NI 43-101 Technical Report on the Preliminary Feasibility Study for the Castle Mountain Project

San Bernardino County

California, USA

Prepared for:

EQUINOXGOLD

Equinox Gold Corp.
Suite 730, 800 West Pender St.
Vancouver, BC V6C 2V6 Canada

Authors and Qualified Persons:

Timothy D. Scott, SME RM, Kappes Cassiday & Associates

Todd Wakefield, SME RM, Mine Technical Services Ltd.

Don Tschabrun, SME RM, Mine Technical Services Ltd.

Terre Lane, MMSA, SME RM, Global Resource Engineering

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1.0 SUMMARY

1.1 Introduction

Equinox Gold Corp. (Equinox Gold) commissioned Kappes, Cassiday & Associates (KCA) to conduct a preliminary feasibility study (Prefeasibility or PFS) for the Project, incorporating all recent drilling, updated geological understanding of the deposits, updated mineral resource estimate and reserve estimate, and recent metallurgical test work. The purpose of this Technical Report is to support the Prefeasibility.

1.2 Property Description and Location

The Castle Mountain Project (Project) is located in the historic Hart Mining District, at the southern end of the Castle Mountains, San Bernardino County, California, approximately 70 mi (112.6 km) south of Las Vegas, Nevada. The Project is located in the high desert area near the Mojave National Preserve and Castle Mountains National Monument. The Project includes 13,276 acres of patented and unpatented lode, placer and mill site claims. The site can be accessed by gravel road year-round.

1.3 Ownership

Equinox Gold (previously Trek Mining Inc.) acquired NewCastle Gold Ltd. (NewCastle) on December 22, 2017 and NewCastle became a wholly-owned subsidiary of Equinox Gold. NewCastle has 100% of the right, title and beneficial interest in and to Castle Mountain Venture (CMV) which owns the Project. A number of net smelter return (NSR) royalty agreements are in place on the Project.

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

1.4 Climate and Physiography

The climate is typical of the arid eastern Mojave Desert area. Most of the precipitation is the result of localized thunderstorms between July and September and infrequent cyclonic storms from December to March. Topographic relief in the Castle Mountains range averages approximately 1,000 ft. (304.8 m) above the adjacent Lanfair Valley floor elevation of approximately 4,100 foot (1,250 m) elevation. Seasonal temperatures can range from approximately 32°F (0°C) to 100°F (38°C).

1.5 Project History

Gold mining began in the Hart Mining District in 1907. Recent exploration was conducted in the area more or less continuously since the late 1960's.

Viceroy Gold Corporation (Viceroy)/MK Gold Corporation commenced gold production on the Project in 1991 and the JSLA deposits were considered 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.

NewCastle (then Castle Mountain Mining Company Limited) acquired the Project in 2012.

In December 2017, NewCastle was acquired by Trek Mining Inc., which was renamed Equinox Gold Corp.

1.6 Geological Setting and Mineralization

The Castle Mountains gold deposit is located in the Hart Mining at an elevation of ~4500 feet (1372 m) in the southern portion of the Castle Mountains Range. The Castle Mountains Range is in the eastern Mojave Desert within the southern Basin and Range Province. 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.

Metamorphic Proterozoic basement is exposed along the northeastern flank of the Castle Mountains and consists of a massive sequence of biotite schist, biotite gneiss and meta-granite. Only local 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 180 feet (55 m) thick locally referred to as PC Seds. Unconformably overlying the PC Seds is the regionally extensive Peach Springs Tuff unit. 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 consists primarily of rhyolitic domes, flows, and felsic tuff, and lesser andesitic, latitic, and basaltic lava emplaced during three intrusive-extrusive episodes between ~18.8 and ~13.5 Ma.

CMV rocks are the primary host of epithermal gold mineralization at the Castle Mountain Project.

1.7 Exploration and Drilling

Prior to 2 December 2015, a total of 1,850 drill holes totaling 1,256,552 feet (392,997 m) were completed on the Castle Mountain Project. A total of 1,762 drill holes totaling 1,185,982 feet

(361,487m) was legacy drilling and 88 drill holes totaling 70,570 feet (31,510m) were completed by NewCastle.

Since December 2015, NewCastle has completed an additional 235,000 feet of drilling in 194 drill holes on the Project in two drill campaigns using angled reverse circulation (RC) and diamond core drilling to improve the grade and the geological understanding of the deposits.

NewCastle Phase I drilling began in June 2016 and by October 2016 had completed 46 exploration and infill resource drill holes, and one hydrological test hole, for a total drilled footage of 65,423ft (19,941m). The program targeted the southern part of the mineralized area known as “Big Chief” and “South Domes” that were considered to have good potential for near-term mineral resource expansion, as well as possible strike extensions of the Lucky John high-grade mineralization encountered in 2014 and 2015.

NewCastle Phase II drilling began in late October 2016 and was essentially a continuation of the Phase I program. A total of 148 core and reverse circulation holes were drilled and these included: 136 resource expansion and infill drilling holes, four water well test holes, four PQ metallurgical test holes, and four PQ holes to test for clays with suitable properties for use as a clay liner. The total drill footage completed was 169,944ft (51,799m) including 160,341ft of resource and infill drilling; 5,620ft of water well test drilling; 3,383ft of PQ metallurgical drilling; and 600ft of clay test hole drilling.

Equinox Gold Phase III drilling included 31 holes aimed at infill drilling in the South Domes area and exploration drilling in other areas of the Project. The total drill footage completed was 30,047 ft (9,158 m) in 31 diamond core and RC holes.

In 2018, a 53-hole RC program totaling 9,680 ft (2,951 m) in the JSLA back-fill, down to the 4300’ elevation and a depth of approximately 182 ft (55 m) on average, has been completed.

Also in 2017 and 2018, a RAB (reverse air blast) drill hole program, designed to test the top 20 ft (6.1 m) of the JSLA back-fill material, was completed with a total of 809 holes at 50 ft (15 m) spacings. An additional 32 holes were completed over an infill grid on 20 ft (6.1 m) spacings, centered on RC hole RC18-1-2. The RAB program was then extended to include drilling portions of the north and south waste dumps, bringing the total RAB drilling program in JSLA backfill to 995 holes.

1.8 Sampling and Verification

Viceroy drill hole samples were analyzed for gold by conventional fire assay methods by Legend or Rocky Mountain Geochemical in Reno, Nevada. Routine duplicate analyses were performed on conventional rotary, reverse-circulation and core drill holes utilizing the same pulp as that used for the initial analyses. Assay precision from the pulp duplicates was variable with gold grade, but generally acceptable. Check assay samples submitted to other commercial labs and the Castle Mountain Mine lab did not indicate any problems with Legend's original assays.

NewCastle drill hole samples were assayed by ALS or Inspectorate in Reno, Nevada. Check assays were completed at American Assay Laboratories in Sparks, Nevada. Gold and silver were assayed by conventional fire assay methods followed by AA analysis. Gold assays returning greater than 10 g/tonne Au were re-assayed by fire assay and gravimetric finish and gold assays returning greater than 0.2 g/tonne Au were analyzed for gold cyanide solubility.

NewCastle employed a QA/QC program that included the analysis of certified reference material (CRMs), blanks, RC field duplicates, and check assays. CRMs, blanks and duplicates were inserted regularly in the sample stream, and a random selection of samples from mineralized intervals were submitted to an umpire laboratory for check assay at the completion of each drill campaign.

Mine Technical Services Ltd. (MTS) reviewed a compilation of the 2017 control sample results and found the assay accuracy and precision to be acceptable for purposes of resource estimation. No significant bias was observed in the CRM results for gold. Check assays showed no significant bias between the ALS and Inspectorate original assays and the AAL check assays. No significant carryover contamination was observed in the blank results.

In the opinion of the Qualified Person, the sample preparation, security, and analytical procedures are adequate for purposes of resource estimation. The assay accuracy and precision are considered acceptable for resource estimation.

1.8.1 JSLA Backfill - RAB Drill Program

For the JSLA Backfill RAB drilling campaigns, the RAB drill collected the sample direct from the top of the drill hole outside the drill string, and then directed the chips to a cyclone where the sample was recovered and bagged. 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.

ALS Laboratories performed assays on the RAB samples with Fire Assay with Atomic Absorption and Gravimetric finish, and Cyanide Digestion.

QA/QC procedures were implemented according to industry best practice and approved by the Qualified Person. Certified reference material was screened for results within 10% of the reported mean, and blank material was screened for results above 10X the detection limit of the analytical method.

In the opinion of the Qualified Person, the sample collection, preparation, security, and analysis of the RAB samples are adequate for the purposes of resource estimation. The assay accuracy and precision are adequate for resource estimation.

1.9 Metallurgical Testing

A significant amount of metallurgical test data has been generated for the Castle Mountain Project, including:

- Initial test work before startup of the mine in 1992;
- Continued test work during operations for process optimization during 1991-2001;
- Actual production statistics from pulp agglomeration plant operations 1991-2001 and post production data (rinsing, etc.);
- 2014-2015 program with crush size vs. recovery column tests, run-of-mine (ROM) column tests, bottle roll tests, grinding / cyanidation testing, gravity recoverable gold tests, comminution tests, and compacted permeability tests;
- 2017 /2018 program with ROM column tests, pulp agglomeration studies, cyanidation testing, gravity recoverable gold tests, crush size vs. recovery column tests, variability testing, CIL testing, compacted permeability testing, gravity sedimentation and filtration tests.

Much is of this work is dedicated to pulp agglomeration studies, both historically and within recent campaigns. As studies progressed during recent campaigns, however, the test work emphasis shifted to evaluating conventional milling with Carbon in Leach (CIL) for higher grade ore within the deposit, and evaluation of ROM heap leaching for lower grade ore. Conventional milling and ROM heap leaching allows high and low-grade ores to be treated independently, which offers more flexibility to processing and mine scheduling as compared to the pulp agglomeration process, which is dependent upon the blending ratios of higher grade mill slurry to lower grade crushed ore in the pulp agglomeration product (1:9 to 1:12 during historical operations).

The current resource defines higher ratios of mill slurry to crushed ore than was the case during historical operations, which presents challenges to scheduling of high and low-grade ore delivery from the mine and also presents additional risk to heap permeability as compared to the historical operation. Recent work focused on de-coupling this limitation to maximize the overall amount of ore for processing by considering a straight mill / CIL for high-grade ores and a conventional ROM heap leach for the low-grade ore.

Because the previous pulp agglomeration process design also utilizes milling of high-grade material, there are significant amounts of test work programs and results from prior pulp agglomeration test work campaigns that support the current mill/CIL design in the PFS.

1.10 Mineral Resource Estimate

The Mineral Resource estimate utilized an inverse distance weighting method bounded by multiple grade shells and updated geologically-interpreted domains. A resource classification was developed based on sample support within various distances. The Mineral Resource estimate presented in Table 1-1 (imperial units) and

Table 1-2 (metric units) shows a range of cutoff grades with the base case (in bold) listed at a gold cutoff grade of 0.005 opt (0.17 g/tonne) and contained within a Lerchs-Grossman (LG) shell based on a gold price of $1,400/oz.

Table 1-1 Mineral Resource Estimate (Imperial Units).

	Measured			Indicated		
Cutoff (Au opt)	**Mtons**	**Gold Grade (opt)**	**Gold Oz (million)**	**Mtons**	**Gold Grade (opt)**	**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**		
Cutoff (Au opt)	**Mtons**	**Gold Grade (opt)**	**Gold Oz (million)**	**Mtons**	**Gold Grade (opt)**	**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

1) The effective date of the Mineral Resource is March 29, 2018.
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 cutoff grade of 0.005 opt.
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 this Technical Report.
7) For further information on backfill see Sections 13.5.13 and 14.3.2

Table 1-2- Mineral Resource Estimate (Metric Units)

	Measured			Indicated		
Cutoff (Au gpt)	**Mtonnes**	**Gold Grade (gpt)**	**Gold Oz (million)**	**Mtonnes**	**Gold Grade (gpt)**	**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**		
Cutoff (Au gpt)	**Mtonnes**	**Gold Grade (gpt)**	**Gold Oz (million)**	**Mtonnes**	**Gold Grade (gpt)**	**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

1) The effective date of the Mineral Resource is March 29, 2018.
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 cutoff grade of 0.17 gpt.
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 this Technical Report.

1.11 Mineral Reserve Estimate

GRE has estimated the Mineral Reserves for the Castle Mountain project using a pit design on a $850 gold price pit shell (generated by a Vulcan pit optimizer), conventional open pit mining methods, and a gold price of $1,250 in the economic analysis. The Mineral Reserve for the Castle Mountain project is effective June 29, 2018.

The Mineral Reserve includes measured and indicated resources to produce proven and probable Mineral Reserves. The mine plan presented in Section 16.0 – Mining Methods – details the production of this reserve. Several cutoff and ROM/mill cutover grades were used at different points in the production schedule to meet production targets. Vulcan generated pit shells at lower gold prices were used to assist in phase design.

The Castle Mountain Mineral Reserves are shown in Table 1-3. The Mineral Reserves are shown using a cutoff of 0.004 oz per ton (0.14 g/tonne) for the JSLA backfill and 0.005 oz per ton (0.17 g/tonne) for fresh ore.

Table 1-3- Castle Mountain Mineral Reserves

	Proven			Probable			Proven & Probable		
Resource Area	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**
JSLA - Rock	62.5 (56.7)	0.015 (0.52)	0.95	1.9 (1.7)	0.027 (0.92)	0.05	64.5 (58.5)	0.016 (0.54)	1.01
JSLA - Pit Fill	0	0	0	18.0 (16.3)	0.010 (0.35)	0.18	18.0 (16.3)	0.010 (0.35)	0.18
Jumbo	9.8 (8.9)	0.022 (0.77)	0.22	2.9 (2.6)	0.011 (0.39)	0.03	12.7 (11.5)	0.020 (0.68)	0.25
Oro Belle	42.7 (38.7)	0.017 (0.57)	0.71	6.8 (6.2)	0.014 (0.48)	0.10	49.6 (45.0)	0.016 (0.56)	0.80
East Ridge	5.6 (5.1)	0.023 (0.80)	0.13	7.1 (6.4)	0.012 (0.42)	0.09	12.8 (11.6)	0.017 (0.59)	0.22
South Domes	29.9 (27.1)	0.018 (0.63)	0.55	30.5 (27.7)	0.018 (0.62)	0.56	60.4 (54.8)	0.018 (0.63)	1.10
Total	**150.6 (136.6)**	**0.017 (0.58)**	**2.56**	**67.2 (61.0)**	**0.015 (0.51)**	**1.00**	**217.8 (197.6)**	**0.016 (0.56)**	**3.56**

Note: The Mineral Reserve estimate with an effective date of June 29, 2018 is based on the Mineral Resource estimate with an effective date of March 29, 2018 that was prepared by Don Tschabrun, SME RM of Mine Technical Services Ltd. The Mineral Reserve was estimated by Global Resource Engineering, LLC with supervision by Terre Lane, MMSA, SME RM. Mineral Reserves are estimated within the final designed pit which is based on the $850/oz pit shell with a gold price of $1,250/oz. The minimum cutoff grade was 0.004 oz/ton (0.14 g/tonne) gold and 0.005 oz/ton (0.17 g/tonne) gold for Stages 1 and 2, respectively. Average life of mine costs are $1.26/ton ($1.39/tonne) mining, $1.56/ton ($1.72/tonne) processing ROM, $8.17/ton ($9.01/tonne) processing Mill/CIL, and $0.73/ton ($0.80/tonne) processed G&A. The average process recovery was 72.4% for ROM and 94% for Mill/CIL. Tons and gold ounces are both reported in millions. Small differences in total tonnage and grade may occur due to rounding. The Mineral Resource estimate is inclusive of Mineral Reserves.

1.12 Mining Methods

The Castle Mountain deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the resource model and reserve estimate described in the previous sections.

The mine plan is based on the extraction of the proven and probable ore in the mineral reserve. The mine plan was designed to deliver 16,425,000 tons (14,901,000 tonnes) of ore per year to the processing facility in two process types. ROM ore in the quantity of 15,476,000 tons (14,040,000 tonnes) per year and mill ore in the quantity of 949,000 tons (861,000 tonnes) per year starting in Year 4 of the mine plan. Prior to Year 4, the mine plan will deliver 5,110,000 tons (4,636,000 tonnes) of ore per year strictly from the JSLA backfill to the heap leach pad.

The mine plan includes:

- Ultimate pit design including benches, ramps, and haul roads;
- Pit phase designs;
- A mine production schedule;
- Waste storage design;
- Yearly mine plan drawings including the pit, exterior waste dumps, and in-pit waste backfill; and,
- Equipment and labor requirement calculations.

The Vulcan pit shell analysis shown in Section 15.0 provides a basis for creating the ultimate pit design. The $850/oz pit (0.68 revenue factor) was selected as the basis for designing the ultimate pit. The ultimate pit design was developed using Geovia GEMS mine design software.

The ultimate pit is comprised of five pit areas: JSLA, Jumbo, Oro Belle, East Ridge, and South Domes. The pit areas progress in the following order:

1. JSLA
2. Jumbo
3. Oro Belle
4. East Ridge
5. South Domes

This pit area order became the basis for the mine plan based on the higher NPV generated by mining JSLA first, and the practicality of mining adjacent/closer pits subsequently. East Ridge and South Domes were planned at the end of the mine life as they constitute a higher strip ratio, a higher risk based on drilling density, and lower incremental NPV.

Production begins in the JSLA backfill in Years 1-3. Ore and waste will be mined by contract mining. During these years, 19,049,000 tons (17,281,000 tonnes) of ore will be mined with contained gold totaling 195,895 ounces. All material mined in Years 1 and 2 consists of material previously-mined and backfilled by Viceroy. Pre-stripping in new areas starts in Year 3.

Production ramps up to full scale in Year 4 and continues in a steady state until ore is depleted near the end of the mine life. Production also switches to a fleet owned and operated by Equinox Gold in Year 4.

JSLA was sequenced early in the mine life to facilitate a backfill waste strategy instead of larger waste dumps. As pits are emptied, each becomes a new target for a backfill. By the end of the mine life, all pits except those of South Domes are backfilled with waste.

Waste rock is placed in dumps adjacent to the pits for the first 8 years of operation, and pit backfilling begins in Year 9. Waste is added to mined-out pits from the top. The upper surface of the backfill grows from the edge of the pit inward as more waste material is dumped into the pit.

1.13 Recovery Methods

Test work developed by KCA and Equinox Gold and carried out by McClelland Laboratories in Reno, NV has indicated that the Castle Mountain ores are amenable to cyanide leaching for the recovery of gold.

The processing plan has been divided into two stages:

- Stage 1 (Years 1-3) considers processing 14,000 tons per day (12,700 tonnes per day) of ROM backfill material from the JSLA pit, where it was stored from the previous operation. Excavated backfill material will be loaded into 100-ton (91-tonne) haul trucks and stacked in 50 ft (15 m) lifts. Quicklime (CaO) will be added to the material in the trucks for pH control before the ore is stacked and leached in two stages using a dilute sodium cyanide solution. Pregnant solution discharging from the heap will flow by gravity to a pregnant solution tank from which it will be pumped to a Carbon-in-Column (CIC) adsorption circuit. Gold and silver values will be loaded onto activated carbon and then be periodically stripped from the carbon in a desorption circuit, electrowon and smelted to produce the final doré product.

- Stage 2 (Years 4+) will be constructed during Year 3 and includes expanding the Stage 1 leach pad, adsorption and desorption circuits, and adding a 2,600 ton per day (2,360 tonnes per day) crushing system and mill for high-grade ore with a Carbon-in-Leach (CIL) circuit for recovery of gold and silver. For Stage 2, ROM production from newly mined ore will increase to 42,500 tons per day (38,600 tonnes per day) for a total processing rate of 45,100 tons of ore per day (40,900 tonnes per day).

During Stage 2, high-grade ore only will be crushed to 100% passing 3/8” (9.5 mm) at an average rate of 144 t/h (131 tonnes/h) in a three-stage mobile / skid mounted crushing circuit. Process solution will be added to the high-grade ore in a single-stage ball mill and ground to 80% passing 100 mesh in closed circuit with hydrocyclones. The gravity concentration system will include a Knelson concentrator, and an intensive leach reactor system to recover metal values. The CIL circuit will have six stages with a total residence time of 36 hours. Loaded carbon from the first tank of the CIL will be processed in the ADR plant shared with the ROM circuit. Tailings from the CIL will then be thickened to 58% solids by weight and will be pumped to the filter feed tank for cyanide detoxification. A Caro’s acid generator will deliver Caro’s acid into the filter feed tank to destroy residual cyanide in the thickened slurry. Detoxified tailings from the agitated filter feed tank will be pumped to two recessed plate filter presses to remove moisture. The resulting filter cake will discharge onto a collecting conveyor and will be conveyed to a filter cake stockpile and truck-stacked at a designated dry tailings disposal impoundment lined adjacent to the leach pad.

A summary of the processing design criteria is presented in Table 1-4.

Table 1-4. Processing Design Criteria Summary

Item	Design Criteria
Annual Tonnage Processed	
Stage 1 (Years 1-3)	5,114,000 tons (4,636,000 tonnes)
Stage 2 (Years 4+)	16,200,000 tons (14,700,000 tonnes)
Grade, oz/t Au (g/t)	
Low-grade Years 1-3	0.0100 oz/ton (0.343 g/tonne)
Low-grade Years 4+	0.0127 oz/ton (0.435 g/tonne)
High-grade Years 4+	0.0926 oz/ton (3.17 g/tonne)
Production Rate	
Stage 1	14,000 tons/day (12,600 tonnes/day), 365 days per year
Stage 2	45,100 tons/day (40,900 tonnes/day), 365 days per year
Processing	
Stage 1	ROM Heap Leach
Stage 2	ROM Heap Leach - Low-grade - 42,500 t/d (38,600 tonnes/d), Mill - High-grade – 2,600 t/d (2,360 tonnes/d)
Recovery Gold	
Low-grade Years 1-3	72.4%
Low-grade Years 4+	72.4%
High-grade	94%
Recovery Silver	
Low-grade	20%
High-grade	TBD
Operation	12 hours/shift, 2shifts/day, 7 days/week, 360 days/year
Heap Leaching Cycle	
Stage 1	80 day primary, 80 day secondary
Stage 2 – ROM Ore	160 days

During Stage 1, on-site natural gas powered generators will be used to supply electric power to all elements of the process plant. Line power will be installed as part of the Stage 2 expansion and the Stage 1 generators will be converted to emergency backup generators.

The heap leach facility will be constructed using a double liner system to prevent release of process solutions to the environment. The liner system consists of two layers of 80-mil linear low-density polyethylene (LLDPE) geomembrane with a 2-foot (0.6 m) thick layer of drainage gravel between them.

Event ponds will be included to contain seasonal accumulations of leach solutions and/or upset conditions that cannot be managed during normal operations. The event ponds will be constructed in two phases. Event solution will be returned to the barren tank as makeup solution as soon as practical.

1.14 Infrastructure

Infrastructure remaining from previous operations at the Castle Mountain project site include the main site access road as well as the west wellfield area which supplied water for the past operations. Water supply for ROM heap leach and mill will primarily be from new wells with the existing west wellfield and water tank being used only for water supply to water trucks for dust suppression.

The existing access road is a two-lane road in Nevada and one-lane road with two-lane passing areas in California, and is sufficient for current exploration and preliminary construction activities. For major construction and operations, road improvements, including road widening will be required.

Buildings and facilities for the project and operations have been considered and will be constructed in two stages. Buildings required during Stage 1 include the administration and mine offices buildings, a small modular laboratory, site gate house, ADR and reagent storage facility, and refinery building. A mill shop and warehouse building will be constructed in Stage 2, along with a larger laboratory.

For Stage 1 operations (Years 1-3), electrical power for the Project will be supplied using propane generators. For Stage 2 operations (Year 4+), electrical power for the Project will be supplied using line power, in a similar configuration to what was provided for historical operations. The plan is to receive retail service to the project by NV Energy (NVE). The Project power estimates for Stage 1 are 1.0 MW peak demand load with an average demand of 0.8 MW, and for Stage 2 are 7.1 MW peak demand load with an average demand of 6.0 MW.

A water balance model was prepared and considers the Project's water demand, water collected from direct precipitation and seasonal evaporation. Additional water consumption allowances in gallons per minute (gpm) were included for road dust suppression (100 gpm or 6.3 L/s), mill tailings moisture loss (65 gpm or 4.1 L/s), and miscellaneous uses (15 gpm or 0.95 L/s). Based on the water balance model plus these allowances, makeup water requirements average 360 gpm (23 L/s) during Stage 1 operations and 650 gpm (41 L/s) during Stage 2. Currently existing wells (five total) are sufficient to supply the necessary makeup requirement for Stage 1, after which additional wells south of the project site will be constructed to supply water for Stage 2.

Lavatory and wash facilities will be located throughout the project site. Sanitary waste from the lavatories will flow by gravity to multiple septic systems for treatment and disposal. A licensed waste management company will transport collected solid wastes to a dedicated offsite, third party controlled landfill site. Hazardous waste will be disposed of in a safe and environmentally sound manner using outside contractors.

1.15 Environmental Studies and Permitting

The number of wells used during the operating period ranged from five to 14 wells. This number includes a combination of monitoring wells and production wells. As part of the permitting requirements, water levels were measured monthly.

During the previous operation, the average annual water use was 400 acre feet per year (248 gpm or 16 L/s). The maximum permitted annual water use for the mine expansion was adjusted downward (1998 EIS\EIR approvals) to 625 acre feet per year or 387 gpm or 24 L/s (in the 1990 EIS/ EIR, the predicted water use was 725 acre feet per year or 449 gpm or 28 L/s) because actual water use was lower than predicted. Water quality measurements were taken at a number of wells throughout the operation. Water quality during operations was within the predicted concentrations.

The 1998 EIS/EIR analyzed the potential for acidic conditions in pit water and found, once again, the Project has very limited acid-generating sulfide minerals, and the natural alkalinity provided by the rock and ground and surface water inflows minimize the potential for acidification of the pit water.

Cultural resources field studies were undertaken as part of the environmental assessment reviews to identify if there were any significant sites to be considered for inclusion in National Register of Historic Places (NRHP) and/or the California Register of Historic Resources (CRHR). The field studies evaluated both historic and prehistoric resources at the Project site. Approximately 48 sites were identified. Mitigation measures excluded certain sites from mine development. A chain link fence was built around the Hart town site cemetery and a 300-foot (91 m) buffer zone separates the cemetery from the North Overburden Site. Future Project design activities will acknowledge and accommodate all historic and prehistoric resources found on the site.

On October 31st, 1994, the Mojave National Preserve was established 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 north, west, east and south of the Project. The Project is bounded on all sides by a buffer zone administrated by the Bureau of Land Management.

On February 12, 2016, Barrack Obama, President of the United States of America, by presidential proclamation, established the Castle Mountains National Monument. The reserved Federal lands and interests in lands encompass approximately 20,920 acres and the boundaries fall between the Project and the aforementioned 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.

All permits were in place when the Castle Mountain Mine was operating. Since 2012, the Project has been maintained on idle status. During this period, the environmental review permits issued after the Project was released from the State and Federal environmental assessment processes were maintained. Also, all fees have been paid and all applicable permits and authorizations have been maintained by NewCastle. The Project was returned to active status in 2017.

1.16 Capital and Operating Costs

Capital and operating costs for the Castle Mountain Project were estimated by KCA, GRE and GLA with input from Equinox Gold. The estimated capital and operating costs are considered to have an accuracy of +/- 25% and +/- 20% respectively.

The capital costs have been estimated primarily by KCA for the process and infrastructure, and GRE for mining. All equipment and material requirements are based on the design information described in this study. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items. Where supplier quotes were not available, a reasonable cost estimate was made based on supplier quotes in KCA's project files and cost guide data.

Operating costs for all areas of the project have been estimated from first principles. Labor costs are estimated using project-specific staffing, salary, wage, and benefit requirements. Unit consumptions of materials, supplies, power, water, and delivered supply costs are also estimated.

The total capital cost for the Project is $488.7 million including all applicable sales tax. The project will be developed in stages with Stage 1 being constructed in Year -1 to process ROM ore from the JSLA pit. Stage 2 will be constructed in Year 3 and includes the addition of a mill and CIL circuit and owner mining fleet, along with significant capitalized mining pre-stripping activities. Sustaining capital for the expansion of the heap leach pad and replacement of equipment is considered throughout the life of the mine. Table 1-5 presents the capital requirements for the Project.

Table 1-5 - Capital Costs Summary

Description	Cost (US$)
Stage 1 Pre-Production Capital	$51,667,000
Stage 2 Expansion Capital	$294,958,000
LOM Sustaining Capital	$142,029,000
TOTAL Capital Costs Including Sales Tax	**$488,654,000**

The total life of mine operating cost for the Project is $ 8.43 per ton of ore processed. Table 1-6 presents the LOM average operating cost requirements for the Castle Mountain Project.

Table 1-6 - Operating Costs LOM Summary

Description	LOM Cost US$/ton ore (US$/tonne ore))
Mine	$5.79 ($6.38)
Process & Support Services	$1.92 ($2.11)
Site G&A	$0.72 ($0.80)
TOTAL Operating Costs	**$8.43 ($9.29)**

1.17 Economic Analysis

Based on the estimated production parameters, revenue, capital costs, operating costs, taxes, and royalties, a cash flow model was prepared by KCA for the economic analysis of the Castle Mountain Project. All of the information used in this economic evaluation has been taken from work completed by KCA and other consultants as described in this report.

The Castle Mountain Project economics were evaluated using a discounted cash flow (DCF), which measures the Net Present Value (NPV) of future cash flow streams. The final economic model was developed with input from Equinox Gold using the following assumptions.

The period of analysis is 20 years, and includes one year of pre-production and investment, 16 years of production, and three years for reclamation and closure). The major inputs to the analysis are as follows:

- Gold price of $1,250/oz.
- Stage 1 design processing rate of 14,000 tpd or 12,600 tonnes/d (Years 1-3, ROM only)
- Stage 2 design processing rate of 45,100 tpd or 40,900 tonnes/d (Years 4-17, 42,500 tpd or 38,600 tonnes/d for ROM and 2,600 tpd or 2,360 tonnes/d for mill).
- Average ROM gold grade of 0.012 oz/ton (0.41 g/tonne).
- Average mill gold grade of 0.094 oz/ton (3.22 g/tonne).
- LOM average opex of $8.43/ton ($7.65/tonne) ore.
- Total LOM capex of $433.7M (not including working capital and reclamation & closure costs).
- Net Smelter Royalties, with an average NSR of 4.31%:
    - 2.65% FNV royalty applied to all ounces;
    - 5.00% Conservation royalty;
    - 2.00% American Standard royalty; and
    - 5.00% Huntington Tile royalty.
- State Income Tax rate of 8.84%.
- Federal Income Tax rate of 21%.
- Gold recoveries of:
    - 72.4% for ROM ore; and

- o 94.0% for mill ore.

Table 1-7 - Life of Mine Summary

Economic Analysis		
Internal Rate of Return (IRR), Pre-Tax	21.7%	
Internal Rate of Return (IRR), After-Tax	20.1%	
Average Annual Cashflow (Pre-Tax)	$54.3	M
NPV @ 5% (Pre-Tax)	$490.8	M
Average Annual Cashflow (After-Tax)	$45.9	M
NPV @ 5% (After-Tax)	$406.5	M
Gold Price Assumption	$1,250	/Ounce
Silver Price Assumption	$17	/Ounce
Pay-Back Period (Years based on After-Tax)	8.8	Years
Capital Costs		
Phase 1 Initial Capital	$51.7	M
Phase 2 Initial Capital	$295.0	M
LOM Sustaining Capital	$142.0	M
Operating Costs (Average LOM)		
Mining	$5.79 ($6.38	/Ton processed /Tonne processed)
Processing & Support	$1.92 ($2.11	/Ton processed /Tonne processed)
G&A	$0.72 ($0.80	/Ton processed /Tonne processed)
Total Operating Cost	$8.43 ($9.29	/Ton processed /Tonne processed)
Total By-Product Cash Cost	$712	/Ounce Au
All-in Sustaining Cost	$763	/Ounce Au
Production Data		
Life of Mine	16.2	Years
Total Ton to Heap	207,057,520 (187,842,582	Tons Tonnes)
Total Ton to Mill	10,744,919 (9,747,791	Tons Tonnes)
Grade Au (Avg.)	0.016 0.56	oz/ton g/tonne
Contained Au oz	3,563,093	Ounces
Metallurgical Recovery Au (Overall)	79%	
Average Annual Gold Production	173,000	Ounces
Total Gold Produced	2,798,173	Ounces
LOM Strip Ratio (W:O)	3.76	

Sensitivity of the project economics to key parameters including gold price, total capital cost and operating was completed to evaluate the relative strength of the project. The sensitivities are based on +/- 25% of the base case. The after-tax sensitivity analysis is presented in Table 1-8, and graphically in Figures 1.1, 1.2, 1.3 and 1.4. The economic indicators chosen for sensitivity evaluation are the internal rate of return (IRR) and NPV at 0, 5, and 10% discount rates.

The sensitivity analysis indicates that the project is robust and is most sensitive to revenue (gold price, ore grade, and recovery), and operating costs.

Table 1-8 - Sensitivity Analysis (After Tax)

Gold price ($/oz)	**-25%**	**-10%**	**$1,250**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	-$21.5	$243.6	**$406.5**	$565.7	$799.7
IRR (after tax)	4.2%	14.0%	**20.1%**	26.3%	35.8%

Capital costs	**-25%**	**-10%**	**$471.0**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	$478.7	$435.6	**$406.5**	$377.5	$333.8
IRR (after tax)	27.1%	22.5%	**20.1%**	18.1%	15.6%

Operating costs	**-25%**	**-10%**	**$1,836.0**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	$624.9	$495.2	**$406.5**	$315.6	$175.2
IRR (after tax)	30.0%	24.0%	**20.1%**	16.4%	11.0%

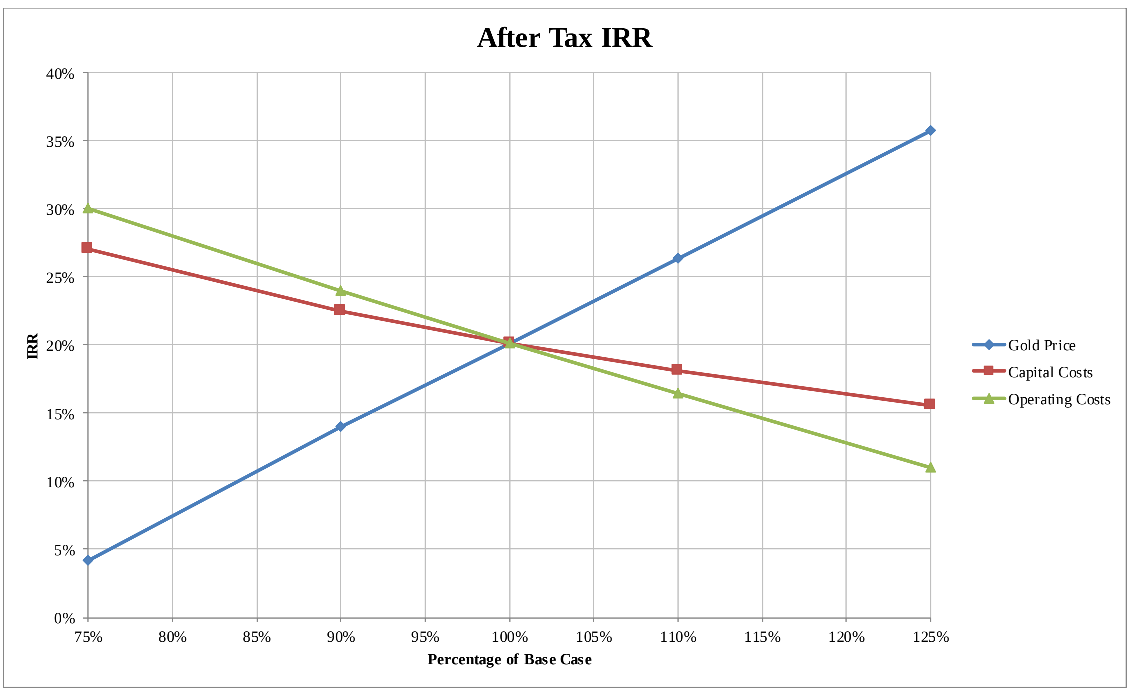

Figure 1-1 - After-Tax IRR vs. Gold Price, Capital Cost, and Operating Cash Cost

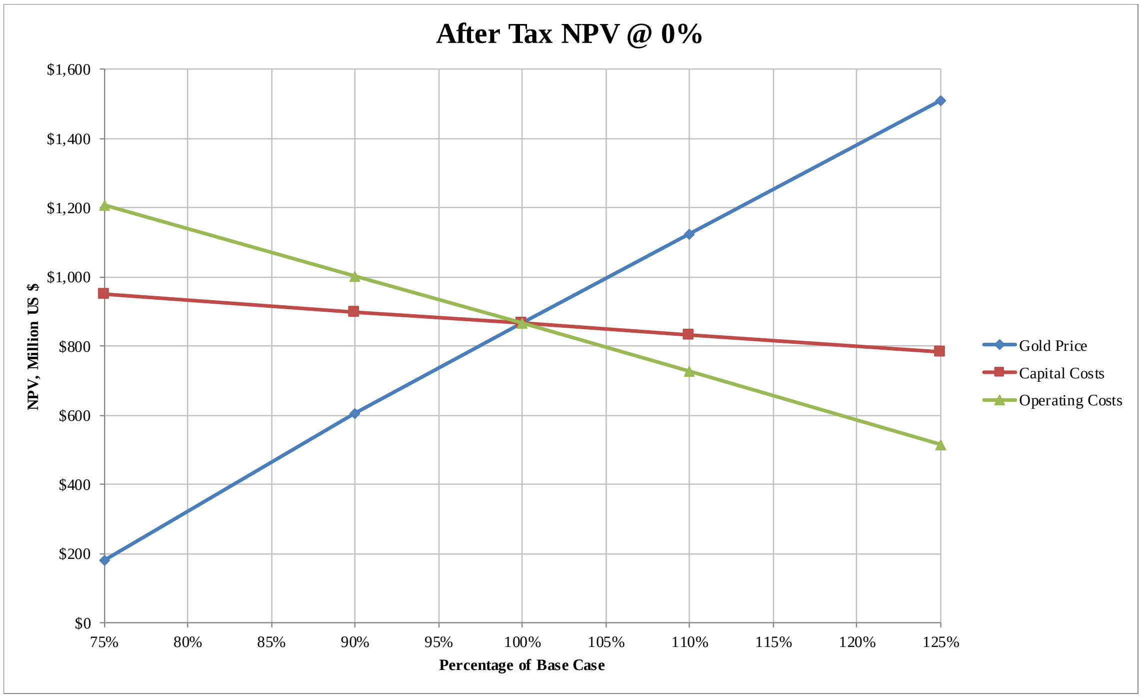

Figure 1-2 - NPV @ 0% vs. Gold Price, Capital Cost, and Operating Cash Cost

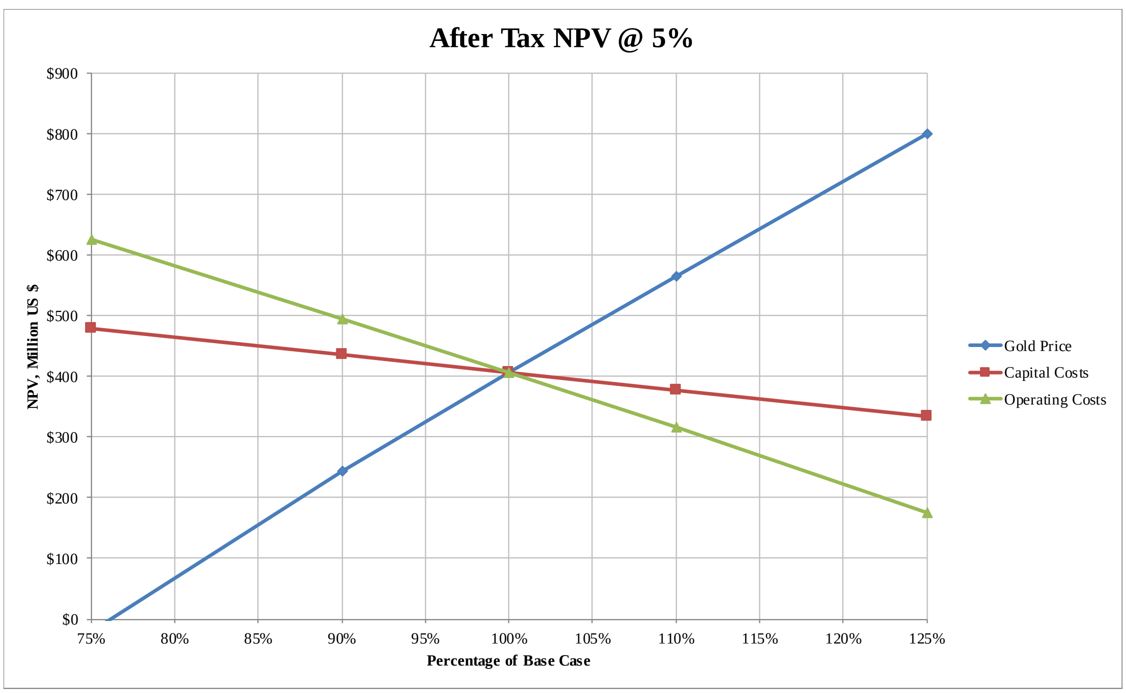

Figure 1-3 - NPV @ 5% vs. Gold Price, Capital Cost, and Operating Cash Cost

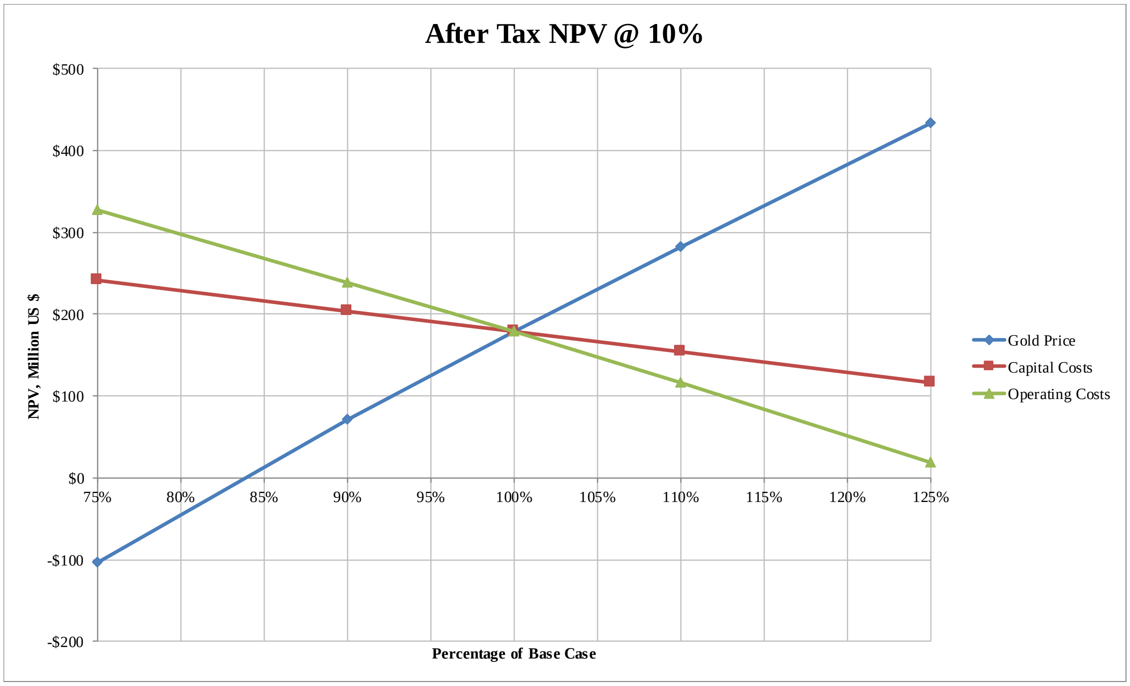

Figure 1-4 - NPV @ 10% vs. Gold Price, Capital Cost, and Operating Cash Cost

1.18 Risks and Opportunities

1.18.1 Opportunities

Resources and Reserves

Within the designed pit, there is an inferred resource of 70.8 Mt (64.2 M tonnes) of material above a 0.005 oz/ton(0.17 g/tonne) cutoff. This portion of the mineral resource presents a significant opportunity for increased reserves, and the conversion of that resource to reserve should enhance project economics as it is currently treated as waste rock. Additionally, the lower portions of the JSLA backfill material represent a similar opportunity to increase reserves from material currently treated as waste. Currently unexplored, the deposition of the backfill is well documented through monthly as-built drawings from Viceroy, and it may have previously considered waste that is now above cutoff.

Silver Credit

Silver is not included in the mineral resource estimate or reserve, and accordingly no credit to silver recovery has been given in the economic analysis. Silver recovery may present an upside potential of up to 700,000 ounces recovered life of mine, or $10 million at current silver prices.

Pit Slope Angles

There is an opportunity that the pit slope angles may be steepened in areas with better three-dimensional controls of where the major through going faults are located. In addition, better geological understanding of where some of the wall rock is clay altered could also allow for better controls on the pit slopes.

1.18.2 Risks

South Domes Metallurgy

Test work in support of South Domes (approximately one-third of the total Project ounces) ore recoveries is currently limited to variability bottle roll tests, which however do indicate similar recoveries to JSLA bottle roll tests. There is a low risk that a lower recovery for South Domes ore may be realized which can negatively impact project economics. This risk can be reduced significantly by running coarse crush column tests on composites of South Domes ores to support the selected recovery.

Water Supply

The processing rate and gold production at Castle Mountain is critically dependent on water supply. The currently existing water supply network (and the planned infrastructure within permitted areas) is sufficient for Stage 1 operations but not Stage 2 operations, and additional hydrological studies and drilling outside of currently permitted areas will be required to secure the necessary Stage 2 water requirement. There is a moderate risk that the necessary water cannot be secured either due to technical or permitting reasons, which would limit the Stage 2 production rate of ore and negatively impact project economics.

Land Title Risks and Designation

Although Equinox Gold may receive title opinions for any mineral properties in which Equinox Gold has or will acquire a material interest, there is no guarantee that title to such properties will not be challenged or impugned. Also, in the United States, claims have been made and new claims are being made by aboriginal peoples that call into question the rights granted by the government. A determination of defective title or restrictions in connection with a challenge to title rights could impact Equinox Gold's ability to develop and operate at Equinox Gold's mineral projects.

The Mojave Desert Preserve established in 1994 surrounds Equinox Gold's patented and unpatented land. In addition, there is an approximately 22,000 acre "buffer zone" surrounding Equinox Gold's lands. On February 12, 2016, Equinox Gold announced that its claim holdings and private land held would not be included in the new Castle Mountains National Monument following the proclamation by then US President Barack Obama under the Antiquities Act of 1906. The Monument surrounds but does not include an approximately 8,340 acre parcel referred to as the 'Castle Mountain Mine Area', consisting of BLM-managed Federal land, State land, and private land. The Castle Mountain Project is included in its entirety within the Castle Mountain Mine Area.

The proclamation also directs that after any such mining and reclamation are completed at the Castle Mountain Project, or after 10 years if no mining occurs, then jurisdiction over federal lands in the Castle Mountain Mine Area is to be transferred to the National Park Service. There can be no assurance that the Castle Mountain Project will not be included in any expansion of the Monument or that the jurisdiction of the Castle Mountain Project will not be transferred to the National Park Service.

In Equinox Gold's view, these land designations do not impede Equinox Gold's plans for developing the Castle Mountain Project. Equinox Gold is not able to provide any assurance regarding any future designation of lands, nor the timing of implementation of any such designations.

1.19 Recommendations

Recommendations for future development work for the Castle Mountain Project include:

Geology and Exploration

- Add fields to the Castle Mountain lithology database for the updated geologic interpretations to inform future statistical analysis and grade modelling efforts.
- Complete the 3D geologic interpretation through the mined-out pits to generate a complete geologic package and to allow for comparison of current grade modelling methods with historical mine results.
- Explore the area outside of the current grade shells, especially to the east, north, and west of the current limits of the deposit.
- Perform detailed mapping of faults and review the geological logs to produce a three-dimensional model of the major faults.
- Perform a detailed analysis of the areas of clay alteration, with special attention to the lithology to determine where clay altered tuffs are located, as these could present challenges for future pit wall stability.

Mine Planning and Design

- Additional optimization of the pit design, phase design, production scheduling, and waste dump design is recommended during the Feasibility and detailed design engineering effort.

Metallurgical Testing

- Conduct variability bottle roll tests with carbon from several samples throughout the JSLA, South Domes, and Oro Belle pit areas, to confirm CIL operating parameters, reagent consumptions and recoveries.
- Conduct Caro's Acid and INCO/SO_2 detoxification tests on a composite slurry sample to confirm detoxification operating parameters, final residual cyanide values, and confirm reagent requirements.
- Conduct at least four column tests, for two crush sizes in duplicate, on composites prepared from the South Domes area to confirm ROM recovery for low-grade ore from the South Domes area.

1.20 Recommended Work Program

Based on the encouraging exploration and development results and updated design and economic results from the PFS, KCA recommends that Equinox Gold initiate a full feasibility study (FS) using the current mineral resource estimate, metallurgical test work, and current PFS process designs as a new foundation. The FS should advance the mine design, metallurgical test work, and process design.

KCA recommends Equinox Gold continue with expanded hydrogeological studies and drilling, environmental baseline, metallurgical and geotechnical studies.

The recommended scope of work is estimated to cost about $8.97 million dollars.

2.0 INTRODUCTION

2.1 Purpose

Equinox Gold commissioned Kappes, Cassiday, and Associates (KCA) to complete an estimate of mineral resources for the Castle Mountain Project (the Project) located in San Bernardino County, California, USA, and to provide the results of this work in a report (Report) that was prepared in compliance with National Instrument 43–101, Standards of Disclosure for Mineral Projects (NI 43–101).

KCA understands that this Report will be submitted to a Canadian stock exchange in support of filings by Equinox Gold. The purpose of this Report is to provide a technical summary of the Castle Mountain Project in support of Equinox Gold's listing on a Canadian stock exchange.

2.2 Source of Information

Information for the Report was obtained from work completed by KCA at the project site and at KCA's offices in Reno, Nevada, USA, and materials provided by, and discussions with, Equinox Gold personnel, along with the other Qualified Persons listed in the report.

2.3 Qualified Persons

The Qualified Persons responsible for preparation of the Technical Report include Mr. Tim Scott, SME RM, Senior Engineer, Mr. Todd Wakefield, SME RM, MTS Principal Geologist, Mr. Don Tschabrun, SME RM, MTS Associate Principal Mining Engineer, and Ms. Terre Lane, MMSA, SME RM, Principal Mining Engineer.

Table 2.1 lists the areas of responsibility for each Qualified Person in this report.

Table 2-1 – QP Areas of Responsibility

Section #	Description	Qualified Person
Section 1	Summary	All QPs
Section 2	Introduction	T. Scott
Section 3	Reliance on Other Experts	T. Scott
Section 4	Property Description and Location	T. Scott
Section 5	Accessibility, Climate, Local Resources, Infrastructure and Physiography	T. Scott
Section 6	History	T. Scott
Section 7	Geological Setting and Mineralization	T. Wakefield
Section 8	Deposit Types	T. Wakefield
Section 9	Exploration	T. Wakefield
Section 10	Drilling	T. Wakefield – All except T. Lane (10.5, 10.7, 10.8)
Section 11	Sample Preparation, Analysis and Security	T. Wakefield – All except T. Lane (11.3)
Section 12	Data Verification	T. Wakefield – All except T. Lane (12.1.2, 12.2.3, 12.3.3, 12.4.3, 12.6.2)
Section 13	Mineral Processing and Metallurgical Testing	T. Scott
Section 14	Mineral Resource Estimates	D. Tschabrun
Section 15	Mineral Reserve Estimates	T. Lane
Section 16	Mining Methods	T. Lane
Section 17	Recovery Methods	T. Scott
Section 18	Project Infrastructure	T. Scott
Section 19	Market Studies and Contracts	T. Scott
Section 20	Environmental Studies, Permitting and Social or Community Impact	T. Scott
Section 21	Capital and Operating Costs	T. Scott - All except T. Lane (21.1.1, 21.2.1)
Section 22	Economic Analysis	T. Scott
Section 23	Adjacent Properties	T. Scott
Section 24	Other Relevant Data and Information	All QPs
Section 25	Interpretation and Conclusions	All QPs
Section 26	Recommendations	All QPs
Section 27	References	All QPs

2.4 Field Involvement of Qualified Persons

Tim Scott of KCA completed a site visit of the Castle Mountain property on 4 to 5 April 2018, accompanied by Equinox Gold representative Mr. Leduc. During his site visit, Mr. Scott reviewed the existing site infrastructure including the water tank and wells in the West Wellfield area, the existing access road, the historical heap, and the areas planned for the new heap, ponds, ADR plant, mill, and crusher. Mr. Scott also visited the JSLA backfill pit area. Additionally, Mr. Scott visited the Searchlight substation and the Walking Box Ranch switching yard where power line infrastructure is planned in Stage 2 Operations.

Mr. Wakefield and Mr. Tschabrun of MTS completed a site visit of the Castle Mountain property on 13 to 16 December 2016 accompanied by NewCastle representatives Mr. Kunkel and Mr. Leduc. During this site visit, Mr. Wakefield and Mr. Tschabrun reviewed the geologic setting and NewCastle's infill drill plans, and visited the Oro Belle, Jumbo, and JSLA pits, and South Domes area. Mr. Wakefield visited the Project an additional 12 days from January to April 2017 to perform database audits, review assay QA/QC results, and review geologic interpretations with NewCastle staff.

Terre Lane of GRE completed a site visit of the Castle Mountain property on 2 February 2018, accompanied by Equinox Gold representative Mr. Leduc (COO), Mr. Roberts (Exploration Manager), and other Equinox Gold staff. During the site visit, Ms. Lane viewed the project site, existing pits, back filled pits, waste dumps, historic leach pad, Rotary Air Blast (RAB) drill, RAB drill cuttings, office and sample prep facilities.

2.5 Previous Technical Reports

Previously filed Technical Reports on the Project include:

- NI 43-101 Report on the Castle Mountain Property, San Bernardino County, California, USA, Prepared for Telegraph Gold Inc. and Foxpoint Capital Corp., Temkin, T., 24 October 2012

- Technical Report on the Mineral Resource Estimate for Castle Mountain Project, San Bernardino County, California, USA, Prepared for Castle Mountain Mining Company Limited, Roscoe Postle Associates Inc., 6 December 2013

- Technical Report on the Preliminary Economic Assessment for Castle Mountain Project, San Bernardino County, California, USA, Prepared for Castle Mountain Mining Company Limited, Roscoe Postle Associates Inc., 30 May 2014

- NI 43-101 Technical Report and Updated Mineral Resource Estimate for the Castle Mountain Project, San Bernardino County, California, USA, Prepared for NewCastle Gold Ltd., Gray, J.N., Singh, R.B., Pennstrom, W.J., Kunkel, K.W., Cunningham-Dunlop, I.R., 2 December 2016

- Castle Mountain Project, San Bernardino County, California, USA, NI 43-101 Technical Report, Prepared for NewCastle Gold Ltd., Wakefield, T. and Tschabrun, D., 26 October 2017

2.6 Terms of Reference

KCA and the qualified persons of this Report are independent from Equinox Gold.

The effective date of this report is 16 July 2018, which represents the date of information used in the report. The effective date of the mineral resource estimate for the Castle Mountain Project is 29 March 2018, which represents the date of exploration information used for mineral resource estimation. The effective date of the mineral reserve estimate for the Castle Mountain Project is 29 June 2018. There has been no material change to the information between the effective date and the signature date of the Technical Report.

Unless stated otherwise, all quantities are in imperial units and currencies are expressed in constant 2018 US dollars.

2.7 Units of Measure

2.7.1 Common Units

Above mean sea level	amsl
Centimeter	cm
Centimeters per second	cm/sec
Cubic Feet	ft2
Cubic Meters	ft3
Day	d
Days per week	d/w, dpw
Days per year (annum)	d/y(a), dpy(a)
Degree	°
Degrees Celsius	°C
Degrees Fahrenheit	°F
Feet	ft
Gallons	gal
Gallons per minute	gpm
Gallons per minute per square foot	gpm/ft2
Gram	g
Grams per tonne	g/t
Greater than	>
Hectare	ha
Hertz (frequency)	Hz
Hour	h, hr
Hours per day	h/d, hpd
Hours per week	h/w, hpw
Hours per year	h/y(a), hpy(a)
Kilo (thousand)	k
Kilogram	kg
Kilometer	km
Kilovolt	kV
Kilowatt	kW
Kilowatt-hour	kWh
Less than	<

Linear foot	LF
Liter	l
Liters per hour per square meter	L/hr/m2
Megawatt	MW
Micrometer (micron)	µm
Milligram	mg
Milligrams per liter	mg/L
Milliliter	mL
Millimeter	mm
Million ounces	Moz
Million tons	Mtons
Million tonnes	Mtonnes
Million	M
Minute (time)	min
Month	mo
Ounce	oz
Ounces per ton	oz/t, opt
Parts per billion	ppb
Parts per million	ppm
Percent	%
Phase (Electrical)	ph
Pound	lb
Pounds per Square Inch	psi
Pounds per ton	lbs/ton
Specific gravity	SG
Square Feet	SF, ft2
Ton	t
Tons per day	t/d, tpd
Tons per month	tpm
Volt	V
Year (annum)	y (a)

2.7.2 Abbreviations

Atomic Adsorption	AA
Adsorption-Desorption-Recovery	ADR
American Society for Testing and Materials	ASTM
Bottle Roll Test	BRT
Bureau of Land Management	BLM
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Carbon in Column	CIC
Carbon in Leach	CIL
Certified Reference Materials	CRM
Conventional Rotary Drill	CR
Diamond Drill	DD
Global Positioning System	GPS
Internal Rate of Return	IRR
Jumbo South-Leslie Ann (Pit)	JSLA
Life of Mine	LOM
Net Present Value	NPV
Net Smelter Return	NSR
Quality Assurance/Quality Control	QA/QC
Reverse Air-Blast (Drilling)	RAB
Reverse Circulation	RC
Rock Quality Designation	RQD
Run of Mine	ROM
Universal Transverse Mercator	UTM

3.0 RELIANCE ON OTHER EXPERTS

KCA disclaims responsibility for information on Equinox Gold's rights to the Castle Mountain property. For this, KCA has fully relied upon an opinion by Gresham Savage Nolan & Tilden LLP, Attorneys at Law of San Bernardino, California dated July 13, 2017 entitled "*Supplemental Title Report/Update, Castle Mountain Project, San Bernardino County, California and Clark County, Nevada*" (Gresham Savage Nolan & Tilden LLP, 2017). This opinion is relied upon in Sections 4.2 and 4.3 of the Report.

KCA disclaims responsibility for information regarding Equinox Gold's legal agreements concerning the Castle Mountain property. For this, KCA has fully relied upon an opinion by Gresham Savage Nolan & Tilden LLP, Attorneys at Law of San Bernardino, California dated July 13, 2017 entitled "*Supplemental Title Report/Update, Castle Mountain Project, San Bernardino County, California and Clark County, Nevada*" (Gresham Savage Nolan & Tilden LLP, 2017). This opinion is relied upon in Section 4.2.1 of the Report.

KCA disclaims responsibility for information regarding the environmental liabilities and risks of the Castle Mountain property. For this, KCA has fully relied upon the opinion of Marc Leduc P.Eng, EVP US Operations, Equinox Gold. This opinion is relied upon in Section 4.4 of the Report.

KCA disclaims responsibility for information regarding the environmental studies and permitting of the Castle Mountain property. For this, KCA has fully relied upon the opinion of Marc Leduc P.Eng, EVP US Operations, Equinox Gold. This opinion is relied upon in Sections 4.5 and 20 of the Report.

KCA is not an expert in tax law, and has relied upon information from Equinox Gold for developing the tax model applied for the economic analysis of the Castle Mountain property, particularly regarding classification of assets for depreciation, depreciation schedules, and depletion. The information used in developing the tax model is relied upon in Section 22 of the Report.

4.0 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 (112.6 km) south of Las Vegas, Nevada (Figure 4.1). The Project is located in the high desert area near the Mojave National Preserve and Castle Mountains National Monument.

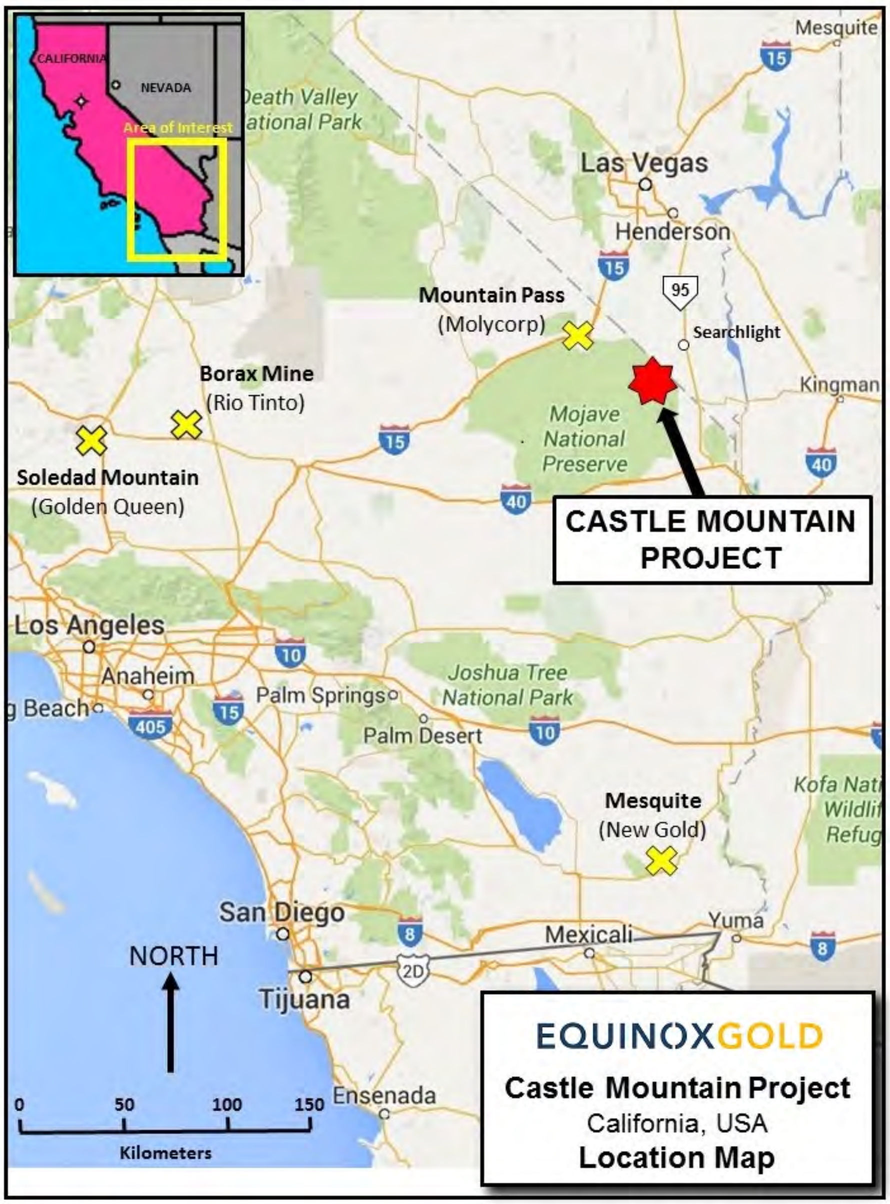

Figure 4-1 – Location Map, Castle Mountain Project

4.2 Land Tenure

The Project, as illustrated in Figure 4.2, includes 13,276 acres of patented and unpatented lode, placer and mill site claims as follows:

a) The Castle Mountain property, located in San Bernardino County, State of California, which comprises an aggregate total of 10,373 acres including:
   - 1,301 acres of patented lode and mill site claims registered in the name of Viceroy Gold Corporation (Viceroy).
   - 3,209 acres of unpatented lode, placer and mill site claims (272 claims) registered in the name of Castle Mountain Venture (CMV).
   - 2,951 acres of unpatented placer claims (19 claims) registered in the name of CMV
   - 1,979 acres of unpatented lode and mill site claims (222 claims) acquired by location in 2016 and registered in the name of CMV or Viceroy.
   - 936 acres of unpatented lode and mill site claims (105 claims) acquired by location in 2016 and registered in the name of Castle Mountain Venture.

b) The Stateline property, located in Clark County, State of Nevada, comprises an aggregate total of 2,903 acres including:
   - 2,903 acres of unpatented lode claims (171 claims) acquired by location in 2016 and registered in the name of Viceroy.

Patented and unpatented claims are located in Townships, Ranges and Sections as shown in Table 4-1.

4.2.1 Acquisition of Castle Mountain Venture

Subject to certain obligations, NewCastle has 100% of the right, title and beneficial interest in and to Castle Mountain Venture (CMV) which owns the Project. 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.

Table 4-1 – Castle Mountain Project Land Tenures by Township and Range

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

Township & Range, all Mt. Diablo Base & Meridian	Sections
T30S R62E	2, 3, 4, 5, 8, 9, 10, 11, 14, 15, 16, 22, 23
T29S R62E	32, 33, 34, 35

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 is 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 on the following terms:

- A first payment in the form of 4,000,000 shares of Telegraph Gold Inc. was made on September 6, 2012 .
- A second payment of C$3,000,000 in cash or shares due upon completion of a Feasibility Study or by September 6, 2015.
- A third payment of C$5,000,000 in cash or shares due upon starting commercial production or by September 6, 2018.

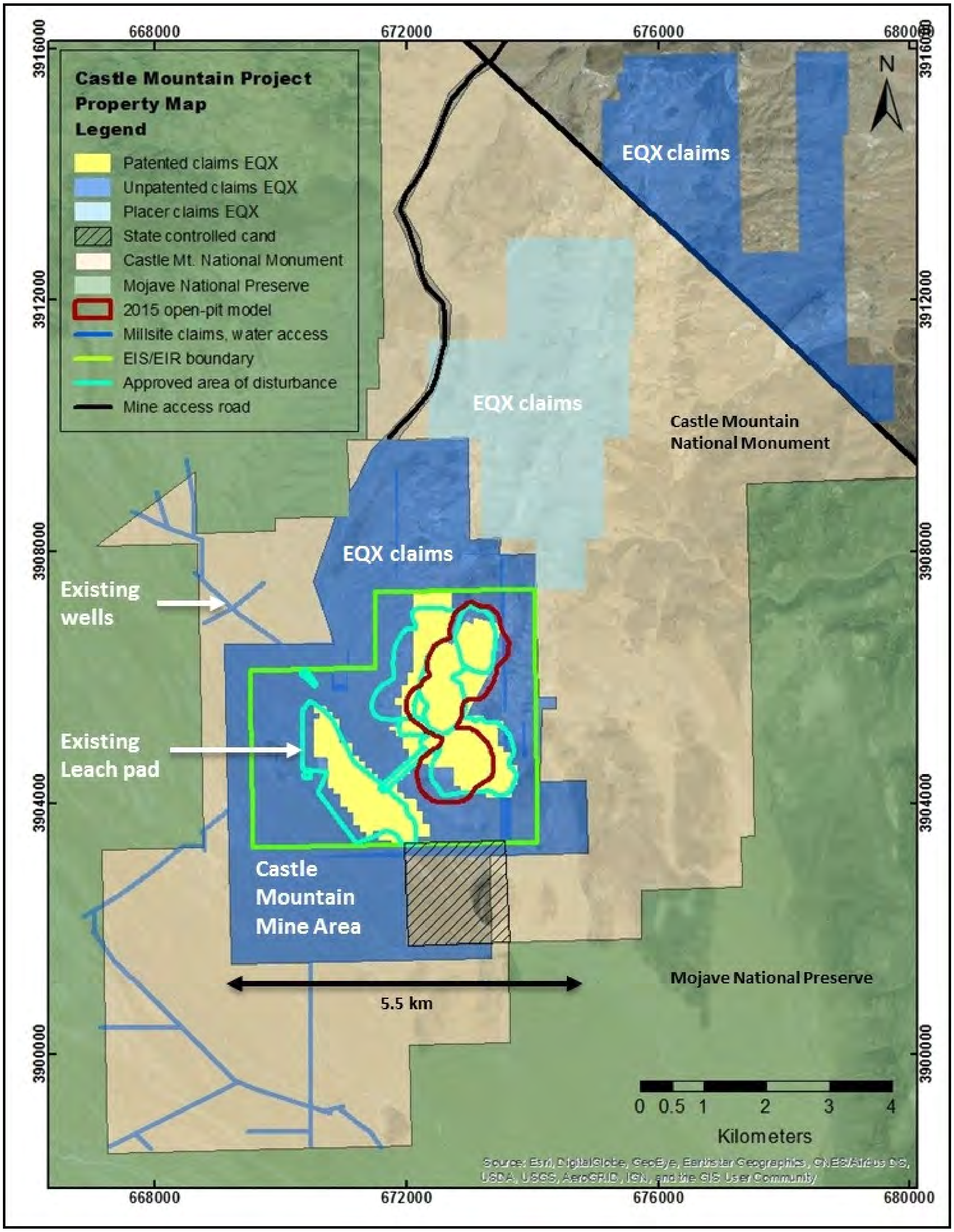

Figure 4-2 – Property Map, Castle Mountain Project

On September 2, 2015, NewCastle and Sprott amended the future property payment terms under the share purchase agreement dated September 6, 2012 as follows:

- The second payment, originally due to Sprott on September 8, 2015, was now due on February 1, 2016 (the "Second Payment").
- The amount of the Second Payment was increased by 5% from C$3,000,000 to C$3,150,000, payable in cash, shares or both at the election of NewCastle.
- The third and final payment of C$5,000,000 remains due to Sprott on September 6, 2018 (the "Third Payment") unless, at any time following September 6, 2017, NewCastle's share price based on its 20-day volume weighted average price (VWAP) exceeds $0.65, in which case the Third Payment becomes immediately due. The Third Payment is payable in cash, shares or both at the election of NewCastle.
- If either the Second Payment or the Third Payment is satisfied by NewCastle with shares, the market price of such shares will be calculated using NewCastle's share price based on its 20-day VWAP (excluding up to nine non-trading days) prior to the date of such payment. If there are more than nine non-trading days in the 20-day period then the VWAP is calculated as the average of (i) the average of the closing bid and ask prices for each day on which there was no trading, and (ii) the VWAP for the days on which there has been trading.
- If, at any time prior to September 6, 2018, the Castle Mountain property is sold, leased or optioned or if NewCastle incurs debt out of the ordinary course or completes a debt financing, the Second Payment (if not already made) and the Third Payment become due to Sprott and must be satisfied in cash.

The Second Payment was satisfied on February 1, 2016, through the issuance of 10,769,230 Common Shares. On June 16, 2016, NewCastle paid a C$500,000 cash installment against the Final Payment to enable it to complete the royalty consolidation transaction with Franco Nevada described below in Section 4.3.

4.2.2 Annual Claim Maintenance Payments to BLM

NewCastle is required to pay an annual federal claim maintenance fees to the United States Bureau of Land Management (BLM) in the amount of $155 per 20 acre section per year in respect of unpatented lode and placer mining claims. These payments are due on September 1st of each year.

- The total for the 272 unpatented lode, mill site and placer claims is $42,160.00 per year, plus $291.00 for the San Bernardino County Affidavit and Notice of Intention to Hold and $75.00 for County Recording Fees.
- The total for the 19 placer claims is $23,560.00 per year, plus $247.00 for the San Bernardino San Bernardino County Affidavit and Notice of Intention to Hold and $75.00 for the County Recording Fees.
- The total for the 105 unpatented lode and mill site claims is $16,275.00 per year, plus $262.00 for the San Bernardino County Affidavit and Notice of Intention to Hold and $75.00 for County Recording Fees.

- The total for the 98 unpatented lode claims is $15,190.00 per year, plus $261.00 for the San Bernardino County Affidavit and Notice of Intention to Hold and $75.00 for the County Recording Fees.
- The total for the 124 unpatented lode and mill site claims is $19,220.00 per year, plus $267.00 for the San Bernardino County Affidavit and Notice of Intention to Hold and $75.00 for County Recording Fees.
- The total for the 164 lode mining claims is $25,420.00 per year, plus $1,983.00 for the Clark County Affidavit and Notice of Intention to Hold (including $12.00/claim) and $75.00 for the County Recording Fees.

All payments have been made in this respect for 2017/2018.

4.2.3 Annual Property Tax Payments to San Bernardino County, CA

Property taxes are also payable to San Bernardino County for the 17 tax parcels (patented & unpatented lode claims, mill site claims and placer claims) that comprise the Project (Table 4.2). Payments are due on a semi-annual basis on November 1st and February 1st and total $14,240.80. Payments have been made by NewCastle for the 2016-2017 tax year.

NewCastle has received revised property tax invoices with respect to one of the parcels increasing the property taxes for four prior tax years. The amounts are not currently due.

4.2.4 Summary of Land Obligations

All unpatented and patented claims are current as of December 31st, 2016 with respect to fees, taxes and levies. Equinox Gold asserts that it has full legal access to the Project with respect to surface and mineral rights. Equinox Gold also reports that there are no known dates of expiration to mining claims pertinent to the Project.

Table 4-2 – Annual Property Tax Payments to San Bernardino County, CA

2016-2017 ANNUAL SECURED PROPERTY TAX BILL
Castle Mountain Venture
c/o Viceroy Gold Corporation
P.O. Box 68
Searchlight, NV 89046

			Installment 1 Oweing		Installment 2 Oweing			
#	Parcel Number	Bill Number	Due Amount	Due Date	Due Amount	Due Date	Total Tax	Status
1	0569-291-04-0-000	160458322	$158.22	11/1/2016	$158.22	2/1/2017	$316.44	Fully Paid
2	0569-291-05-0-000	160458323	$276.13	11/1/2016	$276.11	2/1/2017	$552.24	Fully Paid
3	0569-291-08-0-000	160458324	$102.12	11/1/2016	$102.12	2/1/2017	$204.24	Fully Paid
4	0569-291-09-0-000	160458325	$102.12	11/1/2016	$102.12	2/1/2017	$204.24	Fully Paid
5	0569-291-13-0-000	160458326	$102.12	11/1/2016	$102.10	2/1/2017	$204.22	Fully Paid
6	0569-291-17-X-000	160458327	$33.03	11/1/2016	$33.03	2/1/2017	$66.06	Fully Paid
7	0569-291-17-X-001	160458328	$61.03	11/1/2016	$61.02	2/1/2017	$122.05	Fully Paid
8	0569-291-17-X-003	160458329	$49.54	11/1/2016	$49.54	2/1/2017	$99.08	Fully Paid
9	0569-291-20-0-000	160458330	$887.46	11/1/2016	$887.44	2/1/2017	$1,774.90	Fully Paid
10	0569-291-21-0-000	160458331	$3,293.92	11/1/2016	$3,293.93	2/1/2017	$6,587.85	Fully Paid
11	0569-291-22-0-000	160458332	$270.41	11/1/2016	$270.40	2/1/2017	$540.81	Fully Paid
12	0569-291-25-0-000	160458333	$24.12	11/1/2016	$24.12	2/1/2017	$48.24	Fully Paid
13	0569-301-18-0-000	160458334	$606.98	11/1/2016	$606.98	2/1/2017	$1,213.96	Fully Paid
14	0569-301-22-X-000	160458335	$673.02	11/1/2016	$606.98	2/1/2017	$1,280.00	Fully Paid
15	0569-301-22-X-002	160458336	$175.77	11/1/2016	$175.75	2/1/2017	$351.52	Fully Paid
16	0569-341-01-X-001	160458337	$287.94	11/1/2016	$287.93	2/1/2017	$575.87	Fully Paid
17	0569-351-01-X-001	160458338	$49.54	11/1/2016	$49.54	2/1/2017	$99.08	Fully Paid
17			**$7,153.47**		**$7,087.33**		**$14,240.80**	**Fully Paid**

4.3 Title Report

On July 13, 2017, Gresham, Savage, Nolan & Tilden LLP (Gresham Savage) of San Bernardino, California, prepared an updated title report (Gresham Savage, 2017), 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 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 derived from G.I.S. Land Services (2015).

Table 4-3 – Distribution of Outstanding Net Smelter Royalties (NSR)

Claim/Patent	NSR (%)	Owner of NSR
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

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 all of the existing Project and extends 10 miles from the boundary of the Project. The new royalty does not require any advanced minimum royalty payments.

4.5 Environmental Liabilities

KCA is not aware of any environmental liabilities on the Project.

4.6 Permitting

In July 2013, Castle Mountain was granted a five-year extension to its Mining Conditional Use Permit and Reclamation Plan No. 90M-013 (the "Permit") by San Bernardino County, which is now scheduled to expire in 2025. The Permit allows for open-pit mining up to nine million short tons of mineralized material per year with no pit backfill requirements. Equinox Gold currently maintains the rights to 10 water wells, three of which are operational.

A more comprehensive description of the Existing Permits and the Permitting Process is provided in Section 20.0.

4.7 Other

KCA is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the Project.

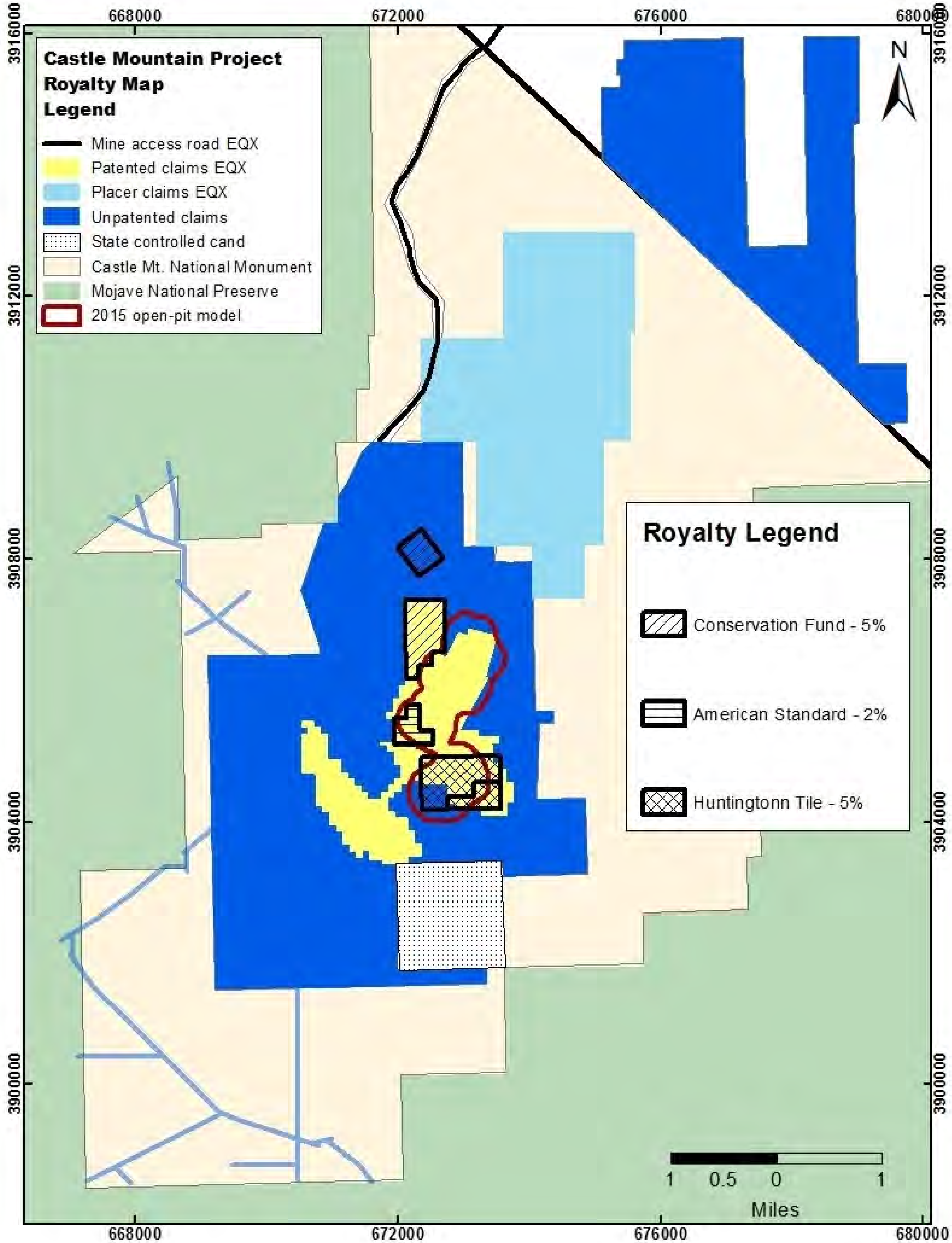

Figure 4-3 – Distribution of Outstanding Net Smelter Royalties

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Project is accessed by travelling south from Las Vegas, Nevada along highway US-95 South for approximately 55 mi (99.8 km) to Searchlight, Nevada. Bearing west for approximately five mi (8 km) along Nevada State Route 164 (Nipton Road), the unpaved Walking Box Ranch Road is intersected. The Project is located approximately 18 mi (29 km) southwest of this intersection along the Walking Box Ranch Road (Figure 5.1).

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

5.2 Infrastructure

Infrastructure from the previous mining operation has largely been removed, but evidence of the past mining remains in the form of the Oro Belle and Jumbo open pits, the back-filled JSLA open pit, the West and South waste dumps, reclaimed heap leach pad, secondary road access, several water wells, a 250,000-gallon (950,000 litre) water holding tank and a portion of a water system. All other areas have undergone full reclamation. An aerial photograph showing the sites of remaining infrastructure from the former operation is presented in Figure 5.2.

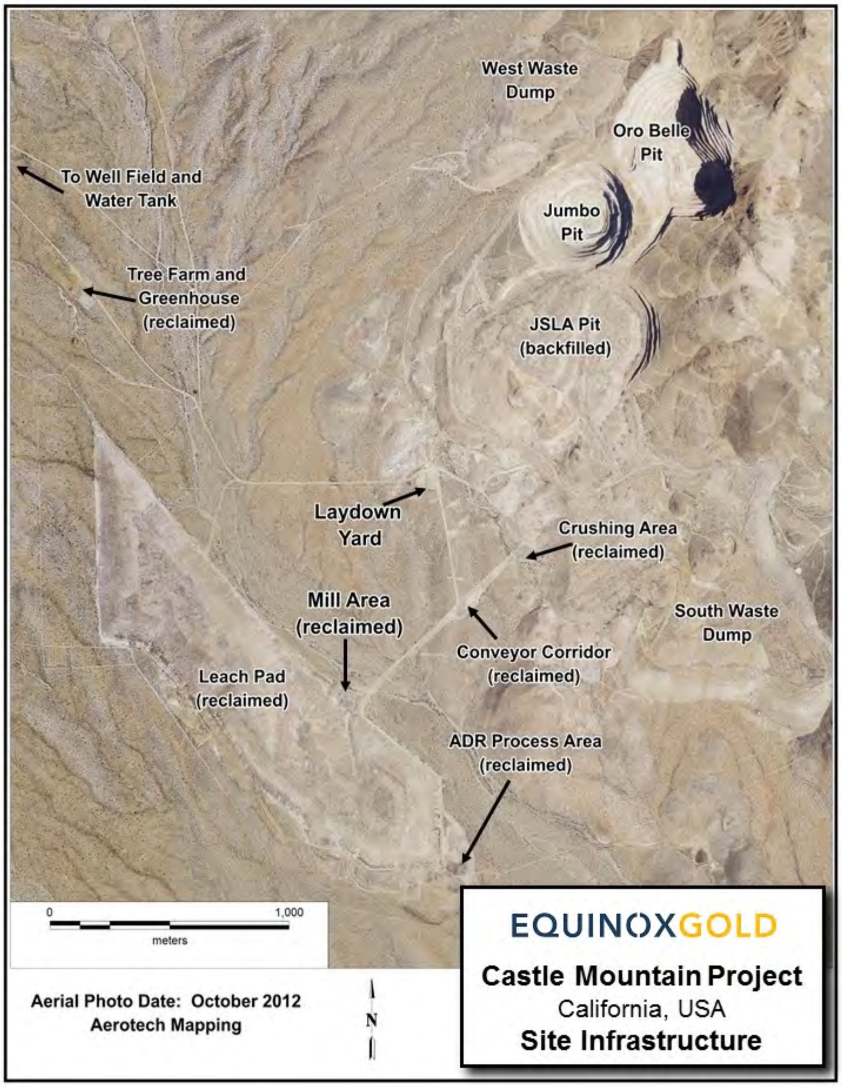

Figure 5-2 – Site Infrastructure Map, Castle Mountain Project

The history of work in the area demonstrates that the Project can accommodate mining operations, waste rock disposal, heap leach pads and processing plant sites. Power and water have previously been established on the site and could be re-established as needed. Equinox Gold currently maintains the rights to 10 water wells, three of which are operational, including the water holding tank.

Nearby southern Nevada has a well-regarded pool of human resources for mining and the Project enjoys all-season access through an established network of roads and close proximity to grid power.

Equinox Gold maintains a temporary office trailer and logging facility at the laydown yard.

5.3 Physiography

California's portion of the Mojave Desert can be divided into three subregions with indistinct boundaries. The western Mojave subregion consists of the triangular Antelope Valley and the central Mojave subregion roughly coincides with the Mojave River Valley. 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 subregions, but differences in elevations are not as pronounced as those seen in Basin and Range terrains (Michaelsen, 2013).

The region hosts the Clark, New York and Providence mountain ranges and many peaks exceed 6,560 fasl (1,999 m) in elevation (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 two to three 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 m), and elevations at the Project site range from about 4,100 fasl (1,250 m) to 5,100 fasl (1,554.5 m).

Topographic relief in the Castle Mountains range averages approximately 1,000 ft. (304.8 m) above the adjacent Lanfair Valley floor elevation of approximately 4,100 fasl (1,250 m) (Temkin, 2012). Evidence of volcanic activity is found in landforms associated with crustal thinning most notably in the Cima Dome area (Michaelsen, 2013). Volcanic plugs and domes form the high relief features.

Hart Peak (5,543 fasl) (1,689.5m), Linder Peak (5,543 fasl) (1,689.5m) and Egg Hill (5,077 fasl) (1,547.5 m) are prominent features. The latter is located at the southern end of the range, nearby the currently known gold mineralization (Temkin, 2012).

Sand, from the Mojave River area, has been carried east by strong winds and deposited against the Providence mountains forming large dunes (Michaelsen, 2013).

Vegetation and wildlife are typical of the Mojave Desert. Many cactus species (cholla and barrel) are found along with woodlands of Joshua trees, blackbrush scrub, creosote bush scrub and desert grasslands (Figure 5.3).

Figure 5-3 – View west of reclaimed heap leach area, laydown yard (center of picture), and Joshua trees and scrub brush in the foreground.

Common mammals found in the area are coyote, jack rabbit, desert woodrat, bighorn sheep and mice. Reptiles include lizards, snakes and the protected desert tortoise which occurs locally in limited numbers below 4,500 fasl (1,372m). The lethal Mojave Green rattlesnake inhabits the area along with various hawk and owl species.

5.4 Local Resources

The nearest primary supply center for goods and services is Henderson, Nevada approximately 70 mi (112.7 km) by road to the north-northeast. Henderson is a center for hospitals and schools and has a history of industrial manufacturing. Based on the 2010 US census, Henderson's estimated population is about 257,000.

McCarran International Airport is located approximately 80 mi (128.7 km) to the north-northeast of the Project in Paradise, Nevada, about seven mi (11.3 km) south of Las Vegas. It is the principal commercial airport in the region and comprises three terminals which service a number of domestic and international carriers.

5.5 Climate

No long-term weather data are available for the immediate site, but the climate is typical of the arid eastern Mojave Desert area. Most of the precipitation is the result of localized thunderstorms between July and September and infrequent cyclonic storms from December to March.

Nearby Searchlight, Nevada, located approximately 15 mi (24.1 km) northeast, reports precipitation of about eight inches (200 mm) per year. Searchlight has an elevation of 3,445 fasl (1.050 m), which is lower in elevation than the Project. Therefore, slightly higher rates of precipitation may occur at the Project. Precipitation is primarily in the form of rain but occasional snowfalls occur. Snow typically melts within days.

Seasonal temperatures can range from approximately 32°F (0°C) to 100°F (38°C). The area is subject to gusty winds that help moderate high temperatures in summer but cause wind chill conditions in winter. In the Author’s opinion, climate does not have the potential to materially impact any exploration work or future production activities. The site can be accessed by gravel road year-round.

6.0 HISTORY

Gold mining began in the Hart Mining District in 1907. Recent exploration was conducted in the area more or less continuously since the late 1960's. A brief summary for each era is provided below. Further details are presented in Temkin (2012), Pressacco (2013) and Gray et al. (2016).

6.1 Historical Mining

6.1.1 Historical Gold Mining

In 1907, three underground mining operations were brought into production: Oro Belle, Big Chief and Jumbo. Operations wound down 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.

6.1.2 Historical Clay Mining

In the 1920s, development began on the clay alteration zones associated with gold deposits. Quarrying for clay started in the area in the 1930's. Clay production was reported to have exceeded 200,000 st by 1957.

6.2 Modern Gold Exploration

Modern exploration in the Hart Mining District began in 1968 and carried on more or less continuously through to the early 2000's. A synopsis of the work carried out is presented below (from Cox et al. 2014) and with areas highlighted in Figure 6.1.

Vanderbilt Gold Corporation

1968: Sampled historical mine dumps and underground workings.

1979: Acquired the Oro Belle patents.

1980: Staked nine lode claims covering the Southern Belle 1 deposit. Completed a 28-hole conventional rotary (CR) drilling program. Collected a 1,980 st bulk sample for vat-leach testing.

Freeport Mineral Ventures

1980-81: Staked 352 "MYO" lode claims. Conducted regional-scale geological mapping and grid-style rock chip and geochemical sampling.

1982-84: Drilled 26 CR drill holes. Exploration ceased, claims allowed to lapse.

B&B Mining and Vanderbilt Gold Corporation

1981: Acquired lode claims adjoining the Vanderbilt Gold Corporation (Vanderbilt) land holdings. Completed four CR drill holes.

1983: Signed a Joint Venture Agreement (JVA) with Vanderbilt and completed geological mapping. Vanderbilt completed 159 CR drill holes for a total of 21,058 ft. Other work during 1981-1984 included:

Surface

- Geologic mapping at a scale of 1 in. = 200 ft.
- Rock chip sampling
- Grid pattern soil mercury survey
- Magnetometer and very low frequency electromagnetic (VLF-EM) geophysical surveys

Underground

- Rehabilitation of underground workings
- Geologic mapping at a scale of 1 in. = 20 ft.
- Rock chip sampling of approximately 3,650 ft. of drifts and crosscuts

1984: B&B Mining amalgamated with Viceroy Petroleum Ltd. to form Viceroy Gold Corporation (Viceroy) and became the U.S. subsidiary of Viceroy Resources Ltd. (Viceroy Resources). In late 1984, Viceroy became the majority partner and operator as Vanderbilt's interest was reduced to below 10%. Eventually, Vanderbilt ceased involvement in the JVA.

Viceroy Gold Corporation

1984: Became operator of the project.

- Geologic mapping at scales of 1 in. =100 ft., 1 in. =200 ft., 1 in. =500 ft.
- Stream sediment and grid-pattern soil sampling
- Rock chip, channel and panel sampling
- Geophysical (Induced Polarization (IP)/Resistivity and magnetic) surveys
- Biogeochemical survey
- Completion of 18 CR drill holes

1986: Detailed geological mapping and drilling of 116 CR drill holes. By the third quarter of 1986, Viceroy had acquired 100% of the Project and progressed toward a Feasibility Study (FS) on the Oro Belle, Jumbo South and Leslie Ann mineralized bodies.

1987: The 1987 Viceroy Feasibility Study (FS) (pre-NI 43-101), conducted by Holt Engineering Ltd. (Holt) of North Vancouver, British Columbia, was prepared concurrent with the exploration work. The FS was based on conceptual guidelines that focused on early production using heap leach methods of extraction. The merits of milling the potential mineralized material was not within the FS's scope of work.

The FS proposed mineralized material extraction by means of open pit with a two-stage crushing circuit followed by heap leach, carbon-in-column gold recovery, electrowinning, and on-site smelting to produce gold bullion. An initial production rate of 5,000 stpd was proposed by Holt with production increase to 8,000 stpd later in the mine's life. Holt concluded that metallurgical studies done at the time indicated that the mineralized material was amenable to heap leaching for gold (Holt, 1987).

Holt concluded that the CMV, as proposed, was economically and technically feasible, so Viceroy sought a major financing with Hemlo Gold Corporation (Hemlo Gold) in 1987. Under terms of the agreement, Viceroy retained 100% interest in the central area while Hemlo Gold acquired a 50% interest in the exploration rights for the remainder of the Castle Mountains. Permitting for mining production began in 1987 and, in 1990, a favorable decision was granted to Viceroy by the BLM.

Concurrent to the permitting process, exploration, condemnation and development drilling increased. In 1987, 189 holes were drilled and in 1988 another 231 holes were drilled over 25 target areas. These holes were generally drilled using a combination of CR and reverse circulation (RC) methods. The majority were initially CR with follow-up holes utilizing RC. An additional ten core holes tested areas where significant gold mineralization was intersected in the RC drilling. The South Extension, Jumbo and Hart Tunnel mineralized bodies were found using these methods. The South Dome mineralized body was discovered during a condemnation drilling program designed to sterilize an area proposed for a waste dump facility.

Due to the time required for the permitting process, Viceroy ceased drilling for a time and focused its efforts on special studies of the gold mineralization including microscopy, petrology and geochemistry.

Hemlo Gold Corporation/Noranda Exploration Company Ltd.

1987-90: Began exploring in the Castle Mountains in 1987. Early work included regional-scale geological mapping and rock chip sampling. Other, more focused work done by Noranda included IP, biogeochemical and microbial surveys over the Northwest Rim target area. These later surveys generated targets which were tested by a number of RC holes but no significant gold mineralization was intersected. Noranda dropped its interests in the area in 1990.

Viceroy Gold Corporation

1990: When drilling resumed in 1990, 16 RC condemnation and exploration holes were drilled in eight separate areas. This drilling resulted in the discovery of gold mineralization in the North Oro Belle area.

Viceroy Gold Corporation and MK Gold Company

1991: In early 1991, MK Gold Company (MK Gold) purchased a 25% interest in the mine and became the contract mining operator. Later that year, mine construction began on the JSLA deposits with the plan to exploit both deposits with one open pit. Commercial production was started at the mine that same year as exploration work continued.

In June 1991, drilling resumed on the Oro Belle and Jumbo deposits in addition to condemnation drilling on other parts of the Project. The Lucky John high-grade mineralized zone was discovered in what was proposed to be the South Clay Pit waste area and the deep, well-mineralized 621 Zone was found in the proposed South Waste Dump area. The persistent discovery of new mineralized areas as the mine was closing in on production necessitated further exploration and condemnation drilling. These discoveries also demonstrated the Project's potential for hosting narrow high-grade zones at depth which could potentially be exploited by underground mining in addition to substantial mineralized shallow zones.

1992-93: Exploration, development and condemnation drilling from 1992 to 1993 totaled 263 RC holes in 15 target areas. The emphasis was on development drilling at the Jumbo and North Oro Belle deposits and exploration/condemnation drilling at South Domes. Condemnation drilling continued at the South Waste Dump area and led to the discovery of the Southeast Egg mineralized zone. Additional drilling outside of this mineralized area, on 400 ft. centers, encountered little mineralization and this area was ultimately designated to be used for waste rock disposal.

1993-94: 252 RC holes were drilled in 19 areas for exploration, development and condemnation. Development drilling was concentrated on the Hart Tunnel and Oro Belle mineralized bodies. Exploration drilling was carried out at Egg Dome, Lucky John and South Extension zones.

1994-95: Significant gold intersections were encountered at Hart Tunnel, Oro Belle and Mountain Top during the RC drilling programs that were conducted between 1994 and 1995.

1996: Viceroy contracted Intermountain Mine Services (IMS) of Salt Lake City, Utah in 1996 to design an underground exploration program that would further define the mineralization found in the surface drilling. Once defined, IMS was to formulate a plan to develop and mine the mineralized material. The underground exploration program called for drilling BQ- diameter (36.5 mm) core holes through the mineralized zones at 50 ft. centers to further define the boundaries and overall grade of mineralization (Intermountain, 1996).

6.3 Modern Gold Mining

Viceroy Gold Corporation/MK Gold Corporation commenced gold production on the Project in 1991 (Figure 6.2) and the JSLA deposits were 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 (>0.1 opt) 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.

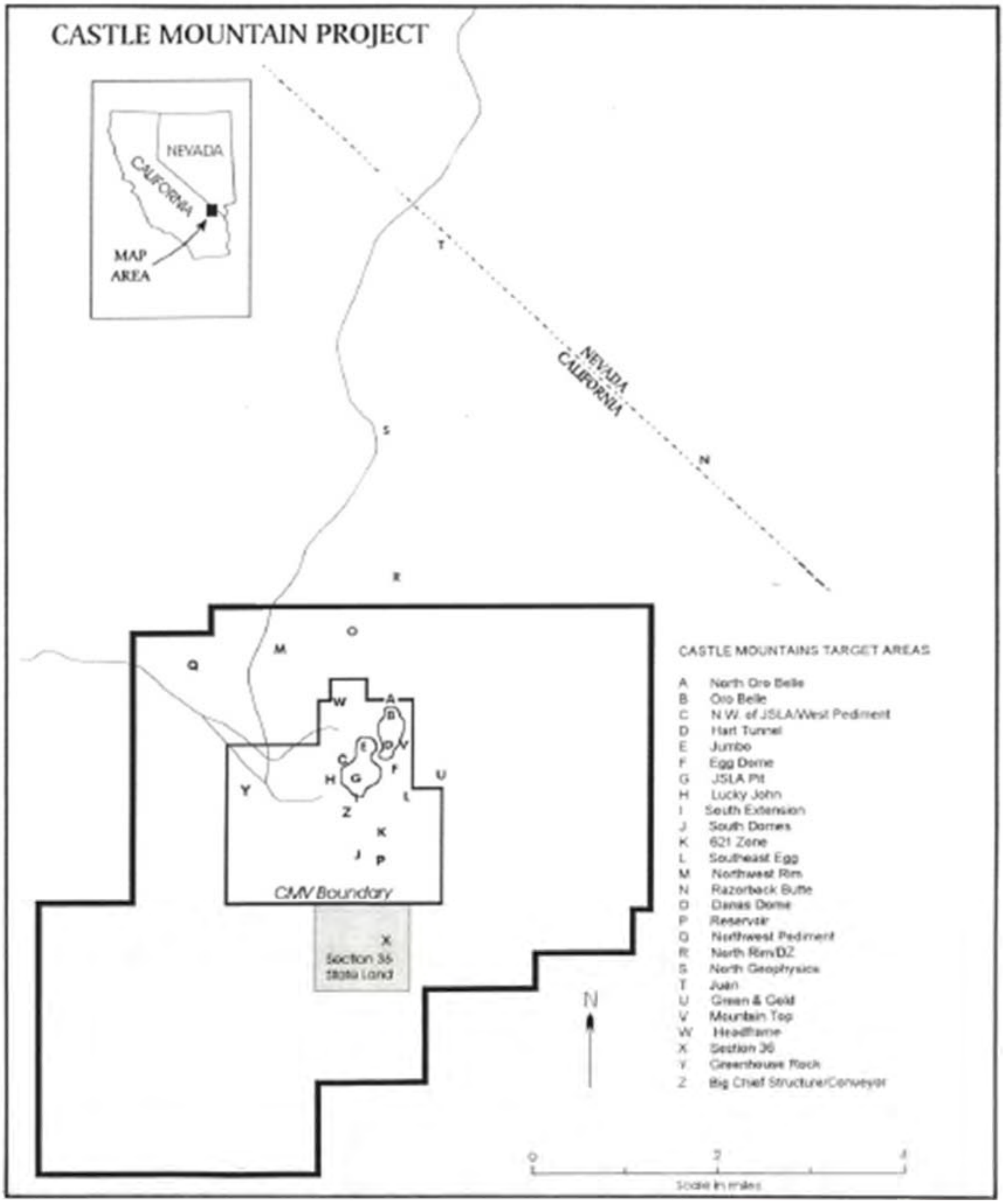

Figure 6-1 – Location Map (dated 2002) of Target Areas described in Section 6.2. **Land boundaries shown on this figure are historic and do not match the current Project (from Temkin, 2012)**

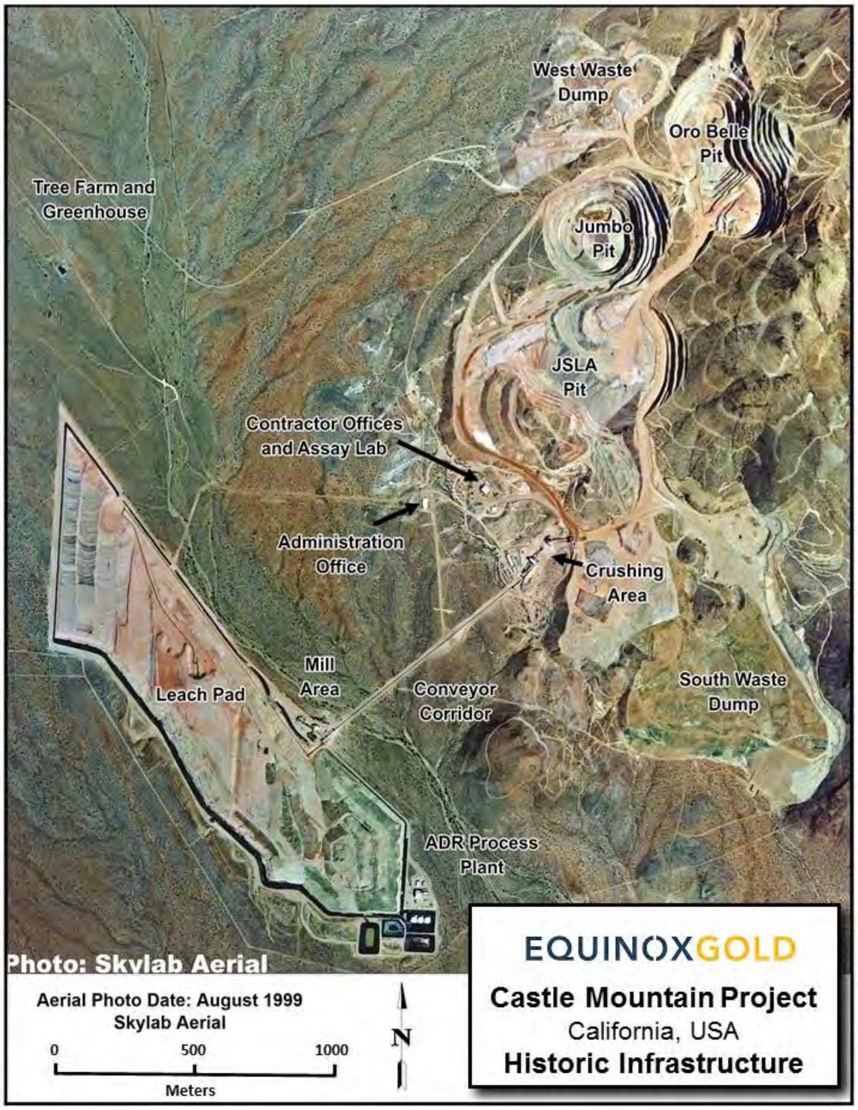

Figure 6-2 – Historic Mining and Processing Infrastructure

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 ~ 116,120 oz of gold, or 12% of the total leach production, recovered after the closure of the pit.

Total gold production from all deposits was in excess of 1.24 Moz with an approximate silver production of 400,000 oz (Table 6.1).

Table 6-1 – Past Production From 1991 to 2004

Activity	Tonnage (000 st)	Grade (oz/st Au)	Contained Ounces Au (000s)	Recovered Ounces Au (000s)	Recovery (%)
Ore Mined	37,683	0.040	1.52		
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. *Some columns may not add up due to rounding.*

6.3.1 2015 Mineral Resource Estimate

An Updated Mineral Resource estimate was prepared and released by NewCastle on December 2, 2015 (Gray et al., 2016). The estimate utilized a 0.20 gram per tonne gold cutoff within an optimized pit shell calculated at a gold price of $1,100 per ounce. Highlights of the Updated Mineral Resource are as follows:

- Measured Mineral Resources of 17.4 million tonnes grading 0.86 g/tonne gold and containing 0.48 million gold ounces.
- Indicated Mineral Resource of 202.5 million tonnes grading 0.57 g/tonne gold and containing 3.711 million gold ounces.
- Inferred Mineral Resources of 40.8 million tonnes grading 0.58 g/tonne gold and containing 0.76 million gold ounces.
- Strip ratio at 0.20 g/tonne cut off is 2.8:1 versus 3.4:1 (at 0.14 g/tonne cutoff) in the prior mineral resource estimate, notwithstanding the higher cutoff applied in the Updated Mineral Resource.

Site investigation, database validation and construction of the geological model was carried out by R. Bob Singh, P.Geo. of North Face Software Ltd. The geological and mineralization model formed the basis for grade estimation that was completed by James N. Gray, P.Geo. of Advantage

Geoservices Limited. The Updated Mineral Resource was based on the results from 352,090 meters of drilling in 1,683 holes (1,637 RC and 46 core). Assays were reviewed statistically by interpreted geologic domain to establish appropriate grade capping levels. Capped grades were composited to a length of three meters within interpreted units. In addition to the capping of assay data, the impact of anomalously high gold values was further controlled by restricting their range of influence in the estimation process. Block grades were estimated by ordinary kriging. Blocks measure 10 x 10 x 10 meters. Tonnage estimates are based on 335 density measurements which were used to assign average values to lithologic units of the block model. Bulk density for the mineralized domains averages 2.20 tonnes per cubic meter.

Table 6.2 summarizes the estimate at a range of cutoff grades.

Blocks classified as Measured Mineral Resources are within 20 meters of at least three holes and the closest is within 10 meters or within 30 meters of three holes and the closest is within 7.5 meters. Indicated blocks have a maximum average distance to three holes of 50 meters and the closest hole is within 10 meters or at least one hole in four octants of a 75 meters spherical search. Inferred Mineral Resource blocks require three holes within 180 meters.

In order to establish reasonable prospects for eventual economic extraction in an open pit context, the mineral resources were defined within an optimized pit shell using a gold price of $1,100 per ounce, pit wall slopes of 48°, estimated gold recovery of 80%, mining costs estimated at $1.42/tonne, processing costs estimated at $4.79/tonne and general and administrative costs estimated at $0.73/tonne. The base case cutoff grade used is 0.20 g/tonne gold. The assumptions listed produced a strip ratio of 2.8:1.

Table 6-2 – Castle Mountain 2015 Mineral Resource Estimate (Gray et al., 2016)

Cut-off (Au g/t)	Measured Tonnes (millions)	Measured Grade Au (g/t)	Measured Ounces Au (millions)	Indicated Tonnes (millions)	Indicated Grade Au (g/t)	Indicated Ounces Au (millions)
0.41	10.1	1.26	0.409	89.3	0.93	2.671
0.34	12.0	1.12	0.432	114.8	0.81	2.991
0.26	14.8	0.97	0.462	157.9	0.67	3.401
0.20	17.4	0.86	0.480	202.5	0.57	3.711
0.17	18.8	0.81	0.489	230.6	0.52	3.855
0.14	20.2	0.76	0.495	263.0	0.48	4.059

Cut-off (Au g/t)	**Measured + Indicated Tonnes (millions)**	**Measured + Indicated Grade Au (g/t)**	**Measured + Indicated Ounces Au (millions)**	Inferred Tonnes (millions)	Inferred Grade Au (g/t)	Inferred Ounces Au (millions)	Strip Ratio
0.41	**99.4**	**0.96**	**3.080**	16.8	1.01	0.545	7.60 : 1
0.34	**126.8**	**0.84**	**3.423**	20.9	0.89	0.599	5.76 : 1
0.26	**172.7**	**0.70**	**3.863**	29.7	0.71	0.678	3.94 : 1
0.20	**219.9**	**0.59**	**4.191**	40.8	0.58	0.760	2.83 : 1
0.17	**249.4**	**0.54**	**4.344**	48.5	0.52	0.811	2.35 : 1
0.14	**283.2**	**0.50**	**4.554**	57.9	0.46	0.856	1.93 : 1

Notes:
1. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. The Mineral Resources in this news release were estimated using current Canadian Institute of Mining, Metallurgy and Petroleum (CIM, 2014) standards, definitions and guidelines.
3. Numbers may not sum due to rounding.
4. Mineral Resources are stated at a cutoff of 0.20 g/t gold.
5. Mineral Resources are contained within an optimized pit shell generated at a gold price of $1,100 per ounce.
6. Capping of high-grade as discussed below.
7. Mineral Resource estimate by James N. Gray P.Geo. of Advantage Geoservices Limited.

6.4 Historical Preliminary Economic Assessment

Following the new Mineral Resource estimate in 2013, NewCastle commissioned RPA to complete a PEA (Cox at al. 2014). The preliminary economic analysis contained in the 2014 PEA was based on the 2013 Mineral Resource Estimate. This information is now considered to be out of date due to the updated 2015 and 2017 Mineral Resource Estimates such that it can no longer be relied upon.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

The Castle Mountains gold deposit is in the Hart Mining District (Tucker and Sampson, 1943; Wright, 1953; Hewett, 1956; Linder, 1989), at an elevation of ~4500 feet (1372 m) in the southern portion of the Castle Mountains Range, eastern San Bernardino County, California.

The Castle Mountains Range are in the eastern Mojave Desert within the southern Basin and Range Province (Theodore, 2007). Castle Mountain rocks form a small range of Miocene volcanic rocks at the northern end of the Lanfair Valley in eastern San Bernardino Co., California, and extend north into Nevada (Figure 7.1). Tectonically, the Castle Mountains are located along the north-western margin of the Colorado River extensional corridor, a regional tectonic feature (Howard et al., 1994). Extensional tectonism in the Colorado River corridor is represented by mid-Tertiary detachment faulting in Nevada, California and Arizona that extend from south of the Las Vegas-Lake Mead shear zone, to just north of the San Andreas fault (Ausburn, 1991, 1995). Miocene volcanism and structural trends of Miocene rocks and faults in the region have been attributed to continental extension. While no regional low-angle normal (detachment) faults crop out in the Castle Mountains (or in the Piute Range to the south), detachment faults are exposed to the west in the Kingston Range and to the east in the Black Mountains of Arizona (Howard et al., 1994). Accordingly, the Castle Mountains Range is interpreted by Ausburn (1991), to be located on the western margin of documented detachment faulting along this segment of the Colorado River corridor. Structures in the Castle Mountains are temporally and spatially consistent with crustal extension in the region, but on a much smaller scale as compared to the highly extended Colorado River extensional terrane to the east (Linder, 1989; Capps and Moore, 1991, 1997; Nielson et al., 1999; Spencer, 1985).

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).

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 recognized volcanic unit above the Proterozoic basement is the regionally extensive Peach Springs Tuff (Young and Brennan, 1974; Glazner et al., 1986; Nielson et al., 1990; Buesch, 1992), dated at ~18.8 Ma (Ferguson et al., 2013). Overlying the Peach Springs Tuff is a sequence of andesitic to basaltic lavas and associated volcaniclastic rocks termed the Jack's Well Formation. These are in turn overlain by the intrusive and extrusive rhyolite domes, flows, associated breccias and tuff, of both pyroclastic and epiclastic depositional emplacement, termed the Linder Peak and Hart Peak Formations. The sequence of rocks overlying the Peach Springs Tuff that includes the Jack's Well, Linder Peak and Hart Peak Formations is herein collectively referred to as the Castle Mountains Volcanic Sequence (CMVS).

Tertiary intrusive rocks (Capps and Moore, 1997) include the following:

1. rhyolite dikes, probably related to Linder Peak rhyolite, occur throughout the Castle Mountains but are most abundant in the northeastern Castle Mountains. Quartz-adularia veins locally cut the dikes;

2. possible pyroclastic dikes and minor sills, one to three feet wide, cut Linder Peak rocks throughout the Castle Mountains. Contained clasts are Proterozoic metamorphic rocks and minor CMVS rocks;

3. fine-grained, unaltered andesite dikes and plugs occur throughout the Castle Mountains, and are probably related to late Hart Peak or Piute Range volcanism; and

4. dark-gray, medium-gray, and dark-brown, coarse- and medium-grained diorite dikes and sills occur in the north-central and northeastern Castle Mountains. The diorites locally intrude low-angle normal faults and are probably related to late Hart Peak volcanism.

Capps and Moore (1997) proposed four episodes of deformation in the Castle Mountains:

1. Proterozoic deformation

2. Mesozoic deformation

3. Miocene dilation associated with growth faults and hypabyssal dike emplacement and

4. cryptic Miocene northwest-striking faulting.

Proterozoic deformation is manifested in a well-developed northwest-striking, northeast-dipping foliation in metamorphic rocks. Other structural events are manifested primarily as north-northeast striking normal faults with dips ranging from low to high angle. Capps and Moore (1997) make note of a cryptic northwest fabric identified through photo lineaments and discontinuities cross-cutting prominent northeast structures.

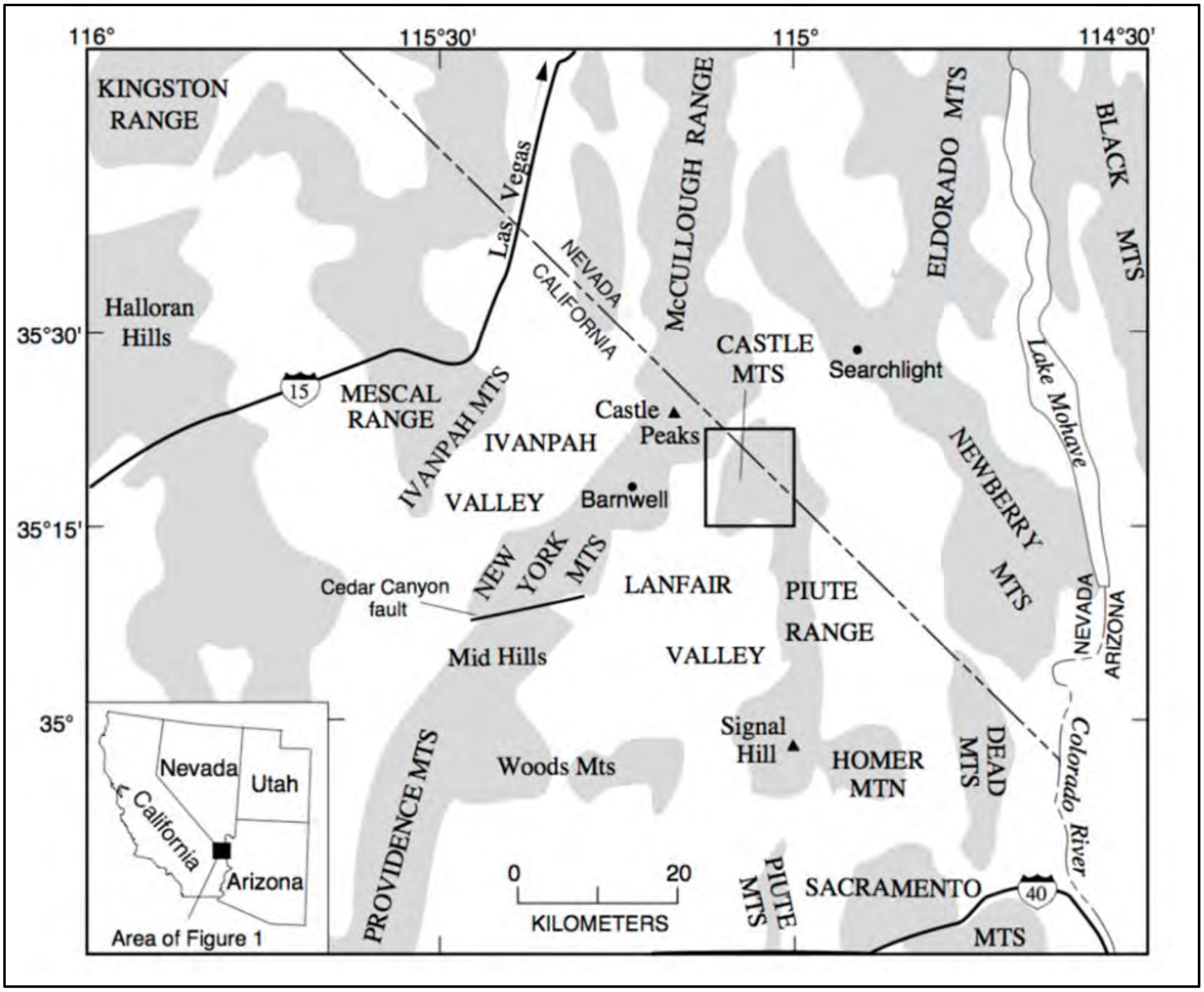

Figure 7-1 – Location map of Castle Mountains in SE California and SW Nevada (Neilson et al., 1999)

7.2 Property Geology

Geologic mapping and stratigraphic correlations were conducted by previous companies that worked in the area. Particularly, work by Viceroy and one of its consulting geologists, R.C. Capps (Capps and Moore 1991, 1994, 1997), helped define the geologic setting of the Castle Mountains.

Metamorphic 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. A massive sequence of biotite schist, biotite gneiss and meta-granite characterize this unit. Only local 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 55 meters thick. Clasts are completely composed of Proterozoic metamorphic rocks, generally range from pebble to cobble size and are rounded to sub-angular. This unit is locally referred to as PC Seds (Figure 7.2). The PC Seds have been recognized in drilling, notably in the northern Oro Belle pit area and underneath the JSLA pit.

Unconformably overlying the PC Seds is the regionally extensive Peach Springs Tuff unit (~18.8 Ma; Nielson et al., 1990; Nielson et al., 1999; Ferguson et al., 2013). The Miocene age Castle Mountains Volcanic Sequence (CMVS), by definition, includes all volcanic units above the Peach Springs Tuff (18.8 Ma) and below the Piute Range volcanic rocks (Figure 7.2; ~13.5 Ma, Nielson and Nakata, 1993; Nielson et al., 1999). The CMVS consists primarily of rhyolitic domes, flows, and felsic tuff, and lesser andesitic, latitic, and basaltic lava emplaced during three intrusive-extrusive episodes between ~18.8 and ~13.5 Ma.

At the Castle Mountain property, the CMVS is generally divided into three informal units.

1. The lower-most unit include trachyandesite to basaltic-andesite flows, minor rhyolite ash-flow tuff locally displaying accretionary lapilli textures, and locally abundant debris flow and epiclastic deposits of the Jacks Well Fm. Capps and Moore (1997) indicate ages of 16.5±0.5 Ma from biotite in trachyandesite, and 15.20±0.03 Ma from andesine from Jack's Well Fm. rocks.

2. These are overlain by porphyritic and aphyric rhyolite flow-dome complexes including both extrusive and intrusive domes and plugs (age range 14.9-16.5 Ma; Capps and Moore, 1997), abundant pyroclastic-surge tuff, and volcaniclastic rock of the Linder Peak Fm.

3. On the Project these are intruded by minor trachyandesite and trachydacite intrusions. Elsewhere, porphyritic rhyolite flows, plugs, welded ash-flow tuff; pyroclastic-surge tuff; and volcaniclastic rocks make up the Hart Peak Fm. (age range 13.8-16.3 Ma; Capps and Moore, 1997).

CMVS rocks are the primary host of epithermal gold mineralization at the Castle Mountain Mine. The oldest volcanic sequence that overlies the CMVS is the Piute Range (age range 13-14 Ma; Nielson and Nakata, 1993; Capps and Moore, 1997), which consists of flows and lahars of intermediate to mafic composition. An idealized, schematic stratigraphic column of the rocks hosting the Castle Mountain gold deposit is presented in Figure 7.2.

7.3 NewCastle Mapping

Beginning in August 2016, a comprehensive mapping, sampling, petrographic and lithogeochemical program was initiated by NewCastle. The approach employed was one based on lithological or lithofacies mapping, testing the hypothesis that mineralization at Castle Mountain is strongly controlled by the primary and/or secondary (structurally-induced) textural characteristics of host rocks (porosity, permeability, etc.), and not specific to any one volcanic unit or series of units. Primary and secondary lithochemical and compositional/mineralogical variations with the aim of developing potential exploration vectors was addressed by a comprehensive lithochemical and petrographic assessment of the host CMVS rocks. This work was reported in Barrett (2016a, b) and Monecke (2017). The most recent product of the mapping effort is presented in Figure 7.3.

7.3.1 Rhyolite Facies

Rhyolite rock types dominate the surface exposures, and where subsurface geology is exposed in the pits and in drill core it is clear that rhyolites are the most volumetrically abundant rock type at the Castle Mountain project area, making up ~65% of the total package. Rhyolite facies rocks occur as a complex package of coherent and monolithic breccia facies, vitrophyre and rhyolite dikes (Figure 7.4). Individual rhyolite bodies are identified using contact relationships with geometries that are vertically continuous and laterally restricted. Subtle compositional and textural changes exist between individual bodies; however, no single body is entirely unique in that they are all variations of the same general rhyolite. Subtypes include quartz porphyritic, quartz-feldspar porphyric, and aphyric rhyolites with an array of textures ranging from flow-banded, massive, vitrophyric/perlitic, spherulitic, and vesicular. The geochemical results show there is little to no chemical distinction between the many individual rhyolite bodies found in the Castle Mountain project area which emphasize the point of a single magma source (Barrett, 2016a).

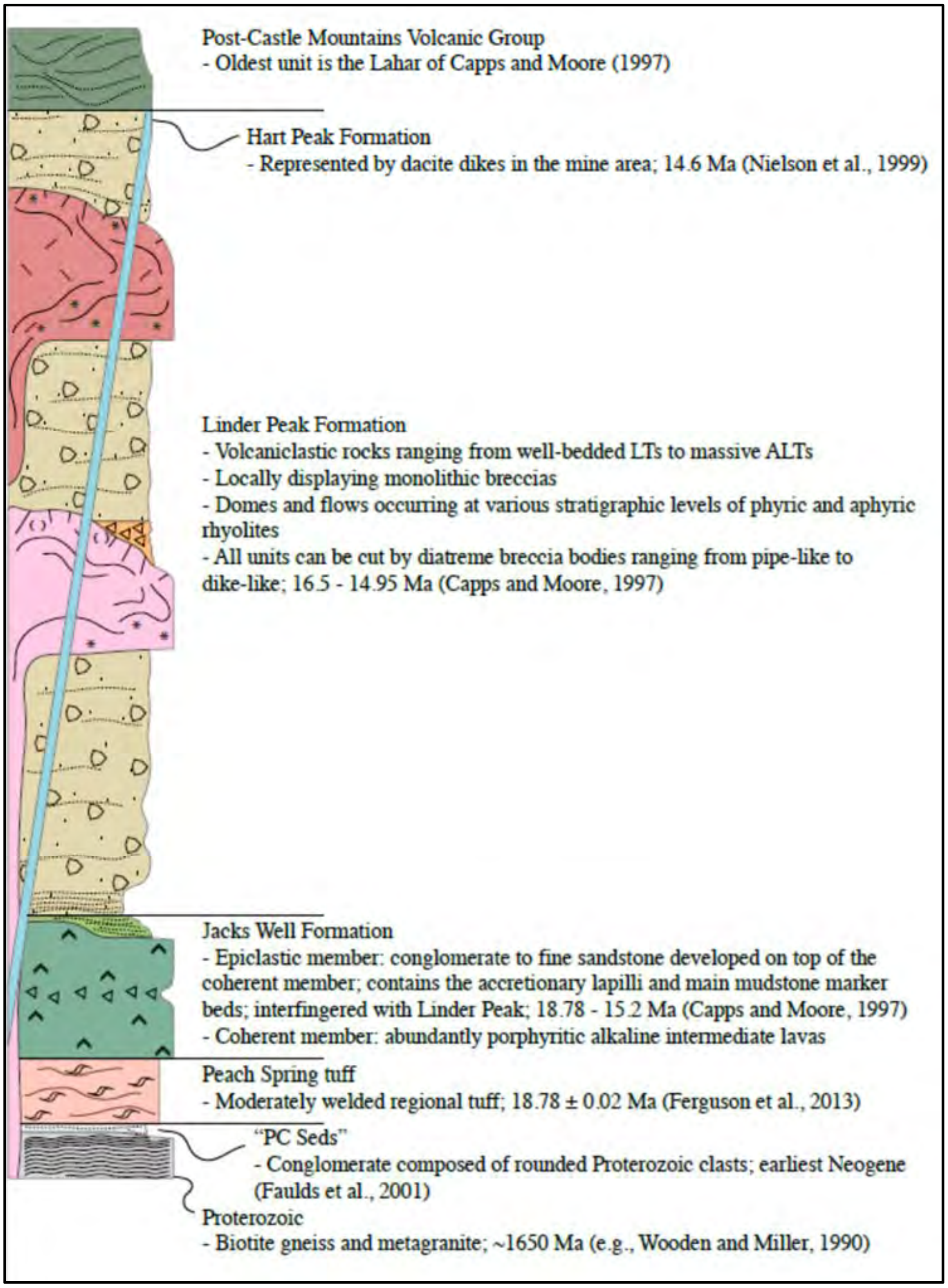

Figure 7-2 – Schematic stratigraphic section for the Castle Mountain property. Modified from Tharalson, 2017.

7.3.2 Volcaniclastic Facies

Volcaniclastic facies are the second most abundant rock types found on the Castle Mountain project and account for 30% of the total package of rocks. In contrast to the coherent rhyolites 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). Continued mapping has refined the basic stratigraphy to include a block-and-ash flow tuff unit, finer-grained stratified lithic and lapilli tuffs (termed LT), as well as both stratified and massive polymictic agglomerate tuffs (termed ALT; Figure 7.5). On a property-wide 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.

7.3.3 Diatreme Facies

Diatreme breccia facies are exposed through the central portion of East Ridge as well as in the JSLA, Oro Belle and Jumbo pits. Identification of diatreme facies at surface is difficult due to their limited surface expression as a result of their vertically continuous and laterally restricted geometry. Additionally, diatreme composition and textures closely resemble ALT, further complicating consistent 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 may also contain pumice fragments, but strong alteration typically obscures much of the original fine textures. Clasts sizes range from <2mm to over 2m in size. The matrix is clay-altered ash with zones of very fine glass shards. There is no internal structure and clast orientations are completely random (Figure 7.6).

Individual diatreme bodies range from <1m to >100m 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.

7.3.4 Trachydacite (Dacite) Dikes

Cross-cutting all felsic lithofacies on the Project, are a series of unaltered, locally columnar-jointed dikes of intermediate composition, termed trachydacite to trachyandesite (Tiha) by Capps and Moore (1997). Geochemically, these dikes are defined as trachydacite due to their high-K content and because they contain ~20% normative quartz based on a CIPQW normative calculation (Barrett, 2016a, b), although historically and colloquially, they have been termed dacite.

Dacite dikes are biotite and feldspar porphyritic with an aphyric groundmass that is locally finely vesicular. Biotite phenocrysts are dark brown to black, <1mm to 3mm in size, and euhedral, making up 2% to 4% by mode. Feldspar phenocrysts are 2mm to 1cm in size, blocky and vitric, and display cleavage. Feldspar phenocrysts make up <1% to 1% of the rock. The groundmass is gray and massive to finely vesicular.

Dacite dikes are sharp and typically chilled against host rocks. Dacite dikes exhibit columnar jointing (Figure 7.7a) oriented perpendicular to the dike contacts that locally can be recognized through the chill margins. Notably, dacite dikes are completely fresh, and thus interpreted to post-date the mineralizing event. Accordingly, dacite dikes are ascribed to the Hart Peak Fm.

At least three distinct dacite dikes have been recognized on the properties. 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 ~045° (Figure 7.7b). A third less prominent dike trends 020° and is discontinuous along the top of East Ridge for ~300 m.

7.3.5 Structure

7.3.5.1. Fault Descriptions

Numerous faults are recognized on the Project and have a significant effect on redefining the original geometry of dome complexes and volcaniclastic stratigraphy. In accordance to the description of faults in Gray et al., (2016), the major structural trend is oriented north-northeast with east-side-down movement. A secondary east-west structural trend is also observed. Offset along the north-northeast faults range from <5m to >100m and the cumulative offset of the entire structural corridor may be significant. Offset along east-west faults is less well constrained owning to relative infrequency, discontinuity and/or truncation, and poor surface exposure.

The north-northeast faults trend between 350° to 040°. These faults generally dip 60° to 80° east. According to Gray et al. (2016), the majority of faults on which orientation data was collected, are high-angle ranging from 70°E to vertical. They are marked by zones of cataclasite with or without gouge and very rarely show any slickensides or other kinematic indicators. These faults range in size from <10cm to >4m wide and larger faults can be traced along bedrock surface exposures over the length of the Project and can be linked across areas of overburden.

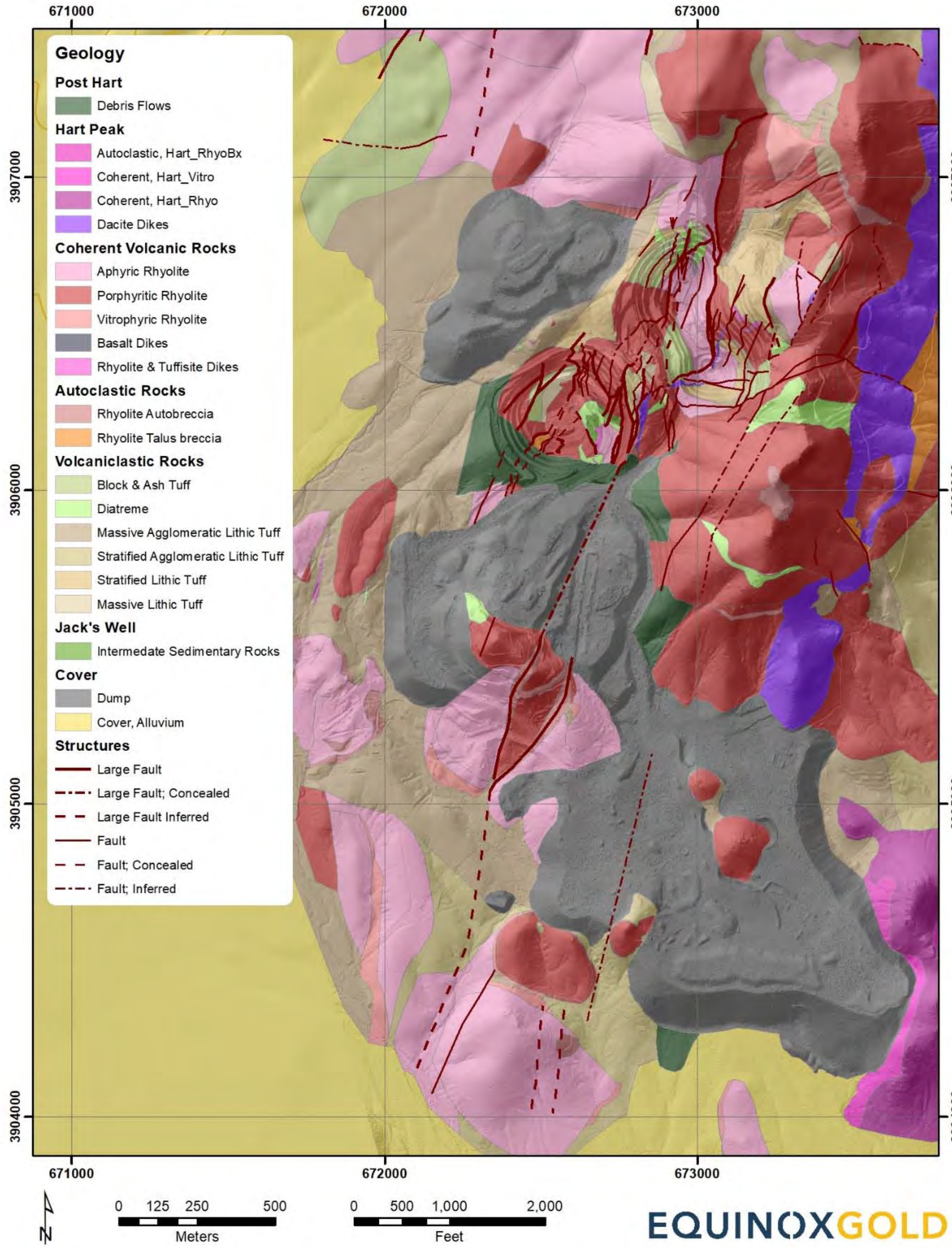

Figure 7-3 – Current version of the geological mapping program (Source: Equinox Gold)

Figure 7-4a – Flow-banded quartz-porphyry rhyolite (QPR), with 1-2 mm quartz phenocrysts in a crystalline groundmass.

Figure 7-4b – Monomict rhyolite breccia consisting of spherulitic QPR fragments in rock flour matrix.

Figure 7-5a – Block-and-ash flow tuff. In the foreground blocks and finer clasts from 2cm to 8cm in size can be seen.

Figure 7-5b – Bedded lithic tuff (LT) with stratified ash matrix and fine-grained lithic clasts <1mm.

Figure 7-6a – Diatreme with 70cm clast of massive QPR and a 10cm clast of andesite. The matrix contains numerous other clasts ranging from 3-6 cm in size.

Figure7-6b – Chaotically distributed clasts in a diatreme with 3-5cm andesite clasts forming a cluster (pencil).

Figure 7-7a – Columnar jointing in dacite dike.

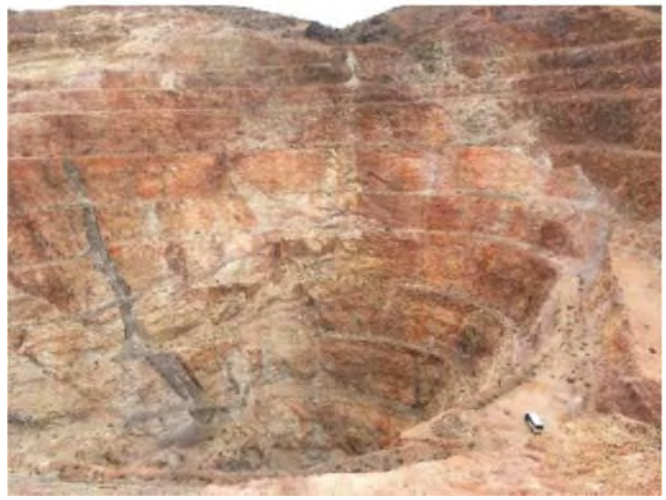

Figure 7-7b – Dacite dike observed cutting through the Oro Belle pit. Note white van in the bottom left corner for scale.

The east-west trending faults trend between 260° to 300°. These faults appear to dip subvertically both north and south. They are marked by cataclasite, and can contain significant amounts of gouge. They show no kinematic indicators. They range in size from <10cm to 1m+ wide. Inferred normal north-side-down movement along the east-west faults have been estimated from stratigraphic offset in drill core and limited exposure in the Oro Belle pit.

7.3.5.2. Fault Timing

Capps and Moore (1997) recognized east-west faults and his map interpretation implies they are pre- to syn- Linder Peak. They are mapped as consistently cutting the Jack Wells andesite and intermediate epiclastic rocks with occasional faults extending into the Linder Peak rocks. The limited exposure of east-west faults mapped by NewCastle geologists indicates that these faults are truncated by the north-northeast faults. The cross-cutting relationships and limited exposure of east-west faults suggest these faults predate the more dominant north-northeast trending faults. Additional evidence for the east-west faults predating the north-northeast 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 volcaniclastics. 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 vertical pathways for diatreme emplacement.

7.3.5.3. Regional Structural Framework

The Castle Mountains lie at the far western edge of the Northern Colorado River Extensional Corridor between the east dipping McCullough Range Fault to the west and the east dipping Newberry Detachment Fault to the east (Faulds et al., 2001). The McCullough Range Fault is the western most detachment fault bounding the Colorado River Extensional Corridor (Spencer, 1985) and the Castle Mountains are part of the brittle deformed upper crustal block in the hanging wall of the McCullough Range Fault (Turner and Glazner, 1990).

The southern portion of the Northern Colorado River Extensional Corridor saw the onset of Early Miocene calc-alkaline volcanism which was accompanied by mild north-south extension due to the collapse of the topographically elevated Kingman arch to the north. North-south extension has been constrained between 20 and ~16 Ma, however the lack of any appreciable north tilting indicates north-south extension was limited (Faulds et al., 2001). By 15.5 Ma, major east-west extension accommodated by major east dipping normal faults and regionally extensive detachments began 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 composition volcanic rocks to more bimodal and felsic compositions. 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. However, felsic volcanic emplacement spanned the entire duration of extension leading to the accumulation of thick, complexly faulted, felsic volcanic piles in half-grabens created during ongoing east-west extension.

7.4 Lithologic Interpretation

The Castle Mountain complex is defined by a succession of complex rhyolite flow-domes and related facies which have intruded into a package of volcaniclastic rocks. Rhyolite flow-domes are also disconformably overlain by similar volcaniclastic rocks. The stratigraphy is transected by north-south and northeast-southwest extensional faulting interpreted as syn- and post-volcanic. Normal faulting focused and facilitated continued both volcanic and later hydrothermal activity. The entire felsic volcanic package is cut by dacite (trachydacite) dikes, of which one is the dominant feature along the eastern margin of the Castle Mountain range and appears to have utilized pre-existing structures for emplacement, as suggested by their orientation and extent of mineralization on/near their margins. A schematic reconstruction of the principal lithostratigraphic elements at Castle Mountain is presented as Figure 7.8 (Tharalson, 2017).

7.5 Mineralization at Castle Mountain

Gold mineralization on the Castle Mountain property occurs in oxidized fractures, faults, discontinuous veins, and breccia matrix; it correlates with iron oxide, but it is not always a reliable indicator as there are many generations of iron oxide. Iron oxide varies in color from pink-red and is fracture controlled and discontinuous in coherent rhyolite facies; to deep red matrix replacement in rhyolite breccias, and wispy selvages and clast haloes in volcaniclastic rocks. In diatremes, iron oxide can be either pervasive or matrix selective and varies in color from a deep red to a brown-red-gray. These iron oxide occurrences can be cut by fracture and vein filling iron oxide that ranges in color from brown-tan to red. Gold appears to correlate best with the deep red, red-brown and brown iron oxide.

Gold mineralization also correlates with moderate to intense silica alteration which includes 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 1m to >10m wide. Amorphous quartz occurs as discontinuous irregular veins and as open space filling quartz. The strongest silica alteration associated with gold mineralization is found along brecciated coherent rhyolite margins; this results in floating angular rhyolite clasts in a hydrothermal silica matrix.

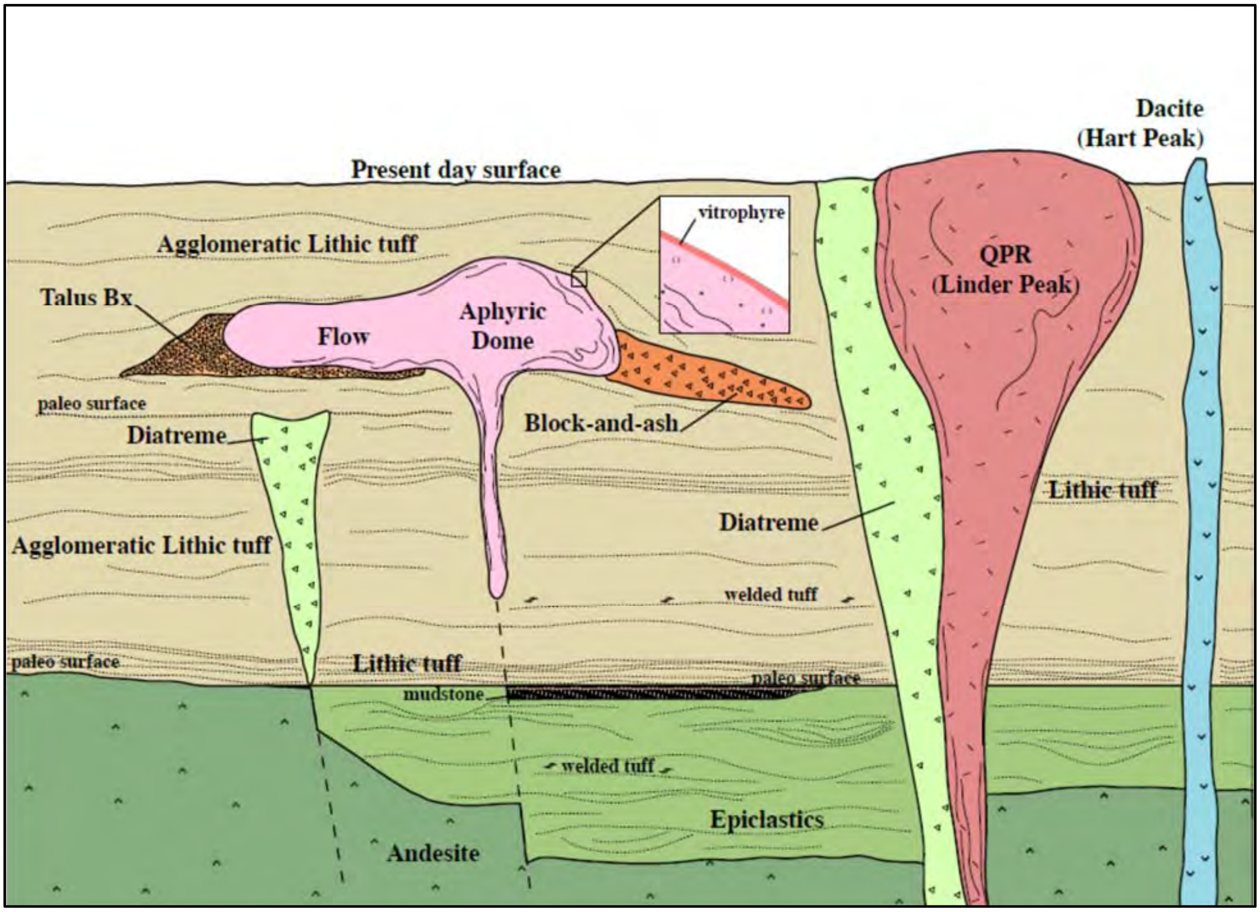

Figure 7-8 – Schematic reconstruction of the principal lithologic and stratigraphic elements - derived from detailed mapping and logging at the Castle Mountain project (not to scale). Modified from Tharalson, 2017.

Silica alteration that contains mineralization typically contains iron oxide also. Like iron oxide, silica alteration is not a reliable indicator, instead it provides a tool to indicate proximity to mineralization. Visible gold is uncommon, but where observed, occurs as very small free gold grains in silica, and in close association with Fe-oxide.

Gold mineralization with associated iron oxide and silica alteration is found in all rocks types. In coherent rhyolite facies gold is typically hosted in joints, fractures, and faults with iron oxide ± quartz. Joints and fractures with mineralization range from 1mm to >5cm wide and are usually found as subparallel sets. Faults that cut coherent rhyolite facies can also contain gold in cataclasite matrix. Rhyolite breccias host gold mineralization in the matrix, are typically strongly silicified, and iron oxide-rich. In volcaniclastic facies, mineralization can be both disseminated within the matrix, and as hosted in fractures and veins like those found in coherent facies. Phreatomagmatic diatremes host gold in structures though the matrix rarely contains gold. Phreatic diatremes that are iron oxide rich contain gold in the matrix.

The Castle Mountain deposits have generally similar host rocks, alteration, chemistry, mode of gold occurrence, and gold to silver ratios. Silicification is common to all mineralization, and despite pervasive low intensity argillic alteration the best mineralization is low in clay content.

Gold mineralization is focused along structures and margins of facies contacts. It is believed that the vertical structures which tapped the magma responsible for the felsic volcanic package also acted as conduits for hydrothermal fluids that carried and deposited gold. Intersections of the steep structures with permeable units in the volcanic package created an environment for gold precipitation from hydrothermal fluids, possibly due to processes of boiling and interaction with meteoric water.

Rock porosity-permeability characteristics was a first-order control on the distribution of gold mineralization. Flow-dome breccia margins, phreatic diatremes, fault cataclasite, and fractures focused and interacted with more mineralizing fluids and contain the highest gold grades. Un-fractured coherent flow-dome facies, clay altered volcaniclastic facies, and clay altered phreatomagmatic diatremes with low or variable permeability are the least mineralized due to limited exposure to hydrothermal fluid. Mineralized rocks with lower permeability are invariably cut by structures e.g. faults, fractures, or phreatic diatremes.

Ore body geometry appears to be the combination of two principal factors:

1. Mineralization can mimic steep to vertical brecciated contacts of flow-domes and phreatomagmatic diatremes; and

2. Mineralization can form broad tabular zones that correlate more closely to the general orientation of variably bedded clastic units, such as flows, lobe and coulée breccias, basal flow breccias or autoclastic carapace and talus breccias, and felsic Agglomerated Lithic Tuff (ALT) units.

Both can occur in the same space and all transitions between them are possible. Felsic ALT units with a more porous matrix proximal to cross-cutting flow-domes and phreatomagmatic diatremes may contain gold mineralization in the matrix. Individual mineralized bodies are also focused around fault zones with the same general vertical geometry but may mushroom where permeability was high. The lateral volume of the mineralized bodies focused around fault zones is dictated by the intensity and extent of fracturing and faulting, and the pre-existing permeability-porosity of the host rocks (Figure 7.9).

Most historic reserves are in the relatively flat-lying, thick and laterally extensive Oro Belle, Jumbo and Jumbo South and Lesley Ann (JSLA) deposits, which are adjacent to high-angle, silicified fracture zones thought to be likely conduits for ore-forming fluids. Lithologic controls are more dependent on rock texture than rock type. Tuff beds, auto-breccias, and hydrothermal breccias have permeable fragmental textures. Brittle rhyolite flows and intrusives exhibit intense fracturing and have cooling joints, vesicular zones, spherulitic vugs, and flow foliations. Mineralization occurs in secondary silica in all of these features. Major fault/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.

Phreatic diatremes are another recognizable source of gold mineralization, and provide an insight into the possible mechanism for broader gold precipitation. In epithermal deposits around the world, phreatic diatremes are both genetically and spatially associated with hydrothermal systems that host gold mineralization (Tamas & Milési, 2003); this is the case at the Castle Mountain. They are recognized to occur within pre-existing phreatomagmatic environments resulting from indirect interaction between magmatic heat and an external water source in higher levels of hydrothermal systems, ranging from 200m to ≤1000m (Tamas & Milési, 2003). Their occurrence at Castle Mountain reaffirms that meteoric water likely interacted with upwelling Au-bearing hydrothermal fluids during the development of an orebody. This interaction, and the concomitant cooling and dilution of the hydrothermal fluids that would result, may well be an important mechanism for gold precipitation. The geometry of the phreatic diatremes also provides a pathway for meteoric water (and oxidation) to reach greater depths, where gold precipitation might not otherwise have occurred.

The consistent characteristic for each mineralized area is permeability-porosity, and where it is highest mineralization can occur. It is possible to have faults and fracture zones that are not mineralized as the structural regimes through the Project were active both pre- and post-mineralization. While faults and fracture zones cut mineralization, multiple phases of mineralization are not observed suggesting gold was precipitated in a single phase within a larger and longer-lived structural and hydrothermal event.

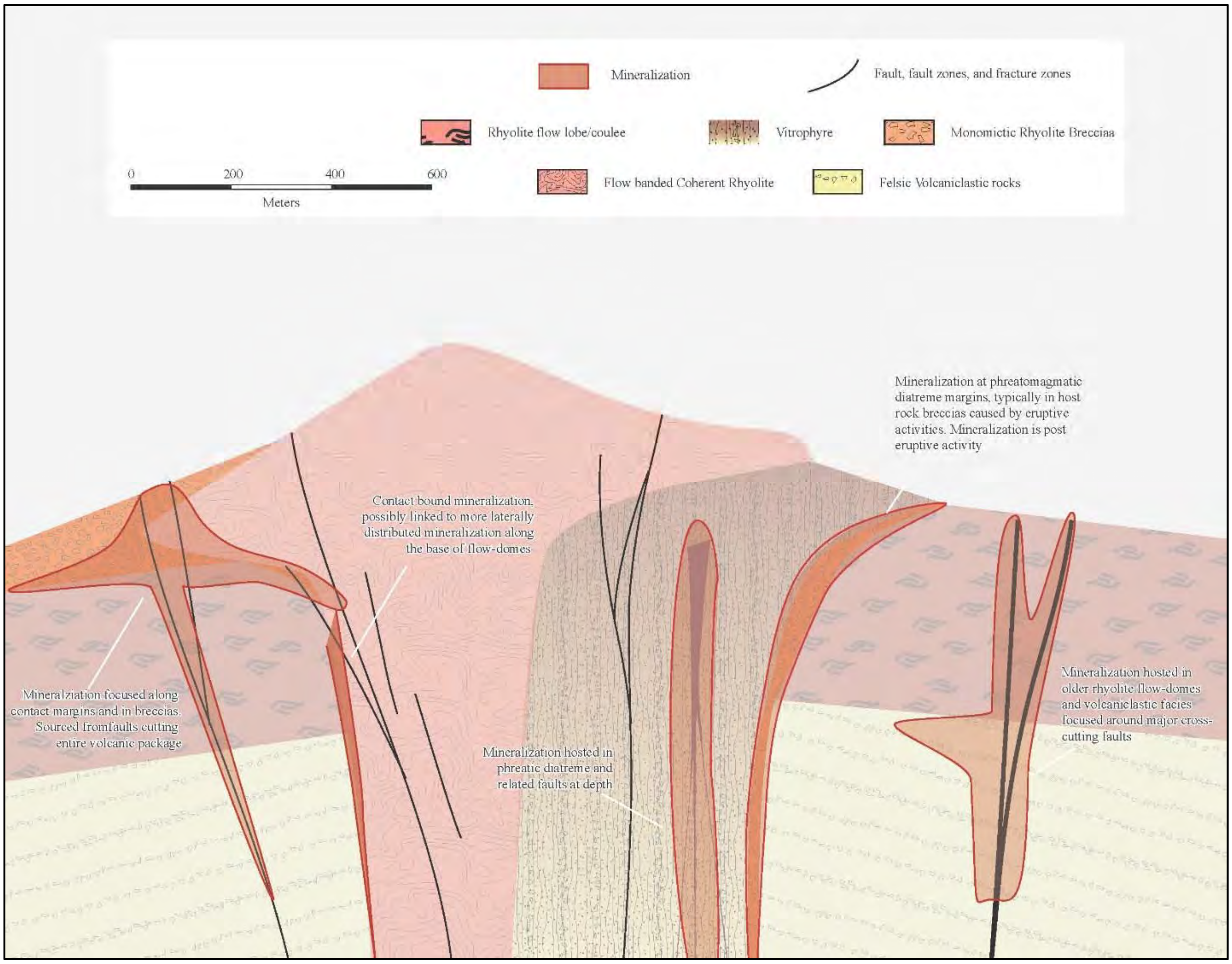

Figure 7-9 – Schematic model for zone of gold mineralization found on the Castle Mountain property (Source: Nicholls et al., 2017)

8.0 DEPOSIT TYPES

8.1 Deposit Model

Gold mineralization at Castle Mountain can be classified as volcanic-hosted low-sulfidation, quartz-adularia epithermal (LSE); it typically occurs within silicified zones, stockwork veins, breccias of tectonic or hydrothermal origin, as disseminations, and as lesser quartz veining within broad zones of hydrothermal alteration. LSE deposits are commonly found in volcanic island arcs, continent-margin magmatic arcs, and continental volcanic fields with extensional structures. Depositional environments include high-level hydrothermal systems from surficial hot spring settings to a depth of about 3,300 ft. to 4,900 ft. A genetic model of LSE deposition is shown in Figure 8.1.

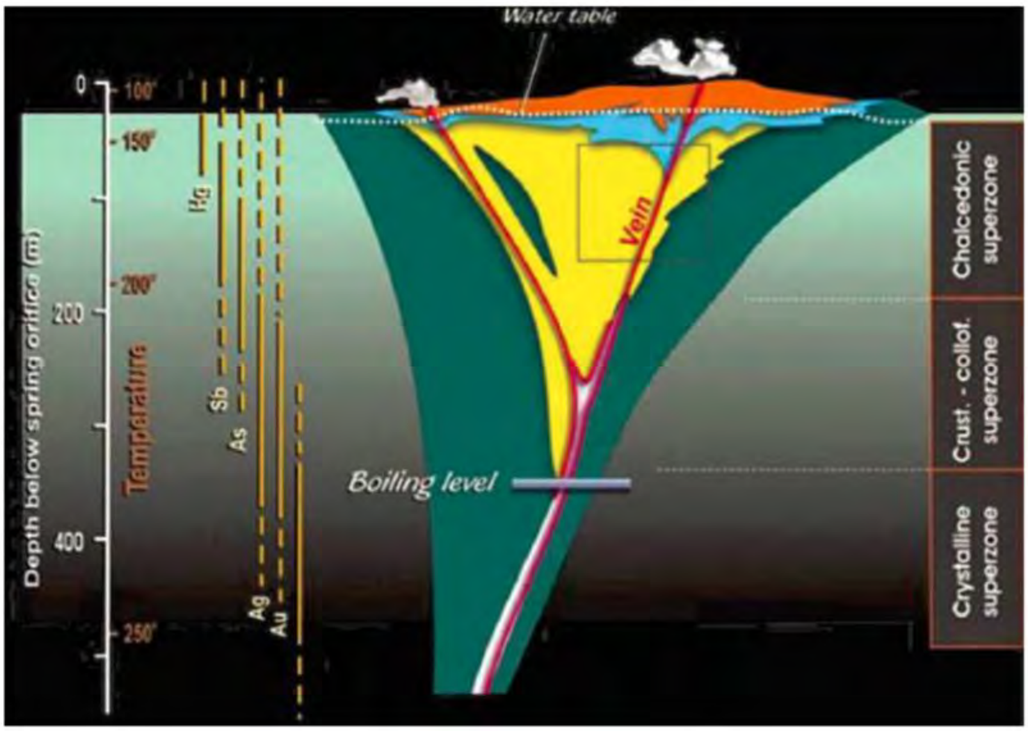

Figure 8-1 – Genetic Model for Low-Sulfidation Epithermal Gold Deposits (from Pressacco, 2013).

Low-sulfidation epithermal deposits are genetically linked to bimodal volcanism and typically are formed from extremely dilute fluids associated with magmas (Sillitoe and Hedenquist, 2003; Figure 8.2); economic grade gold deposition occurs several kilometers above the intrusion. Geothermal fluids with near neutral pH and reduced deep fluids, which are essentially in equilibrium with the altered host rocks, combine and precipitate minerals in veins. The slow ascent of the deep fluids in the rock-dominated hydrothermal system allows for this equilibrium to be achieved. The deep fluids are low- salinity and may be rich in gases such as CO_2 and H_2S.

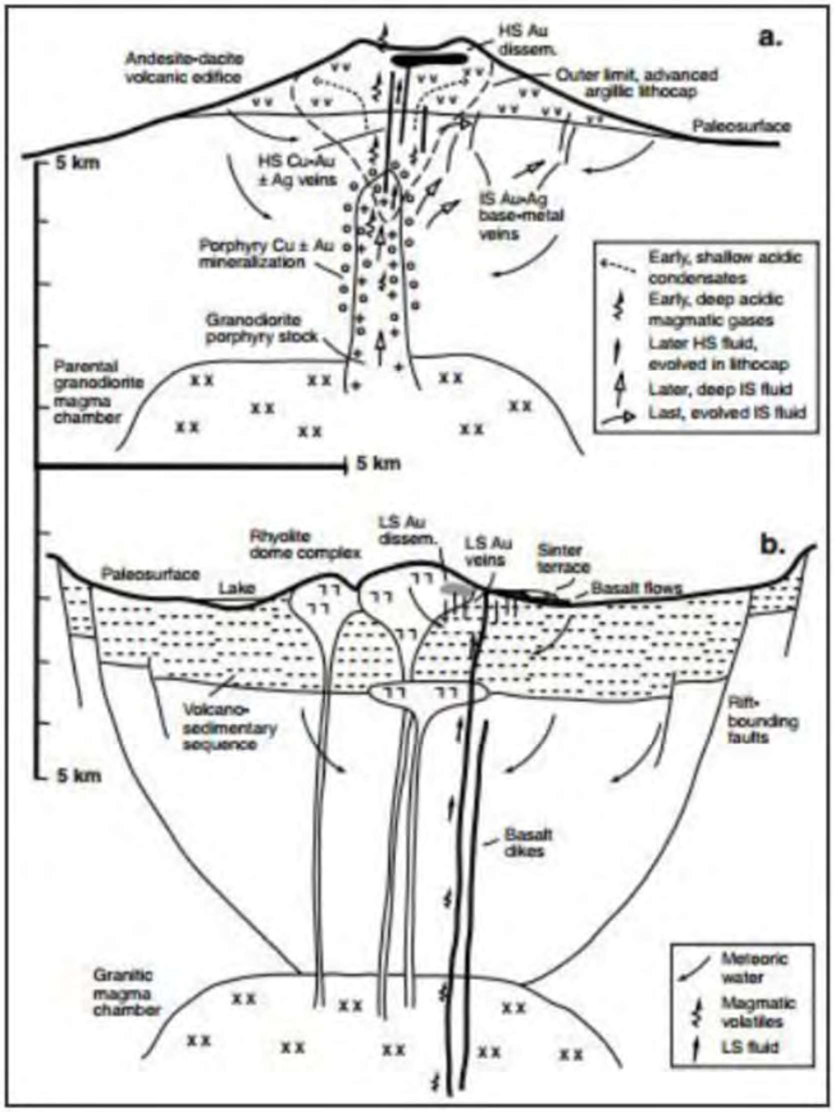

Figure 8-2 – Schematic Sections of End-Member Volcanotectonic Settings and Associated Epithermal and Related Mineralization Types. a. Calc-alkaline volcanic arc with high- and intermediate-sulfidation epithermal and porphyry deposits. b. Rift with bimodal volcanism and low-sulfidation deposits. From Sillitoe and Hedenquist (2003).

Limited dimension alteration zones form and are dominated by minerals produced in neutral pH environments e.g. quartz, adularia, carbonates and sericite. These liquids discharge at the surface as boiling, neutral pH hot springs which deposit silica and metals (MDA, 2009).

Regional-scale fracture systems related to grabens, calderas, and flow-dome complexes are typical of the depositional environment. Extensional normal faults, fault splays, ladder fault systems and cymoid loops are common. Locally, graben or caldera-fill clastic rocks are present. High-level subvolcanic intrusives in the forms of stocks or dikes and pebble diatremes are often present; these underlying intrusive bodies may be related to the presence of resurgent or domal structures (Yukon Geological Survey, 2005).

Economically important LSE gold deposits hosted by Mid-Miocene rhyolites include Round Mountain, Nevada (Sander, 1988; Sander and Einaudi, 1990); Sleeper, NW Nevada (Nash et al., 1995; Conrad et al., 1993, 1996), and Midas, Mule Canyon, Ivanhoe and Buckhorn along the Northern Nevada Rift (Goldstrand and Schmidt, 2000; John et al., 2003; Ioannou and Spooner, 2002; Wallace, 2003; Leavitt et al., 2004).

Most LSE deposits are associated with alteration zones marked by abundant K-feldspar (adularia), various types of secondary silica as well as As-Sb-Hg enrichments; these features are all present at Castle Mountain. In contrast to some LSE deposits the altered rocks at Castle Mountain have very low concentrations of Te and Se (<0.5 and <0.2 ppm, respectively), and base metals (a few tens of ppm of combined Zn+Pb, and usually <10 ppm Cu). Many of the mineralized rocks at and near the open pits show notable mass additions of K and/or Si. The Castle Mountain host rhyolite appears distinctive by virtue of its low-TiO_2, low-Al_2O, low-HFSE, high-K calc-alkaline nature, and alteration typified by additions of K (as adularia), silica, As and Sb.

The Shumake epithermal low-sulfidation gold-silver deposit of southwestern California shows a number of similarities with Castle Mountain (Barrett, 2016a): (i) it is hosted by lower to mid-Miocene felsic-dominated volcanic rocks of calc-alkaline affinity; (ii) the alteration assemblage is mainly adularia-sericite-silica; and (iii) it is enriched in Hg, As and Sb (Blaske et al., 1991). In addition, local surficial zones of acid-sulfate alteration are present in both areas.

In southern Idaho, the Challis region contains Eocene rhyolite tuffs and intrusions of the Twin Peaks caldera which, at least in major element composition (Hardyman and Fisher, 1985), compare closely with rhyolite at Castle Mountain. Some of the rhyolite tuffs and intrusions also host epithermal Au-Ag mineralization (Hardyman, 1985).

In the Goldfields mining district in southwestern Nevada, epithermal gold deposits are hosted by calc-alkaline andesitic to rhyolitic flows, domes and tuffs, but differ from Castle Mountain in their Lower Miocene age and their high-sulfidation style (Rockwell, 2000). Although some mid-Miocene felsic volcanics are present at Goldfields, they are unaltered. At the Hasbrouck Mountain Au-Ag deposit in southwestern Nevada, shallow epithermal mineralization is hosted by mid-Miocene volcaniclastic and pyroclastic rocks; these have been altered to an adularia-quartz-illite assemblage with late-stage development of kaolinite and chalcedonic sinter (Graney, 1986). The Borealis Au-Ag deposit in westernmost Nevada is hosted by silicified-argillized Miocene andesite and overlying kaolinite-alunite hydrothermal breccias that probably formed in a shallow hot-spring setting, but the mineralization itself has been dated as Pliocene (Strachan, 1985).

Felsic-dominated volcanic rocks of mid-Miocene age, but without significant precious metal deposits, are found in several regions in southwestern USA, for example the Woods Mountain volcanic center of southeastern California (McCurry, 1988; Musselwhite et al., 1989); the southwestern Nevada volcanic field (Sawyer et al., 1994); the Kane Springs Wash caldera in southern Nevada (Noble, 1968; Novak, 1984); the McDermitt caldera on the Nevada-Oregon border (Conrad, 1984), and the Snake River Plain of southern Idaho (Knott et al., 2016). However, the rhyolites of these volcanic centers differ from those at Castle Mountain by virtue of their higher concentrations of elements such as P, Zr, Th, Nb, Y and light REEs, with some suites also having higher Fe and higher ratios of (Na+K)/Al, that is, they are alkaline or peralkaline, rather than calc-alkaline (Barrett, 2016a).

9.0 EXPLORATION

Exploration work conducted prior to NewCastle and now Equinox Gold is found in Section 6.0, History and in Section 6.2, Modern Gold Exploration. Additional detail may be found in Temkin (2012). Exploration drilling is found in Section 10.0, Drilling.

9.1 NewCastle

9.1.1 Surveying

NewCastle acquired the Project in 2012 and began exploration activities with an airborne LIDAR survey to construct a detailed digital topographic surface and capture the extent of previous mining activities.

In March 2017, a high-resolution, drone- and fixed-wing based aerial photogrammetry survey was contracted to Compass Tools of Denver, Colorado to provide an updated topographic surface of the Project area, to enhance pit mapping, as well as to provide detailed updated maps of surface disturbance following the completion of the drill programs.

9.1.2 Mapping

Initial exploration in spring 2014 included detailed geologic mapping of the deposit area exposures and critical evaluation of the structural and stratigraphic setting. Further structural studies in spring 2015 focused on the historic open pits and a critical review of diamond drill core to develop a new geological model.

Beginning in August 2016, a comprehensive mapping, sampling, petrographic and lithogeochemical program was initiated by NewCastle with the goals of:

1. developing a consistent framework for the geology at Castle Mountain for use in future mapping and core logging efforts;
2. producing a detailed digital (ArcGIS) map of the Project;
3. enhancing the detailed lithologic controls on mineralization;
4. defining the lithochemical signature of the various volcanic units that make up the CMVS; and
5. to better define the geochemical effects imposed by the hydrothermal mineralizing event(s) onto the host sequences at Castle Mountain.

A detailed (1:2000 scale) mapping and sampling program was initiated in 2016 (Nicholls, et al., 2017). The results of the mapping program are in Section 7.3.

9.1.3 Geochemistry

In late 2016, a comprehensive lithogeochemical and petrographic study (Barrett, 2016a, b) was carried out in order to:

1. better define the volcanic, intrusive and volcaniclastic units as an aid to mapping and correlation; and

2. to quantify the degree of hydrothermal alteration.

To that end, 270 whole-rock samples from recent drill holes and East Ridge outcrops were analyzed using a comprehensive lithogeochemical package that includes major oxides, trace elements, and rare earth elements. These data were used to define the lithology and alteration described in Section 7.3.

A separate study was carried out in early 2017 involving several hundreds of samples analyzed from four selected drill holes located in the South Domes mineralized area, namely holes CMM-079, 080, 081, and 111. The analyses included an extensive suite of trace elements as well as gold and aimed at understanding “pathfinder” metal distributions relative to zones of gold enrichment. While there appears to be no consistent, direct correlation between gold enrichment and any of the other metals plotted, there is a general tendency for antimony to be elevated at or near where Au enrichment occurs. Additionally, it appears that enrichment in antimony, arsenic and bismuth occurs at varying offsets from where the main Au enrichment occurs, but generally at shallower levels. The associated enrichment in antimony, arsenic and bismuth at relatively shallow levels may thus be used as an exploration vector pointing to zones of gold enrichment at somewhat 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.

9.1.4 Geophysics

NewCastle completed a Transient Electromagnetic survey (TEM) using Zonge International between February 2-9, 2015 and included soundings at 50 locations to evaluate alluvial material and identify potential water table levels southwest of the mine. The TEM survey appears to show a strong resistivity contrast between the alluvial material and the bedrock, which is lower resistivity. Correlating with historic water wells, a number of inversion sections indicated gradual thickening alluvial material to the west and a possible marker defining the top of the water table.

Ground-based gravity was also used as a tool to assist in determining the depth of alluvial cover. The first Phase I survey (concurrent with the TEM) was carried out between December 19-30, 2014 and totaled 615 stations over a 300m x 300m grid (Magee, 2014) (Wright, 2015a). In the eastern part of the survey area (closest to the mine), where alluvial cover is thinnest, the gravity is strongly influenced by bedrock geology. Several interpreted north-south and northeast trending structures were indicated. Correlating with drill hole geology, gravity highs strongly suggest the presence of near-surface of the Proterozoic basement. A prominent northeast trending low correlated to an increase thickness of rhyolite, mimicking the modeled rhyolite-andesite contact.

A Phase II gravity survey was performed over the deposit area from June 14-16, 2015 to augment definition of the potential volcanic/mineral system. The survey totaled 779 stations on a 100m x 100m grid (Wright, 2015b). The survey suggests a sharp stepped, western down-drop on the eastern side of the grid.

10.0 DRILLING

10.1 Viceroy Legacy Drilling

Prior to 2012, a total of 1,762 drill holes were completed on the Castle Mountain Project for a footage totaling 1,185,982 feet (361,487 meters) of legacy drilling (Gray et al., 2016).

The legacy drilling on the project was a mix of vertical mud-rotary drilling, and vertical to moderately inclined reverse circulation (RC) drilling and diamond core drilling. Drilling prior to 2012 was dominated by vertical drilling, either mud-rotary or reverse circulation drilling, and was closely spaced in areas of significant mineralization. The results of this drilling are summarized in Gray et al. (2016).

Table 10.1 summarizes the legacy drilling activity from 1968-2001. A plan map showing all drilling completed to date including the 2016-2017 program is illustrated in Figure 10.1. A plan map showing drilling completed from 2013-2015 is shown in Figure 10.2.

Table 10-1 - Drilling Summary from 1968 to 2001

Year	Drilling Description	No. of Holes	Feet drilled	Meters drilled	Notes
1968-2001	Mud-rotary & RC	1,695	1,141,582	347,954	Dominantly vertical holes
	Diamond Core	67	44,400	13,533	
Totals		**1,762**	**1,185,982**	**361,487**	

10.2 NewCastle 2013-2015 Phase 1/2/3 Drilling

Since the start of the Castle Mountain Mining 2013 Phase 1 drilling program, and through to the completion of the 2015 Phase III drilling program, the drill holes were re-oriented and drilled at lower inclination angles to better 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). Table 10.2 summarizes the legacy drilling activity from 2013-2015. A plan map showing all drilling completed to date including the 2016-2017 program is illustrated in Figure 10.1.

In March 2013, NewCastle initiated a Phase 1 drilling program on the Project. A total of 5,514 m (18,091.5 ft.) of core drilling and 2,068 m (6,785 ft.) of RC drilling in 30 holes was completed. The drill program was designed to twin and scissor historical drill holes in and around the previously mined pit areas, as well as, test mineralization in other selected exploration targets. The goal was also 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 rhyolite domes as the principal structural controls of the mineralization. These same features were the initial targets of the 2013 drill program but as early results were returned and interpreted, the importance of the intersections of the north-south and northeast-trending structures was recognized and gold mineralization was recognized to be located at the intersection points of these controlling structures in lithology other than rhyolite. As a result, later holes were re-oriented to test the north-south and northeast trending structures.

In 2014-2015, NewCastle drilled an additional 13,927 m in 47 RC and Core holes in Phases 2 and 3. Drilling targeted areas between and under the existing open pits and was successful in terms of the conversion of Inferred Mineral Resources to Measured/Indicated Mineral Resources and also the conversion of areas previously classified as waste within the proposed pit shells to Inferred Mineral Resources. The intersection of very encouraging high-grade intercepts in the Lucky John target area (CMM-054 and CMM-060) demonstrated potential for strike/depth extensions to known zones and near-term low strip ounces adjacent to, and on trend of the existing mineral resources. Broad intervals of gold mineralization were also encountered around the southern margins of the proposed JSLA pit (CMM-018 and CMM-040). Highlights included:

- CMM-054: 33.3m @ 30.31 g/t Au, incl. 10.6m @ 94.04 g/t Au
- CMM-060: 74.4m @ 9.11 g/t Au, incl. 35.1m @ 18.97 g/t Au
- CMM-018: 137.8m @ 1.5 g/t Au, incl. 16.8m @ 3.07 g/t Au
- CMM-040: 79.2m @ 0.89 g/t Au, incl. 23.8m @ 1.48 g/t Au

The Phase 2 program also included four PQ diameter (85 mm) core holes drilled for metallurgical column testing for an aggregate 1,195.6 meters, with two holes (CMM-012 and 013) under the Jumbo pit, one hole into the Lucky John zone (CMM-014), and one hole into the South JSLA zone (CMM-017).

The follow-up Phase 3 program in Jan-Feb 2015 was compromised of 1,996 m of RC and 1,795 m of HQ core in 10 deeper holes to further follow up on the high-grade Lucky John target intercepts in CMM-054 and CMM-060. Highlights included:

- CMM-068: 62.8m @ 1.26 g/t Au, incl. 1.5m @ 32.9 g/t Au
- CMM-069: 21.3m @ 1.17 g/t Au
- CMM-070: 67.7m @ 1.27 g/t Au, incl. 20.4m @ 2.86 g/t Au
- CMM-071: 123.4m @ 2.43 g/t Au, incl. 6.1m @ 23.25 g/t Au
- CMM-073: 30.5m @ 9.22 g/t Au, incl. 18.3m @ 15.14 g/t, incl. 6.1m @ 31.78 g/t Au
- CMM-074: 66.1m @ 1.5 g/t Au, incl. 2.9m @ 17.48 g/t Au

Total cumulative drilling by NewCastle in the period 2013-2015 is **21,510 m in 77 holes** and shown in Table 10.2 below.

Table 10-2 - Drilling Summary from 2013 to 2015

Year	Drilling Description	No. of Holes	Feet drilled	Meters drilled	Notes
2013	RC/Diamond Core	30	18,091	5,514	RC and PQ core
2014-2015	RC/Diamond Core	47	3,923	13,927	RC and PQ core
Totals		**77**	**70,570**	**21,510**	

10.3 NewCastle Phase I 2016 Drilling

NewCastle Phase I drilling began in June 2016 and by October 2016 had completed 46 exploration and infill resource drill holes, and one hydrological test hole, for a total drilled footage of 65,423ft (19,941m).

The program targeted the southern part of the current resource area known as "Big Chief" and "South Domes". These targets were historically sparsely drilled and lay immediately 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 mineralization encountered in 2014 and 2015 (Gray et al., 2016).

The list of drill holes completed in Phase I is shown in Table 10.3 and drill hole locations are shown in Figure 10.3.

Table 10-3 - Castle Mountain Phase I Drilling Program

Hole Id	Easting NAD83 (m)	Northing NAD83 (m)	Elevation (m)	Azimuth degrees	Inclination degrees	Total Depth (ft)	Core Drilled (ft)	RC Drilled (ft)	Deposit Area	Date Drilling Started	Date Drilling Finished
CMM-078	672,822	3,904,292	1,302	110	-70	1507.0	1507.0		South Domes	06/14/16	06/20/16
CMM-079	672,822	3,904,292	1,302	290	-65	2013.5	2013.5		South Domes	06/21/16	07/03/16
CMM-080	672,623	3,904,369	1,288	290	-65	1447.0	1447.0		South Domes	07/03/16	07/08/16
CMM-081	672,452	3,904,425	1,289	290	-65	1267.0	1267.0		South Domes	07/09/16	07/13/16
CMM-082	673,059	3,904,655	1,335	90	-75	1427.0	1427.0		621 Zone	07/21/16	07/29/16
CMM-083	672,930	3,904,709	1,332	270	-60	1607.0	1607.0		622 Zone	07/14/16	07/21/16
CMM-084	672,578	9,095,098	1,332	110	-55	1400.0		1400	Big Chief	06/28/16	06/30/16
CMM-085	672,580	3,905,100	1,332	290	-60	1045.0		1045	Big Chief	06/27/16	06/28/16
CMM-086	672,440	3,905,212	1,326	290	-70	985.0		985	Big Chief	06/25/16	06/26/16
CMM-087	672,680	3,905,190	1,335	290	-65	1960.0		1960	Big Chief	07/20/16	07/25/16
CMM-088	672,257	3,905,343	1,343	290	-78	1405.0		1405	Big Chief	06/11/16	06/14/16
CMM-089	672,720	3,905,240	1,339	290	-65	1600.0		1600	Big Chief	07/13/16	07/20/16
CMM-090	672,425	3,905,350	1,357	290	-70	1185.0		1185	Big Chief	07/11/16	07/13/16
CMM-091	672,310	3,905,390	1,360	290	-67	1145.0		1145	Big Chief	07/09/16	07/10/16
CMM-092	672,110	3,905,448	1,347	290	-65	1000.0		1000	Big Chief	06/15/16	06/18/16
CMM-093	672,655	3,905,356	1,354	110	-50	1388.0	1388.0		Big Chief	07/20/16	07/25/16
CMM-031	672,635	3,905,380	1,355	270	-70	2288.0	1259.0		Big Chief	07/26/16	08/05/16
CMM-094	672,462	3,905,431	1,395	290	-60	1533.0	1533.0		Big Chief	06/21/16	07/06/16
CMM-095	672,293	3,905,481	1,387	290	-60	1050.0	1050.0		Big Chief	07/08/16	07/15/16
CMM-096A	672,086	3,905,554	1,347	290	-60	598.0	598.0		Big Chief	07/16/16	07/19/16
CMM-097	672,426	3,905,473	1,404	290	-60	1517.0	697.0	820	Big Chief	7/24,7/31	7/27,8/5
CMM-098	672,500	3,905,478	1,377	290	-75	1508.0	1008.0	500	Big Chief	8/3-8/6	8/5-8/12

Hole Id	Easting NAD83 (m)	Northing NAD83 (m)	Elevation (m)	Azimuth degrees	Inclination degrees	Total Depth (ft)	Core Drilled (ft)	RC Drilled (ft)	Deposit Area	Date Drilling Started	Date Drilling Finished
CMM-099	672,350	3,905,632	1,378	290	-75	1227.0	427.0	800	Big Chief	6/19,8/6	6/25 - 8/8
CMM-100	672,180	3,905,697	1,350	290	-75	1000.0		1000	Big Chief	07/01/16	07/07/16
CMM-101	672,747	3,906,008	1,372	110	-75	1600.0		1600	Jumbo	08/19/16	08/28/16
CMM-102	672,688	3,906,030	1,361	290	-70	1155.0		1155	Jumbo	07/07/16	07/08/16
CMM-103	672,743	3,906,041	1,373	290	-75	1630.0		1630	Jumbo	08/06/16	08/17/16
CMM-104	672,920	3,906,172	1,373	110	-60	1818.5	1818.5		East Ridge	08/13/16	08/27/16
CMM-105	673,388	3,906,224	1,502	290	-60	1500.0		1650	East Ridge	09/29/16	10/04/16
CMM-106	673,394	3,906,292	1,497	290	-70	1500.0		1400	East Ridge	10/16/16	10/20/16
CMM-107	673,496	3,906,288	1,445	290	-60	1205.0		1205	East Ridge	08/10/16	08/14/16
CMM-108	673,400	3,906,314	1,497	290	-60	1500.0		1550	East Ridge	09/16/16	09/28/16
CMM-109	673,394	3,906,390	1,500	290	-50	1500.0		1450	East Ridge	09/09/16	09/15/16
CMM-110	673,485	3,906,357	1,453	290	-75	1500.0		1500	East Ridge	08/07/16	08/10/16
CMM-111	672,822	3,904,292	1,302	290	-80	2091.5	2091.5		South Domes	08/09/16	08/19/16
CMM-112	673,470	3,906,395	1,455	290	-75	500.0	975.0	500	East Ridge	08/06/16	08/07/16
CMM-113	673,387	3,906,455	1,502	290	-75	1505.0		1505	East Ridge	08/20/16	08/23/16
CMM-114	673,550	3,906,485	1,465	290	-70	1500.0		1500	East Ridge	08/14/16	08/18/16
CMM-115	673,465	3,906,522	1,502	290	-75	1385.0		1385	East Ridge	08/29/16	09/08/16
CMM-118	673,306	3,906,681	1,494	290	-80	1457.0	1457.0		Oro Belle NEX	08/28/16	09/05/16
CMM-119	673,145	3,906,742	1,456	290	-65	1356.0	1356.0		Oro Belle NEX	08/22/16	08/29/16
CMM-120	673,146	3,906,743	1,456	290	-45	1817.0	1817.0		Oro Belle NEX	08/29/16	09/08/16
CMM-121	673,245	3,906,790	1,450	290	-85	1205.0		1000	Oro Belle NEX	08/29/16	08/30/16
CMM-124	673,138	3,906,868	1,445	290	-75	1325.0		1325	Oro Belle NEX	08/03/16	08/05/16
CMM-125	673,303	3,906,878	1,471	290	-80	1625.0		1625	Oro Belle NEX	07/26/16	08/02/16
CMM-127	673,322	3,906,902	1,477	290	-65	1645.0		1645	Oro Belle NEX	08/31/16	09/08/16
W-31P	672,441	3,904,978		0	-90	1000.0		1000		9/23/16	10/23/16
Totals (ft)						**66,423**	**26,744**	**39,470**			
Totals (m)						**20,246**	**8,151**	**12,030**			

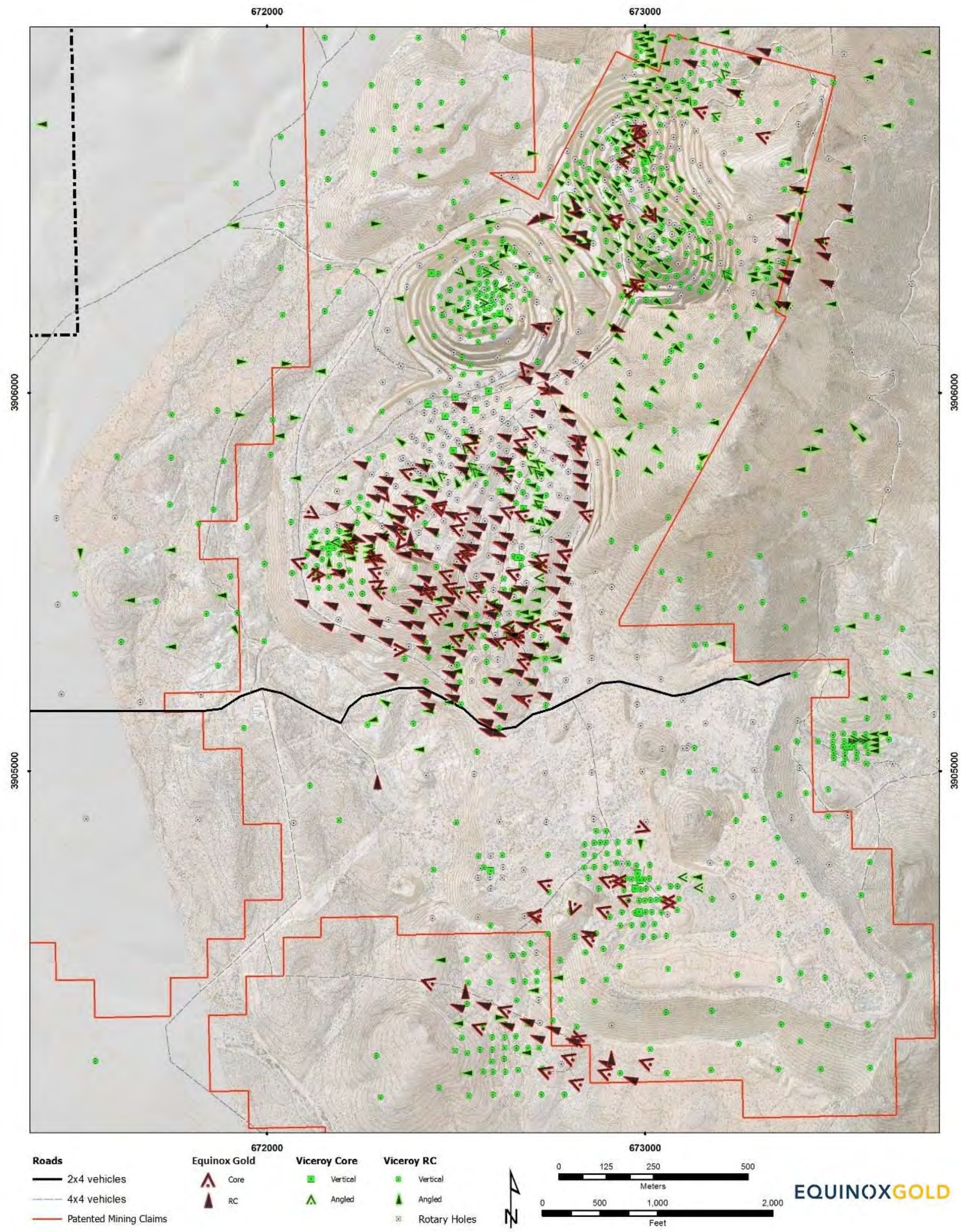

Figure 10-1 – Drill Hole Locations – All 1968-2017

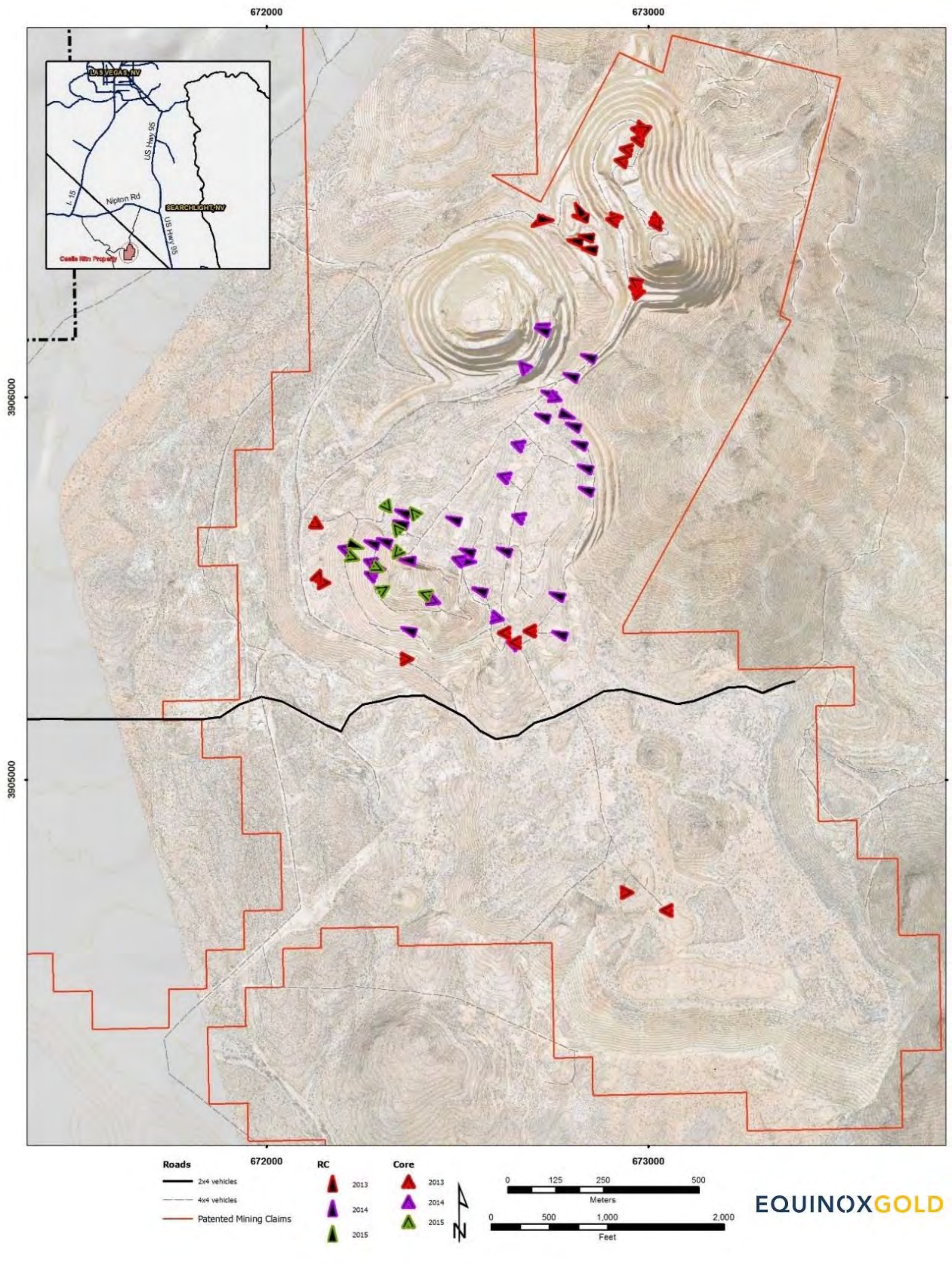

Figure 10-2 – Drill Hole Locations - NewCastle 2013-2015

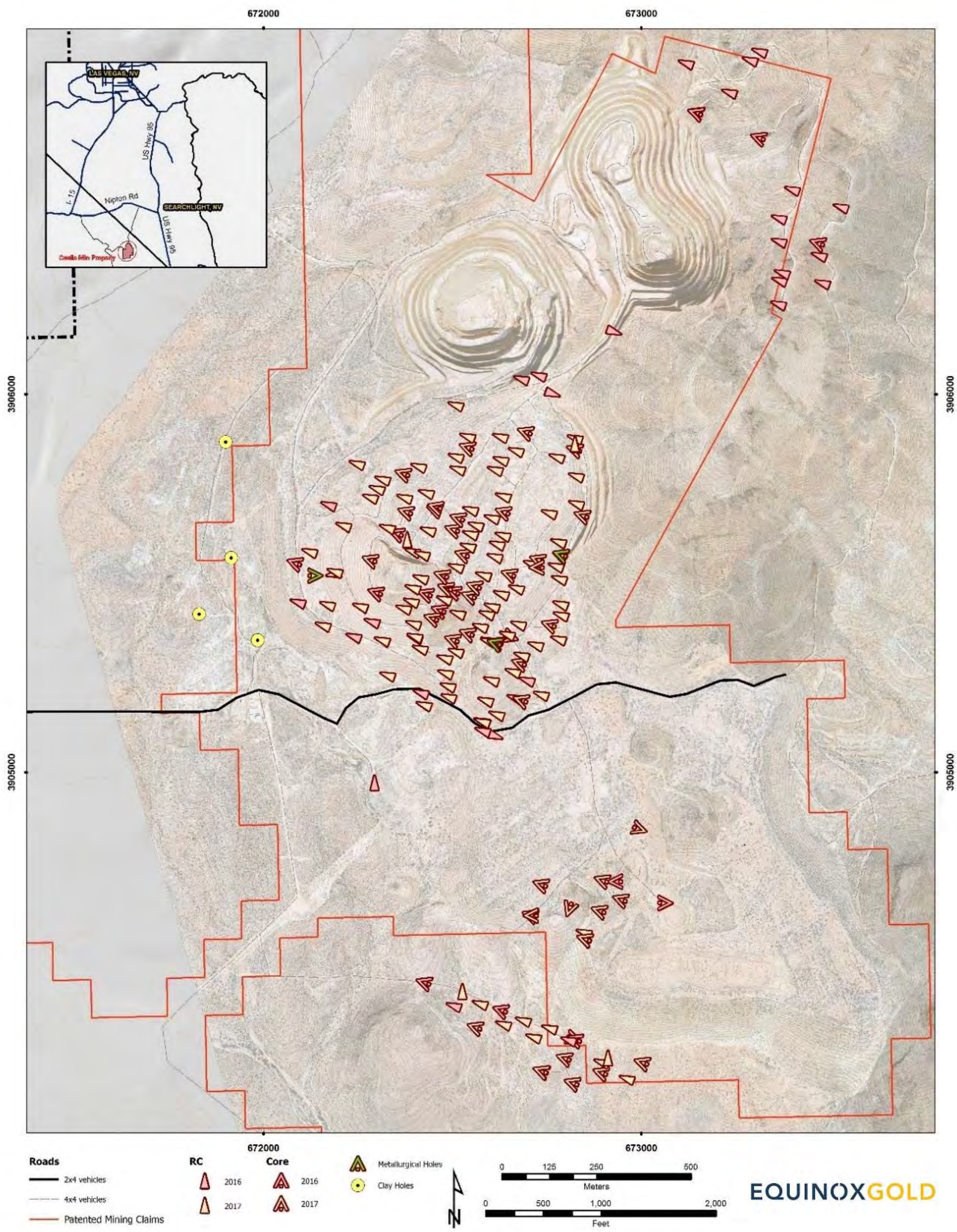

Figure 10-3 – Drill Hole Locations – NewCastle Phase I 2016 and Phase II 2016-17

10.4 NewCastle Phase II 2016-2017 Drilling

NewCastle Phase II drilling began in late October 2016 and was essentially a continuation of the Phase I program that targeted 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). Additionally, drill holes were added to the program to support ongoing metallurgical test work; test for sources of clay on the Project for the leach pad liner, and test for additional water sources for mine development. All holes drilled in 2016 and 2017 are illustrated in Figure 10.3.

A total of 148 core and reverse circulation holes were drilled and these included: 136 resource expansion and infill drilling holes, four water well test holes, four PQ metallurgical test holes, and four PQ holes to test for clays with suitable properties for use as a clay liner. The total drill footage completed was 169,944ft (51,799m) including 160,341ft (48,872m) of resource and infill drilling; 5,620ft (1,713m) of water well test drilling; 3,383ft (1,031m) of PQ metallurgical drilling; and 600ft (183m) of clay test hole drilling. Table 10.3 summarizes the drill footage by hole and area drilled.

10.4.1 Resource Definition Drilling

The resource definition and infill drilling focused on two main areas:

1. The areas encompassed by the 2015 defined block model extent but focusing on JSLA (backfilled) area, Lucky John area, and JSLA South area and the then-defined OB-1, OB-2 and OB-3 mineralized trends; and

2. The South Domes area which at the end of the drill program was extended to include the 621 Zone under the South Dump.

A total of 136 drill holes were completed that were either diamond core drilled, reverse-circulation (RC) drilled, or combination RC pre-collar with diamond core tails. Drill footage totaled 160,341ft (48,872m) that broke down as 47,131ft (14,366m) of diamond drill core and 113,210ft (34,506m) of reverse circulation drilling. A breakdown of the resource definition and infill drilling is contained in Table 10.3.

10.4.2 Metallurgical Drilling

Three large diameter PQ metallurgical test holes for a total of 2,943ft (897m) were completed that were twins of core holes. The twinned holes selected were CMM-033, CMM-161 and CMM-212; details of the holes are shown in Table 10.3. A fourth metallurgical hole was pre-collared but the JSLA dump backfill proved too deep to successfully complete a drill hole twin for metallurgical testing and was abandoned at 440ft.

Four vertical, large diameter PQ diamond core holes were drilled to test for a site source of clay for use in construction of the heap leach pad under-liner. The 4 holes were each completed to a depth of 150ft for a total of 600ft (183m). 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.

10.4.3 Water Well Drilling & Testing

Four reverse circulation holes were drilled, two in the JSLA (backfill) pit area and two in the South Domes area and were drilled to test for sources of water within the patented mining claims; the ultimate goal being to drill production wells that could be used for early stage run-of-mine (ROM) and further mine development. Sites were chosen based upon a combination of data sourced from geophysical (gravity) data, surface geologic mapping, and early stage remodeling of the geology across the resource area. A total of 5,620ft (1,713m) of RC drilling was completed in 4 holes and is summarized in Table 10.4.

Table 10-4 - Phase II 2016-17 Drilling Program Summary

Hole Id	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Drilling Start	Date Drilling Finished
CMM-116	672,679	3,905,191	1334	290	-50	1330		1330	JSLA	10/30/16	11/07/16
CMM-117	672,567	3,904,323	1292	290	-60	1165		1165	South Domes	12/10/16	12/15/16
CMM-122C	672,891	3,904,205	1277	290	-60	1987	1987		South Domes	11/28/16	12/10/16
CMM-123	672,523	3,904,372	1290	290	-60	1105		1105	South Domes	12/05/16	12/09/16
CMM-126	672,654	3,904,324	1283	290	-60	1460		1460	South Domes	11/29/16	12/04/16
CMM-128	672,741	3,904,292	1285	290	-60	1640		1640	South Domes	12/05/16	12/10/16
CMM-129	672,906	3,904,232	1278	290	-60	1600		1600	South Domes	12/13/16	12/16/16
CMM-130C	672,977	3,904,180	1279	290	-60	1505	1505		South Domes	02/19/17	02/25/17
CMM-131	672,592	3,904,379	1285	290	-50	815		815	South Domes	12/01/16	12/04/16
CMM-132	672,703	3,904,339	1287	290	-60	1680		1680	South Domes	11/08/16	11/21/16
CMM-133	672,771	3,904,314	1288	290	-60	1570		1570	South Domes	01/16/17	01/24/17
CMM-134C	673,001	3,904,230	1281	290	-60	1697	1697		South Domes	11/14/16	11/23/16
CMM-135	672,870	3,904,569	1323	290	-60	500		500	South Domes	01/23/17	01/24/17
CMM-136	672,440	3,905,180	1316	290	-60	745		745	JSLA	02/14/17	02/17/17
CMM-137	672,593	3,905,125	1337	290	-60	1000		1000	JSLA	02/18/17	02/20/17
CMM-137A	672,593	3,905,125	1337	290	-60	1000		1000	JSLA	04/13/17	04/16/17
CMM-138	672,351	3,905,245	1326	290	-60	700		700	JSLA	02/28/17	03/02/17
CMM-139	672,516	3,905,185	1331	290	-60	1000		1000	JSLA	02/21/17	02/23/17
CMM-140	672,633	3,905,142	1337	290	-60	1000		1000	JSLA	02/28/17	03/02/17
CMM-141	672,505	3,905,222	1329	290	-60	765		765	JSLA	01/25/17	01/31/17
CMM-142	672,605	3,905,185	1345	290	-60	1000		1000	JSLA	02/22/17	02/28/17
CMM-143	672,178	3,905,373	1344	290	-60	750		750	JSLA	03/19/17	03/20/17
CMM-144	672,505	3,905,254	1334	290	-60	1000		1000	JSLA	03/16/17	03/18/17

Hole Id	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Drilling Start	Date Drilling Finished
CMM-145	672,334	3,905,349	1353	290	-60	1000		1000	JSLA	03/15/17	03/17/17
CMM-146	672,431	3,905,313	1353	290	-60	1000		1000	JSLA	03/21/17	03/24/17
CMM-147	672,494	3,905,291	1342	290	-60	1000		1000	JSLA	02/09/16	02/13/16
CMM-148	672,629	3,905,241	1350	290	-60	1450		1450	JSLA	02/15/17	02/19/17
CMM-149	672,750	3,905,197	1335	290	-60	1380		1380	JSLA	11/28/16	12/06/16
CMM-150	672,190	3,905,434	1346	290	-60	700		700	JSLA	03/21/17	03/23/17
CMM-151	672,425	3,905,349	1362	290	-60	1200		1200	JSLA	03/06/17	03/08/17
CMM-152	672,529	3,905,311	1349	290	-65	1200		1200	JSLA	02/05/17	02/09/17
CMM-153	672,671	3,905,258	1346	290	-60	1500		1500	JSLA	02/20/17	02/24/17
CMM-154	672,277	3,905,434	1371	290	-60	1000		1000	JSLA	03/03/17	03/05/17
CMM-155	672,422	3,905,382	1375	290	-60	1500		1500	JSLA	03/01/17	03/03/17
CMM-156C	672,491	3,905,357	1371	290	-60	1197	1197		JSLA	03/02/17	03/06/17
CMM-157C	672,677	3,905,289	1347	290	-60	1457	1457		JSLA	01/21/17	01/26/17
CMM-158	672,359	3,905,437	1386	290	-60	1160		1160	JSLA	03/27/17	03/29/17
CMM-159C	672,438	3,905,408	1388	290	-60	1420	1420		JSLA	03/08/17	03/13/17
CMM-160C	672,545	3,905,369	1367	290	-60	1417	1417		JSLA	01/26/17	02/03/17
CMM-161A	672,614	3,905,344	1358	290	-60	1150		1150	JSLA	04/09/17	04/13/17
CMM-162	672,700	3,905,299	1345	290	-60	1200		1200	JSLA	12/07/16	12/14/16
CMM-163	672,211	3,905,523	1362	262	-60	285		285	JSLA	03/07/17	03/08/17
CMM-163A	672,211	3,905,523	1362	290	-60	700		700	JSLA	03/08/17	03/09/17
CMM-164	672,399	3,905,455	1399	290	-60	1500		1500	JSLA	03/17/17	03/21/17
CMM-165	672,526	3,905,409	1379	290	-60	1420		1420	JSLA	03/04/17	03/08/17
CMM-166	672,602	3,905,381	1357	290	-60	1460		1460	JSLA	02/24/17	02/27/17
CMM-167	672,670	3,905,356	1351	290	-60	1500		1500	JSLA	03/02/17	03/05/17
CMM-168	672,759	3,905,324	1329	290	-60	1500		1500	JSLA	02/16/17	02/19/17
CMM-169	672,151	3,905,577	1347	290	-60	500		500	JSLA	03/24/17	03/25/17
CMM-170	672,403	3,905,486	1405	290	-60	1500		1500	JSLA	03/21/17	03/24/17
CMM-171	672,484	3,905,456	1388	290	-60	1320		1320	JSLA	03/24/17	03/27/17
CMM-171A	672,484	3,905,456	1388	290	-60	940		940	JSLA	03/29/17	04/09/17
CMM-172C	672,606	3,905,412	1360	290	-60	1324	1324	0	JSLA	03/14/17	03/22/17
CMM-173	672,675	3,905,387	1353	290	-60	1500		1500	JSLA	02/28/17	03/02/17
CMM-174	672,781	3,905,348	1341	290	-60	1025		1025	JSLA	12/16/16	12/19/16
CMM-175C	672,282	3,905,562	1384	290	-60	1127	1127		JSLA	12/11/16	12/17/16
CMM-176	672,417	3,905,513	1403	290	-60	1500		1500	JSLA	03/08/17	03/17/17
CMM-177C	672,487	3,905,488	1385	290	-60	1587	1587		JSLA	02/22/17	03/01/17
CMM-178C	672,546	3,905,466	1366	290	-60	1597	1597		JSLA	12/11/16	12/19/16
CMM-179	672,615	3,905,441	1364	290	-60	1500		1500	JSLA	02/28/17	03/02/17
CMM-180C	672,760	3,905,388	1344	290	-60	1118	1048	70	JSLA	11/29/16	12/03/16
CMM-181	672,410	3,905,548	1388	290	-60	1500		1500	JSLA	03/09/17	03/17/17
CMM-182C	672,486	3,905,520	1380	290	-60	1673	1673		JSLA	03/22/17	03/29/17
CMM-183	672,547	3,905,498	1365	290	-60	1320		1320	JSLA	03/08/17	03/15/17

Hole Id	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Drilling Start	Date Drilling Finished
CMM-184	672,635	3,905,466	1370	290	-60	1455		1455	JSLA	01/11/17	01/24/17
CMM-185	672,792	3,905,409	1341	290	-60	1750		1750	JSLA	02/07/17	02/17/17
CMM-186	672,230	3,905,646	1359	290	-60	1000		1000	JSLA	03/16/17	03/20/17
CMM-187C	672,443	3,905,569	1378	290	-60	1416	1416		JSLA	02/18/17	02/22/17
CMM-188	672,529	3,905,537	1366	290	-60	1500		1500	JSLA	03/05/17	03/08/17
CMM-189	672,605	3,905,510	1365	290	-60	1845		1845	JSLA	03/20/17	03/23/17
CMM-190	672,686	3,905,480	1364	290	-60	1500		1500	JSLA	01/11/17	01/21/17
CMM-191	672,807	3,905,436	1339	290	-60	1470		1470	JSLA	01/25/17	02/04/17
CMM-192	672,334	3,905,641	1376	290	-60	1000		1000	JSLA	03/16/17	03/18/17
CMM-193C	672,526	3,905,571	1369	290	-60	1500	1360	140	JSLA	01/11/17	01/20/17
CMM-194C	672,649	3,905,526	1365	290	-60	1660	1360	300	JSLA	12/19/16	01/13/17
CMM-195	672,451	3,905,630	1375	290	-60	1480		1480	JSLA	02/21/17	02/23/17
CMM-196	672,563	3,905,590	1366	290	-60	1000		1000	JSLA	03/17/17	03/19/17
CMM-197	672,625	3,905,567	1362	290	-60	1500		1500	JSLA	03/23/17	03/26/17
CMM-198	672,701	3,905,539	1364	290	-60	1000		1000	JSLA	12/09/16	01/11/17
CMM-199C	672,801	3,905,503	1343	290	-60	1507	442	1065	JSLA	1/7 - 1/23	1/21-1/24
CMM-200	672,296	3,905,720	1362	290	-60	720		720	JSLA	03/14/17	03/17/17
CMM-202C	672,500	3,905,645	1378	290	-60	1080	715	365	JSLA	2/1-2/9	02/12/17
CMM-203	672,561	3,905,623	1368	290	-60	1000		1000	JSLA	03/17/17	03/18/17
CMM-204	672,633	3,905,597	1365	290	-60	1280		1280	JSLA	03/26/17	03/28/17
CMM-205C	672,742	3,905,557	1358	290	-60	1062	522	540	JSLA	12/4 - 1/14	12/11 - 1/16
CMM-206	672,806	3,905,534	1348	290	-60	1000		1000	JSLA	12/19/16	01/07/17
CMM-207	672,326	3,905,741	1364	290	-60	700		700	JSLA	03/08/17	03/14/17
CMM-208	672,399	3,905,714	1375	290	-60	900		900	JSLA	02/02/17	02/06/17
CMM-209C	672,518	3,905,671	1376	290	-60	1035	535	500	JSLA	2/6-2/9	2/8-2/12
CMM-210	672,586	3,905,646	1364	290	-60	1000		1000	JSLA	03/15/17	03/16/17
CMM-211	672,652	3,905,622	1364	290	-60	1025		1025	JSLA	02/20/17	02/21/17
CMM-212C	672,784	3,905,574	1347	290	-60	1427	1427		JSLA	12/19/16	01/13/17
CMM-213	672,255	3,905,799	1349	290	-60	605		605	JSLA	03/24/17	03/26/17
CMM-214	672,340	3,905,768	1363	290	-60	700		700	JSLA	03/07/17	03/08/17
CMM-215	672,443	3,905,731	1374	290	-65	920		920	JSLA	03/18/17	03/20/17
CMM-216	672,558	3,905,689	1370	290	-60	1115		1115	JSLA	03/11/17	03/14/17
CMM-217	672,632	3,905,662	1364	290	-60	1045		1045	JSLA	02/02/17	02/07/17
CMM-218	672,804	3,905,599	1352	290	-60	1025		1025	JSLA	01/23/17	02/01/17
CMM-219C	672,389	3,905,783	1361	290	-65	938	578	360	JSLA	03/02/17	03/25/17
CMM-220C	672,632	3,905,694	1365	290	-60	1254	594	660	JSLA	01/17/17	01/21/17
CMM-221	672,438	3,905,797	1358	290	-65	880		880	JSLA	02/03/17	02/05/17
CMM-222	672,644	3,905,722	1364	290	-60	1200		1200	JSLA	03/06/17	03/09/17
CMM-223A	672,772	3,905,676	1366	290	-60	1600		1600	JSLA	03/29/17	04/02/17
CMM-224	672,538	3,905,795	1376	290	-60	1000		1000	JSLA	03/20/17	03/23/17
CMM-225	672,854	3,905,678	1357	290	-60	1300		1300	JSLA	02/02/17	02/07/17

Hole Id	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Drilling Start	Date Drilling Finished
CMM-226	672,523	3,905,831	1377	290	-65	890		890	JSLA	03/24/17	03/28/17
CMM-227	672,628	3,905,793	1364	290	-60	1250		1250	JSLA	02/26/17	02/28/17
CMM-228	672,836	3,905,705	1359	290	-60	1500		1500	JSLA	02/11/17	02/17/17
CMM-229C	672,537	3,905,859	1377	290	-60	1477	837	640	JSLA	2/9-2/12	2/10-2/16
CMM-230C	672,644	3,905,820	1366	290	-60	1177	517	660	JSLA	2/8-2/14	2/9-2/17
CMM-231	672,536	3,905,891	1372	290	-60	880		880	JSLA	02/09/17	02/15/17
CMM-232	672,683	3,905,838	1365	290	-60	1000		1000	JSLA	02/18/17	02/22/17
CMM-233C	672,843	3,905,780	1358	290	-60	1087	962	125	JSLA	01/25/17	01/31/17
CMM-234	672,645	3,905,884	1364	290	-60	825		825	JSLA	02/17/17	02/18/17
CMM-235	672,802	3,905,827	1365	290	-60	1200		1200	JSLA	02/19/17	02/22/17
CMM-236	672,531	3,905,958	1350	290	-60	860		860	JSLA	03/26/17	03/28/17
CMM-237C	672,695	3,905,899	1364	290	-60	1168	608	560	JSLA	2/9-2/12	2/11-2/14
CMM-238C	672,800	3,905,860	1367	290	-60	1378	1133	245	JSLA	02/01/17	02/09/17
CMM-241	672,845	3,905,876	1364	290	-60	1045		1045	JSLA	05/01/17	05/06/17
CMM-242C	672,814	3,904,177	1280	290	-60	1496	1496		South Domes	02/26/17	03/05/17
CMM-243C	672,729	3,904,211	1282	290	-60	453	453		South Domes	03/06/17	03/08/17
CMM-243C-A	672,731	3,904,209	1282	290	-60	1247	1247		South Domes	03/10/17	03/14/17
CMM-244C	372,794	3,904,245	1284	290	-60	1737	1737		South Domes	03/15/17	03/22/17
CMM-246C	672,700	3,904,585	1323	290	-55	767	767		South Domes N	04/03/17	04/05/17
CMM-248C	672,837	3,904,557	1335	290	-55	715	714.5		South Domes N	04/06/17	04/08/17
CMM-250C	672,700	3,904,675	1340	200	-60	1057	1057		South Domes N	04/09/17	04/12/17
CMM-252C	672,890	3,904,720	1330	290	-60	1017	1017		621 Zone	04/13/17	04/16/17
CMM-254C	673,000	3,904,657	1333	290	-60	927	927		621 Zone	04/17/17	04/22/17
CMM-255C	672,890	3,904,645	1333	290	-60	927	927		South Domes N	04/23/17	04/27/17
CMM-256	672,458	3,905,693	1378	290	-60	520		520	JSLA	04/25/17	04/27/17
CMM-257C	672,704	3,904,621	1333	270	-55	976	976		South Domes N	04/27/17	05/02/17
CMM-258C	672,733	3,904,701	1334	280	-60	987	987		South Domes N	05/02/17	05/07/17
CMM-259C	672,453	3,905,700	1378	288	-61	1152	672	480	JSLA	05/15/17	05/19/17
Totals (ft)						**160,341**	**47,131**	**113,210**			
Totals (m)						**48,872**	**14,366**	**34,506**			
Water Well Test Drill Holes											
CMM-247 / W1	672,830	3,905,830	1356	n/a	-90	1505		1505	JSLA	04/02/17	04/06/17
CMM-249 / W2	672,920	3,904,220	1274	n/a	-90	1405		1405	South Domes	04/07/17	04/10/17
CMM-251 / W3	672,560	3,904,390	1286	n/a	-90	1505		1505	South Domes	04/10/17	04/12/17
CMM-253 / W4	672,380	3,905,595	1380	n/a	-90	1205		1205	JSLA	04/13/17	04/15/17
Totals (ft)						**5,620**	**-**	**5,620**			
Totals (m)						**1,713**	**-**	**1,713**			
Metallurgical Drill Holes											
Met-033	672,137	3,905,520	1345	90	-50	964	964		JSLA	04/19/17	04/28/17
Met-161	672,614	3,905,344	1358	290	-60	952	951.5		JSLA	04/28/17	05/06/17

Hole Id	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Drilling Start	Date Drilling Finished
Met-858	672,723	3,905,542	1357	270	-65	440		440	JSLA		
Met-212	672,784	3,905,574	1347	290	-60	1027	1027		JSLA	05/07/17	05/14/17
Totals (ft)						**3,383**	**2,943**	**440**			
Totals (m)						**1,031**	**897**	**134**			
Clay Test Drill Holes											
BH-01	671,985	3,905,351	1315	n/a	-90	150	150		West	05/25/17	05/26/17
BH-02	671,842	3,905,419	1330	n/a	-90	150	150		West	05/24/17	05/25/17
BH-03	671,921	3,905,567	1320	n/a	-90	150	150		West	05/23/17	05/24/17
BH-04	671,893	3,905,865	1340	n/a	-90	150	150		West	05/22/17	05/23/17
Totals (ft)						**600**	**600**	-			
Totals (m)						**183**	**183**	-			

10.5 NewCastle 2017 RAB Drilling

In January and February 2017, a Reverse Air-Blast (RAB) drill program was carried out across two areas of mine back-fill and dump material: the mine back-fill material in the former JSLA pit, and the South Dump area. The goal of the program was to test the mine back-fill and dump material for low-grade mineralized material that could be added as run-of-mine (ROM) material.

A total of 6,989ft (2,130m) in 273 RAB drill holes were drilled. A total of 242 drill holes were completed in the JSLA pit back-fill at 100ft (30m) spacing, and 31 drill holes were completed adjacent to an access road on the South Dump. RAB drill holes were drilled to a depth of between 18ft and 30ft depending on the stability of the dump material, and to recover enough material to sample.

10.6 Equinox Gold Phase III 2017/2018 Drilling

In late 2017, Equinox Gold 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 diamond core and RC holes were drilled. Table 10.5 is a summary of the drilling program.

Table 10-5 - Phase III 2017/2018 Drilling Program Summary

Hole ID	East NAD83m	North NAD83m	Elev. (m)	Azimuth degrees	Inclination degrees	Drilled (ft)	Core Drilled (ft)	RC Drilled (ft)	Area	Date Start	Date Finish
CMM-260C	**672315**	3905710	1366	290	-75	**1317.0**	1317.0	0	JSLA	9/5/2017	9/15/2017
CMM-261C	672792	3906031	1372	110	-48	**1432.0**	1432.0	0	Orobelle	9/16/2017	9/25/2017
CMM-262C	672456	3905667	1349	290	-73	**1709.0**	1349.0	360	JSLA	9/22/2017	9/25/2017
CMM-263C	672239	3905968	1354	290	-50	**1097.0**	1097.0	0	JSLA	9/25/2017	9/30/2017
CMM-264	672460	3905888	1354	290	-55	**1000.0**	0.0	1000	JSLA	9/28/2017	10/8/2017
CMM-265C	672809	3905269	1301	290	55	**917.0**	917.0	0	621 South Dump	10/8/2017	10/13/2017
CMM-266	672070	3905796	1318	290	50	**850.0**	0.0	850	West JSLA	10/10/2017	10/11/2017
CMM-267	672147	3905613	1317	290	60	**500.0**	0.0	500	West JSLA	10/11/2017	10/12/2017
CMM-268C	672234	3906061	1340	275	65	**1307.5**	1307.5	800	North JSLA	10/13/2017	10/31/2017
CMM-269C	673074	3905046	1301	290	50	**1017.0**	1017.0	0	West JSLA	10/14/2017	10/18/2017
CMM-270	672365	3905449	1390	290	60	**400.0**	0.0	400	West JSLA	10/18/2017	10/18/2017
CMM-271C	672442	3905344	1365	350	54	**976.0**	976.0	0	West JSLA	10/18/2017	10/23/2017
CMM-272C	672199	3905496	1331	290	55	**312.0**	312.0	0	West JSLA	10/24/2017	10/26/2017
CMM-273C	672496	3906501	1412	110	60	**958.0**	958.0	0	North Jumbo	11/1/2017	11/6/2017
CMM-274C	672301	3905738	1364	225	70	**1208.0**	1009.0	199	North JSLA	11/1/2017	11/11/2017
CMM-275C	672357	3905514	1415	290	65	**1320.5**	484.0	875	Big Chief	11/4/2017	11/16/2017
CMM-276C	673067	3906962	1448	270	60	**1128.0**	1128.0	0	Orobelle North	11/10/2017	11/15/2017
CMM-277C	673115	3907008	#REF!	270	50	**947.0**	947.0	0	Orobelle North	11/16/2017	11/20/2017
CMM-278C	672727	3906500	1407	290	50	**1097.0**	1097.0	0	Jumbo North	11/17/2017	11/27/2017
CMM-279C	673163	3907073	1462	270	50	**1407.0**	1407.0	0	Orobelle North	11/20/2017	12/3/2017
CMM-280C	672712	3906601	1424	290	50	**807.0**	807.0	0	Jumbo North	11/27/2017	11/30/2017
CMM-281C	672555	3906659	1427	275	60	**1215.0**	1215.0	0	Jumbo North	12/1/2017	12/7/2017
CMM-282C	673380	3906457	1503	180	55	**1065.0**	1065.0	0	East Ridge	12/4/2017	12/8/2017
CMM-283C	672741	3906459	1399	225	55	**1437.0**	1437.0	0	Jumbo North	12/7/2017	12/12/2017
CMM-284C	673506	3906456	1461	240	50	**952.0**	952.0	0	East Ridge	12/8/2017	12/13/2017
CMM-285C	672732	3906013	1369	225	55	**1404.0**	1404.0	0	JSLA North	12/13/2017	12/20/2017
CMM-286C	673313	3906998	1476	245	65	**1667.0**	1667.0	0	NE Extension	12/14/2017	12/19/2017

10.7 Equinox Gold 2018 RC Drilling

A 53-hole RC program totaling 9680ft in the JSLA back-fill, down to the 4300' elevation and a depth of approximately 182 ft on average, has been completed. National EWP mobilized a Shramm T450 reverse circulation (RC) drilling rig on January 16th, and drilling began January 18th, 2018. By the end of January, 41 holes for a total of 7,700 ft had been completed. On January 28th, approval was given to complete all 53 holes in the program. The program was successfully completed as planned on February 1st, 2018.

10.8 Equinox Gold 2018 RAB Drilling

A RAB (reverse air blast) drill hole program designed to test the top 20ft of the JSLA back-fill material, planned for a total of 825 RAB holes, began on February 12th, and was completed on March 7th with a total of 809 holes completed at 50 ft spacing; and an additional 32 holes completed over an infill grid on 20ft spacings centered on RC hole RC18-1-2.

The RAB program was extended to include drilling portions of the north and south waste dumps. On the north dump an additional 46 holes were completed as an initial test of ROM material. An additional 107 samples were collected across two lines at the southern end of the south dump – again testing ROM potential and following up on the 2017 RAB ROM test program.

The extended RAB drilling program was completed on Wednesday March 7th for a total of 995 drill holes. The Ledcor rig (an Atlas Copco DM60) was demobilized from site on March 9th.

10.9 Drilling Methods & Equipment Used

10.9.1 Diamond Core Drilling

Phase II diamond core drilling was conducted by Major Drilling of Salt Lake City, Utah. Drilling utilized up to two Boart-Longyear LF-90 drill rigs with conventional PQ and HQ tooling. Diamond tipped face-discharge drill bits were used to increase productivity and recovery. In late January 2017, drilling was converted to HQ3 drill bits and triple tube tooling using inner core tube splits to further improve recovery and preserve core condition. The metallurgical holes were drilled with conventional PQ tooling using face-discharge diamond core drill bits. All drill depths and core runs were recorded in feet.

10.9.2 RC Drilling

Phase II reverse circulation (RC) drilling was conducted by the Layne Christensen Company (Layne) of Chandler, Arizona and National EWP (National) of Elko, Nevada.

Layne provided four RC rigs that typically employed 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.

National employed a single Shramm 685 drill rig using Symmetrix casing and center return, face sampling air hammers for the deeper RC holes. The Symmetrix RC casing system was used in the drill holes where greater than 400ft of backfill was anticipated; the deepest cased hole utilized up to 800ft of Symmetrix casing within the JSLA pit.

All RC drill runs were measured in feet.

10.9.3 RAB Drilling

The Reverse Air-Blast (RAB) drill program utilized an Atlas-Copco D-65 blast hole drill rig from Ledcor CMI Inc. of Reno, Nevada. The RAB drilling used a downhole air-blast button or tricone bit to drill a hole through unconsolidated, dry material. Material with a high clay content used the tricone; rocky materials required the use of the button bit. Drilling foam was added to lubricate the bit and suppress dust. 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.

10.9.4 Downhole Surveys

Downhole surveys for all the drilling covered by this report in 2016 and 2017 was 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); readings were collected at 50ft (15.3m) intervals inside the drill string. The SRG corrects at the time of data collection for the 11.51^0 East magnetic declination. Outputs were provided on paper and as digital files. No down hole surveys were performed on the RAB drilling due to the shallow nature of the holes and they are assumed to be vertical.

10.9.5 Drill Collar Coordinates

Drill collars were located using hand held Garmin or Trimble GPS receivers. After drill holes were completed the drill collar locations were recorded using either hand-held Garmin or Trimble GPS receivers. Periodically throughout the program, and at the end of the program, Mineral Exploration Services of Reno, Nevada collected high precision differential GPS location for existing drill collars using a Trimble R2 dual frequency GPS with a horizontal accuracy of 2cm.

Fore- and back-sights for drilling azimuth were located using hand-held Brunton compass employing a magnetic declination correction of 11.51^0. An azimuth orientation line was sprayed on the ground with orange fluorescent paint prior to arrival of the drill rig. Inclination was checked by either hand-held inclinometer or by Brunton compass inclinometer.

10.9.6 Core Photos

All core was photographed prior to cutting using high definition digital cameras. 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, and a back-up copy is kept at the Henderson, Nevada office.

10.9.7 Geotechnical Logging

Geotechnical logging was performed on all core drilled in 2016 and 2017 and include footage drilled, core recovery, RQD, fracture frequency and joint condition. Each category value was determined for each core tube pulled, or block to block. Data was recorded directly onto Excel spreadsheets via laptop computers.

All geotechnical logging was measured in feet.

10.9.8 Geologic Logging

Geologic logging was carried out on all core and reverse-circulation chips generate by the 2016/17 drill program covered by this report. Logging was carried out either at the core logging facility at the Castle Mountain site or at the Henderson office. Geologic data was recorded directly via laptop computer on to an Excel spreadsheet based core logging template developed specifically by NewCastle geologists for the Castle Mountain Project. Principal data fields collected included lithology, metallurgy (FeOx, MnOx, Py, Au), alteration (silica, clay, chlorite), structure (fractures, faults), veins, and point data.

All core logging was measured in feet.

10.9.9 Specific Gravity

Bulk specific gravity was measured for 186 samples from multiple drill holes; the intervals selected were based upon lithologies for which little or no prior specific gravity measurements existed. Measurements were conducted at the Castle Mountain site core logging facility by NewCastle's geologists. 4-6inch (10-15cm) long core samples were tested using the water immersion method after coating with paraffin wax.

10.9.10 Core Storage

Phase I drill core from 2016 was stored at the Henderson office until July 2017 whereupon it was moved by NewCastle staff to the newly constructed core storage facilities at the Castle Mountain Mine site.

All Phase II core has been stored in the core storage facilities at the Castle Mountain Mine site aside from that shipped to analytical laboratories of analysis.

10.10 Drilling Results and Assays

10.10.1 Drill Results

The Phase I/II definition and exploration drill program which began in June 2016 and ended in June 2017 was successful in:

1. Identifying gold mineralization within areas previously classified as ‘Waste’ within the previously modeled pit shell,

2. Intersecting gold mineralization outside the limits of the previously modeled pit shell (both laterally and at depth),

3. Encountering encouraging zones of high-grade gold mineralization at both JSLA and South Domes, and

4. Gaining a better understanding of the deposit geology and potential controls on gold mineralization through the drilling of angled diamond core holes across perceived steeply dipping, NE-SW ‘gold trends’, leading to the development of a new geological model. The distribution of gold mineralization is controlled by discrete structures and the margins of facies contacts within these trends, and the presence of felsic intrusive bodies and associated breccias.

10.10.1.1. Main Oro Belle Trend (OBT)

Assay results from Phase I/II drilling at the southern end of the Oro Belle Trend (both under and south of the previously mined JSLA backfilled pit) returned many encouraging intercepts including:

- CMM-087: 50.3m @ 3.70 g/t Au, incl. 9.1m @ 17.59 g/t Au
- CMM-161: 103.6m @ 2.77 g/t Au (uncut)/ 1.60 g/t Au (cut), incl. **15.2m @ 16.47 g/t Au** (uncut)/ 8.53 g/t Au (cut), incl. 3.0m @ 73.95 g/t Au (uncut)/34.29 g/t Au (cut)

- CMM-161A: 179.8m @ 1.01 g/t Au, incl. **19.8m @ 5.49 g/t Au**, incl. 3.0m @ 24.05 g/t Au
- CMM-175C: **12.8m @ 31.32 g/t Au** (uncut)/ 14.20 g/t Au (cut), incl. 3.4m @ 99.68 g/t Au (uncut)/ 34.29 g/t Au (cut) (Lucky John Zone)
- CMM-180C: 196.9m @ 1.13 g/t Au, incl. 44.0m @ 2.11 g/t Au
- CMM-190: 196.6m @ 0.36 g/t Au, incl. 33.5m @ 1.39 g/t Au
- CMM-195: **29.0m @ 31.19 g/t Au** (uncut)/ 10.63 g/t Au (cut), incl. 9.1m @ 93.95 g/t Au (uncut)/ 84 g/t Au (cut) (Lucky John Zone)
- CMM-195: 126.5m @ 2.26 g/t Au, incl. 6.1m @ 5.98 g/t Au and 6.1m @ 7.50 g/t Au
- CMM-204: 47.2m @ 1.74 g/t Au, incl. 3.0m @ 17.01 g/t Au

Holes CMM-075C and CMM-180C are highlighted in Figure 10.5 and CMM-195 is shown in Figure 10.6.

Of interest are the results from hole CMM-195 which was targeting the steeply dipping fault structure associated with the Lucky John Zone, approximately 65 metres down-dip from CMM-054 and approximately 150 metres north of CMM-060. The hole was successful in extending the high-grade gold mineralization to depth, where it remains open along strike to the north and south. CMM-195 also intersected a new zone of gold mineralization within the underlying footwall andesite sequence (FW Zone). The FW Zone lies 50m below the currently modeled mineral resource in an area of very sparse drilling, is open in all directions, and represents an opportunity for further follow-up. Figure 10.5.

10.10.1.2. South Domes Target

Phase I/II drilling at the South Domes target began with two key diamond core cross-sections to assess the southern and northern portions of the zone and resulted in the discovery of previously unknown quartz-feldspar porphyritc bodies with proximal hydrothermal breccias near the southern end of the zone. Subsequent step-out drilling for an additional 400 ft. (120 m) to the south extended the limits of gold mineralization outside the 2015 modeled pit shell and encountered broad zones of gold mineralization, and also high-grade gold intercepts associated with steeply dipping structures/breccias. Highlights include:

- CMM-079: 213.70m @ 1.09 g/t Au, incl. 16.00m @ 4.14 g/t Au
- CMM-111: 135.90m @ 1.73 g/t Au, incl. 30.00m @ 3.32 g/t Au
- CMM-122C: 192.9m @ 1.07 g/t Au, incl. 77.3m @ 1.98 g/t Au, incl. 4.6m @ 20.10 g/t Au
- CMM-128: 94.5m @ 1.25 g/t Au, incl. 15.2m @ 3.37 g/t Au
- CMM-129: **275.8m @ 1.63 g/t Au**, incl. 204.2m @ 2.05 g/t Au, incl. **48.8m @ 5.43 g/t Au**
- CMM-130C: **102.7m @ 2.76 g/t Au** (uncut)/2.67 g/t Au (cut), incl. **41.5m @ 6.15 g/t Au** (uncut)/5.92 g/t Au (cut), incl. 4.9m @ 28.30 g/t Au (uncut)/26.29 g/t Au (cut)
- CMM-132: 67.1m @ 1.07 g/t Au, incl. 12.2m @ 2.91 g/t Au

- CMM-133: 38.1m @ 2.10 g/t Au, incl. 9.1m @ 5.03 g/t Au
- CMM-134C: 113.7m @ 0.74 g/t Au, incl. 19.7m @ 2.59 g/t Au, incl. 3.0m @ 10.08 g/t Au
- CMM-242C: 25.6m @ 3.36 g/t Au, incl.6.4m @ 8.14 g/t Au

Hole CMM-130C is shown in Figure 10.4.

10.10.1.3. Exploration Targets

Phase I drilling to the north (OB NEX) and northeast (East Ridge) of the main mineral resource area also indicates the potential for extending the main gold trends along strike and represents a target for further exploration work. Highlights include:

OB NEX

- CMM-119: 167.8m @ 0.49 g/t Au, incl. 12.3m @ 1.12 g/t Au
- CMM-120: 66.8m @ 0.45 g/t Au
- CMM-124: 99.1m @ 0.56 g/t Au, incl. 15.2m @ 1.31 g/t Au

East Ridge

- CMM-109: 18.3m @ 1.95 g/t Au, incl. 10.7m @ 3.08 g./t Au
- CMM-112: 56.4m @ 0.48 g/t Au
- CMM-115: 50.3m @ 0.66 g/t Au, incl. 6.1m @ 3.23 g/t Au

10.10.2 Leapfrog Modelling

Beginning in April 2017, NewCastle geologists embarked on an intense geologic modelling program using Leapfrog 3-dimensional modelling software. Approximately 1,980 drill holes from all prior programs were incorporated into a newly interpreted database using the adopted core logging template. Viceroy, Castle Mountain Mining paper logs, and NewCastle Gold digital logs were reviewed and reinterpreted for the principal features of interest including lithology, structure, alteration, metallurgy and gold assay results.

By the end of June 2017, NewCastle geologists had created a 3-dimensional model that was incorporated into the resource and block modelling program.

Figures 10.4, 10.5, and 10.6, show highlighted sections from the Phase I, 2016 and Phase II, 2016-17 drill program; these sections combine the geologic modelling with drill results and gold assays. The sections clearly illustrate the relationships between property-scale geology and economic mineralization.

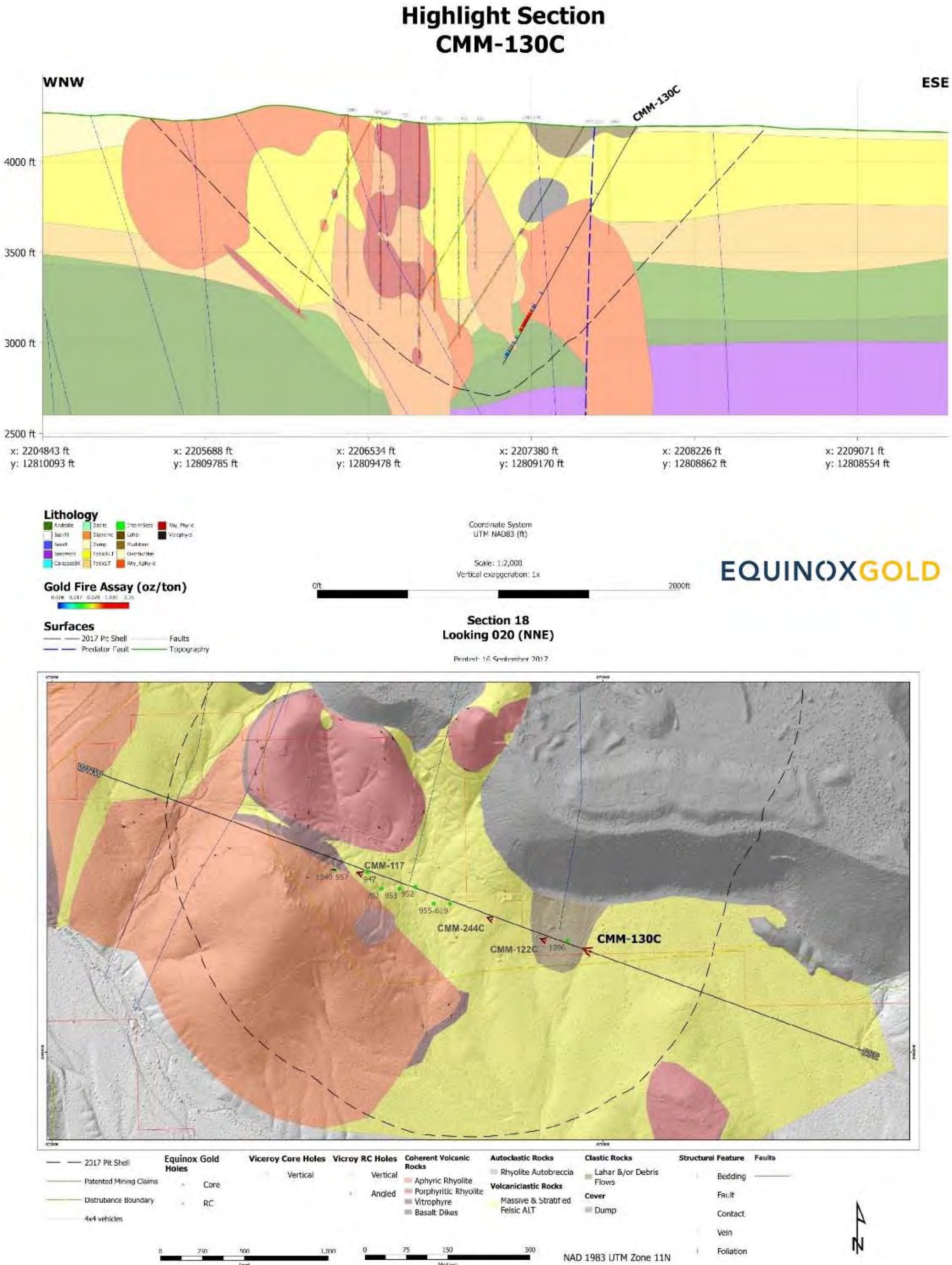

Figure 10-4 – Section 18 Cross Section

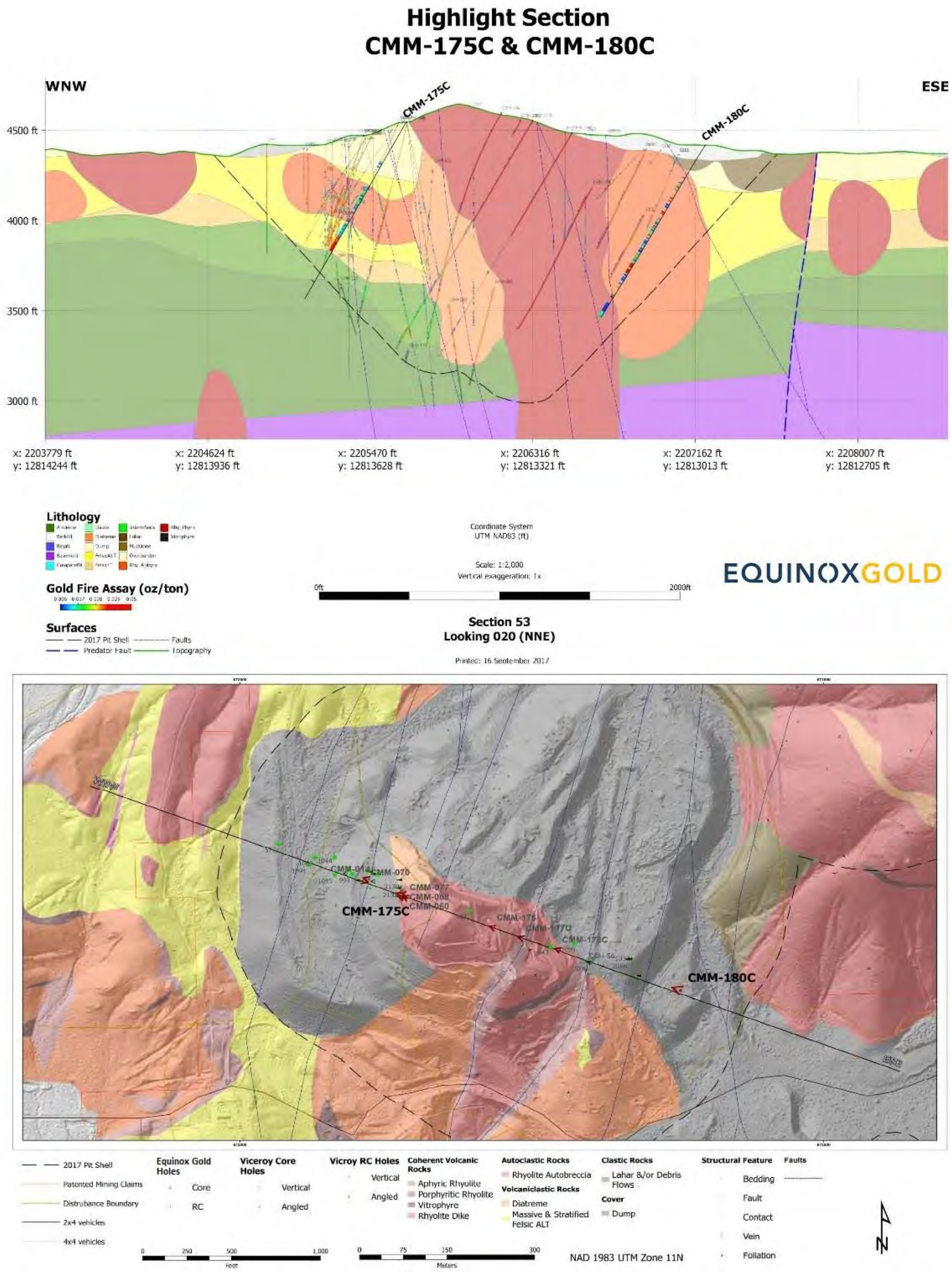

Figure 10-5 – Section 53 Cross Section

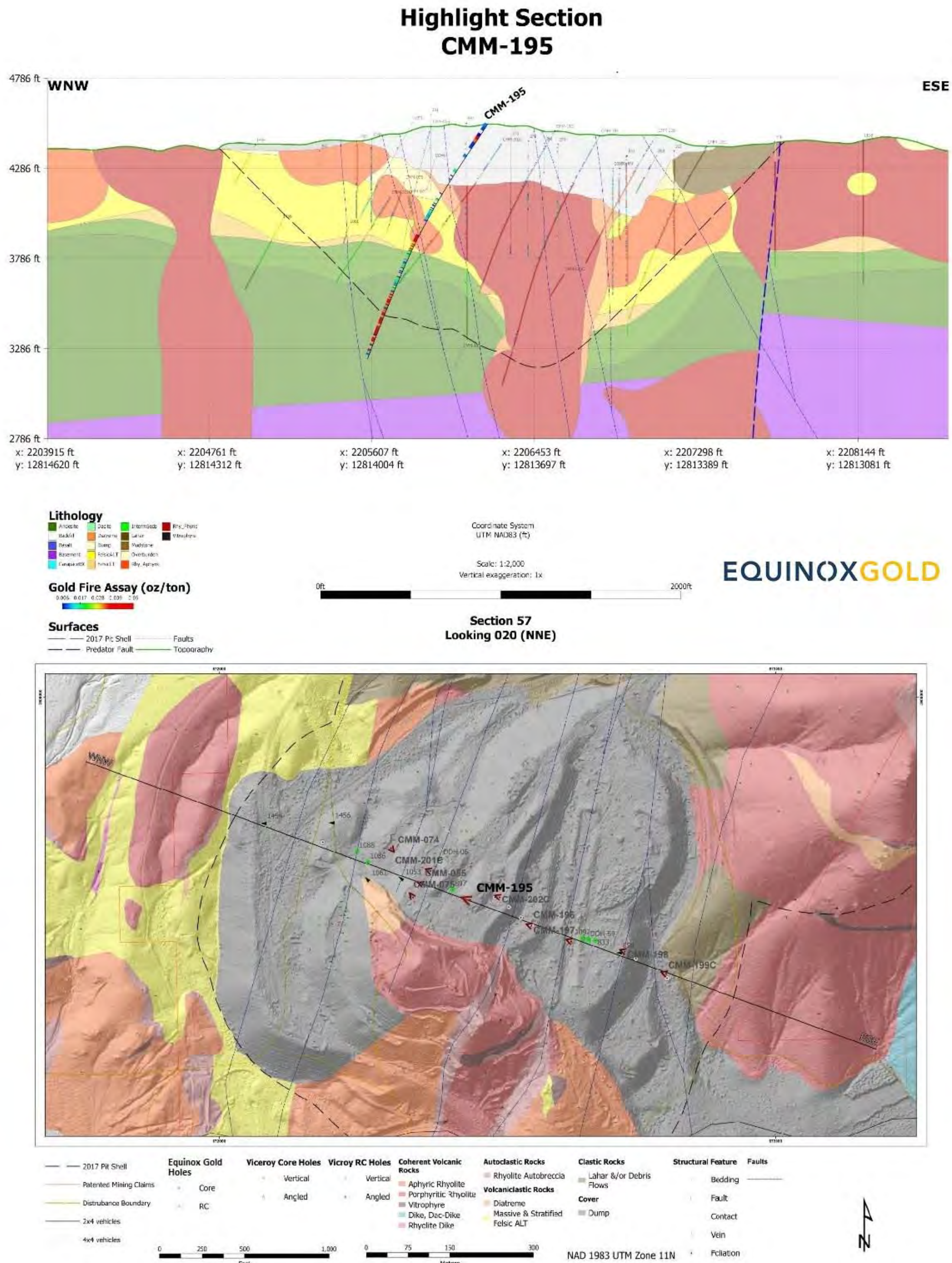

Figure 10-6 – Section 57 Cross Section

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

Discussion of sample preparation and assay procedures, assay QA/QC program results, and security is divided into sections by drill campaign, as defined in Section 10, Drilling. The discussion below focuses on gold assays. Silver and other elements were analyzed for certain drill campaigns but are not considered relevant to the mineral resource at Castle Mountain.

11.1 Viceroy

Temkin (2012) summarized the sampling, analysis, and QA/QC employed during the Viceroy drilling campaigns. MTS reviewed original assay certificates for Viceroy drill holes, but did not review any original information related to sampling, security, or QA/QC. The following description was modified by MTS from Temkin (2012).

11.1.1 Sampling and Security

All core drilled on the Project was completed with either HQ (2.5 in/63.5 mm) or NQ (1.875 in/47.6 mm) diameter tools. Prior to sampling, core was sawn into equal halves utilizing a standard lapidary blade, then measured and marked into five-foot (1.5 m) intervals. Individual samples were prepared utilizing one-half of the sawn core, and collected systematically on five-foot (1.5 m) intervals over the entire length of the drill hole.

Conventional rotary drill holes were drilled utilizing four and one-half inch (11 cm) diameter pipe and five and one-quarter inch (13 cm) diameter bits. The reverse-circulation holes were drilled utilizing five and three-quarter inch (15 cm) diameter pipe and six and three-eighths (16 cm) diameter bits. Samples in both conventional rotary and reverse-circulation holes were collected at five-foot (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 fifteen pounds (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 fifteen pounds (7 kg) of material. Individual samples were collected in five-gallon (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 tied and laid out to dry. Reference samples were collected in plastic containers for logging purposes.

All drill samples were either retrieved from the Project by the assay laboratory or shipped directly to the lab via a contract shipping company.

11.1.2 Analyses

Most Viceroy drill hole samples were analyzed by Legend, Inc. (Legend) in Reno, Nevada. Sample preparation consisted of crushing the entire sample to -10 mesh, splitting out a 200 gm (5.8 oz) subsample, and grinding the subsample to greater than 80% -150 mesh. Gold and silver were determined by fire assay on a one-assay ton subsample followed by atomic absorption finish. Assay values were reported in oz/ton units and the lower detection limits for gold and silver were 0.001 and 0.050 oz/ton respectively. Assays returning gold values greater than 0.100 oz/ton were reassayed by fire assay on a one-assay ton subsample with a gravimetric finish.

Some Viceroy drill hole samples were 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 atomic absorption 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 Au.

Legend and RMG were not certified or an accredited ISO 17025 laboratory for analysis by fire assay with atomic absorption (AA) finish at the time they were performing analytical services for the Project.

All coarse rejects were stored at the labs for 90 days, then discarded. All pulps were stored for one year at the lab, then returned to the Project site for long-term storage. 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

Routine duplicate analyses were performed on conventional rotary, reverse-circulation and core 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 approximately 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 Au) initial samples reported precision of ±10%. Medium-grade (0.010 to 0.100 oz/ton Au) initial samples reported precision of ±17%. And 90% of the high-grade (> 0.100 oz/ton Au) initial samples reported precision of ±25%.

Check assay samples submitted to other commercial labs and the Castle Mountain Mine lab did not indicate any problems with Legend's original assays.

11.2 NewCastle Core and RC

NewCastle sampling and assaying procedures have remained consistent through the drilling campaigns listed in Section 10, Drilling. NewCastle's sampling, assaying, and QA/QC procedures from the 2017 and 2018 drilling campaigns are summarized in sections 11.2.1 to 11.2.4. Significant changes from previous NewCastle procedures include:

- Drill core is sawn in half, sampled, and bagged at the core logging facility on site where previously core was sent to the ALS facility in Reno, Nevada where it was sawn in half and sampled.

- Drill samples are assayed for gold and gold cyanide solubility only where in previous campaigns, drill samples were also assayed for silver.

QA/QC results from previous campaigns are summarized from RPA (2013), RPA (2014) and Gray et al. (2016).

11.2.1 Sampling and Security

NewCastle personnel pick up core from the drill on a daily basis and transport the core to its core facility on site. Most drill holes were completed using HQ (2.5 in/63.5 mm) tools. Where necessary NQ (1.875 in/47.6 mm) diameter tools were used to complete the drill hole to target depth. All core is thoroughly washed and photographed. NewCastle geologists determine sample intervals based on lithology and mineralization contacts and record them on a sample cut sheet. Sample intervals are nominally 5.0 ft in length, but do not cross geological contacts and are typically range between 2.0 and 7.0 ft. Aluminum tags labeled with the beginning and ending footage are stapled into the core box to denote sample intervals.

RC drill cuttings are collected in sample bags by the drill contractor on continuous five-foot (1.5 m) intervals from a rotary cyclone splitter. RC sample weights from the 2017 drill program typically range from 2.0 to 10.0 kg and average about 6.0 kg. A reference subsample of each interval is placed in a chip tray for logging purposes. 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, NewCastle personnel collect the samples from the field and place them in bins at the secured laydown yard. Reference chip trays are transported to the site logging facility for geologic logging with a binocular scope.

Drill samples are retained at the secured site logging facility until they are picked up by the commercial lab at site at regularly scheduled intervals and transported to the laboratory facility in Reno or Elko, Nevada.

11.2.2 Analyses

Drill samples were assayed are either the ALS facility in Reno, Nevada or the Inspectorate facility in Sparks, Nevada. Check assays were completed at American Assay Laboratories in Sparks, Nevada.

11.2.2.1. ALS

Samples submitted to the ALS facility are first dried and then crushed to 70% passing 2 mm. A 250 g sub-sample is taken from the crushed material and pulverized to 85% passing 200 mesh (75 µm) (ALS Method PREP-31). A 30 g aliquot of pulverized material (pulp) is then assayed for gold and silver by conventional fire assay methods followed by AA analysis (ALS Method Au-AA23). Gold assays returning greater than 10 g/t Au (0.292 oz/st Au) are reassayed by fire assay and gravimetric finish on a separate 30 g aliquot (ALS Method Au-GR21). Gold assays returning greater than 0.2 g/t Au 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 AA (ALS Method Au-AA13).

11.2.2.2. Inspectorate

Samples submitted to the Inspectorate facility are dried and crushed to 70% passing 2 mm. A 250 g sub-sample is taken from the crushed material and pulverized to 85% passing 200 mesh (75 µm) (Inspectorate Method PRP70). A 30 g aliquot of pulverized material (pulp) is then assayed for gold by conventional fire assay methods followed by AA analysis (Inspectorate Method FA430). Gold assays returning greater than 10 g/t Au (0.292 oz/st Au) are reassayed by fire assay and gravimetric finish on a separate 30 g aliquot. Gold assays returning greater than 0.2 g/t Au 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 AA (Inspectorate Method CN403).

11.2.2.3. American Assay

Check assays were completed using conventional fire assay methods on a one assay ton aliquot with final gold analysis by AA. Gold assays returning greater than 10 g/t Au (0.292 oz/st Au) were reanalyzed by fire assay with gravimetric finish.

11.2.3 Density

A total of 647 specific gravity measurements have been collected by NewCastle during the 2013 to 2017 drill campaigns to provide density data for mineral resource estimation purposes. A total of 461 measurements from 2013 to 2016 drill core were conducted by ALS on 4 to 6 inch long pieces of split (half-core) HQ (2.50 inch diameter) or PQ (3.35 Inch diameter) size core using the water immersion method after coating with paraffin wax (ALS Method OA-GRA08a). In 2017, 186 specific gravity measurements were determined on site at the Castle Mountain logging facility by NewCastle staff using the paraffin wax coated water immersion method. Measurements were determined on 4 to 6 inch pieces of split HQ or PQ size core selected from 2017 drill intervals based on logged lithology. MTS compared measurements determined by NewCastle to those determined by ALS and found them to be comparable.

11.2.4 Quality Assurance and Quality Control

11.2.4.1. 2013

QA/QC programs performed by NewCastle in 2013 are summarized in RPA (2013). RPA assessed Castle Mountain's QA/QC program and found it to be industry-standard with an acceptable rate of insertion for Certified Reference Materials (CRM) and check assays. The results of the check assays showed good reproducibility with no significant grade bias between the original and the check laboratory. The CRM results showed that ALS assays were of acceptable accuracy. The blank results showed no significant carryover contamination in the ALS assays. RPA considered that the QA/QC program for the 2013 drill campaign was adequate and assay results from the campaign are suitable for use in resource estimation.

11.2.4.2. 2014/2015

Details on QA/QC programs for the 2014 and 2015 drill campaigns can be found in Gray et al. (2016). Gray et al. found that NewCastle conducted an external, industry-standard QA/QC program for its 2014 and 2015 drill campaigns consisting of the insertion of blanks and CRMs into the sample stream and the analysis of field duplicate and pulp duplicate samples.

The insertion rate for CRM, blanks, field duplicates and pulp duplicates was acceptable and the results of the check assays showed good reproducibility with no significant grade bias between the original and check assay laboratories. The insertion of CRM's showed that the accuracy of ALS and Inspectorate assays was acceptable. Results from the insertion of blanks were also within acceptable limits. Gray et al. found that the QA/QC program for the 2014-2015 drill programs by NewCastle did not identify any grade bias and that the assay results are appropriate for use in resource estimation.

11.2.4.3. 2016/2017/2018

NewCastle employed a QA/QC program during the 2016, 2017 and 2018 drill campaigns that included the analysis of CRMs, blanks, RC field duplicates, and check assays. CRMs, blanks and duplicates were inserted regularly in the sample stream, and a random selection of samples from mineralized intervals were submitted to an umpire laboratory for check assay at the completion of each drill campaign. NewCastle used CRMs sourced from Mineral and Exploration Geochemistry (MEG) of Reno, Nevada and blanks sourced locally from coarse crushed construction rock.

NewCastle geologists evaluated the control sample results from the Phase I drilling program in 2016 and reported that all drill sample submittals passed control limits for accuracy and precision.

For the Phase II drilling program in 2016-2017, a total of 1,538 standards, 691 blanks, 2,001 duplicates, and 415 check assays were analyzed. This represents a control sample insertion rate of approximately 11%.

Control sample results were evaluated by NewCastle geologists for each batch of project samples submitted to the assay laboratory. When control sample results returned values outside acceptable limits, the assay laboratory was contacted and the batch of samples was reassayed. All assay batches submitted during the Phase II program were evaluated using this protocol and the original or corrected assays were loaded to the project database.

MTS reviewed a compilation of the Phase II control sample results and found the assay accuracy and precision to be acceptable for purposes of resource estimation. No significant bias was observed in the CRM results for gold. Check assays showed no significant bias between the ALS and Inspectorate original assays and the AAL check assays. No significant carryover contamination was observed in the blank results.

Precision of the RC field duplicate results was marginal but acceptable. MTS considers assay precision to be acceptable for field duplicates where 90% of the duplicate pairs display less than ±30% absolute relative difference (ARD). ARD is calculated as the absolute value of the pair difference in grade, divided by pair's mean grade, and ARD values range between 0 and 200%. However, in cases where high-grade gold is known to occur, MTS considers agreement within 50% to indicate good performance. The calculated precision of the 2017 field duplicate pairs was ±75% ARD at the 90th percentile, marginally outside the threshold of 50% that indicates good performance.

For the Phase III drilling program, a total of 216 standards, 218 blanks, and 42 duplicates were analyzed. This represents a control sample insertion rate of approximately 5%.

MTS reviewed a compilation of the Phase III control sample results and found the assay accuracy and precision to be acceptable for purposes of resource estimation.

No significant bias was observed in the CRM results for gold. Results for 23 standards were outside acceptable limits and the batches were reassayed and Equinox Gold replaced the original assays with the reassays in the exploration database. No significant carryover contamination was observed in the blank results.

11.2.5 Data Adequacy

In the opinion of the Qualified Person, the sample preparation, security, and analytical procedures for the core and RC samples are adequate for purposes of resource estimation. The assay accuracy and precision are considered acceptable for resource estimation.

11.3 NewCastle RAB

NewCastle sampling and assaying procedures are consistent to the description of sample collection in Section 10, Drilling. NewCastle Gold's sampling, assaying, and QA/QC procedures from the 2017 and 2018 RAB drilling campaigns are summarized in sections 11.3.1 to 11.3.4. The RAB drill program includes 1,262 holes, of which 1,057 were located within the JSLA pit backfill. The JSLA RAB drill holes were drilled on an approximate 50ft by 50 ft grid and total 22,350 feet of drilling.

11.3.1 Sampling and Security

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. 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. From the drill, the samples are stored in the laydown yard. Then, bagged samples are sent to the lab for analysis. Samples from the bags are sufficiently mixed to get a representative sample for the drilling interval.

11.3.2 Analyses

ALS Laboratories performed assays on the RAB samples with the following analyses:

2017 Program: Fire Assay with Atomic Absorption and Gravimetric finish, Cyanide Digestion.

2018 Program: Fire Assay with Atomic Absorption finish, Cyanide Digestion.

11.3.3 Quality Assurance and Quality Control

QA/QC procedures were implemented according to industry best practice and approved by the Qualified Person. Certified reference material (CRM) was screened for results within 10% of the reported mean, and blank material was screened for results above 10X the detection limit of the analytical method.

2017 Program: CRM and blank material were included in the sample stream at a ratio of 1:25. The ratio of CRM to blank material was 1:1. Field duplicates were taken at a ratio of 1:50.

2018 Program: CRM and blank material were included in the sample stream at a ratio of 1:14. The ratio of CRM to blank material was 1:1. Field duplicates were taken at a ratio of 1:10.

No control samples for the 2017 program failed QC tests, while three control samples in the 2018 program failed QC tests. All three failures were rerun at the lab along with a bracket of 5 samples on either side of the failure, and all results passed QC tests upon reanalysis. Due to the nature of backfill material, field duplicates were simply used as a measure of material consistency and yielded a per-sample variation of ± 20%.

11.3.4 Data Adequacy

In the opinion of the Qualified Person, the sample collection, preparation, security, and analysis of the RAB samples are adequate for the purposes of resource estimation. The assay accuracy and precision are adequate for resource estimation.

12.0 DATA VERIFICATION

MTS performed verification of exploration data relevant to the Castle Mountain resource estimate including all information from the drill campaigns summarized in Section 10, Drilling. Previous data verification work is modified from Gray et al. (2016).

12.1 Database

12.1.1 Core and RC Drilling

The Castle Mountain project data is stored in a custom Microsoft Access® (Access) database designed for the purpose by MTS in 2016. This database is secure, operated by a single database administrator, and contains data checking routines designed to prevent common data entry errors. An export of the data was provided to MTS for auditing purposes and for import into Micromodel and MineSight mining software for subsequent geologic modeling and mineral resource estimation.

MTS conducted an extensive audit of the Castle Mountain digital database in 2017. This audit consisted of checking the digital data against source documents to ensure proper data entry as well as exhaustive data integrity checks (checking for overlapping intervals, data beyond total depth of hole, unit conversion, etc.).

12.1.2 RAB Drilling

The data for the RAB drilling of the JSLA backfill material was provided to the resource estimators in an Excel spreadsheet that contained the collar coordinates and the fire assay gold grade. The QA/QC assays had been removed from the data prior to delivery to the estimating firm.

12.2 Collar and Down-Hole Surveys

12.2.1 Viceroy

RPA (2013) reviewed the transformation of Viceroy drill hole local mine coordinates (feet) into present UTM NAD83 coordinates (feet) and compared the collar locations to pre-mining, post-mining and present topographic surfaces. Because historical drill hole collars and survey markers are not locatable due to mining activities and post-mining reclamation, RPA created a best-fit solution to match topographic features with the available topographic surfaces. In RPA's opinion, there was reasonable agreement between these surfaces, but there are some local discrepancies up to several tens of feet. MTS accepted the coordinates converted by RPA and these are the coordinates stored in the Castle Mountain Access database.

In 2017, NewCastle contracted Compass Tools in Denver, Colorado to provide an updated topographic surface of the Project area. NewCastle and MTS compared the historic drill hole coordinates to the 2017 Castle Mountain topographic surface and found them to be in close agreement. Where significant discrepancies existed between the historic collar coordinates and the 2017 topographic surface, the drill collar elevation was changed to the elevation of the 2017 topographic surface.

Very few of the Viceroy drill holes were surveyed down-hole. Of the 88 Viceroy drill holes audited by MTS, only one was surveyed down-hole. The database records accurately reflected the original survey records for this drill hole.

12.2.2 NewCastle Core and RC

MTS compared the NewCastle collar elevations to the 2017 Castle Mountain topographic surface and found them to be in close agreement. Where differences between collar elevation and topographic surface exceeded 10 ft, the collar elevation was assigned to the topographic surface elevation.

All NewCastle down-hole surveys were loaded into the Castle Mountain Access database using a custom computer application developed by MTS. Down-hole surveys were loaded from files received directly from the surveying company.

12.2.3 NewCastle RAB

Survey collar locations for the RAB holes were provided by a high-resolution handheld Garmin GPS device and the coordinates were then placed in an Excel spreadsheet and matched with the hole number. The locations were plotted on a map to ensure data consistency. Downhole surveys were not performed on these holes due to their lengths being 20 to 30 feet and they are assumed to be all vertical.

12.3 Drill Logs

12.3.1 Viceroy

MTS randomly selected 5% of the Viceroy drill holes and compared the original drill logs against the records in the Castle Mountain Access database. Of the 88 drill holes selected, the original drill logs were not present in the drill folder for 10 drill holes. Of the 78 drill holes that were compared, the lithology codes in the drill logs matched the records in the database.

In 2017, NewCastle reviewed and reinterpreted over 1,200 original Viceroy paper logs, or about 710,000 feet of the 1,182,500 feet (60%) of historic Viceroy drilling. These reinterpreted logs were used to inform the geological model used in resource estimation.

12.3.2 NewCastle Core and RC

All NewCastle drill logs were loaded into the Castle Mountain Access database using a custom computer application developed by MTS. Drill logs including collar, lithology, alteration, and metallurgy data were loaded into the database from the Microsoft Excel files generated by the NewCastle geological staff.

12.3.3 NewCastle RAB

The sample created from a RAB drill are drill cuttings produced by a hammer drill and are quite fine with mostly fine sand sized material generated for the sample. As a result, RAB drill logs were kept very simple, focusing only on presence or absence of known unmineralized units. These logs are stored in Equinox Gold's server in Henderson, Nevada.

12.4 Assays

12.4.1 Viceroy

In 2013, RPA audited a total of 8,205 assay intervals from 55 drill holes, or 3.4% of the historical database and found no significant data entry errors.

In 2017, MTS randomly selected 5% of the Viceroy drill holes and compared the original assay certificates against the records in the Castle Mountain Access database. Of the 88 drill holes selected, 31 of the original assay certificates were missing from the Project files. Of the 57 drill holes audited, only one error was found.

12.4.2 NewCastle Core and RC

12.4.2.1. 2013-2016 Drill Campaigns

MTS compiled assay results from original assay certificates received directly from the assay laboratories for drill holes completed by NewCastle from 2013 through 2016. This compilation included 21,179 assay records, or 78% of the assay records in the database for these campaigns. MTS compared these records against the assay records in the Castle Mountain Access database and no errors were found.

12.4.2.2. 2017/2018 Drill Campaigns

Assay certificates from the 2017 drill campaign were loaded into the Castle Mountain Access database using a custom computer application developed by MTS. Assays were loaded into the database in their original units from files received directly from the assay laboratories.

12.4.3 NewCastle RAB

For the RAB drilling campaigns, assays and QA/QC protocols were performed in a similar manner to the core and RC drilling programs. For the sample, the entire drill cutting pile was collected at the hole collar and sent to the ALS assay lab in Reno Nevada for gold assaying. This method was used to avoid sample bias due to possible improper field sampling techniques.

QA/QC checks were performed on all the assays in accordance with NewCastle's test protocols. Additionally, these data were compared with other sample sets taken from the JSLA backfill material and it was determined through statistical analysis that different drilling methods produced a data sets that corresponded with one and other and could therefore be used in grade estimation.

12.5 Density

A total of 647 wax-coated, water-immersion specific gravity measurements have been carried out on core samples drilled by NewCastle. From 2013 to 2016, a total of 461 specific gravity measurements were conducted by ALS in Reno, Nevada. In 2017, NewCastle conducted 186 specific gravity measurements at the Castle Mountain site. All measurements were conducted on 4 to 6 inch long pieces of split (half-core) HQ (2.50 inch diameter) or PQ (3.35 Inch diameter) size core.

Table 12.1 summarizes the density data by rock type; four of the 647 density measurements were conducted on rock types not material to the resource estimate and are not included in the table.

Table 12-1 - Castle Mountain Density Data

Rock Type	Count (n)	Density (t/m^3)
Felsic ALT	**127**	**2.14**
Felsic LT	**64**	**2.10**
Rhyolite Aphyric	**172**	**2.22**
Rhyolite Phyric	**104**	**2.25**
Diatreme	**64**	**2.24**
Intermediate Sediments	**30**	**2.28**
Andesite	**71**	**2.22**
Basement	**11**	**2.47**
Total	643	2.24

A bulk density 1.78 t/m^3 (18 ft^3/ton) was used for the historic waste dumps and JSLA pit backfill. This compares well to a 2.24 t/m^3 in-situ density with a 30% swell factor.

12.6 Data Adequacy

12.6.1 Core and RC Drilling

In the opinion of the Qualified Person, the data are adequate for purposes of resource estimation. MTS recommends that NewCastle load the reinterpreted Viceroy drill logs into the Castle Mountain database.

12.6.2 RAB Drilling

In the opinion of the Qualified Person, the sample collection, preparation, security, and analysis of the RAB samples are adequate for the purposes of resource estimation. The assay accuracy and precision are adequate for resource estimation.

13.0 METALLURGICAL TESTING

A significant amount of metallurgical test data has been generated for the Castle Mountain Project, including:

- Initial test work before startup of the mine in 1992;
- Continued test work during operations for process optimization during 1991-2001;
- 2014-2015 test program with crush size vs. recovery column tests, run-of-mine (ROM) column tests, bottle roll tests, grinding / cyanidation testing, gravity recoverable gold tests, comminution tests, and compacted permeability tests;
- 2017-2018 test program with ROM column tests, pulp agglomeration studies, cyanidation testing, gravity recoverable gold tests, crush size vs. recovery column tests, variability testing, CIL testing, compacted permeability testing, gravity sedimentation and filtration tests.

In addition to test data, actual production statistics from the pulp agglomeration plant operations from 1991-2001 and post-production heap leaching data have been used. The historical pulp agglomeration process involved combining and agglomerating partially leached, ground mill ore with coarser crushed ore in an agglomeration drum, followed by conveyor stacking the agglomerates on a heap leach pad.

Much of the available test work is dedicated to pulp agglomeration studies, both historically and within recent campaigns. As studies progressed during recent campaigns, however, the test work emphasis shifted to evaluating conventional milling with Carbon in Leach (CIL) for higher grade ore within the deposit, and ROM heap leaching for lower grade ore. Conventional milling and ROM heap leaching allows high and low-grade ores to be treated independently, which offers more flexibility to processing and mine scheduling as compared to the pulp agglomeration process, which is dependent upon the blending ratios of higher grade mill slurry to lower grade crushed ore in the pulp agglomeration product (1:9 to 1:12 during historical operations). The current resource defines higher ratios of mill slurry to crushed ore than was the case during historical operations, which presents challenges to scheduling of high and low-grade ore delivery from the mine and also presents additional risk to heap permeability as compared to the historical operation. Recent work has focused on de-coupling this limitation to maximize the overall amount of ore for processing by considering a CIL process plant for high-grade ores and a conventional ROM heap leach for low-grade ore.

Because the pulp agglomeration process also utilizes milling of high-grade material, there are significant amounts of test work from prior pulp agglomeration test work campaigns that support the design of the proposed CIL option.

13.1 Test Work Summary

As data was generated over a period of many years, test parameters have varied with regard to particle size distributions, leach times, and leach conditions. These must be considered when analyzing the historical data in support of the current proposed process route.

In addition to test work and production data, prior Technical Reports were prepared by RPA in 2013 and 2014, and by Advantage Geoservices in 2016. KCA has reviewed the Metallurgy and Processing sections (Sections 13 and 17) of those reports.

A summary of all known metallurgical test work for Castle Mountain is shown in Table 13.1 below.

Table 13-1 - Summary of Castle Mountain Metallurgical and Physical Test Work

Cyanide / Leach Testing										
Date	**Lab**	**Period**	**Samples**	**BRTs**	**Sizes, mesh**	**Time, hrs**	**Column Tests**	**Sizes, in**	**Time, days**	**Reference**
February-87	Bateman	Pre-Production	Jumbo South Bulk	3	100, 150, 200	72	5	3, 1½, 1, 3/4, 3/8	33	Bateman, 1987a
November-87	Bateman	Pre-Production	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	Pre-Production	Jumbo South DDH-3	3	100	24	6	3/4, 3/8	58 to 63	Bateman, 1988a
1988	Bateman	Pre-Production	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	Pre-Production	Leslie Ann DDH-10	---	---	---	10	3/8, ¼	69 to 105	Bateman, 1988b
July-89	McClelland	Pre-Production	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	Pre-Production	DDH-18, DDH-3M, DDH-3U	1	100	72	4	3/8	68 +10 rinse	MLI, 1989b
January-93	McClelland	Production	HL Residue	6	100	24	---	---	---	MLI, 1993
March-95	McClelland	Production	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	Production	RC Cuttings - South Extension	4	As Received 100 mesh	120	---	---	---	MLI, 1995b
July-95	McClelland	Production	Bulk Primary Crusher Product	---	---	---	1	6	85	MLI 1995
May-96	McClelland	Production	141 South Ext DDH-56, DDH-57, DDH-58, DDH-59	3	80% -¼ in	240	2	¼	66 to 71	MLI 1996
February-15	McClelland	Post-Production	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	Post-Production	ROM (Oro Belle South, JSLA Backfill)	2	10 (1.7mm)	96	4	ROM (2), 3/8 (2)	157-164	MLI, 2015b

Table 13-1 cont'd

Cyanide / Leach Testing										
Date	**Lab**	**Period**	**Samples**	**BRTs**	**Sizes, mesh**	**Time, hrs**	**Column Tests**	**Sizes, in**	**Time, days**	**Reference**
June-17	RDI	Post-Prod. (PFS)	ROM (JSLA Backfill)	15	6", 2", 3/8"	72	5	ROM	42	RDI, 2017
May-18	McClelland	Post-Prod. (PFS)	ROM (JSLA Backfill)	---	---	---	4	ROM	130-140	MLI, 2018b
May-18	McClelland	Post-Prod. (PFS)	JSLA, LG Master Composite	---	---	---	2	2, 3/8	120-130	MLI, 2018a
May-18	McClelland	Post-Prod. (PFS)	Variability BRT Testing (S Domes, Oro Belle, JSLA Master, Andesite)	20	2" (1), 3/8" (19)	480 (20 days)	---	---	---	MLI, 2018a
May-18	McClelland	Post-Prod. (PFS)	Variability Gravity/Leach Tests (S Domes, Oro Belle, JSLA HG Master, Andesite)	12	100	96	---	---	---	MLI, 2018a
May-18	McClelland	Post-Prod. (PFS)	Gravity/Leach Tests, Variable Grind (JSLA HG Master)	4	48, 65, 100, 150	96	---	---	---	MLI, 2018a
August-18	KCA	Post-Prod. (PFS)	Gravity / CIL Test, High Grade Comp.	4	100	variable	---	---	---	KCA, 2018c

Other Testing					
Date	**Lab**		**Tests**	**Description**	
October-89	Pocock	Pre-Production	Gravity Sedimentation		Pocock, 1989
May-98	Glasgow	Production	Compacted Permeability	Production samples	Glasgow, 1998
February-17	McClelland	Post-Production	Comminution	Crusher Index (Cwi) and Bond Abrasion (Ai) for Four PQ core samples	MLI, 2015a
May-18	Pocock	Post-Prod. (PFS)	Gravity Sedimentation, Vaccum and Pressure Filtration (JSLA HG Master)		MLI, 2018a
July-18	KCA	Post-Prod. (PFS)	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	Post-Prod. (PFS)	Compacted Permeability (JSLA ROM w/ Mill Tails	ROM / Mill Tails Varying Blends for Tailings Disposal Investigations	KCA, 2018b

13.2 Historical Pre-Production Metallurgical Test Work

In support of the design of the historical process plant operated in the 1990s and early 2000s, Bateman completed a total of 19 bottle roll tests (BRTs) and 61 column leach tests using samples from the Oro Belle, Jumbo South and Leslie Ann pit areas, from 1987 to 1989, over ten different test work campaigns. These campaigns evaluated a number of variables including the impact of particle size, leach time, and head grade, and included samples taken from various portions of the deposits and at different depths of the deposits. Generally, samples were grouped by grade into "leach-grade" and "mill-grade" samples for planned treatment by heap leaching and milling respectively.

A significant finding from the column tests was the relationship between particle size and gold extraction. Table 13.2 shows the average recovery by crush size, which is also presented graphically in Figure 13.1.

Table 13-2 - Summary of all Castle Mountain Metallurgical and Physical Test Work

Size (in)	Size (um)	Leslie Ann (%)	Jumbo South (%)	Oro Belle (%)
2½	63,500	29	43	
1½	38,100	51	52	53
3/4	19,050	39	58	59
3/8	9,525	61	74	60
1/4	6,350	68		
100 Mesh (BRT)	149	94	91	

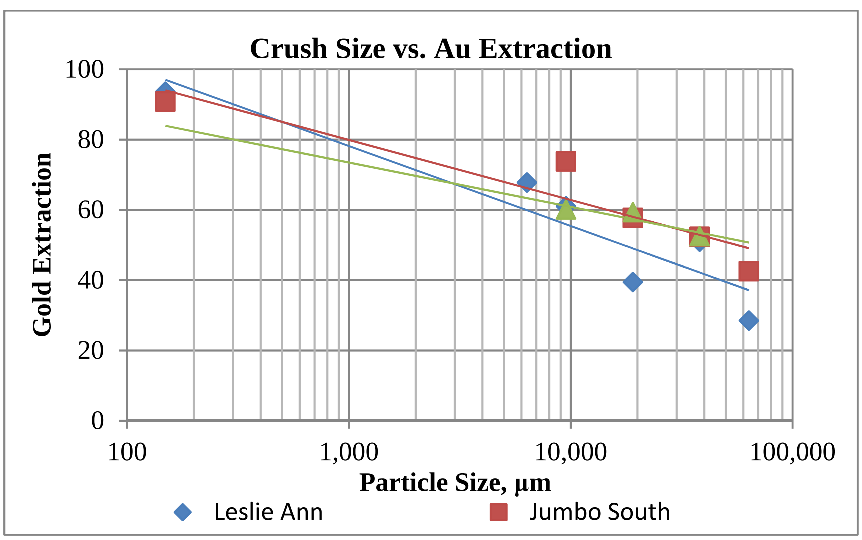

Figure 13-1 - Pre-Production Column Test Recoveries vs. Crush Size

An evaluation of the head grade of the samples and the gold extraction showed no clear correlations between the gold grade and gold extraction, either overall, by crush size, or by deposit.

Additionally, there were no discernable trends in recovery by deposit depth for either the Jumbo South or Lesley Ann deposits (samples from Oro Belle were bulk samples and thus could not be evaluated by depth).

In preparation for the addition of a milling circuit at Castle Mountain, in 1989 McClelland conducted cyanidation, gravity concentration tests, and a test of pulp agglomeration of mill tailings with crushed ore. Thickener settling tests were also conducted by Pocock Industrial.

A BRT on a composite sample that was ground to 100% passing 100 mesh (149 µm) and leached in cyanide for 96 hours achieved a gold extraction of approximately 99%.

Duplicate column tests were then run on heap leach samples that were crushed to 80% passing 3/8 inch (9.5 mm) and agglomerated, one with leached and the other with unleached ground mill-grade material at a target particle size of 80% passing 65 mesh (210 µm). The proportion was 20% milled material and 80% heap leach material (1:4).

The sample that was agglomerated with unleached material achieved a gold extraction of 77% in 44 days and the sample that was agglomerated with leached material achieved a gold extraction of 66% after 42 days. When considering the overall recovery, the pre-leached sample recovery was 78%, nearly identical to the column with unleached material. This suggested the recovery of the ground high-grade portion of ore in the column was essentially 99% as indicated in the BRT test. The gold extraction rate was faster for the sample agglomerated with leached material. It was determined in the column test work that effective agglomeration would be achieved by using 10 lb/ton cement.

Gravity concentration test results suggested that one of the two high-grade samples did not respond particularly well to gravity gold concentration while the other did, with the gold recovery into gravity concentrates 4.5% and 20.3% of the feed respectively.

13.3 Test Work During Production Years

In 1993, McClelland undertook a testing program to determine if heap leached residue could be subsequently milled and leached using agitated cyanidation to recover residual gold. The agitated cyanidation (bottle roll) test was performed on nine production heap leach residue samples at 80% -100 mesh (149 µm) feed to determine residual gold recovery, recovery rate, and reagent requirements. Metallurgical results showed that the residue samples were amenable to subsequent treatment through agitated cyanidation at that feed size. Residual gold recoveries ranged from 69% to 81% and averaged 74% in 24 hours of leaching.

In 1995, three composite samples, collected from three different three-month periods from 3/8 inch (9.5 mm) crusher discharge at the mine, were submitted for column leach tests at McClelland. The as-collected composite samples, with a nominal size of 75% passing 3/8 inch (9.5 mm), were split and crushed finer to produce three composites at three crush sizes each. The test results showed that the gold extraction from column tests using samples that were 75% -3/8 inch (9.5 mm) and 90% -3/8 inch (9.5 mm) were similar but the gold extractions from samples crushed to 80% -1/4 inch (6.4 mm) increased significantly over the coarser crush sizes. A summary of the results is presented in Table 13.3 below.

Table 13-3 - 1994 Quarterly Production Composite Recoveries, Crush Size -3/8 inch

	Gold Recovery (%)		
Feed Size	**Composite A**	**Composite B**	**Composite C**
75% -3/8"	69%	70%	67%
90% -3/8"	69%	73%	68%
80% -1/4" (100% - 3/8")	79%	77%	77%

Also in 1995 a low-grade 0.013 oz/ton (0.45 g/t) Au bulk sample of primary crusher discharge of -6 inch (152 mm) was tested in a pilot-scale column. The results indicated the sample was readily amenable to heap leaching at the 6-inch (152 mm) crush size, with a recovery of 62% after a 72-day leach cycle and 69% after an additional 13 days of rinsing.

In 1995 and 1996, samples from the South Extension deposit area were evaluated.

In 1995 two composites of RC cuttings at leach-grade and mill-grade were run in duplicate bottle roll tests. Leach-grade samples were tested at the as-received size and mill-grade samples were ground to 80% -100 mesh (149 µm), and tested for gravity concentration with cyanidation of the gravity tailings. All samples were leached for 120 hours. Test results indicated leach-grade samples were amenable to cyanidation, recovering 64% and 69% gold. Results indicated mill-grade samples were amenable to gravity concentration with 24% recovery into the concentrate at only 0.03% of the feed weight, and tailings were amenable to cyanidation, with a combined gravity/leach recovery of 96% and 97%.

In 1996, leach-grade drill core composites of Rhyolite and Tuff rock types from the South Extension were tested in columns and BRTs at an 80% passing 1/4-inch (6.4 mm) crush size. Column test results indicated gold recoveries of 65% and 90% for Rhyolite and Tuff composites respectively, after 60 days of leaching. BRT recoveries were 54% and 75% for Rhyolite (average of two composites) and Tuff composites respectively, after 10 days of leaching.

In 1998, Viceroy tested ore permeability on samples of leach grade ore with no mill fines added. Samples were taken from production crushed ore and tested with no cement followed by adding cement incrementally until suitable permeability was achieved, and also on composites of belt cuts from 3-day periods of operation. Test results indicated no cement was required for any of the samples to achieve good percolation up to a simulated heap height of 150 feet (46 meters), with no samples showing a hydraulic conductivity less than 0.018 cm/sec (~650 L/hr/m^2 or 0.27 gpm/ft^2). This work suggested heap-grade ore with no added mill fines could be stacked on the heap with no cement addition.

13.4 Production Data

Castle Mountain was mined for a period of nine years from 1992 to 2001, and continued processing and recovering gold from the heap for an additional three years until 2004.

The initial process plant commissioned in 1992 included heap leaching of ore that was crushed to 100% -3/8 inch (9.5 mm) in a three-stage crushing circuit. The milling circuit was added in 1993 with heap leach ore then agglomerated with partially leached mill tailings. Mill-grade ore was ground to 80% passing 100 mesh (149 μm) in cyanide solution.

Initially, the gold extraction by leaching in the mill averaged 35% in 1993-1994 prior to being agglomerated with the crushed ore. 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 addition of the gravity circuit, the combined recovery in the mill (gravity plus leaching) increased to approximately 45-50%.

The partially leached ore from the milling circuit was used to agglomerate the crushed ore in the proportions of up to 10% mill ore to 90% crushed heap leach ore.

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%. KCA believes this to be a reasonable assumption, based on available test data on BRTs on ground 100 mesh samples and particularly the pulp agglomeration tests conducted by MLI in 1989. The recovery of low-grade ore from the leach pad was then back calculated from the total amount of gold recovered.

Production records show that, over the life of the Mine, 36.2 Mt of ore grading 0.043 oz/t (1.5 g/tonne) Au containing approximately 1.55 M oz (48 M g) gold were loaded onto the heap leach pads.

The 36.2 Mt (32.8 M tonnes) comprises approximately 34.2 Mt (31.0 M tonnes) grading 0.037 oz/t (1.3 g/tonne) Au, approximately 1.27 M oz (39.5 M g) of contained gold was fed directly to the heap leach circuit and 2.0 Mt (1.8 M tonnes) grading 0.144 oz/t (4.9 g/tonne) Au, containing approximately 0.28 M oz (0.25 M g) gold was placed on the leach pad from agglomerated tailings from the milling circuit.

Over the life of the operation, a total of 1.24 M oz (38.6 M g), or 80.2% of 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 cessation of mining. A summary of the ore production and metallurgical recoveries by year is shown in Table 13.4.

A graph showing the cumulative recovery of the heap leach circuit is shown in Figure 13.2.

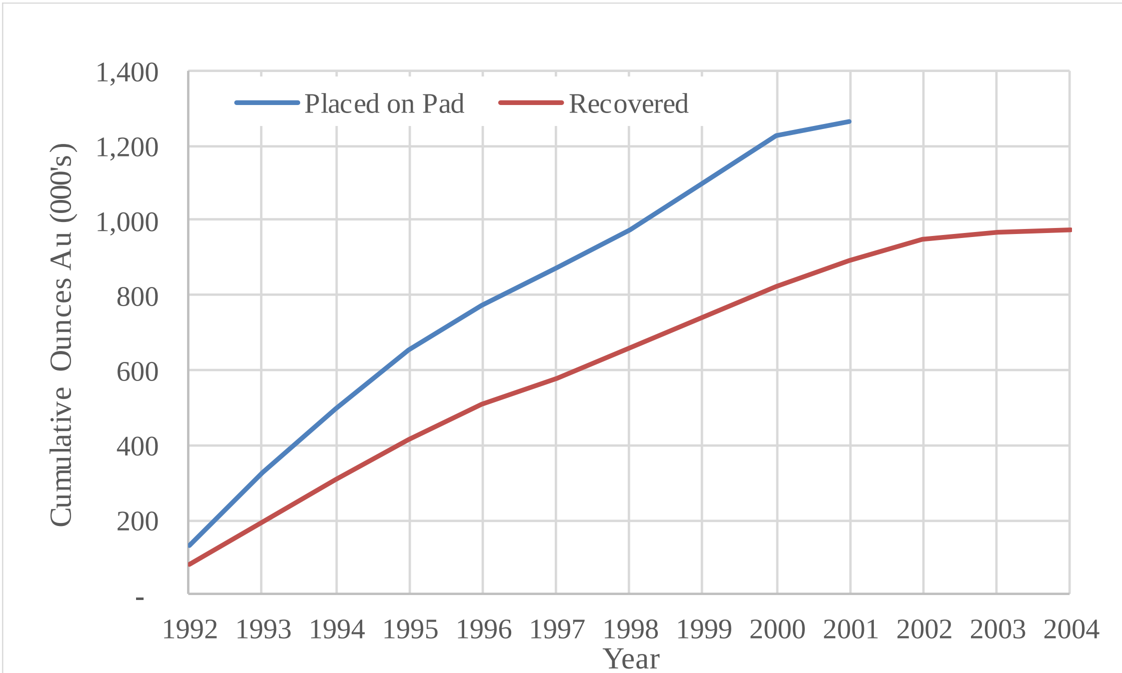

Figure 13-2 - Production Cumulative Ounces Stacked vs. Recovered, Leach-Grade Ore

Table 13-4 - Castle Mountain Production Data, 1992-2004

Item	units	TOTAL	1992	1993	1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	2004
Total to Pad	000 tons	36,193	2,581	3,658	4,083	4,204	4,120	4,103	3,891	4,123	4,120	1,308			
Grade	oz/t	0.043	0.051	0.058	0.056	0.050	0.037	0.038	0.027	0.034	0.040	0.037			
Contained	000 oz	1,550	131.6	211.5	230.0	211.3	151.6	155.9	105.1	140.2	164.8	48.4	-	-	-
Recovered	000 oz	1,243	78.0	133.2	170.3	156.9	122.2	122.4	89.1	95.0	118.7	77.7	56.7	14.8	8.2
Cunmulative Recovery		80.2%	59.3%	61.5%	66.5%	68.6%	70.6%	71.7%	72.9%	72.3%	72.3%	75.0%	78.7%	79.7%	80.2%
Mill Ore	000 tons	1,967	-	88	305	301	296	412	58	90	328	89	-	-	-
Grade	oz/t	0.144	-	0.183	0.210	0.162	0.113	0.138	0.108	0.138	0.108	0.113	-	-	-
Contained	000 oz	283	-	16.1	64.1	48.5	33.4	56.7	6.3	12.4	35.3	10.1	-	-	-
Mill Recovered	000 oz	120	-	5.8	21.5	18.4	14.1	28.2	2.8	6.3	17.2	5.5	-	-	-
Average Gravity Recovery						13.0%	16.0%	16.6%	22.1%	14.3%	18.1%	21.1%			
Total Mill Recovery (Grav+Leach)	% of mill feed	42.3%	0.0%	35.9%	33.5%	37.9%	42.3%	49.7%	45.0%	50.5%	48.7%	54.9%	-	-	-
Mill Tailings	000 oz	163	-	10.3	42.6	30.1	19.3	28.5	3.4	6.1	18.1	4.5	-	-	-
Leach Recoverable	000 oz	149	-	9.5	39.4	27.7	17.6	25.7	3.1	5.5	16.4	4.0	-	-	-
Leach Recovery	% of mill tailings	91.3%	0.0%	92.2%	92.5%	91.9%	91.3%	90.1%	90.9%	89.9%	90.3%	88.9%	-	-	-
Leach Ore		34,226	2,581	3,571	3,778	3,904	3,824	3,691	3,833	4,033	3,792	1,219	-	-	-
Grade	oz/t	0.037	0.051	0.055	0.044	0.042	0.031	0.027	0.026	0.032	0.034	0.031	-	-	-
Contained	000 oz	1,267	131.6	195.4	165.9	162.7	118.2	99.2	98.8	127.8	129.5	38.3	-	-	-
Recovered	000 oz	974	78.0	117.9	109.4	110.8	90.4	68.5	83.2	83.2	85.2	68.2	56.7	14.8	8.2
Cunmulative 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%	76.9%

13.5 Recent Metallurgical Test Work

13.5.1 McClelland 2014/ 2015 - Phase 1

A significant metallurgical testing program was completed by McClelland Laboratories, Inc. (MLI) in 2014 and 2015 to evaluate processing of Castle Mountain oxide material types by heap leach cyanidation for low-grade ore and milling/cyanidation and gravity concentration of high-grade ore. A total of 33 column tests and 52 bottle roll tests were completed. A sample location map for this program is shown in Figure 13.3.

PQ diamond drill core from four holes (CMM-012, 013, 014 and 017) was used for the test work, the locations of which are illustrated in Figure 13.3. These holes were drilled in the Jumbo and south JSLA areas. The heap leach testing consisted of bottle roll and column leach testing on 19 column test composites and bottle roll testing on 33 “material variability composites.” A column test was conducted on each of the 19 composites at a nominal 3/8 inch (9.5 mm) feed size.

Comparative column tests were conducted on seven select composites at nominal 2 inch (50 mm) and 3/4 inch (19 mm) feed sizes. Duplicate column tests were conducted on four select composites. Six selected high-grade composites were used for whole material milling/cyanidation tests, as well as gravity concentration tests with agitated cyanidation of the gravity tailings. Four separate rock type composites were used for comminution testing.

Overall, test results showed that the Castle Mountain oxide material types were amenable to heap leach cyanidation treatment at feed sizes ranging from 2 inch (50 mm) to 3/8 inch (9.5 mm). A summary of the column tests and select related BRTs are shown in Table 13.5. The composites showed a minor sensitivity to crush size with respect to gold recovery. Higher grade material, particularly the relatively deep, conglomerate rock type material from drill hole CMM-014, tended to be not as amenable to heap leach cyanidation, and more sensitive to feed size. All bottle rolls tests on ground composites at 80% -200 mesh (74µm) showed very high gold recoveries (>95%) including those of the conglomerate rock type, and Gravity concentration tests indicated high recoveries into the concentrates, as shown in Table 13.6.

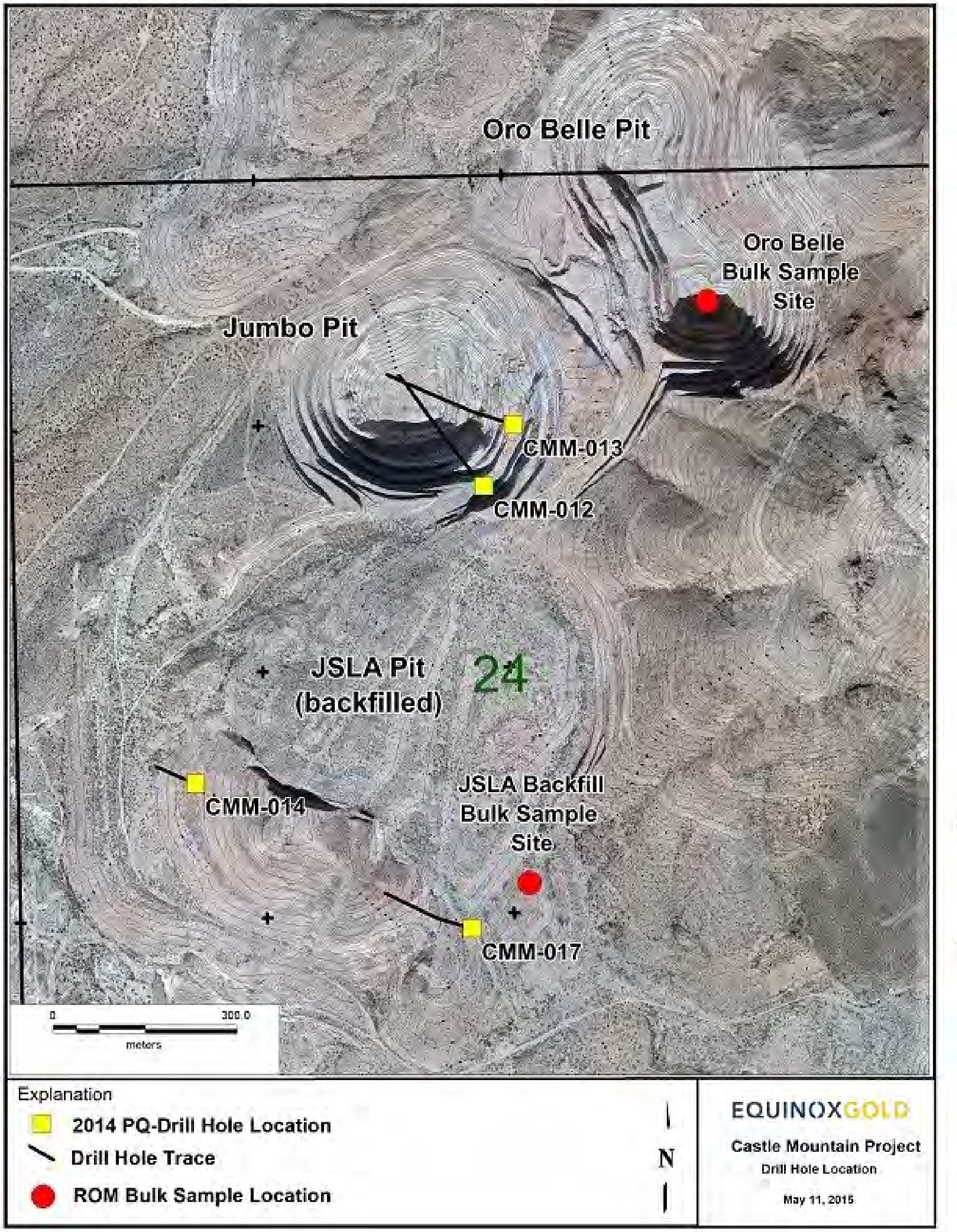

Figure 13-3 - MLI Phase 1 and Phase 2 Sampling Locations

Table 13-5 - MLI 2014/ 2015 Phase I Column and Bottle Roll Test Results

Composite I.D.	Test No.	Test Type	Feed Size	Leach Time, days	Au Rec. %	gAu/mt ore Extracted	Tail Screen	Calc'd. Head	Assay Head	NaCN Consumed, kg/mt ore	Lime Added kg/mt ore
3878-101	P-1	CLT	80%-9.5mm	69	89.3	0.25	0.03	0.28	0.26	1.11	3.6
3878-101	CY-35	BRT	80%-1.7mm	4	87.0	0.20	0.03	0.23	0.26	0.07	4.5
3878-102	P-2	CLT	80%-9.5mm	92	77.8	0.49	0.14	0.63	0.67	1.55	1.6
3878-102	CY-36	BRT	80%-1.7mm	4	72.5	0.37	0.14	0.51	0.67	0.07	2.0
3878-103	P-3	CLT	80%-9.5mm	160	93.4	1.14	0.08	1.22	1.21	2.42	1.5
3878-103	CY-37	BRT	80%-1.7mm	4	71.8	0.74	0.29	1.03	1.21	0.07	1.9
3878-104	P-4	CLT	80%-9.5mm	92	96.1	0.49	0.02	0.51	0.47	1.42	2.0
3878-104	CY-38	BRT	80%-1.7mm	4	71.4	0.35	0.14	0.49	0.47	0.07	2.5
3878-105	P-5	CLT	80%-9.5mm	92	61.9	0.26	0.16	0.42	0.43	1.70	1.5
3878-105	CY-39	BRT	80%-1.7mm	4	51.3	0.20	0.19	0.39	0.43	0.07	1.8
3878-106	P-16	CLT	50mm	89	79.5	0.35	0.09	0.44	0.37	0.67	1.7
3878-106	P-19	CLT	80%-19mm	96	81.1	0.30	0.07	0.37	0.37	0.71	1.7
3878-106	P-22	CLT	80%-9.5mm	56	82.5	0.33	0.07	0.40	0.37	0.90	1.7
3878-106	P-6	CLT	80%-9.5mm	92	84.6	0.33	0.06	0.39	0.37	1.49	1.7
3878-106	CY-40	BRT	80%-1.7mm	4	84.2	0.32	0.06	0.38	0.37	0.07	2.1
3878-107	P-17	CLT	50mm	89	90.6	0.48	0.05	0.53	0.60	0.66	1.8
3878-107	P-20	CLT	80%-19mm	95	91.2	0.62	0.06	0.68	0.60	0.81	1.8
3878-107	P-23	CLT	80%-9.5mm	56	95.4	0.62	0.03	0.65	0.60	0.86	1.8
3878-107	P-7	CLT	80%-9.5mm	92	93.1	0.54	0.04	0.58	0.60	1.44	1.8
3878-107	CY-41	BRT	80%-1.7mm	4	89.7	0.70	0.08	0.78	0.60	0.07	2.3
3878-108	P-8	CLT	80%-9.5mm	79	88.2	0.30	0.04	0.34	0.32	1.01	1.6
3878-108	CY-42	BRT	80%-1.7mm	4	89.7	0.35	0.04	0.39	0.32	0.07	2.0
3878-109	P-18	CLT	50mm	89	62.9	0.39	0.23	0.62	0.74	0.77	1.0
3878-109	P-21	CLT	80%-19mm	69	59.2	0.42	0.29	0.71	0.74	0.57	1.0
3878-109	P-24	CLT	80%-9.5mm	49	64.0	0.48	0.27	0.75	0.74	0.51	1.0
3878-109	P-9	CLT	80%-9.5mm	70	72.1	0.49	0.19	0.68	0.74	0.85	1.0
3878-109	CY-54	BRT	80%-1.7mm	4	66.3	0.59	0.30	0.89	0.74	<0.07	1.0
3878-110	P-10	CLT	80%-9.5mm	157	64.1	4.32	2.42	6.74	6.09	2.18	1.0
3878-110	CY-55	BRT	80%-1.7mm	4	69.9	4.31	1.86	6.17	6.09	<0.07	1.2

Table 13-5 - Continued

Composite I.D.	Test No.	Test Type	Feed Size	Leach Time, days	Au Rec. %	gAu/mt ore Extracted	Tail Screen	Calc'd. Head	Assay Head	NaCN Consumed, kg/mt ore	Lime Added kg/mt ore
3878-111	P-11	CLT	80%-9.5mm	160		8.83	7.93	16.76	17.95	2.45	1.0
3878-111	CY-56	BRT	80%-1.7mm	4	58.6	8.40	5.93	14.33	17.95	<0.07	1.2
3878-112	P-12	CLT	80%-9.5mm	149		2.40	0.90	3.30	3.29	2.30	1.0
3878-112	CY-57	BRT	80%-1.7mm	4	84.5	2.50	0.46	2.96	3.29	0.10	1.3
3878-113	P-13	CLT	80%-9.5mm	118	66.7	0.56	0.28	0.84	0.74	1.65	1.0
3878-113	CY-58	BRT	80%-1.7mm	4	71.0	0.66	0.27	0.93	0.74	0.07	1.3
3878-114	P-14	CLT	80%-9.5mm	64	88.6	0.39	0.05	0.44	0.45	0.70	1.0
3878-114	CY-59	BRT	80%-1.7mm	4	88.9	0.40	0.05	0.45	0.45	0.07	1.2
3878-115	P-15	CLT	80%-9.5mm	89	>97.4	0.38	<0.01	<0.39	0.50	0.71	1.3
3878-115	CY-60	BRT	80%-1.7mm	4	92.5	0.37	0.03	0.40	0.50	0.11	1.6
3878-116	P-25	CLT	50mm	123	89.2	0.66	0.08	0.74	0.62	1.24	1.2
3878-116	P-30	CLT	80%-19mm	102	88.5	0.54	0.07	0.61	0.62	0.96	1.4
3878-116	P-35	CLT	80%-9.5mm	111	88.4	0.61	0.08	0.69	0.62	1.14	1.5
3878-116	CY-61	BRT	80%-1.7mm	4	81.5	0.66	0.15	0.81	0.62	0.16	1.9
3878-117	P-26	CLT	50mm	148		6.18	10.96	17.14	15.54	1.21	0.8
3878-117	P-31	CLT	80%-19mm	116	43.6	7.18	9.29	16.47	15.54	1.27	0.9
3878-117	P-36	CLT	80%-9.5mm	134	45.0	8.14	9.96	18.10	15.54	2.19	1.0
3878-117	CY-62	BRT	80%-1.7mm	4	48.8	9.87	10.37	20.24	15.54	0.09	1.3
3878-118	P-27	CLT	50mm	148	24.3	0.53	1.65	2.18	1.29	1.28	0.8
3878-118	P-32	CLT	80%-19mm	144	59.5	0.66	0.45	1.11	1.29	1.18	0.9
3878-118	P-37	CLT	80%-9.5mm	151	71.8	0.56	0.22	0.78	1.29	1.18	1.0
3878-118	CY-63	BRT	80%-1.7mm	4	62.7	0.89	0.53	1.42	1.29	0.10	1.2
3878-119	P-28	CLT	50mm	84	86.8	0.33	0.05	0.38	0.35	0.71	0.9
3878-119	P-29	CLT	50mm	84	76.2	0.32	0.10	0.42	0.35	1.11	0.9
3878-119	P-33	CLT	80%-19mm	65	91.7	0.33	0.03	0.36	0.35	0.44	1.0
3878-119	P-34	CLT	80%-19mm	77	91.7	0.33	0.03	0.36	0.35	0.57	1.0
3878-119	P-38	CLT	80%-9.5mm	50	97.0	0.32	0.01	0.33	0.35	0.52	1.1
3878-119	P-39	CLT	80%-9.5mm	50	97.1	0.33	0.01	0.34	0.35	0.50	1.1
3878-119	CY-64	BRT	80%-1.7mm	4	86.0	0.37	0.06	0.43	0.35	0.13	1.4

Table 13-6 - MLI 2014 / 2015 Phase I Gravity Concentration and Milling/Cyanidation Test Results

				Au Recovery, % of total			gAu/mt ore					Reagent Req., kg/mt ore	
							Ext'd.			Head Grade			
Composite/Description	Drill Hole	Depth, ft.	Test Type*	Grav. Conc.	CN (Grav. Tail)	Comb.	Grav. Conc.	CN Leach	Tail	Calc'd.	Assay	NaCN Cons.	Lime Added
3878-017 (Rhy-Stg Sil & Stg Ox-HG)	CMM-014	325.5-339	CN	N/A	N/A	98.7	N/A	1.54	0.02	1.56	1.12	0.45	1.5
3878-017 (Rhy-Stg Sil & Stg Ox-HG)	CMM-014	325.5-339	Grav./CN	57.4	41.4	98.8	0.85	0.61	0.02	1.48	1.12	<0.07	2.3
3878-020 (Cong-Stg Sil, Stg Ox & Wk S⁼)	CMM-014	479-503	CN	N/A	N/A	98.6	N/A	9.26	0.13	9.39	9.23	0.16	2.0
3878-020 (Cong-Stg Sil, Stg Ox & Wk S⁼)	CMM-014	479-503	Grav./CN	18.2	80.2	98.4	1.51	6.64	0.13	8.28	9.23	<0.07	2.5
3878-022 (Cong-Mod Sil & Mod Ox)	CMM-014	536-567	CN	N/A	N/A	99.4	N/A	9.94	0.06	10.00	10.56	0.22	2.2
3878-022 (Cong-Mod Sil & Mod Ox)	CMM-014	536-567	Grav./CN	41.2	58.1	99.3	4.16	5.88	0.07	10.11	10.56	<0.07	2.6
3878-023 (Cong-Wk Sil & Mod Ox)	CMM-014	567-594	CN	N/A	N/A	99.1	N/A	7.80	0.07	7.87	7.87	0.15	2.5
3878-023 (Cong-Wk Sil & Mod Ox)	CMM-014	567-594	Grav./CN	21.9	77.1	99.0	1.67	5.91	0.07	7.65	7.87	<0.07	2.7
3878-024 (Cong-Mod Sil & Stg Ox)	CMM-014	599-629	CN	N/A	N/A	99.8	N/A	48.11	0.12	48.23	48.73	0.16	2.2
3878-024 (Cong-Mod Sil & Stg Ox)	CMM-014	599-629	Grav./CN	44.3	55.4	99.7	18.60	23.28	0.14	42.02	48.73	<0.07	2.6
3878-026 (Rhy-Stg Sil, Ox & Mod Sulf-HG)	CMM-017	343-368.5	CN	N/A	N/A	99.0	N/A	8.84	0.09	8.93	12.52	0.20	1.6
3878-026 (Rhy-Stg Sil, Ox & Mod Sulf-HG)	CMM-017	343-368.5	Grav./CN	87.1	12.6	99.7	9.96	1.45	0.03	11.44	12.52	<0.07	1.4

* CN denotes whole ore milling cyanidation. Grav./CN denotes whole ore gravity concentration, with agitated cyanidation of the recombined gravity (cleaner and rougher) tailings.

13.5.2 Compacted Permeability Testing

Seven heap leach column residues were submitted to Applied Soil Water Technologies in Sparks, Nevada for Load vs. Hydraulic Conductivity Testing, and preliminary Load/Permeability Results were tabulated. The results were generally positive with five column residues performing well. One of the seven column residues (Comp. 107 @ 3/4 inch (19 mm)) showed poor permeability from a 50' simulated stack height onwards. Another residue (Comp. 116 @ 3/8 inch (9.5 mm)) showed poor to moderately poor permeability from 75-foot simulated stack height onwards (Gray, 2016).

13.5.3 Comminution Testing

Physical testing including Bond Crusher Work Index (CWi) and Bond Abrasion Index (Ai) was conducted by FLSmidth. The work was conducted on four rock types most commonly associated with mineralization at the Castle Mountain project. The test results based on CWi and Ai classify the rock types as Very Soft to Medium, as shown in Table 13.7 below, suggesting relatively low required crusher energy and wear part replacements.

Table 13-7 - Bond Crusher Index and Bond Abrasion Index Results (2014)

Client Sample ID	Number of Samples Tested	Relative Density	Crusher Index (kwh/short t)	Work (kwh/metric t)	Classification	Bond Abrasion Index (grams)	Classification
Ash Tuff	20	2.11	9.7	10.7	Very Soft/Soft	0.0115	Very Soft
Conglomerate Multi-Lithic	20	2.13	13.6	15.0	Soft/Medium	0.2165	Medium
Rhyolite	20	2.30	2.3	15.3	Soft/Medium	0.2978	Medium
Rhyolite-Breccia	20	2.19	2.19	15.4	Soft/Medium	0.1602	Soft

13.5.4 McClelland 2015 - Phase 2

In 2015, two bulk ROM, -18 inch (457 mm), samples from the Oro Belle and JSLA deposits were tested by MLI for heap leach cyanidation testing. The Oro Belle sample was an in-situ sample and the JSLA sample was from mine backfill in the JSLA pit. A pilot column leach test was conducted on each ROM sample to evaluate amenability to heap leaching without crushing. Comparative column leach tests were conducted on each sample at an 80% -3/8 inch (9.5mm) feed size to determine heap leach amenability of the crushed feeds. Bottle roll tests were also conducted on each sample at an 80% -12 mesh (1.7mm) feed size. Results from the test work are summarized in Table 13.8 and Figure 13.4 (the latter for column tests only).

Table 13-8 - ROM and 9.5mm Crushed Column Tests from Oro Belle and JSLA Areas, MLI 2015 Phase 2

					gAu/mt ore					gAg/mt ore				Reagent Req., kg/mt ore	
Sample	Feed Size	Leach Time, days	Test Type	Au Rec., %	Ext'd.	Tail	Calc'd. Head	Avg. Head	Ag Rec., %	Ext'd.	Tail	Calc'd. Head	Avg. Head	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

Note: CLT denotes column leach test. BRT denotes bottle roll test.

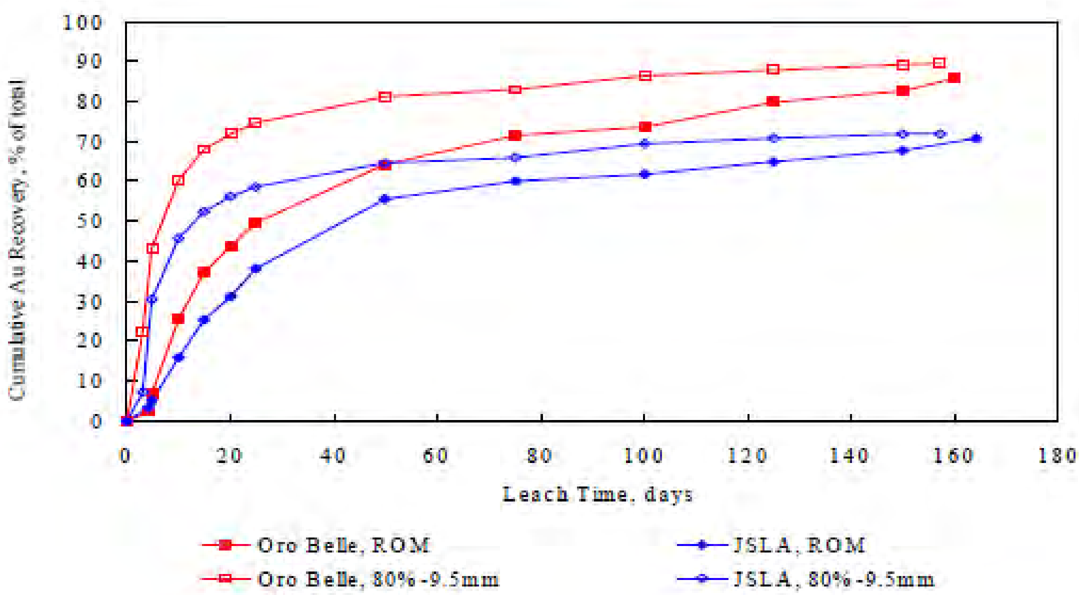

Figure 13-4 - Leach Curves from ROM and 9.5mm Crushed Column Tests from MLI 2015 Phase 2, Oro Belle and JSLA Areas

The column test results indicate high amenability to heap leaching for both samples at both crush sizes. Crushed 3/8 inch (9.5mm) samples showed slightly higher recoveries than ROM samples (about 4% for Oro Belle samples and 1% for JSLA samples) after about 160 days of leaching. However, MLI noted that gold extraction was still progressing at a slow, but significant rate in all samples when leaching was terminated and believed that given a longer leach cycle, gold recoveries may eventually be the same between ROM and crushed samples for both composites.

13.5.5 McClelland 2017 / 2018

Additional work was conducted by McClelland during the development of the PFS, which focused on detailed simulation of gravity concentration, cyanidation, and pulp agglomeration. A total of 212 spatially representative drill core intervals (HQ and PQ) were used for compositing.

The PQ portion of these (82) were used to produce a single JSLA-LG (low-grade composite) for heap leach tests. Column leach tests were conducted on this composite, at 80% passing 2 inch (50 mm) and 3/8 inch (9.5 mm) respectively, the results of which are presented in Table 13.9. Comparative bottle rolls were also conducted at the same feed sizes.

The HQ core intervals (130) were used to composite 18 low-grade, and nine high-grade samples for variability tests. The low-grade samples were bottle rolled at 80% passing 3/8 inch (9.5 mm) for 20 days. The high-grade samples were subjected to gravity concentration and cyanidation of the gravity tails. A tenth high-grade sample was subjected to grind size optimization tests (with gravity and cyanidation) and comminution testing.

Table 13-9 - Summary 2017 / 2018 Column Tests – JSLA Low-grade Master Composite

Feed Size	Leach/Rinse Time, Days	Au Recovery, %	gAu/mt ore Extracted	gAu/mt ore Tail	gAu/mt ore Calc'd. Head	gAu/mt ore Avg. Head	Reagent Requirements, kg/mt ore NaCN Cons.	Reagent Requirements, kg/mt ore Lime Added
80%-50mm	182	77.3	0.51	0.15	0.66	0.68	1.22	0.5
80%-9.5mm	180	85.1	0.57	0.10	0.67	0.68	1.38	0.7

Table 13-10 - Summary 2017 / 2018 Variability Bottle Roll Tests 80% -3/8 inch

Composite	Description	Drill Hole, No.	Interval, ft.	Au Recovery, %	gAu/mt ore Ext'd.	gAu/mt ore Tail	gAu/mt ore Calc'd. Head	gAu/mt ore Head Assay	Reagent Requirements, kg/mt ore NaCN Cons.	Reagent Requirements, kg/mt ore Lime Added
4210-039*	JSLA LG MC	---	---	70.2	0.40	0.17	0.57	0.74	<0.10	0.7
4210-039	JSLA LG MC	---	---	80.0	0.52	0.13	0.65	0.74	<0.10	0.9
4210-040	South Domes LG	CMM-016	637-652	75.6	0.31	0.10	0.41	0.29	<0.10	0.7
4210-041	South Domes LG	CMM-016	777-792	89.1	0.49	0.06	0.55	0.51	<0.10	0.7
4210-042	South Domes LG	CMM-111	838-858	88.9	0.48	0.06	0.54	0.52	<0.10	1.0
4210-043	South Domes LG	CMM-111	910-930	85.1	0.57	0.10	0.67	0.54	<0.10	1.1
4210-044	South Domes LG	CMM-111	930-950	92.0	0.23	0.02	0.25	0.18	<0.10	0.8
4210-045	South Domes LG	CMM-122C	796.5-811.5	81.0	0.34	0.08	0.42	0.32	<0.10	1.0
4210-046	South Domes LG	CMM-122C	997.7-1017	82.2	0.60	0.13	0.73	0.43	<0.10	1.3
4210-047	South Domes LG	CMM-122C	1267-1280.5	84.1	0.58	0.11	0.69	0.27	<0.10	1.6
4210-048	Andesite LG	CMM-016	1089-1109	90.6	0.29	0.03	0.32	0.29	<0.10	1.8
4210-049	Andesite LG	CMM-111	1471-1481	88.4	0.38	0.05	0.43	0.33	0.25	4.3
4210-050	Andesite LG	CMM-130C	1452-1467	35.6	0.16	0.29	0.45	0.32	0.28	2.8
4210-051	Oro Belle LG	CMM-036	470.5-484	67.3	0.37	0.18	0.55	0.52	<0.10	1.1
4210-052	Oro Belle LG	CMM-036	880-893	91.7	0.77	0.07	0.84	0.78	<0.10	1.0
4210-053	Oro Belle LG	CMM-119	407-420	75.0	0.42	0.14	0.56	0.57	<0.10	1.3
4210-054	Oro Belle LG	CMM-119	553-573.5	90.3	1.30	0.14	1.44	2.03	0.16	1.5
4210-055	Oro Belle LG	CMM-119	1030-1040	78.2	0.43	0.12	0.55	0.77	<0.10	2.1
4210-056	Oro Belle LG	CMM-120	542-557	67.2	0.41	0.20	0.61	0.59	<0.10	1.4
4210-057	Oro Belle LG	CMM-120	1304-1324	32.1	0.09	0.19	0.28	0.25	<0.10	0.7

* 80%-50mm feed size.

The lowest bottle roll gold recoveries were from two of the deepest samples tested. Drill holes CMM–130C, interval 1452'-1467', and CMM-120, interval 1304'-1324', had gold recoveries of 35.6% and 32.1% respectively. These also had elevated levels of sulfide sulfur 0.68% and 1.14% respectively. Gold recovery from the remaining 17 samples averaged 82.7%.

Bottle roll recoveries obtained from the JSLA-LG composite at the 2 inch (50 mm) and 3/8 inch (9.5 mm) crush size were 70.2% and 80.0% respectively. These bottle roll recoveries were incrementally lower by 5-7 % than the matching column tests.

Table 13-11 - Summary 2017/2018 Gravity Concentration / Cyanidation Tests- High-grade Cores

Comp.	Feed Size, P_{80}	Description	Drill Hole No.	Interval, ft.	Au Recovery, % of Total Grav. Conc.	Au Recovery, % of Total CN (Grav. Tail)	Au Recovery, % of Total Comb.	gAu/mt Ext'd Grav. Conc.	gAu/mt Ext'd CN Leach	gAu/mt Tail	gAu/mt Head Grade Calc'd.	gAu/mt Head Grade Assay	Reagent Req., kg/mt ore NaCN Cons.	Reagent Req., kg/mt ore Lime Added
4210-058	-150µm	South Domes HG	CMM-111	957.5-986	34.6	54.5	89.1	0.76	1.20	0.24	2.20	3.60	<0.10	1.4
4210-059	-150µm	South Domes HG	CMM-111	1011-1031	38.9	57.0	95.9	1.34	1.97	0.14	3.45	4.67	<0.10	1.2
4210-060	-150µm	South Domes HG	CMM-122C	940.5-988.4	45.6	50.8	96.4	1.14	1.27	0.09	2.50	3.95	<0.10	1.9
4210-061	-150µm	South Domes HG	CMM-130C	1182-1212	20.9	70.2	91.1	0.28	0.95	0.12	1.35	1.76	<0.10	1.4
4210-062	-150µm	South Domes HG	CMM-248C	659-699	33.8	60.9	94.7	0.70	1.26	0.11	2.07	2.55	<0.10	1.8
4210-063	-150µm	Andesite HG	CMM-111	1416.5-1441.5	55.5	35.1	90.6	0.47	0.30	0.08	0.85	1.08	0.12	3.2
4210-064	-150µm	Andesite HG	CMM-130C	1246-1271.5	32.4	58.8	91.2	1.10	2.00	0.30	3.40	3.41	<0.10	2.9
4210-065	-150µm	Oro Belle HG	CMM-036	230-262	17.9	76.9	94.8	1.46	6.31	0.43	8.20	13.77	<0.10	2.1
4210-066	-150µm	Oro Belle HG	CMM-036	644-667	23.8	68.5	92.3	0.62	1.79	0.20	2.61	2.91	<0.10	2.0
4210-067	-150µm	JSLA HG MC*	—	—	26.7	71.5	98.2	0.53	1.42	0.04	1.99	2.17	<0.10	1.1
4210-067	-300µm	JSLA HG MC	—	—	15.8	80.3	96.1	0.28	1.45	0.07	1.80	2.17	<0.10	0.9
4210-067	-212µm	JSLA HG MC	—	—	19.8	76.0	95.8	0.37	1.43	0.08	1.88	2.17	<0.10	0.8
4210-067	-150µm	JSLA HG MC	—	—	18.6	78.7	97.3	0.34	1.43	0.05	1.82	2.17	<0.10	0.8
4210-067	-106µm	JSLA HG MC	—	—	21.3	76.7	98.0	0.42	1.50	0.04	1.96	2.17	<0.10	0.8

* Average of 3 tests.

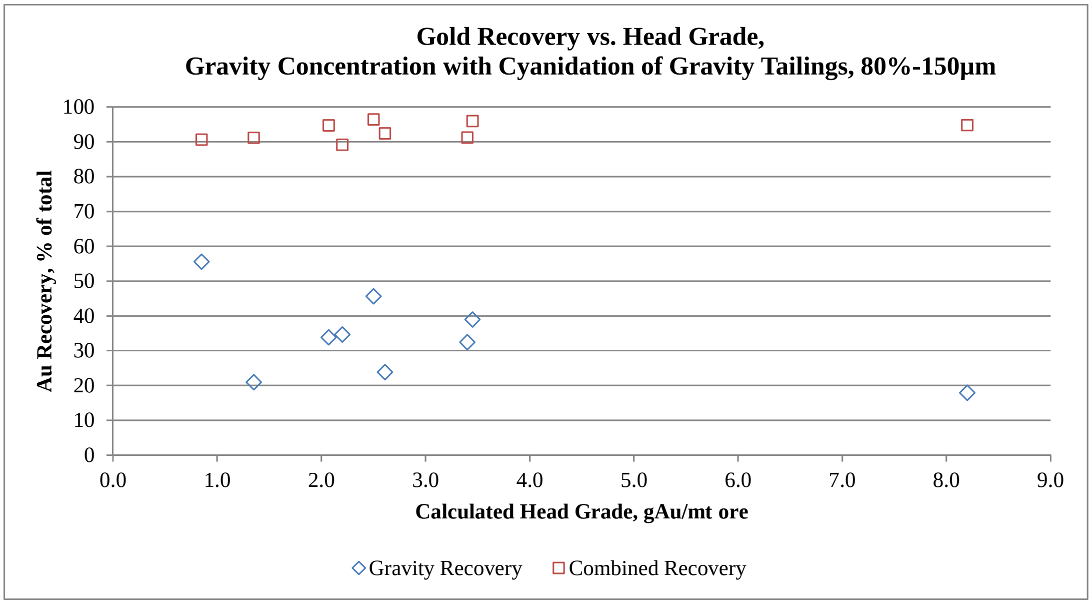

Figure 13-5 - Gravity and Combined Gravity / Cyanidation Recovery vs. Head Grade

The gravity cleaner concentrates ranged from 0.13% to 0.22% of the feed weight, with an average of 0.17%. The gravity concentrate assayed between 5.1 and 26.1 oz/ton (176-896 g/t) Au, and represented gold recoveries of between 17.9 and 55.5% (33.1% average).

13.5.6 Grind vs. Recovery

A grind size optimization series was run on the tenth high-grade master composite. Grind sizes evaluated ranged from 80% -50 mesh (300 µm) to 80% -140 mesh (106 µm). The results of this series are shown in Table 13.12.

Table 13-12 - Grind Size vs. Gold Recovery

					gAu/mt ore				
					Extracted			Head Grade	
	Size	Grav. Conc.	CN (Grav. Tail)	**Combined**	Grav. Conc.	CN Leach	Tail	Calculated	Assayed
CY-32/G-12	80%-300µm	15.8	80.3	**96.1**	0.28	1.45	0.07	1.80	2.17
CY-33/G-13	80%-212µm	19.8	75.9	**95.7**	0.37	1.42	0.08	1.87	2.17
CY-34/G-14	80%-150µm	18.7	78.5	**97.2**	0.34	1.42	0.05	1.81	2.17
CY-35/G-15	80%-106µm	21.3	76.7	**98.0**	0.42	1.49	0.04	1.95	2.17

As can be seen, recoveries are all high and show only slightly higher recoveries with finer grind size.

Cyanide consumptions were very low for all tests, at less than 0.1 kg/mt.

13.5.7 Ore Characterization

A Bond ball mill work index was determined for the high-grade JSLA composite 4210-067. The work index was 18.03 kW-hr/st, indicative of a relatively hard material. The results are presented in Table 13.13 below.

Table 13-13 - Bond Ball Mill Grindability Test Summary

Composite	100% Passing Size Product	80% Passing Size Product	80% Passing Size Feed	Grams per Revolution	Bond Work Index kWh/st	Bond Work Index kWh/mt
4210-067	150um	109um	2667µm	1.02	18.03	19.87

Copper head grades ranged from 5 to 228 ppm (38 ppm average). Cyanide soluble copper ranged from 2% to 45% of the head grade (18% average).

Total Carbon and Sulfur speciation analysis are shown in Table 13.4 below.

Table 13-14 - Carbon and Sulfur Speciation

Composite	Description	Drill Hole No.	Interval, ft.	Total Carbon, %	% S Total	% S Sulfate	% S Sulfide
4210-039	JSLA LG MC	---	---	0.03	0.02	0.01	0.01
4210-040	South Domes LG	CMM-016	637-652	0.02	<0.01	<0.01	<0.01
4210-041	South Domes LG	CMM-016	777-792	0.02	<0.01	<0.01	<0.01
4210-042	South Domes LG	CMM-111	838-858	0.02	0.01	<0.01	<0.01
4210-043	South Domes LG	CMM-111	910-930	0.01	0.01	<0.01	<0.01
4210-044	South Domes LG	CMM-111	930-950	0.01	0.01	<0.01	<0.01
4210-045	South Domes LG	CMM-122C	796.5-811.5	0.02	<0.01	<0.01	<0.01
4210-046	South Domes LG	CMM-122C	997.7-1017	0.01	0.01	<0.01	<0.01
4210-047	South Domes LG	CMM-122C	1267-1280.5	0.01	0.01	<0.01	<0.01
4210-048	Andesite LG	CMM-016	1089-1109	0.08	0.03	0.02	<0.01
4210-049	Andesite LG	CMM-111	1471-1481	0.45	0.09	0.04	0.05
4210-050	Andesite LG	CMM-130C	1452-1467	0.93	1.65	0.51	1.14
4210-051	Oro Belle LG	CMM-036	470.5-484	0.02	0.03	0.01	0.01
4210-052	Oro Belle LG	CMM-036	880-893	0.02	0.03	0.02	0.01
4210-053	Oro Belle LG	CMM-119	407-420	0.04	0.01	<0.01	<0.01
4210-054	Oro Belle LG	CMM-119	553-573.5	0.04	0.02	0.01	<0.01
4210-055	Oro Belle LG	CMM-119	1030-1040	0.02	0.03	0.02	<0.01
4210-056	Oro Belle LG	CMM-120	542-557	0.02	0.02	< 0.01	<0.01
4210-057	Oro Belle LG	CMM-120	1304-1324	0.33	1.14	0.46	0.68
4210-058	South Domes HG	CMM-111	957.5-986	0.06	<0.01	<0.01	<0.01
4210-059	South Domes HG	CMM-111	1011-1031	0.04	<0.01	<0.01	<0.01
4210-060	South Domes HG	CMM-122C	940.5-988.4	0.03	<0.01	<0.01	<0.01
4210-061	South Domes HG	CMM-130C	1182-1212	0.02	<0.01	<0.01	<0.01
4210-062	South Domes HG	CMM-248C	659-699	0.03	<0.01	<0.01	<0.01
4210-063	Andesite HG	CMM-111	1416.5-1441.5	1.11	0.71	0.11	0.60
4210-064	Andesite HG	CMM-130C	1246-1271.5	0.04	0.01	<0.01	0.01
4210-065	Oro Belle HG	CMM-036	230-262	0.05	<0.01	<0.01	<0.01
4210-066	Oro Belle HG	CMM-036	644-667	0.03	<0.01	<0.01	<0.01
4210-067	JSLA HG MC	---	---	0.07	0.02	<0.01	0.01

Total carbon content was less than 1% and sulfide sulfur was very low in all but three of the deepest core intervals.

ICP scans and whole rock analysis were also conducted on all composites.

13.5.8 Pulp Agglomeration Slurry Ratio Tests

Preliminary agglomeration tests were conducted on the JSLA-LG composite at 80% -3/8 inch (9.5 mm) blended with various ratios of thickened slurry of the same composite to simulate pulp agglomeration at these various ratios and optimized cement dosages. Specifically, ratios of 4:1, 6:1, and 9:1 of 3/8" crushed ore to 100 mesh slurry (on a dry basis) were tested, along with a sample of 3/8" crushed ore with no slurry addition. The results are summarized in Table 13.15.

Table 13-15 - Summary Agglomeration Tests – Pulp Agglomerates, 80% -3/8 inch

Test	Blend LG	Blend HG	Ratio LG:HG	Cement, kg/mt	Agglom. Moisture	Agglomerate Description	Bulk Density, kg/m³ Initial	Bulk Density, kg/m³ Day 3	Bulk Density, kg/m³ Final	Slump	Maximum Solution App. Rate, Lph/m²
PA-1	100%	0%	N/A	0.0	5.5%	1)	1,368	1,368	1,395	2.0%	6.70E+03
PA-2	100%	0%	N/A	2.0	5.5%	1)	1,368	1,368	1,368	0.0%	6.70E+03
PA-3	100%	0%	N/A	4.0	5.6%	1)	1,341	1,341	1,341	0.0%	6.70E+03
PA-4	100%	0%	N/A	6.0	5.5%	1)	1,328	1,328	1,328	0.0%	6.70E+03
PA-5	10%	90%	9:1	0.0	9.4%	2)	1,409	1,439	1,570	11.4%	2.10E+04
PA-6	10%	90%	9:1	2.0	7.9%	3)	1,303	1,328	1,381	6.0%	4.19E+04
PA-7	10%	90%	9:1	4.0	8.2%	3)	1,303	1,328	1,368	5.0%	2.10E+04
PA-8	10%	90%	9:1	6.0	8.2%	3)	1,279	1,303	1,303	1.9%	4.19E+04
PA-9	86%	14%	6:1	0.0	8.7%	4)	1,473	1,473	1,539	4.4%	2.10E+04
PA-10	86%	14%	6:1	2.0	8.7%	3)	1,442	1,442	1,505	4.3%	2.10E+04
PA-11	86%	14%	6:1	4.0	8.7%	3)	1,358	1,358	1,385	2.0%	2.10E+04
PA-12	86%	14%	6:1	6.0	8.7%	3)	1,371	1,385	1,385	1.0%	4.19E+04
PA-13	80%	20%	4:1	0.0	11.8%	5)	1,818	1,867	2,031	11.8%	5.24E+03
PA-14	80%	20%	4:1	2.0	11.8%	6)	1,501	1,535	1,606	7.0%	1.05E+04
PA-15	80%	20%	4:1	4.0	11.8%	6)	1,381	1,409	1,501	8.7%	1.05E+04
PA-16	80%	20%	4:1	6.0	11.8%	6)	1,291	1,303	1,354	4.9%	2.10E+04

1) Very few agglomerates formed during agglomeration. There was not much change in the appearance of the feed during testing.
2) Initial agglomerates looked good. Minor degradation of agglomerates during testing.
3) Initial agglomerates looked good. Very little to no degradation of agglomerates during testing.
4) Initial agglomerates looked too wet. Minor degradation of agglomerates during testing.
5) Initial agglomerates looked too wet / poor quality. Significant degradation of agglomerates was observed during solution application at standard rate.
6) Initial agglomerates looked too wet / poor quality. Little to no change in agglomerates was observed during testing.

Agglomeration test results indicate that the crushed low-grade ore without slurry probably does not any require cement. The 9:1 and 6:1 blend ratios required a minimum of 4 lb/st (2 kg/mt) cement for proper agglomeration. The 1:4 blend ratio contained too much solution for quality agglomerates to be formed.

13.5.9 Crush Size vs. Recovery Column Tests

Column tests were run on 80% passing 2 inch (50 mm) and 3/8 inch (9.5 mm) of the low-grade master composite (JSLA-LG) with no cement addition to establish the difference in recovery between a two-stage and three-stage crush.

The 9.5 mm crush size resulted in almost 7% higher gold recovery than the 2 inch (50 mm) crush size, which would suggest a finer crush size is justified for pulp agglomeration. However, note the results presented in earlier test work on JSLA ROM and 3/8" crushed columns (Figure 13-4) suggested a less significant recovery dependence on crush size. The detailed results of the crushed ore column tests are shown in Table 13.16 and Figure 13.6.

Table 13-16 - Column Leach Tests, Size vs. Recovery, Low-grade Master Composite

Feed Size:	80%-50mm	80%-9.5mm
Metallurgical Results	P-1	P-2
Extraction: % of total Au		
1st Effluent	3.1	30.8
in 5 days	32.8	60.6
in 10 days	46.0	71.5
in 15 days	52.1	74.9
in 20 days	56.2	76.8
in 30 days	61.3	78.9
in 40 days	64.6	80.3
in 50 days	66.6	80.9
in 60 days	68.8	81.9
in 75 days	69.6	82.3
in 100 days	73.2	83.6
in 120 days	74.5	84.1
in 140 days	76.0	84.4
in 160 days	77.0	84.8
End of Leach/Rinse	77.3	85.1
Extracted, gAu/mt ore	0.51	0.57
Tail Screen, gAu/mt ore	0.15	0.10
Calc'd. Head, gAu/mt ore	0.66	0.67
Average Head, gAu/mt ore[1]	0.68	0.68
Ag Extraction, % of total	5.0	7.7
Extracted, gAg/mt ore	0.1	0.2
Tail Screen, gAg/mt ore	1.9	2.4
Calc'd. Head, gAg/mt ore	2.0	2.6
Head Assay, gAg/mt ore[2]	1.0	1.0
Extracted, gCu/mt ore	2.4	2.8
Head Assay, gCu/mt ore[3]	15	15
NaCN Consumed, kg/mt ore	1.22	1.38
Lime Added, kg/mt ore	0.5	0.7
Final Solution pH	10.0	9.9
pH After Rinse	9.7	10.0
Leach/Rinse Cycle, Days	182	180

1) Average of all head grade determinations.
2) Average of duplicate assays.
3) Based on ICP scan.

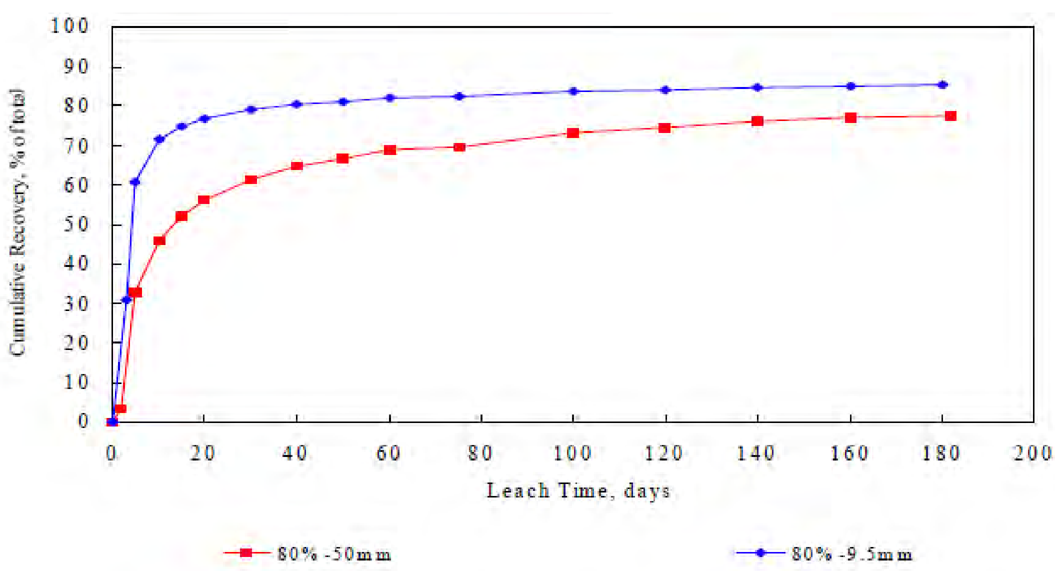

Figure 13-6 - Column Test Leach Curves, Size vs. Recovery, Low-grade Master Composite

A metallurgical balance of the column tests is presented in Table 13.17, which shows very close agreement between the calculated head grades, which implies a very accurate result.

Table 13-17 - Metallurgical Balance, Column Tests, Low-grade Master Composite

	Metallurgical Balance		
	Sol. vs. Tail	Carbon vs. Tail	Head vs. Tail[2]
80%-50mm Feed Size			
Extracted, gAu/mt ore	0.51	0.49	0.53
Tail Assay, gAu/mt	0.15	0.15	0.15
Calc'd. Head, gAu/mt	0.66	0.64	0.68
Deviation, gAu/mt[1]	N/A	0.02	0.02
Recovery, %	77.3	76.6	77.9
Precision, %	N/A	97.0	97.0
80%-9.5mm Feed Size			
Extracted, gAu/mt ore	0.57	0.59	0.58
Tail Assay, gAu/mt	0.10	0.10	0.10
Calc'd. Head, gAu/mt	0.67	0.69	0.68
Deviation, gAu/mt[1]	N/A	0.02	0.01
Recovery, %	85.1	85.5	85.3
Precision, %	N/A	97.0	98.5

1) Deviation from solution versus tail balance.
2) Calculated, based on average of all head grades and tail screen results.

The physical characteristics of the two column tests are shown in Table 13.18.

Table 13-18 - Physical Characteristics, JSLA Low-grade Master Composite Column Tests

Test No.	Feed Size	Ore Charge, kg	Moisture, wt. % As Rec'd.	Moisture, wt. % To Saturate*	Moisture, wt. % Retained	Apparent Bulk Density mt ore/m³ Before	Apparent Bulk Density mt ore/m³ After
P-1	80%-50mm	258.41	0.6	8.3	7.2	1.44	1.44
P-2	80%-9.5mm	66.62	0.5	11.8	8.9	1.42	1.43

* Calculated on a dry ore weight basis.

13.5.10 Thickener and Filtration Testing of High-Grade Slurry

Slurry for thickener and filtration testing was prepared from the high-grade JSLA composite sample. Summary results are presented in Tables 13.19, 13.20 and 13.21.

Table 13-19 - High-Grade Composite Size Distribution for Thickener and Filtration Tests

Material	Average Liquor S.G.	Average Solids S.G.	pH	P_{90} (micron)	P_{80} (micron)	D_{50} (micron)	% Passing 25 micron
CY 30/30B	1.00	2.79	10.8	183	146	88	1.61

Table 13-20 - Summary Results Thickener Testing

Sample Name	Floc Type	Floc Dose (g/MT)(1)	Floc Conc. (g/L)(2)	Max Thk Feed Solids (%)(3)	Minimum Unit Area for Conventional Thickener Sizing (m²/MTPD)(4)	Hydraulic Rate for High-Rate Thickener Sizing (m³/m²•hr)(5)	Estimated U'Flow Density for Standard Thickener (%)(5)	Thickener Type Recommended
CY 30/30B	AN 910 SH	55 - 60	0.1 – 0.2	20% – 30% **Conv. Type** 20% - 25% **High-Rate**	0.125 – 0.150 (min. sizing)	4.50 - 5.00	56% - 60%	Standard Conventional Type or Standard High-Rate

Table 13-21 - Summary Results Filtration Testing

Material	Design Tonnage (MTPD) dry solids(1)	Filter Feed Solids	Dry Bulk Cake Density (kg/m^3)	Recess Plate Depth(2) (mm)	Chamber Spec.(3) (Len./Vol./Area) ($mm/m^3/m^2$)	Air Blow Time(4)	Filter Cake Moist. (%)(5)	Filter Cycle Time (min)	Pressure Filter Chambers Required/ Number of Presses Required(6)
CY 30/30B	10	58.2%	1,439.5	15	500/0.006/0.41	3.0	12.8%	12.0	16 / 1

13.5.11 McClelland and RDi 2017 – ROM Tests on JSLA Backfill

Four bulk samples of JSLA pit backfill material were taken from excavated pits for ROM tests by McClelland. One column test at -18 inch (457 mm) feed size was conducted for each sample. Bulk sample weights ranged from 9-10 tons for each column test. Also, splits of these were used for comparative flooded vat leach tests. Each sample from each test pit was tested with the flooded vat leach test -18 inch (457 mm) and -6 inch (152 mm) sizes. Bottle roll tests were also conducted on each sample at 100% -2 inch (50 mm) and 80% -3/8 inch (9.5 mm).

On a separate campaign, with RDi, five other test pits were excavated for bulk samples to be used also for flooded vat leach tests, and were treated in the same way. Each sample from each pit tested -18 inch (457 mm) and – 6 inch (152 mm) sizes. Bottle roll tests were also conducted on each sample at 100% -2 inch (50 mm) and 80% - 3/8 inch (9.5 mm).

The sample locations for the bulk samples are shown in Figure 13.7, where the numbers in black are the McClelland test pits and the numbers in red are the RDi test pits.

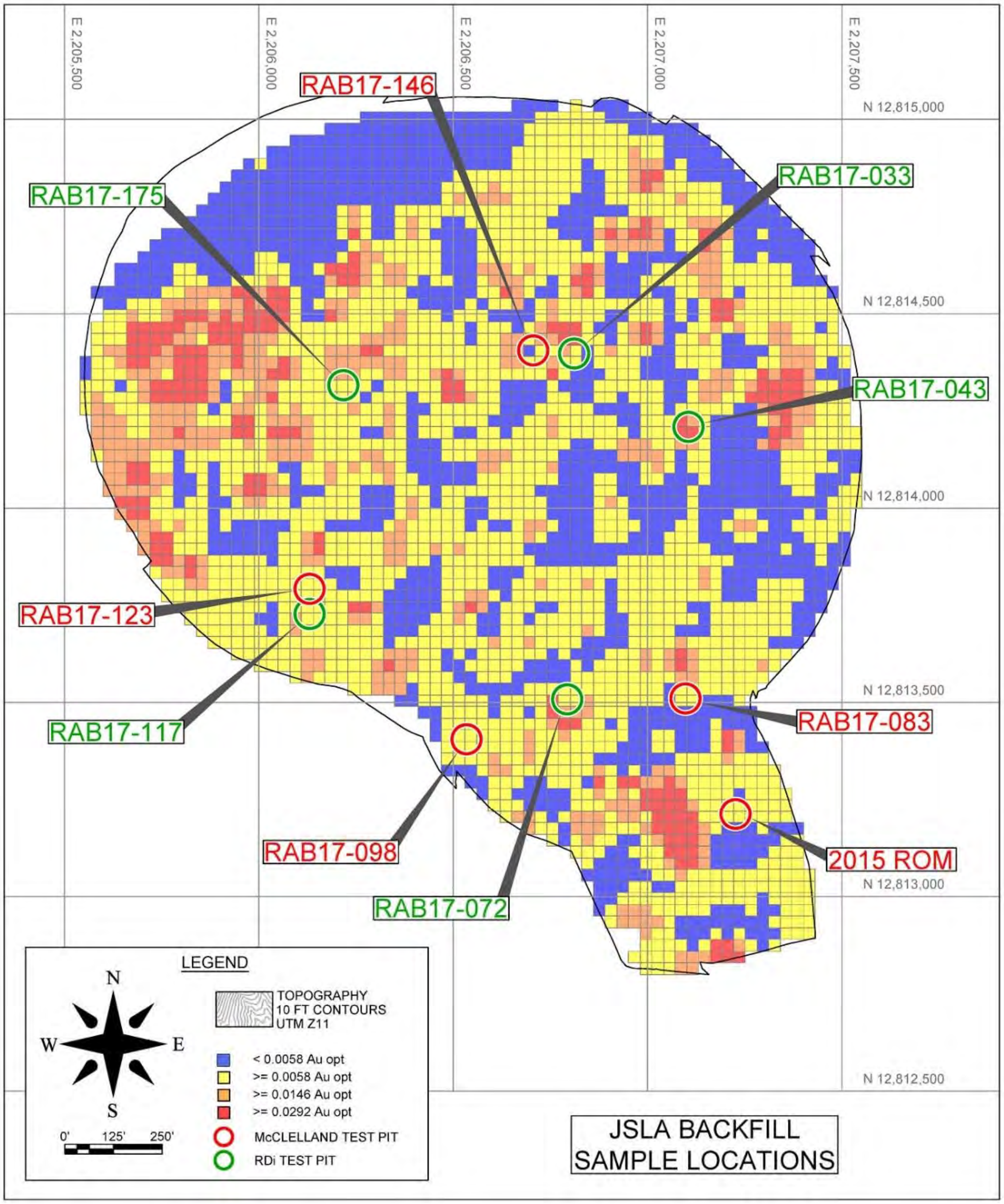

Figure 13-7 - Location Map – JSLA Bulk Sample Pits

13.5.12 McClelland ROM Tests JSLA Backfill

Summary results of the McClelland work, which includes the bulk ROM column tests are presented in Table 13.22.

Table 13-22 - Summary Results JSLA Bulk Samples

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).

For the column tests, gold extraction was continuing at a slow rate when leaching was stopped at 132 days, plus a rinse to 173 days. No significant extraction was noted during the extended 34-day rinse.

The gold recovery for the four column tests ranged from 65.5% to 85.1%, and average 76.7%. The -18 inch (460 mm) bucket tests also showed similar recoveries as the column tests with the exception of sample 098, which showed only 33.3%.

Cyanide consumptions (column tests) ranged from 0.44 to 0.64 lb/st (0.22 to 0.32 kg/mt), and (2.2 to 3.6 lb/st) 1.1 to 1.8 kg/mt lime addition was sufficient to control pH. Field cyanide consumption could be expected to be 0.16 to 0.22 lb/st (0.8 to 0.11 kg/mt).

An analysis and comparison of all of the head grades calculations for the bulk samples are shown in Table 13.23, confirming the ore tested is representative of the ROM grade class.

Table 13-23 - Head Grades – JSLA ROM Bulk Samples

	Head Grade, gAu/mt ore			
Determination Method	RAB 17-083	RAB 17-098	RAB 17-123	RAB 17-146
Direct Assay, Init.	0.341	0.194	0.452	0.116
Direct Assay, Dup.	0.262	0.110	0.340	0.145
Direct Assay, Trip.	0.357	0.140	0.521	0.130
Calc'd. BRT, 100%-50mm	0.36	0.20	0.61	0.13
Calc'd. BRT, 80%-9.5mm	0.42	0.19	0.57	0.13
Calculated, Head Screen, ROM	0.35	0.48	0.45	0.18
Calc'd. Column, ROM (-460mm)	0.39	0.29	0.47	0.21
Calc'd. Bucket, ROM (-460mm)	0.48	0.87	0.52	0.26
Calc'd. Bucket, -150mm	0.72	0.38	0.74	0.26
Average	0.41	0.32	0.52	0.17
Std. Deviation	0.13	0.24	0.11	0.06

Detailed results of the bulk column tests are shown in Table 13.24 and Figure 13.8.

Table 13-24 - ROM Column Test Results, JSLA Bulk Samples -18 inch

Sample:	RAB 17-083	RAB 17-098	RAB 17-123	RAB 17-146
Test No.:	P-4	P-1	P-3	P-2
Metallurgical Results				
Extraction: % of total				
1st Effluent		1.2		3.0
in 5 days	1.3[2]	4.3	2.5[2]	7.6
in 10 days	28.2	17.5	34.7	28.8
in 15 days	49.1	26.2	53.1	42.0
in 20 days	59.8	33.7	63.1	48.3
in 30 days	70.6	44.5	71.9	57.4
in 40 days	75.5	51.5	76.4	62.6
in 50 days	77.8	55.6	78.9	64.2
in 60 days	79.9	58.6	80.9	64.9
in 75 days	81.3	61.0	82.3	67.1
in 100 days	82.1	62.0	82.7	68.7
in 125 days	83.2	63.5	83.7	69.1
in 150 days	84.6	65.5	85.1	71.4
End of Leach/Rinse	84.6	65.5	85.1	71.4
Extracted, g/mt ore	0.33	0.19	0.40	0.15
Tail Screen, g/mt	0.06	0.10	0.07	0.06
Calculated Head, g/mt ore	0.39	0.29	0.47	0.21
Average Head, g/mt ore[1]	0.41	0.32	0.52	0.17
Ag Extraction, % of total	9.1	<9.1	11.1	<14.3
Extracted, gAg/mt ore	0.1	<0.1	0.1	<0.1
Tail Screen, gAg/mt	1.0	1.0	0.8	0.6
Calculated Head, gAg/mt ore	1.1	<1.1	0.9	<0.7
Average Head, gAg/mt ore[1]	<1.1	<1.0	<1.0	<1.0
NaCN Consumed, kg/mt ore	0.22	0.32	0.27	0.23
Lime Added, kg/mt ore	1.3	1.4	1.8	1.1
Final Solution pH	10.3	10.2	10.1	10.1
pH After Rinse	9.4	9.6	9.8	9.5
Leach/Rinse Cycle, Days	173	173	173	173

1) Average of all head grade determinations.
2) First effluent day is day 5.

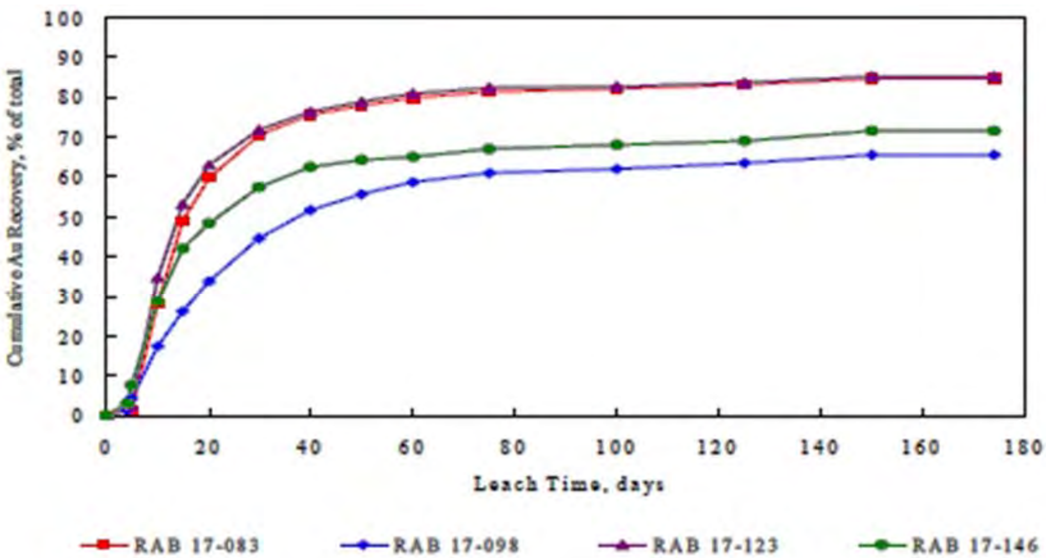

Figure 13-8 - Leach Curves, ROM Column Tests, JSLA Bulk Samples

A metallurgical Balance of the four bulk ROM column tests is presented in Table 13.25, which shows the calculated head grades agree quite closely, attesting to the accuracy of the results.

Table 13-25 - Metallurgical Balance, ROM Column Tests, JSLA Bulk Samples

	Metallurgical Balance		
	Sol. vs. Tail	Carbon vs. Tail	Head vs. Tail[2]
RAB 17-083 (P-4)			
Extracted, gAu/mt ore	0.33	0.29	0.35
Tail Assay, gAu/mt	0.06	0.06	0.06
Calc'd., Head, gAu/mt	0.39	0.35	0.41
Deviation, gAu/mt[1]	N/A	0.04	0.02
Recovery, %	84.6	82.9	85.4
Precision, %	N/A	89.7	94.9
RAB 17-098 (P-1)			
Extracted, gAu/mt ore	0.19	0.19	0.22
Tail Assay, gAu/mt	0.10	0.10	0.10
Calc'd., Head, gAu/mt	0.29	0.29	0.32
Deviation, gAu/mt[1]	N/A	<0.01	0.03
Recovery, %	65.5	65.5	68.8
Precision, %	N/A	>96.6	89.7
RAB 17-123 (P-3)			
Extracted, gAu/mt ore	0.40	0.39	0.45
Tail Assay, gAu/mt	0.07	0.07	0.07
Calc'd., Head, gAu/mt	0.47	0.46	0.52
Deviation, gAu/mt[1]	N/A	0.01	0.05
Recovery, %	85.1	84.8	86.5
Precision, %	N/A	97.9	89.4
RAB 17-146 (P-2)			
Extracted, gAu/mt ore	0.15	0.15	0.11
Tail Assay, gAu/mt	0.06	0.06	0.06
Calc'd., Head, gAu/mt	0.21	0.21	0.17
Deviation, gAu/mt[1]	N/A	<0.01	0.04
Recovery, %	71.4	71.4	64.7
Precision, %	N/A	>95.2	81.0

1) Deviation from solution versus tail balance.
2) Calculated, based on average of all head grades and tail screen results.

The physical characteristics of the samples used in the bulk ROM column tests are shown in Table 13.26.

Table 13-26 - Physical Ore Characteristics, ROM Column Tests, JSLA Bulk Samples

Sample	Test No.	Ore Charge, kg	Moisture, wt. % As Rec'd.	Moisture, wt. % To Saturate*	Moisture, wt. % Retained	Ore Apparent Bulk Density, mt/m³ Initial	Ore Apparent Bulk Density, mt/m³ Final
RAB 17-083	P-4	8,394.56	2.9	13.7	10.2	1.45	1.47
RAB 17-098	P-1	8,265.69	2.7	11.9	9.8	1.44	1.44
RAB 17-123	P-3	8,347.68	3.0	13.8	11.3	1.41	1.43
RAB 17-146	P-2	8,398.71	2.4	10.4	7.5	1.45	1.47

* Calculated on a dry ore weight basis.

The Mercury balances for the bulk ROM tests are shown in Table 13.27.

Table 13-27 - Mercury Balance, ROM Column Tests, JSLA Bulk Samples

	RAB 17-083, P-4	RAB 17-098, P-1	RAB 17-123, P-3	RAB 17-146, P-2
Hg Assays, gHg/mt				
Head	0.222	0.208	0.161	0.184
Tail	0.194	0.237	0.194	0.234
Loaded Carbon*	31.8	45.3	39.3	65.3
Weight, kg				
Column Charge	8,394.6	8,265.7	8,347.7	8,398.7
Loaded Carbon	3.059	3.063	3.071	3.066
Hg Extraction, % of total				
Carbon vs. Tail Balance**	5.6	6.6	6.9	9.3
Calculated Head Grade, gHg/mt ore				
Carbon plus Tail**	0.21	0.25	0.21	0.26

* Weighted average of the three carbon charges.

** Carbon vs. tail balance does not account for Hg in final barren, rinse effluent and solution analytical samples.

Drain down rates for the four ROM bulk column tests are shown in Table 13.28.

Table 13-28 - Drain Down Rates, ROM Column Tests, JSLA Bulk Samples

Drain	Effluent Solution					
	P-1, RAB 17-098, ROM (-460mm)			P-2, RAB 17-146, ROM (-460mm)		
Time, hours	Liters	Cum. L/mt ore	Rate, L/hr/mt	Liters	Cum. L/mt ore	Rate, L/hr/mt
0.08	0.735	0.09	1.11	0.520	0.06	0.77
0.25	2.045	0.34	1.46	1.520	0.24	1.06
0.5	2.090	0.59	1.01	1.660	0.44	0.79
1	3.720	1.04	0.90	3.530	0.86	0.84
2	5.455	1.70	0.66	7.080	1.70	0.84
4	7.740	2.64	0.47	14.870	3.47	0.89
8	10.195	3.87	0.31	29.170	6.95	0.87
24	23.770	6.74	0.18	32.162	10.78	0.24
48	18.170	8.94	0.09	21.300	13.31	0.11
72	12.295	10.43	0.06	13.040	14.87	0.06
96	9.290	11.55	0.05	6.435	15.63	0.03

Drain	Effluent Solution					
	P-3, RAB 17-123, ROM (-460mm)			P-4, RAB 17-083, ROM (-460mm)		
Time, hours	Liters	Cum. L/mt ore	Rate, L/hr/mt	Liters	Cum. L/mt ore	Rate, L/hr/mt
0.08	0.645	0.08	0.97	0.610	0.07	0.91
0.25	1.705	0.28	1.20	1.570	0.26	1.10
0.5	1.895	0.51	0.91	1.760	0.47	0.84
1	3.915	0.98	0.94	3.190	0.85	0.76
2	6.535	1.76	0.78	4.950	1.44	0.59
4	9.810	2.94	0.59	7.280	2.31	0.43
8	14.160	4.63	0.42	11.530	3.68	0.34
24	30.730	8.31	0.23	30.450	7.31	0.23
48	22.150	10.97	0.11	24.010	10.17	0.12
72	12.950	12.52	0.06	14.339	11.88	0.07
96	8.210	13.50	0.04	9.705	13.03	0.05

13.5.13 RDi ROM Tests JSLA Backfill – Submersion Tests and Bottle Rolls

As mentioned above a 2017 campaign with RDi Labs was conducted to simulate ROM leaching using samples from test pits in the JSLA backfill. The sample locations are shown in Figure 13-7,. These tests are complimentary to the full-scale ROM tests, and provide supporting indications that ROM leaching is a valid process for the JSLA backfill material. The tests consisted of bottle roll tests (72 hour) on three separate coarse fractions for each test pit sample, as well as a whole sample subjected to full immersion in cyanide solution for 28 days for each test pit sample.

The results for the 72-hour bottle rolls are shown below in Table 13.29.

Table 13-29 - JSLA Coarse Bottle Rolls Test Results

Test #	Sample	Crush (P_{80})	% Extraction (Au)	Calc. Head Grade (Au g/mt)	Residue Grade (Au g/mt)	NaCN Consumption (kg/mt)	Lime Consumption (kg/mt)
BR1	RAB17-033	6 inch	64.4	0.20	0.072	0.004	1.95
BR2	RAB17-033	2 inch	76.6	0.24	0.055	0.163	2.60
BR3	RAB17-033	3/8 inch	73.6	0.19	0.051	0.089	1.76
			Average	0.21			
BR4	RAB17-043	6 inch	76.2	0.31	0.075	0.090	2.49
BR5	RAB17-043	2 inch	74.0	0.21	0.055	0.211	2.12
BR6	RAB17-043	3/8 inch	79.5	0.22	0.045	0.118	2.13
			Average	0.25			
BR7	RAB17-072	6 inch	74.9	0.27	0.069	0.059	2.14
BR8	RAB17-072	2 inch	74.0	0.28	0.072	0.180	1.85
BR9	RAB17-072	3/8 inch	81.2	0.25	0.048	0.175	2.63
			Average	0.27			
BR10	RAB17-117	6 inch	62.5	0.26	0.099	0.063	1.73
BR11	RAB17-117	2 inch	76.4	0.19	0.045	0.111	1.77
BR12	RAB17-117	3/8 inch	82.2	0.23	0.041	0.240	1.74
			Average	0.23			
BR13	RAB17-175	6 inch	46.6	2.56	1.368	0.053	2.27
BR14	RAB17-175	2 inch	91.6	1.18	0.099	0.046	1.57
BR15	RAB17-175	3/8 inch	94.1	1.17	0.069	0.240	1.57
			Average	1.64			

Table 13-30 - JSLA Full Immersion Whole Samples Test Results

Full Immersion Tests (28 days)- JSLA

Sample	Wt. kg	Head Grade g/t Au	Au Recovery %
RAB-117-033	124.6	0.209	46.6
RAB-117-043	112.0	0.216	78.2
RAB-117-072	113.2	0.267	46.3
RAB-117-117	113.4	0.24	38.1
RAB-117-175	111.4	1.241	51.0

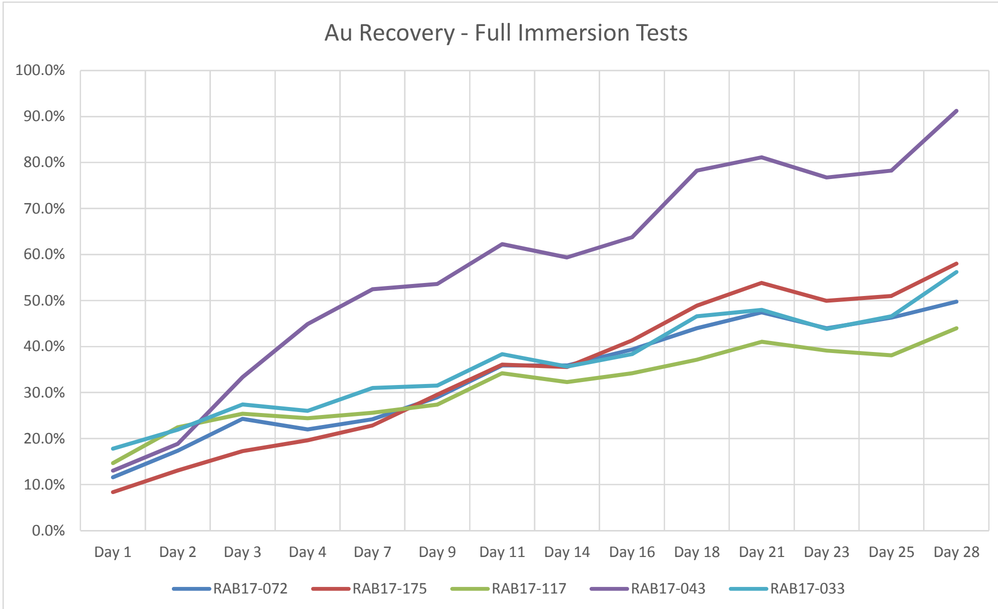

Figure 13-9 - Gold Recovery – Full Immersion Tests

As can be seen, leaching of these samples was incomplete with 28 days of leaching. It is not clear why the tests were terminated prematurely, but as such, this data set is not considered further.

13.5.14 McClelland 2018 – Cyanidation and Gravity / Cyanidation 4217

A small campaign was conducted from three selected RC drill composites, 10 mesh (2 mm), of higher grade samples to further evaluate gravity/ cyanidation vs. cyanidation only. These samples were prepared at a single grind size (80% -200 mesh (74 µm)), using 72-hour bottle roll tests for the cyanidation. The gravity concentrates were examined and all three samples contained some particulate gold, some as coarse as 100 µm. This campaign demonstrated that for these samples, recoveries were nearly identical with or without the gravity concentration step ahead of cyanidation. The results are shown below in Table 13.31.

Table 13-31 - Cyanidation Only vs. Gravity / Cyanidation – Select High-grade RC Drill Composites

		Au Recovery %					
Sample	CMM-195 Interval	As Rec'd	CN Only	Gravity/CN	Avg Calc'd Head g/t Au	CN kg/mt	Lime kg/mt
4217-001	1090-1175	61.2	96.5	97.0	3.27	0.1	2.3
4217-002	1225-1265	55.9	97.6	97.5	3.05	0.1	2.2
4217-003	1350-1370	57.8	97.0	96.8	1.59	0.1	1.8

13.5.15 KCA 2018 – CIL and Gravity / CIL Tests

Carbon-in-Leach (CIL) Tests and Gravity CIL Tests were conducted on a composite made from five high-grade core intervals (1/4 core) as shown in Table 13.32.

Table 13-32 - Sample Composite for CIL Tests

Drill Hole	From	To	kg	Au g/t
CMM-018C	402	412	4.02	2.485
CMM-059C	441.5	446	0.63	3.08
CMM-070C	1134	1139	2.85	2.79
CMM-156C	557.5	562	2.02	3.34
CMM-156C	577	582	1.77	2.78

Four different times, 24, 36, 48, and 96 hours were tested to evaluate the required retention time for CIL circuit design.

Table 13-33 - CIL Test Results

KCA Sample No.	KCA Test No.	Description	p80 Milled Size, mm	Head Average, gms Au/MT	Calculated Head, gms Au/MT	Extracted, gms Au/MT	GAC Extracted, gms Au/MT	Avg. Tails, gms Au/MT	Au Extracted, %	Leach Time, hours	Consumption NaCN, kg/MT	Addition $Ca(OH)_2$, kg/MT
80852 B	80853 A	High Grade Sample - Gravity Tailings	0.15	1.271	1.493	1.370	---	0.123	92%	96	0.05	1.00
80852 B	80854 A	High Grade Sample - Gravity Tailings	0.15	1.271	1.491	---	1.373	0.117	92%	24	0.47	0.75
80852 B	80854 B	High Grade Sample - Gravity Tailings	0.15	1.271	1.609	---	1.497	0.111	93%	36	0.59	1.00
80852 B	80854 C	High Grade Sample - Gravity Tailings	0.15	1.271	1.573	---	1.463	0.108	93%	48	0.75	1.00

The shortest time observed for the highest recovery was 36 hours, at 93% gold recovery.

A direct simulation of gravity extraction followed by CIL resulted in a 95% overall gold extraction, with 31% gravity recoverable gold as shown in Table 13.34.

Table 13-34 - Gravity / CIL Gold Extraction

			Overall Stage 1 + 2		Knelson Stage 1		Tail Leach Stage 2	
KCA Test No.	Description	Target p80 Grind Size, mm	Gold Extraction, %	Gold Tailings, %	Knelson Con., Stage %Au	Knelson Tail, Stage %Au	Leach Ext., Stage %Au	Leach Tail, Stage %Au
77789	High Grade Sample	0.15	95	5	31	69	64	5

13.5.16 KCA 2018 – Tailings Disposal Studies

In support of the ROM and Gravity/CIL project concept, tests were conducted to determine if blending the mill tailings with the ROM material would provide acceptable heap permeability at a range of different blend ratios under simulated pad loadings (stacking heights). From these test results it was concluded that this tailings disposal concept posed a real risk to satisfactory permeability within the ROM heap and that the project would be better served by reserving a section of the lined impoundment to place and store the dry tailings separately. It should be noted that none of these tests had cement added to stabilize the mill tailings. The poor permeability seen in the tests could have been caused by fine fraction migration during the test and this could pose a real problem in the heap if this method of mixing tailings with ROM heap leach ore were used in the operation.

The results of these permeability tests are shown in Table 13.35.

Table 13-35 - Summary of ROM / Tailings Compacted Permeability Tests

KCA Sample No.	KCA Test No.	Sample Description	Test Phase	Cement Added, kg/MT	Effective Height, meter	Effective Height, Feet	Cell Construction, Equiv. feet	Bulk Density, MT/m^3	Flow Rate, LpHr/m^2	Flow Result Pass/Fail	Saturated Permeability, cm/sec	Incremental Slump, %
80846 A	80857 A	Minus 3" Bulk Material	Primary	0	0	0.0	0	1.78	1,626	Pass	4.5E-02	5%
			Stage Load		18.29	50.0		1.91	1,151	Pass	3.2E-02	6%
			Stage Load		36.58	100.0		1.97	1,036	Pass	2.9E-02	3%
			Stage Load		54.86	150.0		2.01	923	Pass	2.6E-02	2%
			Stage Load		73.15	200.0		2.05	777	Pass	2.2E-02	1%
			Stage Load		91.44	250.0		2.07	681	Pass	1.9E-02	1%
			Stage Load		109.73	300.0		2.09	571	Pass	1.6E-02	1%
80846 A	80857 B	Minus 3" Bulk Material	Primary	0	0	0.0	25	1.78	1,200	Pass	3.3E-02	1%
			Stage Load		18.29	50.0		1.96	824	Pass	2.3E-02	9%
			Stage Load		36.58	100.0		2.01	656	Pass	1.8E-02	2%
			Stage Load		54.86	150.0		2.04	543	Pass	1.5E-02	1%
			Stage Load		73.15	200.0		2.07	489	Pass	1.4E-02	1%
			Stage Load		91.44	250.0		2.09	401	Pass	1.1E-02	1%
			Stage Load		109.73	300.0		2.11	355	Pass	9.9E-03	1%
80858 A	80862 A	5:1 ROM:Milled	Primary	0	0	0.0	25	1.75	1,576	Pass	4.4E-02	1%
			Stage Load		18.29	50.0		1.96	170	Pass	4.7E-03	10%
			Stage Load		36.58	100.0		2.02	79	Fail	2.2E-03	3%
			Stage Load		54.86	150.0		2.06	47	Fail	1.3E-03	2%
			Stage Load		73.15	200.0		2.09	26	Fail	7.2E-04	1%
			Stage Load		91.44	250.0		2.11	16	Fail	4.4E-04	1%
			Stage Load		109.73	300.0		2.13	13	Fail	3.6E-04	1%
80859 A	80862 B	10:1 ROM:Milled	Primary	0	0	0.0	25	1.78	493	Pass	1.4E-02	1%
			Stage Load		18.29	50.0		1.97	182	Pass	5.1E-03	10%
			Stage Load		36.58	100.0		2.03	75	Fail	2.1E-03	3%
			Stage Load		54.86	150.0		2.08	39	Fail	1.1E-03	2%
			Stage Load		73.15	200.0		2.10	20	Fail	5.6E-04	1%
			Stage Load		91.44	250.0		2.12	15	Fail	4.2E-04	0%
			Stage Load		109.73	300.0		2.14	12	Fail	3.3E-04	1%
80860 A	80863 A	15:1 ROM:Milled	Primary	0	0	0.0	25	1.80	314	Pass	8.7E-03	2%
			Stage Load		18.29	50.0		1.99	63	Fail	1.8E-03	9%
			Stage Load		36.58	100.0		2.06	25	Fail	6.9E-04	3%
			Stage Load		54.86	150.0		2.11	12	Fail	3.3E-04	2%
			Stage Load		73.15	200.0		2.13	13	Fail	3.6E-04	1%
			Stage Load		91.44	250.0		2.16	7	Fail	1.9E-04	1%
			Stage Load		109.73	300.0		2.18	6	Fail	1.7E-04	1%
80861 A	80863 B	20:1 ROM:Milled	Primary	0	0	0.0	25	1.81	1,095	Pass	3.0E-02	1%
			Stage Load		18.29	50.0		1.91	475	Pass	1.3E-02	5%
			Stage Load		36.58	100.0		2.04	325	Pass	9.0E-03	6%
			Stage Load		54.86	150.0		2.08	221	Pass	6.1E-03	2%
			Stage Load		73.15	200.0		2.10	150	Pass	4.2E-03	1%
			Stage Load		91.44	250.0		2.13	127	Pass	3.5E-03	1%
			Stage Load		109.73	300.0		2.15	79	Fail	2.2E-03	1%

13.6 Interpretations

13.6.1 Pulp Agglomeration

The pulp agglomeration process with mill slurry blended with 3/8 inch (9.5 mm) crush material in ratios of 1:10 to 1:6, after gravity concentration, is well supported as a viable recovery method at Castle Mountain. This is evidenced by large amounts of test results before, during, and after historical production. The production records themselves also attest to this fact, with gravity recoverable gold ranging from 13% to 22%, averaging 18%, and overall pulp agglomeration gold recovery of 76.9%. Mill recovery from production was attributed as 91.3 %, although this is a derived result as the stand-alone mill recovery is not known due to the nature of the combined processes and lack of data regarding intermediate sampling points within the overall pulp agglomeration process.

13.6.2 ROM Heap Leach Recovery

All results of the ROM bulk samples (100% passing 18 inches (457 mm), and approximately 80% passing 12 inches (304 mm) column test results are shown in Figure 13.10 below.

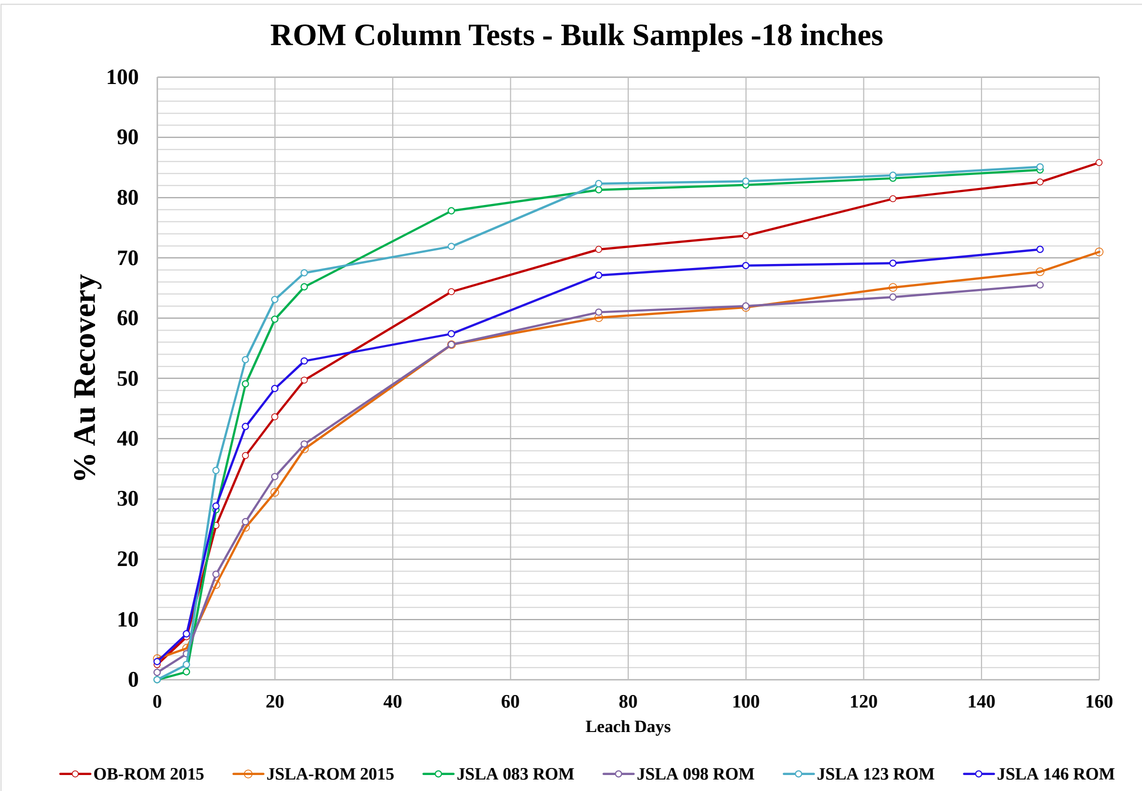

Figure 13-10 - Recovery Curves for ROM Bulk Samples

An overall average lab recovery of the six tests is 77.2%. It should be noted that production ROM will likely be coarser than 100% passing 18 inches (457 mm), and is estimated to be 80% passing 26 inch (660 mm), with a true top size of up to 40 inches (1016 mm). A plot of the relevant recovery size vs. recovery data from the recent metallurgical programs only (historical test work excluded, as the represented ore from such test work is mined and gone) is shown in Figure 13.11.

Note that there are some data omitted, namely the 2015 Phase 1 tests of the deepest high-grade (0.5 opt Au) conglomeritic material from drill hole CMM-014, which showed low column test recoveries (probably due to coarse gold), one of the 12 inch (300 mm) full immersion test results that was anomalously low (33.3%) compared to its parallel sample splits of a ROM column (65.5%), and one 6 inch (152 mm) immersion test (60.5%).

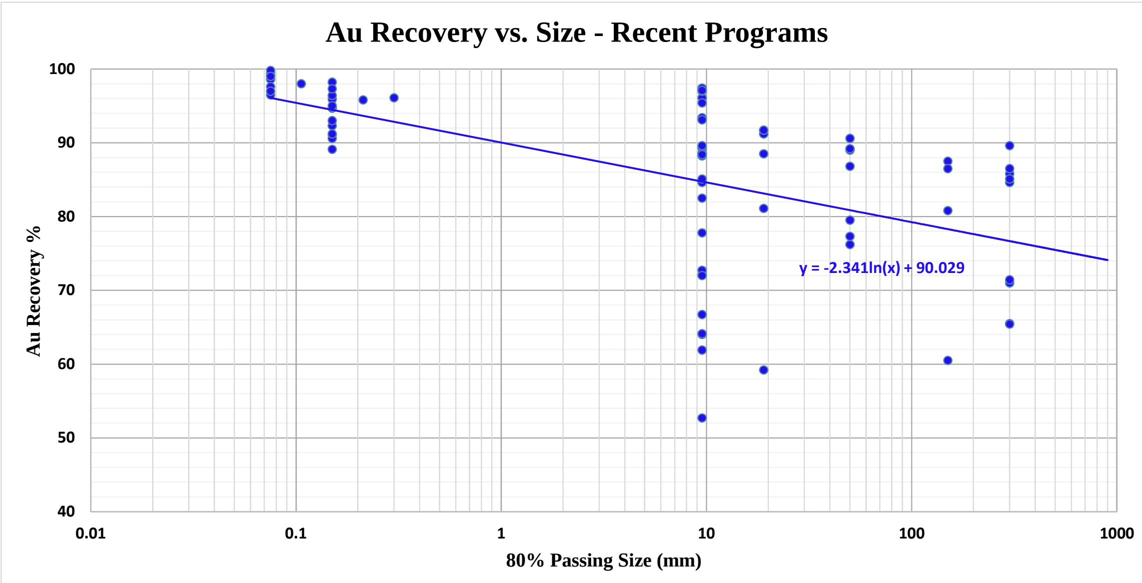

Figure 13-11 - Gold Recovery vs. Size – Recent Metallurgical Programs Only

Projecting a production ROM size distribution of 80% passing 26 inch (660 mm), the lab recovery was adjusted to 74.9%. Further, a lab to field discount of 2.5% was applied, resulting an estimated production gold recovery of 72.4% for ROM, which is used as the recovery in this study.

Reagent requirements are estimated as follows:

- Cyanide consumption is estimated at 0.2 lb/ton
- Lime consumption is estimated at 2.43 lb/ton

The leach cycle for design purposes was selected at 160 days using ROM lift heights of 50 feet (15 meters).

13.6.3 Ore Permeability

From available records, initial pre-production test work established the use of cement at approximately 8-10 lb/ton (4-5 kg/mt) for adequate pulp agglomeration in production. Later in production, after 1997, a mix of cement, fly ash and lime appears to have been used, with the combined average annual usage ranging from 4-7 lb/ton (2-3.5 kg/mt).

The 1998 work showed very good permeability for all 3/8 inch (9.5 mm) samples including those with no cement addition, suggesting that in periods where no mill grade material was processed, and no slurry added, leach-grade ore could be stacked with little or no cement addition. This appears to be confirmed by production records during 1998 and 1999 when annual mill feed tonnages were about 20-25% of the mill capacity and the corresponding cement/fly ash usages were considerably reduced, suggesting that these binders were not added during mill downtime periods.

Three additional test work campaigns examined ore permeability on crushed ore with and without the use of cement in the later years of production and recently in 2015.

The first campaign was conducted in 1998 during production on samples of 3/8 inch (9.5 mm) crushed ore (Glasgow, 1998) with adequate percolation characteristics.

The second campaign was conducted in 2015 on several composites from the Jumbo and South Extension deposits, on 3/8 inch (9.5 mm) and 3/4 inch (19 mm) crush sizes with no cement addition (MLI 2015), although 3/8 inch (9.5 mm) samples were agglomerated by lime and water additions. This work indicated good permeability in 5 of the 7 tested composites at simulated heap heights up to 250 feet (76 meters). For the two composites showing poor permeability, one was at a 3/4 inch (19 mm) crush size (sample 3878-107) and the other at a 3/8 inch (9.5 mm) crush size (sample 3878-116). Both were below KCA's typical recommended flow rate at 75 feet (23 meters), and both composites were from the Jumbo deposit. Neither composite had any reported clay content, and it is not immediately clear why these samples were indicating poor permeability.

The third campaign was part of the MLI 2018 program of agglomeration tests demonstrating permeability at various ratios of slurry blends and cement dosages for pulp agglomeration. Among these, a base-line sample of the master low-grade composite was tested at a 3/8 inch (9.5 mm) crush with no cement addition. This test showed very good percolation characteristics.

As part of the KCA 2018 tailings disposal studies, permeability tests were run on JSLA bulk samples at -3 inch (75 mm) with no cement and these also demonstrated adequate percolation characteristics to simulated stacking heights of 300 feet (91 meters).

During all of the six bulk ROM column tests at -18 inches (457 mm), no percolation problems were observed.

In conclusion, from all the data at hand, there is no reason to believe there will be any significant percolation problems with ROM leaching at Castle Mountain.

13.6.4 Mill Recoveries

13.6.4.1. Grind Size vs. Recovery (Mill-Grade Ore)

Bottle Roll Tests on high-grade samples ground in the range of 100-200 mesh (149-74 µm) were run throughout the history of metallurgical test work. All test results in all periods indicate very good amenability to agitated cyanidation. One study was conducted specifically comparing grind size, in 1987 by MLI, on samples from the Jumbo South deposit, and showed recoveries of 93%, 93%, and 97% for 100 mesh (149um), 140 mesh (105um), and 200 mesh (74um) grinds respectively.

In other test campaigns, agitated BRTs were conducted at a single grind size either at 100 mesh (149 µm) or 200 mesh (74 µm) to test general amenability to agitated leaching or in combination with gravity concentration test work. KCA averaged historical data on BRTs on grind sizes of 100 mesh (149 µm) and 200 mesh (74 µm) and the results are presented graphically in Figure 13.12.

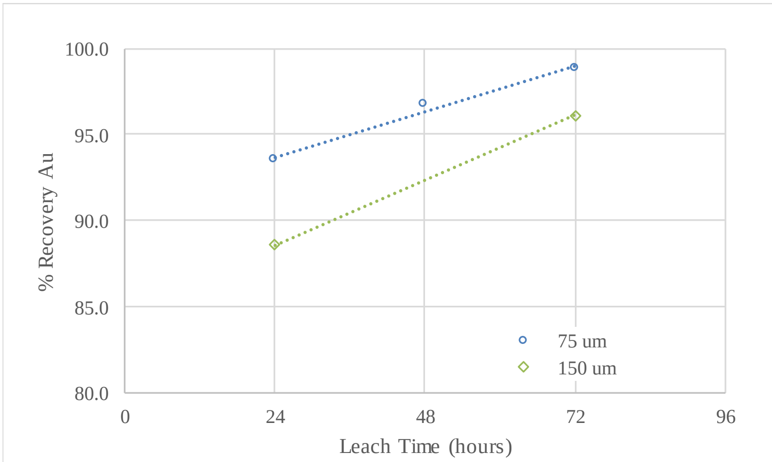

Figure 13-12 - Bottle Roll Test Gold Recovery vs. Leach Time at Two Grind Sizes

The 2017/2018 program tested a high-grade ore composite at four different grind sizes (80% passing 140 mesh (106 µm), 100 mesh (149 µm), 70 mesh (212 µm), and 50 mesh (300 µm)) with gravity concentration and cyanidation of the gravity tailings. The combined recoveries ranged from 95.7% and 98 % as shown in Table 13.12 above, with the highest recovery at the finest 140 mesh (106 um) grind.

13.6.4.2. Gravity Concentration (Mill-Grade Ore)

Generally high-grade samples from all historical and recent test work were amenable to gravity concentration. Recent gravity test work showed good amenability to gravity concentration in all high-grade samples, with recoveries ranging from 16% to 87%, averaging 33.7%. The master high-grade composite from the 2017 / 2018 work averaged 18.9% gravity recoverable gold. This is in close agreement with production records from 1995-2001 that indicate consistent gravity concentration recoveries averaged 18% over the life of mine as shown previously shown in Table 13.6.

13.6.4.3. Leach Time and Projected Mill Recovery

Agitated leach recoveries from lab test work in pre-production and in recent test work was consistently 90% or greater at 24 hours leach time and 95% or greater at 72 hours. Grind sizes from this test work ranged from 75 µm to 300 µm, with most test work conducted at 150 µm.

The shortest agitated leach time observed for the highest recovery was 36 hours, at 93% gold recovery.

A direct simulation of gravity extraction followed by CIL results in a 95% overall gold extraction, with 31% gravity recoverable as shown in Table 13.6.

It is KCA's opinion that 94% is a reasonable total recovery to expect from high-grade ores processed through the mill (grind, gravity, and CIL), using a 36 hour CIL retention time for design purposes.

13.6.5 Head Grade vs. Recovery

In examining the test work, KCA has found no clear or significant grade vs. recovery relationships for any of the low-grade composites (<0.043 oz/ton or 1.5 g/t Au) tested either in pre-production or in recent test work.

High-grade samples (>0.043 oz/ton or 1.5 g/t Au) have generally shown poorer recoveries in column tests as compared to low-grade samples. In particular, for very high-grade samples above 0.145 oz/ton (5 g/t), recoveries averaged less than 60% with no recovery greater than 65%. KCA generally attributes this to slow-leaching coarse gold, and accordingly recommends that a separate grinding circuit be included to treat high-grade ores, as was practiced historically.

With respect to milling of higher grade ores, gold recovery is universally high (89-98%) independent of grade in all tests conducted in the recent metallurgical test programs.

13.6.6 Deposit Depth vs. Recovery

From recent 2015 MLI Phase 1 column test work, the gold recovery vs. down hole depth is shown graphically in Figure 13.13, for each of the four holes tested from Jumbo and South Extension areas. Figure 13.13 suggests there may be an increasing recovery trend with increasing depth for CMM-017 (South Extension) area but the other holes do not show any discernable relationship. Hole CMM-014 might suggest a decreasing recovery with hole depth but this is compounded with the fact that the deep samples are very high-grade (~0.5 oz/ton or 17.1 g/t) and more likely have poor recoveries due to the presence of slow-leaching coarse gold.

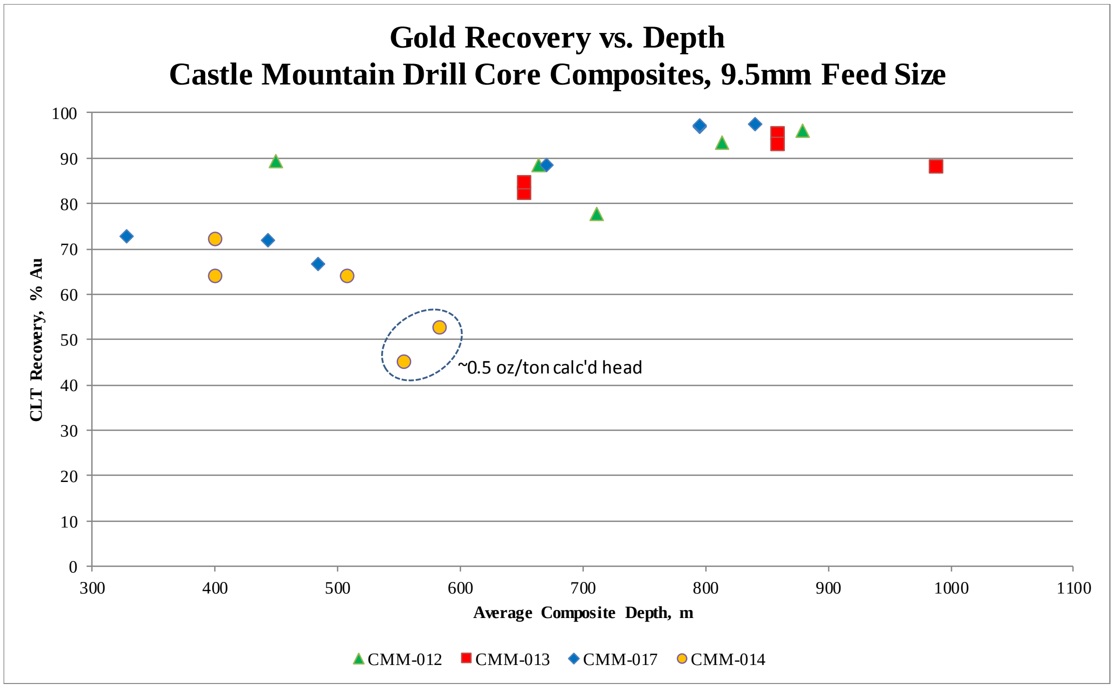

Figure 13-13 - Gold Recovery vs. Depth, Castle Mountain Drill Core Composites, 9.5mm Crush Size

13.7 Recommendations for Additional Tests

Recommendations for future test work include:

- Conduct variability bottle roll tests with carbon from several samples throughout the JSLA, South Domes, and Oro Belle pit areas, to confirm CIL operating parameters, reagent consumptions and recoveries. The samples for these tests should have a grade range that is consistent with the grade range for the mill that is expected in production.
- Conduct Caro's Acid and INCO/SO2 detoxification tests on a composite slurry sample to confirm detoxification operating parameters, final residual cyanide values, and confirm reagent requirements.
- Conduct at least four column tests, for two crush sizes in duplicate, on composites prepared from the South Domes area. No previous column tests have been run on samples from this area. This work will help to confirm whether leaching operating parameters and recovery data are substantially similar to those obtained from the JSLA backfill and hardrock areas, so that a ROM recovery for ore from the South Domes area can be confirmed. The composites should be made to target the average ROM gold grade expected in the South Domes area.

13.8 Conclusions

In the opinion of the QP issuing this report, the metallurgical test work programs were sufficiently detailed to establish the optimal processing methods for the known ores at Castle Mountain and were performed on mineralization that was typical and representative of the deposit. The results support the estimation of recovery factors for the selected process streams at a prefeasibility level or higher. The metallurgical test work programs are adequate to understand ore variability and plant optimization potential, although in the future additional CIL variability testing is recommended. Heap leach gold recovery for ROM is estimated at 72.4%, and 94% for the milling, gravity, and CIL circuit as proposed.

14.0 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The Mineral Resource estimate for the Castle Mountain Project was developed by Don Tschabrun (SME RM), an Associate Principal Mining Engineer of MTS, utilizing MicroMODEL version 9, a commercial mine planning software. The Mineral Resource estimate is based on drill hole data through March 2018.

A three-dimensional (3D) block model was constructed using standard procedures consisting of:

- importing topographic data in order to construct a digital terrain model of the current surface topography;
- conversion and importing of drill hole data into the MicroMODEL software;
- develop geologic 3D block model to provide boundaries for basic statistics and grade modelling;
- develop grade shells to use in conjunction with geologic boundaries to provide domains for basic statistics and grade modelling;
- develop variogram modelling to establish mineral trends and ranges of sample influence during grade modelling;
- 3D block modeling of grades within the respective geologic and grade shell domains; and
- classification of mineral resources into confidence categories of measured, indicated and inferred.

14.2 Resource Estimation Methodology

14.2.1 Topography

The topographic data for the Castle Mountain Project was based on a digital terrain model flown in May 2017 by Compass Tools and provided to MTS by Equinox Gold in the form of 2 ft contour intervals in dxf format. The recent topographic survey by Compass Tools did not cover the entire aerial extent of the block model. The remaining topography was supplemented with post-mining topography data generated by Equinox Gold. MTS converted this surface data into a 2D digital terrain model representing the current surface topography as of May 2017.

14.2.2 Block Model Orientation and Dimensions

A 3D block model was developed to represent the deposit. The block model dimensions and model limits are shown in Table 14.1. The coordinate system used for the 3D modelling was based on

NAD83 Zone 11 using imperial units of feet. The block model maintains a north-south and east-west orientation with no rotation.

Table 14-1 - Block Model Orientation and Dimensions (units in feet)

Parameter	Minimum	Maximum	Unit Block Size	Number of Blocks
Northing	12,807,100	12,820,000	30	430
Easting	2,202,300	2,211,300	30	300
Elevation	2,600	5,400	20	140

14.2.3 Drill Hole Database

The drill hole database contains drill hole assay information dating back to 1968, as well as, all recent drilling by NewCastle from 2013 through March 2018, for a total of 2,044 drill holes. Of this total, 130 are outside the limits of the current resource block model. In addition, 246 rotary air blast (RAB) drill holes were used solely for drilling and evaluating the backfill material within the JSLA pit area and were not used during the hardrock grade modelling process. Grade modelling for the JSLA backfill material is presented in Section 14.2.14. Thus, 1,914 drill holes were available to inform the 3D grade model for the hardrock portion of the deposit. Table 14.2 summarizes the drill holes by type as contained within the database, with the exception of the RAB drill holes, which were available for grade modelling. Figure 14-1 shows the location of the drill holes within the block model limits for the Castle Mountain Project.

Table 14-2 - Drill Holes within Resource Model Limits

Type	Legacy	NewCastle	Total
Rotary	480	0	480
RC	1,203	149	1,352
Core	67	125	192
RC/Core Tail	0	20	20
RAB	0	246	246
Outside Model Limits	(130)	(246)	(376)
Total	**1,620**	**294**	**1,914**

The assay sample length was generally about 5 ft with allowances made for geologic contacts. The database was comprised mostly of gold fire assay values. Although some intervals contained gold cyanide and silver assay values, there were insufficient assays for either gold cyanide or silver to develop a 3D gold cyanide or silver grade model.

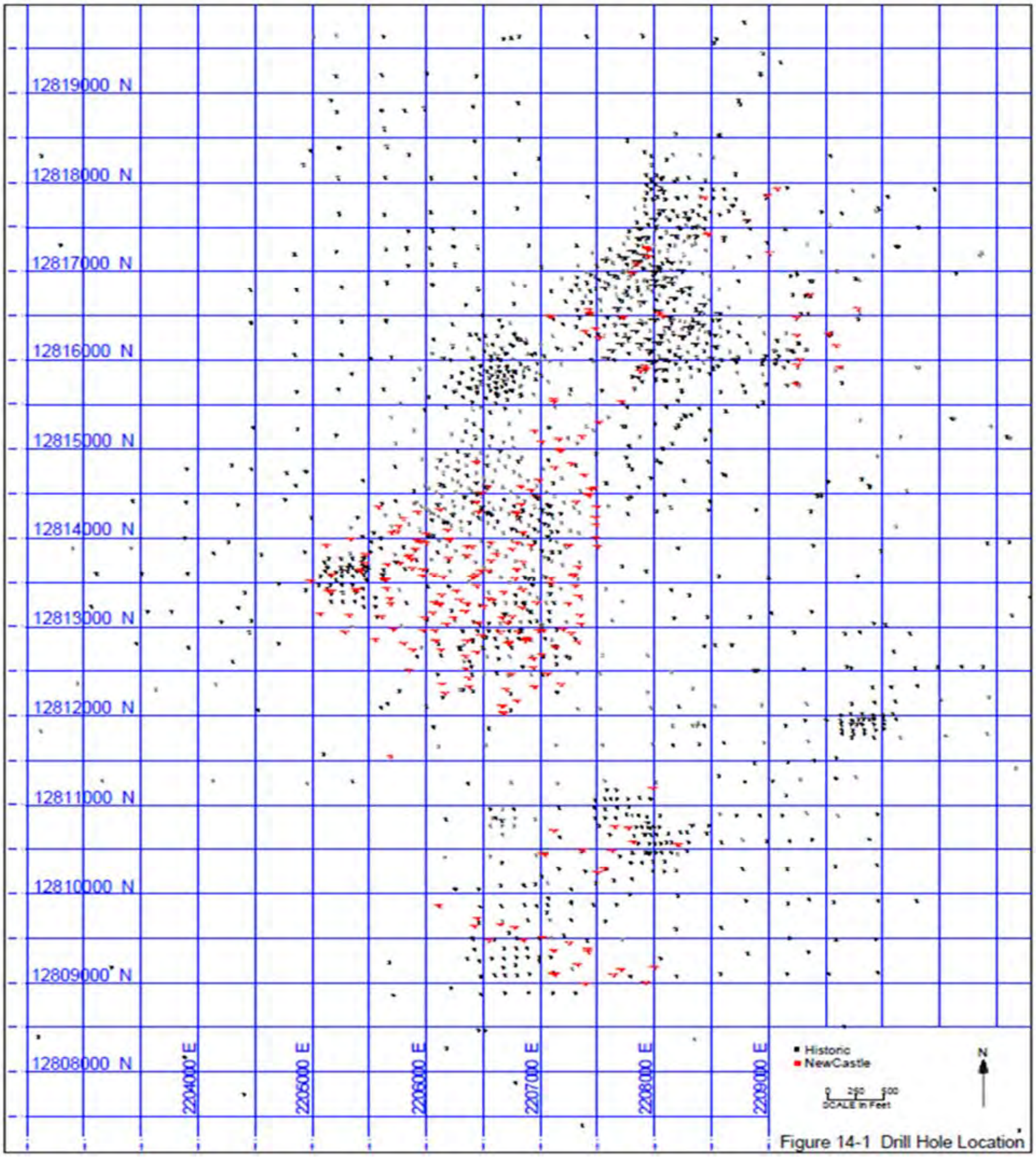

Figure 14-1 – Drill Hole Location Map (Black=pre NewCastle Red=NewCastle/ Equinox Gold drill holes)

14.2.4 Geologic Modelling

A 3D geologic block model was developed by Equinox Gold geologists using Leapfrog software version 4.0. The geologic model represents the state of geologic understanding of the Castle

Mountain project as of March 2018. The model represents RC and diamond drill logging, contact mapping on surface and 3D reviews of lithology and structural data. The geologic model represents a review and compilation of 997,300 feet of drilling from the 1,488,000 feet of total drilling on the Project.

The geologic model is bounded by the present-day topography which reflects historic backfill in the JSLA pit as well as historic mine dumps to the south, west and northwest of the JSLA pit. Geology from the mined-out Oro Belle, Jumbo and JSLA pits is not included in the current 3D geology model; however, a volume for the JSLA backfill is included based on the post-mining topography extending to the current topography. In addition, volumes have been developed for the three existing rock dumps and differentiated from the underlying geology.

The geologic model is comprised of 14 lithologic units, 10 of which are relevant to gold mineralization. The lithologic units are summarized in Table 14-3.

Table 14-3 - Lithologic Units with Relevant Controls on Gold Mineralization

Lithologic Unit	3D Rock Code	Geologic Type	Mineralization Host	Mineralization Style
Overburden	1	Depositional	Non-mineralized	Not Applicable
Basalt	2	Depositional	Non-mineralized	Not Applicable
Lahar	3	Depositional	Non-mineralized	Not Applicable
Dacite	4	Dikes	Non-mineralized	Not Applicable
Rhyolite-Phyric	5	Intrusion & flows	Strong	Disseminated
Diatreme	6	Volcanic vents	Moderate	Structural Control
Rhyolite-Aphyric	7	Intrusion & flows	Very strong	Disseminated
JSLA Backfill	8	Man-made	Variable	Variable
Vitrophyre	9	Intrusion & flows	Poor	Irregular
FelsicALT	10	Depositional	Moderate	Structural Control
FelsicLT	11	Depositional	Moderate	Structural Control
Intermediate Seds	12	Depositional	Poor	Structural Control
Andesite	13	Depositional	Moderate	Structural Control
Basement	14	Depositional	Moderate	Structural Control

Original paper logs from the historic Viceroy drilling were used to determine lithology for those drill holes. In total, 1,200 paper logs from historic Viceroy drill holes were reviewed and reinterpreted, totaling 710,000 feet of drilling out of 1,182,500 feet of historic drilling. Unfortunately, there was insufficient time to re-code the historic Viceroy geologic logging information into the electronic database prior to grade modelling.

Structures were modelled based on surface mapping as well as downhole structural data recording fractures, gouge and cataclasite. The Project contains a large number of individual structural zones which are often networked into irregularly branching and reconnected veins, but which also exhibit discontinuity. As such, this model focuses on a limited and manageable set of structures that appear to offset lithologies, focus gold mineralization or bound gold mineralization. These structures are modelled as planes that may locally represent a broader and more complex structural zone. It became clear from this work that the near vertical structures form a primary control to the gold mineralization.

14.2.5 Grade Shell Development

Grade shells were developed by NewCastle's geologists based on three grade intervals of gold fire assays: 0.006 opt (0.21 g/t), 0.015 opt (0.51 g/t) and 0.035 opt (1.2 g/t). These grade intervals represented the anticipated process groups: ROM, crush and mill components respectively to the project (the crush case is no longer being considered for the PFS). The three zones were prepared mostly on 20 ft plan map intervals corresponding to the block model level. In some areas, where drilling was less dense, the vertical spacing increased to 40 ft intervals. These level zones were transformed into a 3D triangulated model and then converted to a 3D block model. Table 14-4 lists the grade shell code that was assigned to each grade shell group. The grade shells were adjusted slightly to accommodate the 2017 drill hole information.

Table 14-4 - Grade Shell Groups Based on Gold Intervals

Gold Grade Shell Groups	Shell Code	Process Type
Low-grade: 0.006 – 0.015	100	ROM
Medium-grade: 0.015 – 0.035	200	Crush
High-grade: >0.035	300	Mill

The grade shell numeric codes were combined with the lithology codes to form a three-digit rock code. If the lithology codes were outside the lowest grade shell group, the 3D block model code maintained the rock code as defined in Table 14.3.

14.2.6 Compositing

The original uncut 5-ft assay intervals were composited to 20-foot intervals using standard bench compositing. This method computes a length-weighted average of the portions of assay intervals which fall within the 20 foot interval. Composite intervals with less than 10 feet of assayed length were not used for grade estimation. The maximum composite length allowed was 30 feet to allow for inclined holes. The corresponding composite rock code was extracted from the 3D rock model for that composite's centroid location.

Compositing to a 20-foot bench height partially simulates what grade can be expected when actual mining occurs. Because one cannot mine individual samples, but rather a combined group of samples over a constant vertical distance, the operator needs to know the expected dilution that will occur in the given mining interval. If compositing occurs on a smaller interval than the expected mining bench height, there is a high likelihood of over-estimating the gold grade block values. Compositing sample values to their representative bench height, initiates the dilution estimation process.

In addition, compositing to the planned mining bench height honors the volume-variance relationship between the estimating sample (the composite value) and the estimated value of the block's grade. The variance between the composite value and the block grade value will be minimized if the volume of the composite sample is representative of the block volume, in this case the height, which will provide higher confidence in the block value being less variable and hence, a more reasonable estimate.

14.2.7 Statistical Analysis

Basic statistics and cumulative frequency plots were run on composited data. Generally, MTS runs statistics on both sample and composite data for comparative purposes. However, the historic drill hole data in the electronic database was not updated with rock coding from the more recent geologic interpretation, and thus, it was not possible to generate sample statistics that would be comparable to the composite statistics.

Basic statistics were run on the individual lithology types (as listed in Table 14.3), initially split between the northern group of deposits (JSLA, Jumbo and Oro Belle) and the South Domes deposit. Cumulative frequency plots were generated for the various lithology types, and again, split out between the north group and South Domes.

In evaluating the cumulative frequency plots between the north and south groups, it appeared that there were no statistical differences between lithologies from the separate deposit groups. In addition, the lithology types displayed similar characteristics among certain groupings of lithologies. For example, the rhyolite and felsic groups displayed similar gold population characteristics representing the similarities in the geologic host material. The intermediate sediments, andesites and basement rock all displayed similar gold population characteristics to one another, representing the similarities with respect to the geologically vertical structural features. The diatreme material displayed a third distinct gold population separate from the other lithologies. Table 14.5 summarizes the main statistical results for the composite data.

Only the mineralized lithologies are presented in Table 14-5. Basalt, lahar and dacite are all post-mineral lithologies and are considered non-mineralized.

Table 14-5 - Summary for Gold Composite Statistics (No Cap)

Lithologic Unit	Rock Code	No. of Values	Mean Au opt	Max Au opt	Standard Deviation	Coefficient of Variation
Rhyolite-Phyric	5	5,584	0.0148	2.4923	0.0494	3.33
Diatreme	6	1,011	0.0156	0.2713	0.0252	1.62
Rhyolite-Aphyric	7	7,506	0.0176	3.5911	0.0718	4.08
JSLA Backfill	8	3,648	0.0306	2.1302	0.0861	2.81
Vitrophyre	9	18	0.0137	0.0750	0.0215	1.56
FelsicALT	10	3,718	0.0170	1.8534	0.0673	3.95
FelsicLT	11	564	0.0329	2.3802	0.1577	4.79
IntermSeds	12	915	0.0243	2.1106	0.0864	3.55
Andesite	13	890	0.0369	4.3034	0.2033	5.50
Basement	14	422	0.0195	0.3557	0.0378	1.93

It is worth noting that statistics and cumulative frequencies were run within the various grade shell intervals, by lithology type, to check for distinguishing distribution characteristics. While the grade shells displayed a full range of grade values within the distributions, there were no distinguishing features to indicate that lithology types within the individual grade shells should be treated separately with respect to statistical or geostatistical analysis. The grade shells were treated mainly as a physical boundary and sample identifier for gold grade estimation.

14.2.8 Gold Capping

Gold deposits typically display a highly skewed distribution and high-grade outliers, if not controlled, tend to over-estimate the gold metal content of the deposit. While there are a number of methods available to control high-grade gold values, one method applies a grade capping to the high-grade outlier values based on the cumulative frequency plots with respect to lithology type. Any values that deviate from the upper end of the cumulative frequency curve are candidates for grade capping.

Given the significant number of composite values (more than 73,000), there were only 22 composite values that required capping, indicating the robust nature of the gold distributions. Table 14-6 lists the lithology type, the cap value applied and the number of capped values. The capping values are presented in both imperial and metric units for easier comparison to previous technical reports.

Table 14-6 - Gold Grade Capping Limits

Lithologic Unit	Rock Code	Gold Cap Value (opt)	Gold Cap Value (g/tonne)	No. of Capped Values
Rhyolite-Phyric	5	0.900	31.00	1
Diatreme	6	No cap	No cap	0
Rhyolite-Aphyric	7	0.900	31.00	4
Vitrophyre	9	No cap	No cap	0
FelsicALT	10	0.600	21.00	6
FelsicLT	11	0.600	21.00	5
IntermSeds	12	0.400	14.00	4
Andesite	13	0.600	21.00	2
Basement	14	No cap	No cap	0

14.2.9 Variography

Variograms were developed in various directions on composited data to evaluate the preferential direction of mineralization and range of sample influence with respect to grade modelling. As is generally the case with most gold deposits, the variography for the Castle Mountain deposit showed inconsistent variograms.

Relative variography (similar to correlograms, which tend to smooth some of the inconsistencies in sample data differences by producing smoother variograms) was run along structural trends as well as north-south, east-west, vertical and omni-directional to the deposit. Although the variography was generally inconsistent, one feature that stood out was the fact that the variograms displayed a relatively high nugget effect indicating either a sampling error or a relatively higher variance among sample data. While part of the nugget effect could be associated with minor sampling error, MTS believes the high nugget effect is more likely associated with higher variance among the sample data.

The direction producing the most consistent variography was the vertical direction oriented N10^{o}E dipping 90 degrees with a range of 150 ft, which is consistent with the vertical nature of the structural trends. The next most consistent variogram trends appeared to be a N10^{o}E, no dip or rotation, with a range of about 90 ft; and perpendicular to the N10^{o}E direction, (an approximate east-west trend) with a range of 75 ft, likely the result of stockworks being generated from the many structural intersections which allowed the hydrothermal fluids to mix and commingle within the host rocks, with respect to the flat lying nature of the host rocks.

While the variography produced some useful indicators for search range parameters, MTS believes that in general, the variograms were not definitive enough to justify using kriging techniques for gold grade interpolation.

14.2.10 Grade Modelling and Resource Classification

Several grade modelling estimates were interpolated, varying the parameters to determine the most representative estimate. After reviewing the various estimates and comparing to geologic as well as composite data, a 3D grade model was developed using the inverse distance weighting (IDW) estimation method with a weighting power of three. The IDW method was selected as it appeared to best represent the composite values with respect to statistical comparison as well as visual comparisons of mineral distributions in cross section and plan maps.

Search ellipsoids were developed within the guidelines indicated by the variography and discussions with NewCastle's geologists regarding the geologic environment with respect to gold deposition. The search ellipsoids were also developed based on the grade shell orientations as well as the resource class being estimated. Table 14-7 provides the ellipsoid ranges used in conjunction with composite selection depending on the resource class being estimated.

Table 14-7 - Ellipsoid Parameters for Grade Modelling

Resource Class	Range: X/Y/Z (ft)	Minimum Composites for Grade Estimate	Maximum Composites per Drill Hole
Grade Shell: Low-grade (100 series) & Medium-grade (200 series)			
Measured	125/125/100	3	2
Indicated	250/250/150	3	2
Inferred	350/350/200	1	1
Grade Shell: High-grade (300 series)			
Measured	125/90/90	3	2
Indicated	250/180/180	3	2
Inferred	300/200/200	1	1
Outside Grade Shells	300/300/150	3	2

Note: The range indicates the distance for Primary, Secondary and Tertiary axes respectively.

Ellipsoid orientation varied depending on the particular grade shell being estimated. For the low and medium-grade shells (100 & 200 series shell codes), the primary ellipsoid axis was oriented N10^{o}E with no dip or rotation. Composite values outside the low-grade shell as well as composite values inside the low and medium-grade shells were allowed to estimate block grades within the low and medium-grade shells.

For the high-grade shell (300 series shell code), the primary ellipsoid axis was oriented vertically, rotated N10^{o}E with no rotation. Composite values inside the medium and high-grade shells were allowed to estimate block grades within the high-grade shell.

The lithology was combined into one of three groups: 1) the rhyolites and felsic material, 2) the diatreme material and 3) the andesites, intermediate sediments and basement material with associated composites from each lithology group being used to model the grade for the corresponding rock type within the 3D rock model.

For measured and indicated resource classes, a sector search was employed that allowed for a maximum of 12 composite values (two composites per sector in a six-sector search) and a minimum of three composite values (two from any one drill hole), combined with the search ranges previously mentioned, in order to estimate a block grade. For the inferred resource class within the grade shells, a sector search was employed that allowed for a maximum of 12 composite values (two composites per sector in a six-sector search) and a minimum of one composite value, also combined with the search ranges previously mentioned, in order to estimate a block grade.

Grade values were also estimated outside the low-grade shell however, all grade values were tagged as inferred resource class. A sector search was employed that allowed for a maximum of 12 composite values (two composites per sector in a six-sector search) and a minimum of three composite values (two from any one drill hole) in order to estimate a block grade.

Once the appropriate composite values were identified and sorted by lithology and grade shell, an isotropic weighting was used within the anisotropic ellipsoid to estimate block grades. No attempt was made to determine block partials with respect to lithology or grade value.

A resource classification of Measured, Indicated or Inferred was assigned to each block depending on the aforementioned search parameters, drill density and number of associated composites per block. Measured and Indicated blocks were confined to only those blocks within the grade shell boundaries. Inferred blocks could be assigned inside or outside the grade shell boundaries. The block was assigned the value of one, two or three corresponding to the resource class of Measured, Indicated or Inferred.

14.2.11 Density

Each block was assigned a density depending on the block's lithology coding. Table 14-8 lists the density assignment with respect to lithology. The density determinations are discussed in Section 12.5.

Table 14-8 - Castle Mountain Density Assignment

Rock Type	Density (g/m^3)	Tonnage Factor (ft^3/ton)
Alluvium (Overburden)	1.90	16.8
Basalt	2.47	13.0
Lahar	2.00	16.0
Dacite	2.24	14.3
Rhyolite Phyric	2.25	14.2
Diatreme	2.24	14.3
Rhyolite Aphyric	2.22	14.4
JSLA Backfill	1.72	18.6
Vitrophyre	2.22	14.4
Felsic ALT	2.14	15.0
Felsic LT	2.10	15.2
Intermediate Sediments	2.28	14.0
Andesite	2.22	14.4
Basement	2.47	13.0

14.3 Mineral Resource Statement

The Mineral Resource estimate must be based on reasonable prospects of economic extraction as required under NI43-101 and CIM guidelines. An optimized pit shell was generated using the Lerchs-Grossman (LG) algorithm and the parameters listed in Table 14.9.

Although it was discussed earlier in the report regarding the potential for three mineral processes, only ROM was considered for the LG evaluation as crushing is no longer being considered and milling represents a very small portion (5 – 10%) of the total process feed material. Although all backfill material was considered as waste rock with respect to the LG evaluation, the backfill material was separately modelled for gold grade with a methodology discussion provided in Section 14.3.2.

Based on the information provided in Table 14-9, the ROM breakeven gold cutoff grade calculates to be 0.005 opt (0.17 g/tonne).

Table 14-9 - LG Parameters

Parameter	**Unit**	**Value**
Gold Price	$/oz	1,400
Marketing Cost	$/oz	5.00
Royalty – Jumbo, JSLA, Oro Belle	NSR (%)	2.65
Royalty – South Domes	NSR (%)	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	degrees	48

Note: All dollars are US$ and tons are dry tons.

The Mineral Resource estimate presented in Table 14-10 shows a range of cutoff grades with the **base case** (in bold) listed at a gold cutoff grade of 0.005 opt (0.17 g/tonne) and contained within an LG shell based on a gold price of $1,400/oz. Table 14-11 presents the Mineral Resource estimate in the metric system in order to compare with earlier reported estimates. The metric equivalent gold cutoff grade is 0.17 gpt.

The Mineral Resource estimate is also presented by domain in Table 14-12. Although the division between the domains is somewhat arbitrary, as the domains are contained within the same LG shell, the information provides some guidance regarding the distribution of resource material within each domain.

MTS is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other issues that could materially affect the Mineral Resource estimate.

Table 14-10 - Mineral Resource Estimate (Imperial Units)

	Measured			Indicated		
Cutoff (Au opt)	**Mtons**	**Gold Grade (opt)**	**Gold Oz (million)**	**Mtons**	**Gold Grade (opt)**	**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**		
Cutoff (Au opt)	**Mtons**	**Gold Grade (opt)**	**Gold Oz (million)**	**Mtons**	**Gold Grade (opt)**	**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

1) The effective date of the Mineral Resource is March 29, 2018.
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 cutoff grade of 0.005 opt.
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 this Technical Report.

Table 14-11 - Mineral Resource Estimate (Metric Units)

	Measured			Indicated		
Cutoff (Au gpt)	**Mtonnes**	**Gold Grade (gpt)**	**Gold Oz (million)**	**Mtonnes**	**Gold Grade (gpt)**	**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**		
Cutoff (Au gpt)	**Mtonnes**	**Gold Grade (gpt)**	**Gold Oz (million)**	**Mtonnes**	**Gold Grade (gpt)**	**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

1) The effective date of the Mineral Resource is March 29, 2018.
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 cutoff grade of 0.17 gpt.
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 this Technical Report.

Table 14-12 - Mineral Resource Estimate by Domain (Imperial Units)

	Measured			Indicated		
Domain	**Mtons**	**Gold Grade (opt)**	**Gold Moz**	**Mtons**	**Gold Grade (opt)**	**Gold Moz**
JSLA	67.0	0.0149	1.00	6.7	0.0154	0.10
Jumbo	17.7	0.0202	0.36	7.9	0.0167	0.13
Oro Belle	55.2	0.0170	0.94	21.3	0.0132	0.28
South Domes	37.3	0.0188	0.70	35.9	0.0179	0.64
JSLA Backfill	0.0	0.0000	0.00	18.0	0.0101	0.18
Total	**177.1**	**0.0169**	**2.99**	**89.7**	**0.0149**	**1.34**
	Measured + Indicated			**Inferred**		
Domain	**Mtons**	**Gold Grade (opt)**	**Gold Moz**	**Mtons**	**Gold Grade (opt)**	**Gold Moz**
JSLA	73.7	0.0149	1.10	39.7	0.0116	0.46
Jumbo	25.5	0.0191	0.49	25.4	0.0129	0.33
Oro Belle	76.5	0.0159	1.22	58.9	0.0111	0.65
South Domes	73.2	0.0184	1.34	43.2	0.0136	0.59
JSLA Backfill	18.0	0.0101	0.18	21.7	0.0081	0.18
Total	**266.8**	**0.0162**	**4.33**	188.9	0.0116	2.20

1) The effective date of the Mineral Resource is March 29, 2018.
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 cutoff grade of 0.005 opt.
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 this Technical Report.

14.3.1 Grade Model Validation

The IDW grade model interpretation was validated by visual comparison of plan and cross section views of composite drill hole grades to block grade values. The block grades appear to correlate well with the drill hole composite grades. Cross section examples are provided in Figures 14-2 (north area) and 14-3 (South Domes).

A nearest neighbor (NN) grade model was also estimated using similar parameters as the IDW method. The comparison of tonnage and grade is shown in Table 14-13. The comparison only considers Measured and Indicated resource classes given that the search ranges used for the NN grade model were less than those ranges used for the IDW grade model. As seen from the table the gold ounces compare very well, although the NN tonnage is slightly less, with a slightly higher gold grade than the IDW values.

Table 14-13: Block Model Comparison: IDW to NN Estimate

Modelling Method	Mtons (M & I)	Gold Grade (opt)	Gold Moz
IDW(3)	248.8	0.0167	4.1
NN	203.8	0.0200	4.1
Difference	**45.0**	**0.0033**	**0.0**

Additional validation in the form of cumulative frequency plots, as provided in Figure 14.4, shows good correlation between composite, IDW and NN grade values. Again the IDW and NN grades are only for the Measured and Indicated resource classes.

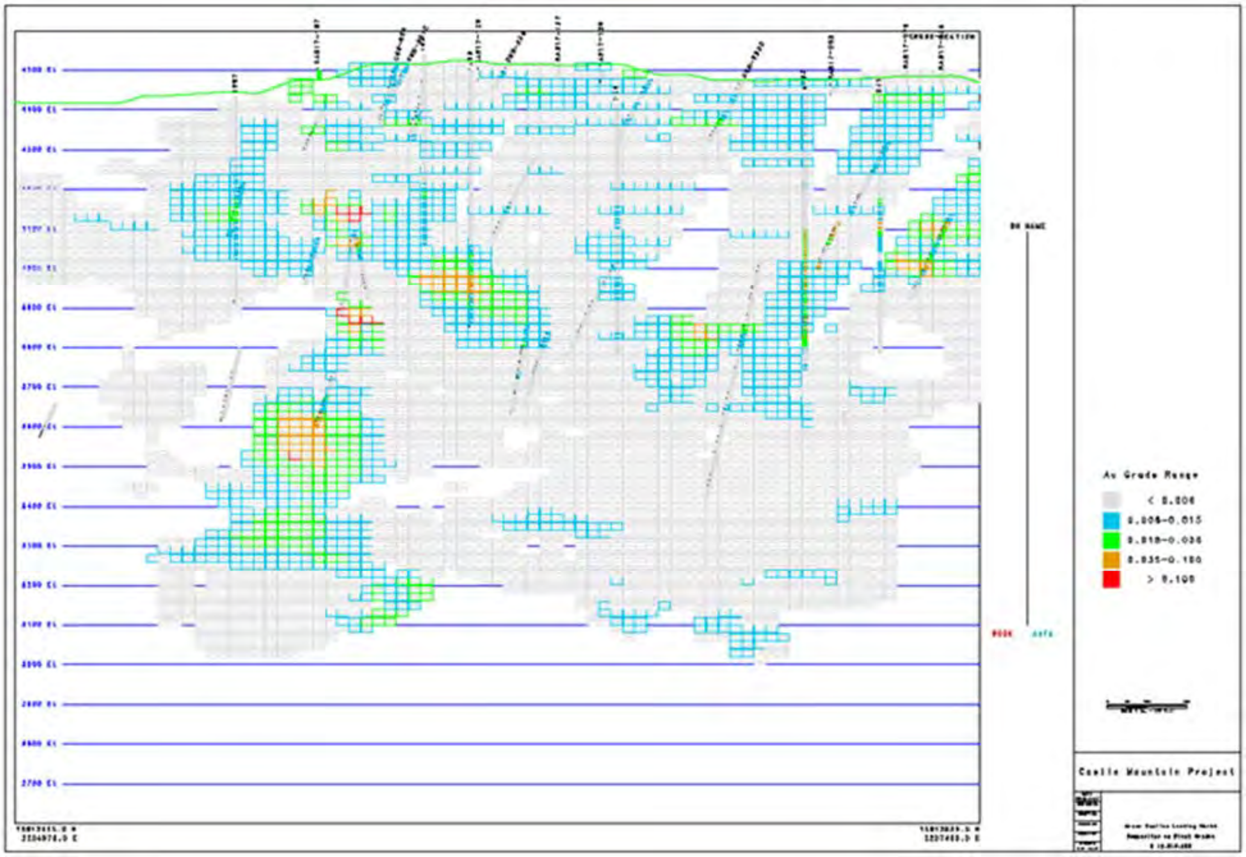

Figure 14-2 – East West Cross Section (12,813,955 N) Composites vs Block Grades – North Group

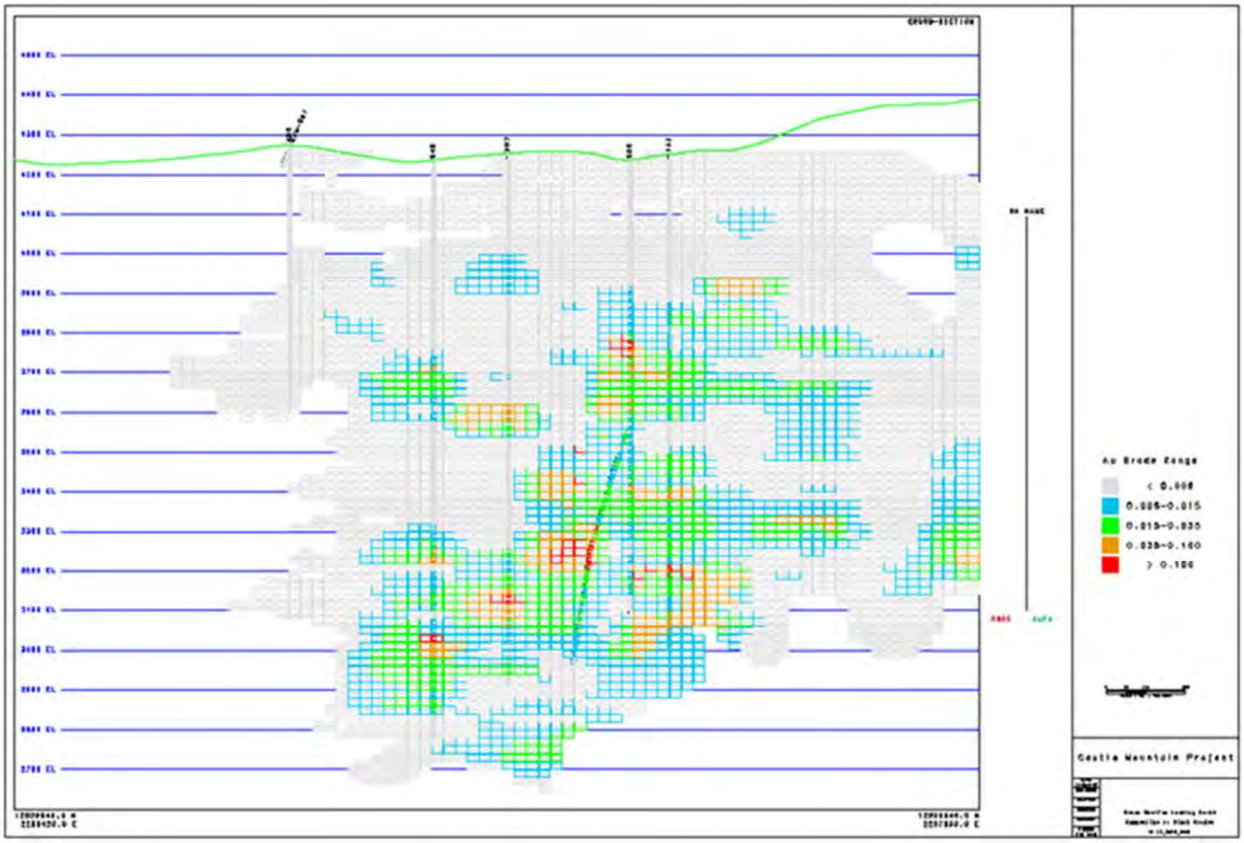

Figure 14-3 – East West Cross Section (12,809,845 N) Composites vs Block Grades – South Domes

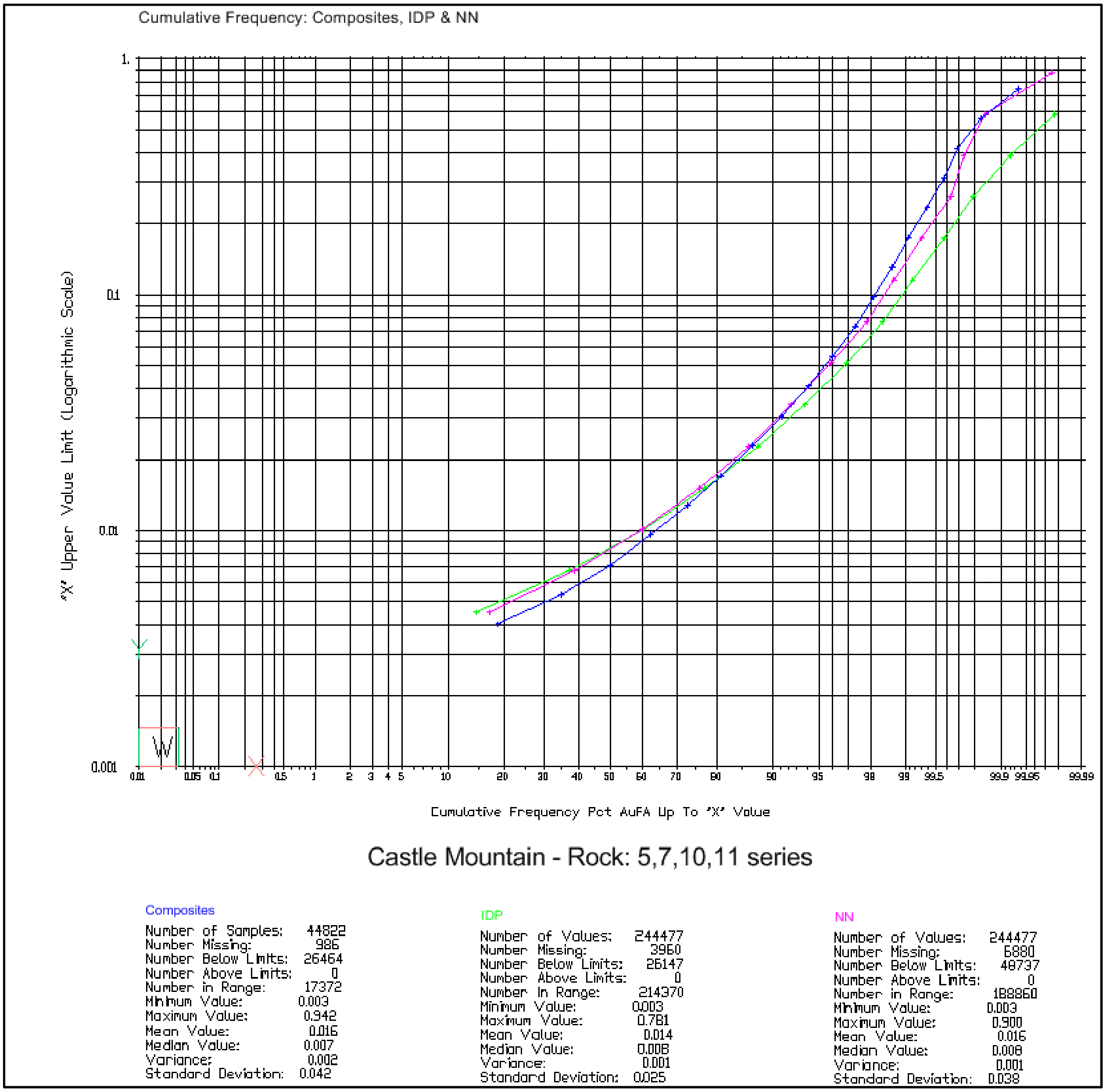

Figure 14-4 – Cumulative Frequency plot of Composite, IDW and NN Grades

14.3.2 JSLA Backfill Grade Modelling

There is material contained within the historic JSLA pit that was backfilled with below cutoff-grade material from the Jumbo and Oro Belle pits as mined by the previous operator from October 1996 to April 2001. The previous operator used a gold cutoff grade of 0.015 opt (0.51 g/t) which was primarily based on a much lower gold price. This backfill material at today's gold price now represents potential ROM heap leach process feed.

Kelsey Stark and Terre Lane of Global Resource Engineering, Ltd. evaluated and prepared a resource estimate for the mineralized ROM material within the JSLA backfill. Don Tschabrun of MTS reviewed and approved the estimation process for the backfill material and has incorporated the backfill resource estimate with the hardrock portion of the Mineral Resource as presented in Tables 14.10 through 14.12.

In summary, a 3D grade model for the JSLA backfill was developed using the inverse distance weighting (IDW) estimation method with a weighting power of three. The IDW method was selected as it appeared to best represent the backfill composite values with respect to statistical comparison as well as visual comparisons of mineral distributions in cross section and plan maps.

The JSLA backfill volume was estimated using the current topography and the JSLA as-built pit shape from historic data as shown in Figure 14-5. The gold grade estimate was contained within the established block model limits using only blocks designated as backfill (rock code 8).

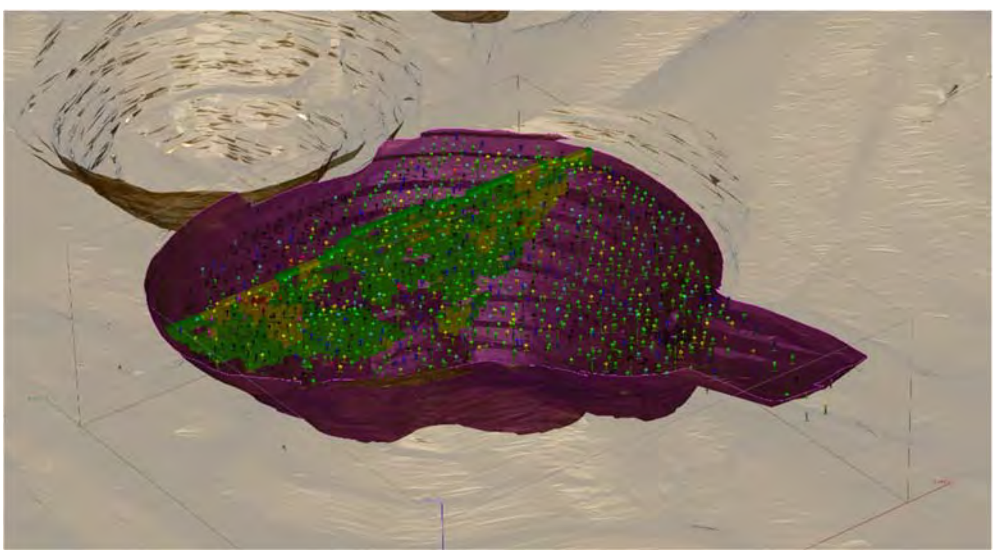

Figure 14-5 – JSLA Backfill Limits

In absence of geologic data controlling the spatial continuity of the grade, the deposition of the backfill material was analyzed for potential trends in mineralized continuity. The monthly as-built drawings from 1996 to 2001 showed the pit filling by dumping over the edge in a general north to south trend at 38 degrees down dip, essentially the angle of repose. The upper portion of the JSLA pit backfill (above the 4,400 ft elevation) was placed in a horizontal tabular lifts. Figure 14-6 shows the progression of the backfill of the JSLA backfill as of January 1998. The series of as-built drawings from Viceroy (the previous operator) are important, because they show a consistency in backfill deposit method as well as suggesting a directional continuity of gold grade.

The backfill mineral resource is based on the drill hole database provide by Equinox Gold for drilling and assaying through May 2018. From January through March of 2018 Equinox Gold embarked on an extensive drilling program aimed primarily at evaluating the backfill material within the historic JSLA pit. The backfill drilling program is summarized in Table 14-14.

Table 14-14 - Backfill Drilling Program Summary

Drilling Program	Number of Drill Holes	Number of Samples	Drill Hole Length (ft)
2018 RAB	988	988	19,760
2018 RC	53	895	9,000
2017 RAB	242	387	6,095
2017 RC	123	3,433	23,657

The 2017 RAB holes averaged about 30 ft in depth where the 2018 RAB holes were limited to a depth of 20 ft, one pass drilling. The 2017 and 2018 RC drilling programs had depths ranging from 150 feet to 230 feet. This had a significant impact on the approach to modeling the deposit. The upper part of the deposit (above 4,400 ft) has significantly more data to draw from and thus, was modeled separately from the lower portion of the backfill.

Based on the gold sample statistics (Table 14-15) and the gold sample cumulative frequency curve (Figure 14-6), gold samples were capped at 0.0875 opt (3.0 g/tonne) before compositing to 20 ft intervals which corresponds to the bench height. In total, twenty sample values were capped prior to compositing and grade modelling. Table 14-16 shows the gold composite statistics.

Table 14-15 - Gold Sample Basic Statistics

Number of Samples	Mean (opt)	Max (opt)	Standard Deviation	Coefficient of Variation
4,954	0.00987	2.0037	0.03181	3.2235

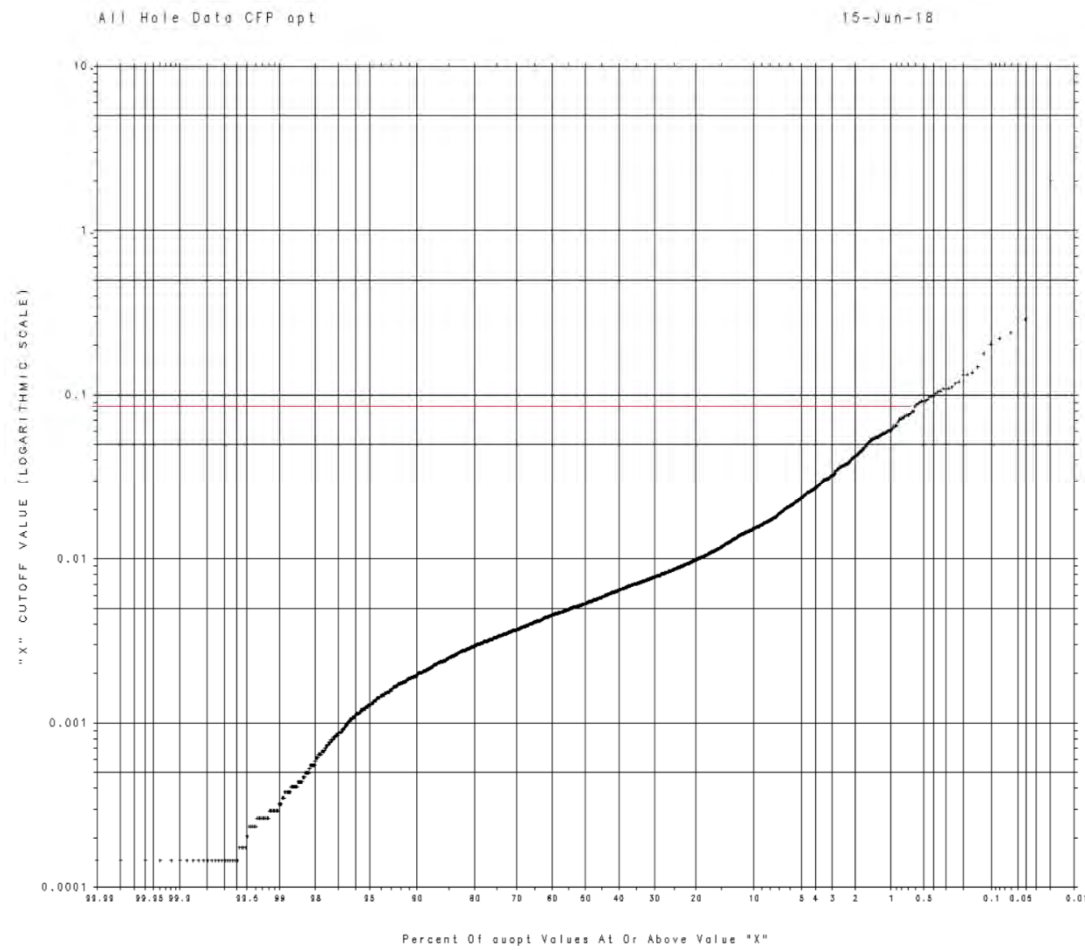

Figure 14-6 – Cumulative Frequency Plot of JSLA Gold Samples

Table 14-16 - Gold Composite Statistics

Number of Samples	Mean (opt)	Max (opt)	Standard Deviation	Coefficient of Variation
2,640	0.00915	0.0875	0.01114	1.2175

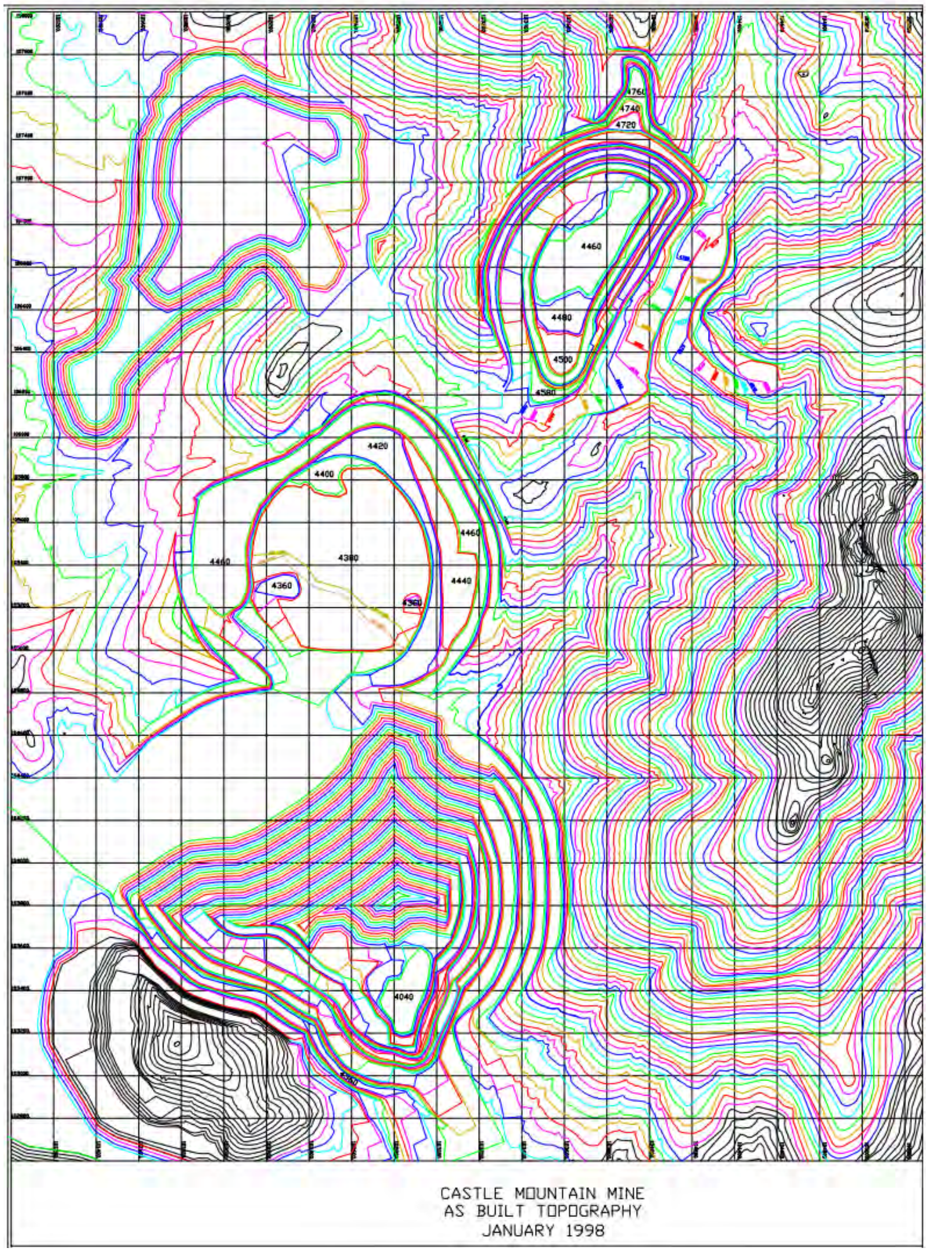

Figure 14-6 – Monthly As-Built Drawing of JSLA Backfill, January 1998

Based on the analysis of the monthly as-built drawings of the JSLA backfill two grade modeling domains were established. The upper 100 feet of the backfill sits at an elevation of 4,400 ft. This area was deposited in flat shapes on top of the already placed sloped backfill. Below the 4,400 ft elevation, the backfill was deposited continuously on the downhill side in a general direction of north to south. The angle from crest to toe was generally 38 degrees inside the pit boundary. The 38 degrees dip direction was analyzed for continuity using variography.

The variography analysis produced inconsistent variogram results. Although unable to provide grade modelling parameters, correlograms generated from variogram analysis provided continuity ranges for the IDW grade modeling as shown in Table 14-17. The maximum search distance was set to 600 feet for the inferred data to fill in areas of the block model with sparse drilling data as bounded by the pit limit.

Table 14-17 - JSLA Backfill Search Parameters

Domain	Primary Axis	Secondary Axis	Tertiary Axis	Primary Range	Secondary Range	Tertiary Range
Above 4400 ft	0/0	90/0	0/-90	95	95	60
Below 4400 ft	160/ -38	70/0	340/-52	100	75	30

Gold grade estimation was performed using the IDW estimation method with a weighting power of three using MicroMODEL version 9. This method was selected as it seemed best suited in representing the overall blend of composite grades while limiting the influence of distant between drilling data. This was important since the deposit was man-made, with no geologic controlling continuity. There is reasonable evidence of continuity as shown in Figure 14-6. The series of monthly as-built drawings show strong evidence of grade continuity within the backfill, as well as evidence of barren/low-grade zones. The search ellipses defined in Table 14-17 share direction with the known shape of the man-made deposition.

No Measured resource was classified for the backfill with respect to the nature of the material. Indicated resource in the domain above 4,400 ft was limited to blocks with a composite sample within 75 ft of the block center. Indicated resource in the domain below 4,400 ft was constrained to blocks with a composite sample within 50 ft of the block center. This was based on likely grade continuity observed within each zone and the availability of data within each zone.

The backfill Mineral Resource is provided in Tables 14-10 through 14-12.

The JSLA backfill grade model was validated visually with cross sections consisting of composite and grade values. The block grades show a reasonable representation of the drill hole data as seen in Figure 14-7. In addition, the grade model is shown to correlate well with the composite data as seen in the cumulative frequency plot in Figure 14.8.

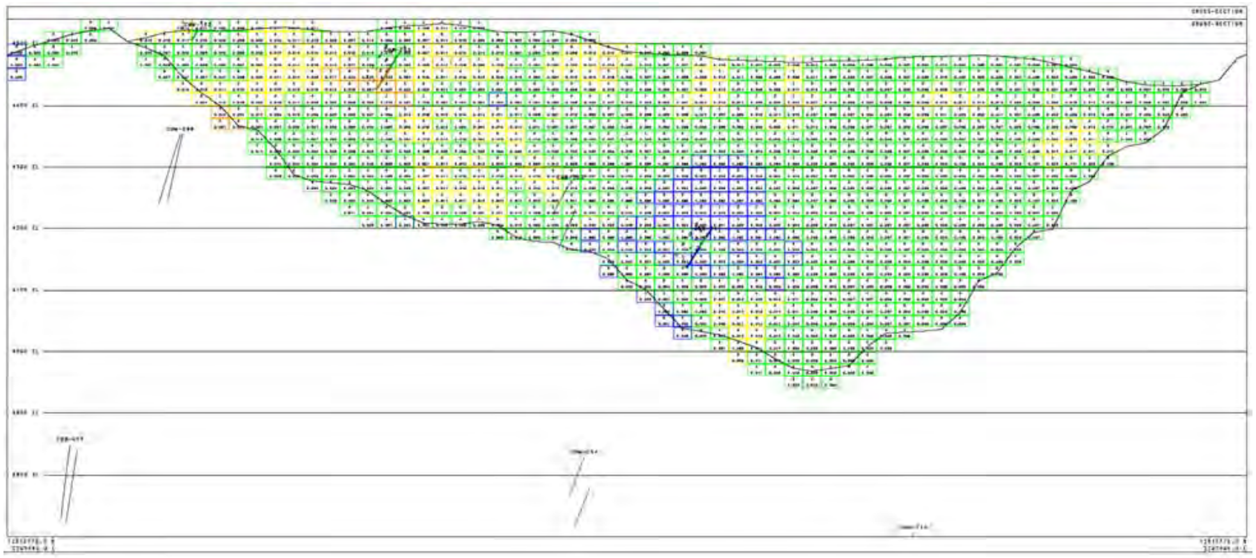

Figure 14-7 – East-West Cross Section (12,813,775 N) showing Composites and Block Grades

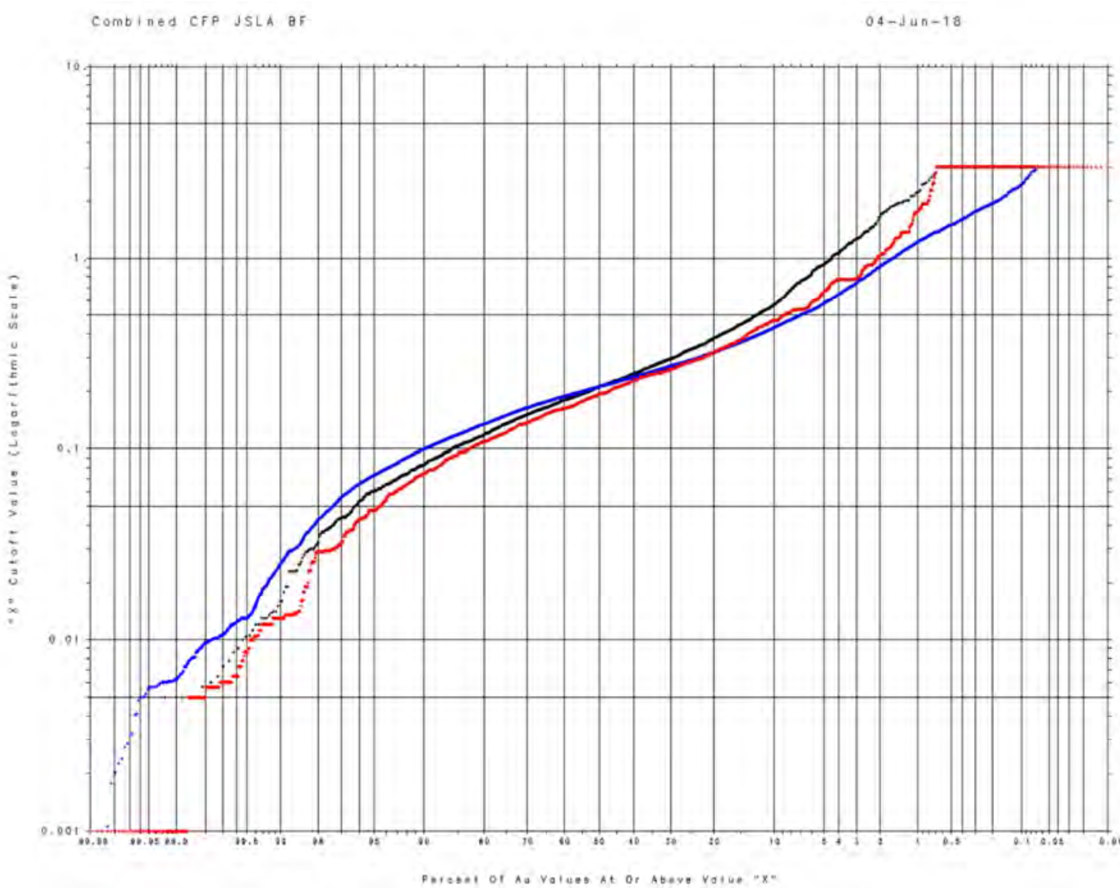

Figure 14-8 – Combined Cumulative Frequency Plots of Composites (Black), IDW Block Model (Blue) and Nearest Neighbor Block Model (Red)

It is worth noting that the northern deposit area contains about 68 million tons of backfill material and South Domes contains an additional 27 million tons of backfill material for a total of 95 Mt of backfill material contained within the LG shell limit based on a gold price of $1,400.

15.0 MINERAL RESERVE ESTIMATES

GRE has estimated the Mineral Reserves for the Castle Mountain project using a pit design on a $850 gold price pit shell (generated by a Vulcan pit optimizer), conventional open pit mining methods, and a gold price of $1,250 in the economic analysis. The Mineral Reserve for the Castle Mountain project is effective June 29, 2018.

The Mineral Reserve includes measured and indicated ore to produce proven and probable Mineral Reserves. The mine plan presented in Section 16.0 – Mining methods – details the production of this reserve. Several cutoff and ROM/mill cutover grades were used at different points in the production schedule to meet production targets. Vulcan generated pit shells at lower gold prices were used to assist in phase design.

15.1 Topography

Topography for the area was made from two sources: an October 2012 Aerotech Mapping aerial survey via Heritage Surveying and a May 2017 dataset from Compass Tools . The newer data was stitched into the previous and a single topography for most of the permitted land was created.

15.2 Geotechnical Parameters - Pit Slope

Slope angles are recommended by an analysis, “Preliminary Slope Angle Recommendations for the Castle Mountain Project,” performed by Call & Nicholas, Inc. on July 7, 2017 (Call & Nicholas, 2017a):

“The slope angle recommendation was based on a combination of overall, interramp, and bench-scale stability concerns. Overall stability calculations were largely driven by back-analyzed clay strengths and generalized rock-mass strengths estimated from point load data, UCS data, and RQD data distribution of drill holes piercing key rock units near analyses areas. A hydrostatic water surface at the 3900-foot elevation was assumed based on water level measurements taken in 1995 and 1996. Interramp stability calculations were based on major structure mapping performed by R.C. Capps in February 1991. Bench-scale stability calculations were based on fabric mapping data collected in 1993 and 2017.”

The result was applied to pit optimization in the following section and to the mine design in Section 16.0.

15.3 Pit Optimization

Vulcan software uses the 3D Lerchs & Grossman algorithm to generate optimized pit shells. The algorithm uses metal prices, average operating cost inputs, process-based metal recovery, royalties, and user-defined slopes by zone to produce an ultimate pit that contains the maximum net economic value for that set of parameters.

The economic inputs for Castle Mountain are summarized in Table 15-1. The result from Vulcan, Figure 15-1, was used as a basis for designing the ultimate pit, Figure 15-2. Figure 15-2 shows the ultimate pit design after haul road and bench designs were completed.

Table 15-1 - Vulcan Pit Optimizer Inputs

Input	Value
Mining Cost per Ton of Rock	$1.43
Mining Cost per Ton of Backfill	$1.14
General and Administrative Cost per Ton of Ore Processed	$0.64
Process Cost per Ton of Ore - ROM	$1.54
Process Cost per Ton of Ore - Mill	$9
Process Recovery - ROM	60%
Process Recovery - Mill	95%
Slope Angles	
Pit 1 Northwest Wall	50 deg
Pit 1 General	51 deg
Pit 2 Clay Zone	35 deg
Pit 2 Northwest Wall	50 deg
Pit 2 General	53 deg
Pit 3 Clay Zone	43 deg
Pit 3 Northwest Wall	50 deg
Pit 3 General below 1200 feet	51 deg
Pit 3 General above 1200 feet	53 deg
Discount Rate	10%
Gold Price	$1,300
Royalty	2.65%
Sales Cost per troy ounce	$5

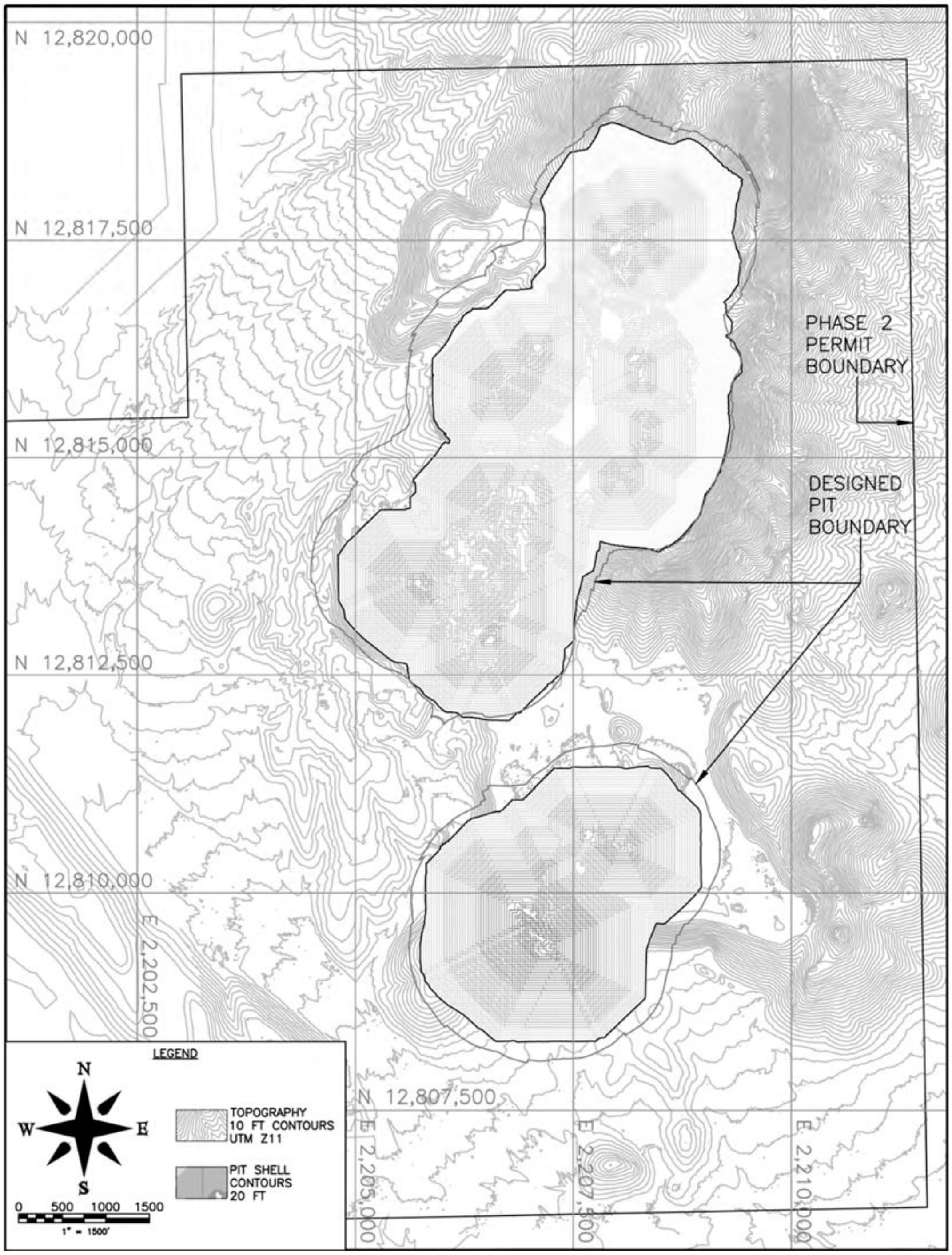

Figure 15-1 – Vulcan Pit Shell 850 $/oz

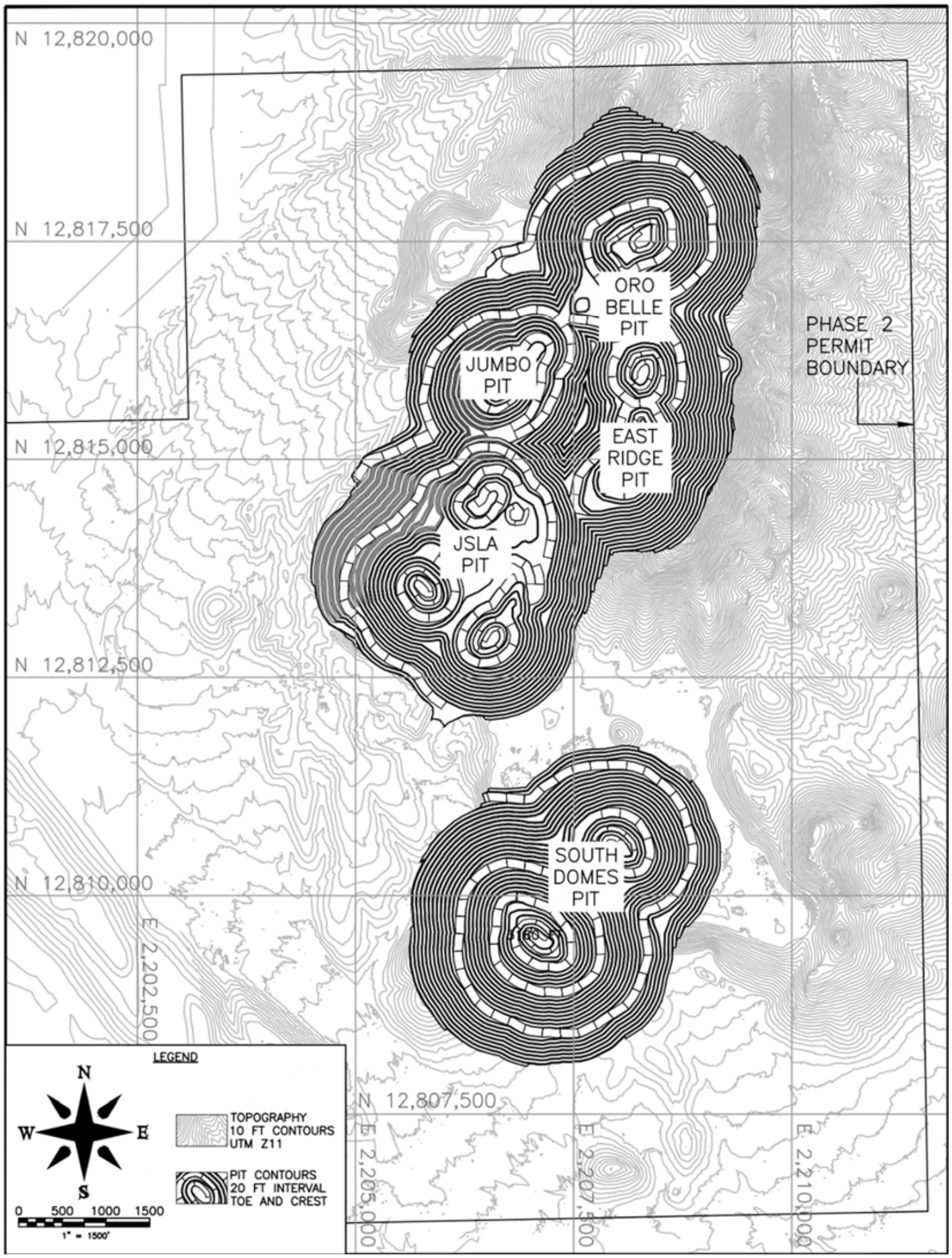

Figure 15-2 – Ultimate Pit

15.4 Pit Shell Selection

The Vulcan optimizer used a gold price of \$1,300/oz as a starting point (Base = revenue factor 1.0), subtracting a royalty of 2.65% to equal \$1,265.55 ounce. A series of 54 pit shells were generated at various revenue factors around this starting point from gold prices of $591.01 oz. to $1,841.38 oz.

Only measured and indicated categories were included. Inferred mineralization was treated as waste. The selected pit shell is shown below. At prices above $850, the additional ore added to the pit is minimal and the added waste offsets the additional revenue, creating a very flat discounted cash flow line. No significant economic value is added to the pit at gold prices above $850.

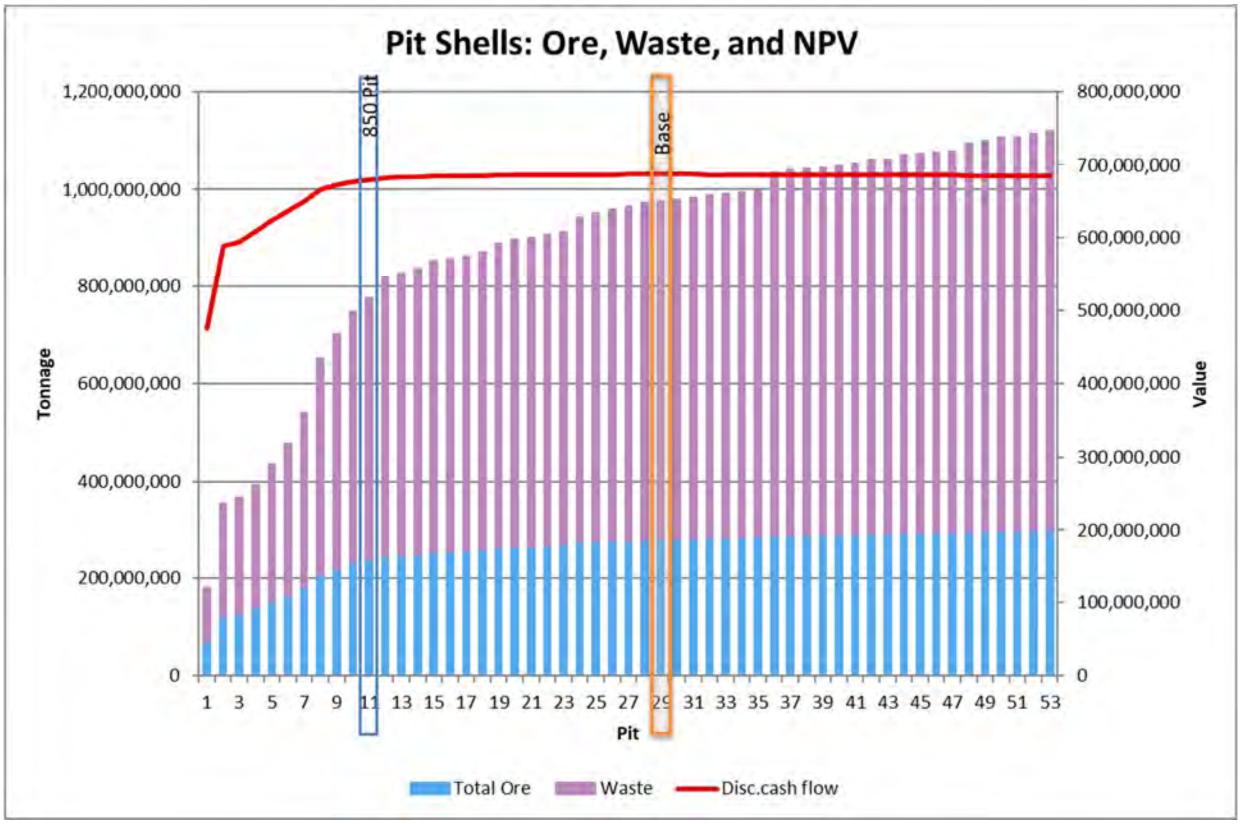

Figure 15-3 – Optimized Pit Shell Comparisons and Selection

15.4.1 Pit Shell Sensitivity

The pit shells show a high tolerance for changes gold price. The lower end of the gold price analysis, $750/oz and below, show a more dramatic change in pit shapes. This low end, however, excludes areas of high profitability and are therefore, not good design guides for the ultimate pit.

15.4.1.1. JSLA

The JSLA pit changes very little in the higher range of gold prices, $850 to $1250. Near the lower end of gold prices $600 to $750, the bottoms of the JSLA pits tend to retreat, but the two main pods of ore in the west and south of JSLA remain in the resource.

15.4.1.2. Jumbo

The Jumbo pit is not highly sensitive to gold price. At the lowest considered gold price, $600/oz, the pit stays nearly the same. This is likely due to an existing pit at the Jumbo site. With a large portion of the prestripping removed in Viceroy’s operation, Jumbo is unburdened by a large stripping volume.

15.4.1.3. Oro Belle and East Ridge

Much of the area around the existing pit at Oro Belle will fall within the lower end pits. The existing pit acts much in the same way as the pit at Jumbo, and the northern part of Oro Belle has already had much of the waste removed. The southern part of Oro Belle and on into East Ridge is highly sensitive to gold prices. Most of the resource in East Ridge remains in the pit shell as the gold price descends, but East Ridge is eliminated from the resource at the $600/oz gold price. The proximity to the mountain range drives the sensitivity for East Ridge. As the pit deepens, so does the waste mining increase substantially. Additionally, East Ridge lacks the exploration drilling density of the other pits. So, with a larger inferred resource, East Ridge is harder to keep in the targeted area for mineral reserve.

15.4.1.4. South Domes

The South Domes deposit is deep and localized. As such, the pit shape changes very little depending on the gold price. The exception is the satellite pit northwest of the main South Domes pit. The satellite pit falls out of the resource when the gold price drops to $755/oz. Because of this sensitivity, the satellite pit was analyzed for two other mining options: underground mining and exclusion from the project. A simple analysis shows that the satellite pit is, indeed, profitable at the base gold price. The underground option was closely examined in Section 15.4.2.

15.4.1.5. Additional Drilling

Much of the inferred resource lies in deep deposits. In order to extract this potential resource, the pits would need to widen and strip more waste rock. With limited space for surface facilities including dumps, the work required to increase the indicated and measured portion of the resource would not likely be fruitful. However, the inferred resource does show some areas of good potential on the east side of the project. There are some large areas east of the mountain ridge near

Oro Belle and East Ridge that have grade above cutoff and are near the surface. Drilling east of the pits could yield ore with lower stripping requirements than any other area in the project. It is recommended to focus there first.

15.4.2 Underground Mine Option for South Dome Satellite Deposit

The current resource model includes a two-cone open pit for the South Dome deposit. Although the second cone that mines the eastern portion of the deposit is viable at current revenue and cost projections, an underground mine would eliminate the large prestripping requirement and could produce better economic results. For that reason, GRE completed an underground mine plan as a trade off alternative to mining the satellite South Dome deposit via surface methods.

15.4.2.1. Mine Layout

The mine plan was developed using a net smelter return selection of $30/ton based on grade of 0.06 Au opt considering the estimated mining cost ($26/ton), process cost ($11.50/ton), recovery (90%), royalty rate (2.65%), and gold price ($1,300/troy ounce). The resource model was viewed in north-south sections and manual boundaries were selected above the target gold grade to determine the mining areas for the underground option considering only blocks defined as measures and indicated resources. Two distinct mining areas were determined from this exercise and are shown in the figure below.

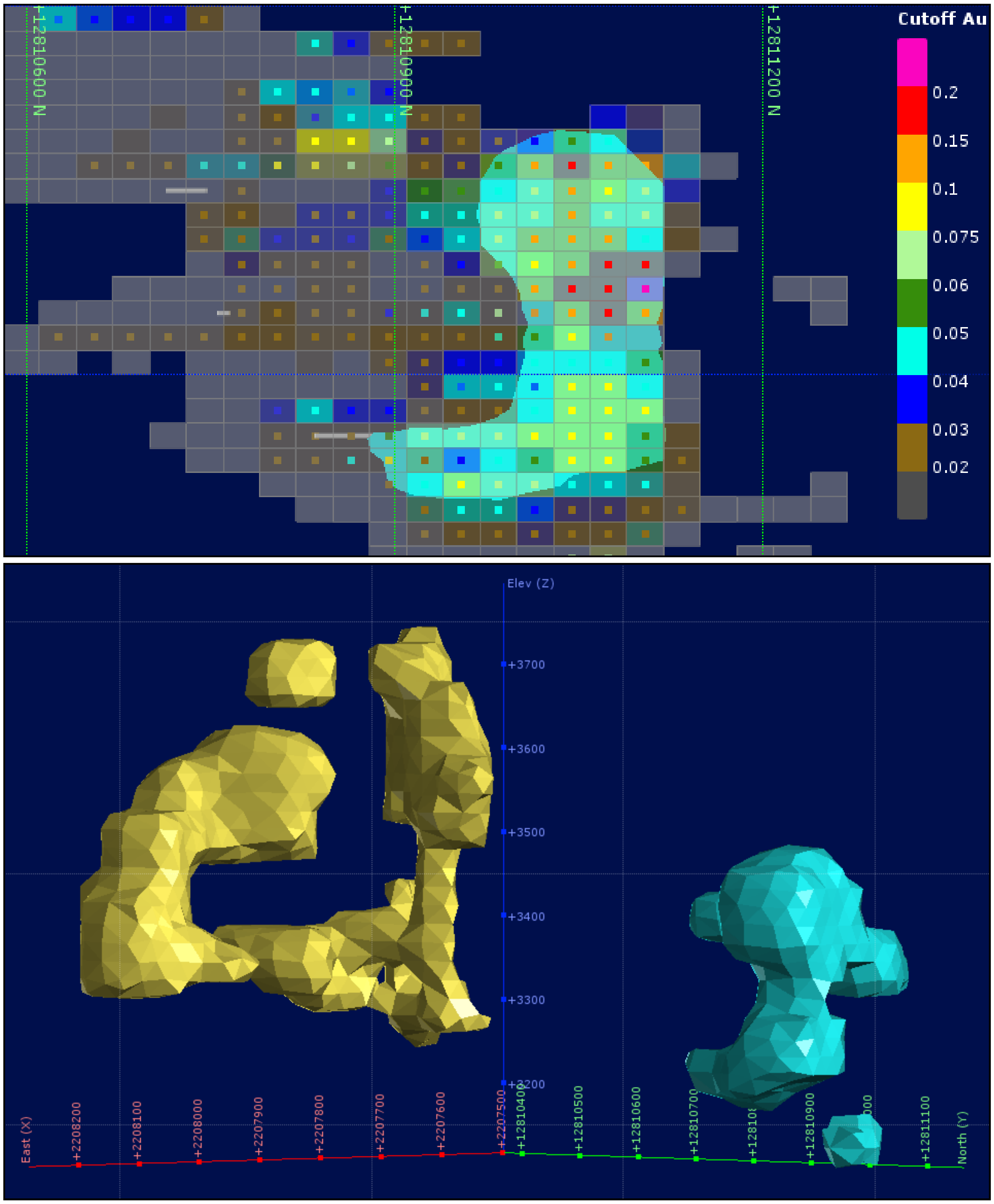

Figure 15-4 – Underground Mining Areas Selection (Top – Block Selection along Section, Bottom – 3D Mining Areas Created from Sections)

All resource blocks contained within the three-dimensional areas were included in the mine plan. Any inferred resource blocks within the solid were treated as waste. The table below summarizes the resource blocks for each mining area, Mining Unit 1 (Cyan), Mining Unit 2 (Yellow).

Table 15-2 – Underground Mining Areas, Tons and Grade

Mining Area	Tons	Au Opt	Au Ounces
Unit 1	381,636	0.101	38,730
Unit 2	799,452	0.099	78,918

Development requirements were laid out as 3D line entities, and included a decline from the adjacent open pit beginning at elevation 3890, and central ramp system from for the entire vertical extent of the mining areas (3750 to 3120), levels for top and bottom access to the stope area at approximately every 60 feet, a central ventilation raise that daylights to the open pit at elevation 4060), and a secondary access/escape way along a level adit access at elevation 3460 that connects to the open pit ramp system. The figure below shows these elements in an isometric view.

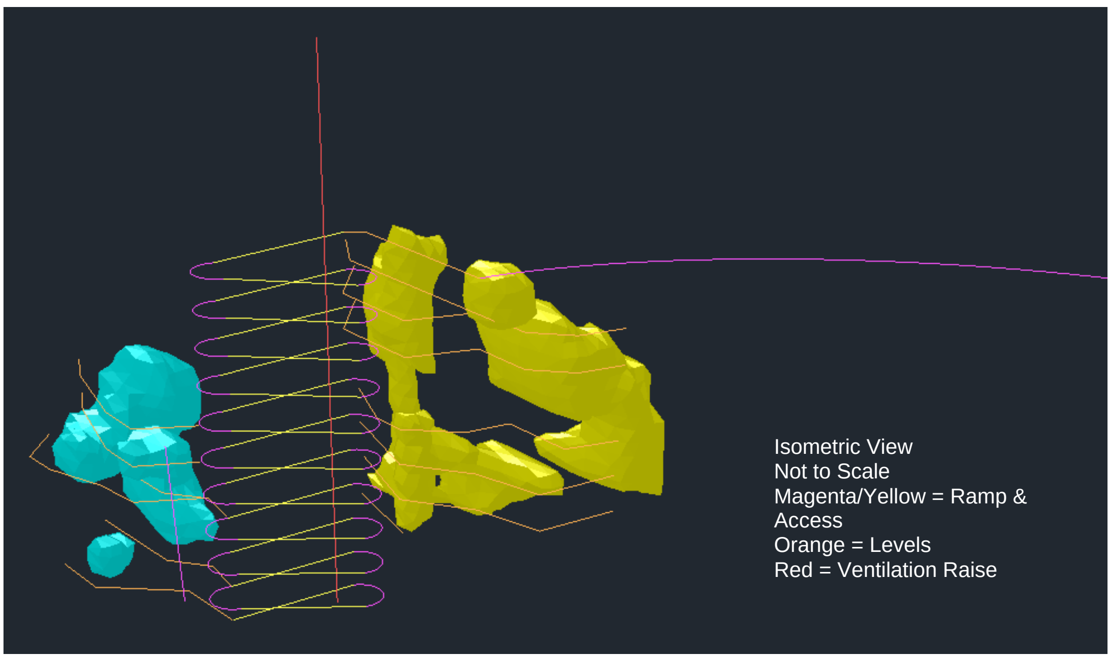

Figure 15-5 – Underground Mining Option, Development Layout

Mining would take place within the defined mining units between each level utilizing the end slice method. This method allows for development of large stope blocks that are mined using down-the-hole blasthole drills and that are subsequently filled. Mining takes places in alternating blocks, in this specific case 30-foot (ft) wide slices, and then filled with cemented tailings/sand fill. The cemented fill allows for 100% extraction of the slices within each cut. A sill pillar is left below the mucking level for each slice. This mine plan considered levels spaced at 60 feet with a 12-ft sill pillar, resulting in an extraction ratio of 80%. It is possible to recover all or a portion of the sill pillar with cemented fill; however, this was not considered for the study. Figure 15-6 shows a conceptual layout of the mining method illustrating the haulage levels, top sill cut, and bottom sill cut. Dilution (additional to that contained in the block model) of 9% at 0.0008 Au opt was added to the resource numbers to account for 6 feet (3 feet each end) of low-grade (0.02 Au opt) material blasted at the end of each slice and 2 feet of the adjacent cut fill material containing no grade.

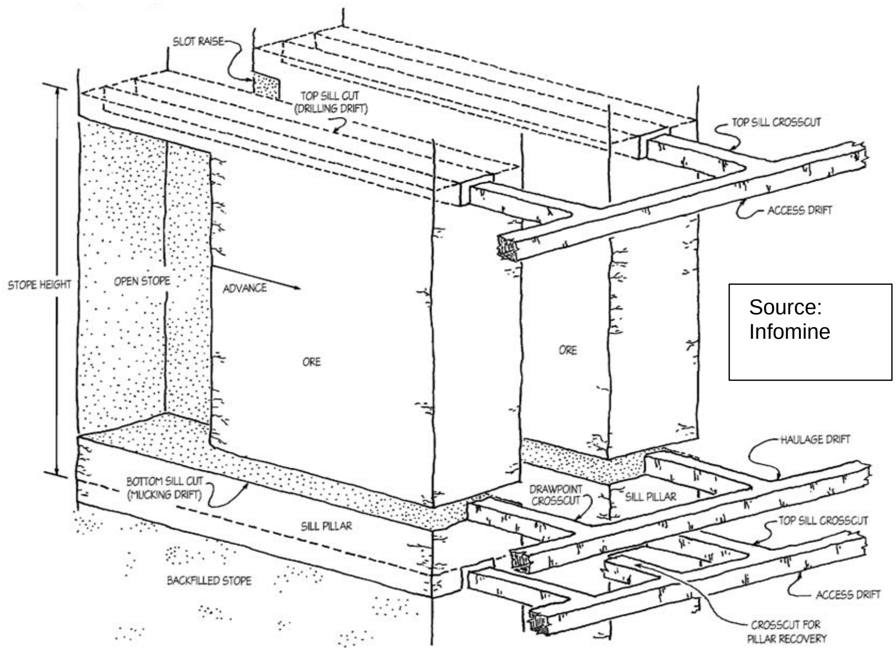

Figure 15-6 – Conceptual Layout, End Slice Mining

15.4.2.2. Conclusion

Although an underground option would eliminate the high cost of stripping the plentiful overburden of the South Domes satellite deposit, the increased cost of underground mining proved to be a financially less economic option. The mining cost ends up being $27.65/ton of ore mined, which is far greater than the surface mining cost associated with the rest of the project.

15.5 Cutoff Grade

The cutoff grade for the Mineral Reserve is calculated from the pit optimization parameters. The final cutoff grades used in the production schedule and economic model are higher to maximize project economics and meet production schedule needs.

15.5.1.1. ROM

The Run of mine cutoff grade is the incremental haulage cost between the waste dump and heap, plus the heap leaching costs, plus pad costs, plus G&A cost, divided by recovery, divided by the gold price (after NSR royalty deduction). Using the pit optimization parameters, the equation is:

($0.25 + $1.54 + $0.64)/0.60/($1,300 X 0.9735) = 0.0028 (0.003) oz/ton

GRE chose a slightly higher cutoff of 0.004 oz/ton for backfill material, and 0.005 oz/ton for the mineral reserve.

15.5.1.2. Mill

The Mill Cutoff Grade is similarly calculated:

($0.25 + $9.0 + 0.64)/.95/($1300 X 0.9735) = 0.0082 (0.008) oz/ton

The ROM to Mill cutover grade is the grade where the revenue generated by ROM processing is exceeded by milling the ore. Again, using the pit optimization values the equation is:

Cutover is where Grade X 0.60 X (1300 X 0.9735) – ($0.25 + $1.54 + $0.64) = Grade X 0.95 X (1300 X 0.9735) - ($0.25 + $9.0 + 0.64) = 0.017 oz/ton

15.6 Mineral Reserve Statement

The Castle Mountain Mineral Reserves are shown in Table 15.3 and Table 15.4. The Mineral Reserves are shown using a cutoff of 0.004 oz per ton (0.14 g/tonne) for the JSLA backfill and 0.005 oz per ton (0.17 g/tonne) for mining fresh ore.

Table 15-3 - Castle Mountain Mineral Reserves

Reserves	Tons (Tonnes), millions	Gold Grade, oz/ton (g/tonne)	Gold Ounces (millions)
Proven	150.6 (136.6)	0.017 (0.58)	2.56
Probable	67.2 (61.0)	0.015 (0.51)	1.00
Total	**217.8 (197.6)**	**0.016 (0.56)**	**3.56**

Note: The Mineral Reserve estimate with an effective date of June 29, 2018 is based on the Mineral Resource estimate with an effective date of March 29, 2018 that was prepared by Don Tschabrun, SME RM of Mine Technical Services Ltd. The Mineral Reserve was estimated by Global Resource Engineering, LLC with supervision by Terre Lane, MMSA, SME RM. Mineral Reserves are estimated within the final designed pit which is based on the $850/oz pit shell with a gold price of $1,250/oz. The minimum cutoff grade was 0.004 oz/ton (0.14 g/tonne) gold and 0.005 oz/ton (0.17 g/tonne) gold for Stages 1 and 2, respectively. Average life of mine costs are $1.26/ton ($1.39/tonne) mining, $1.56/ton ($1.72/tonne) processing ROM, $8.17/ton ($9.01/tonne) processing Mill/CIL, and $0.73/ton ($0.80/tonne) processed G&A. The average process recovery was 72.4% for ROM and 94% for Mill/CIL. Tons and gold ounces are both reported in millions. Small differences in total tonnage and grade may occur due to rounding. The Mineral Resource estimate is inclusive of Mineral Reserves.

Table 15-4 - Castle Mountain Mineral Reserves by Area

	Proven			Probable			Proven & Probable		
Resource Area	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**	**Tons (Tonnes), millions**	**Gold Grade, oz/ton (g/tonne)**	**Gold Ounces (millions)**
JSLA - Rock	62.5 (56.7)	0.015 (0.52)	0.95	1.9 (1.7)	0.027 (0.92)	0.05	64.5 (58.5)	0.016 (0.54)	1.01
JSLA - Pit fill	0	0	0	18.0 (16.3)	0.010 (0.35)	0.18	18.0 (16.3)	0.010 (0.35)	0.18
Jumbo	9.8 (8.9)	0.022 (0.77)	0.22	2.9 (2.6)	0.011 (0.39)	0.03	12.7 (11.5)	0.020 (0.68)	0.25
Oro Belle	42.7 (38.7)	0.017 (0.57)	0.71	6.8 (6.2)	0.014 (0.48)	0.10	49.6 (45.0)	0.016 (0.56)	0.80
East Ridge	5.6 (5.1)	0.023 (0.80)	0.13	7.1 (6.4)	0.012 (0.42)	0.09	12.8 (11.6)	0.017 (0.59)	0.22
South Domes	29.9 (27.1)	0.018 (0.63)	0.55	30.5 (27.7)	0.018 (0.62)	0.56	60.4(54.8)	0.018 (0.63)	1.10
Total	**150.6 (136.6)**	**0.017 (0.58)**	**2.56**	**67.2 (61.0)**	**0.015 (0.51)**	**1.00**	**217.8 (197.6)**	**0.016 (0.56)**	**3.56**

Note: The Mineral Reserve estimate with an effective date of June 29, 2018 is based on the Mineral Resource estimate with an effective date of March 29, 2018 that was prepared by Don Tschabrun, SME RM of Mine Technical Services Ltd. The Mineral Reserve was estimated by Global Resource Engineering, LLC with supervision by Terre Lane, MMSA, SME RM. Mineral Reserves are estimated within the final designed pit which is based on the $850/oz pit shell with a gold price of $1,250/oz. The minimum cutoff grade was 0.004 oz/ton (0.14 g/tonne) gold and 0.005 oz/ton (0.17 g/tonne) gold for Stages 1 and 2, respectively. Average life of mine costs are $1.26/ton ($1.39/tonne) mining, $1.56/ton ($1.72/tonne) processing ROM, $8.17/ton ($9.01/tonne) processing Mill/CIL, and $0.73/ton ($0.80/tonne) processed G&A. The average process recovery was 72.4% for ROM and 94% for Mill/CIL. Tons and gold ounces are both reported in millions. Small differences in total tonnage and grade may occur due to rounding. The Mineral Resource estimate is inclusive of Mineral Reserves.

16.0 MINING METHODS

The Castle Mountain deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the resource model and reserve estimate described in the previous sections.

16.1 Summary

The mine plan is the extraction of the proven and probable ore in the mineral reserve presented in Section 15.0. The mine plan was designed to deliver 16,425,000 tons of ore per year to the processing facility in two process types. ROM ore in the quantity of 15,476,000 tons per year and mill ore in the quantity of 949,000 tons per year starting in Year 4 of the mine plan. Prior to Year 4, the mine plan will deliver 5,110,000 tons of ore per year strictly from the JSLA backfill to the heap leach pad.

The mine plan includes:

- Ultimate pit design including benches, ramps, and haul roads;
- Pit phase designs;
- A mine production schedule;
- Waste storage design;
- Yearly mine plan drawings including the pit, exterior waste dumps, and in-pit waste backfill; and,
- Equipment and labor requirement calculations.

16.2 Ultimate Pit Design

The Vulcan pit shell analysis shown in Section 15 provides a basis for creating the ultimate pit design. The objective of the Vulcan pit optimization was to maximize the economic extraction of the mineral resources contained in the block model. The optimal pit shell selected was based on the combined principles of incremental net present value (NPV) and cash flow, and it included an analysis of a shell-by-shell overall pit value. The $850/oz pit (0.68 revenue factor) was selected as the basis for designing the ultimate pit.

The ultimate pit design was developed using Geovia GEMS mine design software. The final pit design is the result of multiple iterations in which ramp locations and layback configurations have been examined to maximize recovery of the mineral resource and provide for efficient haulage routes for the mobile equipment. Permanent haul roads were placed for maximum use and a pit entrance as close as possible to the mine facilities.

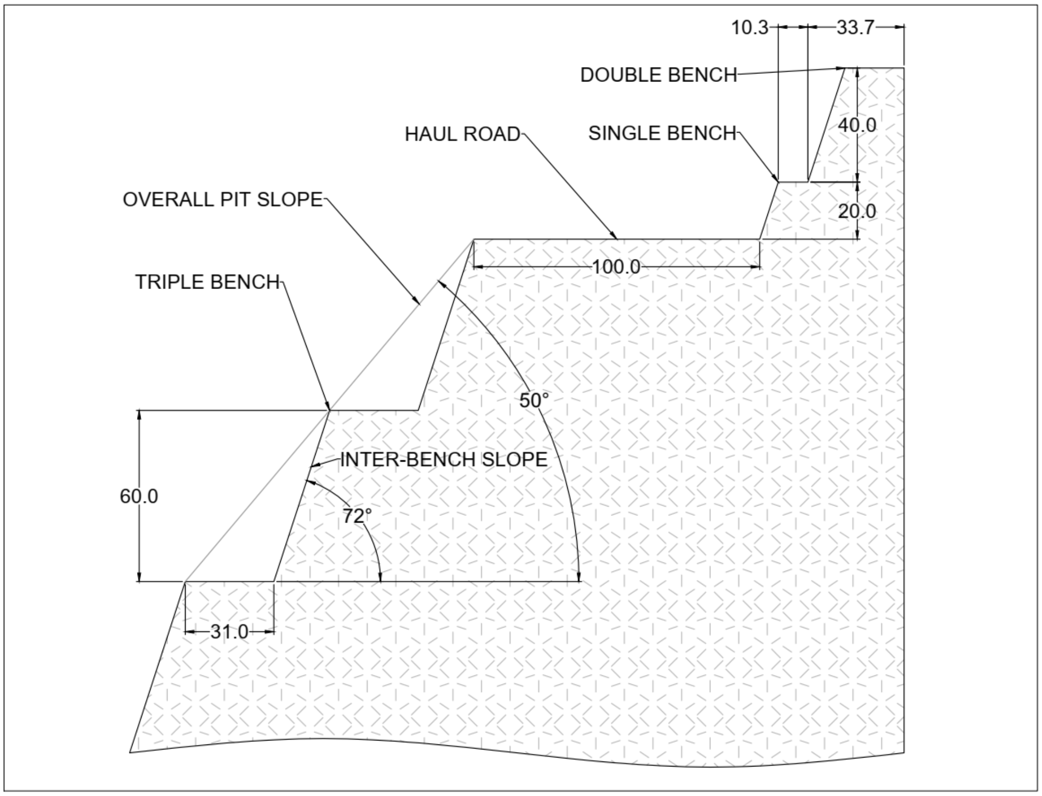

Figure 16-1 – Bench Design Example

Haul roads and ramps were designed to provide access to the pit phases and haulage routes to the ore crusher, leach pad, ore dumps, and truck shop. The roads have generally been laid out with a maximum ore recovery in the pit shell bottoms. The haul road design is based on the following parameters:

- Total width two-way roads: 100 ft
- Total width one-way roads: 50 ft
- Maximum grade: 10%

A memorandum from Call & Nicholas, Inc. provides the basis for pit slope angles. The memorandum, dated July 7, 2017, recommends the slope values in Table 16.1 and provides a plan view for the location of the slope zones in Figure 16.2. The slope zone boundary lines were reproduced in GEMS and used for slope variability during pit design. In practical application, the bench heights and bench widths were normalized to ensure continuous catch benches throughout the pit. The bench face angle was set as the zone configured variable in each bench expansion in GEMS. The bench height was set to 60 feet (triple benches) and the bench width (berm width) was set to 30 feet. Bench face angles remained the same, and overall pit slope angles were adjusted to be within 2 degrees; see Table 16.2.

Table 16-1 – Slope Recommendations from Call & Nicholas

Pit	Location	IRSA - degrees	OSA - degrees	Bench Width - feet	Bench Height - feet	Bench Face Angle - degrees
1	Northwest Wall	50	50	31	60	72
1	General	51	51	43	80	75
2	Clay zone	35	NA	35	60	50
2	Northwest Wall	50	50	31	60	72
2	General	53	53	39	80	75
3	Clay zone	43	NA	30	60	60
3	Northwest Wall	50	50	31	60	72
3	General Above 1200'	51	51	43	80	75
3	General Below 1200'	53	53	39	80	75

Table 16-2 – Slope by Zone Normalized for GEMS

Pit	Location	Bench Width - feet	Bench Height - feet	Bench Face Angle - degrees	OSA - degrees
1	Northwest Wall	29	60	72	51
1	General	29	60	75	53
2	Clay zone	29	60	50	37
2	Northwest Wall	29	60	72	51
2	General	29	60	75	53
3	Clay zone	29	60	60	43
3	Northwest Wall	29	60	72	51
3	General Above 1200'	29	60	75	53
3	General Below 1200'	29	60	75	53

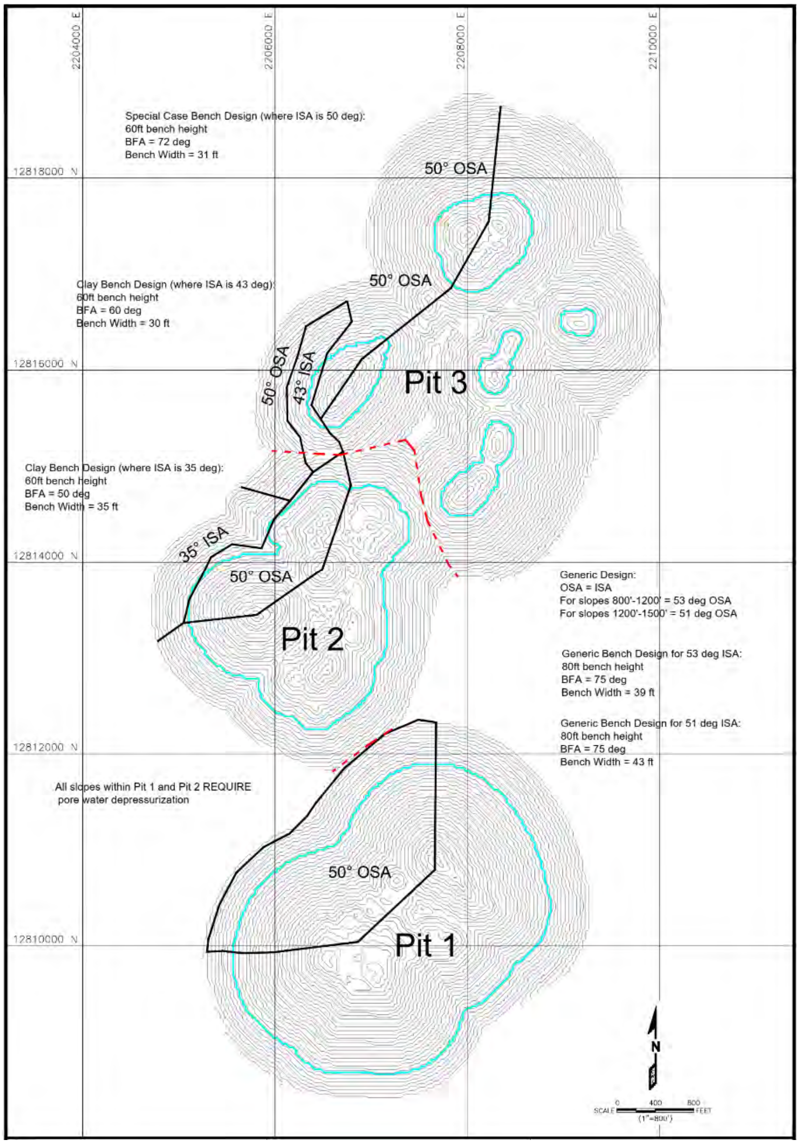

Figure 16-2 – Slope Zones Defined by Call & Nicholas

16.3 Phase Design

The ultimate pit is comprised of five pit areas: JSLA, Jumbo, Oro Belle, East Ridge, and South Domes. Each of the pit areas also has one or more phases in the mine plan. The phase design was based partly on the low revenue factor pit shells from Vulcan, but mostly on the spatial configuration of the pit bottoms. JSLA, Jumbo, and Oro Belle have an old Viceroy excavation to work around, and spatially configured pit phases. East Ridge is a small pit and has only one phase. South Domes has an enormous pre-strip requirement; therefore, South Domes phases use a combination of economic shells and pit bottom for design.

The pit areas progress in the following order:

1. JSLA
2. Jumbo
3. Oro Belle
4. East Ridge
5. South Domes

This pit area order became the basis for the mine plan based on the higher NPV generated by mining JSLA first, and the practicality of mining adjacent/closer pits subsequently. East Ridge and South Domes were planned at the end of the mine life as they constitute a higher strip ratio, a higher risk based on drilling density, and lower incremental NPV.

16.4 Pit Phases

The series of Vulcan pit shells helped guided the design of the pit phases as well as the spatial layout of the previously mined topography. Phases/pushbacks were laid out to provide access to prestrip mining and mining of successive phases and to provide practical operating room for mine equipment. The five pit areas have a combined total of 17 phases summarized in Table 16-3.

Table 16-3 – Pit Area with Phases

Pit Area	Number of Phases
JSLA	8
Jumbo	1
Oro Belle	3
East Ridge	1
South Domes	4

The phases are sequenced with the goal of maximizing cash flow and balancing total material mined. Balancing material includes targeting 15,476,000 ROM tons per year and 949,000 Mill Tons per year.

16.4.1 JSLA Phases

Phase 1 is defined by the previous mine plan. It is strictly a backfill phase with no blasting of fresh rock. Its purpose is to begin mining operations at low cost until the rest of the mine and processing can come online.

Phases 2 and 3 are the tops of two ore "pods" located to the south and west of Phase 1. Both phases include a stripping component around Phase 1 and a large ore producing area. The intent is to mix ore and waste in manageable portions. The driving force for this strategy is the lack of a pre-production year in the mine plan. The mine must go from mining backfill to nearly full production very quickly in year 4. The bottom of both phases matches the lowest bench in Phase 1.

Phase 4 of JSLA is the entire breadth of the pit from the bottom of the three previous phases to where the pit starts to break apart into separate pit bottoms.

The final 4 phases of JSLA are a north pit bottom (Phase 6N), the several benches that cover both the southwest pit bottoms (Phase 6SA), and the two southern pit bottoms (Phases 6SB and 6SC). Each pit bottom was broken into separate phases in anticipation of scheduling requirements.

16.4.2 Oro Belle Phases

The first phase of Oro Belle (Phase 8) is largely prestripping and ramp building. The ramp will provide access to both upper Oro Belle and upper East Ridge pits. This phase is large and designed around the existing Oro Belle pit. The remaining two phases of Oro Belle (Phases 9A and 9B) are the separate pit bottoms. They are broken into separate phases in anticipation of scheduling needs.

16.4.3 South Domes Phases

The first phase of South Domes was designed to be nearly all waste (Phase 11). The second phase takes the pit down to the split between pit bottoms (Phase 12). The last two phases of South Domes are the pit bottoms (Phases 13 and 14).

16.5 Mine Plan

JSLA was sequenced early in the mine life to facilitate a backfill waste strategy instead of larger waste dumps. As pits are emptied, each becomes a new target for a backfill. By the end of the mine life, all pits except those of South Domes are backfilled with waste.

Mining begins in the old Viceroy JSLA pit, which is currently filled with blasted rock from the previous mine operation. Though the material was designated as waste during operations from 1995 to 2001, at the then average gold price below $400/oz, it is now economical at higher current gold prices. This material now represents a low-cost mineral resource.

In the first phase of mining, to span three years, ore production will be 5,110,000 tons per year of ROM material from the JSLA backfill. Prestripping the next phases will start in year 3 and ore production will ramp up in year 4 to 16,425,000 tons per year for the remainder of the mine life. This increase of mining activity will trigger the need for a new phase in mining equipment, processing capacity, and permit requirements. Figures 16-3 through Figure 16-7 show the pit, dump, and permit boundary end of year plans throughout the mine life.

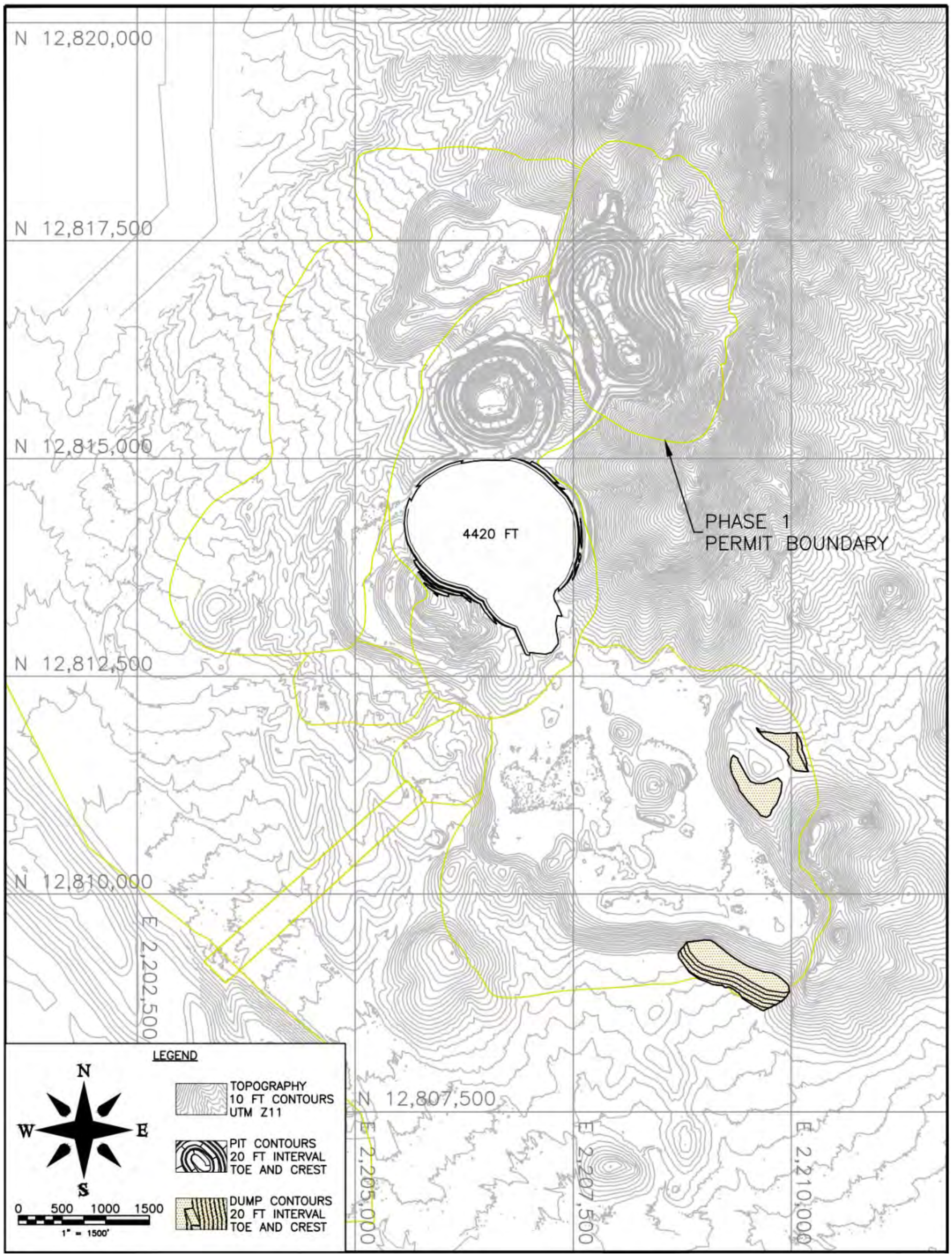

Figure 16-3 – Mine Plan, Year 1

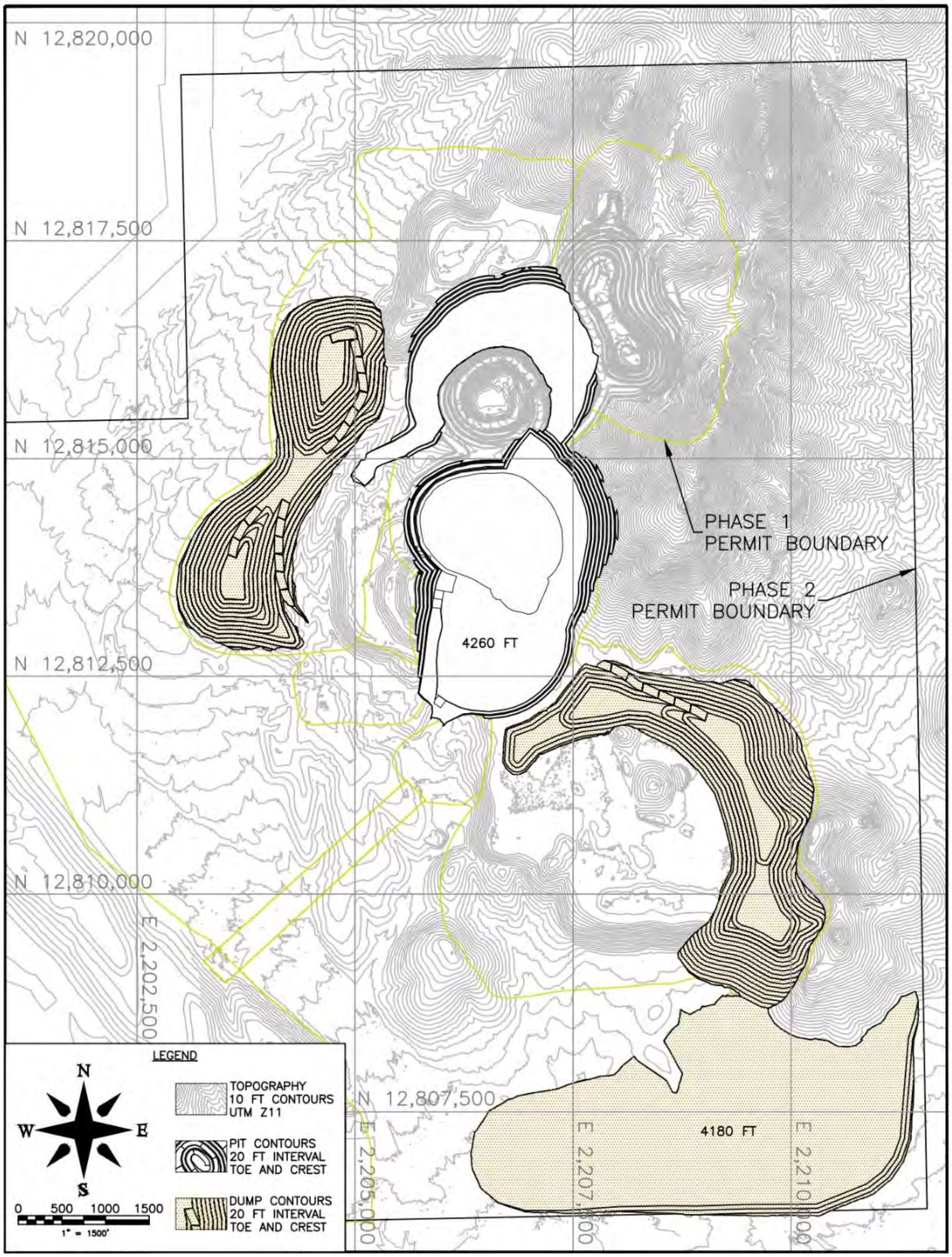

Figure 16-4 – Mine Plan, Year 3

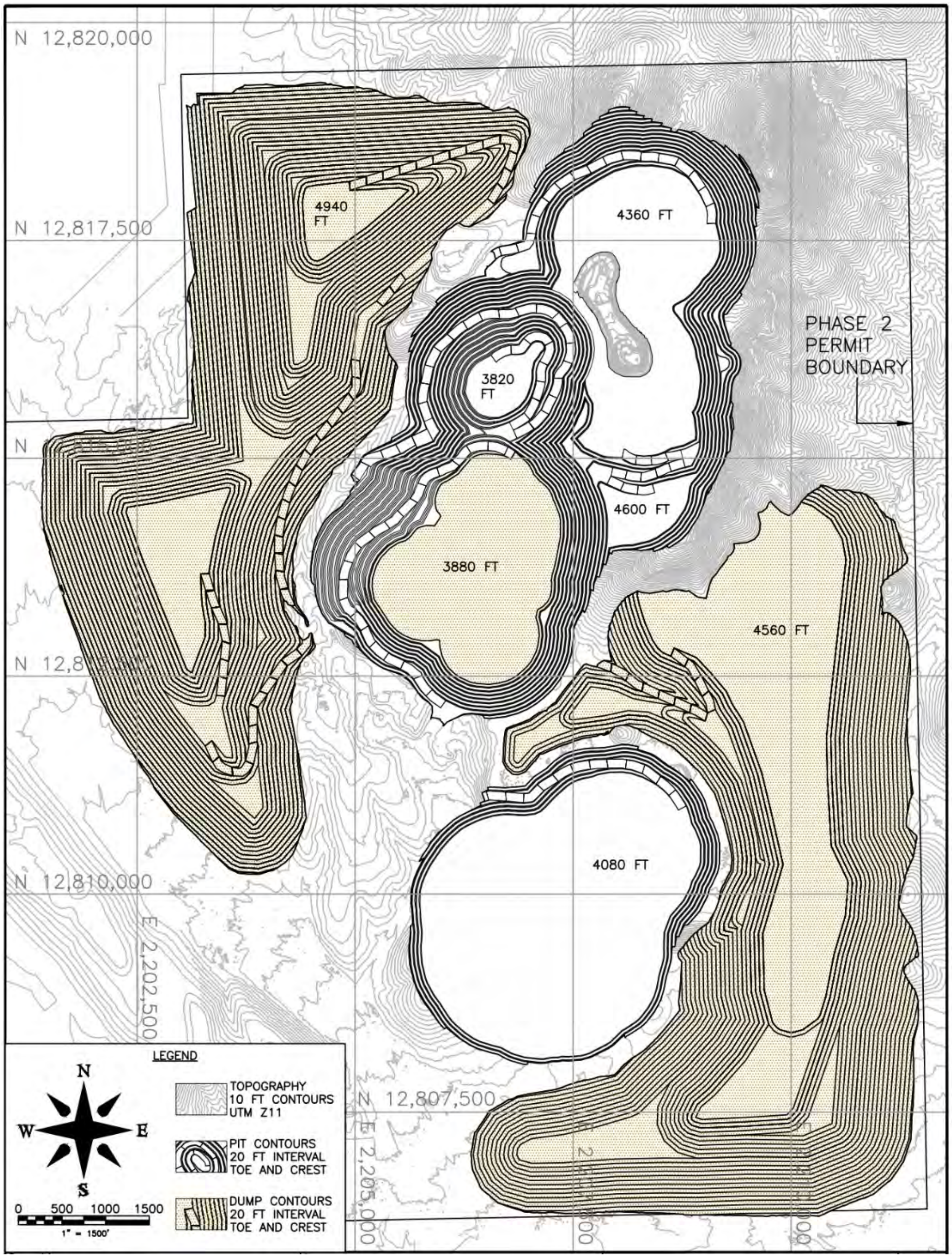

Figure 16-5 – Mine Plan, Year 9

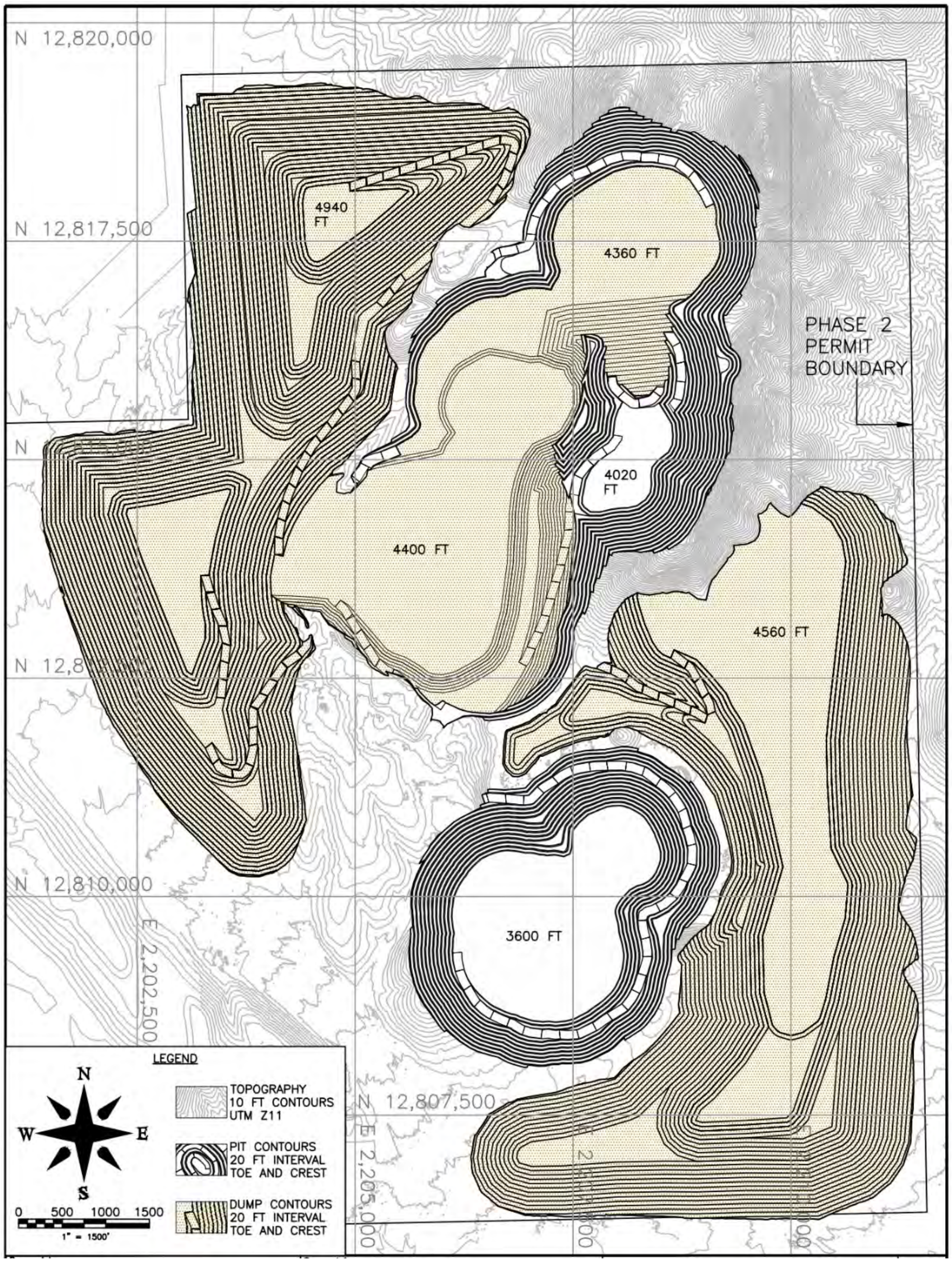

Figure 16-6 – Mine Plan, Year 13

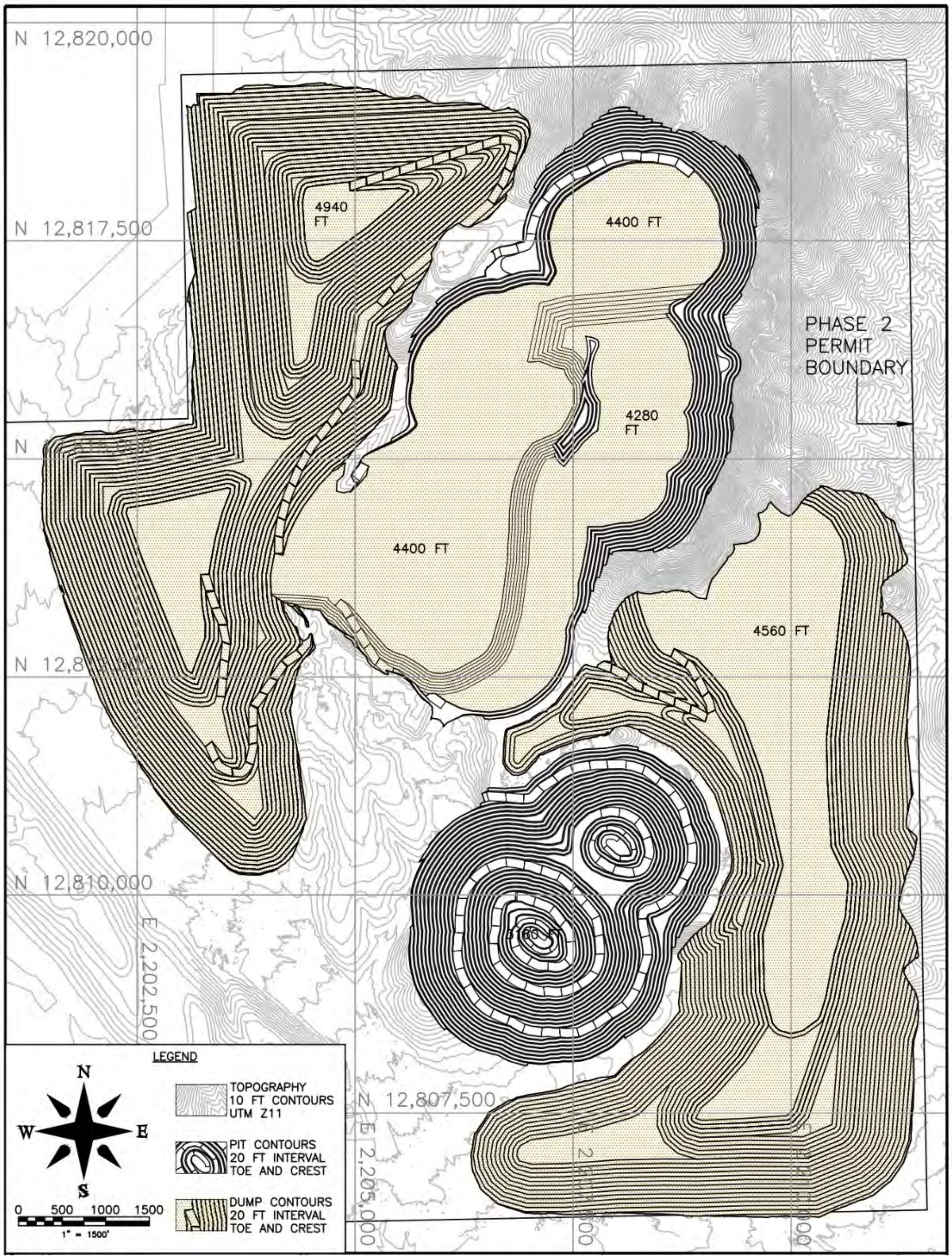

Figure 16-7 – Mine Plan, Year 16 (Final)

16.6 Production Schedule

The mine schedule was developed by volumetric reports of each phase surface from GEMS. The material was discretized in the report by multiple cutoff values that could be adjusted throughout the mine schedule to keep a consistent flow of ore and waste. Only measured and indicated categories were included in the schedule. All inferred and uncategorized material were treated as waste.

16.6.1 Processing Capacity

Available water is the limiting factor in ore throughput during Stage 1. Currently water availability limits the mine to 14,000 tons per day of ROM material (5,110,000 tons per year). Once the necessary water rights can be acquired, as part of the Stage 2 development plan, ore production can be increased to 42,400 tons per day of ROM material (15,476,000 tons per year) and 2,600 tons per day mill material (949,000 tons per year).

16.6.2 Ore and Waste Production Schedule

Material from a phase is mined as prestrip if it is on a bench that has a stripping ratio of 5:1 or greater including all benches in the phase above it. Any bench with a strip ratio less than 5:1 is mined as ore along with all benches below it. Generally benches are 20 feet high, 40 foot double benches are assumed in prestrip when ore is not present. Ore benches that also hold a lot of waste have an ore mining rate much lower than benches of mostly ore. For this reason, up to four multiple pit areas will be mined concurrently. Figure 16-8 shows phases mined by year, and Figure 16-9 shows the breakdown of production benches ore and waste vs prestrip benches ore and waste.

Pit Area	Phase	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
JSLA	1	■	■	■	■												
JSLA	2			■	■												
JSLA	3				■	■											
JSLA	4					■	■	■	■								
JSLA	6N								■								
JSLA	6SA								■								
JSLA	6SB								■	■							
JSLA	6SC								■	■							
Jumbo	7			■	■	■	■	■	■	■	■						
Oro Belle	8						■	■	■	■	■	■					
Oro Belle	9A											■	■				
Oro Belle	9B											■					
East Ridge	10						■	■	■	■	■	■	■	■	■		
South Domes	11					■	■	■	■	■	■	■					
South Domes	12											■	■	■	■		
South Domes	13														■		
South Domes	14														■	■	■

Figure 16-8 – Pit Area Mined by Year

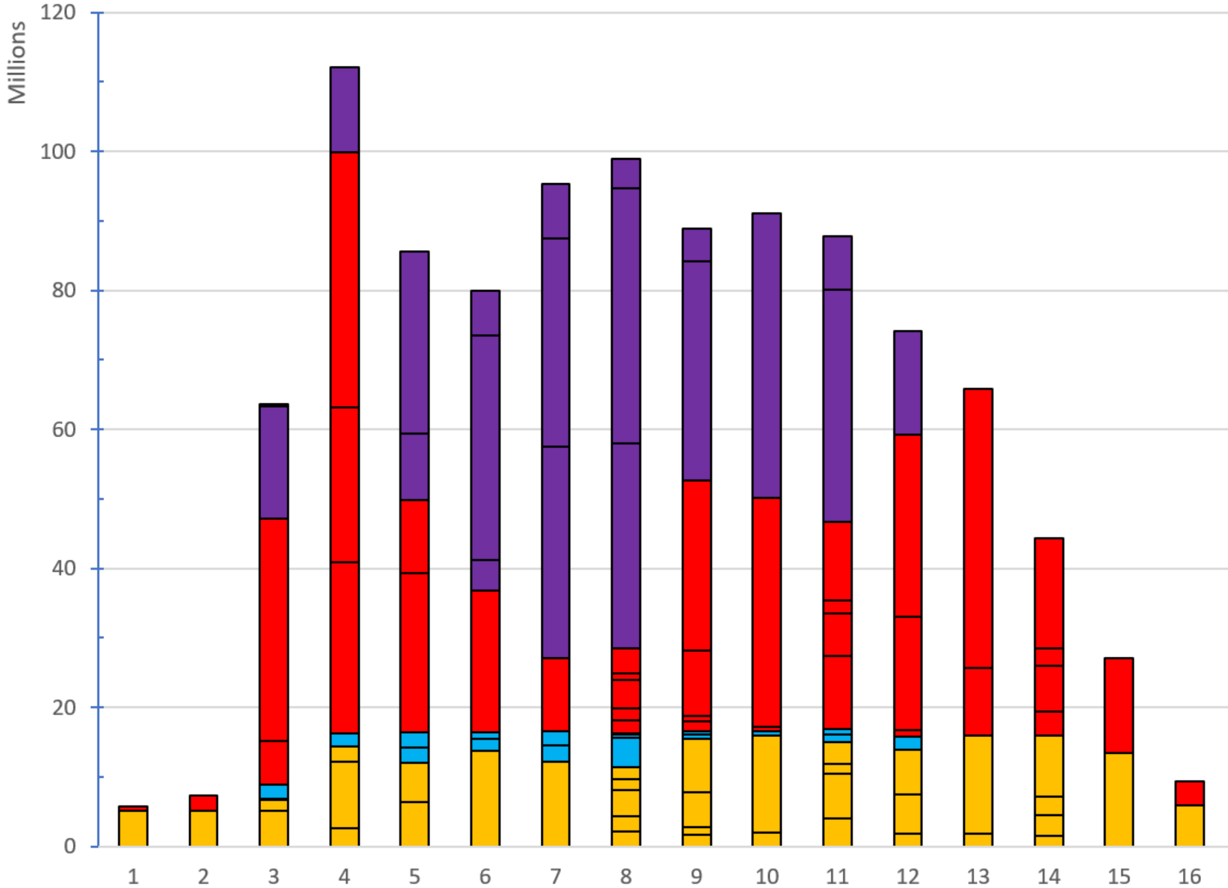

Figure 16-9 – Yearly Production and Prestripping Tons, Ore and Waste

Note: The colors represent the following material types in the figure, also the term prestrip is used he to imply mining of high ratio waste benches and does not imply the financial capitalization of waste mining throughout the operations:

- Yellow is Ore from a production bench
- Red is Waste from a production bench
- Blue is Ore from a prestripping bench
- Purple is Waste from a prestripping bench

16.6.3 Production Schedule Detail

Years 1 Through 3

Production begins in the JSLA backfill. Ore and waste will be mined by contract mining. 19,049,073 tons of ore are mined with contained gold totaling 195,895 ounces. All material mined in years 1 and 2 is exclusively previously-mined and backfilled by Viceroy. Prestripping new areas starts in Year 3.

Years 4 Through 16

Mine production ramps up to full scale in Year 4 and continues in a steady state until ore is depleted near the end of the mine life. Production also switches to a fleet owned and operated by Equinox Gold. Table 16-4 shows the production by year for the entirety of the project.

Table 16-4 – Yearly Production Schedule

Year	Ore kTons	Contained Gold koz	Gold Grade opt	ROM kTons	Contained ROM koz	ROM Gold Head Grade opt	Mill kTons	Contained Mill koz	Mill Gold Head Grade opt	Waste kTons
1	5,114	59	0.011	5,114	59	0.011	0	0	-	630
2	5,114	56	0.011	5,114	56	0.011	0	0	-	2,193
3	8,822	82	0.009	8,798	81	0.009	24	1	0.059	54,875
4	16,244	194	0.012	15,695	155	0.010	549	39	0.071	95,907
5	16,469	273	0.017	15,562	174	0.011	908	99	0.109	69,074
6	16,383	278	0.017	15,441	200	0.013	942	78	0.083	63,586
7	16,496	222	0.013	15,831	180	0.011	665	42	0.064	78,876
8	16,262	209	0.013	15,909	182	0.011	354	27	0.076	82,725
9	16,518	304	0.018	15,536	205	0.013	982	99	0.101	72,300
10	16,518	282	0.017	15,548	202	0.013	970	79	0.082	74,612
11	16,868	317	0.019	15,872	230	0.014	996	87	0.087	70,954
12	15,849	207	0.013	15,286	170	0.011	563	37	0.066	58,301
13	15,893	283	0.018	15,058	196	0.013	835	87	0.105	49,979
14	15,939	350	0.022	14,725	198	0.013	1,214	151	0.125	28,417
15	13,478	247	0.018	12,698	188	0.015	780	60	0.077	13,591
16	5,836	200	0.034	4,873	77	0.016	963	124	0.128	3,457
Total	217,802	3,563	0.016	207,058	2,552	0.012	10,745	1,012	0.094	819,479

16.6.4 Royalty Payments

All gold ounces in the project are subject to a 2.65% royalty that is owned by Franco Nevada. In addition, there are three royalty agreements on the property overlapping the pit. Table 16-5 shows the claim, royalty term, and owner. These royalty payments are all included in the projects financial analysis and represent a 4.31% NSR when all are combined. The location of each royalty claim is shown in Figure 16-10.

Table 16-5 – Royalty

Claim/Patent	NSR (%)	Owner of NSR
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

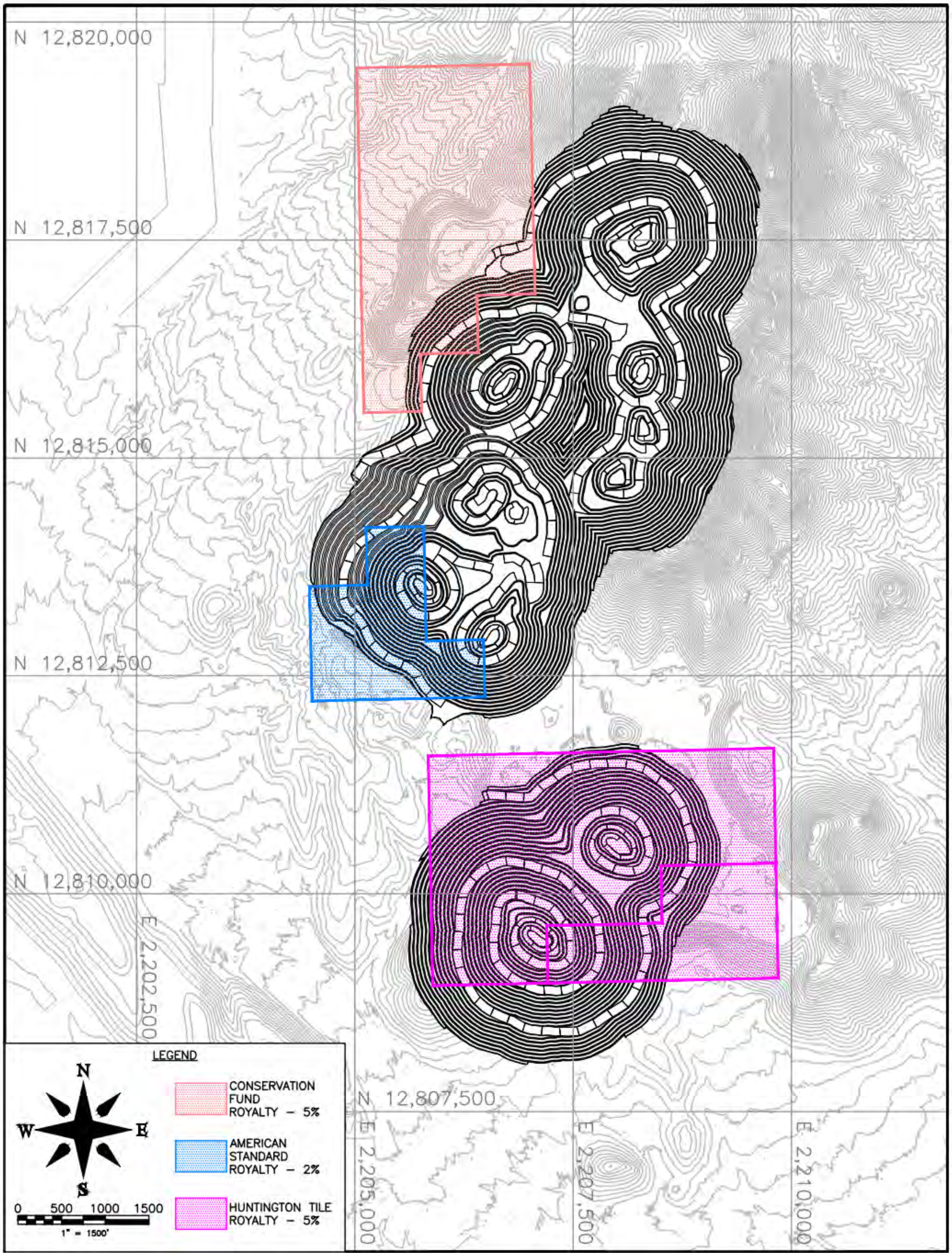

Figure 16.10 – Royalty Claims

16.7 Waste

Waste rock is place in dumps adjacent to the pits for the first 8 years of operation, and pit backfilling begins year 9. Waste is added to mined-out pits from the top. The upper surface of the backfill grows from the edge of the pit inward as more waste material is dumped into the pit.

Table 16-6 shows the backfill areas by year. The only pit without any backfill at the end of the mine life is South Domes.

Table 16-6 – Yearly Dump and Backfill Schedule in Thousands of Cubic Yards

Year	Waste from Pit	East Dump	West Dump	JSLA BF	Jumbo BF	Oro Belle BF	East Ridge BF
1	539	539	-	-	-	-	-
2	1,700	1,700	-	-	-	-	-
3	38,166	23,924	14,243	-	-	-	-
4	65,610	65,610	-	-	-	-	-
5	47,584	47,584	-	-	-	-	-
6	43,804	20,119	23,685	-	-	-	-
7	54,337	-	54,337	-	-	-	-
8	56,988	-	56,988	-	-	-	-
9	49,807	-	20,605	29,202	-	-	-
10	51,399	-	-	51,399	-	-	-
11	48,879	-	-	27,170	21,709	-	-
12	40,163	-	-	-	36,424	3,739	-
13	34,430	-	-	-	-	34,430	-
14	19,576	-	-	-	-	7,373	12,204
15	9,363	-	-	-	-	-	9,363
16	2,382	-	-	-	-	-	2,382

16.8 Mining Equipment

Mine equipment was selected to meet the requirements of the production schedule; shown in Table 16-4. Equipment was selected from the list provided by Komatsu in their fleet study. Mobile equipment unavailable from Komatsu was selected from other manufacturer’s standard units that would likely be available with operating and capital costs from InfoMine.

Most of the mining equipment will be needed starting in year 3 when prestripping begins. Prior to year 3 and continuing in the Phase 1 JSLA backfill, mining will be conducted by a contract mining company.

Production equipment purchased from Komatsu will be covered by a Maintenance and Repair Contract (MARC), and operating costs have been adjusted accordingly.

16.8.1 Drilling

Production drilling will be performed with Komatsu P&H 77XD (DTH) drills using 8.875-inch holes. The ore blasting pattern was estimated by W. A. Murphy to be 17.5’ by 20’, and the waste pattern was 22’ by 25.3’.

16.8.2 Loading

Primary loading will be performed by Komatsu PC4000 hydraulic shovels with a 30.1 cubic yard bucket. A maximum of four shovels for four active mining areas will be needed in the mine schedule. Secondary loading will be done by Komatsu WA1200-6 wheel loaders with a 32 cubic yard bucket. One secondary loader is sufficient to support primary loading.

16.8.3 Hauling

Rock will be hauled by Komatsu 730-E trucks. A maximum of 27 trucks will be needed in the mine plan (in year 11). The increase to 27 trucks in year 11 and the steady decrease of total tons mined from year 12 on provide the necessary replacements for trucks that have reached their maximum expected life through the end of the mine life. Trucks are scheduled to be replaced after 75,000 hours of operation.

Table 16-7 – Ore Haulage Productivity

Year	JSLA tph	Jumbo tph	Oro Belle tph	East Ridge tph	South Domes tph
3	905	772			
4	717	660			
5	664	523			840
6	558		536		820
7	494		527	490	711
8	436		459	477	654
9	392	420	408		605
10		362	357		553
11			403	599	510
12			396	483	448
13				606	393
14				525	350
15					302
16					261

Table 16-8 – Waste Haulage Productivity

Year	JSLA tph	Jumbo tph	Oro Belle tph	East Ridge tph	South Domes tph
3	840	914			
4	771	716			
5	621	493			777
6	575		1326	1043	866
7	457		1588	1304	636
8	361		1011	1123	512
9	326	572	571		1027
10		465	454		842
11			308	793	632
12			290	538	488
13				643	412
14				512	367
15					373
16					305

16.8.4 Support Equipment

The role of the remaining equipment is to support mining activities in the following areas:

- Road maintenance
- General site maintenance
- Loading support and cleanup
- Waste rock dump construction and maintenance
- Large production equipment, support, and maintenance
- Pit dewatering

Support equipment is shown in Table 16-9.

Table 16-9 – Support Equipment

Equipment	Description	Quantity
D735A-6	Track Dozer	6
WD600-6	Wheel Dozer	3
WA1200-6	Standard Boom Wheel Loader	1
Water Truck	Used 100-ton Haul Truck with Water Tank	2
Lube Truck	Mobile Field Lube Truck	2
Mechanics Truck	Mobile Field Mechanics Truck	4
GD655-6	14 ft blade Grader	2
PC360LCi-11	0.35 cu yd Excavator	1
416F2	Backhoe	1
Terex RT35-1	35-ton Crane	1
Light Plants	Trailer Mounted Telescoping Lighting Towers	8
Dewatering Pump	Centrifugal Pump	1
4x4 Pickup	3/4-ton automatic, extra cab, heavy duty	20

16.8.5 Manpower

Mine labor is comprised of salaried supervisory positions and technical positions and hourly laborers and operators. The mine operates three 8 hour shifts per day with 4 crews to ensure steady operation. Table 16-10 shows the hiring needs for salaried positions in Stage 1 (JSLA Backfill) and Stage 2.

Table 16-10 – Salaried Personnel

Salaried Position	Quantity Stage 1	Quantity Stage 2
Manager	1	1
Superintendent	1	2
Foreman	0	4
Maintenance Foreman	0	4
Geologist	1	4
Engineer	1	3
Survey/CAD	1	2
Sampler	2	2

Operations labor was estimated using required equipment operating hours, numbers of equipment in use, and shifts per day. Table 16-11 shows hourly labor requirements by year.

Table 16-11 – Hourly Labor

Hourly Labor Position	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Heavy equipment operator	0	0	36	56	56	60	64	64
Blasters	0	0	0	0	0	0	0	0
Mine Laborer	0	0	8	8	8	8	8	8
Drill operator	0	0	12	12	16	20	20	20
Production Truck Driver	0	0	40	68	88	64	80	104
Oilers/Mechanic	0	0	24	24	24	24	24	24
Total	**0**	**0**	**120**	**168**	**192**	**176**	**196**	**220**
Hourly Labor Position	**Year 9**	**Year 10**	**Year 11**	**Year 12**	**Year 13**	**Year 14**	**Year 15**	**Year 16**
Heavy equipment operator	60	60	56	56	52	48	40	24
Blasters	0	0	0	0	0	0	0	0
Mine Laborer	8	8	8	8	8	8	8	8
Drill operator	20	16	16	16	12	12	8	8
Production Truck Driver	88	104	108	92	92	72	52	24
Oilers/Mechanic	24	24	24	24	24	24	24	24
Total	**200**	**212**	**212**	**196**	**188**	**164**	**132**	**64**

16.8.6 Blasting and Explosives

W. A. Murphy has provided an estimate that includes delivery and loading bulk explosives. Although based on a different bench size and a constant material flow, the estimate is a sufficient cost estimate. Table 16-12 shows the cost estimate and basis from W. A. Murphy.

Table 16-12 – Explosive and Blasting Cost Estimation

Item	Ore	Waste
Hole Diameter	8.875	8.875
Bench Height	30	30
Sub drill	3	3
Stemming	14	14
Tons/yd	2.25	2.25
Powder Factor (lbs/ton)	0.483	0.304
Stripping Ratio	3.7:1	
Tons/year	10,920,000	41,080,000
Pattern	17.5' x 20'	22'x 25.3'
Holes/year	12,491	32,458
Ammonium Nitrate tons/year	2610	6781

Explosive Cost/Ton:	Cost
Non-electric initiation	$0.119
Electronic initiation	$0.122
Blasting Service Labor/Ton	$0.019
Total Cost per ton Non-electric	$0.138
Total Cost per ton Electronic	$0.141

16.8.7 Work Schedule

The work schedule assumes mine production will operate 24 hours per day, 7 days per week, 365 days per year. Operations and mining personnel will work three 8 hours per day shifts.

16.9 Pit Pioneering

Initial estimates put a cost of $500,000 for pioneering a pit.

16.10 Contractor Operations

Contractors provided three separate quotes for conducting mining and hauling operations at an ore mining rate of 15,000 tpd, with one quote assuming a stripping ratio of 1:1 ore to waste, one quote assuming a stripping ratio of 1 to 0.5 ore to waste, and one quote assuming a stripping ratio of 1:0 ore to waste.

This work will take approximately 3 ½ years to complete. The bids were for conducting all mining operations except drilling for grade control. Blasting will not be needed.

Work included in the contractor operations includes maintenance of work areas; excavation, loading, and hauling; haul road maintenance; and providing mining equipment and equipment maintenance. Exclusions included facilities and infrastructure, construction water costs, environmental, rock excavation, structural excavation, backfilling, ore grade control, mine planning, medical services, road establishment, security, camp and accommodations, engineering and surveying, and operating permits.

17.0 RECOVERY METHODS

17.1 Process Design Basis

Test work developed by KCA and Equinox Gold and carried out by McClelland Laboratories in Reno, NV has indicated that the Castle Mountain ores are amenable to cyanide leaching for the recovery of gold and silver.

The processing plan has been divided into two stages:

- Stage 1 (Years 1-3) considers processing 14,000 tons per day of ROM backfill material from the JSLA pit, where it was stored from the previous operation. Excavated backfill material will be loaded into 100-ton haul trucks and stacked in 50-ft lifts. Quicklime (CaO) will be added to the material in the trucks for pH control before the ore is stacked and leached in two stages using a dilute sodium cyanide solution. Pregnant solution discharging from the heap will flow by gravity to a pregnant solution tank from which it will be pumped to a Carbon-in-Column (CIC) adsorption circuit. Gold and silver values will be loaded onto activated carbon and then be periodically stripped from the carbon in a desorption circuit, electrowon and smelted to produce the final doré product.

- Stage 2 (Years 4+) will be constructed during Year 3 and includes expanding the Stage 1 leach pad, adsorption and desorption circuits, and adding a 2,600 ton per day crushing system and mill for high-grade ore with a Carbon-in-Leach (CIL) circuit for recovery of gold and silver. For Stage 2, ROM production from newly mined ore will increase to 42,500 tons per day for a total processing rate of 45,100 tons of ore per day.

During Stage 2, high-grade ore only will be crushed to 100% passing 3/8” at an average rate of 144 t/h in a three-stage mobile / skid mounted crushing circuit. The crushed product will be stockpiled by a fixed stacker and reclaimed at an average rate of 120 t/h using vibrating pan feeders. Reclaimed high-grade ore will be conveyed to the grinding circuit; quicklime will be metered onto the ball mill feed conveyor for pH control.

Process solution will be added to the high-grade ore in a single-stage ball mill and ground to 80% passing 100 mesh in closed circuit with hydrocyclones. The ball mill will discharge to a pump box from which a fixed portion of the ground slurry will be pumped to the vibrating screen in the gravity circuit. The remaining ground slurry will flow to the cyclone feed pump box. Oversize from the gravity screen will be returned to the ball mill feed and the screen undersize will feed the gravity concentrator.

The gravity concentration system will include a Knelson concentrator, and an intensive leach reactor system to recover metal values. Tailings from the gravity concentrator system will flow to the cyclone feed pump box and be pumped to a hydrocyclone cluster. Underflow from the hydrocyclone cluster will be returned to the ball mill feed and the overflow will pass to a trash screen before flowing to the grinding thickener feed mix box. Flocculant will be added at the thickener mix box and the ground material will be thickened to 50% solids by weight in a high capacity thickener. Thickener overflow solution will be collected in the process solution tank and pumped to the grinding circuit. Thickener underflow slurry will be pumped to a CIL circuit for leaching and adsorption of metals from solution.

The CIL circuit will have six stages with a total residence time of 36 hours. Slurry from the thickener underflow will be mixed counter-flow with activated carbon and will flow from one tank to the next through carbon interstage screens. Leached slurry leaving the last tank will be discharged as tailings to the tailings thickener feed mix box. Loaded carbon from the first tank of the CIL will be processed in the adsorption-desorption-recovery (ADR) plant shared with the ROM circuit.

Flocculant will be added to the tailings in the mix box. The tailings will then be thickened to 58% solids by weight. The thickener underflow will be pumped to the filter feed tank. Overflow from the tailings thickener will be collected in the tailings thickener overflow tank from which it will be pumped as rinse solution in the CIL, and also used as rinse solution for the tailings filters. Overflow will also be used for make-up in the process solution tank. Excess overflow will be transferred to the heap leach barren solution tank for make-up.

The agitated filter feed tank will receive tailings thickener underflow. For cyanide detoxification, a Caro's acid generator will deliver Caro's acid into the filter feed tank. Thickened detoxified tailings from the agitated filter feed tank will be pumped to two recessed plate filter presses to remove moisture. After completion of the filter cycle, the filter detoxified filter cake will discharge onto a collecting conveyor and will be conveyed to a filter cake stockpile. The filter cake will be reclaimed by a front-end loader and loaded into dump trucks and transported to the designated dry tailings disposal impoundment adjacent to the leach pad. Filtrate and cloth rinse from the filters will be pumped to the clarifier. Clarifier overflow will discharge by gravity to the tailings thickener overflow tank, and the underflow will be pumped back to the filter feed tank.

Raw water will be used for rinsing in the filtration area to minimize cyanide concentration in the filter feed.

A summary of the processing design criteria is presented in Table 17-1.

Table 17-1 - Processing Design Criteria Summary

Item	Design Criteria
Annual Tonnage Processed	
Stage 1 (Years 1-3)	5,114,000 tons
Stage 2 (Years 4+)	16,200,000 tons
Grade	
Low-grade Years 1-3	0.0100 oz/ton (0.343 g/tonne)
Low-grade Years 4+	0.0127 oz/ton (0.435 g/tonne)
High-grade Years 4+	0.0926 oz/ton (3.17 g/tonne)
Production Rate	
Stage 1	14,000 tons/day, 365 days per year
Stage 2	45,100 tons/day, 365 days per year
Processing	
Stage 1	ROM Heap Leach
Stage 2	ROM Heap Leach (low-grade - 42,500 t/d), Mill (High-grade – 2,600 t/d)
Recovery Gold	
Low-grade Years 1-3	72.4%
Low-grade Years 4+	72.4%
High-grade	94%
Recovery Silver	
Low-grade	20%
High-grade	TBD
Operation	12 hours/shift, 2shifts/day, 7 days/week, 360 days/year
Heap Leaching Cycle	
Stage 1	80 day primary, 80 day secondary
Stage 2 – ROM Ore	160 days

During Stage 1, on-site natural gas generators will be used to supply electric power to all elements of the process plant. Line power will be installed as part of the Stage 2 expansion and the Stage 1 generators will be converted to emergency backup generators.

Event ponds will be included to contain seasonal accumulations of leach solutions and/or upset conditions that cannot be managed during normal operations. The event ponds will be constructed in two stages. Event solution will be returned to the barren tank as makeup solution as soon as practical.

The simplified process flowsheets for Stages 1 and 2 are presented in Figures 17-1 and 17-2. The site general layouts for Stages 1 and 2 are presented in Figures 17-3 and 17-4.

All of the selected processes and equipment are established technologies used in gold processing plants.

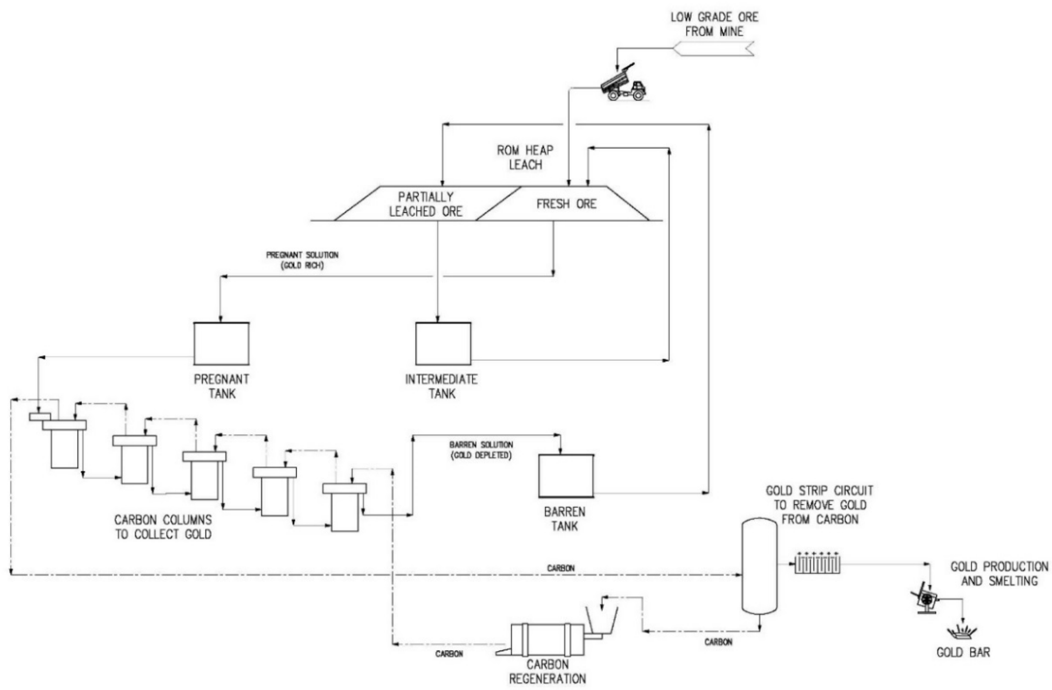

Figure 17-1 - Simplified Process Flowsheet, ROM – Stage 1

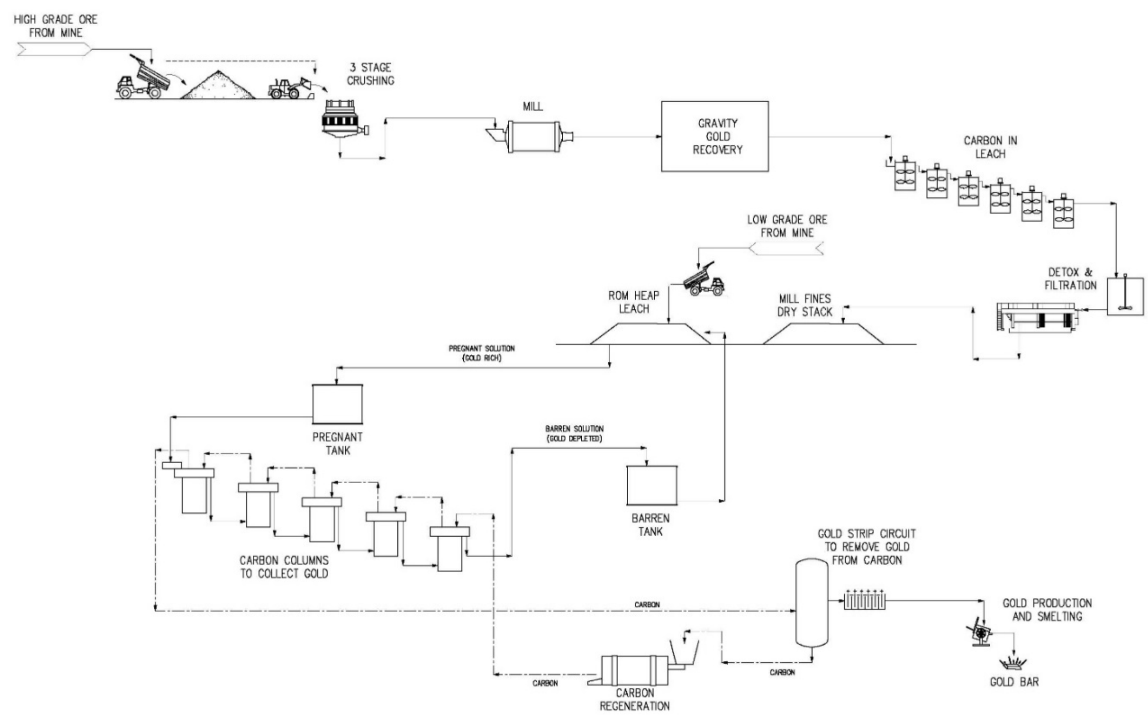

Figure 17-2 - Simplified Process Flowsheet, ROM and Mill – Stage 2

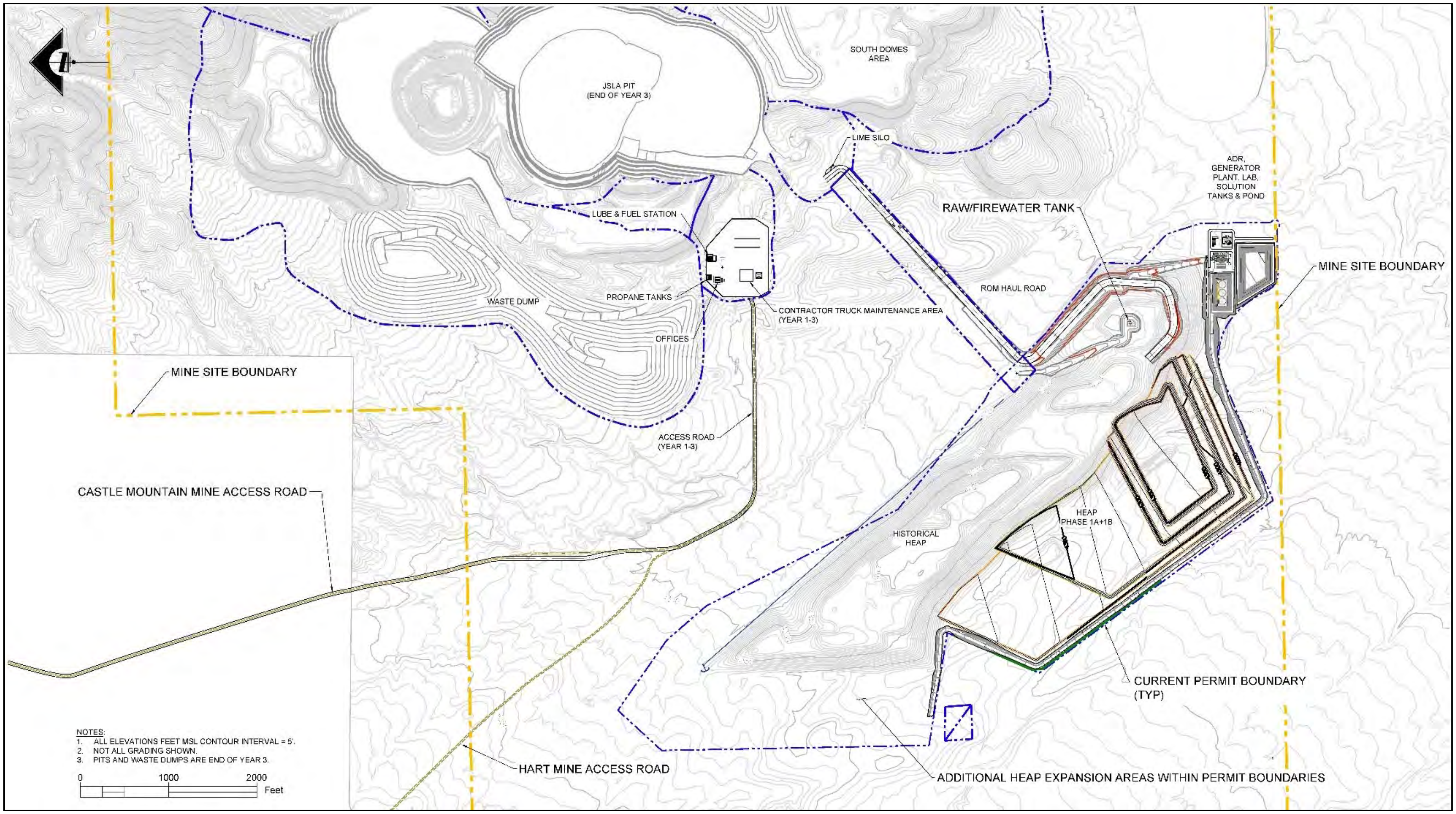

Figure 17-3 - Overall Site General Layout – Stage 1

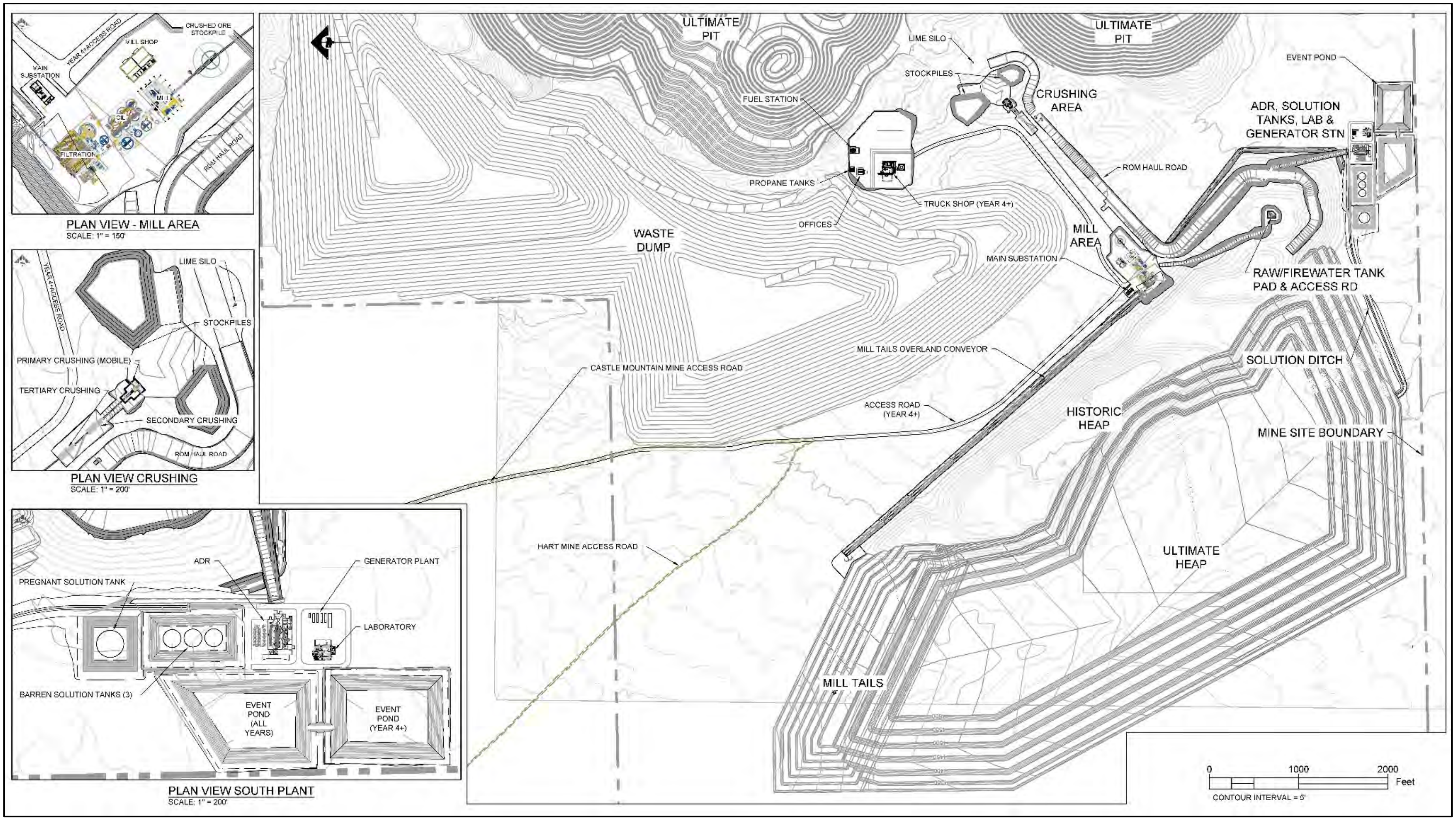

Figure 17-4 - Overall Site General Layout – Stage 2

17.2 ROM Truck Stacking

Excavation, loading, hauling and dumping of ROM material will be conducted by the mining fleet. A contractor mining fleet will be used in Stage 1 production, switching to an Owner mining fleet in Stage 2. In Stage 1, contract loaders will be used to load 100-ton haul trucks. In Stage 2, Komatsu PC4000 shovels will be used to excavate and load 200-ton haul trucks (Komatsu 730E).

Quicklime (CaO) will be used for pH control of the process with an estimated consumption of 2.35 lbs/ton based on metallurgical test work. Pebble quicklime will be stored in a 200-ton silo which will be equipped with a variable speed feeder. Once the haul trucks have been loaded, the lime will be metered directly 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.

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 50-foot high lifts with a maximum ore heap height of 150 feet during Stage 1, and eventually to 300 feet during Stage 2.

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.

The leach pad will be constructed with a protective layer of overliner material immediately on top of the LLDPE liner. The overliner will be two feet thick, and spent crushed leach ore from the existing reclaimed leach pad will be used.

Following stacking, the ore will be drip irrigated with dilute cyanide leach solution and the resulting gold-bearing solutions collected in the pregnant solution tank. The leach pad will be a multiple-lift, single-use type pad.

Stage 1 considers processing approximately 5.1 million tons per year of ROM material, specifically backfilled ore from previous operations from the JSLA pit. During Stage 2, ROM production will continue from newly mined ore and will increase to a nominal 15.3 million tons per year.

17.3 Mill (Stage 2)

For the Stage 2 expansion, a 2,600 ton per day mill will be installed to process high-grade ore. The mill includes a three-stage crushing circuit, a grinding circuit, a gravity concentration circuit, a

CIL circuit, and a detoxification and filtration circuit. Sections 17.3.1 through 17.3.7 describe these circuits in detail.

17.3.1 Crushing

The crushing circuit for high-grade ore will be constructed in Year 3 and has been designed as a three-stage crushing system with the third (tertiary) stage operating in closed circuit, with an overall availability of 75%. ROM high-grade ore will be transported from the mine in surface haul trucks to the primary crusher area. The ore will be stockpiled and fed to the primary crushing dump hopper using a CAT 966 front-end loader. The high-grade ROM stockpile has been sized to accommodate 15 days of production. The crushing plant will process nominally 144 tons of ore per hour. The crushing circuit will consist of modular units mounted on skids or trailers.

The primary crushing module will include a dump hopper, vibrating grizzly feeder, primary jaw crusher and under-crusher conveyor. Ore from the primary dump hopper will be fed to the jaw crusher via the vibrating grizzly feeder. Minus 4” undersize from the vibrating grizzly feeder will bypass the jaw crusher while plus 4” oversize material will feed into the jaw crusher. Undersize from the vibrating grizzly feeder and primary crusher discharge will fall onto the under-crusher conveyor, which in turn will discharge onto the coarse ore transfer conveyor. The primary crushed ore product size is expected to be approximately 80% passing 4.4”. The coarse ore transfer conveyor will be equipped with a cross-belt magnet, metal detector and belt scale and will discharge the material to the secondary crushing module.

The secondary crushing module will include a 300 hp standard cone crusher with a 1.0” closed side setting, which will discharge to the common under-crusher conveyor (shared with the tertiary crusher). The secondary crushed ore product size is expected to be approximately 80% passing 1.2”.

The closed-circuit tertiary crushing unit will include a double deck vibrating screen, a 300 hp short-head cone crusher, under-crusher conveyor and screen fines conveyor. The tertiary crusher discharge together with the secondary crusher discharge will be collected on the common under-crusher conveyor and will be recycled back to the tertiary screen feed conveyor via the closed-circuit transfer conveyor. The tertiary screen oversize (+3/4”) will drop into the tertiary cone crusher with a 3/8” closed side setting, which will discharge to the under-crusher conveyor. Tertiary screen undersize crushed product material will discharge onto the tertiary screen undersize conveyor, which in turn will discharge onto the tertiary crushing discharge conveyor.

The final crushed product size will be 100% passing 3/8”. The crushed ore will be stockpiled using a fixed stacker. A modular control center will be located in a container located proximal to the crushing area. All of the equipment and conveyors will be interlocked so that if one trips out, all upstream equipment will also trip. This interlocking will prevent large spills and equipment damage and is considered necessary to meet the design utilization for the system.

17.3.2 Crushed Ore Reclaim

The high-grade crushed ore will be conveyed using a fixed stacker into the high-grade crushed ore stockpile. The high-grade stockpile has been sized for 24 hours of live capacity (2,600 tons). A dozer will be used if required to maintain feed to the reclaim system.

The high-grade stockpile will have an under-pile reclaim system with a reclaim tunnel, vibrating pan feeders and a reclaim conveyor to reclaim the crushed ore from the stockpile. The high-grade stockpile will be equipped with two vibrating pan feeders (one operating, one standby) which will feed high-grade ore onto the ball mill feed conveyor at a nominal rate of 120 tons per hour. The ball mill feed conveyor will be equipped with a belt scale and sampler. Pebble quicklime for pH control will be metered onto the ball mill feed conveyor from a 30-ton storage silo at a nominal rate of 1.34 lbs/ton ore before discharging to the ball mill.

17.3.3 Grinding & Gravity Concentration

The grinding circuit has been designed as a single-stage closed circuit system with hydrocyclones to grind an average of 2,600 tons/day of crushed ore high-grade ore at a nominal rate of 120 tons per hour from 100% passing 3/8” to 80% passing 100 Mesh (150 microns), with an overall availability of 90% and a 300% recirculating load. A gravity concentration circuit will be coupled with the grinding circuit to recover the coarse gold present in the high-grade ore.

Crushed high-grade ore will be fed to the ball mill by the ball mill feed conveyor where it will be combined with process solution to make a slurry of 70% solids by weight. The slurry will discharge from the ball mill through a trommel to the gravity feed pump box. Process solution will be added to the gravity feed pump box to adjust the slurry density to 60% solids by weight. A fixed amount (150 tons per hour of solids) of the slurry from the gravity feed pump box will be pumped to the gravity concentration circuit by the gravity feed pumps (one operating, one standby). The remaining slurry will overflow from the gravity feed pump box to the cyclone feed pump box where it will be combined with gravity concentration circuit tailings and will be pumped to the hydrocyclone cluster by the cyclone feed pumps (one operating, one standby).

Slurry pumped to the gravity concentration circuit will first pass through a vibrating screen to separate the oversize and undersize material. Oversize material (+12 Mesh) will be returned to the ball mill feed and undersize material (-12 mesh) will discharge to a Knelson Concentrator. The screen will be rinsed with process solution to prevent plugging.

The Knelson Concentrator will concentrate coarse gold from the slurry. The concentrate will be periodically discharged in batches to a concentrate storage cone. The concentrate will be fed in daily batches to an intensive cyanide leach circuit. The nominal intensive leach cycle duration will be 20 hours. Leach solution will be circulated through the concentrate for a pre-set operation cycle. After leaching is complete, the leached concentrate tailings will be pumped to the cyclone feed pump box, and pregnant solution will be transferred to an electrowinning circuit for the recovery of metal values extracted in the leach circuit. The electrowinning sludge will be filtered and periodically transported to the refinery at the ADR plant.

Gravity feed pump box overflow and gravity concentration tailings will be pumped from the cyclone feed pump box by the cyclone feed pumps to the hydrocyclone cluster for classification. Oversize particles will report to the cyclone underflow at nominally 70% solids by weight and will be recycled to the ball mill feed for further size reduction. The cyclone overflow grinding product (80% passing 100 Mesh) will contain nominally 30% solids by weight. The overflow slurry will pass through a vibrating trash screen to remove any tramp material and then will discharge to the grinding thickener feed mix box.

17.3.4 Grinding Thickener & Process Solution

Flocculant will be added to the cyclone overflow slurry at a rate of 0.12 lbs flocculant per ton of ore in the thickener feed mix box before discharging to the center feed well of a 40’ diameter high rate grinding thickener. The overflow slurry will be thickened to 50% solids by weight with the underflow pumped to the CIL circuit via the thickener underflow pumps (1 operating, 1 standby), and the overflow will discharge to the process solution tank.

The process solution tank will receive grinding thickener overflow solution and make-up solution from the tailings thickener overflow. The process solution will be pumped via the process solution pumps (1 operating, 1 standby) to the grinding and gravity circuits for dilution and rinsing.

17.3.5 CIL

Grinding thickener underflow at 50% solids by weight will be pumped to the first of six identically sized CIL tanks which will operate in series. Each tank will provide six hours residence time for a total of 36 hours at the nominal rate of 120 tons per hour. The leach tanks will be fitted with axial flow impeller agitators. Compressed air will be injected into each tank using a “down shaft” aeration system. Sodium cyanide solution will be added to any of the first three tanks in series.

The CIL tanks will require air at a nominal flow of 390 scfm. This will be provided by a dedicated low-pressure compressor (1 operating, 1 warehouse spare).

The CIL tanks will be maintained at a concentration of 10 g/L of activated carbon. As soon as the gold is extracted it will be adsorbed onto the carbon. The carbon will be advanced counter-current to the slurry at a nominal advance rate of 1.23 tons of carbon per day. Each of the tanks will be equipped with an interstage screen to prevent the carbon from flowing from tank to tank with the slurry. Carbon advance will take place approximately 3 hours per day from each tank. Carbon advanced from the first CIL tank, loaded with precious metals up to approximately 5,000 grams gold and silver per tonne carbon (approximately 150 oz/ton), will be directed to the recovery screen where the slurry will be rinsed through with process solution. The rinsed carbon will be recovered as oversize into the carbon tanker truck, and the undersize slurry will be returned to the first CIL tank. The tanker truck will transport the loaded carbon from CIL to the desorption circuit.

Slurry overflowing from the last CIL tank will discharge via the carbon safety screen into the tailings thickener feed mix tank.

17.3.6 Tailings Thickener

Flocculant will be added to the CIL discharge slurry at a rate of 0.12 lbs flocculant per ton of ore in the thickener feed mix box before being discharged to the center feed well of a 40’ diameter high rate tailings thickener. The CIL discharge slurry will be thickened to 58% solids by weight with the underflow pumped via the thickener underflow pumps (1 operating, 1 standby) to the agitated filter feed tank. The thickener overflow will discharge to the tailings thickener overflow tank. Tailings thickener overflow will be pumped via the tailings thickener overflow pumps (1 operating, 1 standby) to the CIL circuit for rinse solution. Tailings thickener overflow will also be used as make-up to the process solution tank. All excess tailings thickener overflow will be transferred to the heap leach barren solution tank for make-up.

17.3.7 Tailings Filtration, Cyanide Detoxification & Clarification

Tailings underflow slurry will be pumped to the agitated filter feed tank, which has been sized for eight hours of retention (half full) at the nominal flow. A Caro’s acid (H_2SO_5) generator will be used to prepare Caro’s acid and deliver it directly to the filter feed tank. The detoxified slurry will be fed to the tailings filters by the filter feed pumps to remove most of the remaining moisture from the thickened tailings slurry. Each filter will have a dedicated filter feed pump. The filtration circuit consists of two recessed plate filter presses each with 51 filter plates which have been sized to produce a final filter cake that will be at least 85% solids by weight. Each filter cycle will require approximately 12 minutes including 3 minutes for filling, 3 minutes for cake squeeze and air blow, 1 minute for cake discharge and 5 minutes of technical time. After completion of a filter cycle, the plates will be opened and the cake will be discharged onto the filter discharge conveyor (one per filter) which will feed the filter cake onto the filter cake collection conveyor. The filter discharge conveyor speed will be controlled in order to maintain an even feed on the filter cake collection conveyor. After discharging the filter cake, the filter cloths will be rinsed using raw water via the rinse pumps (1 operating, 1 standby) after every filter cycle. The filter cloths will be washed with raw water using a high-pressure cloth wash system once per day or as needed.

Filter cake on the filter cake collection conveyor will be sampled and conveyed via an overland conveyor and a radial stacker to a filter cake stockpile located adjacent to a dedicated tailings impoundment area. From the stockpile a front-end loader will load 40-ton articulated dump trucks to deliver the tailings to the tailings impoundment area, which is located adjacent to the heap leach pad.

The filtrate and cloth rinse will flow to the clarifier feed tank where it will be pumped via the clarifier feed pump (1 operating, 1 standby) to the clarifier to remove any suspended solids. A small amount of flocculant will be added at the clarifier feed mix box. Overflow from the clarifier will discharge to the tailings thickener. Clarifier underflow will be returned to the filter feed tank using the clarifier underflow pump (1 operating, 1 warehouse spare).

17.3.8 Tailings Impoundment

The tailings impoundment area will be constructed using a double liner system to prevent release of any seepage from the mill tailings to the environment. The liner system will be constructed the same as with the ROM heap leach facility, consisting of two layers of 80-mil linear low-density polyethylene (LLDPE) geomembrane with a 2-foot thick layer of drainage gravel between them.

In summary, the tailings impoundment will be composed of the following lining system from top to bottom:

- 2 feet (minimum) gravel overliner (previously leached and rinsed ore).
- 80-mil LLDPE geomembrane liner (textured side face down).
- 2 feet leak detection drainage gravel (previously leached and rinsed ore).
- 80-mil LLDPE geomembrane liner (textured side face down).
- Prepared subgrade.

The gravel overliner will act as a protective layer that resides above the LLDPE geomembrane. The main purpose of this material will be to protect the geomembrane layer and collection piping from damage during ore placement.

The mill tailings will be placed on 20-feet thick lifts, to reach an ultimate height of approximately 140 feet above the overliner. The overall slope of the mill tailings dry-stack will remain within a 4H:1V envelope. A layer of gravel (previously leached ore) will be placed on top of each lift, to provide lateral drainage and relief pore pressure build out, as well as to provide a better surface for truck traffic during operation.

The impoundment facility for stacking mill tailings will be constructed in multiple phases, as shown in Table 17-3.

Table 17-2 - ROM Heap Leach Construction Phases

Construction Phase	Area (acre)	Tailings Capacity (million tons)	Construction Year
Phase 1	45	4.9	3
Phase 2	22	3.3	7
Phase 3	29	4.6	10
Total	96	12.7	

In the later years of production, ROM ore will begin to be placed over the boundary of the tailings impoundment areas, against the sideslope of previously placed tailings. This approach will minimize the ultimate area of the heap leach facility that will need to be constructed for ROM ore placement. Where this overlap is planned, a single liner system will be installed over the mill tailings sideslope (and underneath the ROM ore to be stacked), consisting from bottom to top, of a 2-foot gravel layer, an 80-mil LLDPE geomembrane, and a 2-foot gravel overliner, such that all active leach solutions applied to the ROM ore in this area will be directed to the heap leach facility solution collection system and not into the tailings impoundment area.

Gravity solution collection pipes will be placed over the tailings liner and will consist of 4-inch diameter double-walled corrugated polyethylene (CPE) lateral pipes spaced 16 feet apart, flowing into larger double walled CPE header pipes up to 18-inch in diameter. These collection pipes will direct any seepage from the tailings over to the heap leach facility solution collection system.

The leak detection gravel layer in the tailings impoundment area is to detect, monitor, and control solution in case the primary liner is damaged during construction or operation. A network of pipes will be used to convey the liquids within the leak detection layer to a collection and recovery sump, where the solution can be pumped out, sampled and recirculated into the system as necessary.

17.4 Heap Leaching Solution Handling & Storage

Following truck stacking, the ROM ore will be irrigated with barren leach solution and the resulting gold-bearing solutions will be collected into a pregnant solution tank. During Stage 1, ROM ore will be leached in two stages with recycle of intermediate solutions back to the heap leach. Barren solution from the barren solution tank will be used to irrigate partially leached ore and the intermediate solution will be collected in the intermediate solution tank. Intermediate solution will then be used to irrigate fresh ROM ore with the resulting pregnant solution draining to the pregnant solution tank. During Stage 2, ROM ore will be leached in a single stage with all pregnant solution reporting to a newly constructed pregnant solution tank. The pregnant, intermediate, and barren tanks constructed during Stage 1 will be interconnected using large pipes and repurposed as a single common barren solution tank. Barren solution from the combined barren solution tank will be used to irrigate ROM ore.

The heap will be irrigated using drip tubes spaced at three-foot intervals. Reusable Yelomine pipes will be used to distribute the solution to the drip tubes on top of the heap. Antiscalant will be added at the barren and pregnant solution tanks to reduce the potential for scaling problems within the system.

The total leach cycle for the ROM ore is estimated to be 160 days. During Stage 1 the leach cycle will include an 80-day primary and an 80-day secondary leach to concentrate pregnant solution values and reduce the size of the Stage 1 adsorption and recovery plant requirements. During Stage 2, ROM ore will be leached in a single 160-day stage. Estimated leach cycles are based upon metallurgical test work with appropriate field adjustments made.

Leach solutions will be applied to the ore at a nominal application rate of 0.004 gpm/ft^2 with an approximate maximum cyanide concentration of 250 ppm to the heap. Vertical turbine pumps mounted on the barren tank(s) (or intermediate solution tank) will be used for solution application to the heap leach. High-strength cyanide and an antiscalant will be added to the tanks by metering pumps. During Stage 1 the nominal flow for barren and intermediate solution to the ROM ore is estimated at 1,792 gpm each, which will be supplied by vertical turbine pumps at each tank (one operating, one standby). The nominal barren solution flow to the ROM ore during Stage 2 is estimated at 10,880 gpm which will be provided by new vertical turbine pumps installed at the combined barren tank (three operating, one standby).

Header pipes from the barren pumps will supply the solution to the active irrigation areas on the leach pad. Valved tees at the header will supply leach solution to re-usable sub-header pipes. Distribution pipes connected to the sub-header pipes will feed networks of drip tubes that will distribute solution to the top of the stacked ore.

Gold bearing solutions draining from the leach pad will be collected at the bottom of the heap by a network of perforated drainage pipes and directed to the pregnant (or intermediate) solution tank.

Pregnant solution will be pumped from the pregnant solution tank to the CIC circuit. Intermediate solution during Stage 1 will be pumped to the leach pad to irrigate fresh ore.

During Stage 1 the heap leach will process approximately 5.1 million tons of ROM ore annually. During Stage 2, the heap will process approximately 15.5 million total tons per year.

The barren, intermediate and pregnant solution tanks constructed during Stage 1 will each have a volume of 85,000 ft^3. During Stage 2 these tanks will be combined using pipes for a single integrated tank with a volume of 255,000 ft^3, which will be used as the barren solution storage for Stage 2. A new 255,000 ft^3 pregnant solution tank will be constructed for storage of pregnant solution during Stage 2.

The solution storage tanks and event pond have been sized to ensure that all the leach solutions can be managed in a controlled manner.

17.5 Heap Leach Facility

The heap leach facility for stacking ROM ore will be constructed in multiple phases, as shown in Table 17-3.

Table 17-3 - ROM Heap Leach Construction Phases

Construction Phase	Area (acre)	ROM Capacity (million tons)	Construction Year
Phase 1A	70	10.7	Pre-Production
Phase 1B	88	26.9	Year 2
Phase 2	113	61.2	Year 4
Phase 3	100	56.5	Year 7
Phase 4	85	46.7	Year 10
Total	456	207.9	

Excess storm water falling within the leach pad area will be handled through the solution collection system, and routed to an Event pond. By the end of Year 3 of operation, the ore production will increase from 5.1 million tons per year to 15.5 million tons per year. At this stage, Phase 1A will be completed and Phase 1B will be partially filled and active. During Year 3 a second event pond will be constructed to increase the capacity of the event pond at the beginning of Year 4.

Based on the current permit boundaries, the maximum leach pad capacity is 37.6 million tons, equivalent to approximately 4.4 years of operation. Equinox Gold will apply to increase the permitted area in order to accommodate the full 208 million tons of ROM ore for the project, which is required for completion of Stage 2 operations.

17.5.1 Heap Leach Pad Construction

The heap leach facility will be constructed using a double liner system to prevent release of process solutions to the environment. The liner system consists of two layers of 80-mil linear low-density polyethylene (LLDPE) geomembrane with a 2-foot thick layer of drainage gravel between them.

In summary, the leach pad liner will be composed of the following lining system from top to bottom:

- 2 feet (minimum) gravel overliner (previously leached and rinsed ore).
- 80-mil LLDPE geomembrane liner.
- 2 feet leak detection drainage gravel (previously leached and rinsed ore).
- 80-mil LLDPE geomembrane liner.
- Prepared subgrade.

The ore cover will act as a protective layer that resides above the LLDPE geomembrane. The main purpose of this material will be to protect the geomembrane layer and collection piping from damage during ore placement.

Gravity solution collection pipes on the leach pad liner will consist of 4-inch diameter double-walled corrugated polyethylene (CPE) lateral pipes spaced 16 feet apart flowing into larger double walled CPE header pipes up to 18-inch in diameter. The whole pad is divided into smaller collection subareas, each one draining to a dedicated header pipe. Two-foot high separation berms are used to define the collection subareas and a 5-feet high perimeter berm encloses the total area of each construction phase.

Each header pipe runs from northwest to southeast, ending at the main solution channel located along the southern edge of the pad. Each header pipe is connected to two solid 32-inch diameter high-density polyethylene (HDPE) pipes through a distribution manifold equipped with valves. The HDPE pipes or main pregnant leach solution / intermediate leach solution (PLS/ILS) pipes will carry the pregnant solution collected from the pad to the solution tanks.

The leak detection gravel layer is to detect, monitor, and control solution in case the primary liner is damaged during construction or operation. A network of pipes will be used to convey the liquids within the leak detection layer to a collection and recovery sump, where the solution can be pumped out, sampled and recirculated into the system as necessary.

17.5.2 Event Pond

The project considers two event ponds interconnected with a spillway that will be constructed at different stages of the operation. The event pond #1 will be constructed as part of the initial construction along with the leach pad Phase 1A. This pond has a designed capacity of about 19.0 million gallons, and has been sized to accommodate the 24 hour heap drain down plus the 100 year 24 hour storm even over the full lined area of Phase 1A. The event pond #2 will be constructed as part of the construction of the leach pad Phase 1B. This pond has a designed capacity of about 20.3 million gallons, and the entire complex (event pond #1 and event pond #2) has been sized to accommodate in conjunction the 24 hour heap drain down plus the 100 year 24 hour storm even over the full lined area.

Each event pond will have a composite lining system with a leak detection system and will be composed of the following from top to bottom:

- 80-mil textured HDPE geomembrane liner.
- HDPE geonet leak detection layer.
- 60-mil textured HDPE geomembrane liner.
- Prepared subgrade.

Leak detection will be utilized in both event ponds. Leak detection riser pipes will be provided beneath the primary and secondary liners of the pond and will allow for monitoring and pumping of solutions from within the leak detection sumps if the liner is found to be leaking.

The leak detection risers will include the use of a 12-inch diameter SDR-32 HDPE pipe that will run down the slope of the pond into a sump between the primary and secondary liners. The sump will be filled with drain gravel that will be free-draining and porous in nature. The 12-inch diameter pipe will be perforated within the sump to accumulate fluids, if they exist in the sump.

The non-perforated portion of the pipe will boot through the anchor trench and terminate at the crest of the pond.

17.6 Solution Management

Solution management for Castle Mountain has been determined by GLA based on a monthly basis water balance model which approximates the circulation of fluids within the heap, solution tanks and event pond(s) as well as the introduction of daily precipitation and evaporation as a function of time. The heap leach facility and processing facilities have been designed as zero discharge facilities to both surface and ground water; therefore, the liner was modeled as a no flow boundary.

17.6.1 Pregnant and Barren Solutions

Pregnant solution will be collected in a pregnant solution tank. The pregnant solution will be pumped to the CIC circuit and then will be returned to the barren solution tanks as barren solution. Barren solution from the barren solution tank will be used to irrigate the leach pads as well as solution for miscellaneous process uses. Make-up water for the process will be added primarily at the barren solution tank as needed.

Solution management for the system will be generally simple. The pregnant and barren solution tanks will be operated approximately half full to allow for some surge capacity within the tanks. The event pond is intended for emergency solution storage only, and will be kept empty or at low levels whenever possible. When solution is diverted to the event pond, it should be pumped back to the leach system as soon as practical. Every effort should be made to avoid storing solution in the event pond over an extended period of time.

Make-up raw water for the system will be supplied from water wells around the project site as required. The water supply and distribution system is described in further detail in Section 18.

17.6.2 Event Pond

The fluid management system is intended to operate as a zero-discharge system. It will include provisions to accommodate upset conditions such as severe storms and temporary loss of electric power or pumps. During upset conditions, the solution volume in the pregnant tanks may exceed the available storage, causing the solution to overflow to the event pond. The required event pond storage volumes have been estimated based on the 24 hour heap drain down and 100–year, 24-hour storm event over the lined area. The event pond will be constructed in two Phases by constructing an additional pond connected by an overflow channel. The event pond #1 has been sized to handle solutions collected over the Phase 1A of the leach pad. The additional event pond expansion will be constructed concurrently with the leach pad expansion for Phase 1B.

Table 17-4 shows the approximate event pond volumes.

Table 17-4 - Event Pond Storage Capacity

Description	Capacity (gal)
Event Pond #1	18,985,558
Event Pond #2	20,294,350

17.7 Process Water Balance

A monthly water balance model has been developed for the heap leach operation including pad construction phases 1A, 1B, 2, 3, and 4. The model was used to evaluate the annual heap leach facility performance, evaluate the event pond complex utilization, and estimate average make-up water flowrates. The water balance simulation model is incremented on a monthly basis to reflect seasonal variations in precipitation and evaporation. The model has been developed using an Excel spreadsheet, with versions for low, average, and high precipitation years.

The model considers the discrete increments of lined area as the leach pad phases are constructed throughout the life of the project, the increment of areas covered with fresh ore, and the areas under irrigation.

The water balance model is based upon the leach facility operating for 365 days a year. Inflow to the system consists of precipitation on all lined areas including all leach pad areas and collection ponds, applied leach solution, and moisture in the ore heap as stacked on the pad. Outflow from the system consists of evaporation from the lined areas and heaps, and solution flow to the collection ponds. Moisture consumed in taking the ore heap from the as-stacked condition to its field capacity is included as a loss.

17.7.1 Climate Data

Water balance computations included different precipitation datasets, including considerations for above and below average precipitation to estimate the system accumulation of process solution and the makeup water required. The wet year reflects the most water likely to affect the system, and the dry year will indicate the most makeup water needed. For this study, the main objective is to confirm that the capacity of the existing pond is sufficient.

The monthly rainfall data for the site was estimated from two sets of precipitation data obtained from two nearby stations: Mountain Pass (045890), and Searchlight (267369). Mountain Pass station is located approximately 26 miles northwest from the mine site. Searchlight station is located approximately 18 miles northeast from the mine site. The site rainfall data was obtained utilizing the arithmetic average method, determining the precipitation profile for the average, the driest and wettest years, as shown on Table 17-5.

Table 17-5 - Rainfall data estimated for the site

Month	Mean (in)	Wet year (in)	Dry year (in)
January	0.92	2.15	0.39
February	0.93	1.92	0.00
March	0.83	2.35	0.00
April	0.44	0.80	0.12
May	0.24	0.00	0.00
June	0.16	0.00	0.00
July	0.98	1.21	0.28
August	1.16	4.11	0.04
September	0.60	0.69	0.04
October	0.53	0.94	0.77
November	0.56	2.81	0.00
December	0.71	1.67	0.00

Monthly average pan evaporation data was obtained from Boulder city station, which is located approximately 50 miles northeast from the mine site, as shown on table 17-4. For the purposes of this model, monthly pan evaporation data was not varied for wet and dry years in the above cases. Instead, the evaporation from the heap is limited to the lesser of the amount of precipitation that has fallen in that period, or the factored pan evaporation. Pan evaporation assumes water is available for evaporation, but in dry months only the amount of precipitation is available.

Table 17-6 - Pan Evaporation data

Month	inches
January	3.71
February	4.68
March	7.56
April	10.67
May	13.79
June	16.57
July	16.45
August	14.41
September	11.51
October	8.11
November	4.87
December	3.69

The water balance model does not account for other climatological factors such as wind speed, humidity and cloud cover.

17.7.2 Water Balance

The model shows that the normal operation of the facility is characterized by a regular deficit on the water balance manly due to the climate characteristics such as high evaporation and low rainfall throughout the years. During dry to average years, the event pond #1 is expected to be utilized at a very minimum with low chance of accumulating water, and therefore, the event pond #2 will not be utilized. During wet years some accumulation of water is expected into the event pond #1 during the wettest months.

Water deficit will be balanced with the addition of make-up raw water supplied from nearby water wells. The flowrate requirement for the heap leach will vary month to month depending mainly on the climate patterns, with average flowrates ranging from 250 gpm during the first stage of operation to 685 gpm for the latter stage on dry years. Table 17-7 shows the summary of estimated average and maximum flowrates required for make-up water for each of the three scenarios modeled. Note these make-up requirements are for the heap leach only, and do not include other requirements such as road dust suppression, and miscellaneous building and infrastructure use.

Table 17-7 - Estimated flowrates for make-up water

		Average year (gpm)	Driest year (gpm)	Wettest year (gpm)
Phase 1A	Average	186	211	145
	Maximum	242	252	252
Phase 1B	Average	424	474	335
	Maximum	742	764	764
Phase 2+	Average	567	685	371
	Maximum	729	766	766

17.8 Recovery Plant

The recovery plant at Castle Mountain has been designed to recover gold and silver values using an adsorption-desorption-recovery (ADR) process and will be constructed in Stage 1 of operations, with an expansion of some of the circuits in Stage 2 to accommodate an increase in gold and silver production.

Pregnant leach solution from the heap leach will be pumped to the carbon in column circuit (CIC) and adsorbed onto activated carbon (adsorption). Loaded carbon from the CIC circuit (and from the CIL circuit) will then be desorbed in a high-temperature elution process coupled to an electrowinning circuit (desorption), followed by retorting to remove mercury and smelting of the resulting sludge to produce doré bullion (recovery). After elution, each batch of carbon will be acid washed to remove any scale and other inorganic contaminants that might inhibit gold adsorption on carbon and a portion of the carbon will be thermally reactivated using a rotary kiln.

17.8.1 Adsorption

Adsorption of gold and silver onto carbon will occur in the carbon adsorption circuit which will be constructed in two stages.

The Stage 1 (Years 1-3) adsorption circuit will consist of one train of five, cascade type open-top up-flow mild-steel carbon adsorption columns (CICs). Each of the carbon columns will be sized to hold 3.5 tons of carbon.

Pregnant solution from the pregnant solution tank will be pumped to the adsorption circuit at a nominal rate of 1,600 gpm. Barren solution exiting the last carbon adsorption column in the train will flow through a static screen to separate any floating carbon from the solution, then flow by gravity into the barren tank.

Stage 2 of the adsorption circuit will be constructed during Year 3 (for Year 4 operation) and will include the addition of three new trains of five, cascade type open-top up-flow mild steel adsorption columns, with each column each sized to hold 7 tons of carbon. The three additional column trains will be operated in parallel with the existing column train constructed in Stage 1. Pregnant solution from the newly constructed Stage 2 pregnant solution tank will be pumped to the adsorption circuit at a rate of 2,900 gpm for the three new column trains and 1,600 gpm for the Stage 1 column train, for a total of approximately 10,300 gpm. Each train will be supplied by a separate pump from the pregnant solution tank. Each column train will be equipped with a static safety screen to separate any floating carbon from barren solution leaving the columns, then the solution will flow by gravity back to the barren solution tank (which will be the combined and repurposed Stage 1 pregnant, intermediate and barren tanks).

During Stage 1 and Stage 2, antiscalant will be added at the pregnant solution tank to prevent scaling of carbon and reduction of the carbon loading capability. Magnetic flowmeters equipped with totalizers will measure solution flow to the adsorption circuit. Pregnant solution will flow by gravity through each set of five columns in series, exiting the lowest column as barren solution. Pregnant and barren solution continuous samplers will be installed at the feed and discharge end of each carbon column train, respectively. Solution samples will be used to measure pregnant and barren solution gold concentrations.

Adsorption of gold and silver from pregnant leach solutions from the heap circuit will be a continuous process. Once the carbon in the lead column achieves the desired precious metal load it will be advanced to the elution (desorption) circuit using screw type centrifugal pumps. Carbon in the remaining columns will be advanced counter current to the solution flow to the next column in series. New or acid washed/regenerated carbon will be added to the last column in the train.

Generally, the stripping of carbon will occur each day.

17.8.2 Desorption

Castle Mountain will use a pressure Zadra hot caustic desorption circuit for the stripping of metal values from carbon, which requires 18 hours or less to complete a cycle. During the elution cycle, gold and sliver are continuously extracted by electrowinning from the pregnant eluate concurrently with desorption.

The desorption circuit at the Castle Mountain ADR plant will be constructed in two stages. Stage 1 (Years 1-3) will include a single elution circuit sized to process 3.5 tons of loaded carbon from the adsorption circuit. A second identical 3.5-ton circuit will be added in Year 3 which will operate in parallel with the Stage 1 circuit.

Each desorption circuit will be equipped with a strip solution tank, strip solution pump, primary (heat up), secondary (heat recovery) and tertiary (cooling) heat exchangers, hot water heater, elution column and elution column drain.

In Stage 1, loaded carbon will be transferred batchwise from the lead carbon column to the elution column. In Stage 2, carbon will be transferred both from the carbon columns and from the CIL circuit to one of the two elution columns.

After carbon has been transferred to the elution column, barren strip solution (eluant) containing sodium hydroxide and sodium cyanide will be pumped through the heat recovery and primary heat exchangers, and introduced to the elution vessel at a nominal temperature of 275°F and a nominal operating pressure of approximately 65 psig. The final stripped-carbon gold and silver content will be typically less than six troy ounces per ton of carbon.

Under normal operating conditions, barren eluant solution from the solution storage tank will pass through the heat recovery exchanger to be preheated by hot pregnant eluate leaving the elution column. The barren eluant solution then passes through the primary heat exchanger to raise the temperature up to 275°F using pressurized hot water (~360°F) from the hot water heater system.

The elution column will contain internal stainless-steel inlet screens to hold carbon in the column and to distribute incoming stripping solution evenly in the column. Pregnant eluate leaving the elution column will pass through two external stainless steel screens before passing through the heat recovery exchanger and the cooling heat exchanger to reduce the temperature to about 90°F (to prevent boiling). The cooled pregnant eluate solution will flow to the electrowinning cells.

After desorption is complete, the stripped carbon will be transferred to the carbon acid wash circuit using the "pressure potting" method, where the elution column will be filled and slightly pressurized with process solution to facilitate carbon transfer. Acid washing will be performed after every strip.

17.8.3 Electrowinning

The electrowinning circuit will be operated in series with the elution circuit. Solution will be pumped continuously from the barren strip solution tank through the elution column, then through the electrowinning cells, and back to the strip solution tank in a continuous closed loop process.

The Stage 1 electrowinning circuit will include two 70 ft^3 electrowinning cells, each equipped with a rectifier. Two additional, identical electrowinning cells (plus associated ancillary equipment) will be added in Stage 2 as part of the Stage 2 elution circuit expansion.

The gold and silver-laden solution exiting the elution column will be filtered to trap any carbon escaping from the column; will pass through the heat recovery exchanger and the cooling exchanger to reduce the solution temperature to 90ºF, then will flow to the electrowinning circuit.

Gold and silver will be won from the eluate in the electrowinning cells using stainless steel cathodes using a current density of approximately 5 amperes per square foot of anode surface. Caustic soda (sodium hydroxide) in the eluate solution will act as an electrolyte to encourage free flow of electrons and promote the precious metal winning from solution. To keep the electrical resistance of the solution low during desorption and the electrowinning cycle, make-up caustic soda will sometimes be added to the strip solution tank. Barren eluant solution leaving the electrolytic cells will discharge to the E-cell discharge tank from which it will be pumped back to the eluate storage tank for recycle through the elution column.

Periodically, all or part of the barren eluant will be dumped to the barren solution tank. Typically, about one-third of the barren eluant will be discarded after each elution or strip cycle. Sodium hydroxide and sodium cyanide will be added as required from the reagent handling systems to the barren eluant tank during fresh strip solution make-up.

The precious metal-laden cathodes in the electrolytic cells will be removed about once per week and processed to produce the final doré product. Loaded cathodes will be transferred to a cathode wash box where precipitated precious metals will be removed from the cathodes with a pressure washer. The resulting sludge will be pumped to a plate-and-frame filter press to remove water and the filter cake will be loaded into pans for retorting. After mercury has been removed in the retort and, the dried gold sludge will be mixed with fluxes and smelted in a natural gas-fired furnace to produce doré bullion.

In Stage 2, filtered sludge will be produced both by the electrowinning cells at the ADR plant and by the electrowinning cell at the gravity/intensive leach circuit, for retorting and smelting.

17.8.4 Carbon Acid Wash

Acid washing will consist of circulating a dilute acid solution through the bed of carbon to dissolve and remove scale from the carbon. Acid washing will be performed on a batch basis after every desorption cycle.

A single 3.5-ton acid wash circuit will be constructed during Stage 1. During Stage 2, an identical 3.5-ton acid wash circuit will be added to match the 3.5-ton elution circuit also being added in Stage 2.

After carbon has been transferred into the acid wash column, but before any acid is introduced, fresh water will be circulated through the bed of carbon to remove any entrained caustic/cyanide solution. The rinse solution will be pumped to the carbon safety screen using the acid wash circulation pump. A dilute acid solution will then prepared in the mix tank, and circulation established between the acid wash vessel and the acid mix tank. Concentrated acid will be injected into the recycle stream to achieve and maintain a pH ranging from 1.0 to 2.0. Completion of the cycle will be indicated when the pH stabilizes between 1.0 and 2.0 without acid addition for a minimum of one full hour of circulation.

After acid washing has been completed, the acid wash pump will pump spent acid solution from the acid mix tank and wash vessel to the carbon safety screen. The carbon will then be rinsed with raw water followed by rinsing with dilute caustic solution to remove any residual acid. Total time required for acid washing a batch of carbon will be four to six hours. After acid washing has been completed, a carbon transfer pump will transfer the carbon to either the carbon storage tank or the carbon regeneration circuit.

17.8.5 Carbon Handling and Regeneration

The carbon handling and regeneration circuit will include all equipment required to store, prepare, transfer and regenerate carbon and will be constructed during Stage 1 to handle the ultimate carbon regeneration capacity required in Stage 2.

The carbon preparation and storage system will include a one ton agitated carbon attritioning tank, a 7.5-ton carbon storage tank, carbon dewatering screen, carbon fines storage tank, carbon fines filter press and carbon transfer pumps. New and acid washed/regenerated carbon will be stored in the carbon storage tank to be returned to the CIC circuits or CIL circuit as makeup carbon. Carbon being transferred to the carbon storage tank will pass to a carbon fines/dewatering screen in order to remove any carbon fines from the system. Carbon fines will stored in a carbon fines storage tank, which will be periodically pumped through the carbon fines filter press; carbon fines from the filter press will be stored in bulk bags for removal from the system.

New carbon being added to the system will first be attritioned in the carbon attritioning tank before being pumped to the carbon dewatering screen to remove carbon fines and then being transferred to the carbon storage tank.

Thermal regeneration will consist of drying the carbon thoroughly and heating it to approximately 1400ºF for ten minutes in order to maintain carbon activity levels. During Stage 1, 100% of the carbon will be regenerated after every elution cycle. During Stage 2, the regeneration circuit will have the capacity to regenerate approximately one third of the carbon from each elution cycle.

Carbon from the acid wash circuit to be thermally reactivated will be dewatered on a static screen, transferred to the regeneration kiln feed hopper and fed to the regeneration kiln by a screw feeder. Hot, regenerated carbon leaving the kiln will pass into a water-filled quench tank for cooling before being transferred to the carbon dewatering screen and carbon storage tank. Ultimately, quenched regenerated carbon will be pumped to the CIC tanks or transported to the CIL circuit to be loaded with precious metals.

17.8.6 Refining, Smelting and Mercury Control

Cathode sludge from the filter press will be dried and treated in a mercury retort to remove and recover any mercury that may be present. The sludge will be placed into pans and heated in the retort for a minimum of 6 hours at 900ºF to volatilize the mercury. A vacuum system will remove mercury vapor from the retort and pass the vapor through a water-cooled mercury condenser. Condensed mercury will be collected in a trap, and then transferred and stored in flasks. Cooled, mercury-depleted vapor leaving the trap will be passed through a sulfur-impregnated carbon scrubber to remove any residual mercury.

After mercury removal, fluxes will be mixed with the cathode sludge and then fed to a tilting natural gas-fired furnace. After melting, slag will be poured off into cast iron molds until the remaining molten furnace charge will be mostly molten metal (doré). Doré will poured off into bar molds, cooled, cleaned, and stored in a vault pending shipment to a third-party refiner. The Doré poured from the furnace will represent the final product of the processing circuit.

Periodically, slag produced from the smelting operation will be re-smelted on a batch basis to recover residual metal values. Alternatively, slag can be returned to the grinding circuit.

The mercury retort and smelting furnace will be sized and installed in Stage 1 to treat the ultimate required capacity of Stage 2 of the project.

17.8.7 Mercury Abatement System

In addition to the mercury retort, the ADR facility will be fitted with an exhaust gas handling system to treat mercury emissions from the various pieces of equipment. The exhaust system will be designed to combine mercury-containing exhaust streams and treat them in two separate sulfur-impregnated carbon beds prior to discharge to the atmosphere.

The exhaust gas handling system will be sized and installed in Stage 1 to treat the ultimate required capacity and controls of Stage 2 of the project.

The first carbon bed will be dedicated to treat fumes from the smelting furnace. The smelting furnace will be fitted with a hood which will collect fumes and direct them to a scrubber, which will remove suspended particles from the gas and cool the gas before passing through the carbon bed. The carbon bed will collect traces of mercury vapor before exhausting the gas to atmosphere.

The second carbon bed will treat the combined exhaust gas streams from the electrowinning cells, eluant solution storage tank, elution vessel, and carbon regeneration kiln. The kiln exhaust gas will be first treated through a wet scrubber to remove particulates and cool the gas, which will then be combined with the remaining exhaust gas streams and pass through the carbon bed.

17.9 Process Reagents and Consumables

The reagent handling system will include equipment used to mix and/or store all reagents required for the Castle Mountain process. Reagent mixing and storage will be at ambient temperature and pressure.

Average estimated annual reagent and consumable consumption quantities for the processing area are shown in Tables 17-8 and 17-9.

Table 17-8 - Projected Annual Reagents and Consumables, Stage 1 (Year 1-3)

Item	Form	Annual Usage	On Site Storage
Sodium Cyanide	Liquid Delivery, 30% NaCN by Weight	898 t NaCN solid	23,490 gallons
Lime (Pebble CaO)	Bulk Delivery Trucks	6,004 t	200 t
Activated Carbon	1100 lb kg Supersacks	11.2 t	2 t
Sodium Hydroxide	Liquid Delivery, 50% Caustic by Weight	11 t NaOH solid	5,000 gallons
Antiscalant	Liquid Tote 1 m^3 Bins	68 t	10 t
Hydrochloric Acid 32%	Liquid Tote 1 m^3 Bins	75 t	5 m^3
Fluxes	Dry Solid Sacks	2450 lb	500 lb

Table 17-9 - Projected Annual Reagents and Consumables, Stage 2 (Year 4+)

Item	Form	Annual Usage	On Site Storage
Sodium Cyanide	Liquid Delivery, 30% NaCN by Weight	3,245 t NaCN solid	29,950 gallons
Lime (Pebble CaO)	Bulk Delivery Truck	18,863 t	230 t
Activated Carbon	1100 lb Supersacks	53.3 t	10 t
Sodium Hydroxide	Liquid Delivery, 50% Caustic by Weight	60 t NaOH solid	5,000
Flocculant	Dry Solid Sacks	114 t	10 t
Antiscalant	Liquid Tote 1 m^3 Bins	280 t	20 t
Hydrochloric Acid	Liquid Tote 1 m^3 Bins	357 t	10 m^3
Fluxes	Dry Solid Sacks	14,200 lb	1000 lb
Sulfuric Acid, 93%	Liquid Delivery, 93% H_2SO_4 by Weight	659 t	5000 gallons
Hydrogen Peroxide, 70%	Liquid Delivery, 70% H_2O_2 by Weight	121 t	2000 gallons
Grinding Balls	Bulk Delivery Truck	1040 t	50 t
Ball Mill Liners	Bulk Delivery Truck	104 t	1 set

17.9.1 Sodium Cyanide

Sodium cyanide (NaCN) will be used in the leaching process and will be delivered in tanker trucks as a liquid at 30% concentration by weight (1.15 s.g.). Sodium cyanide will be stored in a 23,500-gallon steel tank at the ADR area and a 6,500-gallon tank at the mill area within concrete containment, and will be distributed by metering pumps to points of use.

All cyanide distribution lines will be double-containment, either by "pipe-within-pipe" or "pipe-over-liner" containment systems. Cyanide will be consumed at an estimated 0.35 lb/ton solid NaCN for the ROM ore and 1.00 lb/ton solid NaCN for the high-grade mill ore, plus 75 lb per strip and 200 lb per intensive leach batch.

17.9.2 Lime

Pebble quicklime (CaO) will be used to treat the ore prior to cyanide leaching to maintain the alkaline pH. Lime will be delivered in bulk by 20-ton trucks, which will be off-loaded pneumatically into storage silos. Stage 1 will include a 200-ton ROM lime silo with a variable speed feeder that will meter lime directly onto the ore being carried by haul trucks to the heap leach and will added in proportion to the tonnage of ore in each truck. A second lime silo (30-ton storage capacity) for high-grade mill ore will be added in Stage 2, with lime being metered onto the ball mill feed conveyor.

Lime will be consumed at an estimated 2.35 lb/ton for ROM (low-grade) ore and 1.34 lb/ton for mill (high-grade) ore.

17.9.3 Activated Carbon

Activated carbon will be used to adsorb precious metals from the leach solution in the adsorption columns and CIL circuit. Make-up carbon will be 6 x 12 mesh and will be delivered in 1,100 lb supersacks. It is estimated that approximately 3% of the carbon stripped will have to be replaced due to carbon fines losses. Stage 1 consumption has been estimated at 11.2 tons per year during Stage 1 and 53.3 tons per year during Stage 2.

17.9.4 Sodium Hydroxide (Caustic)

Sodium hydroxide (caustic) will be delivered to site as a liquid at 50% caustic by weight (1.53 s.g.). Liquid caustic will be stored in a 5,000-gallon steel tank and metered to the strip solution tank and acid wash circuits by a caustic metering pump. Caustic has been estimated to be consumed at 300 lb per strip and 41 lb per intensive leach batch. The estimated consumption of sodium hydroxide will be approximately 11 t solid per year during Stage 1 increasing to 60 t solid per year during Stage 2.

17.9.5 Flocculant

Flocculant will be used to aid settling in the thickeners and clarifier. Flocculant will be received in 50 lb bags on pallets. Estimated consumption will be at 0.12 lb per ton solids for the thickeners and 0.070 lb per ton solids for the clarifier. Estimated consumption will be 114 tons per year during Stage 2 only.

17.9.6 Antiscalant

Antiscalant will be used to prevent the build-up of scale in the process solutions and heap irrigation lines. Antiscalant will be added directly into pipelines or tanks, and consumption will vary depending on the concentration of scale-forming species in the process stream. Delivery will be in liquid form in 1 m^3 tote bins.

Antiscalant will be added directly from the supplier tote bins into the pregnant, barren, and desorption pumping systems using variable speed, chemical-metering pumps. On average, antiscalant consumption is expected to be about 6 ppm of for leach solutions and 10 ppm for strip solutions to be treated, which equates to 374 lbs per day during Stage 1 and 1,523 lbs per day during Stage 2.

17.9.7 Hydrochloric Acid

Hydrochloric acid (32%) will be used in the acid wash section of the elution circuit prior to desorption. Hydrochloric acid (32% by weight, 1.16 s.g.) will be delivered in 1 m^3 tote bins. Acid washing consists of circulating a dilute acid solution through the bed of carbon to dissolve and remove scale from the carbon. Acid washing of carbon will be done before each desorption cycle. Annual consumption of 32% HCl has been estimated at 75 tons per year during Stage 1 and 357 tons per year during Stage 2.

17.9.8 Fluxes

Various fluxes will be used in the smelting process to remove impurities from the bullion in the form of a glass slag. The normal flux components are a mix of silica sand, borax, and sodium carbonate (soda ash). The flux mix composition is variable and will be adjusted to meet individual project smelting needs: fluorspar and/or potassium nitrate (niter) are sometimes added to the mix. Dry fluxes will be delivered in 50 lb bags. Average consumption of fluxes has been estimated to be 1.0 lbs per pound of gold and silver produced.

17.9.9 Sulfuric Acid

Sulfuric acid (93%) will be used in to prepare Caro's acid for cyanide detoxification. Sulfuric acid (93% by weight, 1.84 s.g.) will be delivered in tanker trucks. Annual consumption of 93% H_2SO_4 has been estimated at 659 tons per year during Stage 2 only.

17.9.10 Hydrogen Peroxide

Hydrogen peroxide will be used in to prepare Caro's acid for cyanide detoxification. Hydrogen Peroxide (70% solution by weight, 1.29 s.g.) will be delivered in tanker trucks. Annual consumption of 70% H_2O_2 has been estimated at 121 tons per year during Stage 2 only.

17.9.11 Process Make-up Water Results

Make up water requirements for the process facilities have been estimated based on the water balance calculation by GLA, and miscellaneous mine, mill, and infrastructure requirements estimated by KCA.

During Stage 1 operations, the average monthly heap leach make-up water demand will be 360 gpm with the largest average monthly make-up water demand of 620 gpm. During Stage 2 operations, the average monthly heap leach make-up water demand will 650 gpm with the largest average monthly make-up water demand of 950 gpm. Make-up water demand for the mill will be nominally 65 gpm, and is included in the above totals.

17.10 Process Power Requirement

Power usage for the process equipment and infrastructure has been estimated based on connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation are assigned and coupled with estimated on-stream times to determine the average energy usage.

KCA's scope for power estimation includes the heap leach, mill, recovery plant, and infrastructure. The Project power estimates for Stage 1 are:

- Attached Power: 1.6 MW
- Peak Demand Load (for genset sizing): 1.0 MW
- Average Demand Load: 0.8 MW

The Project power estimates for Stage 2 are:

- Attached Power: 9.0 MW
- Peak Demand Load (for genset sizing): 7.1 MW
- Average Demand Load: 6.0 MW

18.0 PROJECT INFRASTRUCTURE

18.1 General Infrastructure

18.1.1 Existing Installations

Infrastructure remaining from previous operations at the Castle Mountain project site include the main site access road as well as the west wellfield area which supplied water for the past operations. The west well field area includes three water wells, each equipped with a diesel powered well pump (well pumps W-45P, W-18P and W-14P), and a 50-foot diameter water storage tank. Water supply for ROM heap leach and mill will primarily be by new wells with the existing west wellfield and water tank being used only for water supply to water trucks for dust suppression.

All other infrastructure from the previous operations, including site buildings, electrical power line and water distribution systems were removed as part of site reclamation efforts.

18.1.2 Access Road

The Castle Mountain project is located in San Bernardino County, California and is approximately 70 miles south from Las Vegas, Nevada. Access to the project site is by The Walking Box Ranch Road, approximately 18 miles of unpaved road off of Nevada State Route 164, approximately 5 miles west of Searchlight, Nevada. The existing access road is a two-lane road and is sufficient for current exploration and preliminary construction activities. For major construction and operations, road improvements, including road widening in certain areas, will be required.

18.1.3 ROM Haul Road

For Stage 1 operations a haul road will be constructed from the JSLA backfill area along the historical conveyor corridor, which is within the currently permitted boundaries. The haul road will ramp up onto the historical heap area, around the southern edge and over to the newly constructed heap leach pad area for ROM ore stacking. For Stage 2 the haul road will be expanded as necessary to reach both the southern and northern areas of the expanded heap leach pad.

18.1.4 Project Buildings

Buildings and facilities for the project and operations have been considered and will be constructed in two stages. Buildings required during Stage 1 include the administration and mine offices buildings, a small modular laboratory, site gate house, ADR and reagent storage facility, and refinery building. A mill shop and warehouse building will be constructed in Stage 2, along with a larger laboratory.

During Stage 1 the mining contractor will operate and maintain mining production and support equipment with minimal Equinox Gold-supplied infrastructure. A 10,000-gallon diesel tank will be supplied by Equinox Gold in Stage 1. For Stage 2, a mine truck shop building and lube area will be constructed for owner mining activities, along with the addition of a second 10,000-gallon diesel tank and mine offices.

The site facilities layout during Stages 1 and 2 are shown in Figure 17-3 and Figure 17-4 respectively.

18.1.5 Administration & Mine Offices Buildings

The administration and mine offices building will consist of a modular office complex and will be constructed during Stage 1. The admin office building will be 48' by 60' and is sized to accommodate approximately 20 persons including private and open office spaces and a conference room.

In Stage 2 an additional complex will be added to accommodate approximately 45 permanent administrative and mine management personnel.

18.1.6 Laboratory Building

For Stage 1 operations, a small starter laboratory will be installed on site. The lab will consist of three 8' by 20' containerized units for sample preparation, fire assay, and a wet lab for managing assays for ore control. Metallurgical testing and other process control tests, and bullion assaying will be contracted off-site.

A full-service assay and metallurgical laboratory will be installed on site in Stage 2 to run all sample analyses required for mining and process operations. The laboratory will include sample preparation, fire assay, bullion analyses, AA spectroscopy, particle size distribution analyses, metallurgical testing (bottle roll and column leach testing) and personnel offices. The laboratory includes a finished steel building with a foot print of approximately 5,000 ft^2, dust collection system for the sample preparation area and a ventilation system with a wet scrubber for the wet lab area. Laboratory metallurgical chemical wastes will be stored temporarily on site and to the heap leach for disposal in the process systems.

The Stage 2 laboratory is designed to process 200 solids samples daily.

18.1.7 Gate House

A gatehouse will be located at the entrance to the mine site. The gate house will include a reception desk and waiting area.

18.1.8 ADR & Reagent Storage Facility

The ADR area includes the carbon adsorption, elution, acid wash, carbon handling and thermal reactivation circuits as well as the cyanide and caustic storage area. The desorption and refinery areas and buildings will be sized and constructed in Stage 1 to contain the equipment expansions in Stage 2 and have a combined footprint of approximately 16,000 ft^2. A corrugated steel roof with open sides will be constructed over the desorption area, which includes the elution, acid wash, and carbon handling areas. The refinery building includes the electrowinning, mercury retort and smelting equipment. For security reasons, the refinery building will be constructed with reinforced CMU blocks and the entire ADR facility will be inside a fenced area. The refinery building will include a security office, washroom and break room.

The cyanide and caustic storage area includes separate tanks for storing liquid caustic and liquid cyanide. Each tank is installed within a concrete containment area sized for containment of 110% of the respective tank volumes.

18.1.9 Mill Workshop / Warehouse

The mill workshop and warehouse building will be constructed during Stage 2 and will be approximately 6,500 ft^2. This building will be located adjacent to the mill and will include a tool room, offices, a meeting and break room, and a bathroom.

18.1.10 Mine Truck Shop

The mine truck shop will be constructed by the owner during Stage 2. The mine truck shop will be an 18,000 ft^2 steel building with five primary work bays, including a truck wash bay. The truck shop will include a 10-t overhead crane, lubrication pumps, high pressure wash system and an oil skimmer.

18.1.11 Fuel Storage & Dispensing

The diesel fuel storage and dispensing area will be constructed during Stage 1 by the mining contractor and will service heavy and light vehicles for the mine and process equipment. Diesel fuel will be delivered to the mine site via tanker trucks and stored in storage tanks. The fuel storage facility has a storage capacity of 10,000 gallons of fuel and is equipped with all necessary fuel dispensing equipment.

All fuel storage tanks will be placed in concrete containment with capacity to hold 110% for the fuel storage tank volume to assure no fuel is leaked to the environment.

A second 10,000-gallon tank will be installed in Stage 2 to accommodate the increase in mine production and support equipment.

18.1.12 Mill Building

In Stage 2, a mill building will be constructed to cover the grinding and gravity concentration circuits. The main building will consist of a corrugated steel roof with open sides, includes a 10-ton overhead crane for maintenance activities and is sized at approximately 8,000 ft^2. The gravity concentration circuit is located in an enclosed and secure area within the mill building footprint.

18.1.13 Filtration Building

In Stage 2, a filtration building will be constructed to cover the mill tailings filters and associated equipment. The filtration building will consist of a corrugated steel roof with open sides, includes a 3-ton overhead crane for maintenance activities and is sized at approximately 12,000 ft^2.

18.1.14 Security and Fencing

Access to the site will be limited by fences around the process areas. A security gate and gate house are also included at the project site entrance and will be manned 24 hours per day.

18.2 Power Supply & Communication Systems

18.2.1 Power Supply – Stage 1

For Stage 1 operations (Years 1-3), electrical power for the Project will be supplied using generators.

The ADR plant, laboratory and process solution pumps will be powered using four 360kW gas generators running on propane. This system will operate with three generators running and one generator on standby for maintenance. The generator plant will output at the Stage 1 distribution voltage of 460V, 60Hz and will not require a step-up transformer.

Power for the mine and administration building complex and fuel station will be supplied by a single diesel generator. The mine contractor will supply diesel generators as necessary for maintenance and operation of the mining fleet.

The five remote well pumps used for providing site make-up water will be powered by small individual diesel generators, one located at each well head.

18.2.2 Power Supply – Stage 2

For Stage 2 operations (Year 4+), electrical power for the Project will be supplied using line power, in a similar configuration to what was provided for historical operations. The plan is to receive retail service to the project by NV Energy (NVE). A preliminary study was conducted with the plan to provide metering to site at NVE's Searchlight substation, connect to an existing 69 kV overhead line running approximately 9 miles from the Searchlight substation to a switching station at Walking Box Ranch, and construct 15.5 miles of new 69 kV overhead line to the Project site along Walking Box Ranch road (the main access road). The study also indicated that the additional load on NVE's upstream transmission infrastructure will require rebuilding of eight miles of NVE-owned 69 kV overhead line. A 69 kV primary substation (owned and maintained by Equinox Gold) will be constructed adjacent to the mill area to step down to 13.8 kV for power distribution on site. 13.8 kV overhead line power will be run to all facilities including the mill, crushing plant, mine and administration area, ADR plant, laboratory, pumping system, and also the newly constructed well field located south of the Project site. The ball mill will operate at medium voltage (4,160V) but all other facilities will operate at low voltage (480V and below) with appropriate transformers installed at each area.

The Stage 1 process generators will be used as emergency backup power generation for critical process equipment once line power is available, and will be supplemented by the purchase of an additional 2,000 kW diesel generator to support the necessary backup load. The primary equipment requiring backup power are the barren and pregnant solution pumps, and mill CIL and filter tank agitators.

18.2.3 Site Power Distribution & Consumption

Power distribution around the process plant and facilities will be at 480 V, 3 Ph, 60 Hz during Stage 1 and will be achieved by a combination of underground and overhead power distribution lines.

Power distribution during Stage 2 will be at 13.8 kV, 3 Ph, 60 Hz and will be further stepped down to 4,160 V, 480 V, 220 V and 120 V accordingly. Large operating motors will be supplied power at 4,160 V and smaller operating motors will use 460 V. Electrical outlets will be 120 V.

The estimated power attached power and consumption by area for Stages 1 and 2 is presented in Table 18-1.

Table 18-1 - Castle Mountain Power Demand

Stage 1 Operations	Average Demand (kW)	Peak Demand (kW)
Area 22 - Heap Leach Pad & Ponds	396	410
Area 28 - Carbon Adsorption	5	7
Area 29 - Carbon Desorption & Reactivation	146	191
Area 31 - Refinery	91	123
Area 34 - Reagents	6	13
Area 38 - Laboratory	73	98
Area 60 - Power	0	0
Area 62 - Water Supply, Storage & Distribution	48	145
Area 66 - Facilities	14	16
Total	**781**	**1003**
Stage 2 Operations	**Average Demand (kW)**	**Peak Demand (kW)**
Area 13 Crushing	485	647
Area 15 - Crushed Ore Stockpile & Reclaim	15	18
Area 16 - Grinding	1515	1683
Area 17 - Gravity Separation	87	116
Area 18 - Grinding Thickener & Process Solution	74	82
Area 19 - CIL	293	325
Area 20 - Tailings Thickener	28	37
Area 21 - Filtration, Clarification & Detox	689	918
Area 22 - Heap Leach Pad & Ponds	1599	1642
Area 28 - Carbon Adsorption	15	20
Area 29 - Carbon Desorption & Reactivation	171	226
Area 31 - Refinery	111	149
Area 34 - Reagents	13	26
Area 38 - Laboratory	131	175
Area 60 - Power	0	0
Area 62 - Water Supply, Storage & Distribution	210	448
Area 65 - Compressed Air	453	504
Area 66 - Facilities	48	53
Total	**5936**	**7069**

18.2.4 Communication Systems

The site will be connected to the local phone and internet data network using a microwave or other through the air method. A microwave network was used by Viceroy during the previous operations for phone and internet connections. Currently, the site uses a satellite system to support basic internet and phone communications. A sitewide VHF radio network is currently on site with multiple channel capacity. This radio network will be maintained, and additional repeaters put in place if needed.

18.3 Water Supply and Distribution

18.3.1 Water Balance

A water balance model was prepared by GLA and is discussed in greater detail in Section 17 of this Report. The water balance considers the Project's water demand, water collected from direct precipitation and seasonal evaporation. Additional water consumption allowances were included by KCA for road dust suppression (100 gpm), mill tailings moisture loss (65 gpm), and miscellaneous uses (15 gpm). Based on the water balance model plus these allowances, makeup water requirements average 360 gpm during Stage 1 operations and 650 gpm during Stage 2.

18.3.2 Site Water Management – Stage 1

In Stage 1, site water requirements will be met by two separate and isolated systems.

The first system consists of three production wells from historical operations, W-14, W-18, and W-45, which are already connected via existing underground pipelines to an existing 250,000-gallon (50' diameter) water tank. This system is referred to as the West Wellfield area and is located to the northwest of the Project site. The water tank feeds an existing underground line that terminates near the planned mine and administration office complex and facilities. This system will be exclusively used to provide road dust suppression water.

A preliminary water supply investigation was conducted by Geo-Logic Associates (Geo-Logic, 2018) and determined that the three production wells W-14, W-18, and W-45 can provide at least 100 gpm of sustained water production for the first five years of the Project, which is sufficient for the road dust suppression demand.

The second system consists of two production wells CMM-W-01 and CMM-W-02 recently drilled and completed in 2017, which are planned for supplying all remaining site water make-up requirements, including the heap leach and all other process and infrastructure demands. These wells are located at the edge of the JSLA backfill and South Domes pit areas, respectively, and are bedrock wells.

The wells will be fitted with new well pumps and an underground pipeline will be constructed to carry water from the wells to a new 440,000-gallon water tank located on the historical heap area just northwest of the ADR plant. The water tank will be constructed as a combined fire water and process water storage tank, and will gravity feed make-up water to the ADR plant and barren tank for the heap leach.

The water supply investigation conducted by GLA noted above determined that the two production wells CMM-W-01 and CMM-W-02 can provide at least 200 gpm of sustained water production each (400 gpm total) for Stage 1 operations, which is sufficient for the 260 gpm required for make-up to the heap leach and miscellaneous uses.

18.3.3 Site Water Management – Stage 2

In Stage 2, additional production wells will be required to meet the increased demand from the ROM processing rate expansion and the addition of the mill. Additionally, it has been assumed from the current mine plan in Stage 2 that production wells CMM-W-01 and CMM-W-02 will be depleted or partially depleted in Year 3 and Year 7 respectively, requiring additional replacement wells for this supply volume.

The water supply investigation and preliminary modeling conducted by GLA proposed a number of options in the vicinity of the project site that could potentially meet the site water demand in Stage 2, based on historical production and exploration drilling. The selected option for the study was for production wells to be installed approximately 3 miles southwest of site, termed the South Wellfield. Additional investigation including exploration drilling and pump tests in the area will be required to confirm the actual capacity of the South Wellfield.

In the pre-feasibility study, five production wells are assumed to be drilled in the South Wellfield area, SWW-01 through SWW-05. It is estimated that each well will be able to produce a sustained 200 gpm during Stage 2 operations, for a total of approximately 1,000 gpm. An underground pipeline will be installed to connect these wells to a booster tank near SWW-02 in the South Wellfield area. From the booster tank water will be pumped approximately 3 miles to the 440,000-gallon water storage tank installed in Stage 1. The water storage tank will continue to gravity feed make up water to the ADR plant and barren tank and will also supply make up water to the newly constructed mill.

The West Wellfield wells will continue to supply road dust suppression water in Stage 2, and a small line will be installed to also feed the crusher for dust suppression.

18.3.4 Construction Water

Water for earthmoving and other construction activities will be provided by the existing West Wellfield system, which is already installed and operating.

18.3.5 Fire Water & Protection

A dedicated water system will be installed in Stage 1 and expanded in Stage 2 to provide fire protection to all areas of the project site.

The 440,000-gallon water tank installed in Stage 1 includes a dedicated fire water reserve of 180,000 gallons (1,500 gpm for up to 2 hours of continuous flow).

In Stage 1, a modular fire pump system will be installed at the 440,000-gallon water tank which includes two diesel-powered fire pumps, a jockey pump, and piping, valves and controls.

The fire pump system will feed a pipe network consisting of buried 10” steel and HDPE mains, which will feed the ADR plant, laboratory, generator plant, mine and administration offices, and fuel station. Hydrants and foam stations will be installed for proper coverage in these areas.

In Stage 2 the buried pipe network and associated hydrants will be expanded to provide coverage to the crusher, mill complex, and truck shop.

18.4 Sewage & Waste

18.4.1 Effluents

Other than treated effluent rom the site septic systems, the project will have no water discharge to the environment.

18.4.2 Sanitary Waste (Sewage)

Lavatory and wash facilities will be located throughout the project site. Sanitary waste from the lavatories will flow by gravity to multiple septic systems for treatment and disposal. Each septic tank and drainfield are sized for the building occupancy.

18.4.3 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. On-site burning of any waste materials, vegetation, domestic waste, etc. will not be allowed.

18.4.4 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.

19.0 MARKET STUDIES AND CONTRACTS

No market studies or contracts were conducted for the Project.

Gold doré bars are routinely sold to third party refiners and the pre-feasibility study assumes the refining and transportation costs will be $1.51 per ounce of gold and the refiner will pay 99.95% of the value for the gold.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The following Environmental Studies and Permitting sections are from Cox et al., 2014 (modified in Gray et al., 2016). In 2017 and 2018 several baseline flora, fauna and cultural resources studies have been performed to establish that the original baseline studies performed in support of the 1990 Environmental Impact Study / Environmental Impact Review (EIS/EIR) and the 1998 update to the EIS/EIR are still representative of the environmental and cultural resources of the site. These studies have all shown that there have been no significant changes to site baseline conditions so the EIS/EIR still properly represent the approved impacts of the site.

20.1 Environmental Studies

20.1.1 Biophysical Environment

The Castle Mountain Project occupies the southern end of the Castle Mountain range and forms part of the northeastern margin of the Lanfair Valley catchment basin. The northern New York Mountains lie to the northwest, and the Piute Range is three mi (4.8 km) to the east. The slopes of the opposite crest of the New York Mountains are part of the Ivanpah Valley catchment basin.

The Lanfair Valley supports widespread and well-developed Joshua tree woodland. Undisturbed habitats range from bedrock and colluvial slopes to wash bottoms and alluvial fan surfaces. Elevations range from slightly over 5,100 ft. (1,550 m) along the mountain crest to about 4,100 ft. (1,250 m) at the southeastern boundary of the site. The highest peak in the region is part of the New York Mountain range rising to 7,532 ft. (2.295 m) and is situated about 10 mi (16 km) to the west of the Project.

Plant species are distributed according to many factors, including soil type, climate and available water. In the Mojave Desert, species types are often represented along an elevation gradient. Plant species vary as the finer grained soils of valley floors change to coarser deposits on lower slopes and to rocky substrate on mountain slopes. Local climatic conditions which affect species distribution are in part influenced by increases in elevation and related lower temperatures and greater precipitation. In general, a gradual change in plant species occurs from lower to higher elevations, with a different combination of species represented at any point.

The Castle Mountain Project site is located in the eastern San Bernardino County portion of the Mojave Desert Air Basin (MDAB). The climate of the Basin is arid to semi-arid, generally hot and dry in the summer and mild in the winter. Little climatic variation exists throughout the Basin. Arid weather conditions are present with low precipitation of five to ten inches (13 to 25 cm) per annum. Precipitation most frequently occurs during winter months, but a significant portion of the annual rainfall can occur as summer thunderstorms, which may result in heavy rainfall and flash floods. There is a correlation between rainfall and elevation. Precipitation in Lanfair Valley is expected to range between six inches (15.2 cm) at the lower elevations to over 11 inches (28 cm) (at the mountain peaks).

Wind speeds average between 5 mi (8 km) and 13 mi (20.9 km) per hour. Summer and winter winds are similar, generally blowing from the south and west. Vertical air dilution is generally good because of the area's high surface temperatures, creating strong daytime thermal mixing. Thermal mixing and moderate winds generally tend to disperse occasional nighttime inversions. From late fall to early spring, average daytime temperatures are moderate, averaging 60°F to 85°F. (15.5°C to 29.4°C). Nights are cooler, with temperatures averaging 40°F to 60°F (4.4°C to 15.6°C). Winter temperatures are occasionally below freezing, and can be lower than 10°F. During the summer, daily temperatures are often 100°F to 110°F (37.8°C to 43.3°C) and 80°F (26.7°C) at night. Seasonal changes in the eastern MDAB are marked principally by large seasonal and diurnal temperature differences rather than by precipitation.

20.1.2 Hydrology and Hydrogeology

The Lanfair Valley surface water drainage area is approximately 340 mi^2 (881 km^2) in size. The maximum basin dimensions are approximately 20 mi (32.2 km) (east to west) and 17 mi (27.4 km) (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 (201 km^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 Piute Springs, located about 20 miles south from the mine. The Colorado River lies 28 mi (45 km) east of the Project. The water source for operations is groundwater. Most of the water required for the Project was and will be obtained from wells drilled into the alluvium strata at the area known as the West Well Field and from hard rock wells located along large fault zones. A minor amount of water may also be obtained from wells designated as the East Well Field also located within the Project Area and from water ingress into the pits themselves.

The number of wells used during the operating period ranged from five to 14 wells. This number includes a combination of monitoring wells and production wells. As part of the permitting requirements, water levels were measured monthly.

During the previous operation, the average annual water use was 400 acre feet per year. The maximum permitted annual water use for the mine expansion was adjusted downward (1998 EIS\EIR approvals) to 625 acre feet per year (in the 1990 EIS/ EIR, the predicted water use was 725 acre feet per year) because actual water use was lower than predicted. Water quality measurements were taken at a number of wells throughout the operation. Water quality during operations was within the predicted concentrations.

As part of the decommissioning of the heap leach pad, one additional monitoring well was established and maintained until 2010. The purpose of these wells was to assess if seepage occurred from the heap leach pad. All monitoring wells were decommissioned in Q4, 2010 after the regulatory authorities determined the heap leach pad was successfully decommissioned, closed and reclaimed.

20.1.3 Acid Rock Drainage Potential

During the initial environmental review, samples of the mineralized material, waste rock and overburden were subjected to geochemical testing to determine the acid generation potential and extractable metals. The average neutralization potential (tons $CaCO_3$/1,000 tons of material) was 54.3. The acid generating potential was 2.4 (tons $CaCO_3$/1,000 tons of material) resulting in an NP:AP ratio of 22.6. A NP:AP ratio greater than 3 or 4 is considered to have sufficient buffering capacity to negate hazards associated with acid rock drainage. The 1998 EIS/EIR analyzed the potential for acidic conditions in pit water and found, once again, the Project has very limited acid-generating sulfide minerals, and the natural alkalinity provided by the rock and ground and surface water inflows minimize the potential for acidification of the pit water. Additional mitigation in the current permits requires analysis for ARD potential in the pit water and if any pit contains water where poor water quality would develop this pit must be backfilled. The surfaces of these backfilled pits would consist of coarse material to allow infiltration of meteoric waters to minimize ponding.

During the previous operations, there was no evidence of acid rock drainage. Also, the ponded water at the bottom of Leslie Anne Pit did not show any signs of sulfidic oxidation. Based on the analytical data and the previous operational experience, the potential of acid rock drainage is extremely low and this matter has been addressed.

20.1.4 Vegetation and Wildlife

In June 2010, CMV released a report entitled “Biological Resources Assessment Castle Mountain Venture Land holdings County of San Bernardino, California” (Lilburn Report). The purpose of the report was to evaluate desert tortoise habitat within the vicinity of the Project as a potential offset (compensatory) location for a nearby renewable energy project. The government agencies determined that the project area did not offer sufficient quality of habitat to be considered as a valuable offset as a compensatory measure.

20.1.4.1. Vegetation

The Lilburn Report is the most recent review of the status of biological resources found within and adjacent to the Project area and includes literature data base searches from the Natural Diversity Data Base and U.S. Fish and Wildlife Service listings (January 2010). In summary, initial surveys and reports completed between 1987 and 1995 (Draft EIR/EIS BLM 1989, Plan Amendment Application Viceroy Gold 1995) indicated no listed endangered or threatened plant species known or expected to occur on or near the Project. However, the initial special species studies listed a

Category 2 "candidate" species, Stephens' beardtongue (Penstemon stephensii), as occurring on ground disturbed by road building on the northeastern portion of the claims. In addition, two other "non-candidate" species were observed, and three other species were listed that may occur in the area. The most recent database list no longer includes Stephens' beardtongue as a candidate species. During the review, four sensitive plant species were found on the Project.

20.1.4.2. Wildlife

The Project area has a wide variety of habitat types from high elevation rocky slopes to desert washes that can support a variety of wildlife species. The Lilburn Report identified approximately 44 species that were observed or whose habitat is found at the site. Ravens, owls, bobcats, coyotes, lizards, snakes and bighorn sheep frequently visit the Project area. Nelson's Bighorn Sheep are often seen at the Project. The bighorn sheep prefer steep, rocky terrain as a means to escape predators. The re-vegetated area of the Project provides bighorn with a food source. The water guzzler at the Project provides the sheep and other wildlife with a water supply through the summer months.

The Federal and State listed "threatened" desert tortoise habitat is found in some portions of the Project area. 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 to listed species and provides significant penalties for violations. However, the law also provides procedures for legally impacting threatened or endangered species under certain circumstances.

The need to protect the desert tortoise during construction and operation of the former mine was a major concern during the environmental permit reviews. The previous operators committed to specific Project changes in the 1990 and 1998 EIS/EIRs. For example, mine access roads were re-aligned to avoid higher quality desert tortoise habitat and other Project facilities were also located away from areas of high tortoise population density. Additionally, any tortoises found on the site prior to disturbance of an area are safely relocated to a location providing suitable habitat; the site has been partially fenced, and fencing was extended when mining activity expanded; potential raven predation is controlled, and employee vans and buses will continue to be used as needed to reduce traffic volumes through tortoise habitat. An education program outlining information on desert tortoises and the responsibilities of Equinox workers and subcontractors is required to be implemented. Mitigation included providing alternative road access to the site, the installation of tortoise proof fencing, surveying all undisturbed areas prior to construction, and restrictions on vehicle traffic use and volumes. Further mitigation included the retirement of grazing rights for CMM held parcels of land purchased by NewCastle and the acquisition of 745 acres of private land (Crescent Peak) to be managed for the benefit of desert tortoises. An additional Environmental fund of $2.8 million was established as well. The land in the Ivanpah Valley was sold to Clark County through the

Nature Conservancy. Consequently, during both environmental assessment reviews (1990 and 1998) of the Project, the Fish and Wildlife Service (USFWS) determined that the Project would not result in “jeopardy” or harm to the desert tortoise population. As noted above the Project and areas adjacent to the portion of the access road in California comprise low quality desert tortoise habitat (i.e., because of the high elevation of the site).

20.1.5 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 mine development in proximity to the current Castle Mountain 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 resources field studies were undertaken as part of the environmental assessment reviews to identify if there were any significance and potential 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. Approximately 48 sites were identified. Mitigation measures excluded certain sites from mine development. A chain link fence was built around the Hart town site cemetery. Three hundred foot buffer zone separated the cemetery from the North Overburden Site. Future Project design activities will acknowledge and accommodate all historic and prehistoric resources found on the site.

During previous operations, no paleontological or archaeological deposits were uncovered during the construction and operational phases of the mine.

20.1.6 Mojave National Preserve

On October 31st, 1994, the Mojave National Preserve was established 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 north, west, east and south of the Project. The Project is bounded on all sides by a buffer zone administrated by the Bureau of Land Management.

20.1.7 Castle Mountain National Monument

On February 12, 2016, Barrack Obama, President of the United States of America, by presidential proclamation, established the Castle Mountains National Monument. The reserved Federal lands and interests in lands encompass approximately 20,920 acres and the boundaries fall between the Project and the aforementioned 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.

The Secretary also continues to manage the Federal lands and interests in lands within the "Castle Mountain Mine Area" through the Bureau of Land Management, pursuant to applicable authorities. Upon the determination of the Secretary that either:

1. all mining and mining-related activities have terminated and reclamation has been completed; or

2. a period of 10 years from the date of the proclamation has elapsed during which no commercial mining activities have occurred pursuant to a Bureau of Land Management approved plan of operations. The Secretary shall, consistent with applicable legal authorities, transfer jurisdiction of the lands within the Castle Mountain Mine Area to the National Park Service and ensure that the lands are managed in a manner compatible with the proper care and management of the objects identified above.

20.1.8 Environmental Monitoring

The main monitoring elements have included the following parameters:

- Ground water elevations in water production wells and regional monitoring wells;
- Groundwater quality in monitoring wells located near the leach pad and mining operations;
- Vegetation inventories and propagation of indigenous plant communities;
- A dedicated program surveying the movements and tracking of desert tortoise; and
- Wildlife monitoring surveillance programs.

During the previous operations, there was a complete set of management plans addressing regulatory requirements. The EIS/EIR documents included predicted ambient air quality concentrations at the zone of impingement. PM-10 air quality monitoring was conducted at the site during operations, though no exceedance of any standards are known to have occurred due largely because discharge limits were based on point source emission factors and the isolated location of the Project area.

20.1.9 Supplemental Monitoring

In 2014 NewCastle undertook biological inventories along the power corridor and the southern portion of the Project. These studies confirmed the result of the previous studies and will be used in future permit applications.

20.2 Project Permitting and Permitting Process

20.2.1 Completion of the Federal and State Environmental Assessment Review Process

All permits were in place when the Castle Mountain Mine was operating. Since 2004, the Project has been maintained on idle status. During this period, the environmental review permits issued after the Project was released from the State and Federal environmental assessment processes were maintained. Also, all fees have been paid and all applicable permits and authorizations have been maintained by NewCastle.

There are two environmental assessment processes required to assess the potential project effects resulting from mine development: The California Environmental Quality Act (CEQA) and the Federal National Environmental Policy Act (NEPA). The federal lead agency with responsibility for the Project is the Bureau of Land Management (BLM). The California State lead agency for the Project is the County of San Bernardino (County).

One environmental review document, the Environmental Impact Statement/ Environmental Impact Review (EIS/EIR) is prepared to address the requirements of both agencies. Federal, State, County and municipal officials as well as the public review the documents. As part of the public involvement process, a Notice of Intent on the proposed action is published in the Federal Register. If the Project is approved then each of the lead agencies prepares their respective approvals. The County issues a Conditional Use Permit and Mining and Reclamation Plan. The BLM issues a Record of Decision. Once these permits have been granted, the Proponent can apply to agencies who issue “operational” permits and authorizations.

In 1990, the EIS/EIR was approved for the Castle Mountain Mine. In 1998, the previous operator underwent another joint review process to allow for the expansion of the operations. In both cases, a series of environmental component reports were completed covering all aspects of the biophysical and socio-economic environment of the Project and the potential impacts were assessed.

In the case of the 1998 environmental assessment review, two public scoping meetings were conducted. Over 330 copies of the Draft EIS/EIR were distributed. During the 60-day review period, two public hearings were held on the Draft EIS/EIR. The Final EIS/EIR was distributed to the public, agencies and organizations who had expressed an interest in the Project. The availability of the documents and issues raised during the public meetings were published in local and regional media. The issues of primary concerns which arose during the 1998 public review process included reclamation and backfilling of the open pits; compliance under the current permit; phasing of operations and changes to groundwater monitoring at Piute Springs. These concerns were addressed in the Final EIS/EIR.

The County and BLM consult with each other to ensure that the Project meets applicable State of California, San Bernardino County laws and regulations and Federal government legislation prior to approving the Project. After the 1998 environmental review was completed, the approvals and permits granted by the authorizing agencies were updated.

The environmental assessment permits issued by the County and BLM are shown below:

- San Bernardino County, Department of Lands: Conditional Use Permit. SAMR/88-003/DN585-1145N; Reclamation Plan 90M-013;
- 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; and
- 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, July 1, 1998.

These permits are still in effect. In July 2013, the County extended the operational term of the Conditional Use Permit from 2020 to 2025. Under these approvals, there are a number of documents that provide the basis of other permits or explain the reasons for conditions stipulated in the Conditional Use Permits and Records of Decision. These documents include:

- The Castle Mountain Mine Expansion Project Draft EIS/EIR (March 1997) which incorporates the Castle Mountain Project Draft EIS/EIR (February 1989) and the Castle Mountain Project Final EIS\EIR (August 1990));
- Castle Mountain Mine Expansion Project Final EIS/EIR (October 1997);
- The Castle Mountain Mine Plan Amendment Application, including the Reclamation Plan (August 14, 1997);
- Preliminary Report of National Register of Historic Places/California Register of Historical Resources Eligibility Evaluation of Archaeological Site CA SBR-3060/H (August 1997);
- U.S. Fish and Wildlife Biological Opinion (August 1990) (1-6-90-F-24);
- U.S. Fish and Wildlife Biological Opinion (August 1990) (l -6-90-F-24R); and
- U.S. Fish and Wildlife Biological Opinion (February 1998) (1-8-97-F-37).

An outline of NewCastle's Updated Mining and Reclamation Plan and Plan of Operations is shown in Figure 20.1.

20.2.2 Exploration Permits

In 2013, NewCastle obtained permission from the Bureau of Land Management to undertake an exploration drilling program (Decision Record: DOl-BLM-CA-D090-2013-0105- DNA).

Phase I was conducted early in 2013 and consisted of 30 drill holes and approximately 6.3 acres of disturbance.

Phase II approval was obtained on September 13, 2013 allowing for sixty (60) drill holes to be drilled on patented and unpatented lands, and will generally re-impact areas and roads previously disturbed.

The reclamation bond for 30 acres has been issued to the County and the BLM.

20.2.3 Operating Permits

Once a project has been released from the environmental assessment process, the proponent can apply for the construction and operating permits. The Castle Mountain Mine acquired all relevant operating permits commencing in 1990 until the Mine closed in 2004. Dismantling and active closure continued until 2007 followed by a three to five-year performance monitoring period. Because of the uncertainty of when the Mine would reopen, all infrastructure, save for a few minor items, were dismantled and removed from the site. While the site was successfully reclaimed, the previous operator retained certain permits in the event of re activation of the mine.

Prior to re-starting operations, the County of San Bernardino Land Use Services Department will be notified and a revised Plan of Operations will be prepared. Most "operational" permits take three to six months to obtain after the application is submitted. Once the Project schedule is defined, permit applications for permits that require a long lead will be submitted as early as possible so as to not impede the construction schedule. NewCastle has been in contact with the key regulatory authorities on the content of the permit applications and process.

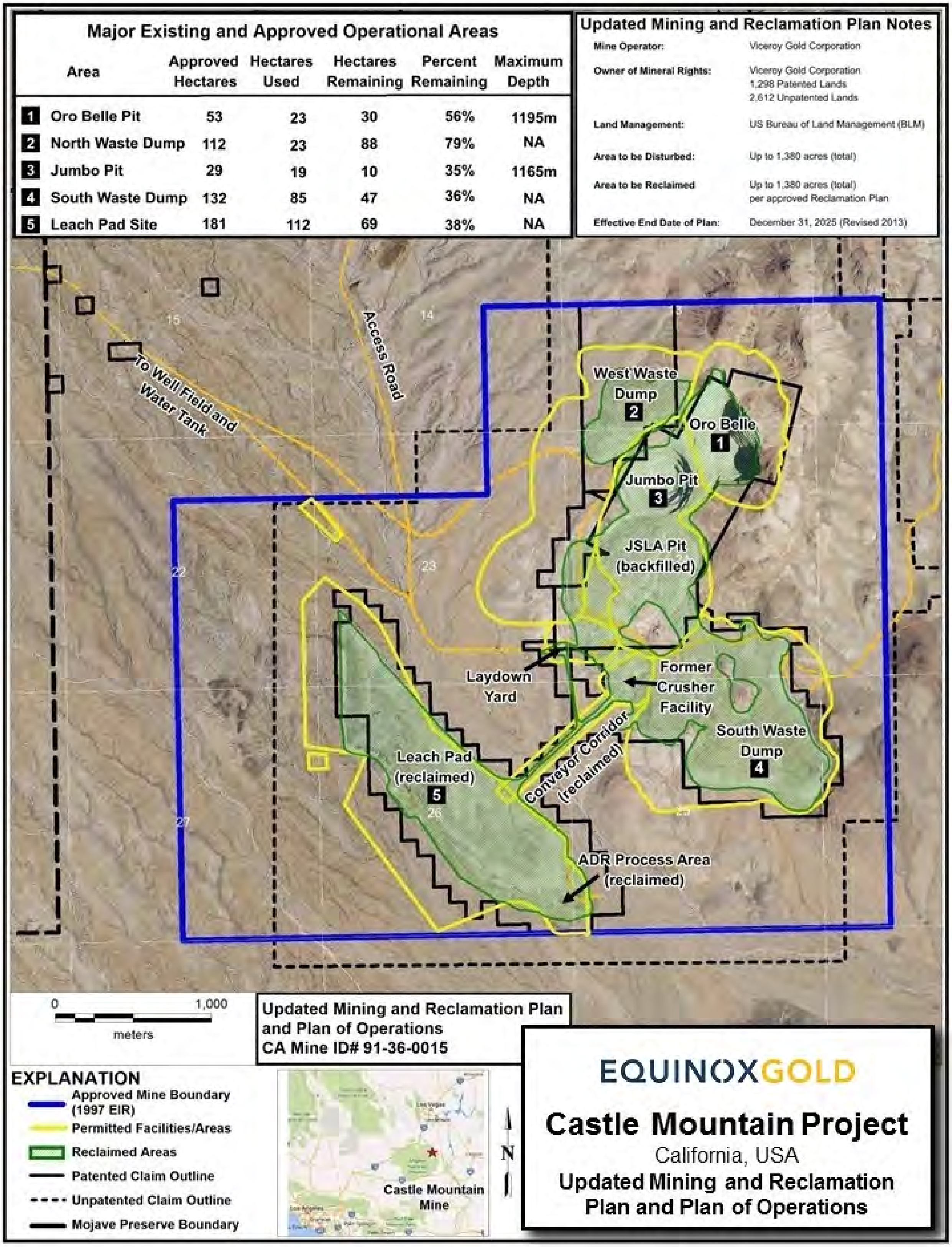

Figure 20.1 – Updated Mining and Reclamation Plan and Plan of Operations

20.2.4 BLM Right of Ways

Rights of ways are required for the water pipelines, power line transmission corridors and in some cases access roads. Most of the rights-of-ways permits have expired. Castle Mountain has maintained right of way access of BLM roads through their inclusion in the 1998 Record of Decision and the Decision Record for the exploration drilling program in 2013. NewCastle will need to reapply for these rights of way; the plan calls for re-establishing the infrastructure in the same location as it was previously found when the mine was in operation.

20.2.5 Air Management

The Mojave Desert Air Quality Management District (MDAQMD) requires all equipment with the potential to emit air pollutants (including air toxins and hazardous air pollutants) to be permitted prior to construction and operations. These permits are known as “Authorities to Construct” (ATCs) and “Permits to Operate” (PTOs). A New Source Review (NRS) is undertaken pursuant to District Regulation XIII. Once Equinox Gold is aware of the types of equipment and air emission devices that will be used, an application is made to MDAQD. MDAQD will inform the applicant within 30 days if the application is complete, then the permit is issued within the following 30 days. If the permit is complicated then a longer time period may be required. Equinox Gold expects the permits will be issued six months after the application is submitted.

The previous air permits held by the previous operators have expired. Air permits are renewed annually. The permit’s conditions of approval are based on source emissions standards. Therefore, the previous air quality modeling is sufficient. Permit conditions are derived from Federal, State and District laws, rules and regulations, as well as site specific operating procedures.

20.2.6 Water Management

The California Regional Water Quality Control Board, Colorado River Basin Region (RWQCB) regulates water quality, pursuant to California Water Code, for a point source discharge of wastewater to land and surface waters. The RWQCB also manages use and application of water on the heap leach pad. If the discharge could affect California’s surface waters, an NPDES Permit is required. For discharges of wastewater to land, Waste Discharge Requirements will be issued. Each Permit contains liquid effluent limitations which ensure the protection of the quality of the receiving waters. It should be noted that the process plant and heap leach are design as a zero-discharge facility, but not withstanding that the site still permit to operate from the RWQCB.

In the case of the heap leach pad approval, provisions incorporated in the licensing conditions may include the requirement for impermeable clay or synthetic liners for process solution basins and heap leach pads, sealed drainage and collection facilities to transport or contain leaching solution, diked leach pads to confine and control drainage from the leach piles, storage basins with adequate freeboard to safely contain storm run-off from within the heap leach system, drain down of solution from the leach pads in the event of a pumping failure and leakage detection monitoring systems for the leach pads, emergency solution storage and storm water storage basins.

In some cases, the RWQCB may waive site specific discharge permits if the activities can be handled through enrollment in an existing general permit.

When the previous mining operation began, the initial Water Discharge Permit (no. 91-002) was issued on January 16, 1991. This original Board Order was modified on June 10, 1999 (no. 99-015) upon completion of the environmental assessment for the site expansion and finally in June 2005 another amendment was made to address the reclamation activities.

The RWQCB approved the detoxification of the heap leach pad in September 2003. Nutrients were added in 2003 and 2004 for detoxification of the pad. Solution circulation pumps were shut off in August 2005. Decommissioning of the solution storage facilities occurred in 2005. In 2010, RWQCB concluded all closure requirements outlined in Board Order No. R7-2005-0092 had been completed for the Castle Mountain Site. The monitoring wells were properly abandoned later that year.

Upon discussions with the RWQCB, Equinox Gold will seek re-activating the previous permit (Order NO 99-015: Waste Discharge Requirements).

Mobile equipment permits and air emission permits for equipment used on a temporary basis (i.e. one year) are issued by the California Environmental Protection Agency, Air Resources Board (ARB).

20.2.7 Water Extraction Rights

There were approximately 39 historic bore or well holes drilled on the Project between 1987 and 2000 that recorded water depth and pumping information (Table 20.1). A portion of those present in the East, the Castle Mountain Mine area, the southern Valley and West Well Fields are shown in Figure 20.2.

Per California laws pertaining to groundwater rights, NewCastle has maintained its historic water rights to ground water aquifers directly underlying and adjacent to NewCastle's privately held land. Annual Notices of Groundwater Extraction are filed each year with the State Water Resources Control Board for all existing water wells (Table 20.2).

Table 20.1 – Summary of Historic Boreholes and Wells (See Figure 20.1)

#	Borehole or Well Name	Well Field	X_NAD83	Y_NAD83	Borehole Completion Date	Final Pumping Rate (gpm)	Notes	Current Status	#
1	W-25P	East	674059	3905012	2/4/1988		Test Hole	Reclaimed	
2	W-26	East	674425	3904650	1/7/1988		Condemnation Hole	Reclaimed	
3	W-27	East	674376	3904132	2/14/1988		Condemnation Hole	Reclaimed	
4	W-31P	East	672441	3904978	2/5/1988		WL Monitoring	Reclaimed	
5	W-32	East	672643	3904506	1/13/1988		Condemnation Hole	Reclaimed	
6	W-33	East	672255	3904238	3/5/1988		Condemnation Hole	Reclaimed	
7	W-35	East	672104	3905265	1/26/1988		Condemnation Hole	Reclaimed	
8	W-36	East	674458	3903655	2/23/1988		Condemnation Hole	Reclaimed	
9	JSLA	Mine/East	Unavailable	Unavailable	Feb-00	50	Production	Active Water Right	1
10	M-1	Mine/East	Unavailable	Unavailable	Unavailable		WL Monitoring	Reclaimed	
11	W-1 (L-1)	Mine/East	670741	3905205	2/18/1987		Test Hole	Reclaimed	
12	W-41 (M-3)	Mine/East	670608	3906002	11/27/1994		Production	Reclaimed	
13	PS-1	Piute Spring	679890	3886243	6/20/1987			Reclaimed	
14	PS-2	Piute Spring	681004	3887482	6/22/1987			Reclaimed	
15	PS-3	Piute Spring	680028	3888885	6/23/1987			Reclaimed	
16	W-2 (L-2)	Valley	667794	3902874	2/23/1987		Test Hole	Reclaimed	
17	W-8	Valley	663312	3899013	8/27/1987		Drill Bit Lost	Reclaimed	
18	W-3 (P-3)	West	666580	3908000	3/6/1987		Test Hole	Reclaimed	
34	W-3P	West	666561	3907703	7/26/1987		WL Monitoring	Reclaimed	
19	W-4P	West	668470	3907581	7/12/1987	50	Production	Reclaimed	
20	W-5	West	669505	3906768	7/31/1987		Test Hole	Reclaimed	
21	W-6	West	665387	3906552	8/7/1987		Test Hole	Reclaimed	
22	W-7P	West	668000	3908913	9/30/1987	40	Production	Active Water Right	2
23	W-11P	West	668182	3908321	1/11/1988	30	Production	Active Water Right	3
24	W-14P	West	668840	3907357	11/18/1987	30	Production	Currently in use	4
25	W-15P	West	669098	3907138	2/24/1988	13	Production	Reclaimed	
26	W-17	West	669278	3907916	2/11/1988		WL Monitoring	Reclaimed	
27	W-18P	West	669528	3907600	4/8/1988	25	Production	Currently in use	5
28	W-19P	West	668724	3906962	3/8/1988		WL Monitoring	Reclaimed	
29	W-20	West	668605	3906547	1/27/1988			Reclaimed	
30	W-24P	West	668755	3907619	4/5/1988	0	Production	Reclaimed	
31	W-37	West	666149	3902641	Sep-90		WL Monitoring	Reclaimed	
32	W-38	West	672264	3897943	Jul-90		WL Monitoring	Reclaimed	
33	W-39	West	667655	3908058	11/19/1994	35	Production	Active Water Right	6
35	W-40	West	667942	3907595	1/12/1996	12	Production	Active Water Right	7
36	W-42	West	668121	3907779	1/16/1996	60	Production	Active Water Right	8
37	W-43	West	668528	3909365	1/20/1996	30	Production	Active Water Right	9
38	W-44	West	667753	3908525	1/24/1996	25	Production	Reclaimed	
39	W-45	West	668595	3908752	1/28/1996	30	Production	Active Water Right	10
39	**Total Wells**					**430**		**Active Water Rights**	**10**

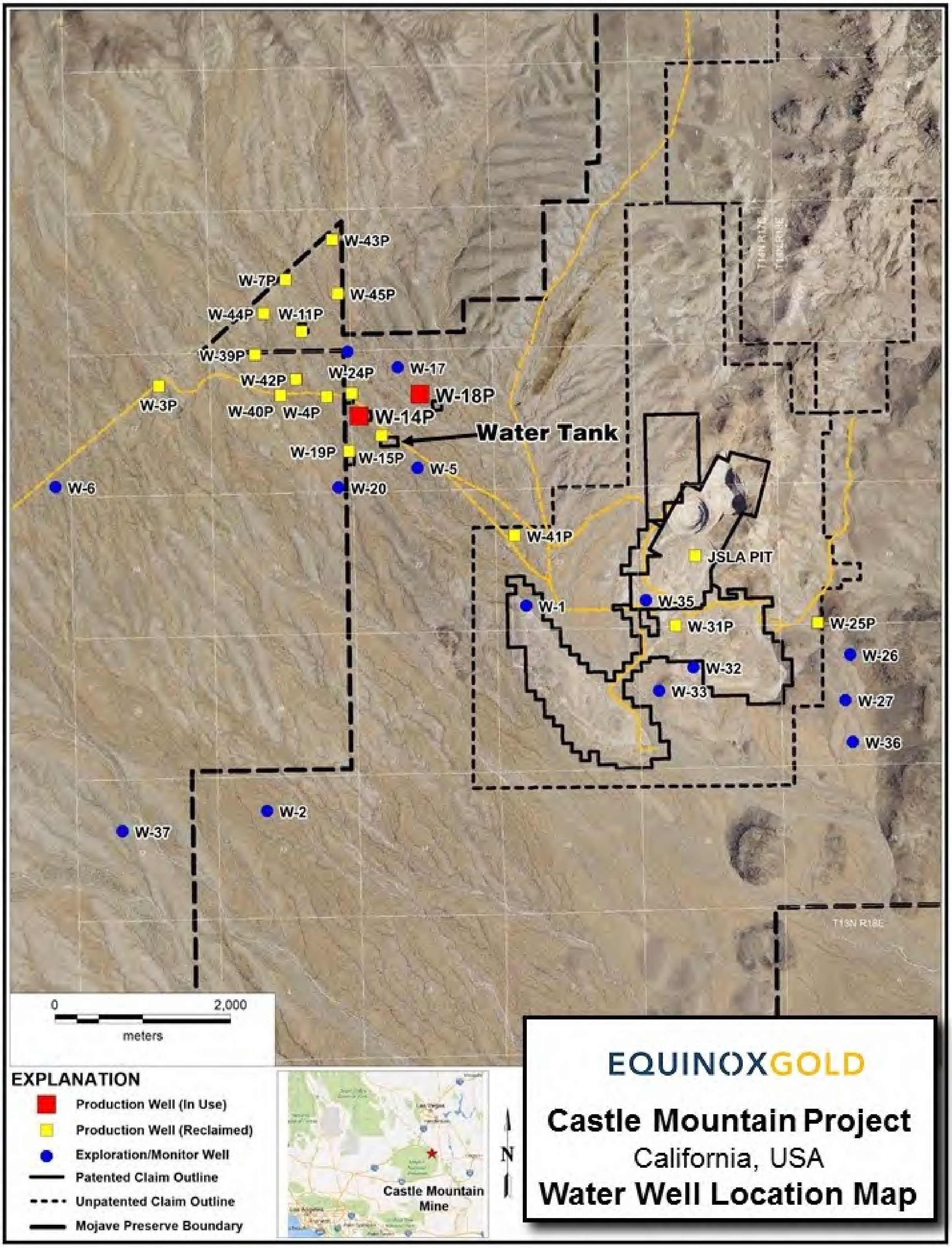

Figure 20.2 – Location Map of Historic Boreholes and Wells

Table 20.2 – Summary of Active Water Rights, Castle Mountain Project

# Rights	Recordation #/ Water Right #	Well Designation	Operating Wells	GPM	Reporting Fee	Reporting Date	Last Reported
1	G363178	W-18	**Yes**	**250**	$50	30-Jun-15	30-Jun-15
2	G363181	W-14	**Yes**	**250**	$50	30-Jun-15	30-Jun-15
3	G363586	W-7	No	0	$50	30-Jun-15	30-Jun-15
4	G363587	W-11	No	0	$50	30-Jun-15	30-Jun-15
5	G363603	JSLA PIT	No	0	$50	30-Jun-15	30-Jun-15
6	G363634	W-39	No	0	$50	30-Jun-15	30-Jun-15
7	G363635	W-42	No	0	$50	30-Jun-15	30-Jun-15
8	G363636	W-40	No	0	$50	30-Jun-15	30-Jun-15
9	G363643	W-43	No	0	$50	30-Jun-15	30-Jun-15
10	G363645	W-45	No	0	$50	30-Jun-15	30-Jun-15
10					**$500**		

20.2.8 Site Management

Listed below are some of the permits, authorizations, codes and regulations that will be required for a potential future operation. A complete list of permitting requirements will be prepared during the next stage of Project development.

- Use of above ground fuel storage tanks requires the preparation of a contingency plan for secondary containment of potential leaks, as well as inclusion in a County Hazardous Materials Business Plan.
- The San Bernardino County Fire Department, in conjunction with California Department of Toxic Substances Control (DTSC) issues a permit for the safe use of hazardous materials or when hazardous wastes are generated.
- The previously approved water production wells are still available for use. Castle Mountain requires approval from the County of San Bernardino Department of Public Health Services for installing new wells.
- Equinox Gold will comply with the rules and regulations prepared by the County of Environmental Health Services (DEHS).
- Equinox Gold will comply with the California Building Standard Code administrated by the California Building Standards Commission.
- Equinox Gold will require Building permits issued by the San Bernardino County Land Use Services and Building and Safety California code of Regulations Title 24.
- Nation-wide permits are required for construction and maintenance of roads.
- A Streambed Alteration Permit was issued for the previous operation and has since expired. An update will be required, as well as a new Jurisdictional Determination from the Army Corps. of Engineers regarding Section 404 permitting.
- The explosive powder magazine will be constructed, and explosives will be stored and used, in accordance with Federal and local requirements.

20.2.9 Waste Materials Management

As part of the reclamation activities, demolition permits were obtained and septic systems were demolished as per County regulations. An application for the site sewage system will be submitted for approval to the County of San Bernardino Department of Environmental Health and Safety.

Non-hazardous waste will be hauled off site by licensed contractor. Hazardous wastes will be removed from site using procedures approved by the DTSC and the San Bernardino County Fire Department and handled by an approved hazardous waste handler. Waste oil will be hauled off site for recycling. The Viceroy Castle Mountain Mining Toxic Release Inventory number used during previous operations is TRI Facility – 92309VCRYG11557.

20.2.10 Management Plans

Equinox Gold has a Storm Water Pollution Prevention Plan (SWPPP) and a Spill Prevention Control and Countermeasure Plan (SPCC) in place at the site. As part of the permit approval process all management plans including the SWPPP and SPCC will be updated as part of the permit applications.

20.2.11 Social and Community Impact Assessment

The Project site is located near the California/Nevada border. As noted above, it is essentially surrounded by Mojave Desert Preserve and the more recent Castle Mountains National Monument, both of which restrict the amount of activities allowed within the region. Low density land use activities prevail in the desert areas of eastern San Bernardino County, California and in the adjacent southern Clark County, Nevada. Livestock grazing, mining, recreation/tourism, and recently solar farms occur in the surrounding area. Transportation and transmission facilities, including interstate and State highways and County and local roads, railroads, power transmission lines, utility pipelines, and communication stations, are also located throughout the region. Mining has been a continuous activity in eastern San Bernardino County and southern Clark County for the past century. Many existing and former towns were founded as mining communities, including Searchlight and Crescent in Nevada, and Hart, Nipton and Ivanpah in California. Recreation activities involve casual use, oriented toward the observation and enjoyment of the area's scenery and natural or historic resources. Activities include off highway vehicle touring, sightseeing, hiking, bird watching, hunting, and rock collecting.

Small communities are scattered throughout this region of the Mojave Desert. For the most part, these communities are closely tied to major transportation corridors, such as Interstate 15, Interstate 40, and the Union Pacific and Southern Pacific Railroads. Towns such as Baker and Needles provide services to highway travelers, and are long-established railroad and trade/service centers for the surrounding desert region. Some communities within the area are becoming increasingly tourist oriented, gambling destinations, and retirement communities. Privately owned lands are interspersed in the desert area although these residences are becoming less common. Public services in the desert communities of eastern San Bernardino County are limited. In California, Baker and Mountain Pass have educational, limited fire response, and police services. In Nevada, Searchlight and Laughlin have libraries, fire departments, some educational and police, and ambulance services and community medical facilities. Other services, including hospital and high school and college educational services are available only within the larger urban centers provide a full range of public services and facilities. These urban centers include the communities within Las Vegas Valley and Barstow and Victorville/Apple Valley on the western edge of the Mojave Desert.

There are no housing or public services at the Project site. During the previous operations, most of the employees lived in Nevada. However, goods and services were obtained from both Clark County and San Bernardino County.

Other than mining employment, most workers are employed in industries such as tourism, highway trade and services or, in the case of southern Clark County, the gambling and hospitality industry.

Equinox Gold has initiated contact with government officials. Once further Project details are established, Equinox Gold will begin consultation with adjacent communities, environmental associations and other interested parties.

20.2.12 Mine Closure and Rehabilitation

Both San Bernardino County (No. 90M-013) and BLM have reclamation obligations included in their approvals. The Surface Mining and Reclamation Act (SMARA) outlines the State's regulatory requirements. The State Office of Mine Reclamation provides assistance to the County for reclamation planning. BLM reclamation requirements are outlined in 43 CFR 3809.1 -5 c (5) and in the Records of Decision.

Equinox Gold has filed a series of updated Reclamation Plans pursuant to the permit requirements. The initial Reclamation Plan was submitted in September 1990, a revision was approved in January 1998. During the previous operations, reclamation activities included a formal revegetation research program; salvaging plants and cacti for later transplantation; 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 by the previous operators. Research topics included seed treatment and germination; plant propagation; pest management; plant salvage; soil stockpile management; plant hormone use; vesicular-arbuscular micorrhizae use; plant/water relationships; plant spacing patterns; and density, diversity, herbivory, and irrigation design. The previous operators also established native nurseries including greenhouses to support research of site-specific revegetation efforts, such as plant propagation for revegetation. The nursery-grown plants and salvaged plants grown in the Castle Mountain Mine greenhouse were transplanted onto rehabilitation areas around the Project.

As part of the licensing requirements, the previous operators established a Revegetation Review Committee in 1991. The Committee consisted of an arid lands revegetation expert, a geologist/hydrologist and an arid lands ecologist, three representatives of the environmental community and one representative each of the County, the BLM and the State Division of Mines and Geology, and the company. The Committee reviewed the annual revegetation reports filed by the company, interpreted the information contained in these reports, recommended actions to increase the success of revegetation efforts, and recommended to the County and the BLM modifications to the revegetation standards.

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, a 250,000-gallon water tank, some of the water production wells and various access roads (as part of the region's fire prevention efforts) were not reclaimed.

Site reclamation commenced during the fall of 2000 and continued through to 2005. Overburden piles were re-contoured to conform to requirements of the regulating agencies. The heap leach piles were constructed to provide vertical relief by stacking at different total heap heights. 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 undertaken. The addition of cyanide, lime or sodium hydroxide to the leach pads was discontinued in 2003. Leach pad reclamation (rinsing, detoxification and certification as waste rock) occurred until 2005 when the heap leach pad received Class "C" waste designation. Then, all solution tanks, storage basins, netting and collection ditches were reclaimed in 2005. All mining equipment was removed from the site. In 2010, the power transmission line was removed.

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 or hand-broadcast seeded with native seeds. By 2005, the previous operators had satisfied the revegetation objectives for plant density, diversity and aerial extent of cover.

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.

20.2.13 Financial Surety

Throughout the operations, the pervious operators posted financial assurance jointly with the County of San Bernardino (the "County") and the BLM, to ensure compliance with all of the conditions of the Plan of Operations and the Mine and Reclamation Plan. The bond or a portion of the bond is released to the previous operators once certification that the reclamation required is complete. The County administers the bond for both agencies. The amount of the financial surety is reviewed annually.

Equinox Gold currently holds two surety bonds in the amounts of $50,527 and $356,391 to complete reclamation requirements for the ongoing exploration activity. The BLM is also included in coverage of the bond.

21.0 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Castle Mountain Project were estimated by KCA, GRE and GLA with input from Equinox Gold. The estimated capital and operating costs are considered to have an accuracy of +/- 25% and +/- 20% respectively, and are discussed in greater detail in this Section.

The total capital cost for the Project is $488.7 million including all applicable sales tax. The project will be developed in stages with Stage 1 being constructed in Year -1 to process ROM ore from the JSLA pit. Stage 2 will be constructed in Year 3 and includes the addition of a mill and CIL circuit and owner mining fleet, along with significant capitalized mining pre-stripping activities. Sustaining capital for the expansion of the heap leach pad and replacement of equipment is considered throughout the life of the mine. Table 21-1 presents the capital requirements for the Project.

Table 21-1 - Capital Costs Summary

Description	Cost (US$)
Stage 1 Pre-Production Capital	$51,667,000
Stage 2 Expansion Capital	$294,958,000
LOM Sustaining Capital	$142,029,000
TOTAL Capital Costs Including Sales Tax	**$488,654,000**

The total life of mine operating cost for the Project is $8.43 per ton of ore processed. Table 21-2 presents the LOM average operating cost requirements for the Castle Mountain Project.

Table 21-2 - Operating Costs Lom Summary

Description	LOM Cost (US$/ton ore)
Mine	$5.79
Process & Support Services	$1.92
Site G&A	$0.72
TOTAL Operating Costs	**$8.43**

Sales tax is not included in the operating cost estimate.

21.1 Capital Cost Summary

The required capital expenditures for the Castle Mountain Project are summarized in Table 21-1. These costs are based on the design outlined in this study and are considered to have an accuracy of +/-25%. The scope of these costs includes all mining equipment, process facilities, and infrastructure for the project.

The costs presented have been estimated primarily by KCA for the process and infrastructure and GRE for mining. Costs for the heap leach facility have been estimated by GLA, which were reviewed by KCA and determined to be reasonable. All equipment and material requirements are based on the design information described in previous sections of this study. Capital cost estimates have been made primarily using budgetary supplier quotes for all major and most minor equipment items. Where supplier quotes were not available, a reasonable cost estimate was made based on supplier quotes in KCA's project files and cost guide data.

All capital cost estimates are based on the purchase of equipment quoted new from the manufacturer or estimated to be fabricated new.

Table 21-3 presents the capital cost summary by area for the Stage 1 pre-production capital requirements. Table 21-4 presents the capital summary by area for the Stage 2 mill expansion. All costs are in first quarter 2018 US dollars ($).

The detailed Capital Cost estimate and vendor quotes for selected capital items are referenced in Section 27.0 - References of this Report.

Table 21-3 - Summary of Stage 1 Pre-Production Capital Costs By Area

Plant & Infrastructure Totals Direct Costs Phase 1	Total Supply Cost	Freight & Sales Tax	Install	Grand Total
	US$	US$	US$	US$
Area 22 - Heap Leach Pad & Ponds	$10,622,000	$274,000	$3,793,000	$14,689,000
Area 28 - Carbon Adsorption	$0	$0	$0	$0
Area 29 - Carbon Desorption & Reactivation	$5,524,000	$429,000	$4,645,000	$10,598,000
Area 31 - Refinery	$0	$0	$0	$0
Area 34 - Reagents	$270,000	$40,000	$100,000	$410,000
Area 38 - Laboratory	$374,000	$19,000	$85,000	$478,000
Area 52 - Mining	$0	$0	$0	$0
Area 60 - Power	$2,561,000	$203,000	$212,000	$2,977,000
Area 62 - Water Supply, Storage & Distribution	$1,652,000	$250,000	$755,000	$2,656,000
Area 66 - Facilities	$933,000	$10,000	$226,000	$1,169,000
Area 08 - Plant Mobile Equipment	$935,000	$166,000	$0	$1,101,000
Plant & Infrastructure Total Phase 1 Direct Costs	**$22,872,000**	**$1,391,000**	**$9,816,000**	**$34,079,000**
Spare Parts	$475,000			$475,000
Sub Total with Spare Parts				**$34,554,000**
Contingency	$4,251,000			$4,251,000
Plant & Infrastructure Total Phase 1 Direct Costs with Contingency				**$38,805,000**

Plant & Infrastructure Totals Indirect Costs				Grand Total
Indirect Field Costs				$1,311,000
Indirect Field Costs Contingency				$328,000
Plant & Infrastructure Total Indirect Costs				**$1,639,000**

Plant & Infrastructure Totals Other Owner's Costs				Grand Total
Other Owner's Costs				$4,270,000
Other Owner's Costs Contingency				$1,068,000
Total Other Owner's Costs				**$5,338,000**

Initial Fills	**$225,000**
Sub Total Phase 1 Cost Before EPCM & Mining	**$46,007,000**
EPCM	**$582,000**
Mining & Support Equipment	**$1,514,000**
Mining & Process Preproduction + Working Capital	**$3,564,000**
Sub Total Phase 1 Project Capital Cost	**$51,667,000**
Sustaining Cost with Contingency	**$0**
TOTAL COST	**$51,667,000**

Table 21-4 - Summary of Stage 2 Expansion Capital Costs By Area

Plant & Infrastructure Totals Direct Costs Phase 2	Total Supply Cost	Freight & Sales Tax	Install	Grand Total
	US$	US$	US$	US$
Area 13 Crushing	$4,343,000	$605,000	$395,000	$5,342,000
Area 15 - Crushed Ore Stockpile & Reclaim	$833,000	$144,000	$404,000	$1,381,000
Area 16 - Grinding	$5,706,000	$528,000	$953,000	$7,186,000
Area 17 - Gravity Separation	$1,096,000	$193,000	$1,656,000	$2,944,000
Area 18 - Grinding Thickener & Process Solution	$852,000	$101,000	$334,000	$1,287,000
Area 19 - CIL	$2,576,000	$286,000	$1,129,000	$3,992,000
Area 20 - Tailings Thickener	$643,000	$88,000	$272,000	$1,003,000
Area 21 - Filtration, Clarification & Detox	$13,090,000	$1,705,000	$1,871,000	$16,665,000
Area 22 - Heap Leach Pad & Ponds	$8,709,000	$560,000	$3,165,000	$12,433,000
Area 28 - Carbon Adsorption	$0	$0	$0	$0
Area 29 - Carbon Desorption & Reactivation	$3,449,000	$270,000	$2,842,000	$6,562,000
Area 31 - Refinery	$0	$0	$0	$0
Area 34 - Reagents	$340,000	$50,000	$245,000	$635,000
Area 38 - Laboratory	$1,471,000	$55,000	$582,000	$2,109,000
Area 52 - Mining	$0	$0	$0	$0
Area 60 - Power	$12,561,000	$1,111,000	$257,000	$13,929,000
Area 62 - Water Supply, Storage & Distribution	$1,404,000	$245,000	$3,960,000	$5,609,000
Area 65 - Compressed Air	$1,151,000	$204,000	$490,000	$1,845,000
Area 66 - Facilities	$1,010,000	$79,000	$188,000	$1,278,000
Area 08 - Plant Mobile Equipment	$3,922,000	$696,000	$0	$4,618,000
Plant & Infrastructure Total Phase 2 Direct Costs	**$63,156,000**	**$6,920,000**	**$18,742,000**	**$88,817,000**
Spare Parts	$1,162,000			$1,162,000
Sub Total with Spare Parts				**$89,979,000**
Contingency	$10,964,000			$10,964,000
Plant & Infrastructure Total Phase 2 Direct Costs with Contingency				**$100,943,000**

Plant & Infrastructure Totals Indirect Costs	**Grand Total**
Indirect Field Costs	$2,085,000
Indirect Field Costs Contingency	$521,000
Plant & Infrastructure Total Indirect Costs	**$2,606,000**

Plant & Infrastructure Totals Other Owner's Costs	**Grand Total**
Other Owner's Costs	$1,979,000
Other Owner's Costs Contingency	$495,000
Total Other Owner's Costs	**$2,474,000**

Initial Fills	**$323,000**
Sub Total Phase 2 Cost Before EPCM & Mining	**$106,346,000**
EPCM	**$11,200,000**
Mining & Support Equipment Expansion Capital - Year 3	**$80,019,000**
Mining Pre-Stripping - Year 3	**$46,828,000**
Mining & Support Equipment Expansion Capital - Year 4	**$37,025,000**
Mining & Process Preproduction + Working Capital	**$13,541,000**
Sub Total Phase 2 Project Capital Cost	**$294,958,000**
TOTAL COST	**$294,958,000**

21.1.1 Mining Capital Costs

Capital costs for the project were estimated using budgetary quotes from:

- Caterpillar and Komatsu, including Maintenance and Repair Contracts (MARC);
- Quotes for explosives, tires/chains, diesel fuel, and for light equipment;
- InfoMine USA, Inc. (InfoMine) resources, including Mining Cost Service 2018 (InfoMine, 2018) and 2017 Mine and Mill Equipment Cost Guide (InfoMine, 2017); and,
- GRE experience and data from similar mining projects.

21.1.1.1. Capital Cost Estimate

Because the JSLA pit will be mined by a contractor, the majority of the capital costs required for the remainder of the project occur during year 3, i.e., during the year preceding the start of owner operations. However, production equipment capital costs also occur throughout the project life as both new and replacement pieces of equipment are required. The capital costs by year are summarized in Table 21-5. Major components of the capital costs are discussed below.

21.1.1.2. Production Equipment Capital Costs

Anticipated production equipment includes Komatsu 730-E haul trucks, Komatsu PC4000 shovels, Komatsu D375A-6 track dozers, and P&H 77XD drills. The quantity of each type of equipment was calculated based on required operating hours and equipment effective operating hours per year.

21.1.1.3. Support Equipment Capital Costs

Anticipated support equipment includes Komatsu WD600-6 wheel dozers, Komatsu WA1200-6 wheel loaders, water trucks, lube trucks, mechanics trucks, graders, Komatsu 330 excavators, backhoes, a small crane, light plants, dewatering pumps, and 4x4 pickup trucks.

The quantity of each type of equipment was calculated based on knowledge of similarly sized mining projects. All support equipment capital costs occur in year 3 except for 10 of the pickup trucks, which occur in year 1.

21.1.1.4. Equipment Replacement

Production and support equipment replacement quantities were calculated based on expected life of the equipment and scheduled operating hours.

21.1.1.5. Office Equipment and Software

Expected capital costs for software, computers, survey equipment, and office equipment were estimated based on experience with similarly sized mining projects. Initial capital costs for these items occur in year 1 since the owner will need these equipment even when the contractor is operating. The office equipment and software capital costs were estimated as follows:

- Computers - $50,000
- Software - $150,000 in pre-production plus $15,000 per year for the life of mine
- Survey equipment - $25,000
- Office equipment - $100,000

21.1.1.6. Infrastructure

Infrastructure includes haul road construction, construction of a crossing over an arroyo, and pit pioneering.

21.1.1.7. Other Mining Capital Cost Items

Other capital cost items in the estimate include:

- Mechanic shop - $3 million
- Fuel lube bay - $1.5 million
- Dispatch system - $325,000
- Mining Sustaining capital - $2.5 million in Year 2 and $2.7 million in Year 3
- Mining Working capital - $5.8 million in Year 1 (recovered in the cash flow at the end of the project)
- Mining Capital contingency – 5%

Table 21-5 – Summary of Mining Capital Costs (000s)

Description	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Total
Production Equipment	$0	$0	$0	$57,700	$34,750	$20,250	$4,000	$0	$13,000	$0	$0	$3,250	$0	$0	$0	$0	$0	$132,950
Support	$231	$0	$0	$14,344	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$14,575
Capital Replacement	$0	$0	$0	$0	$230.5	$462	$0	$0	$231	$3,753	$0	$0	$4,588	$0	$0	$0	$0	$9,263
Shop	$0	$0	$0	$3,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$3,000
Fuel/Lube/Wash	$500	$0	$0	$1,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$1,500
Office Equipment & Software	$325	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$15	$565
Infrastructure	$386	$0	$500	$150	$500	$1,069	$115	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$2,720
Contingency	$72	$1	$26	$3,810	$1,775	$1,090	$207	$1	$662	$188	$1	$163	$230	$1	$1	$1	$1	$8,229
Total Capital	**$1,514**	**$16**	**$541**	**$80,019**	**$37,270**	**$22,885**	**$4,337**	**$16**	**$13,908**	**$3,956**	**$16**	**$3,428**	**$4,833**	**$16**	**$16**	**$16**	**$16**	**$172,801**

21.1.2 Process & Infrastructure Capital Costs

Process and infrastructure costs have been estimated by KCA with input from GLA for the heap leach facility. Capital cost estimates have been made using budgetary quotes for all major and most minor equipment items. Where supplier quotes were not available a reasonable cost estimate was made based on costs in KCA's project files and cost guide information.

21.1.3 Process Cost Basis

The capital costs for each process area shown in the capital cost summary tables presented in Table 21-3 and Table 21-4 have been built up from the following disciplines, where applicable:

- major earthworks (includes pad/pond liner);
- civils (concrete);
- structural steel;
- platework;
- mechanical equipment;
- piping;
- electrical;
- instrumentation; and
- infrastructure.

Supply, freight, sales tax and installation costs are included in the capital cost buildup for each discipline and are discussed in the following sections.

Engineering, procurement, and construction management (EPCM), indirect costs, other owner's costs, spare parts and initial fills inventory are added to the total direct costs.

21.1.4 Freight

The process equipment freight costs are estimated based on loads as bulk freight at an average percentage of mechanical equipment costs. The factor of 10% used is based on experience with similar projects in California/Nevada and the United States.

21.1.5 Sales Tax

Sales tax for San Bernardino County in California is 7.75% and has been applied to the supply cost of all equipment and materials.

21.1.6 Installation

Installation estimates for items not quoted turn-key from the supplier have been estimated based on the equipment type and include all installation labor and equipment usage. Installation costs are based on KCA's experience from recent projects and includes $55 per hour for labor, $25 for light vehicles, $45 per hour for forklifts and $157 per hour for cranes. An allowance of $5 per hour for small equipment and tools has also been included.

Installation hours for fork lifts and light vehicles assume six-man crews. Crane hours have been estimated on an item-by-item basis. Excluding crane use, the average installation cost is approximately $71.67 per hour. This basis was applied to installation of mechanical equipment, piping, electrical, and instrumentation.

21.1.7 Major Earthworks & Liner

Earthworks and liner quantities were estimated by GLA for the heap leach pad and solution ponds and by KCA for the remaining process and infrastructure areas based on the preliminary site layout and heap leach facility design.

Unit rates for earthworks and lining activities were estimated by GLA based on recent contractor costs in GLA's files. These unit rates were compared by KCA to those from other recent Nevada and California projects that KCA has recently worked on, and were deemed to be reasonable estimates. Earthworks unit rates are presented in Table 21-6.

Table 21-6- Earthworks & Liner Unit Rates

Description	Unit	Unit Cost (US$)
Clearing & Tree Removal	AC	$4,000.00
Topsoil Strip & Stockpile	CY	$3.00
Subgrade Prep	SF	$0.11
Cut Type B (Cut to fill)	CY	$4.00
Cut Type B (Cut to stockpile)	CY	$2.00
Structural Fill (95% Standard Proctor)	CY	$4.00
80 mil LLDPE Liner (supply & Install)	SF	$0.80
80 mil HDPE Liner (supply & Install)	SF	$0.80
60 mil HDPE Liner (supply & Install)	SF	$0.60
HDPE Geonet (supply & Install)	SF	$0.50

21.1.8 Civils

Civils include detailed earthworks and concrete. Concrete quantities have been estimated from takeoffs based on drawings and calculations where possible, or estimated from previous similar equipment installations. Average supply and placement rates for concrete have been estimated at $783 per cubic yard for f'c 250 concrete and $1,006 per cubic yard for f'c 300 concrete are based on recent quotations from local contractors. The concrete unit costs include all formwork, footing excavation, concrete supply, rebar and curing costs.

21.1.9 Structural Steel

Structural steel, which includes miscellaneous tank platforms, hand rails, grating and the mill and filter building structures for Stage 2 have been estimated from takeoff lists and quantities for similar items form recent projects based on layout drawings and equipment loads. Unit costs for steel, including installation labor and equipment requirements, were provided by a local contractor and are presented in Table 21-7.

Table 21-7 - Structural Steel Unit Rates

Description	Unit	Unit Cost (US$)
Grating	SF	$1.95
Structural Steel	lb	$2.34
Handrails	LF	$150
Stair Treads	ea	$40
Roof Sheeting	SF	$3.25

Structural steel for the remaining areas such as the recovery plant, laboratory, and truckshop were included in the overall turnkey package quotations and cost estimates for these areas.

21.1.10 Platework

Platework includes the supply and installation of steel tanks, bins and chutes. Costs were estimated primarily from budgetary supplier quotes or estimated based on recent similar tanks from recent KCA projects. Where similar tanks and quotes were not available, costs were estimated based on material takeoffs of the tank based on its preliminary design.

21.1.11 Mechanical Equipment

Costs for all mechanical equipment are based on an equipment list developed for all major process areas of the Project. Costs for most major equipment items are based on budgetary quotes from vendors for new equipment. Costs for minor equipment items are based on budgetary quotes, costs found in supplier’s catalogs, KCA’s in-house database of recent quotes or mining cost guide data.

Installation estimates are based on equipment type. Each type of equipment is assigned an installation rate of hours per unit cost allowing the total installation hours to be estimated.

21.1.12 Piping and Valves

Piping, fittings, and valve costs for major areas including water supply and distribution, heap leach irrigation and drainage, and fire water distribution are based on material take offs and vendor budget quotes for supply of major items. Installation costs are calculated estimates based on factored installation time and assumed hourly rates of contractors.

Piping, fittings, and valve costs for other areas such as those in the mill complex and ADR plant have been estimated on a percentage basis of the mechanical equipment costs or included in the vendor's scope of supply.

21.1.13 Electrical

Major electrical items such as transformers, substations, power lines, motor control centers (MCCs) and variable frequency drives (VFDs) have been included in the electrical cost estimate, based on preliminary electrical layouts and site distribution single lines. Costs for these items are based on budgetary supplier quotes or recent project data.

Costs for the power line installation in Stage 2 are also included under the electrical discipline. A brief description of the construction of the power line is included under Section 18.2.2. Costs for rebuilding of upstream NVE infrastructure and updating of the Searchlight substation are based on NV Energy's preliminary study. Costs for construction of 15.5 miles of new power line and the main substation on-site are based on KCA's estimates from project data and outside consultants.

Miscellaneous electrical costs, including electrical supply and installation from MCCs to individual motorized equipment, have been estimated as a percentage of the mechanical equipment supply cost for each process area.

21.1.14 Instrumentation

Instrumentation costs are estimated as a percentage of the mechanical equipment supply cost for each process area.

21.1.15 Spare Parts

Spare parts for most mechanical equipment have been estimated based on 4% of the mechanical equipment supply costs for each process area, plus freight costs and taxes.

Spare parts for the ADR plant in Stage 1 and Stage 2 were quoted as part of the turnkey supply package.

21.1.16 Infrastructure Capital Costs

Costs for the site infrastructure for the Castle Mountain Project have been estimated based on supplier budgetary quotes or information from recent similar projects. Where appropriate, it has been assumed that some existing site infrastructure will be used for construction and operations, or an allowance to modify or repair the existing infrastructure was made. Existing infrastructure includes site roads and well pumps for Stage 1 operation.

New infrastructure for the project for Stage 1 includes a starter laboratory, mine and administration offices, gate house, process office trailers, site fencing around the heap leach, septic system, and communications system. Infrastructure for Stage 2 includes the full-service laboratory, new water well construction, additional site fencing, and additional septic systems.

Other infrastructure items, including electrical power supply and distribution and water supply and distribution (outside of well construction) are included in other discipline cost estimates.

21.1.17 Indirect Capital Costs

Indirect costs include costs for items such as equipment rentals, hotels and lodging, temporary construction facilities, tools, quality control, survey support, warehouse and fenced yards, consumables such as fuel and power, security, and vendor commissioning support for certain equipment items. These costs have been estimated based on KCA’s experience with recent projects in Nevada and California. Indirect costs for Stage 1 are based on an 8-month construction schedule; indirect costs for Stage 2 are based on a 13-month construction schedule.

21.1.18 Initial Fills

Initial fills consist of critical consumable items purchased and stored on site at the onset of operations. Initial fills items include sodium cyanide (NaCN), pebble lime, activated carbon, antiscalant, caustic soda, hydrochloric acid, fluxes (silica, borax, niter, and soda ash), etc. This inventory of initial fills ensures adequate consumables are available for plant commissioning. Details of the initial fills for Stages 1 and 2 are presented in Table 21-8 and Table 21-9, respectively.

Table 21-8 - Stage 1 Initial Fills

Item	Basis	Needed Weight lb or gal	Quantity to Order lb or gal	Unit Price (Including Freight) US$	Total Cost US$
NaCN	Full Tank	23,490	90,171	1.32	$118,575
Lime (pebble)	Full Silo	400,000	400,000	0.10	$38,000
Carbon	Full Circuit	35,000	40,000	0.88	$35,200
Antiscalant	2 weeks	825	8,847	1.04	$9,201
Caustic Soda	Full Tank	5,000	31,899	0.23	$7,309
Hydrochloric Acid	Full Tank	5,000	13,424	0.31	$4,200
Propane	1 month	8,108	10,000	1.23	$12,250
Flux					
SiO2	1 month	121	150	0.37	$56
Borax	1 month	193	200	1.32	$263
Niter	1 month	97	100	2.22	$222
Soda Ash	1 month	73	100	0.34	$34
				TOTAL	**$225,310**

Table 21-9 - Stage 2 Initial Fills

Item	Basis	Needed Weight lb or gal	Quantity to Order lb or gal	Unit Price (Including Freight) US$	Total Cost US$
NaCN				1.32	$0
Lime (pebble)	Full Silos	60,000	60,000	0.10	$5,700
Mill Media	Full Charge + 25%	204,026	200,000	0.60	$120,000
Carbon	Full Circuit	210,000	210,000	0.88	$184,800
H2SO4	2 weeks	56,133	60,000	0.15	$8,700
Hyrogen Peroxide	2 weeks	10,368	12,000	0.30	$3,600
				TOTAL	**$322,800**

21.1.19 Engineering, Procurement and Construction Management

The estimated cost for engineering, procurement and construction management (EPCM) for the development, construction and commissioning of the Castle Mountain Project are based on a percentage of the direct capital cost. EPCM costs during stage one are expected to be minimal and are estimated at $582,000. EPCM during Stage 2 is estimated at 11% of the direct costs, or $11.2 million.

The EPCM costs cover the services and expenses for the following areas:

- Project Management;
- Detailed engineering;
- Engineering support;
- Procurement;

- Construction Management;
- Commissioning; and
- Vendor representatives.

21.1.20 Owners Costs

Owner's costs are considered which include the owner's construction team and other G&A costs during construction.

Pre-production G&A costs include office operating expenses, legal fees, phones/internet service, office supplies, insurance, IT services and computers, travel, community assistance and environmental expenses. Labor G&A costs include office, safety, security and warehouse staff.

Owner's costs have been estimated at $5.3 million for Stage 1 and $2.5 million for Stage 2 by KCA with input from Equinox Gold.

21.1.21 Contingency

Contingency has been applied to each discipline ranging from 10% to 20% of the direct cost. The overall process direct contingency is estimated at $4.3 million or 12.3% of the direct costs for Stage 1 and $10.0 million or 12% of the direct costs for Stage 2.

Contingency for indirect and owner's construction costs have been applied at 25% of these costs.

21.1.22 Working Capital

Working capital for the Project is estimated to be $3.6 million for Stage 1 and $13.5 million for Stage 2. The working capital is the capital required for operations before any revenue is produced by the mine and is based on the operating costs for the mine, process and G&A costs for the Project. The working capital is based on 60 days of operation for Stage 1 and 30 days of operation for Stage 2.

21.1.23 Process Sustaining Capital

Sustaining capital is required throughout the life of the project for the expansion of the heap leach pad and replacement of process support equipment. LOM sustaining capital is presented in Table 21-10. The Mill expansion during Year 3 of operations is not considered as part of the project sustaining capital.

Table 21-10 - Process Sustaining Capital By Year

Cost Area	Sustaining Capital (US$)
Mine equipment	$51,672,000
Leach pad	$49,815,000
Plant mobile equipment	$6,962,000
EPCM	$1,993,000
Construction indirects	$1,993,000
Mining and Process Contingency	$9,385,000
Reclamation and Closure	$20,210,000
Total	**$142,029,000**

Mining equipment sustaining capital includes replacement of mine production and support equipment, software and minor infrastructure items. Major sustaining capital expenses for the mine occur in Year 5 ($22.9 million), Year 6 ($4.3 million), Year 8 ($13.9 million), Year 9 ($3.9 million), Year 11 ($3.4 million), Year 12 ($4.8 million), with the remaining minor capital for software and small infrastructure items.

Leach pad sustaining costs occur with major pad expansion stages 1B, 2, 3, and 4, in Year 2 ($9.7 million), Year 4 ($12.9 million), Year 7 ($14.9 million), and Year 10 ($12.3 million) respectively. These totals do not include EPCM, indirects, or contingency.

Plant mobile equipment sustaining costs include replacement of light vehicles and heavy equipment for the process such as dozers and maintenance equipment. These costs occur in Year 10 ($5.7 million) and Year 14 ($0.9 million).

EPCM and construction indirect costs were applied to leach pad expansion activities and were estimated at 4% of direct costs. A contingency of 12% of direct costs was applied to leach pad expansions and plant mobile equipment, and a contingency of 5% of direct costs was applied to replacement mine equipment.

21.1.24 Reclamation & Closure Costs

Reclamation and closure costs include rinsing of the heap (minimal labor and operation of pumping systems) during Years 17 and 18, and reclamation construction activities including heap smoothing and contouring and revegetation, construction of drainage basins, and post-closure environmental monitoring.

Heap rinsing is included under process operating costs in Year 17 and 18. Reclamation construction activities include $3.44 million in Year 17, $3.44 million in Year 18, and $13.3 million in Year 19.

21.1.25 Exclusions

The following capital costs are excluded from KCA's scope of supply and estimate:

- Finance charges and interest during construction.
- Escalation costs.

21.2 Operating Cost Summary

Operating costs for the Castle Mountain Project have been based on the information presented in earlier sections of this study. Mine operating costs have been developed by GRE and are estimated to average U$5.79/ton ore processed over the life of the mine. Process, laboratory and support operating costs have been estimated by KCA with a LOM average of $1.92/ton ore. G&A costs have been developed by KCA with input from Equinox Gold and are estimated at $0.72/ton ore over the life of the mine. Mining and process operating costs are presented in more detail in Sections 21.2.1 and 21.2.2, respectively. A summary of projected operating costs is presented in Table 21-11.

Table 21-11 - Operating Costs LOM Summary

Description	LOM Cost (US$/ton ore)
Mine	$5.79
Process & Support Services	$1.92
Site G&A	$0.72
TOTAL Operating Costs	**$8.43**

Operating costs for all areas of the project have been estimated from first principles. Labor costs are estimated using project-specific staffing, salary, wage, and benefit requirements. Unit consumptions of materials, supplies, power, water, and delivered supply costs are also estimated.

The operating costs presented are based upon ownership of all project production equipment and site facilities as well as the owner employing and directing all operating, maintenance and support personnel.

The operating costs have been estimated and are presented without any added contingency allowances. The mine, processing, support and general and administrative operating costs are considered to have an accuracy range of +/-20%.

Operating cost estimates have been based upon information obtained from the following sources:

- GRE mining costs;
- Project metallurgical test work and process engineering;
- Budgetary quotations from potential suppliers for reagents and other consumable and material items;
- Recent KCA project file data; and
- Experience of KCA staff with other similar operations.

Where specific data do not exist, cost allowances have been based upon consumption and operating requirements from other similar properties for which reliable data exists. Freight costs have been estimated where delivered prices were not available.

All costs are presented in first quarter 2018 US dollars (US$).

21.2.1 Mining Operating Costs

The operating costs for the Castle Mountain Project were estimated for open pit mining, drilling and blasting, contractor operations, manpower, processing, G&A, and contingency. The operating costs were estimated based on InfoMine 2017 Mining Cost Service data, estimated productivities, and recent GRE experience with other mines.

A summary of the estimated operating costs is provided in Table 21-12.

21.2.1.1. Production Equipment Operating Costs

Mining production equipment hours were estimated from the equipment productivity estimates, the production schedule, and the number of pieces of equipment required. Mine Equipment Selection

Komatsu performed a fleet study to determine the most economically viable equipment solution for the Castle Mountain mine. The result recommended a 200-ton truck (Komatsu 730E) paired with a 29 cubic yard shovel (Komatsu PC4000-6). Compared to a 100-ton truck and a 150-ton truck, the 730E had the greater productivity advantage despite the higher cost. The overall savings in the quantity of equipment meant an overall, lower cost.

21.2.1.2. Mine Equipment Productivity

Haul routes were estimated for each year. Each rock destination – ROM ore, mill ore, and waste – had a designated route for each pit active during a given year. Each route was mirrored and an operating time was calculated based on the distance and slope for each segment of each haul. The times were calculated based on the rimpull and retarding graphs of Komatsu’s 730E 200-ton haul truck. This was the basis for the ore and waste productivity used to calculate total operating hours per year for the truck fleet and the total demand for calculating truck units for necessary capital. Table 21-13 shows the productivity for a truck hauling ROM ore, mill Ore, or waste to a given destination in tons/hour throughout the mine life.

21.2.1.3. Blasting

The drilling and blasting costs include explosive costs by explosive contractor quote at $0.141 per ton mined.

21.2.1.4. Mine Manpower

Manpower for the mine and processing facility includes hourly-rate employees and salaried employees, who are generally superintendents and professional personnel. The number of required equipment operators was estimated using the quantities of equipment required, the quantity of personnel per piece of equipment, and the number of shifts per day. Numbers of required processing and salaried personnel were estimated based on GRE's experience. A burden factor of 40% was added to all labor. The burden includes fringe benefits, holidays, vacation and sick leave, insurances, etc.

Table 21-12– Summary of Mining Operating Costs ($'000s)

Description	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	Total
Mine Ops	$348	$395	$37,020	$57,382	$61,907	$52,858	$61,828	$72,024	$64,934	$68,572	$70,348	$62,182	$58,420	$46,015	$30,887	$15,635	$0	$0	$760,755
Hourly Labor	$0	$0	$13,065	$18,293	$21,278	$19,617	$21,306	$23,840	$22,151	$22,968	$23,390	$21,246	$20,373	$17,807	$14,368	$10,048	$770	$770	$271,289
Salaried Labor	$798	$798	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,814	$2,309	$1,000	$0	$0	$38,673
Drilling-Blasting	$0	$0	$9,081	$15,913	$12,162	$11,376	$13,548	$14,057	$12,623	$12,949	$12,483	$10,555	$9,388	$6,354	$3,917	$1,410	$0	$0	$145,817
Contractor Ops	$13,564	$15,416	$22,785	$39,067	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$90,832
Total Mine Operating Cost	**$14,711**	**$16,609**	**$84,764**	**$133,470**	**$98,160**	**$86,664**	**$99,495**	**$112,736**	**$102,522**	**$107,303**	**$109,035**	**$96,797**	**$90,995**	**$72,991**	**$51,481**	**$28,093**	**$770**	**$770**	**$1,307,366**

Table 21-13– Truck Haulage Productivity by Year, Pit, and Rock Destination in Tons/Hour

Haul Route	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
JSLA ROM Ore	-	-	905	704	648	545	485	433	379	-	-	-	-	-	-	-
JSLA Mill Ore	-	-	-	1,190	1,071	835	718	620	522	-	-	-	-	-	-	-
JSLA Waste	-	-	840	771	621	575	457	361	326	-	-	-	-	-	-	-
Jumbo ROM Ore	-	-	768	658	510	-	-	-	407	357	-	-	-	-	-	-
Jumbo Mill Ore	-	-	1,331	1,063	740	-	-	-	578	489	-	-	-	-	-	-
Jumbo Waste	-	-	914	716	494	-	-	-	572	465	-	-	-	-	-	-
Oro Belle ROM Ore	-	-	-	-	-	535	526	455	407	351	393	388	-	-	-	-
Oro Belle Mill Ore	-	-	-	-	-	813	810	666	577	477	567	498	-	-	-	-
Oro Belle Waste	-	-	-	-	-	1,326	1,588	1,011	571	454	307	290	-	-	-	-
East Ridge ROM Ore	-	-	-	-	-	-	490	477	-	-	599	478	576	462	-	-
East Ridge Mill Ore	-	-	-	-	-	-	-	-	-	-	1,130	781	1,117	774	-	-
East Ridge Waste	-	-	-	-	-	1,043	1,304	1,123	-	-	793	538	643	513	-	-
South Domes ROM Ore	-	-	-	-	840	820	711	654	605	553	509	443	387	344	297	251
South Domes Mill Ore	-	-	-	-	-	-	-	-	-	-	847	693	575	491	406	327
South Domes Waste	-	-	-	-	777	866	637	512	1,027	842	632	488	412	367	373	305

21.2.1.5. Contractor Operations Costs

GRE prepared a request for proposal (RFP) for the stage 1, JSLA pit backfill excavation. Four companies provided budgetary bids for this mining. The RFP requested that the contractors provide three separate quotes for conducting mining and hauling operations at an ore mining rate of 15,000 tpd, with one quote assuming a stripping ratio of 1:1 ore to waste, one quote assuming a stripping ratio of 1 to 0.5 ore to waste, and one quote assuming a stripping ratio of 1:0 ore to waste.

This work will take approximately 3 ½ years to complete. The bids were for conducting all mining operations except drilling for grade control. Blasting will not be needed. GRE used an average of these four rates and experience with similar working mines to estimate the contractor operating costs for the JSLA pit. These costs are summarized in Table 21-14.

Table 21-14– Summary of Contractor Operations Costs

Description	Year 1	Year 2	Year 3	Year 4	Total
Contractor Ops Rate ($/ton)	$2.30	$2.30	$2.30	$1.43	
Contractor Ops Cost ($‘000s)	$13,560	$15,416	$22,785	$39,067	$90,832
Total Contractor Operations Costs ($‘000s)	$13,560	$15,416	$22,785	$39,067	$90,832

21.2.2 Process and Support Services Operating Costs

Process operating costs requirements for the Castle Mountain Project were estimated by KCA based upon unit consumption and, where possible, have been broken down by area. The overall average life of mine operating cost for the process, lab, support services and process area costs during reclamation is estimated at $1.92 per ton ore. The LOM average cost for ROM ore is estimated at $1.33 per ton of ROM ore processed on the heap leach pad. The average LOM cost for ore processed in the mill is estimated at $12.62 per ton mill ore.

Table 21-15 Presents the process and support service operating costs by area and ore type (ROM or mill). Shared costs, including labor, laboratory and support services, and recovery and refining costs have been split between the ROM and mill ores based on tons or stripping batches, as appropriate.

Table 21-15 - Summary Process & Support Operating Costs

Process Operating Costs - ROM Ore	LOM Total ($/ton)
Labor - All Process Areas	$0.42
Area 22 - Heap Leach Pad & Ponds	$0.11
Area 28 - Carbon Adsorption	$0.02
Area 29 - Carbon Desorption & Reactivation	$0.05
Area 31 - Refinery	$0.02
Area 34 - Reagents	$0.57
Area 38 - Laboratory	$0.05
Area 60 - Power	$0.02
Area 62 - Water Supply, Storage & Distribution	$0.02
Area 66 - Facilities, Area 08 - Plant Mobile Equipment	$0.05
TOTAL - ROM Ore	**$1.33**

Process Operating Costs - Mill Ore	LOM Total ($/ton)
Labor - All Process Areas	$3.31
Area 13 Crushing	$1.04
Area 15 - Crushed Ore Stockpile & Reclaim	$0.06
Area 16 - Grinding	$2.84
Area 17 - Gravity Separation	$0.10
Area 18 - Grinding Thickener & Process Solution	$0.10
Area 19 - CIL	$0.33
Area 20 - Tailings Thickener	$0.07
Area 21 - Filtration, Clarification & Detox	$1.22
Area 29 - Carbon Desorption & Reactivation	$0.21
Area 31 - Refinery	$0.04
Area 34 - Reagents	$2.15
Area 38 - Laboratory	$0.23
Area 60 - Power	$0.02
Area 62 - Water Supply, Storage & Distribution	$0.02
Area 65 - Compressed Air	$0.24
Area 66 - Facilities Area 08 - Plant Mobile Equipment	$0.64
TOTAL Milled Ore	**$12.62**

21.2.3 Reagents and Consumables

Reagents and other consumable items, including lime, cyanide, etc., are required to recover gold and cast it into doré.

Reagent consumptions are derived from metallurgical test work and from the design criteria information. Other costs have been estimated by KCA based on past experience with similar projects. Table 21-16 Shows the annual consumption and storage capacity for all major process consumables.

Table 21-16 - Process Consumable Items

Item	Form	Avg. Annual Usage Stage 1 (ton/yr)	Avg. Annual Usage Stage 2 (ton/yr)
Sodium Cyanide	Bulk Delivery, Liquid	511	2,025
Lime (Pebble)	Bulk Delivery, Truck	6,124	19,221
Activated Carbon	Supersacks	80	949
Flocculant	Dry Solid Sacks	N/A	70
Antiscalant	Liquid Tote Bins	86	351
Sodium Hydroxide (Caustic)	Bulk Delivery, Truck	26	109
Hydrochloric Acid	Liquid Tote Bins	13	53
Mill Media		N/A	983
Mill Liners		N/A	98
Fluxes	Dry Solid Sacks	3	15
Propane	Bulk Delivery, Liquid	332,000	3,912,000

Operating costs for these items have been distributed based on tonnage and gold production, or smelting batches, as appropriate.

21.2.4 Sodium Cyanide

Sodium cyanide consumption is estimated at 0.2 lbs/ton ore and 1.0 lbs/ton ore for the ROM and mill operations, respectively. The supply cost delivered to site is quoted at $1.32 per pound. Sodium cyanide is delivered to site as a bulk liquid and 30% concentration by weight.

21.2.5 Lime

Pebble lime is used for both the ROM heap leach and mill circuits for pH control. Pebble lime is added at 2.4 lbs/ton for ROM ore and 1.4 lbs/ton for mill ore. The supply cost of lime delivered to site is quoted at $0.10 per lb.

21.2.6 Carbon

Carbon is used for the adsorption of gold and silver from pregnant solution for the heap and CIL circuits. Carbon consumption is estimated at 4% per strip batch due to attrition. Carbon supply cost is estimated at $0.88 per lb based on recent supplier quotes in KCA's database.

21.2.7 Flocculant

Flocculant is added to the grinding thickener, tailings thickener and clarifier at a rate of 0.15 to 0.25 lbs per ton ore during Stage 2 operations. A delivered price of $1.90 per lb is estimated based on recent supplier quotes in KCA's database.

21.2.8 Antiscalant Agent

Antiscalant agent is added to the barren and pregnant solution tanks to prevent the build up of scale in the process solution pipes. Antiscalant addition ranges between 0 to 20 ppm depending on water quality. The supply cost of antiscalant delivered to site is estimated at $1.04 per lb based on recent supplier quotes in KCA's data base.

21.2.9 Caustic Soda (NaOH)

Caustic is delivered to site as a liquid at 50% concentration by weight. Caustic is used in the ADR and is consumed in the strip and acid wash circuits. Caustic consumption is based on a 2% caustic strip solution with approximately one third of the solution being discarded each strip. Caustic supply cost is estimated at $0.23 per lb based on recent supplier quotes in KCA's database.

21.2.10 Hydrochloric Acid

Hydrochloric acid is used in the acid wash circuit to remove scale from the carbon which inhibits the adsorption of gold and silver. Hydrochloric acid consumption is estimated at 145 gallons per strip with an estimated supply cost of $0.31 per lb based on recent supplier quotes in KCA's database.

21.2.11 Mill Media

Mill media includes replacement balls for the ball mill in Stage 2 operations. Mill media is consumed at an estimated rate of 2.1 lbs per ton ore and is suppled at an estimated cost of $0.60 per lb.

21.2.12 Smelting Fluxes

Smelting fluxes include borax, silica, niter and soda ash. Flux consumption is estimated at 1 lbs of flux per lb of metal bearing sludge from the electrowinning. Flux supply cost is estimated at $1.11 per lb and is based on recent quotes in KCA's database and an assumed ratio for the flux mixture.

21.2.13 Hydrogen Peroxide and Sulfuric Acid

Hydrogen peroxide and Sulfuric acid are used to generate Caro's acid for the destruction of cyanide in the mill tailings in Stage 2 operations. On average, hydrogen peroxide and sulfuric acid are consumed at a rate of 31 lbs per hour and 167 lbs per hour with quoted supply costs of $0.30 per lb and $0.15 per lb, respectively.

21.2.14 Propane

Propane is required for the process kiln, boiler and smelting furnace. In Stage 1, propane is also used to power the generator plant near the ADR plant. Propane will be delivered to site by truck and stored in a 30,000-gallon horizontal tank at the ADR. Propane supply is estimated at $1.23 per gallon.

21.2.15 Process Power

Power usage for the process and process related areas was derived from estimated connected loads assigned to powered equipment from the mechanical equipment list. Equipment power demands under normal operation were assigned and coupled with estimated on-stream times to determine the average energy usage and cost.

The total attached power for the process and infrastructure is estimated at 1.6 MW for Stage 1 and 9.0 MW for Stage 2, with average draws of 0.8 MW and 6.0 MW, respectively. During Stage 1 power will be supplied by natural gas gensets (running on propane) for the process areas and small diesel gensets at the water well pumps with estimated power generation costs of $0.15/kWh and $0.18/kWh, respectively, not including costs for generator maintenance. Line power will be provided during Stage 2 with an average cost of $0.067/kWh based on preliminary information from NV Energy.

Power requirements are summarized in Table 21-17.

Table 21-17 - Process Power and Consumption - Average

Project Area	Stage 1 kWh/year	Stage 2 kWh/year
Area 13 Crushing		4,250,000
Area 15 - Crushed Ore Stockpile & Reclaim		127,000
Area 16 - Grinding		13,272,000
Area 17 - Gravity Separation		762,000
Area 18 - Grinding Thickener & Process Solution		646,000
Area 19 - CIL		2,566,000
Area 20 - Tailings Thickener		246,000
Area 21 - Filtration, Clarification & Detox		6,033,000
Area 22 - Heap Leach Pad & Ponds	3,473,000	14,005,000
Area 28 - Carbon Adsorption	48,000	129,000
Area 29 - Carbon Desorption & Reactivation	1,276,000	1,501,000
Area 31 - Refinery	799,000	970,000
Area 34 - Reagents	56,000	114,000
Area 38 - Laboratory	641,000	1,148,000
Area 62 - Water Supply, Storage & Distribution	956,000	1,838,000
Area 65 - Compressed Air		3,971,000
Area 66 - Facilities	124,000	420,000
TOTAL	**7,372,000**	**51,999,000**

21.2.16 Process Labor

Staffing requirements for the process personnel have been estimated by KCA with input from Equinox Gold. Wage, salary, and burden information for personnel was estimated by KCA based on KCA's experience with similar projects in California and Nevada. The work force will consist of approximately 32 persons in the plant areas and 5 persons in the laboratory during Stage 1 and 86 persons in the plant areas and 23 persons in the laboratory during Stage 2. The total yearly staffing costs for the process and laboratory are estimated at $3.2 million during Stage 1 and $8.6 million during Stage 2.

21.2.17 Process Support Equipment Costs

Numerous pieces of support equipment are required for the processing areas. Costs to operate and maintain the process support equipment has been estimated primarily using published information, otherwise reasonable allowance have been made based on experience with similar operations. A summary of the support equipment required for Stage 1 and Stage 2 is presented in Table 21-18.

Table 21-18 - Process Support Equipment Operating Costs

Equipment	Quantity	Total US$/hr
Fork Lift	1	6.87
Boom Truck	1	10.00
Mechanic Service Truck	1	8.90
Backhoe/Loader	1	15.96
Crew Van, personnel transportation on site	2	10.00
Pickup Truck	6	11.88
Ambulance	1	10.00
CAT 966		43.25
45 ton articulated trucks	2	37.32
Heap Leach Dozer	1	54.38
Telehandler	1	11.13
Flatbed Truck	1	11.77
Skid Steer / Bobcat Loader	1	10.45
Bus, Personnel Transport On-Site	2	10.00
Pickup Truck (Mine, additional process & G&A)	10	11.88

21.2.18 Miscellaneous Operating and Maintenance Supplies

Overhaul and maintenance of equipment, along with miscellaneous operating supplies for each area, have been estimated based on a unit cost per ton of ore processed based on KCA's experience with similar operations.

21.2.19 General Administrative Costs (G&A)

G&A expenses represent the cost of activities that are necessary to the operation of the business as a whole, but for which a direct relationship to any particular cost objective cannot be shown. Annual average G&A costs for the Castle Mountain Project are shown in Table 21-19.

Table 21-19 - G&A Cost Summary

Cost Area	Stage 1 (Year 1-3)	Stage 2 (Year 4-16)
G&A Labor	$1,521,840	$3,688,320
G&A Expenses	$2,977,414	$6,254,872
G&A Total	**$4,499,254**	**$9,943,192**

21.2.20 General Administrative Costs – Labor

The G&A labor is presented in Table 21-20.

Table 21-20 - G&A Staffing Levels & Salary Schedules

	Stage 1 (Year 1-3)		Stage 2 (Year 4-16)	
Job Title	**# at Position**	**TOTAL COST**	**# at Position**	**TOTAL COST**
General Manager	1	$285,600	1	$285,600
Purchasing Manager	1	$136,000	1	$136,000
Purchaser			2	$168,640
Chief Accountant	1	$136,000	1	$136,000
Accounting Clerk	1	$84,320	2	$168,640
Human Resources/Relations Manager	1	$136,000	1	$136,000
Human Resources/Payroll Clerk			2	$136,000
Security/Safety/Training Manager	1	$122,400	1	$122,400
Safety Officer			4	$299,200
Environmental Supervisor	1	$163,200	1	$163,200
Environmental Technicians	1	$84,320	2	$168,640
Logistics Administrator			1	$74,800
IT Manager			1	$122,400
Warehouseman ON SITE	2	$149,600	4	$299,200
Accounts Payable Clerk	1	$74,800	1	$74,800
Receptionist/Secretary			2	$149,600
Guards			8	$598,400
Drivers	2	$149,600	2	$149,600
Laborers / Janitorial ON SITE			4	$299,200
Subtotal G&A	13	$1,521,840	41	$3,688,320

21.2.21 Non –Labor G & A Costs

Non-labor G&A costs include maintenance on the access road, donations to community projects, insurance and the costs of operating the administrative offices. The costs are detailed in Table 21-21.

Table 21-21 - G&A Non-Labor Costs

Cost Area	Stage 1 (Year 1-3)	Stage 2 (Year 4-16)
Maintenance Supplies	$76,092	$184,416
Office Supplies/Software	$114,138	$276,624
Transportation	$60,000	$180,000
Light Vehicle Operating Costs	$150,000	$450,000
Mancamp	$0	$0
Crew Rotations	$0	$0
Vancouver Office	$250,000	$750,000
Local Office Rental	$72,000	$144,000
Public Relations Expense	$76,092	$184,416
Communications	$76,092	$184,416
Insurance, Misc. Taxes, Fees, Licenses	$300,000	$700,000
Safety Supplies	$25,000	$50,000
Environmental (Testing, etc)	$150,000	$300,000
Training Supplies	$10,000	$25,000
Outside Audit (Accounting, Metallurgy, etc)	$100,000	$200,000
Travel	$100,000	$200,000
Legal	$600,000	$1,000,000
Data Processing	$60,000	$120,000
Access Road Maintenance	$250,000	$350,000
Security (Night Shift)	$100,000	$100,000
Cleaning	$20,000	$40,000
Miscellaneous (15%)	$388,000	$816,000
TOTAL	**$2,977,414**	**$6,254,872**

22.0 ECONOMIC ANALYSIS

22.1 Summary

Based on the estimated production parameters, revenue, capital costs, operating costs, taxes, and royalties, a cash flow model was prepared by KCA for the economic analysis of the Castle Mountain Project. All of the information used in this economic evaluation has been taken from work completed by KCA and other consultants as described in previous sections of this report.

The Castle Mountain Project economics were evaluated using a discounted cash flow (DCF), which measures the Net Present Value (NPV) of future cash flow streams. The final economic model was developed by KCA with input from Equinox Gold using the following assumptions.

The period of analysis is 20 years, and includes one year of pre-production and investment, 16 years of production, and three years for reclamation and closure). The major inputs to the analysis are as follows:

- Gold price of $1,250/oz.
- Stage 1 design processing rate of 14,000 tpd or 12,600 tonnes/d (Years 1-3, ROM only)
- Stage 2 design processing rate of 45,100 tpd or 40,900 tonnes/d (Years 4-17, 42,500 tpd or 38,600 tonnes/d for ROM and 2,600 tpd or 2,360 tonnes/d for mill).
- Average ROM gold grade of 0.012 oz/ton (0.41 g/tonne).
- Average mill gold grade of 0.094 oz/ton (3.22 g/tonne).
- LOM average opex of $8.43/ton ore ($7.65/tonne).
- Total LOM capex of $433.7 million (not including working capital and reclamation & closure costs).
- Net Smelter Royalties, totaling a 4.31% NSR for all combined royalties:
    - 2.65% FNV royalty applied to all ounces;
    - 5.00% Conservation royalty;
    - 2.00% American Standard royalty; and
    - 5.00% Huntington Tile royalty.
- State Income Tax rate of 8.84%.
- Federal Income Tax rate of 21%.
- Gold recoveries of:
    - 72.4% for ROM ore; and
    - 94.0% for mill ore.

Capital and operating costs used for this model are described in greater detail in Section 21 of this report.

The project economics based on these criteria from the cash flow model are summarized in Table 22-1.

Table 22-1 - Life of Mine Summary

Economic Analysis		
Internal Rate of Return (IRR), Pre-Tax	21.7%	
Internal Rate of Return (IRR), After-Tax	20.1%	
Average Annual Cashflow (Pre-Tax)	$54.3	M
NPV @ 5% (Pre-Tax)	$490.8	M
Average Annual Cashflow (After-Tax)	$45.9	M
NPV @ 5% (After-Tax)	$406.5	M
Gold Price Assumption	$1,250	/Ounce
Silver Price Assumption	$17	/Ounce
Pay-Back Period (Years based on After-Tax)	8.8	Years
Capital Costs		
Phase 1 Initial Capital	$51.7	M
Phase 2 Initial Capital	$295.0	M
LOM Sustaining Capital	$142.0	M
Operating Costs (Average LOM)		
Mining	$5.79 ($6.38	/Ton processed /Tonne processed)
Processing & Support	$1.92 ($2.11	/Ton processed /Tonne processed)
G&A	$0.72 ($0.80	/Ton processed /Tonne processed)
Total Operating Cost	$8.43 ($9.29	/Ton processed /Tonne processed)
Total By-Product Cash Cost	$712	/Ounce Au
All-in Sustaining Cost	$763	/Ounce Au
Production Data		
Life of Mine	16.2	Years
Total Ton to Heap	207,057,520 (187,842,582	Tons Tonnes)
Total Ton to Mill	10,744,919 (9,747,791	Tons Tonnes)
Grade Au (Avg.)	0.016 (0.56	oz/ton g/tonne)
Contained Au oz	3,563,093	Ounces
Metallurgical Recovery Au (Overall)	79%	
Average Annual Gold Production	173,000	Ounces
Total Gold Produced	2,798,173	Ounces
LOM Strip Ratio (W:O)	3.76	

22.2 Methodology

The Castle Mountain Project economics are evaluated using a discounted cash flow (DCF) method. The DCF method requires that annual estimated cash inflows and outflows are converted to equivalent dollars in the year of evaluation. Considerations for this analysis include the following:

- The cash flow model was prepared by KCA with input from Equinox Gold on taxes and depreciation and other consultants working on the Project.
- The cash flow model is based on the GRE mine production schedule which includes only proven and probable reserves.
- Recovery of gold values from the heap leach are delayed to reflect the time required to recover gold from the heap; recovery of gold from the mill is assumed to occur in the same year in which the ore is processed.
- The period of analysis is 20 years including one year of pre-production and investment, 16 years of production, and three years for reclamation and closure.
- All cash flow amounts are in US dollars (US$). All costs are considered to be first quarter 2018 costs. Inflation is not considered in this model.
- The Internal Rate of Return (IRR) is calculated as the discount rate that yields a Net Present Value (NPV) of zero.
- The NPV is calculated by converting annualized cash streams to Year -1 at different discount rates. All annual cash flows are assumed to occur at the end of each respective year.
- The Payback Period is the amount of time, in years, required to recover the initial construction and expansion capital costs.
- Working capital is considered in this model as the capital required for operations until the project achieves a positive cashflow. Working capital is considered for both Stages 1 and 2.
- Applicable taxes have been included in the model. A sales tax rate of 7.75% has been applied to material supply costs for all construction periods. State income taxes and federal income taxes are applied at a rate of 8.84% and 21%, respectively, to income after allowed deductions are applied.

- Royalties totaling 4.31% (NSR). Royalties include:
    - 2.65% FNV royalty applied to all ounces;
    - 5.00% Conservation royalty;
    - 2.00% American Standard royalty; and
    - 5.00% Huntington Tile royalty.
- 100% equity financing is assumed.
- Sustaining Capital, Reclamation and Closure costs are included in the model.

The economic analysis is performed on a before and after-tax basis in constant dollar terms, with the cash flows estimated on a project basis.

22.3 General Assumptions

A description of the general assumptions for cost inputs, parameters, royalties, and taxes used in the economic analysis are included in the following subsections.

22.3.1 Project Timing

The financial analysis assumes that spending begins in 2019 (Year -1). The mill expansion (Stage 2) will be constructed in 2022 (Year 3). Reclamation and closure will take place in Years 17, 18 and 19.

22.3.2 Smelting and Refining Terms

The smelting and refining terms have been estimated based on information provided by a refiner in the region. The gold returned from the refinery is to be 99.95% payable. A refining charge of $1.51 per ounce has also been applied, which includes transportation and insurance costs.

22.3.3 Gold Price and Revenue

A gold price of $1,250/oz is used as the base case commodity price.

Gold production and revenue for the heap leach portion of the project in the model are delayed to reflect the time required to recover gold from the heap, which is based on field-adjusted leach curves for the ore stacked on the heap. Approximately 12% of the heap recoverable gold is deferred to the following year of production. All of the gold recovery and revenue from ore processed in the mill is assumed to be realized in the year the ore is processed.

Silver sales and revenue are not considered in this model.

22.3.4 Operating Costs

Operating costs were developed on an annual basis based on the production schedule and other operating parameters. The life-of-mine average operating costs are $5.79/ton ($6.38/tonne) ore for mining (with a contractor fleet Years 1-3, and owner fleet Years 4+), $1.92/ton ($2.11/tonne) ore for processing including ROM and mill ore, and $0.72/ton ($0.80/tonne) ore for G&A. The specific annual operating costs as applied to the cash flow model are included in the overall cash flow model which is presented in Table 22-6.

22.3.5 Capital Costs

The initial capital costs for project construction are incurred in the first year of development (Year -1) for the initial ROM production. The mill expansion and purchase of the Owner mining fleet occurs in Years 3 and 4. The following sustaining capital is also included:

- Replacement of mining equipment and mining supplies through the life of the project;
- Expansion of the heap leach in Years 2, 4, 7 and 10;
- Replacement of process mobile equipment in Years 5, 10 and 14;
- Barren solution header expansion in Year 7;

The distribution of the estimated project capital costs (including working capital, initial fills and reclamation & closure costs) are included below. Refer to Section 21 for capital cost details.

Table 22-2 - Capital Cost Summary

Description	Cost (US$)
Stage 1 Pre-Production Capital	$51,667,000
Stage 2 Expansion Capital	$294,958,000
LOM Sustaining Capital	$142,029,000
TOTAL Capital Costs Including Sales Tax	**$488,654,000**

22.3.6 Royalties

The Castle Mountain Project is subject to several royalties which are payable to different parties resulting in an overall average 4.31% NSR. Royalties payable include:

- 2.65% FNV royalty applied to all ounces;
- 5.00% Conservation royalty;
- 2.00% American Standard royalty; and
- 5.00% Huntington Tile royalty.

Using base case prices, the current financial model estimates the total value of royalty payments as $150.6 million over the life of the mine.

22.3.7 Taxation

Revenue from the Castle Mountain project is subject to an 8.84% California State Income Tax and 21% Federal Income Tax. Income subject to state income taxes is calculated based on net revenue, less royalties, operating expenses, reclamation and closure costs, depreciation, depletion and any net operating losses. The project is subject to a minimum California state income tax of $800 regardless of net losses. California State income taxes from the previous year are deducted from the taxable income for the current year for determining income subject to Federal Income Taxes.

22.3.8 Reclamation and Closure for Tax Estimation

Reclamation and closure costs have been estimated at $20.2 million. To approximate the tax accounting method of amortizing reclamation and closure costs, the reclamation and closure costs have been amortized on a straight-line basis over the life of the project.

In addition to the above depreciation rate schedules, certain qualifying assets acquired and placed in service after September 27, 2017 and before January 1, 2027 were eligible for "bonus depreciation" allowing for 100% depreciation of the capital cost in the year it is incurred through Year 2022 with Bonus depreciation tapering off by 20% each following year through 2026.

22.3.9 Depreciation

Depreciation of capital items, including $17.9 million in capitalized historical exploration costs, have been based on asset classes as defined in Table 22-3 and depreciation rate schedule as shown in Table 22-4. Capitalized exploration costs are depreciated based on a 10-year straight-line amortization schedule.

Table 22-3 - Depreciation Asset Groups

Item	Recovery Period
Mine Production Equipment	7
Mine Support	7
Mine Capital Replacement	7
Mine Shop	15
Mine Fuel/Lube/Wash	15
Mine Office Equipment & Software	7
Mine Infrastructure	15
Mining Contingency	7
Mine Waste Pre-stripping	Immediate
Major Earthworks	Immediate
Liner (Supply & Install)	Immediate
Civils (Supply & Install)	Immediate
Structural Steel (Supply & Install)	Immediate
Platework (Supply)	Immediate
Platework (Install)	Immediate
Mechanical Equipment (Supply)	7
Mechanical Equipment (Install)	7
Piping (Supply & Install)	15
Electrical (Supply)	20
Electrical (Install)	20
Instrumentation (Supply & Install)	Immediate
Infrastructure (Supply & Install)	15
Spare Parts	7
Process Contingency	Immediate
EPCM	Immediate
Indirect Costs (incl. contingency)	Immediate
Owner's Costs (incl. contingency)	Immediate

Table 22-4 - Depreciation Rate Schedule

Recovery Period	3	5	7	10	15	20	Immed.	UoP*
Year 1	33.30%	20.00%	14.30%	10.00%	5.00%	3.80%	100.00%	1.30%
Year 2	44.50%	32.00%	24.50%	18.00%	9.50%	7.20%		1.50%
Year 3	14.80%	19.20%	17.50%	14.40%	8.60%	6.70%		2.00%
Year 4	7.40%	11.50%	12.50%	11.50%	7.70%	6.20%		5.10%
Year 5		11.50%	8.90%	9.20%	6.90%	5.70%		7.80%
Year 6		5.80%	8.90%	7.40%	6.20%	5.30%		7.70%
Year 7			8.90%	6.60%	5.90%	4.90%		6.20%
Year 8			4.50%	6.60%	5.90%	4.50%		5.60%
Year 9				6.60%	5.90%	4.50%		8.40%
Year 10				6.60%	5.90%	4.50%		7.80%
Year 11				3.30%	5.90%	4.50%		8.60%
Year 12					5.90%	4.50%		6.20%
Year 13					5.90%	4.50%		7.90%
Year 14					5.90%	4.50%		9.00%
Year 15					5.90%	4.50%		7.70%
Year 16					3.00%	4.50%		6.20%
Year 17						4.50%		0.90%
Year 18						4.50%		0.00%
Year 19						4.50%		0.00%
Year 20						4.50%		0.00%
Year 21						2.20%		0.00%
Total	**100.00%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**	**100.00%**

*UoP = Depreciation based on Units of Production

In addition to the above depreciation rate schedules, some asset classes are eligible for "bonus depreciation", allowing for 100% depreciation of the capital cost in the year it is incurred through Year 2022, with bonus depreciation tapering off by 20% each following year through 2026. In this model, bonus depreciation has been claimed for asset classes that are assigned a seven-year recovery, which primarily includes mining and process equipment, infrastructure, and spare parts.

22.3.10 Depletion

Castle Mountain is eligible for tax deductions based on depletion. As a proxy for the annual depletion deductions, the percentage depletion deduction is calculated as either 15% of the gross income (net revenue less royalties) or 50% of the taxable income, whichever is less.

22.3.11 Net Operating Losses

Net operating losses (NOL) for the Castle Mountain project can be used to offset taxable income in any given year. The cash flow considers a starting NOL pool of $9.0 million based on historical losses.

22.3.12 Working Capital and Initial Fills

Working capital is included in the model for both Stage 1 and the Stage 2 expansion to ensure sufficient funds for operations before enough revenue is generated to cover all applicable operating costs. A working capital of 60 days of operating cost (mine, process and G&A) is included for Stage 1 and 30 days of operating costs is included for Stage 2. The total LOM working capital estimate is $17.1 million.

Initial fill of reagents (such as cyanide, lime, carbon, etc) are also included as part of the process capital. The capital for initial fills is estimated at $548,000. Initial fill requirements for Stages 1 and 2 are estimated at $225,000 and $323,000, respectively.

Working capital and initial fills are assumed to be fully recovered during the final two years of operations, with two-thirds of the total recovered during Year 16 and one-third recovered in Year 17.

22.3.13 Closure Costs

Closure costs have been estimated at $20.2 million and will occur in Years 17, 18, and 19, with the total cost distributed at 20%, 20%, and 60% over these years respectively. Closure costs have been estimated by GLA based on the disturbed area and do not include costs for heap rinsing and labor costs during closure, which are included as part of the process operating costs.

22.3.14 Average Cash Cost and All-In-Sustaining Costs

The average cash cost per ounce payable for the life of the mine is calculated by adding all the mining, process and G&A operating costs, refining and transportation costs and royalties and dividing that number by the total ounces payable. The average cash cost is calculated at $712 per ounce.

The all-in-sustaining cost per ounce payable includes the mining, process and G&A operating costs, refining and transportation costs, royalties, the life of mine sustaining costs and costs for reclamation and closure divided by the payable ounces. The LOM sustaining costs do not include capital costs for the Stage 2 expansion which is constructed during Year 3 with additional mine production equipment being purchased during Year 4 And pre-stripping of waste in Year 3. The all-in-sustaining cost is calculated at $763 per ounce.

22.4 Financial Model and Results

A discounted cash flow (DCF) method was used to evaluate the economics of the Castle Mountain project. The DCF method measures the Net Present Value (NPV) of future cash flow streams. This financial model has been developed by KCA with input from Equinox Gold for the tax and depreciation model, GRE for the mine production schedule and GLA for the reclamation and closure costs.

Table 22-5 shows the key financial parameters derived from the cash flow analysis. Table 22-6 presents the cash flows.

Table 22-5 - Key Financial Parameters

	Stage 1 (Years 1-3)	**Stage 2** (Years 4-16)	**Total Project**
Gold Price, $/oz			1,250
Ore, M tons (M tonnes)			217.8 (197.6)
Grade, oz/ton (g/tonne)	0.010 (0.34) - ROM	0.013 (0.43) - ROM 0.094 (3.23) - Mill	0.016 (0.56)
Proven & Probable Reserve, oz			**3,563,093**
Average annual production, oz	44,930	202,979	173,345
Recoverable LOM gold, oz			**2,798,173**
Throughput, tpd (tonnes/d)	14,000 (12,700)	45,100 (41,000)	
Strip ratio			3.76
Recovery	72% (ROM)	72% (ROM) 94% (mill)	79%
Mine life, Years	3	13	16
Initial capex, $M	$52	$295	$347
Sustaining capex, $M			$142
Cash cost (including royalties), $/oz	$889	$703	$712
AISC, $/oz	**$980**	**$752**	**$763**
Pre-tax NPV 0%, $M			$1,034.2
Pre-tax NPV 5%, $M			$490.8
Pre-tax IRR			21.7%
After-tax NPV 0%, $M			**$865.5**
After-tax NPV 5%, $M			**$406.5**
After-tax IRR			20.1%

Table 22-6 - Cash Flow Analysis

			Year									
			Stage 1				Stage 2					
	Total	Units	-1	1	2	3	4	5	6	7	8	9
Mined Tonnes (Ore)	217,802	k tons	0	5,114	5,114	8,822	16,244	16,469	16,383	16,496	16,262	16,518
	197,589	k tonnes	0	4,639	4,639	8,003	14,736	14,941	14,862	14,965	14,753	14,985
Mined Tonnes (Waste)	819,479	k tons	0	630	2,193	54,875	95,907	69,074	63,586	78,876	82,725	72,300
	743,426	k tonnes	0	571	1,989	49,783	87,007	62,663	57,685	71,556	75,048	65,590
Mined Tonnes (Waste + Ore)	1,037,281	k tons	0	5,743	7,306	63,697	112,151	85,543	79,969	95,373	98,987	88,818
	941,015	k tonnes	0	5,210	6,628	57,786	101,743	77,604	72,547	86,521	89,801	80,575
Strip Ratio	3.76	t:t	0.00	0.12	0.43	6.22	5.90	4.19	3.88	4.78	5.09	4.38
Mined Gold	3,563,055	oz	0	58,608	56,296	82,424	193,971	272,764	277,832	222,260	209,355	303,499
Ore processed	217,802	k tons	0	5,114	5,114	8,798	16,268	16,469	16,377	16,502	16,262	16,472
	197,589	k tonnes	0	4,639	4,639	7,981	14,758	14,941	14,857	14,971	14,753	14,943
Processed Grade	0.016	oz/ton	0.000	0.011	0.011	0.009	0.012	0.017	0.017	0.013	0.013	0.017
	0.56	g/tonne	0.00	0.39	0.38	0.32	0.41	0.57	0.58	0.46	0.44	0.58
Gold Processed	3,563,093	oz	0	58,609	56,297	80,989	195,409	272,767	277,336	222,761	209,357	298,918
Recovery %	78.50%	%	0.00%	72.40%	72.40%	72.40%	76.89%	80.26%	78.45%	76.56%	75.18%	79.20%
Recoverable Gold	2,798,173	oz	0	42,433	40,759	58,636	150,251	218,918	217,567	170,557	157,398	236,747
Produced Gold	2,798,173	oz	0	36,492	41,808	56,491	143,840	217,290	215,296	172,283	157,172	234,801
All-in-sustaining Costs	763	US$	0	803	1,042	1,048	1,339	787	654	946	1,077	666
Pre-tax cashflow	1,034,153	US$000s	-51,667	16,301	8,691	-246,555	-49,861	100,390	128,165	52,192	27,132	137,055
After-tax cashflow	865,476	US$000s	-51,667	16,300	8,690	-246,556	-49,862	98,855	124,646	51,048	26,698	131,863
Mill Tonnes Processed	10,745	k tons	0	0	0	0	573	908	936	671	354	936
	9,748	k tonnes	0	0	0	0	520	823	849	609	321	849
ROM Tonnes Processed	207,058	k tons	0	5,114	5,114	8,798	15,695	15,562	15,441	15,831	15,909	15,536
	187,841	k tonnes	0	4,639	4,639	7,981	14,238	14,117	14,008	14,362	14,432	14,094
Total Ore Tonnes Processed	217,802	k tons	0	5,114	5,114	8,798	16,268	16,469	16,377	16,502	16,262	16,472
	197,589	k tonnes	0	4,639	4,639	7,981	14,758	14,941	14,857	14,971	14,753	14,943

Table 22-6 (Continued) - Cash Flow Analysis

			Year									
			Stage 2									
	Total	**Units**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**
Mined Tonnes (Ore)	217,802	k tons	16,518	16,868	15,849	15,893	15,939	13,478	5,836	0		
	197,589	k tonnes	14,985	15,303	14,378	14,418	14,460	12,227	5,295	0		
Mined Tonnes (Waste)	819,479	k tons	74,612	70,954	58,301	49,979	28,417	13,591	3,457	0		
	743,426	k tonnes	67,687	64,369	52,891	45,341	25,780	12,330	3,136	0		
Mined Tonnes (Waste + Ore)	1,037,281	k tons	91,130	87,822	74,151	65,872	44,356	27,069	9,293	0	0	0
	941,015	k tonnes	82,673	79,672	67,269	59,759	40,240	24,557	8,431	0		
Strip Ratio	3.76	t:t	4.52	4.21	3.68	3.14	1.78	1.01	0.59	-		
Mined Gold	3,563,055	oz	281,647	316,784	206,975	283,100	349,605	247,450	200,485	0		
Ore processed	217,802	k tons	16,484	16,808	15,989	15,893	15,661	13,634	5,809	149		
	197,589	k tonnes	14,954	15,248	14,505	14,418	14,207	12,369	5,270	135		
Processed Grade	0.016	oz/ton	0.019	0.014	0.018	0.020	0.020	0.033	0.128	0.000		
	0.561	g/tonne	0.63	0.47	0.61	0.69	0.69	1.13	4.40	0.00		
Gold Processed	3,563,093	oz	279,715	311,096	219,189	283,103	314,915	272,774	190,735	19,125		
Recovery %	78.50%	%	78.38%	78.03%	77.26%	79.06%	80.40%	79.14%	85.29%	94.00%		
Recoverable Gold	2,798,173	oz	219,241	242,757	169,337	223,829	253,196	215,887	162,683	17,977		
Produced Gold	2,798,173	oz	219,461	240,351	174,555	221,581	252,980	216,813	172,302	24,657		
All-in-sustaining Costs	763	US$	808	680	864	677	532	508	444	680		
Pre-tax cashflow	1,034,153	US$000s	96,833	136,841	67,277	126,912	181,517	160,693	150,583	19,915	-11,727	-16,533
After-tax cashflow	865,476	US$000s	92,035	116,402	58,375	103,782	144,071	128,495	121,988	18,575	-11,728	-16,534
Mill Tonnes Processed	10,745	k tons	936	936	704	835	936	936	936	149		
	9,748	k tonnes	849	849	638	758	849	849	849	135		
ROM Tonnes Processed	207,058	k tons	15,548	15,872	15,286	15,058	14,725	12,698	4,873	0		
	187,841	k tonnes	14,105	14,399	13,867	13,660	13,358	11,520	4,421	0		
Total Ore Tonnes Processed	217,802	k tons	16,484	16,808	15,989	15,893	15,661	13,634	5,809	149		
	197,589	k tonnes	14,954	15,248	14,505	14,418	14,207	12,369	5,270	135		

22.5 Sensitivity Analysis

Sensitivity of the project economics to key parameters including gold price, total capital cost and operating was completed to evaluate the relative strength of the project. The sensitivities are based on +/- 25% of the base case. The after-tax sensitivity analysis is presented in Table 22-7, and graphically in Figures 22.1, 22.2, 22.3 and 22.4. The economic indicators chosen for sensitivity evaluation are the internal rate of return (IRR) and NPV at 0, 5, and 10% discount rates.

The sensitivity analysis indicates that the project is robust and is most sensitive to revenue (gold price, ore grade, and recovery), and operating costs.

Table 22-7 - Sensitivity Analysis (After Tax)

Gold price ($/oz)	**-25%**	**-10%**	**$1,250**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	-$21.5	$243.6	**$406.5**	$565.7	$799.7
IRR (after tax)	4.2%	14.0%	**20.1%**	26.3%	35.8%

Capital costs	**-25%**	**-10%**	**$471.0**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	$478.7	$435.6	**$406.5**	$377.5	$333.8
IRR (after tax)	27.1%	22.5%	**20.1%**	18.1%	15.6%

Operating costs	**-25%**	**-10%**	**$1,836.0**	**10%**	**25%**
$NPV_{5\%}$ (after tax), $M	$624.9	$495.2	**$406.5**	$315.6	$175.2
IRR (after tax)	30.0%	24.0%	**20.1%**	16.4%	11.0%

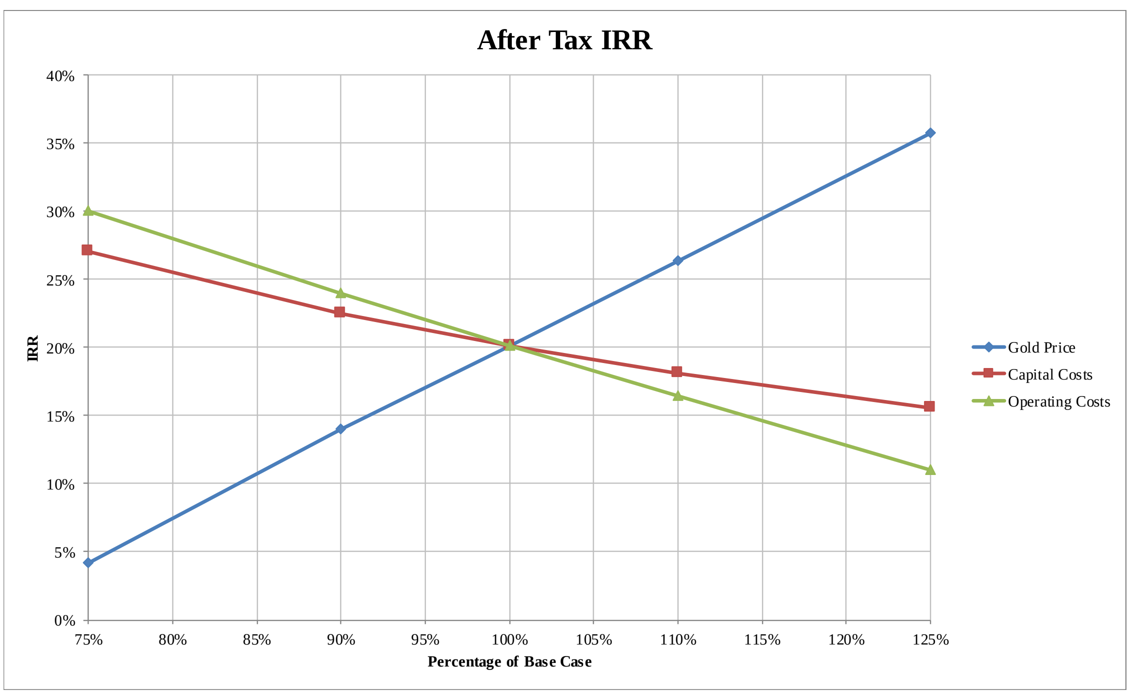

Figure 22-1 - After-Tax IRR vs. Gold Price, Capital Cost, and Operating Cash Cost

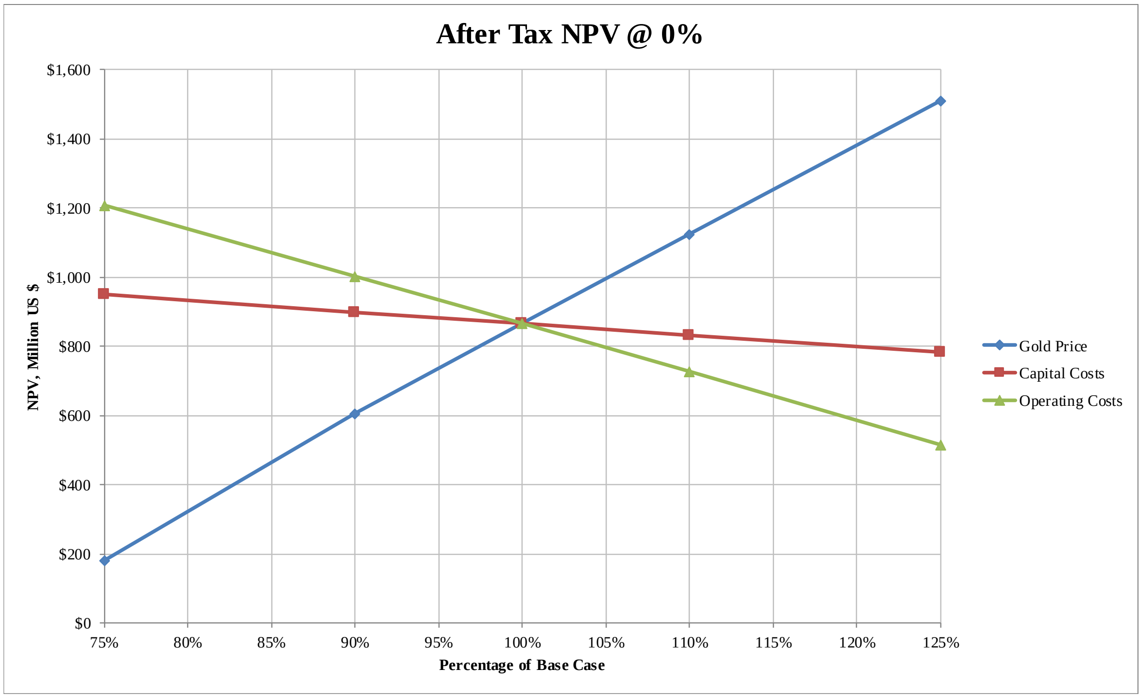

Figure 22-2 - NPV @ 0% vs. Gold Price, Capital Cost, and Operating Cash Cost

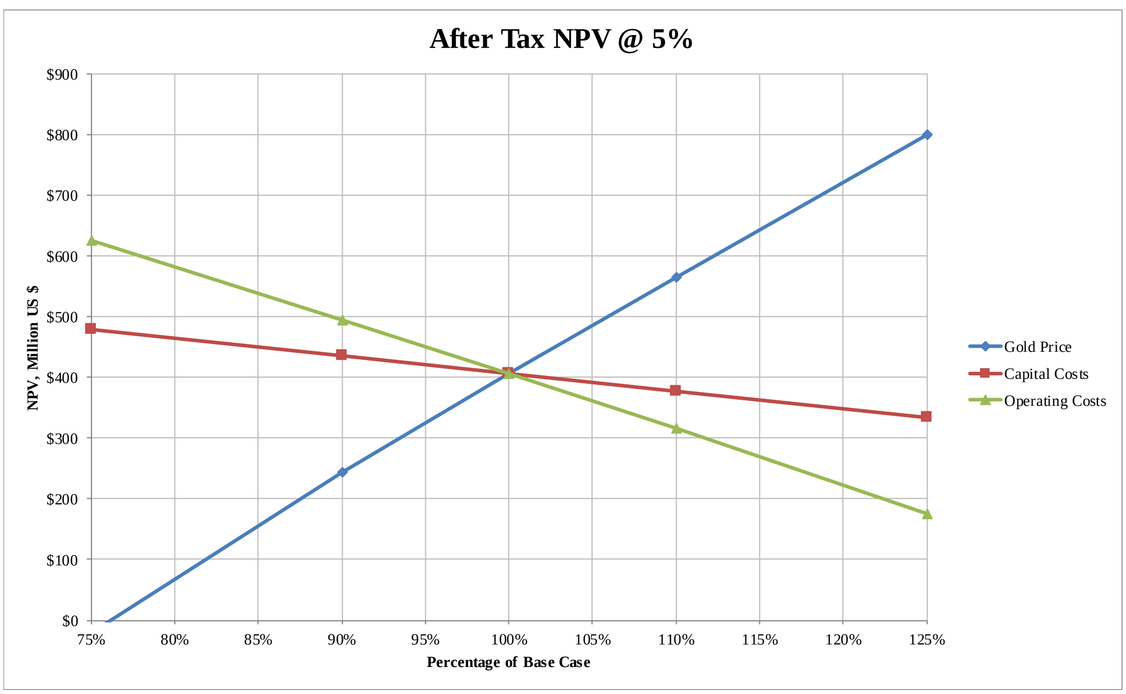

Figure 22-3 - NPV @ 5% vs. Gold Price, Capital Cost, and Operating Cash Cost

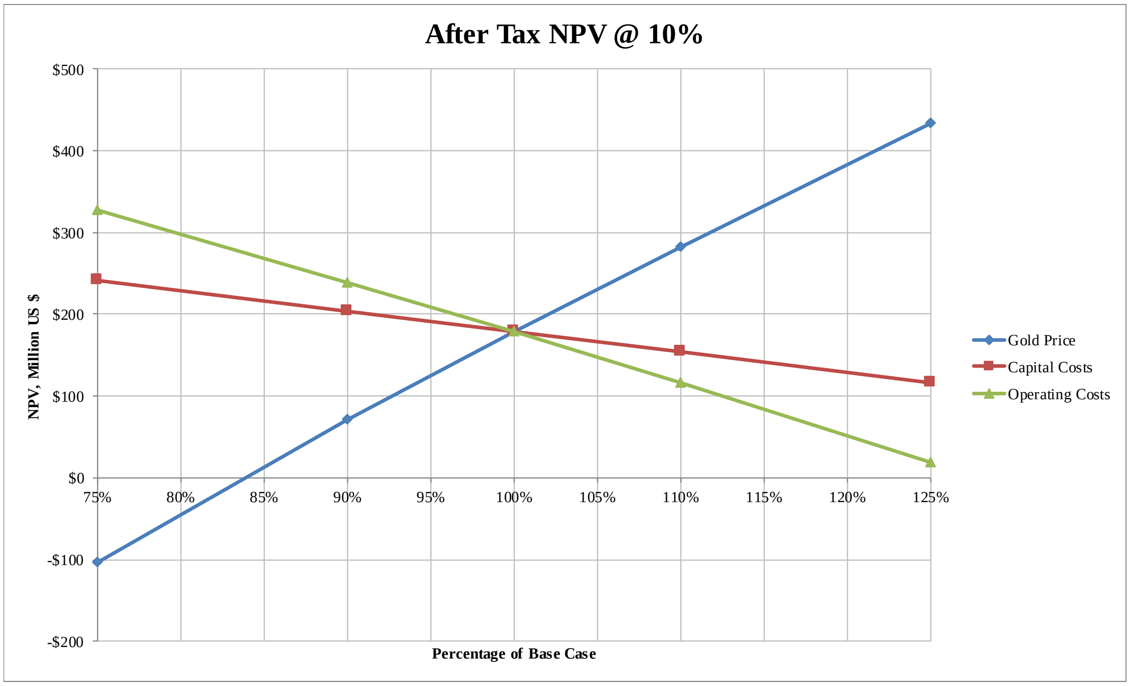

Figure 22-4 - NPV @ 10% vs. Gold Price, Capital Cost, and Operating Cash Cost

23.0 ADJACENT PROPERTIES

There are no relevant mineral properties adjacent to the Castle Mountain Project.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Opportunities

Resources and Reserves

Within the designed pit, there is an inferred resource of 70.8 Mt of material above a 0.005 oz/t cutoff. This portion of the mineral resource presents a significant opportunity for increased reserves, and the conversion of that resource to reserve should enhance project economics as it is currently treated as waste rock. Additionally, the lower portions of the JSLA backfill material represent a similar opportunity to increase reserves from material currently treated as waste. Currently unexplored, the deposition of the backfill is well documented through monthly as-built drawings from Viceroy, and it may have previously considered waste that is now, above cutoff.

Silver Credit

Silver is not included in the mineral resource estimate or reserve, and accordingly no credit to silver recovery has been given in the economic analysis. Available historical metallurgical test work and limited historical production records (dore assays) suggest that up to 0.2 oz of silver per ounce of gold may be recovered in operations. In the current estimate of recovered gold ounces this may present an upside potential of up to 700,000 ounces of silver recovered life of mine, or $10 million at current silver prices.

Pit Slope Angles

There is an opportunity that the pit slope angles may be steepened in areas with better three-dimensional controls of where the major through going faults are located. In addition, better geological understanding of where some of the wall rock is clay altered could also allow for better controls on the pit slopes.

24.2 Risks

South Domes Metallurgy

The selected ROM recovery for all ores in Phase 2 operations is 72.4%, based on bulk ROM column tests conducted in the JSLA pit area. Test work in support of South Domes (approximately 1/3 of the total Project ounces) ore recoveries is currently limited to variability bottle roll tests, which however do indicate similar recoveries to JSLA bottle roll tests. Still there is a low risk that a lower recovery for South Domes ore may be realized which can negatively impact project economics. This risk can be reduced significantly by running coarse crush column tests on composites of South Domes ores to support the selected recovery.

Water Supply

The processing rate and gold production at Castle Mountain is critically dependent on water supply. The currently existing water supply network (and the planned infrastructure within permitted areas) is sufficient for Phase 1 operations but not Phase 2 operations, and additional hydrological studies and drilling outside of currently permitted areas will be required to secure the necessary Phase 2 water requirement. There is a moderate risk that the necessary water cannot be secured either due to technical or permitting reasons, which would limit the Phase 2 production rate of ore and negatively impact project economics.

Land Title Risks and Designation

Although Equinox Gold may receive title opinions for any mineral properties in which Equinox Gold has or will acquire a material interest, there is no guarantee that title to such properties will not be challenged or impugned. In some countries, the system for recording title to the rights to explore, develop and mine natural resources is such that a title opinion provides only minimal comfort that the holder has title. Also, in the United States, claims have been made and new claims are being made by aboriginal peoples that call into question the rights granted by the government. A determination of defective title or restrictions in connection with a challenge to title rights could impact Equinox Gold's ability to develop and operate at Equinox Gold's mineral projects.

The Mojave Desert Preserve established in 1994 surrounds Equinox Gold's patented and unpatented land. In addition, there is an approximately 22,000 acre "buffer zone" surrounding Equinox Gold's lands. On February 12, 2016, Equinox Gold announced that its claim holdings and private land held would not be included in the new Castle Mountains National Monument following the proclamation by then US President Barack Obama under the Antiquities Act of 1906. The Monument surrounds but does not include an approximately 8,340 acre parcel referred to as the 'Castle Mountain Mine Area', consisting of BLM-managed Federal land, State land, and private land. The Castle Mountain Project is included in its entirety within the Castle Mountain Mine Area.

The proclamation also directs that after any such mining and reclamation are completed at the Castle Mountain Project, or after 10 years if no mining occurs, then jurisdiction over federal lands in the Castle Mountain Mine Area is to be transferred to the National Park Service. There can be no assurance that the Castle Mountain Project will not be included in any expansion of the Monument or that the jurisdiction of the Castle Mountain Project will not be transferred to the National Park Service.

In Equinox Gold's view, these land designations do not impede Equinox Gold's plans for developing the Castle Mountain Project. Equinox Gold is not able to provide any assurance regarding any future designation of lands, nor the timing of implementation of any such designations.

25.0 INTERPRETATION AND CONCLUSIONS

MTS reviewed the exploration and geological data and estimated the mineral resources for the Castle Mountain Project. A summary of MTS interpretations and conclusions by area are as follows:

Geology

- Surface mapping by NewCastle during the 2016-2017 field season greatly improved the understanding of the geological and mineralization setting at Castle Mountain.
- Focused angle core drilling during the 2016-2017 field season greatly improved the geological and resource models for Castle Mountain.
- NewCastle's geologic modelling program using Leapfrog 3-dimensional modelling software clearly illustrates the relationships between property-scale geology and economic mineralization.

Exploration

- Infill drilling completed as part of the Phase I and Phase II drill programs at Castle Mountain have improved the confidence in defining mineralization in the JSLA and South Domes areas.

QA/QC

- Viceroy drill samples were analyzed for gold by conventional fire assay methods and a routine program of duplicate analyses and check assays was used to control assay precision and accuracy.
- Viceroy assay precision from the pulp duplicates was variable with gold grade, but generally acceptable.
- Viceroy check assay samples did not indicate any significant bias in the original assays.
- The company employed an industry standard QA/QC program that included the analysis of CRMs, blanks, RC field duplicates, and check assays.
- MTS reviewed the 2017 control sample results and found the assay accuracy and precision to be acceptable for purposes of resource estimation.
- The sample preparation, security, and analytical procedures are adequate for purposes of resource estimation.
- The assay accuracy and precision are considered acceptable for resource estimation

Mineral Resources

- Grade interpolations were estimated using the inverse distance weighting (IDW) method. Mineral Resources estimated for the Castle Mountain Project total 148.8 Mt at a gold grade of 0.0188 oz/t in the Measured (M) category, 63.1 Mt at a gold grade of 0.0187 oz Au/t in the Indicated (I) category, for a total M&I of 212.0 Mt at a gold grade of 0.0187 opt using a 0.006 gold cutoff grade. Contained gold in the Measured category is about 2.8 million ounces and in the Indicated category, about 1.2 million ounces, totalling about 4.0 million ounces.

- The Inferred category estimated 112.8 Mt at a gold grade of 0.014 opt.

- In MTS' opinion, the block grades appear to correlate well with the drill hole composite grades. MTS believes the 3D grade model is suitable for mine planning purposes.

Mineral Reserves

- The current mine plan includes 217.8 Mt of ore and 3.56 Moz at a gold grade of 0.016 oz/t using a 0.005 oz/t cutoff. This is divided into 150.6 Mt of proven ore and 2.56 Moz at 0.017 oz/t and 67.2 Mt of probable ore and 1.00 Moz at 0.015 oz/t.

- Measured and indicated resources contained within the design pit were used to determine final pit limits and thus converted into proven and probable reserves, respectively.

- Mining of the deposit will be accomplished using straightforward, conventional open pit hard rock mining techniques.

Metallurgy and Process

- The metallurgical test work programs completed to date are sufficiently detailed to establish the optimal processing methods for the known ores at Castle Mountain and were performed on mineralization that was typical and representative of the deposit.

- The test work results support the estimation of recovery factors for the selected process streams at a prefeasibility level or higher. The metallurgical test work programs are adequate to understand ore variability and plant optimization potential.

- Heap leach gold recovery for ROM is estimated at 72.4%, and 94% for the milling, gravity, and CIL circuit as proposed.

- All processing, recovery methods, and plant designs selected for Castle Mountain are consistent with industry standard gold processing.

Project Economics

- The base case economic analysis indicates a favorable return for the Project, with an after-tax IRR of 20.1% and an NPV (at 5% discount rate) of $406.5 million.

- The sensitivity analysis indicates that the Project is robust, and is most sensitive to revenue (gold price, ore grade, and recovery), and operating costs.

26.0 RECOMMENDATIONS

26.1 Recommended Future Activities

26.1.1 Geology and Exploration

MTS submits the following recommendations resulting from the review of the Castle Mountain geology and exploration data:

- Add fields to the Castle Mountain lithology database for the updated geologic interpretations to inform future statistical analysis and grade modelling efforts.

- Complete the 3D geologic interpretation through the mined-out pits to generate a complete geologic package and to allow for comparison of current grade modelling methods with historical mine results.

- Explore the area outside of the current grade shells, especially to the north, west and east of the known limits of the deposit.

26.1.2 Mining Methods

- Additional optimization of the pit design, phase design, production scheduling, and waste dump design is recommended during the Feasibility and detailed design engineering effort.

26.1.3 Metallurgical Test Work and Processing

KCA submits the following recommendations resulting from the review of the Castle Mountain metallurgical and process data:

- Conduct variability bottle roll tests with carbon from several samples throughout the JSLA, South Domes, and Oro Belle pit areas, to confirm CIL operating parameters, reagent consumptions and recoveries.

- Conduct Caro's Acid and INCO/SO_2 detoxification tests on a composite slurry sample to confirm detoxification operating parameters, final residual cyanide values, and confirm reagent requirements.

- Conduct at least four column tests, for two crush sizes in duplicate, on composites prepared from the South Domes area. No previous column tests have been run on samples from this area. This work will help to confirm whether leaching operating parameters and recovery data are substantially similar to those obtained from the JSLA backfill and hardrock areas, so that the ROM recovery for ore from the South Domes area can be confirmed.

26.2 Recommended Work Program

Based on the encouraging exploration and development results and updated design and economic results from the PFS, KCA recommends that Equinox Gold initiate a full feasibility study (FS) using the current mineral resource estimate, metallurgical test work, and current PFS process designs as a new foundation. The FS should advance the mine design, metallurgical test work, and process design.

KCA recommends Equinox Gold continue with delineation and exploration drilling to augment the current mineral resources, along with expanded hydrogeological studies and drilling, environmental baseline, metallurgical and geotechnical studies.

The recommended drilling program is summarized in Table 26-1.

Table 26-1 – Recommended Drill Program

Area/Purpose	Holes	Average Depth (ft)	Total Length (ft)
Reverse Circulation Drilling of the low part of the JSLA Backfill	64	395	25,280
Diamond Core Drilling (East Ridge resource conversion)	18	700	12,600
Reverse Circulation Drilling (East Ridge resource conversion)	18	700	12,600
Diamond Core Drilling (Green and Gold condemnation/evaluation)	13	600	7,800
Reverse Circulation Drilling (Green and Gold condemnation/evaluation)	10	600	6,000
Hydrological Drilling (testing for new water sources)	4	1500	6,000
Total	**128**		**70,780**

A recommended budget of **$8,968,000** for the drilling program and Feasibility Study is shown in Table 26-2.

Table 26-2 – Recommended Castle Mountain Project Budget

Item	$US Amount
Land and site costs (Annual BLM fees, County taxes, Water Board fees)	245,000
Regional geological mapping, rock sampling	100,000
RC drilling and assays - JSLA backfill (64 holes @ 395 feet)	1,264,000
RC drilling and assays – hard rock targets (28 holes @ 664 feet)	930,000
Core drilling and assays (31 holes @ 852 feet)	1,734,000
Drill materials and supplies	250,000
Geology labor/consultants	250,000
Field Support Costs (Camp, travel, gas, meals)	95,000
Mineral resource and modeling optimization studies	100,000
Subtotal	**4,968,000**
Hydrogeology surveys and studies	300,000
Hydrogeology based geophysics	200,000
Hydrogeology drilling (4 holes @ 1,500 ft)	1,200,000
Environmental and permitting baseline studies and legal	500,000
Metallurgical testing (variability CIL, detox, column tests)	200,000
Geotechnical study (slope stability)	100,000
Feasibility study (Mine design, process design, infrastructure)	1,400,000
FS 43-101 reporting	100,000
Subtotal	**4,000,000**
TOTAL	**8,968,000**

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