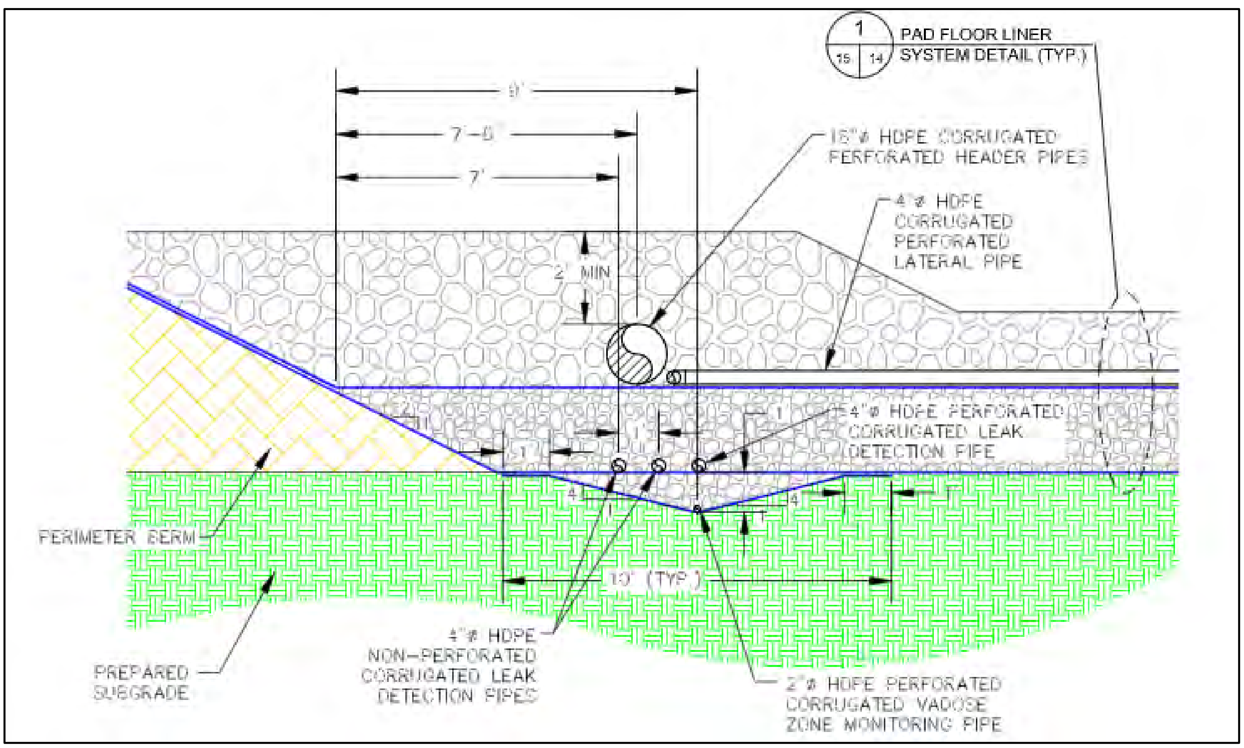


Figure 17-4: Heap Leach Pad Design Layers

The Phase 2 leach pad footprint will encompass an area of approximately 480 acres within the Project's permitted 3,910 acre Mine Property boundary. The top of the heap leach pad will extend to about 300 ft above adjacent native grades, and the maximum elevation of the heap leach pad will be approximately 4,600 ft amsl. Based on the difference between the proposed bottom grades and the proposed top of heap grades, the gross volume for the Phase 2 extension of the heap leach pad will be approximately 230 Mton.

17.3.1.3 Event Pond Design

Two event ponds will be used to manage storm solution at the heap leach facility. The Phase 1 Event Pond (23.5 M gal) is used for the current operations while the Phase 2 Event Pond (100 M gal) will be constructed as part of the expansion to handle the larger potential flows coming from the expanded heap leach pad. These ponds will not be used during normal operations remaining mostly dry and are intended to only receive solution during abnormal events such as a storm.

The Phase 2 Event Pond is designed to handle a 100-year, 24-hour precipitation event on the heap pad and pond areas and is sized based on the final ultimate leach pad area added as part of the Phase 2 expansion. Solution collected in the event pond will be returned to the process as soon as practical via portable pump. The event pond will be double-lined with a primary (upper) 80 mil HDPE geomembrane liner, a geonet and a secondary (bottom) 60 mil HDPE geomembrane liner. The pond will include a conventional leak detection system comprised of a sump and a set of riser pipes. Pond liner system details are shown in Figure 17-5.

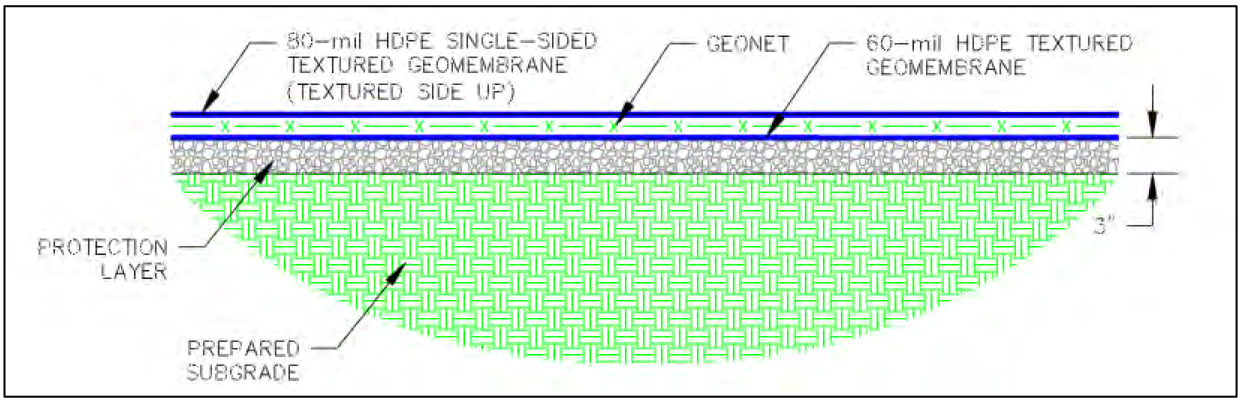


Figure 17-5: Event Pond Layers

17.3.1.4 Stacking Progression

The heap leach pad will be developed as a single pad abutted to the Phase 1 heap in four distinct campaigns throughout the course of the mine life. To minimize haul distances, the heap leach cells will be accessed via two haul roads: the existing haul route to the South or a new haul road providing access to the Northern extents of the pad. ROM ore from JSLA, Jumbo and Oro Belle pits will mostly be sent to cells 2A, 2B and 2C while ROM ore from South Domes will mostly be sent to cells 2C and 2D.

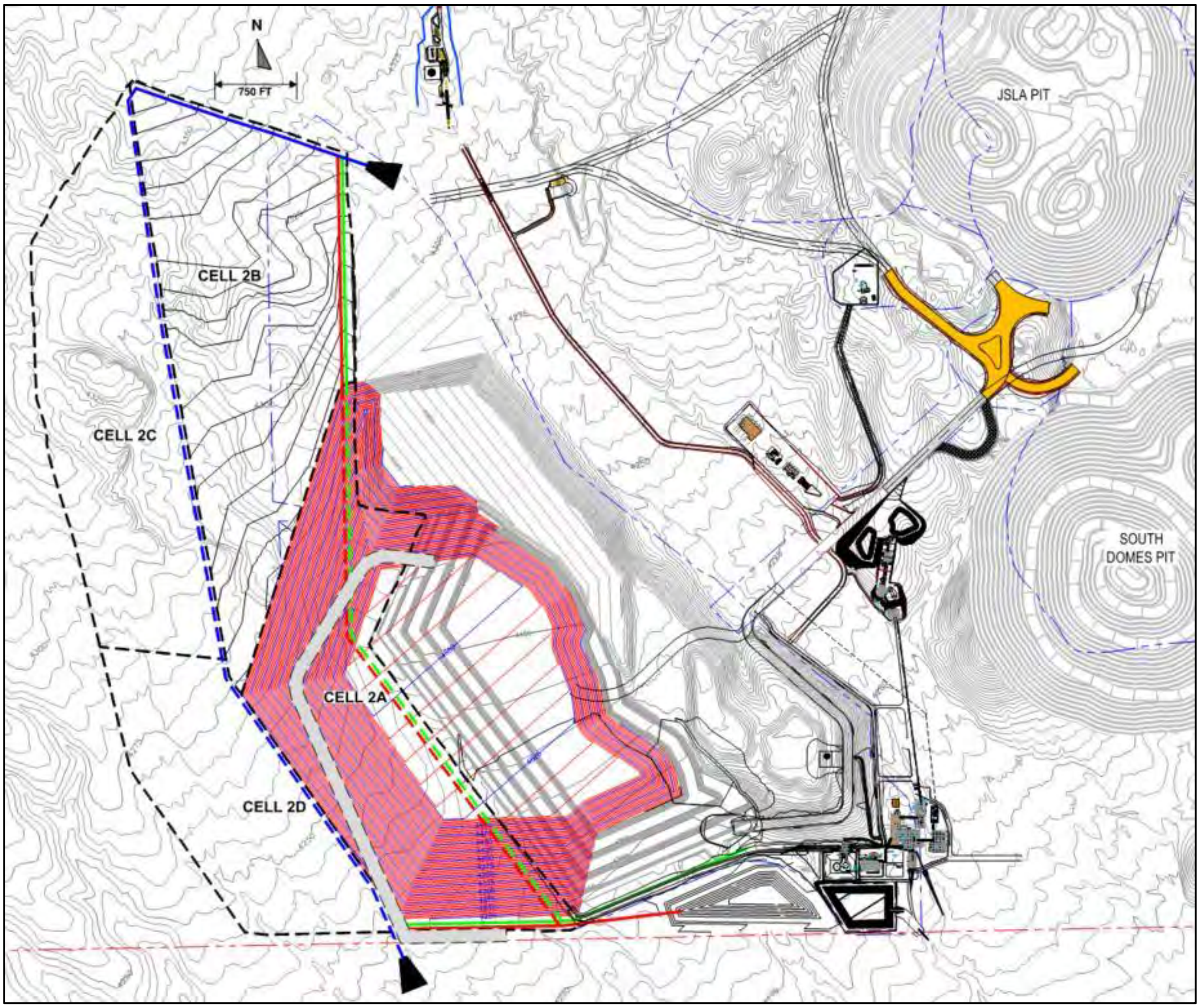


Figure 17-6: Stacking Cell 2A; (Phase 2 Years 1-4)

The initial capital project will include development of the first cell, cell 2A, which will accommodate 78 Mton of heap leach ore. This first cell will provide enough capacity for project operation through the first four years after Phase 2 startup. Ore will be stacked initially in the area just West of the historic heap leach pad over the top of the newly constructed solution collection channel and slowly expand upgradient to the West throughout the first year. Once material from the newly lined cell 2A area is up to the elevation of the Phase 1 pad, material will be placed on top of the newly combined Phase 1 pad and heap leach cell 2A. Heap leach cell 2A stacking plan is illustrated in Figure 17-6.

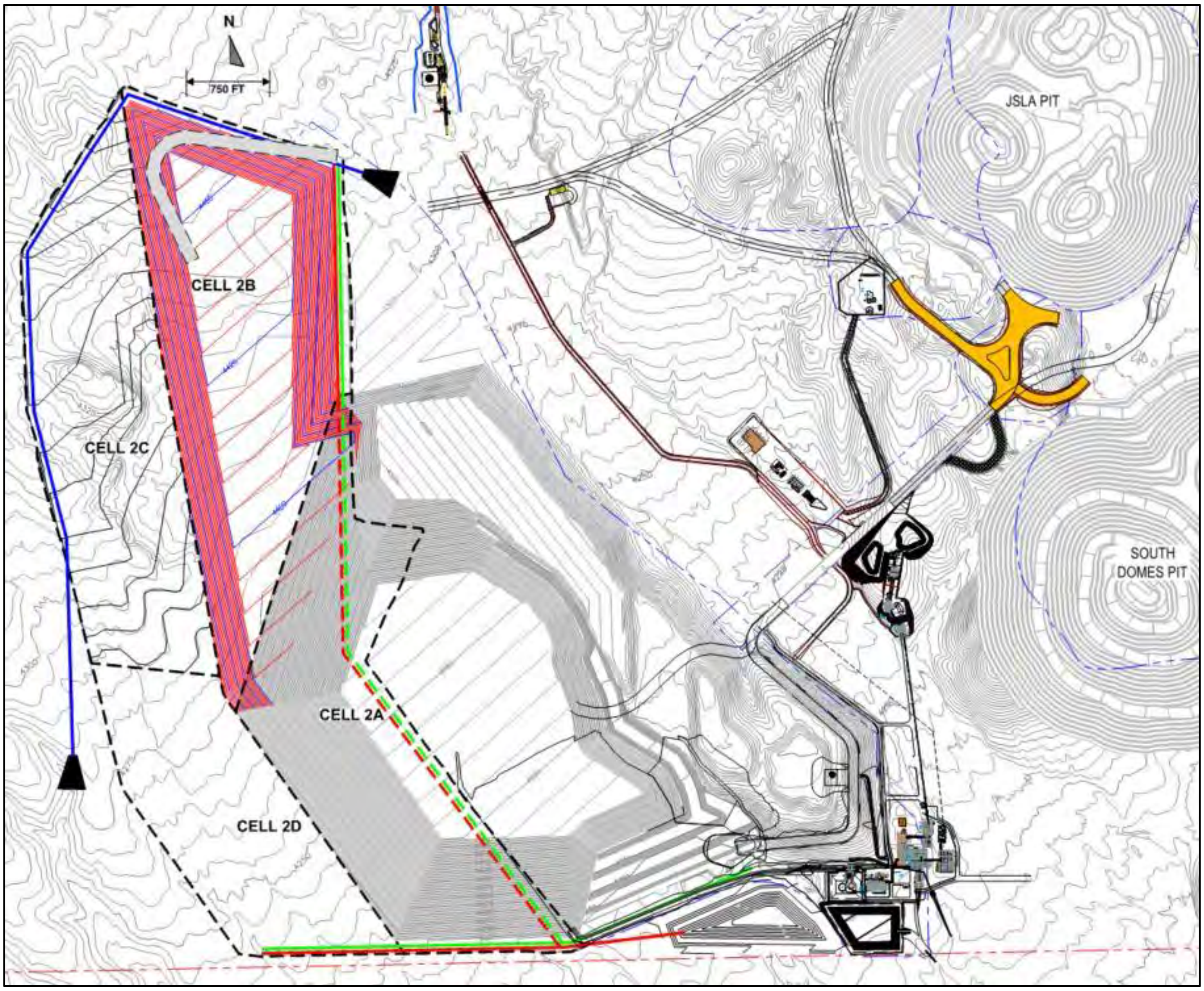


Figure 17-7: Stacking Cell 2B; (Phase 2 Years 5-7)

Cell 2B will accommodate 29 million tons of heap leach ore. The addition of the second cell will provide capacity for project operation through approximately 7 years of Phase 2 production. Early phases of Cell 2B will see ore stacked adjacent to the Cell 2A western slope. Material placements will then move upgradient to the North and West. Cell 2B stacking plan is illustrated in Figure 17-7.

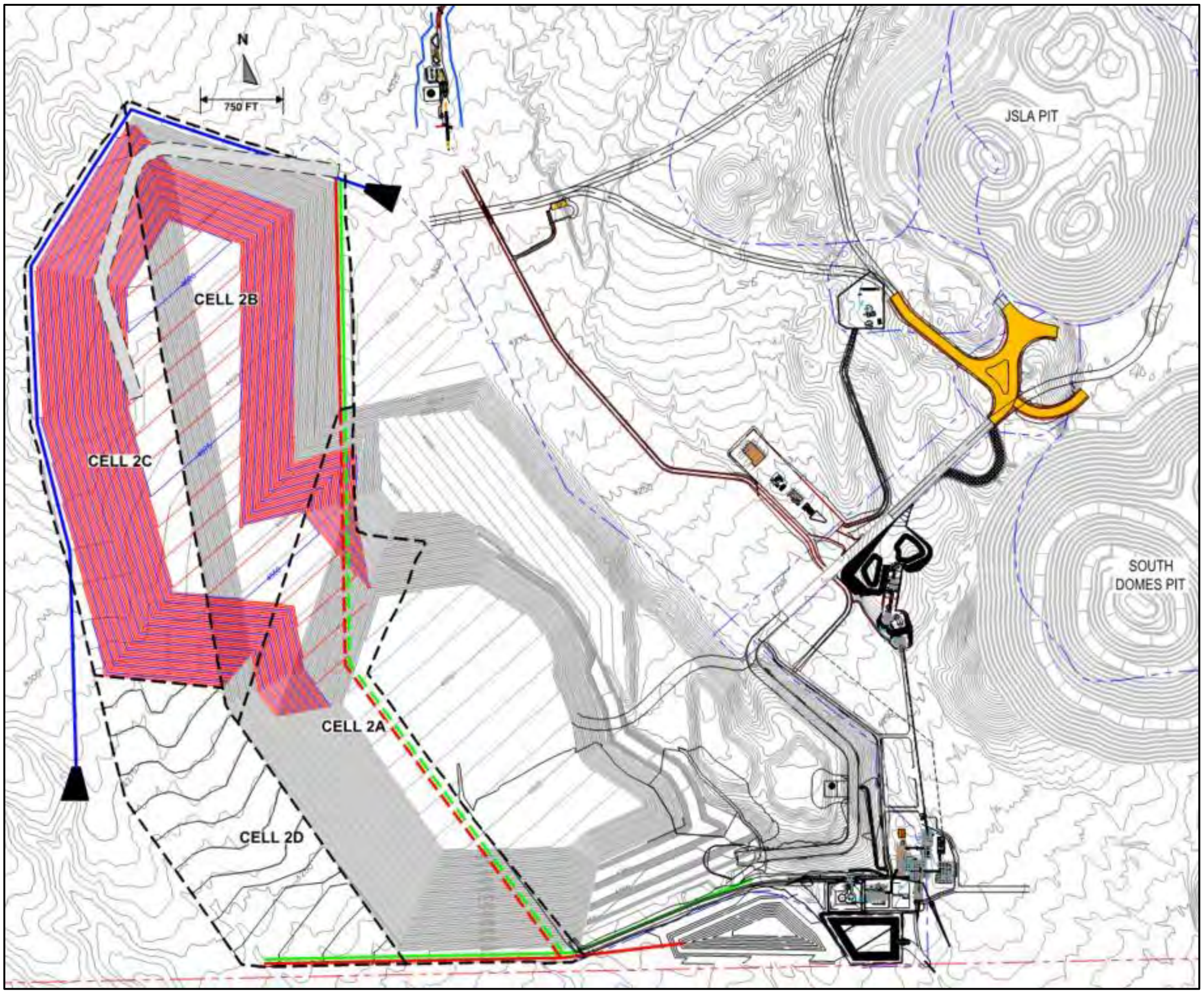


Figure 17-8: Stacking Cell 2C; (Phase 2 Years 7-10)

Cell 2C will accommodate 74 Mton of heap leach ore. This third cell will provide capacity for project operation through approximately 10 years of Phase 2 production. Early phases of Cell 2C will see ore stacked adjacent to the Phase 2B pad moving upgradient to the West. Upon matching the cell height of Cell 2B, Cell 2C material will be placed on top of Cell 2B until Cell 2C matches the Cell 2A elevation. At this point, each of the three Phase 2 cells will be merged into one pad surface. Cell 2C stacking plan is illustrated in Figure 17-8.

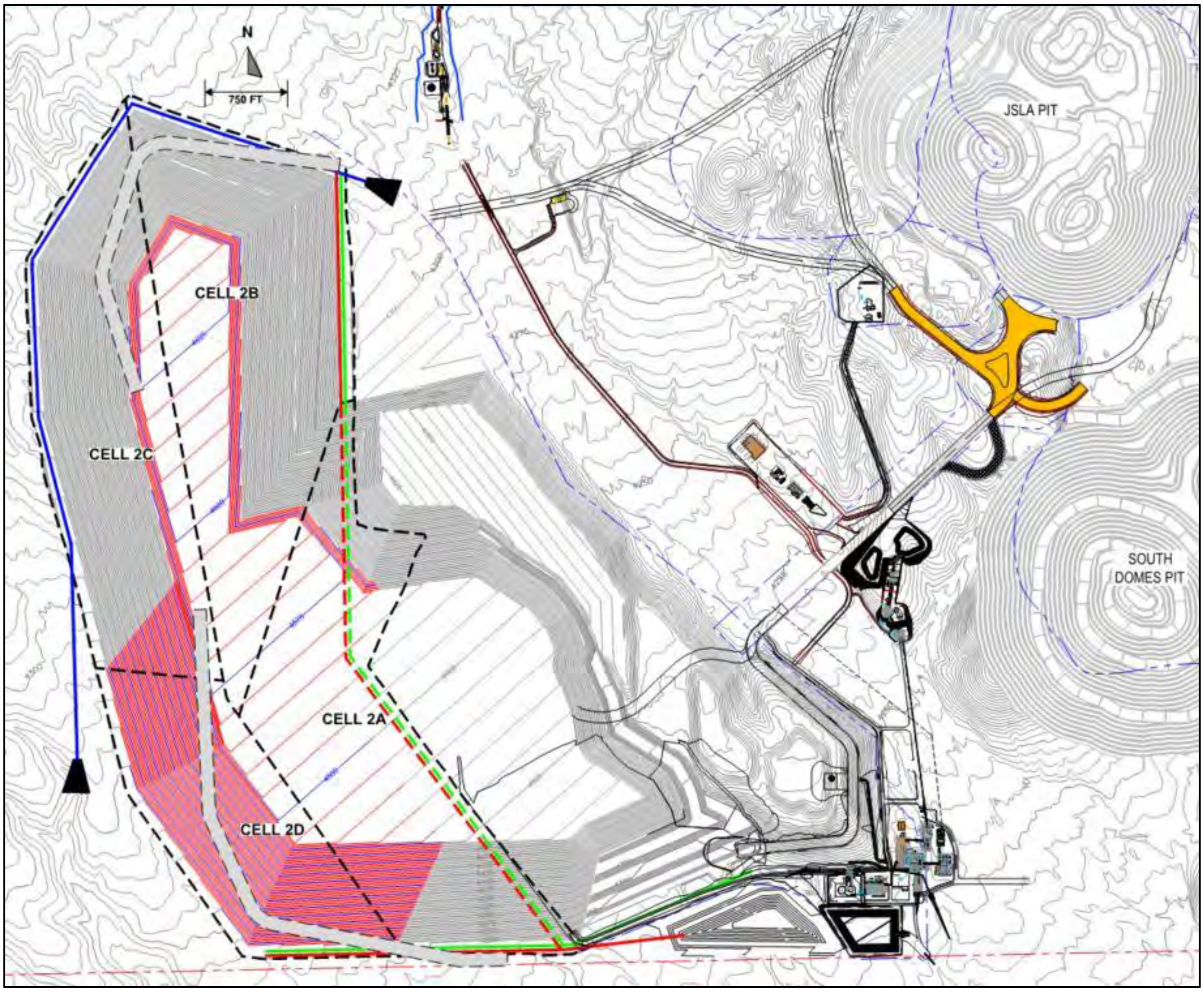


Figure 17-9: Stacking Cell 2D; (Phase 2 Years 11-14)

Cell 2D will accommodate 50 Mton of heap leach ore. This fourth cell will provide capacity for project operation through currently planned Phase 2 production. Early phases of Cell 2D will see ore stacked adjacent to the Phase 2A and 2C pads moving West upgradient towards the Western property boundary. Upon matching the adjacent cell height, material will continue to be placed to an ultimate height of approximately 300 ft above surrounding natural grade. Cell 2D stacking plan and ultimate heap is illustrated in Figure 17-9.

17.3.1.5 Leaching and Solution Handling

After a leach cell lift has been stacked and cross ripped, the irrigation system will be installed. Dripline emitters will be used to apply a dilute cyanide solution at an application rate of 0.004 gpm/ft^2 to the material. A primary leach cycle of 80 days has been selected for the ROM material based on the testwork. After primary leaching, the heap will freely drain and dripline emitters will be removed prior to cross ripping and subsequent stacking. Each lift is cross ripped both before and after leaching.

Barren solution exiting the CIC circuit will be recirculated to the heap leach pad. Make-up water for the heap leach pad will come from the mill process water tank which is used to recycle dilute cyanide solution. Barren leach solution pH will be maintained at a minimum value of 10.

Barren solution will be pumped to the top of the heap leach pad from the barren solution surge tank located near the leach pad, by four vertical turbine pumps at a nominal flow of 12,800 gpm. Cyanide solution and antiscalant are added to the barren sump as needed. Barren solution will pass through an in-line filter to remove any carbon or other foreign material from the solution to prevent plugging of the emitters and distribution system. This solution will be carried by a new HDPE pipeline to the base of the heap and then to a network of sub-headers and risers to the top of the heap where it is finally applied to the material by drip emitters.

Solution passing through the heap will dissolve the precious metal values and be collected in a network of perforated solution collection pipes, which feed to a common discharge point at the base of the heap. The solution will then be carried by gravity to a pregnant solution tank. Pregnant solution will be pumped from the pregnant tank to the adsorption carbon column circuit at the recovery plant.

17.3.1.6 Carbon Columns

The carbon adsorption circuit will consist of two trains of five cascading carbon columns. The pregnant solution will be pumped to the carbon adsorption circuit across a stationary trash screen for removal of any debris from the heap leach facility. The solution will flow by gravity from column 1 to column 5; the carbon will be pumped countercurrent to the main solution flow. The barren solution overflow from the final column will discharge via a safety screen to recover any carbon that may be flushed from the circuit and will flow by gravity to the barren solution surge tank. On average, 12 tonnes of loaded carbon from the first carbon columns (6 tonnes from each train) will be pumped to the acid wash and stripping circuits each day. The carbon will be advanced up the train, with reactivated barren carbon added to the fifth column. Existing carbon columns on site will be used during project start up to allow for time to transition and develop sufficient flow to new carbon columns.

17.3.2 Crushing and Crushed Ore Storage

Ore from the open pit mine will feed a crushing plant that consists of two stages of crushing. The plant will process 3,500 ton/d of ore, operate 24 hours per day and produce a final product with a P_{80} of ½ in. The crushing circuit will be designed for higher throughput than the rest of the plant to allow for catch up due to anticipated lower overall crushing circuit availability.

17.3.2.1 Crushing and Screening

ROM ore will be dumped onto stockpiles and a front-end loader (FEL) will be used to reclaim ore via a static grizzly with 24 inch openings. Ore will flow into a 20 yd^3 dump pocket, while any boulders will be removed for later reduction using a backhoe with a rock breaker attachment. An apron feeder will draw ore from the dump pocket and discharge onto a vibrating grizzly feeder. The vibrating grizzly oversize ore will feed directly into the 49 in x 37 in jaw crusher with an installed power of 175 hp. The minus 4 inch ore will bypass the crusher and feed directly onto the coarse ore transfer conveyor. The primary crushing stage will produce a product P_{80} of approximately 4 in with a closed side setting (CSS) of 4 in. A magnet will be installed over the coarse ore transfer conveyor to ensure any tramp metal is removed ahead of fine crushing.

Jaw crusher discharge will be combined with the vibrating grizzly undersize on the coarse ore transfer conveyor and will feed an 8 ft x 20 ft double deck inclined screen. The top deck of the secondary screen will have an aperture size of 1 inch and the bottom deck will have an aperture size of ½ in. The plus 1 inch material from the top deck and the plus ½ inch ore from the bottom deck will be conveyed to a surge bin ahead of the secondary cone crusher. The undersize from the bottom deck, minus ½ inch ore, will be the final product that will discharge onto the screen undersize transfer conveyor and ultimately report to fine ore bin via the final crushing product conveyor.

A belt feeder will withdraw ore from the bottom of the surge bin and feed it into a standard cone crusher with an installed power of 500 hp. The secondary crusher will operate in closed circuit and will reduce the ore to produce a product P_{80} of approximately 0.8 in. Crushed product will be combined with crushed material from the primary crushing stage on the coarse ore transfer conveyor and return to the secondary screen.

Between the screen undersize transfer conveyor and the final crushing product conveyor, a diverter gate will be installed to allow for recovery of crushed product material should the bin be temporarily unavailable.

A water spray system will be installed to suppress dust at the crusher dump pocket while cartridge type dust collectors will be installed to capture and control dust from the crushing and screening plant.

Any ore spillage will be returned to the nearest belt conveyor for processing. Precipitation falling on the crusher will be collected in a sump in the lowest level of the crusher structure and will be pumped to the process plant for use in the process.

The primary crushing facility will utilize a 24 ft high mechanically stabilized earth retaining structure to build up elevation to allow for proper vertical clearance between equipment within this facility. The retaining wall is backfilled with select structural backfill material while further extension of the pad is achieved using mine waste from Phase 2 mine pre-development activities.

A two stage crushing system was selected after completion of a high-level trade off study which investigated 2-3 stages of crushing as well as use of a semi-autogenous grinding mill (SAG) and pebble handling circuit. Cost implications from an operating and capital standpoint were investigated and it was determined based on preliminary pricing, that two stages of crushing offered optimized economics.

17.3.2.2 Fine Ore Bin and Reclaim

The crushing plant product will be conveyed to a fine ore bin. The bin will be fully enclosed and of steel construction and will provide 3,500 tons, or approximately 24 hours, of live storage capacity. Two draw points under the bin will provide ore to two reclaim belt feeders located underneath the bin. The reclaim feeders will discharge onto the ball mill feed conveyor. The feeders will be installed with variable frequency drives (VFD) to control the ore feed rate to the grinding circuit. Each belt feeder will be capable of providing the total plant throughput. Either or both feeders may be operated at any time. The speed of the feeders will be controlled by a control signal provided by a belt scale mounted on the conveyor downstream of the feeders. A clean up bin will be installed over the tail end of the ball mill feed conveyor to allow for recovery and clean up of any dribble material at the reclaim.

A cartridge type dust collector will be installed to capture and control dust from the reclaim feeders within the reclaim tunnel.

17.3.3 Grinding and Gravity Concentration

The grinding circuit will consist of a single stage ball mill with a gravity concentration circuit to recover any gravity recoverable gold and silver. The mill will operate in closed circuit with cyclones. A portion of the cyclone underflow will be processed through the gravity circuit. The grinding circuit will be able to process a nominal throughput of 162 ton/h (fresh feed), producing a final product P_{80} of 100 mesh (150 µm).

The grinding and gravity circuit will be located in an open building structure. The area will be serviced by a 50 ton mobile crane which will be shared across the plant site. The floor will be concrete with containment walls to contain process upsets within the grinding area.

17.3.3.1 Grinding

Reclaimed ore from the fine ore stockpile will feed a 16.5 ft x 21 ft long overflow ball mill via the ball mill feed conveyor. The mill will be supplied with rubber liners, a single 3,300 hp wound rotor induction motor with a VFD, a trommel screen, and a retractable feed spout/chute. Pebble lime will be slaked, and milk of lime will be added to the ball mill feed for pH control. An automated feeder will supply grinding media to the ball mill. Ground slurry will overflow from the ball mill onto the trommel screen attached to the discharge end of the mill. The trommel screen oversize will discharge into a trash bin for removal from the system, while the undersize will flow into the cyclone feed pump box.

Slurry from the cyclone feed pump box will be pumped to a cluster of four (2 operating/2 standby) 12 inch cyclones for size classification. Process water will be added to the ball mill feed and the cyclone feed sump to achieve the appropriate pulp density. The coarse cyclone underflow from one operating cyclone will flow by gravity back to the ball mill for additional grinding while the underflow from the second operating cyclone will flow by gravity to the gravity circuit. The fine cyclone overflow, at a final target product P_{80} of 100 mesh (150 µm), will flow by gravity to a vibrating trash screen ahead of the pre-leach thickener. The cyclones have been designed for a 300% circulating load. The grinding mill can operate at a pulp density range of 55%-70% solids without affecting grinding performance or water balance.

17.3.3.2 Gravity Concentration and Intensive Leach

Cyclone underflow from one operating cyclone will flow by gravity to the gravity concentrator scalping screen. With an aperture size of 10 mesh, the feed screen will remove any oversize particles prior to gravity concentration. The screen undersize will feed a semi-continuous batch gravity concentrator. Gravity recoverable gold will collect in the concentrate cone, while lower density material will flow out of the gravity concentrator tailings discharge port to the cyclone feed pump box. The scalping screen oversize will flow by gravity to the ball mill feed chute.

The gravity concentrator will be a batch operation. During a cycle, gravity recoverable gold will collect in the concentrate cone. At the end of the cycle the concentrate cone will be flushed with water, sending the concentrate to a storage hopper ahead of an intensive leach reactor (ILR). Once per day, the concentrate will be transferred from the storage hopper to the ILR for processing. After being transferred the concentrate will be pre-washed to remove slimes which are returned to the cyclone feed sump. The leach solution including concentrated cyanide, caustic solution, and the leach accelerant LeachAid will be mixed and the leach cycle will be started. At the end of the leach cycle, the pregnant solution will be recovered, and the leach residue will be sent to the cyclone feed pump box. The pregnant solution will be directed to a dedicated EW cell where precious metals will be recovered.

17.3.4 Leaching

17.3.4.1 Pre-Leach Thickening

Cyclone overflow will flow onto a vibrating trash screen for removal of trash material. Oversize material will discharge into a trash bin, while screen undersize will flow by gravity to a 68 ft diameter pre-leach thickener. Flocculant and process water will be added as needed to the thickener feed to assist in the settling of fine solids and thickening. The flocculant addition rate will be adjusted by a variable speed metering pump. The high-rate thickener will thicken the slurry to 45-50% solids and a variable speed thickener underflow pump will pump the slurry to the leach circuit. The thickener overflow will flow by gravity to the non-cyanide solution tank to be used as make-up water in the grinding circuit.

The pre-leach thickener will be mounted on steel legs on foundations in a concrete containment area with slab on grade and cast-in-place walls will contain precipitation and process spills. A sump pump will transfer the contained water back to the pre-leach thickener.

17.3.4.2 Leach/CIL

The selected hybrid Leach/CIL circuit will consist of seven agitated tanks. The precious metals are leached from the ore and adsorbed onto activated carbon that is mixed within the slurry. The first one or two tanks will operate without any carbon to improve leaching efficiency. The overall circuit will provide approximately 30 hours of total retention time at 45% solids. It is expected that the slurry will arrive at the Leach/CIL circuit at a pH of 10.5. Milk of lime may be added to the circuit to adjust pH to maintain the desired alkalinity. Sodium cyanide (NaCN) may be added to any of the tanks and its addition rate will be controlled based on cyanide analyses. Process air will be piped to all tanks to maintain dissolved oxygen set points.

Slurry will advance from tank to tank using submerged vertical carbon pumper interstage screens and will exit the last tank reporting to the CIL carbon safety screen.

The Leach/CIL tanks will nominally contain 10 to 15 g/L of 6 x 12 mesh granular activated carbon to adsorb the dissolved precious metals from the slurry. Fresh activated carbon and regenerated carbon from the regeneration circuit will be pumped into the last Leach/CIL tank daily in 3 ton batches. Loaded carbon will be sent to the desorption plant daily, also in 3 ton batches.

Slurry discharging from the last Leach/CIL tank will flow to a CIL carbon safety screen fitted with 28 mesh screen panels. The purpose of this screen is to prevent accidental losses of activated carbon.

The oversize carbon from the safety screen will be collected and sent to the regeneration circuit. Slurry that passes through the screen will be pumped using a fixed speed pump to the cyanide recovery thickener.

The Leach/CIL tanks will be uncovered and the interstage screens will be serviced by a 5 ton gantry crane. A concrete containment slab on grade and containment walls will contain precipitation and process spills. A sump pump will transfer this material back to the process. The concrete pad and containment area are sized to allow for an additional leach tank to increase leach residence time if it is required in the future.

17.3.5 Tailings Management

17.3.5.1 Cyanide Recovery Thickener and Tailing Detoxification

The CIL carbon safety screen undersize stream will report to the cyanide recovery thickener feed box. Dilution water and flocculant are added to the slurry to aid in settling which is then thickened to approximately 60% solids. Overflow solution containing cyanide from the cyanide recovery thickener is recycled to the barren solution tank for reuse in the process. The thickener will be operated to maximize underflow density and minimize the cyanide remaining in the slurry.

The cyanide recovery thickener underflow will be pumped by variable speed thickener underflow pump to the cyanide detoxification circuit, where cyanide will be destroyed using the SO_2/Oxygen process. The cyanide recovery thickener underflow will be sampled in the detox feed sampler.

In the cyanide destruction tank, residual free and Weak Acid Dissociable (WAD) cyanide will be oxidized to the relatively non-toxic form of cyanate by the SO_2/Oxygen process using sodium metabisulfite and oxygen, with copper sulfate as a catalyst as needed. An oxygen generator system supplying will be installed local to the detox tank. Milk of lime will also be added as needed to maintain a slurry pH in the range of 8.0 to 9.0. The more stable iron cyanides will be removed from the solution as an insoluble ferrocyanide precipitate. The cyanide levels will thereby be reduced to an environmentally acceptable level.

The detoxification will be accomplished in a single agitated tank that will provide a residence time of approximately 120 minutes based on the Cyanco test work report (Cyanco, 2020).

Discharge from the cyanide detox tank will be final plant tailing and will be pumped to the tailing filter feed tank.

A concrete containment slab on grade and containment walls will contain precipitation and process upsets in the cyanide recovery thickener and cyanide detoxification area. A sump pump will transfer the material back to the process.

17.3.5.2 Tailing Filtering

Detoxified slurry will be pumped to an agitated tailing filter feed tank. Slurry from the filter feed tank will be pumped using variable speed horizontal centrifugal pumps to three tailing filters (2 operating/1 standby). The tailing filters each utilize 8.2 ft (2.5 m) square mixed pack polypropylene plates (alternating recessed plates and membrane plates) that when closed provide 64 enclosed chambers for formation of filter cake. Mixed pack plates are recommended to optimize and assure moisture content specifications for material minerology characteristics expected to be processed. Each full cycle is estimated to take 16 minutes which includes: filling, filtration, air drying, cake discharge, cloth shaking and cloth rinsing. Filters will have the capability of cake squeezing and blowing if required.

The filter cake at approximately 18% moisture (by weight, wet basis) will discharge to a filter cake stockpile to be reclaimed by front end loader and loaded into articulated trucks for haulage to the filtered tailings facility east of the heap leach pad atop of the former Viceroy operations reclaimed heap leach pad. Filtrate will discharge into a tailing filtrate tank and be pumped using a fixed speed horizontal centrifugal pump back to the early stages of the grinding process for re-use.

The filters are housed within a building with siding extended down the building sides an adequate length to protect equipment from climatic conditions, specifically ultraviolet exposure. The filters will be serviced by a dedicated overhead bridge crane and crane loading/unloading area adjacent to the process and within the filtration building.

A concrete containment slab on grade and containment walls will contain precipitation, wash down and process upsets in the area. A sump pump will transfer the material back to the process.

17.3.6 Carbon Handling

17.3.6.1 Acid Wash

Each day, six tons of loaded carbon is transferred from each CIC train and three tons of loaded carbon is transferred from the CIL circuit to a 6 ton carbon storage tank at different times of the day.

Loaded carbon will be transferred from each CIC train to the loaded carbon dewatering screens sitting above the acid wash tanks. The 30 mesh screen cloth will capture most of the carbon, which then will flow into the acid wash tank while the solution returns to the CIC circuit. Loaded carbon will be transferred from the CIL circuit to the CIL loaded carbon screen where the slurry will be washed from the carbon and returned to the CIL circuit while the loaded carbon will flow by gravity to a 6 ton carbon storage tank. Every other day 6 tons of CIL carbon will be treated.

Loaded carbon from both the CIC and the CIL circuits is treated in one of two 6 ton capacity acid wash tanks constructed of fiber-reinforced plastic (FRP). The carbon will be acid washed with a circulating 3% hydrochloric acid (HCl) solution to remove inorganic contaminants (i.e., calcium deposits, magnesium, sodium salts, silica, and fine iron particles). Organic foulants such as oils and fat are unaffected by the acid and will be removed after the elution step in the regeneration circuit using a horizontal propane fired kiln.

During the acid wash cycle, HCl solution will be pumped from the acid mix tank upward through the acid wash vessel, overflowing back into the acid mix tank. The carbon will then be rinsed with a caustic solution to neutralize any residual acid. After neutralization, the carbon will then be rinsed with fresh water to remove the remaining acid and any mineral impurities before transferring to the strip vessel.

A recessed impeller pump will transfer the acid washed carbon from the wash vessel into the strip vessel using transport water. The carbon transfer water comes from the closed circuit carbon transfer water system. Carbon slurry will discharge directly into the top of the strip vessel. Under normal operation, two acid wash and two strip cycles will take place per day. Every other day, a third acid wash and strip cycle will take place.

A concrete containment slab independent of the carbon strip area containment slab on grade and containment walls will contain precipitation and process spills in the acid wash area. A sump pump will transfer the material back to the process.

17.3.6.2 Stripping (Elution)

The carbon stripping process will be a pressure Zadra circuit utilizing barren strip solution to strip (desorb) the loaded carbon. Barren strip solution is prepared in the strip solution tank by adding cyanide, caustic (NaOH), antiscalant and water as needed. Gold will be removed from the carbon by circulating 290ºF caustic cyanide solution upward through the partially fluidized bed of loaded carbon, creating a pregnant gold and silver solution which will be pumped through the electrowinning cells for precious metal recovery. The solution exiting the electrowinning cells will be circulated back to the strip solution tank for reuse.

More specifically, the strip vessels will be stainless steel tanks, each with a capacity to hold approximately six tons of carbon. The loaded carbon from the acid wash circuit will be pumped into the top of the strip column and the excess water will be drained to the floor sump and returned to the process using a sump pump.

After the complete batch of carbon has been transferred, the strip cycle will be initiated by pumping solution containing approximately 1.25% (25 lb/ton) NaOH and 0.15% (3.0 lb/ton) NaCN from the strip solution tank through two heat exchangers, which will raise the temperature to 290ºF, into the bottom of the strip vessel.

After rising through the bed of carbon, the solution exiting the top of the vessel will be cooled below its boiling point by passing through the heat recovery exchanger prior to discharging to the EW distribution box. Heat from the outgoing solution will be transferred to the incoming cold barren solution prior to passing through the solution heater. The hot side of the final heat exchanger will be piped to a propane fired thermal fluid heating system.

Approximately 10 Bed Volumes (BV's) at a rate of 2 BV/h will be passed through the carbon to remove all precious metals. A Bed Volume is the volume of solution that occupies the space in the vessel when containing carbon. A final 2 BV of hot water will be used to wash the carbon at the end of the stripping cycle.

After the stripping circuit has been cooled down, the depleted carbon will be transferred with water to the reactivation circuit using a horizontal recessed impeller pump. The strip cycle will be complete in approximately 10 hours, allowing additional strips as needed.

Ammonia and mercury fumes from the strip solution tank will be combined with exhaust from the electrowinning cells and will be collected in a mercury abatement system.

A concrete containment slab on grade and containment walls, independent of the acid wash area, will contain precipitation and process spills in the carbon strip area. A sump pump will transfer the water back to the process.

17.3.6.3 Carbon Regeneration

The carbon regeneration circuit will thermally regenerate the stripped carbon, reactivating the pores and removing any organic foulants, such as oils and fats. Fresh activated carbon will be added to account for any carbon lost during the adsorption, desorption and regeneration processes.

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the kiln dewatering screen. Oversize carbon from the screen will discharge by gravity into the kiln feed bin. Screen undersize carbon, containing carbon fines and water, will drain by gravity into the carbon fines tank.

An indirect propane fired horizontal carbon regeneration rotary kiln will be utilized to treat 18 tons of carbon per day, equivalent to 100% regeneration of stripped carbon. The kiln will be heated to temperatures as high as 1,500ºF to remove organics from the surface of the carbon. The regenerated carbon from the kiln will flow by gravity into the carbon quench tank where it is cooled by fresh water and/or carbon fines water and is pumped over a carbon dewatering screen. The dewatering screen doubles as a dewatering screen and a carbon sizing screen where fine carbon particles will be removed. The coarse carbon particles in the screen oversize will drop into the regenerated carbon storage tank. Regenerated carbon will be pumped back to the carbon columns. The fine carbon particles and the bulk of the transfer water will pass through the screen and flow to the carbon fines tank.

Kiln off gas will be collected through ductwork and passed through a scrubber containing sulfur impregnated carbon to remove mercury vapor from the process gas before discharging to atmosphere.

Periodically, new activated carbon will be added to the system to replenish fine carbon losses. Carbon may be added to the agitated carbon attrition tank. The agitating action quickly wets the surfaces of the carbon and attritions the carbon particles to break up any lumps and remove loose edges.

Carbon fines from the carbon fines tank are recovered using a plate and frame filter to produce damp carbon cake. The filter cake may be sold for its metal or thermal value.

A concrete containment slab on grade and containment walls will contain process upsets in the carbon regeneration area. A sump pump will transfer the material back to the process.

17.3.7 Electrowinning, Retorting and Refining

Pregnant solution from the strip circuit will be pumped to the refinery for electrowinning (EW), producing a gold and silver sludge. The sludge will then be filtered, dried and refined in an electric induction furnace producing gold and silver doré bars.

The pregnant strip solution produced in the carbon strip circuit is sent to EW cells to recover the precious metals. EW is used to recover the precious metals from the pregnant solution. It is an electrolytic process where the precious metals are recovered from the solution by passing direct electrical current between electrodes (anodes and cathodes) immersed in the solution. As the current passes from the anode to the cathode, the precious metals loosely plate onto the cathode as a sludge.

17.3.7.1 Electrowinning

Electrowinning is accomplished in four EW cells operating in parallel. Each cell contains anodes (stainless steel punched plate) and cathodes (stainless steel mesh held in place by stainless steel bayonets and wire frames and suspended in cross linked polyethylene baskets). Each cell has a DC rectifier capable of delivering a current of 0 to 2000 amps at 0 to 9 volts.

The flow rate of pregnant solution through each cell is approximately 50-75 gpm. During electrowinning, the solution flows by gravity to the EW discharge box. From there, the EW discharge pump delivers the solution to the strip solution tank. To prevent a build-up of impurities, a 15% daily bleed of barren solution will be pumped to the Leach/CIL circuit.

The sludge from the carbon circuit and ILR will be periodically washed off the cathodes using high pressure water and recovered as a damp cake in a plate and frame filter press. The filter cake will be dried in a retort furnace prior to smelting.

Vapor from the barren strip solution tank and the electrowinning cells will pass through a sulfur impregnated carbon scrubber before discharging a clean off-gas to the atmosphere.

17.3.7.2 Retorting

Filtered cake will be loaded into retort boats, or pans, for treatment in a retort furnace. The pans will be placed in a mercury retort system for several hours. The retort will heat the filtered cake to approximately 1,200ºF to vaporize mercury, which may be present in low concentrations. Retort vapor will be withdrawn from the retort by a vacuum pump, which will pull the vapor through a condenser where the mercury will condense and flow into a mercury collection compartment/tank. Mercury will be recovered from the tank periodically. Exhaust from the retort vacuum pump will pass through a sulfur impregnated carbon (SIC) filter before venting to atmosphere.

The retort, mercury condenser system, mercury collection tank, vacuum pump, and carbon filter will be supplied as a complete packaged system ready for utility hook-up and operation.

Following the retort cool down cycle, the dry sludge will be manually loaded into a furnace charge mixer along with fluxing material. The blended material will be manually charged to an electric induction furnace.

17.3.7.3 Refining

The dried filter cake (gold sludge) will be processed, along with flux, in an electric induction melting furnace. When the sludge and flux mixture become fully molten, the components will separate into two distinct layers: slag (on the top) and metal (on the bottom). The slag layer, containing most of the impurities, will be poured-off first into a conical slag pot. The remaining molten metal, containing the precious metals and minor impurities, will then be poured into bar molds. Doré will be sampled using vacuum tubes during pouring.

After cooling and solidifying, the metal bar (doré) will be dumped from the mold and slag will be knocked off by hand. The resulting doré bar will be further cleaned of residual slag and finished as required with a needle gun. The cleaned bars are then weighed and stamped with an I.D. number and weight. Doré bars, each weighing a total of approximately 35 to 40 pounds, will be the final product of the operation and will be stored in a vault awaiting shipment. This process will take place within a secure and supervised area, and the precious metal product will be stored in a vault until shipped off site.

Slag will be collected and returned to the ball mill.

Fumes from the melting furnace will be collected through ductwork and cleaned in a high temperature bag house dust collector before discharging through an exhaust control system, a vessel with sulfur impregnated carbon, to the atmosphere.

The refining building will be enclosed by concrete block walls with a steel framed roof and metal roofing. Water used for cleanup and any spills will be collected and pumped back into the process.

17.3.8 Reagents and Grinding Media

Reagents used within the plant will be mixed on-site and distributed via the reagent handling systems. These reagents include:

- Lime (CaO)
- Sodium cyanide (NaCN)
- Hydrochloric acid (HCl)
- Leach Aid
- Caustic soda (NaOH)
- Activated carbon
- Sodium metabisulfite (SMBS)
- Copper sulfate ($CuSO_4$)
- Flocculant
- Antiscalant

All reagent areas will be contained with sump pumps which will transfer any spills to their respective storage tanks. The reagents will be mixed, stored and then delivered to the heap leach, mill, Leach/CIL, acid wash, elution and cyanide destruction circuits. The capacity of the mix tanks will be sized to handle one day of production and the storage tanks will be sized to handle one and one-half days of production.

17.3.8.1 Cyanide (NaCN)

Sodium cyanide briquettes will be delivered to site by 20 ton (40,000 lb) isotainer trucks. The briquettes will be dissolved by circulating barren solution through the ISO container and transferring the dissolved cyanide into the heap leach cyanide dissolving tank and subsequently to the heap leach cyanide storage tank or to the mill cyanide storage tank. Caustic soda (sodium hydroxide) will be added as needed during dissolution to provide protective alkalinity. Cyanide will be mixed to a 24% solution concentration for use in the process and will be added to the ore on the heap leach pad and to ore in the Leach/CIL tanks to leach gold and silver. Cyanide solution will also be used to promote the removal of gold and silver from the carbon in the carbon stripping procedure.

The concentrated cyanide solution will be added to the barren solution pump box or the barren solution surge tank at a rate of 0.60 lb/ton of ore for the heap leach. The concentrated cyanide solution will be added to the intensive leach reactor and the Leach/CIL tanks at a rate of 0.05 lb/ton and 1.25 lb/ton ore, respectively. Cyanide will be used in the carbon strip circuit at a concentration of 0.2%.

17.3.8.2 Lime (CaO)

Quicklime will be delivered to the site in bulk by trucks. The bulk trucks will be pneumatically offloaded, using a blower, directly to two 100 ton cone bottom lime silos for use at the heap leach and one 100 ton cone bottom silo for use at the mill. The silos will be equipped with a bin vent type dust collector. The lime specification is 90 to 100% CaO.

Lime will be used in the ROM ore leach system for pH control. Lime will be metered from a silo into a clamshell by a screw conveyor. The clamshell will discharge the lime directly into each ore truck hauling ore to the ROM leach pad. The lime will be delivered at a rate of approximately 2.6 lb/ton of heap leach ore.

Lime will also be used in the mill for pH control. Lime will be fed from a silo into a detention slaker by a screw conveyor. Process raw water will be used to slake the lime to an 18% concentration of milk of lime (MOL). Slaked lime will discharge to the milk of lime mix tank and will be transferred to the milk of lime distribution tank. Lime will be delivered at a rate of approximately 3.0 lb/ton and 0.20 lb/ton (1g $Ca(OH)_2$/g CN_{WAD}) of mill feed ore to Leach/CIL and Detox circuit, respectively .

17.3.8.3 Hydrochloric Acid (HCl)

Hydrochloric acid will be delivered in bulk trucks at 30 to 33% concentration and will be offloaded directly into a hydrochloric acid storage tank. Acid will be pumped from the tank to the carbon acid wash circuit when needed.

Hydrochloric acid will be used in the carbon handling circuit to acid wash carbon. Raw water will be used to dilute the HCl from approximately 33% solution to 3 to 5% solution in the acid mix tank.

17.3.8.4 Sodium Hydroxide (Caustic Soda) (NaOH)

Sodium hydroxide (caustic soda or caustic) will be delivered to site in bulk trucks at 50% concentration and will be offloaded directly into a caustic storage tank.

Caustic soda solution will be used in the carbon stripping area to neutralize acidic solutions after acid washing the carbon and as reagent in the carbon elution process. In addition, caustic will be added as needed to the cyanide mixing system to maintain the proper pH of the cyanide solution thus preventing the generation of hydrogen cyanide gas.

17.3.8.5 Activated Carbon

Activated carbon (6 x 12 mesh) will be delivered in 1,100 lb bulk bags. The bulk bags will be stored on pallets. The carbon will be introduced into the carbon attrition tank where it will be slurried with water and conditioned to remove the friable edges of the carbon particles and the adhering carbon dust generated in transport. The slurry will be pumped over the carbon sizing screen with the coarse carbon particles added to the CIC and CIL circuits as needed. Carbon consumption is expected to be approximately 0.02 lb/ton heap leach ore and 0.11 lb/ton mill feed ore.

17.3.8.6 Sodium Metabisulfite (SMBS)

Dry SMBS will be delivered in 2,000 lb bags and will be stored on pallets in a reagent building. SMBS will be mixed using raw water in sodium metabisulfite mixing tank. After mixing is complete, the solution will be transferred to the sodium metabisulfite distribution tank. SMBS will be mixed to a 20% solution strength. SMBS will be pumped to the detox circuit using a metering pump. SMBS will be delivered at a rate of approximately 1.21 lb/ton of mill feed ore (4 g SO_2/g CN_{WAD}).

SMBS will be added to the tailing detoxification circuit as the primary source of SO_2 to oxidize free cyanide and weak acid dissociable (WAD) metal cyanide complexes (SO_2/air process).

17.3.8.7 Copper Sulfate ($CuSO_4$)

Dry copper sulfate will be delivered in 50 lb bags and stored in a dry area. The copper sulfate system will include an agitated mixing tank and a storage tank. Copper sulfate will be mixed to a 10% solution concentration. Copper sulfate will be added at 0.08 lb/ton ore (0.2 g $CuSO_4$/g CN_{WAD}), to the cyanide destruction circuit as needed to catalyze the reaction.

Copper sulfate will be pumped to the detox feed box tank using a metering pump.

17.3.8.8 Antiscalant

Antiscalant agents are used to prevent the build-up of scale in the process solution and heap irrigation lines. Antiscalant will be delivered in 250 gal totes supplied by the vendor. Totes will be stored on pallets in a containment area at the reagent storage building/warehouse.

Antiscalant will be added at the point of use to various clear water pump suctions using metering pumps directly coupled to suppliers' tote boxes to prevent scaling.

Antiscalant will be added to the suction of the barren solution pumps to mitigate scaling in the barren lines and drip emitters. Antiscalant will be used during the stripping process to prevent scaling of the heat exchanger plates.

17.3.8.9 Flocculant

Powdered flocculant will be delivered to site in 50 lb bags. Two vendor packaged mixing and dosing systems will be installed, one at each thickener, which will include a flocculant storage hopper, screw feeder, blower, wetting head, and mixing tank. Flocculant is mixed with raw water to produce a 0.1% mix strength. Flocculant solution will be aged in the flocculant mixing tank for a pre-set period before transfer to the flocculant storage tank for dosing to the thickener. Flocculant will be diluted at the point of use with raw water to a 0.01% strength.

The flocculant area will be serviced by a sump pump. Spillage generated within this area will be pumped to the thickener feed box.

17.3.8.10 Miscellaneous Reagents

Other chemicals such as dust suppressants will be received in totes and Leach Aid will be received in pails or other approved containers and stored as directed. In addition, fluxes (silica, nitre and borax) are required for smelting charge preparation and will be received in 50 lb bags and stored in a dry area at the reagent storage building/warehouse.

17.3.8.11 Consumption Rates

Heap leach and mill reagent consumption rates are based on metallurgical testwork.

Liners for the crushers and grinding mill have been estimated based on ore hardness and on experience at similar operations. Grinding media consumption for the grinding mill has been estimated on a lb/ton basis, mill power draw and abrasion index.

Table 17-7 summarizes the reagents used in the process plant and their estimated daily consumption rates. The table also includes other major process consumables.

Table 17-7: Summary of Process Area Consumables

Reagents	Consumption (lb/ton ore)	Package	Frequency	Storage Days
Heap Leach				
Pebble Lime; 90% active	2.58	20 ton Bulk truck	4 trucks/day	3
Sodium Cyanide	0.60	20 ton iso container	1 truck/day	4
Activated Carbon	0.02	1,100 lb. Super sack	40 bags/month	-
Antiscalant	0.04	20 totes per truck; 250 gal Tote	2 trucks/month	38
Mill				
Pebble Lime; 90% active	3.21	20 ton Bulk truck	Included above	7
Sodium Cyanide	0.47	20 ton iso container	Included above	*
Activated Carbon	0.11	1,100 lb. Super sack	Included above	-
Antiscalant	0.03	20 totes per truck; 250 gal Tote	Included above	-
Sodium Hydroxide	0.17	5,000 gal tanker	1 tanker/every 4 days	4
Flocculant	0.14	40 bags per pallet; 50 lb. bag	8 pallets/month	33
Hydrochloric Acid	0.10	5,000 gal tanker	1 tanker/every 3 days	3
Leach Aid (Est.)	0.003	-	-	-
Sodium Metabisulfite	1.21	20 supersacks per truck; 2,000 lb. Supersack	1 truck every 10 days	10
Copper Sulfate	0.08	40 bags per pallet; 50 lb. bag	5 pallets/month	36
Primary Crusher Liners	0.061	-		-
Secondary Crusher Liners	0.057	-		-
Grinding Mill Liners (Rubber)	0.08	-		-
Grinding Balls – 3 inch chrome steel	1.4	Direct dump to concrete bunker		-

* Sodium Cyanide solution for the mill will come from the heap leach storage system.

17.3.9 Process Water Balance

A water balance was developed for the Castle Mountain process facilities to characterize water flows within the process plant and heap leach to understand monthly usage rates. This results in determining the water available for recycling throughout the process and make-up water requirements. Make-up water needs will fluctuate with the amount of evaporation and precipitation events seen in each month specifically for the heap leach.

ROM ore is assumed to be relatively dry with a moisture content of 3%, while the saturation moisture of the ore is expected to average 10% and must be sustained for solution to begin flowing from the pad. This results in an average consumption of 668 gpm.

Climatic data, discussed in Section 18.5.1, is utilized in accordance with required irrigation moisture and the staged development of the lined heap leach pad to characterize the water required to keep the heap leach properly irrigated and process solutions balanced within the process plant. A summary of anticipated evaporation losses through pan evaporation on the leach pad surface and from emitters as well as losses associated with ore wetting is shown below in Table 17-8.

Table 17-8: Summary of Heap Leach Water Demands

Month	Ore Wetting (gpm)	Net Evaporation* (gpm)	Total (gpm)
January	668	158	826
February	668	209	877
March	668	326	994
April	668	476	1,144
May	668	593	1,261
June	668	693	1,361
July	668	684	1,352
August	668	558	1,226
September	668	489	1,157
October	668	334	1,002
November	668	203	871
December	668	146	814
Average	**668**	**406**	**1,074**

* Net Evaporation is the combined effect of evaporation losses and precipitation gain.

In addition to the heap leach, the processing plant also requires make-up water to operate. Make-up water may be introduced to various locations within the process including the non-cyanide solution tank, acid wash circuit, tailing filter water tank, cyanide recovery thickener feed tank as well as for mixing of dry reagents. Water will also be used throughout the plant as seal water for pumps. Water is collected and recycled throughout the plant via area sump pumps. The average make-up water requirement of the mill is 110 gpm. See Table 17-9.

Table 17-9: Mill Water Demand Summary

Area	Percent Solids		Water Addition (gpm)	Water Recycle Flows (gpm)
	In	Out		
Crushing	97	96.5	3	-
Grinding	96.5	30	1338	-
Pre-Leach Thickener	30	45	-	647
CN Recovery Thickener	45	60	-	324
Tailings Filters	60	82	-	260
Total	**97**	**82**	**1341**	**1231**
Total (Make-Up)			**110**	

Raw make-up water will be pumped from several sources, including historic and newly developed well sites. The raw water for the process plant will be primarily pumped to the existing process area raw/fire water tank for gravity distribution to the process plant. Make-up water sourcing and sitewide water management which includes usage for dust suppression, mining and other uses is further discussed in Section 18.5.

17.3.10 Air Supply and Distribution

Air compressors and receivers will provide air for operation and maintenance at the primary crushing area, the fine crushing area, and fine ore reclaim area. Plant air compressors will be located in a common Air Systems building central to the mill facility.

A plant air compressor will provide service and instrument air throughout the process plant. An air dryer will remove moisture in instrument air. Plant air and instrument air receivers will be provided. A dedicated compressor will provide air to the bottom of the Leach/CIL tanks.

An oxygen generator located at the tailing facility will provide oxygen to the cyanide destruction tank.

Tank mounted reciprocating air compressors will be installed for operation and maintenance at the truck shop and at the mill maintenance building.

17.3.11 Electrical Power Requirements

The estimated average running power load for process facilities is shown in Table 17-10. Electrical power supply is discussed in Section 18.7.1.

Table 17-10: Summary of Process Area Power Consumption

Code and Area Description	Estimated Running Load MW
Area 220 - Solution Handling	2.4
Area 280 - Carbon Adsorption	0.1
Area 340 - Reagents	0.2
Area 410 - Primary Crushing	0.2
Area 420 - Fine Crushing & Crusher Ore Storage	0.6
Area 430 - Grinding	2.2
Area 440 - Gravity Concentration	0.2
Area 450 – Leach/CIL	0.5
Area 460 - Tailing Handling and Filtration	0.8
Area 510 - Desorption and Carbon Regeneration	0.2
Area 520 - Electrowinning and Refinery	0.5
Area 620 - On Site Water Supply and Distribution System	0.2
Area 630 - Air Systems	0.2
Area 680 - Fuel Infrastructure	0.1
Total	**8.4**

17.3.12 Sampling

Samplers will be installed where samples are required for metallurgical accounting and process control purposes. Installation location and type of sampler are listed in Table 17-11.

Table 17-11: Sampler Location & Type

Location	Sampler Type
Mill Feed	Belt crosscut
Leach/CIL Feed	Pipe/Vezin
Detox Feed	Pipe/Vezin
Detox Discharge	Pipe/Vezin
Barren Solution to Pad	Pipe
CIC Feed (pregnant solution)	Wire
CIC Discharge (barren solution)	Wire
Electrowinning Feed (pregnant eluate)	Wire
Electrowinning Discharge (barren eluate)	Wire

These samplers will be installed in conjunction with totalizing flow meters and, where necessary, density meters to allow process control and allow a full plant gold balance to be completed.

In addition to the samplers above, sample ports will be available for equipment (e.g. electrowinning cells) to allow for manual grab samples. Access to equipment (e.g. carbon columns, cyclone) will allow for operators to grab samples as required.

17.4 INSTRUMENTATION AND CONTROLS

The key design criteria for the instrumentation and controls are to provide and implement sufficient supervisory and control to achieve design production rates, to enable stable process operations within design limits and to facilitate safe operation of all process and equipment.

Equipment and process will primarily be monitored and controlled remotely from a new central control room located at the grinding building, new satellite control room situated in the crusher facilities, and the existing control room from Phase 1. The control room operator will be able to input set points, open/close valves, start/stop motors/pumps/conveyors/equipment and visualize all alarms and interlocks via the process control systems human machine interface system.

Equipment and process parameters will be monitored and automated when it is deemed critical for process productivity and quality or is required to support human, equipment or environmental safety functions. Equipment will be field operated, where it is only required for infrequent actuation or activation with no significant impact on process, equipment or safety.

Intelligent type motor control centers will be located in electrical rooms throughout the facilities. A digital communication interface to the process control system will facilitate remote operation and monitoring of motor control center equipment. Field instrumentation and devices will be hardwired to the process control system via input and output modules.

The site-wide process communication system will be an industrial Ethernet CAT6a cable and fiber optic network, providing communication between the process controllers, the motor control centers, remote input and output modules, vendor supplied skids, the control room(s) operator's workstations, and graphical interface consoles.

The following areas and associated process will be monitored and controlled:

- Comminution and conveying
- Leach/CIL
- Adsorption, desorption, regeneration
- Cyanide detox
- Reagents
- Process and fire water
- Process and instrument air
- Dewatering, filtering, and tailing disposal

18 PROJECT INFRASTRUCTURE

Existing infrastructure at Castle Mountain is well established with the current Phase 1 operations described in Section 5. The Phase 2 expansion will continue to utilize historic facilities and the recently built Phase 1 facilities to the greatest extent possible. Phase 2 infrastructure will increase in size to meet the expanded project parameters and include new site improvements to support the operation of the required new process and mining facilities. This section describes the new facilities required for Phase 2.

18.1 ACCESSIBILITY

18.1.1 Off-site Access Roads

The Castle Mountain Project is accessed primarily from Nevada State Route 164, also referred to as Nipton Road. The Nipton Road connects the town of Searchlight, NV to Interstate 15 in California via the town of Nipton. 7 miles Northwest of the town of Searchlight, NV is a signed access to Walking Box Ranch which also acts as site access for the Castle Mountain Project. From this point, a well maintained 25 to 30 foot wide dirt and gravel access road brings traffic 18.1 miles to the main access gate for Phase 1 operations.

Phase 2 operations will continue to utilize the same main access road with no further upgrades needed. Phase 2 will have a new access point/gatehouse which is located approximately 17.2 miles from Nipton Road.

The project overall site plan is shown in Figure 18-1.

18.1.2 Security Gatehouse

The security gatehouse is a 75 ft x 20 ft pre-engineered steel building with a security check-in space, a safety training area, offices, restrooms and a medical emergency area. Covered parking is included for housing of site medical transport. Other features include a helipad for emergency evacuation and a truck scale.

Entry is permitted with key cards. Light vehicle parking is provided offsite, immediately adjacent to the gatehouse for visitors, and truck parking is provided to stage deliveries prior to inspection and weigh in at the truck scale. The truck scale currently being used on site for Phase 1 will be relocated to the new gatehouse location with the Phase 2 expansion.

Mine and process plant staff will primarily report to site via mine site bussing from Searchlight, Nevada. Adequate space for safe and efficient loading and unloading is accounted for in the gatehouse staging area. Project personnel will be expected to change into work attire prior to arrival at the bus pick up point as no change house is included within the project plan.

18.1.3 Site Fencing

The project site will be completely fenced with 25,000 linear feet (LF) of a combination of 6 ft and 8 ft high chain link fencing surrounding the main process facilities, expanded heap leach pad and new filtered tailings facility. In addition to chain link fencing, 25,000 LF of desert tortoise fencing will be installed on a similar linear route as the chain link in all areas at elevations less than 4,400 ft amsl. 32,000 LF of expanded wire ranch fencing with appropriate signage will be placed around exterior boundaries bordering the pit and waste areas.

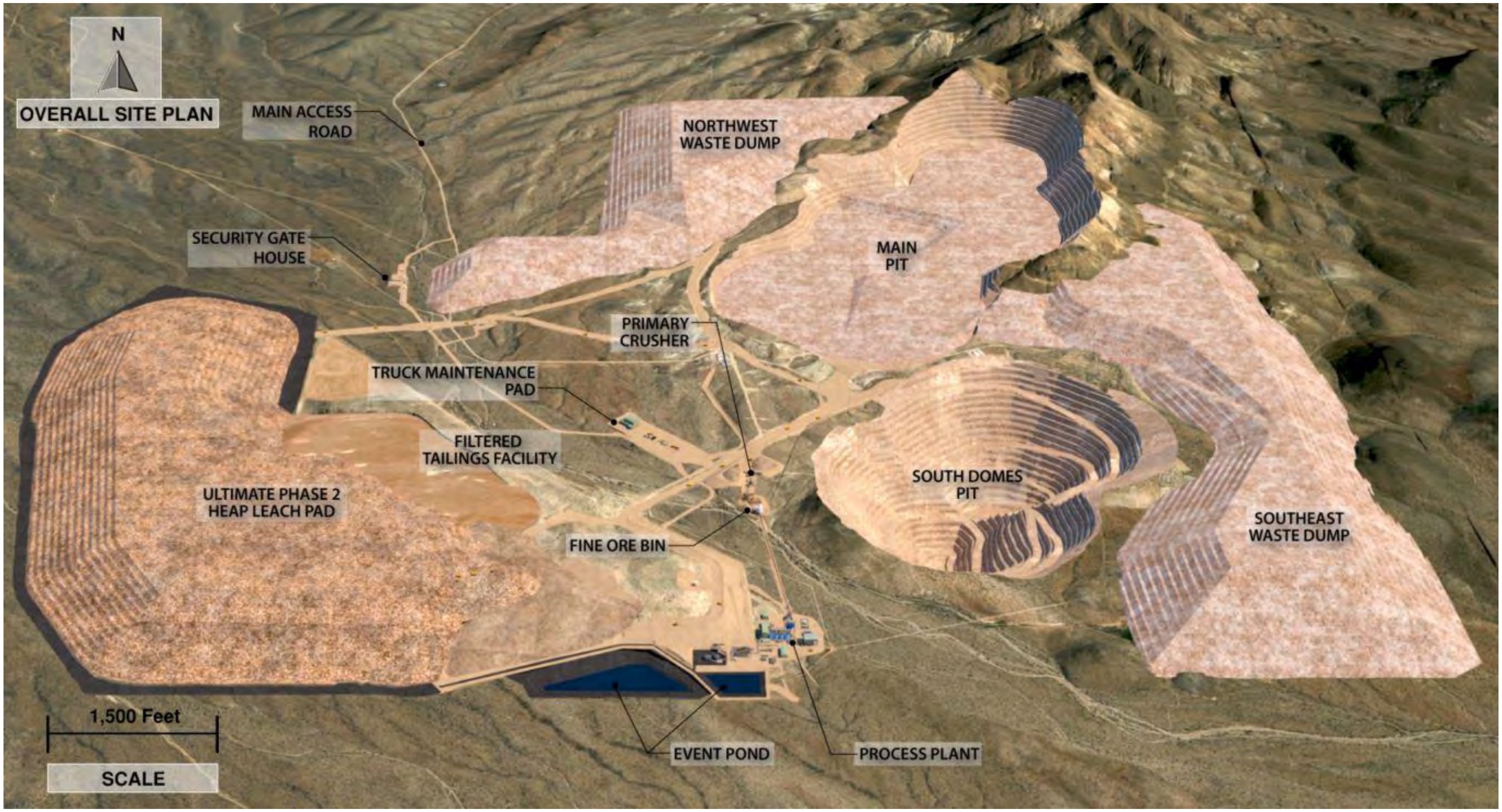


Figure 18-1: Overall Project Site Plan

18.1.4 On-site Access Roads

Upon entry through the gatehouse, the main on-site access road is designed to allow light vehicle traffic and truck deliveries to enter and have minimal exposure to the mining fleet bringing ore from the pits to the heap leach pad. The new northern haul road which connects the mine to the northern extents of the expanded Phase 2 leach pad area will cross over above the main on-site vehicle access road by means of a 40 ft wide corrugated multiplate tunnel crossing. The Phase 1 haulage corridor to the south will continue to operate in Phase 2 with the existing access road crossing; however, additional access roads on site have been added in a way to minimize interactions between haul traffic and light vehicles or truck deliveries.

18.2 SITE DEVELOPMENT

18.2.1 Mill Siting Study

As part of the feasibility study, four options were short-listed and considered for the location of the processing facilities including the pre-feasibility design with a centrally located mill and a tailings impoundment north of the heap. The effort focused on reducing capital and operating costs as well as improving operations and maintenance functionality.

The pre-feasibility design basis was determined to have higher capital and operating cost requirements partly due to the use of mechanical conveyance to deliver tailings uphill to the north side of the new expanded heap leach facility.

Of the other options, locating the mill adjacent to the current Phase 1 plant made the most sense, allowing for direct transfer of carbon between CIL/CIC to the desorption plant without needing to pump pregnant solution a substantial distance uphill. Operations, maintenance, traffic flow and security will also benefit by having a single consolidated processing plant on the southern end of the property. These advantages led to selecting the current general site plan layout, as shown in Figure 18-2, which illustrates the Phase 2 expansion facilities with respect to currently operating Phase 1 facilities.

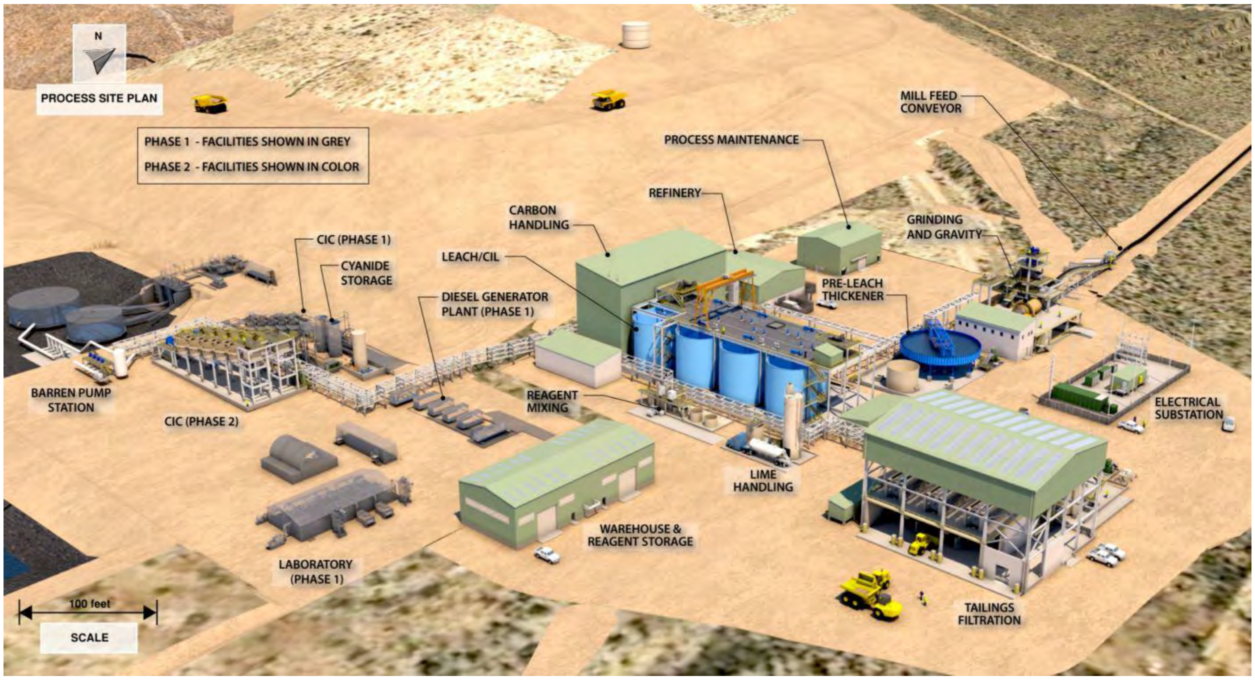


Figure 18-2: South Processing Plant 3D View Looking Northwest

The selected site plan was developed with most facilities located in close proximity to each other as well as the Phase 1 facilities to provide a compact layout which is easily accessible for operators and maintenance personnel. Emphasis was placed on minimizing new disturbance in the existing watershed/arroyo areas to allow for adequate natural drainage.

18.2.2 Topsoil Handling and Stockpile

Project topsoil will be stockpiled in designated topsoil reserve areas over the course of Phase 2 expansion development. Areas will be designated around the perimeter of the Phase 2 heap leach pad extents for topsoil removed as a result of the expanded heap leach pad while topsoil from the crushing plant, process plant and filtered tailings facility will primarily report to local areas in close proximity to their source near the South process plant. It is estimated that the expansion project initial heap leach pad, filtered tailings and plant area will generate approximately 425,000 yd^3 of topsoil which will be stockpiled for utilization during final reclamation.

18.2.3 Surface Plant Area Geotechnical Investigation

Two site geotechnical studies on site at Castle Mountain were completed in order to determine key project foundation design criteria, excavation characteristics and general recommendations for site earthworks in site areas specific to development of Phase 1 as well as facilities associated with the Phase 2 expansion.

A total of 29 test pits and 3 geotechnical borings were conducted in 2019-2020 to characterize project subsurface characteristics and how they may affect design and construction relating to facilities on site. These test pits and borings were specifically located in anticipated locations of major process facilities. Samples were collected from the boring and test pits and tested for expansion index, soil gradation, soil pH, resistivity and soluble contents. Based upon the laboratory data and field evaluation, no geotechnical fatal flaws were identified. The local soil materials were characterized as suitable for construction materials for the various facility components.

The following contains a general summary of the findings of the investigation relating to Phase 2 facility locations:

- **Phase 2 North Site (Crushing and Ore Storage)** – This area was underlain by a relatively thin mantle of Colluvium over Tertiary Rhyolite Tuff in various stages of weathering. The Colluvium was described as dark yellowish brown, dry, loose silty sand with localized clayey areas, angular gravel and cobble-size volcanic fragments to a maximum of 7 inches in diameter. This layer ranged in thickness from 18 in to 30 in. Underlying Volcanic Rhyolite Tuff (considered the bedrock unit in this area) was encountered below these depths, of which, the top 24 in to 30 in was generally rippable with an excavator bucket.

- **Phase 2 South Site (Mill, Processing Plant and Refinery)** – This area was underlain by a thicker section of Alluvium and a relatively thinner mantle of Colluvium over Tertiary Volcanic Rhyolite Tuff in various stages of weathering. Test pit 5, located near the proposed process plant facility, showed an Alluvium characterized as dark yellowish brown and dry. Loose fine to coarse silty sand with sub-angular gravel and cobble-size volcanic fragments up to 6 inches in size was encountered to the total depth explored in this area of 9.5 ft. In the proposed location of the grinding mill, a Colluvium characterized as dark yellowish brown, soft clay with a trace of sand and angular volcanic fragments to 6 in was encountered to a depth of approximately 3.5 ft followed by Volcanic Rhyolite Tuff which was rippable to an additional 5 ft in depth.

Test pits sampled for characterization of Phase 2 facilities are shown in Figure 18-3.

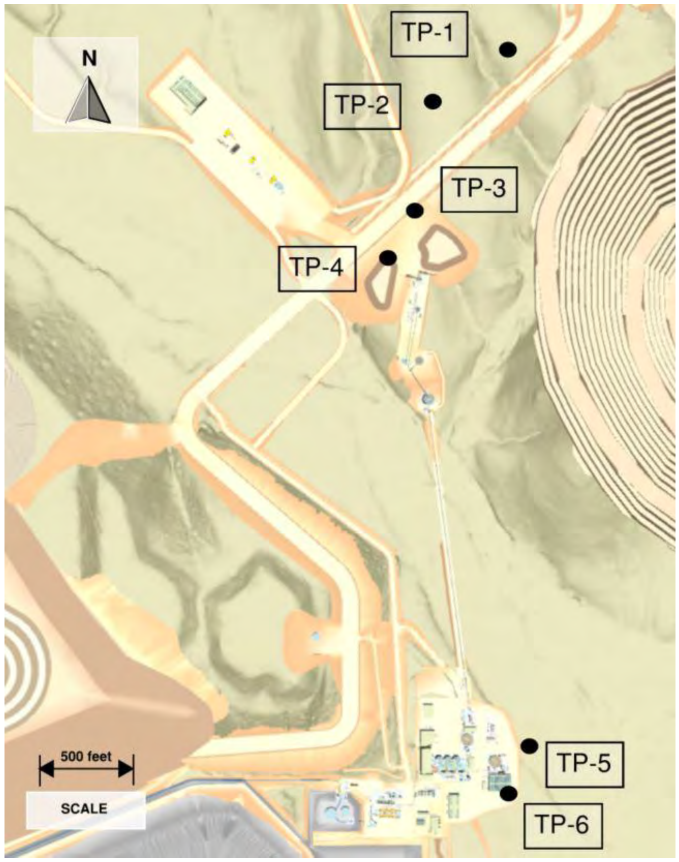


Figure 18-3: Phase 2 Test Pit Map (Process Plant)

18.2.4 Mine Geotechnical Investigation

Mine geotechnical recommendations have been provided by Call and Nicholas, Inc. (CNI) and were discussed in Section 15. (Call and Nicholas, 2020). As part of the wall slope recommendations, CNI recommended a crest offset of 300 ft (1.2 factor of safety) for critical infrastructure constructed on the western side of the South Domes pit. Stability analyses indicate that location and orientation of the Maverick fault could potentially lead to instability within 300 ft of the crest.

Crushing infrastructure, conveyors and fine ore storage all located on the western side of the South Domes pit adhere to this offset as the secondary crusher is the closest plant infrastructure and is at least 500 ft away from the ultimate pit edge.

18.3 MINE INFRASTRUCTURE

18.3.1 Truck Shop

The truck shop is a 185 ft x 100 ft shop facility and parts warehouse for maintenance of the property's mobile fleet. The truck shop includes 3 truck bays, 10 ton capacity bridge crane, tool crib, electrical shop and a second-floor mezzanine level with office space, restrooms and conference area. The truck shop has translucent panels to reduce lighting requirement.

18.3.2 Truck Fuel Storage and Dispensing

Phase 1 fuel storage and dispensing facilities will be relocated to the new truck shop pad and a new 20,000 gal off road diesel storage tank will be added to account for higher fuel consumption under expanded project requirements. Phase 1 on road diesel and gasoline equipment and storage will be relocated to the Phase 2 truck shop pad and are assumed to be substantial enough to service the expanded project needs.

18.3.3 Tire Pad

The tire pad is a 60 ft x 50 ft concrete pad specifically designed to allow for efficient change out and servicing of haul fleet tires. The pad includes a small tool room and electrical room as well as compressed air and nitrogen for haul tires.

18.3.4 Truck Wash

The truck wash is a 65 ft x 118 ft facility designed to allow for efficient washing of the property's mobile fleet. The truck wash includes a loader accessible sediment clean up bay, sludge drying pad, overflow basins and oil skimming system.

18.4 PROCESS PLANT INFRASTRUCTURE

18.4.1 Administration and Offices

Existing administrative and mine office facilities set up for current Phase 1 operations will continue to be utilized in similar capacity. These facilities consist of a modular office complex including office spaces and a conference area.

18.4.2 Process Maintenance Building

The process maintenance building is a 50 ft x 69 ft pre-engineered steel building intended to house maintenance activities associated with process equipment on site. The building is home to a 10 ton capacity bridge crane as well as a small parts and tool storage room and office space for maintenance supervision and technicians and restrooms.

18.4.3 Warehouse and Reagents Storage Building

The warehouse and reagents storage building is a 54 ft x 132 ft pre-engineered steel building housing 2,900 ft^2 of warehouse space and 2,900 ft^2 of storage space designated for process reagents. The building also includes secure storage areas, restrooms and a second-floor mezzanine with office space and a small conference area.

18.4.4 Laboratory

The existing lab facilities for Phase 1 are outfitted for assay work as well as metallurgical work and will continue to be utilized in similar capacity for Phase 2. The lab facilities have been sized to allow for the additional samples generated with the expanded operation and the addition of lab equipment with respect to the high-grade mill.

18.5 SITE WIDE WATER BALANCE AND SUPPLY

A site wide water balance was developed for the Castle Mountain property to use in planning for additional raw water. This analysis considers multiple scenarios and climatic conditions for evaporation and precipitation in wet and dry conditions. Castle Mountain is a zero discharge facility with the main water loss occurring via evaporation from the surface of the heap leach pad and filtered tailings facility. Water is also used in saturating the heap leach pad. Additional losses are via dust control mitigation for onsite roads and small projects, as necessary. The project water balance is shown in Figure 18-4.

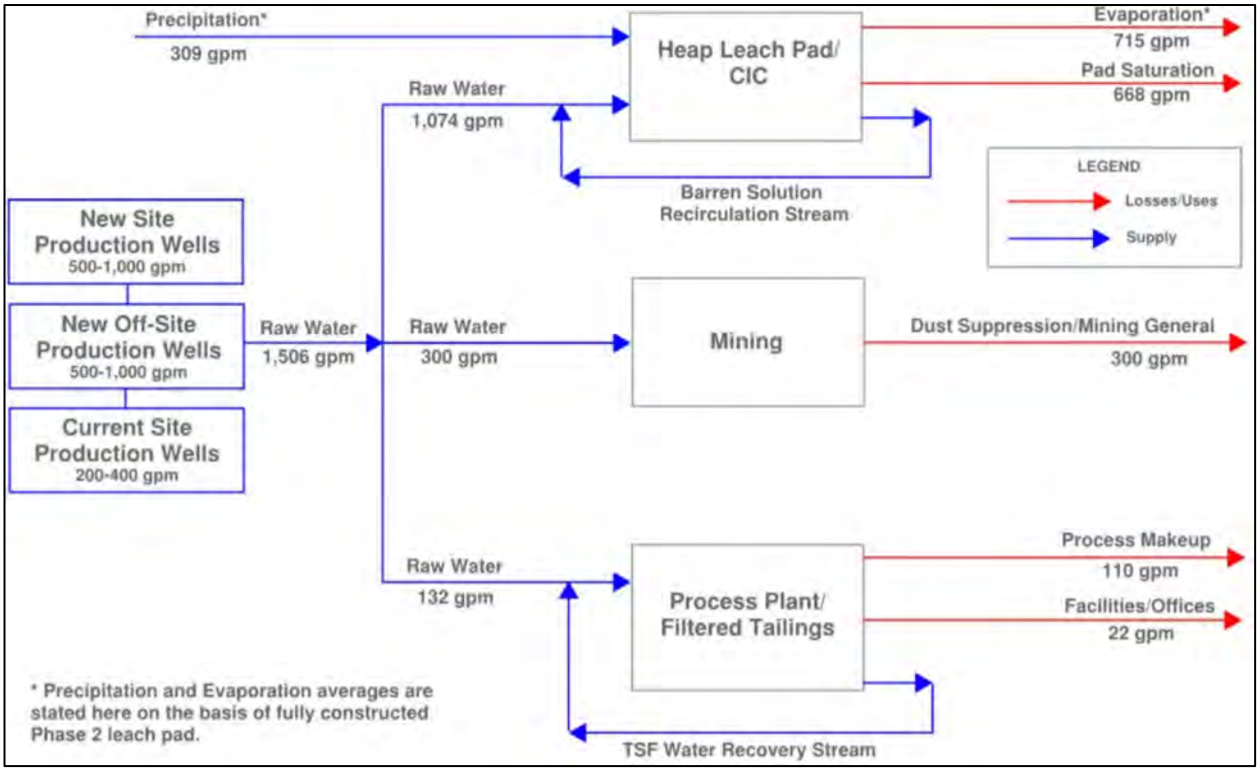


Figure 18-4: Sitewide Water Balance (Annual Average)

18.5.1 Climatic Data

The meteoric record used for this study was synthesized from multiple sources, since complete weather data for all the necessary variables was not available from the onsite weather station. The sources for the synthetic meteoric record were:

1. Onsite weather data from Castle Mountain (3 partial years of data)
2. Weather data from the Mountain Pass Mine (a nearby property with 27 years of temperature)
3. Surrounding regional weather stations (nearly 50 years of data)
4. PRIME grid estimate from the Prism Climate Group at Oregon State University (computer generated models based on multiple weather data sources)

The synthetic record for local weather was used to generate a deterministic model for temperature, precipitation, and evaporation. That data, along with the leach pad make-up water monthly average, is summarized in Table 18-1.

Table 18-1: Modelled Monthly Climate Data Summary

Month	Mean Monthly Precipitation (mm)	Mean Monthly Pan Evaporation (mm)	Mean Monthly Temperature (°C)	Leach Pad Make-up Water (gpm)
January	27.9	68.5	5.7	826
February	26.4	87.1	8.4	877
March	24.1	135.6	11.4	994
April	11.3	186.0	15.9	1,144
May	10.8	244.6	19.9	1,261
June	9.6	294.5	27.1	1,361
July	20.9	297.5	30.3	1,352
August	31.3	259.1	30.3	1,226
September	25.6	206.7	25.4	1,157
October	15.5	148.4	18.9	1,002
November	18.2	89.4	10.3	871
December	19.5	68.5	6.1	814
Average	**20.1**	**173.8**	**17.5**	**1,074**

The resulting climatic synthetic record was used in conjunction with the phasing of the leach pad liner and stacking plan to produce resulting ingress/egress flows to the project water balance around the heap leach pad and filtered tailings facility. These flows will vary year by year based on the leach pad evolution and for the purposes of the study a deterministic weather record was used to predict longer term conditions over the mine life. Climatic conditions such as wind speed, days of sunlight and humidity were not considered within this model.

18.5.2 Water Demand

In addition to the water demands listed for the process plant and heap leach described in Section 17, water usage for Phase 2 expanded operations was estimated for mining dust control, mining and general water usage at project facilities. These were then totaled with process needs to estimate site water make-up demand low, average and peak values. Table 18-2 shows the resulting summary of project water demands.

Table 18-2: Project Total Water Demands

Description	Low Demand (gpm)	Average Demand (gpm)	Peak Demand (gpm)
Heap Leach Pad Saturation	668	668	668
Heap Leach Net Evaporation*	146	406	693
Mill Water Demand	110	110	110
Mine Water Demand	200	300	400
Ancillary Facilities/Offices	22	22	22
Total	**1,146**	**1,506**	**1,893**

* Net Evaporation is the net combined effect of evaporation losses and precipitation gain.

To account for uncertainties in the weather model, a contingency has been applied to the average demand to assure adequate continuous design basis of pumping and delivery systems. The maximum evaporative loss was calculated as one standard deviation from the mean. The value of one standard deviation, estimated at 220 gpm, was incorporated into the water balance as an additional requirement for average make-up water. The added contingency of 220 gpm together with calculated average make-up results in an average annual demand of 1,726 gpm with an expected range from approximately 1,150 gpm to 1,900 gpm. Site water supply systems are sized to handle both peak instantaneous demands as well as continuous supply of the average including this contingency.

Opportunities to further reduce the water demand through greater use of onsite dust suppressants, strategic construction planning to construct leach pad cells during the wet season, and optimization of the heap leach make-up water requirements through efficiency improvements will continue to be studied and utilized through life of mine site development. Pumping and process monitoring systems will be designed to accommodate irrigation turn down at the heap leach pad and allow process management to select lower strategic water need set points. These adjustments can be made until dry and demanding conditions subside.

18.5.3 Water Supply

18.5.3.1 Site Water Supply

Phase 2 operations will require a continuous supply of water, generally averaging 1,500 gpm and up to 1,900 gpm under peak demand in the dry season. Supply of this make-up water will be met by a combination of sources from near site and off-site shown in Figure 18-5.

Three historical production wells, W-14, W-18, and W-45, which are already connected via existing underground pipelines to an existing 250,000 gal water tank, are estimated to provide 150 gpm total. This is referred to as the West Wellfield area located to the northwest of the Project site. The water tank feeds an existing underground line that terminates near the mine and administration office complex and facilities. This system is primarily used for dust suppression water under current operations and will continue in similar service for the expanded Phase 2 plans.

Two production wells, W-01 and W-02, were drilled and completed in 2017 and had pumps installed in 2019 at the start of Phase 1 construction. They are currently providing the majority of site water make-up for current operations. These wells are located at the edge of the JSLA pit and what will eventually be the South Domes pit, respectively, and are bedrock wells which are estimated to be able to produce a combined total of up to 400 gpm. The wells have been fitted with new well pumps and serve a pipeline feeding the recently constructed 300,000 gal raw water tank located on top of the historical Viceroy heap area. The wells will continue to operate in the expanded Phase 2 plans.

Additional mine site production wells will be located in an area to the south and southwest of the process plant to provide additional make-up water for Phase 2. A series of five monitoring wells have been drilled in 2020 in the area south of the property and these holes have also served to evaluate groundwater potential and to improve the understanding of the hydrogeology in the basin fill and bedrock of Lanfair Valley. This recent work has been reported on by Clear Creek Associates, (Clear Creek, 2020a) and supplements surface geophysical surveys conducted in 2015, 2017, and 2018.

The results of the near site water investigation suggest that there are two aquifer systems in Lanfair Valley, a younger alluvial system that the West Wellfield taps, and a deep alluvial-volcanic aquifer that extends to depth of about 1,450 ft. Two of the five recent drill holes appear to be favorable for production well installations: one that penetrates the deep alluvial-volcanic aquifer, and another located in the bedrock aquifer. For the first one in the alluvial aquifer, the water table is at a depth of 490 ft and drilling indicates nearly 1,000 ft depth of aquifer showing high probability for a productive water zone. It is anticipated that, once developed, wells in this area will produce between 500 and 1,000 gpm of water. Further detailed investigations are recommended going forward, including an update to the local hydrogeological model and construction of a 14 in diameter steel-cased test well which could eventually serve as a production well. A groundwater model will be used with the information gained from the water exploration program to assess any impact that pumping the new well at its maximum rate may have on Piute Spring over the life of the mine and beyond.

18.5.3.2 Off-Site Water Supply

It is anticipated that supplemental make-up water will be required in addition to that reasonably available from the Lanfair basin in order to mitigate the stress that may be put on the local aquifers. Equinox has recently investigated the groundwater production potential of an area of the Ivanpah Valley near the town of Nipton, California, approximately 16 miles northwest of the mine. The investigation included conducting surface geophysical surveys using audio-frequency magnetotellurics (AMT) and gravity, the drilling and construction of an observation well to a depth of 605 ft, and the testing of an existing production well. The results were evaluated and reported by Clear Creek Associates, (Clear Creek, 2020b) and show that the aquifer is about 3,000 ft deep, comprised of unconsolidated basin fill alluvial materials, and is uniform in character both vertically and horizontally.

The aquifer was recently pump tested as part of the investigation and provided excellent results demonstrating a high hydraulic conductivity; however, testing was limited to low flow rates. Additional data from historical well field pump tests in other parts of the Ivanpah basin indicate a high probability for good water production and coupled with the fact that there does not appear to be a salt layer in the location tested as is encountered in the center of the basin that would act as a bottom boundary for the aquifer, it is expected that water production would be good. Production wells would likely yield between 500 and 1,000 gpm and possibly more. It is recommended to further evaluate this area with the installation of a production well to a depth of 1,500 ft.

Ultimately, it is expected that two new production wells will be installed near this location. These new exploration wells will be fully outfitted with new casing and pumps and serve an overland pipeline extending 30 miles following Nipton Road and then into site along the main access road. The pipeline will be pressurized by a 10,000 gal booster tank equipped with 350 hp booster pumps feeding the new 190,000 gal north tank located on top of the historical Viceroy heap area. This system will serve as the second major source for the expanded Phase 2 plans, contributing up to 1,000 gpm in make-up water capacity. Table 18-3 shows a water source summary.

Table 18-3: Water Source Summary

Description	Operating Wells	Capacity (gpm)
Current Site Production Wells (Phase 1)	4-5	200-400
New Site Production Wells (Phase 2)	1-2	500-1,000
New Off-site Wells (New – Phase 2)	2	500-1,000
Total	**7-9**	**1,200-2,400**

Figure 18-5 illustrates the general areas intended for water production.

Figure 18-5: Water Sourcing Map

18.5.4 Potable Water Distribution

Raw water will be treated using an on-site chlorinator system to meet applicable potable water standards. Treated water will be stored in a 10,000 gal polypropylene potable water tank. This water will be utilized throughout the plant site for general potable uses (i.e. restrooms, break areas, etc.) and will be sized appropriately for building occupancy. The estimated size for this demand based on current building plans and occupancy is 22 gpm.

18.5.5 Fire Water Protection System

The fire water system for project expansion will be divided into two areas: the process plant and the crushing and ancillary areas. The process area fire water system will be tied into the existing 10 in fire water gravity fed pipeline built as part of Phase 1 which feeds a hydrant at the generator station; this branch will then distribute throughout the new process plant and mill area to use in hydrants spaced at 300 ft.

The crushing and ancillary fire water system will consist of a new separate 34 ft x 32 ft water tank with 190,000 gal of capacity which will be fed from the new raw water system. Due to hydrant residual/static pressure requirements by the fire code, the Raw/Fire Water Tank built as part of the first phase of construction will not provide the necessary pressure at the crusher and ancillary facilities. Therefore, there is a need for the new tank as well as a listed/approved automatic diesel water pump package. The new tank will be located next to the fine ore bin and will be able to provide two hours of fire water retention as well as feed dust suppression systems relating to the primary dump pocket.

18.5.6 Non-Contact Water Management

Analysis was conducted to understand peak discharge and runoff volumes to determine requirements for managing of surface flows around and through the Castle Mountain site as they relate to the proposed improvements for Phase 2 expansion. The hydraulic analysis was conducted using the San Bernardino Hydraulic Manual Guidelines as basis for the review on the premise of a 24-hour, 100-year storm event. Watershed volumes and maximum surface water elevations were evaluated to develop criteria for effective handling of the water and to minimize erosion concerns. Figure 18-6 illustrates the contributing watershed areas which will require management around and through the project site.

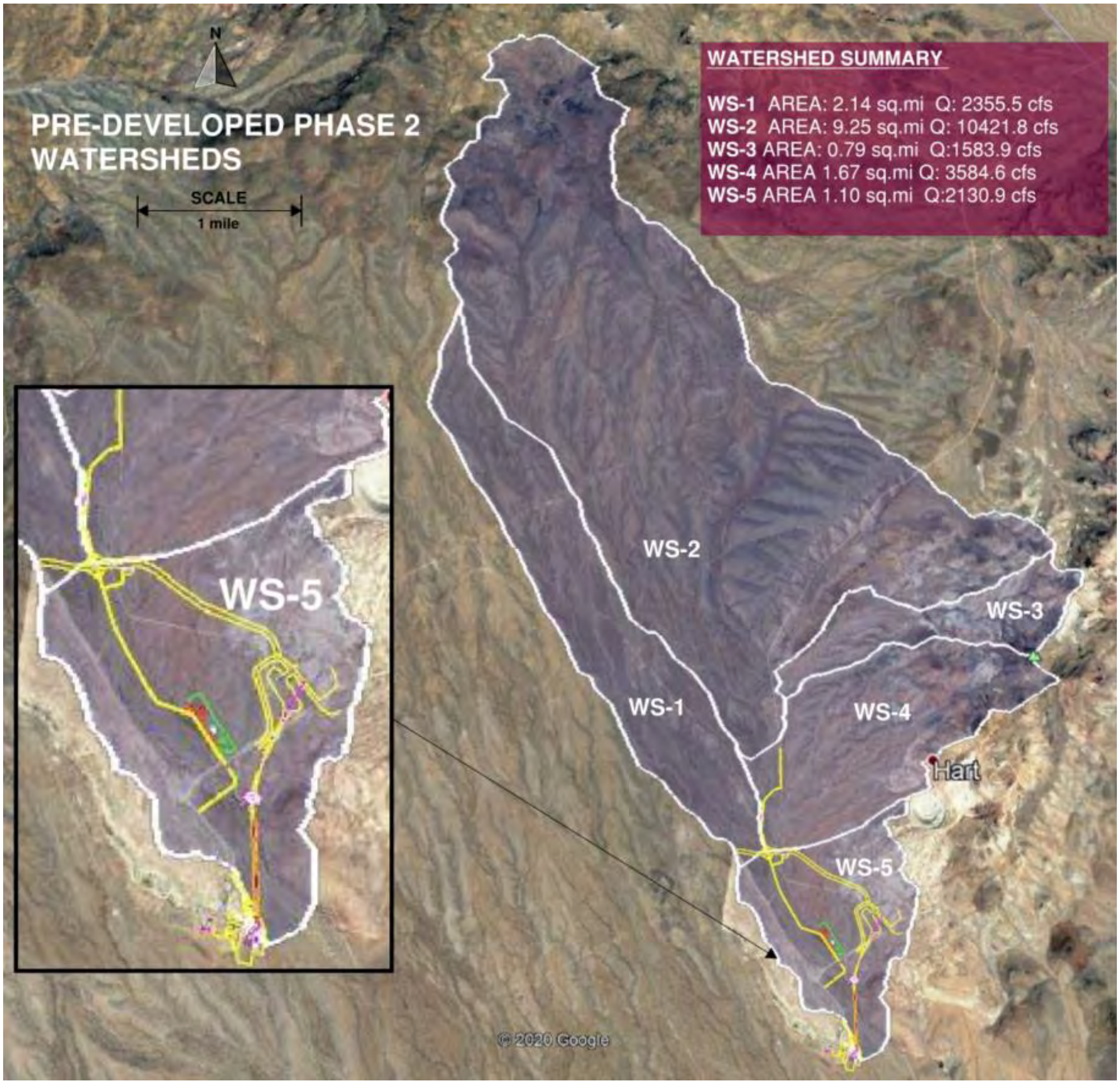


Figure 18-6: Castle Mountain Watershed Summary

Combined contributing watersheds considering a 24-hour, 100-year storm event of 5.19 in result in the need for handling of approximately 20,000 cfs (566 cms). Diversion channels and culverts around the North end of the heap leach pad and the Northern extents of the project site will direct flow around and through re-graded surfaces to the natural channel which has handled much of this flow historically. As mitigation for erosion control, channel armoring in the form of Reno/Gabion mattresses will be utilized at areas sharing a border with new process facilities. Figure 18-7 shows locations requiring reinforcement as part of the Phase 2 expansion.

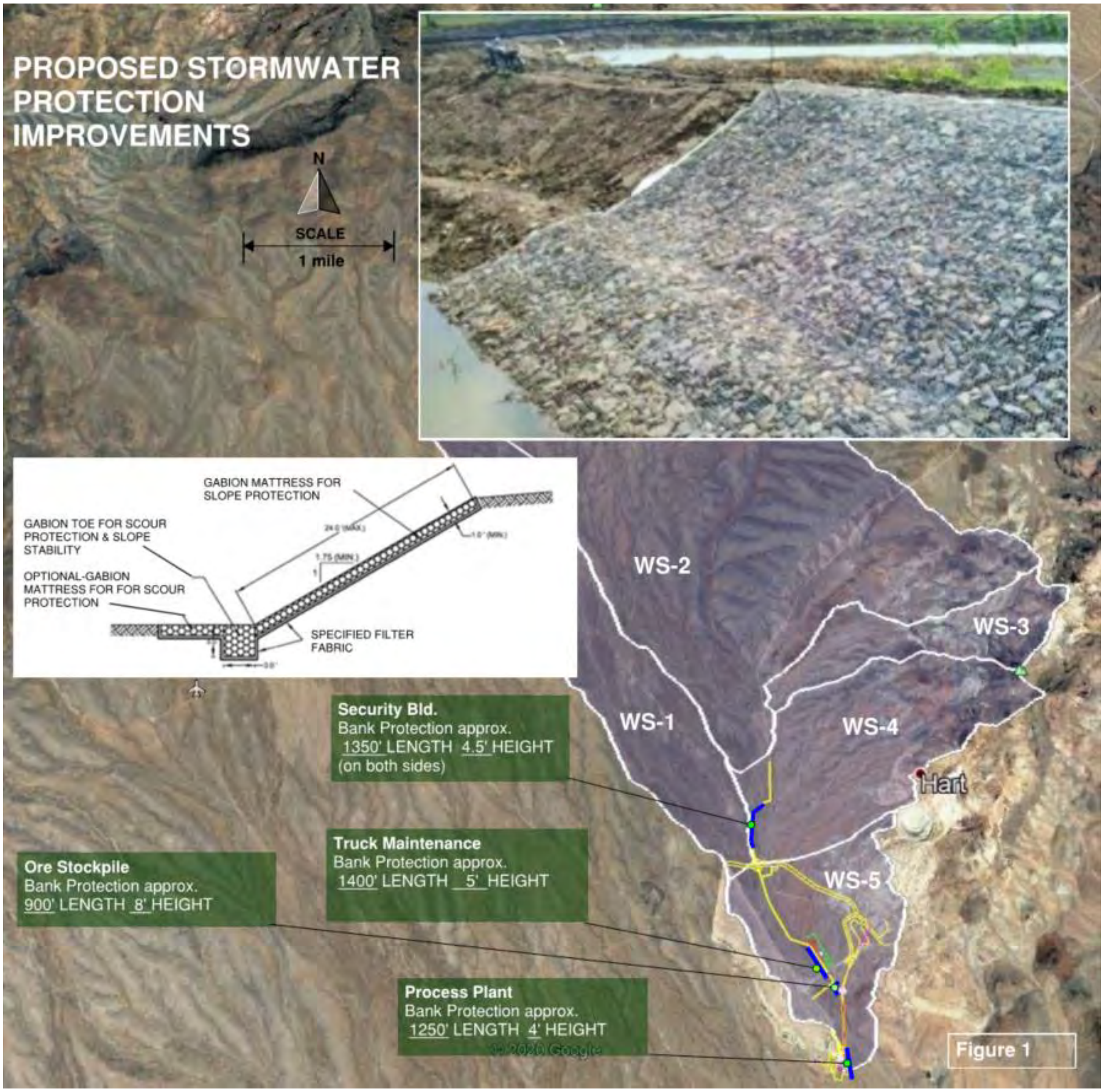


Figure 18-7: Stormwater Protection Plan

18.6 FILTERED TAILINGS FACILITY

18.6.1 Tailings Stacking Operation

Filtered tailings will be produced at a moisture content of 19% to 22% by dry weight basis (16-18% wet basis) and will be discharged into one of three reclaim chambers beneath the pressure filter deck of the tailings filtration building. This material will be recovered via front end wheel loader for loading into 40 ton articulated dump trucks. Filtered tailings material will then be transported along haul roads approximately 1 mile north of the main processing plant and dumped within a lined facility for spreading by dozer atop the reclaimed former Viceroy heap leach pad. Development of the filtered tailings facility will occur in four stages to allow for both the placement of appropriate volumes of material to match production and the rinsing of heap leach side slopes which will be covered by the final filtered tailings facility footprint. Rinsing is required to allow for recovery of residual remaining gold ounces within the heap as well as reduction of cyanide levels to compliant levels within the placed heap leach material prior to final reclamation.

Stacking of filtered tailings is considered best available technology for handling and placing of this type of material. By placing filtered tailings abutted to the new heap leach facility and on top of the historic leach pad the area of disturbance on the site will be minimized. This will increase the long-term stability on the western edge of the facility and allow integrated management of solution between the tailings and heap leach facility, allowing for further recycle of cyanide.

Upon Phase 2 mill start-up, tailings will be placed initially in a 1.5 million ft^2 graded and lined starter cell, V1. This facility will stand alone initially on top of the historic Viceroy heap leach pad. In order to allow for the facility to drain (primarily from precipitation events) into the adjacent heap leach pad, 600,000 yd^3 of gravel material from the pad will be removed from under the footprint of this cell and graded accordingly sloped towards the heap. This material is to be utilized in construction of the heap leach pad underliner and overliner as it expands in size and will be staged accordingly. Starter cell, V1 will contain 2.2 million yd^3 of tailings capacity and service mill production through the first 2 years. The starter cell is strategically sized to allow for placement of tailings while allowing for operations to begin the rinse process on the eastern slope of the Phase 1 heap. Starter cell V1 stacking plan is illustrated in Figure 18-8.

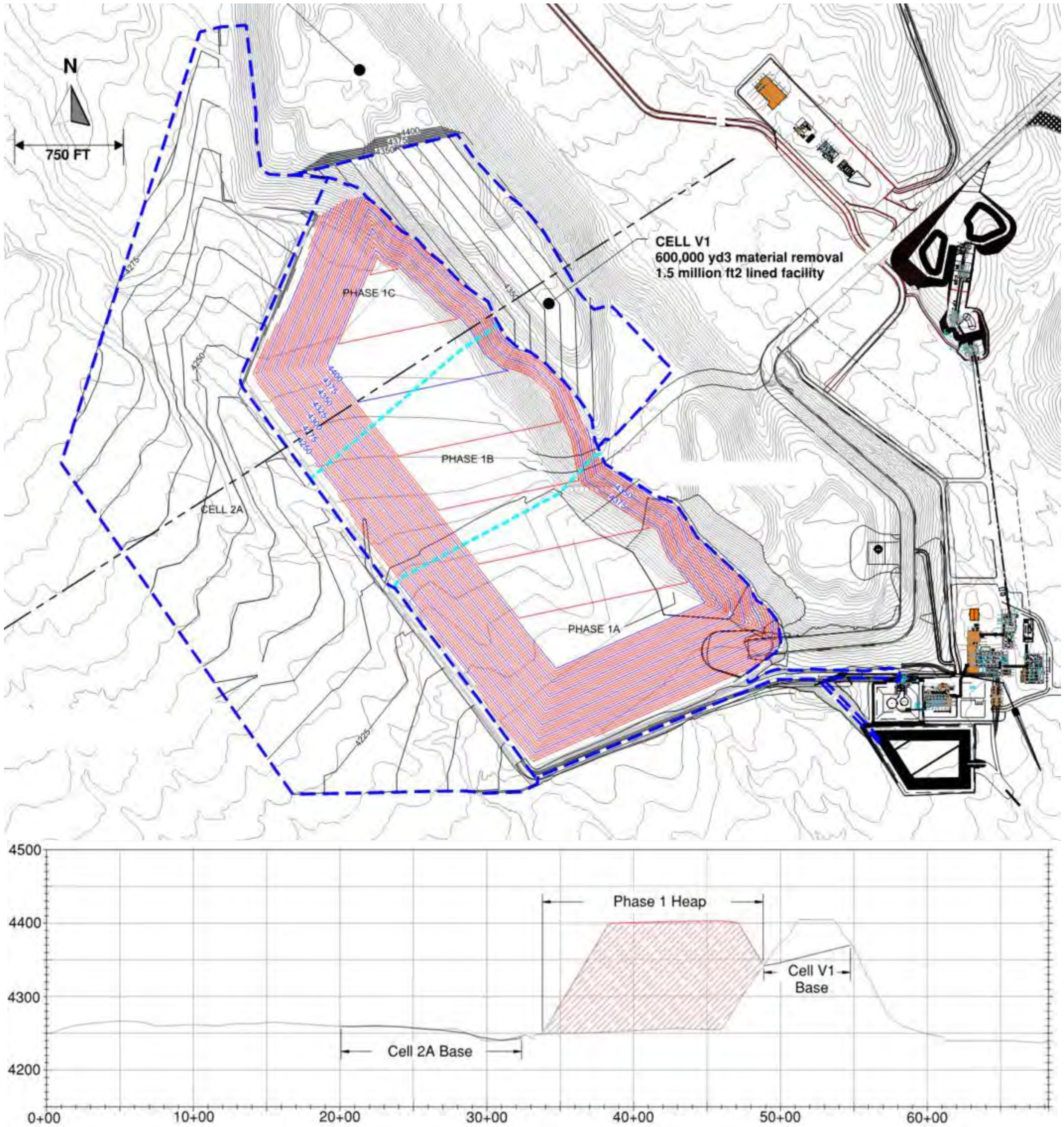


Figure 18-8: Tailings Stacking Cell V1 (Phase 2 Start-up)

Expansion of the tailings facility will occur in Year 3 after Phase 2 start-up and result in a newly lined and adjacent tailings placement cell V2 to the North of Cell V1. The combined capacity for tailings of cells V1 and V2 is 3.7 million yd^3 without extending up the sides of the Phase 1 heap and will allow for an additional 2 years of tailings placement (4 years total post Phase 2 start-up). Historical data has indicated that operations at Viceroy took approximately 3 years of rinsing to achieve acceptable residual levels of cyanide within the heap. Assuming a similar rinse period, these two cells will allow for adequate capacity to rinse the Phase 1 side slopes and allow for further placement of tailings on top of the Phase 1 heap slopes moving forward. Cells V1/V2 stacking plan is illustrated in Figure 18-9.

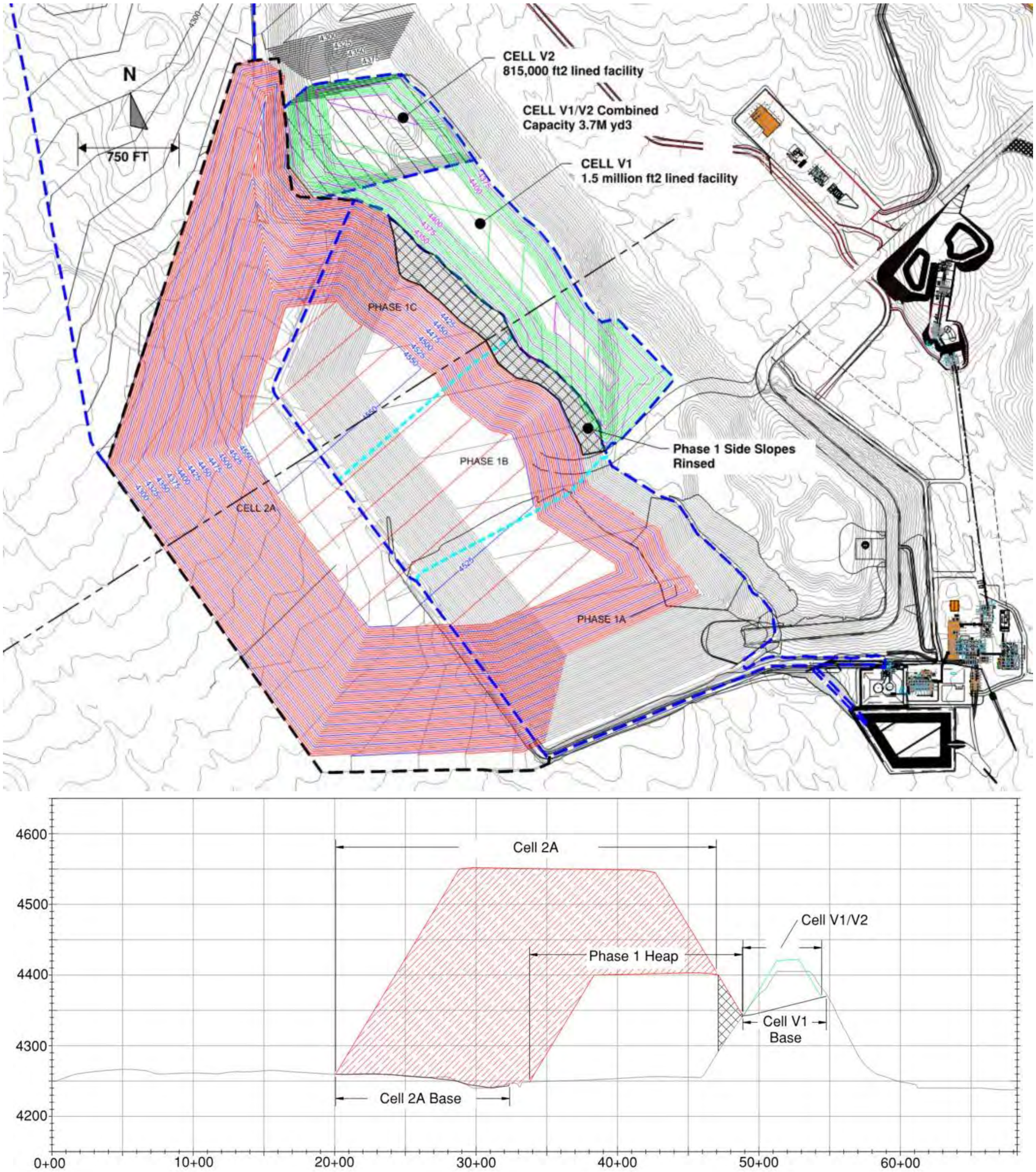


Figure 18-9: Tailings Stacking Cell V1+V2 (Through Year 4)

Upon completion of rinsing of the Phase 1 side slopes, placement of tailings in between cells V1 and V2 and the Phase 1 side slope can commence. This will increase the overall total capacity to 6.6 million yd^3 and provide adequate storage capacity through Year 7 of Phase 2 operations. Throughout utilization of these earlier cells, the Phase 2 Cell 2A heap leach pad will have been built up to final elevation. Once the eastern slopes of the pad come available and have been leached, rinsing will commence to allow for further stacking of the tailings abutted to this slope. Filtered tailings stacking plan through Year 7 is illustrated in Figure 18-10.

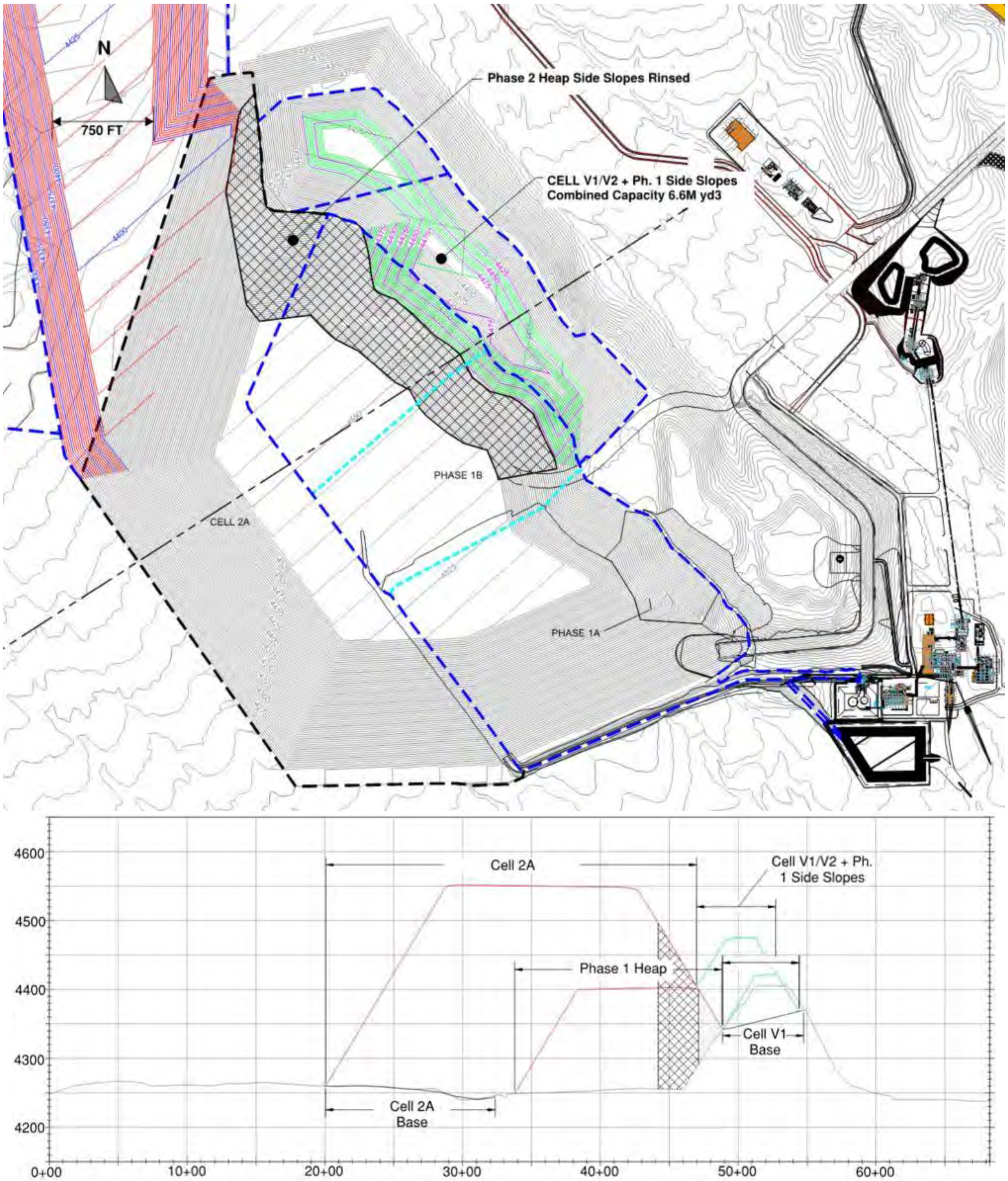


Figure 18-10: Tailings Stacking Cell V1+V2+Ph. 1 Slopes (Through Year 7)

Upon completion of rinsing of the Phase 2A side slopes, placement of tailings in between stacked filtered tailings cells and the Phase 2A side slope can commence. This will increase the overall total potential capacity to 16 million yd^3 and provide adequate storage capacity through the current life of Phase 2 operations (11 million yd^3) as well as the ability to handle future extensions to the life of mine plan. Filtered tailings stacking plan through Year 14 is illustrated in Figure 18-11.

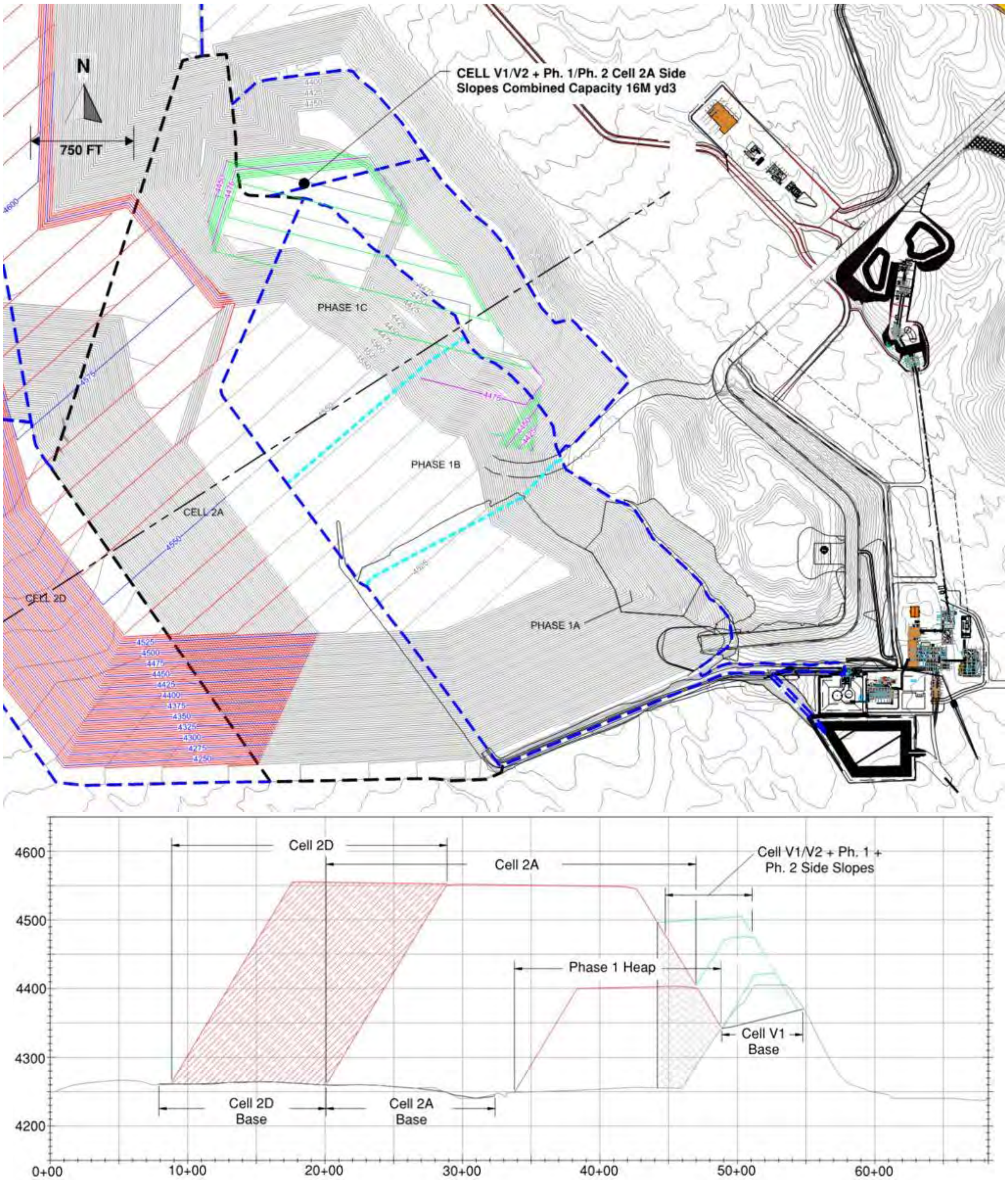


Figure 18-11: Tailings Stacking Cell V1+V2+Ph. 1+Ph. 2 Cell 2A Slopes (Through LOM)

The filtered tailings storage facility requires the management of partially dewatered tailings. Filtered tailings comprise an unsaturated cake with saturation levels ranging from about 50% to 85% and they typically are delivered to the pad at a moisture level above the optimum moisture required for compaction. Therefore, the tailings are spread and allowed to dry either through drainage and/or evaporation, prior to compaction.

Compaction will be handled by a padfoot vibratory steel drum compactor (CAT CP533 or equivalent). Studies of evaporation potential and compressibility were performed to identify an optimized lift thickness of approximately 20 in to 28 in which provides the greatest likelihood of achieving a minimum required density of 100 pounds per cubic foot (pcf), and assures stable slopes at 3 horizontal to 1 vertical to a maximum height of 125 ft above surrounding grade for the initial East facility or up to 205 ft above the spent ore of the historic heap leach pad. The drying and compaction of the tailings will assure that the stacked tailings will not experience strength loss from “contractive behavior”, either due to static liquefaction or to seismic liquefaction (i.e. from earthquakes).

Filtered tailings have a number of advantages over wet tailings disposal facilities including:

1. A reduction in overall water consumption.
2. Storage of a larger, denser volume of tailing on a smaller footprint.
3. The ability to perform concurrent reclamation.
4. Improved long term stability and reduction of stability risk.

18.6.2 Tailings Material Characterization

Filtered tailings facility design is generally a function of key material properties. In order to define the design criteria a series of material characterization tests were conducted on the two master composite samples described in Section 13.7.1, called the Main Pit composite and South Domes composite. The tests included hydrometer and sieve analysis, plasticity, consolidation properties, hydraulic conductivity, specific gravity, compaction testing, and triaxial shear strength testing, and results were provided by the MINES Group (MINES Group, 2020).

The two material samples were similar in many ways, but significantly different in others. The South Domes composite acted much like a non-plastic silty sand and the Main Pit composite acted more like a plastic clay. The design criteria were selected to enable proper storage within the facility for both types of material at varying times during mine operation.

18.6.3 Tailings Facility Design

Key resulting design criteria for the filtered tailings facilities can be seen in Table 18-4.

Table 18-4: Filtered Tailing Facility Design Criteria

Description	Units	Value
Tailings Storage Volume Required	ton	17,700,000
Average Delivery Rate	ton/day	3,500
Recommended Lift Height	m	0.5 - 1
Lift Cycle Times	days	6-35
Total Stack Height	ft	150
Side Slopes Required	-	3H:1V
Static Factor of Safety	-	⩾ 1.3
Facility Max. Volume Available	ton	22,200,000
Facility Base Area	ft^2	2,315,000

The filtered tailings facility will be a multiple-lift, single-use type pad utilizing a single-liner system consisting of prepared subgrade atop the historic Viceroy heap leach pad. The Viceroy heap facility will utilize a primary 80 mill LLDPE geomembrane liner allowing drainage to the Phase 2 heap leach lined facility. Figure 18-12 shows the Viceroy heap filtered tailings facility liner concept.

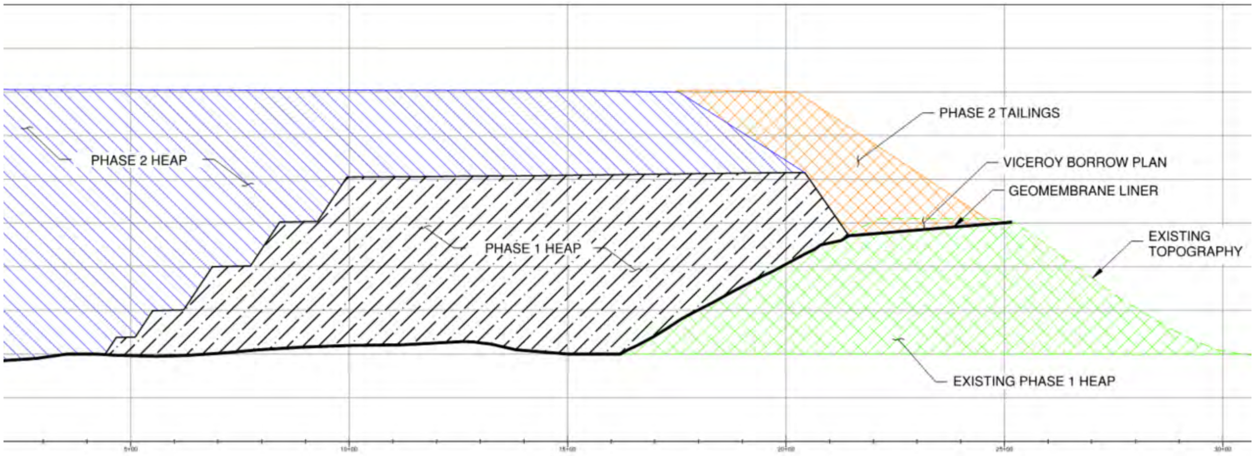


Figure 18-12: Filtered Tailings Design Layers (Viceroy Heap Facility)

For any given phase, it is assumed that about 90% of the lined area will be available for the stacking of tailings. The remaining 10% will be dedicated to access and drainage corridors with either bare liner or liner with a thin liner cover layer. Compaction to the minimum density of 100 pcf assures the existence of a stable structural embankment zone around the perimeter of the tailing facility.

The lift cycle is a function of the tailings production rate, lift thickness, and the available surface area for stacking and is a measure of the number of days the newly deposited tailings will be open and exposed to drying through evaporation prior to being covered by another layer of tailings.

The facility layout and phasing provide adequate room for the containment of any abnormally wet or weak tailing materials from occasional filter press upset conditions in a central core area, well behind the structural embankment.

The facility has been designed for the management of erosion from the exposed tailings surface. The Viceroy heap facility will be sloped and connected to the Phase 2 heap leach pad so as to allow drainage of rain events. Rain reporting from this facility will ultimately consolidate with solution flows within the heap and report to the solution collection pipe or lined channel and event pond.

Wind erosion is also a concern on the exposed tailings surface and will be managed during normal operations with the application of water and/or the use of chemical palliatives. Control of both water and wind erosion during reclamation and closure will be accomplished through the application of a surface armoring layer. Figure 18-13 illustrates the final life of mine filtered tailings footprint and site plan looking west.

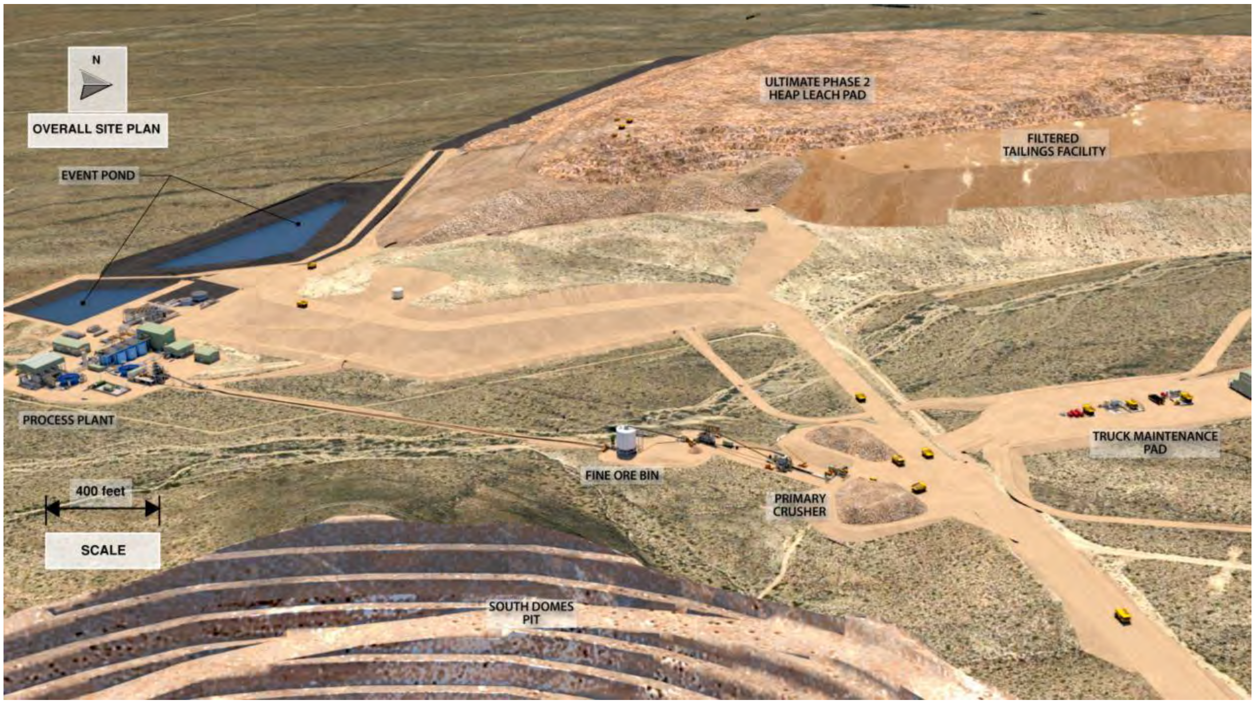


Figure 18-13: Site Plan Looking West

18.7 POWER

18.7.1 Power Supply

The project requirements are approximately 10 MW of power. Electrical power will be provided by connection to the grid at an existing Nevada Energy (NVE) sub-station near Searchlight, NV and routed to site via a new transmission line. The transmission line is expected to be managed in a co-operative arrangement between Southern California Edison (SCE) and NVE.

Power from the Searchlight sub-station will be provided via a 69 kV transmission line re-built by NVE and SCE to follow the existing right of way to the west occupied by active NV Energy equipment supplying Walking Box Ranch approximately 6.5 miles. It will then head South for approximately 17 miles along the site access road via newly constructed overhead line to the project sub-station on site. Figure 18-14 shows the intended transmission line route from Searchlight, NV.

Figure 18-14: Power Transmission Line Route

18.7.2 Alternative Power Supply

Several alternatives have been examined for power supply using a natural gas-fired power plant. The options considered included a 3rd party power supply through a private power agreement (PPA) with a newly constructed gas power plant near Searchlight. In this case power would be routed to site on a 138 kV transmission line following a similar right of way to the permanent utility line described above. Another alternative considered is building an on-site gas power generating plant utilizing liquified natural gas (LNG) delivery from Topock, AZ.

Both options provide a feasible solution for the project, however connecting to the regional grid provides the added benefit of allowing incorporation of renewable power sources such as solar and provides for the lowest operating power rate for the life of the project.

The location and climate of the project area provides an ideal opportunity to implement solar power and take advantage of a plentiful renewable resource while reducing the project's carbon footprint. This potential opportunity has been investigated and is presented in Section 26.

18.7.3 Project Electrical Loads

In addition to the loads for the main process plant described in Section 17.3.11, power demands were estimated for the off-site demands and site ancillaries to determine a total demand load. Table 18-5 below summarizes the total anticipated Phase 2 estimated running load of 9.9 MW. Under the current plan of service by SCE, off site loads relating to the sourcing of water to the North would be serviced by SCE from their existing distribution line near Nipton, CA.

Table 18-5: Summary of Process Area Power Consumption

Code and Area Description	Estimated Running Load (MW)
On-site Process Facilities	8.4
On-site Ancillary Facilities	0.3
Off-site facilities (Water Sourcing)	1.2
Total	**9.9**

Project Phase 1 operations currently utilize four (4) Tier 4F, 455 kW diesel generators on site. These generators would be installed to act as emergency power back-up for key process needs such as agitator drives, the carbon regeneration kiln and ventilation needs.

18.7.4 Power Distribution

Power distribution on site will consist of a 4,160 V overhead distribution line from the main substation to the north process facilities including primary crushing, fine crushing and ore storage as well as ancillary facilities and existing facilities located towards the access gate and mine. 4,160 V power will also be distributed via underground duct bank to the process area electrical houses and transformed to 480 V and 120 V as required to power equipment, utility and instrumentation needs.

Uninterruptable power supplies will be used to provide back-up power to critical control systems. This equipment would be sized to permit operations to shut down and back up the computer and control systems and to facilitate start-up on restoration of normal generator power. Battery power packs would supply back-up power to fire alarm systems and egress lighting fixtures.

18.8 SEWAGE AND WASTE

18.8.1 Sewage Handling

On-site sewage will flow by gravity to local septic systems located in proximity to points of use. Septic systems and leach fields will be sized appropriately for building occupancy.

18.8.2 Solid Waste

Solid waste will be managed in dumpsters or other appropriate waste containers. All containers will be covered (or covered and weighted if covers are not attached) to reduce the potential for blowing trash and to prevent access by wildlife. Containers used on site will be labeled. Trash from office and lunch areas will be bagged. A licensed waste management company will transport collected waste to a dedicated offsite third party-controlled landfill site.

18.8.3 Hazardous Waste

Hazardous waste will be placed in drums, put on pallets, and stored in secure, impermeable, and appropriately sized containers, providing the required secondary containment, until being hauled offsite by a licensed contractor. Hazardous waste will be disposed of in a safe and environmentally sound manner using outside contractors.

18.9 COMMUNICATION & CONTROLS

The Castle Mountain Project currently has connection to an Internet Service Provider (Century Link) via the Crown Castle Tower, East Ridge Repeater, Big Chief Tower, and to the operating Phase 1 Administration Building. The Phase 2 expansion project will continue to utilize this connection.

The Phase 1 Administration Building is the starting point for the layout of the new Phase 2 Fiber Optic campus backbone for site communications including process, process cameras, security, fire, and VoIP/Data. The main fiber optic trunk will be routed from the Phase I Administration Building to the Phase 2 Grinding Area Server Room through an IT firewall to the Process Historian and on to the Process firewall. The Grinding Area Server Room will be the hub for distribution for other areas.

Further consideration was included for cyber security by the utilization of a UTM (Unified Threat Management) unit, a NIDS (Network Intrusion Detection System) unit, and a SEM (Security Event Management) linked to the Network Administration.

The existing control system utilizes a Rockwell Automation solution and Rockwell will be used for the Phase 2 expanded control system to ensure a common platform and the ease of merging the two systems. The plan is to leave the Phase 1 control system intact and undisturbed so that it may continue to function as operations requires, and it would then interface with the Phase 2 controls to allow operation from either control room.

Phase 2 servers that are located in the Grinding Area Server Room will be the central area for various areas' Programmable Logic Controller (PLC) to report to with the Human Machine Interfaces (HMI(s)) being located in the Grinding Control Room. Other HMI(s) would be located at the Primary Crusher Control Room to allow control of the Primary Crusher, Secondary Crusher, and Screen/Transfer up to Fine Ore Storage. All required information from the primary crushing area would still be trended and historized in the Grinding Servers.

19 MARKET STUDIES AND CONTRACTS

No market studies or contracts were conducted for the Project.

The process facility will produce gold doré bars which are routinely sold to third party refiners. The feasibility study assumes the refining and transportation costs will be $1.51 per ounce of gold and the refiner will pay 99.5% of the value for the gold.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Castle Mountain Mine encompasses both public and private land, accordingly, the County of San Bernardino and the United States Bureau of Land Management (BLM) have historically served as co-leading agencies for implementing environmental review. The 1990 Environmental Impact Statement / Environmental Impact Report (1990 EIS/EIR), the 1998 Castle Mountain Mine Expansion project Environmental Impact Statement / Environmental Impact Report (1998 EIS/EIR), and the 2020 BLM Environmental Assessment cumulatively provided the impact analysis, mitigation and resulting lead agency authorization for current mine operations.

The Castle Mountain Mine historically acquired all relevant lead agency (discretionary) and local (non-discretionary) operating permits commencing in 1990 until activity at the mine was largely suspended by 2005 (after which some discretionary permits were relinquished). Reclamation became a significant focus beginning in 2001 and continued until about 2006, followed by a reclamation monitoring period of approximately six-years. Monitoring concluded after the lead agencies determined that reclamation success standards were achieved by 2012. During periods of suspended activity, the site operated through implementation of an approved Interim Management Plan (IMP) which preserved the validity of certain permits until termination of the IMP concurrent with resumed mining at the project site.

In January 2019, CMV submitted an application to the co-lead agencies to consider and approve minor modifications to the approved Mine Plan and Reclamation Plan (Mine and Rec Plan). Through their review of the plan amendment application, both agencies determined the proposed changes to be minor in scope, and the resulting environmental impact was adequately analyzed both through the 2020 Environmental Assessment (EA) and during previous California Environmental Quality Act (CEQA) and National Environmental Policy Act (NEPA) reviews conducted to date.

On August 23rd, 2019, the County Land Use Services Department approved minor revisions to the Mine and Reclamation Plan and issued a revised Mining Conditional Use Permit (CUP) and Reclamation Plan 90M-013 which expires December 31st, 2035. This revised mining CUP, in addition to the 1998 CUP and associated environmental review, provide the basis and authorization for current mine operations at the project site. While mineral exploration can sometimes be considered synonymous with “active mining”, for the sake of this discussion, active mining operations resumed in 2020 with active open pit mining and cyanide heap leaching.

Regarding the most recent BLM NEPA analysis, the 2020 EA, most of this analysis was specific to the installation of five new groundwater elevation monitoring wells at the project site. After completing the EA and responding to public comments, the BLM issued a Decision Record and Finding of No Significant Impact (FONSI) and approved the revised Mine and Reclamation Plan on February 27th, 2020.

Historic and current resource monitoring on the Project site include the following:

- Groundwater elevations in water production wells and regional monitoring wells.
- Flow and water quality monitoring at Piute Springs.
- Groundwater quality in monitoring wells located near the leach pad and mining operations.
- Stormwater surveillance.

- Vegetation inventories and propagation of indigenous plant communities.
- Dedicated programs surveying the movements and tracking of desert tortoise, bighorn sheep, bats, golden eagle, etc.
- Additional wildlife monitoring programs as applicable.
- PM-10 perimeter air monitoring.
- Meteorological weather station monitoring.

Mine expansion is expected to require new or updated environmental review (likely in the format of an EIS/EIR) as well as several new state and federal permits and amendments.

20.1 ENVIRONMENTAL STUDIES

In 2017 and 2018, several flora, fauna, hydrology, and cultural resource studies were performed to confirm whether the original baseline studies are still representative of the environmental, ecological, and cultural resources at the Project site. To date, these studies have all indicated that there have been no significant changes to site baseline conditions, so a combination of the previous EIS/EIR(s) analysis, combined with ongoing updates associated with mine expansion will properly characterize potential resource effects from mine expansion activity.

20.1.1 Flora and Fauna

Floral and faunal assessments generally target species that are state or federally protected and identified as potentially present at the Project site. Assessments are completed by expert consultants respective to each specific resource, and in some instances, multiple individual assessments are required to completely document a species and its abundance (or absence) at the Project site. Local, state and federal protections may include:

- Federal listing to the Endangered Species Act (ESA)
- State listing to the California Endangered Species Act (CESA)
- The Bald and Golden Eagle Protection Act
- BLM Special Status Species
- State protected species
- Local county ordinances

20.1.1.1 Flora

A multitude of botanical reports spanning over 30 years have confirmed that no state or federal threatened or endangered plant species are known or expected to occur on or adjacent to the Project.

There are multiple plant species listed as a BLM Special Status Species (SSS) with the potential to occur at the Project site. These species are in part defined through a rare plant ranking system developed by the California Native Plant Society. One species on the list that is known to occur on the Project site is the rosy two-tone beardtongue (penstemon bicolor ssp. roseus). The presence of this species is attributed to the success of reclamation work performed to date at the Project site. The species is most frequently documented as occurring on reclaimed mine land because of its inclusion in the revegetation seed mixture used at the Project site. In addition, it appears this species requires land disturbance for successful seed germination[1]. Ongoing mitigation used at the Project site includes transplantation and annual seed collection of the species, these practices will continue indefinitely.

Though not a state or federally protected species, the eastern Joshua tree is also prevalent throughout the Project area. Nonetheless, impact to this species is mitigated through salvage and transplant to onsite plant nurseries, salvage and donation to third parties, and collection of seed for use in future revegetation.

20.1.1.2 Fauna

The Project area has a wide variety of habitat types from high elevation rocky slopes to desert washes and grasslands that can support a variety of wildlife species. Wildlife that is most often the focus for resource assessments include:

- Desert Bighorn Sheep (ovis canadensis nelsoni)[2]
- Desert Tortoise (Gopherus agassizii)
- Bats
- Raptors (Golden Eagle)
- Mountain Lion

Nelson's Bighorn Sheep is listed as a state fully protected species though not listed as either state or federally protected as threatened or endangered. Sheep are often seen at the project and prefer steep, less vegetated rocky terrain as a means to avoid and escape predators. The original water guzzler at the Project site provides the sheep and other wildlife with a water supply through the dry summer months. As of late 2020, a second wildlife guzzler has been installed north of the mine pits. This guzzler is the result of a cooperative project with the Society for the Conservation of Bighorn Sheep (SCBS) and it provides a secondary option for sheep access to water that has historically been provided by the main guzzler. Additional cooperation is also ongoing with California Dept. of Fish & Wildlife experts regarding monitoring of the sheep including annual health assessments, collaring and species tracking.

The desert tortoise is listed as a "threatened" species by Federal and State authorities. Desert tortoise habitat is found in some portions of the Project area, though no Project lands are considered critical habitat. Tortoises occupy a variety of habitats from flats and slopes dominated by creosote bush scrub at lower elevations to rocky slopes in higher elevations dominated by black brush scrub and juniper. They can be found in elevations from sea level to 7,300 ft. and prefer elevations between 1,000 ft. to 3,000 ft. Federal law prohibits activities resulting in harm or "take" of listed species and provides significant penalties for violations. However, the law also provides procedures for legally impacting threatened or endangered species under certain circumstances. Two Biological Opinions[3,4] permit the disturbance of up to 1,128-acres of tortoise habitat and a "take" permit in the form of direct mortality to ten desert tortoises at the Project site and access roads. Historic records indicate two desert tortoise mortalities were reported along

[1] All known occurrences at the project site are located in conjunction with recent land disturbance activity.
[2] Peninsular bighorn sheep (ovis canadensis nelsoni) and Sierra Nevada bighorn sheep (ovis canadensis sierrae) are the only subspecies protected at both the federal and state level as threatened under their respective endangered species acts. Neither of these species are found at the Project site.
[3] 1990 U.S. FWS Biological Opinion #1-6-90-F-24
[4] 1997 U.S. FWS Biological Opinion #1-8-97-F-37

the access roads in the 1990s, though it is unclear whether these were attributed to mine operations or other causes related to public use of the open mine access road.

The need to protect the desert tortoise during construction and operation at the Project has always been important to CMV. CMV has committed previously to specific project changes, specifically during the 1990 and 1998 EIS/EIRs. For example, mine access roads were re-aligned to avoid higher quality desert tortoise habitat, almost five miles of desert tortoise fence was installed to exclude the tortoise from the most active areas of the Project, as well as additional compensatory mitigations provided by CMV. Between 1990 and 1998, over 920-acres of mitigation land was purchased by CMV and transferred to the U.S. BLM, as well as a deposit of $92,000 by the mine operator to U.S. BLM. Cumulatively, these concessions permit disturbance to 1,128 acres of desert tortoise habitat that would potentially be impacted by project operations. Other mitigations were also incorporated into County, BLM, and U.S. FWS permit conditions such as construction of perimeter exclusion fencing, speed limits on area roads, raven monitoring, the requirement of biological monitors during mine activity outside of fenced areas, tortoise awareness training, and annual reporting on species activity, to name but a few protective/compensatory mitigations required at the Project to lessen impact to desert tortoise. These mitigations coupled with the low density of tortoise found at the Project site throughout 30 years of monitoring and studies, provided for the U.S. FWS determination that the Project will not result in “jeopardy” or harm to the desert tortoise population.

Other faunal species that were identified for further assessment include bats, raptors (particularly the golden eagle), and potentially, mountain lion. Mitigation measures include, but are not limited to, fencing, netting, lighting placement, and strategic powerline design were considered as part of the wildlife impact mitigation strategy.

While additional faunal assessment surveys are planned for 2021, to date, all recent surveys have supported the baseline characteristics formed by previous impact studies conducted at the Project site. Therefore, any significant new information or changed circumstance regarding any state or federally protected species that has the potential to suffer project related impact is not anticipated.

20.1.2 Air Quality and Greenhouse Gas Emissions

Sources of air pollution are those with potential to emit (PTE) criteria pollutants or toxic air contaminants. All potential sources of air emissions, including both stationary and mobile sources, are reviewed, inventoried, and modeled during the lead agency NEPA and CEQA process. The NEPA and CEQA process also includes an evaluation of greenhouse gas (GHG) emissions associated with the proposed Project. Following NEPA and CEQA impact analysis and upon subsequent approvals, stationary sources of criteria and toxic air contaminant emissions are evaluated for local air district permitting.

Baseline air quality levels for particulate matter 10 microns or smaller (PM_{10}) at the project fence-line were developed during original project permitting in the 1990’s. Renewed monitoring began in 2020 with the installation of four air monitoring stations located at one upwind and two downwind fence-line locations. All locations are equipped with a Met One Instruments E-BAM model (E-BAM) using a PM_{10} Omni-directional Inlet. One of the downwind locations is also outfitted with a Met One Instruments E-BAM Plus model unit (E-BAM Plus). The E-BAM Plus has been designated and approved by the United States Environmental Protection Agency (USEPA) as a Federal Equivalency Method (FEM) for measurement of PM_{10}, as set forth in 40 CFR Part

53. The addition of the E-BAM Plus unit affords comparison and traceability to a PM_{10} FEM monitoring method.

Meteorological data is also collected at the Project site by a modeling quality 10-meter meteorological tower. Parameters recorded include:

- Wind speed (avg. and max. gust) and direction
- Temperature (at 2 and 10 meters above ground level)
- Relative humidity
- Barometric pressure
- Solar radiation
- Precipitation
- Evaporation (pan) depth and rate

Project GHG emissions are defined by the EPA in two different areas as follows:

- Scope 1 emissions are the most significant and are generated by the mobile mining equipment fleet through the burning of diesel fuel.
- Scope 2 emissions are less significant and are generated indirectly through the generation of power off-site by the utility.

Project GHG may be offset by purchasing renewable energy credits or by direct connection to a renewable energy power plant. The use of renewable energy has been extensively investigated and the Castle Mountain site offers high potential for reducing reliance on conventional power generation. A solar plant using photovoltaic technology has been examined in detail and presents an excellent opportunity to the Project to reduce GHG emission and lower reliance on electrical power provided by utilities. Utilization of a solar plant prior to Phase 2 start-up would also have the added benefit of eliminating the need for operating the current diesel generators on site. The solar plant opportunity is summarized further in Section 26. Options will be further assessed to lessen project impacts related to GHGs. Criteria pollutants or toxic air contaminants will be further reduced for mine expansion through continued use of mobile equipment classified as meeting Tier 4 emission standards.

20.1.3 Hydrology and Hydrogeology

The Lanfair Valley surface water drainage area is approximately 340 mi^2 in size. The maximum basin dimensions are approximately 20 miles (east to west) and 17 miles (north to south). The topographic relief on the basin floor is relatively low with gradients varying from 50 ft. to 200 ft. per mile. The mountain slopes lying above the alluvial floor represent approximately 80 mi^2, or about 24% of the total watershed. Streams (washes) within the valley are ephemeral and flow only in direct response to precipitation or snow melt. One exception is the perennial Piute Springs, located about 15 miles southeast from the mine. The Colorado River lies 28 miles east of the Project.

20.1.3.1 Piute Springs

Piute Springs is a significant perennial spring located approximately 15 miles southeast from the Project site, in the Piute Mountain range. Potential mine related impact to Piute Spring has been extensively studied since the late 1980's at the Castle Mountain region. Multiple hydrogeologic studies completed to date have arrived at the same conclusion, which is that available evidence suggests water use at the mine site is not expected to significantly impact water flow at Piute Springs. Historical data spanning many years does show that spring flow naturally fluctuates, and this is generally attributed to seasonal and climactic effects experienced throughout the entire Mojave region. CMV maintains a network of groundwater monitoring wells that provide an early warning system[5] for any future potential impact to Piute Springs. Groundwater elevations are routinely recorded from a network of five monitoring wells.

The potential for future impact to Piute Spring, specifically related to mine expansion, will again be addressed by renewed hydrogeologic investigation and groundwater modeling to account for an expected increase in water use at the Project site.

20.1.3.2 Water Production and Use

The water source for operations is groundwater. All water required for the current mine operation is produced from two historical well fields, the West Well Field (WWF) and the East Well Field (EWF). WWF wells predominantly source water from wells drilled into saturated alluvium northwest from the main project area, whereas EWF wells source water from deep bedrock wells located along large fault zones throughout the mining area. The latter will also function as pit dewatering wells as mine pit depth increases. Five wells are currently available from the WWF and the EWF. The WWF wells are predominantly used to provide potable water to mine facilities.

The current permitted annual water use for the Project is 625 acre-feet per year. Proposed mine expansion activity anticipates an average water use of up to four times the current rate of water extraction during the dry season. The Project would extract water from multiple well fields, both existing and new, located in the Lanfair water basin and adjacent basins to the Project. The project will continue to conserve water by use of low evaporation drip emitters, burying drip emitters when feasible, limiting water in ponds with larger evaporative losses, use of binders and dust collectors that limit water needs for dust suppression and using extensive water recycling in the process. In addition, by drawing water from multiple well fields in different water basins, water is drawn from a larger area which lessens the demand on a single well field/aquifer. Further groundwater mathematical modeling is being undertaken to evaluate potential impacts. Water production and use is further discussed in Section 18.5.3.

20.1.3.3 Water Monitoring

Ten groundwater monitoring wells are located throughout the project area. Some wells are compliance (detection) monitoring wells associated with operation of the heap leach pad and ponds, while the primary function of others is for monitoring groundwater depth. Groundwater depth monitoring serves as an early warning detection system for potential impacts on the regional aquifer systems and the more distant Piute Springs. Water quality measurements were taken at several wells throughout the operation. Water quality sampling and monitoring resumed in 2020 under the direction of the California Regional Water Quality Control Board (RWQCB), Colorado River Basin Region, which will continue until final closure of the Project site. Currently, there are no known water quality impacts at or adjacent to the Project site, as a result of past or present commercial scale gold mining at the Project site.

Additional hydrological work has been conducted and/or is in progress to support both current mine operations and mine expansion. Groundwater models for the Project area continue to be

[5] Supplemental Plan for Groundwater Monitoring, Geo-Logic Associates, February 2019.

updated to estimate potential impacts associated with mine expansion. Additional monitoring wells have been installed south of the leach heap. These wells are used both for groundwater elevation monitoring and to provide additional aquifer characterization which may be used to guide the design of a potential water supply for mine expansion.

20.1.4 Cultural Resources

Cultural resources are places or objects that are important for scientific, historic and/or religious reasons to cultures, communities, groups, or individuals. Cultural resources include historic and prehistoric archaeological sites, architectural remains, structures, and artifacts that provide evidence of past human activity and places of importance in the traditions of societies or religions. Section 101 of the National Historic Preservation Act (NHPA) establishes procedures for determination of eligibility for listing historic and archaeological sites on the National Register of Historic Places (NRHP).

The earliest dated period of human occupation in the eastern Mojave Desert is estimated to be over 10,000 years following the last period of glaciation. As the climate became warmer and more arid, subsistence practices caused the inhabitants to change their way of life and became a more migratory society. The final period of human occupation in the region prior to Euro-American expansion was the Shoshonean Period. Southern Piute groups migrated southward replacing the Mojave groups.

In 1907, historic gold mining development in proximity to the current Project area created the town of Hart, one of several towns established in the Lanfair Valley. The population of the town ranged from 400 to 700 within two months of its founding. The 1910 census listed 40 residents and shortly thereafter was abandoned. Cultural resource field studies were undertaken as part of past environmental reviews to identify if there were any significant sites to be considered for inclusion in NRHP and/or the California Register of Historic Resources (CRHR). The field studies evaluated both historic and prehistoric resources at the Project site.

A multitude of surveys were completed between 1987 and 1998 on project and adjacent land. The former Hart townsite was itself found ineligible for listing to the NRHP in 1998. However, within the 3,910-acre approved project area, seven sites were deemed significant enough to warrant potential eligibility to the NRHP. All seven sites have been avoided as mitigation. Additional mitigation measures include a chain link fence built around the Hart town site cemetery and a 300-foot buffer zone separating the cemetery from the North Overburden Site. The remains of three individuals from the old Hart town are presumed buried at the cemetery. More recent Class III cultural resource surveys were completed in 2018 and produced an additional two sites that could be eligible for NRHP listing. Additional Class III investigation was completed in 2020 and a final report expected in 2021. Preliminary results have yielded one new and potentially eligible site. Regarding the ten potentially significant sites that have been avoided, at least two are not expected to be impacted by mine expansion, four may be removed as eligible sites, and the remaining four sites are scheduled for further investigation and (potentially) treatment plans may be warranted.

During previous and current operations, no paleontological or archaeological deposits were uncovered during the construction and operational phases of the Project.

20.1.5 Acid Rock Drainage Potential

Samples of mineralized ore, waste rock, and overburden, were subjected to geochemical testing during prior environmental review to evaluate the acid rock drainage (ARD) potential, acid generation potential, and extractable metals. The average neutralization potential (NP) was 54.3 tons $CaCO_3$/1,000 tons of material. The acid generating potential (AP) was 2.4 tons $CaCO_3$/1,000 tons of material resulting in an NP:AP ratio of 22.6. An NP:AP ratio greater than 3 or 4 is considered to have enough buffering capacity to mitigate hazards associated with acid rock drainage.

The 1998 EIS/EIR analyzed the potential for acidic conditions in pit water and found, once again, the acid-generating sulfide minerals found in project ore and waste rock are very limited, and the natural alkalinity provided by the rock, soils, and surface water inflows further minimize the potential for acidification of the pit water. The current permits require analysis for ARD potential in the pit water. Any pit containing poor water quality water would have to be backfilled. The surfaces of these backfilled pits would consist of coarse material to allow infiltration of meteoric waters to minimize ponding.

There was no evidence of ARD during the previous operations, and mine pit water at the bottom of Leslie Anne Pit did not show any signs of sulfidic oxidation. Based on historic analytical data and the previous operational experience, the potential for ARD is quite low. Notwithstanding the historical record documenting a low potential for acid rock drainage impacts, additional studies will occur to further document this potential within any underlying areas of potential mine expansion.

20.1.6 Visual Impact

Visual impact to the viewshed of the Lanfair valley has been assessed during each of the previous EIS/EIR reviews that concluded in 1990 and 1998. Viewshed impact was found not significant (after mitigations) in the 1990 EIS/EIR, but then found significant and unavoidable[6] in the 1998 EIS/EIR. The actual viewshed impact resulting from Viceroy operations at the project site was less than anticipated based on the very successful land reclamation and revegetation programs conducted at the Project site. It is anticipated that mine expansion will create significant and unavoidable visual impacts during active mining, even if they are found less than significant post mining. Further visual impact assessments will be undertaken to determine significance.

Irrespective of the final significance determination, public lands at the southern end of the Castle Mountain range, and encompassing the current and proposed mine expansion area, are classified as Class IV visual resource[7] lands. By definition, this ranking class assumes (permits) a high degree of modification to the natural landscape that may be a major focus of viewer attention. In summary, a finding of significant and unavoidable impact during active mining follows the findings of past assessments at the Project (1998 EIS/EIR) and is well within the land use planning objectives assigned to Visual Resource Class IV lands.

[6] Visual Impacts and Air Quality were the only two resource categories found as significant and unavoidable by the 1998 EIS/EIR report.
[7] BLM's regional (and most recent) land use plan covering public lands surround the project site is known as the Desert Renewable Energy Conservation Plan (DRECP). This land use planning amendment established a visual resource ranking classification from Class I (most protective against modification of the natural environment) to Class IV (least protective).

Visual impact mitigation measures will be taken during project design development such as selection of neutral desert paint colors for all facilities, tanks, and infrastructure, and design specifications requiring use of dark sky friendly plant and access lighting to minimize light pollution. Mitigation for impact to visual resources is further discussed in section 20.5, Mine Closure and Reclamation.

20.2 PROJECT PERMITTING AND PERMITTING PROCESS

20.2.1 Federal and State Environmental Reviews

Environmental impact review of mine development projects in this jurisdiction must comply with both state and federal impact review and planning analysis.

At the state level, the CEQA provides the mechanism for environmental review and additional planning and reclamation requirements are afforded through County implementation of California's Surface Mining and Reclamation Act (SMARA). Environmental review is conducted on the federal level through the NEPA process.

The federal lead agency with responsibility for the Project is the BLM. The California state lead agency for the Project is the County of San Bernardino (County). Both agencies cooperate to prepare a single environmental review document. The EIS/EIR, federal, state, county, and local agency officials review and comment on the analysis provided through the CEQA and NEPA process.

There are also multiple public review and comment periods as part of the scoping and public involvement process initiated by a Notice of Intent on the proposed action published in the Federal Register. Once the co-lead agencies complete and publish a Final EIS/EIR, then subsequently, each of the lead agencies prepare their respective approvals. The BLM issues a Record of Decision (ROD) and associated project stipulations to satisfy project specific mitigation measures adopted by the agency to lessen project impacts. The County will hold a final (public) hearing where the Planning Commission will ultimately vote to certify the EIR and approve (or deny) the CUP for the Project and associated conditions of approval, which similar to the BLM, provide mitigation to lessen project impacts.

Once lead agency operating permits have been granted, CMV can apply to remaining local, state, and federal agencies who issue further discretionary and non-discretionary permits.

20.2.2 Discretionary Operating Permits

Discretionary permits are those which can be denied following the completion of the specific permitting process; that is to say, completion of all required elements of the permitting process does not guarantee project approval by the lead agency. Project lead agency operating permits are generally issued following successful completion of the discretionary project review process that includes impact analysis, comparison to alternative actions, and public participation and comment. Aside from the lead agency discretionary permits (discussed above), additional discretionary authorization is required from the California regional Water Boards for operation of mine waste units; the resulting permit is known as a Waste Discharge Requirements (WDR). This is also a discretionary permit process that ends with a public hearing and vote by the regional water quality control board membership; all before a permit can be issued.

The Colorado River Basin Region RWQCB regulates surface and groundwater quality for waters of the state and waste discharges to land/water for the Project area, pursuant to California Water Code, and Title 27 California Code of Regulations (CCR). Specific to the type of activity at the Project site, the RWQCB regulates discharges of mining waste, operation of a waste management unit for treatment, storage, or disposal of a mining waste (mining unit), and for stormwater discharges associated with industrial or construction activity.

For a mine operation, the water board typically considers the heap leach pad and detoxified, filtered mill tailings (and ponds) as a Group B mining unit, and overburden rock piles as Group C mining units (or mining waste), per Title 27 of the CCR. While overburden is technically a recognized waste, a Group C mine waste is not expected to affect surface or groundwater quality.

A mine operator receives RWQCB approval to operate a mining unit after submitting a Report of Waste Discharge (ROWD) report and application. Once the ROWD application is deemed complete, a draft WDR permit will be issued for a 45-day public comment period. Following a response to applicable comments (and making any corresponding changes to the draft WDR permit), the RWQCB will assign a hearing date to the full regional water board membership where staff (and the applicant) will review the case file with the regional water board members during a public hearing. The hearing culminates with a vote of the board members either in favor of the RWQCB staff recommendation (to approve) the WDR permit, or alternatively, to deny it.

The RWQCB regulated mining units currently in operation at the Project site are operated under recently approved (June 2020) WDR Permit No. R7-2020-0004. A new or revised permit will be required for mine expansion facilities documented in this technical report.

20.2.3 Non-Discretionary Operating Permits

Nondiscretionary permits are generally those types of permits whereby once the permit process is successfully completed by an applicant, the issuing agency must issue a permit, there is no discretion for them not to issue if all requirements have been satisfied. Aside from the lead agency permits, most remaining local, state, and federal agency permits are non-discretionary type operating permits which are obtained for a project after the lead agency environmental analysis has been completed, mitigation measures assigned, and the proposed action authorized by the lead agencies. Noteworthy permits that will be necessary to construct and operate mine expansion facilities are found on Table 20-1. It should be noted that all the permits found on Table 20-1 are active at the current Project site; so, in some cases a new permit will be required, while other permits might be amended, or retained in current form.

A few of the more substantial non-discretionary permit processes are discussed in more detail in the ensuing sections 20.2.4 to 20.2.6 found below.

Table 20-1: Castle Mountain Mine Permit Requirements

Agency	Permit	Action Authorized	Permit Modification Required (For Mine Expansion)
U.S. Bureau of Land Management	Record of Decision (ROD)	Mining[8] – Federal	YES – Additional NEPA review; new ROD required
San Bernardino County	Mining Conditional Use Permit (CUP)	Mining[9] – State/County	YES – Additional CEQA review; new CUP required
National Park Service	Decision Record	Continued Existing Use (WWF) and access agreement	Temporary NPS Authorization[10] is currently provided.
U.S. Bureau of Land Management	Right-of-Way Lease (ROW)	Construction of roads/utilities	YES – New or amended ROWs required
U.S. Fish & Wildlife Service	Biological Opinion (BO) Authorizing Take	Take of Desert Tortoise (federal)	YES – Additional US FWS Section 7 Consultation; new Biological Opinion
Colorado River Basin Regional Water Quality Control Board (Region 7)	Waste Discharge Requirements (WDR)	Operation of: Heap Leach Pad and Ponds, Overburden	YES – Amended Waste Discharge Requirements for expanded heap leach facility (and support), and expanded overburden piles
Mojave Desert Air Quality Management District	Authority to Construct and Operate	Permit to emit air pollutants	YES – New ATC permits for new sources with the potential to emit air pollutants.
California Dept. of Fish and Wildlife	1602 Lake and Streambed Alteration Agreement	Disturbance to state jurisdictional drainage features	YES – Delineation of new land disturbance
California Dept. of Fish and Wildlife	2081 Incidental Take Permit	Take of Desert Tortoise (state)	POTENTIALLY[11]
U.S. Army Corps. of Engineers	Jurisdictional Determination[12]	Disturbance to Waters of the United States	POTENTIALLY – Additional delineation of new land disturbance is required to determine jurisdiction
Colorado River Basin Regional Water Quality Control Board (Region 7)	Industrial and/or Construction Stormwater General Permits	Discharge of pollutants or the prevention thereof, relating to industrial/construction activity	YES – Amended Stormwater Pollution Protection Plans will be required

[8] Including activities ancillary to mining.
[9] See note 8.
[10] Once NPS completes review of the pending Plan of Operations, and issues a permanent authorization, no further permitting modification is expected in association with mine expansion.
[11] At this project, CDFW historically has accepted the U.S. FWS Biological Opinions and not required a separate 2081 permit, nonetheless, this will be reviewed during mine expansion.
[12] If necessary, a Section 404 permit, likely a Nationwide Permit

Agency	Permit	Action Authorized	Permit Modification Required (For Mine Expansion)
San Bernardino County Fire Dept., Hazardous Materials Division	Hazardous Materials Business Plan	Storage and use of regulated hazardous materials	YES – HMBP will have to be amended to account of new/increased material quantity
San Bernardino County Fire Dept., Hazardous Materials Division	Hazardous Waste Generator/Handler	Handle/generate hazardous waste	YES – HMBP will have to be amended to account of new/increased material quantity
San Bernardino County Fire Dept., Hazardous Materials Division	California Accidental Release Plan (CalARP), Risk Management Plan	Above threshold storage of Sodium Cyanide	YES – HMBP will have to be amended to account of new/increased material quantity
San Bernardino County Public Health, Environmental Health Services	Public Water System Permit	Operate of state public water system	NO – Potential for minor modification
San Bernardino County Public Health, Environmental Health Services	Domestic Waste Discharge Permit	Install/operate a domestic sewage system	NO
San Bernardino County Building and Safety Dept and Office of the Fire Marshal	San Bernardino Building & Safety, and Fire Dept. construction permits	New/modified building construction	YES – New building and safety and fire permits will be required prior to construction

20.2.4 BLM Right-of-Way Permits

Right-of-way (ROW) permits may be required for certain access roads, water pipelines, and power line transmission corridors if changes to current systems are included with mine expansion. CMV has maintained right-of-way access across BLM managed roads through their inclusion in the current mining plan of operation and resulting 1998 Record of Decision and the 2020 Decision Record. Other, new, ROW lease agreements with the BLM, and potentially other agencies, will likely be required for installation of new infrastructure such as power or water lines associated with mine expansion.

20.2.5 Local Air District

The Mojave Desert Air Quality Management District (MDAQMD) is the local air district and enforces local district rules, California Air Resources Board (CARB) rules and regulations, and federal USEPA rules and regulations, as they apply to stationary sources of air pollution, including sources of fugitive dust.

20.2.5.1 Mobile Sources

Mobile sources most common at mine sites are heavy, diesel-fueled equipment which are regulated under a CARB rule known as the In-Use Off-Road Diesel-Fueled Fleets Regulation. Reporting and compliance are managed through the state’s Diesel Off-Road On-Site Reporting System (DOORS), which is an online registration and management system that tracks mobile fleets. Regulated mobile fleets must be registered with DOORS. With some exceptions, all mobile equipment engines at the current Project site are classified as Tier 4 Interim or Tier 4 Final, which produce the lowest available level of PTE. The mobile fleet proposed for the mine expansion will continue to use mostly Tier 4 equipment.

20.2.5.2 Stationary Sources

Stationary sources (including what are commonly referred to as point sources) with the potential to emit air pollutants (including criteria and toxic air contaminants) must be permitted through the air district prior to construction and operation. The initial permits are known as Authorities to Construct (ATC) and transition into Permits to Operate (PTO) following construction and startup of operations. Once the operator is aware of the types of equipment (with PTE) and air emission control devices that will be used, applications and supporting documentation are submitted to the air district. MDAQMD staff will inform the applicant within 30 days if the application is complete, and permits are generally issued within the following 90 days. However, for more complicated project permits, a longer time period may be required.

Operating permits issued under Title V of the Clean Air Act (Title V permits), are also procured when potential emissions are above specified thresholds (often referred to as “major source” facilities), for specified non-major sources subject to regulation under the National Emissions Standards for Hazardous Air Pollutants (NESHAP), or in other specialized cases. For the proposed mine expansion operations, the Project would be subject to NESHAP Subpart EEEEEEE, and thus would be required to obtain a Title V operating permit, regardless of the level of emissions. Applications for Title V permits must be submitted within 12 months of commencing applicable operations.

GHG emissions from stationary sources are evaluated under CARB’s Mandatory Reporting Rule (MRR), CARB’s Cap-and-Trade Program, and USEPA’s Greenhouse Gas Reporting Program

(GHGRP). Depending on the level of emissions, reporting may be required under one or both of the CARB MRR program and USEPA GHGRP, and the procurement of carbon allowances may be required under the CARB Cap-and-Trade Program. Both reporting and allowance procurement are annual requirements, as applicable. GHG emissions from stationary sources may also be covered under the Prevention of Significant Deterioration (PSD) and Title V Operating Permit Programs when GHG emissions are above specified levels.

Fugitive dust air quality issues are addressed both by individual ATCs and District Rule 403 that requires the operator to maintain a district approved Dust Control Plan, which the Project maintains. Additional mitigation measures are also prescribed by the lead agency operating permits that focus on dust suppression primarily due to vehicular traffic, drilling, crushing, screening, and stockpiling of mined materials. Fugitive dust in the form of PM_{10} is currently monitored at the Project site fence-line (project boundary) through continuous operation of four E-BAM air particulate monitoring stations.

20.2.6 Water Extraction Rights

Per California laws and judicial precedent pertaining to groundwater rights, CMV has maintained its historic overlying water rights to groundwater aquifers in the Lanfair basin by using extracted water for beneficial use within the basin. Additionally, Annual Notices of Groundwater Extraction are filed each year with the RWQCB for all existing water wells (Table 20-2). Historical borehole and well locations are shown in Figure 20-1.

Table 20-2: Active Water Rights

#	Record #	Well ID	In Use	GPM (avg.)	Proof of Use (Annual Report of Water Extraction)
1	G363178	W-18P	Yes	50	July 2020
2	G363181	W-14P	Yes	50	July 2020
3	G363645	W-45P	Yes	15	July 2020
4	G363635	W-42	No	NA	July 2020
5	G363603 (new Rec. # pending)	W-001	Yes	150	Pending new Rec. #
6	(new Rec. # pending)	W-002	Yes	150	Pending new Rec. #

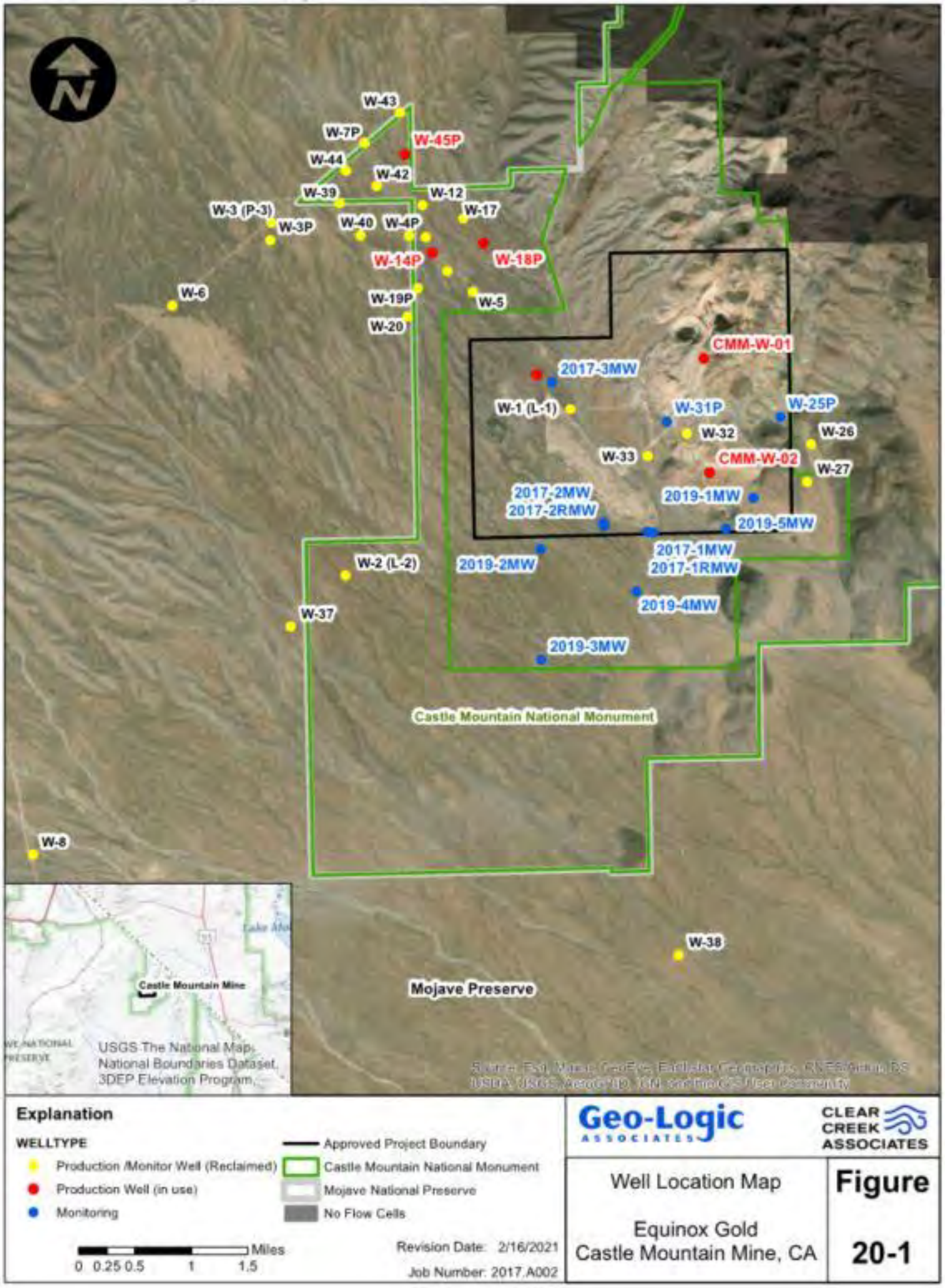


Figure 20-1: Historical Borehole and Well Locations

20.3 LAND USE AND PROPERTY

20.3.1 Mojave National Preserve

The Mojave National Preserve (the Preserve) was established on October 31st, 1994 through the California Desert Protection Act. The Preserve is managed by the National Park Service and is comprised of 1.6 million acres to the west, east, and south of the Project. Current mine facilities are not located inside the Preserve though historically some water production wells, and groundwater monitoring wells were located within the Preserve boundary. These facilities and the related operation of the WWF are recognized by and included in the approved Management Plan for the Mojave National Preserve. Mine expansion at the Project site will not conflict with public lands in the Preserve managed by the National Park Service.

20.3.2 Castle Mountains National Monument

The Castle Mountains National Monument (CAMO) was established on February 12, 2016, through an executive order signed by President Barack Obama, as authorized under the Antiquities Act. The reserved Federal lands encompass approximately 20,920 acres and the boundaries fall between the Project site, and the Mojave National Preserve on all four sides. The Secretary of the Interior manages these lands through the National Park Service, pursuant to applicable authorities, consistent with the purposes and provisions of the proclamation.

Shortly after the creation of CAMO, CMV began cooperation with the National Park Service to correct errors evident of the proclamation process, most significant being the presence of the Project's WWF facilities located partially within CAMO. A Plan of Operation describing these facilities, and their continued operation, was prepared and submitted to park staff in 2018 and a final decision is expected in 2021. However, until that time, current Project operations within the monument are regulated and authorized by the National Park Service through a Temporary Authorization permit granted by the National Park Service. It is expected that mine expansion activity (namely continued use of the WWF water system) co-located within the monument will operate under a park service approved Plan of Operations.

20.4 SOCIAL AND COMMUNITY IMPACT

The Project site is located near the California-Nevada border in eastern San Bernardino County. As noted above, it is essentially surrounded by both the Mojave National Preserve and the more recent Castle Mountains National Monument, both of which place restrictions on the activities allowed within these conservation focused park units. Aside from conservation, generally low impact land use activities prevail in the undeveloped desert areas of eastern San Bernardino County, California and in the adjacent southern Clark County, Nevada. One exception is that in many instances, historic land use activities such as livestock grazing, and mining have been replaced more recently by large commercial scale solar power generating plants. Transportation and utility corridors are also located throughout the region including Interstate and State highways, county and local roads, railroads, power transmission lines, utility pipelines, and communication stations.

Mining has been a continuous activity in eastern San Bernardino County and southern Clark County for the past century, even though many small- to intermediate-sized mine operations have shut down in the surrounding Mojave region (e.g. Vanderbuilt, Colosseum, and Morning Star mines). Many existing and former towns were founded as mining communities, including Searchlight, NV, Mountain Pass, Ivanpah, and Hart in California, and Henderson in Nevada.

Recreational use of land adjacent to the Project site is similar to the activities found throughout the Mojave desert region and includes casual use enjoyment of the area's natural and historic resources, off-highway vehicle (OHV) touring, sightseeing, hiking, bird watching, hunting, stargazing and rock collecting.

Small communities are scattered throughout this region of the Mojave Desert along major transportation corridors, such as Interstate 15, Interstate 40, and the Union Pacific and BNSF Railroads. Towns such as Baker and Needles (California) provide services to highway travelers and are long-established railroad and trade/service centers for the surrounding desert region. Privately owned lands are interspersed in the desert regions although these residences are becoming less common as large tracts of land are set aside for renewable power generation, land conservation usually in the form of a National Monument, an Area of Critical Environmental (ACEC), or restricted access Department of Defense land.

The nearest communities are Searchlight, NV, located about 16 miles by direct line to the northeast, and Nipton, CA approximately 16 miles by direct line to the northwest. The Project site is not visible by California or Nevada communities and is only visible by a small handful of private residencies located throughout the Mojave National Preserve as private inholdings. The inholdings are mostly undeveloped, though some have developed structures that are mostly habituated only part of the year (usually not during the hot summer months). The nearest developed inholding lies approximately five miles south of the project. CMV maintains an active relationship with most of these private residences through the Mojave Landowners Association.

Of the two nearest population centers, Searchlight is a municipality but Nipton is an unincorporated community with less than a dozen residents in the local RV park and one separate private residence west of the railroad tracks that bisects the town limits. Public services in the desert communities are limited. Baker is the nearest California town with educational, fire, ambulatory, and police services and is located 54 miles due west from the Project site. Searchlight and Laughlin, NV have limited educational, fire, ambulatory, and police departments, and community medical facilities. Other goods and services are available from the larger regional urban centers of Las Vegas/Henderson, NV and Barstow and Victorville/Apple Valley on the western edge of the Mojave Desert in California.

There are no housing or public services at the Project site. Most employees live in Nevada with a small number residing in California.

Other than mining employment, most workers are employed in industries such as tourism, trade, and services or, in the case of southern Clark County, the gambling and hospitality industry.

Local community stakeholders include local private landowners (Mojave Landowners Association); the towns of Searchlight, NV, and Nipton, CA; the National Park Service; the BLM and a diverse group of conservation focused non-governmental organizations. CMV has existing and planned communications and outreach with all parties.

20.5 MINE CLOSURE AND RECLAMATION

Mine closure and reclamation are addressed in the Mine and Reclamation Plan approved by the project lead agencies (San Bernardino County and BLM). SMARA outlines the State's regulatory and statutory requirements that are implemented by San Bernardino County as the project lead agency. The State Division of Mine Reclamation (DMR) enforces SMARA but often defers primacy to the local lead agency to enforce SMARA, in this case San Bernardino County. BLM reclamation requirements are outlined in 43 CFR §3809.5 and in the project-specific Records of Decision. While both lead agencies have an oversight role regarding mine reclamation, the State (county) generally takes a lead role. The BLM and State/County relationship is managed through multiple active Memorandum of Understanding (MOU) agreements between the BLM California State Office and the California State Mining and Geology Board.

The Project has filed a series of updated Mine and Reclamation Plans pursuant to state and federal requirements. The original Mine and Reclamation Plan was approved in September 1990, a revision was approved in January 1998, and the most recent revision is dated July 15, 2019.

During the previous operations, reclamation activities included a formal revegetation research program; salvaging plants and cacti for later transplantation back in the reclaimed areas; establishing a greenhouse/nursery; creating a propagation area for native plants; local seed collection; and maintaining several control areas at various representative re-vegetated sites across the mine. A research program to identify and test for successful desert revegetation and reclamation techniques was instituted until about 2001. Research topics included seed treatment and germination; plant propagation; pest management; plant salvage; soil stockpile management; plant hormone use; vesicular-arbuscular mycorrhizae use; plant/water relationships; plant spacing patterns; and density, diversity, herbivory, and irrigation design. The nursery-grown plants and salvaged plants temporarily stored at the Castle Mountain Mine greenhouse and nurseries were transplanted onto rehabilitation areas around the Project. The successes (and failures) from these research initiatives formed the backbone of the ultimately successful revegetation program at the Project site. Completed reclamation and revegetation at the Project site is often recognized as one of the best regional examples of successful mine reclamation, and many of these successful strategies have been retained for future use.

Interim reclamation activities commenced in June 2001 and continued through 2005. Most site infrastructure including maintenance and electrical shops, administration and warehouse buildings, primary, secondary, and tertiary crusher facilities, laboratory, refinery and change buildings were removed from the site. At the request of the government agencies, several facilities were not reclaimed including a 250,000 gal water tank, some of the water production wells, and various access roads (as part of the region’s fire prevention efforts). The heap leach pad, process plant and lined ponds were the last facilities reclaimed and areas revegetated from 2005 to 2007 after the regional water board recertified the heap leach pad material from a Group B to a Group C mine waste. Group C mine waste poses no impact to groundwater or surface water quality.

Overburden piles were recontoured to conform to requirements of the approved Mine and Reclamation Plan. The heap leach piles were constructed to provide vertical relief by stacking at different total heap heights instead of one uniform upper elevation. The north and south overburden piles had mounds added to the surface to create vertical relief. Placement of overburden to cover the north and south clay pits was also undertaken. This was a mitigation the operator agreed to conduct, to remove the blight on the land from previous clay quarries that were abandoned and not reclaimed. In 2010, the section of the power transmission line from site to Walking Box Ranch was removed. Future mine reclamation related to mine expansion will follow these similar reclamation and revegetation strategies that have produced successful results.

Over 2,000 plants were transplanted in the rehabilitation areas in 1996 and an additional 8,203 plants were transplanted in 2001. Native seed was collected in the immediate area of the mine. In 2005-2006, 1,242 plants were transplanted to the heap leach area. Areas of the reclaimed mine site were aerially seeded or hand-broadcast with native seeds. By 2007, most plant transplants and seeding were complete and by 2012 the Project had satisfied the revegetation objectives for plant density and diversity which defined the release criteria for successful reclamation of the Project site. The extensive efforts made to revegetate the site, as near to its natural state as possible, have produced excellent vegetation communities that closely match the species richness and diversity of the surrounding natural landscape.

The current reclamation plan is contained in the document titled “Mine Plan and Reclamation Plan Ver.2.1 (90M-013)”. The reclamation plan includes a statement of purpose, goals, schedule, and metrics for success. The plan focuses on revegetation and restoration of the Project site to a condition as close as possible to that of the site before disturbance. This includes natural plant assemblages, visual compatibility with natural land contours, conditions of safety for the public, and restoration of habitat for indigenous wildlife species. The plans include decommissioning of mining facilities and structures, maintenance and management of reclaimed areas, and post-mine considerations for hydrology, groundwater, and drainage, erosion, and sedimentation. The plan envisages returning the land for recreational and wildlife use after mine closure and reclamation.

All Water Board regulated facilities will be closed in accordance with CCR Title 27.

Future amendments to the mine and reclamation plan to account for mine expansion are expected to include facility decommissioning, land recontouring, and revegetation as has been described herein, due to the success of the previously documented reclamation effort. This project site is not statutorily required to backfill mined pits, as the approved reclamation plan and financial assurance dates before state implementation of the Backfill Regulation on December 18, 2002. However, voluntary backfilling of certain parts of the mine will be included in the future amendment to the mine and reclamation plan.

20.5.1 Financial Assurance

CMV is required to post financial assurance mechanisms with the various state and federal agencies pursuant to the governing laws and regulations. Most financial assurance mechanisms are in the form of surety bonds issued to the Project by commercial carries. The agencies usually have a requirement for the project proponent to produce an annual Financial Assurance Cost Estimate (FACE) report and increase or decrease the total assurance accordingly. At the Project site, the County administers the reclamation bond for both the project lead agencies (County and BLM). This bond was most recently increased in 2021 to approximately $3,742,611. An additional FACE report was recently approved by the Colorado River Basin RWQCB for cleanup and closure of water board regulated facilities, namely the heap leach pad and ponds. This bond will be established in 2021 at $4,137,230 and will bring the total 2021 anticipated project bonds to $7,879,841.

Bonds will continue to be reviewed on an annual basis and increased or decreased based on new liability and inflation or to account for completed mine reclamation, respectively. Bonds will similarly be increased based on new liability associated with the mine expansion.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

Capital costs for the Castle Mountain Project have been estimated by M3, NMS, GLA, and The MINES Group with appropriate input from Equinox. All costs are in US dollars ($). The estimate was broken down by Work Breakdown Structure (WBS) and organized into specific areas of responsibility as follows:

- M3 – Site Infrastructure, Heap Leach Solution Handling, Crushing and Ore Storage, Process Plant and Gold Refinery, Ancillary Facilities, Raw Water Pumping and Distribution, Main Substation and Power Distribution, Indirect Costs (Contractor Indirects, EPCM, Commissioning, Working Capital, First Fills, Sales Tax etc.) Contingency, Reclamation
- NMS – Mining Equipment Costs, Slope Monitoring Equipment, Mining Clearing, Grubbing and Pre-Strip
- GLA – Heap Leach Pad Expansion and Filtered Tailings Facility
- Equinox – Owner's Cost (Pre-Production Labor, Insurance, Permitting, Legal etc.), Utility Transmission Line (2018 PFS estimate)

The initial and sustaining capital costs are summarized in Table 21-1.

Table 21-1: Capital Cost Summary

Item	Initial ($M)	Sustaining ($M)	Total ($M)
Mine Mobile Equipment[1]	154	70	224
Mine Development	41	11	52
Mine Total	**195**	**81**	**276**
General Siteworks	11	-	11
Heap Leach and Solution Handling	38	56	94
Process Plant	62	-	62
Tailings Filtration and Storage	16	1	17
Infrastructure	41	-	41
Freight	8	-	8
Direct Plant and Infrastructure Total	**176**	**57**	**233**
EPCM, Vendor Support and Other Indirects	51	-	51
Transmission Line	15	-	15
Owner's Cost, Working Cap and Taxes	40	-	40
Sub-total Plant and Infrastructure	**282**	**-**	**-**
Contingency[2]	33	9	42
Total CAPEX	**510**	**147**	**657**
Less Leased Mining Equipment	***121***	**-**	***(121)***
Total CAPEX (with Leased Mining Equipment)	**389**	**-**	**536**

Note 1: Mining equipment includes all applicable sales tax.
Note 2: Contingency not included for mining and working capital.

The initial capital costs by main area are shown in Table 21-2.

Table 21-2: Summary of Initial Capital Costs by Area

Area	Cost ($M)
TOTAL Mining	**195**
General Site	11
Heap Leach Solution Handling	10
Carbon Adsorption	6
Heap Leach	28
Crushing and Ore Storage	16
Grinding and Gravity	12
Leach/CIL	10
Tailings Handling and Filtration	15
Filtered Tailing Storage Facility	2
Desorption and Carbon Regeneration	8
Electrowinning and Gold Refinery	4
On-site Water Supply, Storage, Distribution	3
Off-site Water Supply	18
Sub-station and Electrical Distribution	5
Process Plant Utility, Mobile Equipment and Ancillaries	9
Reagent Mixing and Storage	5
Mining Fleet Service Facilities	6
Freight	8
TOTAL Direct Plant and Infrastructure Costs	**176**
Construction Indirects (i.e. mobilization, bussing, facilities)	15
EPCM	31
Vendor Supervision and Commissioning Support	2
Spare Parts (Capital and Commissioning)	1
First Fills and Operating Spares	2
Transmission Line	15
Working Capital	16
Owner's Cost	16
County Sales Tax	8
TOTAL Indirect Plant, Infrastructure and Owner's Costs	**106**
Contingency	33
TOTAL CAPEX	**510**
Less Leased Mining Equipment	***(121)***
Total CAPEX (with Leased Mining Equipment)	**389**

The initial capital costs by commodity for Process and Infrastructure are shown in Table 21-3.

Table 21-3: Summary of Process Plant Initial Capital Costs by Commodity[1]

Item	Supply Cost ($M)	Labor Costs ($M)	%
Sitework	15.3	14.4	16.9
Concrete	5.7	8.1	7.8
Structural & Architectural	8.7	6.1	8.4
Mechanical Equipment	39.2	5.4	25.3
Piping	14.1	11.4	14.5
Electrical Equipment	9.1	1.0	5.7
Electrical Bulks	4.9	3.3	4.7
Instrumentation	3.1	1.7	2.7
Sub-Total	**100.1**	**51.4**	
Construction Equipment	16.7		9.5
Freight	8.0		4.5
Total			**100.0**

Note 1: Excludes Transmission Line Direct Cost

Total initial capital cost is estimated at $389 million excluding the mining equipment fleet which is estimated at $121 million and expected to be leased to own over five years, or a total of $510 million considering the fleet purchased upfront. Sustaining capital costs for the project are primarily accounting for mining and additional stages of the heap leach pad and filtered tailings facility development. Total sustaining capital costs during production until closure are $147 million. Closure costs totaling $22 million are included separately for the end of mine life.

The estimated capital costs for Phase 2 of the Castle Mountain Project are considered to have an accuracy of -10% to +15% and are estimated in Q4 2020 US dollars. Allowance for escalation and foreign exchange fluctuation has not been included. The Project will be primarily sourced from the United States (> 90%) and have minimal exposure to currency risk.

The estimated costs are based on this project being executed by an experienced EPCM contractor(s) in the hard rock mining industry. In addition, it is assumed that all contracts and subcontracts are based on a lump-sum basis or a competitively bid unit cost basis, such as per cubic yard of concrete placed.

21.1.1 Design Basis

The cost estimate is based on preliminary engineering including 250 feasibility level design drawings for the mill & infrastructure, heap leach pad expansion, filtered tailings facility, and supporting ancillary facilities covering all engineering disciplines. The entire facility has been developed in AutoCAD® Plant 3D. The drawings and the process design criteria were used as basis for detailed material take-offs for all disciplines and development of an equipment register to define the mechanical, electrical and instrumentation needs of the project. Table 21-4 summarizes the reference documents available for estimation purposes.

Table 21-4: Reference Documents Summary

	Drawings and/or Documents
Engineering Design Drawings:	
Flowsheets	19
General Arrangements	53
Architectural	sketches
Civil	22
Concrete	4
Structural Steel	Yes
Piping and Instrumentation Diagrams (P&IDs)	123
Electrical Schematics	35
Instrumentation Schematics	Yes
Engineering Specifications:	
Project Standards and Site Conditions	Yes
Process Design Criteria	Yes
Project Scope of Facilities	Yes
Equipment Specifications	Data Sheets Only
Engineering Lists and Logs:	
Equipment List (with Buildings)	Yes
Valve List	Yes
Cable Schedule	Yes
Instrument Log	Yes
Other References:	
Material Take-Off's:	
Civil	Yes
Concrete	Yes
Structural Steel	Yes
Mechanical sketches	No
Piping	Partial*
Electrical	Partial*
Instrumentation	Yes

* Piping drawings developed for large solution lines and raw/fire water distribution. Electrical physical drawings developed for main substation. No in-plant electrical physical drawings were developed.

21.1.2 Installation Cost Basis (Labor Rates and Construction Equipment)

Labor rates for the process plant and infrastructure estimate are based on a weighted average of prevailing non-union shop wages from a published source (Davis-Bacon; 50%) and actual labor rates on site for Phase 1 construction (50%). Craft labor has been estimated at the overall average rates illustrated in Table 21-5 and this includes both Direct and Indirect Costs of the contractor such as fringe benefits, social burden, overtime adjustment, contractor supervision, overhead and profit.

Table 21-5: Labor Rates

Craft	Labor Rate ($/h)
Bricklayer	51
Carpenter	59
Cement Mason	51
Electrician	60
Ironworker	70
Laborer	37
Millwright	74
Operator	60
Pipefitter	66

Resulting blended crew labor rate for the project, by discipline or commodity and based on a weighted average of crew mix is shown in Table 21-6. The rates below are not all-inclusive rates and exclude, subsistence, travel time, construction equipment, mobilization, and demobilization. These items are estimated separately within the capital estimate. Labor rates are based on 50-hour work weeks.

Table 21-6: Blended Crew Rate

Discipline	Labor Rate ($/h)
Civil	45
Concrete	59
Reinforcing Steel	70
Structural	70
Mechanical Equipment Installation	74
Piping	69
Electrical	59
Instrumentation & Controls	57

Labor rates for the heap leach pad and filtered tailings estimates are based on current and proposed labor rates from Phase 1 construction.

Construction equipment costs were estimated according to the tasks performed and the hours involved. This results in a direct cost for the process plant (excluding the heap leach and filtered tailings facility) of approximately $11/h based on 700,000 man-hours. This rate varies significantly based on discipline from very low for electrical and instrumentation to above $15/h for architectural and steel.

Construction equipment for the heap leach and filtered tailings facility requires use of larger equipment and results in a direct cost of approximately $45/h based on 200,000 man-hours. Overall, this totals $16.7M. Small tools, man-lifts, construction generator sets etc. are included as an allowance within the contractor field labor rate.

21.1.3 Process & Infrastructure Equipment Cost Basis

Major process mechanical and electrical equipment were identified based on the project design basis and appropriate criteria. Prices were then solicited for the major equipment depicted on the flow sheets and/or the equipment register. M3 obtained budgetary quotations from a minimum of three qualified suppliers for all major packages. Major capital equipment categories for this project included mechanical, structural, and electrical. Received proposals were then evaluated for technical compliance and an average or median cost used in the estimate. The selected mid-point price is intended to allow for some growth allowance. For other equipment, M3 used firm and budgetary pricing data from recent M3 projects.

Pricing sources for major equipment (>$50,000 capital cost anticipated) in this estimate were as follows:

- 89% budgetary quotes (140 of 157 equipment items)
- 3% informal quotes (4 of 157 equipment items)
- 8% from M3 historical data for similar equipment (13 of 157 equipment items)

21.1.4 Bulk Commodities Cost Basis

Bulk material pricing was estimated through a combination of budgetary proposals, data from recent projects in the region, and actual pricing from Phase 1 construction as well as check reviews by local contractor.

<u>Earthworks</u> – Earthworks and liner quantities for the expanded heap leach pad, filtered tailings facility, new event pond and solution collection channel were estimated by GLA based on design drawings and specifications. Geomembrane liner includes a 12% provision for waste. Contingency includes an allotment of $4.1M to screen the material, which was not required for Phase 1 and may not be required for Phase 2.

Earthworks quantities for the process plant and project infrastructure were estimated by M3 on a similar basis. Unit costs were based on costs for recently completed Phase 1 development construction. These rates were then validated by M3 using their recent project internal database for projects local to the area. Earthworks unit rates are shown in Table 21-7.

Table 21-7: Earthworks and Liner Unit Rates

Description	Unit	Unit Cost ($/Unit)
Pad Clearing and Grubbing	acre	1,350
Topsoil Removal (Heap Leach Pad; 1 ft. thick)	acre	4,840
Pad Cut – From Cell and Ponds	yd^3	2.75
Pad Fill – Haul and Place in Cell	yd^3	3.00
Liner Subgrade Preparation (3 in depth)	yd^3	4.60

Description	Unit	Unit Cost ($/Unit)
80 mil LLDPE Geomembrane (Supply & Install)	ft^2	0.65
80 mil HDPE Geomembrane (Supply & Install)	ft^2	0.65
60 mil HDPE Geomembrane (Supply & Install)	ft^2	0.60
5 mm Geonet (Supply & Install)	ft^2	0.45
Overliner (Sourced from Historic Heap)	yd^3	4.60
Filtered Tailings Facility Surface Preparation	acre	7,421
Diversion Channel Excavation	yd^3	3.59
Channel Rip-Rap Lining	ft^2	4.00
Main Process Plant Cut to Fill	yd^3	5.74
Process Plant Finish Grading	yd^2	0.92

Concrete – Concrete quantities for the process plant were estimated by M3 based on developed design drawings, specifications and equipment loading diagrams from budgetary proposals. Quantity estimates, site layouts and plans were provided to two general contractors for review and they provided budgetary estimates. The resulting average supply and placement rate has been estimated at $1,250/$yd^3$ for an all-in concrete unit rate. This rate includes formwork, concrete supply, reinforcement steel and curing. A factor of 5% has been applied for waste and miscellaneous pads.

Structural Steel – Structural steel quantities for the process plant and ancillary buildings were estimated by M3 based on design drawings. Material take-off values were determined for light, medium and heavy steel as well as grating, handrail, stairs, etc. Unit costs for steel including installation labor and equipment requirement were based on recently completed M3 projects in the area. These estimated costs are shown in Table 21-8.

Table 21-8: Structural Steel Unit Rates

Description	Unit	Unit Cost ($/Unit)
Light Steel (Supply and Install)	ton	7,025
Medium Steel (Supply and Install)	ton	4,825
Heavy Steel (Supply and Install)	ton	4,185
Steel Grating	ft^2	34
Handrail	ft	61
Stairs	ft (vert.)	688
Steel Detailing	lb	0.20

Piping – Piping, fittings and valve costs are based off material take-offs from site plans and project layouts as well as project P&IDs developed for all major systems throughout the processing plant and ancillary supporting areas. Piping material costs in general were sourced from a combination of quotes from local suppliers and M3 internal database of recently completed projects in the region.

Electrical and Instrumentation – Major electrical equipment such as transformers, switchgear, motor control centers (MCCs) and variable frequency drives (VFDs) costs are based on budgetary supplier proposals as described above in 21.1.3 for equipment costs. Electrical bulks such as cables, cable tray and conduit have been estimated based on material take offs generated using site plans and project layouts as well as project single line diagrams and electrical house preliminary plans. Instrumentation and control costs have been estimated in similar fashion.

21.1.5 Cost Basis (Freight)

Freight, customs, and duties have been included as a percentage (8%) of equipment and material costs in the estimate for domestically sourced equipment. It is recognized that bulks and smaller equipment will likely have a much lower percentage, but the 8% also comprises factory and/or in-transmit warehousing, inspections, port charges, road and rail charges, freight forwarding as well as actual freight costs. Freight costs were further validated using vendor budgetary freight pricing on major equipment.

21.1.6 Cost Basis (Indirect Costs)

Indirect costs included as part of the project capital estimate are as follows:

- Mobilization/Demobilization
- Contractor Personnel Bussing
- Temporary Construction Facilities and Power
- EPCM
- Vendor Supervision of Specialty Construction
- Vendor Pre-Commissioning and Commissioning Services
- Capital Spares
- Commissioning Spares
- First Fills and Operating Spares
- Working Capital
- Sales Tax
- Owner's Costs
- Contingency

Mobilization/Demobilization – Cost is included at 1.5% of total direct costs except for civil which is included at 5% of total related direct costs.

Contractor Personnel Bussing – Bussing cost unit rates were estimated by Equinox and M3 applied the total direct manhours required for project construction to obtain an overall value. Bussing service using 50 plus passenger will be contracted out of Henderson, NV. Contractor personnel would be bussed to site from a gathering/parking area near Searchlight, NV.

Temporary Construction Facilities and Power – Cost is included at 0.5% of total direct costs for temporary construction facilities and 0.1% of total direct costs for supplemental temporary construction power for the facilities.

EPCM – EPCM costs are factored as percentages of constructed costs as shown in Table 21-9. except heap leach pad and filtered tailings design. Heap leach pad and filtered tailings design costs are included based on budgetary estimate from GLA, which is $375,000. Constructed costs include direct costs plus mobilization and construction utilities.

Table 21-9: EPCM Indirect Costs

Activity	% of Constructed Costs
Management	0.75%
Engineering[1]	6.00%
Project Services	1.00%
Project Controls	0.75%
Construction Management	6.50%
Total EPCM (before fee)	**15.00%**
EPCM Fee	1.50%
EPCM Construction Trailers	0.20%

Note 1: % Basis not utilized for heap leach pad and tailings facility engineering as noted.

Vendor Supervision, Pre-Commissioning and Commissioning – Cost is included at 1% of total direct costs for each of these elements for a total of 3%.

Capital Spares – Cost is estimated using budgetary proposals from equipment suppliers. The following equipment is anticipated:

- Cone Crusher Motor
- Ball Mill Motor
- Pinion Assembly including bearing
- Trunnion Bearings (Pad and Thrust)
- Ball Mill VFD Module
- Cyclone Feed Pump
- Interstage Screen
- Conveyor/Feeder Common Motors
- Barren and Pregnant Solution Pump VFD Phase Modules

Commissioning Spares – Cost is included at 0.5% of plant equipment costs.

First Fills and Operating Spares – First fill requirements were determined by project design criteria to service the project for a period of 3 months. In similar fashion, 3 months of operating spare parts were assumed to carry the project well into full operations.

Sales Tax – Combined State of California and San Bernardino County sales tax is included at 7.75% applied to plant equipment and material cost without freight.

Working Capital - Working capital for the Project is estimated to be $15.5 million for Phase 2. The working capital is the capital required for operations before any revenue from Phase 2 ounces is produced by the mine and is based on the operating costs for the mine, process, and G&A costs for the Project.

Owner's Cost – Owner's cost was developed by Equinox and has been estimated specifically for the execution phase of the project including detailed engineering, procurement, and construction support. The detailed breakdown is shown in Table 21-10.

Table 21-10: Owner's Cost Summary

Description	Cost ($000)
Preproduction Labor - Plant	2,211
Preproduction Labor - Mine	2,173
Total Owner's Project Mgmt. Team	1,489
Construction/Commissioning Power	400
Access Road (Improvements)	400
Construction Support Vehicles	248
Additional Trucks (F-150 Supercrew)	360
Safety	360
First Aid and Medical (Construction)	400
Road Signage	60
Temporary Trailers – Owner's Team	40
Sanitation Facilities	360
Insurance (COC, Liability, Marine)	4,200
Environmental and Permitting	450
Community Relations, Lobbying	450
Consultants	180
Operations Training	400
Operations Readiness	200
Geotech Testing	150
Metallurgical Testing	150
Legal Fees	540
Accounting Fees	180
Communications (Upgrades)	360
IT Purchases	50
SAP	200
Furniture	75
Travel (Additional to Project Team)	90
TOTAL Owner's Cost	**16,176**

Contingency – Cost Contingency is an amount added to an estimate to allow for items, conditions, or events for which the area, occurrence, and/or effect is uncertain, and that experience shows will likely result in additional costs. These costs are typically estimated using statistical analysis or judgment based on past asset or project experience.

Contingency excludes:

- Major scope changes such as changes in product specification, capacities, building sizes, and location of the asset or project,
- Extraordinary events such as major strikes and natural disasters,

- Management reserves, and
- Escalation and currency effects.

Contingency is included at a percentage of the total contracted cost including commissioning and spare parts. Contingency is calculated on a discipline basis, and then applied across each area. The values assigned have been established through an analysis of the level of detail included in estimating the value. For example, as a very high percentage of equipment has been quoted it is considered to have a high level of accuracy and therefore 10% contingency is considered reasonable. The basis for contingency for each discipline is shown in Table 21-11.

Table 21-11: Project Contingency Basis

Activity	% of Constructed Costs	Contingency ($M)
Civil/Sitework	15.0	4.4
Concrete	12.5	1.7
Structural Steel	12.5	1.6
Architectural	12.5	0.3
Mechanical	10.0	4.4
Piping	15.0	3.8
Electrical	12.5	2.3
Instrumentation	12.5	0.6
Construction Equipment	10.0	1.7
Contractor Indirects	10.0	8.4
Heap Leach Overliner	Estimated by GLA	4.1
Total		**33.3**

*Contingency is also applied to the transmission line cost and is not included in the costs listed above.

21.1.7 Mining Capital Cost

Initial mining capital costs are based on converting to an Equinox owned mining fleet from the contract-based fleet being utilized for Phase 1 operations, necessary parts and spares for the fleet, as well as slope monitoring equipment and mine development and pre-stripping. A major part of the mining equipment fleet could be leased which results in a reduction of $121 million of initial capital. Leasing the mining equipment adds to the operating cost; however, the net impact is an improvement to the Internal Rate of Return (IRR). Initial capital and fleet are summarized in Table 21-12. Costs for the mobile mining fleet have been based on budgetary quotations from a minimum of two qualified suppliers and often three or four.

Table 21-12: Mining Initial Capital Summary

Description	Cost ($M)
Mobile Equipment	146.4
Spares and Inventory	5.9
Slope Monitoring Equipment	1.9
Clearing and Grubbing (Mine)	4.5
Initial Pre-Strip	36.8
Total	**195.5**

Key mining equipment is summarized in Table 21-13. Details on the mining equipment fleet can be found in Section 16. The initial capital costs included here are for the mobile equipment purchased in the first 3 years of the expansion.

Table 21-13: Mining Equipment Summary

Equipment	Details	Total
Production Blasthole Drill	8 7/8"	2
Wall Control Drill	4 1/2' - 9"	2
Hydraulic Shovel	2996 hp 44.5 yd^3	2
Wheel Loader	1739 hp 28 yd^3	2
Haul Truck	2650 hp 250 ton	17
Track Dozer	600 hp	5
Wheel Dozer	620 hp	2
Grader	290 hp 16 ft	3
Water Truck	1450 hp 32,000 gal	2
Wheel Loader	541 hp 10 yd^3	1
Haul Truck	825 hp 61 ton	3
Excavator	524 hp 6 yd^3	1
Tire Manipulator	Large Tire	1
Vibratory Compactor	130 hp 7.5 ft	1
Backhoe	105 hp 1.3 yd^3	1
Articulated Truck	450 hp 40 ton	1
Fuel and Lube Truck	100 ton 8,000 gal	1
Tractor and Low Bed	160 ton	1
Flatbed Hiab Truck	10 ton	1
Rough Terrain Forklift	32 ton	1
Shop Forklift	18 ton	1

21.1.8 Sustaining Capital Cost

Sustaining capital costs for the expansion of the Castle Mountain Project have been estimated and are primarily for continued development of the heap leach pad and filtered tailings as well as mining activities. Sustaining capital costs for the heap leach pad and filtered tailings include indirect costs and contingency. Mining sustaining capital costs reflect the cost of operating the new mobile mining fleet to handle the provision of ore to the process plant as well as stripping and placement of waste material. Equipment for slope monitoring, fleet spares and inventory, and development costs are included as well. Sustaining capital costs allotted for Phase 2 are shown in Table 21-14.

Table 21-14: Summary of Sustaining Capital Costs

Ph. 2 Year	Mining ($M)	Plant & Infrastructure ($M)	Leach Pad/Event Pond ($M)	Filtered Tailings ($M)	Total Cost ($M)
1	1.8	-	3.4	-	5.2
2	8.1	-	-	-	8.1
3	13.6	-	-	1.3	14.9
4	16.5	-	27.0	-	43.5
5	3.3	-	-	-	3.3
6	0.7	-	-	-	0.6
7	7.3	-	22.2	-	29.5
8	12.9	-	-	-	12.8
9	15.5	-	-	-	15.5
10	0.2	-	12.6	-	12.8
11	0.7	-	-	-	0.7
12	0.2	-	-	-	0.2
13	0.1	-	-	-	0.1
14	Reclamation – 3.5				3.5
15	Reclamation – 3.5				3.5
16	Reclamation – 15.0				15.0
Total	**80.9**	**-**	**65.2**	**1.3**	**169.4**[1]

Note 1: Including reclamation

The estimated sustaining capital costs for the expansion of the Castle Mountain Project are considered to have an accuracy of -10% to +15% and are estimated in Q4 2020 dollars.

21.1.9 Mining Sustaining Capital Cost

Table 21-15 shows the mining sustaining capital cost summary including both planned initial capital and sustaining capital requirements.

Table 21-15: Mining Sustaining Capital Cost Summary

Description	Units	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Mobile Equipment	$000	-	$7,543.6	$10,509.4	$12,565.9	$462.9	$489.8	$7,190.5	$12,067.7	$15,312.9	$75.0	$75.0	$75.0	$75.0
Spares & Inventory	$000	-	-	-	-	-	-	-	-	-	-	-	-	-
Slope Monitoring Equipment	$000	-	$587.3	$160.6	$160.6	$587.3	$160.6	$160.6	$762.9	$160.6	$160.6	$587.3	$160.6	-
Development & Pre-stripping	$000	$1,845.8	-	$2,899.7	$3,742.9	$2,291.9	-	-	-	-	-	-	-	-
Total	**$000**	**$1,845.8**	**$8,130.9**	**$13,569.7**	**$16,469.3**	**$3,342.0**	**$650.4**	**$7,351.1**	**$12,830.6**	**$15,473.5**	**$235.6**	**$662.3**	**$235.6**	**$75.0**

21.2 OPERATING COST ESTIMATE

21.2.1 Summary – Operating Cost

Operating costs are broken down by area including mining, process plant and G&A. G&A staffing plans and wages were estimated by Equinox. Processing and tailings haulage were estimated by M3 and include the combined heap leach and mill facilities. Mining was estimated by NMS. Mining equipment purchase costs are all considered capital costs and excluded from operating costs. Table 21-16 shows a summary of operating cost elements over the course of Phase 2.

Table 21-16: Operating Cost Phase 2 Summary

Description	Unit Cost ($/ton mined)
Mining	1.75
Description	**$/ton ore**
Mining	6.20
Processing (Total)	2.45
G&A	0.65
Sub-Total	**9.30**
Refining and Transportation	0.02
Total	**9.32**

Table 21-17 shows a summary of operating cost elements over the course of the mine life.

Table 21-17: Summary of Annual Operating Cost

Phase 2 Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Total
Units	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M/y**	**$M**
Mining	100.0	118.5	120.9	128.0	130.4	123.5	122.0	134.4	137.1	124.0	128.5	115.6	69.2	14.8	-	-	**1,567**
Processing	44.9	45.5	45.5	45.5	45.3	45.5	45.6	45.6	45.3	45.6	45.6	44.8	41.4	25.8	4.7	3.5	**620**
G&A	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.4	11.1	9.8	3.4	2.5	**164**
Annual Cost	**156.8**	**175.3**	**177.7**	**184.8**	**187.2**	**180.3**	**178.8**	**191.2**	**193.9**	**180.8**	**185.3**	**171.7**	**121.7**	**38.2**	**8.1**	**6.0**	**2,351**
Units	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton**	**$/ton processed**
Mining	5.18	6.07	6.13	6.58	6.87	6.33	6.13	6.97	7.15	6.32	6.58	6.17	4.14	4.77	-	-	**6.20**
Processing	2.31	2.33	2.33	2.33	2.33	2.34	2.33	2.33	2.33	2.35	2.33	2.38	2.58	6.89	-	-	**2.45**
G&A	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.59	0.60	0.69	2.62	-	-	**0.65**
Annual Cost	**8.10**	**8.99**	**9.05**	**9.50**	**9.79**	**9.25**	**9.05**	**9.89**	**10.07**	**9.24**	**9.50**	**9.15**	**7.41**	**14.28**	**-**	**-**	**9.30**

21.2.2 Mining Operating Cost

Mining Operating costs were developed by NMS using the developed mine plan for the project. The mining fleet and operations will be fully self-performed by Equinox. Labor costs were developed based on a staffing plan and rate schedule from Equinox based on current Phase 1 operations and forecasted rates for personnel required for staffing. No contingency is applied to the estimated operating costs generated. Table 21-18 summarizes the Phase 2 mining operating cost including mining labor, energy, consumables (parts, explosives etc.), contract services and consumables tax.

Table 21-18: Operating Cost Summary

Description	Units	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Total	Average ($/t mined)
General Mine	$/y	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.9	6.8	5.1	1.7	89.5	0.10
Drilling	$/y	6.0	7.0	8.0	9.9	9.7	10.3	10.0	10.2	10.5	9.2	9.6	8.1	3.0	0.4	111.8	0.12
Blasting	$/y	8.9	10.5	11.1	12.8	11.9	12.5	12.5	13.5	13.8	12.0	12.7	10.8	5.1	1.0	149.0	0.17
Loading	$/y	16.0	23.4	21.0	24.9	21.7	19.6	24.2	27.2	23.5	21.9	17.7	15.2	6.2	1.2	263.6	0.29
Hauling	$/y	39.8	51.4	48.5	50.3	57.1	54.0	47.6	57.8	63.0	54.2	60.7	55.8	35.5	7.4	683.1	0.76
Roads & Dumps	$/y	20.4	19.1	22.5	19.5	20.7	20.2	20.6	18.6	19.3	19.6	20.7	18.8	14.0	3.0	257.0	0.30
Contract Services	$/y	2.0	0.1	3.0	3.9	2.4	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	12.8	0.01
Total	$/y	**100.0**	**118.5**	**120.9**	**128.0**	**130.4**	**123.5**	**122.0**	**134.4**	**137.1**	**124.0**	**128.5**	**115.6**	**69.2**	**14.8**	**1,566.9**	**1.75**

21.2.2.1 Mining Labor

Mining labor costs were estimated by NMS for the expanded project. Mining activities including supervision, engineering, and geology as well operators and maintenance personnel will begin ramping up before the start of Phase 2, increasing to 240+ quickly and topping out at 284 employees in Year 9. Figure 21-1 shows the anticipated manpower throughout development of the mine. Table 21-19 shows the anticipated peak staff list (Year 9).

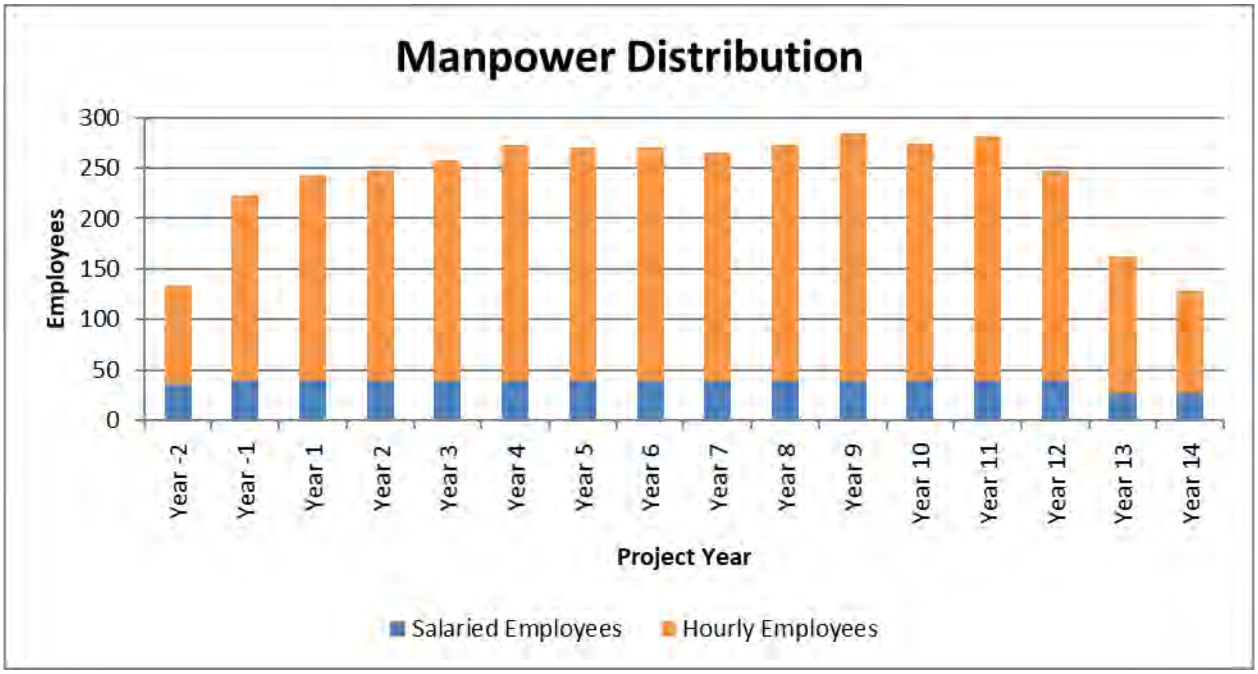


Figure 21-1: Mining Labor Plan

Table 21-19: Mine Staffing (Peak – Year 9)

Description	Qty
Mine Manager	1
Mine Superintendent	1
Mine Foreman	4
Drill and Blast Foreman	2
Mine Training Coordinator	1
Mine Shifter & Dispatcher	4
Mine Clerk	1
Mine Supervision (Total)	**14**
Mine Maintenance Superintendent	1
Electrical Foreman	1
Maintenance Foreman	4
Maintenance Planner	2
Maintenance Clerk	1
Mine Maintenance (Total)	**9**

Description	Qty
Chief Mine Engineer	1
Senior Mine Engineer	1
Drill & Blast Engineer	1
Geotechnical Engineer	1
Senior Surveyor	2
Surveying Technician	2
Senior Mine Geologist	1
Mine Geologist	2
Environmental Coordinator	1
Grade Control Tech	4
Engineering and Geology (Total)	**16**
Operations (Hourly)	167
Maintenance (Hourly)	78
TOTAL	**284**

21.2.2.2 Mining Equipment Fuel

Mining operating costs assumes $2.75/gal diesel. Diesel costs by mining operation and equipment were estimated. Total diesel consumption for the owners fleet during Phase 2 is estimated at 113.8 million gal with an annual peak of 9.7 million gal.

21.2.2.3 Drilling and Blasting

Unit operating costs ($/ton mined) relating to drilling and blasting in Phase 2 Year 1 to 14 are shown in Table 21-20 including appropriate fuel costs. Blasting includes emulsion handling, charging and accessories.

Table 21-20: Drilling and Blasting Unit Costs

	Labor ($/ton)	Energy ($/ton)	Consumables ($/ton)	Tax ($/ton)	Total Unit Cost ($/ton)
Drilling	$0.043	$0.025	$0.061	$0.005	$0.133
Blasting	Supply Contract		$0.163	$0.013	$0.176

21.2.2.4 Loading and Hauling, Roads and Dumps

Unit operating costs ($/ton mined) in Phase 2 Years 1 to 14 relating to loading, hauling, roads and dumps are shown in Table 21-21 including appropriate fuel costs.

Table 21-21: Loading, Hauling, Roads & Dumps Unit Costs

	Labor ($/ton)	Energy ($/ton)	Consumables ($/ton)	Tax ($/ton)	Total Unit Cost ($/ton)
Loading	$0.057	$0.067	$0.173	$0.013	$0.311
Hauling	$0.165	$0.189	$0.412	$0.032	$0.798
Roads & Dumps	$0.135	$0.069	$0.104	$0.008	$0.316

21.2.2.5 Contract Services

Clearing and grubbing as well as supply of magnesium chloride for dust control in Phase 2 Years 1 to 14 are assumed to be on a contract basis and are estimated at $0.114/ton mined.

21.2.3 Process Plant Operating Cost

Operating costs were developed using reagent, grinding media and power consumptions based on the process flow sheet. Labor costs were developed based on a staffing plan and rate schedule from Equinox based on current Phase 1 operations and forecasted rates for personnel required for staffing. No contingency is applied to the estimated operating costs generated (see Table 21-22 and Table 21-23).

Table 21-22: Summary of Process Operating Costs by Area (Typical Processing Year)

Area Description	Typical Annual Cost ($M/y)	Typical Unit Cost ($/ton processed)
Heap Leach/Solution Handling	20.7	1.07
Crushing & Ore Storage	2.3	0.13
Mill/Leach/Detox/Filtration	13.3	0.66
Desorption, Regeneration and Refinery	3.8	0.20
Reagents, Utilities, Ancillaries	5.3	0.27
TOTAL	**45.5**	**2.33**

Table 21-23: Summary of Process Operating Cost by Use ($M/y)

Phase 2 Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	Total
Labor	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	11.6	1.6	1.2	165.2
Process Plant Power[1]	6.8	7.1	7.1	7.1	7.0	7.1	7.1	7.1	7.0	7.1	7.1	6.9	6.4	4.0	0.6	0.4	95.9
Reagents & Consumables	22.9	23.2	23.2	23.2	23.1	23.2	23.3	23.3	23.1	23.3	23.3	22.7	20.0	7.7	2.3	1.7	309.5
Maint. Parts & Repairs	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.5	1.9	0.2	0.1	35.9
Process Mobile Equip Fuel	0.9	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.9	0.6	0.1	0.1	13.5
Annual Cost	44.9	45.5	45.5	45.5	45.3	45.5	45.6	45.6	45.3	45.6	45.6	44.8	41.4	25.8	4.8	3.5	**620.0**
$/ton	2.31	2.33	2.33	2.33	2.33	2.34	2.33	2.33	2.33	2.35	2.33	2.38	2.58	6.89	-	-	**$2.45**

Note 1: Raw Water Systems and Ancillaries included in G&A.

21.2.3.1 Process Plant Labor

Process plant labor costs were estimated by M3 for the expanded project increasing the Phase 1 workforce to 121 resulting in an increase to $11.6 M/y annually ($0.59/ton of processed ore). These costs include process plant operations and maintenance labor, lab technicians, operators, mechanics, and electricians etc. Process plant labor staffing plan is summarized in Table 21-24.

Table 21-24: Process Plant Staffing

Description	Qty
Process Manager	1
Process Superintendent	1
Metallurgist/Process Engineer	1
Process Foreman	4
Heap Leach Supervisor	1
Crusher Supervisor	1
Clerk	1
Total Process Admin	**10**
Heap Leach - Piping ROM Ore	12
Heap Leach Helper	4
Primary Crusher Operator	4
Loader Operator	4
Crushing Helper	4
Grinding/Gravity Operator	4
CIL & Thickening	4
Filtration & Clarification	4
Filter Tails - Truck Stacking	12
Grinding/Tailings Helper	4
ADR Operator	8
Refinery Operator	2
Total Process Operations	**57**
Maintenance Supervisor	2
Maintenance Planner	1
Mechanics	12
Electrician	2
Instrument Tech	1
Total Maintenance	**18**
Lab Manager	1
Assayers	4
Sample Preparation Labor	16
Met Tech	2
Total Lab	**23**
TOTAL	**121**

21.2.3.2 Electrical Power

Power consumption is based on developed project equipment list connected kW discounted for operating time per day and anticipated equipment loading level. The cost of power is $0.103/kWh based on construction of a utility transmission line to site. Alternative power supply options that may improve these costs are being investigated (see Table 21-25).

Table 21-25: Power Cost Summary

Area Description	Annual Power Consumption (MWh/y)	Annual Cost ($M/y)	Power ($/ton processed)
Heap Leach/Solution Handling/CIC	20,700	2.0	0.12**
Crushing & Ore Storage	4,700	0.5	0.39***
Mill/Leach/Detox/Filtration	31,100	3.2	2.50***
Desorption & Regeneration/Refinery	10,600	1.1	0.06
Reagents/Lab	1,600	0.2	0.01
Ancillaries and Utilities (G&A)	4,700	0.5*	0.02
Raw Water Systems	11,500	1.2*	0.06
TOTAL	**84,900**	**8.7**	**-**

* Included with G&A costs.
** Based on only heap leach ore; all others based on total ore unless noted.
*** Based on only mill ore.

21.2.3.3 Reagents and Consumables

Reagents and consumables consumption rates were determined from metallurgical test data, current on-site Phase 1 consumption rates and/or industry best practice. Budget quotations and current existing contracts were used as basis for reagent supply from local sources including estimated freight. Table 21-26 and Table 21-27 summarize annual consumptions and costs of reagents and process plant consumables.

Table 21-26: Reagent Cost Summary

Description	Consumption (lb/ton)	Annual Consumption (ton/y)	Annual Cost ($M/y)
Pebble Lime (Heap)	2.58	23,543	3.5
Pebble Lime (Mill)	2.99	1,910	0.3
Pebble Lime (Detox)	0.22	143	< 0.1
Sodium Cyanide (Heap)	0.60	5,475	11.4
Sodium Cyanide (Mill)	0.47	300	0.6
Antiscalant (Heap)	0.04	365	1.1
Antiscalant (Mill)	0.002	1	< 0.1
Carbon (CIC)	0.02	175	0.4
Carbon (CIL)	0.11	70	0.2
Flocculant (Grinding)	0.07	45	0.1
Flocculant (Detox)	0.07	45	0.1
Caustic Soda (Grinding)	0.008	5	< 0.1
Caustic Soda (ADR)	0.16	102	0.1
Leach Aid	0.003	2	< 0.1
Hydrochloric Acid	0.10	64	< 0.1
Sodium Metabisulfite	1.21	775	0.7
Copper Sulfate	0.08	51	0.1
Fluxes	0.001	1	< 0.1
TOTAL			**18.9**
TOTAL ($/ton - Heap)		**0.93**	
TOTAL ($/ton - Mill)		**1.82**	

Table 21-27: Consumables Cost Summary

Description	Consumption	Annual Consumption (unit/y)	Annual Cost ($M/y)
Crushing Liners (Prim/Sec)	0.06 lb/ton	78 ton	0.2
Grinding Liners	0.08 lb/ton	53 ton	0.1
Mill Grinding Media	1.83 lb/ton	1,170 ton	1.2
Filter Cloth	-	-	0.5
Propane	1.7 gpm	894,000 gal	0.9
TOTAL			**2.9**
TOTAL ($/ton – Mill)		**2.35**	
Drip Emitters	-	-	0.4
TOTAL			**0.4**
TOTAL ($/ton – Heap)		**0.02**	

21.2.3.4 Maintenance Parts and Outside Repairs

Parts are estimated using 5% of a project area's mechanical equipment cost and 2% of a project area's electrical equipment cost on an annual basis. Outside repairs are estimated at 10% of an area's parts cost on an annual basis. Maintenance labor is included in the process plant labor count and cost mentioned previously.

21.2.4 General and Administrative Costs

Equinox provided an estimate for the G&A costs for the expanded project of $9.7 M/y. These costs include general and administrative labor, property costs, legal fees, outside services, insurance, public relations, recruiting and other general costs. G&A costs are summarized in Table 21-28 and G&A staffing plan is summarized in Table 21-29.

Table 21-28: General and Administrative Cost Summary

Description	Annual Cost ($M/y)
Labor and Fringes	4.5
Insurances	1.0
Property Taxes	1.5
Community Relations	0.5
Legal Fees	0.5
Consultants	0.3
General Site Power and Maintenance	1.9
Other General Expenses	1.3
TOTAL	**11.5**
TOTAL $/ton	**0.59**

Table 21-29: General and Administrative Staffing

Description	Qty
General Manager	1
Administrative Assistant	1
Controller	1
Assistant Controller	1
Senior BI Specialist	1
Accountant	1
Accounts Payable Clerk	1
Accountant Analyst	1
Administration & Accounting	**8**
Materials Superintendent	1
Purchasing Specialist	1
Data/Inventory Analyst	1

Description	Qty
Repairables and Cores Coordinator	1
Warehouse Supervisor	1
Warehouse Technician	1
Warehouse/Purchasing	**6**
IT Specialist	1
IT Specialist Jr	2
IT	**3**
Human Resources Manager	1
Senior Human Resources Generalist	1
Human Resources Generalist	1
Recruiter	1
Benefits Specialist	1
Payroll Specialist	1
Human Resources Assistant	1
Human Resources	**7**
Safety Supervisor	1
Senior Health and Safety Coordinator	1
Senior Health and Safety Specialist	1
Health and Safety Coordinator	1
Health & Safety Specialist	1
Security	4
Health and Safety	**9**
Environmental Manager	1
Environmental Sr. Coordinator	1
Environmental Coordinator	1
Environmental Specialist	1
Environmental Admin Technician	1
Environmental	**5**
TOTAL	**38**

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

This section presents the cash flow forecast model for the Phase 2 project. This is used in the financial evaluation to determine the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Phase 2 expansion project.

Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost, and sales revenue. The sales revenue is based on the production of a gold bullion. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

For the purposes of this cash flow forecast, the Phase 2 project is considered to have a construction time frame of approximately 24 months. The project is complete once the Phase 2 ROM heap leach pad and the process plant have been constructed, commissioned and are operating.

The following key parameters were used in the construction of the cash flow model and the economic results:

- Gold price at $1,500/oz,
- 100% equity financing with no debt component, and
- Revenues and costs reported in constant Q4 2020 U.S. dollar terms without escalation.

This analysis was completed primarily utilizing a Microsoft Excel-based discounted cash flow model. Currency is provided in US dollars.

22.2 SUMMARY ECONOMIC ANALYSIS

Table 22-1 presents the summary economic analysis results for the Phase 2 project at $1500/oz gold price.

Table 22-1: Summary Phase 2 Financial Results

Category	Units	Value	
Production Summary			
Phase 2 Ore material mined	Mton	894	
Phase 2 Ore tons processed	Mton	253	
Phase 2 Life (Processing)	y	14	
Phase 2 Life (Processing + Rinsing)	y	17	
Heap Leach Ore	Mton	235	
Head grade	oz/ton	0.0119	
Recovery	%	74	
Recovered Gold	koz	2,095	
Mill Ore	Mton	18	
Head grade	oz/ton	0.0665	
Recovery	%	94	
Recovered Gold	koz	1,108	
Total Recovered Gold	koz	3,203	
Total Payable Gold	koz	3,187	
Capital Costs			
Phase 2 Initial Capital	$M	510	
Sustaining Capital	$M	147	
Operating Costs			
Mining	$/ton mined	$1.75	
Mining	$/ton processed	$6.20	
Processing	$/ton processed	$2.45	
G&A	$/ton processed	$0.65	
Refining and Transportation	$/ton processed	$0.02	
Total Operating Cost	$/ton processed	$9.32	
Total Production Cost	$/ton processed	$806	
All-In Sustaining Cost	$/oz Au	$858	
Economic Indicators			
		Without Leasing	**With Leasing**
Internal Rate of Return (IRR), Pre-tax	%	18.9	19.7
Internal Rate of Return (IRR), After-tax	%	17.5	18.3
Undiscounted Cashflow, Pre-tax	$M	1,550	1,539
Undiscounted Cashflow, After-tax	$M	1,280	1,268
Net Present Value (NPV) @5%, Pre-tax	$M	784	784
Net Present Value (NPV) @5%, After-tax	$M	639	639
Payback Period (Based on After-tax)	y	5.3	5.4

22.3 MINE PRODUCTION STATISTICS

Mine production is reported as ROM heap leach grade ore, Mill grade ore and waste from the mining operation. The annual production figures were obtained from the mine plans as reported earlier in this report. Phase 2 production is defined as all ounces placed and processed after the beginning of LOM Year 6 within the project mine plan.

The life of mine ore, waste quantities and ore grade are presented in Table 22-2.

Table 22-2: Phase 2 Ore, Waste and Metal Grades

	Material Moved (Mton)	Gold Grade (oz/ton)
Heap Leach Ore	235	0.0121
Mill Ore	18	0.0685
Waste	641	-
Total	**894**	**0.0157**

22.4 PLANT PRODUCTION STATISTICS

ROM heap leach grade ore will be processed on a conventional heap leach pad, and the mill ore will be processed using a crushing and grinding circuit. Gold will be recovered using carbon columns for the heap leach ore and a hybrid leach/CIL circuit for the mill ore. A conventional carbon desorption, electrowinning, and refinery plant will produce a gold doré bar.

The estimated gold recoveries which were used are presented below:

- The average recovery for the ROM heap leach ore with rinsing: 74%
- The average recovery for the mill ore: 94%

Marketing Terms

A doré bar will be produced and sent to a precious metal refinery. The refining charges are negotiable at the time of the agreement. The refining terms and transportation charges used in this analysis are shown in Table 22-3.

Table 22-3: Marketing Terms

Description	Term Basis
Payable Gold (%)	99.5%
Refining Charge ($/oz)	$0.56
Transportation Charges ($/oz)	$0.95

22.5 PRODUCTION SCHEDULE PARAMETERS

The Phase 2 project from an economic analysis perspective begins with the commitment to detailed engineering activities and procurement of major equipment in preparation for construction which is expected to start 2.5 years ahead of start-up. The period of project execution resulting in significant capital spend with construction activities will begin approximately 2 years prior to full operations starting. Figure 22-1 and Table 22-4 below illustrates Phase 2 project related activities through project start up..

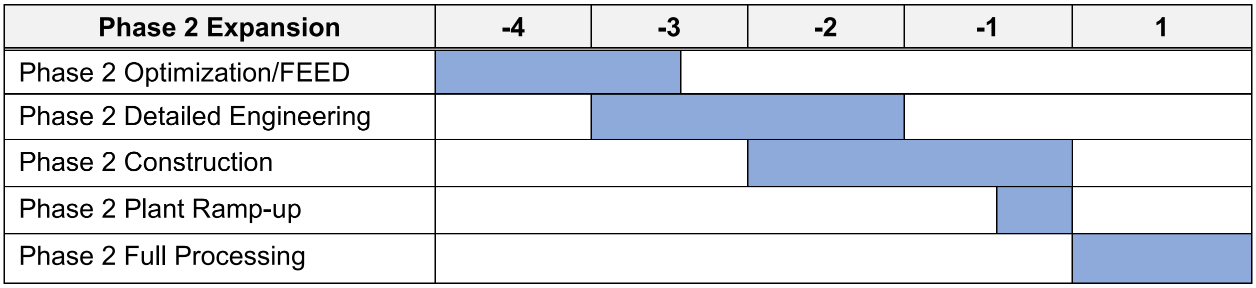


Figure 22-1: Phase 2 Schedule

Table 22-4: Phase 2 Initial Capital Spend Plan

Phase 2 Year	Ore Production (kton/y)	Mining Initial Capital Spend ($M)	Plant Initial Capital Spend ($M)	Working Capital ($M)	Total ($M)
Pre-Prod Year -3	5,150	-	28	-	**28**
Pre-Prod Year -2	5,150	109	204	-	**313**
Pre-Prod Year -1	11,150	62	68	14	**144**
Phase 2 Prod Year 1	19,300	24	-	1	**25**
Expanded Operations	19,500	Sustaining	Sustaining	N/A	-

22.6 CAPITAL EXPENDITURE

22.6.1 Initial and Sustaining Capital

22.6.1.1 Initial Capital

The financial indicators have been determined with 100% equity financing of the initial capital. The initial capital costs included in the financial model are shown below and detailed in Section 21. Table 22-5 summarizes the initial capital cost of the Phase 2 expansion. The planned capital cost spend by quarter is shown in Figure 22-2.

Table 22-5: Initial Capital Cost Summary

Area	Initial Capital ($M)
Mine	195
Process Plant and Infrastructure	282
Contingency	33
Total CAPEX	**510**
Less Leased Mining Equipment	*(121)*
Total CAPEX (with Leased Mining Equipment)	**389**

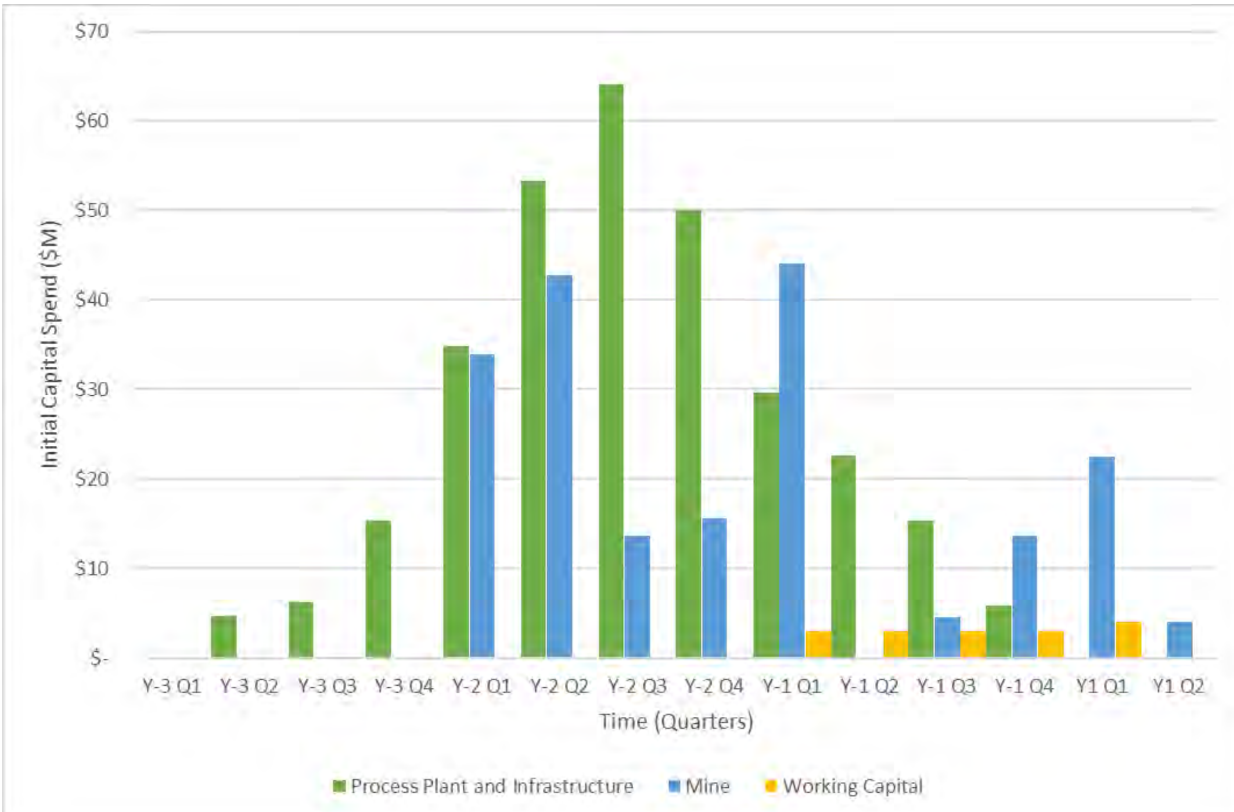


Figure 22-2: Initial Capital Spend Plan (Quarterly)

22.6.1.2 Sustaining Capital

Sustaining capital expenditures during the production period have been included in the financial analysis. The sustaining capital contained in the financial model is estimated at $147 million.

22.6.2 Working Capital

Working capital for the Project is estimated to be $15.5 million for Phase 2. The working capital is the capital required for operations before any revenue from Phase 2 ounces is produced by the mine and is based on the operating costs for the mine, process, and G&A costs for the Project. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.6.3 Salvage Value

An allowance for salvage value has been included in the cash flow analysis which was based on 5% of the capital cost of equipment and is estimated at $2.5 million.

22.6.4 Reclamation/Closure Costs

Reclamation and closure costs are estimated to be $22.0 million and account for activities required to comply with anticipated future amendments to the mine and reclamation plan for mine expansion. These activities include facility decommissioning, land recontouring and revegetation.

22.7 NET REVENUE

Net revenue was determined by applying estimated gold prices to the payable gold estimated for each year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the value of payable metals sold minus treatment and transportation charges. The gold sales price used in the evaluation is $1,500/oz.

22.8 ROYALTIES

Royalty payments are included for several royalties; the estimated royalty payments for the life of the mine totals $213.6 million and are shown in Table 22-6.

Table 22-6: Royalties Summary

Claim/Patent	%	Phase 2 Total Royalty ($000)	Owner
Turtle Back	5	288	Conservation Fund
Milma	5	288	Conservation Fund
Golden Clay	5	78,303	Huntington Tile
All Claims	2.65	126,569	Franco-Nevada
Pacific Clay	2	8,132	American Standard

22.9 OPERATING COST

Life of mine Cash Operating Costs include mine operations, process plant operations, general administrative cost and refining/transportation charges. Table 22-7 shows the estimated operating cost by area per ton of ore processed.

Table 22-7: Operating Cost Summary

Description	Unit Cost ($/ton mined)
Mining	1.75
Description	**$/ton ore**
Mining	6.20
Processing (Total)	2.45
G&A	0.65
Sub-Total	**9.30**
Refining and Transportation	0.02
Total	**9.32**

22.10 TAXATION

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation, and depletion. Income tax rates for state and federal are as follows:

- State rate: 8.8%

- Federal rate: 21.0%

Income taxes were calculated on the taxable income described above using the federal and state rates.

22.10.1 Depreciation

Depreciation was calculated using the MACRS (Modified Accelerated Cost Recovery System) which is the tax depreciation system used in the United States. Under MACRS, fixed assets are assigned to a specific asset class, which has a designated depreciation period. The majority of fixed assets were assigned a 7 year depreciation period.

22.10.2 Depletion

The percentage depletion method was used in the evaluation. It is determined as a percentage of gross income from the property, not to exceed 50% of taxable income before the depletion deduction. The gross income from the property is defined as metal revenues minus downstream costs from the mining property (smelting, refining and transportation). Taxable income is defined as gross income minus operating expenses, overhead expenses, depreciation, and state taxes.

22.11 PROJECT FINANCIAL INDICATORS

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the project. The evaluation shows the following financial indicators without leasing of the mining equipment:

- Undiscounted Cashflow, After-Tax: $1,280 million
- NPV @ 5%, After-Tax: $639 million
- IRR %, After-Tax: 17.5%
- Payback (y): 5.3

The evaluation shows the following financial indicators should mining equipment be leased:

- Undiscounted Cashflow, After-Tax: $1,268 million
- NPV @ 5%, After-Tax: $639 million
- IRR %, After-Tax: 18.3%
- Payback (y): 5.4

Table 22-8 shows the Phase 2 detailed financial model while Figure 22-3 shows the cost and revenue summary by Phase 2 project year.

Table 22-8: Phase 2 Detail Financial Model

	$/ton mined	$/ton ore	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Mining Operations																						
ROM Ore (kton)			236,350	-	-	-	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	18,250	14,775	2,576	-	-
Gold Grade (oz/ton)			0.0121	-	-	-	0.0108	0.0112	0.0112	0.0106	0.0102	0.0104	0.0127	0.0111	0.0121	0.0120	0.0136	0.0128	0.0187	0.0176	-	-
Contained Gold (koz)			2,851	-	-	-	196	205	204	193	186	190	232	202	220	218	248	234	277	45	-	-
Mill Ore (kton)			16,169	-	-	-	1,058	1,259	1,480	1,217	723	1,256	1,662	1,022	916	1,364	1,273	493	1,922	524	-	-
Gold Grade (oz/ton)			0.0685	-	-	-	0.0644	0.0473	0.0515	0.0703	0.0462	0.0721	0.0649	0.0616	0.0559	0.0854	0.0929	0.0932	0.0825	0.0643	-	-
Contained Gold (koz)			1,108	-	-	-	68	60	76	86	33	91	108	63	51	117	118	46	159	34	-	-
Waste (kton)			641,245	-	-	-	43,830	43,676	44,238	53,694	54,808	53,307	53,284	59,179	59,449	58,744	60,354	42,516	11,989	2,178	-	-
Total Material Mined (kton)			893,764	-	-	-	63,138	63,186	63,968	73,161	73,781	72,812	73,196	78,450	78,615	78,358	79,877	61,258	28,686	5,278	-	-
Stockpile (kton)			1,710	-	-	-	145	18	-	61	370	-	-	256	128	-	4	83	-	645	-	-
Total Material Moved (kton)			895,473	-	-	-	63,283	63,204	63,968	73,222	74,151	72,812	73,196	78,706	78,743	78,358	79,881	61,341	28,686	5,923	-	-
Process Plant																						
ROM Ore (kton)			235,208	-	-	-	18,250	18,250	18,250	18,250	18,066	18,228	18,250	18,250	18,016	18,250	18,250	17,548	14,775	2,576	-	-
Gold Grade (oz/ton)			0.0119	-	-	-	0.0108	0.0112	0.0112	0.0106	0.0100	0.0104	0.0127	0.0111	0.0117	0.0120	0.0136	0.0116	0.0187	0.0176	-	-
Contained Gold (koz)			2,804	-	-	-	196	205	204	193	180	190	232	202	210	218	248	204	277	45	-	-
Gold Recovery (%)			74.0%	0.0%	0.0%	0.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%	67.0%			
Recovered Gold (koz)			2,095	-	-	-	132	137	137	129	121	127	155	136	141	146	166	137	185	95	86	65
Mill Ore (kton)			17,702	-	-	-	1,203	1,277	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,278	1,277	1,278	1,278	1,169	-	-
Gold Grade (oz/ton)			0.0665	-	-	-	0.0635	0.0474	0.0515	0.0695	0.0466	0.0714	0.0649	0.0622	0.0544	0.0854	0.0928	0.0650	0.0825	0.0743	-	-
Contained Gold (koz)			1,177	-	-	-	76	61	66	89	60	91	83	79	69	109	119	83	105	87	-	-
Gold Recovery (%)			94.0%	0.0%	0.0%	0.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	94.0%	0.0%	0.0%
Recovered Gold (koz)			1,108	-	-	-	72	57	62	84	56	86	78	75	65	103	112	78	99	82	-	-
Total Ore (ROM & Mill)			252,910	-	-	-	19,453	19,527	19,528	19,528	19,343	19,506	19,528	19,528	19,294	19,527	19,528	18,825	16,052	3,745	-	-
Gold Grade (oz/ton)			0.0157	-	-	-	0.0140	0.0136	0.0138	0.0144	0.0124	0.0144	0.0161	0.0144	0.0145	0.0168	0.0188	0.0152	0.0238	0.0353	-	-
Contained Gold (koz)			3,982	-	-	-	273	265	270	282	240	281	315	282	279	327	366	287	382	132	-	-
Gold Recovery (%)			80.5%	0.0%	0.0%	0.0%	74.6%	73.2%	73.6%	75.6%	73.7%	75.8%	74.1%	74.7%	73.8%	76.0%	75.8%	74.9%	74.5%	133.8%	0.0%	0.0%
Total Recovered Gold (koz)			3,203	-	-	-	203	194	199	213	177	213	233	210	206	249	278	215	285	177	86	65
Payable Metal																						
Payable Gold (koz)			3,187	-	-	-	202	193	198	212	176	212	232	209	205	248	276	214	283	176	86	64
Income Statement ($000)																						
Metal Prices																						
Gold ($/oz)			$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00	$1,500.00
Revenues																						
Gold Revenue ($000)			$4,780,994	$0	$0	$0	$303,672	$289,766	$296,809	$317,665	$264,054	$317,838	$348,463	$313,988	$307,560	$371,552	$414,468	$320,531	$424,735	$264,227	$128,952	$96,714
Refining Charges			$1,785	$0	$0	$0	$113	$108	$111	$119	$99	$119	$130	$117	$115	$139	$155	$120	$159	$99	$48	$36
Transportation			$3,028	$0	$0	$0	$192	$184	$188	$201	$167	$201	$221	$199	$195	$235	$262	$203	$269	$167	$82	$61
Net Revenues			$4,776,182	$0	$0	$0	$303,366	$289,475	$296,511	$317,346	$263,788	$317,518	$348,113	$313,672	$307,250	$371,178	$414,051	$320,208	$424,307	$263,961	$128,822	$96,617
Operating Cost	**$/ton mined**	**$/ton ore**																				
Mine	$1.75	$6.20	$1,566,941	$0	$0	$0	$100,003	$118,476	$120,933	$128,005	$130,418	$123,540	$121,970	$134,403	$137,146	$124,005	$128,523	$115,557	$69,164	$14,797	$0	$0
Process Plant - Heap Leach		$1.44	$365,189	$0	$0	$0	$27,259	$27,689	$27,562	$26,984	$27,303	$26,884	$27,401	$27,248	$27,241	$26,843	$26,919	$26,357	$23,179	$8,144	$4,672	$3,504
Process Plant - CIL		$1.01	$254,575	$0	$0	$0	$17,622	$17,793	$17,932	$18,520	$17,988	$18,615	$18,135	$18,299	$18,032	$18,725	$18,660	$18,381	$18,210	$17,664	$0	$0
G&A		$0.65	$164,102	$0	$0	$0	$11,444	$11,452	$11,452	$11,452	$11,433	$11,450	$11,452	$11,452	$11,428	$11,452	$11,452	$11,379	$11,092	$9,819	$3,368	$2,526
Total Operating Cost (Minus Refining)		$9.30	$2,350,808	$0	$0	$0	$156,328	$175,410	$177,879	$184,961	$187,142	$180,488	$178,958	$191,402	$193,848	$181,025	$185,553	$171,674	$121,645	$50,423	$8,040	$6,030
Royalties																						
FNV			$126,569	$0	$0	$0	$8,039	$7,671	$7,858	$8,410	$6,990	$8,414	$9,225	$8,312	$8,142	$9,836	$10,972	$8,486	$11,244	$6,995	$3,414	$2,560
Conservation Fund			$575	$0	$0	$0	$0	$0	$57	$382	$132	$4	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Huntington Tile			$78,303	$0	$0	$0	$0	$0	$0	$0	$93	$151	$197	$505	$1,569	$7,008	$18,145	$17,382	$28,102	$5,152	$0	$0
American Standard			$8,132	$0	$0	$0	$266	$2,019	$2,569	$1,894	$407	$977	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value			-$2,500																-$2,500			
Reclamation/Closure			$22,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,500	$3,500	$15,000
Gross Margin			$2,192,294	$0	$0	$0	$138,732	$104,375	$108,149	$121,699	$69,023	$127,484	$159,732	$113,453	$103,691	$173,308	$199,380	$122,666	$265,816	$197,891	$113,868	$73,026
Depreciation			$642,495	$0	$84,942	$38,442	$189,290	$46,072	$39,674	$39,111	$35,208	$27,942	$27,345	$22,859	$18,374	$19,113	$14,864	$11,079	$9,835	$8,027	$6,125	$3,390
Operating Income after Depreciation			$1,549,799	$0	-$84,942	-$38,443	-$50,557	$58,302	$68,475	$82,588	$33,815	$99,542	$132,387	$90,594	$85,317	$154,196	$184,517	$111,587	$255,981	$189,864	$107,743	$69,636
Taxes			$270,196	$0	$0	$0	$0	$1,602	$1,887	$2,282	$917	$2,927	$11,773	$12,529	$11,791	$27,983	$35,183	$18,578	$54,481	$43,247	$25,573	$19,443

	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Net Income after Taxes	$1,279,603	$0	-$84,942	-$38,443	-$50,557	$56,700	$66,588	$80,306	$32,898	$96,615	$120,615	$78,065	$73,526	$126,213	$149,333	$93,009	$201,500	$146,617	$82,171	$50,193
Cash Flow																				
Operating Income after Depreciation	$1,549,799	$0	-$84,942	-$38,443	-$50,557	$58,302	$68,475	$82,588	$33,815	$99,542	$132,387	$90,594	$85,317	$154,196	$184,517	$111,587	$255,981	$189,864	$107,743	$69,636
Add back Depreciation	$642,495	$0	$84,942	$38,442	$189,290	$46,072	$39,674	$39,111	$35,208	$27,942	$27,345	$22,859	$18,374	$19,113	$14,864	$11,079	$9,835	$8,027	$6,125	$3,390
Working Capital																				
Accounts Receivable (0 days)	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Accounts Payable (30 days)	$0	$0	$0	$0	$12,849	$1,568	$203	$582	$179	-$547	-$126	$1,023	$201	-$1,054	$372	-$1,141	-$4,112	-$5,854	-$3,484	-$165
Working Capital	$0				-$15,500															$15,500
Total Working Capital	$0	$0	$0	$0	-$2,651	$1,568	$203	$582	$179	-$547	-$126	$1,023	$201	-$1,054	$372	-$1,141	-$4,112	-$5,854	-$3,484	$15,335
Capital Expenditures																				
Initial Capital																				
Mine	$195,522	$0	$108,978	$62,397	$24,146	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$299,546	$28,126	$203,512	$67,909	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital																				
Mine	$80,872	$0	$0	$0	$1,846	$8,131	$13,570	$16,469	$3,342	$650	$7,351	$12,831	$15,473	$236	$662	$236	$75	$0	$0	$0
Process Plant	$66,555	$0	$0	$0	$3,404	$0	$1,295	$26,965	$0	$0	$22,249	$0	$0	$12,641	$0	$0	$0	$0	$0	$0
Total Capital Expenditures	$642,495	$28,126	$312,490	$130,306	$29,396	$8,131	$14,865	$43,434	$3,342	$650	$29,601	$12,831	$15,473	$12,877	$662	$236	$75	$0	$0	$0
Cash Flow before Taxes	$1,549,799	-$28,126	-$312,490	-$130,306	$106,685	$97,812	$93,487	$78,846	$65,860	$126,286	$130,006	$101,645	$88,419	$159,377	$199,090	$121,290	$261,629	$192,037	$110,385	$88,361
Cumulative Cash Flow before Taxes		-$28,126	-$340,616	-$470,922	-$364,237	-$266,425	-$172,938	-$94,091	-$28,231	$98,055	$228,061	$329,707	$418,125	$577,503	$776,593	$897,883	$1,159,512	$1,351,548	$1,461,933	$1,550,294
Taxes	$270,196	$0	$0	$0	$0	$1,602	$1,887	$2,282	$917	$2,927	$11,773	$12,529	$11,791	$27,983	$35,183	$18,578	$54,481	$43,247	$25,573	$19,443
Cash Flow after Taxes	$1,279,603	-$28,126	-$312,490	-$130,306	$106,685	$96,210	$91,600	$76,565	$64,943	$123,359	$118,233	$89,116	$76,628	$131,394	$163,907	$102,712	$207,148	$148,790	$84,812	$68,918
Cumulative Cash Flow after Taxes		-$28,126	-$340,616	-$470,922	-$364,237	-$268,027	-$176,427	-$99,862	-$34,919	$88,440	$206,674	$295,790	$372,418	$503,812	$667,719	$770,431	$977,579	$1,126,369	$1,211,181	$1,280,099
Financial Indicators before Taxes																				
NPV @ 0%	**$1,549,799**																			
NPV @ 5%	**$783,721**																			
IRR	**18.9%**																			
Payback	**5.2**			1.0	1.0	1.0	1.0	1.0	0.2	-	-	-	-	-	-	-	-	-	-	-
Financial Indicators after Taxes																				
NPV @ 0%	**$1,279,603**																			
NPV @ 5%	**$638,627**																			
IRR	**17.5%**																			
Payback	**5.3**			1.0	1.0	1.0	1.0	1.0	0.3	-	-	-	-	-	-	-	-	-	-	-

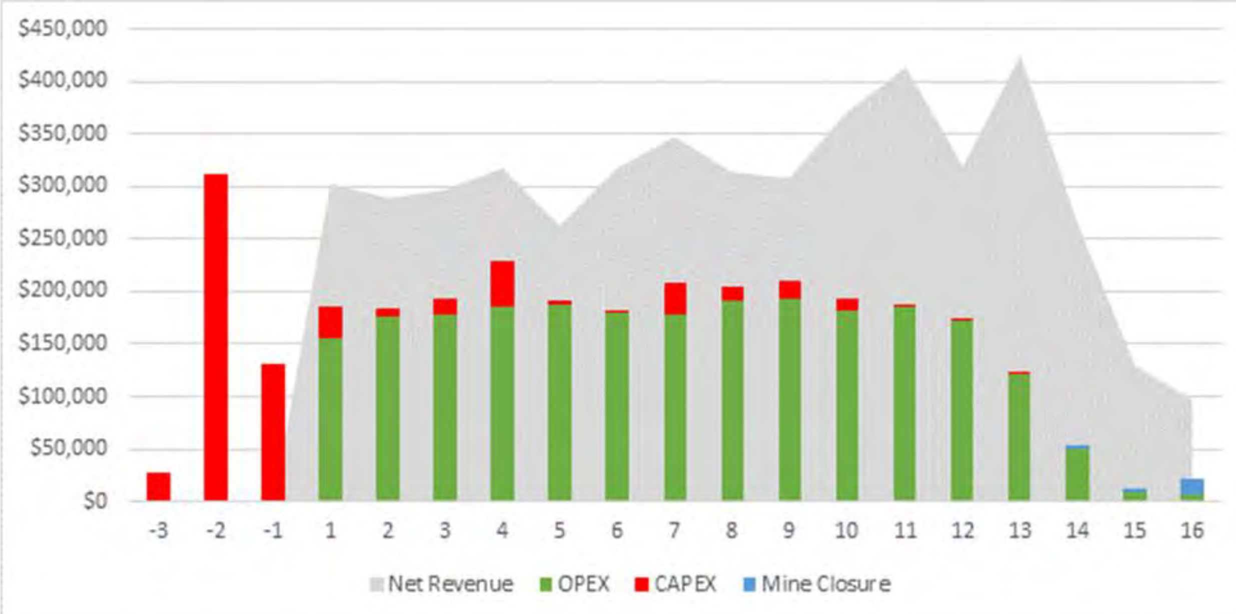


Figure 22-3: Phase 2 Cost and Revenue Summary

22.11.1 Sensitivity Analysis

The following tables illustrate the Base Case project economics and the sensitivity of the project to changes in the base case gold prices, operating costs, and capital costs. As is typical with precious metal projects, the Castle Mountain Project is most sensitive to gold prices, followed by operating cost and then initial capital costs. Presented in Table 22-9 to Table 22-13 are the sensitivities. Figure 22-4 shows the sensitivity analysis.

Table 22-9: Gold Price Sensitivity – After Taxes

Variance	Gold Price ($)	NPV @ 5% ($M)	IRR (%)	Payback (y)
-20%	1,200	143.3	7.9	10.3
-10%	1,350	393.9	12.9	6.9
0%	**1,500**	**638.6**	**17.5**	**5.3**
10%	1,650	874.6	21.7	3.9
20%	1,800	1,104.2	25.4	3.1
33%	2,000	1,408.8	30.1	2.5

Table 22-10: Operating Cost Sensitivity – After Taxes

Variance	NPV @ 5% ($M)	IRR (%)	Payback (y)
-20%	878.4	22.0	3.8
-10%	761.3	19.9	4.4
0%	**638.6**	**17.5**	**5.3**
10%	510.1	15.0	6.1
20%	378.4	12.4	7.4

Table 22-11: Initial Capital Sensitivity – After Taxes

Variance	NPV @ 5% ($M)	IRR (%)	Payback (y)
-20%	735.9	21.9	4.0
-10%	687.3	19.5	4.8
0%	**638.6**	**17.5**	**5.3**
10%	590.0	15.8	5.7
20%	541.3	14.3	6.1

Table 22-12: Cyanide Consumption Sensitivity – After Taxes

Variance	NPV @ 5% ($M)	IRR (%)	Payback (y)
-20%	654.8	17.8	5.2
-10%	646.7	17.7	5.2
0%	**638.6**	**17.5**	**5.3**
10%	630.5	17.3	5.3
20%	622.4	17.2	5.4

Table 22-13: Heap Leach Recovery Sensitivity – After Taxes

Variance	Recovery (%)	NPV @ 5% ($M)	IRR (%)	Payback (y)
-2.7%	72	556.4	16.1	5.6
-1.4%	73	597.5	16.8	5.5
0%	**74**	**638.6**	**17.5**	**5.3**
1.4%	75	679.8	18.2	5.1
2.7%	76	720.9	18.9	4.9

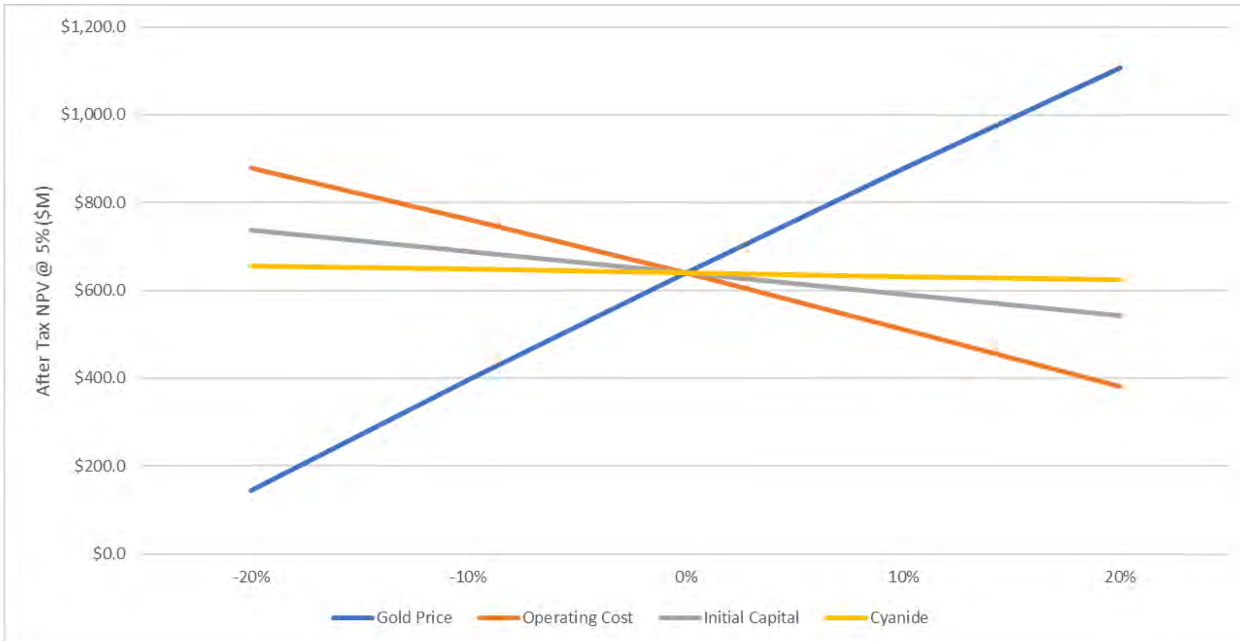


Figure 22-4: Sensitivity Analysis NPV @ 5%

23 ADJACENT PROPERTIES

There are no adjacent properties to the Project.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT IMPLEMENTATION

Project execution of the Castle Mountain Phase 2 expansion is expected to be completed in several stages including basic engineering or front-end engineering design (FEED) which are expected to progress in parallel with project permitting exercises. Further stages will include detailed engineering, project construction, commissioning, and project start-up and ramp-up.

There is expected to be some overlap between each stage; however, the near-term focus will be on permitting the project and optimization. The overall project timeline depends largely on attaining the discretionary operating permits from the co-lead agencies, the BLM and the County. The project permitting process and modifications to existing permits required for the Phase 2 project are discussed in Section 20.2. This permitting phase will be a main factor in the project timeline from the completion of the feasibility study to project completion and start of Operations. Phase 2 construction will not begin until the key operating permits are attained.

Considering the above, the Phase 2 expansion is anticipated to require an overall timeline of approximately 4 years from the effective date of this report to achieve full expanded operations. This timeline coincides with the completion of current operations' mining activity relating to JSLA backfill material and a requirement to move into fresh ore regions to sustain production.

24.1.1 Optimization and FEED

Throughout the course of completing the feasibility study, several areas have been identified that may provide environmental and economic benefits to the project or mitigate risk. A few key potential improvements will be studied to inform final project basis decisions prior to release of full-scale engineering activities. These studies will consist of specific evaluations of areas within the proposed process plant, alternative plant layouts and arrangements, reduction in overall project footprint, potential use of modularized equipment and structures etc. Additional testwork is planned to help inform these decisions.

Optimization work is expected to extend into a FEED program which will advance the feasibility study design to basic engineering. This will include development of long lead equipment specifications as well as receipt of detailed quotations, further engineering detailing, updates to cost estimates and implementation of project controls. During this stage, further development of important project infrastructure elements such as the utility transmission line and water sourcing will be undertaken. This stage is expected to be impacted to some extent by the permit application process which may evolve into additional engineering modifications. As the project timeline is refined, a detailed project execution plan (PEP) and project schedule will be developed. It is expected that during this stage the project organization structure will be developed and different project execution strategies will be reviewed. Further reviews by third parties will be incorporated.

The goal of this stage is to have a clear scope definition to start the detailed engineering stage and result in improved engineering timeline and deliverable quality. This stage is anticipated to continue for approximately 12 months and precedes full release of project engineering.

24.1.2 Basic and Detailed Engineering

The full project engineering scope of work begins with development of foundational deliverables such as project detailed design criteria, detailed project flow sheets, detailed piping and instrumentation diagrams, equipment lists, long lead equipment procurement, PEP, detailed process description and control philosophy.

Once the project has been fully defined during this basic phase, updated project cost estimates will be developed to act as the final project control point as the project moves into stages requiring more substantial financial investment.

Detailed engineering will then commence for all disciplines with priority on early works packages which may be required to support the construction on site. Basic and Detailed Engineering for this project, including the heap leach and filtered tailings facility, are anticipated to take approximately 15 months.

24.1.3 Contract Development

As detailed engineering reaches a point of substantial advancement and project work packages become available, a detailed project contracting plan is developed, work packages assembled and requests for services from Contractors are begun. Selected project contractor(s) will be engaged to complete a constructability review and establish firm expectations and execution methods prior to mobilization into the construction phase. This stage is expected to be carried primarily over the course of 3-6 months.

24.1.4 Project Construction

Project construction is anticipated to continue for approximately 24 months based on 50-hour work weeks. This includes construction of new process facilities, heap leach Phase 2A, the initial starter cell for the filtered tailings facility, the truck shop and wash, process maintenance and warehousing building and all supporting infrastructure to operate the project. This construction timeframe also includes a duration for commissioning and ramp up of the plant over the last 6 months of construction with handover to the Owner’s team.

Within the two-year construction period, care will be taken to schedule construction activities during optimized seasons and conditions to minimize things like construction water usage and to coincide best with on site operations.

Figure 24-1 shows a summary of the anticipated overall project schedule.

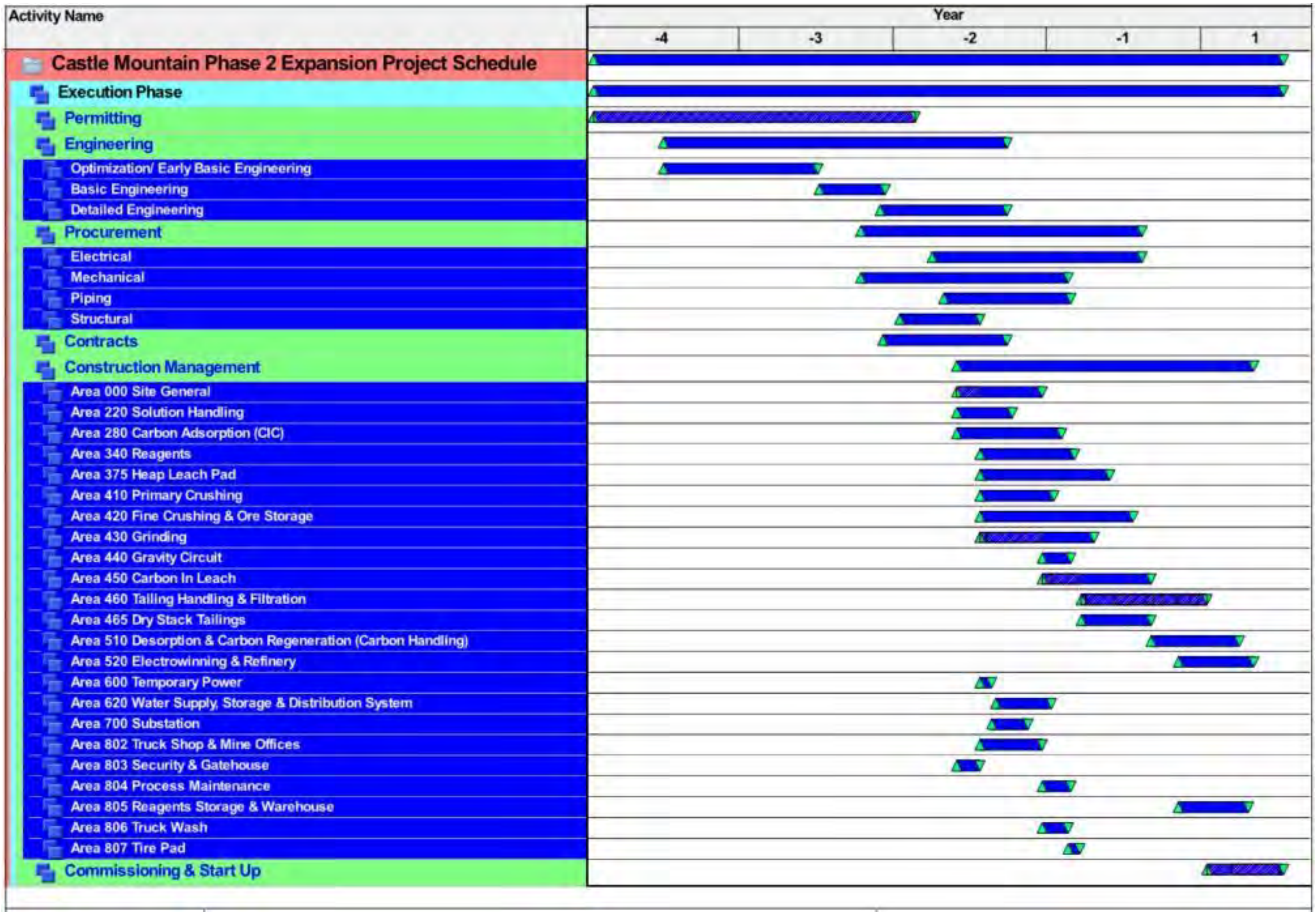


Figure 24-1: Anticipated Overall Project Schedule

25 INTERPRETATION AND CONCLUSIONS

25.1 CONCLUSIONS

This Technical Report provides a summary of study results and conclusions for each major element of investigation conducted. These include, but are not limited to, resource exploration, project metallurgy and interpretation, resource estimation, mine planning and design, process plant and infrastructure design and requirements, environmental status and mitigation, capital and operating costs, and economic analysis.

The feasibility study results provide a clear conclusion that the Castle Mountain Phase 2 Project is an economically viable project based on an evaluation and interpretation of the available data and work carried out in undertaking the study. The QPs have drawn the following principal conclusions:

25.1.1 General

The level of investigation for all elements of this study, as confirmed by all Technical Report QPs, is consistent and typical of a feasibility level study.

As of the effective date of this Technical Report, Equinox holds a 100% interest in the Castle Mountain Project.

25.1.2 Geology and Mineral Resource Estimate

The deposits within the Castle Mountain Project are part of a low sulfidation epithermal system characterized by gold mineralization commonly occurring with silica alteration and iron oxide minerals. Mineralization is controlled by first order porosity and permeability of the lithological units that form the host volcanic complex and structural zones which can provide the conduit for hydrothermal fluids that carry mineralization.

The Project has now combined Measured and Indicated mineral resources exclusive of mineral reserves that are amenable to open pit mining that total 82 Mton at 0.018 oz/ton gold for 1,470 koz contained gold. These mineral resources occur dominantly within the oxide portion of the ore body and include portions of transition and sulfide ore. The Measured mineral resources exclusive of mineral reserves that are amenable to open pit mining total 861,000 tons at 0.020 oz/ton gold for 17 koz contained gold. Indicated mineral resources exclusive of mineral reserves that are amenable to open pit mining that total 81 Mton at 0.018 oz/ton gold for 1,453 koz contained gold.

Contributions to the changes to the current mineral resources are predominantly due to the differences between criterion used for Mineral Resource classification which relies dominantly on drillhole spacing.

25.1.3 Mining and Mineral Reserve Estimate

The current LOM plan was developed based upon Proven and Probable Mineral Reserves of 284.3 Mton with a grade of 0.015 oz/ton at a cut-off grade of 0.005 oz/ton Au. The total waste mined will be 701.9 Mton. The strip ratio average is 2.47:1 waste:ore tons.

Mining for project expansion will be by Owner operated conventional diesel-powered truck and shovel operation. Current operations are focused on mining of previously backfilled material in the JSLA pit while expanded operations will focus on expansion of current pits and new pit development.

25.1.4 Mineral Processing and Metallurgical Testing

Significant metallurgical testwork from 2015 to 2020 has been completed on both heap leach and mill material to establish the design criteria for the Phase 2 expansion. The testwork program completed provided the necessary data to define a process flowsheet and engineering parameters. This allowed for design and cost estimation of a conventional process plant for the feasibility study as well as defining the metal recoveries and operating consumables. Recovery methods proposed for the expansion consist of proven and well-known technologies in the industry.

After evaluating the column leach tests by feed size, ore zone, and lithology, the arithmetic average gold recovery was 81%, 80% and 80%, respectively. A weighted gold recovery based on ounces per lithology was calculated as 82%. The average lab recovery for Castle Mountain low grade ore was 80%.

To estimate the Castle Mountain gold recovery for the production heap leach from the lab data, operating and environmental conditions were considered. This includes ROM particle size distribution, permeability, effective leaching of the side slopes, etc. The ROM material for the Castle Mountain Project is predicted to have an F_{80} of 152 mm to 203 mm. When considering the ore size and other data, a lab to field deduction of 6% was applied to the average lab recovery of 80% for an expected LOM heap leach gold recovery of 74% after solution application is stopped. To account for the typical time impact in recovering gold from a large leach facility at closure, the expected gold recovery during LOM operations is considered to be 67% with a final recovery of 74% attained only after extracting residual gold values and reducing cyanide levels in the heap. This is expected approximately three years after mining has ceased.

For this Feasibility Study, gravity followed by gravity tail leach in a CIL circuit was selected for the process based on economics. An overall gold recovery of 94% is expected from mill grade ores processed through the mill after 24-hour hybrid leach/CIL retention time. The plant has been sized conservatively with 30 hours retention time in the leach/CIL tanks.

25.1.5 Project Infrastructure

The existing infrastructure including current operations has been integrated into the proposed designs to support the expansion operations to the greatest extent possible. Further Phase 2 infrastructure developments include a security gatehouse, expanded site fencing, on site access roads, truck fleet service facilities, and dedicated process storage and warehousing facilities.

A site wide water balance was developed considering multiple scenarios based on historic and current climatic conditions at site and in the surrounding areas. Resulting make-up water demands were evaluated and used to inform water supply requirements. Recently completed groundwater studies on and off site indicate a high likelihood that sufficient capacity for fresh water can be attained through a combination of on site and off-site sources.

Mill tailings were tested and analyzed, and it was concluded they are amenable to filtered stacking. To limit new land use on the property, it was determined that adequate capacity is available on the historic Viceroy heap leach pad and adjacent to the expanded heap leach facility to serve the LOM requirement. This results in optimized footprint while minimizing new disturbance on the property.

Several options for power supply were studied in detail with each being determined to provide viable alternatives for power supply to the expanded operations. The selected basis for the feasibility study is connection to the grid via a new powerline serviced by NV Energy and SCE. Other alternatives including supplementing with renewable power, specifically solar, have been investigated and show potential opportunity.

25.1.6 Environmental Studies and Permitting

The Castle Mountain Mine is located on both public and private land and historically has been environmentally permitted by co-lead agencies, the County of San Bernardino at the state level, and the United States Bureau of Land Management (BLM) at the federal level. The current operation was issued a revised Mining CUP by the County in August 2019 while the BLM issued a Decision Record and FONSI in February 2020 approving the revised Mine and Reclamation Plan. These key permits along with others provide the authorization for current mine operations at the project site.

Significant resource monitoring and environmental analyses have been conducted and continue to be completed on site to assure compliance and environmental stewardship of the project site.

Mine expansion as considered in this feasibility study is expected to require new or updated environmental review (likely in the format of an EIS/EIR) as well as several new state and federal permits and amendments. Future amendments to the mine and reclamation plan to account for mine expansion are also expected and will include facility decommissioning, land recontouring, and revegetation.

25.1.7 Cost Estimates

Detailed capital and operating cost estimates have been developed including consideration for all direct and indirect costs associated with execution of the expansion project and required supporting infrastructure as well as sustaining costs, and reclamation and closure costs.

The cost estimate is based on preliminary engineering including 250 feasibility level design drawings covering all engineering disciplines, design criteria and detailed material take-offs. Mechanical and electrical equipment pricing was obtained for all major equipment (>$50,000) including 89% (or 140 items) through budgetary quotes. Bulk material pricing was estimated from recent projects budget quotes, local contractor budgetary review and actual pricing from Phase 1 construction. Mining equipment costs are based on quotes from major suppliers. Labor rates have been estimated using a weighted average of prevailing non-union shop wages from a published source (Davis-Bacon; 50%) and actual labor rates on site for Phase 1 construction (50%).

Project operating cost is based on a detailed build up of staffing requirements, reagent and fuel consumptions, mining activities, maintenance, and power demand. All major consumables have been specifically quoted with delivery to site.

The costs reflect a joint effort conducted by M3, NMS and specialist sub-consultants to adequately define project cost to a -10% to +15% accuracy level.

Analysis of the resulting economic parameters shows the expansion project to be economically viable.

25.2 RISKS

As with most projects at the feasibility stage, there are risks that could affect the economic potential or the continued viability of the project. There are several risks associated with external factors such as permitting, political, social, and metal prices which are generally considered inherent general risks to all mining projects. Throughout the development of the feasibility study, specific risks have been identified and mitigated through testing, analysis, and engineering practice. However, some risks remain and to a large degree there will always be risk associated with a mining project. Not all of these are listed here but a few key ones are described below.

In going forward it is expected that the project will develop a formal detailed Project Risk Register as well as conduct several risk assessments at various stages including Hazard and Operability Analysis (HAZOP). Risk mitigation will be a continual aspect that gets further refined through the next stages of project development.

25.2.1 Permitting

As described in detail in Section 20, there are significant regulatory hurdles and several different agencies involved. Time frames include time to prepare and submit environmental review documents, additional time for environmental studies, as well as multiple public review and comment periods. Key permits must be obtained in order that other non-discretionary permit processes can begin. There is a risk that the agencies will not meet reasonable time frames for permitting approvals. Equinox has developed good relationships with all permitting agencies through the recent permitting of the Phase 1 operations and continued dialogue through the next phase of work will mitigate permitting concerns.

25.2.2 Gold Recovery

The LOM gold recovery for heap leach grade ore is anticipated to be 67% and increasing to 74% after rinsing. It is based on column tests which have been completed using material finer than the predicted ROM with a top size of 432 mm (F_{80} of 152-203 mm). There are several factors within a heap that could affect performance. Gold recovery could be less than anticipated despite extensive column and bottle roll testwork that has been conducted on various crush sizes and on ROM; therefore, additional ROM testing on in-situ ore paired with 9.5 mm, 25 mm and 50 mm column tests is recommended.

25.2.3 Water Supply

The source of project make-up water is based on recent groundwater potential studies and small-scale pump testing which have indicated the high probability to service the project's demand needs from a combination of current and new on site as well as off-site wells (Lanfair Basin). Should further planned pump testing and groundwater modelling result in contrary results, alternative sources for water and/or adjustments to water demand will need to be investigated which could impact project economics.

25.2.4 Filtered Tailings

Mill tailings will be filtered and additionally allowed to dry before being systematically layered in place in a dry stack facility which is lined and will be abutted to the coarse ROM ore heap leach pad. The material has been geotechnically studied to ensure it will be structurally stable when placed. Filtered tailings is considered best available technology in terms of tailings placement. By abutting mill tailings with heap leach, residual cyanide solution is collected and recycled to the process.

Mill tailings have been shown through testwork to be readily amendable to detoxification and meeting recent USA EPA recommendations for the state of California for effluent cyanide concentration. Should lower levels of resulting cyanide be determined to be required to meet acceptable final residual levels within the lined facility, detox reagent consumptions and operating costs could be affected.

25.2.5 Geotechnical

During Viceroy operation, there were three significant wall failures. Geotechnical evaluation discovered that each failure was associated with weak clay altered materials and geological evaluation shows that the weak clay altered materials are within close vicinity of major mapped fault zones. Slope recommendations implemented in the mine plan include reduced slope angles within specified widths to mapped fault zones. Additional wall failures may occur again, but the risk is significantly mitigated based on lessons learned from previous operations.

25.3 OPPORTUNITIES

Throughout the feasibility study, opportunities for improvement of project economics were identified and investigated. Many of these opportunities have been incorporated into the study while others will be more fully examined in the next stage of project development. A few key ones are summarized here.

25.3.1 Supplemental Solar Power

The use of renewable energy has been investigated and the Castle Mountain site offers high potential for reducing reliance on conventional power generation. A solar plant has been examined in detail and provides an excellent opportunity to the Project to reduce GHG emission and lower reliance on electrical power provided by utilities.

The Project is situated in an area with substantial recent development of solar power generating plants and is well placed to utilize this as a renewable energy source on site to offset utility power supply costs. An analysis of solar power and its project economics has been prepared. This analysis focused on an 11.5 MW_{DC} photovoltaic solar power production plant, typically providing 10 MW_{AC} of power to the site and located on 42 acres at the northwestern corner of the current mine property. Figure 25-1 below shows the proposed location.

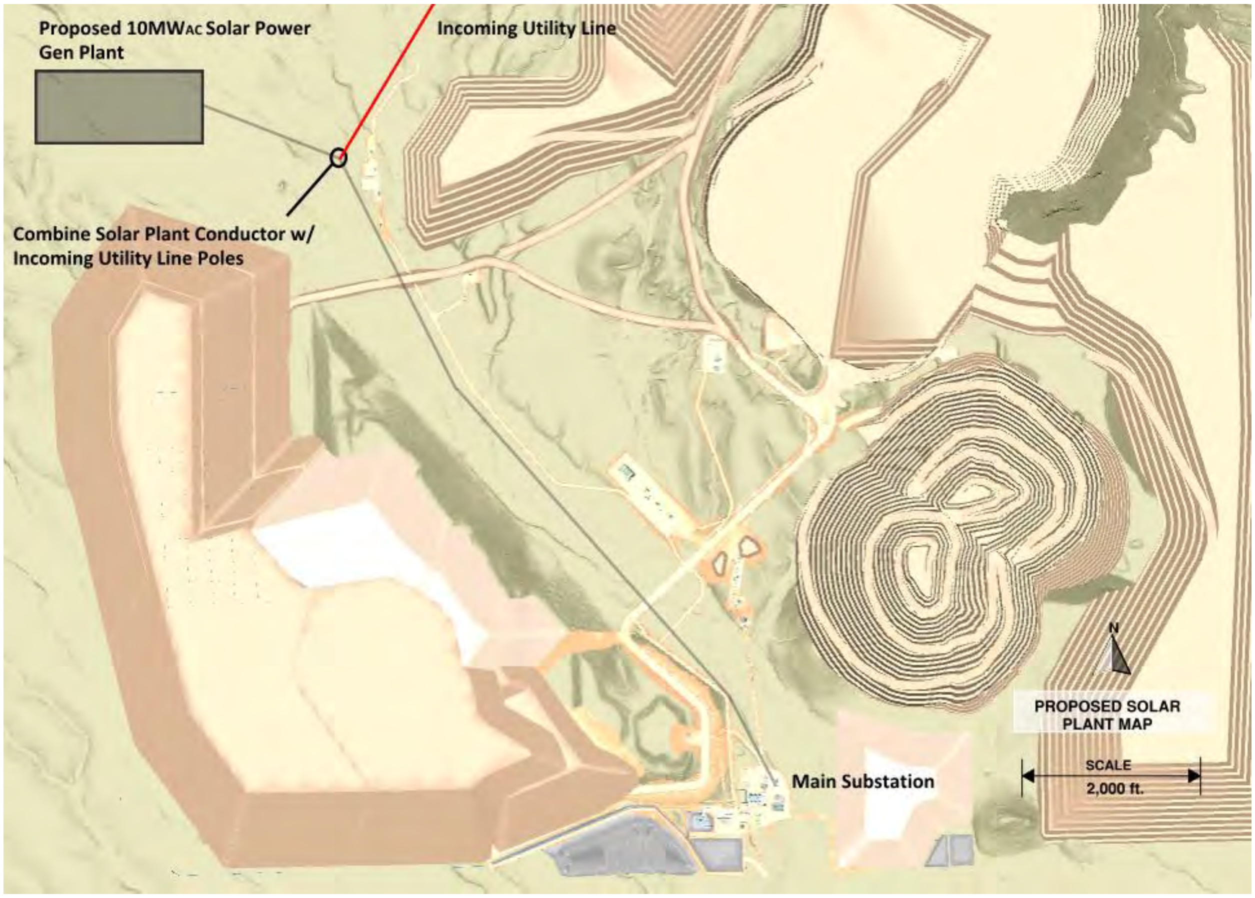


Figure 25-1: Proposed Solar Plant Map

The proposed solar power generating plant has shown to lower the energy cost for Castle Mountain if Castle Mountain subscribes to Net Metering. Net Metering is a program run by the California Public Utilities Commission providing customer-generators with credit or compensation for electricity generated by their renewable facilities to balance the costs of the renewable electrical generation facility. Net Metering is a direct incentive for solar power generation, and it has been indicated that Castle Mountain would likely qualify.

The proposed plant would generate approximately 40% of the typical expanded project demand load and reduce all-in average power costs from $0.103/kWh to $0.073 kWh. Initial capital of the 10 MW_{AC} is estimated at $12.8 million (including 15% contingency) with an operating cost of $140,000 annually resulting in a net power savings of $1.88 million annually, a simple payback of 6.8 years and an IRR of 12% when considering Net Metering.

Potential opportunity also exists for current operations on site in the potential addition of a pilot solar facility to offset or eliminate diesel power generation on site. Further studies to optimize are likely to be progressed throughout 2021.

25.3.2 Exploration

As resources are depleted throughout the current project life, exploration targets exist in the form of anomalous mineralized trends on East Ridge, East Flats, Egg Hill, Northwest Rim and Benson.

25.3.3 Plant and Infrastructure Design

There are several project and plant design aspects that offer opportunities to improve the project. These include reducing water usage and reagent consumption while there are others that may provide economic benefits through reduced equipment or footprint. This includes modularization of some equipment and selecting different types of similar equipment such as vertical carbon columns or alternative grinding mill or filter designs.

Grinding size sensitivity tests have indicated that coarser particle sizes show gold recovery by cyanide to be similar to the tests completed at 150 microns. Grinding to a coarser size would require both less power and smaller equipment.

Carbon loadings for the carbon columns are estimated to be 100 oz (Au + Ag) per tonne of carbon. Should current operations and further evaluation validate improved loadings, the daily amount of carbon movement may be able to be reduced.

Tailings have been shown to readily detoxify to low cyanide levels. There may be an opportunity to store tailings in an unlined facility which would allow for backfilling tailings into pits and co-mingling tailings with waste rock.

26 RECOMMENDATIONS

Based on the results from the study conducted, the following recommendations are made:

26.1 GENERAL

- The Castle Mountain Project Phase 2 expansion is a feasible project with good economics and should be progressed to the next stage which includes permitting along with optimization and front-end engineering ahead of detailed engineering.

26.2 ASSAY MANAGEMENT AND GEOLOGIC MODELING

- Implement a blind QA/QC program for ongoing monitoring of grade control and production sampling with control samples and duplicates analyzed at the mine laboratory.

- Implement in-pit bench mapping to inform short-range models in concert with grade control sampling.

26.3 MINERAL RESOURCE

- On-going grade control and reconciliation processes are recommended to measure the performance of the long-term model against production, to identify areas for improvement of the production cycle and to possibly refine the long-term model based on these results.

- Additional characterization of alteration and mineralization to better inform and constrain existing alteration and oxidation models is recommended.

26.4 MINING

- Continued optimization of phase development, pit waste backfill sequencing and blast patterns with respect to fragmentation for ROM leaching is recommended.

26.5 HEAP LEACH

- Observation of current operations to confirm design criteria such as saturation moisture percentage within the pad, performance of overliner gravel, evaporation losses etc.

26.6 METALLURGICAL AND PROCESS PLANT

- Conduct additional ROM testing on in-situ ore paired with 9.5 mm, 25 mm and 50 mm column tests to validate optimal size for heap leach.

- Conduct further mill grind size vs recovery with whole ore leach and gravity with gravity tails leach to validate optimal grind size.

- Further investigate potential clay impact on rheology and filtration characteristics. Specifically add East Ridge samples to confirm thickening and filtration criteria.

- Further investigate comparative leach/CIL tests to confirm effect of cyanide concentration on gold dissolution rates, final extraction, and reagent consumption.

- Head analyses on heap leach and mill composites from 2020 testwork indicated high levels of copper associated with andesite lithology. High solubility of the copper can impact the efficiency of carbon and the consumption of cyanide and detox reagents. The addition of copper and sulfur to the project mine plan is recommended.

- Undertake gravity recoverable gold (GRG) tests and modelling to determine optimum percentage of material to send to the gravity circuit. In addition, carry out intensive leach tests on gravity concentrate to confirm recovery and reagent consumption.

- Conduct further leach/CIL tests to validate reagent consumption.

- Gravity and tails leach testwork has not been done on the East Ridge ore zone which accounts for 17% of the LOM gold ounces to be processed. This testwork is recommended to be completed.

- Detoxification tests using oxygen are recommended to validate reagent consumption.

- Agitator testing to observe mixing performance and to determine design parameters with the variability expected in the solids density from different ore types for both CIL and Detox.

26.7 PROJECT INFRASTRUCTURE

- Update the Lanfair Valley hydrogeological model based upon recent well testing results and surface geophysical information.

- Install larger diameter wells in the Lanfair Valley and Ivanpah Valley to validate sustainable availability of make-up water. Each of the wells could eventually serve as production wells.

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

Gabriel A. Secrest

I, Gabriel A. Secrest, P.E., do hereby certify that:

1. I am an Assistant Vice President of:

   M3 Engineering and Technology Corporation
   2051 W Sunset Rd.
   Tucson, AZ 85704

2. I graduated with a Bachelor of Science degree in Mechanical Engineering from the University of Arizona in 2006.
3. I am a Professional Engineer in good standing in Arizona (Cert. # 52776), Nevada (Cert. # 21989) and South Carolina (Cert. # 30039).
4. I have worked as an engineer and project manager for industrial and mining projects a total of 15 years. My experience includes detailed engineering projects, project management and execution, engineering management, construction planning and management and technical analysis and study for industrial and mineral processing projects. I have worked on scoping, pre-feasibility and feasibility level studies for mining projects within the United States and abroad.
5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.
6. I am the principal author for the preparation of the technical report titled “Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA” (the “Technical Report”), dated effective February 26, 2021, prepared for Equinox Gold Corp. (“Equinox Gold”); and am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.14, 1.15, 1.16, 1.17, 1.18.1, 1.18.5, 1.18.6, 1.18.7, 1.19, 2, 3, 4, 5, 13.8.5, 17.3.1, 18 (except for 18.5.3), 19, 20, 21 (except for 21.1.7, 21.1.9 and 21.2.2), 22, 23, 24, 25.1.1, 25.1.5, 25.1.6, 25.1.7, 25.2, 25.3.1, 25.3.3, 26.1, 26.5, 26.7, and 27.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Equinox Gold and all their subsidiaries as defined in Section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15th day of March, 2021.

Signature of Qualified Person

Gabriel A. Secrest
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled: "Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA" (the "Technical Report"), dated effective February 26, 2021, prepared for Equinox Gold Corp.

I, Laurie Tahija, MMSA-QP, Principal Consultant (Processing), do hereby certify that:

1. I am currently employed as Senior Vice President by M3 Engineering & Technology Corporation, 2051 W. Sunset Road, Ste. 101, Tucson, Arizona 85704, USA.
2. I am a graduate of Montana College of Mineral Science and Technology, in Butte, Montana and received a Bachelor of Science degree in Mineral Processing Engineering in 1981.
3. I am recognized as a Qualified Professional (QP) member (#01399QP) with special expertise in Metallurgy/Processing by the Mining and Metallurgical Society of America (MMSA).
4. I visited the Castle Mountain Gold property on November 19, 2020, specifically to review the Phase I processing plant which was not the subject of this Technical Report.
5. I have practiced mineral processing for 39 years. I have over twenty (20) years of plant operations and project management experience at a variety of mines including both precious metals and base metals. I have worked both in the United States (Nevada, Idaho, California) and overseas (Papua New Guinea, China, Chile, Mexico) at existing operations and at new operations during construction and startup. My operating experience in precious metals processing includes heap leaching, agitation leaching, gravity, flotation, Merrill-Crowe, and ADR (CIC & CIL). My operating experience in base metal processing includes copper heap leaching with SX/EW and zinc recovery using ion exchange, SX/EW, and casting. I have been responsible for process design for new plants and the retrofitting of existing operations. I have been involved in projects from construction to startup and continuing into operation. I have worked on scoping, pre-feasibility and feasibility studies for mining projects in the United States and Latin America, as well as worked on the design and construction phases of some of these projects.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I am independent of the issuer as defined by Section 1.5 of NI 43-101.
8. I accept professional responsibility for Sections 1.9, 1.13, 1.18.4, 13, 17 (except for 17.3.1), 25.1.4, and 26.6 of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.

Dated this 15th day of March, 2021.

(Signed) (Sealed)
Signature of Qualified Person

Laurie M. Tahija
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Eleanor Black

I, Eleanor Black, P.Geo, do hereby certify that:

1. I am Partner, Senior Geologist of:

   Equity Exploration Consultants Ltd.
   1238-200 Granville Street
   Vancouver, BC, Canada, V6C 1S4

2. I graduated with a Bachelor of Science in Geology from the University of British Columbia in 2004.
3. I am a Professional Geologist, license #42632, in good standing with the Engineers and Geoscientists of British Columbia.
4. I have worked as geologist for a total of 16 years. I have sufficient relevant experience having been directly involved in managing exploration programs focused on identifying and delineating orogenic gold, porphyry, VMS, and other deposit types in British Columbia, Nunavut, Ontario and Yukon, Canada as well as Finland, Brazil, and the United States. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
5. I am a contributing author for the preparation of the technical report titled "Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA" (the "Technical Report"), dated effective February 26, 2021, prepared for Equinox Gold Corp. ("Equinox Gold"); and am responsible for Sections 1.5, 1.6, 1.7, 1.8, 6, 7, 8, 9, 10, 11, 12 (except for 12.4), 25.3.2, and 26.2. I visited the project site from August 13 -15, 2019.
6. I have not had prior involvement with the property that is the subject of the Technical Report.
7. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15th day of March, 2021.

(Signed) (Sealed)
Signature of Qualified Person

Eleanor Black
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Trevor Rabb

I, Trevor Rabb, P.Geo, do hereby certify that:

1. I am Partner, Resource Geologist of:

   Equity Exploration Consultants Ltd.
   1238-200 Granville Street
   Vancouver, BC,
   Canada, V6C 1S4

2. I graduated with a Bachelor of Science in Geology from Simon Fraser University in 2009.

3. I am a Professional Geologist, license #39599, in good standing with the Engineers and Geoscientists of British Columbia.

4. I have worked as geologist for a total of 11 years. I have sufficient relevant experience having been directly involved in managing exploration, practicing geostatistics and resource modelling and estimating resources. I have practiced as a geologist since 2009 and have worked managing exploration programs focused on identifying and delineating copper porphyry, VMS, orogenic gold, nickel and other deposits in British Columbia, Yukon, Ontario, Australia, and Brazil. I have specialized in geochemistry, geostatistics and resource modelling for six years on various underground and open pit base metal and gold deposits in Canada, the United States, Central and South America. I have practiced mineral resource estimation for four year on various underground and open pit base metal and gold deposits in Canada, the United States, Central and South America. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

5. I am a contributing author for the preparation of the technical report titled "Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA" (the "Technical Report"), dated effective February 26, 2021, prepared for Equinox Gold Corp. ("Equinox Gold"); and am responsible for Section 1.10, 1.18.2, 12.4, 14, 25.1.2, and 26.3. I visited the project site from August 13 -15, 2019.

6. I have not had prior involvement with the property that is the subject of the Technical Report.

7. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

8. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15th day of March, 2021.

(Signed) (Sealed)
Signature of Qualified Person

Trevor Rabb
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

John Nilsson, MSc., P.Eng.

I, John Nilsson, MSc., P.Eng., do hereby certify that:

1. I am a Professional Engineer, President of:

   Nilsson Mine Services Ltd.
   20263 Mountain Place
   Pitt Meadows, B.C.
   Canada

2. I graduated from Queen's University with a Bachelor of Science degree Geology in 1977 and subsequently a Master of Science degree through the Department of Mine Engineering in 1990.
3. I am a member in good standing of the Engineers & Geoscientists British Columbia (License #20697).
4. I have worked as a geologist and then a mining engineer for a total of 43 years on mining related precious and base metal projects in North America, Central America, South America, Africa, Europe and Asia.
5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am a contributing author for the preparation of the technical report titled "Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA" (the "Technical Report"), dated effective February 26, 2021, prepared for Equinox Gold Corp., Castle Mountain Venture; and am responsible for Sections 1.11, 1.12, 1.18.3, 15, 16, 21.1.7, 21.1.9, 21.2.2, 25.1.3, and 26.4. I have not visited the project.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15th day of March, 2021.

(Signed) (Sealed)
Signature of Qualified Person

John W. Nilsson, MSc., P.Eng
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

R. Douglas Bartlett

I, R. Douglas Bartlett, PG, CPG, do hereby certify that:

1. I am Principal Hydrogeologist of:

   Geologic Associates, Inc.
   6155 E. Indian School Rd., Suite 200
   Scottsdale, AZ 85251

2. I graduated with a Master of Science degree in Geology from Colorado State University.

3. I am a Professional Geologist and Certified Hydrogeologist in good standing in California (CA PG 8809, CA CHG 965). I am also a Certified Professional Geologist with the American Institute of Professional Geologists (CPG No. 8433).

4. I have worked as a geologist/hydrogeologist for a total of 43 years. My experience includes assessing groundwater supplies for mining properties in the southwestern U.S.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Technical Report on the Castle Mountain Project Feasibility Study, San Bernardino County, California, USA" (the "Technical Report"), dated effective February 26, 2021, prepared for Equinox Gold Corp. ("Equinox Gold"); and am responsible for Section 18.5.3. I last visited the project site on April 28, 2020.

7. I have prior involvement with the property that is the subject of the Technical Report. I have overseen the design of a groundwater exploration program intended to define the groundwater resources in and around the mine property.

8. I have also evaluated groundwater resources in Ivanpah Valley, west of the mine and have assisted Equinox Gold in assessing the condition of existing production wells at the site.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 15th day of March, 2021.

(Signed) (Sealed)
Signature of Qualified Person

R. Douglas Bartlett
Print Name of Qualified Person