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CONSTRUCTION MANAGEMENT

Haile Gold Mine Project

NI 43-101 Technical Report Project Update

Lancaster County, South Carolina

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DATE AND SIGNATURES PAGE

The effective date of this report is July 31, 2016. The issue date of this report is October 25, 2016. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

HAILE GOLD MINE PROJECT
FORM 43-101F1 TECHNICAL REPORT

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1 SUMMARY

1.1 IMPORTANT NOTE

This technical report is based on the *Haile Gold Mine Technical Report* that was issued in October of 2015 by OceanaGold Corporation. This is a re-issue of the Technical report which includes the results of a Preliminary Economic Assessment (PEA) on the underground mining potential at the Haile Gold Mine in Section 24 of this report. The intent of the PEA is to present a scoping level study which evaluates the mining of selected areas of mineralization below the reserve pit floor by underground mining methods. This would not preclude mining any of the reported open pit reserves. Further work is required to assess the comparative merits of mining this mineralization via underground versus open pit mining methods. Until such time, the underground mining scenario described in the PEA scoping study is considered as a technically viable alternative to open pit mining scenarios below the current reserve pit design. For this reason, the resource below the reserve pit floor remains classified as open pit resource. The subset of this resource selected for this underground option study in the PEA, has been reported only in section 24 and is not excised from the open pit resource inventory presented in Table 1-2.

The authors of this report have updated open pit capital and mining costs data for this issuance of the Technical Report.

1.2 SUMMARY

This section briefly summarizes the findings of the Haile Gold feasibility project update. The proposed project is an open pit gold mine that delivers sulfide ore to a 7,000 tpd (short tons per day) grinding, flotation, cyanide leach, carbon handling and refining facility. The project is located near Kershaw, South Carolina which has a balance of remoteness and close proximity to infrastructure. Over the life of the project, 1,681,500 ounces (troy ounces) of gold are projected to be produced.

Following a Plan of Arrangement completed on October 1, 2015 between Romarco Minerals Inc. and OceanaGold Corporation, Haile Gold Mine Inc. (HGM) is a wholly owned subsidiary of OceanaGold Corporation. References in this document to OceanaGold refer to the parent company together with its subsidiaries, including HGM and Romarco Minerals Inc.

HGM selected third-party consultants that are well known and respected in the industry. These consultants performed the design, engineering, reserve calculations, and environmental studies used for this report. All consultants have the capability to support the project, as required and within the confines of expertise, from feasibility study to full operation.

M3 Engineering & Technology Corporation (M3), and other HGM consultants, developed more than 2,500 engineering detailed design drawings since the completion of the feasibility study in 2011. A large portion of the mining and process equipment has been purchased and is either on site or awaiting fabrication.

The Haile deposit will be mined using conventional open pit methods. Pre-stripping began in the second quarter of 2015 with the first ore scheduled to arrive at the mill in the fourth quarter of 2016. Annual high-grade ore production from the mine is 2.555 million tons and total material moved averages 63,000 tpd with daily mill production averaging 7,000 tpd. Low-grade material between the mill cutoff and a breakeven cutoff is stockpiled in years "-1" through year 7 for a total of 4.9 million tons, and this material is processed at the end of the mine life. The life-of-mine (LOM) stripping ratio is 7.2:1 (overburden to ore).

1.3 OPEN PIT KEY DATA

Key project data are presented in Table 1-1 including a summary of the project size, production, operating costs, metal prices, and financial indicators.

The financial analysis for the base case metal pricing provides an after-tax NPV of $290.8 million at a 5% discount rate, an IRR of 16.7% and a payback period of 4.4 years. The financial indicators are most sensitive to the gold price and gold grade. The base case assumptions and other sensitivity analyses are summarized below and in the financial section at the end of this report.

Table 1-1: Open Pit Key Project Data

Open Pit Mine Life (years)	13		
Milling of Low Grade stockpile (years)	3		
Total Life (years)	14 (low grade processed in Year 14)		
Mine Type:	Open Pit		
Process Description:	Crushing, Grinding, Flotation, Cyanide Leach		
Mill Throughput (Short tons per day)	7,000		
Initial Capital Costs ($US Millions)	$380.0 (includes $30.8 sunk costs)		
Sustaining Capital Costs ($US Millions)	$138.5		
Reclamation Remediation Costs ($US Millions)	$74.9		
Mitigation Costs ($US Millions)	$41.9 (Includes $8.5 sunk costs)		
Payable Metals	Gold		
Average Ore Grade, Au (troy ounces/ton)	0.060		
Average Mill Recovery %	83.73		
Average Annual Gold (troy ounces)	126,700 (For 13.25 years)		
First Year Gold (troy ounces)	172,000		
Average Annual Gold first 4 years (troy ounces)	155,000		
Byproduct	Silver		
Grade	1.5X the grade of gold		
Recovery	70.0%		
Unit Operating Cost:			
Mining Cost per total ton material	$1.45		
Mining Cost per processed ore ton	$11.18		
Milling Cost per processed ore ton	$10.11		
G&A per processed ore ton	$3.56		
Refining Cost per processed ore ton	$0.18		
Total cost per processed ore ton	$25.03		
Total including $1.24 By-product Credit per processed ore ton	$23.79		
Average Cost Per Ounce of Gold:			
Operating Cost	$476.74 (including refining & by product credit)		
Royalties Cost	NA		
Total Cash Cost	$564.12 (includes mine development, salvage value, mitigation and reclamation closure)		
Financial Indicators:	Base Case	Low Case (-20%)	High Case (+20%)
Gold (price per troy ounce)	$1250	$1000	$1500
Pre-Tax Project Internal Rate of Return (IRR)	18.7%	8.0%	27.9%
Pre-Tax NPV at 5% Discount Rate ($ Millions)	$371.8	$73.8	$669.8
Benefit Cost Ratio at 5% Discount Rate	2.4	1.3	3.5
Pre-Tax Payback (years)	4.2	8.2	2.9
After Tax Project Internal Rate of Return (IRR)	16.7%	6.5%	25.2%
After Tax NPV at 5% Discount Rate ($ Millions)	$290.8	$35.1	$536.4
Benefit Cost Ratio at 5% Discount Rate	2.1	1.1	3.0
After Tax Payback (years)	4.4	8.3	3.0

1.4 SCOPE

M3 prepared this feasibility study update on behalf of HGM including the open pit mining data and relevant information on a potential underground mining operation. The purpose and scope of the open pit study was to report M3's findings as to the economic and technical feasibility of the project. M3's scope of work included:

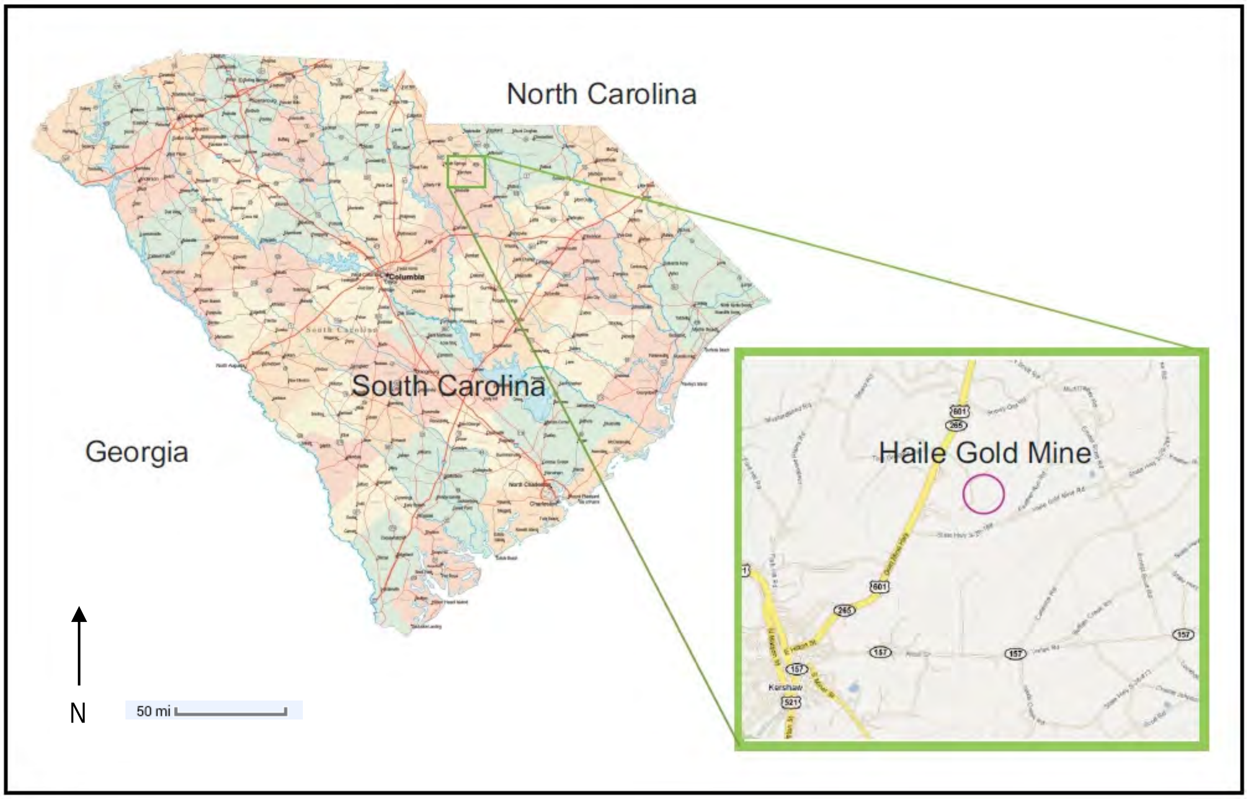
- Overall study report project management
- Detailed level engineering design including equipment specifications and procurement
- Development of drawings to describe the project and support the equipment and material takeoffs
- Solicitation of firm equipment and material costs from vendors
- Preparation of capital estimates and the master capital cost estimate
- Review of processing operating cost estimates
- Development of the economic analysis
- Review of metallurgical testing
- Development of process flow sheets

HGM and its consultants developed:

- Geological interpretation and mineral resource estimation
- Reserve calculation including ore tons and grade plus waste tons
- Mine plans
- Mine manpower and equipment requirements
- Mine capital and operating cost estimates
- Metallurgical testing to support process design and design criteria
- Tailing deposition studies and design
- Environmental and reclamation studies and environmental permits
- Land positions and ownership
- Water supply and hydrogeological studies
- Owner's costs
- Tax Guidelines
- G&A costs

1.5 PROPERTY AND LOCATION

The Haile project property site is located 3 miles northeast of the town of Kershaw in southern Lancaster County, South Carolina (Figure 1-1). Lancaster County lies in the north-central part of the state. The HGM property site is approximately 17 miles southeast of the city of Lancaster, the county seat, which is approximately 30 miles south of Charlotte, North Carolina. It is also approximately 50 miles north east of Columbia, South Carolina.



(Source: State-Maps.org and Google Maps, 2014)

Figure 1-1: Property Location Map

1.6 SITE LAYOUT

The overall project consists of mine development, overburden storage areas, surface water management, process facilities, ancillary buildings, infrastructure and a tailing storage facility. A simplified layout that was used for the detailed engineering design can be seen in Figure 1-2 and Figure 1-3.

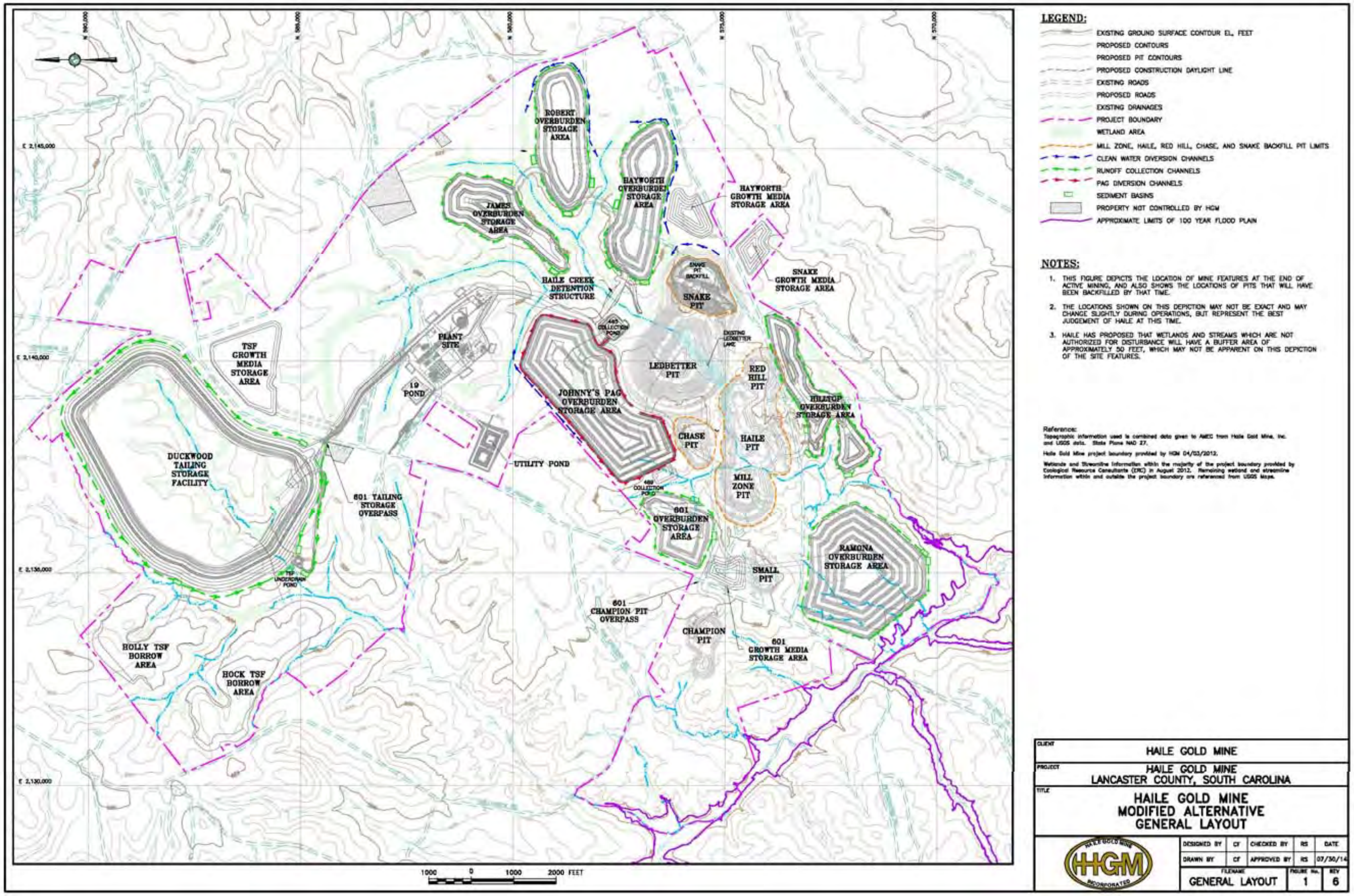


Figure 1-2: Overall Site Layout

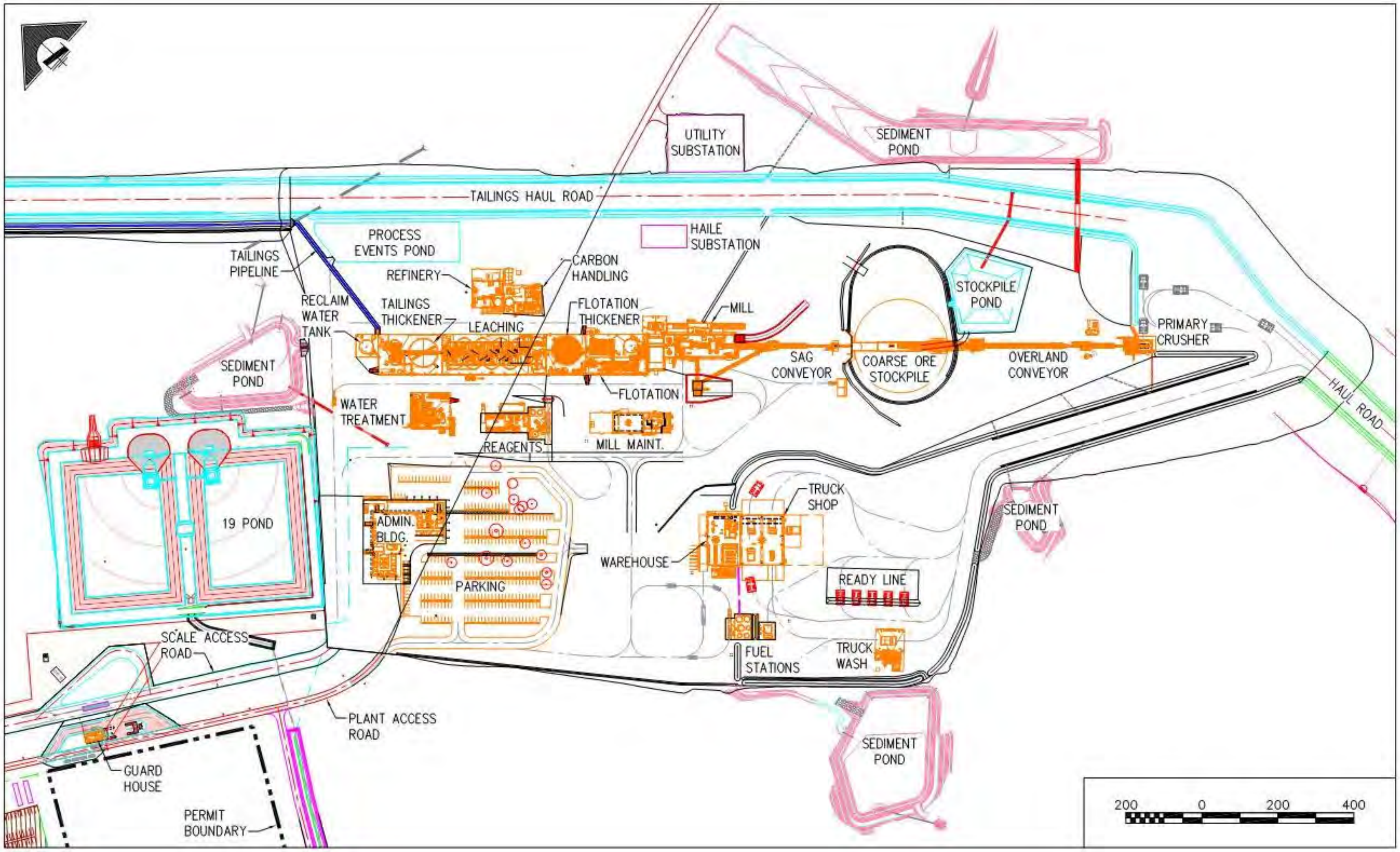


Figure 1-3: Process Area Site Plan

1.7 INFRASTRUCTURE

The HGM property is situated three miles northeast of the Town of Kershaw in Lancaster County, South Carolina, USA. The site is roughly one-hour south of Charlotte, North Carolina and one and one half hour northeast of Columbia, South Carolina. The proximity to existing infrastructure reduces project costs because the project is easily accessible, and there is adequate housing, power, phone, and water. It has the benefit of being bordered by U.S. Highway 601 to the west with the main access to the site provided via Snowy Owl Road. Natural gas, sanitary sewer, and potable water lines run along Highway 601. Power for the Haile property will be provided from Duke Energy, Central Electric Power Cooperative and Lynches River Electric Cooperative. The power transmission infrastructure is well established. A new 69 kV (Lynches River) service will be required.

High annual average precipitation allows for surface water that comes in contact with mining facilities to be used for mill and tailing makeup water. Pit dewatering and pit depressurization wells will provide the remainder of the water makeup. A municipal tap is also planned to provide fresh/firewater to the project.

1.8 OWNERSHIP

HGM, a wholly owned subsidiary of OceanaGold Corporation, acquired the Haile property from Kinross and another private party in October of 2007. After transferring approximately 4,388 acres of land into mitigation projects, HGM owns approximately 5,719 acres of land associated with the project in total, of which approximately 368 acres have been ear-marked for conservancy purposes. HGM owns all land associated with the project as fee simple land, including the surface and mineral rights with no associated royalty.

1.9 GEOLOGY

The north central portion of South Carolina is geologically situated in the Carolina superterrane or Carolinia. This composite terrane consists of the Carolina terrane, the Charlotte terrane, the Augusta-Dreher Shoals terrane and the Kings Mountain terrane. This exotic, volcanic arc terrane formed adjacent to the African continent and was accreted to the North American craton during the Late Ordovician-Silurian (Hibbard et al., 2010) or in the Mid to Late Paleozoic (Hatcher et al., 2007). The Haile gold mine is located within the Carolina terrane which has formerly been called the Carolina slate belt.

The gold mineralization at the Haile property occurs along a trend of moderately- to steeply-dipping ore bodies within a regional corridor which runs from the west-southwest (WSW) to the east-northeast (ENE). The corridor is approximately 3,500 ft (1 km) wide (NNW to SSE) and is over 2 miles (3.4 km) long (WSW to ENE). Most of the mineralization at Haile is restricted to the laminated metasiltstone of the Richtex Formation. The gold mineralized zones within the laminated metasediments can vary in distance from the metavolcanic contact, and can appear at different stratigraphic levels within the metasediments.

The gold mineralization is disseminated and occurs in silica-rich, pyrite-pyrrhotite bearing metasediments. Alteration in the mineralized zones consists of intense quartz-pyrite-sericite with occasional potassium feldspar that grades outward to weak quartz-sericite-pyrite. The unaltered metasediments consist of pyrite bearing, sericite-quartz-chlorite-carbonate phyllites. Within the mineralized zones, quartz is dominant (greater than 80 percent), pyrite is subordinate (generally 3 to 10 percent), and sericite is variable. Moving away from the center of a mineralized zone, quartz and pyrite decrease while sericite increases in abundance. Multiple silicification events have occurred in the mineralized zones. The earliest silicification is massive and penetrative, whereas later silicification appears as re-healed broken angular rock fragments (breccias) followed by a scattered wormy stringer veinlet phase.

Gold mineralization is associated with pyrite, pyrrhotite, and molybdenite mineralization. Detailed ore microscopy and scanning electron microscope mapping indicate that the gold is found as native gold, electrum, and within gold bearing tellurides (Honea, 1992 and Thompson, 2009). These minerals are found as inclusions and along fractures within pyrite. The pyrite is usually present as either disseminated euhedral to subhedral grains or as euhedral to subhedral aggregates. Additional petrologic work has yet to be done within mineralized zones that contain abundant pyrrhotite. Arsenopyrite, chalcopyrite, galena, and sphalerite are also associated with the mineralization. Molybdenite occurs primarily on foliation surfaces or as dispersed fine-grained aggregates in silicified zones. The Haile molybdenite has been dated by Re-Os isotopes at 553.8 $\pm$ 9 and 586.6 $\pm$ 3.6 million years (Ma) (Stein et al., 1997). The first Re-Os age closely approximates the zircon crystallization age of 553 $\pm$ 2 Ma reported by Ayuso et al. (2005) indicating that molybdenite mineralization was concurrent with Persimmon Fork deposition. Seven recent Re-Os molybdenite ages from Haile (Mobley et al., 2014) yield ages ranging 529 to 564 Ma. Four of these samples give a weighted age of 548.7$\pm$2 Ma, indicating that gold mineralization is closely linked to Neoproterozoic volcanism.

1.10 MINERAL RESOURCES AND RESERVES

The mineral resources at HGM are comprised of both potential open pit and underground ores. The open pit component was developed by Independent Mining Consultants, Inc. (IMC) using conventional block model procedures and floating cone pit geometry to determine the component of the deposit that has “reasonable prospects of economic extraction”. John Marek, P.E. of IMC acted as the Qualified Person for the development of the model and the open pit mineral resource estimate.

The resources reported as underground resources in the previous (October 2015) NI 43-101 have been removed from the resource table, given the PEA underground study presented in section 24 evaluates an underground mining option. These previously reported underground resources were defined within underground resource volumes developed by Snowden Mining Industry Consultants in December 2011 and comprised Measured and Indicated Resources of 929 kst at 0.129 oz/st for 119 koz and Inferred Resources of 773 kst at 0.122 oz/st for 94 koz.

The open pit mineral resource is contained in a computer generated open pit (floating cone) to assure reasonable prospects of economic extraction. The open pit cutoff was 0.012 oz/ton.

Table 1-2: Haile Gold Mine Inc. Open Pit Mineral Resource as of 1 January 2012 and 1 November 2014
Resources on this table include the published mineral reserve

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000
Measured	0.012	40,529	0.052	2107.0
Indicated	0.012	36,995	0.049	1813.0
Measured + Indicated	0.012	77,524	0.051	3920.0
Inferred Resource	0.012	21,411	0.33	707.0
Notes: Tonnages are short tons of 2000 lbs Grades are in Troy ounces per short ton Gold price of $1,200 per troy ounce was applied Mineral Resources in this table include the mineral reserve				

The qualified person for the open pit mineral resources is John Marek, P.E. of IMC.

Metal price changes could materially change the estimated mineral resources in either a positive or negative way.

At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Haile mineral resource at a higher level of risk than any other developing resource within the United States.

Mineral reserves for HGM will be produced from an open pit and were developed from the block model and the feasibility mine plan. The mineral reserve is the total of all proven and probable category mineralization planned for processing during the course of the feasibility mine plan. The block model and determination of the mineral reserves were completed by IMC, with John Marek, P.E. acting as the qualified person for the calculation. The mineral reserves are summarized in Table 1-3. The mineral reserves are included within the mineral resource stated in Table 1-2.

Table 1-3: Haile Mineral Reserves as of 1 January 2012, and 1 November 2014

Category	Gold Cutoff oz/t	Tons x 1000	Head Grade Troy Oz/ton	Contained Oz x 1000	Recov Grade Troy Oz/ton	Recovered Oz x 1000
Proven	0.014	21,596	0.064	1,382.1	0.054	1,166.2
Probable	0.014	12,034	0.053	635.7	0.043	515.3
Proven+Probable	0.014	33,630	0.060	2,017.8	0.050	1,681.5

Notes:

Tonnages are short tons of 2000 lbs
Grades are in Troy ounces per short ton
Mineral Reserve Based on $950 / Troy Ounce Gold Price

1.11 MINING

The Haile Gold Mine is planned to be mined using conventional open pit mining methods. A combination of hard rock and soft rock will be encountered in the deposit during the mining process. The majority of the material from the mine will be hard rock which will be drilled and blasted prior to loading.

The mine plan produces 2,555 ktons of gold bearing ore per year for delivery to the process plant (7,000 tpd for 365 days/year). After an 18-month preproduction period, total material movement ramps up to 22,100 ktons/year (60,500 tpd) for the first three years followed by 35,000 ktons/year (95,900 tpd) for four years.

Mining will utilize 20 ft benches. Drilling and blasting will be required for the hard rock units at Haile. The coastal plain sands will not require blasting. Saprolite will require drilling in ore zones for ore control but will require only localized blasting near the bedrock contact.

The major mine equipment that was used as the basis of the study is summarized in Table 1-4.

Table 1-4: Major Mine Equipment

Unit	Initial Fleet for 3 Years	Fleet, Year 4 and Beyond
4 ½” Blast Hole Drills	2	2
6 ½” Blast Hole Drills	2	3
17 Cubic Yd Front Loader	1	1
14.4 Cubic Yd Hyd Shovel	1	1
15.7 Cubic Yd Hyd Excavator	1	2
100 ton Trucks	12	24

Appropriate mine ancillary and support equipment is also planned and scheduled.

The mine production schedule is summarized on Table 1-5. The mine schedule is based on proven and probable mineral material, and the total of material planned for processing is the mineral reserve. The annual mine plan and waste storage drawings are summarized in Section 16 of this document. Quarterly mine plans were developed for the preproduction period and the first 2 years of the mine plan.

Table 1-5: Mine Production Schedule

Year	Recov Cutoff oz/ton	Ore Ktons	Head Grade oz/ton	Recov Grade oz/ton	LG Stkp Ktons	Head Grade oz/ton	Recov Grade oz/ton	Waste Ktons	Total Mat Ktons
ppQ1								150	150
ppQ2								600	600
ppQ3	0.017	8	0.025	0.019	18	0.019	0.014	1,154	1,180
ppQ4	0.017	29	0.027	0.021	27	0.019	0.015	2,834	2,890
ppQ5	0.017	38	0.035	0.028	27	0.018	0.013	5,460	5,525
ppQ6	0.017	79	0.092	0.080	27	0.018	0.014	5,419	5,525
yr1Q1	0.017	325	0.091	0.079	55	0.018	0.013	5,145	5,525
yr1Q2	0.017	638	0.093	0.080	97	0.018	0.013	4,790	5,525
yr1Q3	0.017	638	0.085	0.073	80	0.018	0.013	4,807	5,525
yr1Q4	0.017	639	0.076	0.065	91	0.018	0.014	4,795	5,525
yr2Q1	0.019	639	0.076	0.065	102	0.020	0.015	4,784	5,525
yr2Q2	0.019	639	0.064	0.054	106	0.019	0.014	4,780	5,525
yr2Q3	0.019	639	0.055	0.046	183	0.019	0.014	4,703	5,525
yr2Q4	0.019	638	0.054	0.045	185	0.020	0.015	4,702	5,525
3	0.012	2,555	0.075	0.064	88	0.015	0.011	19,557	22,200
4	0.017	2,555	0.071	0.061	662	0.018	0.014	30,783	34,000
5	0.022	2,555	0.061	0.052	1,366	0.021	0.016	31,079	35,000
6	0.014	2,555	0.062	0.053	209	0.016	0.012	32,236	35,000
7	0.022	2,555	0.068	0.057	1,527	0.021	0.016	29,918	34,000
8	0.010	2,555	0.063	0.054				25,912	28,467
9	0.010	2,555	0.074	0.064				6,563	9,118
10	0.010	2,555	0.073	0.062				5,209	7,764
11	0.010	2,555	0.051	0.042				4,832	7,387
12	0.010	836	0.023	0.018				1,128	1,964
Total		28,780	0.066	0.056	4,850	0.020	0.015	241,340	274,970

Table 1-6: Mill Feed Schedule

Year	Cutoff oz/ton	Ore Ktons	Contd Grade oz/ton	Recov Grade oz/ton
yr1Q1	0.017	479	0.082	0.071
yr1Q2	0.017	638	0.093	0.080
yr1Q3	0.017	638	0.085	0.073
yr1Q4	0.017	639	0.076	0.065
yr2Q1	0.019	639	0.076	0.065
yr2Q2	0.019	639	0.064	0.054
yr2Q3	0.019	639	0.055	0.046
yr2Q4	0.019	638	0.054	0.045
3	0.012	2,555	0.075	0.064
4	0.017	2,555	0.071	0.061
5	0.022	2,555	0.061	0.052
6	0.014	2,555	0.062	0.053
7	0.022	2,555	0.068	0.057
8	0.010	2,555	0.063	0.054
9	0.010	2,555	0.074	0.064
10	0.010	2,555	0.073	0.062
11	0.010	2,555	0.051	0.042
12	0.010	2,555	0.021	0.016
13	0.010	2,555	0.020	0.015
14	0.010	576	0.020	0.015
Total		33,630	0.060	0.050

Note: 1,719 Ktons in Year 12 come from the low grade stockpile. In years 13 and 14, all of the ore comes from the low grade stockpile.
Note: Tonnages are Dry Short Tons.

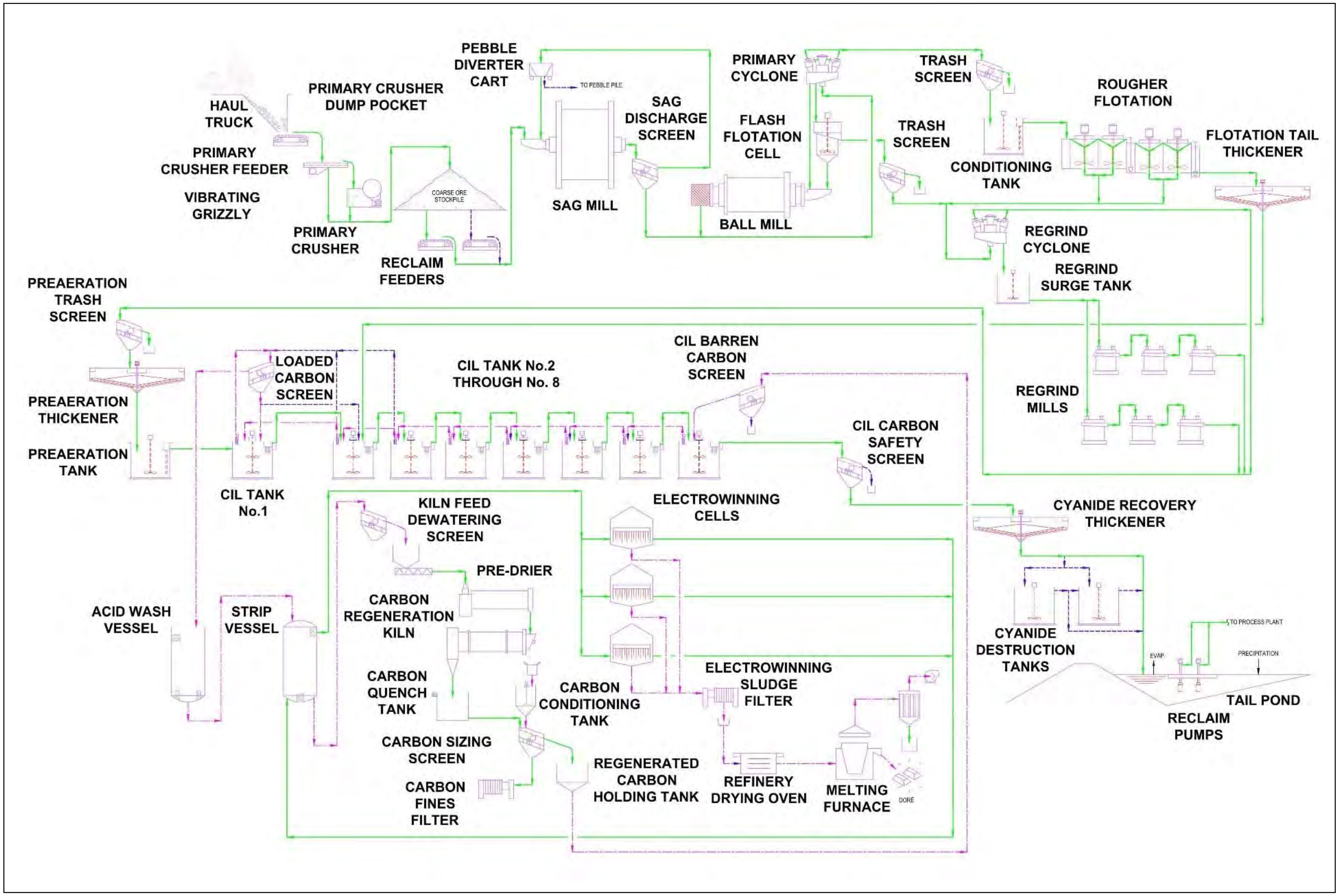
1.12 METALLURGY AND PROCESS PLANT

Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet has been used as the basis of process design. The relative low variability of flotation test work indicates that the mineralized zones are relatively similar in terms of ore grindability, chemical and mineral compositions, and flotation and cyanide leaching response.

The data developed in the metallurgical test programs has been used to establish a relationship between overall gold recovery and mill head grade that has been described by an equation and graph. For example, at a mill head grade of 0.060 opt the recovery equation and graph predicts a gold recovery of 83.7%.

The plant will consist of the following major process steps:

- Crushing and conveying
- Stockpile reclaiming
- Grinding
- Flotation
- Regrinding
- Carbon in leach (CIL) leaching of flotation concentrate
- CIL leaching of flotation tailing
- Acid washing of carbon
- Stripping of carbon
- Electrowinning and refining
- Carbon regeneration
- CIL tailing thickening, cyanide recovery, detoxification and storage



(Source: M3, 2014)

Figure 1-4: Simplified Process Flow Sheet

1.13 TAILING FACILITY

Tailing slurry will be pumped from the mill to a geosynthetic-lined tailing storage facility (TSF). The TSF will be constructed from local materials and utilize the downstream construction method. Process water will be reclaimed from the TSF by utilizing self-priming centrifugal pumps placed on an access ramp in the southeast corner of the facility. Water collected from within the TSF basin and piped to the underdrain collection pond will also be reclaimed. Reclaim water is sent back to the process facilities for re-use.

1.14 ENVIRONMENTAL AND PERMITTING

The project is somewhat unique in that it occurs wholly on private land owned or controlled by HGM and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies. In addition, there is no potential for the federal government to impose a royalty by an amendment to the 1872 Mining Law (General Mining Act of 1872).

Since the property has been mined in the past, a significant amount of background and environmental baseline data existed while additional data was collected through the Environmental Impact Statement (EIS) process. This data continues to be collected. Major permits/certifications obtained include Mine Operating Permit, 404 Dredge and Fill Permit, 401 Water Quality Certification, air quality permit. NPDES Permits (wastewater discharge, wastewater treatment system construction, and stormwater).

The environmental and permitting assessment of the open pit component was developed by Gochnour & Associates, Inc. (G&A). Lee "Pat" Gochnour, MMSA QP Environmental Permitting and Compliance (Member # 01166 QP) of G&A who acted as the Qualified Person for the development of the environmental and permitting assessment associated with the open pit mineral resource estimate.

The environmental and permitting overview/review for the underground mining assessment in Section 24 was prepared by Scott McDaniel.

1.15 CAPITAL COSTS

Initial capital costs have been estimated for the Haile Gold project based on equipment quotations, detailed engineering design and material quantities utilizing unit rates from historic data, published sources and local contractors. The estimate includes all evaluated portions of the project including the process, tailing, and mining facilities. The costs also include pre-production mining, owner's costs and contingency. A more detailed breakdown can be found in Section 21 of this report.

Note that in February 2016, the estimate of initial capital for the project at $380M was increased from the initial estimate of $333M, as published in the previous NI 43-101 report issued 13 October 2015. The increase has principally been due to design enhancements to the project, including:

- Additional loading equipment to reduce the potential for ore dilution during mining
- The addition of a run-of-mine (ROM) pad to allow for more effective ore blending to the mill
- Installation of a crushed ore bin to minimize dust from the crushing circuit
- Installation of a larger flash flotation cell to improve metallurgical recoveries
- Enhancement of control systems for the process plant
- Upgrades to IT systems

A summary of the initial capital cost estimate update is provided in Table 1-7.

Table 1-7: Updated Initial Capital Costs Summary

Area of Discipline	February 2016 ($Millions)	October 2015 ($Millions)
Direct Costs	$222	$197
Owner's Costs	$28	$18
EPCM	$40	$30
Mining Capital Equipment	$53	$46
Mining Pre-Production Opex	$33	$25
Contingency	$0	$17
Total	**$380**	**$333**

1.16 OPERATING COST

The operating and maintenance costs for the HGM operations have been estimated in detail and are summarized by areas of the project. Cost centers include Mine operations, Process Plant operations, and General and Administration (G&A). Operating costs were determined for the life of mine (LOM), based on an annual ore tonnage of 2.5 million tons. The unit operating costs are shown in Table 1-8.

Table 1-8: Unit Operating Cost (LOM)

Item	$ per ton ore
Mining	$11.18
Processing	$10.11
General and Administration	$3.56
Shipping/Refining	$0.18
Total	**$25.03**

1.17 ECONOMIC ANALYSIS

The Haile Gold Project economics were done using a discounted cash flow model. The financial indicators examined for the project included the Net Present Value (NPV), Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections were estimated over the life of the mine based on capital expenditures, production costs, transportation and treatment charges and sales revenue. The life of the mine is 13 years.

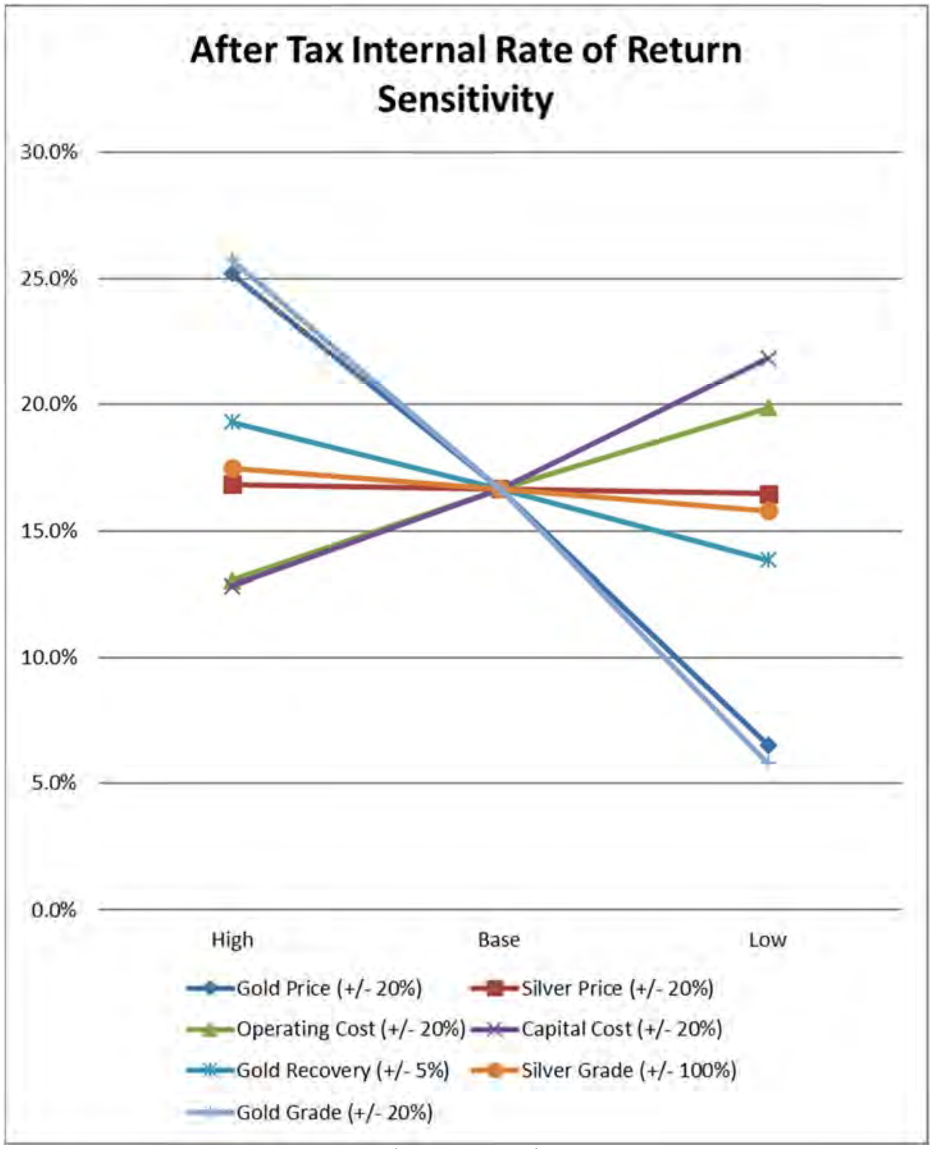
As of the November 2014 Technical Report, HGM has spent $30.8 Million of capital on the project. Those costs are considered "sunk" in the economic model.

The financial indicators based on a 100% equity case are summarized as follows:

Table 1-9: After-Tax Financial Indicators at $1250 Gold

	After-Tax
IRR	16.7%
NPV @ 0%	$561.5 Million
NPV @ 5%	$290.8 Million
NVP @ 10%	$129.5 Million
Payback Period	4.4 Years

Sensitivities were run for seven variables as in Figure 1-5. The sensitivity analysis indicates that the project is most sensitive to gold price.



(Source: M3, 2016)

Figure 1-5: Financial Sensitivities

1.18 PROJECT EXECUTION PLAN

The proposed project execution plan incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the project at the lowest possible capital cost, on schedule. Primary objectives during construction will include safety, quality, and environmental compliance.

Table 1-10: Haile Key Pre-Production Milestones

Milestone	Date
Detailed Engineering	90+% Complete
Equipment Procurement	90+% Complete
Began Construction	Second Quarter 2015
Began Pre-Production Mining	Second Quarter 2015
Start Up	Fourth Quarter 2016

1.19 AUTHORS' CONCLUSIONS AND RECOMMENDATIONS

HGM and its consultants have developed near detailed level design and the project is currently in construction. The results of the Report confirm that the Haile project is technically feasible. The mining and process methods are typical and do not require any specialized technology. Project economics are favorable when $1250/troy ounce gold price is used.

The project is located in a relatively populated region, which greatly favors project execution and operation. The climate is moderate and the project location is relatively flat. The project schedule is reasonable. Procurement of long lead mining and processing equipment has begun with the SAG mill and Ball mill on site and much of the mining equipment purchased. A significant amount of the remaining equipment has been procured and is awaiting fabrication.

See Section 25 for more interpretations and conclusions.

2 INTRODUCTION

Site visits and areas of responsibility are summarized in Table 2-1 for the Qualified Persons ("QP").

Table 2-1: Dates of Site Visits and Areas of Responsibility

Name	Last Site Visit Date	Area of Responsibility
Daniel H. Neff, PE	September 2016	Sections 1, 2, 3, 4, 5, 6, 18, 19, 21 (except 21.4, 21.5), 22, 23, 25, 26, and 27.
Erin L. Patterson, PE	October 2016	Sections 1, 13, 17, 25, 26 and 27.
Lee "Pat" Gochnour, MMSAQP	November 2012	Sections 1, 20, 25, 26 and 27.
John Marek, PE	June 2015	Sections 1, 7, 8, 9, 10, 11, 12, 14, 15, 16, 21.4, and 21.5
Carl John Burkhalter, PE	August 2016	Section 1, 18, 25, 26, and 27.
Jonathan Moore, BSc (Hons)Geology, MAusIMM(CP)	October 2016	Sections 24.7, 24.8, 24.9, 24.10, 24.11, 24.12, 24.14, and corresponding items in 24.1, 24.24, 24.25
David Carr, BEng Metallurgical	October 2016	Sections 24.13, 24.17, 24.19, 24.21, and corresponding items in 24.1, 24.24, 24.25
John Tinucci, PhD, PE	N/A	Sections 24.16.2 and corresponding items in 24.1, 24.24, 24.25
Robert P. Shreiber, PE, D.WRE, BCEE	N/A	Sections 24.16.3 and corresponding items in 24.1, 24.24, 24.25
Patrick Williamson, MSc Geology, MMSAQP	N/A	Sections 24.16.4 and corresponding items in 24.1, 24.24, 24.25
Joanna Poeck, BEng Mining, SME-RM, MMSAQP	N/A	Sections 24.2, 24.3, 24.4, 24.5, 24.6, 24.15, 24.16.1, 24.16.5, 24.16.6, 24.16.7, 24.16.8, 24.16.9, 24.23, 27 and corresponding items in 24.1, 24.24, 24.25
Jeff Osborn, BEng Mining, MMSAQP	April 2016	Sections 24.16.5, 24.16.6, 24.16.7, 24.16.8, 24.16.9, 24.18, 24.21, and corresponding items in 24.1, 24.24, 24.25
Scott McDaniel, BSc. Metallurgical Engineering	Stationed On-Site	Sections 24.20 and corresponding items in 24.1, 24.24, 24.25
Grant Malensek, MEng, PEng/PGeo	N/A	Sections 24.21, 24.22, and corresponding items in 24.1, 24.24, 24.25

2.1 PURPOSE

This document was prepared in order to provide a technical evaluation consistent in format with the NI 43-101 standard and to present data and information developed to substantiate technical and economic viability of the Haile Project in Lancaster County, South Carolina.

This report provides an independent Technical Report, compliant with the Canadian National Instrument 43-101 - Standards of Disclosure for Mineral Projects (NI 43-101).

This report was prepared by M3 Engineering & Technology Corporation (M3) at the request of Haile Gold Mine Inc., a wholly-owned subsidiary of OceanaGold Corporation.

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2.2 SOURCES OF INFORMATION

This report is based in part on internal company technical reports, previous feasibility studies, maps, published government reports, company letters and memoranda, and public information as listed in the references section in the conclusion of this report.

2.3 TERMS OF REFERENCE

The important terms used in this report are presented in Table 2-2.

Table 2-2: Terms and Definitions

Abbreviation	Unit or Term
%	percent
°	degree (degrees)
°F	Temperature in Degrees Fahrenheit
2D	two-dimensional
3D	three-dimensional
AGP or AP	acid generating potential
ARD	acid rock drainage
ASTM	American Society for Testing and Materials
AT	after tax
Au	gold
BT	before tax
BTS	Brazilian tensile strength
cfm	cubic feet per minute
CIL	Carbon-In-Leach
CoG	cut-off grade
CPS	Coastal Plan Sand
CRF	cemented rock fill
DHEC	Department of Health and Environmental Control
DSS	direct shear strength
ELOS	equivalent linear overbreak/slough
EPCM	Engineering, Procurement and Construction Management
FF/m	frequency fracture per meter
ft	foot (feet)
ft^3	Cubic feet
GPa	gigapascal
gpm	gallons per minute
HGM	Haile Gold Mine
HDPE	height density polyethylene
hp	horsepower

Abbreviation	Unit or Term
in	inch
IMC	Independent Mining Consultants
IRR	initial rate of return
IRS	intact rock strength
ISRM	International Society of Rock Mechanics
Ja	joint alteration
Jn	joint number
Jr	joint roughness
kN	kilonewton
kN/m^3	kilonewton per cubic meter
koz	thousand troy ounce
kt	thousand tonnes
kV	kilovolt
kW	kilowatt
lb	pound
LHD	long-haul-dump
LoM	life-of-mine
m	meter
m	meter
m^3	cubic meter
M3	M3 Engineering & Technology Corporation
ML	metal leaching
MOA	Memorandum of Agreement
MPa	megapascal
Mst	million short tons
MW	million watts
New Fields	New Fields LLC, Denver CO
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NNP	net neutralization potential

Abbreviation	Unit or Term
NPV	net present value
OP	open pit
OSA	overburden storage area
oz	troy ounce
PAG	potential acid generating
PEA	preliminary economic assessment
PLT	point load test
PMP	Probable Maximum Precipitation
ppb	parts per billion
ppm	parts per million
Q	rock mass rating (according to the Barton 1974 criteria)
Q'	Barton's (1974) Q with the JW and SRF both set to a value of 1
QA/QC	Quality Assurance/Quality Control
RMI	Romarco Minerals, Inc.
RMR	rock mass rating (according to the Bieniawski 1989 criteria)

Abbreviation	Unit or Term
RoM	run-of-mine
RQD	rock quality designation
sec	second
S.G.	Specific Gravity
SRF	stress reduction factor
st	short ton (2,000 pounds)
st/d	short tons per day
STD	standard deviation
t/d	metric tonnes per day
TCC	total cash costs
TCR	total core recovery
TCS	triaxial compressive strength
TSF	tailings storage facility
UCS	uniaxial compressive strength
UG	underground
US$	United States Dollar
V	volts
VFD	variable frequency drive
W	watt
y	year

2.4 UNITS OF MEASURE

This report uses English Units expressed in short tons (2,000 pounds), feet, and gallons consistent with US standards. The monetary units are expressed in US Dollars.

3 RELIANCE ON OTHER EXPERTS

M3 relied upon contributions from a range of technical and engineering consultants as well as HGM. M3 has reviewed the work of the other contributors and finds this work has been performed to normal and acceptable industry and professional standards. In conclusion, M3 is not aware of any reason why the information provided by these contributors cannot be relied upon.

Owner's environmental and permitting costs were supplied by HGM staff. In addition, HGM provided all Owner's costs in the capital cost estimate.

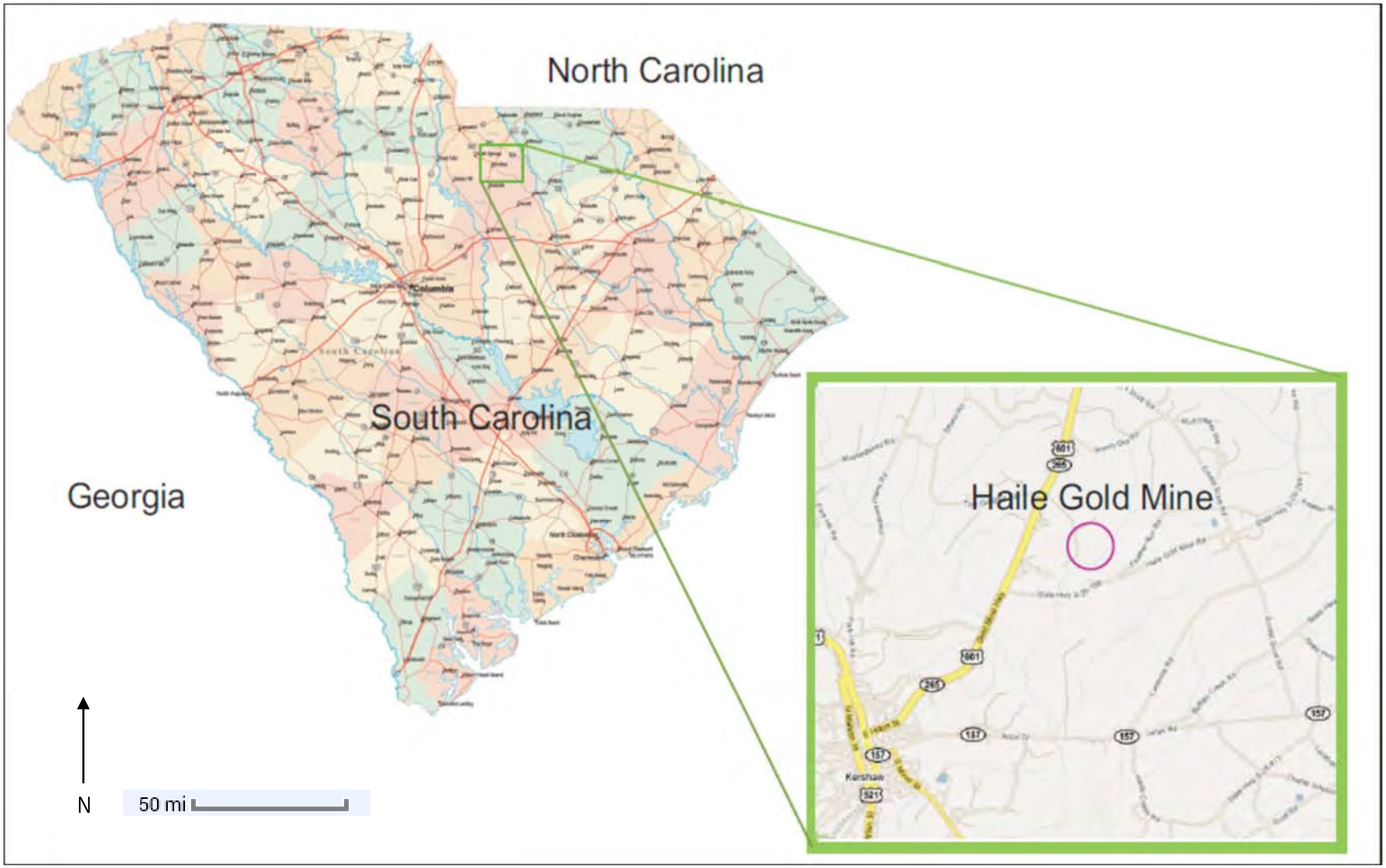
An independent verification of land title and tenure was not performed. M3 has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties. Likewise, HGM has provided data for and verified water rights, land ownership, and claim ownership.

A draft copy of the report has been reviewed for factual errors by HGM. Any changes made as a result of these reviews did not involve any alteration to the conclusions.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

The Haile property site is located 3 miles northeast of the town of Kershaw in southern Lancaster County, South Carolina, Lancaster County lies in the north-central part of the state. The Haile Gold Mine is approximately 17 miles southeast of the city of Lancaster, the county seat, which is approximately 30 miles south of Charlotte, North Carolina. The approximate geographic center of the property is at 34° 34' 46" N latitude and 80° 32' 37" W longitude. The mineralized zones at Haile lie within an area extending from South Carolina state plane coordinates 2136300 E to 2142300 E, and from 573700 N to 576300 N, (1927 North Datum).



(Source: State-Maps.org and Google Maps, 2014)

Figure 4-1: General Location Map of the Haile Gold Mine

4.2 OWNERSHIP

HGM provided an inventory of property that is owned both within the project boundary and as a buffer and land for other purposes outside the project boundary. After transferring approximately 4,388 acres of land into mitigation projects, HGM owns approximately 5,719 acres of land in total, of which approximately 368 acres have been ear-marked for conservancy purposes.

HGM owns additional land that is not associated with the project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Haile property is accessible by car or truck by taking U.S. Highway 601 northeast from the town of Kershaw for approximately 2 miles, with the main access via Snowy Owl Road, following the partial closure of Haile Gold Mine Road.

5.2 CLIMATE

This portion of South Carolina has a humid subtropical climate. Summers are hot and humid with daytime temperatures averaging 85°F to 95°F. Winters are mild and wet, but overnight temperatures can be below freezing. Average annual precipitation approaches 50 inches while annual evaporation is only 30 inches. Precipitation is abundant throughout the year with March being the wettest month. Snowfall annually is often insignificant and averages less than 3 inches per year. Regionally, South Carolina averages approximately 50 days of thunderstorm activity and 14 tornadoes per year. The operating season is considered to be year-round.

5.3 LAND RESOURCES AND INFRASTRUCTURE

Local resources (labor force, manufacturing, housing, etc.) and infrastructure are already in place and available for the operation of the Haile project. Several small and modest-sized communities exist in every direction from and in close proximity to the Haile project area. Equipment and sources of both logistical and professional expertise can be obtained from the major cities of Charlotte, N.C., and Columbia, S.C., which are both within one-hour travel. More than one large industrial contractor is within close proximity to the site and can provide a skilled workforce for the construction project.

Power is available in the area via an existing 44 kV transmission grid or a 69 kV transmission grid.

5.4 PHYSIOGRAPHY

The Haile Gold Mine and its surroundings occur within the Sand Hills sub-province of the Piedmont physiographic province of the southeastern United States. This province trends from southwest to northeast and is bounded by the Coastal Plain to the southeast and the southern Appalachian Mountains to the northwest. Gentle topography and rolling hills, dense networks of stream drainages, and white sand to red-brown lateritic soils characterize the Sand Hills sub province.

The elevation of the property ranges approximately from 400 ft (122 m) to 550 ft (168 m) above mean sea level. The topography is the result of dissection by the perennial, southwest-flowing Haile Gold Mine Creek and by its intermittent, southeast and northwest-flowing tributaries. The surface ground slopes within the drainages are gentle to moderate (approximately 9 to 13%) and the slopes above the drainages are gentle to nearly flat (less than 1%). Haile Gold Mine Creek enters the southeast-flowing Little Lynches River at a point approximately 1 mile (1.6 km) southwest from the mine site. The property is heavily wooded with both pine and hardwood forests. Pine timber harvesting occurs frequently in and around the property area as each harvestable tract matures.

5.5 LABOR

There are large highly industrial population centers near the project site. There is adequate labor for both construction and operations.

6 HISTORY

Gold was first discovered in 1827 near Haile by Colonel Benjamin Haile, Jr. in the gravels of Ledbetter Creek (now the Haile Gold Mine Creek). This led to placer mining and prospecting until 1829, when lode deposits at the Haile-Bumalo pit site were found. Surface pit and underground work continued at the Haile-Bumalo site for many years. In 1837, a five-stamp mill was built on site (Newton et al., 1940). Gold production and pyrite-sulfur mining for gun powder continued through the Civil War. General Sherman's Union troops invaded the area and burned down the operations near the war's end (Culvern, 2006).

In 1882, a twenty-stamp mill was constructed by E.G. Spilsbury and operated continuously until a fatal boiler explosion killed the mine manager in 1908. During that time, Capt. Adolph Thies developed the Thies barrel chlorination extraction process on site and improved gold recovery from Haile sulfides (Pardee & Park, 1948). During this 26-year operation period, mining grew to include the Blauvelt, Bequelin, New Bequelin, and Chase Hill areas. In 1913, an attempt to operate a cyanide plant to extract gold from mine tailings turned out to be unsuccessful. Pyrite used to produce sulfuric acid was mined at Haile from 1914 to 1918 (Newton et al., 1940).

From mid-1937 to 1942, larger-scale mining was undertaken on site by the Haile Gold Mines Company. The property then consisted of owned or leased ground totaling about 3,300 acres (1,335 hectares). The operator was financed out of New York by the Barlowe Corporation (Newton et al., 1940). Most of the main pits were mined to the 150-ft level with some underground operations at Haile-Bumalo reaching the 350-ft level (Pardee & Park, 1948). This period was also significant because the Red Hill Deposit was discovered by crude induced polarization techniques next to the Friday pyrite diggings (Newton et al., 1940). This fairly large operation was shut down by presidential decree (L208) in 1942 because of World War II. By this time, the Haile Mine had produced over US$6.4 million worth of gold (in 1940 dollars) (Newton et al., 1940).

From 1951 to the present, the Mineral Mining Company (Kershaw, South Carolina) has mined Mineralite® from open pits around the Haile property. This industrial product is a mixture of sericite, kaolinite, quartz, and feldspar and is used in manufacturing insulators and paint base.

In 1966, Earl M. Jones conducted exploration work in the area and eventually interested Cyprus Exploration Company in the project. Cyprus worked Haile from 1973 to 1977. Following this, many companies explored the area around the Haile mine, including Amselco, Amax, Nicor, Callaghan Mining, Westmont, Asarco, Newmont, Superior Oil, Corona, Cominco, American Copper and Nickel, Kennecott, and Hemlo.

Between 1981 and 1985 Piedmont Land and Exploration Company (later Piedmont Mining Company), explored the historic Haile Mine and surrounding properties. Piedmont mined the Haile deposits from 1985 to 1992, producing 85,000 ounces of gold from open pit heap leach operations that processed oxide and transitional ores. New areas mined by Piedmont included the Gault Pit (next to Blauvelt), the 601 pits (by the US 601 highway), and the Champion Pit. They also expanded the Chase Hill and Red Hill pits and combined the Haile-Bumalo zone into one pit. They also discovered the large Snake deposit sulfide gold resource and mined its small oxide cap. Piedmont extracted gold ores from a mineralized trend a mile long, from east to west.

In June of 1991, Amax signed an agreement to evaluate the site to determine if it should enter a joint venture on the Haile property. During that evaluation period, core drilling that stepped north of the Haile-Bumalo area resulted in the discovery of the new sulfide resource at the Mill zone (under the old 1940's mill). With the satisfactory verification of Piedmont data, Amax and Piedmont entered into a Joint Venture agreement and established the Haile Mining Company (HMC) in May 1992.

From 1992 to 1994, HMC completed a program of exploration/development drilling, property evaluation, mineral resource estimation, and technical report preparation. During this period, the Ledbetter resource zone was discovered under a mine haul road. At the end of the Amax / HMC program in 1994, the gold reserve was stated by HMC as

780,000 ounces of gold contained within 8,736,000 tons with an average grade of 0.089 opt Au. A qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. HGM is not treating the historical estimate as current mineral reserves. Because of unfavorable economic conditions at the time, Amax did not proceed with mining, but began a reclamation program to mitigate ARD conditions at the site.

Kinross acquired Amax in 1998, assumed Amax's portion of the Haile joint venture, and later purchased Piedmont's interest. Because Haile was a low priority compared to larger and more profitable prospects, Kinross decided not to reopen the mine but did continue the closure/reclamation effort. The closure/reclamation has proceeded through the present and has been considered successful.

HGM acquired the Haile property from Kinross in October of 2007 and began a confirmation drilling program in late 2007. HGM completed the confirmation drill program in early 2008 and began infill and exploration drilling. The drill program was accelerated in early 2009 with a major reverse circulation drilling program. That program was continued into 2013. Data from the drill program that was available as of November 17, 2011 has been used in this update of the mineral resource estimate.

HGM submitted a Feasibility Study on the project in February of 2011. An updated resource estimate was submitted in March of 2012. HGM has completed a large portion of detailed engineering for the project.

In May of 2015, construction of the project began. In October of 2015, OceanaGold Corporation acquired Romarco Minerals Inc.

7 GEOLOGICAL SETTING AND MINERALIZATION

This section was written by James Berry, former Chief Geologist at the Haile Gold Mine. John Marek of IMC has reviewed this text and has sufficient comfort with the information to act as the Qualified Person under NI 43-101.

7.1 REGIONAL, LOCAL AND PROPERTY LOCATION

The north central portion of South Carolina is geologically situated in the Carolina superterrane or Carolinia (Hatcher et al., 2007 and Hibbard et al., 2007). The Carolina superterrane or Carolinia consists of the Carolina terrane, the Charlotte terrane, the Augusta-Dreher Shoals terrane and the Kings Mountain terrane. These exotic, volcanic arcs formed adjacent to the African continent and were accreted to the North American craton during the Late Ordivician–Silurian (Hibbard et al., 2010) or Mid to Late Paleozoic (Hatcher et al., 2007). The Haile gold mine is located within the Carolina terrane which has formerly been called the Carolina Slate Belt.

The Brewer gold mine is located approximately ten miles to the northeast of the Haile mine and the Ridgeway mine is located thirty miles to the southwest. All of the deposits are hosted in a similar geologic setting within the Carolina terrane. The Haile, Ridgeway, Brewer, and Barite Hill gold mines are hosted at the contact between metamorphosed volcaniclastic and metamorphosed sedimentary rocks of Neoproterozoic to Early Cambrian age. This volcanic arc assemblage was deposited in a back arc or fore arc setting. The metamorphosed volcaniclastic and interbedded epiclastic lithologies are called the Persimmon Fork Formation, and the metamorphosed sedimentary-dominated sequence is termed the Richtex Formation (Maher et al., 1991). The Persimmon Fork Formation was derived from volcanic material that contains a continuous range of compositions from basaltic to rhyodacitic and a transitioning geochemical signature from tholeiitic to calc-alkaline (Shelley, 1988), indicating a mature arc setting on an older arc sequence or thinned continental crust. The Carolina superterrane was metamorphosed to amphibolite grade conditions in the Charlotte, Kings Mountain and Augusta-Dreher Shoals terranes and to greenschist grade conditions within the Carolina terrane (Secor and Snoke, 2004). Dennis and Wright (1997) have possibly constrained the timing of this metamorphic/deformational event between 550 and 535 Ma based on the presence of synkinematic deformational fabrics within the Longtown metagranite and the absence of foliation within the Mean Crossroads igneous complex. They also propose that this early deformational event resulted from intra-arc collision. Hibbard et al. (2010) report evidence of a Late Ordivician-Silurian tectonothermal event in central North Carolina. The extent of deformation during the Alleghanian orogeny (320 to 270 Ma) within Carolinia is localized to mylonitic zones with normal and dextral strike-slip sense of shear (Secor et al., 1986). Alleghanian deformation and metamorphism are documented in the Augusta-Dreher Shoals terrane which is several miles south of the Haile mine area. Post-tectonic granites intruded the Carolina superterrane at the end of the Alleghanian orogeny. These granites have variably developed contact metamorphic aureoles. Alleghanian-aged granites are exposed to the northeast and west of the Haile mine property. Intermediate dikes of Carboniferous age (Mobley et al., 2014) and Mesozoic diabase dikes also intrude the Carolina terrane. The diabase dikes were produced when North America rifted from Africa during the Mesozoic. Deep erosion and extensive weathering have occurred within the region since the Mesozoic, due to a near tropical, humid paleo-environment. The intensity of this weathering event has significantly altered the original composition and textures of the rocks. Regional submersion during the Cretaceous resulted in the deposition of sands and clay above the saprolite. Continental uplift and regression of the Atlantic have led to continued and ongoing erosion.

Figure 7-1 (after Hibbard et al. 2006) showing the locations of significant gold deposits within the Carolina terrane.

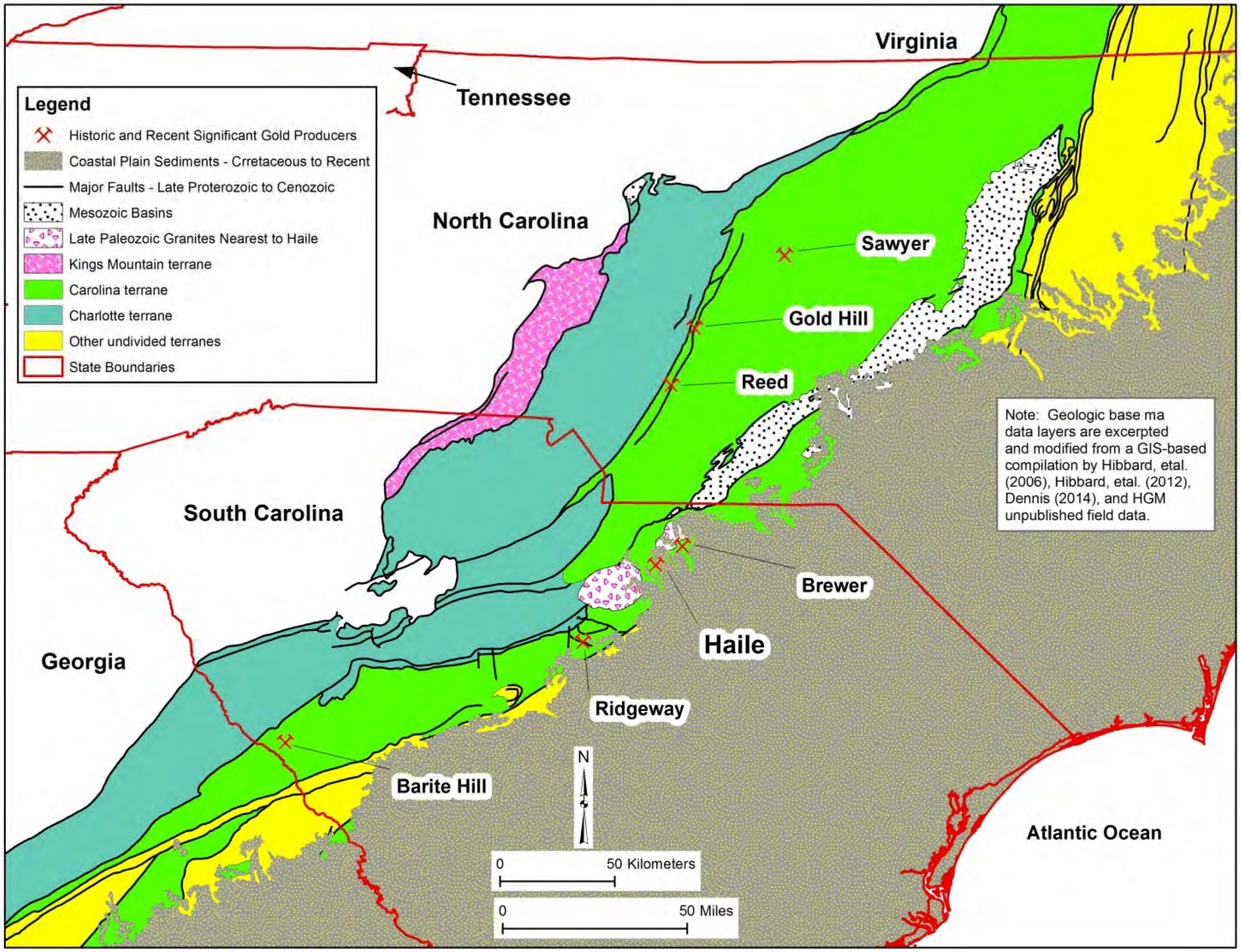


Figure 7-1: Gold Deposit Locations within the Carolina Terrane

7.1.1 Lithology

Two Neoproterozoic to Cambrian-aged rock units are found in the project area. The Persimmon Fork and the Richtex Formations were deposited in an arc-related environment and are known to be complexly folded with local shearing. The accompanying metamorphism has obscured some of the primary depositional or volcanic textures making the exact geologic history difficult to interpret. These units are crosscut by northwest-trending, Triassic to Jurassic age diabase dikes in the mine area, and Carboniferous granites have intruded the Neoproterozic units within a few miles of the site. Saprolite of variable thickness has developed within the crystalline rock. The bedrock and saprolite are overlain by Coastal Plain sediments. Figure 7-2 is a schematic geologic map of the Haile property reflecting bedrock patterns beneath the Coastal Plain sediments and saprolite.

7.1.1.1 Richtex Formation

The Richtex Formation is the primary host rock for gold mineralization and is dominated by sedimentary lithologies. The unit is characterized by thin, alternating rhythmic bands of silt, clay, and sand, which are metamorphosed into a finely banded phyllitic metasiltstone with a "poker chip" appearance. The Richtex Fm. is generally well foliated and crenulation surfaces are common. When strongly mineralized, the metasiltstone is highly silicified and has a pale, steel gray color. The unit often contains strong penetrative cleavage, and is colored light gray, green, tan, or brown. When weathered, the unit is very light gray or pink. Laminae and bedding are often folded, and sometimes disrupted by passive-slip shearing or dissolution. The mineral composition is comprised of quartz, white mica (up to 50 percent), pyrite (generally less than 10 percent), pyrrhotite, and chlorite, with lesser amounts of biotite and calcite. The unit contains lenses of greywackes, sandstones, and conglomerates that contain clasts of volcanic rock or siltstone. The coarser clastic units are poorly sorted and less likely to be as strongly foliated as the siltstones. The coarser grained lithologies of the Richtex Fm. exhibit cleavage development and flattening of clasts. Recent detrital zircon ages indicate that the Richtex was derived from a peri-Gondwanan source (Mobley et al., 2014). The contact between the Richtex and the Persimmon Fork is typically gradual but sharp contacts have also been observed.

7.1.1.2 Persimmon Fork Formation

The Persimmon Fork Formation consists of felsic volcanic and volcaniclastic rocks that are rhyodacitic to andesitic in composition. The unit is generally buff, gray, white, or green in color and is distinctive due to the lack of bedding and the presence of feldspar clasts. Albite, quartz, white mica, biotite, and chlorite are the dominant mineralogy and the unit locally contains calcite and epidote. The unit is more massive in appearance than the adjacent metasediments, but has a well-developed, penetrative cleavage. The Persimmon Fork Fm. contains variable amounts of sub-rounded or sub-euhedral albite grains in a quartz-mica matrix. Portions of this unit contain poorly sorted, rounded to angular volcanic clasts. Overprinting of primary textures by alteration, metamorphism, and weathering events has made interpretation of this unit difficult. The textures present within the Persimmon Fork indicate that it may be a syneruptive volcaniclastic sediment although pyroclastic flows and shallow intrusives cannot be ruled out. Uranium-lead weighted ages from zircons in the metavolcanic units have yielded crystallization ages of 553 $\pm$2 Ma (Ayuso et al., 2005). A portion of the spot zircon ages in some samples are younger and may be attributed to later metamorphic events.

7.1.1.3 Lamprophyre Dikes

These dikes intrude the previous units, are medium to fine-grained with porphyritic, spheroidal, or mottled texture and they are sometimes strongly altered. The dikes are gray, buff, tan, and green in color. Below the saprolite zone, the dikes can contain biotite, plagioclase, clay, chlorite, and carbonate. Some of the dikes contain distinctive biotite phenocryts and are lamprophyric in composition. These dikes either trend with, or are normal to the foliation. These non-foliated dikes are Alleghanian in age based on recent $^{40}Ar/^{39}Ar$ geochronology (Mobley et al., 2014).

7.1.1.4 Mesozoic Diabase Dikes

The diabase dikes are basaltic in composition, medium- to fine-grained, dense, black, green, or brown in color, magnetic, and they can also have talc vein fillings. Some of the dikes exhibit narrow chilled margins, and they also produce local contact metamorphism in the adjacent wallrock. Diabase dikes are occasionally associated with the earlier lamprophyre dikes. The Mesozoic dikes trend north or northwest throughout the Carolina terrane and generally have steep dips. Large amounts of displacement are not seen across the diabase dikes in the mine area, and some dike trends consist of subparallel sets of dikes.

7.1.1.5 Saprolite

Saprolite is a thick, structureless, unconsolidated, kaolin-rich, red-brown to white residuum that has been derived from intense weathering of the underlying bedrock. Saprolite development is usually thickest in near-surface occurrences of metavolcanic rocks and thinnest in silicified metasediments. The saprolite also thins where it has been eroded in incised stream drainages.

7.1.1.6 Coastal Plain Sand

The Cretaceous Middendorf Formation can have thicknesses of up to 75 feet (23 m) on the Haile property and generally thins to the west. The upper layer is clean, tan, quartz sand; the middle layer is white to red sand with abundant clay, while the lower contact is iron oxide-cemented coarse gravel and sand. The lower portion sometimes contains layers of red-brown ferricrete that vary in thickness from a few inches to 2 feet. The ferricrete consists of iron-oxide cemented quartz vein fragments and angular sand clasts. Ferricrete cementation is sometimes sub-parallel to bedding indicating that its formation was related to groundwater movement.

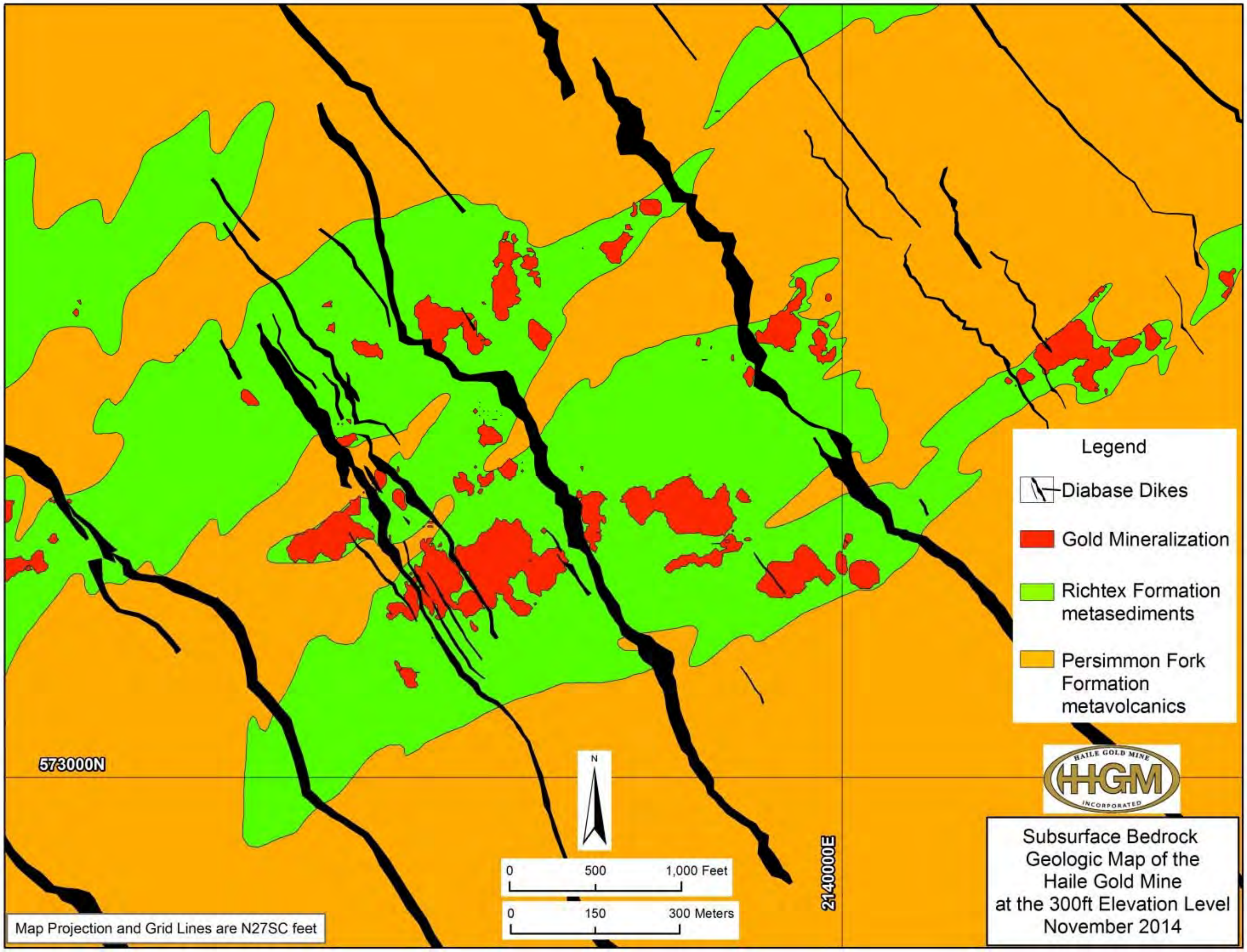


Figure 7-2: Schematic Geologic Map of Haile Property, November 2014

7.1.2 Structure

Deformation of the rocks at Haile, have created a structurally complex deposit. Penetrative strain is present within all of the Neoproterozoic to Early Cambrian aged units. This deformation manifests itself as spaced to well-developed foliation, tight to isoclinal folding, and local shearing. The foliation surface results from alignment of mica minerals and consequently, rocks that are more micaceous often have better developed foliation. The more massive portions of the Persimmon Fork Fm. are less foliated but micas within them are generally aligned. The foliation as mapped and taken from oriented core generally strikes northeast and dips moderately to the northwest. Bedding is more variably oriented than the foliation but commonly strikes east-northeast and dips to the north-northwest. Tight to isoclinal folds are present at the thin section, outcrop, and map scale. Most of the mapped fold axes have shallow to moderate plunges towards the northeast or east. Plunge reversals are also present as well as folds that plunge down-dip to the northwest or north. Many of the folds are asymmetric with moderately dipping northwest limbs and steep to overturned southeastern limbs. Shear textures have been observed in thin section and outcrop, and they may also be present at the map scale. Observed shear textures include pressure shadows, passive-slip planes, ribbon quartz along slip planes, mica fish, and anastomozing foliation surfaces. These features indicate ductile shearing but only minor offsets have been observed to date. Small scale, brittle-offsets are observed in the folded units and are parallel to the axial planar foliation. Indicators of brittle deformation such as slicken-sides are occasionally observed but do not show substantial offset of the major units.

Lithologic contacts encountered during drilling and mapping at Haile indicate that the deposit is situated within a large scale antiform that plunges shallowly to the northeast. This general pattern is complicated by lateral facies changes and interbedding of the lithologic units. Regional stratigraphy and recent zircon ages indicate that the section has been overturned at Haile. To date, major shear offsets and large scale shear structures have not been encountered.

7.2 MINERALIZATION

This section has been written by James Berry, Chief Geologist at the Haile Gold Mine. John Marek of IMC has reviewed this text and has sufficient comfort with the information to act as the Qualified Person under NI 43-101.

7.2.1 General Characteristics

The gold mineralization at the Haile property occurs along a trend of moderately- to steeply-dipping ore bodies within a regional corridor which runs from the west-southwest (WSW) to the east-northeast (ENE). The corridor is approximately 3,500 ft (1 km) wide (NNW to SSE) and over 2 miles (3.4 km) long (WSW to ENE). Most of the mineralization at Haile is restricted to the laminated metasiltstone of the Richtex Formation. The gold mineralized zones within the laminated metasediments can vary in distance from the metavolcanic contact, and can appear at different stratigraphic levels within the metasediments.

The gold mineralization is disseminated and occurs in silica-rich, pyrite-pyrrhotite bearing metasediments. Alteration in the mineralized zones consists of intense quartz-pyrite-sericite with occasional potassium feldspar, that grades outward to weak quartz-sericite-pyrite. The unaltered metasediments consist of pyrite bearing, sericite-quartz-chlorite-carbonate phyllites. Within the mineralized zones, quartz is dominant (greater than 80 percent), pyrite is subordinate (generally 3 to 10 percent), and sericite is variable. Moving away from the center of a mineralized zone, quartz and pyrite decrease while sericite increases in abundance. Multiple silicification events have occurred in the mineralized zones. The earliest silicification is massive and penetrative, whereas later silicification appears as re-healed broken angular rock fragments (breccias) followed by a scattered wormy stringer veinlet phase.

Gold mineralization is associated with pyrite, pyrrhotite, and molybdenite mineralization. Detailed ore microscopy and scanning electron microscope mapping indicate that the gold is found as native gold, electrum, and within gold bearing tellurides (Honea, 1992 and Thompson, 2009). These minerals are found as inclusions and along fractures within pyrite. The pyrite is usually present as either disseminated euhedral to subhedral grains or as euhedral to subhedral aggregates. Additional petrologic work has yet to be done within the mineralized zones that contain abundant pyrrhotite. Arsenopyrite, chalcopyrite, galena, and sphalerite are also associated with the mineralization. Molybdenite occurs primarily on foliation surfaces or as dispersed fine-grained aggregates in silicified zones. The Haile molybdenite has been dated by Re-Os isotopes at 553.8 ± 9 and 586.6 ± 3.6 million years (Ma) (Stein et al., 1997). The first Re-Os age closely approximates the zircon crystallization age of 553 ± 2 Ma reported by Ayuso et al. (2005) indicating that molybdenite mineralization was concurrent with Persimmon Fork deposition. Seven recent Re-Os molybdenite ages from Haile (Mobley et al., 2014) yield ages ranging from 529 to 564 Ma. Four of these samples give a weighted age of 548.7 ± 2 Ma, indicating that the gold mineralization is closely linked to Neoproterozoic volcanism.

7.2.2 Mineralized Zones

Mineralized zones at Haile can strike (trend) northeast to southwest and east to west. The mineralized zones dip at variable angles and directions at the site. The interpreted dips of the ore zones range from 25° at the western end of the property to steeply southeast at the eastern end of the known trend. In several areas, multiple mineralized zones exist. Their formation may be due to multiple favorable ore horizons having developed adjacent to feeder systems, or the repetition of mineralized zones due to isoclinal folding. The higher grade, core portions of the mineralization have trends that are sometimes different than the overall ENE trend. These trends range from E-W, NE-SW, NW-SE, and N-S. The mineralized zones are confined to Richtex sediments except for minor mineralization within transitional volcanic rocks. Portions of the mineralization are folded and fault offsets have not been observed at this time. Mineralized zones have been found to be intruded by diabase dikes, but are not altered or offset by them. Contiguous ore bodies have been found to occur on both sides of some diabase dikes.

8 DEPOSIT TYPES

This section was written by James Berry, former Chief Geologist at the Haile Gold Mine. John Marek of IMC has reviewed this text and has sufficient comfort with the information to act as the Qualified Person under NI 43-101.

Several gold deposits are located along a northeasterly trend that extends from eastern Georgia to Virginia. Many of these deposits are located at or near the contact between felsic volcanic and sedimentary dominated sequences. Various metal associations and mineralization styles indicate that this is a complex metalogenic province. Brewer has many features of an acid-sulfate mineralization system such as the presence of aluminosilicates, topaz, and enargite. Gold mineralization at Barite Hill contains the assemblage of pyrite-chalcopyrite-galena-sphalerite and is characteristic of a submarine, high-sulphidation volcanogenic massive sulfide deposit. Haile and Ridgeway are similar in that the mineralization is hosted within silicified siltstones. Both deposits contain molybdenite and the mineralization correlates with anomalous silver, arsenic, antimony, molybdenum, and tellurium.

The genesis of Haile and Ridgeway are quite controversial and both deposits have been proposed to have been formed by conflicting models. This controversy has been exacerbated by poor exposures, overprinting deformation, metamorphism, and intense weathering. Submarine hot springs have been suggested for the gold mineralization by several geologists (Worthington and Kiff, 1970; Spence et al., 1980; and Kiff and Spence, 1987). Foley et al. (2001) and Ayuso et al. (2005) have presented additional evidence in support of this model which include geochemistry of sulfide phases and geochronology. The exhalative model stipulates that gold deposition occurred when “black smokers” on the sea floor fumed out silica, gold, and sulfide bearing fluids and the minerals precipitated in a wide area over a uniform seafloor. The precipitated minerals were buried by later sedimentation. The resulting mineral deposits are typically classified as being stratiform and lenticular in shape, and the concentration of mineralization dissipates away from the source.

Alternatively, several workers have proposed the mineralization is structurally controlled and was caused by deformation. Tomkinson (1990) proposed that shearing was responsible for the mineralization at Haile and Ridgeway. This model invokes shears as the conduit for focusing gold bearing fluids into the metasiltstones. Drops in pressure during faulting are speculated to be responsible for gold precipitation. Nick Hayward (1992) proposed that folding of the phyllites controlled the gold mineralization. This genetic model proposes that gold was emplaced within the dilational zones of fold hinges during deformation.

Gillon et al. (1995) proposed a model which invoked both early mineralization and remobilization during deformation. O’Brien et al. (1998) proposed that the deposits were generated during the Neoproterozoic by the arc related volcanic activity in a hydrothermal system. This is supported by the close spatial associations between Haile and the felsic volcanic rocks. Pressure shadows around pyrite grains within the mineralized zones, folded mineralized zones, and flattened hydrothermal breccias indicate that the mineralization is pretectonic and rules out that the mineralization is related to deformation as proposed by Tomkinson and Hayward. Hydrothermal breccias containing well bedded clasts, silicification fronts cross-cutting bedding, and multiple phases of silicification indicate that the mineralization is post depositional and invalidate the submarine hot springs or exhalative model.

9 EXPLORATION

This section is based on the *Haile Gold Mine Technical Report* that was issued in October of 2015 by OceanaGold Corporation.

9.1 PRE-ROMARCO

Modern exploration, development, and mining activity on the Haile property began during the 1970s. Between 1973 and 1977, Cyprus Exploration Company (Cyprus) carried out an extensive exploration program consisting of surface geophysical surveys, trenching, geologic mapping, auger drilling, core drilling, air-track drilling, and metallurgical testing. Cyprus calculated the Haile resources at 186,000 ounces (5,785 kg) of gold with an average grade of 0.062 opt (2.13 g/t). Resources that are reported in this section do not conform to the standards of NI43-101 and are included only as part of the historic record, as a qualified person has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves. HGM is not treating the historical estimate as current mineral resources or mineral reserves.

During the late 1980s, Westmont/Nicor drilled out a small, low-grade oxide resource immediately west of the property.

Between 1981 and 1985, Piedmont explored the historic Haile Mine and surrounding properties with various drilling methods (including core and reverse circulation), surface geophysics, soil sampling, trenching, and rock-chip sampling. Piedmont's total drilling footage was 228,500 ft (69,647 m), much of which was for mine development. Piedmont mined several Haile property deposits from 1985 to 1992, producing about 86,000 ounces (2,675 kg) of gold.

In 1991, Amax performed an extensive exploration program on the Haile property under an exploration option with Piedmont. In 1992, Amax and Piedmont formed HMV as a joint venture, and from 1992 to 1994 HMC (the operating company) completed a program of exploration/development drilling (using core and reverse circulation), property evaluation, mineral resource estimation, and technical report preparation (Wells and Wolverson, 1993). The Ledbetter area was discovered and the Mill and Snake areas were expanded with this effort.

Kinross acquired Amax in 1998, assumed Amax's portion of the HMC joint venture, and later purchased Piedmont's interest. Kinross performed no exploration activities on the property and limited their operations to a highly successful reclamation program from 1998 to 2007.

9.1.1 Geologic Mapping

Numerous workers have performed geologic mapping on and around the Haile Mine area. The mapping adjacent to the mine area is complicated by very poor exposure of bedrock due to extensive-saprolitic weathering, coastal plain sand cover, and thick vegetation. Most of the better quality mapping has been focused within the excavations related to mining. H. Bell completed a preliminary geologic map for the Kershaw quadrangle in 1980. This map includes the Haile Mine site and the surrounding area and is mapped at a regional scale. Also, more detailed mapping has been done in the Haile Mine area. W.T. Spence, I.T. Kiff, and J. Maye constructed a detailed geologic map for the mine site in 1975. Subsequent detailed geologic mapping has been done by D. Taylor in 1985 and D.R. Cochrane in 1986. In addition, a dissertation completed by M.J. Tomkinson in 1985 included geologic mapping as did a Master's thesis completed by N. Hayward in 1988.

The HGM geologic team has scanned and loaded the mapping of N. Hayward, D. Taylor, D. R. Cochrane, and H. Bell into the Vulcan software for structural interpretation, exploration planning, and geologic modeling. The use of the structural dataset in conjunction with the drilling dataset has allowed the HGM geologic team to create a more substantive computerized geologic model. This model has been used successfully to expand the resource at the Haile property. Structural data interpretation, study of mineralization control, and deposit genesis is ongoing.

9.1.2 Geophysics

Because of the limited bedrock exposure in the Haile Mine area, numerous geophysical surveys have been conducted at the site in the quest for additional ore. These efforts are summarized in reports by A. Larson for Piedmont and led to the discovery of the Snake ore zone.

Geophysical surveys conducted by Piedmont include ground magnetics and dipole-dipole IP/resistivity. The ground magnetic data was acquired in a patchwork fashion and was not corrected for diurnal changes. The magnetic data is capable of mapping the Mesozoic diabase dikes but is not capable of mapping older units. The dipole-dipole IP/resistivity data has been reprocessed and is assisting with drill targeting and geologic modeling. Airborne EM has been gathered on the mine property by HGM in order to identify additional drill targets.

9.1.3 HGM Exploration Program

Romarco completed the Haile property acquisition on October 17, 2007. Romarco, by February 2008, confirmed the quality of historical drilling and assay data and turned their effort to exploration and resource expansion. During its ownership, Romarco has significantly expanded the resource and reserve of the property. This report documents the results of the drill program achieved to date with assay data available through November 17, 2011.

10 DRILLING

This section is based on the *Haile Gold Mine Technical Report* that was issued in October of 2015 by OceanaGold Corporation.

Drilling at the Haile property commenced in the 1970's and has continued intermittently to the present by several different companies. The data base that was used for this resource estimate was transferred to IMC on 17 November 2011. At that time there were a total of 3,747 drill holes in the data base totaling 1,511,912 feet of drilling. However, not all of this drilling was used for estimation of the block model.

Drilling has continued in a limited fashion since the November 2011 time period. That information has not been incorporated into the resource model or into the determination of mineral resources or mineral reserves. Property, permit, and other constraints are such that the additional drilling would not constitute a material change to the mineral resources or mineral reserves.

As of November 19, 2015 drill holes that fire assay above a grade of zero amounted to 2,297 drill holes, containing 302,088 assay intervals amounting to 1,404,716 ft of drilling information.

As of November 19, 2015, Romarco had drilled a total 1,647,637 ft on the property. In December 2015, a two phase infill drilling program for the Horseshoe deposit commenced. The program is expected to be completed by September 2016. As at July 17, 2016, 42,150 feet had been drilled. Prior to Romarco, 370,879 ft of fire assayed drilling was completed by previous property holders including Cyprus, Gold Fields Mining Corp, Piedmont, Westmont Mining, and a joint venture between Piedmont and Amax called Haile Mining Company. A portion of the early drilling has actually been mined out and has little impact on the estimation of the remaining in-ground mineralization. Some of the Piedmont and Cyprus drill holes were assayed by Cyanide soluble methods to determine Cyanide amenability of the mineralization. That information has not been used in the determination of resources and only those intervals with Fire assay from those previous property holders have been used.

IMC has completed a comparison of historic drilling to Haile-Romarco drilling and has found that the old and new data can be commingled if it has been fire assayed.

Within the fire assayed data, 28% of the holes are core and 72% are RC. There are very few fire assays (301) that are from air track drilling and "doodle bug" as recorded in the data base. They amount to 0.2% of the data base and are not a significant sample set.

Drilling completed by Haile since RC hole number 1502 and all DDH holes since hole number 289 have received down hole surveys. That amounts to 32% of the RC holes, 100% of the core-tail holes, and 89% of the diamond drill holes within the database have down-hole surveys. Since all of the surveyed drill holes deflect to the southeast, the Haile staff has developed an algorithm as a function of depth to adjust the down-hole survey of the historical drill holes to reflect their likely deviation toward the southeast from the collar orientation.

The foliation dip at Haile is to the northwest. Consequently, the drill hole deviation generally turns perpendicular to the foliation dip.

Figure 10-1 is a drill hole location map of the Haile project as of November 17, 2011.

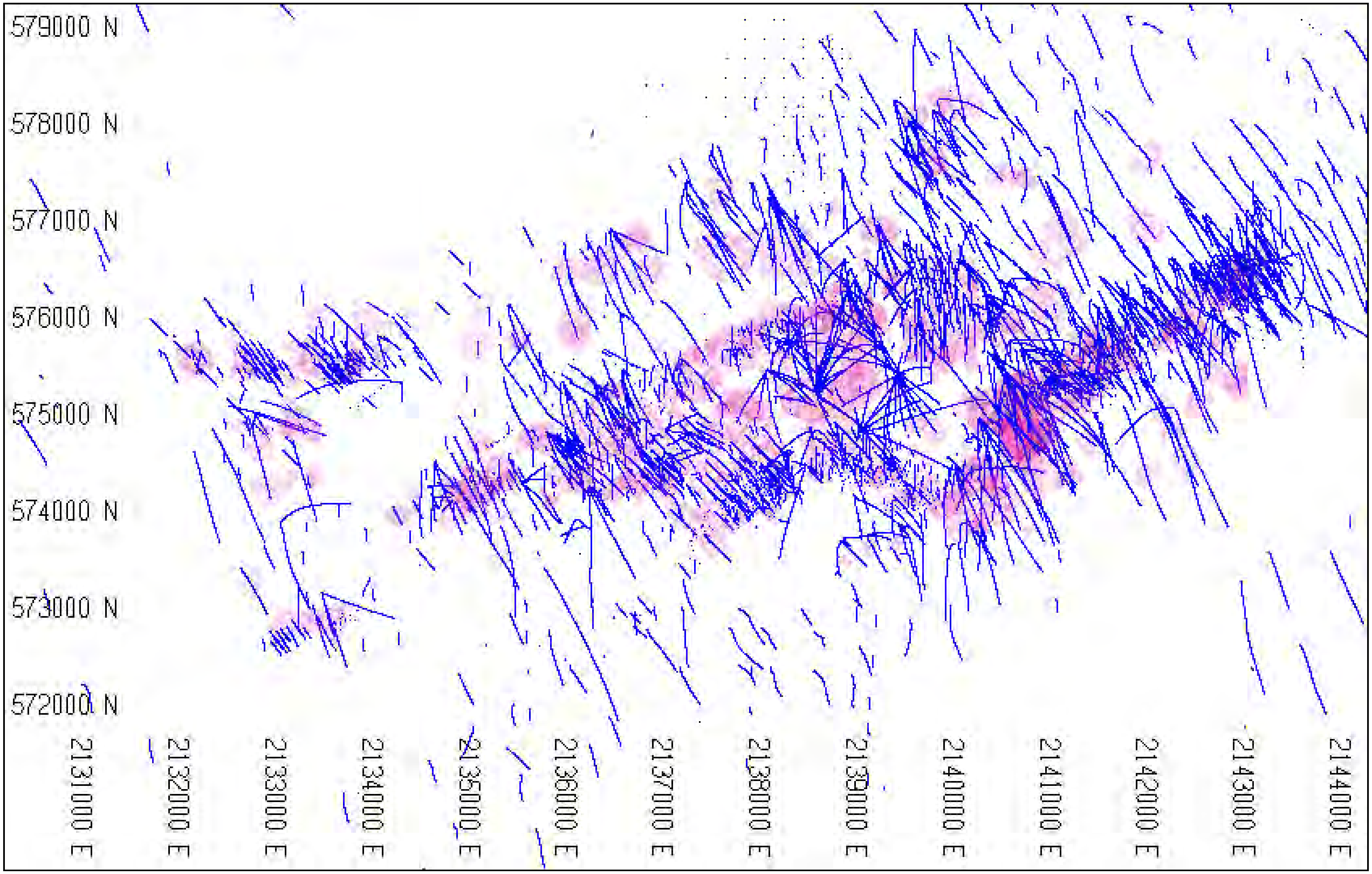


Figure 10-1: Drill Hole Location Map – Holes with Fire Assay as of November 2015, with Modelled Mineralization in Pink

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is based on the *Haile Gold Mine Technical Report* that was issued in October of 2015 by OceanaGold Corporation. In October of 2015, Romarco Minerals was acquired by OceanaGold Corporation.

11.1 SAMPLE COLLECTION

Romarco has been drilling both Reverse Circulation (RC) and Diamond Drilling (DDH) at Haile. This section will describe the sampling procedures applied to both data collection techniques. The sample procedures applied to the historic drilling at Haile are not well known. IMC has completed a statistical comparison between the historic information and the recent drilling to provide verification of the reliability of the historic drilling.

Romarco has been drilling at the Haile project since 2007. The techniques described in this section reflect the procedures applied by Romarco during the period up to November of 2011.

John Marek, the qualified person for this section, has reviewed the sample preparation, analysis, and security utilized by HGM and find the procedures to be proper for determination of mineral reserves and mineral resources. The results of quality control sampling that are reported in this Section are summarized in Section 12 that follows.

<u>Reverse Circulation (RC) Drilling</u>

The reverse circulation drilling at Haile typically uses 6.25-inch drill bits. The RC rigs are equipped with a cyclone and a rotary splitter. Most RC drilling at Haile is in wet conditions. Water injection is typically 4 to 5 gpm above the water table and decreases to 1 gpm when groundwater is encountered.

Sample sizes are between 20 and 30 lbs with a minimum requirement of 15 lbs. The standard size reflects a 15 to 20% split of the total drilled volume. Drill intervals are generally 5 ft intervals.

The following paragraphs describe sample procedures as reported by Romarco personnel. IMC observations during the site visit confirmed the application of these techniques.

For each 5-foot interval, a sample container is placed on top of the splitter table to catch the flow from the sample splitter. Labeled, sample bags measuring 20" by 24" are placed in five to seven-gallon plastic buckets. Multiple quarter-inch holes are predrilled in the plastic buckets to reduce the suction of a full sample bag and allow limited water drainage. The top of the sample bag is folded securely over the edge of the bucket. This is the sample container that is placed under the splitter to catch the sample discharge. Flocculant is added to each sample bag as it is placed on the splitter table to aid in precipitating fine material from the sample. As one sample container fills, another sample bag is prepared in advance and staged near the splitter table. On the driller's signal, the sample containers are switched instantaneously at the break between 5-foot drill intervals.

Sampling during advancement of each twenty-foot rod is a continuous process. Sample timing is metered by the count of the driller, as determined by drill speed and sample return rate. After each rod break, a new rod is attached and the borehole is thoroughly flushed. The driller should raise the bit slightly off bottom and blow the borehole clean before beginning the next interval. Once the sample return is clean, the bit is lowered and drilling begins on the next twenty-foot rod. Then, the driller counts the time it takes for the discharge water to turn from clear to muddy, which approximates the return rate of samples to the surface. Markings on the drilling rig feeder cable denote five foot intervals. When the feeder cable indicates the completion of the 5-foot sample interval, the driller counts the measured return rate to allow the last sample material to reach the surface.

The rod break depth is determined by the drilling rig set-up and may vary with every drill hole. The rod break generally occurs within a 5-foot sample interval. The sample collected over a rod break should be removed from beneath the sample splitter during borehole flushing. Following the addition of a new rod and subsequent flushing, the sample container is replaced and drilling continues. During the rod break, the sampler should clean the splitter, check the splitter plates, measure the pH and temperature of discharge water, and keep current with logging. For rod breaks occurring at shift changes, the crew is mindful of the incomplete sample and communicates its location to the next crew. Rod additions, timing, and bit changes are recorded in the drilling progress log. Filled sample bags are typically kept at the drilling rig during each shift. The samples can be stored on the ground or in the bed of a pickup truck to begin water drainage. At the end of each shift, the samples are transported to the sample storage area for initial drying.

During each drilling interval, a metal mesh-screened strainer (rice/pasta strainer) is placed on the splitter table beneath the waste stream to obtain a representative chip sample for geologic logging. The lithologic sample is collected from the waste discharge material to avoid biasing the assay sample partition. A portion of the lithologic sample is kept within a ten or twenty compartment, plastic chip tray for logging. Chip trays are labeled with the drill hole number and depth intervals in permanent marker.

Sample bags are collected at the end of each shift and transferred to the sample storage area for initial drying.

Diamond Drilling

Diamond core drilling is by wireline methods and generally utilizes HQ and NQ size core (2.5 inch and 1.9 inch core). Core is transferred from the core barrels to plastic core boxes at the drill rig by the driller. Core is broken as required to completely fill the boxes. Drill intervals are marked on the core boxes and interval marker blocks are labeled and placed in the core box. Whole core is transported to the sample preparation area by Romarco personnel.

11.2 ON SITE SAMPLE PREPARATION

RC Samples

The reverse circulation (RC) sample bags from the truck are transferred to the Haile sample handling facility where they are prepared for shipment to a lab. RC samples are prepared at either the Kershaw Mineral Lab (KML) in Kershaw, SC or the AHK Geochem (AHK) preparation facility in Spartanburg, SC.

Samples follow one of two paths:

1) Some samples are weighed and sample number tags added to the bags. The samples are poured through a Jones splitter to reduce the size to roughly 6 pounds for shipment to the sample lab. Coarse rejects are kept in their original sample bags and stored on site on pallets.

2) Alternatively, samples are staged at the Haile site and placed in containers for direct shipment to KML or AHK.

Core Samples

At the core logging facility, the core is cleaned, measured, and photographed. Geotechnical and geologic logging is completed on the whole core. Rock Quality Data (RQD) and core recovery are recorded as part of the geotechnical suite of data.

The logging geologist assigns the sample intervals and sample numbers prior to core sawing. Core is either sawed or split with a putty knife if soft. The saw or knife is cleaned between each sample. A brick or barren rock sample is sawed with the diamond saw between intervals to minimize cross-contamination. The cooling water for the saw is not recycled.

Split core is delivered to the sample preparation facilities. Core is prepared at the either the Kershaw Mineral Lab (KML) facility in Kershaw, South Carolina or at the AHK Geochem preparation facility in Spartanburg, South Carolina.

11.3 OFF SITE SAMPLE PREPARATION

The AHK and KML sample preparation and assay facilities that are discussed in this section are independent of HGM.

AHK Geochem (AHK)

Once the samples arrive at AHK in Spartanburg, the following procedures were applied:

Sample Preparation

1) Inventory and log samples into the laboratory LIMS tracking system
2) Print worksheets and envelope labels
3) Dry samples at 150 degrees F
4) Jaw crush samples to 80% passing 2 mm
5) Clean the crusher between samples with barren rock and compressed air
6) Split sample with a riffle splitter to prepare the sample for pulverizing
7) Pulverize a 250 g sample to 90% passing 150 mesh (0.106 mm)
8) Clean the pulverizer between samples with sand and compressed air
9) Ship about 125 g of sample pulp for assay
10) Coarse rejects are returned to Haile for storage
11) The 125 gm reserve pulps are stored at the AHK facility in Spartanburg with a seal. They represent an independent chain of custody sample library.

Sample pulps were shipped to the AHK Laboratory in Fairbanks, AK for analysis.

Kershaw Mineral Laboratory (KML)

Once the samples arrived at KML, the following procedures are applied:

Sample Preparation

1) Inventory and log samples into the laboratory LIMS tracking system
2) Print worksheets and envelope labels
3) Dry samples at 200 degrees F
4) Jaw crush samples to 70% passing 10 mesh (2 mm)
5) Clean the crusher between samples with barren rock and compressed air
6) Split sample with a riffle splitter to prepare the sample for pulverizing
7) Pulverize a 450 g sample (+/- 50 g) to 85% passing 140 mesh (0.106 mm)
8) Clean the pulverizer between samples with sand and compressed air
9) Approximately 225 g of pulp sample is sent for fire assay
10) Coarse rejects and reserve pulps are returned to Haile for storage.

Sample pulps from KML were shipped to the AHK Laboratory in Fairbanks, AK for analysis.

11.4 ANALYTICAL DETERMINATIONS

The procedures applied at AHK Geochem in Fairbanks, AK for assay were as follows:

1) Inventory the samples and create worksheets
2) Insert Quality Control samples of 2 duplicates, 1 Lab Standards, and 1 Blank in each batch of 40 samples.
3) Fire assay a 30gm aliquot for gold with 4 acid digestions and Atomic Absorption finish.

4) Analyze 0.50 gm samples for Multi-Element by ICP-MS as requested.
5) Review the internal QC results and check as required.
6) Review and sign off on final values including the internal check assays.
7) Issue the final report and certificate of assay.
8) Deliver the certificate to the client.

AHK Geochem is 17025 accredited for all facilities that handle Haile samples.

Early in the Romarco drill program, samples were sent to the Inspectorate Lab in Reno, Nevada for preparation and assay. Inspectorate is an ISO-9001 certified laboratory.

Check assays were sent to ALS-Chemex in Reno. ALS-Chemex is also ISO-9001 certified and 17025 accredited. Coarse rejects and returned samples are stored at Haile where they are under the control of Romarco personnel. During off-shift hours, a Deputy Sherriff is on site providing security for the site and sample storage facility.

The procedures currently applied at KML for assay are as follows:

1) Inventory the samples and create worksheets.
2) Insert Quality Control samples of 1 duplicates, 1 Lab Standards, and 1 Blank in each batch of 24 samples.
3) Fire assay 30gm of pulp sample for gold, with Atomic Absorption finish.
4) If the gold assay result from step 3 is greater than or equal to 0.09 opt, an additional 30gm of pulp sample is fire assayed for gold using gravimetric finish, and 0.50gm of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
5) Multi-Element ICP analysis is performed as requested.
6) Carbon and Sulfur determinations are performed as requested.
7) Review the internal QC results and perform check assays as required.
8) Review and sign off on final values including the internal check assays.
9) Issue the final report and certificate of assay.
10) Deliver the certificate to the client.

KML is ISO/IEC 17025:2005 accredited for gold and silver assays through the Standards Council of Canada.

Ore grade results produced by KML were not used in mineral resource calculations. Samples where KML reported above 0.015 oz/ton were sent to a third party lab for verification, and the third party results were used in assembly of the block model. Grades below 0.015 oz/ton from KML may be used in the model assembly process.

Early in the Company's drill program, samples were sent to the Inspectorate lab in Reno for prep and assay. Inspectorate is an ISO-9001 certified laboratory.

Check assays were sent to ALS-Chemex in Reno. ALS-Chemex is ISO-9001 certified and 17025 accredited.

Coarse rejects and returned samples are stored at Haile where they are under the control of Romarco personnel. During off-shift hours, a Deputy Sherriff is on site providing security for the site and sample storage facility.

12 DATA VERIFICATION

This section is based on the *Haile Gold Mine Technical Report* that was issued in October of 2015 by OceanaGold Corporation. In October of 2015, Romarco Minerals was acquired by OceanaGold Corporation.

The Haile drill hole data base was verified by IMC in late 2010 and the results published in the Technical Report titled "Haile Gold Mine Project, NI43-101 Technical Report Feasibility Study" dated 10 February 2011. The data base was later updated during November of 2011. That update and data verification was reported in 2012 and again in the Oceana 2015 document noted above.

This section focuses on verification of the drilling, sampling, and assaying completed from October 2010 thru 16 November 2011. The verification of the late 2011 data when added to the historic data base constitutes the complete data base used in the assembly of the block model and corresponding mineral resource estimate.

The data base verification at Haile utilized the following major steps:

1) A check of the Haile data base against assay certificates from the laboratory.
2) A statistical analysis of the quality control data that is collected by Romarco and their assay laboratory.
3) A comparison of Romarco drilling and assay information versus closely spaced historic information.
4) A comparison of diamond drilling versus reverse circulation drilling (DDH vs RC).
5) During the site visit, in 2009, the qualified person observed the sample procedures and quality control data handling as described in this text.

John Marek of Independent Mining Consultants, Inc. (IMC) acted as the qualified person for the data verification and determination of mineral resources. As a result of the data verification work that is summarized in this section, Mr. Marek and IMC find that the Haile data base is reliable for the determination of mineral resources and mineral reserves.

The approach presented above is to verify that the Romarco data is reliable based on the QAQC information that is collected with the data. Once that is established, the applicability of the historic information is established by a nearest neighbor statistical analysis of old versus Romarco drilling.

12.1 ROMARCO DATA VERIFICATION

The following checks have been applied to the Romarco data by IMC.

1) A comparison of certificates of assay from the laboratory versus the Romarco computerized data base to check the reliability of data entry.
2) Statistical analysis of the standards samples that are inserted by Romarco for analysis by the assay lab.
3) Statistical analysis of the blank samples that are inserted by Romarco for analysis by the assay lab.
4) Statistical analysis of the check samples that are submitted by Romarco to a third party laboratory

12.1.1 Certificate Check

Certificate checks have been completed by IMC in two iterations that correspond to block model updates in October 2010, and November 2011. IMC established a list of drill hole certificates and requested them to be scanned and sent to IMC for a spot check of the data base.

During the October 2010 check, IMC requested the original certificates of assay for 46 drill holes completed by Romarco. The selection of holes was established by IMC to cover the entire life of the Romarco drill program from 2007 through the most recent drilling in the third quarter of 2010. Of the 46-hole selection, 25 were drill holes completed between late 2009 and 2010.

Within the October 2010 data base the 46 holes contained 10,055 assay intervals. Within those intervals, IMC found 11 intervals where the Haile data base did not match the certificate of assay. All 11 discrepancies were in the low grade or trace range. In some cases, they were assigned as no assays in the data base and in others they were assigned as zero values.

IMC obtained certificates of assay for 42 holes that were drilled between the end of 2010 and the close out date for the November 2011 model update. There were 11,046 assay intervals within those holes. There was one interval in drill hole RC1914 where the assay data base did not match the certificate data.

The certificates were missing for 306 intervals out of the total or about 2.8% of the requested files. Most of the missing intervals were isolated single pages missing out of multiple pages of certificates, implying they were simply skipped in the copy process.

12.1.2 Statistical Analysis of Romarco Standards

Certified standards are inserted by Haile geologists with each laboratory submission of samples. The standards were purchased from Rock Labs and CDN Resource Labs Ltd, which reflect a range of gold grades that span the grade range at Haile. Since the lab does the sample preparation, and the standard is a pulp, the lab obviously knows that the samples are either blanks or standards. However, they are not informed of the value of the inserted standard or blank.

Drill hole data is initially stored as Excel files at Haile, with each hole reporting the results of the standards, blanks, and duplicates at the bottom of each file. IMC obtained these files and assembled a working spreadsheet of the QAQC data for statistical analysis. During 2016, Oceana have loaded data from all primary assay files, including all meta-data and associated QC data, into an Acquire™ drill hole database.

The 2011 data set that was used for the resource estimate by IMC contained 4,261 standards (not including blanks). This amounts to roughly 1 standard insertion for every 17 to 18 assay values collected by Romarco drilling during 2011.

Figure 12-1 is a summary plot of the certified sample value on the X axis versus the laboratory reported result on the Y axis. The graph indicates that there a few sample swaps where it is likely that the wrong standard was either recorded or inserted in the sample submission. There are several points on the X axis where blanks have likely been inserted by mistake rather than standards. This swap rate is acceptable although not ideal.

The graph does not indicate any substantial bias in the results from the project assay lab. The 2011 drill program utilized 36 individual standards with the highest grade standard (SN50) being 0.2533 oz/ton.

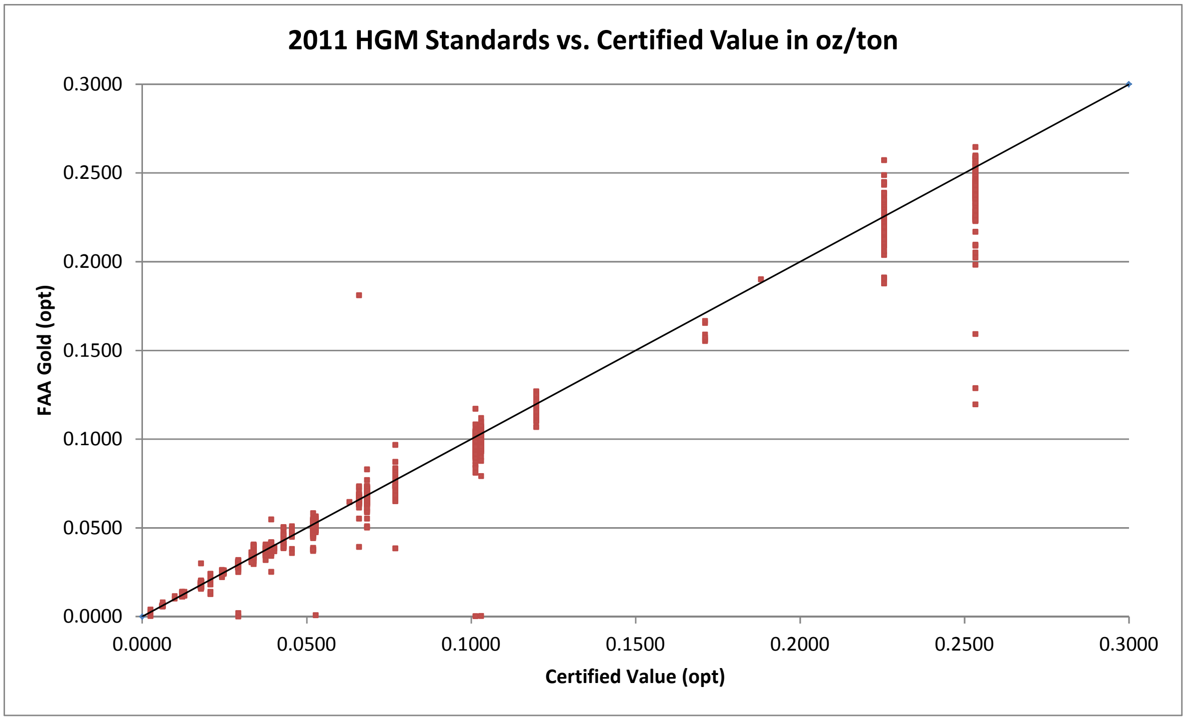


Figure 12-1: 2011 HGM Standards vs. Certified Value in oz/ton

12.1.3 Statistical Analysis of Romarco Blanks

Blanks are inserted by Haile geologists with each laboratory submission of samples in order to test for contamination. The blanks are purchased from a vendor of materials known to contain no gold. Three types of blank materials were utilized in the 2011 drilling campaign, Marble, Quartz Pebble, and sand.

Drill hole data is initially stored as Excel files at Haile, with each hole reporting the results of the standards, blanks, and duplicates at the bottom of each file. IMC obtained these files and assembled a working spreadsheet of the QAQC data for statistical analysis.

In summary, the IMC standards data set contained 3,587 blanks (not including standards). This amounts to roughly 1 blank insertion for every 20 assay values collected by Romarco drilling during 2011.

Figure 12-2 summarizes the results of the blank insertions by sample number. There were 11 occurrences out of 3,587 where blanks were reported as assays greater than 0.001 oz/ton.

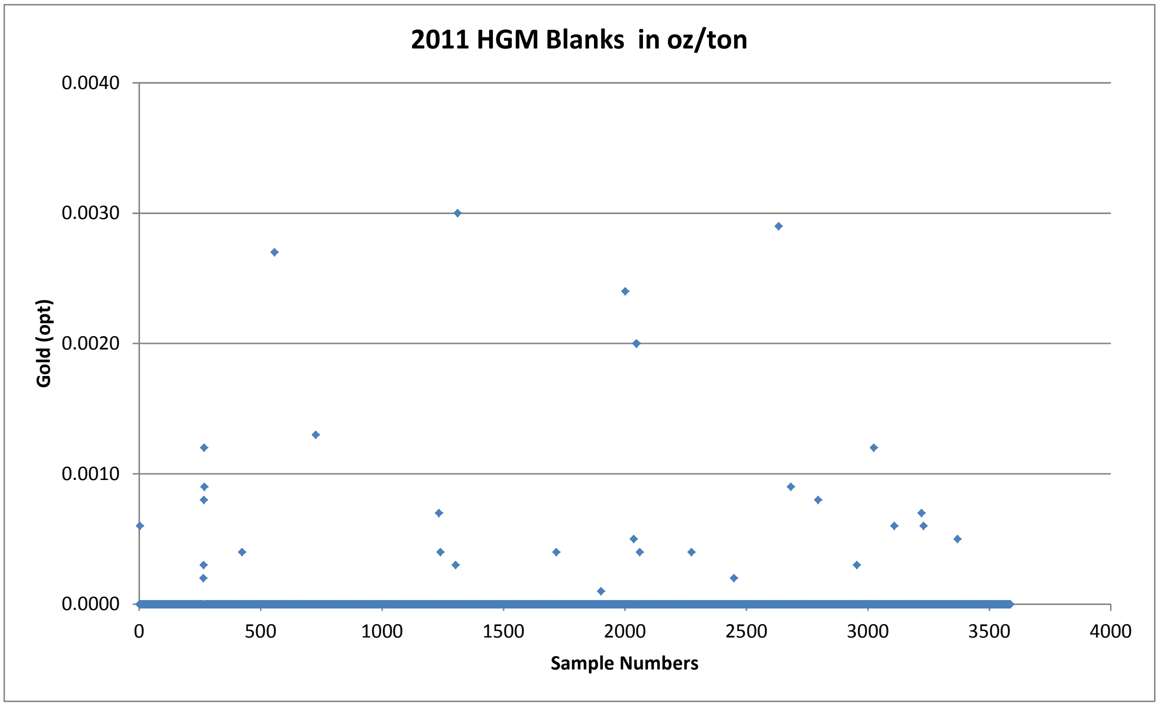


Figure 12-2: 2011 HGM Blanks in oz/ton

12.1.4 Statistical Analysis of Check Assays

Romarco has consistently been sending pulps and duplicates to an outside third party laboratory. During 2011 this outside check lab was ALS Chemex.

- Pulps are prepared pulps from AHK and KML that are sent to ALS Chemex as a check on the laboratory analytical procedures.
- Duplicates are ¼ core, or a second split from RC cuttings that are submitted to ALS Chemex for both sample preparation and assay.

IMC obtained 276 pulp check assays and 76 duplicate results from the 2011 drilling. Figure 12-3 summarizes the results with an XY plot of the AHK and KML assay versus the Chemex check assay on pulps. Figure 12-4 illustrates the XY plot of the duplicate samples.

The Chemex checks actually average slightly higher than the AHK and KML gold results as evidenced in the range between 0.030 and 0.050 oz/ton on the graph as observed previously during 2010.

The mean of the pulp and duplicate values for fire assay are shown below.

Table 12-1: Basic Statistics of Pulp and Duplicate Check Assays

Sample Type	Number of Pairs	AHK/KML Mean	ALS Mean	T Test Result
Pulp	276	0.110	0.114	Pass
Duplicate	75	0.162	0.197	Pass

There was one outlier value of 26 oz/ton within the duplicate checks that was removed from the check statistics by IMC. Values of that level were capped during the block model estimation process to be discussed in Section 14.

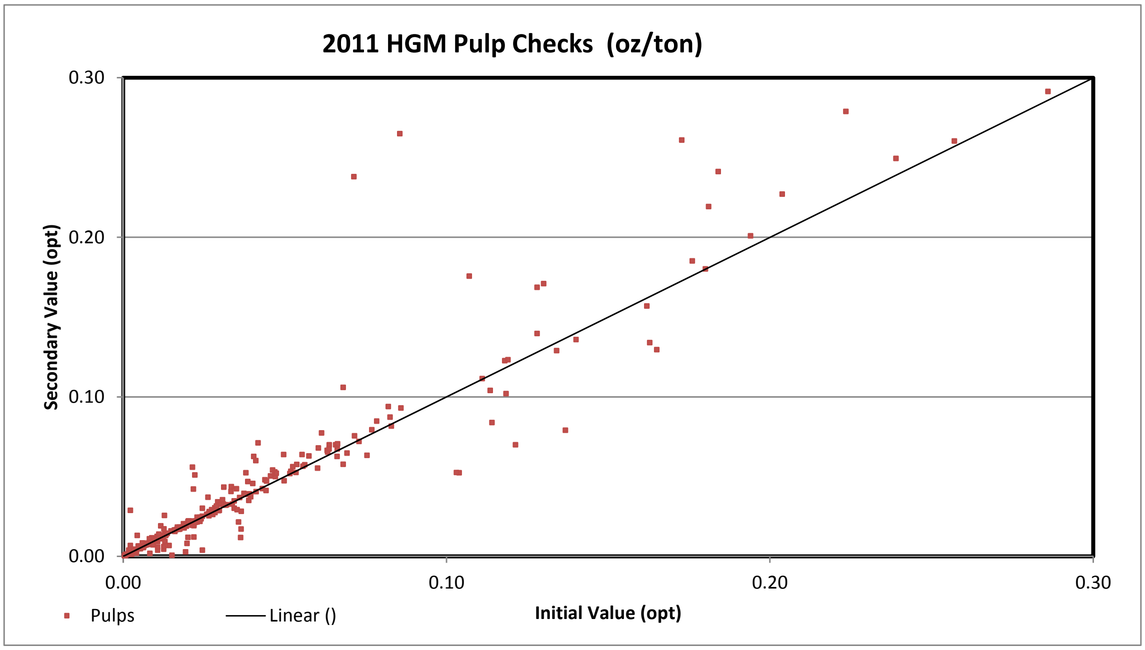


Figure 12-3: AHK/KML Gold Assays versus Chemex Pulp Assays

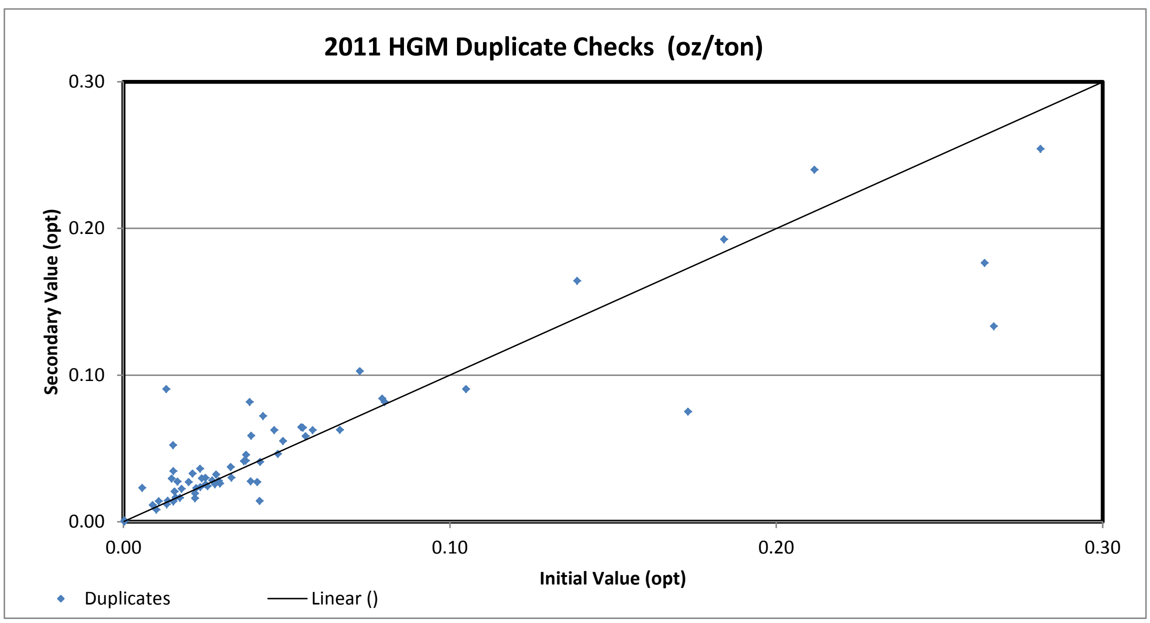


Figure 12-4: AHK/KML Gold Assays versus Chemex Duplicate Preparation and Assay

12.2 NEAREST NEIGHBOR COMPARISONS

12.2.1 Romarco Drilling versus Historical Drilling

In order to gain some comfort with the historical drilling at Haile, IMC completed a nearest neighbor comparison of old drilling versus new drilling on a 20 ft composite basis. The entire data base of Romarco drilling was used in this analysis rather than just the 2011 component.

The procedure was as follows:

1) Drill hole data was composited to 20 ft down hole intervals
2) Drill holes were tagged with the company that drilled them. In this case, Romarco drilling versus all previous drill holes.
3) The data was sorted so that old samples that were within a specified distance of the Romarco composites were selected and paired with the Romarco composite data.
4) Only metasediments and saprolite were used in the analysis as they represent the majority of the ore.
5) The result is a paired data set where statistical tests can be applied to check that the data represents the same population.

The table below summarizes the results of the statistical hypothesis tests for composites spaced 25 ft and 50 ft apart. The distances represent 1 model block and 2 model blocks respectively.

Table 12-2: Old Drilling versus New Drilling, Statistical Comparison

				Hypothesis Tests			
Sample Separation ft	Number of Pairs	New Mean	Old Mean	T Test	Paired-T	Binomial	KS
25 ft	297	0.019	0.027	Pass	Fail	Fail	Pass
50 ft	878	0.020	0.024	Pass	Pass	Pass	Pass

The hypothesis tests listed above all indicate that the data could represent the same population with 95% confidence. The purpose of each test is:

T-Test	Comparison of sampled mean values
Paired-T	Comparison of differences between pairs of samples
Binomial	Test that errors are unbiased
KS	Komologorov-Smirnoff test on the overall population

This test did not apply a sort on drill type so that both RC and DDH holes are in the comparison. The comparison of RC vs DDH will be addressed in the next sub-section.

12.2.2 Diamond Drilling vs RC Drilling

The data base at Haile consists of a mix of diamond drilling (DDH) and reverse circulation drilling (RC). IMC has compared the results of these two drill methods to confirm that they are not biased relative to one another.

A similar procedure was applied as outlined in the previous section. The 20 ft composites were coded by drill type, even if both methods were used in the same hole. For example, there are several holes where the top portion was RC drilled, cased, and then deepened with DDH methods.

A nearest neighbor analysis was completed with sample spacings of 25 and 50 ft. Table 12-3 summarizes the results.

Table 12-3: DDH Drilling versus RC Drilling, Statistical Comparison

				Hypothesis Tests			
Sample Separation ft	Number of Pairs	DDH Mean	RC Mean	T Test	Paired-T	Binomial	KS
25 ft	504	.026	.025	Pass	Pass	Fail	Pass
50 ft	1277	.026	.025	Pass	Pass	Fail	Pass

12.3 CYANIDE SOLUBLE GOLD ASSAYS

Early drilling by Cyprus and Piedmont applied cyanide soluble methods to the assay intervals. Much of this effort was directed at measuring the cyanide amenability of the ore to heap leach processing.

IMC completed a comparison between the cyanide data in the historic data base and fire gold assays where they both existed for the same assay interval. There are 9,417 intervals where both cyanide and fire assay data exist. Within those pairs, the cyanide data averages about 67% of the fire assay results. Statistical hypothesis tests do not support commingling of the data.

As a result, IMC has chosen to ignore the cyanide data within the historic data base and apply fire assay information only to the determination of mineral resources and mineral reserves.

IMC did complete a test to see if the use of cyanide soluble data could add additional information to the determination of inferred mineral resources. The results could have been potentially conservative, but there was the potential to add tonnage in areas where only cyanide data exists.

The result of the test was that there was no addition of contained inferred ounces with the incorporation of the cyanide data. The low bias in grade offset any gain that might have occurred in tonnage.

Consequently, the cyanide soluble data was not used in any of the analysis discussed within this document.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 GENERAL

Sample preparation and characterization, grinding studies, gravity concentration tests, whole ore leach tests, flotation tests and leaching of flotation tailings and flotation concentrate tests were completed to determine the metallurgical response of the ore. Samples of ore were collected by HGM for metallurgical testing. A series of metallurgical testing programs have been completed by independent commercial metallurgical laboratories. The test work indicated that the ore will respond to flotation and direct agitated cyanide leaching technology to extract gold. The results of these test programs are available in the following reports:

1. Resource Development Inc., (RDi), Wheat Ridge, Colorado, September 16, 2009, *Romarco Minerals, Inc. Haile Gold Project, Metallurgical Report*.
2. Metso Minerals Industries, Inc., (Metso), York, Pennsylvania, December 7, 2009, *Test Plant Report No. 20000134-135.*
3. Resource Development Inc., (RDi), Wheat Ridge, Colorado, March 31, 2010, *Romarco Minerals, Inc. Work Index Data For Haile Composite Sample.*
4. Resource Development Inc., (RDi), Wheat Ridge, Colorado, March 31, 2010, *Romarco Minerals, Inc. Metallurgical Testing Of Ledbetter Extension Samples.*
5. Resource Development Inc., (RDi), Wheat Ridge, Colorado, May 27, 2010, *Romarco Minerals, Inc. Flash Flotation, Cyanide Destruction & Leaching Of Concentrate and Tailing for Haile Composites.*
6. Resource Development Inc., (RDi), Wheat Ridge, Colorado, September 27, 2010, *Romarco Minerals, Inc. Optimization of Leaching Of Flotation Concentrate.*
7. KML Metallurgical Services, (KML), Kershaw, South Carolina, December 27, 2012, *HGM Years 1 – 3 Silver Characterization Project Test Report.*

The metallurgical test results were used to develop process design criteria and the flow sheet for processing the ore.

13.2 METALLURGICAL TESTING

Comminution test work was performed by RDi, Phillips Enterprises, LLC (Phillips), and Metso Minerals Industries, Inc. (Metso). Comminution parameters are shown in Table 13-1.

Table 13-1: Comminution Parameters

Parameter	Range of Values	Average Value
Abrasion Index	0.14 - 0.35	0.27
Crushing Work Index	7.46 - 13.89	11.17
Rod Mill Work Index	11.3 - 12.71	12.09
Ball Mill Work Index, 100-mesh	5.13 - 10.39	8.75
Ball Mill Work Index, 200-mesh	8.17 - 9.81	8.92
Regrind Mill Requirements, kwh/mt	37.8 - 43.0	40.4

Bond rod mill and ball mill work indices were determined for six selected composite samples. The bond mill work index for each composite was determined at 100 and 200 mesh for each of the composites. Metso performed ultra-fine grinding testing on bulk flotation concentrate to determine specific energy requirements. Two additional composite samples of the Red Hill ore zone were tested to determine the 100 and 200 mesh bond ball mill work index. The results for this work are presented in Table 13-2.

Table 13-2: Bond's Rod and Ball Mill Work Indices for Haile Composite Samples

Composite No.	Sample Description	RM Wi (KW-hr/t)	BM Wi @ 100 mesh (KW-hr/t)	BM Wi @ 200 mesh (KW-hr/t)
2.	Mill Zone-Average Grade	11.08	8.21	7.78
6.	Mill Zone-High Grade	11.30	8.21	8.17
8.	Haile-Average Grade	12.49	9.47	8.92
20.	Ledbetter-Average Grade	12.18	8.95	8.42
24.	Ledbetter-High Grade	12.56	9.47	9.03
34.	Red Hill-Average Grade	-	8.73	9.47
54.	Red Hill- Low Grade	-	8.83	9.50
83.	Ledbetter Extension Composite Samples (60-62)	12.71	10.21	9.81

Phillips performed comminution studies on samples from the Ledbetter Extension ore zone. The rod mill and ball mill indices were determined and an abrasion index for an ore composite and a waste composite was determined. The results of this work are presented in Table 13-3.

Table 13-3: Abrasion, Rod and Ball Mill Work Indices for Composite No. 83 and 84 Samples

Abrasion Index for Composite 83	0.0914
Abrasion Index for Composite 84	0.2055
Rod Mill Work Index (kW-hr/t)	12.71
Ball Mill Work Index at 100 mesh for Composite 83(kW-hr/t)	10.21
Ball Mill Work Index at 200 mesh for Composite 83 (kW-hr/t)	9.81

RDi performed gravity concentration testing to determine if coarse free gold could be recovered in a concentrate that could be direct smelted. Tests results indicated that a gravity concentrate would be too low grade to treat separately and since there does not appear to be coarse gold in the ore, a gravity circuit is not considered to be applicable as part of the ore treatment scheme.

RDi performed whole-ore cyanide leach tests on ore samples to examine the effect of ore grind size and leach time on gold recovery. The test work indicated that gold extraction from the samples was variable ranging from 40 to 79%. Most of the gold was leached from the ore in 6 hours of leach time and extraction generally increased with increasing fineness of grind. A summary of the test work is presented in Table 13-4.

Table 13-4: Whole-Ore Leach Test Results

Composite No.	Grind Size (P_{80}, mesh)	% Gold Extraction, Leach Time			NaCN Consumption at 48hrs, lbs/t
		6-Hour	24-Hour	48-Hour	
Mill Zone Average	100	56.97	65.02	64.73	0.50
Mill Zone Average	200	64.74	65.69	65.89	0.42
Mill Zone Average	325	68.04	69.25	68.40	0.84
Haile Average	200	67.54	71.28	71.52	0.52
Haile Average	325	69.03	73.75	75.33	0.96
Ledbetter Average	200	72.17	75.60	75.80	0.24
Ledbetter Average	325	70.43	80.27	79.13	1.40

RDi performed flotation test work to investigate the recovery of gold and silver to a sulfide mineral concentrate. The tests indicated that a reagent suite of potassium amyl xanthate (PAX), AERO 404 (or equivalent), and methyl isobutyl carbinol (MIBC), along with a laboratory flotation time of 6-minutes and a grind size of 200 mesh or finer will result in the highest gold recovery values. A summary of the flotation test work is presented in Table 13-5 and Table 13-6.

Table 13-5: Flotation Test Results

Composite No.	Grind Size (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (opt)	
		% wt	Au	Ag	Au	Ag
Mill Zone Average	100	18.2	92.7	50.9	0.516	0.341
Mill Zone Average	200	14.2	91.7	58.7	0.630	0.679
Mill Zone Average	325	12.6	90.8	61.6	0.779	0.846
Red Hill Average	200	16.8	82.6	75.2	0.493	1.420
Red Hill Average	325	15.6	82.3	73.1	0.557	1.053
Ledbetter Average	200	10.3	91.8	57.7	1.234	0.749
Ledbetter Average	325	10.5	88.6	42.8	1.301	0.674
Haile Average	200	12.8	86.7	59.9	0.519	0.752
Haile Average	325	11.3	86.4	65.6	0.618	0.834
Snake Average	200	15.4	90.2	50.4	0.665	0.475
Snake Average	325	15.0	91.6	49.0	0.636	0.446

Table 13-6: Flotation Test Results

Composite No.	Grind Size (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (opt)	
		% wt	Au	Ag	Au	Ag
Mill Zone Average-Grade	200	13.5	93.4	77.1	0.674	1.012
Mill Zone Average-Grade	325	12.9	90.7	70.8	0.697	0.992
Mill Zone High-Grade	200	13.3	92.1	83.5	1.374	1.274
Mill Zone High-Grade	325	12.7	94.8	60.4	1.461	1.015
Red Hill Average-Grade	200	16.6	76.6	83.1	0.338	1.409
Red Hill Average-Grade	325	15.2	82.1	77.8	0.347	0.662
Red Hill High-Grade	200	20.0	93.9	94.3	1.569	3.228
Red Hill High-Grade	325	18.2	93.2	80.5	1.496	2.633
Ledbetter Average-Grade	200	12.2	90.7	68.9	0.703	0.624
Ledbetter Average-Grade	325	14.1	89.5	44.2	0.563	0.271
Ledbetter High-Grade	200	8.0	95.7	57.5	3.071	1.534
Ledbetter High-Grade	325	7.9	87.5	53.3	2.033	1.175
Haile Average-Grade	200	12.2	84.9	65.1	0.365	0.726
Haile Average-Grade	325	11.2	86.5	64.0	0.402	0.682
Haile High-Grade	200	14.8	91.8	86.0	1.595	1.858
Haile High-Grade	325	12.5	87.6	67.3	1.423	1.371
Snake Average-Grade	200	16.4	96.1	53.5	0.472	0.432
Snake Average-Grade	325	17.1	89.1	38.4	0.382	0.350
Snake High-Grade	200	19.0	96.2	69.9	1.575	0.962
Snake High-Grade	325	17.1	95.3	65.6	1.560	0.688

RDi performed a flotation tailing cyanide leach tests to investigate the extraction of gold from the flotation tailing. The test results indicate that gold can be extracted from the flotation tails. A summary of the test work is presented in Table 13-7.

Table 13-7: Flotation Tailing Leach Test Results

Composite No.	Grind Size (P_{80}, mesh)	% Gold Extraction Leach Time – 24 hours	NaCN lbs/t Consumption	Lime $Ca(OH)_2$ - lbs/t Addition
Mill Zone Average-Grade	200	52.86	0.14	-
Mill Zone Average-Grade	325	62.97	0.50	-
Mill Zone High-Grade	200	71.70	0.16	-
Mill Zone High-Grade	325	71.87	0.44	-
Red Hill Average-Grade	200	68.51	0.74	13.19
Red Hill Average-Grade	325	67.53	1.22	12.83
Red Hill High-Grade	200	74.08	2.56	15.76
Red Hill High-Grade	325	81.17	1.40	15.30
Ledbetter Average-Grade	200	68.58	0.44	6.35
Ledbetter Average-Grade	325	70.73	0.24	5.65
Ledbetter High-Grade	200	71.98	0.20	-
Ledbetter High-Grade	325	76.50	0.16	-
Haile Average-Grade	200	62.75	0.16	13.68
Haile Average-Grade	325	62.22	0.26	13.70
Haile High-Grade	200	75.65	0.22	6.71
Haile High-Grade	325	77.10	0.18	6.31
Snake Average-Grade	200	62.38	0.02	8.53
Snake Average-Grade	325	66.34	0.16	8.45
Snake High-Grade	200	70.00	0.20	6.39
Snake High-Grade	325	70.90	0.24	6.29

Master composite samples were prepared and tested at Phillips and reported by RDi to confirm the RDi results on individual ore composite samples, establish process design criteria, and generate bulk flotation concentrate for regrind leach and other studies. Flotation results indicated 91% gold recovery into a concentrate representing 8.8% weight of the flotation feed in 13.5 minutes of flotation time. Flotation tail leach results indicated 50% gold extraction in 16 hours of leaching with a cyanide consumption of 0.20 lb/t. Leaching of composites samples of flotation concentrate indicated that at a grind of 80% passing 15 microns, with 24-hours of slurry pre-aeration, and a leach time of 24 hours, 85% gold extraction can be achieved. Additional tests on concentrate treatment indicated that a higher gold extraction rate for the concentrate would require consideration of an oxidation process.

In addition, concentrate samples were evaluated by Gekko Systems to determine the amenability of leaching in an intensive cyanide environment. The maximum gold extraction achieved was less than the 85% extraction target achieved by fine grinding and pre-aeration in the Phillips-RDi work.

Testing of master composite material provided samples of products for thickening and filtration testing. This work was performed by Pocock Industrial. Thickener and filter design parameters were determined for different types of equipment and different process samples. The results are summarized in Table 13-8.

Table 13-8: Thickening and Filtration Test Summary

Sample Material	Equipment Type	Equipment Design Parameter	
		Value	Units
Flotation Tailing (No CN)	Thickener (Conventional)	1.95 – 2.93	ft^2/stpd
Flotation Tailing (No CN)	Thickener (High Rate)	1.43 – 1.84	gal/min/hr
Flotation Tailing (CN Leach)	Vacuum Filter - Belt	151	lbs/hr/ft^2
Flotation Tailing (CN Detox'd)	Vacuum Filter - Belt	101	lbs/hr/ft^2
Flotation Tailing (No CN)	Pressure Filter	0.051	lbs/ft^3
Flotation Tailing (CN Detox'd)	Pressure Filter	0.055	lbs/ft^3

Cyanide destruction tests were run on process slurry samples from the master composite tests. The SO_2/air process (with sodium meta-bisulfite addition as the SO_2 source) was successful in destroying cyanide in the concentrate leach slurry samples. A test performed on a flotation tailing slurry sample gave an anomalous result and additional testing was recommended.

RDi performed slurry rheology tests on flotation concentrates after regrinding to 80% passing 15 microns. The work determined that the slurry could be suspended at 40 to 50% solids.

The Philips test work described in their September 17, 2008 report was performed on composite ore samples of average grade material from the Haile and Mill Zone pit areas. The testing was conducted to confirm results from sulfide flotation and cyanide leaching of flotation tailings, investigate oxidation methods for enhancing gold extraction from sulfide concentrate, determine thickening and filtration design parameters for flotation tailings, establish tailing neutralization requirements, and provide tailings material for both environmental and tailing disposal engineering studies.

The Phillip's flotation tests on the Haile composite indicate that 66% of the gold was separated into a flotation concentrate that represented 6.7% of the flotation feed. Tests on the Mill Zone composite indicated 89% of the gold was separated into a flotation concentrate that represented 13.6% of the flotation feed. Leach tests indicated leaching of the concentrate extracted 67% of the gold and after regrinding the concentrate to 80% passing 15 microns extraction was improved to 80% with most of the gold in solution within 10 hours of leaching.

Philips reported that leach tests on flotation tail indicated 82% gold extraction for both the Haile and Mill Zone composite flotation tails. For the Haile composite the total gold recovery (flotation concentrate plus gold extracted from the flotation tailing) was 94%. For the Mill Zone composite the total gold recovery (flotation concentrate plus gold extracted from flotation tailing) was 98%. It is important to note that these recovery numbers do not include gold losses associated with flotation concentrate leaching.

Thickening tests indicate leached tailings could be dewatered to 60% solids (w/w) in thickeners using a unit area factor of 1.2 ft^2/stpd. The tailing could be further dewatered to 18 to 20% moisture by vacuum filtration in filters using a filtration rate of 200 to 250 lbs/hr/ft^2.

Phillips processed tailing through filtering, re-pulping, and cyanide detoxification to generate the tail sample for environmental and tailing disposal studies. Detoxification was conducted using SO_2/air technology (sodium sulfite added as the source of SO_2).

Acid-Base accounting procedures were conducted to determine the net neutralizing potential of flotation tailing and flotation concentrate samples. The results indicated values of from -9.6 lbs $CaCO_3$/t for leached flotation tailing to -2,260 lbs $CaCO_3$/t for flotation concentrate leach tailing.

RDi was commissioned by Romarco Minerals, Inc. to perform flotation testwork on twenty-three (23) drill core composite samples from the Ledbetter Extension ore zone. Gold recovery ranged from 70% to 94% and averaged 86% for the 100-mesh grind samples, from 79% to 94% and averaged 87% for the 150-mesh grind samples, and from 81% to 95% and averaged 89% for the 200-mesh grind samples. Silver recovery ranged from, 52% to 79% and averaged 68% for the 100-mesh grind samples, from 46% to 80% and averaged 67% for the 150-mesh grind samples, and from 52% to 80% and averaged 69% for 200-mesh grind samples.

RDi performed flotation tailing cyanide leach tests to investigate the extraction of gold from tailings of the flotation tests. The tailing samples were leached for 24 hours at 40% solids and at pH 11 with 0.0167 lbs/gal sodium cyanide.

The gold extractions ranged from 44% to 85% and averaged 66% for 100-mesh grind samples, from 52% to 85% and averaged 68% for 150-mesh grind samples, and from 44% to 87% and averaged 69% for 200-mesh grind samples.

RDi was commissioned by Romarco Minerals to perform additional metallurgical testing on duplicate ore samples from the 2009 test program. Additional composite samples were made to evaluate carbon loading, cyanide destruction, flash flotation, conventional flotation time, and leaching of concentrate and tailing samples.

The carbon loading tests indicated that gold loads on carbon preferentially over copper.

The cyanide destruction test results indicate that the SO_2/air cyanide destruction process destroys WAD cyanide very effectively, as well as free cyanide, which was below the detection limit in the RDi laboratory.

A procedure was developed and used to evaluate “flash flotation” technology for the treatment of slurry in grinding mill circuit streams before the slurry particles have been completely ground to the final product size. The test results indicate that flash flotation is a viable option for gold recovery and that the flash flotation tailing can either be sent to conventional flotation followed by leaching of the conventional flotation tail, or the flash flotation tail can be directly sent to the leach circuit. Flash flotation was shown to recover 62 to 66% of the gold in 2 minutes of flotation time. Conventional flotation improves the total flotation gold recovery to about 80% and leaching of flotation tailing extracts 76 to 80% of the gold from the flotation tailing.

Fifteen samples were selected for the generation of flotation concentrate in one cubic foot flotation cell tests. The fifteen samples were identified as low grade, average grade, and high grade from the different ore zones (Red Hill, Snake, Ledbetter, and Mill Zone). The flotation tests were followed by leaching tests conducted on the flotation concentrates and flotation tailings. The results of these tests are presented in Table 13-9.

Five samples were selected for the generation of flotation concentrate in small scale laboratory flotation cell tests. The five samples were identified as average grade material from the different ore zones (Red Hill, Haile, Snake, Ledbetter, and Mill Zone). The flotation tests were followed by leaching tests conducted on the flotation concentrates and flotation tailings. The results of these tests are presented in Table 13-9.

Table 13-9: Test Results for Flotation and Flotation Tail Leaching

Test No.	Pit	Grade	Comp. No.	Flotation			Conc. Leaching			Tail Leaching			% Tot. Recovery Au
				Head Grade Au (opt)		% Au Recovery	Head Grade Au (opt)		% Au Recovery	Head Grade Au (opt)		% Au Recovery	
				Assay	Calc		Assay	Calc		Assay	Calc		
1/2	RH	L	49	0.027	0.033	91.5	0.172	0.140	62.7	0.003	0.005	83.8	64.5
7/8	H	L	47	0.010	0.011	64.7	0.093	0.190	82.6	0.004	0.006	85.9	83.8
17/18	S	L	51	0.015	0.015	84.0	0.230	0.245	79.8	0.003	0.003	66.0	77.6
19/20	L	L	43	0.021	0.020	86.7	0.248	0.207	71.9	0.003	0.005	61.3	70.5
25/26	MZ	L	H290	0.024	0.035	95.4	0.152	0.190	77.4	0.002	0.004	72.5	77.2
15/16	RH	A	34	0.080	0.095	92.0	0.589	0.513	83.3	0.009	0.010	67.2	82.0
11/12	H	A	8	0.085	0.064	85.5	0.455	0.467	74.8	0.010	0.012	60.3	72.7
9/10	S	A	39	0.056	0.052	89.6	0.735	0.583	64.2	0.006	0.006	77.7	65.8
¾	L	A	23	0.059	0.073	89.6	1.009	0.752	80.4	0.008	0.013	71.8	79.5
13/14	MZ	A	2	0.057	0.059	92.6	0.423	0.382	69.3	0.005	0.006	69.2	69.3
C34	RH	A	-	0.073	0.072	86.0	-	0.370	80.0	0.012	0.012	80.2	80.0
C28	H	A	-	0.086	0.085	68.1	-	0.580	59.7	0.030	0.029	79.6	66.0
C31	S	A	-	0.051	0.056	93.7	-	0.166	58.5	0.005	0.005	45.1	57.7
C61	L	A	-	0.048	0.047	86.1	-	0.341	80.7	0.007	0.008	81.4	80.4
C5	MZ	A	-	0.073	0.078	92.2	-	0.292	69.5	0.008	0.008	67.0	69.3
27	RH	H	35	-	0.429	94.1	2.601	2.094	73.6	0.030	0.038	77.5	73.8
28	H	H	9	0.180	0.194	90.5	1.394	1.321	88.5	0.021	0.024	64.5	86.2
5/6	S	H	53	0.304	0.312	95.2	2.365	1.875	75.2	0.017	0.020	68.3	74.9
23/24	L	H	71	0.240	0.274	94.7	2.622	2.222	74.0	0.015	0.034	81.5	74.4
21/22	MZ	H	12/3	0.168	0.199	96.0	1.563	1.155	79.7	0.009	0.020	73.3	79.4

The overall recovery for the individual sample zones is presented in Table 13-10.

Table 13-10: % Gold Recovery by Ore Zone and Ore Grade

	% Au Recovery – Combined				Average % Au Recovery
Ore Zone	Low Grade	Average Grade		High Grade	
Red Hill	64.5	82.0	80.0	73.8	75.1
Haile	83.8	72.7	66.0	86.2	77.2
Snake	77.6	65.6	57.5	74.9	68.9
Ledbetter	70.5	79.5	80.8	74.4	76.3
Mill Zone	77.2	69.3	69.3	79.4	73.0
Average	74.7	72.3		77.8	74.3

RDi was commissioned by Romarco Minerals Inc. to perform additional leach tests on flotation concentrates to attempt to improve gold extraction from that reported in RDi May 27, 2010. This new work was to determine if better results could be obtained by improving the aeration of the leach pulp or by increasing the sample size tested.

The results of the pulp aeration tests indicated that when the standard 4-hour leach slurry preaeration procedure was performed there was sufficient oxygen available in the leach pulp to complete the leaching reaction. Therefore, lack of oxygen was not necessarily a reason for poor leach results. Additional testing was recommended to determine if an 8 or 16 hour preaeration procedure would provide an improvement in gold extraction and reagent consumption.

The results of performing leach tests in concentrate samples twice the size as those used in previous leach tests indicated that there was a significant improvement in gold and silver extraction when using a larger sample size. Therefore, all new leach tests were run with larger samples.

Concentrate samples were ground to a size distribution of 80% passing 15 to 18 microns and slurried to 40% solids by weight slurry density. The slurry was then preareated for 4 hours, lead nitrate was added at 0.40 lbs/t for the final 3 hours of preareation. The preareated slurry was then leached at pH 11 for 48 hours with 0.167 lbs/gal carbon and 0.0167 lbs/gal sodium cyanide and with 0.61 in^3/minute air being added.

The leach test results indicated that for concentrate from the low grade ore samples, the gold extraction ranged from 77% to 88% and averaged 82%. For the concentrate from the average grade samples, the gold extraction ranged from 79% to 96% and averaged 83%. For the concentrate from the high grade ore samples, the gold extraction ranged from 83% to 95% and averaged 91%. Silver extraction averaged 80% for concentrate from low grade ore samples, 80% for concentrate from average grade ore samples, and 97% for concentrate from high grade ore samples.

A summary of the leach test results is presented in Table 13-11.

Table 13-11: CIL Test Results for Fine Ground Flotation Concentrate

Test No.	Pit	Composite No.	Grind Size (P80, microns)	48-hr Leach Time % Extraction Au	Ag	NaCN Consumption lbs/t
Concentrate From Low Grade Ore Samples						
37	Red Hill	49	16.64	80.9	71.1	2.00
36	Haile	47	14.07	77.2	49.5	4.99
38	Snake	51	15.79	81.0	94.4	10.83
35	Ledbetter	43	16.37	88.3	91.9	5.09
21	Mill Zone	Hole 290	-	79.8	91.0	5.59
26	Mill Zone	Hole 290	-	85.0	82.3	4.75
	Average			82.0	80.0	
Concentrate From Average Grade Ore Samples						
33	Red Hill	34	16.40	85.8	77.2	4.60
31	Haile	28	17.63	95.6	97.4	4.36
22	Haile	8	-	81.6	93.2	3.62
32	Snake	31	17.97	58.8	18.2	4.26
24	Snake	39	-	84.7	96.4	5.25
40	Ledbetter Ext	61	14.34	89.8	98.3	1.96
27	Mill Zone	2	16.20	81.5	96.2	4.77
28	Mill Zone	5	16.95	79.2	50.0	4.72
41	Ledbetter Ext	73	16.04	83.7	93.4	3.30
23	Ledbetter	23	-	88.3	79.9	4.72
	Average			82.9	80.0	
Concentrate From High Grade Ore Samples						
34	Red Hill	35	16.49	92.6	95.9	3.66
29	Haile	9	19.91	93.7	97.7	3.46
39	Snake	53	15.93	83.4	97.4	5.03
30	Mill Zone	12/3	19.26	88.7	95.9	4.00
25	Ledbetter Ext	71	-	94.9	95.6	12.3
	Average			90.7	96.5	

RDi investigated grinding the flotation concentrate finer than 80% passing 15 to 18 microns to improve gold extraction. The results of these tests indicate that grinding to 80% passing 10 to 13 microns increased extraction by 0.6 (high grade, Ledbetter) to 3.3 (low grade, Red Hill) percentage points and averaged 1.7 percentage points. In the finer grind tests, cyanide consumption increased from 40% to 250% (2 to 5 lbs/t) of the consumption measured in the 80% passing 15 to 18 micron leach tests.

RDi investigated leaching flotation concentrate at a lower leach pulp density to improve gold extraction. The tests indicated no beneficial effect in leaching at 30% solids by weight pulp density instead of 40% solids by weight pulp density.

KML was commissioned by Romarco Minerals Inc. to perform additional flotation and leach tests on 29 composites which represent the initial three years of operation in the mill zone and snake pits. Each composite was subjected to bulk flotation. The flotation concentrate was reground to a P_{80} of approximately 13 microns and leached for 48 hours. The flotation tailing was also leached for 48 hours. The overall gold recoveries ranged from 71.6% to 91.0% and overall silver recoveries ranged from 32.9% to 81.9%.

Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet can be used as the basis of process design. The relative low variability of flotation test work indicates that the mineralized zones are relatively similar in terms of ore grindability, chemical and mineral compositions, and flotation and cyanide leaching response.

The composite samples tested responded favorably at a moderately fine feed size range of 80% passing 200 mesh (74 microns). Therefore, a primary grind size of 80% passing 200 mesh was recommended for process circuit design development.

The flotation testing indicated that gold can be recovered in a flotation concentrate that will also contain the majority of the silver in the ore. The tailing from the flotation circuit can then be processed by cyanide leaching to recover gold onto activated carbon.

The test work indicated that the circuit should include regrinding of the flotation concentrate before leaching. A regrind circuit product size of 80% passing 15 microns is an appropriate target for regrind circuit design.

Leaching of the flotation concentrate can extract 82% to 91% of the gold and 80% to 96% of the silver. Leaching of the flotation tailing can extract 45% to 86% of the gold in the flotation tailings. It appears that overall gold recovery will be in the range of 65% to 92% dependent primarily on head grade to the mill and less dependent on which ore zone the ore is mined.

The unit operations that determine gold extraction are flotation, flotation concentrate leaching, and flotation tailing leaching. The metallurgical testing performed by RDi in 2009 developed flotation and flotation tailing leach data. The work performed by RDi in 2010 provided data on flotation concentrate leaching. The data developed in the two test programs has been used to establish a relationship between overall gold recovery and mill head grade as shown in the graph in Figure 13-1. The graph and the equation for the “best-fit” line that describes the head grade recovery relationship can be used to estimate gold recovery from a predicted mill head grade. For example, at a mill head grade of 0.060 opt the recovery equation graph predicts a gold recovery of 83.7%.

The results of grade and recovery data analysis is shown in Figure 13-1.

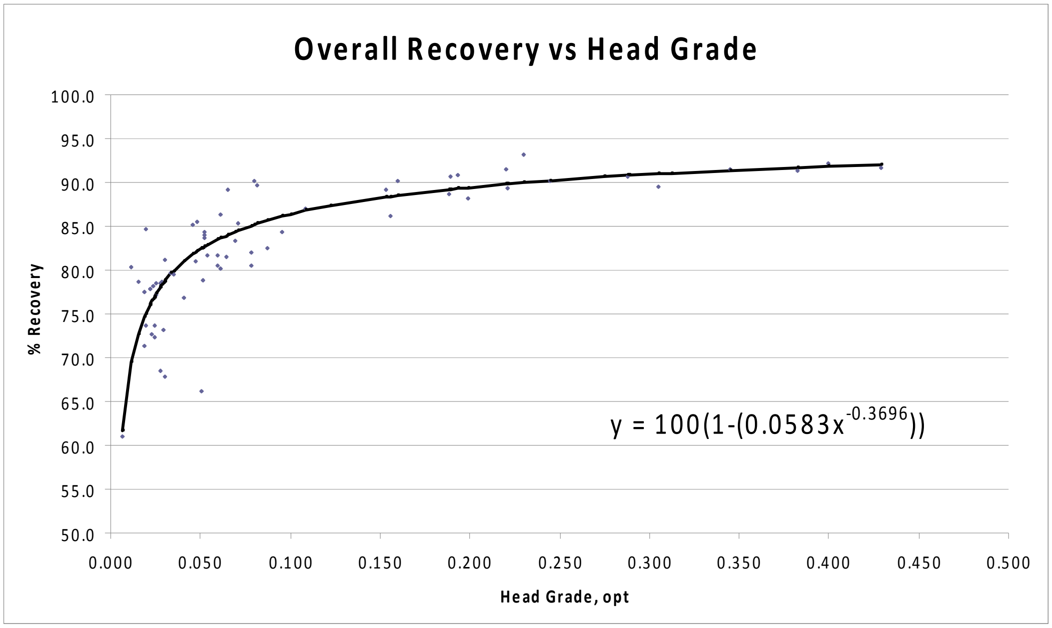


Figure 13-1: Overall Percent Recovery vs. Head Grade

Reagent consumption rates and grinding media consumption rates for full scale plant operation have been estimated from the results of the RDi test work. The estimated reagent consumption rates are presented in Table 13-12. Grinding media consumption rates are presented in Table 13-13.

Table 13-12: Process Reagents

Item	Rate lbs/ton ore
Collector, Potassium Amyl Xanthate	0.05
AERO 404 (or equivalent)	0.05
Frother, Methyl Isobutyl Carbinol	0.03
pH Modifier, Lime	2.07
Sodium Cyanide	1.07
Flocculant	0.13
Antiscalant	0.03
Sulfuric Acid	0.01
UNR 811A (or equivalent)	0.01
Hydrochloric Acid	0.21
Lead Nitrate	0.02
Copper Sulfate	0.02
Ammonium Bisulfite	0.40
Carbon	0.03

Table 13-13: Grinding Media

Item	Rate lbs/ton whole ore
Grinding Balls, SAG Mill	0.99
Grinding Balls, Ball Mill	0.63
Grinding Media, Regrind Mill	0.37

14 MINERAL RESOURCE ESTIMATES

The PEA in section 24 presents a scoping level study which evaluates mining selected areas of the mineralization below the reserve pit floor by underground mining methods. Further work is required to assess the comparative merits of mining this mineralization via underground versus open pit mining methods. Until such time, the scoping study is considered as a technically viable alternative. For this reason, the resource below the reserve pit floor remains classified as open pit resource. The subset of this resource selected for this underground option study in the PEA, has been reported only in section 24 and is not excised from the open pit resource inventory presented in Table 14-5.

The resources reported as underground resources in the previous (October 2015) NI 43-101 have been removed from the resource table, given the PEA underground study presented in section 24.

This open pit resource section was originally published in the technical report titled "Haile Gold Mine Project, Resource Estimate NI 43-101, Lancaster County, South Carolina," dated March 13, 2012. The open pit resource section is presented in its entirety without change since that date.

Section 24 presents a discussion of sub-models that were used for the estimation of potentially minable material within the underground PEA. The resulting underground material target is 95% contained within the volume defined in this section as an open pit mineral resource. The sub-model procedures were modified from those discussed in this section to provide a better estimate for underground mining methods.

John Marek, P.E. of IMC acted as the Qualified Person for the development of the model and the open pit mineral resource estimate.

14.1 BLOCK MODEL

The block model was developed using blocks sized 25 x 25 ft on plan with a 20 ft bench height. The small block size in plan was selected in order to provide a reasonable method of modeling the interpreted geology with particular emphasis on the late barren dykes that cross the Haile deposit.

The bench height of 20 ft was selected based on a combination of planned production equipment sizes and on the results of a bench height dilution study completed to confirm the 20 ft selection. A bench height of 20 ft is common in many open pit gold mines in the U.S.

The block model is assembled in the project coordinate system that aligns with true north. There is no rotation in the model. Table 14-1 summarizes the block model location and size. The model extends some distance to the east beyond current drill intercepts. This is to provide sufficient topographic coverage for open pit back walls that may result from the deep Horseshoe area on the east side of the district.

Table 14-1: January 2012 Haile Model Area – Block Corners

	Southwest	Northwest	Northeast	Southeast
Easting	2131550.00	2131550.00	2146000.00	2146000.00
Northing	57200.00	579000.00	579000.00	572000.00
Elevation Range		-2,500.00	600.00	
No Model Rotation, Primary Axis= 0 degrees North-South				
Model578 Blocks in North - South Size 280 Block in East - West 25 x 25 x 20 foot block size 155 Levels				

14.1.1 Data Base

The data base for the block model assembly was provided by the Romarco Staff and verified by IMC prior to application of model assembly. There are 3,747 drill holes in the Haile data base as of 17 November 2011. However, not all of the holes were used for assembly of the block model. Although geologic information was available in many of the drill holes that were used for geologic interpretation, only those drill holes with fire assay information were used for block grade estimation.

Drill holes with fire assay were used for block grade estimation. Altogether there are 2,102 drill holes with fire assay, including zero valued assays. The count of drill holes with fire assay greater than zero is as follows:

Number of Drill Holes	2,039
Feet of Drilling	1,372,473 ft
Number of assay intervals in those holes	254,681

All drill holes with fire assay data were used including historic drilling, current Romarco drilling, both diamond drilling and reverse circulation drilling. There are 21 drill holes in the data base that are labeled as air track holes or "doodle bug" holes. These are short holes that have minor impact on the estimate of remaining mineralization as most were mined out by the historic mining.

14.1.2 Rock Types and Estimation Boundaries

Geologic surfaces were interpreted by the Haile geology staff. Those electronic files were transferred to and checked by IMC. The rock types were assigned to the block model on a whole block basis and checked again for completeness.

The rock type codes that are assigned to the mode are:

Code
100 = Meta-Sediments
200 = Meta-Volcanics
400 = Diabase Dikes
500 = Saprolite
600 = CPS beach sand

IMC added the following codes to model blocks:

700 = Fill where current topo is above pre-reclamation topo
800 = Old leach pads based on the topographic maps
1100 = Old tailings based on recent augur drilling

The last three codes reflect the material that has been placed back into historic pit excavations and are used primarily for mine planning cost estimation. No gold grades have been assigned to codes 700, 800, or 1100.

Haile personnel provided surfaces for redox and pre-reclamation topography. They were also assigned to the model.

Statistical checks on the rock type boundaries were completed to determine if they should be respected as hard boundaries in the grade estimation process. This procedure utilized 20 ft down hole composites (discussed later) and applied statistical hypothesis tests on samples from opposite sides of boundaries to determine if they were of the same population. As a result, the following boundaries were established for grade estimation:

Boundary	Boundary Type
Meta-Sediments vs Saprolite	Soft -Transitional over about 50 ft vertically
Meta-Seds vs Meta-Volcanics	Hard boundary

Sand to Saprolite	Hard boundary
Diabase	Barren and not estimated
All other rock boundaries	Hard Boundaries

IMC also assigned a code to the model to represent the historic mining areas at Haile. These reflect the historic names of: Mill Zone, Haile, Red Hill, Ledbetter, Snake, Chase Hill, Champion, and the Horseshoe and Mustang zones that were added during 2011. These were used primarily for reporting purposes as they may or may not reflect changes in the mineralization of the deposit.

Zones for control of the variogram parameters were developed based on the orientation of the Meta-Sediment Vs Meta-Volcanic surface. The assumption is that the contact broadly reflects the general orientation of the folded foliation in the Meta-Sediments. Since the mineralization pre-dated the deformation, the orientation of the foliation appears to be indicative of the primary access of the mineralization. Consequently, the new zones represent a structural overlay that is superimposed on the rock type information.

Figure 14-1 is a map of the model area showing the numbered variogram – structural zones. The first two digits of the number reflect the dip of the foliation to the northwest. For example, 303 dips 30 degrees to the northwest and represents the Ledbetter area of the deposit. The 60 zone represents Snake, and the 45 zone represents the South Pit.

Figure 14-2 illustrates the named mining areas in the block model. These areas are not boundaries or controls on mineralization, but they are used as traditional reporting zones for mineral resources. The Palomino and Mustang zones were added to the November 2011 block model.

The assay data was "dipped" into the block model to assign rock type codes and variogram zone codes to each assay based on the rock type of the block that contained the assay. The variogram – structural zones were used to change search orientation for grade estimation but they were not treated as hard boundaries within a rock type.

Population statistics were developed for each of the zones on the map. Cumulative frequency plots were developed to understand the populations. The cumulative frequency plots were also used to set levels to cut individual assays. The cap values for assays are summarized below:

Assay Cap Values

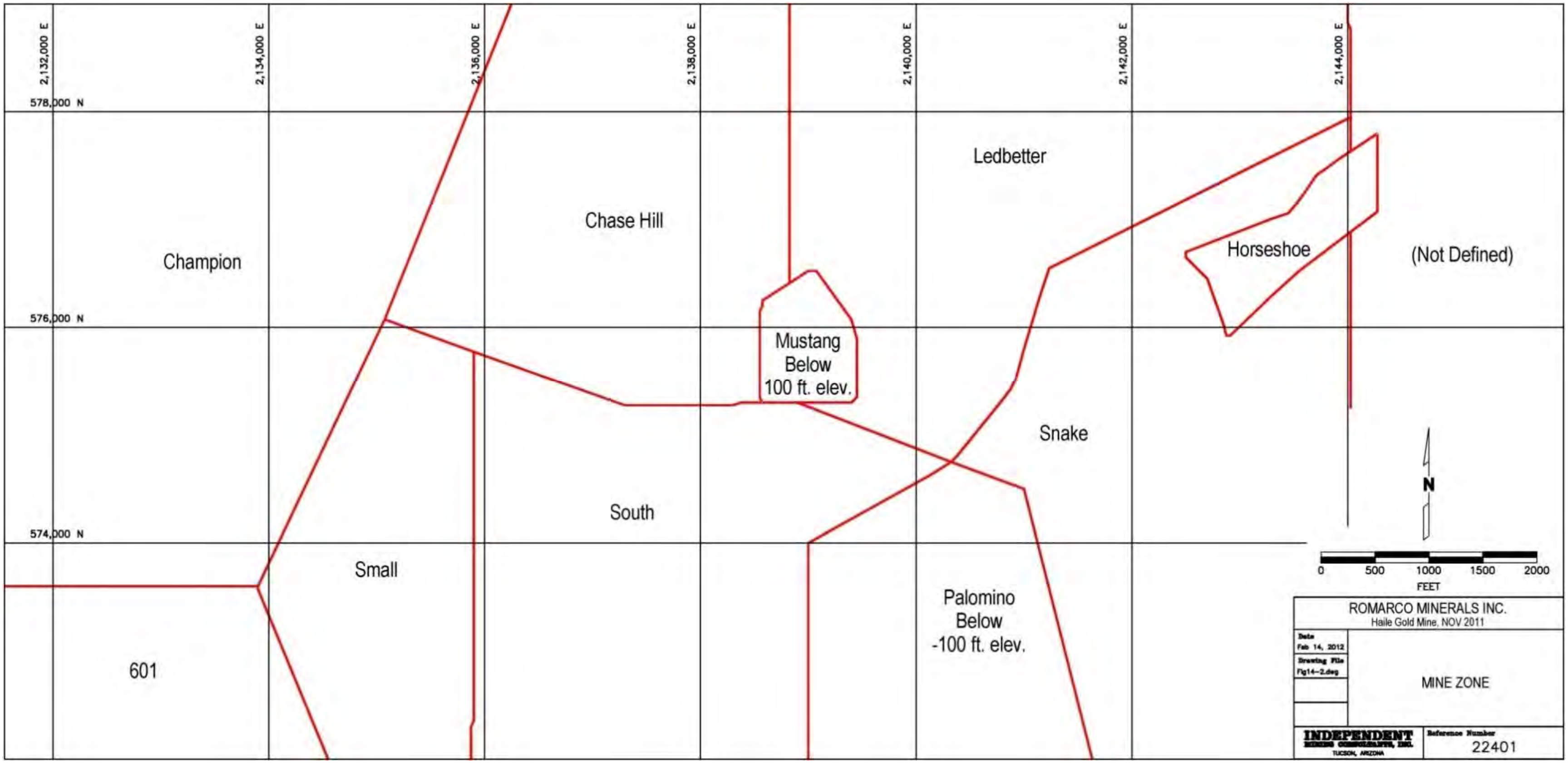
Meta-Sediments: Rock = 100

VarioZone	Description	Cap
45	Haile- South	0.70 oz/ton
303	Ledbetter	1.00 oz/ton
60	Snake	1.00 oz/ton
602	Horseshoe	3.00 oz/ton
Everywhere Else		0.40

Meta-Volcanics, Sand, Saprolite
All Zones 0.32 oz/ton.

(Source: IMC, 2010)

Figure 14-1: Variogram – Structural Zones



(Source: IMC, 2012)

Figure 14-2: Named Mining Zones used in Resource Reporting

14.1.3 Composites

The assay information was composited to 20 ft down hole composite intervals. The lith values that were assigned to the assay intervals by back assignment from the model were composited to the nearest whole rock type during the composite process. A minimum composite length of 10 ft was required to calculate a composite value.

The variogram-structural zone codes were assigned to the composites by “dipping” or back assignment from the model block zone codes.

Figure 14-3 summarizes the composite statistics by rock type and variogram zone across the Haile Deposit. The Haile drilling contains many zero valued or trace valued assay intervals that are correctly coded into the drill hole data base. However, for illustration, the zero valued composites have been removed from the calculations on Figure 14-3. They were however, kept in place for block grade estimation.

Cross Tabulation of 20 ft Composites
Number of Composites and Mean Fire Gold Grade of Composites (oz/ton) in Each Rock Type and Variogram Zone

Rock Type ↓	Variogram Zone 45	60	201	202	301	303	304	452	602	Row Total
Meta-Sediments	15,435 0.012	3,699 0.023	1,033 0.004	387 0.005	206 0.005	3,917 0.015	1,011 0.010	281 0.009	1,303 0.034	27,272 0.014
Meta-Volcanics	1,841 0.002	914 0.002	400 0.001	752 0.000	113 0.001	1,640 0.001	659 0.001	101 0.001	280 0.001	6,700 0.001
Diabase	749 0.002	101 0.003	22 0.004	33 0.000		175 0.001	17 0.001	13 0.000	27 0.006	1,137 0.002
Saprolie	2,216 0.008	437 0.012	139 0.001	196 0.006	25 0.000	357 0.000	125 0.000	21 0.000	37 0.000	3,553 0.007
CPS	171 0.004	26 0.003	16 0.000	10 0.001	2 0.000	49 0.000	4 0.000	4 0.000	23 0.000	305 0.003
Column Totals	20,412 0.010	5,177 0.018	1,610 0.003	1,378 0.002	346 0.003	6,138 0.010	1,816 0.006	420 0.006	1,670 0.027	38,967 0.011

Figure 14-3: Basic Statistics of 20 foot Composites, By Rock Type and Variogram Zone, Zero Values Have Been Removed

14.1.4 Density Assignment

Density was assigned to each block in the model based on the rock type code. Density information was based on the average results by rock type as recorded by Romarco from their analysis of core results. The saprolite density was based on field engineering test results completed by the geotechnical contractors. This information was the result of tailing impoundment design requiring in-situ density data and compaction results.

Density information recorded by Romarco between November 2009 and November 2011 was not specifically used in the calculation of mean density by rock type. The 2011 results were checked by IMC and the difference so minor that the density information established for earlier model trials by IMC were maintained for consistency.

The following dry densities were assigned to each rock type in the block model.

The dry density assignments are:

Rock Type	Sp.G	Lbs / Cubic Ft
Meta-Seds	2.77	172.93
Meta-Volcanic	2.60	162.32
Diabiase Dikes	2.91	181.66
Saprolite	2.14	133.60
Sand	1.89	117.98
Fill	2.14 assumed	133.50
Old Heaps	1.89 assumed same as Sand	

14.1.5 Block Grade Estimation

Block grades were estimated using the statistical procedure of kriging, limited by rock type and by grade range. The grade range limits were established with an indicator at the low grade range and with a search limit on high grades. Rock types were respected as stated earlier, and variogram orientations were changed based on rock type and the variogram-structural zone.

Cumulative frequency plots were developed on the 20 ft composites and population breaks or changes were recorded for each area. There is a distinct low grade break in all of the frequency graphs between 0.004 and 0.010 oz/ton. IMC opted to apply a 0.010 oz/ton discriminator to minimize over estimation of tonnage in that grade range.

The frequency plots also indicated changes in population at high grade values of 0.100 to 0.150 oz/ton. Within the Horseshoe zone (Variogram zone 602) the population break was interpreted at 0.50 oz/ton. These values later received limitations on search radius when assigning grades to the blocks.

Variography was completed in each of the zones to evaluate the potential search distance and orientation. Both indicator variograms at 0.010 oz/ton and gold grade variograms were run and interpreted.

As a result of this work, a single stage indicator kriging approach was used in each of the population zones. The approach is a follows:

1) Indicator kriging was completed in Meta-Sediments, Meta-Volcanics, and Saprolite applying a 0.010 oz/ton discriminator.
2) The resulting fractions between 0 and 1 were sorted on the 0.50 level so that the deposit was divided into two zones: a) those blocks with better than 50% chance of being above 0.010 oz/ton and, b) those blocks with less than a 50% chance of being lower grade than 0.010.
3) Composites were assigned the same indicator code as the block that contained them.

4) Composites were also assigned the same variogram zone as the block that contained them.
5) Grade assignment was then completed using ordinary linear kriging within each rock type respecting the indicator as a hard boundary. The Saprolite-Meta-Sed contact was made a soft boundary, all other rock types were hard boundaries.
6) Variogram zones were not hard boundaries.

Table 14-2 and Table 14-3 summarize the kriging parameters for both the indicator run and the grade runs in both indicator domains.

A limit on the search radius applied to the high grade values was utilized as summarized on Table 14-3. The high grade search was typically limited to 100 or 125 ft or about 2/3 of the total search radius applied to all other mineralization. The purpose is to limit the smearing of high grade over low grade that occurs with most grade estimation techniques.

The variogram – structural zones were used to change search orientation for grade estimation but they were not treated as hard boundaries within a rock type. For example, blocks contained in Zone 45 could use composites in Zone 303 if they were in the Zone 45 search orientation and within the same rock type.

The diabase dykes were not estimated because they are essentially barren.

Table 14-2: Haile Model Indicator Estimation Parameters

Rock Types Estimated	Code		Method
Meta-Sediments	100		1 Stage IK and 0.010 Discriminator
Meta-Volcanics	200		1 Stage IK and 0.010 Discriminator
Saprolite	500		1 Stage IK and 0.010 Discriminator
CPS Sand	600		Ordinary Linear Kriging

Notes:
Variogram zones are soft boundaries within a rock type
Meta-Sediments and Saprolite is a soft boundary
The rest of the rock type boundaries are hard bounds
Max of 10 composites, Min of 2 comps, Max/hole = 4
All search paramters are in feet
Inferred used 50 ft additional search for all areas

Indicator Kriging Parameters

Meta-Sediments

Vario Zone	Bearing Degrees	Plunge Degrees	Range and Search Plunge	Range and Search Strike	Range and Search Cross	Variogram Nugget	Variogram Sill	Discrim oz/ton
201	345	20	150	150	112	0.1	0.9	0.010
202	345	20	150	150	112	0.1	0.9	0.010
204	345	20	150	150	112	0.1	0.9	0.010
301	330	30	150	150	112	0.1	0.9	0.010
303	315	30	150	150	112	0.1	0.9	0.010
304	315	30	150	150	112	0.1	0.9	0.010
45	335	45	165	120	165	0.1	0.9	0.010
60	315	60	180	150	112	0.1	0.9	0.010
452	315	45	165	120	165	0.1	0.9	0.010
602 HS	325	60	180	150	112	0.1	0.9	0.010

Meta-Volcanics

Vario Zone	Bearing Degrees	Plunge Degrees	Range and Search Plunge	Range and Search Strike	Range and Search Cross	Variogram Nugget	Variogram Sill	Discrim oz/ton
201	345	20	150	150	25	0.1	0.9	0.010
202	345	20	150	150	25	0.1	0.9	0.010
204	345	20	150	150	25	0.1	0.9	0.010
301	330	30	150	150	25	0.1	0.9	0.010
303	315	30	150	150	25	0.1	0.9	0.010
304	315	30	150	150	25	0.1	0.9	0.010
45	335	45	165	120	25	0.1	0.9	0.010
60	315	60	180	150	25	0.1	0.9	0.010
452	315	45	165	120	25	0.1	0.9	0.010
602 HS	325	60	180	150	25	0.1	0.9	0.010

Saprolite

Vario Zone	Bearing Degrees	Plunge Degrees	Range and Search Plunge	Range and Search Strike	Range and Search Cross	Variogram Nugget	Variogram Sill	Discrim oz/ton
201	345	20	150	150	112	0.1	0.9	0.010
202	345	20	150	150	112	0.1	0.9	0.010
204	345	20	150	150	112	0.1	0.9	0.010
301	330	30	130	150	50	0.1	0.9	0.010
303	315	30	130	150	50	0.1	0.9	0.010
304	315	30	130	150	50	0.1	0.9	0.010
45	335	45	117	120	50	0.1	0.9	0.010
60	315	60	90	150	50	0.1	0.9	0.010
452	315	45	117	120	50	0.1	0.9	0.010
602 HS	325	60	90	150	50	0.1	0.9	0.010

Table 14-3: Haile Model Grade Estimation Parameters

Rock Types Estimated	Code		Method
Meta-Sediments	100		1 Stage IK and 0.010 Discriminator
Meta-Volcanics	200		1 Stage IK and 0.010 Discriminator
Saprolite	500		1 Stage IK and 0.010 Discriminator
CPS Sand	600		Ordinary Linear Kriging

Notes:
Variogram zones are soft boundaries within a rock type
Meta-Sediments and Saprolite is a soft boundary
The rest of the rock type boundaries are hard bounds
Max of 10 composites, Min of 2 comps, Max/hole = 4
Indicator is a hard boundary in each rock type
Same variogram used for the both indicator zones
High Grade Indicator Zone uses a search limit on high grade noted below
All search parameters are in feet
Inferred utilized 50 ft additional Radius for all Searches

Grade Kriging Parameters

Meta-Sediments

Vario	Bearing	Plunge	Range and Search			Variogram		Discrim	High Grade Limit	
Zone	Degrees	Degrees	Plunge	Strike	Cross	Nugget	Sill	oz/ton	Grd oz/ton	Max Srch
201	345	20	150	150	112	0.1	0.9	0.010	0.100	100
202	345	20	150	150	112	0.1	0.9	0.010	0.100	100
204	345	20	150	150	112	0.1	0.9	0.010	0.100	100
301	330	30	150	150	112	0.1	0.9	0.010	0.100	100
303	315	30	150	150	112	0.1	0.9	0.010	0.150	100
304	315	30	150	150	112	0.1	0.9	0.010	0.100	100
45	335	45	165	120	165	0.1	0.9	0.010	0.100	125
60	315	60	180	150	112	0.1	0.9	0.010	0.150	100
452	315	45	165	120	165	0.1	0.9	0.010	0.100	100
602 HS	325	60	180	150	112	0.1	0.9	0.010	0.500	50

Meta-Volcanics

Vario	Bearing	Plunge	Range and Search			Variogram		Discrim	High Grade Limit	
Zone	Degrees	Degrees	Plunge	Strike	Cross	Nugget	Sill	oz/ton	Grd oz/ton	Max Srch
201	345	20	150	150	25	0.1	0.9	0.010	0.100	100
202	345	20	150	150	25	0.1	0.9	0.010	0.100	100
204	345	20	150	150	25	0.1	0.9	0.010	0.100	100
301	330	30	150	150	25	0.1	0.9	0.010	0.100	100
303	315	30	150	150	25	0.1	0.9	0.010	0.100	100
304	315	30	150	150	25	0.1	0.9	0.010	0.100	100
45	335	45	165	120	25	0.1	0.9	0.010	0.100	100
60	315	60	180	150	25	0.1	0.9	0.010	0.100	100
452	315	45	165	120	25	0.1	0.9	0.010	0.100	100
602 HS	325	60	180	150	25	0.1	0.9	0.010	0.100	100

Saprolite

Vario	Bearing	Plunge	Range and Search			Variogram		Discrim	High Grade Limit	
Zone	Degrees	Degrees	Plunge	Strike	Cross	Nugget	Sill	oz/ton	Grd oz/ton	Max Srch
201	345	20	150	150	112	0.1	0.9	0.010	0.100	100
202	345	20	150	150	112	0.1	0.9	0.010	0.100	100
204	345	20	150	150	112	0.1	0.9	0.010	0.100	100
301	330	30	130	150	50	0.1	0.9	0.010	0.100	100
303	315	30	130	150	50	0.1	0.9	0.010	0.150	100
304	315	30	130	150	50	0.1	0.9	0.010	0.100	100
45	335	45	117	120	50	0.1	0.9	0.010	0.100	100
60	315	60	90	150	50	0.1	0.9	0.010	0.150	100
452	315	45	117	120	50	0.1	0.9	0.010	0.100	100
602 HS	325	60	90	150	50	0.1	0.9	0.010	0.150	100

14.1.6 Classification

Blocks were coded as measured, indicated or inferred based on the gold grade estimate, the kriged standard deviation (square root of the kriged variance), and the number of composites used to estimate the block. The classification was completed with two kriging passes:

1) The indicator and grade kriging procedures were applied as stated on Table 14-2 and Table 14-3. After which, the classification criteria below were applied to the blocks.
2) A second indicator and grade kriging run was completed where an additional 50ft was applied to all of the search parameters. Any blocks that were assigned during this pass that were not assigned with the first pass were added to the model and coded as "inferred".

The criteria for the first pass assignment were as follows:

Measured:
Kriged Standard Deviation < = 0.77
Minimum Number of Composites = 10

Indicated:
Kriged Standard Deviation < = 1.00
Minimum Number of Composites = 5 (Two Holes)

Inferred:
If not assigned a code above but received a gold grade from the first pass.

Second Pass Inferred:
A second kriging was completed with 50 ft additional search.
Any block assigned a grade in the second kriging run that was not already assigned becomes an additional inferred block

14.2 MINERAL RESOURCES

14.2.1 Open Pit Mineral Resources

The component of the block model that qualifies as an open pit mineral resource was estimated using the floating cone algorithm that is normally used as a guide for open pit mine planning. The intent of the application of the floating cone is to establish the component of mineralization that has reasonable prospects of economic extraction.

Table 14-4 summarizes the economic parameters that were applied to the resource floating cone. Table 14-5 summarizes the resulting open pit resources.

A variable recovery function was applied to model the process response as shown on Table 14-4. This equation is the result of process test work and analysis and reflects the current best estimate of the process recovery that will occur at Haile. As a result, the recovered gold grade was calculated and stored in the block model and used for the floating cone analysis. The calculation back to a true gold grade was completed and is shown on Table 14-4 and Table 14-5.

Slope angles on Table 14-4 are based on recommendations by Golder Associates as part of the feasibility study of the Haile project that was completed in December of 2010. The resource slope angles are the optimistic case presented by Golder, that assume slope dewatering and control blasting practices are successfully applied. Allowances for haul roads are accounted for in the slopes.

Operating costs on Table 14-4 are based on the assumption that the mill production rate would be increased from the 7,000 tpd feasibility rate to 10,000 tpd due to the increased tonnage in the resource.

The block model and the determination of the open pit mineral resources were completed by IMC with John Marek, P.E. acting as the qualified person for the calculation. Mr. Marek is independent of HGM, Romarco Minerals Inc. and OceanaGold and has been working on mineral resource and mineral reserve estimates for precious metals projects for over 34 years.

A component of the Horseshoe mineralization is included within the resource pit. The Horseshoe mineralization is contained within the 602 variogram zone and provides a potential target for both surface and underground evaluation. A portion of Horseshoe and deep Snake zones, along with a zone named Palomino, will be addressed with underground stopes that are outside of the resource cone.

A formal economic analysis has not been applied to the statement of resources. The floating cone was applied in order to establish that there is reasonable prospect of economic extraction.

The reader is cautioned that mineral resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the mineral resources will be realized or that they will convert to mineral reserves.

Table 14-4: Floating Cone Input Parameters for Resource

Mining Cost		$1.19/ton material		
Incremental Haul Cost		$0.01/ bench below 440		
Process Cost		$7.22 /ton ore		
G&A		$1.87 /ton ore		
Total		$9.09 /ton ore		
Process Recovery		100 x (1-(0.0583 x Grd ^ -0.3696))		
Refining Cost		$3.00 / ounce		
Slope Angles Overall Angles to Include Roads				
North Wall		48 Degrees		
South Wall		40 Degrees		
Saprolite		40 Degrees		
Sand		27 Degrees		
Calculated Cutoff Grades				
Open Pit Mineral Resources are tabulated at Internal Cutoff				
Price $/oz	Recovered Au oz/t		In-Place Au oz/t	
	Breakeven	Internal	Breakeven	Internal
1200	0.009	0.008	0.013	0.012

Table 14-5: Haile Gold Mine Open Pit Mineral Resource as of 1 January 2012 and 1 November 2014

Resources on this Table Include the Published Mineral Reserve

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000
Measured	0.012	40,529	0.052	2107.0
Indicated	0.012	36,995	0.049	1813.0
Measured + Indicated	0.012	77,524	0.051	3920.0
Inferred Resource	0.012	21,411	0.33	707.0
Notes: Tonnages are short tons of 2000 lbs Grades are in Troy ounces per short ton Gold price of $1,200 per troy ounce was applied Mineral Resources in this table include the mineral reserve				

The qualified person for the mineral resources are John Marek, P.E.

Metal price changes could materially change the estimated mineral resources in either a positive or negative way.

At this time, there are no unique situations relative to environmental, socio-economic or other relevant conditions that would put the Haile mineral resource at a higher level of risk than any other developing resource within the United States, or that would materially affect the mineral resource estimates.

The mineral resources on Table 14-5 contain 95% of the tonnage addressed as potentially minable in the underground PEA presented in Section 24. The 5% excluded mineralization is not material to the total mineral resources.

Table 14-6 is a breakdown of the open pit mineral resources disclosed in Table 14-5 by named zone.

Table 14-6: Open Pit Mineral Resource by Zone, $1200 Gold, 0.012 oz/ton Cutoff

	Measured		Indicated		Measured + Indicated		Inferred	
Zone	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton
South	14,914	0.047	10,955	0.035	25,869	0.042	5,421	0.030
Small	1,458	0.019	500	0.020	1,958	0.019	545	0.021
Chase Hill	618	0.039	1,929	0.031	2,547	0.033	4,045	0.026
Ledbetter	6,470	0.075	6,432	0.057	12,902	0.066	2,779	0.038
Snake	8,464	0.062	4,710	0.047	13,174	0.057	2,223	0.049
Champion	2,339	0.029	1,169	0.024	3,508	0.027	2,037	0.025
601	749	0.027	675	0.024	1,424	0.026	644	0.018
Horseshoe	-	-	5,717	0.090	5,717	0.090	440	0.095
Mustang	5,517	0.050	4,908	0.043	10,425	0.046	3,277	0.034
Totals	40,529	0.052	36,995	0.049	77,524	0.051	21,411	0.033

Table 14-7 summarizes the impact of gold price on open pit mineral resources.

Table 14-7: Open Pit Mineral Resource, Sensitivity to Gold Price

		Measured		Indicated		Measured + Indicated		Inferred	
Metal Price	Cutoff Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton
$1,200	0.012	40,529	0.052	36,995	0.049	77,524	0.051	21,411	0.033
$1,400	0.010	44,169	0.050	41,802	0.046	85,971	0.048	28,285	0.030
$1,500	0.010	45,372	0.049	45,707	0.045	91,079	0.047	41,286	0.032
$1,700	0.008	52,244	0.044	54,292	0.040	106,536	0.042	51,332	0.028

The $1,200 case on Table 14-7 is the published Open Pit Mineral Resource as shown on Table 14-5. John Marek, the qualified person for the open pit mineral resource believes that in light of gold market conditions during 2014, that the $1,200 case is appropriate to establish the mineral resource.

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE SUMMARY AND STATUS

The mineral reserve for the Haile project was developed as part of the feasibility study that was summarized in the Technical Report "Haile Gold Mine Project, NI43-101 Technical Report, Feasibility Study" dated 10 February 2011. The mineral reserve has not changed since that time and is based on a block model and mine plan that was completed in late 2010 as documented in that Technical Report. The mineral reserve is based on open pit mining only using conventional hard rock open pit mining techniques.

Section 14.1 discussed the assembly of a block model dated January 1, 2012 and the development of an updated mineral resource from that block model that incorporates both open pit and underground components.

The impact of the January 2012 block model on the mineral reserve was determined by reporting the tonnage and grade from the January 2012 model that was contained in the February 2011 mine plan. Using the identical mine plan geometries and cutoff grades, the following changes would occur with the latest block model.

Potential Changes between February 2011 and January 2012 Block model

Within the Mineral Reserve Mine Plan

Proven Tonnage increased10.1%	Proven Contained Ounces increased 6.7%
Probable Tonnage reduced 4%	Probable Contained Ounces reduced 8.9%
Total Prov+Prob Tonnage increased 5.2%	Prov+Prob Contained Ounces increased 1.7%

In summary, there would be an increase in confidence with ore moving from probable to proven categories due to additional drilling in 2011. However, the net change to the reserve is minor and it is the opinion of John Marek (qualified person) that the changes are not material.

Based on the checks described above, the mineral reserve is presented on Table 15-1 which is unchanged since February 10, 2011.

Preproduction waste mining at the Haile project during 2015 has encountered minor tonnages of ore. That material has been stockpiled and has not been processed at the time of this writing. Since the material has not been processed, no reduction of reserve is required due to ore production.

The qualified person for the mineral reserve is John Marek, P.E. of IMC.

Metal price changes could materially change the estimated mineral resources in either a positive or negative way. At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Haile mineral reserve at a higher level of risk than any other developing resource within the United States.

Table 15-1: Haile Gold Mine Inc. Mineral Reserves as of February 2011 and November 2014
Romarco Minerals, Inc. now owned by OceanaGold

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000	Recov Grade Troy Oz/ton	Recovered Oz x 1000
Proven	0.014	21,596	0.064	1,382.1	0.054	1,166.2
Probable	0.014	12,034	0.053	635.7	0.043	515.3
Proven + Probable	0.014	33,630	0.060	2,017.8	0.050	1,681.5

Notes:
Tonnages are short tons of 2000 lbs
Grades are in Troy ounces per short ton
Mineral Reserve Based on $950 / Troy Ounce Gold Price

15.2 MINERAL RESERVE SUPPORT FROM THE FEASIBILITY STUDY

The following information was presented in the feasibility study Technical Report dated 10 February 2011. It is presented here for completeness and the convenience of the reader.

Mineral reserves for the Haile Gold mine were developed from the block model and the feasibility mine plan. The mineral reserve is the total of all proven and probable category mineralization planned for processing during the course of the feasibility mine plan.

Economic benefit was applied to measured and indicated mineralization which when incorporated into the economic mine plan became proven and probable mineral reserves.

The mine plan utilizes practical working geometries with all necessary access roads and appropriate working room for equipment. Mine plan drawings are presented in Section 16.

Figure 15-1 illustrates the final pit geometry that results in the production of the mineral reserve. The open pits will never look like the drawing on Figure 15-1 because there will be pit back fill, and concurrent reclamation throughout the mine life. However, Figure 15-1 does illustrate the extent of excavation that is required to produce the mineral reserve.

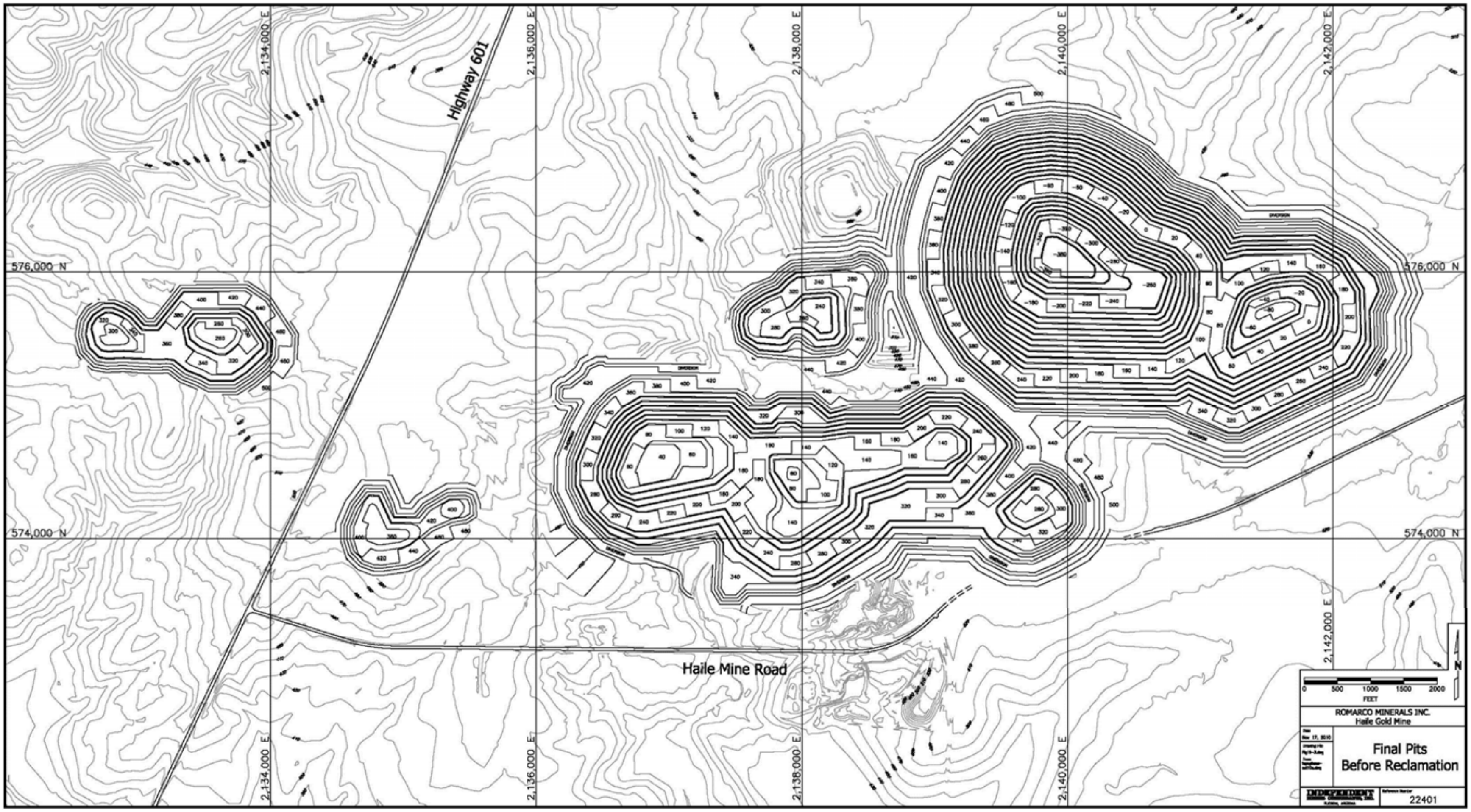
The floating cone algorithm provided guidance to the design of the pushbacks and the final pits. Multiple cones at a range of metal prices were run in order to determine the best place to start mining, initial pit openings, and guidance to final pit geometries. Table 15-2 summarizes the input economics and recoveries that were applied to the development of mineral reserves. Slope angles on Table 15-2 reflect estimated overall angles that would include mine haul roads.

Once phase (or pushback) designs were complete, the mine schedule was developed that integrated the equipment productivity and practical operating constraints. The mine schedule and mine plan is presented in Section 16.

The reserve calculations presented in this text are based on metal prices and project costs that were estimated during 2011. Since that time, operating costs have not escalated as rapidly as precious metal prices. Consequently, the statement of mineral reserves is still reasonable and likely conservative.

Table 15-2: Floating Cone Input Parameters to Guide Mineral Reserve

Mining Cost	Adjust Fuel and Lime		$1.29/ ton material	
Add Sustaining Capex			$0.16/ ton material	
			$1.44/ ton material	
Process Cost			$7.55/ ton ore	
G&A	$5,629 k$/yr		$2.20/ ton ore	
Process Recovery			100x(1-(0.0583xGrd^-0.3696))	
Refining Cost			$3.00/ ounce	
Incremental Haul Cost			$0.01/ bench below 440	
Bench Discount Rate			1.00% /bench	
Slope Angles Overall Angles to Include Roads				
		North Wall	41 Degrees	
		Deep South Wall	35 Degrees	
		Shallow South Wall	32 Degrees	
		Saprolite	40 Degrees	
		Sand	27 Degrees	
Calculated Cutoff Grades				
Price $/oz	Recovered Au oz/t		In-Place Au oz/t	
	Breakeven	Internal	Breakeven	Internal
$950	0.012	0.010	0.016	0.014



(Source: IMC, March 17 2010)

Figure 15-1: Mineral Reserve Pits Before Reclamation

16 MINING METHODS

16.1 SUMMARY

The feasibility mine plan for the Haile Gold Mine was developed by Independent Mining Consultants, Inc. (IMC). John Marek of IMC acted as the Qualified Person for the development of the feasibility mine plan.

The Haile Gold Mine is planned to be mined using conventional open pit mining methods. A combination of hard rock and soft rock will be encountered in the deposit during the mining process. The majority of the material from the mine will be hard rock which will be drilled and blasted prior to loading.

The mine plan produces 2,555 ktons of gold bearing ore per year to the process plant (7,000 tpd for 365 days/year). After a one-year preproduction period, total material movement ramps up to 22,100 ktons/year (60,500 tpd) for the first three years followed by 35,000 ktons/year (95,900 tpd) for four years.

Mining will utilize 20 ft benches. Drilling and blasting will be required for the hard rock units at Haile. The Coastal Plain Sands (CPS) will not require blasting. Saprolite will require drilling in ore zones for ore control but will require only localized blasting near the bedrock contact.

The major mine equipment has changed since the completion of the feasibility study in November of 2011. The listed equipment in Table 16-1 represents the final selection by the Haile mine engineering staff.

Table 16-1: Major Mine Equipment

Unit	Initial Fleet for 3 Years	Fleet, Year 4 and Beyond
4 ½" Blast Hole Drills	2	2
6 ½" Blast Hole Drills	2	3
17 Cubic Yd Front Loader	1	1
14.4 Cubic Yd Hyd Shovel	1	1
15.7 Cubic Yd Hyd Excavator	1	2
100 ton Trucks	12	24

Appropriate mine auxiliary and support equipment is also planned and scheduled.

The mine production schedule is summarized on Table 16-2. The annual mine plan and overburden storage drawings are summarized on Figure 16-5 through Figure 16-13. Quarterly mine plans were developed for the preproduction period and the first 2 years of the mine plan. However, only a subset of the annual plans is presented in this section for brevity.

There have been changes to the equipment and mine plan since this plan was developed. The equipment changes are noted in this text. Oceana personnel have informed John Marek and IMC that there have been no changes to the life of mine plan. A detailed review of the mine plan is anticipated next year during an integrated open pit/underground optimization study. John Marek was last on site on 1 June 2015. A minor change to the first stage design and Mill Zone pit has recently been completed during initial waste stripping. John Marek and IMC have relied on the observations of other qualified persons that are employed by Oceana that the changes have not been material.

Table 16-2: Mine Production Schedule

Year	Recov Cutoff oz/ton	Ore Ktons	Head Grade oz/ton	Recov Grade oz/ton	LG Stkp Ktons	Head Grade oz/ton	Recov Grade oz/ton	Waste Ktons	Total Mat Ktons
ppQ1								150	150
ppQ2								600	600
ppQ3	0.017	8	0.025	0.019	18	0.019	0.014	1,154	1,180
ppQ4	0.017	29	0.027	0.021	27	0.019	0.015	2,834	2,890
ppQ5	0.017	38	0.035	0.028	27	0.018	0.013	5,460	5,525
ppQ6	0.017	79	0.092	0.080	27	0.018	0.014	5,419	5,525
yr1Q1	0.017	325	0.091	0.079	55	0.018	0.013	5,145	5,525
yr1Q2	0.017	638	0.093	0.080	97	0.018	0.013	4,790	5,525
yr1Q3	0.017	638	0.085	0.073	80	0.018	0.013	4,807	5,525
yr1Q4	0.017	639	0.076	0.065	91	0.018	0.014	4,795	5,525
yr2Q1	0.019	639	0.076	0.065	102	0.020	0.015	4,784	5,525
yr2Q2	0.019	639	0.064	0.054	106	0.019	0.014	4,780	5,525
yr2Q3	0.019	639	0.055	0.046	183	0.019	0.014	4,703	5,525
yr2Q4	0.019	638	0.054	0.045	185	0.020	0.015	4,702	5,525
3	0.012	2,555	0.075	0.064	88	0.015	0.011	19,557	22,200
4	0.017	2,555	0.071	0.061	662	0.018	0.014	30,783	34,000
5	0.022	2,555	0.061	0.052	1,366	0.021	0.016	31,079	35,000
6	0.014	2,555	0.062	0.053	209	0.016	0.012	32,236	35,000
7	0.022	2,555	0.068	0.057	1,527	0.021	0.016	29,918	34,000
8	0.010	2,555	0.063	0.054				25,912	28,467
9	0.010	2,555	0.074	0.064				6,563	9,118
10	0.010	2,555	0.073	0.062				5,209	7,764
11	0.010	2,555	0.051	0.042				4,832	7,387
12	0.010	836	0.023	0.018				1,128	1,964
Total		28,780	0.066	0.056	4,850	0.020	0.015	241,340	274,970

Table 16-3: Mill Feed Schedule

Year	Cutoff oz/ton	Ore Ktons	Head Grade oz/ton	Recov Grade oz/ton
ppQ1				
ppQ2				
ppQ3	0.017			
ppQ4	0.017			
ppQ5	0.017			
ppQ6	0.017			
yr1Q1	0.017	479	0.082	0.071
yr1Q2	0.017	638	0.093	0.080
yr1Q3	0.017	638	0.085	0.073
yr1Q4	0.017	639	0.076	0.065
yr2Q1	0.019	639	0.076	0.065
yr2Q2	0.019	639	0.064	0.054
yr2Q3	0.019	639	0.055	0.046
yr2Q4	0.019	638	0.054	0.045
3	0.012	2,555	0.075	0.064
4	0.017	2,555	0.071	0.061
5	0.022	2,555	0.061	0.052
6	0.014	2,555	0.062	0.053
7	0.022	2,555	0.068	0.057
8	0.010	2,555	0.063	0.054
9	0.010	2,555	0.074	0.064
10	0.010	2,555	0.073	0.062
11	0.010	2,555	0.051	0.042
12	0.010	2,555	0.021	0.016
13	0.010	2,555	0.020	0.015
14	0.010	576	0.020	0.015
Total		33,630	0.060	0.050

Note: 1,719 Ktons in Year 12 come from the low grade stockpile. In years 13 and 14, all of the ore comes from the low grade stockpile.
Note: Tonnages are Dry Short Tons.

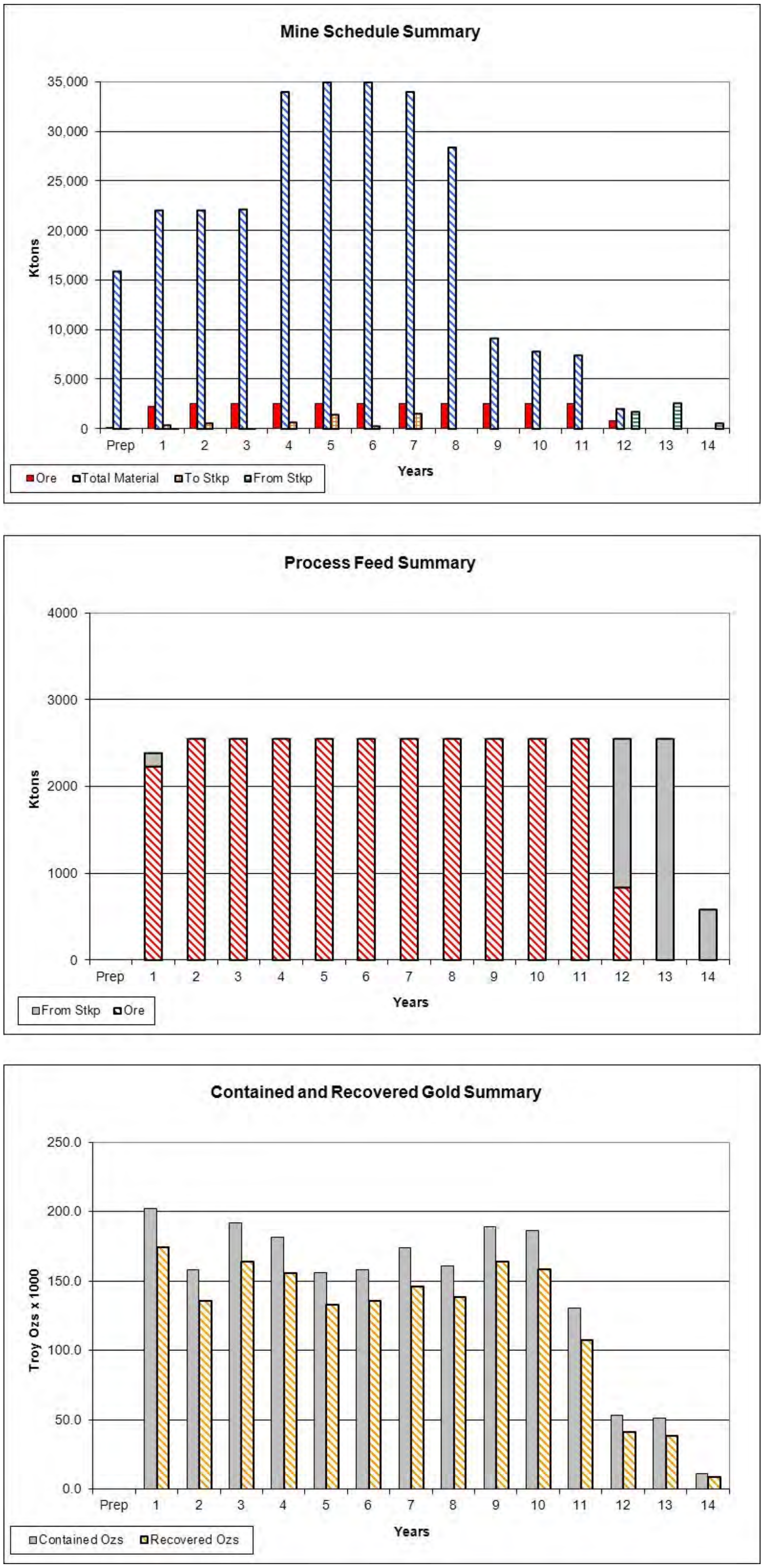


Figure 16-1: Mine Production Schedule, Graphical Summary

16.2 ECONOMIC PIT LIMITS

As discussed in Section 15, the floating cone algorithm was used as a guide to the design of the mine pushbacks and the final pit walls. The economic and process recovery information that was used as input to the floating cones is summarized on Table 15-2. A gold price of $950/oz was used as the design metal price for the mine plan. For reference, the three year trailing average of gold price was $975/oz when the modeling and mine planning effort was commenced on October 1st, 2010.

The mining cost inputs to the floating cone were based on earlier mine planning work completed by IMC during early 2010 with updates for fuel costs and lime blending costs for moderate Acid Rock Drainage (ARD) material being stored in the pits (discussed later).

Process costs and recoveries were provided by the project process team. The variable recovery equation on Table 15-2 was applied to the entire mine planning economic analysis. Economic benefit has been applied to measured and indicated category material only for development of the feasibility mine plan. Any inferred category material that is incurred in the feasibility mine plan is treated as overburden.

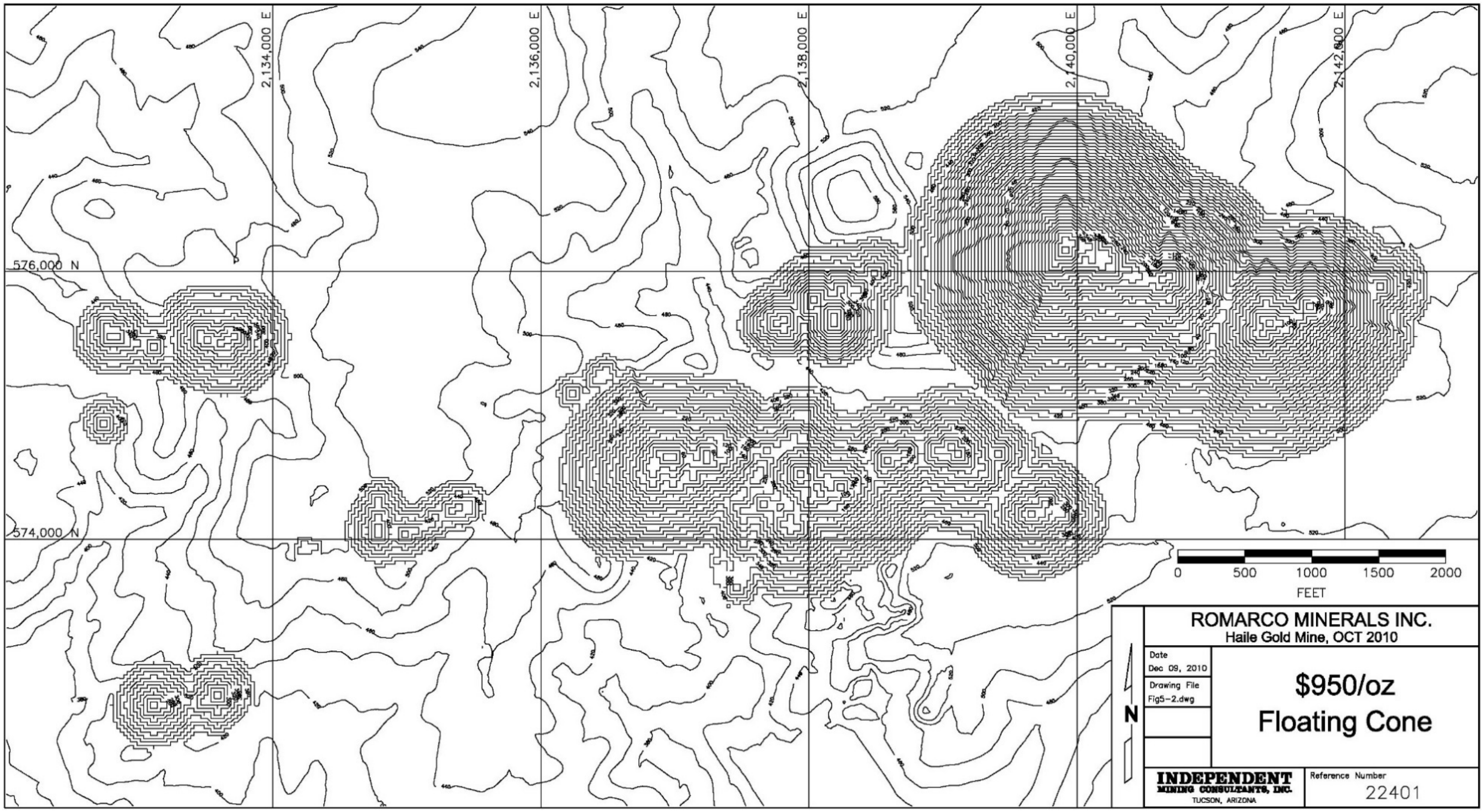
Slope angles for the floating cones and later phase designs were recommended by Golder Associates, Inc., in their report titled “Feasibility Level Pit Slope Evaluation”, March 2010. The interramp angles recommended by Golder were reduced to reflect overall angles for input to the floating cone program. The average reduction for haul roads was based on the haul road geometries within a previous iteration of Haile mine planning by IMC.

Figure 16-2 illustrates the $950/oz floating cone that was used as a guide for design of the final pits. Figure 16-2 is at the same scale and can be compared with the final pit designs that were presented on Figure 15-1.

There is a small pit area on the floating cone plot in the southwest corner of the map. This area is referred to as the 601 Pit because it underlies highway 601. The 601 is not included in the mine plan or mineral reserves as illustrated on Figure 16-2.The floating cone in this area resulted in 1.238 ktons at an average grade of 0.027 ounces per ton with a total pit volume of 3.304 ktons. This material is included in the report resources.

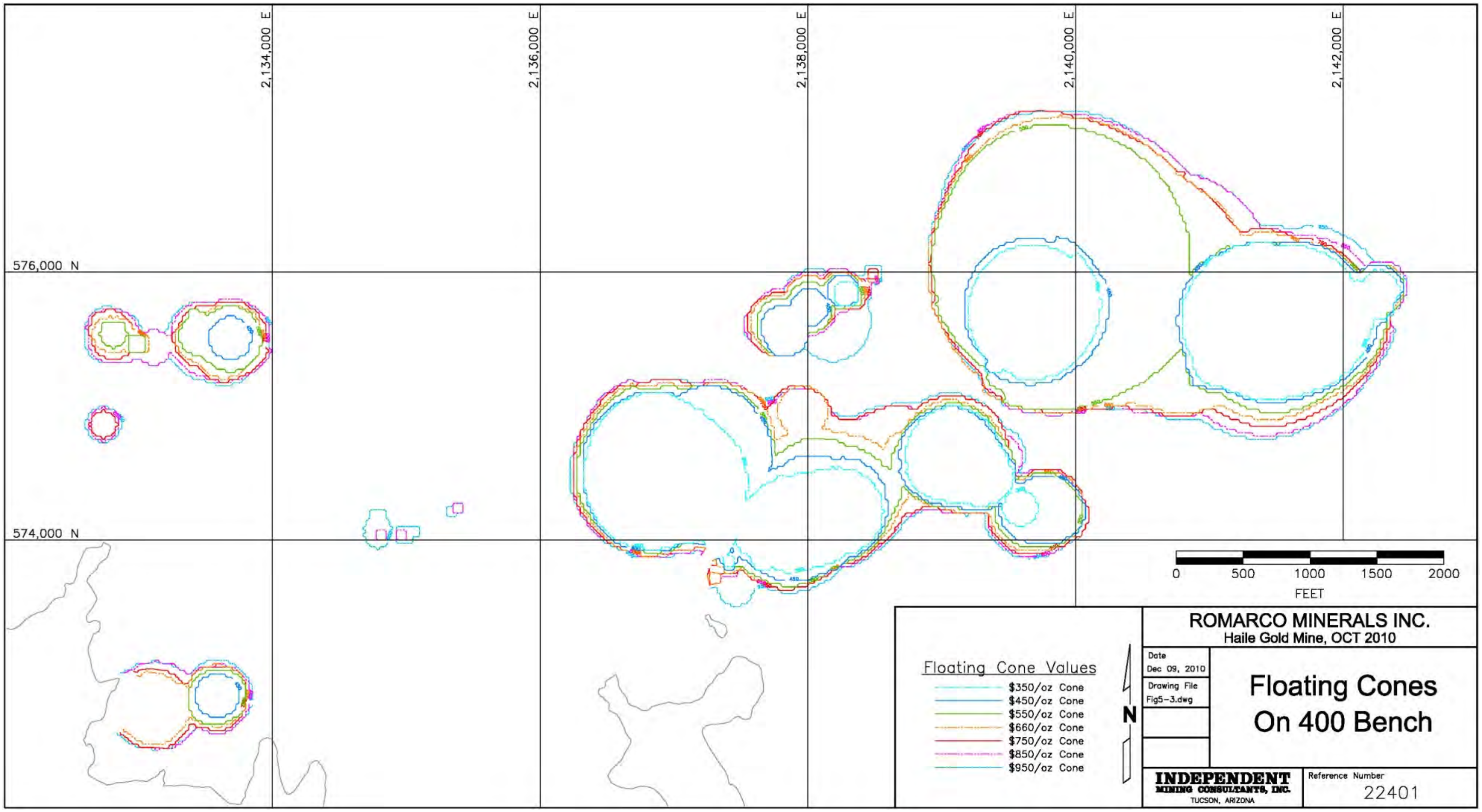
In addition to the $950/oz cone, IMC completed multiple floating cones at metal prices ranging from $350/oz to $950/oz. These floating cones were used as guidance for the development of pushback designs at Haile.

Figure 16-3 is a bench map through all of the floating cones on the 400 ft elevation. The illustration shows the location of the high valued pits relative to the lower value final pit outlines. The guidance for internal phases at Snake and Ledbetter is illustrated on the plot.



(Source: IMC, 2010)

Figure 16-2: $950/oz Floating Cone Guide to Final Pit



(Source: IMC, 2010)

Figure 16-3: Multiple Metal Price Cones On 400 Bench

16.3 PHASE DESIGN

Phases or pushbacks are practical expansions of an open pit. Their designs incorporate proper equipment operating room and all necessary in-pit mine access roads. They are designed independent of time. At any point in time, two or three pushbacks will be in production. For example, overburden will be mined from Phase 2 while ore is still being produced from Phase 1. Phase 2 overburden stripping must be complete prior to exhaustion of ore from Phase 1 in order to guarantee sustained ore feed to the mill.

A total of ten primary pushbacks were designed for the development of a practical mine production schedule at Haile. A sub-phase was added as part of the first pit opening that is used to schedule the removal of the historic 188 overburden storage. In addition, the second phase at Ledbetter was split into two sub-phases for scheduling in order to assure proper access to that working area. Consequently, a total of 12 phase designs were developed for input to mine scheduling.

The sequence of phase extraction parallels the extraction sequence suggested by the floating cone results. Modifications to extraction order were sometimes required for environmental and practical access constraints.

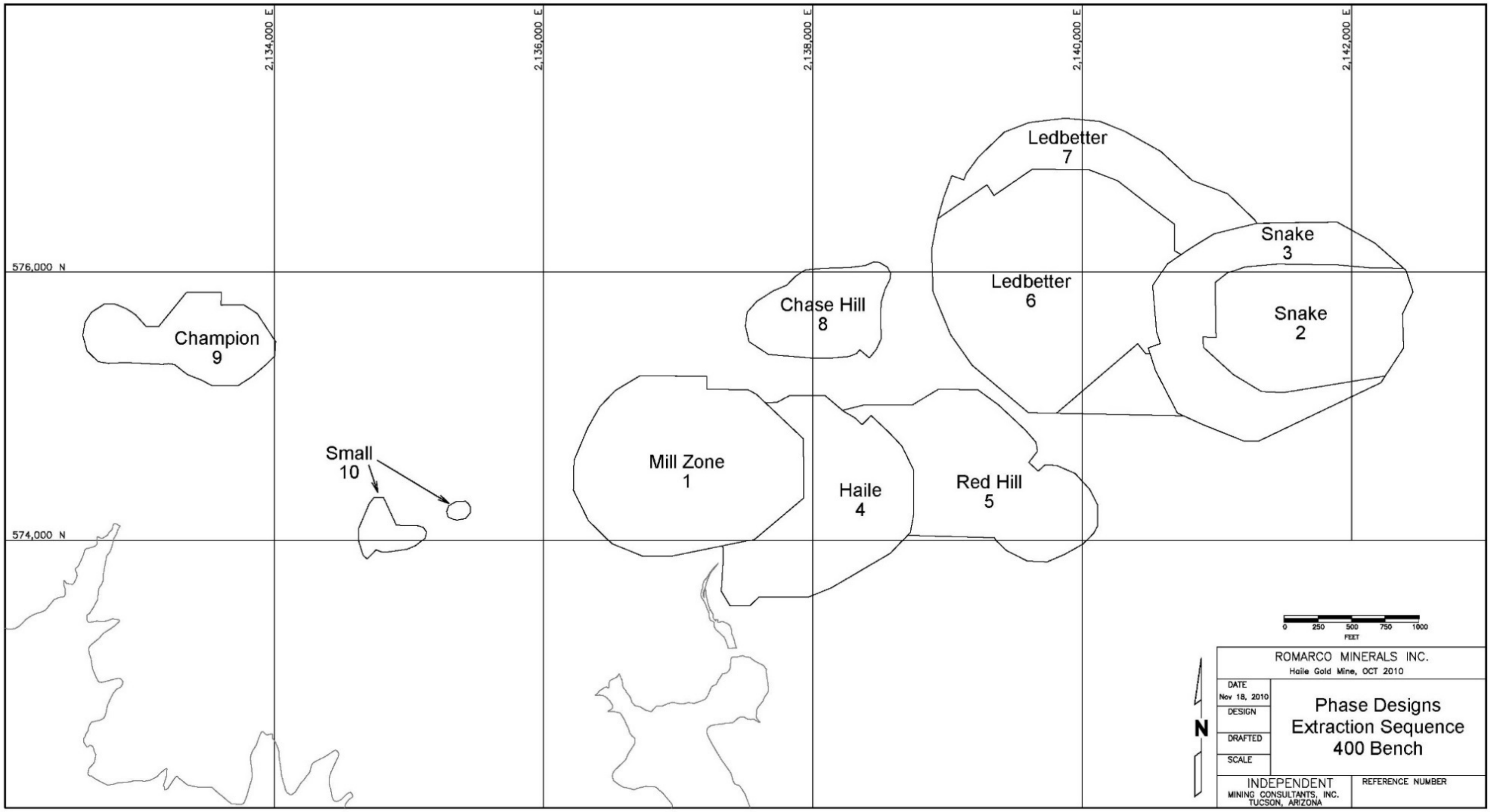
The following design criteria were incorporated into the phase designs:

Bench Height	20 ft
Road Width with Ditch and Berm	95 ft
Maximum Road Gradient	10%
Typical Pushback Width	300 ft

Interramp slope angles are as follows based on recommendations from Golder:

Zone	Interramp Angle Degrees
Sand	27
Saprolite	40
South Pit	
North Side	49
South Side	38
Ledbetter Pit	
North Side	49
South Side	42
Snake Pit	
North Side	49
South Side	45

Figure 16-4 illustrates phase locations and extraction order by slicing through all the phase designs on the 400 bench. Figure 16-4 is at the same scale and bench elevation as the multiple cone plot on Figure 16-2. The two can be compared to understand the difference between the theoretical extraction sequence and the practical sequence as established by the pushback designs.



(Source: IMC, 2010)

Figure 16-4: Phase Designs – Extraction Sequence – 400 Bench

16.4 MINE PLAN AND PRODUCTION SCHEDULE

The mine production schedule is presented on Table 16-2. This schedule was developed from the model and the mine phase designs. The schedule was developed to deliver 7,000 tpd (2,555 kt/yr) of ore to the process plant located northeast of the pit.

Table 16-2 illustrates the mill ore, low grade, and overburden scheduled for movement by time period over the planned mine life. Preproduction and the first two years of ore production are broken into quarters to show greater detail.

The mine schedule must meet several criteria.

1) Ore feed must meet mill requirements.
2) Sufficient overburden movement must be planned to assure continued ore release throughout the mine life.
3) Practical limitations on the number of bench drop cuts per pushback must be respected (12 benches/year).
4) Equipment capacities are part of the input criteria to the schedule.

Once these constraints were met, IMC adjusted the total material rate and the cutoff grade in an effort to maximize the project return on investment. For any specific total material movement schedule, there is a given amount of ore that can be released and processed. Higher cut off grades result in higher total material mined, with corresponding elevated head grades. The trade-off between the capital and operating cost of overburden stripping versus the benefit of higher grade ores is balanced using a net present value analysis to compare cutoff grade and total material rate alternatives.

Based on the economics presented on Table 15-2 as floating cone inputs, the breakeven cutoff grade is 0.016 oz/ton and the internal (sometimes called “marginal”) cutoff grade is 0.014 oz/ton. Cutoff grades for mine planning are presented in terms of “recoverable gold grade” which incorporates the variable process recovery equation presented on Table 15-2. The cutoff grades in recoverable gold terms are: 0.012 oz/ton breakeven and 0.010 oz/ton internal.

The mine schedule on Table 16-2 utilizes cutoff grades that are above the breakeven level for all but 1 of the first 7 years of production. After that time, the internal cutoff grade is applied. The elevated cutoff grades shown on Table 16-2 result in a substantial improvement in project net present value as compared to operating the mine at a fixed breakeven or internal cutoff for the project life.

The breakeven cutoff grade is that grade at which the value of a ton of ore pays for the mining and processing (including recovery, post property costs and reclamation, and property G&A) of that ton of ore. The internal cutoff grade is that grade at which the value of a ton of ore pays for just the processing (including recovery, post property costs and reclamation, and property G&A) of that ton of ore.

The variable cutoff grade strategy that is incorporated into the production schedule on Table 16-2 improves the project NPV at 10% discounting by roughly $50 to $55 million dollars over operating the mine at constant breakeven or internal cutoff grades.

During the period of elevated cutoffs, low grade material above the internal cutoff is stockpiled in the eastern portion of Johnny’s Overburden for eventual processing at the end of the mine life. The stockpile eventually grows to 4.8 million tons of low grade ore that is planned for rehandle and processing in years 12 through 14.

Figure 16-5 through Figure 16-13 illustrates the mine plan at the end of the following years:

Preproduction, 1, 2, 3, 4, 5, 7, 10, 12, and End of Mine Life

The overburden storage plan is also illustrated on the figures for the same time periods.

Mining commences in the Mill Zone pit in the western portion of the South Pit area. Approximately 15,870 ktons of total material is moved during preproduction of which 154 ktons are ore that will be temporarily stockpiled for processing in Year 1.

The design of the ramp in the Mill Zone pit is under review, with a view to reducing geotechnical risk. An option to relocate the ramp within the pit, and to have the ramp exit on the south side of the pit is being investigated. Any changes that may result to Ore Reserves are expected to be minor to negligible.

During the first three quarters of preproduction, the mine will produce sand and saprolite. Some of this material will be used to build roads. The remaining sand will be sent to the 601 Overburden and the initial tonnage of saprolite will be used to establish the liner system in the eastern third of Johnny's Overburden. Once the liner is established at Johnny's Overburden, the mine will encounter 104 ktons of old tailing that will be delivered to Johnny's Overburden in the third and fourth quarters of preproduction. The 188 Overburden containing 336 ktons of old overburden will be removed in the third quarter of preproduction and stored on Johnny's Overburden.

Once preproduction is complete, the mine plan will continue to produce ore and overburden from the Mill Zone while overburden stripping is started during Year 1 in the internal phase at the Snake Pit. The second Snake phase will be started in Year 2 and will continue through Year 4.

The initial openings in the Haile pit area will be started in Year 3. The Haile and Red Hill areas will be in joint production up through Year 7.

The first Ledbetter phase is started in Year 4, with the second Ledbetter phase commencing in Year 5. Ledbetter is in continuous operation until the end of the mine life. The Champion, Chase Hill, and Small pits are produced between Years 7 and 12.

The mine intercepts two major natural drainage features in the district. Wide benches are established around the west end of the Mill Zone pushback and around the east end of the Snake and Red Hill pushbacks to function as water handling structures.

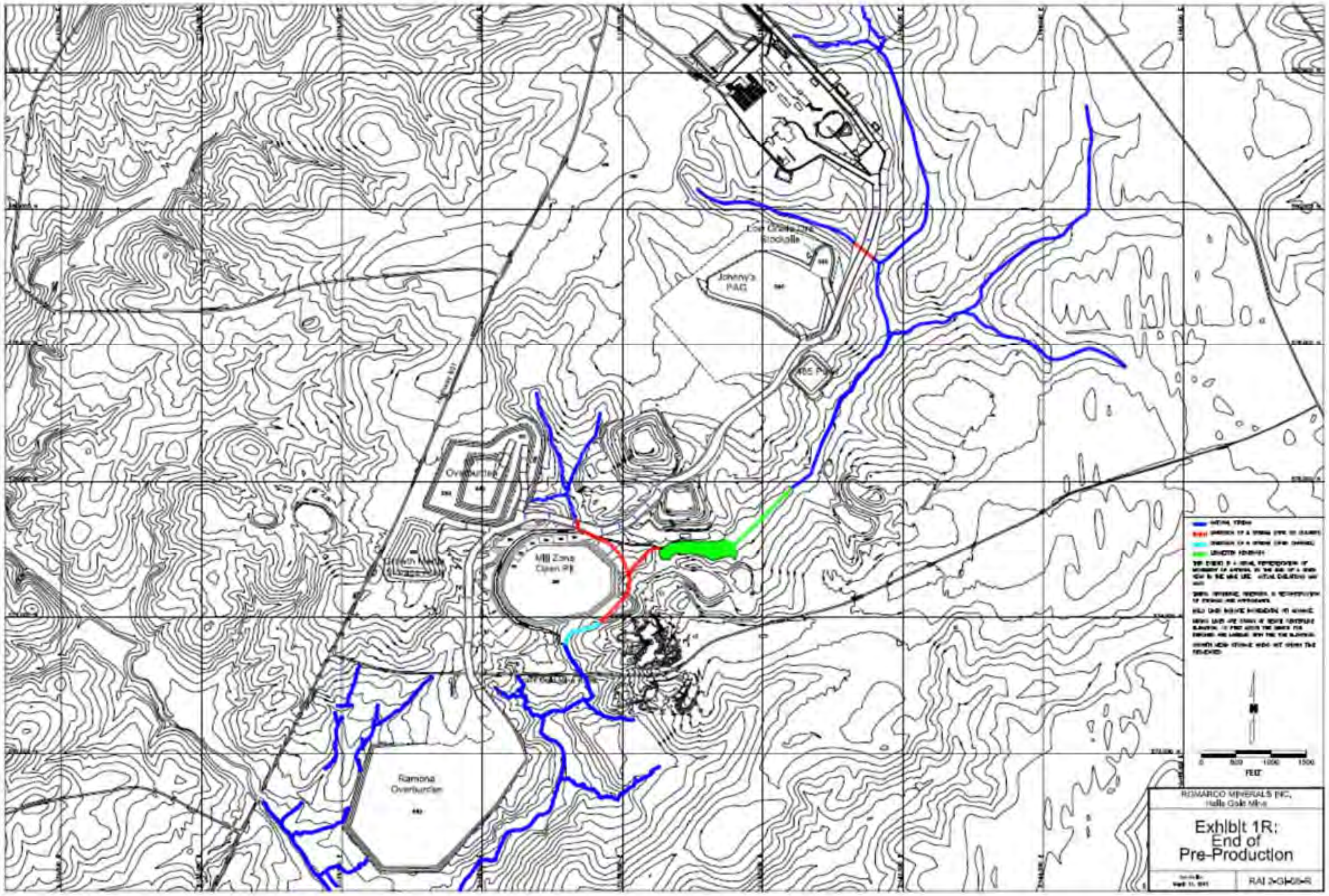


Figure 16-5: End of Preproduction

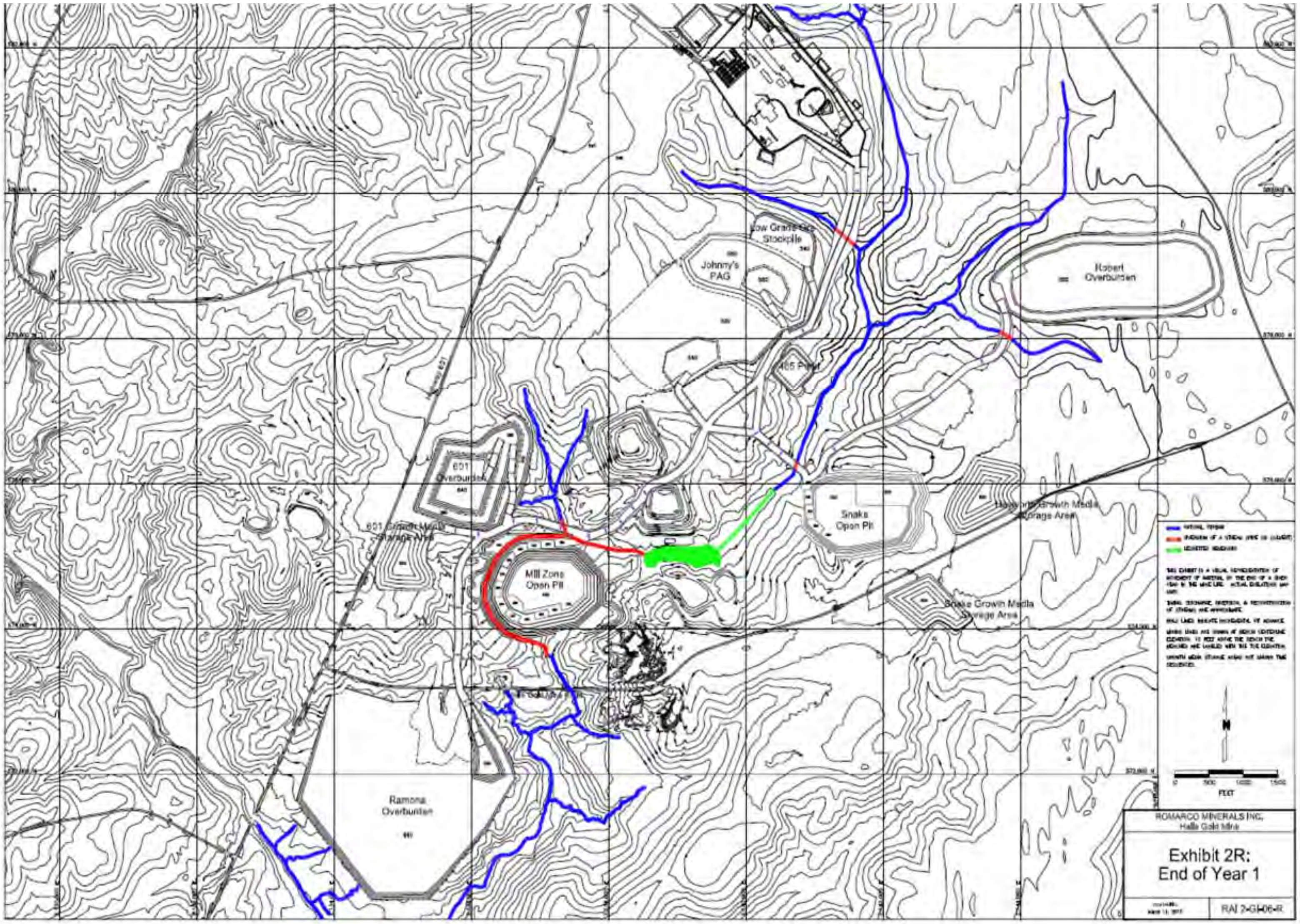


Figure 16-6: End of Year 1

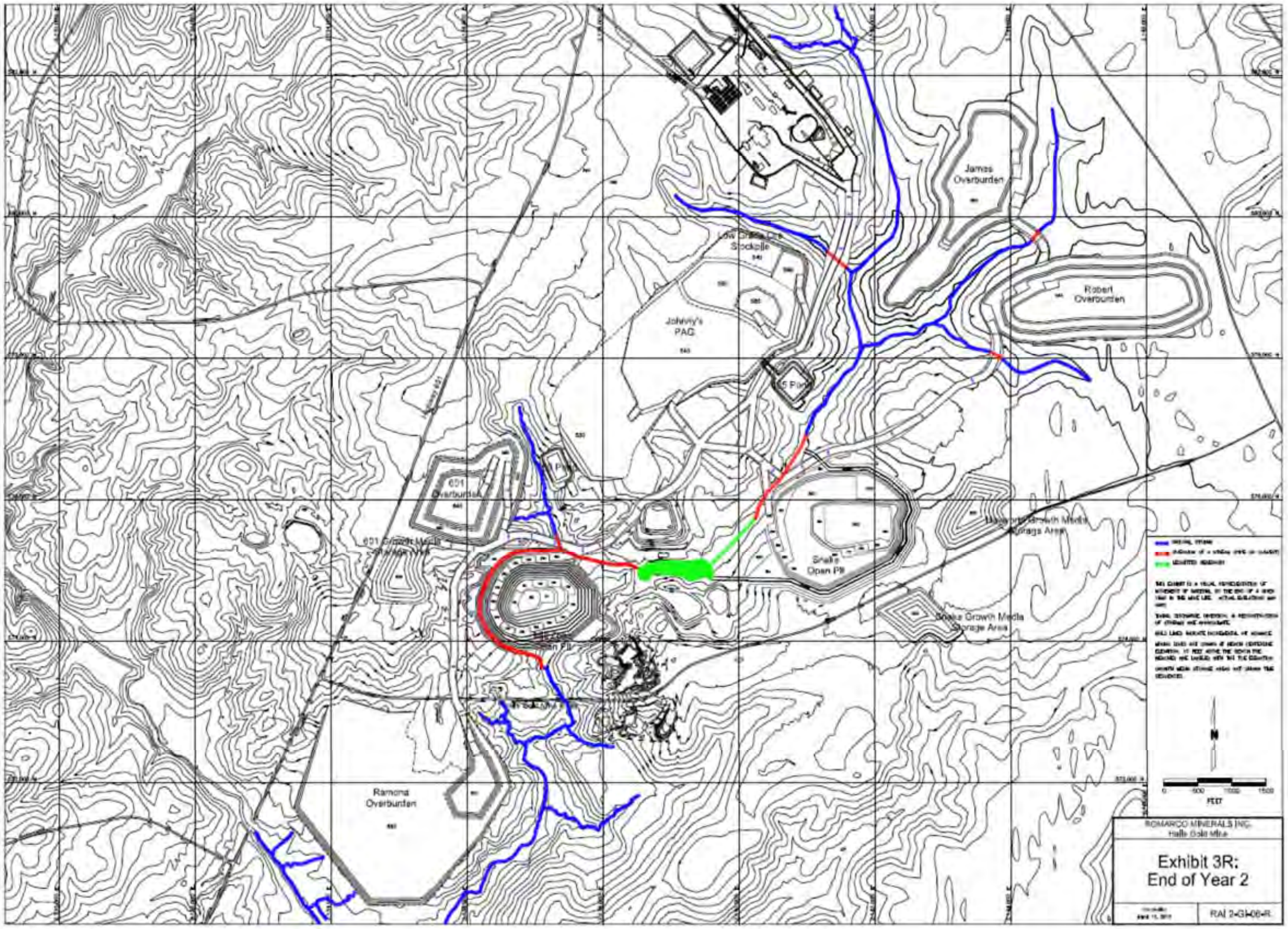


Figure 16-7: End of Year 2

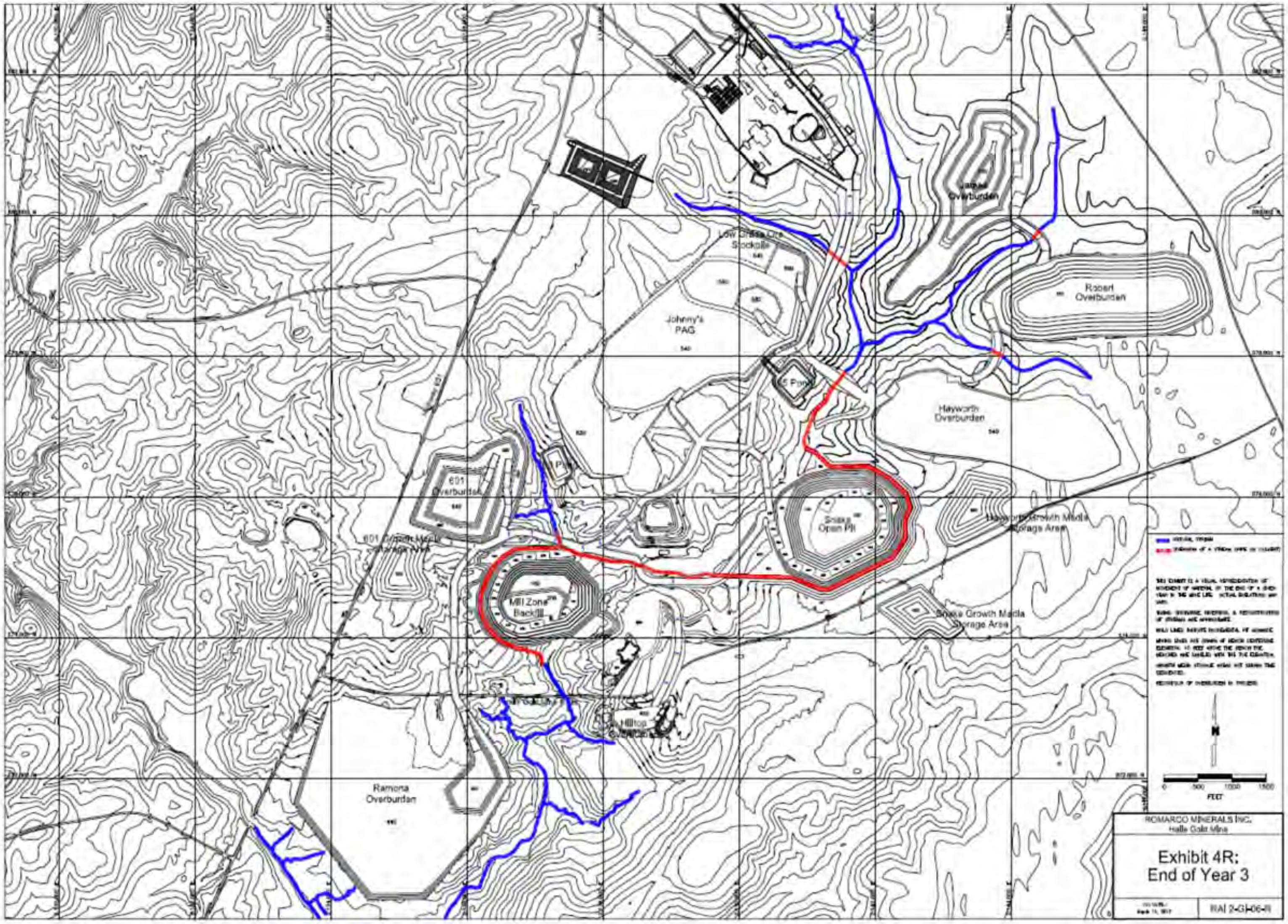


Figure 16-8: End of Year 3

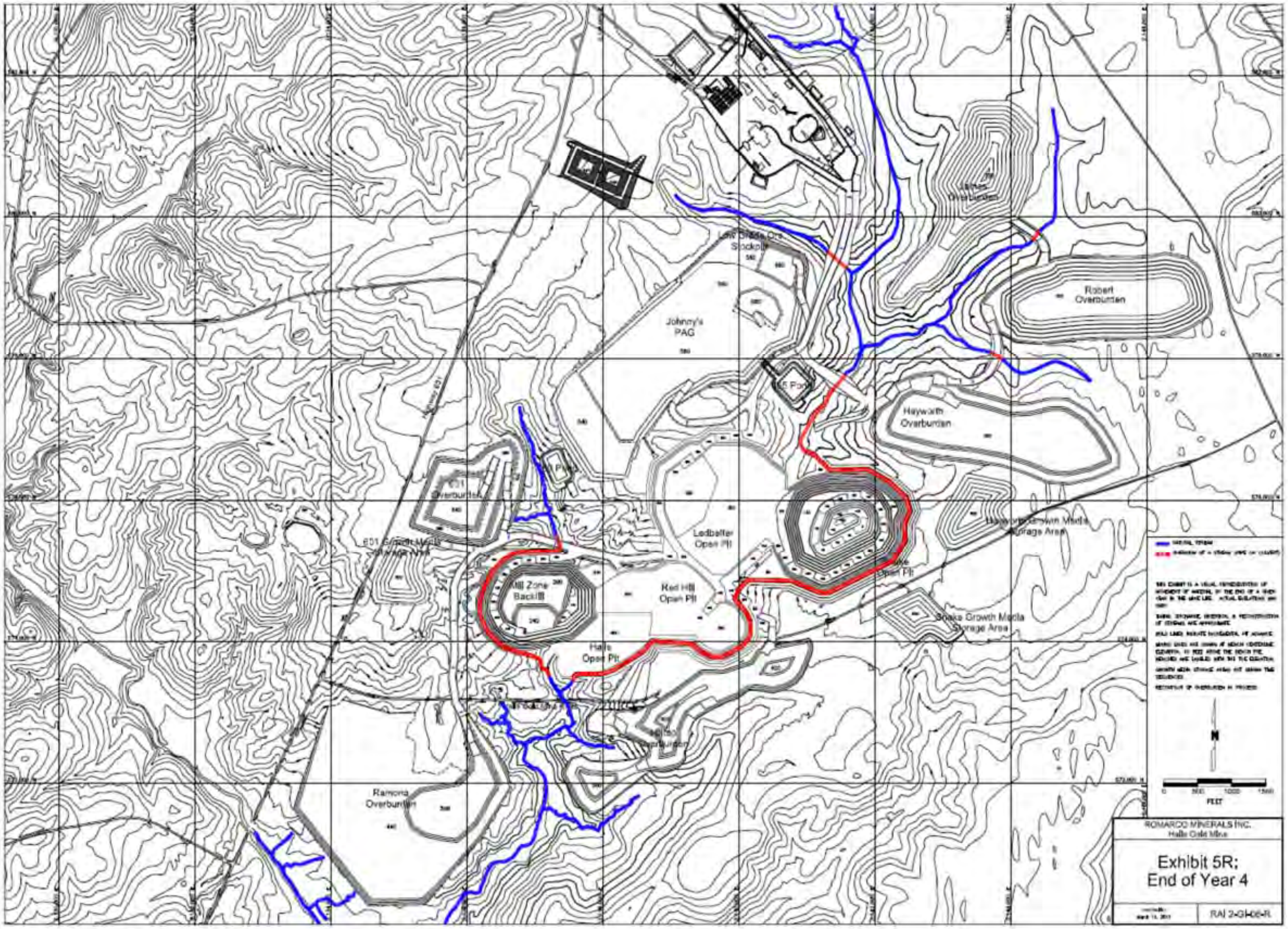


Figure 16-9: End of Year 4

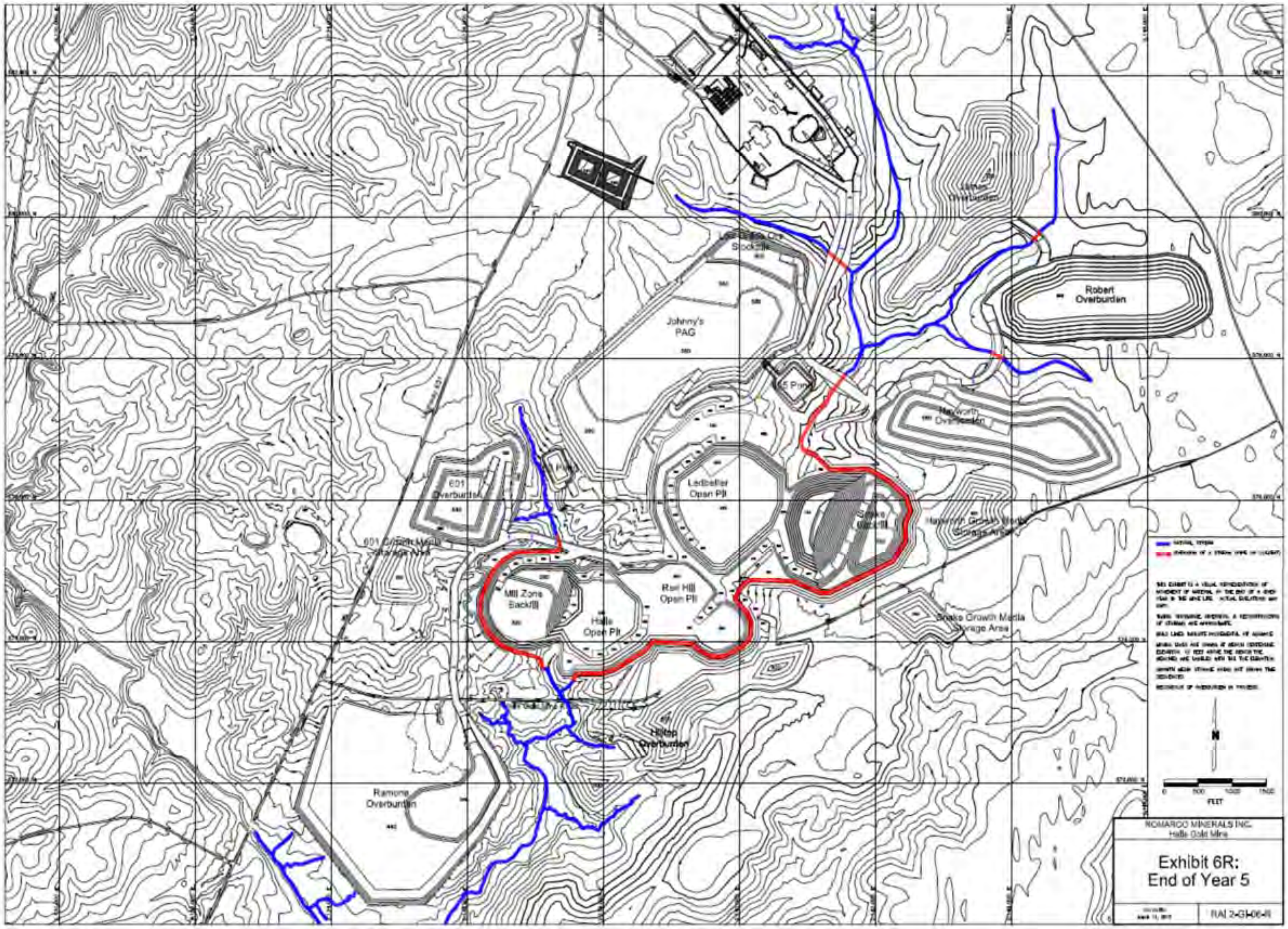


Figure 16-10: End of Year 5

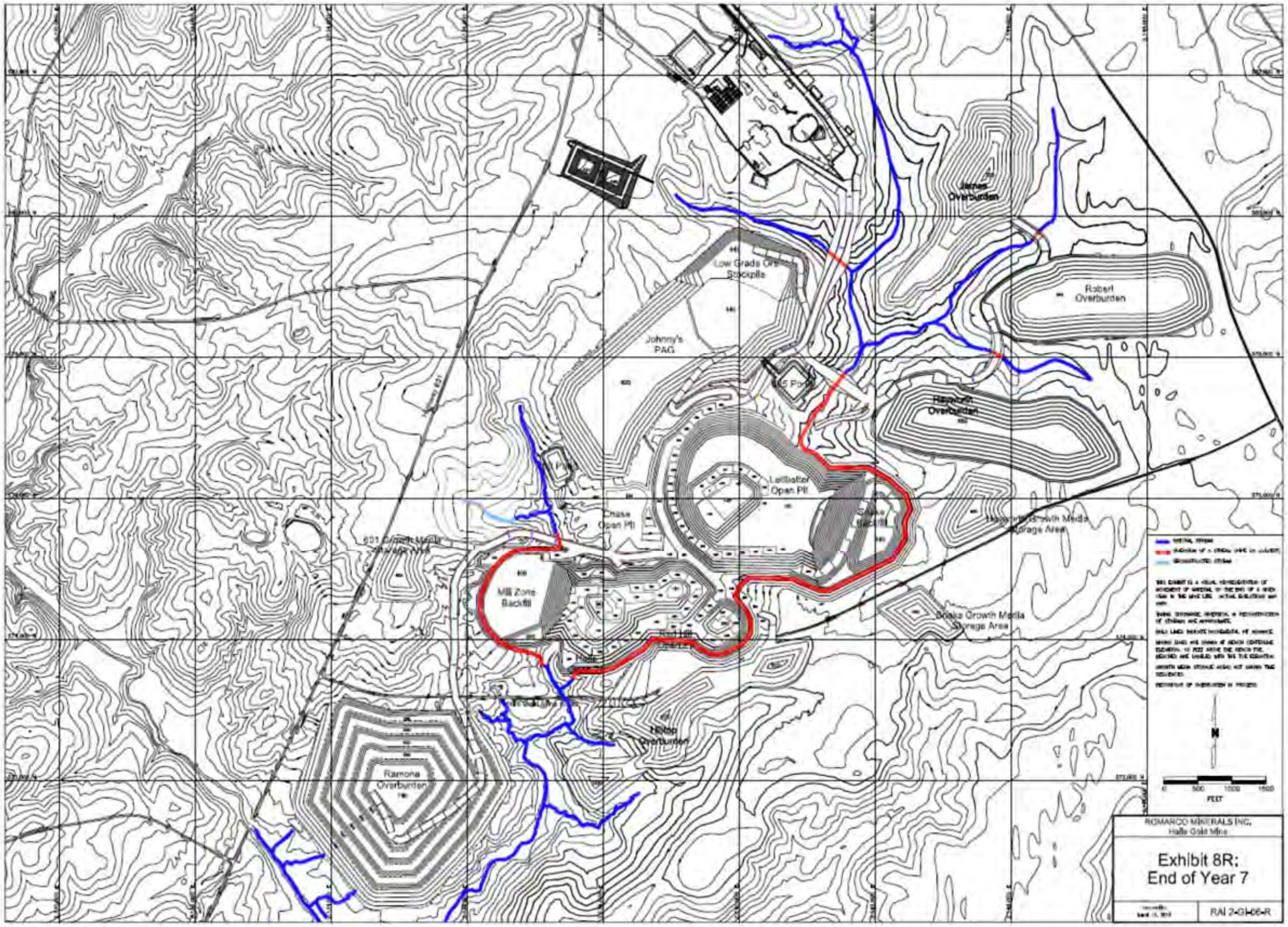


Figure 16-11: End of Year 7

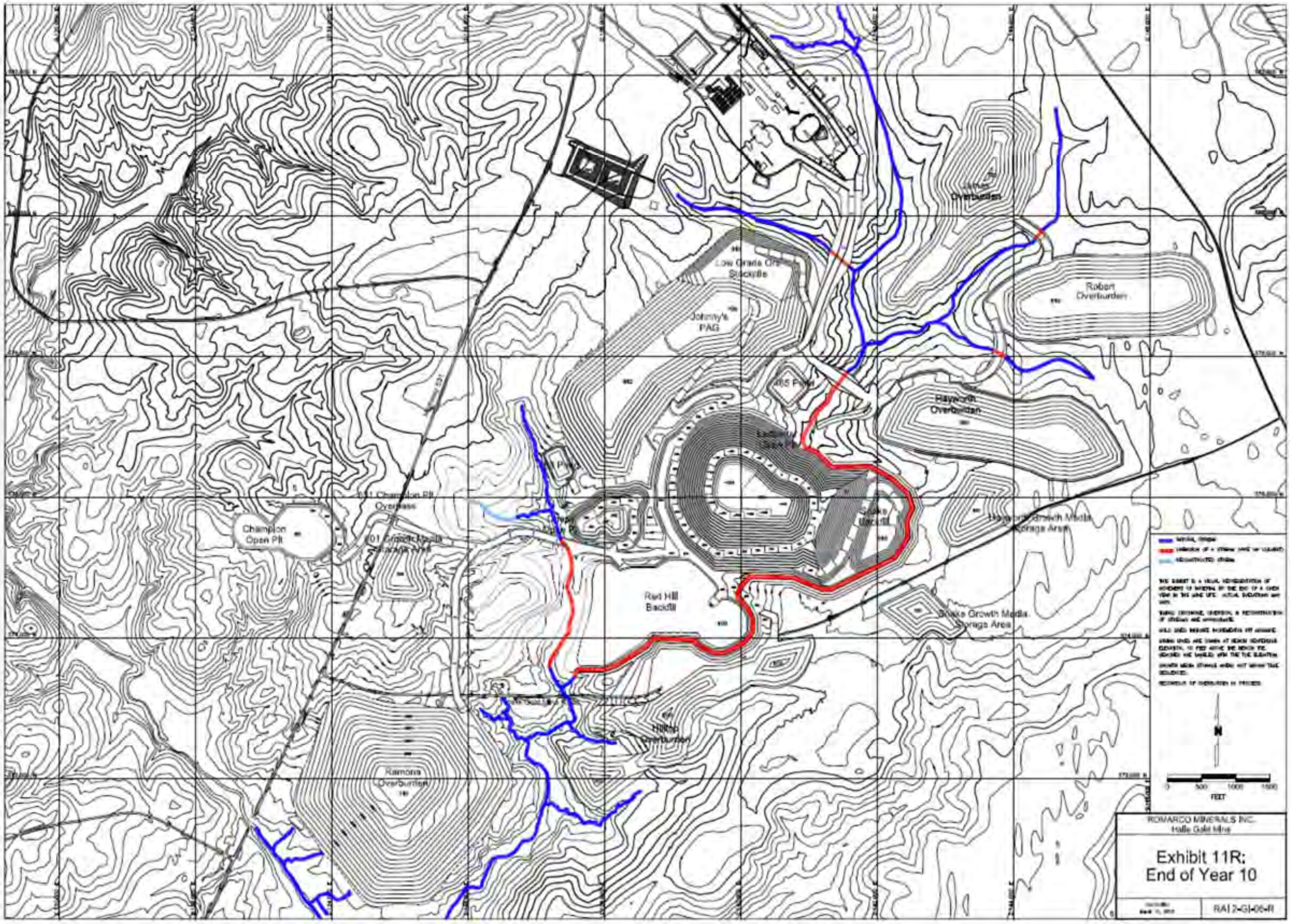


Figure 16-12: End of Year 10

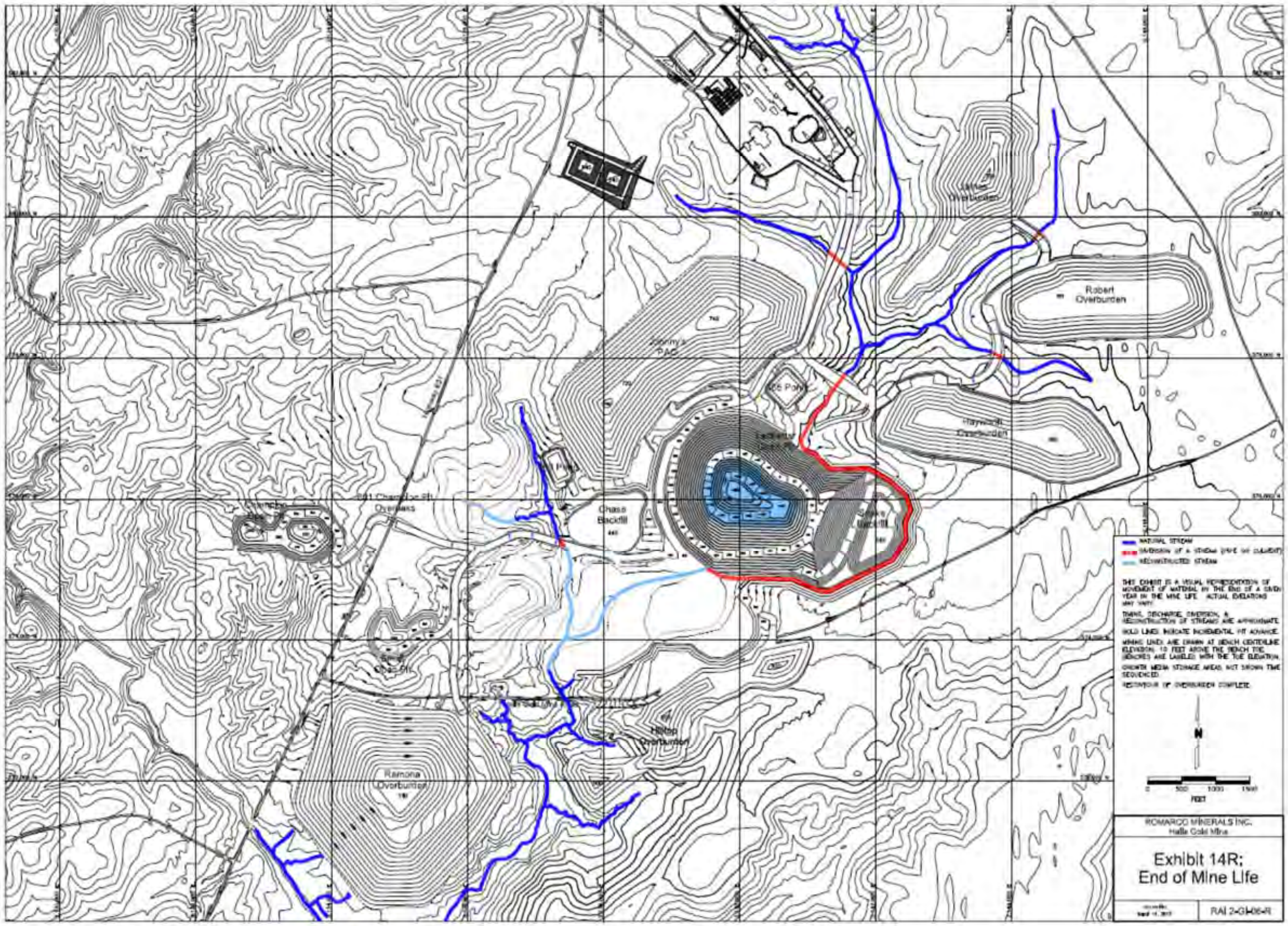


Figure 16-13: End of Mine Life

16.5 OVERBURDEN STORAGE PLAN

Disposal of mine overburden at the Haile project is complex and makes use of most usable land area within the property ownership limits. Overburden is generally stored on the tops of plateaus in the project area, keeping many of the natural drainages and wetlands open for the project life.

The acid generation potential of the overburden material at Haile has been evaluated by other contractors. As a result of that work, there are three categories of overburden material at Haile: Green, Yellow and Red in order of increasing potential for acid generation. Each rock type has been segregated into the three categories with the following allocations, as shown in Table 16-4.

Table 16-4: Overburden Material ARD Classification

Rock Type	Percentage in Each Category		
	Green	Yellow	Red
Meta-Sediments	25	30	45
Meta-Volcanics	90	10	0
Diabase	90	10	0
Saprolite	100	0	0
Sand	100	0	0
General Pit Backfill	100	0	0
188 Dump	0	0	100
Old Heap Leach			To Red Storage
Old Tailing			To Red Storage

There are a number of constraints that must be maintained within the overburden storage plan at Haile. These constraints or rules are summarized below:

1) All red material is sent to Johnny's Overburden storage area which will be a lined facility.
2) Yellow material can be stored at Johnny's.
3) Johnny's Overburden storage area must be built in 20 ft lifts. The outside 21 to 22 ft of each lift must be saprolite. Once a lift is complete, the saprolite will be dozed down to cover the exposed rock. A catchment structure will be left on every 20 ft lift.
4) Yellow material can be stored in the pits below a prescribed water table. Yellow material in-pit must be mixed with 2 lbs of lime per ton of rock before placement. Yellow in-pit must be placed in 40 ft lifts, and 5 ft of every 40 ft lift must be a layer of saprolite.
5) Yellow storage in each pit area cannot exceed the following elevations: Mill Zone = 400 ft, Snake = 440 ft, Ledbetter = 440 ft, Haile = 400 ft, Red Hill = 400ft, Chase = 470 ft, Small = 434 ft. Above these elevations green material can be stored.
6) Green material can be stored anywhere. Green storage facilities will be built in 60 ft lifts at 3 to 1 average slope. The angle of repose face of each 60 ft lift will eventually be dozed to reclamation angle.
7) The historic heap leach material, the historic 188 overburden storage, and the historic tailings that will be incurred in the pit will be placed on Johnny's lined overburden storage area.
8) The low grade stockpile will be placed on the southeast corner of Johnny's storage area for eventual re-mining to the crusher in years 12 through 14.

The northeastern portion of Johnny's storage facility was lined early during the first quarter of preproduction stripping. The mine delivered the necessary saprolite as required for the liner. A second phase of Johnny's storage facility will be lined during year 1 so that the mine can take advantage of a short haul to the southwestern portion of Johnny's during mining in the Mill Zone pit.

Backfill of the Mill Zone pit starts in year 3. Prior to that time, any yellow overburden that is incurred must be hauled to Johnny's storage facility.

The allocation of yellow material to the pit backfill has been limited to the Mill Zone, Red Hill, and Chase Hill pits. This was a result of haul distance allocation of material rather than a conscious effort to utilize one pit area over another.

16.6 MINE OPERATIONS AND EQUIPMENT

Mine mobile equipment was sized to meet the production requirements as outlined in Table 16-2. The mine equipment was selected on the basis of establishing a safe, efficient, and low cost mine operation.

The work schedule at Haile utilizes two (2) 12-hour shifts per day during production and was established in conjunction with Haile operations staff. However, during pre-production the work schedule ramps up the shifts per period utilizing one shift per day, five days per week, with one crew for the first two quarters then 1 shift per day, seven days per week, with 2 crews for the next two quarters. The final two pre-production quarters are at the full two shifts per day with four crews. The work schedule for equipment calculations and mine cost estimation within the feasibility study is as shown in Table 16-5.

Table 16-5: Work Schedule

Period	Days/Period	Shifts/Day	Lost Shifts	Shifts/Period
Preproduction Quarter 1	60	1	1	59
Preproduction Quarter 2	60	1	1	59
Preproduction Quarter 3	90	1	1	89
Preproduction Quarter 4	91	1	1	90
Preproduction Quarter 5	92	2	2	182
Preproduction Quarter 6	92	2	3	181
Year 1 Through Mine Life	365	2	10	720

The lost shifts are an allowance to account for weather delays.

Equipment efficiency within a given shift was based on 11 hours of 50 minutes each averaging 550 minutes per shift of actual equipment productive time. This value is sometimes referred to as the effective time per shift.

IMC calculated the production capacity per shift for hard rock, saprolite, and sand for each of the major equipment types. The tonnage requirement divided by the productivity per shift sets the number of operating shifts required. Further dividing by the shifts/period and application of reasonable estimates of availability and utilization result in the calculated equipment fleet.

Productivity for haulage equipment was estimated by haul time simulation. Haul profiles were measured for each material type from each pushback to every active destination for every time period. Roughly 380 haul profiles were measured for the mine life at Haile. Haul times and productivities over all of those profiles were calculated by haul time simulation.

Blast hole drilling in waste rock is planned with rotary down hole hammer drills equipped to drill 6.5 inch holes. Two drills will be needed initially followed by an addition later in the mine life. In the ore, drills equipped to drill 4.5 inch holes will be utilized; two drills will be needed to maintain ore delivery profiles.

During the first three years of production, loading equipment will be a mixed fleet of two hydraulic excavators and 1 front end loader. The hydraulic excavators are a 14.4 cubic yard shovel and a 15.7 cubic yard backhoe. The front end loader is 17 cubic yard units. The front end loaders are selected due to versatility and high mobility. The hydraulic excavators are provided to assure successful loading of the saprolite unit, as well as superior digging performance in shot rock. During periods of heavy rain, rubber tired equipment on saprolite is difficult to operate. Also, most of the saprolite will not be blasted. The excavators also have a better break-out force for digging material that has not been blasted. The loading fleet will provide required performance and flexibility for the conditions encountered at Haile.

A second excavator will be added to the loading fleet prior to the year 4 ramp up of total material.

Haul trucks are planned to be 100 ton units of which the Cat 777F is a typical example.

In addition to the drilling, loading, and hauling equipment, the auxiliary and support equipment were estimated that will be required to keep the primary units efficient and to keep the mine in good working order.

Track dozers will be both D-9 and D-10 sized units. The D-10 sized tractors will have the primary function of recontouring the overburden storage facilities.

A wheel dozer of the 834 class is provided for haul road and loading area maintenance. Motor graders will have 14 ft moldboards as typified by the Cat 14M. A water truck will be used for dust control. Utility trucks and a utility loader are within the auxiliary equipment list as well as a backhoe-excavator for general drainage maintenance and utility functions.

Table 16-6 is a summary of the mine equipment that will be on site at Haile through the mine life.

Table 16-6: Mine Mobile Equipment

Mine Major Equipment Fleet On Hand (Units owned based on fleet build up and replacement)																										
Equipment Type	PPQ1	PPQ2	PPQ3	PPQ4	PPQ5	PPQ6	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	3	4	5	6	7	8	9	10	11	12	13	14
Blast Hole Drill (6½")		3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	5	5	5	5	5	5	5	5	5	5
Blast Hole Drill (4½")						2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Hitachi 14.4 cu m Hyd. Shovel	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 777G Haul Truck															2	7	7	7	12	12	12	12	12	12	12	12
CAT DT9 Track Dozers																				2	2	2	2	2	2	2
Cat D10 Track Dozers						1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2
Cat 834H Wheel Dozer																	1	1	1	1	1	1	1	1	1	1
Cat 14M Motor Grader							1	1	1	1	1	1	1	1	1	1	1	1	2	3	3	3	3	3	3	3
Cat 773 Water Truck											1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Cat 336DL Excavator																		1	1	1	1	1	1	1	1	1
Cat 6020 Excavator					1	1	1	1	1	1	1	1	1	1	1	2	2	2	4	4	4	4	4	4	4	4
Bomag BW-213DH-40 Compactor			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

16.7 MANPOWER REQUIREMENTS

Mine operations and maintenance labor manpower are provided to operate and maintain the equipment listed previously. The labor rates shown on the table were provided by Haile personnel. IMC has established a ratio of maintenance to operating personnel in the range of 0.40 to 0.60 for most of the mine life.

Table 16-7 summarizes the mine hourly personnel requirements. Table 16-8 summarizes the mine salaried and supervisory staff for the mine life.

Table 16-7: Hourly Personnel

Mine Hourly Labor Requirements																										
JOB TITLE	PPQ1	PPQ2	PPQ3	PPQ4	PPQ5	PPQ6	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	3	4	5	6	7	8	9	10	11	12	13	14
MINE OPERATIONS:																										
Drill Operator	0	0	1	1	4	8	8	6	4	3	5	5	4	8	9	4	13	13	15	12	4	3	3	1	0	0
Shovel Operator	0	0	1	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	2	2	1	1	1
Loader Operator	0	1	1	4	7	5	3	2	5	8	3	6	7	6	5	12	10	11	9	8	1	2	2	1	1	1
Haul Truck Driver	1	3	7	20	43	43	40	39	39	42	43	40	44	46	51	72	64	90	90	74	25	22	20	7	3	3
Track Dozer Operator	1	1	3	3	7	8	8	7	7	7	7	7	7	9	8	10	12	12	12	11	5	4	5	3	2	7
Wheel Dozer Operator	0	0	1	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	2	2	1	1	1
Grader Operator	1	1	2	2	3	3	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	3	3
Service Crew	6	8	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	6	6
ECR Techs	5	5	6	6	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	10	6	6
Operations Total	13	19	34	52	94	97	95	90	91	96	94	94	98	105	109	134	135	162	162	141	69	63	62	42	23	28
MINE MAINTENANCE:																										
Mechanic	4	4	5	8	16	18	15	16	17	16	16	16	16	18	19	23	23	27	27	24	13	12	12	10	7	8
Mechanic's Helper	2	2	2	4	7	8	7	7	7	7	7	7	7	8	8	10	10	11	11	10	6	5	5	4	3	4
Welder	2	2	2	3	6	7	6	6	6	6	6	6	6	7	7	8	8	10	10	9	5	5	5	4	3	3
Electronics Technician	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	2	2	2	1	1	1
Fuel & Lube Man	0	0	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	4	4
Laborer	0	0	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Total	9	9	16	22	43	47	42	43	44	43	43	43	43	47	48	55	55	63	63	58	38	36	36	31	22	24
VS&A at 10.0%	2	3	5	7	14	14	14	13	14	14	14	14	14	15	16	19	19	23	23	20	11	10	10	7	5	5
TOTAL LABOR REQUIREMENT	24	31	55	81	151	158	151	146	149	153	151	151	155	167	173	208	209	248	248	219	118	109	108	80	50	57
Maint/Operations Ratio	0.69	0.47	0.47	0.42	0.46	0.48	0.44	0.48	0.48	0.45	0.46	0.46	0.44	0.45	0.44	0.41	0.41	0.39	0.39	0.41	0.55	0.57	0.58	0.74	0.96	0.86

Table 16-8: Mine Supervisory Personnel

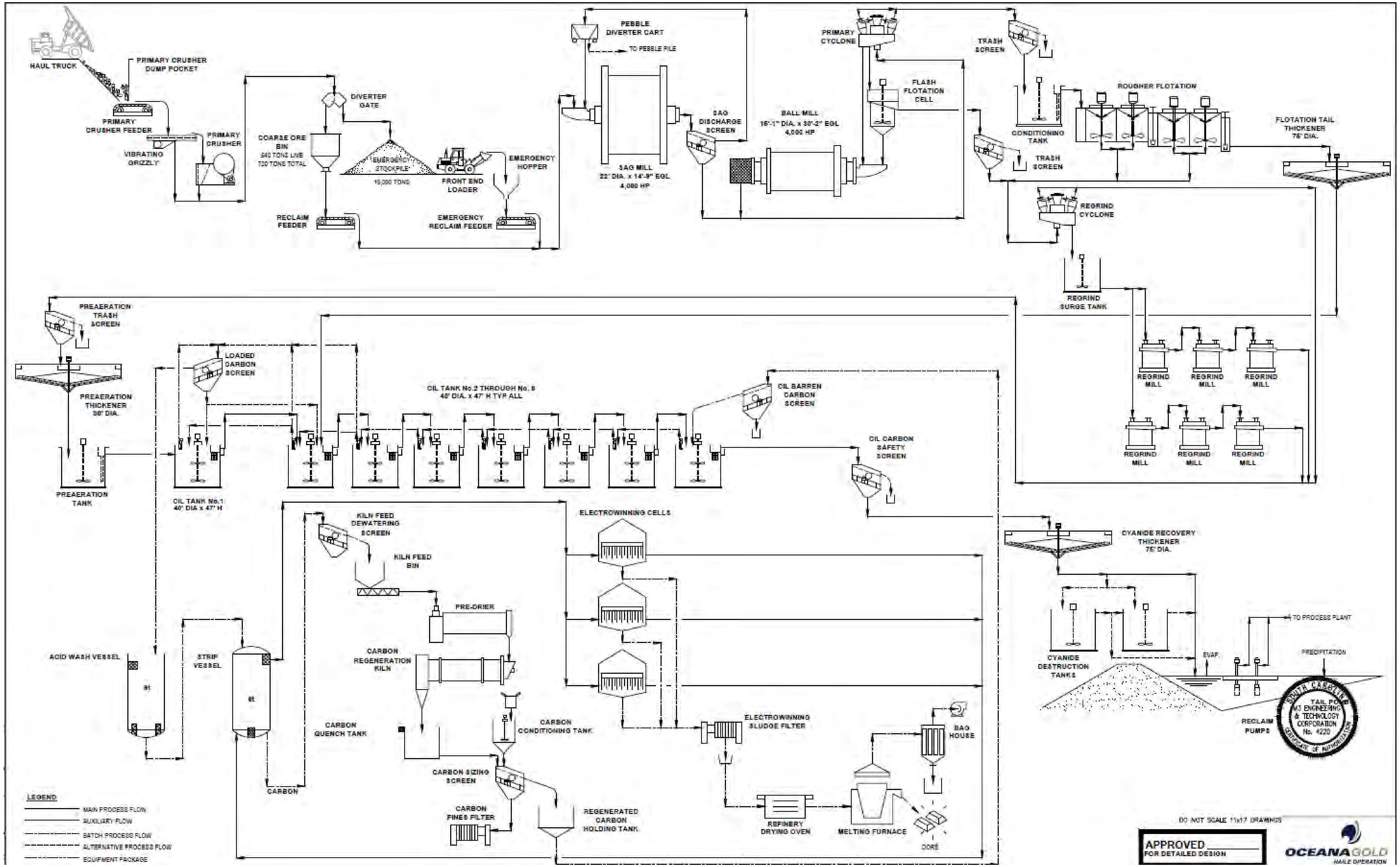
Mine Salaried and Supervisory Staff Labor Requirements																										
JOB TITLE	PPQ1	PPQ2	PPQ3	PPQ4	PPQ5	PPQ6	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	3	4	5	6	7	8	9	10	11	12	13	14
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
MINE OPERATIONS:																										
Mine Superintendant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Operations Shift Foreman	2	2	2	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2
Drill & Blast Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Trainer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1			
Mine Operations Total	6	6	6	6	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	7	4	4
MINE MAINTENANCE:																										
Mine Maint Superintendant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maint. Shift Foreman	1	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	2	2	2
RCM Supervisor					1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Buyer					1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Maintenance Clerk				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Mine Maintenance Total	3	5	6	7	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	9	7	7	5
MINE ENGINEERING:																										
Tech Services Superintendant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Ore Control Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
Geology Technician	1	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	1		
Planning Engineer			1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	
Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1		
Surveyor	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Data Management	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Engineering Total	7	8	9	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	11	8	5	4
TOTAL PERSONNEL	17	20	22	23	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	29	23	17	14

17 RECOVERY METHODS

The following items summarize the process operations required to extract gold and silver from the Haile ore. The plant was designed to process 7,000 TPD.

- ROM area for either direct tipping of ore to crusher, or storage and rehandling to the crusher.
- Size reduction of the ore by a primary jaw crusher to reduce the ore size from run-of-mine (ROM) to minus six (6) inches.
- Stockpiling primary crushed ore in coarse ore storage bin or an emergency stockpile and then reclaiming by feeders and conveyor belt.
- Grinding ore in a SAG mill – ball mill circuit prior to processing in a flotation circuit. The SAG mill will operate in closed circuit with a vibrating discharge screen and a pebble return circuit. The ball mill will operate in closed circuit with hydrocyclones to produce the desired grinding product size of 80% passing 200 mesh (74 microns).
- Grinding will occur with flotation reagents present. A portion of the grinding circuit circulating load will be treated in a flash flotation cell with the concentrate going to a regrind circuit.
- The flotation circuit will consist of rougher flotation.
- Regrinding of combined flash and rougher flotation concentrate to a desired grinding product size of 80% passing 13 microns.
- Thickening of reground concentrate prior to cyanide leaching of the slurry in agitated leach tanks. Concentrate leach discharge will be processed in a carbon in leach circuit to dissolve gold and silver contained in the slurry and to adsorb the dissolved metals from the solution onto activated carbon.
- Thickening of flotation tailing to recycle water to the grinding circuit. Thickened tails will be combined with the leached concentrate and processed in a carbon in leach circuit to dissolve gold and silver contained in the slurry and to adsorb the dissolved metals from the solution onto activated carbon.
- Removal of the loaded carbon from the CIL circuit and further treatment by acid washing, stripping with hot caustic-cyanide solution, and thermal reactivation of stripped carbon.
- Recovery of precious metal by electrowinning.
- Mixing electrowon sludge with fluxes and melting the mixture to produce gold-silver doré bars, which are the final product of the ore processing facility.
- Thickening of the leached tail stream and recovery of the cyanide solution prior to detoxification of residual cyanide as needed in the leached tail stream using sulfur and oxygen, with copper sulfate as a catalyst prior to disposal in a tailings pond.
- Water from the tailings pond will be recycled for reuse in the process. Plant water stream types include: reclaim water, internal reclaim water, fresh water, and potable water.
- Storage, preparation, and distribution of reagents to be used in the process. Anticipated reagents which require storage and distribution include: sodium cyanide, caustic soda, flocculant, copper sulfate, ammonium bisulfite, hydrochloric acid, lime, antiscalant, UNR 811 A, Aero 404, potassium amyl xanthate (PAX), lead nitrate, sulfuric acid, and MIBC.

The overall process flow sheet is shown in Figure 17-1.



(Source: M3, 2014)

Figure 17-1: Overall Process Flowsheet

17.1 OPTIONS FOR FUTURE EXPANSION

The layout provides sufficient room for future incremental expansion that may allow for a 30% increase in the overall process plant throughput. A secondary cone crusher and screen plant can be added near the primary crusher to reduce SAG mill feed size. A pebble crusher can be added to the SAG mill grinding circuit to facilitate increased SAG mill throughput rate. In conjunction with this addition, the SAG mill discharge grate openings would be enlarged and the ball charge level increased. The changes to the SAG mill circuit will result in increased load in the ball mill circuit. Accordingly, the ball mill speed and ball charge level can be increased to accommodate the increased load. Space in the layout has been provided so additional cyclones may be added to the primary and secondary cyclone clusters, and two additional regrind mills may be installed to handle the increased throughput. In addition, there is room to the east of the plant for the addition of future processing equipment such as thickeners, grinding mills, and leach tanks as needed.

A major expansion (i.e. doubling throughput) can be achieved by mirroring the plant and adding a complete second grinding/flotation/leaching line to the East of the current design.

The process facility layout includes a ROM pad, to provide flexibility for ore storage when the primary crusher is unavailable, and options to blend ore as it is fed to the crusher. The ROM pad and coarse ore storage bin are additions to the original design and layout for the processing facilities. The ROM pad design is shown in Figure 17-2.

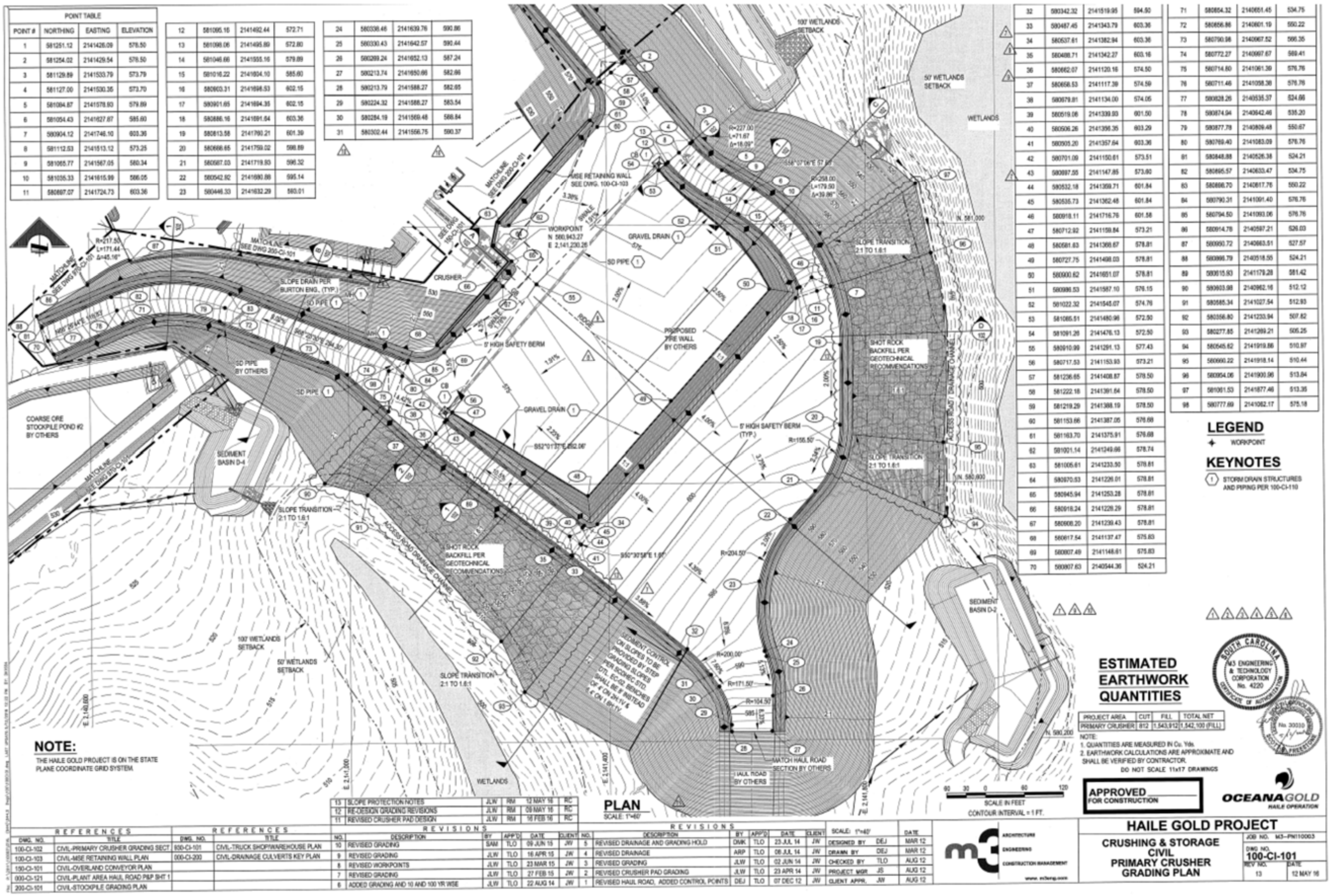


Figure 17-2: ROM Pad Design

18 PROJECT INFRASTRUCTURE

18.1 TAILING STORAGE FACILITY

The Tailing Storage Facility (TSF) was selected after evaluating 21 potential locations. The 3-year starter dam and the ultimate embankment were designed to hold 7.65 and 40.85 million tons tailing respectively. The TSF has been designed to fully contain the Probable Maximum Precipitation (PMP storm event).

Tailing slurry material (approximately 55 percent solids and 45 percent liquids by weight) will be pumped to the TSF in a HDPE pipeline. The pipeline will run along the TSF Haul Road and travel over Highway 601 on the bridge overpass for the haul road. The pipe will then be routed to the crest of the tailing facility where tailing material will be spigotted into the tailing facility via several spigots placed around the east, north and west sides of the facility. Process water will be reclaimed from the tailing facility utilizing vertical turbine pumps mounted on a floating barge in the south east corner of the facility. Process water will be recycled back to the Mill for makeup water.

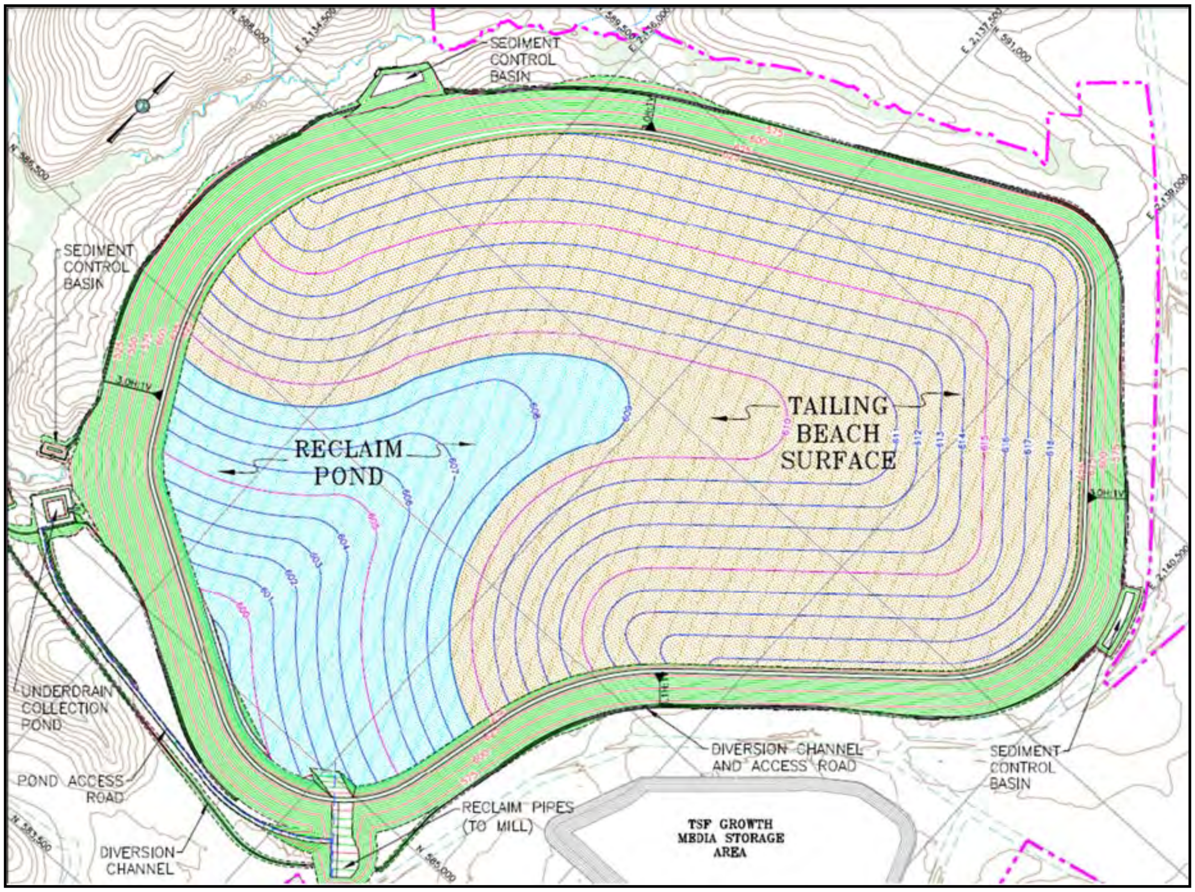


Figure 18-1: Tailing Storage Facility Layout

The TSF is a zoned earthfill embankment of random fill, coastal plain sand (chimney drain) and saprolite (low-permeability layer) within the maximum limits of the reclaim pond. The embankment interior slopes are 2.5:1 and exterior slopes are 3:1 to facilitate concurrent reclamation. All interior slopes of the embankment will be lined with geomembrane. The entire TSF basin will be lined with a composite liner system consisting of a low permeability soil liner overlain with 60-mil HDPE geomembrane.

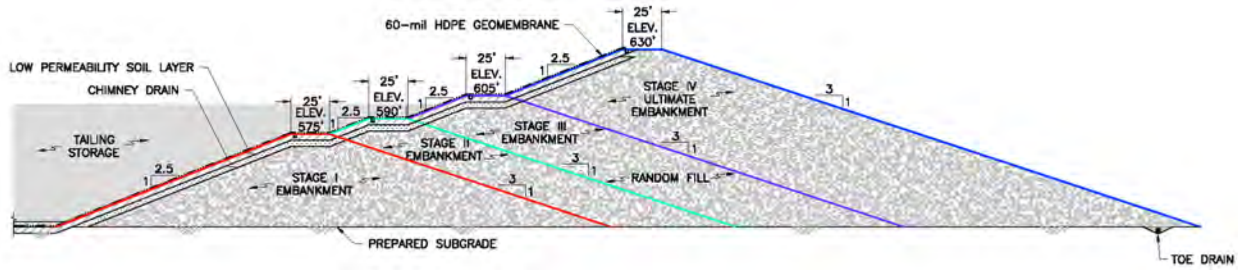


Figure 18-2: Tailing Storage Facility Typical Section

The tailing is drained through a sand layer on top of the HDPE and is collected in a series of pipes that culminate at the low point of the basin. An HDPE geomembrane double-lined pond with a leak collection and recovery system (LCRS) will be constructed downstream of the embankment toe to collect all underdrainage flows from the basin that exit through a concrete-encased series of outlet pipes.

Stability analyses were conducted under both static and seismic loading conditions. Pseudo-static based analyses are commonly used to apply equivalent seismic loading on earthfill structures. Results of the stability analyses show that even under the most extreme conditions, the TSF embankment is expected to perform as designed and prevent any catastrophic loss of tailing. The TSF can contain the PMP and still maintain 4 feet of freeboard.

18.2 OVERBURDEN STORAGE

During the mine life, seven different overburden storage areas (OSAs) will be utilized for the storage of approximately 241 million tons of material generated from the pit development. The material generated will be classified as either potentially acid generating (PAG) or non-acid generating overburden material from the development of the pits. The PAG material and Low Grade Ore Stockpile will be stored exclusively within Johnny's PAG OSA. The other six OSAs are designated as 601, Ramona, Hayworth, Hilltop, James and Robert. The OSAs will be developed according to the pit progression.

Prior to construction of the OSA's, the footprints will be timbered.

Grass lined sediment collection control channels will be constructed around the footprint of each OSA. Sediment control structures will be constructed at the outfall of the sediment control channels for each facility. Water retained within the ponds is routed through a low level riser pipe to an adjacent drainage.

All of the OSA's will be developed with a final reclaimed overall 3(H):1(V) slope. Stability analyses were completed based upon a series of field and laboratory investigations used to define the subsurface conditions and engineering characteristics of the materials, respectively. The stability analyses completed for the OSAs indicate that they are all stable as the computed factors of safety meet or exceed the prescriptive values.

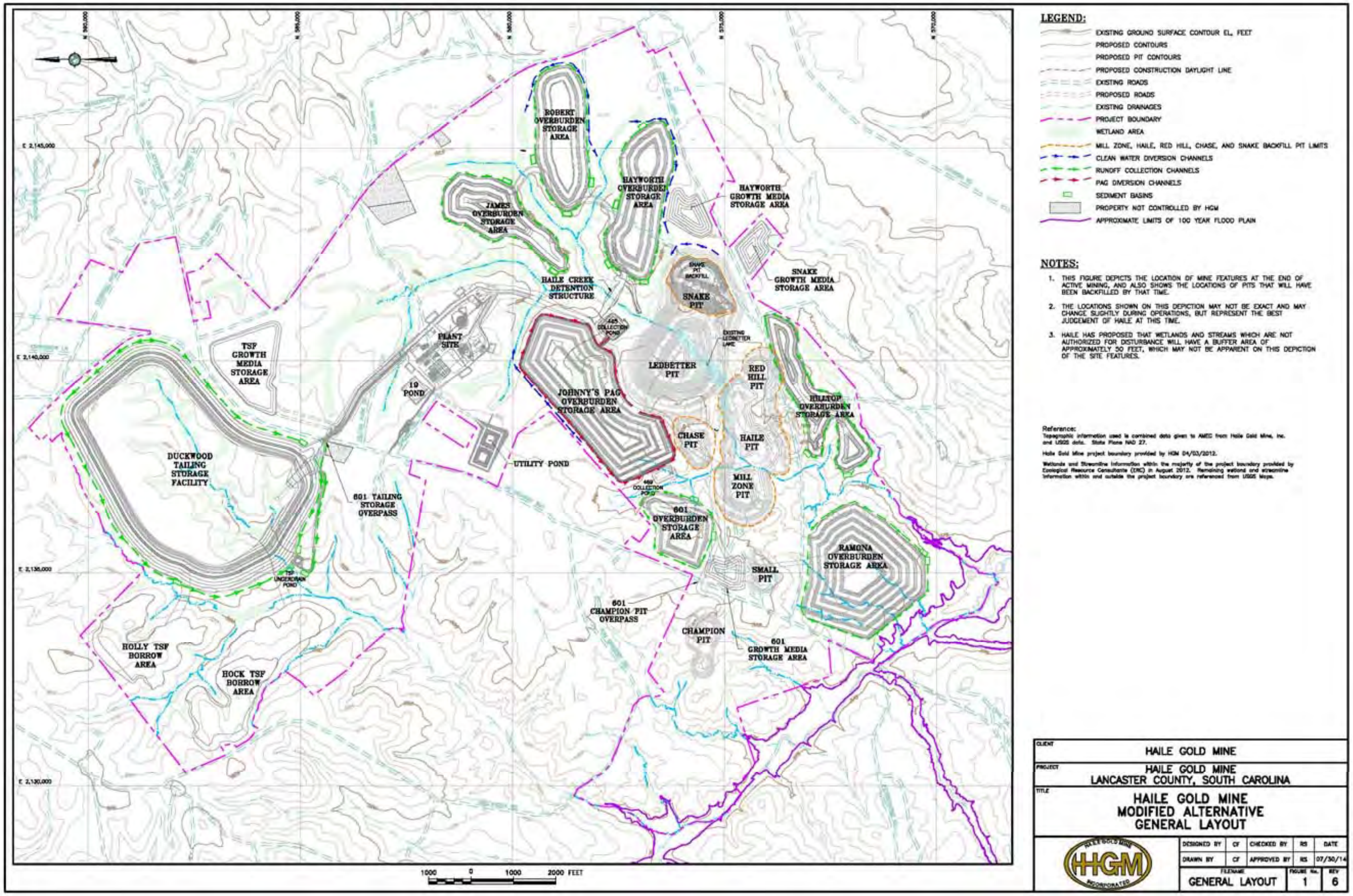


Figure 18-3: Overburden Storage Areas Plan

18.2.1 Johnny's PAG OSA

Figure 18-3 shows the general location of Johnny's PAG OSA with respect to the other facilities. Johnny's PAG OSA is a fully geomembrane-lined facility, which will contain PAG overburden material and low grade ore (LGO) and will route impacted precipitation runoff and seepage to either the 469 or 465 Ponds which are both HDPE geomembrane double-lined with a LCRS. The ultimate facility will have an overall footprint of approximately 159 acres and a capacity of approximately 28 million cubic yards, or 46 million tons, of PAG and LGO material. Material loading within Johnny's PAG OSA will be constructed with an overall slope of 3(H):1(V) and built to a maximum height, measured from the toe to the crest, of approximately 270 feet. Prior to construction of Johnny's PAG OSA, the footprint will be stripped of vegetation and topsoil. Topsoil materials will be stored in growth media area for future use in accordance with HGM reclamation plan.

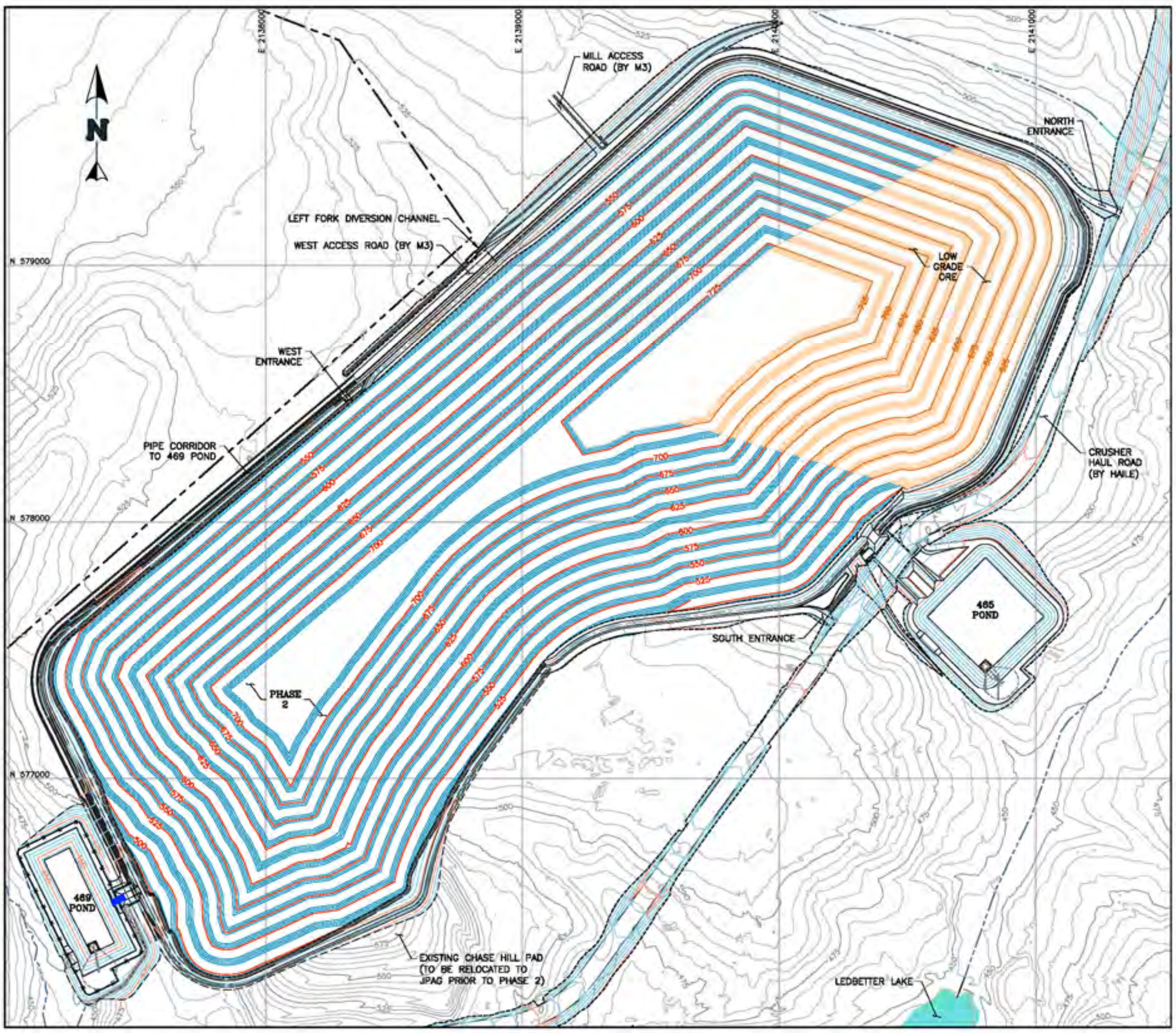


Figure 18-4: Johnny's PAG OSA Area Plan

Groundwater collection pipes will be installed below the low permeability soil liner along existing drainages to route groundwater from beneath the facility. Groundwater will be routed outside the limits of the facility into the lined collection ponds. Geomembrane-lined runoff collection channels will circumnavigate the OSA to divert potentially impacted runoff to lined collection ponds sized to capture the 100- year, 24-hour event. Perforated pipes will be installed above the geomembrane to collect precipitation infiltrating through the overburden and direct it into the collection ponds.

18.3 SURFACE GEOTECHNICAL INVESTIGATION

Nine geotechnical field and laboratory investigations have been conducted from previous design efforts and as part of the current feasibility design. Geotechnical borings and test pits were advanced to define the subsurface conditions, groundwater hydrogeology and potential borrow sources for each of the design elements. Select bulk samples, disturbed samples, and relatively undisturbed samples were collected from test pits and geotechnical borings during the field investigations for index, consolidation, hydraulic and strength characterization.

Based upon the laboratory data and field evaluation, no geotechnical fatal flaws were identified. The local soil materials have been characterized as suitable for construction materials for the various facility components. A general summary of the findings of the investigation are found below:

- Tailing Storage Facility – Consists of dense Coastal Plain Sands (CPS) varying in depth between 5 feet and 40 feet. Stiff saprolite underlies the CPS to an average depth of 70 feet below grade which is underlain with weathered bedrock. The water table was found in some areas to be 10 to 20 feet below grade.
- James, Robert and Hayworth OSA – Comprised of medium dense CPS and hard saprolite to depths of 0 to 40 feet and 40 to 65 feet respectively. Heavily weathered bedrock underlies the saprolite. The water table was not encountered during the geotechnical evaluation.
- Johnny's PAG OSA – Dense CPS resides to a depth of approximately 45 feet underlain by 20 to 30 feet of very stiff saprolite. Below the saprolite lies heavily weathered bedrock at approximately 60 to 65 feet below grade. A shallow groundwater was encountered at 45 feet below grade.
- Ramona OSA – Consists of very stiff to hard saprolite overlain by a thin layer of CPS. The shallow groundwater was encountered at 15 to 25 feet.
- 601 OSA – Consists of medium dense to dense CPS to a depth of 20 feet. Weathered bedrock was encountered at a depth of 35 feet.
- Hilltop OSA – The subsurface investigation indicates medium dense, clean to slightly silty CPS to depths varying between 14 and 35 feet underlain by very stiff saprolite. Bedrock was not encountered in the test pits or borings advanced.
- Plant Site – An extensive investigation was completed under the main plant site in 2012. The geotechnical borings indicate the interface between the CPS and saprolite varies between 11 and 50 feet. Loose sand exists in the top 5 feet that will have to be removed below all structures. The CPS at depth was medium dense to dense. Cone penetration Testing showed some thin layers of soft saprolite at depth with the majority of the clay exhibiting very stiff to extremely stiff characteristics.

18.4 SITE WIDE WATER MANAGEMENT

Across the project site a detention structure, diversion channels, culverts, conveyance pipes, sediment collection channels and sediment control basins are proposed for erosion protection and sediment control for site-wide surface water runoff due to stormwater and diversion of existing streams around the facilities. Preproduction water management will effectively route runoff around the project elements and initial pit development, while reducing sediment load as water is released back into natural drainages. The preproduction site wide water management plan is shown on Figure 18-5. Subsequent production year water management will take into consideration the dynamic design life of the facility by evaluating each water management structure at the most critical design phase with the greatest peak discharge.

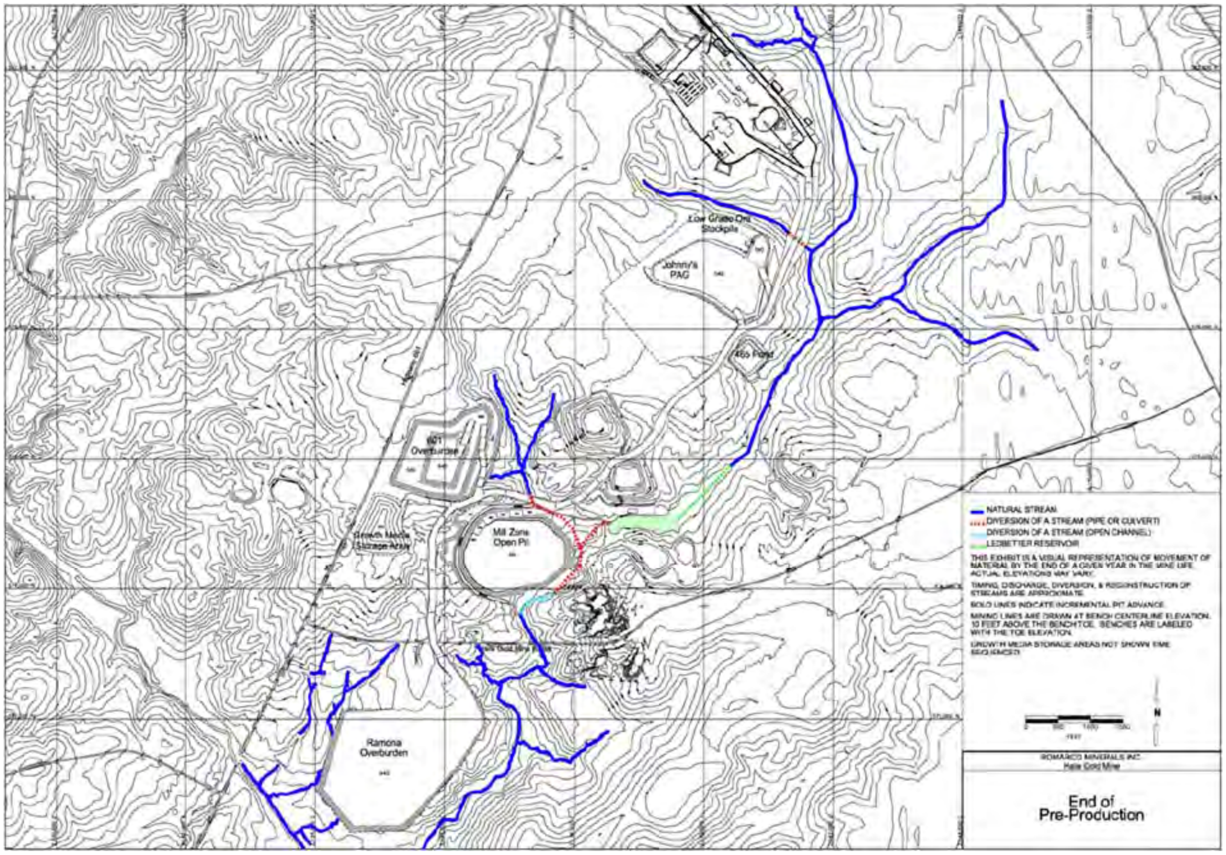


Figure 18-5: Preproduction Site Wide Water Management Plan

18.5 SITE WIDE WATER BALANCE

The site wide water balance was developed for the proposed Haile operations as a tool to aid in the planning, design and operations of the Mill, TSF and water management facilities. These probabilistic analyses looked at multiple possible scenarios covering a range of potential occurrences. Results from the study provide a variety of potential outcomes allowing risk-based decision making. The balance includes all major facilities that are expected to add water to the system, facilities that store water facilities that use water and facilities for water treatment.

Sources of water can be considered to the balance via three sources: process water, contact water and non-contact water. Contact water requires treatment before it can be released, but can be used in the process.

Process water can come from:

- Free water in the TSF
- Underdrain from the TSF
- Any water in the Mill process stream
- Natural moisture in the processed ore after it enters the process circuit

Contact water can come from:

- Runoff and underdrain from PAG OSA and Low Grade Ore Stockpile
- Direct precipitation and runoff accumulating in the active and inactive pits
- Crusher pad and coarse ore stockpile containment areas

Contact water can be used in the process as make up water, or be treated in a water treatment facility and discharged.

Non-contact water will also exist that does not require treatment, sources of non-contact water include:

- Groundwater from pit depressurization
- Surface water from Ledbetter Reservoir
- Municipal water
- Runoff from Topsoil Stockpiles
- Runoff from Overburden Storage Areas
- Groundwater from underground depressurization
- Runoff from Undisturbed Ground
- Run-on from Upgradient Areas
- Runoff from TSF Outer Perimeter
- Runoff from the Plant Site (process water is contained within the process)

The results of the site wide water balance analysis indicate that under normal and moderately extreme conditions, there would likely be adequate water storage in the TSF and delivered from the municipal source, surface water and pit depressurization wells to maintain the process functions.

18.6 OVERPASS ON HIGHWAY 601

A new concrete bridge type overpass will be constructed over Highway 601. The primary purpose of the overpass is to facilitate the haulage of construction fill material from the Mine to the TSF. The bridge will be designed for fully loaded haul trucks. In addition, the bridge will be used to carry the tailings delivery line across Highway 601 from the Process Plant.

18.7 ANCILLARY FACILITIES

In addition to process facilities, the project will construct many support facilities to support the mill and mine facilities. These facilities include:

- Administration Building
- Truck Shop and Warehouse Facility
- Mill Maintenance Building with Showers and Change Rooms
- Guard House and Security Gate and Truck Scale
- Gasoline and on-road diesel Fuel Station
- Off-road diesel Fuel Storage
- Hazardous Material Storage Building
- Mine Operations Line-Out Area
- Truck Wash
- Regional Geology Building (not part of this project, but available to Haile Gold Mine)
- Regional Laboratory (not part of this project, but available to Haile Gold Mine)

18.8 POWER SUPPLY

The main onsite electrical substation will be fed from the existing 69 kV transmission Grid. A new 5-mile-long 69kV power line will be constructed to connect the main HGM substation with the grid. The cost for the construction of the power line is included in the power rate schedule as shown in Table 18-1.

Table 18-1: Power Rate Schedule

Year	Power Rate ($/kW-H)
1	0.0477
2	0.0492
3	0.0519
4	0.0555
5	0.0577
6	0.0699

19 MARKET STUDIES AND CONTRACTS

Gold doré bars are typically delivered via armored transport from mine site to refinery.

HGM received a written quote for transportation and refining terms used in this report. This report used a treatment charge of $0.65 per ounce of net weight received. In addition, transportation costs per shipment are dependent on the weight and the rates per ounce are shown below:

- 200kg $0.475 per oz
- 400kg $0.275 per oz
- 600kg $0.205 per oz
- 800kg $0.175 per oz
- 1,000kg $0.155 per oz

Also, it was assumed that the doré bars were 95% pure with minimal or no deleterious elements. Percentage for the metal return are gold 99.95% and silver 99.00%.

There are several large gold refineries in North America that have a long history of service to the mining industry. The primary refineries that will likely be considered are as follows:

- Johnson Matthey – Salt Lake City, Utah or Brampton, Ontario
- Canadian Mint – Ottawa, Ontario
- Metalor – North Attleboro, Massachusetts

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Discussions with regulatory agency personnel over the past seven (7) years and the successful completion of an Environmental Impact Statement by the Corps of Engineers have resulted in obtaining all of the major permits necessary to construct, operate and close a new operation at Haile. In addition, regulatory discussions do not reveal any new legislation or regulations that are being contemplated that could have an adverse impact on mine construction schedules, operations or anticipated costs. The regulatory agencies have also acknowledged that they are encouraged that successful reclamation has been completed previously at the site (documented through successful partial bond release) and that successful reclamation can be performed again in the future.

The project is unique in that it occurs wholly on private land owned or controlled by HGM and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies. In addition, there is no potential for the federal government to impose a royalty by an amendment to the 1872 Mining Law.

Since the property has been mined in the past, a significant amount of background and environmental baseline data existed while additional data was collected through the Environmental Impact Statement (EIS) process. This data continues to be collected. Major permits/certifications obtained include 404 Dredge and Fill Permit, 401 Water Quality Certification, air quality permit, and NPDES Permits (wastewater discharge, wastewater treatment system construction, and storm water). The last remaining permit, the Mine Operating Permit, was modified to accommodate the project in November 2014 and made final (following the resolution of an appeal by the Sierra Club) in January of 2015.

The permits currently held by the Haile Mine may be kept, modified, terminated, or replaced during the life of the mine. Current permits are listed in Table 20-1.

Table 20-1: Mine Permits

Agency	Permit/Authorization Number	Description
US Army Corps of Engineers	Permit – SAC-1992-24122-4IA	Permit to fill wetlands and streams per the plans submitted on August 19, 2014
U.S. Army Corps of Engineers	Permit 2004-1G-157	Permit to fill a portion of the old North Fork Creek
Mine Safety and Health Administration (MSHA)	MSHA ID: 38-00600	Operate mine within MSHA standards
Federal Communications Commission	Call Sign: WQJB814	One base station frequency, six local frequencies
South Carolina Department of Health and Environmental Control (SCDHEC), Bureau of Water	401 Water Quality Certification	Water Quality certification to construction and operate a gold mine on Haile Gold Mine Creek, Camp Branch Creek, unnamed tributaries and adjacent wetlands.
South Carolina Department of Health and Environmental Control (SCDHEC), Division of Mining and Solid Waste Management	Mining/Operating Permit No. I-000601	Mine Operating Permit – Regulation of closure and reclamation.
SCDHEC, Division of Mining and Solid Waste Management	Mining/Operating Permit No. 214	Mine Operating Permit – Regulation of closure and reclamation of Hilltop Pits (permit cancelled April 12, 2011; Haile Gold Mine, Inc. performed reclamation of Hilltop II Pit, and total acreage has been incorporated into proposed modification of Permit No. I-000601).

Agency	Permit/Authorization Number	Description
SCDHEC, Division of Mining and Solid Waste Management	Mining/Operating Permit No. 440	Mine Operating Permit – Regulation of closure and reclamation of Parker Pit (permit cancelled on April 12, 11; Haile Gold Mine, Inc. stabilized Parker Pit, and the total acreage has been incorporated into proposed modification of Permit No. I-000601)
SCDHEC, Bureau of Drinking Water Protection	Public Water Permit No. 2930013	Former onsite water supply; closed on June 23, 2011
SCDHEC, Bureau of Solid and Hazardous Waste Management	Permit No. SCD987596806	Conditionally exempt small quantity generator
SCDHEC, Industrial Wastewater (IW) Permitting Section	National Pollutant Discharge Elimination System Discharge Permit No. SC0040479	Permit to discharge treated water from the mine operation / reclamation areas. Outfall 002 & 003
SCDHEC, Industrial Wastewater Permitting Section	Operating Permit #18,731-IW	Addition of pH adjustments to 002 outfall discharge for various units. Modified as needed during mine operations and closure.
SCDHEC, Industrial Wastewater Permitting Section	Operating Permit #18,873-IW	Permit to construct and operate semi- passive SRBR (best management practice cells)
SCDHEC, Industrial Wastewater Permitting Section	Operating Permit #19,830-IW	Permit to construct a new wastewater treatment facility including a treatment plant, collection ponds and pipelines
SCDHEC, Industrial Wastewater Permitting Section	ND Discharge Permit No. ND0085561	Permit to discharge sulfate-reducing bioreactor (SRBR) water to two percolation basins
SCDHEC, Industrial Wastewater Permitting Section	General Storm Water Permit for Non-Metal Mining Facilities, Permit No. SCG730398	Stormwater permit for Hilltop II Pit (permit cancelled on June 7, 2011; stormwater now managed pursuant to SCR004763)
SCDHEC, Industrial Wastewater Permitting Section	General Stormwater Permit for Non-Metal Mining Facilities, Permit No. SCG730217	Stormwater permit for Parker Pit (cancelled on June 7, 2011; stormwater now managed pursuant to SCR004763)
SCDHEC, Industrial Wastewater Permitting Section	WTR-Wastewater Construction Permit Permit No. 19852-IW	Permit to construct sewer lines
SCDHEC, Bureau of Water, Industrial, Agricultural, and Storm Water Permitting Division	Dams & Reservoirs Safety Permit 29-0007 (Issued October 7, 2013)	Dam Safety Permit – Significant Hazard (Construction). Stability during earthquake-induced ground motion was evaluated by SCDHEC prior to issuance of the TSF construction permit. Seismic stability was evaluated pursuant to the International Commission of Large Dam (ICOLD) seismic design and performance standards; www.icold- cigb.org
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) SCR100000	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – requires submittal of Storm Water Pollution Prevention Plan (SWPPP) and public notice prior to construction
SCDHEC, NPDES Program, Water Facilities Permitting Division	Stormwater discharges associated with industrial activity SCR000000, Permit No. SCR004763	Discharge of stormwater in connection with industrial activities, Industrial General Permit
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) (SCR10N593)	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – Temporary Trailers

Agency	Permit/Authorization Number	Description
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) (SCR10S309)	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – Plant Site Construction
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) (SCR10U660))	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – Construction City and Laydown Area
SCDHEC, Office of Environmental Quality, Bureau of Air Quality	Bureau of Air Quality, State Construction Permit No. 1460-0070-CA	Authorizes construction of the proposed facility and equipment specified in Haile Gold Mine, Inc.'s application for a Department of Army permit; a permit to operate also is required.
Lancaster County Council	Floodplain Development Permit June 27, 2013	Floodplain Administrator oversees and implements the provisions of the Flood Damage Prevention Ordinance.
Lancaster County Council	Ordinance 2013-1207	Rezoned the Haile property within the permit boundary to the M, Mining District designation.
SCDOT	186924	Encroachment Permit
SCDOT	186924 (Revised)	Encroachment Permit
SCDOT	187151	Encroachment Permit
Lancaster County	2015-CP-29-00497	Road Closure
Lancaster County	2015-CP-29-01463	Road Closure

All permits necessary to start construction have been received. Haile reports that all Construction and Building Permits needed to meet Operations Start-up/Schedule have been issued or are pending.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The project was estimated in Q4 2014 dollars. There were some 2,500 drawings generated for the process plant, infrastructure and tailing facility. They include detailed engineering level design of all disciplines including process, civil, structural steel, concrete, electrical, mechanical and instrumentation. Equipment quotations were received for most of the equipment. A significant amount of equipment has been purchased. Material take offs (MTO's) were used to estimate quantities of materials required to construct the facility. The estimate used labor rates gathered from means, and local contractors.

A significant percentage of process and mine equipment has been procured and is scheduled to arrive on site to meet the project schedule. Mine capital costs reflect the cost of mine mobile equipment required to complete the mine related tasks. Allowances are included for initial spare parts inventory and shop tools. Also included in mine capital are pre-production stripping costs. The initial capital estimate is considered to be +/-10% accuracy.

Table 21-1: Summary of Initial Capital Costs

Area	Description	($ Millions)
General Site	General Site, Site Water Diversion, Power Transmission, Main Substation, Ancillary Facilities	39.1
Mine	Mine Equipment and Preproduction, Mine Dewatering, Overburden Stockpiles	100.6
Process Facilities	Primary Crushing, Grinding, Flotation, Cyanide Leach, Carbon Handling and Refinery	97.8
Tailing	Tailing Thickening, Detox, Tailing Starter Dam Civil, Highway Overpass	62.3
Indirect Costs	Freight, Mobilization, EPCM, Vendor Commissioning and Spare Parts	51.1
Owners cost	Project development and construction consultants, owner's insurance, first fill of reagents, lubes and fuel, early staffing, construction management, strategic operating supplies, environmental monitoring, maintenance tools and other general Items	29.1
Contingency	Calculated based on each sub area of the project.	0
Escalation	Not included in this estimate	0
Total		380.0

Note that in February 2016 the estimate of initial capital for the project at $380M was increased from the initial estimate of $333M, as published in the previous NI 43-101 report issued 13 October 2015. The increase has principally been due to design enhancements to the project, including:

- Replacement loading equipment (excavator) to reduce the potential for ore dilution during mining
- The addition of a run-of-mine (ROM) pad to allow for more effective ore blending to the mill
- Installation of a crushed ore bin to minimize dust from the crushing circuit
- Installation of a larger flash flotation cell to improve metallurgical recoveries
- Enhancement of control systems for the process plant
- Upgrades to IT systems

A summary of the capital estimate update is provided in Table 21-2.

Table 21-2: Updated Initial Capital Costs Summary

Area of Discipline	February 2016 ($Millions)	October 2015 ($Millions)
Direct Costs	$222	$197
Owner's Costs	$28	$18
EPCM	$40	$30
Mining Capital Equipment	$53	$46
Mining Pre-Production Opex	$33	$25
Contingency	$0	$17
Total	$380	$333

21.2 SUSTAINING CAPITAL COST ESTIMATE

Sustaining capital costs were also evaluated for the project. Costs were estimated for future sustaining costs as shown in Table 21-3.

Table 21-3: Summary of Sustaining Costs (in $Millions)

Year	Mining	Surface Water Management	Overburden Storage Areas	Tailing and Process Water Management	Advanced Process Controls	Mine Area Piping Allowance	Future Overpass Highway 601 for Champion	Total
1	1.71	0.71	5.27					7.68
2	1.71	6.05	0.30	11.79	0.60	1.00		21.46
3	4.07	3.23	0.84					8.14
4	16.70	0.20	7.86	7.67				32.44
5	3.10			7.33		0.50		10.93
6	12.15							12.15
7	5.31	0.73		11.94				17.98
8	3.24			21.17		0.50		24.91
9	0.03						1.40	1.43
10	0.49							0.49
11	0.04	0.68						0.72
12	0.23							0.23
Total	48.77	11.61	14.27	59.90	0.60	2.00	1.40	138.54

Sustaining capital costs include indirect costs, but do not include contingency. All sustaining costs are in Q4, 2014 dollars.

21.3 OPERATING COST ESTIMATE

Operating costs were developed using reagent, grinding media, and power consumptions based on the process flow sheet. These costs are summarized in Table 21-4 below. Labor costs were developed based on a staffing plan and rate schedule.

Table 21-4: Process Plant Operating Cost

Year	1		2		3		4		5		6-11	
Tons Processed (Millions)	2.394		2.555		2.555		2.555		2.555		2.555	
Power Rate ($/kW-H)	0.0477		0.0492		0.0519		0.0555		0.0577		0.0699	
	Processing Cost By Type											
	$M	$/Ton	$M	$/Ton	$M	$/Ton	$M	$/Ton	$M	$/Ton	$M	$/Ton
Operating & Maintenance Labor	5.19	2.17	5.19	2.03	5.19	2.03	5.19	2.03	5.19	2.03	5.19	2.03
Power	4.63	1.93	4.77	1.87	5.04	1.97	5.39	2.11	5.60	2.19	6.78	2.65
Liners & Grinding Media	4.29	1.79	4.50	1.76	4.50	1.76	4.50	1.76	4.50	1.76	4.50	1.76
Reagents	5.07	2.12	5.41	2.12	5.41	2.12	5.41	2.12	5.41	2.12	5.41	2.12
Municipal Water	0.45	0.19	0.45	0.18	0.45	0.18	0.45	0.18	0.45	0.18	0.45	0.18
Maintenance	2.23	0.93	2.23	0.87	2.23	0.87	2.23	0.87	2.23	0.87	2.23	0.87
Water Treatment	0.57	0.24	0.57	0.22	0.57	0.22	0.57	0.22	0.57	0.22	0.57	0.22
Laboratory Services	0.28	0.12	0.28	0.11	0.28	0.11	0.28	0.11	0.28	0.11	0.28	0.11
Supplies & Services	0.98	0.41	0.98	0.38	0.98	0.38	0.98	0.38	0.98	0.38	0.98	0.38
Total	23.68	9.89	24.38	9.54	24.64	9.64	24.99	9.78	25.20	9.86	26.39	10.33
	Processing Cost By Area											
Primary Crushing & Conveying	0.87	0.36	0.87	0.34	0.88	0.34	0.88	0.35	0.89	0.35	0.90	0.35
Grinding & Classification	8.57	3.58	8.82	3.45	8.94	3.50	9.11	3.56	9.21	3.60	9.77	3.82
Flotation and Concentrate & Flotation Tailing Treatment	8.15	3.41	8.52	3.34	8.61	3.37	8.73	3.42	8.80	3.44	9.19	3.60
Elutions and Refinery	1.33	0.56	1.35	0.53	1.36	0.53	1.38	0.54	1.40	0.55	1.47	0.57
Tailing Systems, TSF & Reclaim, and Water Management	2.42	1.01	2.48	0.97	2.50	0.98	2.54	0.99	2.56	1.00	2.68	1.05
Laboratory	0.28	0.12	0.28	0.11	0.28	0.11	0.28	0.11	0.28	0.11	0.28	0.11
Water Treatment and Reagents	1.14	0.48	1.14	0.45	1.14	0.45	1.15	0.45	1.15	0.45	1.15	0.45
Ancillary Services	0.92	0.38	0.92	0.36	0.92	0.36	0.93	0.36	0.93	0.37	0.95	0.37
Total	23.68	9.89	24.38	9.54	24.64	9.64	24.99	9.78	25.20	9.86	26.39	10.32

21.4 MINING COST SUMMARY

Mine capital costs reflect the cost of mine mobile equipment required to complete the mine related tasks that were listed on the mine operating cost section (see Table 21-5).

Costs are reported quarterly for preproduction and years 1 and 2. They are reported annually after that time period.

Mine capital costs do not include truck shops or mine offices. Those costs have been developed by other team members.

Mine major equipment capital costs are based on vendor quotes as of January 2014. The Blanchard and hydraulic shovel line items on Table 21-5 are payments for equipment that has already been delivered. The mine capital cost table does not include contingency as the equipment is now on site. A single contingency has been applied to the project within the financial analysis section. Allowances are included for engineering/geology equipment and shop tools.

Table 21-7 illustrates both mine capital and operating costs and has moved the mine preproduction stripping cost to a separate category so that it can be capitalized. The operating costs on the table included concurrent reclamation costs.

Table 21-5: CAPEX 1 – Mining Capital Cost Summary

Mine Equipment Capital Costs	Unit Cost	Life	PP Q1		PP Q2		PP Q3		PP Q4		PP Q5		PP Q6		Yr1 Q1		Yr1 Q2		Yr1 Q3		Yr1 Q4		Yr2 Q1		Yr2 Q2	
	($1000)	Hours	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)
MINE MAJOR EQUIPMENT:																										
Blanchard Payments	28,997		1	31,950																						
Blast Hole Drill (6½")	910	60,000			3	2,718																				
Blast Hole Drill (4½")	575	60,000											2	1,200												
Hitachi 14.4 cu m Hyd. Shovel	1,400	80,000	1	2,593																						
Hitachi Bucket wear Package	20																									
Cat 6020 Excavator	2,225	70,000									1	2,225														
Cat 777G Haul Truck	1,819	55,000																								
Cat D9T Track Dozer	1,081	30,000																								
Cat D10T Track Dozer	1,453	30,000											1	599												
Cat 834H Wheel Dozer	1,165	30,000																								
Cat 14M Motor Grader	557	55,000													1	557										
Cat 773 Water Truck	1,148	30,000																					1	1,148		
Cat 336D Excavator	291	20,000																								
Bomag BW-213DH-40 Compactor	126	20,000					1	126																		
Subtotal Major Equipment				34,543		2,718		126		-		2,225		1,799		557		-		-		-		1,148		-
MINE SUPPORT EQUIPMENT:		Years																								
Fuel/Lube Truck (4,000 gal)	887	6		65				7																		
Flatbed Truck (8 - 10 ton)	110	6	1	55					1	8	1	27														
Crane Truck (8 - 10 ton)	65	6	1	92				40			1	90														
Rough Terrain Crane (40 ton)	366	12									1	3														
Mechanics Truck	110	6	1	157							1	50														
Welding Truck	168	6																								
Mechanics Truck and Shop Equip.	448	6		14					1	3	1	448														
Tractor & Lowboy (75 T)	1,152	12															1	1,152								
Shop Forklift (Hyster H100XM)	40	6	1		1	45		70																		
RT Forklift (Sellick SD-100)	120	6																								
Fire Suppresion systems mobile					1	110		80	1	76																
Man Van	38	6	1	30																						
Pickup Truck (4x4)	38	4	15	526					2	59	1	97														
Light Plants	20	4	1	21	1	21	2	42																		
Mine Communications Network	50	12																								
Mine Radios	1	12	1	7	8	8	1	51	1	4																
Mine Dewatering	0	12	1	1968	1	18	1	516	1	655	1	1,067														
Spare Shovel Bucket	365																						1	365		
Lime Silo & Dispensing System	200	12																					1	200		
Temporary Maintenance Shop	200		1	9	1	346	1	196	1	16	1	250														
Temporary Fueling Facility	150				1	35			1	6																
Information Systems/Software							1	134	1	217	1	1,915														
Subtotal Mine Support Equipment				2,944		583		1,136		1,044		3,947		-		-		1,152		-		-		565		-
Engineering/Geology Equipment	150	6											1	36												
Operator Training Program	396		1	74	1	57	1	54	1	48																
Shop Tools					1	51	1	112	1	89	1	500														
Contingency (0%)																										
TOTAL EQUIPMENT/FACILITIES CAPITAL				37,561		3,409		1,428		1,181		6,672		1,835		557		1,152		-		-		1,713		-
						40,970		42,398		43,579		50,251		52,086		52,643		53,795		53,795		53,795		55,508		55,508

Table 21-6: CAPEX 2 – Mining Capital Cost Summary

Mine Equipment Capital Costs	Unit Cost	Life	Yr2 Q3		Yr2 Q4		Year 3		Year 4		Year 5		Year 6		Year 7		Year 8		Year 9		Year 10		Year 11		Year 12		Project
	($1000)	Hours	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	Total
MINE MAJOR EQUIPMENT:																											
Blanchard Payments	28,997																										31,950
Blast Hole Drill (6½")	910	60,000									2	1,820															4,538
Blast Hole Drill (4½")	800	60,000																									1,200
Hitachi 14.4 cu m Hyd. Shovel	1,400	80,000																									2,593
Hitachi Bucket wear Package	20																										-
Cat 6020 Excavator	2,225	70,000							1	2,268					2	4,536											9,029
Cat 777G Haul Truck	1,819	55,000					2	3,638	5	9,095			5	9,095													21,828
Cat D9T Track Dozer	1,081	30,000								-							2	2,162									2,162
Cat D10T Track Dozer	1,453	30,000							1	1,453																	2,052
Cat 834H Wheel Dozer	1,165	30,000									1	1,165															1,165
Cat 14M Motor Grader	557	55,000												-	1	557	1	557									1,671
Cat 773 Water Truck	1,148	30,000																									1,148
Cat 336D Excavator	291	20,000											1	291													291
Bomag BW-213DH-40 Compactor	143	20,000																									126
																											-
Subtotal Major Equipment				-		-		3,638		12,816		2,985		9,386		5,093		2,719		-		-		-		-	79,753
MINE SUPPORT EQUIPMENT:		Years																									
																											-
Fuel/Lube Truck (4,000 gal)	887	6											1	887													959
Flatbed Truck (8 - 10 ton)	110	6											2	220													310
Crane Truck (8 - 10 ton)	65	6											1	65													287
Rough Terrain Crane (40 ton)	366	12																									3
Mechanics Truck	110	6											2	220													427
Welding Truck	168	6											1	168													168
Mechanics Truck and Shop Equip.	448	6																									465
Tractor & Lowboy (75 T)	1,152	12																									1,152
Shop Forklift (Hyster H100XM)	40	6											1	40													155
RT Forklift (Sellick SD-100)	120	6											1	120													120
Fire Suppresion systems mobile																											266
Man Van	38	6							4	152							4	152									334
Pickup Truck (4x4)	38	4							8	304							8	304							6	228	1,518
Light Plants	20	4					1	20																			104
Mine Communications Network	50	12																									-
Mine Radios	1	12							30	30							30	30									130
Mine Dewatering	0	12						408		3,399		117		897		215		31		31		486		35			9,843
Spare Shovel Bucket	365																										365
Lime Silo & Dispensing System	200	12																									200
Temporary Maintenance Shop	200																										817
Temporary Fueling Facility	150																										41
Information Systems/Software																											2,266
Subtotal Mine Support Equipment				-		-		428		3,885		117		2,617		215		517		31		486		35		228	19,930
Engineering/Geology Equipment	150	6											1	150													186
Operator Training Program	396																										233
Shop Tools																											752
Contingency (0%)																											
TOTAL EQUIPMENT/FACILITIES CAPITAL				-		-		4,066		16,701		3,102		12,153		5,308		3,236		31		486		35		228	100,854

Table 21-7: Summary of Mine Capital and Operating Costs

SUMMARY OF MINE CAPITAL AND OPERATING COSTS
($US x 1000)

Year	Mine Equipment Initial Capital Cost	Mine Equipment Sustaining Capital Cost	Mine Preprod. Development	(1) Total Mine Capital	(2) Operating Cost	TOTAL COST
PP Q1	37,561		3,361	37,561	3,361	40,922
PP Q2	3,409		2,387	3,409	2,387	5,796
PP Q3	1,428		4,970	1,428	4,970	6,398
PP Q4	1,181		5,539	1,181	5,539	6,720
PP Q5	6,672		7,777	6,672	7,777	14,449
PP Q6	1,835		7,023	1,835	7,023	8,858
Yr1 Q1		557		557	7,754	8,311
Yr1 Q2		1,152		1,152	7,190	8,342
Yr1 Q3				0	7,183	7,183
Yr1 Q4				0	7,157	7,157
Yr2 Q1		1,713		1,713	7,089	8,802
Yr2 Q2				0	7,455	7,455
Yr2 Q3				0	7,500	7,500
Yr2 Q4				0	8,627	8,627
3		4,066		4,066	35,218	39,284
4		16,701		16,701	38,199	54,900
5		3,102		3,102	44,905	48,007
6		12,153		12,153	50,313	62,466
7		5,308		5,308	50,496	55,804
8		3,236		3,236	44,350	47,586
9		31		31	20,383	20,414
10		486		486	18,579	19,065
11		35		35	17,927	17,962
12		228		228	10,601	10,829
13				0	6,523	6,523
14				0	1,938	1,938
TOTAL	52,086	48,768	31,057	100,854	430,446	531,300

(1) Mine preproduction development cost carried as an operating cost in this table. If financial analysis requires this cost to be a capital number, subtract from "Operating Cost" column and add to "Total Mine Capital" column.
(2) Includes concurrent reclamation costs

21.5 MINE ASSAY COST

The mine assay cost was calculated using a unit rate of $7.46 per sample. A summary of the annual cost is presented in Table 21-8.

Table 21-8: Mine Assay Cost

Year	Annual Cost ($000)
1	$283
2	$287
3	$515
4	$212
5	$711
6	$716
7	$809
8	$637
9	$185
10	$152
11	$147
12	$30
Total	$4,684

21.6 G&A COSTS

HGM provided an estimate for the G&A cost for the project of $9.03 Million per year. These costs include labor, property costs, utilities, external assays, legal fees, outside services, insurance and other general costs. Table 21-9 shows a summary of these costs.

Table 21-9: General and Administrative Costs

Item	Cost ($000)
Salaries and Wages	$3,129
Property and Other Insurance	$1,640
Property Taxes	$1,300
Outside Services	$1,433
Security	$647
Operating Expenses	$462
Computer and Communications	$419
Total G&A	$9,030

22 ECONOMIC ANALYSIS

The Haile Gold Project economics were done using a discounted cash flow model. The financial indicators examined for the project included the Net Present Value (NPV), Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections were estimated over the life of the mine based on capital expenditures, production costs, transportation and refining charges and sales revenue. The life of the mine is 13 years.

The economic analysis of the Haile Gold Project at a gold price of $1250/oz shows an after tax Net Present Value (NPV) of $290.8 million at a discount rate of 5%. This results in an IRR of 16.7% and a payback period of 4.4 years to recapture the initial capital investment.

All project costs spent through 2014, $30.8 million, are considered “sunk” and are included in the project costs.

A sensitivity analysis was conducted for the project. The results are included in Table 22-1. The project IRR is most sensitive to variation in gold grade and gold price, followed by operating costs and capital costs.

Table 22-1: Sensitivity Analysis (After Tax)

		NPV @ 0%	NPV @ 5%	IRR	Payback
Base Case		$561,526	$290,784	16.7%	4.4
Gold Price	+20%	$903,459	$536,549	25.2%	3.0
	-20%	$205,724	$35,002	6.5%	8.3
Operating Cost	+20%	$419,846	$191,380	13.0%	5.4
	-20%	$696,742	$385,551	19.9%	3.8
Gold Recovery	+5%	$665,242	$365,121	19.3%	4.4
	-5%	$455,844	$214,986	13.8%	5.2
Gold Grade	+20%	$928,628	$554,325	25.8%	2.9
	-20%	$182,673	$18,708	5.8%	8.5
Silver Price	+20%	$568,479	$295,772	16.8%	4.4
	-20%	$554,573	$285,797	16.5%	4.5
Capital Cost	+20%	$493,775	$223,530	12.8%	5.5
	-20%	$627,516	$356,652	21.8%	3.4
Silver Grade	+100%	$593,633	$313,859	17.5%	4.2
	-100%	$529,316	$267,628	15.8%	4.6

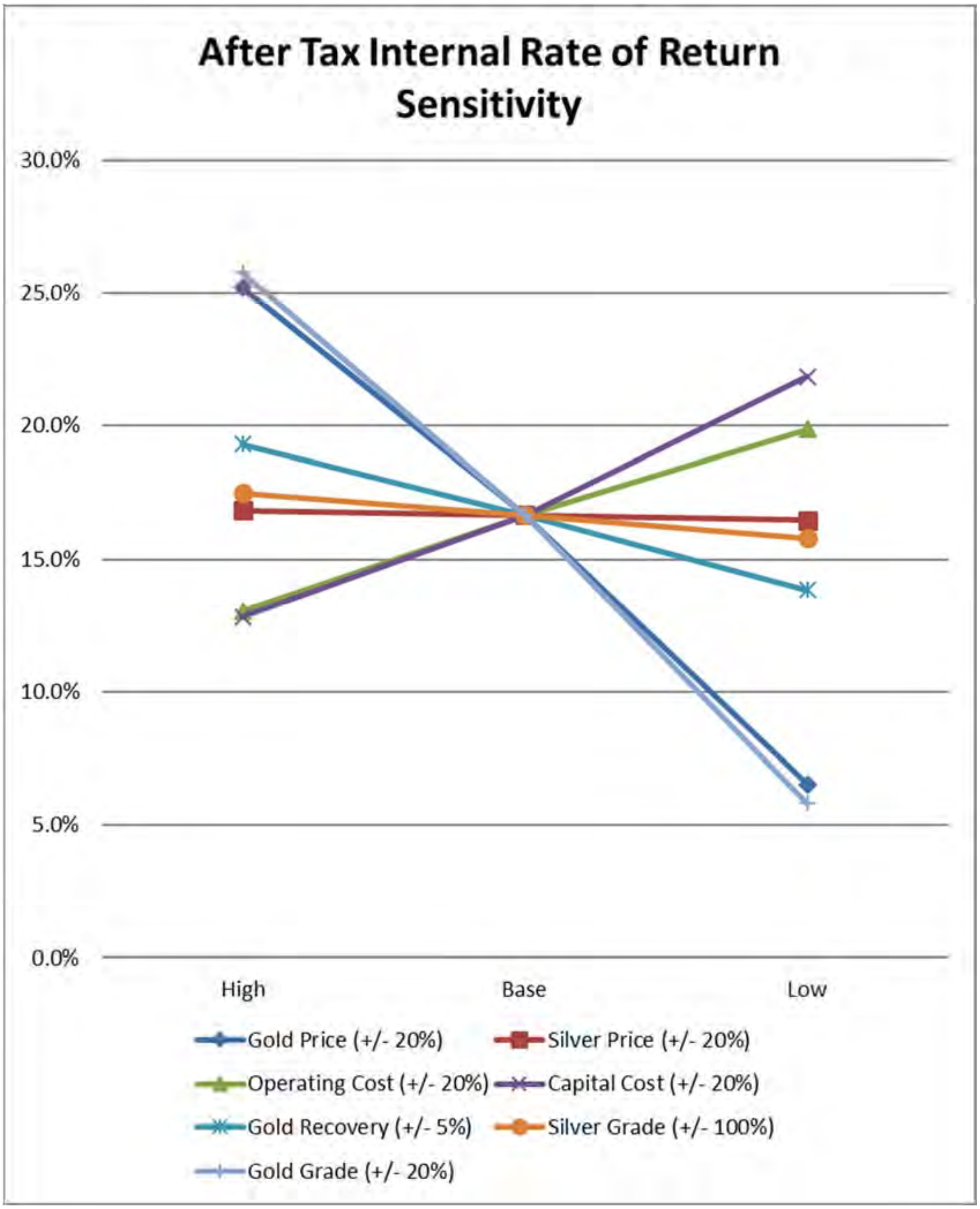


Figure 22-1: Sensitivity Analysis

Table 22-2: Cash Flow Model (Base Case)

Base Case	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mining Operations																	
Open Pit Ore																	
Beginning Inventory (kt)	28,780	28,780	28,780	28,626	26,386	23,831	21,276	18,721	16,166	13,611	11,056	8,501	5,946	3,391	836	-	-
Mined (kt)	28,780	-	154	2,240	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	836	-	-
Ending Inventory (kt)	-	28,780	28,626	26,386	23,831	21,276	18,721	16,166	13,611	11,056	8,501	5,946	3,391	836	-	-	-
Gold Grade (oz/t)	0.066	-	0.062	0.086	0.062	0.075	0.071	0.061	0.062	0.068	0.063	0.074	0.073	0.051	0.023	-	-
Silver Grade (oz/t)	0.099	-	0.093	0.128	0.093	0.113	0.107	0.092	0.093	0.102	0.095	0.111	0.110	0.077	0.035	-	-
Contained Gold (kozs)	1,907	-	10	192	159	192	181	156	158	174	161	189	187	130	19	-	-
Contained Silver (kozs)	2,861	-	14	288	239	287	272	234	238	261	241	284	280	195	29	-	-
Low Grade Stockpile																	
Beginning Inventory (kt)	4,850	4,850	4,850	4,751	4,428	3,852	3,764	3,102	1,736	1,527	-	-	-	-	-	-	-
Mined (kt)	4,850	-	99	323	576	88	662	1,366	209	1,527	-	-	-	-	-	-	-
Ending Inventory (kt)	-	4,850	4,751	4,428	3,852	3,764	3,102	1,736	1,527	-	-	-	-	-	-	-	-
Gold Grade (oz/t)	0.020	-	0.018	0.018	0.019	0.015	0.018	0.021	0.016	0.021	-	-	-	-	-	-	-
Silver Grade (oz/t)	0.030	-	0.028	0.027	0.029	0.023	0.027	0.032	0.024	0.032	-	-	-	-	-	-	-
Contained Gold (kozs)	96	-	2	6	11	1	12	29	3	32	-	-	-	-	-	-	-
Contained Silver (kozs)	144	-	3	9	17	2	18	43	5	48	-	-	-	-	-	-	-
Combined Ore																	
Beginning Inventory (kt)	33,630	33,630	33,630	33,377	30,814	27,683	25,040	21,823	17,902	15,138	11,056	8,501	5,946	3,391	836	-	-
Mined (kt)	33,630	-	253	2,563	3,131	2,643	3,217	3,921	2,764	4,082	2,555	2,555	2,555	2,555	836	-	-
Ending Inventory (kt)	-	33,630	33,377	30,814	27,683	25,040	21,823	17,902	15,138	11,056	8,501	5,946	3,391	836	-	-	-
Gold Grade (oz/t)	0.060	-	0.045	0.077	0.054	0.073	0.060	0.047	0.059	0.050	0.063	0.074	0.073	0.051	0.023	-	-
Silver Grade (oz/t)	0.089	-	0.068	0.116	0.082	0.110	0.090	0.071	0.088	0.076	0.095	0.111	0.110	0.077	0.035	-	-
Contained Gold (kozs)	2,004	-	11	198	170	193	193	185	162	206	161	189	187	130	19	-	-
Contained Silver (kozs)	3,005	-	17	296	255	289	290	277	243	309	241	284	280	195	29	-	-
Overburden																	
Beginning Inventory(kt)	241,340	241,340	241,340	225,723	206,186	187,217	167,660	136,877	105,798	73,562	43,644	17,732	11,169	5,960	1,128	-	-
Mined (kt)	241,340	-	15,617	19,537	18,969	19,557	30,783	31,079	32,236	29,918	25,912	6,563	5,209	4,832	1,128	-	-
Ending Inventory (kt)	-	241,340	225,723	206,186	187,217	167,660	136,877	105,798	73,562	43,644	17,732	11,169	5,960	1,128	-	-	-
Total Open Pit Material Mined (kt)	274,970	-	15,870	22,100	22,100	22,200	34,000	35,000	35,000	34,000	28,467	9,118	7,764	7,387	1,964	-	-
Rehandle	4,850	-	-	-	-	-	-	-	-	-	-	-	-	-	1,720	2,555	575
Process Plant Operations																	
Beginning Ore Inventory (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined Ore to Concentrator (kt)	33,630	-	-	2,394	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	576
Mined Ore - Processed (kt)	33,630	-	-	2,394	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	2,555	576
Ending Ore Inventory		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (oz/t)	0.060	-	-	0.084	0.062	0.075	0.071	0.061	0.062	0.068	0.063	0.074	0.073	0.051	0.021	0.020	0.020
Silver Grade (oz/t)	0.089	-	-	0.126	0.093	0.113	0.107	0.092	0.093	0.102	0.095	0.111	0.110	0.077	0.031	0.030	0.028

Base Case	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Contained Gold (kozs)	2,004	-	-	201	159	192	181	156	158	174	161	189	187	130	54	51	12
Contained Silver (kozs)	3,005	-	-	302	239	287	272	234	238	261	241	284	280	195	80	77	16
Recovery Gold (%)	83.73%	0.00%	0.00%	85.44%	83.73%	84.81%	84.50%	83.60%	83.70%	84.25%	83.80%	84.73%	84.66%	82.48%	75.67%	75.24%	75.24%
Recovery Silver (%)	70.00%	0.00%	0.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%	70.00%
Recovered Gold (kozs)	1,678	-	-	172	133	163	153	130	133	146	135	160	158	107	41	38	9
Recovered Silver (kozs)	2,104	-	-	211	167	201	190	164	166	182	169	199	196	137	56	54	11
Payable Metals																	
Payable Gold (kozs)	1,678			172	133	162	153	130	133	146	135	160	158	107	41	38	9
Payable Silver (kozs)	2,083			209	165	199	189	162	165	181	167	197	194	135	56	53	11
Income Statement ($000)																	
Metal Prices																	
Gold ($/oz)	$1,250.00			$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$1,250.00
Silver ($/oz)	$20.00			$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00	$20.00
Revenues																	
Gold Revenue ($ 000)	$2,096,916			$214,860	$166,384	$203,045	$191,511	$162,795	$165,659	$182,878	$168,524	$200,159	$197,275	$134,281	$50,683	$48,035	$10,829
Silver Revenue ($ 000)	$41,656			$4,185	$3,307	$3,984	$3,771	$3,240	$3,293	$3,612	$3,346	$3,931	$3,878	$2,709	$1,115	$1,062	$223
Total Revenues	$2,138,572	$0	$0	$219,044	$169,691	$207,029	$195,282	$166,035	$168,952	$186,490	$171,870	$204,090	$201,152	$136,990	$51,798	$49,097	$11,052
Operating Cost																	
Mining - Open Pit	$376,093			$27,696	$29,306	$33,371	$36,142	$42,792	$48,263	$49,252	$41,957	$18,474	$16,885	$15,964	$8,897	$5,605	$1,489
Process Plant	$339,943			$23,680	$24,379	$24,641	$24,990	$25,204	$26,388	$26,388	$26,388	$26,388	$26,388	$26,388	$26,388	$26,388	$5,949
General Administration	$119,639			$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$9,030	$2,252
Treatment & Refining Charges																	
Dore'																	
Treatment Charges	$2,581			$262	$205	$248	$235	$201	$204	$224	$207	$245	$241	$167	$66	$63	$14
Gold Refining Charges	$1,049			$107	$83	$102	$96	$81	$83	$91	$84	$100	$99	$67	$25	$24	$5
Silver Refining Charges	$421			$42	$33	$40	$38	$33	$33	$36	$34	$40	$39	$27	$11	$11	$2
Transportation	$2,075			$210	$165	$200	$189	$161	$164	$180	$167	$197	$194	$134	$53	$51	$11
Total Operating Cost	$841,801	$0	$0	$61,028	$63,201	$67,631	$70,719	$77,502	$84,165	$85,202	$77,866	$54,472	$52,876	$51,776	$44,470	$41,171	$9,722
Mine Development	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine Development G&A/Mitigation	$50,247	$28,747	$10,780	$1,480	$1,480	$1,480	$1,480	$480	$480	$480	$480	$480	$480	$480	$480	$480	$480
Salvage Value	-$4,575				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation & Closure	$74,893	$0	$30,200	$5,237	$5,226	$8,445	$10,868	$3,461	$3,484	$6,483	$4,970	$1,940	$2,412	$1,340	$1,524	$2,142	$454
Total Production Cost	$962,367	$28,747	$40,980	$67,745	$69,907	$77,555	$83,067	$81,443	$88,129	$92,166	$83,316	$56,892	$55,767	$53,596	$46,474	$43,792	$10,656
Operating Income	$1,176,205	$(28,747)	$(40,980)	$151,299	$99,784	$129,473	$112,215	$84,593	$80,823	$94,324	$88,554	$147,197	$145,385	$83,394	$5,324	$5,305	$396
Initial Capital Depreciation	$380,000			$54,302	$93,062	$66,462	$47,462	$33,934	$33,896	$33,934	$16,948	$0	$0	$0	$0	$0	$0
Mine Development	$0			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital Depreciation	$138,541			$1,098	$4,948	$7,762	$11,340	$14,294	$13,704	$14,835	$17,334	$16,519	$12,341	$8,694	$5,872	$4,739	$3,326
Total Depreciation	$518,541	$0	$0	$55,400	$98,010	$74,224	$58,802	$48,228	$47,600	$48,769	$34,282	$16,519	$12,341	$8,694	$5,872	$4,739	$3,326
Net Income After Depreciation	$657,664	-$28,747	-$40,980	$95,899	$1,774	$55,250	$53,413	$36,364	$33,224	$45,555	$54,272	$130,678	$133,044	$74,699	-$549	$566	-$2,930
Income Taxes	$126,938		$0	$1,559	$98	$5,181	$9,545	$5,541	$4,993	$8,610	$8,724	$27,852	$36,087	$18,750	$0	$0	$0

Base Case	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Net Income After Taxes	$530,726	-$28,747	-$40,980	$94,340	$1,676	$50,069	$43,868	$30,824	$28,231	$36,945	$45,548	$102,826	$96,957	$55,949	-$549	$566	-$2,930
	19.3%																
Cash Flow																	
Operating Income	$1,176,205	-$28,747	-$40,980	$151,299	$99,784	$129,473	$112,215	$84,593	$80,823	$94,324	$88,554	$147,197	$145,385	$83,394	$5,324	$5,305	$396
Add Back Cost Depletion	$0			$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Working Capital																	
Account Receivable (30 days)	$0	$0	$0	-$18,004	$4,056	-$3,069	$965	$2,404	-$240	-$1,441	$1,202	-$2,648	$241	$5,274	$7,002	$222	$3,127
Accounts Payable (30 days)	$0	$0	$0	$5,016	$179	$364	$254	$557	$548	$85	-$603	-$1,923	-$131	-$90	-$601	-$271	-$2,585
Inventory - Parts, Supplies	$0	$0	-$2,400	-$6,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$8,400
Total Working Capital	$0	$0	-$2,400	-$18,988	$4,235	-$2,705	$1,219	$2,961	$308	-$1,356	$599	-$4,571	$110	$5,183	$6,402	-$49	$8,942
Capital Expenditures																	
Initial Capital																	
Mine	$100,611	$54,718	$45,893	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$250,267	$63,465	$183,750	$3,052	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owners Cost	$29,122	$7,785	$13,574	$7,763	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Land Acquisition	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine Development	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sunk Cost	-$30,800	-$30,800															
Sustaining Capital																	
Mine	$48,768	$0	$0	$1,709	$1,713	$4,066	$16,701	$3,102	$12,153	$5,308	$3,236	$31	$486	$35	$228	$0	$0
Process Plant	$89,773	$0	$0	$5,974	$19,745	$4,071	$15,735	$7,825	$0	$12,673	$21,669	$1,400	$0	$681	$0	$0	$0
Total Capital Expenditures	$487,741	$95,168	$243,217	$18,498	$21,458	$8,137	$32,436	$10,927	$12,153	$17,981	$24,905	$1,431	$486	$716	$228	$0	$0
Cash Flow Before Taxes	$688,464	-$123,915	-$286,597	$113,813	$82,561	$118,632	$80,999	$76,627	$68,978	$74,987	$64,248	$141,195	$145,009	$87,861	$11,497	$5,255	$9,338
Cumulative Cash Flow Before Taxes		-$123,915	-$410,512	-$296,699	-$214,138	-$95,506	-$14,508	$62,119	$131,098	$206,084	$270,332	$411,528	$556,537	$644,398	$655,895	$661,151	$670,489
				1.0	1.0	1.0	1.0	0.2	-	-	-	-	-	-	-	-	-
Taxes																	
Income Taxes	$126,938	$0	$0	$1,559	$98	$5,181	$9,545	$5,541	$4,993	$8,610	$8,724	$27,852	$36,087	$18,750	$0	$0	$0
Cash Flow After Taxes	$561,526	-$123,915	-$286,597	$112,254	$82,463	$113,451	$71,454	$71,086	$63,985	$66,377	$55,524	$113,343	$108,923	$69,111	$11,497	$5,255	$9,338
Cumulative Cash Flow After Taxes		-$123,915	-$410,512	-$298,258	-$215,795	-$102,344	-$30,890	$40,196	$104,181	$170,558	$226,083	$339,426	$448,349	$517,460	$528,957	$534,212	$543,551
Economic Indicators before Taxes																	
NPV @ 0%	0%	$688,464															
NPV @ 5%	5%	$371,796															
NPV @ 10%	10%	$183,143															
IRR		18.7%															
Payback	Years	4.2															
		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
	-$272,949	-$123,915	-$272,949	$103,232	$71,319	$97,598	$63,465	$57,180	$49,022	$50,754	$41,415	$86,682	$84,784	$48,924	$6,097	$2,654	$4,492
Benefit Cost Ratio @ 5%	2.4																
Economic Indicators after Taxes																	
NPV @ 0%	0%	$561,526															
NPV @ 5%	5%	$290,784															
NPV @ 10%	10%	$129,549															
IRR		16.7%															
Payback	Years	4.4															

22.1 TAXES

Taxable income for income tax purposes is defined as metal revenues minus operating expenses, royalty, property and severance taxes, reclamation and closure expense, depreciation, tax loss carry forwards and percentage depletion. Income tax rates for state and federal are as follows:

- State rate 5.0%
- Federal rate 35.0%
- Combined tax rate 38.3%

The combined statutory tax rate was calculated as follows (use decimal format to calculate): state rate (5.0%) + federal rate 35.0 %*(1-state rate 5.0%).

Income taxes were calculated on the taxable income described above using the statutory federal and state rates.

22.2 ROYALTIES

There are no royalties for this project.

23 ADJACENT PROPERTIES

The Carolina Slate Belt (CSB) is host to many mines and mining districts. Most of these deposits were discovered in the 1800's. Nearby deposits include the Ridgeway, Brewer, and Barite Hill Mines in South Carolina and numerous mines of the Gold Hill and Cid Mining Districts in North Carolina. Each of these deposits have similar geologic and mineralization features to Haile, several are polymetallic.

Four of these gold mines were put into production in the 1980's. These mines in order of size of deposit and contained gold were; Ridgeway, Haile, Brewer, and Barite Hill. It is apparent that Haile now exceeds each of the other deposits in contained gold and will possibly have the greatest gold production. M3 has not independently verified the following information, and the information is not necessarily indicative of the mineralization on the Haile project.

23.1 RIDGEWAY MINE

The Ridgeway Mine is located approximately 5 miles (8 km) east of Ridgeway, South Carolina and 25 miles north of Columbia, South Carolina in the Carolina Slate Belt. Kennecott Ridgeway Mining Company (Kennecott) mined low grade oxide and sulfide ore from siliceous deposits with the ultimate production of approximately 1.5 million ounces (46,655 kg) of gold produced from 1988 to 1999. The mine was composed of two open pits. The mine and mill had a production capacity of 15,000 tons (13,608 tonnes) per day.

Ore was milled to minus 200 mesh then fed into a modified carbon-in-leach circuit. Carbon was stripped of gold; the gold was electroplated onto steel wool cathodes then transferred to electro-refining cells where gold was plated onto stainless steel plates.

As stated earlier in this report the Ridgeway deposit has strong similarities to Haile. The saprolite, volcanic and metasedimentary rocks are quartz-sericite-pyrite altered in mineralized areas. Post mineral mafic and felsic dikes cross-cut the deposit, and are often accompanied by shearing and/or faulting. Gold grade is related to lithology, cleavage development, pyrite grain size and abundance, and silica content. Molybdenite is also associated with the mineralization.

23.2 BREWER MINE

The mine is located 10 miles (16.1 km) northeast of Haile on a small north-south ridgeline that divides Little Fork Creek and Lynches River. The Brewer Gold Mine is reported to be one of the oldest gold mines in the U.S, with production rumored from Native American placer production dating to the 1500's. The area was mined for iron prior to the Revolutionary War, before the first documented gold discovery in 1828 by Burrell Brewer. Like Haile and other mines in the CSB, the mine produced gold intermittently, first as a placer, then as a surface and underground mine, and finally as a low-grade cyanide treated heap leach operation in the 1980's.

The most recent production was from 1987 to 1995 by Westmont Mining/Costain Ltd Group. In 1990, a failure of an overflow pond released water containing sodium-cyanide solution, copper, mercury, chromium, cobalt, nickel, and selenium, killing fish along 49 miles of the Lynches River. Unlike the closure of the previously discussed Haile and Ridgeway mines, Brewer suffered from poor planning and closure, and became a Superfund site in 1999. US EPA now controls the property.

In 1987, Westmont Mining estimated a non NI 43-101 compliant reserve for Brewer of 5.1 million tons (4.6 Mt) grading 0.042 opt (1.4 g/t) gold. Ore was mined using conventional truck and loader open pit methods and ore was processed using cyanide leaching.

Lithologies at the mine include schist, volcanics, and granite intrusives which are commonly overlain by 40-60 feet of saprolite and sand. The mineralization is reported to be associated with quartz-sericite-pyrite altered schist. Gold ore was produced from a breccia body of hydrothermal origin and a related smaller body of fault-controlled ore. Pyrite content is generally 2-5%, unevenly distributed as aggregates and individual crystals in quartz veins. Gold grades were reported in the 0.045 to 0.13 opt range with associated silver, copper, tin, and bismuth.

23.3 BARITE HILL MINE

The Barite Hill Mine is located about 2.5 miles (4 km) southwest of the town of McCormick and about 0.75 miles northwest of the intersection of Highways S-33-44 and S-33-30. It is within the Lincolnton-McCormick Mining District, which includes other small mines and prospects for gold, silver, copper, zinc, lead, kyanite, and manganese.

The Barite Hill deposit was mined from 1989 to 1994 by Nevada Goldfields, Inc. The mine produced approximately 59,000 oz of gold (1.8 million grams) and 109,000 oz (3.4 million grams) of silver, mainly from oxidized ore in the 20 acre (8 ha) Main Pit and the 3.93 acre (1.6 ha) Rainsford Pit. The mine used conventional open pit mining methods and an on/off heap leach process.

In June 1999, Nevada Goldfields Inc. filed for Chapter 7 bankruptcy, and the following month abandoned the property. The property came under control of the South Carolina Department of Health and Environmental Control and the site became part of the Superfund program. Reclamation and closure work began in October 2007. The site is now under the control of the US EPA.

The Barite Hill deposit is hosted by sericite altered felsic metavolcanic and metasedimentary rock of the Persimmon Fork Formation. The deposit occurs stratigraphically below an overturned contact between upper and lower pyroclastic units. Mafic to intermediate post-mineralization dikes and sills cross-cut NE trending mineralized zones. Multiple Main Pit ore zones are associated with lenses of siliceous barite rock and pyrite-quartz altered breccias, some of which are offset by normal faulting. Rainsford Pit ore zones are associated with silicified rock and chert. The Barite Hill deposit is interpreted to be the result of a Kuroko-type submarine volcanogenic base-metal sulfide system followed by epithermal precious metal deposition.

24 OTHER RELEVANT DATA AND INFORMATION

This section of the report was prepared for OceanaGold (Oceana) to assess the underground mining potential at the Haile Gold Mine. As the property is currently under development and any potential underground operation would share infrastructure with the existing mine, this underground potential is included in Section 24, other relevant data and information within the existing Canadian National Instrument 43-101 (NI 43-101) Technical Report. This section includes all items required by NI 43-101 and discusses the information relevant to a potential underground operation. Sections where there is no difference between the current open pit feasibility and potential underground are noted as such.

All information described in Section 24 is at a scoping or preliminary economic assessment (PEA) level. A PEA is preliminary in nature in that it includes Inferred Mineral Resources. There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources, or that the production target and forecast financial information derived from the production target will be realized. The production target and forecast financial information as stated in this Section 24 is based on the company's current expectations of future results or events and should not be solely relied upon by investors when making investment decisions. Further evaluation work and appropriate studies are required to establish sufficient confidence that this target will be met.

24.1 SUMMARY

The key concepts of this underground study include:

- An underground longhole stoping mining method, with cemented rock backfill, is used to mine material left in-situ below/adjacent to the designed feasibility study open pits;
- Underground material is processed through the existing process facility which assumes several upgrades to increase plant throughput to a total of 9,120 st/d;
- The current air permit allows for processing 9,120 st/d. Minimal additional permitting would be required for the presented underground scenario assuming no changes to the mining footprint boundary, tailings/waste rock stockpile locations and sizes; and
- The underground mine would share surface infrastructure with the open pit.

24.1.1 Key Project Data

Table 24-1 summarizes the key project data for the underground PEA project plan. Table 24-2 summarizes the after-tax economics of the project.

Table 24-1: Key Underground Project Data

Description	Technical Input
Pre-Production Period	2 years
UG Mine Life	7 years
Mine/Mill Operating Days per Year	365 days/yr
Maximum UG Designed Production Rate	2,120 st/d
UG Construction Start	2018
UG Commercial Production Year	2020
Initial Capital (US$000's)	$73,953
Total Operating Cost (UG-specific Mine, Mill and G&A) US$/st	$44.70

Source: SRK, 2016

Table 24-2: Combined Open Pit and Underground Project Economics (US$000's)

Description	OP+UG
Metrics	
Pre-Tax Free Cash Flow	$1,097,815
After-Tax Free Cash Flow	$860,565
Pre-Tax NPV @: 5%	$670,602
After-Tax NPV @: 5%	$509,650
Pre-Tax IRR	26.8%
After-Tax IRR	23.2%
BT Undiscounted Payback from Start of Comm. Prod. (Years)	3.2
AT Undiscounted Payback from Start of Comm. Prod. (Years)	3.4
Total Cash Costs (TCC) $/payable oz	$554

Source: SRK, 2016

24.1.2 Property Description and Ownership

The property description and ownership are the same as that stated in Section 4. There are three underground targets included in this PEA, referred to as Horseshoe, Mustang, and Mill Zone Deep. They are located near planned open pits.

24.1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The accessibility, climate, infrastructure and physiography are the same as that stated in Section 5. There is available labor in the area, which is currently being used for the open pit development. Specialized underground skill sets will be required for an underground operation. Key trained personnel will be relocated to the area and a training process will develop the overall workforce.

24.1.4 History

The history is the same as that stated in Section 6.

24.1.5 Geologic Setting and Mineralization

The geological setting and mineralization is the same as that stated in Section 7.

24.1.6 Deposit Types

The deposit type is the same as that stated in Section 8.

24.1.7 Exploration

A two phase program, totaling 72,200 ft of surface-collared diamond drilling has been undertaken at the Horseshoe area. Approximately 45,900 ft have been completed to-date. The Horseshoe drilling results to-date from this drilling were not received in time to update the estimate for this underground study. An updated resource estimate is expected to be completed in Q4 of 2016.

24.1.8 Drilling

The drilling is the same as that stated in Section 10.

24.1.9 Sample Preparation, Analyses and Security

The sample preparation, analyses and security are the same as that stated in Section 11.

24.1.10 Data Verification

The data verification is the same as that stated in Section 12.

24.1.11 Mineral Processing and Metallurgical Testing

Process recoveries are anticipated to follow the same recovery curve and sufficient metallurgical test work exists across the property to support these assumptions at a PEA level. Mill feed will be a blend of the open pit and underground material.

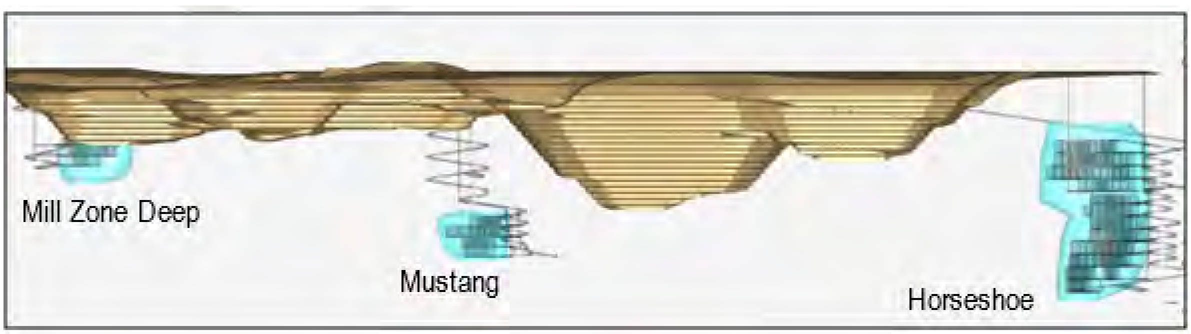
24.1.12 Mineral Resource Estimates

The resources discussed in this section, termed the underground target resources, are almost entirely located below the reserve pit design but within the US$1,200/oz open pit resource reporting shell. In other words, the underground target resources presented here cannot be added to the open pit resource inventories presented in Sections 1.1 and 14.2.1 of this report. Clarification is provided as follows.

Importantly, the mining of the underground resources discussed in this section would not preclude mining of the open pit reserves presented in this report.

After due consideration of CoG, geometric and grade continuity and drill spacing, only Mill Zone Deep, Mustang and Horseshoe, as shown in Figure 24-1 were targeted for this study. Mineralization is interpreted as being open in several areas, with opportunities for delineation of additional mineralized material with additional drilling.

The underground target resource is tabulated in Table 24-3, and is constrained within volumes manually created to envelope the underground stope designs discussed in the mining section. Typically, the resource-constraining volumes extend 20 m (65 ft) beyond the design volumes. All the underground target resource is classified as Inferred.



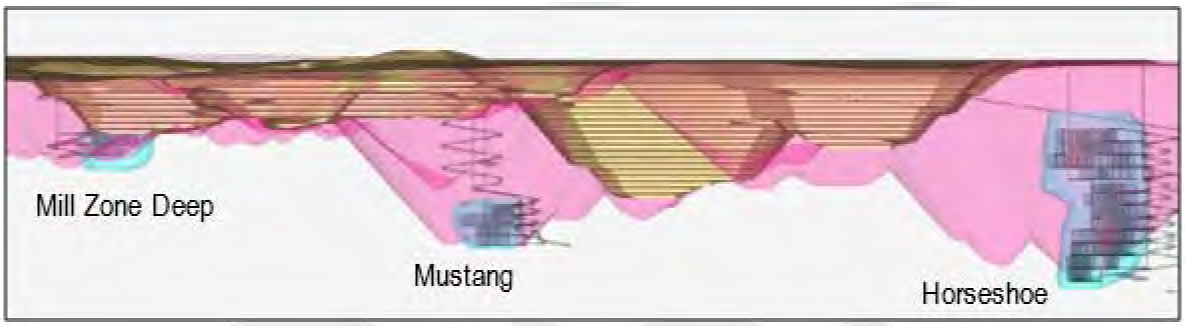
Source: Oceana

Figure 24-1: EW Section with Reserve Pit, UG Development and Resource Volumes (blue)

Other notes regarding the resource:

- 95% of the underground target resource presented in Table 24-3 is already reported as open pit resource in sections 1.1 and 14.2.1 of this report, as it occurs within the US$1,200/oz open pit resource reporting shell (Figure 24-2). This component is presented in Table 24-4 and is classified as Inferred.

- Mining of the underground resources discussed in this section would not preclude mining of the open pit reserves presented in this report and defined by the reserve shell. The underground plan presented here mines below/outside of the reserves shell and pillars are left where required to allow for open pit mining.
- In nearly all cases, the corresponding open pit classification for the same volume of mineralization is of a higher category (generally Indicated, and in a few cases Measured). This demotion of classification category for the underground target resource relates to the higher CoG applied as well as the need for higher locational certainty of grade.
- Only 5% of the resource reported in Table 24-3 is additional to the Open Pit resource presented in sections 1.1 and 14.2.1 of this report. That is to say that only 5% of the underground target resource is located below the US$1,200/oz open pit resource reporting shell. This component is presented in Table 24-5.



Source: Oceana

Figure 24-2: EW Section with Reserve Pit, UG Development / Volumes and US$1,200/oz Shell (pink)

Table 24-3: Total Underground Target Resource by Deposit – Inferred Classification

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	4,056	0.16	645
Mustang	847	0.13	111
Mill Zone Deep	337	0.15	49
Total	5,241	0.15	805

Source: Oceana
Resources constrained to volumes expanded around underground design volumes.

Table 24-4: Component of Underground Resource within US$1,200/oz Shell

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	3,752	0.16	600
Mustang	847	0.13	111
Mill Zone Deep	337	0.15	49
Total	4,937	0.15	760

Source: Oceana
Resources constrained to volumes expanded around underground design volumes.

Note the resources presented above in Table 24-4 are fully included in the reported open pit resource inventory.

Table 24-5: Component of Underground Resource Below US$1,200/oz Shell

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	304	0.15	45
Mustang	0	0.00	0
Mill Zone Deep	0	0.00	0
Total	304	0.15	45

Source: Oceana
Note that only a very small proportion of the Horseshoe resource, presented above in Table 24-21, is additional to the open pit resource inventory.

24.1.13 Mineral Reserve Estimates

No underground mineral reserves have been estimated for the Project.

24.1.14 Mining Methods

The available data indicate that underground operations using longhole stoping methods with cemented backfill are viable for the Project. The production rate from the underground is 2,120 st/d at an average life-of-mine (LoM) grade of 0.14 oz/st Au. There are three areas included in the design named Horseshoe, Mustang, and Mill Zone Deep. The stopes will be 49 ft wide and stope length will vary based on mineralization grade. A spacing of 82 ft between levels has been used in the Horseshoe area and 66 ft in the Mustang and Mill Zone Deep areas to best fit the mineralization and minimize dilution. The areas are mined bottom to top with the use of sill pillars where necessary. A cemented rock fill is used when necessary (70% of stopes) and non-cemented waste rock fill is used in the remaining stopes. The cemented backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars.

Mine design using Vulcan™ software was completed based on an estimated CoG of 0.05 oz/st. Stope optimization was used to determine mine plan resource areas. Table 24-6 summarizes the mine plan resources. These numbers include a 95% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to 0% to 7.5% unplanned waste dilution. An additional development allowance of 20% was applied to main ramps and 10% to level accesses to account for detail currently not in the design. These percentages were determined based on typical level layout, geotechnical ELOS factors, and practical mining factor assumptions. Waste dilution for stopes was applied with grade based on an analysis of the material around stopes in a representative area.

Table 24-6: Mine Plan Resource Classification (1)

Description	Tons (kst)	Au (oz/st)
Inferred	4,777	0.14

(1) Includes inferred material reported using a 0.05 oz/st Au CoG. A PEA is preliminary in nature that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.
Source: SRK, 2016

The design was then scheduled using iGantt software to generate a LoM production schedule summarized in Table 24-7. The mining schedule discussed in this item must be read in conjunction with the cautionary statement in the introduction to Section 24, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the realisation of the production target.

Table 24-7: Annual Mining Schedule (includes Horseshoe, Mustang, & Mill Zone Deep Areas)

Year	Mineralized Tons (kst)	Au (oz/st)	Waste Tons (kst)	Backfill Volume (Mft^3)
2018			154.5	
2019	533.5	0.13	430.7	5.7
2020	773.8	0.16	183.8	8.6
2021	773.8	0.13	55.7	8.9
2022	773.8	0.15	246.2	8.9
2023	773.8	0.15	176.3	8.6
2024	773.8	0.12	216.7	9.5
2025	374.5	0.10	18.2	4.2
Total	4,777.0	0.14	1,482.2	54.3

Source: SRK

Mining operations within a stope include establishing top and bottom accesses, developing a slot raise at the far end of the stope and using a fan shaped drilling pattern to blast rings on retreat toward the level access. Main ramps are sized as 16 ft x 18 ft openings with an arched back. In stope development is sized as 15 ft x 15 ft with a flat back. All drifting work is developed using two boom jumbos. Ramps are designed at a maximum gradient of 14% with an 80 ft turning radius which is suitable for any underground truck. Stope and development material is mucked using 15 st load-haul-dumps (LHD) into 44 st underground trucks for haulage. All three mining areas are accessed via declines from planned open pits. An emergency escape system is included in the ventilation raises.

A cemented rock fill is used when necessary (70% of stopes) and non-cemented waste rock fill is used in the remaining stopes. The cemented backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars. Current cure time assumptions are seven days prior to driving on CRF (cemented rock fill) and 14 days prior to mining adjacent to a CRF filled stope.

A 5.4 ft oval duct would provide sufficient air for the length of development and would accommodate the drift size and equipment. The fan system for the duct will incorporate three fans at the portal operating in series. The fans are added as the length of the development increases. The same development ventilation system would be used for all three mining areas.

The primary ventilation scheme has a fresh air intake and secondary egress, and an exhaust raise. Each level is connected to both the fresh air raise and the exhaust raise. Fresh air comes down the intake raise, blows across the footwall access of a level, and exhausts through the exhaust raise. The ramp is on intake and air flows onto working levels from the ramp and exhausts through the exhaust raise. The ramp air is used for development of lower levels prior to connecting levels to the main ventilation system. The minimum dimensions and airflows are shown in Table 24-8.

Table 24-8: Airway Dimensions and Airflows

Airway	Airflow (cfm)	Dimensions (ft)	Velocity (ft/sec)
Exhaust	435,000	13.1 ft diameter	53.54
Decline	108,000	16 ft wide x 18 ft high	6.23
Fresh Air Raise	327,000	16.4 ft diameter	25.69

Source: SRK

Various areas of the mine are expected to produce varying amounts of water based on the geology and sequencing of the underground and open pits. The underground mine is predicted to produce the amounts of water shown in Table 24-9. In the Horseshoe area, a surface dewatering well is located near the main ramp which helps to lower the quantity of water pumped from underground. Designed pumping capacities to handle the predicted peak inflows are also shown in Table 24-9.

Table 24-9: Predicted Water Inflows

Area	Predicted Peak Inflow (gpm)	Designed Pumping Capacity (gpm)
Horseshoe (1)	300	500
Mustang	75	200
Mill Zone Deep	120	250

(1) Considers an active dewatering well on surface near the main ramp.
Source: SRK, CDM Smith

The large peak inflows are expected to be encountered in the decline near surface. The weathered metavolcanics in this area have an enhanced permeability as compared to the other units. Once through the weathered zone, lower amounts of water are anticipated.

24.1.15 Recovery Methods

The underground feed material will be processed the same way as the open pit material. Process recoveries are anticipated to follow the same recovery curve and sufficient metallurgical test work exists across the property to support these assumptions at a PEA level. Mill feed will be a blend of the open pit and underground material.

A preliminary review of the plant design criteria and equipment sizing has been undertaken envisaging a process plant capable of treating up to 9,120 st/d of material. The main items identified to date that are likely to be required at the higher capacity include:

- ROM Pad upgrade;
- Recycle Crusher;
- Flash Flotation Cleaner Cell installation;
- Rougher Cell installation;
- Regrind Tower Mill installation;
- Flotation Tailings Thickener replacement;
- Leach Tank installation;
- Cyanide Recovery Thickener replacement;
- Tailings Pump upgrade;
- Tailings Line replacement; and
- Slurry Pump Motor upgrades

24.1.16 Project Infrastructure

The underground mine will utilize the existing open pit infrastructure. An additional 4,000 ft of above ground power line will supply the underground mines power from the existing site substation. A new change house/shower and administrative building will be added for underground mine use on the surface near the Horseshoe mine portal. Dewatering lines, pumps, and associated infrastructure will move water from the underground to the existing water treatment facility near the processing plant.

24.1.17 Market Studies and Contracts

Market studies and contracts are the same as stated in Section 19.

24.1.18 Environmental Studies, Permitting and Social or Community Impact

Underground mining operations at the OceanaGold Haile Operation (Haile) would require a modification of Haile's state Mine Operating Permit (Mine Permit) issued by the South Carolina Department of Health & Environmental Control (DHEC), and Haile's associated Mine Plan and Reclamation Plan. Due to the cost of reclaiming the underground operations, Haile would likely be required to increase its current USD$65 million reclamation bond. DHEC would also determine whether a modification of any of Haile's other state-issued permits was required. If the underground mining operation would impact Waters of the United States not currently authorized to be impacted by Haile's federal 404 Clean Water Act Permit (404 Permit), issued by the U.S. Army Corps of Engineers (the Corps), then Haile would be required to obtain another 404 Permit. If a 404 Permit were required, the Corps would determine whether the National Environmental Policy Act (NEPA) of 1970 was applicable. Other state and federal agencies, nongovernmental organizations, and the public would be afforded an opportunity to participate in the permit modification process.

The permitting modification time frame would decrease/increase depending on the level of stakeholder participation, including participation by other state and federal agencies, as well as the nature of the anticipated impacts to the human environment.

24.1.19 Capital and Operating Costs

Table 24-10 contains a summary of capital costs for the underground development and operations of the Project. Capital costs contain the design, procurement and construction of the underground mine and necessary additions to the existing surface processing plant, auxiliary facilities, and infrastructure. At this level of study, and with the work performed to-date, the underground capital cost estimate is at an accuracy of ±40%. An overall LoM contingency of 7.3% was determined by applying 10% and 15% contingency to mobile and fixed mine equipment respectively, 0% contingency for mine development costs, and 15% for process plant expansion. The majority of the capital is mining equipment where the basis for cost was budgetary pricing for single pieces of equipment. During actual purchase fleet discounts would be applied reducing the prices of individual units. As such, a low contingency has been applied to offset the cost basis. Development costs were calculated from first principles and do not include contingency.

Table 24-10: Underground Capital Cost Summary

Description	Initial (US$000's)	Sustaining (US$000's)	LoM (US$000's)
Mining Equipment/Assets	33,017	8,719	41,736
Mine Development	20,068	26,324	46,392
Equipment Rebuilds	0	9,499	9,499
Subtotal Mining Capital	53,085	44,542	97,627
Processing Expansion to 9,120tpd	14,000	0	14,000
Subtotal Capital Before Contingency	67,085	44,542	111,627
Contingency	6,868	1,399	8,267
Total Capital	$73,953	$45,941	$119,894

Source: SRK, 2016

The overall LoM operating cost for the underground Project is estimated at US$44.70/st mineralized material milled. A summary of the operating costs for the Project is shown in Table 24-11. All costs presented in this section are in US dollars per underground mineralized material milled.

Table 24-11: Underground Operating Cost Summary

Description	US$/t Processed	US$/st Processed	LoM (US$000's)
UG Mine	30.80	27.94	133,465
Process (UG tons only)	10.58	9.60	45,859
G&A	7.89	7.16	34,210
Total	$49.27	$44.70	$213,534

Source: SRK, 2016

The capital and operating expenditure discussed in this sub-section must be read in conjunction with the cautionary statement in the introduction to Section 24, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realized.

24.1.20 Economic Analysis

The economic results of the combined open pit and underground scenario and a comparison to the open pit only scenario are presented in Table 24-12. The Project has an after-tax NPV (5%) of US$510 million which is 75% higher than US$291 million for the open pit only scenario. The after-tax IRR of 26.8% is 39% higher than the open pit only scenario of 16.7%.

Table 24-12: Indicative Economic Results (US$000's)

Description	OP+UG	OP Only	Variance
Market Prices			
Gold (US$/oz)	$1,250	$1,250	--
Silver (US$/oz)	$20.00	$20.00	--
Revenue			
Payable Gold (koz)	2,287	1,678	36%
Payable Silver (koz)	2,083	2,083	--
Total Gross Revenue	$2,900,083	$2,138,572	36%
Operating Costs			
Mining	(543,769)	(376,093)	45%
Processing	(388,861)	(339,943)	14%
Site G&A	(136,521)	(119,639)	14%
Selling/Refining	(4,916)	(6,126)	(20%)
Indirects	(125,140)	(125,140)	--
Total Operating Costs	($1,199,207)	($966,942)	24%
Operating Margin (EBITDA)	$1,700,876	$1,171,631	45%
Taxes			
Income Tax	(237,249)	(126,938)	87%
Total Taxes	($237,249)	($126,938)	87%
Working Capital	0	0	--
Operating Cash Flow	$1,463,627	$1,044,692	40%
Capital			
Initial Capital	(423,153)	(349,200)	21%
Sustaining Capital	(184,483)	(138,541)	33%
Reclamation/Salvage Capital	4,575	4,575	--
Total Capital	($603,061)	($483,166)	25%
Metrics			
Pre-Tax Free Cash Flow	$1,097,815	$688,464	59%
After-Tax Free Cash Flow	$860,565	$561,526	53%
Pre-Tax NPV @: 5%	$670,602	$371,796	80%
After-Tax NPV @: 5%	$509,650	$290,784	75%
Pre-Tax IRR	26.8%	18.7%	43%
After-Tax IRR	23.2%	16.7%	39%
BT Undiscounted Payback from Start of Comm. Prod. (Years)	3.2	4.2	(24%)
AT Undiscounted Payback from Start of Comm. Prod. (Years)	3.4	4.4	(23%)
Total Cash Costs (TCC) US$/payable oz	$554	$590	(6%)

Source: SRK, 2016
BT=before tax
AT-after tax

Given the herein presented underground scenario, Oceana has a robust business case for having a 2,120 st/d underground mining operation run concurrently with the proposed open pit mine at the Haile project. With the combined LoM operation producing an average of 200 koz/y of gold at a TCC of US$554/oz, SRK estimates that the operation would be comfortably within the second cash cost quartile of gold producers.

24.2 INTRODUCTION AND TERMS OF REFERENCE

The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in SRK's services, based on:(i) information available at the time of preparation, (ii) data supplied by outside sources and (iii) the assumptions, conditions and qualifications set forth in this report. This report is intended for use by Oceana subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Oceana to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Oceana. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This section provides underground estimates of Mineral Resources within a PEA design mine plan, and a classification of resources prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards – For Mineral Resources and Mineral Reserves, May 10, 2014.

The US System for weights and units has been used throughout this report. Tons are reported in short tons of 2,000 lbs. Unless otherwise stated, all tons are reported as dry tons. All currency is in U.S. dollars (US$) unless otherwise stated.

24.3 RELIANCE ON OTHER EXPERTS/QUALIFICATIONS

The Consultant's opinion contained herein is based on information provided to SRK by Oceana throughout the course of the investigations. SRK has relied upon the work of other consultants and Oceana staff. The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. The QP's are responsible for specific sections of Section 24 as follows:

- Jonathan Moore, BSc (Hons) Geology, DipGrad Physics, (OceanaGold Chief Geologist), is the QP responsible for geology and resources Sections 24.7 through 24.14 (except 24.13), and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- David Carr, BEng Metallurgical (Hons), (OceanaGold Chief Metallurgist), is the QP responsible for metallurgy and processing Sections 24.13, 24.17, 24.19, 24.21 (co-authored), and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- John Tinucci, PhD, PE (SRK Principal Consultant, Geotechnical Engineer), is the QP responsible for geotechnical Sections 24.16.2, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- Robert P. Schreiber, PE, D.WRE, BCEE (CDM Smith Inc., Vice President) is the QP responsible for hydrological Sections 24.16.3, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- Patrick Williamson, MSc Geology, MMSAQP (SRK Principal Consultant, Geochemistry and Hydrogeology), is the QP responsible for geochemical Sections 24.16.4, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- Joanna Poeck, BEng Mining, SME-RM, MMSAQP (SRK Senior Consultant, Mining Engineer), is the QP responsible for mining planning Sections 24.2 through 24.6, 24.15, 24.16.1, 24.16.5 through 24.16.9 (co-authored), 24.23, 27, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- Jeff Osborn, BEng Mining, MMSAQP (SRK Principal Consultant, Mining Engineer), is the QP responsible for mining and infrastructure of Sections 24.16.5 through 24.16.9 (co-authored), 24.18, 24.21 (co-authored), and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.
- Scott McDaniel, BSc. Metallurgical Engineering (Haile Environmental Manager), is the QP responsible for environmental, permitting and social/community Sections 24.20, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.

- Grant Malensek, MEng, PEng/PGeo (SRK Principal Consultant, Mineral Economics), is the QP responsible for economic Sections 24.21 (co-authored), 24.22, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of this Technical Report.

SRK was reliant upon information and data provided by Oceana including historic data inherited from previous owners.

SRK has been provided with adequate copies of necessary data in digital format. SRK has, where possible, verified data provided independently, and completed a site visit to review physical evidence for the Project.

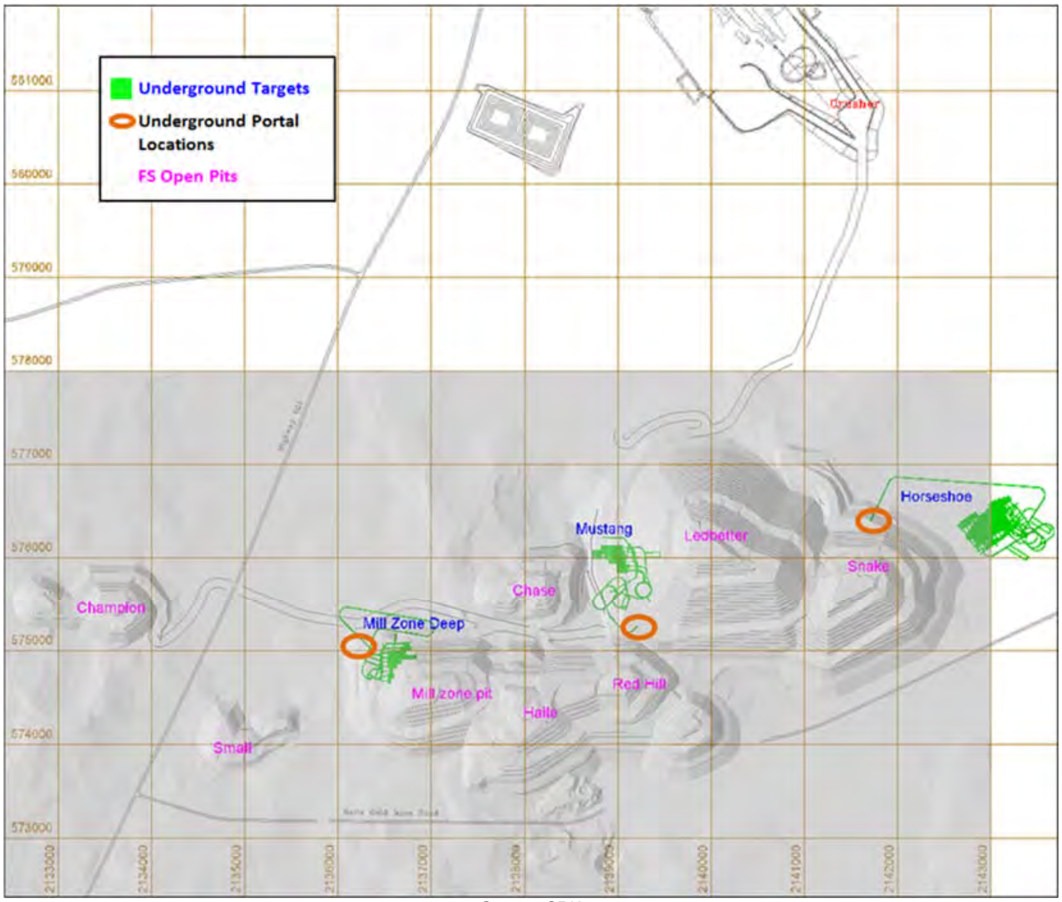
SRK has relied upon information supplied by Oceana (Mr. Tom Cooney) during this current study. Land titles and mineral rights for the Project have not been independently reviewed by SRK.

This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

Jeff Osborn (QP) visited the Project property on April 13 and 14, 2016. The site visit included a general site tour. Specific site locations included visiting the core storage area and review of the Horseshoe area core, visits to the Mill Zone open pit, Horseshoe mine area, and project processing plant facilities and infrastructure currently under construction. The visit included meeting with site supervisory personnel, geology team, and technical services personnel. A meeting concerning mine dewatering was also attended.

24.4 PROPERTY DESCRIPTION AND LOCATION

The property description and location are the same as that stated in Section 4. There are three underground targets included in this PEA, referred to as Horseshoe, Mustang, and Mill Zone Deep. Their locations and planned underground access locations are shown in Figure 24-3.



Source: SRK

Figure 24-3: Underground Targets Location

Additional property and description location is available in Section 4.

24.5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The accessibility, climate, infrastructure and physiography are the same as that stated in Section 5. There is available labor in the area, which is currently being used for the open pit development. Specialized underground skill sets will be required for an underground operation. Key trained personnel will be moved to the area and a training process will be conducted to upgrade the skills of the local workforce.

24.6 HISTORY

The history is the same as that stated in Section 6.

24.7 GEOLOGICAL SETTING AND MINERALIZATION

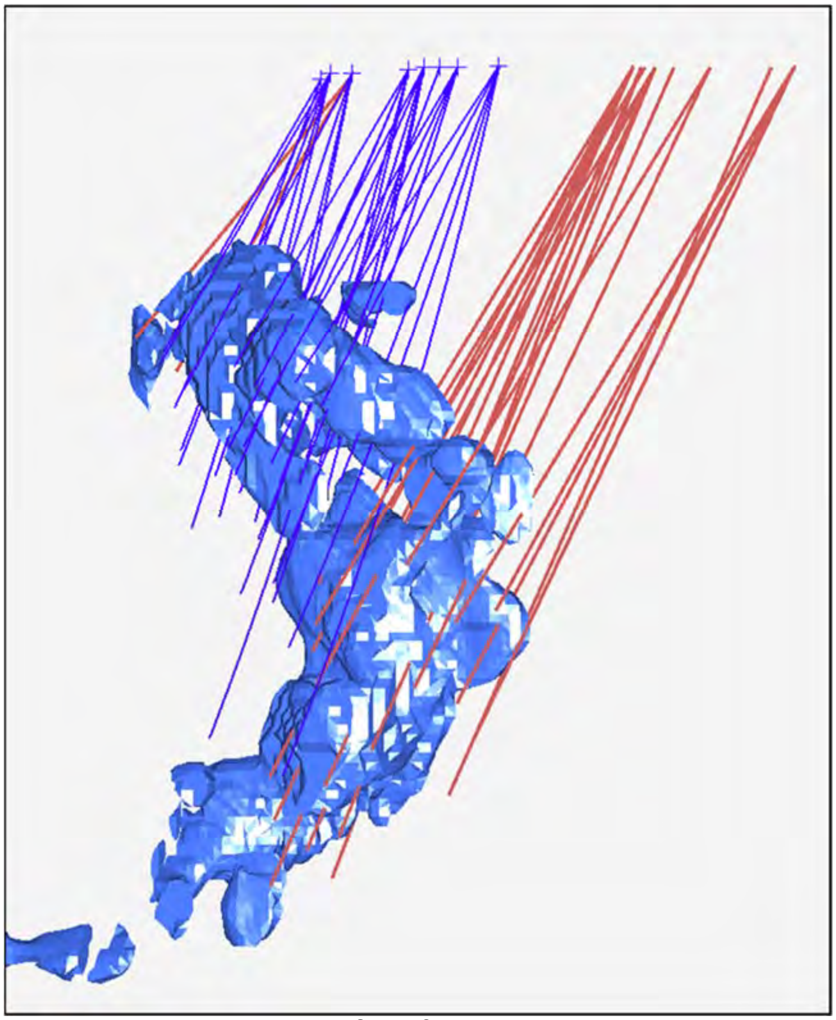
The geological setting and mineralization is the same as that stated in Section 7.

24.8 DEPOSIT TYPES

The deposit type is the same as that stated in Section 8.

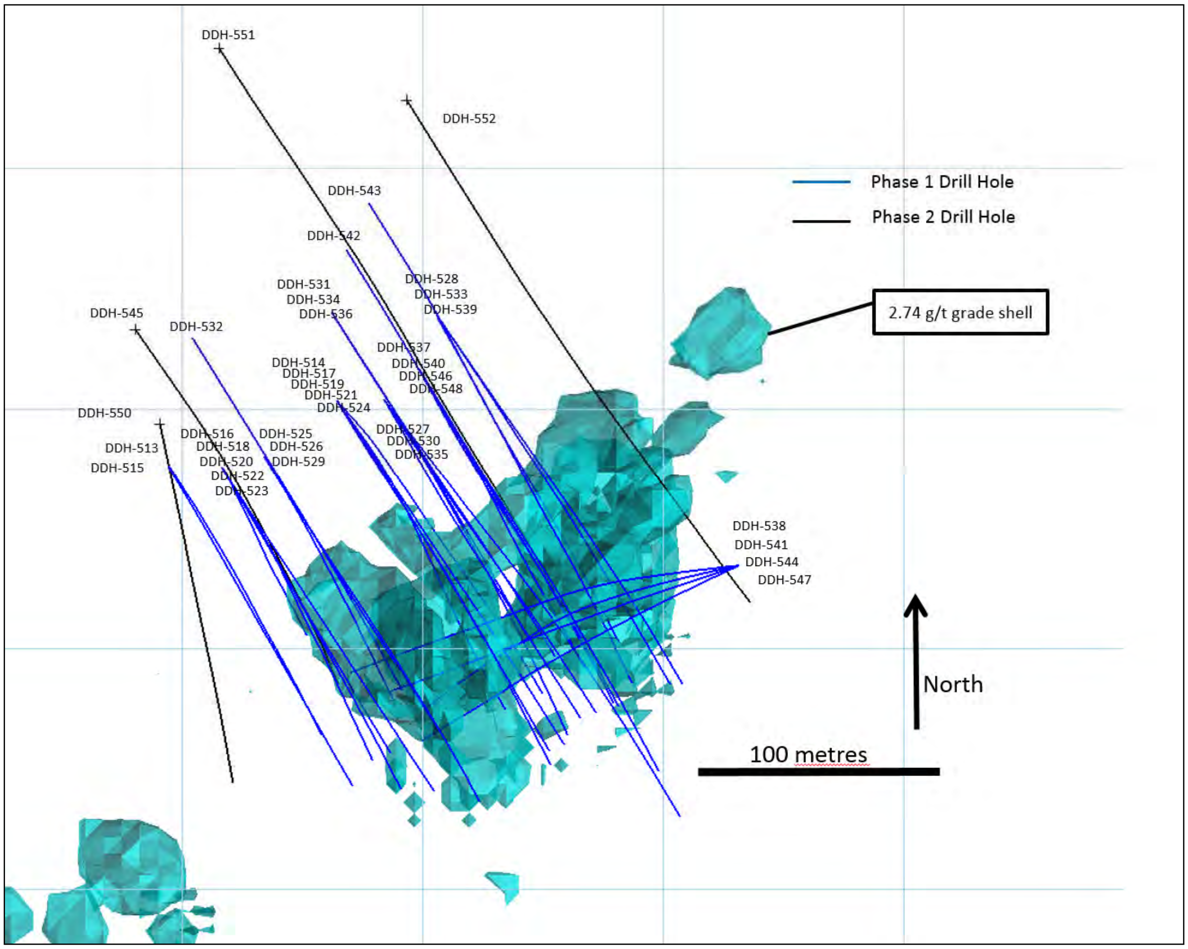
24.9 EXPLORATION

A two phase program, totaling 72,200 ft of surface-collared diamond drilling has been undertaken at the Horseshoe area, as shown in Figure 24-4. Approximately 45,900 ft have been completed to-date as shown in Figure 24-3. The drilling targets approximately 60% of the Horseshoe resource and is intended to provide 65 ft x 65 ft coverage. This is expected to convert at least 350 koz of the Horseshoe resource (discussed in Section 24.14) to Indicated classification.



Source: Oceana

Figure 24-4: Looking NW, Planned Horseshoe Infill Drilling (Phase 1 blue, Phase 2 red)



Source: Oceana

Figure 24-5: Plan View – Horseshoe Infill Drilling Completed To-date

The Horseshoe drilling results to date from this drilling were not received in time to update the Horseshoe estimate for this study; however, they are summarized in Table 24-1. An updated resource estimate is expected to be completed in Q4 of 2016. Over the coming months, an exploration strategy to grow the underground resource base will be developed.

Table 24-13: Horseshoe Drilling Results To-date

Hole ID	Interval (ft)	From (ft)	To (ft)	Grade (opt)
DDH-513	13.1	565	578.1	0.052
	71	594	665	0.111
including	25.7	621.6	647.3	0.207
DDH-514	25	585	610	0.096
including	7.7	587.3	595	0.161
	30.6	640	670.6	0.152
including	10.6	660	670.6	0.308
	25	700	725	0.025
DDH-515	210.2	561	771.2	0.053
including	10	600	610	0.225
DDH-516	20	560	580	0.031
	15	620	635	0.118
including	10	625	635	0.167
	11.7	666.1	677.8	0.174
including	7.5	666.1	673.6	0.26
DDH-517	134.3	577.9	712.2	0.138
DDH-518	38	556	594	0.071
	42.8	602	644.8	0.159
	43.7	662.3	706	0.308
including	14.7	688.1	702.8	0.829
DDH-519	70	570	640	0.632
	77	650	727	0.191
	60	770	830	0.038
DDH-520	35.6	539.4	575	0.04
	45.5	604.2	649.7	0.033
	46.7	665	711.7	0.045
DDH-521	18.5	635	653.5	0.178
	12.7	665	677.7	3.671
	18.6	876.4	895	0.055
DDH-522	217.8	539.5	757.3	0.135
including	17	594	611	0.176
including	56	701.3	757.3	0.333
DDH-523	70	667.5	737.5	0.117
including	14.2	715.3	729.5	0.380

Hole ID	Interval (ft)	From (ft)	To (ft)	Grade (opt)
DDH-525	8.1	563.5	571.6	0.216
	92.4	617.6	710	0.218
DDH-526	200	567	767	0.379
DDH-527	40.1	615	655.1	0.257
including	1.3	619.4	620.7	6.964
DDH-528	104	696	800	0.277
	18	812	830	0.267
DDH-529	43.9	555	598.9	0.082
	164.1	636.8	800.9	0.546
	15.8	813	828.8	0.253
DDH-531	80	510	590	0.073
	47.9	668.8	716.7	0.038
DDH-532	11.4	643.7	655.1	0.068
	9.5	686.2	695.7	0.090
	190.2	718.1	908.3	0.509
DDH-533	No significant assays			
DDH-534	39	526	565	0.089
	13.8	581.6	595.4	0.070
	7.3	980.3	987.6	0.131
DDH-535	11.7	682.5	694.3	0.042
	20	710	730	0.069
DDH-536	47.7	621.5	669.2	0.035
	18.9	700.1	719	0.052
	16.3	1120.5	1136.8	0.097
DDH-537	14.4	655.6	670	0.027
DDH-538	148.9	556.4	705.3	0.150
	15	765	780	0.053
DDH-540	41.9	785	826.9	0.105

Source: Oceana

24.10 DRILLING

The drilling is the same as that stated in Section 10.

24.11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The sample preparation, analyses and security are the same as that stated in Section 11.

24.12 DATA VERIFICATION

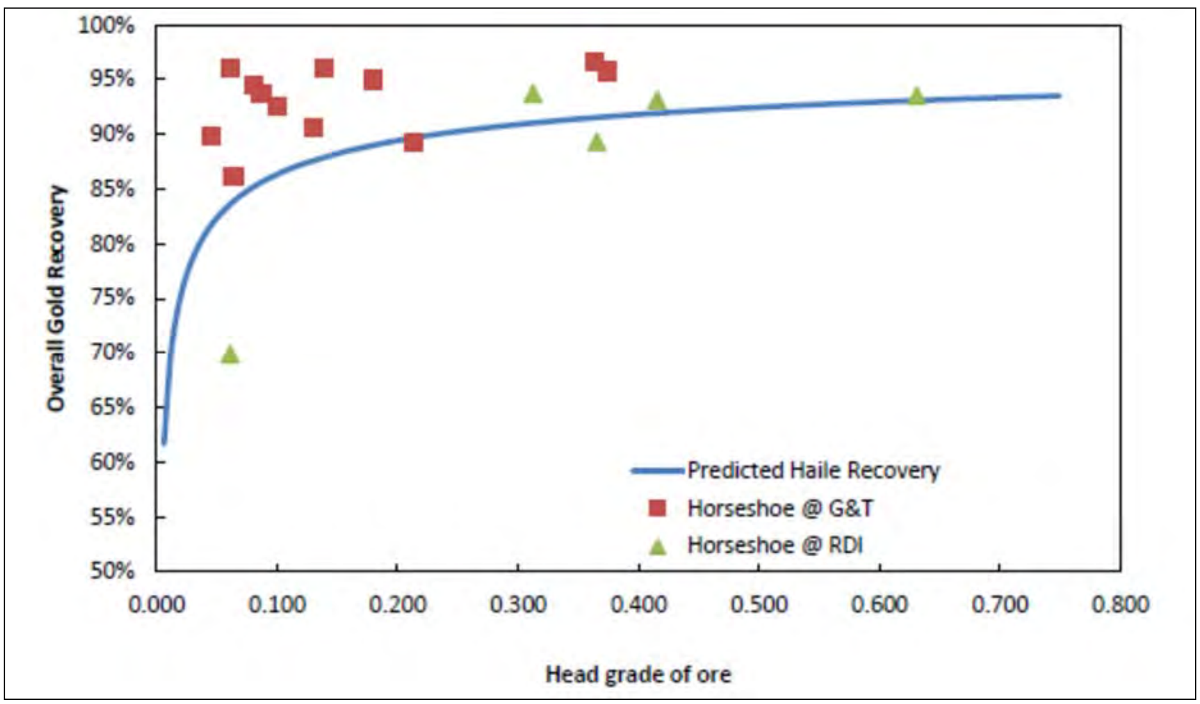
The data verification is the same as that stated in Section 12.

24.13 MINERAL PROCESSING AND METALLURGICAL TESTING

A total of 12 variability composites from historical drilling of the Horseshoe deposit were tested by RDi and G&T to establish the response of the mineralization to the Haile flowsheet. Overall the results were in line with or exceeded the recovery model for the Haile deposit by up to 2% for gold. Overall flotation recovery was high with leachable gold remaining in the flotation tail also amenable to leaching. The existing model as published in the NI 43-101 predicting gold recovery as a function of head grade is a suitable model to continue to use for the evaluation of the Horseshoe deposit.

$$\text{Gold Recovery (\%)} = (1 - (0.053 \times \text{oz/t Au}^{-0.3697})) \times 100$$

The performance of the variability composites against the model is demonstrated in Figure 24-6.



Source: Oceana

Figure 24-6: Gold Recovery Variability Composites for Horseshoe and Predicted Recovery

The underground estimated mill feed is of a higher grade than the samples tested (~0.14 oz Au/st). To progress to the next level of study additional higher grade samples should be tested to confirm the recovery assumptions.

24.14 MINERAL RESOURCE ESTIMATES

24.14.1 Geology and Domaining

The majority of mineralization at Haile is hosted within a single, complexly folded metasedimentary unit and is associated with intense silica-pyrite-sericite alteration. Two major structural orientations are present, broadly outlining a major ENE trending antiformal structure, with shallow northerly dips on the northern limb and steep dips on the southern limb. However, the local scale relationship between structure and mineralization is not fully understood. In lieu of model-able geological controls on mineralization there is little alternative to using grade envelopes to constrain estimates of gold. Geological work is planned for later in 2016 to trial three dimensional form line modeling of bedding using orientated drill core measurements. This will be superimposed over logged lithology and foliation to test the extent to which grade distribution is related.

Gold grades at Haile are typically highly skewed with a high nugget effect, and continuity is poor at elevated cut-offs although at a low threshold (0.073 oz/st Au) mineralization is broadly continuous. For this reason, the resource classified as Indicated for open pit reporting was demoted to Inferred, and in some cases, unclassified for underground resource reporting.

Implicit modelling was used to generate a mineralization envelope at this threshold.

As a first pass, gold grades were estimated directly inside the 0.0073 oz/st Au envelope. Estimated grades are necessarily smoothed, and may not reflect the tonnage and grade that will be defined from close spaced drilling at likely underground mining cut-offs.

The approach used for estimation was to create geometry models of volume above indicator thresholds using implicitly modelled indicator interpolants. Samples were then flagged inside geometry volumes and grades were estimated within. Using this approach, the indicator volumes used to constrain estimation appear to provide a better representation of the likely geometry of mineable material.

It must be recognized that the continuity of mineralization is as yet poorly characterized, and even in the well drilled areas of the deposit, interpretation of continuity is not straightforward. This uncertainty is reflected in the level of classification applied to the estimates.

The estimated tonnage and grade is highly sensitive to the choice of CoG. The modelling approach adopted effectively fixes the tonnage at the chosen indicator threshold (i.e. that becomes the CoG). Deciding on which CoG to use for resource definition will necessitate an iterative approach between resource modelling and mine-planning. A resource model at a cut-off of 0.044 oz/st was used for the mining study.

Note that a set of post-mineralization NNW-SSE striking diabase dykes cut across the deposit. Haile geologists have developed wireframe solid models for the larger and more continuous dykes, and the grades associated with these volumes have been nullified in the block model. There are numerous smaller and less continuous dykes which have not been modelled. While these will reduce the volume of recoverable mineralization, the effect of these on estimated resource is not likely to be material.

24.14.2 Input Data

The resource modeling was completed in metric units, with coordinates directly converted from feet to meters. The parameters are presented in the metric units, as modeled, but converted to imperial in this report where required.

The drilling dataset used for this estimate was provided by Haile Gold Mine exploration personnel on July 21, 2015. It consists of a set of comma delimited ASCII files entitled:

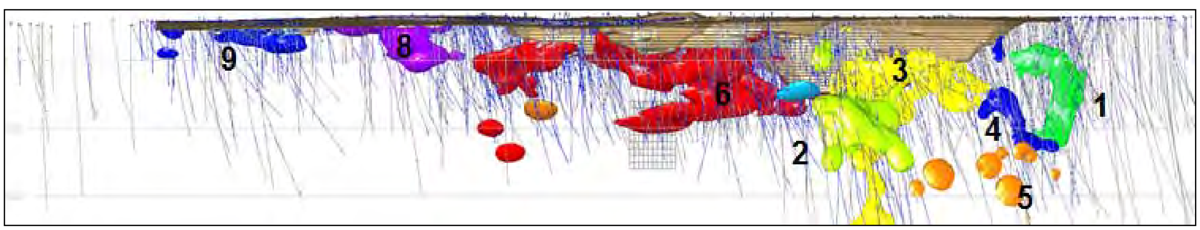
- 15_master_collar.csv
- 15_master_survey.csv
- 15_master_assay.csv
- 15_master_lith.csv
- 15_master_density.csv
- 15_master_geotech.csv
- 15_master_hydro.csv
- 15_master_ARD.csv
- 15_master_isotope.csv
- 15_master_trace.csv

Au assays are provided as either fire assay or cyanide leach. A combined Au field is provided where fire assay values are taken as priority and cyanide values are used only if fire assay is missing. The vast majority (94%) of the assaying used in estimation is by fire-assay. Drilling was imported to both MineSight® and Leapfrog® Geo softwares.

A compositing length of 6 m (20 ft) was used in definition of the 0.25 g/t (0.073 oz/st) mineralization envelope in order to smooth underlying variability in grade prior to modeling, and improve definable continuity. This composite length matches the block size in the north-south and vertical dimensions. The dominant underlying sample length is 1.53 m (5 ft). At a threshold grade of 1.5 g/t (0.044 oz/st) Au, around 66% of the total length of contiguous mineralized intercepts (all assays>1.5 g/t, no internal waste) occur in intervals 6.1 or more meters in length.

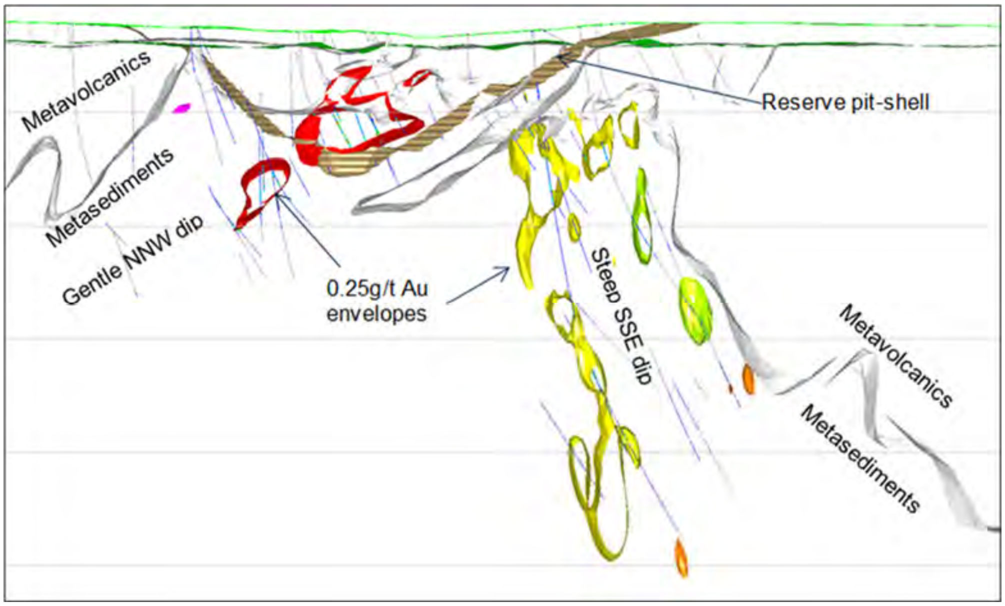
24.14.3 Volume Models

Geometry models of mineralization were created using implicit indicator modeling in Leapfrog Geo software. The deposit was broken up into two broad domains with different direction and anisotropy of spatial continuity applied. The resultant geometry shapes were broken up into groups for subsequent grade modeling. The codes applied to these domain volumes are shown in long section view (looking N) in Figure 24-7. An illustrative cross-section showing the generalized dip of the mineralization domains with respect to the meta-sediment/meta-volcanic boundary is shown in Figure 24-8. Finally, the names attached to different mine areas by the Haile Gold Mine staff and used in previous feasibility studies are shown with respect to the domain solids in Figure 24-9. Note that the Horseshoe area corresponds (mostly) to domain code 1, Palomino to domain code 2, and that Mustang corresponds to a small part of domain code 6.



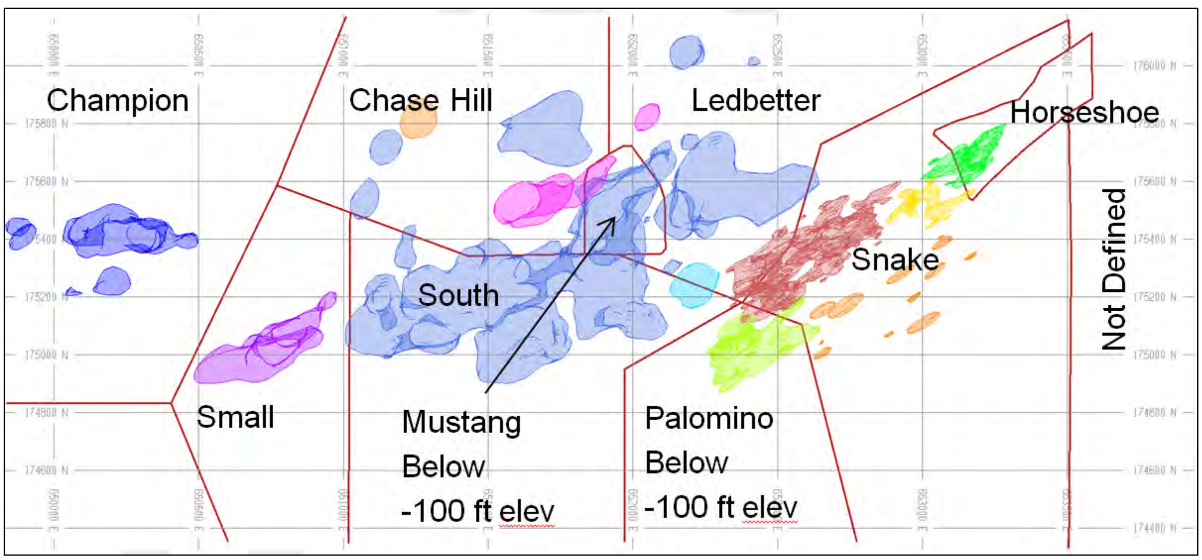
Source: Oceana

Figure 24-7: Long-Section View of Mineralization Envelopes Relative to Reserve Pit Shell



Source: Oceana

Figure 24-8: Cross-Section A-A' Showing Drilling and Main Structural Domains

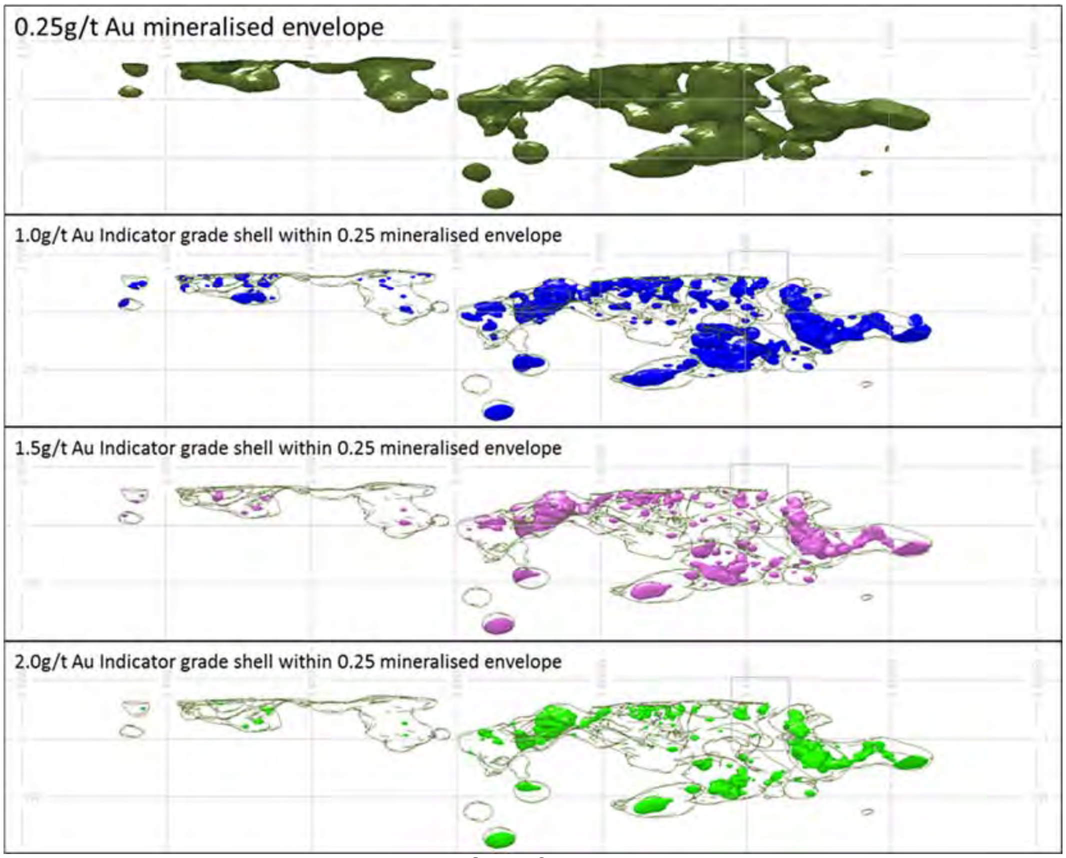


Source: Oceana

Figure 24-9: Mine Area Names Relative to Modelled Grade Domains (Code Values Indicated)

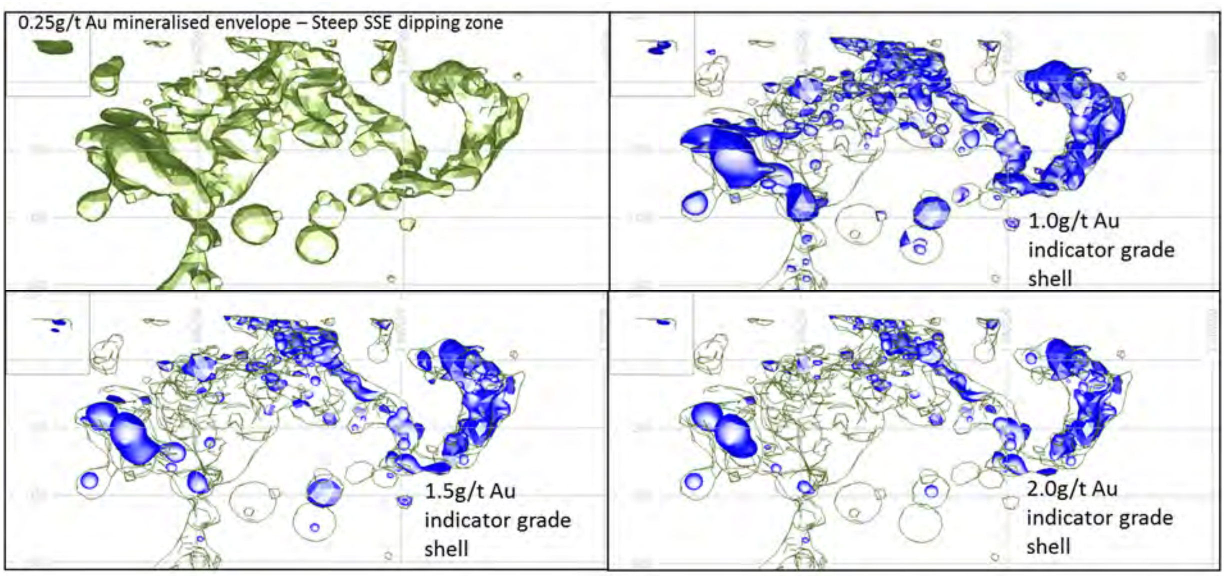
24.14.4 Indicator Grade Shell Geometry Models

Indicator grade shells were constructed within the constraint of the larger 0.25 g/t Au (0.0073oz/st) mineralization envelope at thresholds of 1.0, 1.5 and 2.0 g/t Au (0.029 oz/st, 0.044 oz/st and 0.058 oz/st respectively). The resultant volumes are illustrated below in long-section.



Source: Oceana

Figure 24-10: Long Section Views of Indicator Volumes (1.0, 1.5 and 2.0 g/t) within 0.25 envelope – NNW Zone Only



Source: Oceana

Figure 24-11: Long Section Views of Indicator Volumes (1.0, 1.5 and 2.0 g/t) – SSE Zone Only

24.14.5 Grade Estimates within Indicator Envelopes

Grades were estimated by ordinary kriging with a top cut applied to restrict the influence of high grades. The outlier threshold value, and the distance at which it is applied are shown in Table 24-14 below.

The estimate for domain 1 used an anisotropic search of 4:2:1 (maximum direction vertical) while for all other domains the search anisotropy was 2:2:1. A minimum of 4, maximum of 12, and a maximum of 4 composites per hole were used.

Table 24-14: Search Parameters (Distances in Meters)

Domain	1	2	3	4	5	6	7	8	9	10
Min comps (1)	4	4	4	4	4	4	4	4	4	4
Max comps	12	12	12	12	12	12	12	12	12	12
Max per hole	4	4	4	4	4	4	4	4	4	4
Search Maj	150	150	150	150	150	150	150	150	150	150
Search Int	75	150	150	150	150	150	150	150	150	150
Search Min	35	75	75	75	75	50	50	75	75	75
Rot N	144	70	60	131	145	25	340	340	0	333
Dip N	-85	-25	0	-60	-82	0	-31	-31	-30	-33
Dip E	0	-75	-86	-19	0	25	0	0	0	0
Outlier threshold	60	25	30	60	10	70	1,000	1,000	1,000	1,000
Outlier Distance	-2.5	-5	-5	-2.5	-5	-2.5	1,000	1,000	1,000	1,000

Source: Oceana

(1) Note that the objective of the search strategy was to ensure that all blocks within the domains are estimated, without excessive smoothing. While the minimum composite number requirement is small, there is a post modeling process applied in classification to restrict estimates to appropriately informed blocks.

Variogram parameters used are tabulated in Table 24-15.

Table 24-15: Variogram Parameters (Ranges in Meters)

Domain	1	2	3	4	5	6	7	8	9	10
Nugget	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
Sill 1	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4	0.4
S1 R1	30	25	25	25	25	25	25	25	25	25
S1 R2	20	25	25	25	25	25	25	25	25	25
S1 R3	15	15	15	15	15	15	15	15	15	15
Sill 2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
S2 R1	60	50	50	50	50	50	50	50	50	50
S2 R2	50	50	50	50	50	50	50	50	50	50
S2 R3	30	30	30	30	30	30	30	30	30	30

Source: Oceana

To place the estimates in context, the statistics of input composites are summarized below (Table 24-16). The Horseshoe domain is significantly higher grade than other areas of the deposit, and consequently also contains the highest proportions of composite grades above thresholds of 1.0, 1.5 and 2.0 g/t Au (0.029 oz/st, 0.044 oz/st and 0.058 oz/st respectively).

Grades were only estimated inside grade shell volumes that displayed continuity between holes. Prior to estimation, all grade shells that contained single isolated intersections, were manually removed.

Table 24-16: Composite Grade Grades by Domain within 0.25 g/t Mineralization Envelope

Domain	Length (m)	Max Au	Mean Au	Std. Dev.	cv	>1.0g/t Au	% Composites >1.5 g/t Au	% Composites >2.0 g/t Au
1	3,340	231.8	4.04	12.55	3.11	57%	48%	40%
2	1,929	21.1	1.52	2.38	1.56	40%	30%	20%
3	11,092	110.2	1.68	3.89	2.32	35%	25%	18%
4	788	16.6	1.84	2.54	1.38	43%	34%	26%
5	435	10.1	1.29	1.88	1.46	37%	23%	15%
6	28,185	87.8	1.48	3.66	2.47	34%	24%	18%
7	1,011	4.7	0.58	0.67	1.15	15%	9%	6%
8	2,679	4.1	0.49	0.41	0.84	8%	3%	1 %
9	4,519	6.8	0.59	0.70	1.19	15%	7%	4%
10	192	3.7	0.58	0.67	1.15	20%	6%	40%

Source: Oceana

Table 24-17: Composite Grades by Domain within 1.5 g/t (0.044 oz/st) Grade Shell

Domain	Length	Min Au	Max Au	Mean Au	Std. Dev.
1	1,395.1	0.00	113.7	7.59	11.34
2	262.5	0.76	21.1	4.75	3.74
3	1,881.2	0.00	110.2	6.12	7.78
4	173.5	1.65	17.0	4.85	3.30
5	24.4	1.45	6.7	3.26	2.10
6	5,074.1	0.00	82.7	5.16	6.46
9	22.9	0.72	6.8	4.20	2.19

Source: Oceana

Grades were also estimated into the 0.25 g/t (0.0073 oz/st) envelope outside of the higher grade indicator shells. Blocks outside of the low grade envelope were not estimated.

Blocks on the boundary between the 0.25 g/t (0.0073 oz/st) envelope (LG) and the 1.5 g/t (0.043 oz/st) indicator shell (HG) were diluted by weighting together the LG and HG estimates into a combined Au item (AU15).

Blocks where the 1.5 g/t indicator and 0.25 g/t (0.0073 oz/st) envelope coincide (i.e. where high grade abuts background) are diluted with a background grade of 0.01 g/t.

Bulk density is assigned using lithology based average values adopted from the open pit resource estimates as follows:

Table 24-18: Bulk Density by Lithology

Material Type	In-place dry density dmt/m³	lb/ft³	ft³/t	t/ ft³	t/yd³
Metasediments	2.77	172.93	11.57	0.086	2.33
Metavolcanics	2.60	162.31	12.32	0.081	2.19
Diabase Dike	2.91	181.66	11.01	0.091	2.45
Saprolite	2.14	133.60	14.97	0.067	1.80

Source: Oceana

A block model size of 5 m x 5 m x 5 m was used (16.4 ft x 16.4 ft x 16.4 ft). Two models were generated, a non-rotated model to cover the Mustang and Mill Zone Deep areas and a model rotated to 55° for the Horseshoe area. The respective model orientations allowed for closer modeling of the main mineralization trend and align with the stope orientation directions.

24.14.6 Classification

Estimates were classified taking into account the uncertainty of grade continuity, locational uncertainty in drilling, data spacing, estimation confidence and grade. A cut-off of 1.5 g/t (0.044 oz/st) Au was used. Classifications have been applied in relation to the interpreted continuity of mineralization at this CoG.

Domain 1 (Horseshoe) was classified as Inferred, with the exception of a small zone at the top, eastern part of the domain, where the mineralization is unclassified (not reportable).

Two small portions of domain 6 (Mustang and Mill Zone Deep) were also classified as Inferred. This was limited to areas where drill-holes were dominantly perpendicular to structural trend, and the average distance to the closest three drill-holes was less than 30 m (98 ft). The remainder of the indicator volume is unclassified.

24.14.7 Validation

The models have been visually checked in section and 3D against the input composited sample grades. Additionally, for Horseshoe, swath plots are provided, which compare modelled versus composited sample grade, by bench, for low grade and high grade indicator volumes (Figure 24-12 and Figure 24-13 respectively).

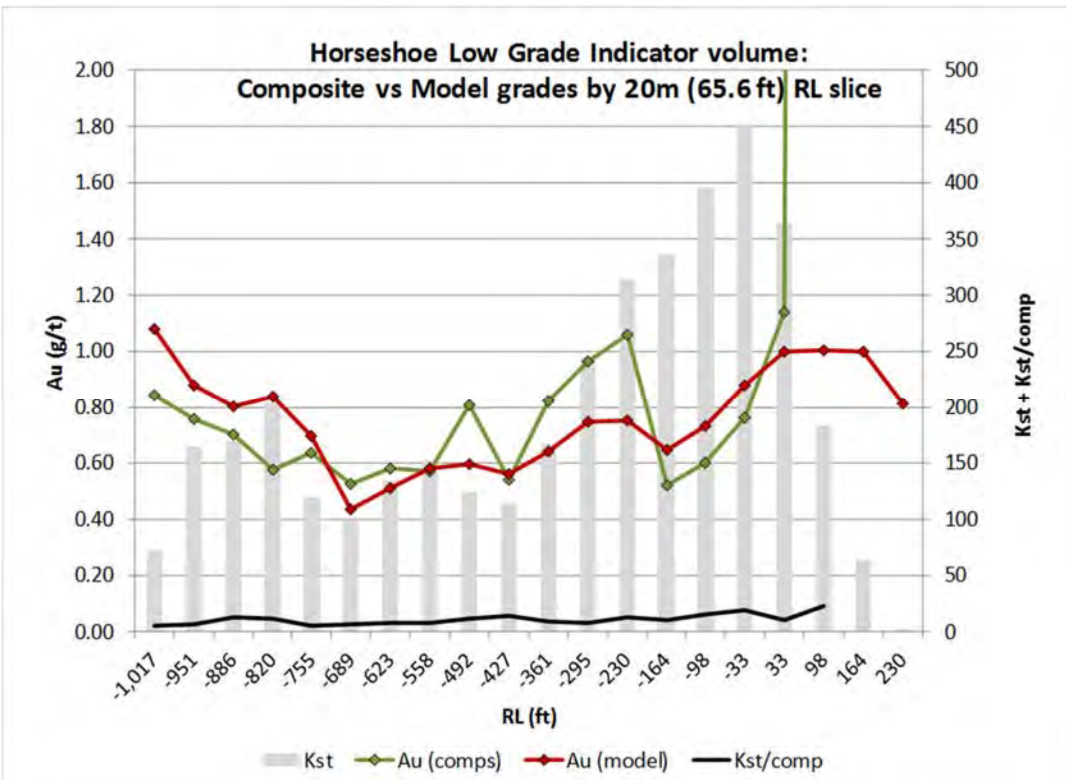


Figure 24-12: Composites vs Modelled Grade for Low Grade Indicator Volume

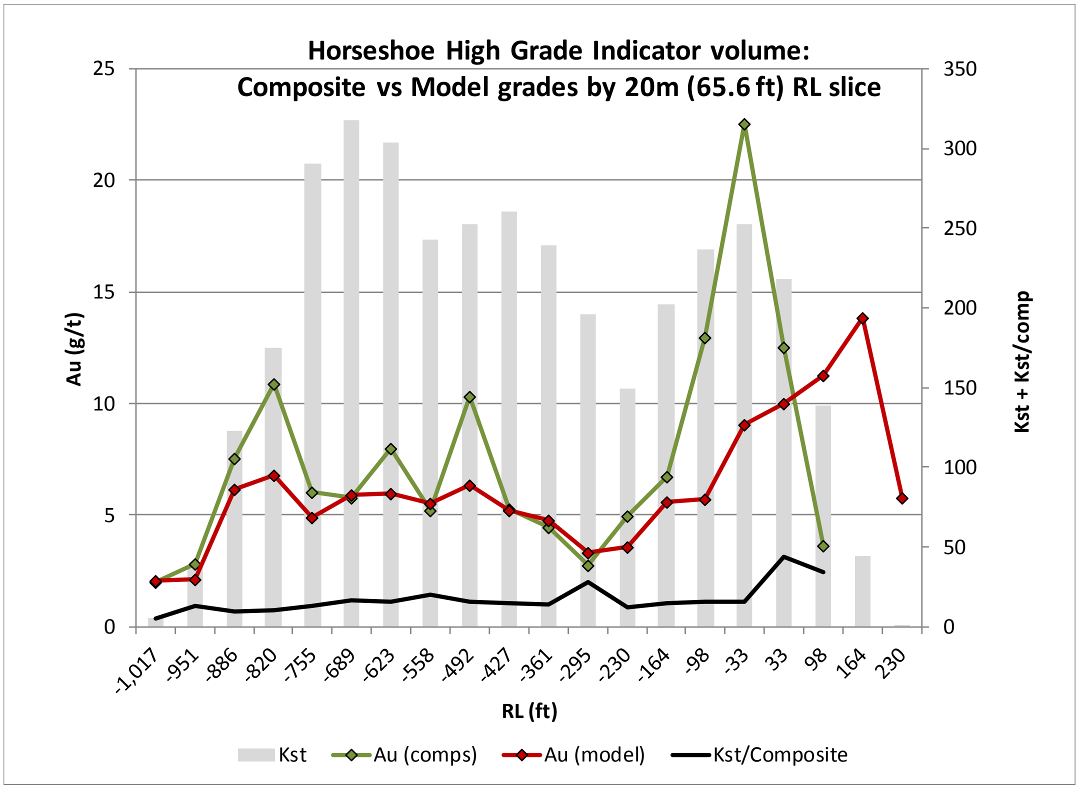


Figure 24-13: Composites vs Modelled Grade for High Grade Indicator Volume

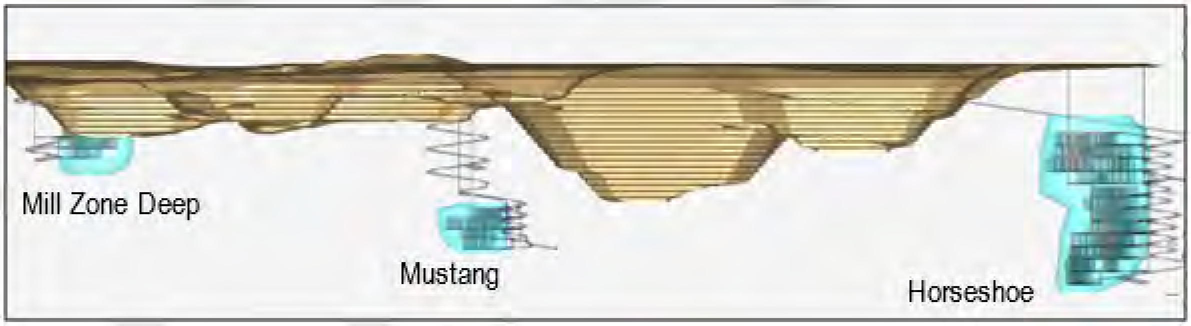
24.14.8 Underground Resource Estimate – A Subset of the Open Pit Reported Resource

The resources discussed in this section, termed the underground target resources, are entirely located below the reserve pit design and almost entirely within the US1,200/oz open pit resource reporting shell. In other words, the underground target resources presented here overlap with the open pit resources and cannot be added to the open pit resource inventories presented in sections 1.1 and 14.2.1 of this report. Importantly, the mining of the underground resources discussed in this section would not preclude mining of the open pit reserves presented in this report. Clarification is provided below.

The intention of section 24 is to evaluate the potential to mine the following subset of the open pit resource by underground methods. However, before any open pit resources would be redefined as underground resources, comparative open pit versus underground evaluations need to be completed.

After due consideration of CoG, geometric and grade continuity and drill spacing, only Mill Zone Deep, Mustang and Horseshoe, as shown in Figure 24-14 were targeted for this study.

The underground target resource is tabulated below in Table 24-19, and is constrained within volumes manually created to envelope the underground stope designs discussed in the mining section. Typically, the resource-constraining volumes extend 20 m (65 ft) beyond the design volumes. All the underground target resource is classified as Inferred.

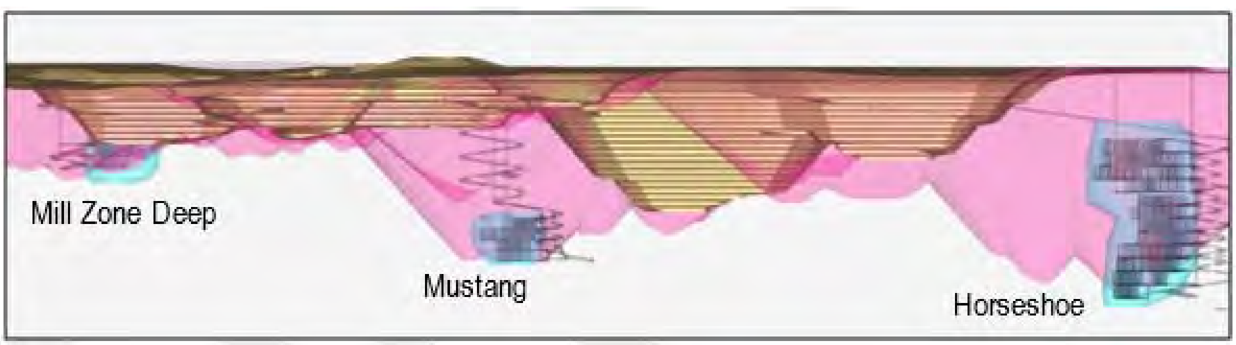


Source: Oceana

Figure 24-14: EW Section with Reserve Pit, UG Development and Resource Volumes (blue)

Other notes regarding the resource:

- 95% of the underground target resource presented in Table 24-19 is already reported as open pit resource in sections 1.1 and 14.2.1 of this report because it lies within the US1,200/oz open pit resource reporting shell (Figure 24-15). This component is presented in Table 24-20 and is classified as Inferred.
- Mining of the underground resources discussed in this section would not preclude mining of the open pit reserves presented in this report and defined by the reserve shell in Figure 24-14. The underground plan presented here mines below/outside of the reserves shell and pillars are left where required to allow for open pit mining.
- In nearly all cases the corresponding open pit classification for the same volume of mineralization is of a higher category (generally Indicated, and in a few cases Measured). This demotion of classification category for the underground target resource relates to the higher CoG applied as well as the need for higher locational certainty of grade.
- Only 5% of the resource reported in Table 24-19 is additional to the Open Pit resource presented in sections 1.1 and 14.2.1 of this report. That is to say that only 5% of the underground target resource is located below the US$1,200/oz open pit resource reporting shell. This component is presented in Table 24-21.



Source: Oceana

Figure 24-15: EW Section with Reserve Pit, UG Development / Volumes and US$1,200/oz Shell (Pink)

Table 24-19: Total Underground Target Resource by Deposit – Inferred Classification

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	4,056	0.16	645
Mustang	847	0.13	111
Mill Zone Deep	337	0.15	49
Total	5,241	0.15	805

Source: Oceana
Resources constrained to volumes expanded around underground design volumes.

Table 24-20: Component of Underground Resource within US$1,200/oz Shell

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	3,752	0.16	600
Mustang	847	0.13	111
Mill Zone Deep	337	0.15	49
Total	4,937	0.15	760

Source: Oceana
Resources constrained to volumes expanded around underground design volumes.

Table 24-21: Component of Underground Resource below US$1,200/oz Shell

0.038 oz/st Au cut-off	kst	oz/st	koz
Horseshoe	304	0.15	45
Mustang	0	0.00	0
Mill Zone Deep	0	0.00	0
Total	304	0.15	45

Source: Oceana
Note that only a very small proportion of the Horseshoe resource, presented above in Table 24-21, is additional to the open pit resource inventory

Infill drilling at Horseshoe is expected to be completed in early September and is designed to convert a proportion of the Horseshoe Inferred Resource to Indicated.

24.15 MINERAL RESERVE ESTIMATES

No underground mineral reserves have been estimated for the Project.

24.16 MINING METHODS

The Project is currently being mined as an open pit mine. The mineralization extends down at depth and outside of the pit extents. This mineralization, not mined by the current feasibility study open pit described earlier in this document is assessed here as an underground mine. As the property is currently under development, the potential underground operation would share infrastructure with the existing mine.

The stopes will be 49 ft wide and stope length will vary based on mineralization grade. A spacing of 82 ft between levels has been used in the Horseshoe area and 66 ft in the Mustang and Mill Zone Deep areas to best fit the mineralization and minimize dilution. The areas are mined bottom to top with the use of sill pillars where necessary. A cemented rock fill is used when necessary (70% of stopes) and non-cemented waste rock fill is used in the remaining stopes. The cemented backfill will have sufficient strength to allow for mining adjacent to filled stopes, thus eliminating the need for dip pillars.

All three underground mining areas will be accessed via decline. Mineralization will be transported from the stopes to the process facility via underground diesel trucks. At Horseshoe and Mustang, two ventilation raises will be installed by conventional raisebore, one serving as intake and secondary egress and one as a dedicated exhaust airway. As levels are developed lower in the deposit, short slot raises will be developed connecting levels for ventilation purposes. At Mill Zone Deep, a single raise is used.

The mine design process used stope optimization within Vulcan™ software to determine potentially mineable areas based on a CoG and minimum mining dimensions. Dilution and recovery were added to the designed tonnage to account for unplanned stope dilution and unrecoverable material within the stope.

The current air permit allows for mining 9,120 st/d. The open pit plan mines 7,000 st/d and therefore the underground operation has been sized to produce 2,120 st/d making up the balance of the permitted capacity. The process facility will require modifications to handle the increased throughput as discussed in Section 24.13.

Access and infrastructure development underground was designed to support the mining method and sized based on mining equipment and production rate requirements. Surface infrastructure is shared with the open pit when possible.

24.16.1 Cut-Off Grade Calculations

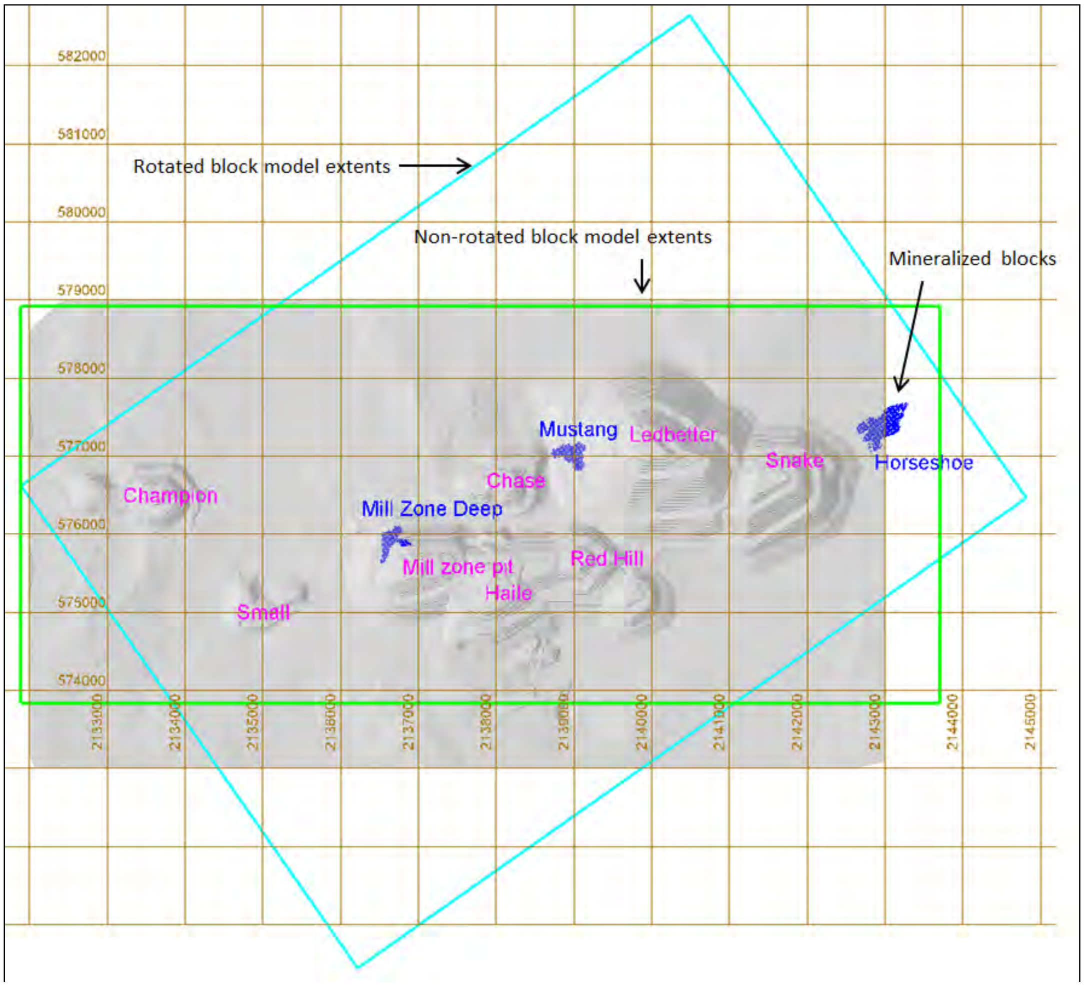
Current estimated project costs and calculated Au CoG are shown in Table 24-22. For mine design purposes, a minimum cut-off of 0.05 oz/st was used.

Table 24-22: Underground Cut-Off Grade Calculation

Parameter	Amount	Unit
Mining cost (1)	45.00	US$/st
Process cost	11.57	US$/st
G&A	3.03	US$/st
Total Cost	**$59.60**	**US$/st**
Gold price	1,300.00	US$/oz
Average AU mill Recovery (2)	88%	
Smelting & Refining	1.28	US$/oz
Cut-off grade (3)	0.05	oz/st

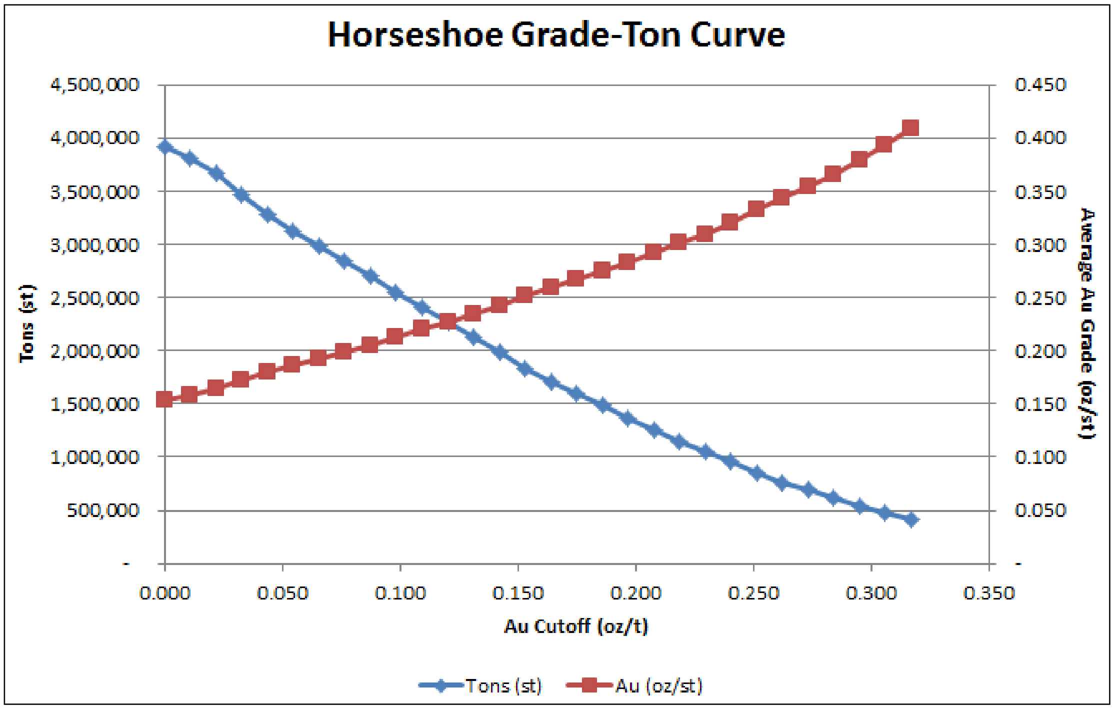
(1) Includes Backfill
(2) Average stated. Variable recovery is expected based on head grade based on the following equation: 100*(1-(0.0583 * (head grade) ^-0.3696)).
(3) 1.5 g/mt
Source: SRK, Oceana

As discussed in Section 24.14, two underground resource block models were generated for the underground potential; one rotated, which was used for the Horseshoe area and the second, non-rotated model, which was used for Mustang and Mill Zone Deep. The rotations of the models fit the mineralization trends in each area. Figure 24-16 shows the two model orientations and the mineralized block locations. Figure 24-17, Figure 24-18, and Figure 24-19 show a grade-tonne curve for the deposit using various Au CoGs for each area. All underground material shown here is classified as Inferred.



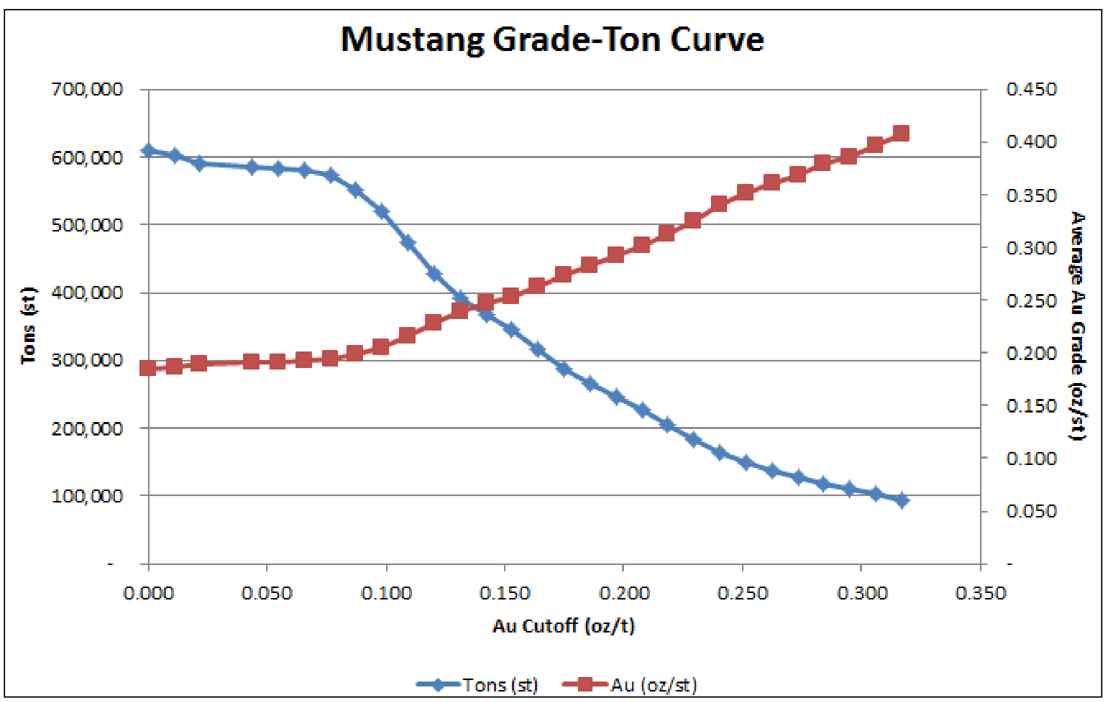
Source: SRK

Figure 24-16: Haile Underground Block Model and Mineralization Extents



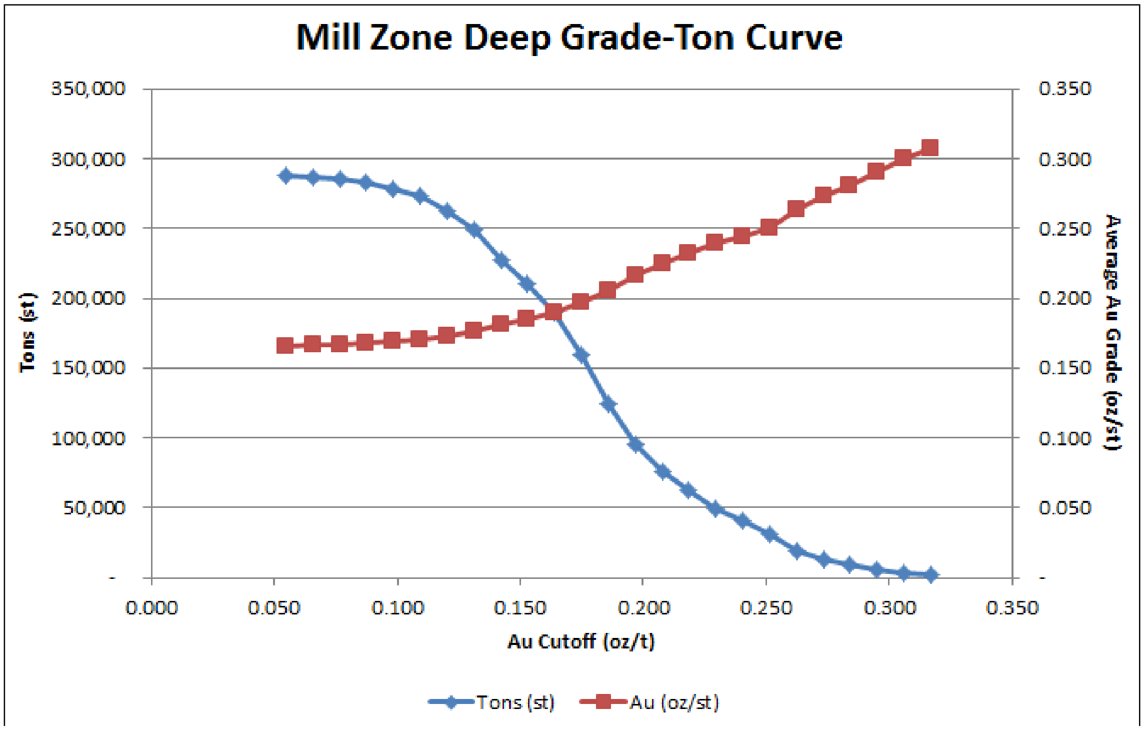
Source: SRK

Figure 24-17: Horseshoe Underground Model Grade/Ton Curve based on Au Cut-off



Source: SRK

Figure 24-18: Mustang Underground Model Grade/Ton Curve Based on Au Cut-off



Source: SRK

Figure 24-19: Mill Zone Deep Underground Model Grade/Ton Curve based on Au Cut-off

24.16.2 Geotechnical

Golder completed an open pit testing program in 2010 (Golder, 2010) consisting of ten boreholes for a total of 6,880 ft of core. Of this, three holes (2,244 ft of core) located in the Snake pit area are considered in close proximity to the Horseshoe underground area and have been considered in this analysis. Of Golder's 2010 characterization program, there were seven UCS tests and 114 PLT tests.

A follow-up characterization program was completed by Golder in 2012 specifically for the underground design program. This program included four boreholes for a total of 7,110 ft of core, and there were 12 UCS tests and 321 PLT tests.

In 2016, SRK collected 70 samples as part of Oceana's exploration drilling program. This program included six boreholes for a total of 5,135 ft of core. There were 21 UCS tests, eight TCS tests and two BTS tests totaling 31 samples. Information in this section is based on SRK's review of the Golder data and the SRK-tested samples.

<u>Upper Fracture Zone</u>

Three different geotechnical zones were defined using the RQD parameters and Bartons Q' rock mass classification from the core logging data. This data came from three sources: the Golder UG report (2012), the existing drillhole data and the current exploration drilling.

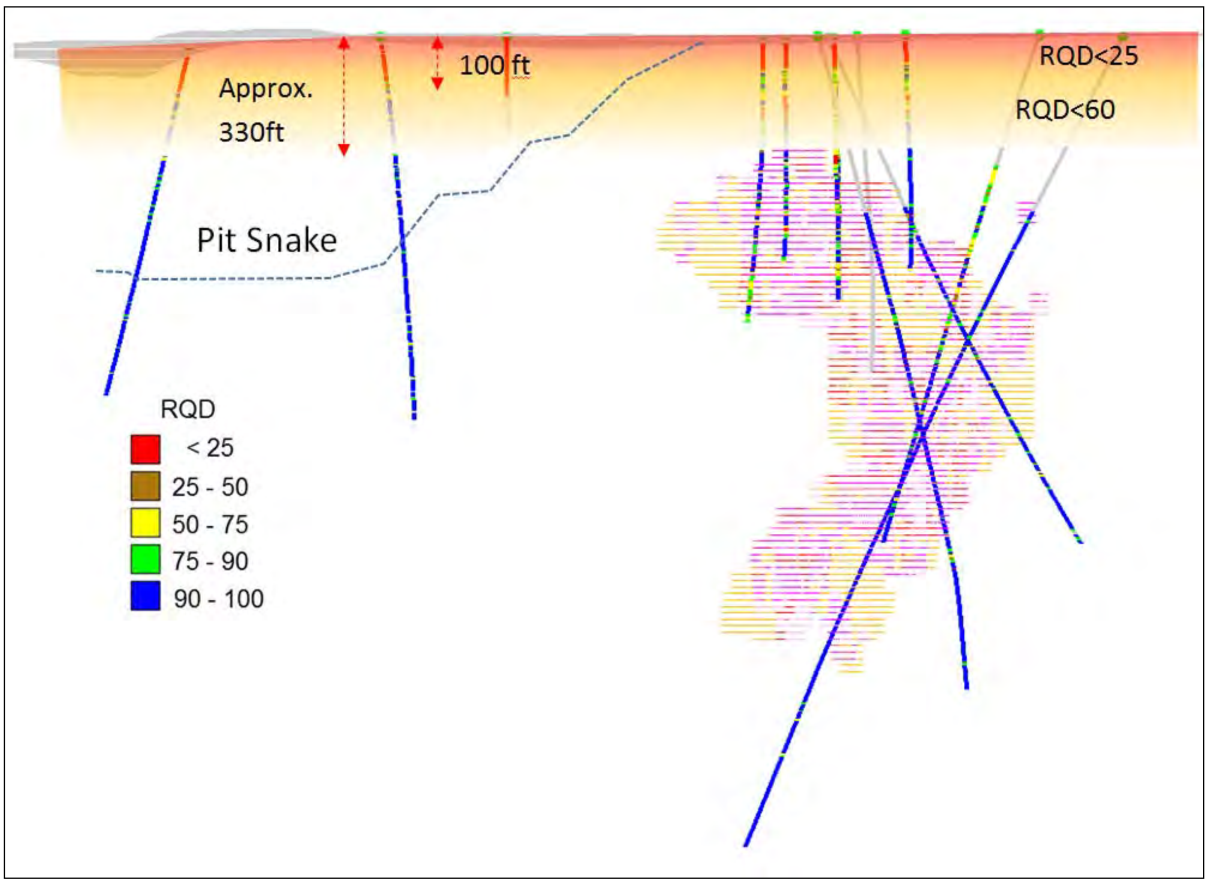
RQD parameters and Barton's Q' rock mass classification was used to define these zones. Table 24-23 shows the variation in RQD and Q' values for two of these zones, the upper fracture zone (up to 330 ft in depth) and non-fracture zone (below 330 ft in depth). The third zone is the near-surface saprolite zone representing material shallower than 100 ft.

Figure 24-20 shows a representative NW-SE cross section indicating the distribution of core holes. The different colors along the core holes represent the RQD parameters, where the red zone represents RQD values lower than 20 (saprolite), brown and yellow represent RQD values between 25 and 75 typically extending up to about 330 ft in depth, and the green and blue represents RQD values over 75 typically extending deeper than 330 ft.

Table 24-23: RQD and Q' Barton Fracture Zone

Date Report	Zone	RQD (%)			Q' (1)		
		Min (2)	Mean (4)	Max (3)	min (2)	Mean (4)	Max (3)
Golder UG 2012	Non Fracture	89	94	100	11.1	24.2	33.3
Resource Hole	Fracture	23	50	73			
	Non Fracture	61	82	94			
Oceana 2016	Fracture	40	55	76	3.9	6.8	9.7
	Non Fracture	77	88	96	7.1	13.0	17

(1) Q' is calculated by setting the Jn = 6 in the Q equation.
(2) The minimum value corresponds to of the Percentile 20% unweighted
(3) The maximum value corresponds to of the Percentile 80% unweighted
(4) Calculate the weighted average values.
Source: SRK



Source: SRK

Figure 24-20: Diagram 2D of Geomechanical Domain

Stope Geotechnical Design

Empirical methods of stope design have been employed to estimate stability conditions. The Stability Graph Method (Mathews et al., 1981) as modified by Potvin et al. (2001) and based on more than 480 case histories worldwide, has been used to size stopes so they remain stable during mining.

The stability graph method plots the stability number (N') on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope wall or back on the x axis. The stability number is calculated based on the rock mass quality (Q' system), geologic structure and induced stress conditions. The estimated stability condition is then determined from the graphs. The various stability graph methods have slight variations in the position of the stability lines and the terms used to describe the stability zones, however, overall the approaches are similar.

The stable stope dimensions were estimated using the Potvin (2001) method. The sized of the stopes in this study are:

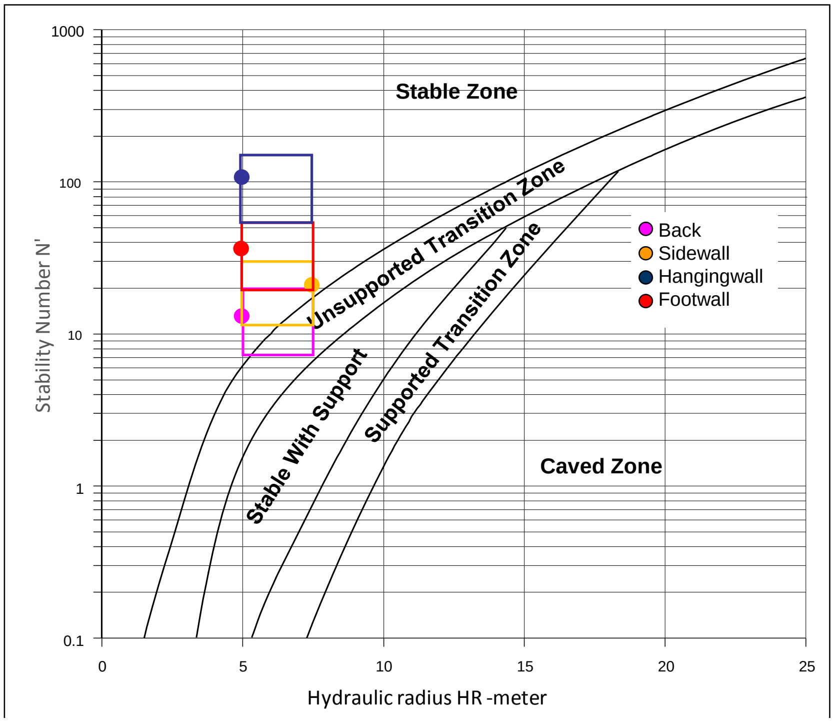
- 100 ft high;
- 49 ft wide; and
- 100 ft long.

Stability of the back, hangingwall (HW), footwall (FW) and side walls were checked considering the range of rock mass properties. Table 24-24 provides a summary of the stability numbers used in the analysis. Examples of this are shown in Figure 24-21 and Figure 24-22, where results show the stope stability condition for different depths (490 ft and 1,310 ft). The rectangle zone represents the lower boundary (Q' = 7), median (Q'=13) and upper boundary (Q' = 17) estimates of rock mass quality (N') while the vertical line represents the different hydraulic radius, in this case 5 m (16 ft) (back, HW and FW) and 7.5 m (25 ft) (side). These charts demonstrate that the selected size of the stopes are sufficient to maintain stability during mining.

Table 24-24: N' Parameters Used in Stope Design

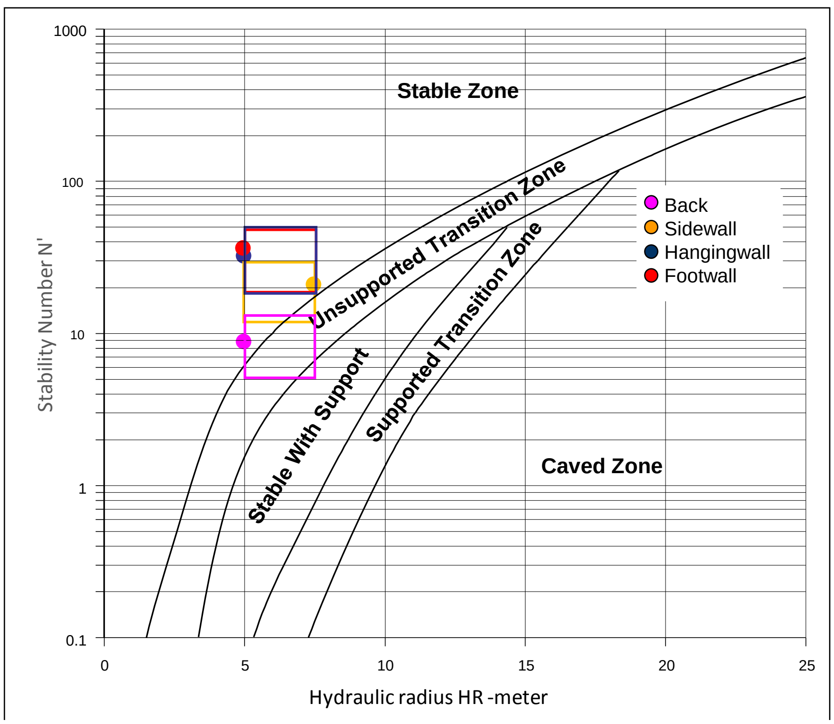
Stope		Factor			Stability Number N'		
Depth (ft)	Zone	A	B	C	Min (Q' = 7)	Mean (Q' = 13)	Max (Q' = 17)
490	Back	1	0.5	2	7.0	13.0	17.0
	Side	1	0.2	8	11.2	20.8	27.2
	Hangingwall	1	1	8	56.0	104.0	136.0
	Footwall	1	0.5	5.5	19.3	35.8	46.8
985	Back	0.9	0.5	2	6.5	12.1	15.9
	Side	1	0.2	8	11.2	20.8	27.2
	Hangingwall	0.5	1	8	25.3	47.0	61.5
	Footwall	1	0.5	5.5	19.3	35.8	46.8
1,310	Back	0.7	0.5	2	4.7	8.7	11.4
	Side	1	0.2	8	11.2	20.8	27.2
	Hangingwall	0.3	1	8	17.2	32.0	41.8
	Footwall	1	0.5	5.5	19.3	35.8	46.8

Source: SRK, 2016



Note: Figure shown in meters
Source: SRK, 2016

Figure 24-21: Empirical Stope Design Chart (Potvin, 2001): Haile Stope 490 ft Depth



Note: Figure shown in meters
Source: SRK, 2016

Figure 24-22: Empirical Stope Design Chart (Potvin, 2001): Haile Stope 1,310 ft Depth

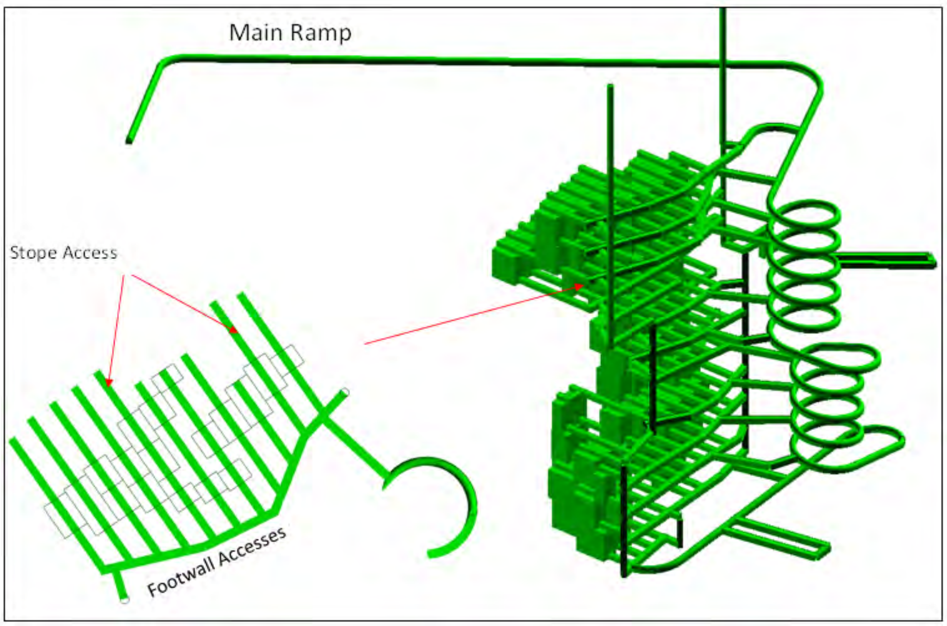
Ground Support Barton Method

Ground support requirements have also been estimated using empirical support charts developed by Barton (1974). The method relates the rock mass quality (Q) to the equivalent dimension of the excavation (De). De is the ratio of the excavation width (D) to the excavation support ratio (ESR) index. The ESR value index relates to the use of the excavation from the degree of safety required. Values ranges from 0.8 for underground nuclear power plants and facilities with high public traffic to temporary mining excavations (3 to 5).

SRK considered ESR of 1.6 to 2.0 for permanent mining excavations (Barton, 1980). The equivalent dimension is determined from the formula:

$$\boldsymbol{D_e = \frac{D}{ESR}}$$

Figure 24-23 show the main underground layout with the different type of excavation: main ramps and ventilation raises, footwall access drifts, and short-term stope access drifts. Table 24-25 summarizes the equivalent dimension used to estimate ground support requirements.



Source: SRK, 2016

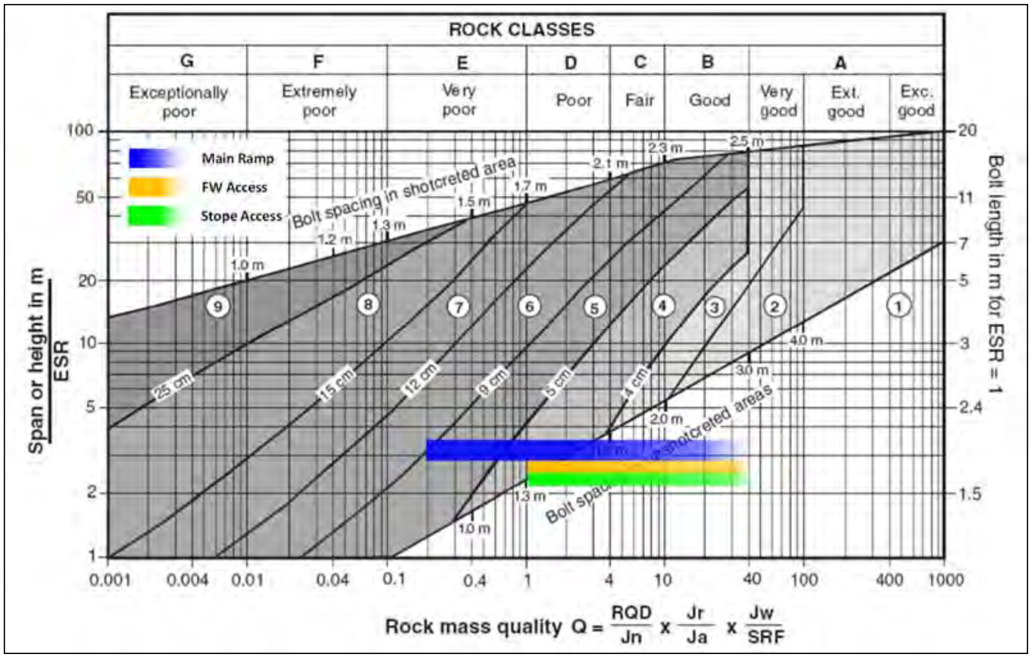
Figure 24-23: Example Underground Layout

Table 24-25: Barton Parameters for Different Types of Excavations

Excavation	Type of Excavation	Opening Dimensions W x H (ft)	ESR Min	ESR Max	D	D_e Min	D_e Max
Main Ramps	Long Term (6 year)	16 x 18	1.6	2.0	5.5	3.4	2.8
Footwall Accesses	Medium Term (1 year)	16 x 16	1.6	2.0	5.0	3.1	2.5
Stope Accesses	Short Term (1 month)	15 x 15	1.6	2.0	4.5	2.8	2.3

Source: SRK, 2016

Estimated support categories for the various development types are shown in Figure 24-24 and summarized in Table 24-26. Note that the low Q values for the main ramp consider that the ramp goes through the upper fracture zone less than 330 ft from the surface.



Source: SRK, 2016Note: Figure shown in meters

Figure 24-24: Estimated Support Categories Based on Index Q

Table 24-26: Preliminary Support – Barton Method

Q’	Rock Classes	Support Categories	Support Recommendation	Excavation
>10	Good	CS1	Unsupported	-
4 - 10	Fair	CS1	Spot bolting	-
1 - 4	Poor	CS4	Bolt Spacing 4.9 to 5.9 ft and 4 ft x 4 ft spacing for #6 mesh	Main Ramp FW Access
0.6 - 1	Very Poor	CS4	Bolt Spacing 3.3 to 3.9 ft and 4 ft x 4 ft spacing for #6 mesh	Main Ramp
0.2 – 0.6	Very Poor		Bolt Spacing 4 ft x 4 ft spacing for #6 mesh, 2 to 3 inch shotcrete	Main Ramp

Source: SRK, 2016

The length of the bolts is estimate through using Barton (1998) with the following equation:

$$L_b = 2 + 0.15\frac{D}{ESR}$$

The required bolt lengths are summarized in Table 24-27.

Table 24-27: Required Bolt Length – Barton Method

Excavation	Dimensions W x H (ft)	ESR		D	Lb (ft)	
		Min	Max		Min	Max
Main Ramp	16 x 18	1.6	2.0	5.5	7.9	8.2
Footwall Accesses	16 x 16	1.6	2.0	5.0	5.9	7.9
Stope Access	15 x 15	1.6	2.0	4.5	4.9	5.9

Source: SRK, 2016

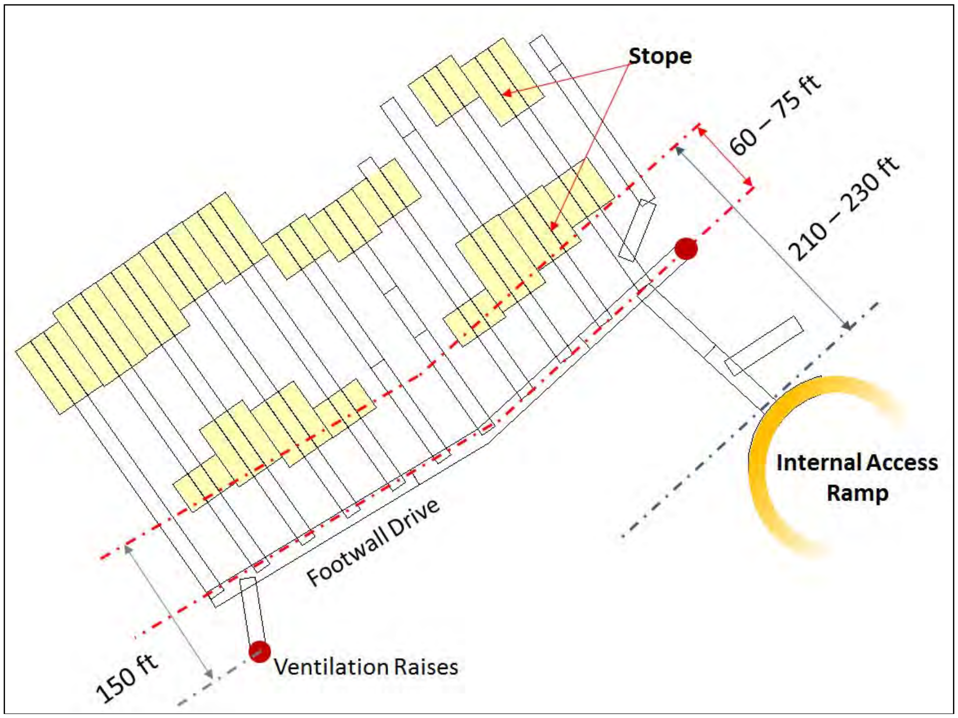
Offset Distances and Crown Pillars

Offset distances for development and raises were estimated based on anticipated rock mass quality and stress conditions. The offset distances from the nearest stope are shown in Figure 24-25 and summarized in Table 24-28 for the various infrastructure components. The distances should be evaluated in the next study level through 3D stress analyses.

Table 24-28: Minimum Offset Distances for Mine Design

Infrastructure	Minimum Distance (ft)
Internal Access Ramps	210 - 230
Footwall Drives	60 - 75
Ventilation Raises	150

Source: SRK, 2016



Source: SRK, 2016

Figure 24-25: Example Mine Design Level

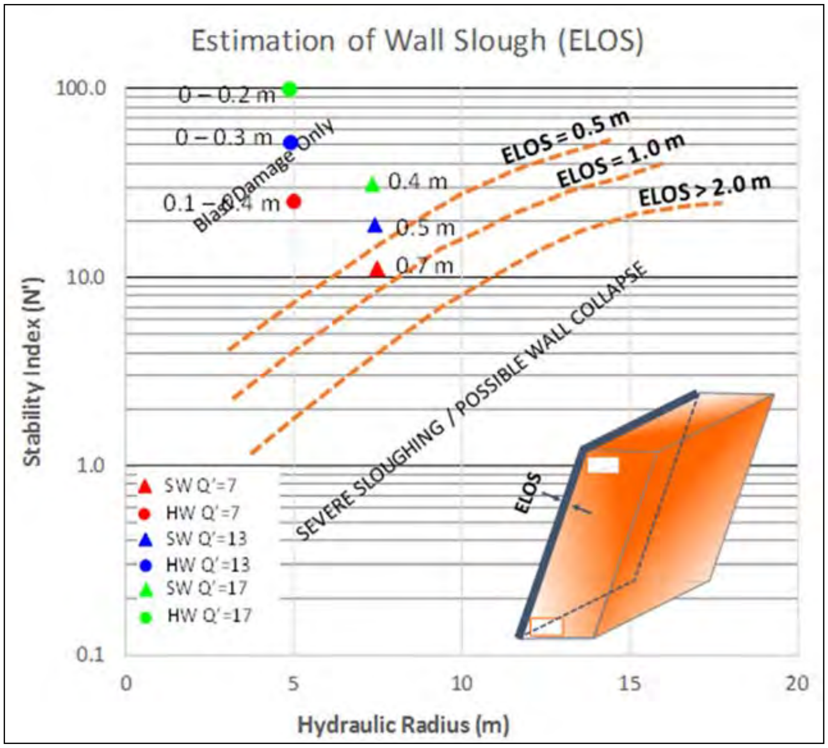
Stability of the crown pillar above mining depends on the competency of the rock mass compared to the exposed mining area. The shallowest mining is currently planned at be approximately 330 ft below ground. The mined exposure area is approximately 400 to 500 ft along strike (NE-SW direction) and 100 to 230 ft perpendicular to strike (NW-SE direction). The mine plan assumes sequential mining of the stopes (HRB=5) with tight backfill and therefor limits the exposure area.

The rock quality of the crown pillar area consists of 100ft of weak soil-like saprolite followed by 230 ft of fair quality fractured rock (Q = 6.8, N'=8-20). Given the stope width of 49 ft, the width to height ratio of the crown pillar in competent ground is greater than 4.5, which is considered stable. SRK recommends additional analyses at the next design phase to assess stability of the crown pillar and any anticipated surface settlement. With a competent crown pillar, any surface settlement is anticipated to be minimal.

Dilution

The potential for dilution has been estimated for the stope hangingwall and sidewall using the equivalent linear overbreak/slough (ELOS) method developed by Clark (1998). The method is similar to the empirical stope stability charts with a stability number (N') plotted on the vertical axis against the hydraulic radius of the stope wall or back on the horizontal axis. The chart includes lines of potential dilution in terms of average thickness over the entire hangingwall or sidewall.

Figure 24-26 and Table 24-29 show the ELOS estimation for the hangingwall and sidewall zones and the estimated ELOS dilution.



Source: SRK, 2016
Note: Figure shown in meters

Figure 24-26: Empirical ELOS Estimate – Hangingwall and Sidewall

Table 24-29: ELOS Dilution for Each Zone and Different and Depths

Slope		ELOS			Dilution (ft)
Depth (ft)	Zone	Min Q'=7	Mean Q'=13	Max Q'=17	
490	Hanging Endwall	0.33 ft	0 ft	0 ft	0 – 0.33
	Sidewall	2.30 ft	1.64 ft	1.31 ft	1.31 – 2.30
980	Hanging Endwall	1.0 ft	0.66 ft	0.33 ft	0.33 – 1.00
	Sidewall	2.30 ft	1.64 ft	1.31 ft	1.31 – 2.30
1,300	Hanging Endwall	1.31 ft	1.0 ft	0.66 ft	0.66 – 1.31
	Sidewall	2.30 ft	1.64 ft	1.31 ft	1.31 –2.30

Source: SRK, 2016

Secondary stopes would experience sidewall dilution from backfilled stopes. For this study, the backfill ELOS dilution is estimated to be 1.64 ft.

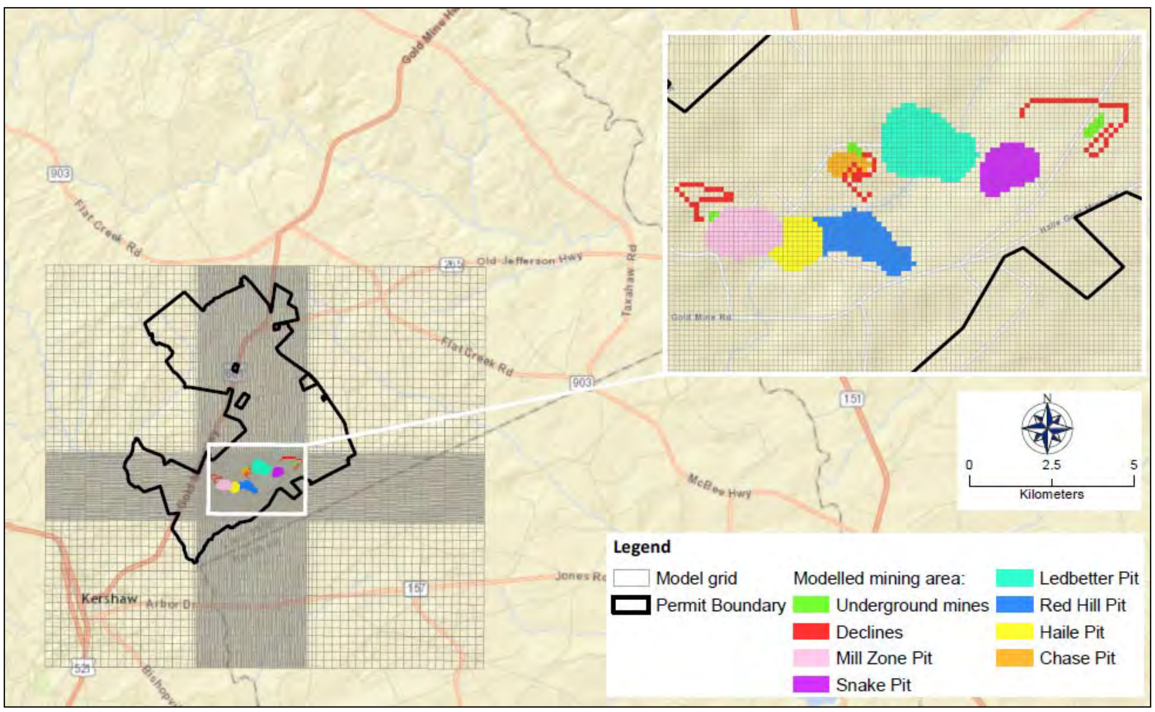
24.16.3 Hydrological

Four aspects of mine hydrology associated with the proposed underground development have been assessed:

- Groundwater inflow and management requirements during sinking of declines and mining of stopes;
- Extent of water table drawdown in response to groundwater management for the proposed underground operation;
- Stormwater control and management to protect the mine portals; and
- Potential water quality issues associated with mine water control.

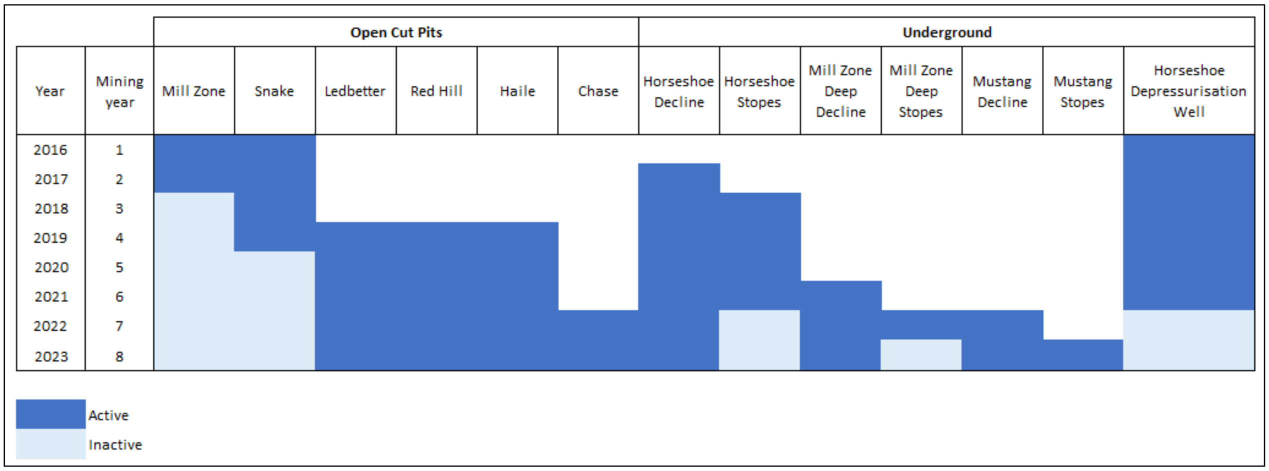
Groundwater Inflow Management

Numerical modeling of the three proposed underground areas (Horseshoe, Mustang and Mill Zone Deep) has been undertaken to assess possible ranges in groundwater inflows to the mines. MODFLOW-SURFACT code has been used to represent the sub-regional groundwater system, including interaction of the underground operations with open pit mine developments at Haile. A detailed description of the numerical model is presented in CDM Smith, 2016. Figure 24-27 presents a locality plan and the numerical model domain, and the simulated mining schedule is presented as Figure 24-28.



Source: CDM Smith

Figure 24-27: Numerical Model Domain



Source: CDM Smith

Figure 24-28: Proposed Haile Mining Schedule (Underground and Open Pit)

The model is a simplified representation of Haile hydrogeology, and is based on the existing mine geological model, hydrogeological conceptualizations developed from work completed for existing open pit dewatering and depressurization works, and mine designs presented in this document. The model has not been calibrated, but hydraulic properties for the different hydrostratigraphic units (HSUs) represented have been applied consistent with the results of aquifer testing conducted on site since 2009.

In general terms the following interactions will occur between open cut and underground mine developments (refer to Figure 24-28 for the mine schedule):

- The Horseshoe development is located very near Snake Pit and dewatering of Snake Pit will impose on groundwater inflows to the upper levels of the decline and stopes, where weathered metavolcanics will be encountered, which are expected to have a higher permeability compared to weathered metasediments and unweathered basement.
- The Mill Zone Deep development is located very near Mill Zone Pit, but dewatering of Mill Zone Pit will have ceased prior to commencement of the proposed underground development. The decline and upper-most stoping level is expected to predominantly encounter weathered metasediments and unweathered basement.
- The Mustang development is located very near Chase Pit (between Snake and Mill Zone Pits), but dewatering of Chase Pit may not have commenced prior to commencement of the proposed underground development. It can be expected that dewatering activities at other pits and underground developments will impose to some extent on groundwater inflows to the Mustang underground area. The upper decline is expected to encounter weathered metavolcanics.
- The stopes will be backfilled with waste rock (cemented and uncemented) as mining progresses, but the declines will remain open. Stoping will proceed bottom up at all three proposed mines.

Table 24-30 presents details of the proposed underground mines used in development of the numerical mode.

Table 24-30: Underground Mine Details

Parameter	Horseshoe	Mustang	Mill Zone Deep
Mine life (yrs)	4	1	4
Elevation – top of highest stope (ft bgl) (1)	510	1,000	510
Elevation – top of lowest stope (ft bgl)	1,410	1,250	675
Stopes per level (2)	15	4	4
Stope dimensions (2) – length / width / height (ft)	98 / 49 / 66	197 / 49 / 66	98 / 49 / 66
Stoping levels	12	4	3
Time to mine each stope (2) (days)			7
Stopes open at any one time			3 to 4

Source: CDM Smith

(1) Feet below ground level
(2) On average

Table 24-31 presents brief details of underground mining scenarios simulated using the numerical model (other scenarios have been simulated, refer to CDM Smith, 2016 for details).

Table 24-31: Model Scenarios

Scenario	Model Description	Purpose
1	Base case (underground developments and open pits)	Predictions for open pit and underground developments combined
2	With Horseshoe decline depressurization well	Water management option to reduce the generation of contact water which will require treatment
3	With dike represented at Horseshoe	To provide an upper bound estimate of groundwater inflow

For the base case scenario (Scenario 1, Table 24-31), the following presents numerical model predictions in relation to groundwater inflows to the proposed underground mine developments.

Horseshoe

The shallow decline (from 0 to around 425 ft vertical depth) is predicted to be the source of most groundwater entering the underground, peaking at an average of around 376 gpm (2,050 m^3/day) at around day 480 LoM [1](Figure 24-29), because the development encounters primarily weathered metavolcanics in this depth range. Inflows to the decline are predicted to decrease from the peak flow to around 184 gpm (1,000 m^3/day) through to the end of mining at Horseshoe.

Stoping occurs predominantly in weathered metasediments or unweathered basement, and the model predicts comparatively minor groundwater inflows to the stopes and drainage to the decline due to these rocks having a lower permeability (with the exception of stoping level 1, which occurs in weathered metavolcanics; Figure 24-29, around day 1,080 LoM).

[1] Life of Mine (LoM) for the proposed underground development

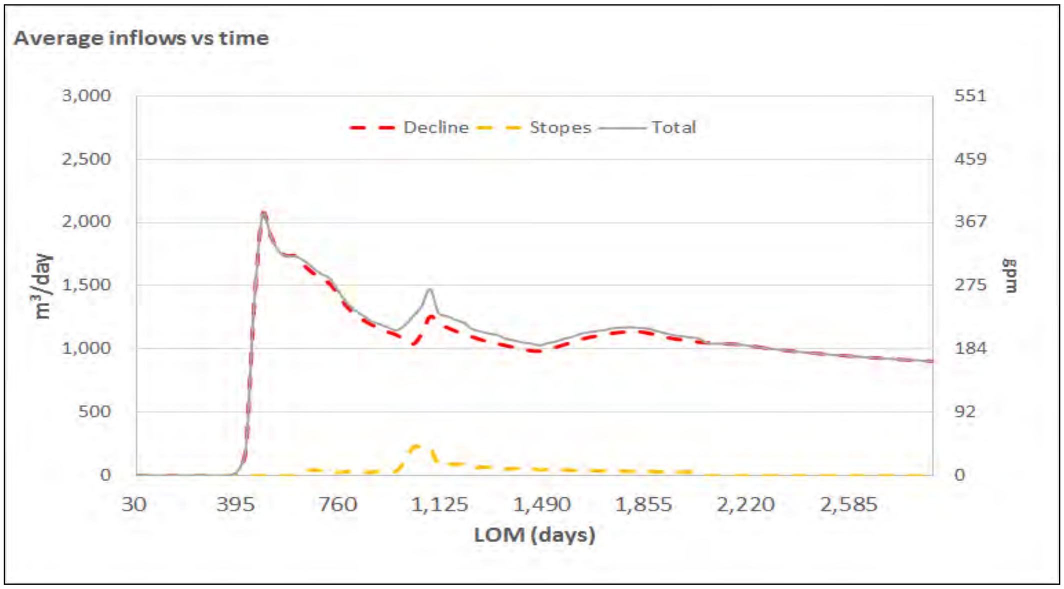


Figure 24-29: Predicted (base case) Groundwater Inflow Rates to the Horseshoe Development

<u>Mustang</u>

As for the Horseshoe development, the shallow Mustang decline (from 0 to around 425 ft vertical depth) is predicted to be the source of most groundwater entering the proposed mine due to weathered metavolcanics being encountered, peaking at around 60 gpm (330 m^3/day) at around day 2,580 LoM (Figure 24-30). The predicted inflow rate to the mine (shallow decline in particular) is much lower than for Horseshoe due the effect of dewatering and depressurization activities at the Ledbetter pit and other nearby open pit mines.

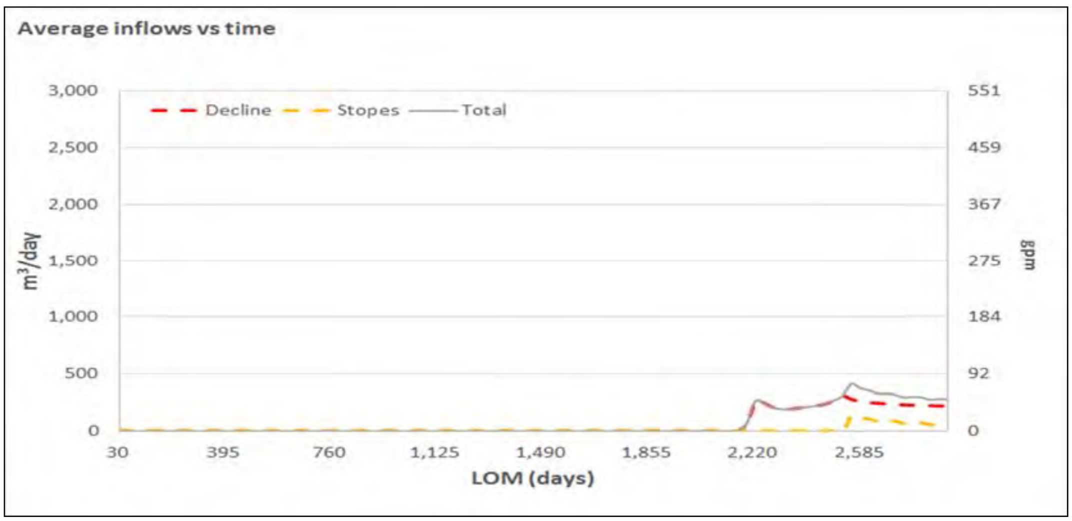


Figure 24-30: Predicted (base case) Groundwater Inflow Rates to the Mustang Development

Mill Zone Deep

The shallow decline (from 0 to around 425 ft vertical depth) is predicted to be the source of most groundwater entering the proposed mine, peaking at an average of around 125 gpm (700 m^3/day) at around day 2,220 LoM (Figure 24-31), again because the development encounters primarily weathered metavolcanics in this depth range. Inflows to the decline are predicted to decline from the peak flow to around 92 gpm (500 m^3/day) through to the end of mining at Mill Zone Deep.

The predicted inflow rate to the mine (shallow decline in particular) is higher than that predicted for the Mustang development because mining at the Mill Zone pit will have been completed a number of years prior to commencing mining of Mill Zone Deep, and groundwater recovery will have commenced in response to this. The groundwater inflow effect of mining of stoping level 1, which occurs in weathered metavolcanics, can be seen in Figure 24-31 at around day 2,490 LoM.

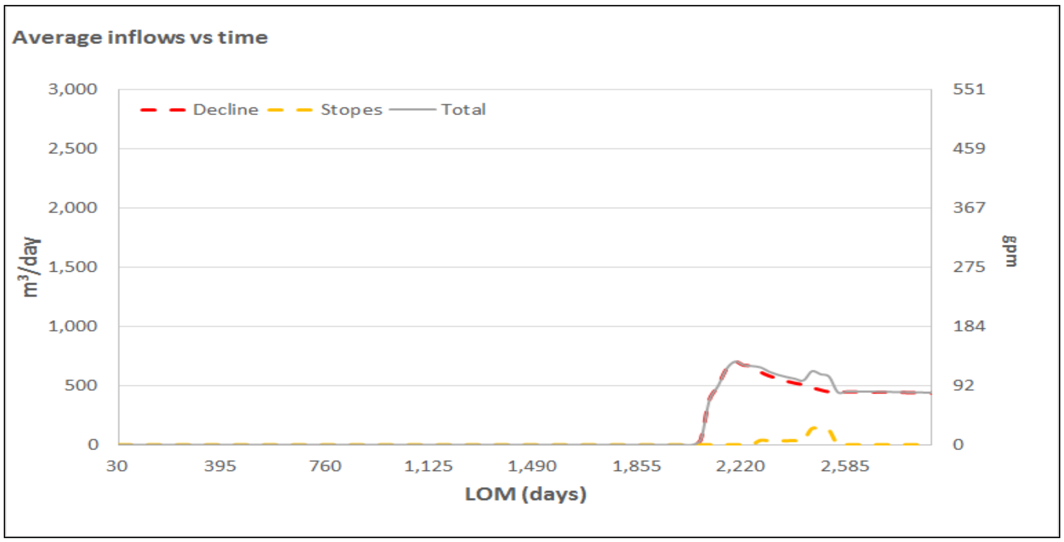


Figure 24-31: Predicted (Base Case) Groundwater Inflow Rates to the Mill Zone Deep Development

Horseshoe Depressurization Wells

A water management scenario (Scenario 2, Table 24-31), involving depressurization of Horseshoe's shallow decline by targeted pumping from the weathered metavolcanic unit (Figure 24-32 for the depressurization well locality plan) has been simulated.

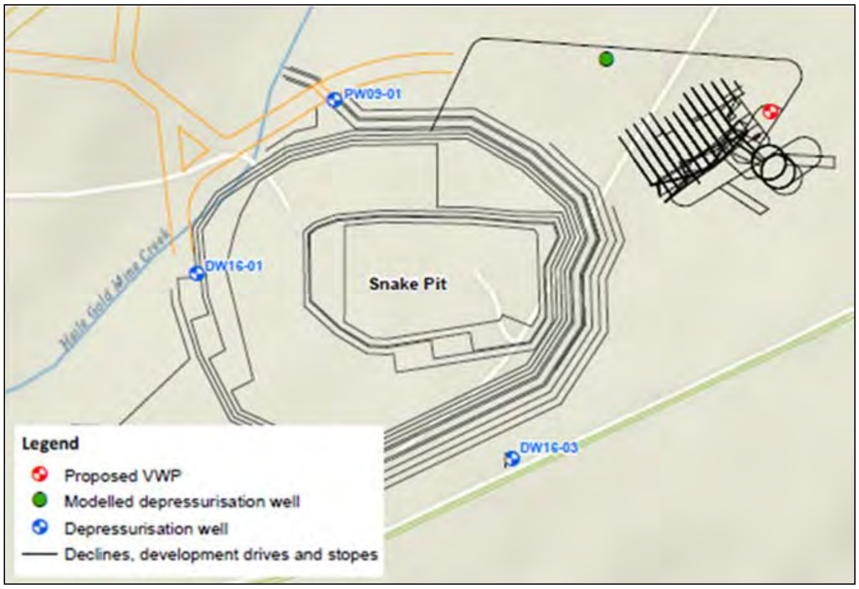


Figure 24-32: Location of Modeled Decline Depressurization Well (Scenario 2)

The model predicts that inflows to the Horseshoe decline and stopes will be reduced by 30 to 10%, equivalent to 92 to 46 gpm (500 to 250 m³/day; Figure 24-33). However, to achieve this outcome the depressurization well would need to pump at around 184 and 92 gpm (1,000 and 500 m³/day; Figure 24-33) for a period prior to commencing the decline. The decline in pumping rate (and decline inflow capture) shown on Figure 24-33 over time occurs because the decline will essentially consume the available drawdown at the depressurization well resulting in well abstractions declining to below 10 gpm (50 m³/day) within around 3 years.

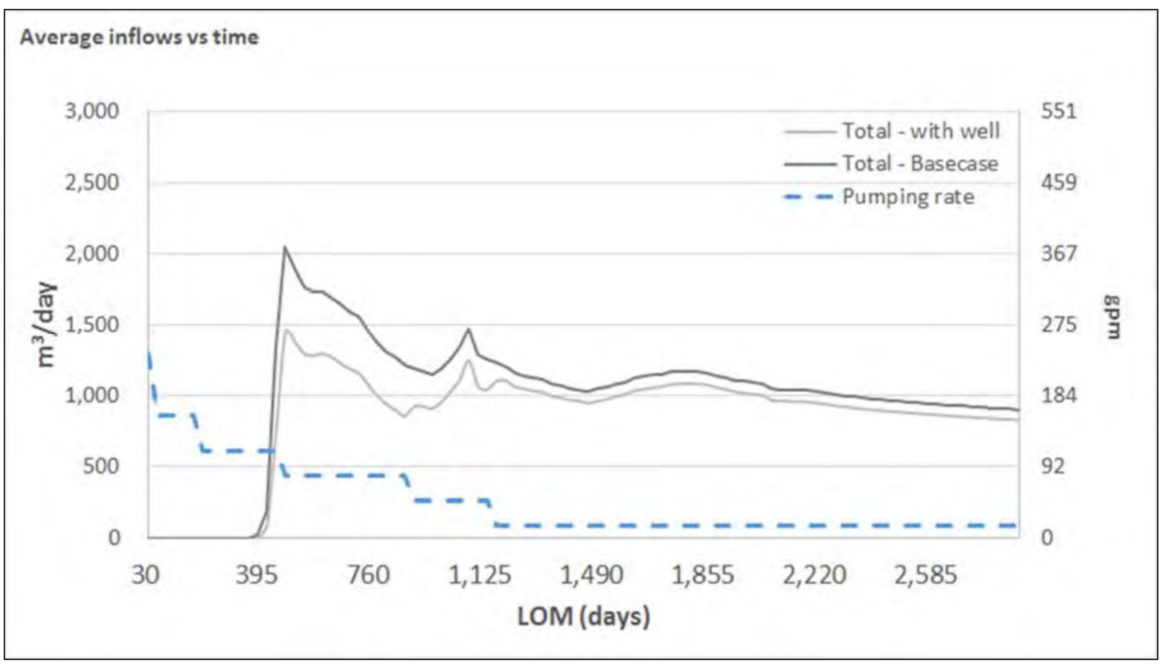


Figure 24-33: Predicted Groundwater Inflow Rates to the Horseshoe Development with Effect of Depressurization Well (Scenario 2)

Dikes

There is the potential for dikes encountering mineralization targeted by the proposed underground mines to be associated with enhanced secondary porosity (as a halo of native basement alteration around the dikes and/or weathering of the intrusive rocks themselves). Model scenario 3 was simulated to test the pattern and scale of groundwater inflows if a dike having the same hydraulic properties as weathered metavolcanics were encountered by the Horseshoe development. Figure 24-34 presents the predicted inflow rates.

As shown on Figure 24-34, the effect of a dike (representing enhanced secondary porosity) on groundwater inflows to an underground development at Haile is predicted to give rise to higher and more erratic inflows to the decline and stopes when compared to the base case, particularly in regard to the lower stopes and deeper decline intersections. The volumes of water that could be generated by intersecting dikes in the underground developments will of course depend on the degree to which secondary porosity is present within the intrusive rocks and any alteration zone in the native rocks.

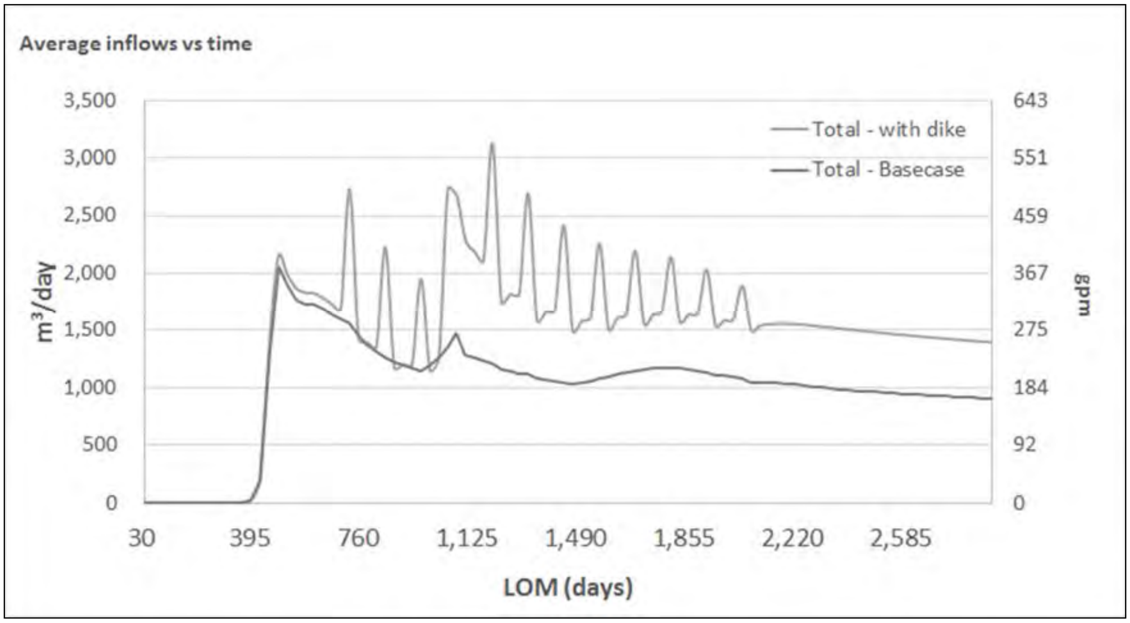


Figure 24-34: Predicted Groundwater Inflow Rates to the Horseshoe Development with Effect of a Dike (Scenario 3)

Mine Water Management Summary

Based on the predictions of the Base Case (Table 24-31) groundwater modeling, Table 24-32 presents summary details of the predicted inflow rates to the three underground mines.

Table 24-32: Predicted Water Flows (gpm) (1)

Area	Minimum Flow	Maximum Flow
Horseshoe	165	375
Mustang	35	70
Mill Zone Deep	80	125

(1) Does not include flow ramp-up at commencement of decline construction.

Water Table Drawdown Extent in Response to Underground Water Management

The zone of influence of water drawdown of the combined open pit and underground mine scenario will likely:

- Extend further to the east and northeast, due largely to the proposed Horseshoe development; and
- Remain relatively static to the south and west.

Further, more detailed modelling will be required in subsequent stages of project evaluation.

Stormwater Control

Surface water modeling has been undertaken to provide estimates of potential stormwater generation (flow rates and depths) for each of the portal catchments so as to provide a basis for understanding the level of 'flood' protection required for portals and vent shafts. Stormwater generated from the catchments above these facilities, including outside the pit rim will need to be diverted away from the facilities into the mine pits, and these broader catchments are excluded from the analysis. Figure 24-35 shows the portal catchments of Horseshoe, Mustang and Mill Zone Deep, as well as the positioning of bench sumps that will divert higher pit catchment flows to the mine pits.

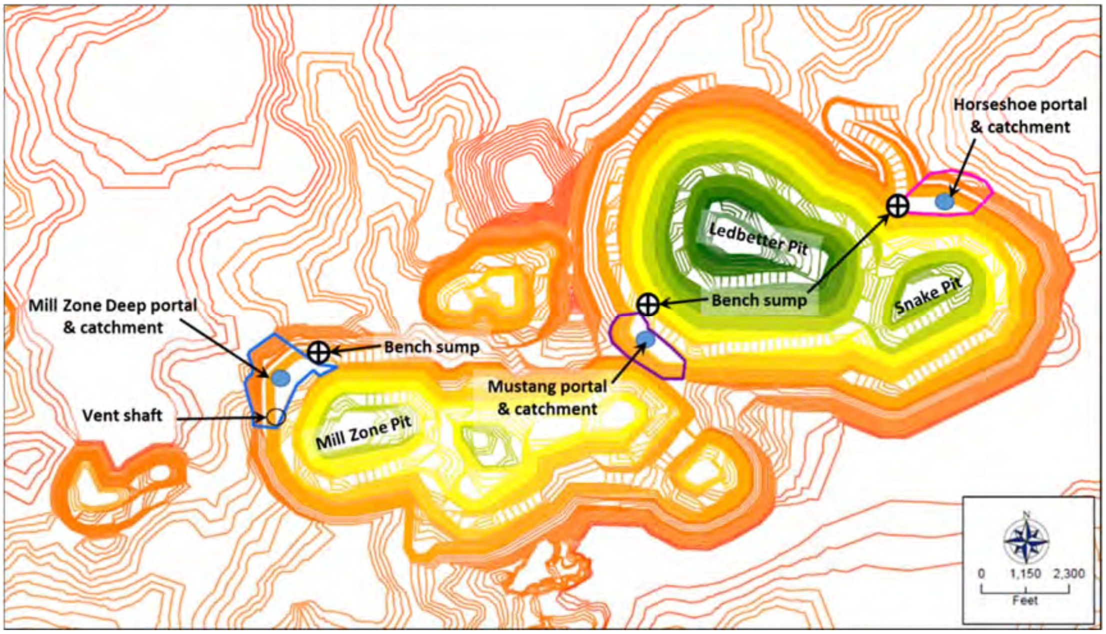


Figure 24-35: Catchment Plan for Mine Portals

The modeling has been undertaken using a rational method calculation to establish the peak flow at portal locations for a range of annual recurrence intervals (ARI) (CDM Smith, 2016). Table 24-33 presents predicted flow rates for each portal catchment.

Table 24-33: Predicted Water Flows[(1)] and Time of Concentration

Portal catchment	Area (acre)	Flow rate (cfm) [(1)]			
		2 yr ARI	10 yr ARI	50 yr ARI	100 yr ARI
Horseshoe (Snake Pit)	47	12,080	15,680	18,435	19,705
Mustang (Ledbetter Pit)	42	11,865	15,255	18,010	19,280
Mill Zone Deep (Mill Zone Deep)	64	16,740	21,615	25,640	27,335

Source: CDM, 2016
(1) Rounded to nearest 5 cfm

A 1:10, or flatter bund, depending on operational plant requirements, should be considered at the portal entrance and vent shafts to divert water into diversion drains, if used. An alternative management approach is to rely on levelling benches at each portal so that stormwater drains toward the pit.

Groundwater Quality

Overall, the quality of groundwater occurring in hydrostratigraphic units HSU3 and HSU4 (upper weathered and lower unweathered basement, respectively) can be described as “fresh”, with concentrations of total dissolved salts (TDS) below 500 mg/L, with the average concentration reported in NewFields (2015) being around 125 mg/L. pH measurements tend to indicate slightly acidic conditions ranging from approximately 5 to 7 pH units, although there appears to be an overall increase in groundwater alkalinity (pH increasing above 7) with depth.

An assessment of the Haile Gold Mine exploration geochemistry database along with consideration of the hydrogeological setting suggests there is low to moderate potential for ARD generation during construction, operation and closure of the proposed underground development.

24.16.4 Geochemical

The objective of the geochemical program was to determine the metal leaching and acid rock drainage (ML/ARD) potential of waste rock that would be generated from the underground operation at the three proposed locations.

Mine waste generated from surface mining at Haile has a range of ML/ARD potential, depending on lithology. The source of ARD is sulfide minerals, principally pyrite, deposited by the hydrothermal alteration that also deposited gold. Starting in 2008, an extensive geochemical characterization program of existing and future mine wastes was performed by Schafer Limited LLC (Schafer, 2015) to identify, manage and mitigate geochemical risks at Haile. The current waste management plan employs three categories, based on combinations of the total sulfur content (ST) and net neutralization potential (NNP). The NNP is a measure of overall acid generation potential calculated as the difference between the neutralization potential (NP) and acid generating potential (AP). The categories of potentially acid generating (PAG) waste rock include:

- Red PAG – Strongly acid generating:
    - NNP < -31.25 kg $CaCO_3$/t;
- Yellow PAG – Moderately acid generating:
    - ST > 0.2 % or NNP between 0 and -31.25 kg $CaCO_3$/t; and
- Green – Not acid generating:
    - ST < 0.2 % or NNP > 0 kg $CaCO_3$/t.

Note the categories are based on metric units.

To evaluate the ARD/ML potential of the waste rock that might be generated from the infrastructure during development of the Mustang, Horseshoe and Mill Zone Deep deposits, SRK reviewed the available ARD data and preliminary mine plan. The Horseshoe and Mustang deposits had almost no ARD data. ARD data does exist in the vicinity of the Mill Zone Deep, but is limited to samples above the mineralization. In the absence of samples specifically selected and analyzed to evaluation ARD/ML potential, a sampling and analysis plan was developed and implemented (SRK, 2016).

Sample Collection

For the purposes of a PEA level characterization, sample collection was limited to 24 samples (SRK, 2016). Samples were selected to provide the full range of the principal lithologies and to cover the vertical range of each area. All proposed samples are within 165 ft of the anticipated mining infrastructures. Table 24-34 lists the SRK-collected samples by area, borehole, sample interval and lithology. The lithology of all borings that pass within 165 ft of the target mining areas are principally the Persimmon Fork Fm metasediments with minor metavolcanics.

Table 24-34: Summary of Total Metal Concentrations in 2016 SRK Geochem Sampling Program

Hole ID	Down-hole depth of sample	Location	Unit	Rock Type	Mo	Cu	Pb	Zn	Ag	Ni	Co	Mn	Fe	As	U	Au	Th	Sr	Cd	Sb	Bi	V	Ca	P	La	Cr	Mg	Ba	Ti	B	Al	Na	K	W	Sc	Tl	S	Hg	Se	Te	Ga
	(ft)				ppm	ppm	ppm	ppm	ppb	ppm	ppm	ppm	%	ppm	ppm	ppb	ppm	ppm	ppm	ppm	ppm	ppm	%	%	ppm	ppm	%	ppm	%	ppm	%	%	%	ppm	ppm	ppm	%	ppb	ppm	ppm	ppm
Limit of Detection					0.01	0.01	0.01	0.1	2	0.1	0.1	1	0.01	0.1	0.1	0.2	0.1	0.5	0.01	0.02	0.02	2	0.01	0.001	0.5	0.5	0.01	0.5	0.001	20	0.01	0.001	0.01	0.1	0.1	0.02	0.02	5	0.1	0.02	0.1
DDH379	761-770	Horseshoe	DB	db	1.06	92.63	1.27	54.7	40	185.6	32.3	781	3.89	0.9	0.2	1.4	1.1	125	0.04	0.07	0.04	69	2.44	0.034	5.7	132.1	2.8	31.2	0.076	20	4	0.504	0.11	0.1	5.4	0.14	0.15	5	0.1	0.02	6.8
DDH368	645-653	Horseshoe	L	lt	2	35.28	8.81	35.6	70	3	3.4	761	1.52	2.5	0.5	0.3	7	82.7	0.06	0.02	0.09	2	1.8	0.051	15.4	39.8	0.3	25.1	0.002	20	0.72	0.009	0.19	0.1	0.5	0.06	0.34	7	0.2	0.06	1.4
DDH368	810-815	Horseshoe	L	l	0.94	13.31	38.46	40.5	300	2.3	3	594	1.13	3.7	0.6	252.5	6.6	65.1	0.08	0.02	0.43	2	1.01	0.038	13.1	28	0.31	30.3	0.002	20	0.6	0.005	0.28	0.1	0.4	0.11	0.12	14	0.2	0.28	1.1
DDH370	1,385-1,390	Horseshoe	L	lt	1.66	31.05	5.2	10.4	83	2.1	1.9	428	0.46	1.2	0.8	235.4	6.9	76.6	0.12	0.02	0.09	2	1.39	0.009	16.5	55.8	0.01	39	0.003	20	0.21	0.021	0.19	0.1	0.3	0.06	0.2	5	0.2	0.09	0.4
DDH369	675-680	Horseshoe	LC	lw	1.2	18.81	62.87	25.5	326	1.4	3.2	641	0.69	0.9	0.8	3.9	7.3	77.7	0.07	0.02	0.81	2	1.77	0.023	15.5	42.3	0.22	25.1	0.001	20	0.42	0.009	0.18	0.1	0.3	0.06	0.06	5	0.1	0.36	0.8
DDH378	675-680	Horseshoe	LC	lw	1.53	21.09	11.37	60	63	2	2.4	641	1.9	0.1	0.4	0.2	6.6	57.1	0.04	0.02	0.08	2	1.4	0.048	19.5	32	0.64	23.7	0.002	20	1.16	0.009	0.15	0.1	0.5	0.04	0.02	6	0.1	0.02	2.2
DDH378	1,060-1,070	Horseshoe	MV	mv	3.11	28.63	6.39	25.5	59	4.1	2	273	0.64	0.5	0.7	0.2	4.3	45	0.04	0.05	0.11	2	0.63	0.018	14.6	127.3	0.08	32.5	0.031	20	0.26	0.04	0.16	0.1	0.5	0.16	0.02	5	0.1	0.02	0.8
DDH379	1,213-1,219	Horseshoe	MV	mv	2.15	12.14	11.91	30.9	185	2.2	1.6	383	0.71	0.6	0.8	0.4	5.3	74.3	0.03	0.03	0.34	3	0.76	0.024	16.7	92.5	0.1	33.5	0.019	20	0.33	0.044	0.15	0.1	0.4	0.13	0.02	6	0.1	0.04	1.1
DDH336	402-406	Mill Zone Deep	L	l	7.73	20.24	8.36	35.4	234	5.2	5.7	24	1.88	65.1	0.5	65	3.9	2.6	0.14	0.43	0.19	2	0.03	0.009	6.6	49.9	0.01	9.8	0.001	20	0.19	0.008	0.13	0.1	0.3	0.11	2.13	175	0.2	0.41	0.4
DDH350	413-418	Mill Zone Deep	L	lpy	340.88	48.34	24.71	54.7	4,985	12.9	9.7	43	13.32	790.3	0.9	6743.6	4.3	2.4	0.47	7.4	0.27	9	0.01	0.001	3	58.4	0.01	9.8	0.001	20	0.22	0.008	0.15	1.5	0.4	2.31	>10.00	1603	9.7	6.91	0.7
DDH350	540-545	Mill Zone Deep	L	l	25.22	9.95	16.12	43	2,298	2.2	5.7	440	1.64	23.4	1	215.9	6.4	28.7	0.35	0.3	0.23	2	1.02	0.073	8.3	33.3	0.23	18.9	0.001	20	0.26	0.006	0.2	0.1	0.4	0.1	1.69	8	0.2	2.27	0.4
DDH442	625-630	Mill Zone Deep	L	l	7.33	68.06	10.4	77.8	111	21.1	15.4	1,023	4.02	19.7	0.4	33.6	4.4	13.9	0.06	0.42	0.3	9	0.86	0.068	7.3	42.6	0.77	23.1	0.001	20	1.49	0.016	0.12	0.1	2.4	0.07	2.23	22	0.4	0.15	3.7
DDH336	335-340)	Mill Zone Deep	LC	lw	1.93	19.34	2.71	66.5	21	2.7	2.8	360	1.02	3.3	1	0.2	7.2	3.8	0.07	0.02	0.02	2	0.07	0.026	11	72.9	0.19	14.9	0.001	20	0.43	0.017	0.12	0.1	0.4	0.08	0.37	11	0.1	0.02	1
DDH352	388-393	Mill Zone Deep	LC	lw	13.86	46.31	8.12	41.3	942	16.7	15.3	218	3.72	148.1	0.3	2312.7	3.1	4.1	0.08	1.4	0.26	4	0.11	0.037	6.8	63.3	0.21	12.5	0.001	20	0.45	0.008	0.14	0.2	0.8	0.27	3.54	118	1.9	1.02	1
DDH406	375-380	Mustang	L	lm	33.53	40.54	9.71	34.5	643	8.3	7.7	570	2.67	80.9	0.5	282.5	3.5	24.8	0.09	0.56	0.11	2	0.75	0.097	3.2	43.3	0.3	9.3	0.001	20	0.25	0.004	0.18	0.1	0.7	0.12	2.69	18	1.1	3.19	0.7
DDH461	1,315-1,320	Mustang	L	lt	0.95	14.17	6.81	74.6	180	2.8	4.1	1,716	1.5	4.4	0.3	0.9	2.6	33.1	0.03	0.13	0.08	7	1.86	0.033	7.8	37.7	0.71	20.7	0.037	20	1.04	0.018	0.1	0.1	1.3	0.02	0.42	5	0.1	0.05	2.2
DDH389	1,315-1,320	Mustang	LC	lw	4.88	13.46	8.56	61.3	340	2.2	3.3	664	1.58	23.9	0.2	46	1.7	57.7	0.09	0.2	0.09	2	1.02	0.03	5.4	51.5	0.03	19.5	0.002	20	0.2	0.004	0.14	0.1	0.4	0.06	1.75	16	0.1	0.79	0.4
DDH396	815-820	Mustang	LC	lc	1.21	60.27	3.47	99.3	188	3.7	5.7	1,422	2.01	15.7	0.1	39.4	1.7	41.8	0.28	0.1	0.04	2	2.54	0.05	4.5	32.2	0.8	17.2	0.001	20	0.25	0.003	0.19	0.1	1.4	0.08	0.77	6	0.6	1.09	0.5
DDH436	1,115-1,120	Mustang	LC	lw	0.43	186.86	2.19	64.1	199	31.9	29.9	1,127	5.67	0.6	0.1	15	0.7	210.3	0.09	0.03	0.02	150	5.54	0.052	3.9	41.1	3.05	5.5	0.015	20	4.11	0.007	0.03	0.1	13.4	0.02	0.02	5	0.1	0.05	9.1
DDH389	820-825	Mustang	LS	lwsk	8.26	20.72	13.84	106.9	6,261	3.8	2.1	119	1.61	11.6	0.3	1627.2	1.7	24.5	0.28	0.84	0.11	2	0.08	0.006	5.9	136	0.03	12.1	0.001	20	0.12	0.008	0.12	0.1	0.4	0.08	1.75	123	0.3	14.01	0.3
DDH389	1,140-1,142	Mustang	LS	lwsk	27.82	10.23	9.55	6.6	907	3.3	2.3	611	1.11	8.1	0.1	159.8	0.8	61.8	0.04	0.23	0.07	2	0.41	0.021	3.6	109.4	0.08	16.1	0.002	20	0.17	0.004	0.16	0.1	0.5	0.08	1.15	8	0.1	4.8	0.4
DDH396	860-865	Mustang	LS	lsm	7.14	18.26	7.32	18.5	1,781	3.5	1.4	163	0.63	9.7	0.1	479	1.1	33	0.04	0.39	0.04	2	0.13	0.002	4.3	117.9	0.06	11.2	0.001	20	0.1	0.005	0.11	0.1	0.4	0.06	0.58	28	0.2	6.63	0.3
			High Calcium Granite		10	300	150	600	0.51	150	70	5,400	29.6	19	30	0.04	85	4,400	1.3	2	D	880	25.3	9,200	450	220	9.4	420	3.4	D	82	28.4	25	13	140	7.2	0.3	800	0.5	#N/A	170
			Sandstones		2	50	70	160	0.5	20	3	500	9.8	10	4.5	0.05	17	200	0.5	0.5	D	200	39.1	1,700	300	350	7	500	1.5	350	25	3.3	10.7	16	10	8.2	0.24	300	0.5	#N/A	120

Source: SRK

- Metric PPM units used.
- Values below the reporting limit are shown in red text.
- Metal concentrations in metasediment samples were compared to ten times the average crustal concentrations for sandstone. Concentrations exceeding 10 times the crustal concentration are shown in tan.
- Metal concentrations in metavolcanic samples were compared to ten times the average crustal concentrations for high calcium granite. Concentrations exceeding 10 times the crustal concentration are shown in orange.

Core samples were selected from recently completed borings. The distribution of samples approximates the volumetric distribution of lithologies expected from the infrastructure zones of the combined mining areas. Certain lithologies with high ARD potential (such as pyritic laminated metasediment) were selected for analysis, even though they make up a small percentage of the total anticipated waste rock stream.

Two kilogram samples of the specified intervals were collected from the recently drilled exploration borings, placed in plastic bags, labeled, and shipped to SGS Vancouver in water tight 5-gallon plastic buckets.

Sample Analysis

Samples were analyzed for static geochemical characteristics using methods similar to those used for the overburden geochemical characterization program (Schafer, 2015). A range of methods were used for sulfur species and sources of neutralization potential (NP) to evaluate the various sources of AP and NP, and potential differences between analytical methods. Samples were analyzed for the following parameters:

- Paste pH (Sobek, 1978);
- Sulfate via HCl digestion with ICP-OES analysis;
- Sulfide via 1:7 HNO3 solution with ICP-OES analysis;
- Total sulfur (LECO);
- Total carbon (LECO);
- Total inorganic carbon (HCl leach);
- Modified NP (Lawrence, 1989);
- NP (Sobek, 1978);
- Net Acid Generation (NAG) test (AMIRA, 2002);
- NAG (AMIRA, 2002); and
- Trace metal analysis, using ICP-OES/MS following aqua regia digestion (Al, Sb, As, Ba, Cd, Ca, Cl, Cr, Cu, F, Fe, Pb, Mg, Mn, Hg, Mo, Hi, K, Se, Si, Ag, K, Th, Zn).

All analyses were performed with the method specific QA/QC protocols, which included blanks, duplicates and reference standards. The laboratory data are considered generally complete and representative.

Results of the sample analyses are summarized in Table 24-35. Lab generated data is indicated by headings in bold, whereas calculated values are shown in italics. Values below the reporting limit are shown in red text.

Table 24-35: Static Geochemical Data (1) (2)

Sample Information					Neutralization Potential Data							Acid Potential Data					Calculated ARD Potential Values					NAG Test				
Hole ID	Down-hole depth of sample	Location	Unit	Rock Type	Paste pH	TIC	*Equiv. CaCO3*	Total C	Modified NP	Sobek NP	Fizz Test	Total S (3)	Sulfate	Sulfide	Insoluble S	AP	Net Modified NP (4)	Net Sobek NP (4)	Modified NP/AP (5)	TIC NP/AP (5)	Sobek NP/AP (5)	NAG pH after	Vol. of 0.1 N NaOH		NAG (kg H2SO4/tonne)	
	(ft)				Std. Units	% C	*kg CaCO3/t*	%C	kg CaCO3/t	kg CaCO3/t		%S	%S	%S	%S	kg CaCO3/t	kg CaCO3/t	kg CaCO3/t				Reaction (6)	to pH 4.5	to pH 7.0	to pH 4.5	to pH 7.0
Limit of Detection					0.20	0.01	#N/A	0.005	0.5	0.5	#N/A	0.005	0.01	0.01	#N/A	#N/A	#N/A	#N/A	#N/A	#N/A						to pH 7.0
DDH378	761-770	Horseshoe	DB	db	9.79	0.15	12.5	0.152	41.7	55.9	Slight	0.162	0.01	0.1	0.05	3.1	38.6	52.8	13.3	4.0	16.9	6.97	#N/A	0.05	#N/A	0.1
DDH368	645-653	Horseshoe	L	lt	9.46	0.52	43.3	0.526	42.7	45.6	Slight	0.339	0.01	0.25	0.09	7.8	34.9	37.8	5.5	5.5	4.8	5.87	#N/A	1.60	#N/A	3.1
DDH368	810-815	Horseshoe	L	l	9.13	0.34	28.3	0.351	26.5	29.8	Slight	0.134	0.01	0.09	0.04	2.8	23.7	27.0	9.4	10.1	9.6	5.86	#N/A	3.10	#N/A	6.1
DDH370	1,385-1,390	Horseshoe	L	lt	9.62	0.43	35.8	0.444	36.6	37.0	Slight	0.199	0.01	0.07	0.13	2.2	34.4	34.8	16.7	16.4	15.9	6.17	#N/A	1.55	#N/A	3.0
DDH369	675-680	Horseshoe	LC	lw	9.48	0.58	48.3	0.598	48.0	49.0	Slight	0.066	0.01	0.07	0.01	2.2	45.8	46.8	21.9	22.1	21.4	6.18	#N/A	4.10	#N/A	8.0
DDH378	620-625	Horseshoe	LC	lw	9.15	0.43	35.8	0.434	38.4	40.7	Slight	0.018	0.01	0.01	0.02	0.3	38.4	40.7	128.0	119.4	135.7	6.20	#N/A	4.15	#N/A	8.1
DDH378	1,060-1,070	Horseshoe	MV	mv	9.82	0.18	15.0	0.197	16.8	18.3	Slight	0.005	0.01	0.01	0.01	0.3	16.8	18.3	56.0	50.0	61.0	6.01	#N/A	5.50	#N/A	10.8
DDH379	1,213-1,219	Horseshoe	MV	mv	9.87	0.22	18.3	0.226	19.4	21.2	Slight	0.006	0.01	0.01	0.01	0.3	19.4	21.2	64.7	61.1	70.7	6.09	#N/A	6.15	#N/A	12.1
DDH336	402-406	Mill Zone Deep	L	l	5.13	0.01	0.8	0.013	0.8	0.6	None	2.22	0.03	2.12	0.07	66.3	-65.5	-65.7	0.0	0.0	-1.0	2.42	24.30	27.85	47.6	54.6
DDH350	413-418	Mill Zone Deep	L	lpy	3.90	0.01	0.8	0.013	-1.1	-2.9	None	15.2	0.08	14.7	0.42	459.4	-460.5	-462.3	0.0	0.0	-1.0	2.12	101.70	128.80	199.3	252.4
DDH350	540-545	Mill Zone Deep	L	l	8.29	0.38	31.7	0.396	30.3	32.6	Slight	1.71	0.03	1.62	0.06	50.6	-20.3	-18.0	0.6	0.6	-0.4	3.16	3.55	5.70	7.0	11.2
DDH442	625-630	Mill Zone Deep	L	l	8.24	0.23	19.2	0.223	20.6	24.7	Slight	2.3	0.02	2.09	0.19	65.3	-44.7	-40.6	0.3	0.3	-0.6	2.68	15.35	18.90	30.1	37.0
DDH336	335-340	Mill Zone Deep	LC	lw	3.85	0.01	0.8	0.063	-1.3	0.0	None	0.352	0.12	0.24	0.01	7.5	-8.8	-7.5	-0.2	0.1	-1.0	3.21	2.40	5.90	4.7	11.6
DDH352	388-393	Mill Zone Deep	LC	lw	6.64	0.01	0.8	0.046	1.8	4.2	None	3.63	0.02	3.51	0.10	109.7	-107.9	-105.5	0.0	0.0	-1.0	2.37	34.50	39.25	67.6	76.9
DDH406	375-380	Mustang	L	lm	6.28	0.31	25.8	0.341	25.0	24.3	None	2.73	0.05	2.61	0.07	81.6	-56.6	-57.3	0.3	0.3	-0.7	2.60	17.30	21.20	33.9	41.6
DDH461	1,315-1,320	Mustang	L	lt	9.22	0.56	46.7	0.561	48.4	49.0	Slight	0.431	0.01	0.42	0.01	13.1	35.3	35.9	3.7	3.6	2.7	6.10	#N/A	1.95	#N/A	3.8
DDH389	1,380-1,385	Mustang	LC	lw	8.82	0.31	25.8	0.313	26.0	26.4	Slight	1.81	0.01	1.75	0.06	54.7	-28.7	-28.3	0.5	0.5	-0.5	2.91	7.45	9.85	14.6	19.3
DDH396	815-820	Mustang	LC	lc	9.30	1.47	122.5	1.51	99.4	86.1	Slight	0.783	0.01	0.75	0.03	23.4	76.0	62.7	4.2	5.2	2.7	6.63	#N/A	0.20	#N/A	0.4
DDH436	1,115-1,120	Mustang	LC	lw	8.89	1.72	143.3	1.77	149.7	173.6	Moderate	0.05	0.01	0.01	0.04	0.3	149.4	173.3	479.0	458.7	554.5	6.72	#N/A	1.25	#N/A	2.5
DDH389	820-825	Mustang	LS	lwsk	7.99	0.04	3.3	0.038	1.3	3.1	None	1.77	0.02	1.67	0.08	52.2	-50.9	-49.1	0.0	0.1	-0.9	2.63	16.35	19.35	32.0	37.9
DDH389	1,140-1,142	Mustang	LS	lwsk	8.74	0.17	14.2	0.167	13.4	10.7	Slight	1.15	0.01	1.08	0.07	33.8	-20.4	-23.1	0.4	0.4	-0.7	3.03	4.95	7.10	9.7	13.9
DDH396	860-865	Mustang	LS	lsm	8.70	0.07	5.8	0.074	5.4	5.9	None	0.624	0.01	0.56	0.05	17.5	-12.1	-11.6	0.3	0.3	-0.7	3.20	2.65	5.55	5.2	10.9

Source: SRK

(1) Metric units used for lab reporting
(2) Values below the reporting limit are shown in red text.
(3) Haile Mine PAG Classification for total sulfur: Yellow= Moderately acid generating (> 0.2% total sulfur), Green = non-acid generating (<0.2% total sulfur)
(4) Haile Mine PAG Classification for NNP: Red = Highly acid generating (< -31.3 kg CaCO3/t), Yellow = Moderately acid generating (between 0 and < -31.3 kg CaCO3/t), Green= non-acid generating (>0 kg CaCO3/t)
(5) Industry Criteria: Red=Potential acid generating (NPR < 1), Yellow= Uncertain acid generating potential (NPR ≤2 and ≥1), Green=Not acid generating (NPR > 2)
(6) Industry Criteria: Red=Potential acid generating (NAG pH < 4.5), Green=Not acid generating (NAG pH > 4.5)

Sulfur

Sulfur species were determined by a range of methods to provide a comprehensive evaluation of potential sources of acidity. Observations based on the sulfur data include:

- Total sulfur ranged from 0.005 to 15.2%;
- Almost all sulfur occurs as sulfide, especially at higher concentrations;
- Sulfide determined analytically shows an excellent correlation (R = 0.999) to sulfide derived as the difference between total sulfur and sulfate sulfide;
- The highest sulfide value was measured in a sample of pyritic metasediment;
- Sulfide values were uniformly low in the samples from Horseshoe (<0.004 to 0.33%); and
- Sulfide values in Mill Zone Deep and Mustang were mostly between 0.5 to 3%, with a range of 0.01 to 14.7%.

Neutralization potential was evaluated using three methods: the Sobek method (conservative), the Modified method, and total inorganic carbon (TIC). Total carbon was also measured to evaluate the potential presence of organic carbon. Observations related to factors controlling NP include:

- Total carbon is almost equal to TIC. At higher concentrations, total carbon was no more than 3% greater than TIC;
- Sobek NP values are slightly higher than Modified NP by an average of 17%; and
- NP determined by the Modified method and calculated from TIC are very similar, with the exception of three samples that are being reanalyzed.

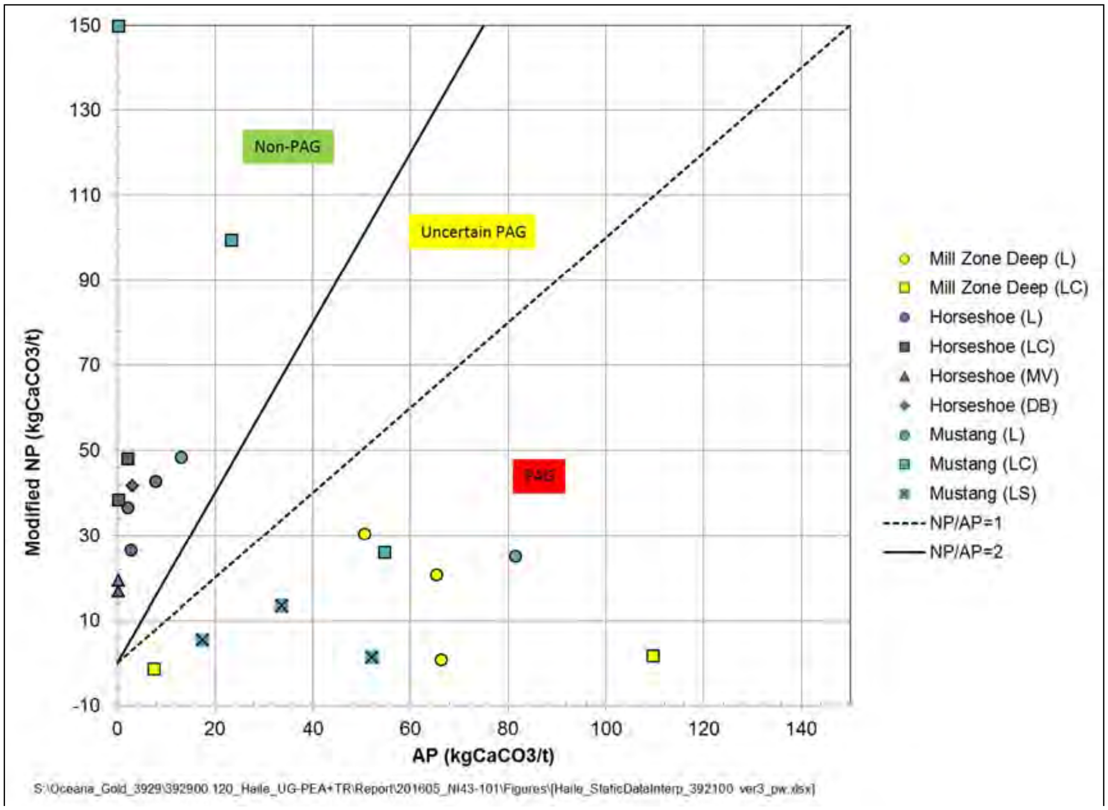
These data indicate that virtually all the available NP is derived from carbonate minerals. Sobek NP slightly overestimates NP since it tends to include NP derived from silicate minerals.

Observations regarding the ARD potential of the samples include:

- There is no apparent correlation between NP/AP and depth of samples;
- For the Horseshoe samples, the metavolcanic rocks had significantly lower NPs compared to the metasediments;
- LC samples from Mustang had the highest NP for that group of samples;
- Conversely, the LC samples from Mill Zone Deep had the lowest NP for that group of samples; and
- NAG pH values ranged from 2.12 to 6.97. NAG pH values below 4.5 (indicative of PAG rock [Amira, 2002]) were measured for all the samples from Mill Zone Deep and five of eight samples for Mustang.

The acid generating potential of the samples was determined using the NNP methods as well as literature criteria for NPR and NAG pH. The waste classification of the samples is summarized in Table 24-35 via the color coding scheme used by Haile.

Figure 24-36 plots the Modified NP vs AP (from measured sulfide sulfur) by deposit and lithology. Plots of NP/AP using the other measurements for NP and AP give very similar results. The commonly used industry criteria for acid generating material (Price, 2009) uses the following categories based on the NPR: <1 is PAG, between 1 and 2 is uncertain acid generation potential, and > 2 is not PAG.



Source: SRK

Figure 24-36: NP/AP for Mill Zone Deep, Horseshoe and Mustang Mine Waste Rock

Observations related to the potential for acid generation include:

- All samples from Horseshoe are non PAG;
- All samples from the Mill Zone Deep area and most of the samples from Mustang (five of eight) are PAG based on NP/AP and NAG pH;
- Samples from Mustang included one sample that was non PAG based on all criteria; and
- As shown on Table 24-34, waste rock classification using the mine criteria occasionally differs from the industry criteria. Specifically, two samples from Mill Zone Deep and three samples from Mustang would be classified as moderately acid generating by the mine but PAG using the industry criteria. These samples have total sulfur values ranging from 0.352 to 1.81%.

Total Metals

Total metals for the samples from the three mining areas were compared to ten times the crustal averages for similar rocks (Price, 1997). The metavolcanic rocks were compared to high calcium granites (an intrusive analog to andesite), while the metasediments were compared to sandstones. The objective of the comparison was to identify rock enriched in metals relative to crustal averages to identify those elements that might leach from the waste rock during oxidation of the sulfides. This metal leaching process can occur under both neutral and acidic conditions.

As shown on Table 24-35, some metasediment samples were enriched in Sb, As, Cu, Au, Mo, Ni, Sc, Se, Ag and Sr. The metavolcanic samples were only enriched in Ag. Trends or significant metal concentrations included:

- The pyritic metasedimentary sample was significantly enriched with respect to As, Ag, Hg and Mo; and
- Enrichment of As, Sb, Co, Hg and Se indicate that leaching of these metals might have environmental impacts if the waste rock is not properly managed.

Summary

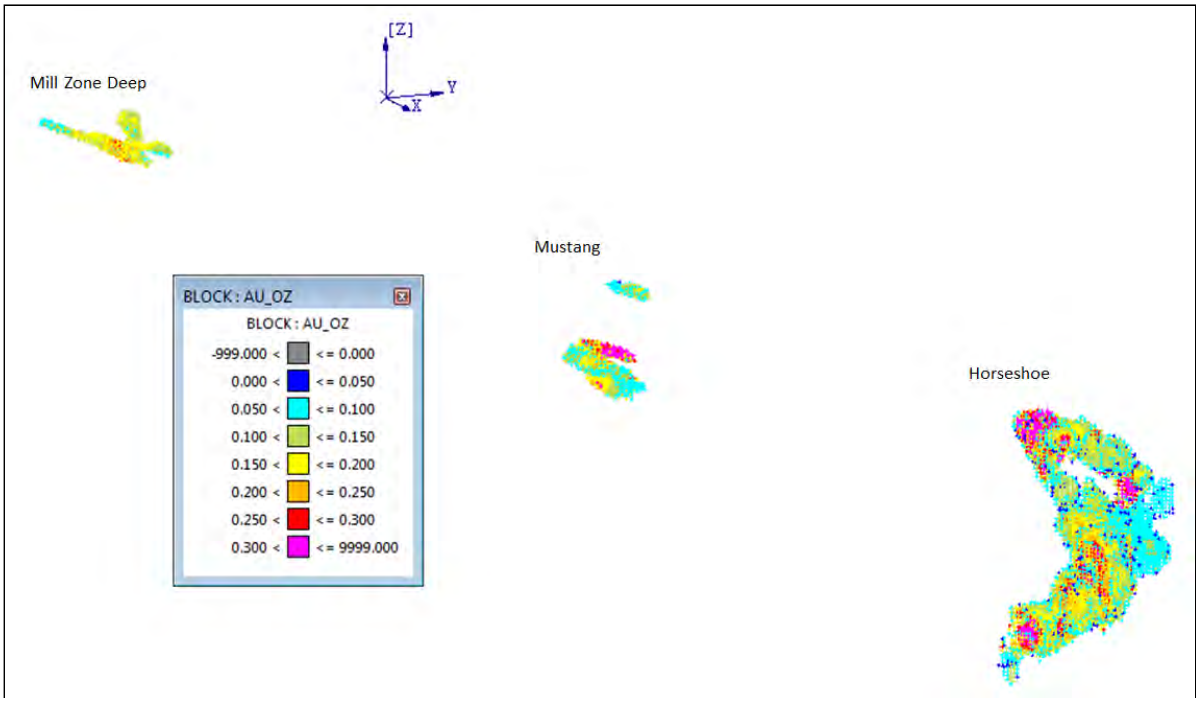
Results from the initial geochemical characterization of waste rock that would be generated by underground development of the Mustang, Horseshoe and Mill Zone Deep areas indicate the following:

- Samples from the Horseshoe area are classified as not acid generating, based on both the Haile and industry classification criteria;
- All samples from the Mill Zone Deep area are classified as potentially acid generating or uncertain acid generation potential based on industry classification criteria. Based on the Haile classification criteria, the samples are strongly (red) and moderately (yellow) acid generating;
- Five of the eight samples from the Mustang area are classified as potentially acid generating or uncertain acid generation potential based on industry classification criteria;
- Based on the Haile classification criteria, three of the samples from Mustang would be classified as moderately (yellow) acid generating, but classified as PAG using industry criteria;
- Many of the metasedimentary samples are enriched in As, Sb, Co, Hg and Se, and might have environmental impacts if the waste rock is not properly managed; and
- The pyritic metasedimentary rock should only account for a small percentage of the total waste rock, but has to be given careful consideration during mine operations due to its very high sulfide content (15%) and high concentrations of As, Ag, Hg and Mo.

The results of the PEA level geochemical characterization should be considered preliminary, as they are based on a very small number of samples.

24.16.5 Mine Design

Figure 24-37 is a rotated view of the resource block model blocks above a cut-off of 0.05 oz/st Au. All underground mineralized material is classified as inferred. Horseshoe has the highest overall higher grade, followed by Mustang and Mill Zone Deep. There is a higher grade portion of mineralization near the top of Horseshoe. The general dimensions of the Horseshoe area are approximately 700 ft along strike (NW – SW), 300 ft perpendicular to strike and ranging over a depth of 400 ft below surface to 1,500 ft below surface. The Mustang area is approximately 450 ft east to west, 300 ft north to south, and ranges over a depth of 900 ft to 1,200 ft in below surface. The Mill Zone Deep area is approximately 250 ft east to west, 400 ft north to south, and ranges over a depth of 400 ft to 600 ft below surface.



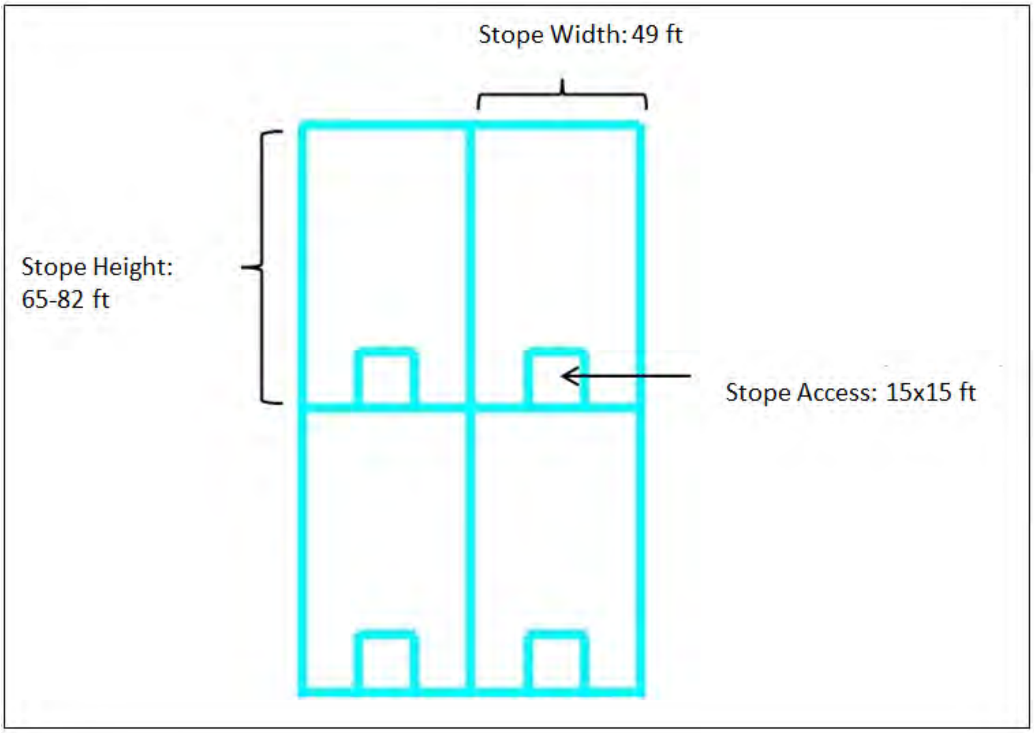
Source: SRK

Figure 24-37: Rotated View of Blocks above 0.05 oz/st (Blocks colored by Au)

Stope optimization was completed in Vulcan™ using a minimum mining stope width of 49 ft, a stope height of 65 to 82 ft, variable length along strike and an Au CoG of 0.05 oz/st. The stopes and block model are oriented to match the direction of mineralization.

Stope Design

Stope optimizer shapes were used as a basis for the design work. A typical level is made up of approximately 5 to 10 stopes. Stopes height at Horseshoe is 82 ft, and 65 ft at Mustang and Mill Zone Deep. Stope heights were selected to fit the mineralization and minimize dilution. Stopes at Horseshoe are oriented perpendicular to the foliation angle and the general mineralization strike. Length of the stopes varies based on mineralization extents; however, the maximum open length for a single stope is limited to 100 ft for geotechnical constraints and to minimize dilution. Each stope has a 15 ft x 15 ft access located at the bottom of the stope as shown in Figure 24-38. Top accesses are designed on most levels to give access to stopes on the next level and to allow for backfilling. For upper most stopes in a block or where there is no mining above, if the stope must be filled, it is assumed a hole can be drilled from adjacent development into the stope for backfilling purposes. The stopes are drilled top down and rings are blasted from the end of a stope (hangingwall) toward the access (footwall). The blasted material is remotely mucked from the stope access.



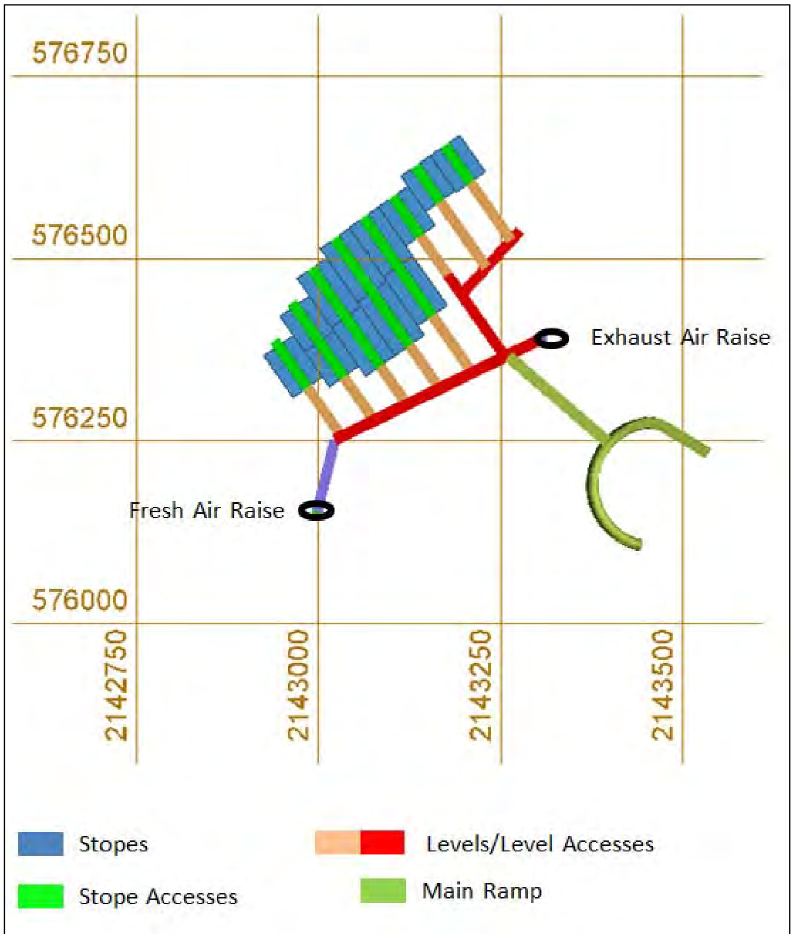
Source: SRK, 2016

Figure 24-38: Horseshoe* Stope Cross Section

A primary/secondary stoping sequence will be used where, on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined, backfilled and have had time to cure. Backfilling will be an integral part of the mining cycle as there is a limited quantity of stopes available on a level. At times mining will need to occur on multiple levels to sustain production.

Development Design

The stope accesses are connected to a level access located in waste material. The level accesses are offset approximately 65 ft from the end of stopes. The level accesses connect to the main ramp which is offset approximately 230 ft from stoping into the metavolcanic material located in the footwall. On the southwest side of each level the level access connects to a fresh air ventilation raise and on the northeast side connects to an exhaust air raise. Figure 24-39 shows a typical level section.



Source: SRK, 2016

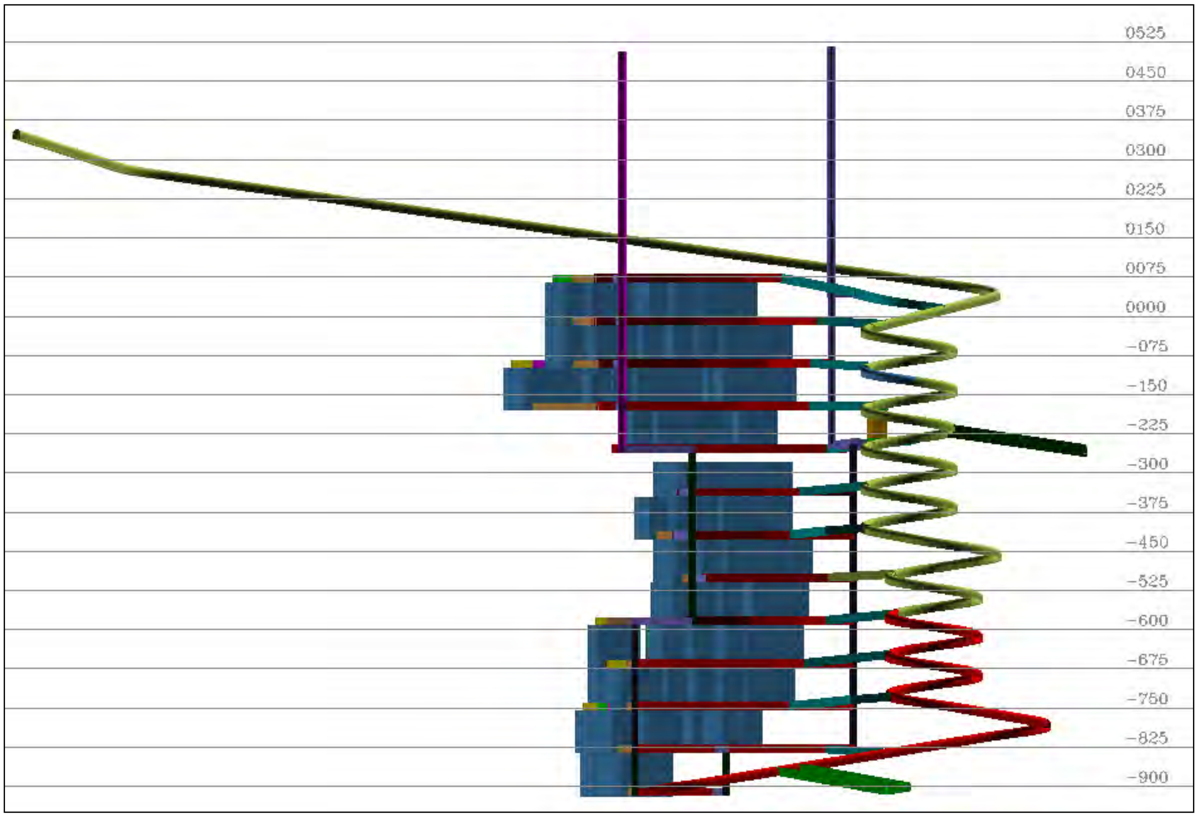
Figure 24-39: Typical Level Section (Horseshoe)

A backfill waste rock storage and cement mixing area is included at an elevation of -260 ft in the Horseshoe area and also at Mustang. As Mill Zone Deep is a smaller area an underground waste rock storage area has not been included at this time and the mixing with cement will occur at the surface.

In all three areas parallel drifts, connected at the ends, serve as water collection and pumping stations underground. Development of a small utilitarian underground shop and some minimalistic offices are included as an allowance in the design and use of existing surface facilities is assumed. All accesses and infrastructure, where possible, have been designed to be located in the metavolcanics and away from existing known dikes.

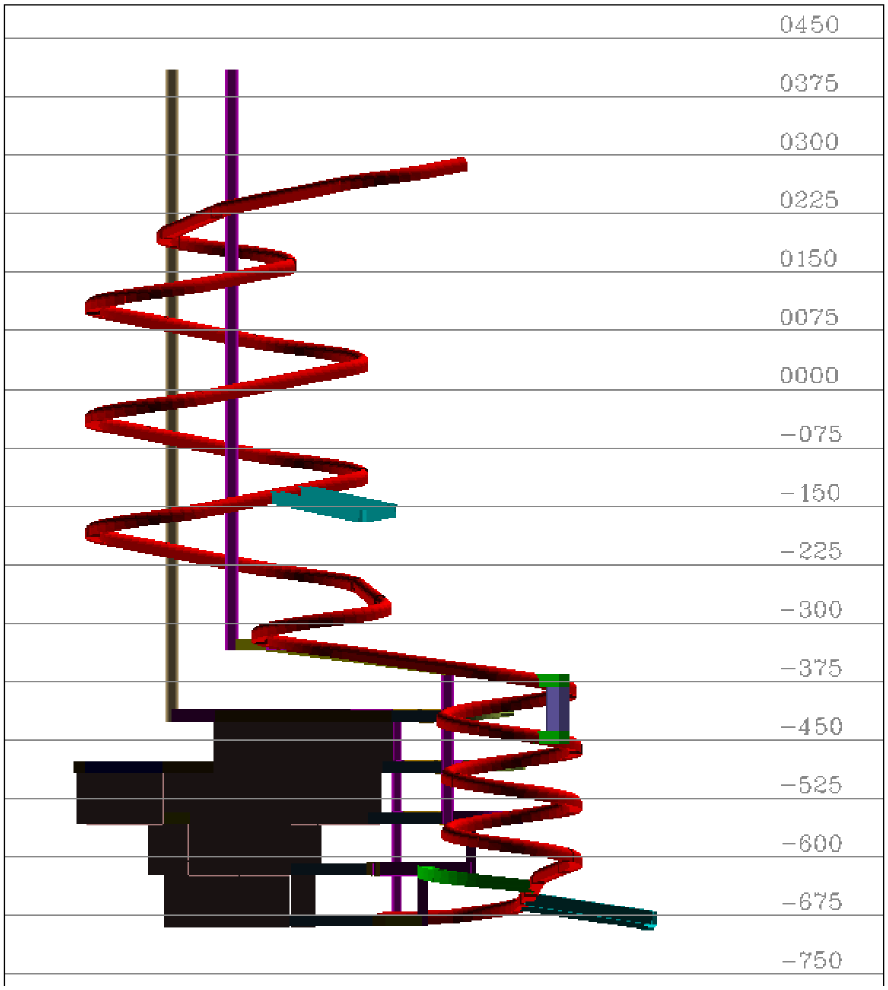
Figure 24-40, Figure 24-41, and Figure 24-42 show the completed design for each area.

Figure 24-43 shows rotated views of the mine design colored by Au.



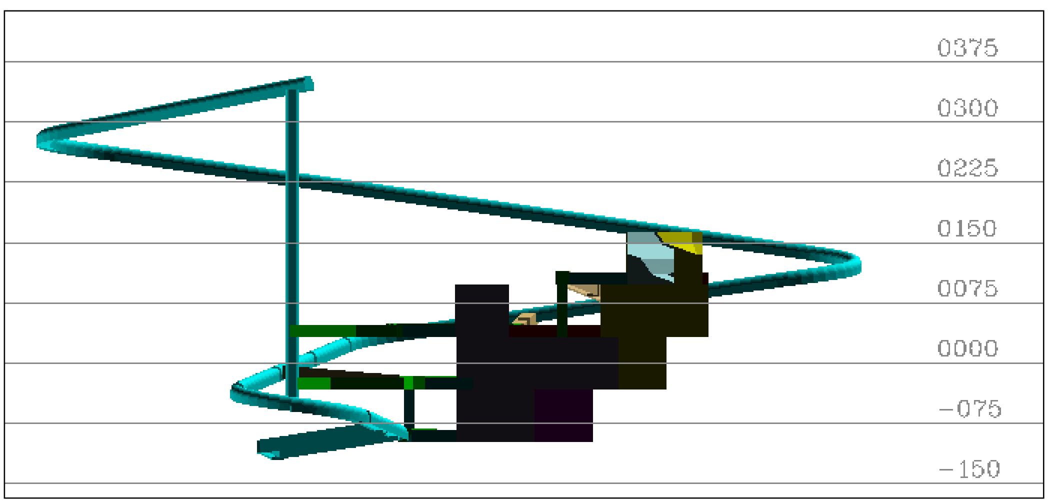
Source: SRK, 2016

Figure 24-40: Completed Mine Design – Horseshoe (Looking North)



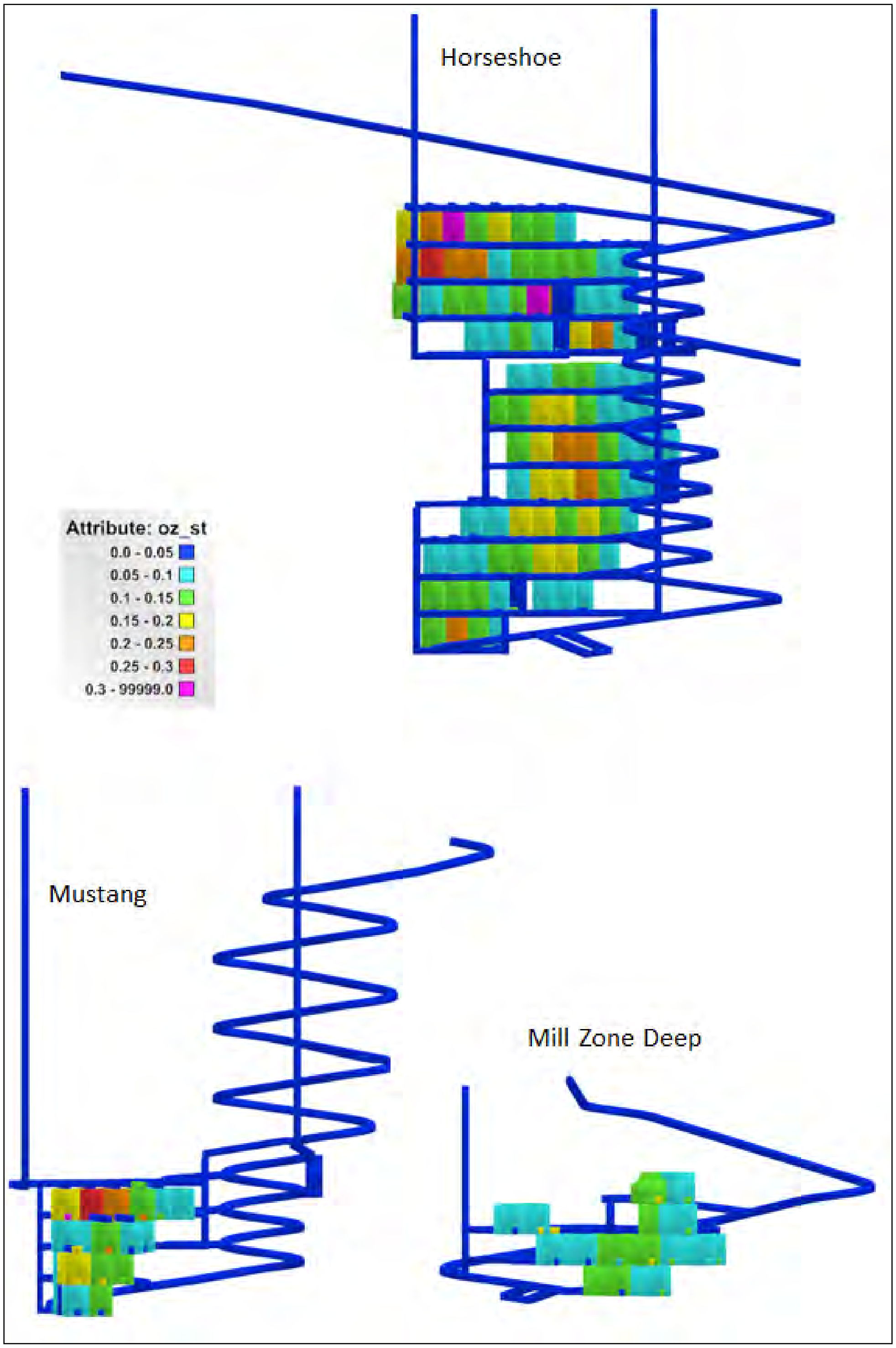
Source: SRK, 2016

Figure 24-41: Completed Mine Design – Mustang (Looking North)



Source: SRK, 2016

Figure 24-42: Competed Mine Design – Mill Zone Deep (Looking North)



Source: SRK, 2016

Figure 24-43: Mine Design (Rotated Views) Colored by Au Grade (oz/st)

24.16.6 Mine Plan Resource

The underground mine design process results in mine plan resources of 4.78 million short tons (Mst) (diluted) with an average grade of 0.14oz/st Au.

This estimate is based on a mine design using a 0.05 oz/st Au cut-off. These numbers include a 95% to 100% mining recovery based on type of opening (stope, development, etc.) to the designed wireframes in addition to a 0% to 7.5% unplanned waste dilution. An additional development allowance of 20% was applied to main ramps and 10% to level accesses to account for detail currently not in the design. These percentages were determined based on typical level layout, geotechnical ELOS factors, and practical mining factor assumptions. Waste dilution for stopes was applied with grade based on an analysis of the material around stopes in a representative area.

Table 24-36 summarizes the mine plan resources. A PEA is preliminary in nature in that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

Table 24-36: Mine Plan Resource Classification (1)

Description	Tons (kst)	Au (oz/st)
Inferred	4,777	0.14

(1) Includes inferred material reported using a 0.05oz/st Au CoG
Source: SRK, 2016

24.16.7 Production Schedule

The production schedule is based on the rate assumptions shown in Table 24-37.

Table 24-37: Production Rates (1)

Activity Type	Dimensions	Rate (1)
Main Ramps – 1st year of mining (single headings)	16 ft x 18 ft	18.1 ft/d
Main Ramps – subsequent years (single headings)	16 ft x 18 ft	21.4 ft/d
Level Development (single headings)	16 ft x 18 ft	21.6 ft/d
Drifting top/bottom stope accesses (multiple headings)	15 ft x 15 ft	43.4 ft/d
Stoping (2)	-	2,864 st/d
Raisebored Raise	16 ft diameter	13.3 ft/d
Slot Raises	13 ft x 13 ft	32.8 ft/d
Other mass excavations	-	9,464 ft^3/d
Backfilling	-	52,972 ft^3/d

(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.
(2) Includes slot, drilling, blasting, and mucking.
Source: SRK, 2016

A delay of seven days was used prior to driving on CRF and a 14-day delay prior to mining adjacent to a CRF filled stope. The mining operation schedule is based on 365 days/year, 7 days/week, with two 12 hour shifts each day. A production rate of 2,120 st/d was targeted with ramp-up to full production as quickly as possible.

Table 24-38 presents the annual mining scheduled based on these assumptions among others (including permitting). The annual schedule was completed using iGantt scheduling software. The iGantt scheduling work included backfill and its associated delays.

Table 24-38: Annual Mining Schedule (includes Horseshoe, Mustang, & Mill Zone Deep Areas)

Year	Mineralized Tons (kst)	Au (oz/st)	Waste Tons (kst)	Backfill Volume (Mft^3)
2018			154.5	
2019	533.5	0.13	430.7	5.7
2020	773.8	0.16	183.8	8.6
2021	773.8	0.13	55.7	8.9
2022	773.8	0.15	246.2	8.9
2023	773.8	0.15	176.3	8.6
2024	773.8	0.12	216.7	9.5
2025	374.5	0.10	18.2	4.2
Total	4,777.0	0.14	1,482.2	54.3

Source: SRK

Underground mine development at Horseshoe begins in January of 2018 with first production from stopes occurring in Q2, 2019 and lasting through the end of 2024. Development of the Mustang area begins in early in 2022 with first production from stopes occurring in Q3, 2023 and lasting into Q1, 2025. Development of the Mill Zone Deep area begins early in 2024 with first production from stopes occurring in Q4, 2024 to the end of the mine life.

This PEA is preliminary in nature and is based on technical and economic assumptions, which will be further evaluated in more advanced studies. The PEA is based on a resource model that contains Measured, Indicated and Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

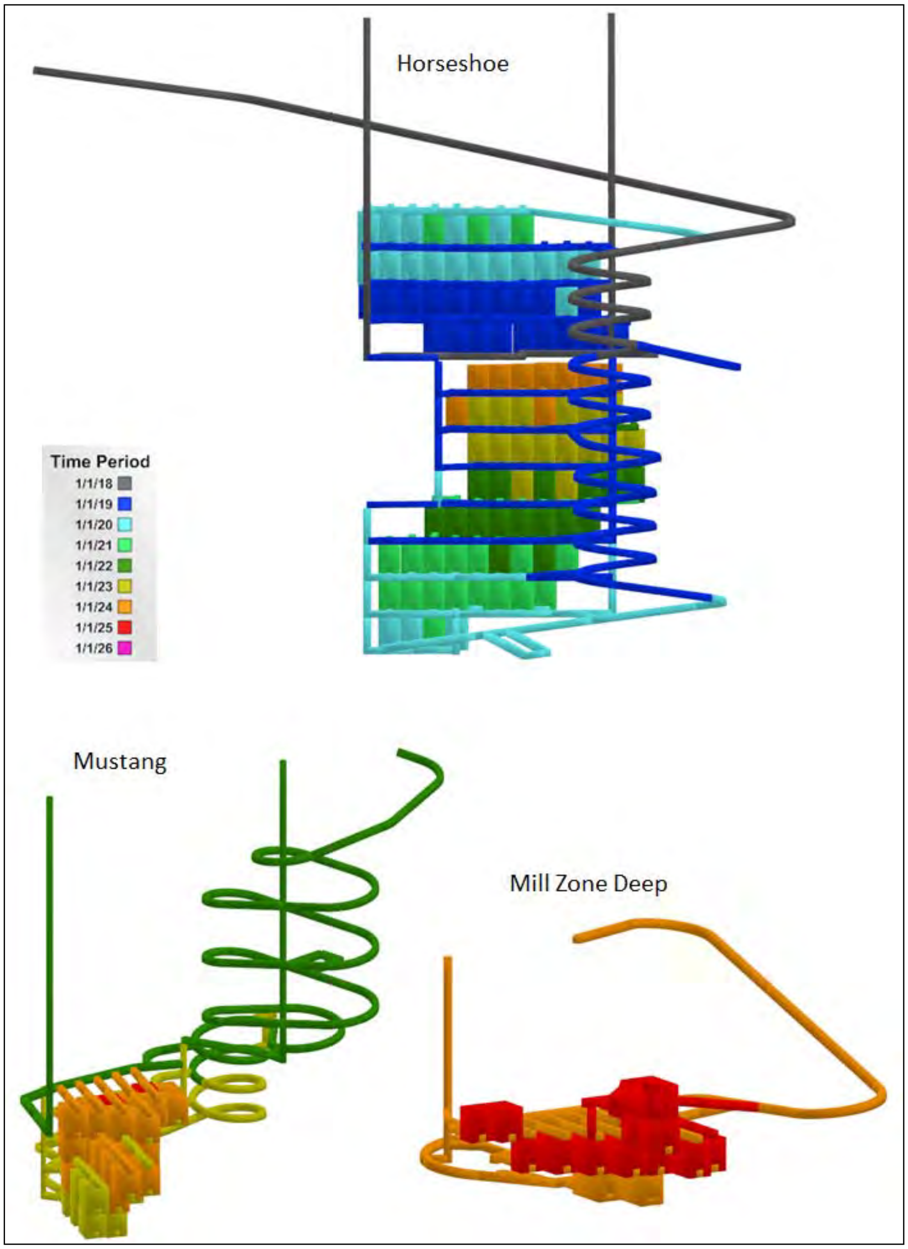
Table 24-39 summarizes the production schedule totals by development type.

Table 24-39: Production Schedule Totals by Activity Type

Develop/Production Type (w x h)	Length (ft)	Total Tons (kst)
Main Ramp (16 ft x 18 ft)	23,215	514.2
Drifting – Stope Access (15 ft x 15 ft)	39,990	746.1
Drifting - Level Accesses (16 ft x 16 ft)	16,804	378.1
Slot Raises (13 ft x 13 ft)	1.591	22.7
Raisebored Raises	3,866	65.8
Stoping		4,527.3
Other Excavations		4.9
Total	85,467	6,259.2

Source: SRK, 2016

Figure 24-44 shows the mine production schedule colored by year.



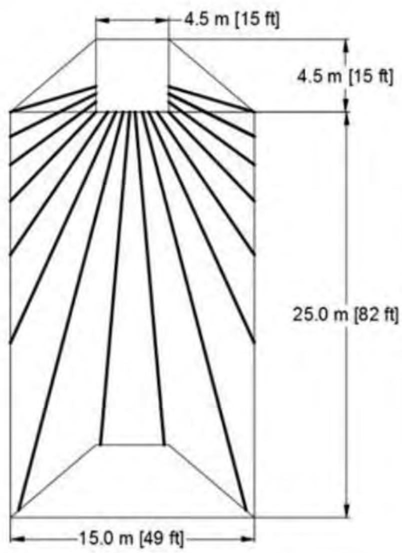
Source: SRK

Figure 24-44: Underground Mine Production Plan, Colored by Year

24.16.8 Mining Operations

Stoping

Stope lengths vary throughout the deposit ranging from 30 ft to a maximum of 100 ft giving a range of approximately 11,000 to 38,000 st per stope. After bottom and top accesses are established a slot raise will be developed at the far end of the stope. Drilling will continue with the longhole drill using a fan shaped pattern as shown in Figure 24-45. Holes will be loaded with bulk emulsion and stope blasting will commence in the slot and subsequently rings will be blasted retreating toward the level access.



Hole	Stemming	Drilled Length (ft)	Stemming Length (ft)	Charged Length (ft)
1	Ts	17.7	5.83	3.6
2	Tm	17.4	14.58	0.9
3	Ts	19.7	5.83	4.2
4	Tm	24.9	14.58	3.2
5	Ts	40.4	5.83	10.5
6	Tl	70.5	36.46	10.4
7	Tm	68.6	14.58	16.5
8	Tl	67.3	36.46	9.4
9	Tm	67.3	14.58	16.1
10	Tl	68.6	36.46	9.8
11	Tm	70.5	14.58	17.1
12	Ts	40.4	5.83	10.5
13	Tm	24.9	14.58	3.2
14	Ts	19.7	5.83	4.2
15	Tm	17.4	14.58	0.9
16	Ts	17.7	5.83	3.6
Ring Blast Total		652.9	246.5	123.9

Source: SRK, 2016

Figure 24-45: Typical Stope Drilling Section

Remote mucking will be required for the majority of stope mucking so the LHD operator can remain behind the brow of the stope. Stope material will be mucked into a muck bay near the level access. The material will then be loaded into trucks and hauled to surface. Once the stope is emptied a bulkhead will be placed in the 15 ft x 15 ft access and the stope void will be filled with cemented or non-cemented waste rock fill. Backfilling will occur from the top stope access.

Development

Drifting development such as main ramps and level accesses are sized as 16 ft wide x 18 ft high openings with an arched back. Drifting top/bottom stope accesses are sized as 15 ft x 15 ft flat back openings. These dimensions provide enough room for equipment, ventilation ducting, and utilities where necessary. Main ramps are typically a single heading environment. Level accesses are also typically single heading environments. Drifting top/bottom stope accesses are multiple heading environments. All development will be mined using a double boom jumbo taking 14 ft rounds. Blasted material will typically be mucked into a muck bay near the heading. The waste muck will subsequently be loaded into trucks and transported to surface or to the underground waste rock storage bin for use as backfill.

The ramp system is designed at a maximum gradient of 14%. A turning radius of 80 ft was used, which is suitable for any underground truck.

Mine Access

The mine will be accessed through separate ramps from surface for each area. Secondary egress for each area will be via raisebored ventilation raises equipped with emergency escape systems. The portals for all areas are located approximately 50 ft below the saprolite material. The portals will have a concrete formed entrance with shotcrete and mesh around the portal as required. A large culvert will be installed in the mouth of the portal and extend approximately 100 ft into the rock (length of culvert will depend on results of future study). Prior to developing the portals geotechnical drilling should be completed near the portal/ramp area to determine ground conditions prior to excavation.

From the portal, the open pit surface haul roads are used meaning both surface and underground trucks will be present.

Haulage

The underground mine will use 15 st LHD's that muck from the stopes and development headings to a muck bay or into 44 st underground trucks for haulage to surface crushing facilities. Approximate haulage distances from the various mining areas to the crusher/plant facilities are summarized in Table 24-40. The approximate number of trucks required to meet the demands of the production schedule for each area are also summarized in Table 24-40. The truck count includes waste haulage and backfill.

Table 24-40: Approximate Haul Distances and Haul Truck Requirements

Block	Haul Distance (ft)	Number of Trucks
Horseshoe – above sill	9,600	4
Horseshoe – middle portion (below sill)	12,800	5
Horseshoe – lower portion	15,000	6
Mustang	13,800	5
Mill Zone Deep	12,100	5

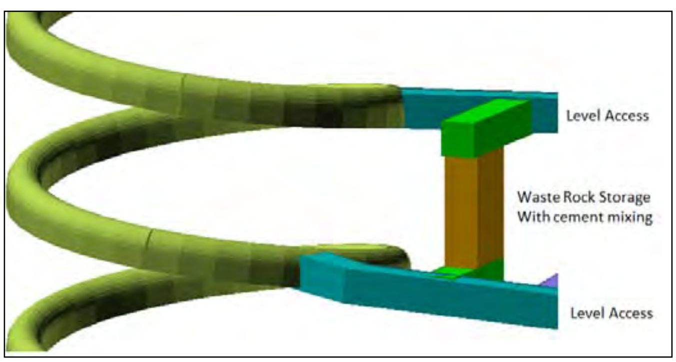
Source: SRK

Stockpiling or sorting of the underground material has not been considered at this time. It is assumed the mill will have sufficient capacity on a daily basis to process the material from the underground plan.

Backfilling/Waste Rock

All stopes will be backfilled using either a cemented rock fill (CRF) or waste rock. The CRF will be made up of sized rock (-3 inches) mixed with a 4% cement. Testwork on the material has not been completed at this time. Future testwork should include grain size distribution screening and variation to maximum aggregate sizes, cement/water ratios, and variable cure time. Strength tests should target UCS strengths in the range of 0.5 to 1.0 Mpa.

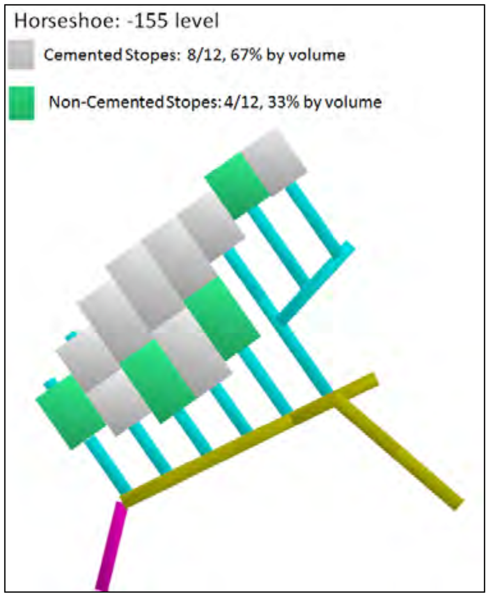
A waste rock storage area has been included underground where a top access is used for depositing sized material in the bin, and a lower access is used to load the haul trucks with CRF as shown in Figure 24-46. The storage area has a place for cement addition and an auger mixes the cement and waste rock mixture to provide a consistently mixed cemented rock fill to the stopes.



Source: SRK

Figure 24-46: Example Waste Rock / CRF Storage Area

A primary/secondary stoping sequence will be used where, on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the adjacent primary stopes have been mined, backfilled and have had time to cure. On a given level, approximately 70% of the stopes are considered primary (requiring cemented fill) and 30% are secondary (waste rock fill only). This is due to limiting the length of the stopes for geotechnical stability and therefore creating more primary stopes as shown in Figure 24-47.



Source: SRK

Figure 24-47: Example Level Showing Primary/Secondary Stopes

Backfilling will be an integral part of the mining cycle as there is a limited quantity of stopes available on a level. At times mining will need to occur on multiple levels to sustain production. Current cure time assumptions are seven days prior to driving on CRF and 14 days prior to mining adjacent to a CRF filled stope. These cure times should be verified with lab testwork using site specific material.

In the first two years, it has been assumed that waste rock from the Horseshoe area will be placed on surface in the PAG storage area. In subsequent years the mine is able to accommodate all the generated waste into open stopes. A temporary stockpile will be used when necessary, located near the main waste dump north of the Horseshoe area. The open pit shot pattern will be modified slightly when backfill material is required to provide appropriately sized material to supplement the underground development material. Table 24-41 shows a material balance by year.

Table 24-41: Waste/Backfill Material Balance (yd^3)

All Waste Movement Summary-Simple Mass Balance	2018	2019	2020	2021	2022	2023	2024	2025	Total
Development Waste	69,703	193,293	81,577	24,353	105,964	75,449	92,694	7,814	650,849
Swelled Waste Produced (40% swell)	97,585	270,611	114,209	34,093	148,350	105,629	129,772	10,940	911,189
Backfill Volume Required		210,594	317,981	328,685	329,549	318,482	350,006	154,096	2,009,395
Net Extra or (Shortage) for Stope Backfill	97,585	60,015	(203,772)	(294,592)	(181,200)	(212,853)	(220,234)	(143,157)	(1,098,205)
Stockpile Amount	97,585	157,602							
Net Needed from Other Source			203,772	294,592	181,200	212,853	220,234	143,157	1,098,205

Source: SRK

Geochemical samples from the three underground mining areas were collected and tested to determine the metal leaching and acid rock drainage (ML/ARD) potential of waste rock. Additionally, an open pit ARD model exists and is based on samples in/near the pit area. As the underground development begins near the pits, a combination of the open pit model and results from the underground sampling program were used to predict the ARD potential of underground development workings. Table 24-42 shows the estimated PAG material categories for each area where Category 1 material is classified as red or strongly acid generating, Category 2 is yellow or moderately acid generating, and Category 3 is green or non-acid generating.

Table 24-42: Estimated PAG Classification for UG Waste Material

Area	Cat 1 – Red (%)	Cat 2 – Yellow (%)	Cat 3 – Green (%)
Horseshoe	0.2	1.0	98.8
Mustang	24.6	36.8	38.6
Mill Zone Deep	66.2	28.1	5.7

Ground Support

The current knowledge of geotechnical characteristics indicate that ground support will be required in the ramps and accesses. Support is currently not required inside the stopes. The ground support plan in the ramps and main access drifts includes use of 8 ft long split sets with mesh with bolting on a 4 ft spacing. Cable bolting will supplement the standard pattern at the intersections and at the brow of the stopes. Shotcrete and other forms of ground support will be used as required. The current plan includes ground support in excess of that required by the geotechnical data evaluated to date (Table 24-27). The ground support plan is consistent with current best practices.

Table 24-43 shows the expected ground control systems for the various locations in the mine.

Table 24-43: Ground Control System

Location	System
Ramp/Level Access drifts	Systematic bolting with split sets and mesh with cable bolts at intersections
Through the stope development drift	Systematic bolting with split sets
Stopes	Spot bolting as necessary with cable bolts at brow
Other locations	Ground support as needed

Source: SRK, 2016

Grade Control

As the main ramp is developed, drill stations from the main ramp allow for fan drilling of the deposit prior to developing levels. This confirmatory drilling should be used to update the long term block model and provide confidence in expected tonnage and grades prior to level development.

The definition drilling will be very important at depth in the Horseshoe area where there is currently limited drilling. The Mustang area is also quite deep and would require drilling through historic stockpiles if drilling from surface. Underground definition drilling should occur as early as possible and changes to the location of the main ramp may need to be considered depending on drilling results.

Once a level is being developed, stope accesses will be sampled to determine material destination. This sampling can occur through use of a handheld XRF instrument or, samples can be taken from longhole cuttings and tested in the on-site lab.

Once stope accesses are developed vertical holes will be drilled through the anticipated stopes and cuttings will be sampled to determine stope extents and estimated stope grades. Any samples tested in the lab should be used to update short term planning block models to better estimate tonnages and grades in the short term mine plan.

24.16.9 Mine Services

Development Ventilation

The main declines at the Horseshoe and Mustang areas are approximately 4,000 ft long prior to tying into a ventilation raise to surface. This distance is developed using vent tubing and a fan system. In both cases the distance prior to tying into the ventilation raise could be shortened if need be by breaking up the raise to surface into two pieces.

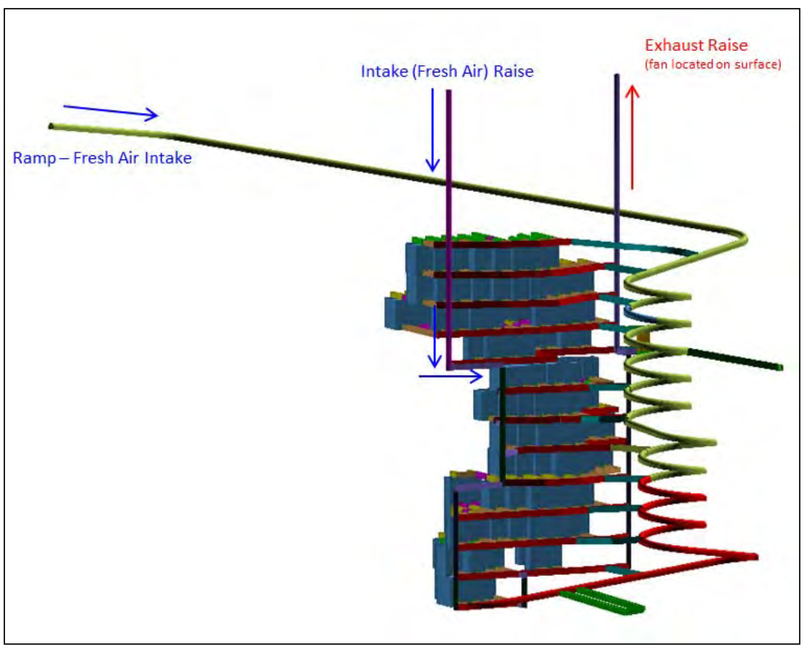
A 5.4 ft oval duct would provide sufficient air for the length of development and would accommodate the drift size and equipment. The fan system for the duct will incorporate three fans at the portal operating in series. The fans are added as the length of the development increases requiring modifications of the fan mounting arrangement as development progresses. A fiberglass duct could be installed for the first 1,000 ft of development with a booster fan located at the rigid duct/flexible duct transition. Rigid ducting would not decrease the overall pressure requirement; however, it would spread the pressure out along the duct system so that extreme pressures are not experienced directly at the fan installations. Required airflow during development is approximately 60,000 cfm based on two trucks and one loader operating in the development heading.

Once development at Horseshoe is complete the same development ventilation system can be used at Mustang, and later at the Mill Zone Deep area.

Primary Ventilation

Both Horseshoe and Mustang areas have two raises to surface, along with a ramp. One raise serves as a fresh air intake and secondary egress, and the other is an exhaust raise. Each level is connected to both the fresh air raise and the exhaust raise. Fresh air comes down the intake raise, blows across the footwall access of a level, and exhausts through the exhaust raise. The ramp is on intake and air flows onto working levels from the ramp and exhausts through the exhaust raise. The ramp air is used for development of lower levels prior to connecting levels to the main ventilation system. In the Mill Zone Deep area, a similar ventilation scheme is assumed however a single raise is used to surface meaning secondary egress is in the exhaust raise and the ramp is the single source of intake air.

The primary ventilation system in each area consists of a single fan installation on surface at the exhaust raise collar. A schematic of the ventilation at Horseshoe is shown in Figure 24-48.



Source: SRK

Figure 24-48: Horseshoe Airflow Directions

The main ventilation requirements at Horseshoe are estimated to be 435,000 cfm at 0.88 inches of water gauge (205 m^3/s at 2.2 kPa). This is based on projected airflow distribution through both the shafts and decline. Equipment assumed for the ventilation calculation is shown in Table 24-44. It was assumed that the whole of the airflow is drawn to the base of the mine and then exhausted to surface. This provides a measure of conservativeness for the calculation that would offset shock losses and other minor considerations.

Table 24-44: Equipment List used for Airflow Calculations(1)

Type of Equipment	Equipment / Manufacturer	Power (hp)	LoM Qty	Utilization (%)	Required Airflow (cfm)
LHD-3.9 yd^3	Sandvik LH307	201	1	75%	19,070
LHD-8.4 yd^3	Sandvik LH517	978	3	75%	278,039
Haul Trucks – 55st	Sandvik TH551	2,716	5	75%	1,287,220
Downhole Drill	Sandvik DL431-7 with LF706	295	2	25%	18,646
Slot and Raise Drill	Sandvik DTH DU311-T_6200				
Jumbo (2 boom)	Sandvik DD321-40	590	4	25%	74,585
Bolter	Sandvik DS411-C	295	2	25%	18,646
Scissor Lift	Getman - A64 Pipe Hanger/Fan Handler	175	1	10%	2,212
Scissor Lift	Getman - A64 Scissor Lift	175	1	10%	2,212
Shotcrete transmixer	Getman A64 HD R60	175	1	10%	2,212
Shotcrete equipment	Getman - Shotcrete DMA	201	1	10%	2,541
Anfo loader	Getman - A64 Anfo Charger	175	1	10%	2,212
Emulsion loader	Getman - A64 Emulsion Charger	175	1	10%	2,212
Road Grader	Getman - RDG1504C	147	1	10%	1,858
Fuel / Lube Truck	Getman - Lube/Fuel Truck	175	1	10%	2,212
Boom truck	Getman - Knuckle Crane Truck	175	1	10%	2,212
Tractors	Kubota - RTV1140CPX -2 seat	500	20	10%	126,404
Jackleg/Stoper	General Jackleg drill and leg		5		
Telescopic Handler	CAT 406C - Telescopic Handler	124	1	10%	1,570

(1) List used for airflow requirements will somewhat differ from final planned equipment list
Source: SRK

The minimum dimensions and airflows are shown in Table 24-45. The fresh air raise diameter was selected to ensure velocity is low enough to allow for secondary egress. Alternatively, a smaller raise (13.1 ft diameter) could be used with a VFD to control the airflow. Note that the exhaust raise has been designed to 16.4 ft diameter to allow for use of the same raisebore machine, thus reducing the velocity in the exhaust raise.

Table 24-45: Airway Dimensions and Airflows

Airway	Airflow (cfm)	Dimensions (ft)	Velocity (ft/s)
Exhaust	435,000	13.1 ft diameter	53.54
Decline	108,000	16.4 ft wide x 18 ft high	6.23
Fresh Air Raise	327,000	16.4 ft diameter	25.69

Source: SRK

Airflow quantities and general ventilation schemes at Mustang and Mill Zone deep are assumed to be similar and have not been modeled at this time.

Auxiliary Ventilation

An auxiliary ventilation system is required to provide ventilation from the level access to the stopes. A production heading is assumed to only have one 1,000 hp LHD requiring 40,000 cfm of airflow. A 45 hp fan could be used with ducting to provide this airflow.

Ramp and level development are assumed to have one 1,000 hp LHD and a 540 hp haul truck requiring 110,000 cfm of airflow. These types of headings can be ventilated with a 405 hp fan and ducting.

Dewatering

Various areas of the mine are expected to produce varying amounts of water based on the geology and sequencing of the underground and open pits. If underground mining occurs near an active open pit area, the expected water inflow would be lower. Based on the CDM Smith work discussed in Section 24.16.3, the mine is expected to produce the amounts of water shown in Table 24-46. In the Horseshoe area, a surface dewatering well is located near the main ramp which helps to lower the quantity of water underground. Designed pumping capacities to handle the predicted peak inflows are also shown in Table 24-46.

Table 24-46: Predicted Water Inflows

Area	Predicted Peak Inflow (gpm)	Designed Pumping Capacity (gpm)
Horseshoe (1)	375	500
Mustang	70	200
Mill Zone Deep	125	250

(1) Considers an active dewatering well on surface near the main ramp.
Source: SRK, CDM Smith

The large peak inflows are expected to be encountered in the decline near surface. The weathered metavolcanics in this area have an enhance permeability as compared to the other units. Once through the weathered zone, lower amounts of water are anticipated.

The general dewatering plan includes a surface system from the underground portals to the water treatment area located approximately 1,300 ft north of the crusher area. Main decline development is completed using an underground mine development dewatering system (portable development system). The portable development system is planned to be used for the Horseshoe development and re-used for the Mustang and Mill Zone Deep development. Separate permanent underground dewatering systems have been designed for the Horseshoe, Mustang and Mill Zone Deep areas.

Surface Systems

The surface systems include water tanks at the underground portals for the underground dewatering system to pump into. Surface water pumps then pump to a lined pond near the road to the crusher. The piping from the portals to the lined pond is assumed to be HDPE. Another set of surface water pumps then pump from the lined pond to the water treatment area. Piping from the lined ponds to the water treatment area is again assumed to be HDPE. The surface piping for the Horseshoe area is separate from that piping used for the Mustang and Mill Zone Deep areas. This is due to the duration of Horseshoe area production and concurrent Horseshoe/Mustang production. The surface piping for the Mustang and Mill Zone Deep areas is merged into one pipe north of the Mustang area.

Portable Development System (Underground)

The portable development system consists of submersible pumps, steel piping and tanks capable of advancing 1,100 ft along a 14% decline (165 ft of vertical advance). As the system is advanced down the decline, permanent steel piping is installed and the portable development system is moved down the decline.

A portable skid-mounted two-stage pumping system capable of pumping up to a 640 ft head follows behind the advance pumping system to pump water through the permanent steel piping to surface. The portable skid-mounted two-stage pumping system consisted of heavy duty centrifugal pumps pumping from a tank filled by the advance pumping system.

As the development advances permanent pumping stations are installed at planned locations.

Underground Dewatering Systems

For the Horseshoe area, permanent underground pumping stations are installed at -230 ft and -870 ft elevation as part of water collections drifts. In the Mustang area, the pumping stations are located at elevations of -130 ft and -650 ft. For the Mill Zone Deep area, a single underground pumping station is used located at an elevation of -100 ft. The permanent two-stage pumping systems consist of heavy duty centrifugal pumps pumping from a collection tank.

Table 24-47 summarizes various parameters concerning the Horseshoe mine dewatering systems. The 500 gpm design capacity assumes an active dewatering well on surface near the main decline.

Table 24-47: Horseshoe Mine Dewatering System Parameters (Design Capacity 500 gpm)

Horseshoe Mine Dewatering Area (with Wells)	Maximum Pipeline Length (ft)	Pipe Size Diameter (inches)	No. of Active Pumps	Pump Rating (hp)
Dewatering Wells	Installed	Installed	Installed	Installed
Surface System	7,200	8	2	40
Portable Development System – Advance Development System	1,300	6	1	75
Portable Development System – Portable Station	Uses Permanent U/G Piping	6	2 (1)	75
U/G Dewatering System – Station – 230 Elevation	4,250	6	2 (1)	75
U/G Dewatering System – Station – 870 Elevation	4,750	6	2 (1)	75

(1) Two-stage pumping required
Source: SRK, 2016

Table 24-48 summarizes various parameters concerning the Mustang mine dewatering systems, without any installation of dewatering wells from surface.

Table 24-48: Mustang Mine Dewatering System Parameters (Design Capacity 200 gpm)

Mustang Mine Dewatering Area	Maximum Pipeline Length (ft)	Pipe Size Diameter (inches)	No. of Active Pumps	Pump Rating (hp)
Dewatering Wells	None	None	None	None
Surface System	9,200	6	2	40
Portable Development System – Advance Development System	1,300	6	1	75
Portable Development System – Portable Station	Uses Permanent U/G Piping	6	2 (1)	75
U/G Dewatering System – Station – 130 Elevation	3,100	4	2 (1)	50
U/G Dewatering System – Station – 650 Elevation	3,900	4	2 (1)	50

(1) Two-stage pumping required
Source: SRK, 2016

Table 24-49 summarizes various parameters concerning the Mill Zone Deep dewatering systems, without any installation of dewatering wells from surface.

Table 24-49: Mill Zone Mine Dewatering System Parameters (Design Capacity 250 gpm)

Mill Zone Mine Dewatering Area	Maximum Pipeline Length (ft)	Pipe Size Diameter (inches)	No. of Active Pumps	Pump Rating (hp)
Dewatering Wells	None	None	None	None
Surface System (joins part of Mustang surface system)	5,575	6	2	40
Portable Development System – Advance Development System	1,300	6	1	75
Portable Development System – Portable Station	Uses Permanent U/G Piping	6	2 (1)	75
U/G Dewatering System – Station – 100 Elevation	3,275	6	2 (1)	50

(1) Two-stage pumping required
Source: SRK, 2016

It is anticipated the following personnel would be required to manage the dewatering systems:

- One person for surface dewatering operations for Horseshoe, and later an additional person for Mustang/Mill Zone Deep;
- Two persons for the Portable Development System;
- Two persons for each Underground Dewatering System (as required annually); and
- There would a total of four crews covering the two 12-hour shift rotation schedule of the full-time mine operations.

Other items Included in dewatering design/cost include:

- Lined surface pond for water management/surge control;
- Pumps, electric motors, piping, pipe couplers, tanks, and pumping control systems;
- Full back-up pumps and electric motors available, but no replacements estimated during LoM (overhaul costs included);
- Temporary and permanent pumping station underground development (excluding main water collection drifts in mining costs);
- General contingency for items required for physical installation, electric wiring, lighting etc.;
- General maintenance costs for dewatering system equipment;
- Estimated dewatering costs based on estimated flows with contingency allowance (not design flow capacities); and
- Estimated dewatering crew labor cost of US$31.50/hour with a 40% burden.

Underground developments of drifts for water collection systems are included in the mining cost. Water crew personnel service trucks are included in the general mining equipment list. Periodic sediment cleanout of water collection drifts will be required, assumed to be included in the mining cost. Labor for permanent piping installation is also assumed to be part of the mining cost.

Electrical Supply

Power to the underground mine will be supplied by a 4,000 ft new 13.8 kV overhead power line that will feed from the existing processing plant substation to a substation near the underground mine portal. The 13.8 kV power will be transformed to 4,160 V and will feed throughout the mine to main load centers where the power will be stepped down to 480 V for underground equipment use. Feeds will be provided at 110 V and 220 V for auxiliary use such as fans, pumps, and auxiliary lighting.

A diesel backup generator at the surface will supply backup power for the emergency hoist systems and required ventilation systems to maintain minimum ventilation requirements in the case of emergency.

The connected electrical load is estimated to be 2.0 MW during development and increasing to approximately 5.0 MW over the life of the mine. Table 24-50 summarizes the range of estimated loads.

Table 24-50: Typical Loads During Selected Time Periods (kW)

Load Category	Developing Horseshoe	Full Production Horseshoe	Horseshoe + Mustang	Mustang + Mill Zone Deep
Connected Load	2,117	4,798	5,371	5,371
Connected Load (94% Electrical Efficiency)	1,990	4,511	5,049	5,049
Maximum Demand (85% Demand Factor)	1,691	3,834	4,292	4,292
Average Demand (90% Load Factor)	1,522	3,451	3,863	3,863

Source: SRK, 2016

The main drivers of electrical consumption are the ventilation mine, mine pumps, and mine mobile equipment. These systems account for 88% of the load.

Health and Safety

The mine design incorporates MSHA safety standards and includes an emergency hoist in the fresh air raise. The hoist is connected to backup power generation. Additionally, a 12-person and an 8-person mobile refuge chamber are included that will be located in active working areas over the LoM.

The mine will have a communications system that has both mine phones and wireless communication through a leaky feeder system. A mine rescue team will support the operation. The mine safety program will integrate with local providers in case of any mine emergency. A stench gas emergency warning system should be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Manpower

Manpower levels are estimated based on the production schedule and equipment needs. The estimate is based on owner mining using an operating schedule consisting of 12 hours per shift, two shifts per day, and 7 days/week. The 12 hour shift is supported by a four crew rotation on a rotating 14 day per month schedule that averages 168 hours on the job per month. The management and technical team are planned to work five 8-hour days per week.

Table 24-51 shows the required workforce.

Table 24-51: Typical Mining Labor by Shift

Day Shift	Qty
Mine Superintendent	1
Mine Planner	1
Maintenance Superintendent	1
Maintenance Planner	1
Maintenance Technician	1
Senior Mining Engineer	1
Geotechnical Engineer	1
Mine Planning Engineer	3
Surveyor	1
Mine Technologist	1
Geologist	3
Environmental Specialist	1
Total	16

Rotating Shift	Per Shift (Qty)	Total (Qty)
Mine Supervisor/Shift Boss	2	8
Safety / Mine Rescue / Training Supervisor	1	4
Bolter Operator	2	8
Blasters	2	8
Utility Crew	3	12
Fuel/Lube/Boom/Grader/Telehandler	2	8
LHD & Truck Operator	10	40
Longhole and Jumbo Operator	5	20
Laborer	9	36
Diamond Driller	2	8
Mine Maintenance Supervisor/Lead Hand	1	4
Mechanic	3	12
Mechanic Helper	3	12
Electrician	2	8
Grade Control Geologist	1	4
Total	48	192
Grand Total	64	208

Source: SRK

The workforce will vary from a low of 79 people in the first year of development to a high of 208 people starting in year 2 of production.

<u>Equipment</u>

The underground equipment needs were based on the production schedule are summarized in Table 24-52.

Table 24-52: Yearly Mobile Equipment Summary

			2018	2019	2020	2021	2022	2023	2024	2025	
Type of Equipment	Diesel (kW)	Electric (kW)	-1	1	2	3	4	5	6	7	Totals
LHD-3 m^3	150			1							1
LHD-6.4 m^3 (14T)	243		2	1	1						4
Haul Trucks - 40T	405		2	2	1	1					6
Scissor Lift	130		1								1
Scissor Lift	130			1							1
Shotcrete transmixer	130		1								1
Shotcrete equipment	150		1								1
Emulsion loader	130		1	1							2
Road Grader	110			1							1
Fuel / Lube Truck	130		1	1							2
Boom truck	130		1	1							2
Tractors	19		2	16							18
Skid Steer	53		1								1
Fork Lift	97			1							1
5 t forklift - all terrain high lift telehandler	97		1								1
Downhole Drill	110	80	0	2							2
Slot and Raise Drill		37	0	1							1
Jumbo (2 boom)	110	135	2	1	1						4
Bolter	110	70	1	1							2
Diamond Drill (Exploration)		40		1	1						2

Source: SRK

The estimate uses an equipment availability of 85% and an operator efficiency factor (job factor) of 90%. Each shift of 12 hours is reduced by 1.83 hours to represent shift change, lunch, miscellaneous operational delays, and travel to and from working areas. The delays equate to an operational utilization of 85%. This provides an equivalent working day of 20.34 hours or 10.17 hours per shift. This nets approximately 5,700 hours per year of mining time.

Equipment is shared between the mining areas where appropriate (i.e., trucks). Other equipment which is not as mobile is dedicated to a mining area thus reducing the overall utilization of the equipment fleet.

The equipment totals by pre-production and production year are summarized in Table 24-53. The later years include additional trucks and LHD’s due to increasing haul distance and mining of multiple areas at once.

Table 24-53: Fixed Equipment Listing by Year

			2018	2019	2020	2021	2022	2023	2024	2025	
Type of Equipment	Diesel (kW)	Electric (kW)	-1	1	2	3	4	5	6	7	Totals
Auxiliary Pumps		15	3.0	3.0	3.0						9.0
Refuge Chambers (12 person)		2		1.0				1.0			2.0
Refuge Chambers (8 person)		2	1.0					1.0			2.0
Mobile 200 amp welders		6	1.0	1.0							2.0
Mobile 400 amp welder		12	1.0	1.0							2.0
Portable Water Pump-small		7	3.0	2.0	2.0						7.0
Portable Water Pump-medium		11	2.0	2.0	1.0						5.0
HDPE Pipe Welder (8" and smaller)		5	1.0	1.0							2.0
Auxiliary Fans (Electric)		35	3.0	5.0	5.0						13.0
Mine Pumps/Piping/Install		283	1.0	0.5		0.5	0.8	0.6	0.5		3.9
Mine Dewatering Well installation (Surface)			1.0								1.0
Surface Offices and Changehouse		1			1.0						1.0
UG Shop		1			1.0						1.0
UG Offices		5			1.0						1.0
Shipping Container Storage			2.0								2.0
Communications Infrastructure and Mine automation		10		1.0	-			0.3			1.3
Sump Construction				1.0	1.0		1.0	1.0	1.0		5.0
Tool Allowance (UG)				1.0							1.0
Engineering Equipment Allowance				1.0							1.0
Emergency Hoist		29		1.0							1.0
Ventilation System		1,990	0.4	0.6	-						1.0
Powder/Primer Storage			1.0								1.0
Power System (transmission, subs, switchgear, plus UG)			1.0								1.0
Compressor (Electric)		480	1.0								1.0
Shipping Container Storage			2.0								2.0
Backup Generation	1,799			1.0							1.0
Backfill Cement Mixing/ Feeder System		15		1.0				1.0	1.0		3.0

Source: SRK

24.17 RECOVERY METHODS

The underground feed material will be processed the same way as the open pit material. Process recoveries are anticipated to follow the same recovery curve and sufficient metallurgical test work exists across the property to support these assumptions at a PEA level. Mill feed will be a blend of the open pit and underground material.

24.17.1 Mineral Processing

A preliminary review of the plant design criteria and equipment sizing has been undertaken envisaging a process plant capable of treating up to 9,120 st/d of material. The main items identified to date that are likely to be required at the higher capacity include:

- ROM Pad upgrade;
- Recycle Crusher;
- Flash Flotation Cleaner Cell installation;
- Rougher Cell installation;
- Regrind Tower Mill installation;
- Flotation Tailings Thickener replacement;
- Leach Tank Installation;
- Cyanide Recovery Thickener replacement;
- Tailings Pump upgrade;
- Tailings Line replacement; and
- Slurry Pump Motor upgrades.

<u>ROM Pad</u>

The current ROM pad is designed for the initial 2.5 Mst/y (2.3 M metric t/y) throughput utilizing the permitted space available. With the increased throughput and grade variation with underground sourced material additional blending will be required. The ROM layout will be reviewed in subsequent phases of project evaluation to ensure that this can be accommodated.

<u>Recycle Crusher</u>

Installation of a pebble crusher on the scats transfer system and increase in grate slot size would allow removal of critical size material at higher milling rates to be crushed below discharge screen size. This would be incorporated into the existing structures as much as possible. This will be reviewed in subsequent phases of project evaluation to ensure that this can be accommodated, and requirements relating to Haile’s air permit would be reviewed.

<u>Flash Flotation Cleaner</u>

Installation of a flash cleaner cell after the flash flotation rougher would allow increased recovery from the flash circuit without increasing the mass flow to the regrind circuit. Increasing the grade of the flash flotation concentrate will proportionately reduce the mass to be reground in the existing regrind circuit and minimize overall energy consumption.

<u>Rougher Cell</u>

The current rougher circuit consists of four 65 yd3 rougher cells with a nominal residence time of 16 minutes. The increased throughput would decrease the residence time to under 12 minutes, and pending kinetic performance data, an additional rougher cell would be required to maintain residence time and recovery.

<u>Regrind Tower Mill</u>

The current regrind circuit consists of six Stirred Media Detritors (SMD) mills in two trains with a total of 2,850 hp (2,130 kW) of installed power. The increased tonnage of concentrate will exceed the capacity of the regrind to achieve the 13 um target grind size from the effect of mass and feed size. The lowest capital cost option is likely to be a tower mill ahead of the existing regrind circuit to reduce the feed size to around 38 µm.

Flotation Tailings & Cyanide Recovery Thickeners

The current thickeners are 75 ft diameter units (22 m) with a design flux rate of 0.0727 st/ft^2/h (0.71 mt/m^2/h) with this increasing to around 0.113 st/ft^2/h (1.1 mt/m^2/h) at the higher throughput rate. At this point in time, without proven settling tests and thickener operation, it may not be possible to accommodate the increased throughput with the existing units, requiring either a parallel second unit for each duty or a larger 92 ft (28 m) diameter unit to achieve the same flux rate.

Leach Tank Capacity

The current leach circuit is based on 24 hours of leach residence time for the reground concentrate and a further 20 hours with the addition of the flotation tailings. At the higher throughput cases two to three additional leach tanks would be required to maintain the overall residence time to maintain overall recovery. These would be constructed adjacent to the current pre-aeration tank at an elevated level to flow through the existing leach train.

Tailings Pump Upgrade

The current tailings line is based on an 8 inch HDPE line with two stage pumps with 250 HP of installed power. The increased flow rate would require a motor upgrade to around 550 HP for the base case and a third stage of pumps to overcome the head. Increasing the pipe line size to 10 inch would allow the existing two stage pumps to handle the increased flow with a minor motor upgrade for the third lift of the tailings embankment. As required, requirements relating to Haile's TSF/air permit would be reviewed. This will be investigated further in the next phase of project evaluation.

Slurry Pump Upgrades

A number of the slurry pumps in the flotation and thickening sections as currently designed to utilize 80% or more of their installed motor power. In most cases a preliminary review of pump curves indicate the pump wet ends are suitable for the increased duty but will require larger motors and variable speed drives to handle the increased load. In some cases, the discharge lines may need to be enlarged as a more cost effective upgrade. The majority of the motors in question are below 50 hp.

24.18 PROJECT INFRASTRUCTURE

Sections of project infrastructure are the same as that stated in Section 18. Additions regarding the underground are described in the following sections.

24.18.1 Tailing Storage Facility

The Tailings Storage Facility (TSF) has a design capacity of 40 Mst of tailings storage at full capacity. The open pit mine plan deposits approximately 34 Mst of tailings over the LoM. The incremental storage of approximately 4.8 Mt of additional tailings from the underground mining operation will fit within the currently designed facility, with additional water management efforts late in the life of the facility.

24.18.2 Overburden Storage

The existing overburden storage areas will be used for the storage of an additional 250,000 st of overburden from the underground in the first two years of development of the underground mine. After this period, overburden from the surface, approximately 2.1 Mst, will be required to provide supplemental backfill material for the underground mine. Table 24-54 shows the incremental material to the stockpile (positive number) and into the underground. The addition of the underground operation results in a net 1.8 Mst reduction in overburden storage.

Table 24-54: Incremental Overburden Movement to (-) or from (+) the Underground

Year	Tons (000's)
2018	154
2019	96
2020	(328)
2021	(481)
2022	(301)
2023	(355)
2024	(368)
2025	(239)
Total	(1,786)

Source: SRK

24.18.3 Backfill

Mining Backfill is discussed in detail in Section 24.16.

24.18.4 Site Wide Water Management

Dewatering lines, pumps and associated infrastructure (including power) will move water from the underground areas to the existing contact water treatment plant (CWTP) via the contact water lined pond (the 19 pond). Suitable storage capacity will be maintained in the contact water lined pond to allow continued operation of the underground area after storm events give rise to additional generation of contact water collecting in operating open mine pits that will require treatment at the CWTP.

Where depressurization wells are employed to reduce inflows into the declines, the groundwater abstractions will be pumped via pipelines to storages from where water can be released directly to the environment or used for various purposes in the mine, such as dust suppression and process plant water supply.

Additional monitoring infrastructure (e.g. wells and vibrating wire piezometers) may be required to complement the existing groundwater and surface water monitoring network that is installed within and outside the mine Permit Boundary. The existing monitoring plan will be updated to incorporate the monitoring requirements (water levels and flow, and quality) for the new infrastructure.

The site wide water balance for the mine will also be updated to incorporate water-related information for the underground areas.

24.18.5 Ancillary Facilities

The underground mine will utilize the existing open pit facilities. The underground design envisions the addition of a mine dry and change house as well as some offices in a facility that will be located in the general area of the Horseshoe portal. An allowance for this facility has been included in the capital cost estimate.

24.18.6 Power Supply

Power to the underground mine will be supplied via a new 4,000 ft 13.8 kV overhead power line that will feed from the existing processing plant substation to a substation near the mine portal. The incremental power system is discussed in Section 24.16.9.

24.19 MARKET STUDIES AND CONTRACTS

Market studies and contracts are the same as stated in Section 19.

24.20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL AND COMMUNITY IMPACT

Underground mining operations at the OceanaGold Haile Operation (Haile) would require a modification of Haile's state Mine Operating Permit (Mine Permit) issued by the South Carolina Department of Health & Environmental Control (DHEC), and Haile's associated Mine Plan and Reclamation Plan. Due to the cost of reclaiming the underground operations, Haile would likely be required to increase its current USD$65 million reclamation bond. DHEC would also determine whether a modification of any of Haile's other state-issued permits was required, including Haile's Air Permit and TSF Dam Safety Permit, as well as its 401 Water Quality Certification. If the underground mining operation would impact Waters of the United States not currently authorized to be impacted by Haile's federal 404 Clean Water Act Permit (404 Permit), issued by the U.S. Army Corps of Engineers (the Corps), then Haile would be required to obtain another 404 Permit. If a 404 Permit were required, the Corps would determine what level of the National Environmental Policy Act (NEPA) of 1970 review was applicable. Other state and federal agencies, nongovernmental organizations, and the public would be afforded an opportunity to participate in the permit modification process.

The permitting modification time frame would decrease/increase depending on the level of stakeholder participation, including participation by other state and federal agencies, as well as the nature of the anticipated impacts to the physical and human environment. Haile's application would require a high degree of technical confidence. If a 404 Permit was required, and NEPA applied, it is anticipated that the Corps would again be the lead federal agency, in keeping with Haile's previous 404 Permit application. The timeframe to complete the 404 Permit application process would be primarily dependent upon the Corps' determination as to what level of NEPA review applied. Although NEPA statistics are Agency-dependent and relatively difficult to come by, an Environmental Impact Statement (EIS) may take several years to complete, an Environmental Assessment (EA) may take a year or more, and a Mitigated Finding of No Significant Impact may take several months to a year or more.

24.21 CAPITAL AND OPERATING COSTS

Capital and operating costs were generated for the underground mine. The open pit costs have not been altered and are discussed in Section 21. Where expansions to the open pit process plant, infrastructure, etc. are required to support underground mining, costs have been included here in the underground section.

The capital and operating expenditure discussed in this section must be read in conjunction with the cautionary statement in the introduction to Section 24, explaining that there is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realized.

24.21.1 Underground Capital Cost Estimate

Table 24-55 contains a summary of capital costs for the underground development and operations of the Project. Capital costs contain the design, procurement and construction of the underground mine and necessary additions to the existing surface processing plant, auxiliary facilities, and infrastructure. At this level of study, and with the work performed to-date, the underground capital cost estimate is at an accuracy of ±40%. An overall LoM contingency of 7.3% was determined by applying 10% and 15% contingency to mobile and fixed mine equipment respectively, 0% contingency for mine development costs, and 15% for process plant expansion. The majority of the capital is mining equipment where the basis for cost was budgetary pricing for single pieces of equipment. During actual purchase fleet discounts would be applied reducing the prices of individual units. As such a low contingency has been applied to offset the cost basis. Development costs were calculated from first principles and do not include contingency.

Table 24-55: Underground Capital Cost Summary

Description	Initial (US$000's)	Sustaining (US$000's)	LoM (US$000's)
Mining Equipment/Assets	33,017	8,719	41,736
Mine Development	20,068	26,324	46,392
Equipment Rebuilds	0	9,499	9,499
Subtotal Mining Capital	53,085	44,542	97,627
Processing Expansion to 9,120tpd	14,000	0	14,000
Subtotal Capital Before Contingency	67,085	44,542	111,627
Contingency	6,868	1,399	8,267
Total Capital	**$73,953**	**$45,941**	**$119,894**

Source: SRK, 2016

Mining Capital Cost

Mining capital cost contains an estimate for the underground development, underground mining equipment and infrastructure, underground pumping stations, waste rock backfill storage/mixing area, ventilation excavations and mechanicals. Initial capital requirements in the pre-production years is US$57.9 million and sustaining capital requirements total an additional US$45.9 million throughout the LoM as shown in Table 24-56.

Table 24-56: Underground Mining Capital Cost Summary

Description	Initial (US$000's)	Sustaining (US$000's)	LoM (US$000's)
Mine Capital	33,017	8,719	41,736
Mine Capital Contingency @ 15%	4,768	1,399	6,167
Subtotal Mine Capital	37,785	10,118	47,903
UG Main Ramp	8,722	11,959	20,681
Level Access Drifts	6,128	6,807	12,934
Slot Production Raise	206	281	487
Raise Bore	4,242	6,493	10,735
Other - Excavations	22	24	46
Ventilation Connection Drifts	748	762	1,510
Subtotal UG Development Capital	20,068	26,324	46,392
Equipment Rebuilds	0	9,499	9,499
Total Capital	**$57,853**	**$45,942**	**$103,795**

Source: SRK, 2016

Processing Capital Cost

A preliminary estimate has been made by Oceana for each area of process facility upgrades based on the equivalent equipment cost and installation factor from the current plant and is summarized in Table 24-57. SRK added a 15% contingency to this estimate to reflect the early stage of the estimate.

Table 24-57: Process Facility Expansion Capital Cost Estimate

Area	Equipment Estimate (US$ millions)	Installed Estimate (US$ millions)
ROM Pad	$0	$1
Recycle Crusher	$1	$3
Flash Flotation Cleaner	$0.4	$1.2
Rougher Cell Upgrade		
Regrind Tower Mill	$1.2	$3.6
Thickener replacements		
Leach Tank Additions	$0.8	$2.4
Tailings Line Upgrade		
Slurry Pump Upgrades	$0.2	$0.4
EPCM @ 20%		$2.4
Total Before Contingency		$14
SRK Contingency @ 15%		$2.1
Total		$16.1

Source: SRK, 2016

24.21.2 Underground Operating Cost Estimate

The operating costs are based on processing 2,120st of mineralized material per day. The operating costs are based on 2016 costs, and the estimate has been broken down into three main areas: mining costs (mine), processing costs (process), and general & administration (G&A).

The mine operating cost is estimated at US$27.94/st of the mineralized material delivered to the processing operation and includes the manpower, energy, spares and maintenance supplies required for the underground development and production of the mineralized material, as well as underground backfill, underground pumping systems, and ventilation.

First order process operating budgets for Haile are still in progress. Preliminary estimates based on the split of fixed and variable cost components indicate expected baseline cost, with full cyanide kill, of US$10.88/st for the open pit only (7,000 st/d). This cost decreases to US$9.60/st with the addition of the underground material and throughput increase to 9,120 st/d.

The general & administration operating cost associated with the underground operation is estimated at US$7.16/st of the mineralized material milled. This includes all of the underground project's operating costs which are not related to the open pit mining and processing plants. The underground G&A includes mine supervision.

The overall LoM operating cost for the underground Project is estimated at US$44.70/st mineralized material milled. A summary of the operating costs for the Project is shown in Table 24-58. All costs presented in this section are in US dollars per underground mineralized material milled.

Table 24-58: Operating Cost Summary

Description	US$/t Processed	US$/st Processed	LoM (US$000's)
UG Mine	30.80	27.94	133,465
Process (UG tons only)	10.58	9.60	45,859
G&A	7.89	7.16	34,210
Total	$49.27	$44.70	$213,534

Source: SRK, 2016

24.22 ECONOMIC ANALYSIS

The economic analysis presented in this section combines the open pit feasibility study cash flow presented in Section 21 with the underground capital and operating costs discussed in Section 24.21. The open pit production schedule, capital, and operating costs were not altered in any way. Where expansions or additions were required to the open pit plant/infrastructure to support the underground mining operation, costs have been included in the underground mine plan. The cash flow numbers presented here reflect the combined production from the open pit and underground.

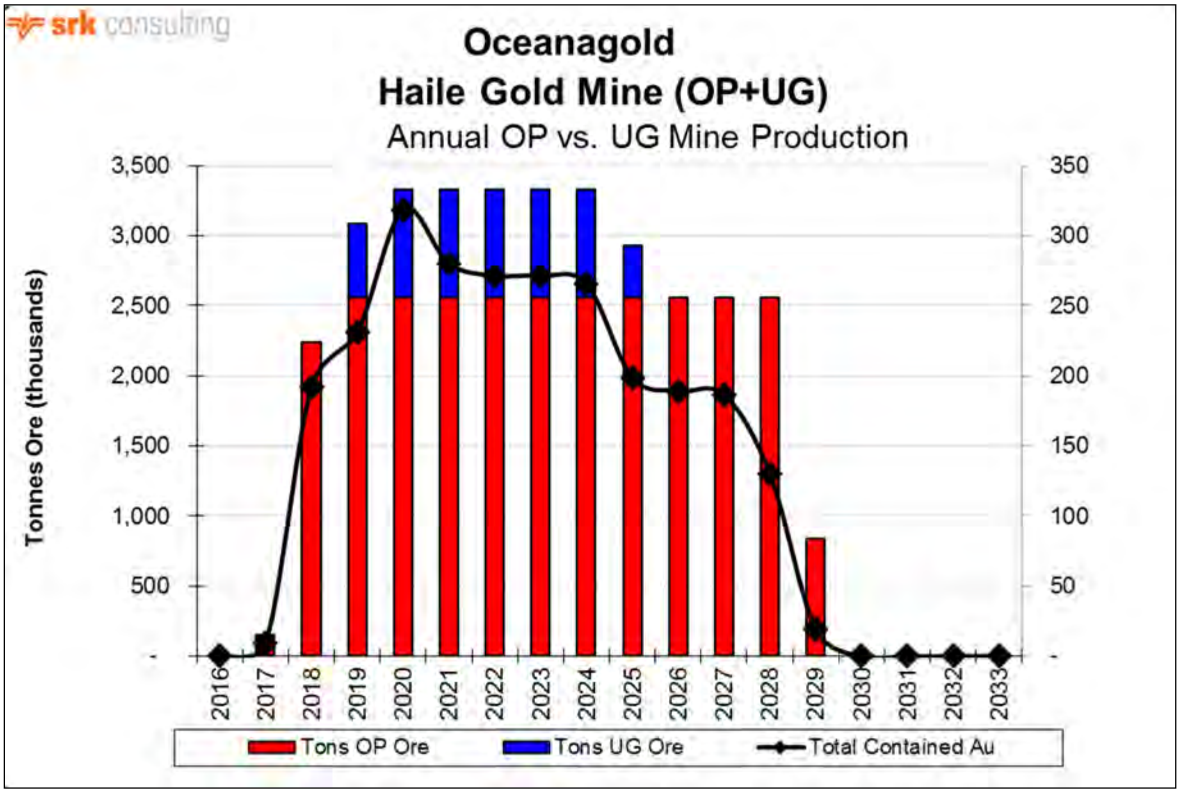
Physicals

Table 24-59 shows the LoM production summary from both the open pit and underground while Figure 24-49 shows the annual breakout of the open pit and underground mine tons. The proposed mine schedule has the highest gold ounce output in the first years of the underground operation which is optimal for the Net Present Value metric.

Table 24-59: LoM Mine Production Summary

Description	Value	Units
OP Mineralization Mined	28,780	kst
OP Waste Mined	241,340	kst
OP Total Material Mined	270,120	kst
OP Mined Grade	0.066	oz/st
OP Contained Gold	1,907	koz
LG Stockpile Mineralization Movement	4,850	kst
LG Stockpile Grade, Gold	0.020	oz/st
LG Stockpile Contained Gold	96	koz
UG Mineralization Mined	4,777	kst
UG Mined Grade	0.137	oz/st
UG Contained Gold	655	koz
Total Mineralization Mined	38,407	kst
Waste Mined	241,340	kst
Total Material Mined	279,747	kst
Daily Mining Capacity	98,010	st/d
Total RoM Grade	0.069	oz/st
Total Contained Gold	2,660	koz

Source: SRK, 2016



Source: SRK, 2016

Figure 24-49: Annual Open Pit vs. Underground Mine Production

Economic Results

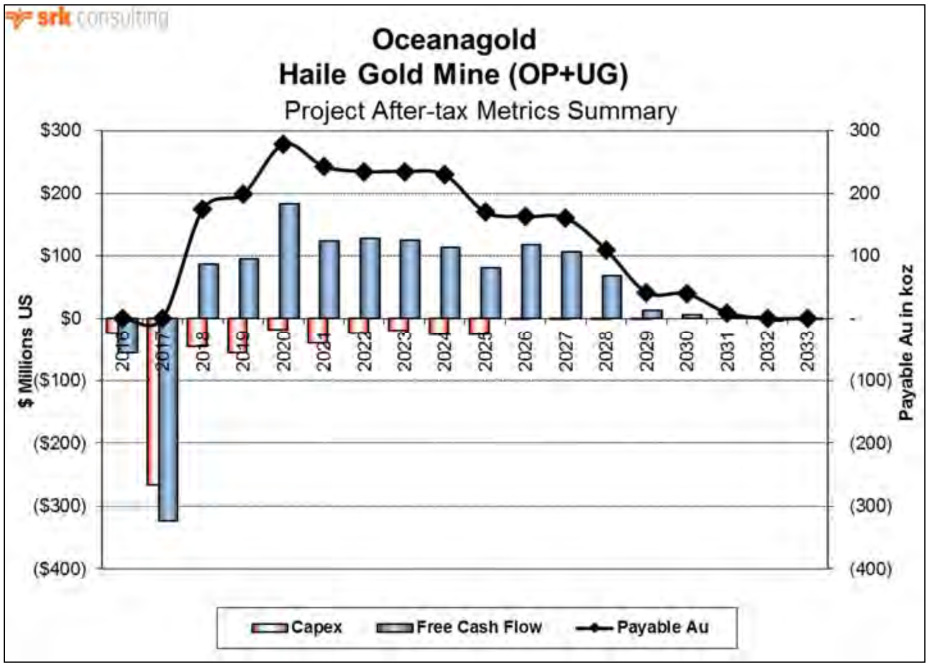
The economic results of the combined open pit and underground scenario and a comparison to the open pit only scenario are presented in Table 24-60. The Project has an after-tax NPV (5%) of US$510 million which is 75% higher than US$291 million for the open pit only scenario. The after-tax IRR of 23.2% is 39% higher than the open pit only scenario of 16.7.

Table 24-60: Indicative Economic Results (US$000's)

Description	OP+UG	OP Only	Variance
Market Prices			
Gold (US$/oz)	$1,250	$1,250	--
Silver (US$/oz)	$20.00	$20.00	--
Revenue			
Payable Gold (koz)	2,287	1,678	36%
Payable Silver (koz)	2,083	2,083	--
Total Gross Revenue	$2,900,083	$2,138,572	36%
Operating Costs			
Mining	(543,769)	(376,093)	45%
Processing	(388,861)	(339,943)	14%
Site G&A	(136,521)	(119,639)	14%
Selling/Refining	(4,916)	(6,126)	(20%)
Indirects	(125,140)	(125,140)	--
Total Operating Costs	($1,199,207)	($966,942)	24%
Operating Margin (EBITDA)	$1,700,876	$1,171,631	45%
Taxes			
Income Tax	(237,249)	(126,938)	87%
Total Taxes	($237,249)	($126,938)	87%
Working Capital	0	0	--
Operating Cash Flow	$1,463,627	$1,044,692	40%
Capital			
Initial Capital	(423,153)	(349,200)	21%
Sustaining Capital	(184,483)	(138,541)	33%
Reclamation/Salvage Capital	4,575	4,575	--
Total Capital	($603,061)	($483,166)	25%
Metrics			
Pre-Tax Free Cash Flow	$1,097,815	$688,464	59%
After-Tax Free Cash Flow	$860,565	$561,526	53%
Pre-Tax NPV @: 5%	$670,602	$371,796	80%
After-Tax NPV @: 5%	$509,650	$290,784	75%
Pre-Tax IRR	26.8%	18.7%	43%
After-Tax IRR	23.2%	16.7%	39%
BT Undiscounted Payback from Start of Comm. Prod. (Years)	3.2	4.2	(24%)
AT Undiscounted Payback from Start of Comm. Prod. (Years)	3.4	4.4	(23%)
Total Cash Costs (TCC) US$/payable oz	$554	$590	(6%)

Source: SRK, 2016
BT=before tax
AT-after tax

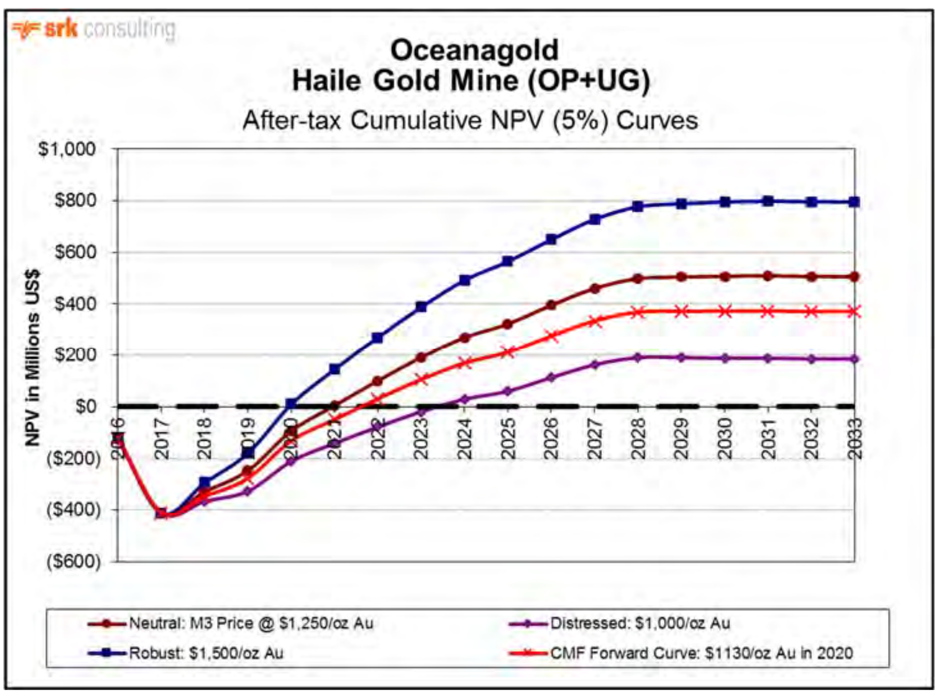
The Project's annual free cash flow and payable gold production is presented in Figure 24-50 which shows highest free cash flow (FCF) and gold production at the beginning of the underground mine life which is optimal for Net Present Value metrics. It is noted that the Project cash flow is fairly consistent through the mine life producing an average of 200 koz/y over the LoM until the last three years where it drops to 30 koz/yr.



Source: SRK, 2016

Figure 24-50: Total Project Annual After- Tax Metrics

The combined OP+UG Project as designed for this report is also robust in a range of gold prices as shown in Figure 24-51 which shows positive cumulative NPV (5%) curves above US$1,000/oz Au along with payback periods (starting at 2018) ranging from 2.7 years at $1,500/oz Au to 3.4 years at US$1,250/oz Au to 5.2 years at US$1,000/oz.



Source: SRK, 2016

Figure 24-51: After-Tax Cumulative NPV Curves

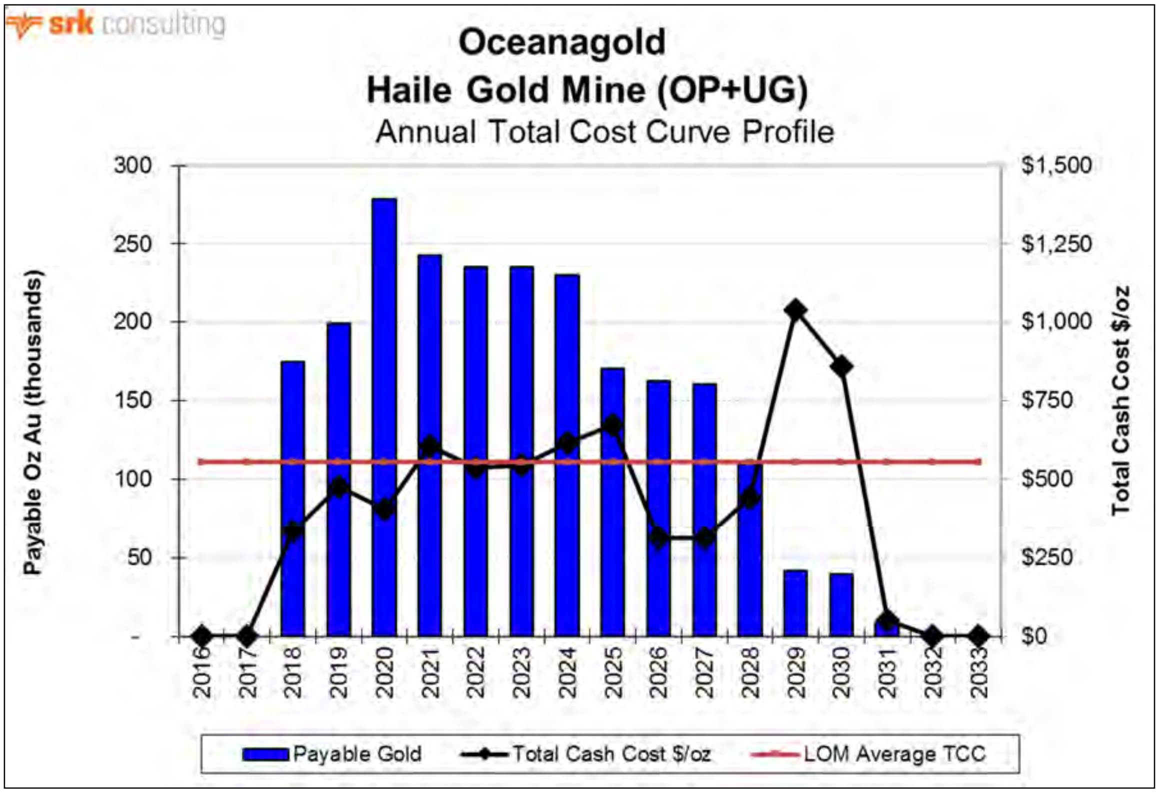
Cash Costs

Total Cash Costs (TCC) covers the period from the start of commercial production through end of mine (EOM) but also includes post closure reclamation/closure costs. The Project has a TCC of US$554/payable oz Au that includes an US$18/oz by-product credit as presented in Table 24-61. This figure is a 6% decrease from the open pit scenario TCC of US$590/oz. This reduction is driven mainly by the 36% increase in gold production over the LoM offset by 24% higher operating costs and 33% higher sustaining capital. The annual TCC cost profile is presented in Figure 24-52 which shows a consistent average TCC through 2025 followed by a drop in 2026-2028 resulting from the end of underground production and less mine development sustaining capital required. The dramatic increase in the last three years of operation (2029-2031) is common where gold production decreases at the end of mine life while still incurring a lot of costs.

Table 24-61: Total Cash Cost Contribution

Description	OP+UG		OP Only	
	Costs	$/oz	Costs	$/oz
Mining	543,769	238	376,093	224
Processing	388,861	170	339,943	203
Site G&A	136,521	60	119,639	71
Selling/Refining	4,916	2	6,126	4
Direct Cash Costs Before By-Product Credits	$1,074,067	$470	$841,801	$502
By-Product Credits	(41,656)	(18)	(41,656)	(25)
Direct Cash Costs Net of By-Product Credits	$1,032,411	$451	$800,145	$477
Royalties	-	-	-	-
Expensed Preproduction Mine Development	-	-	-	-
Mitigation	10,720	5	10,720	6
Concurrent Reclamation/Closure Costs	44,693	20	44,693	27
Indirect Cash Costs	$55,413	$24	$55,413	$33
Sustaining Capex	184,483	81	138,541	83
Reclamation/Closure/Salvage Capex	(4,575)	(2)	(4,575)	(3)
Sustaining Capex	$179,909	$79	$133,966	$80
Total Cash Costs	$1,267,733	$554	$989,525	$590
Total Payable Gold (koz)	2,287		1,678	

Source: SRK, 2016



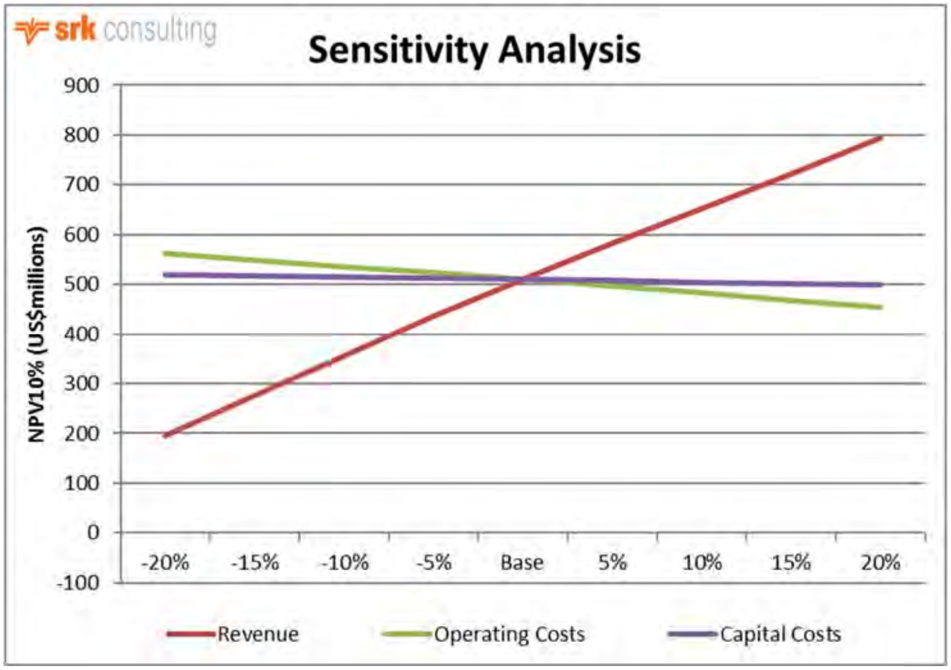
Source: SRK, 2016

Figure 24-52: Annual Total Cash Cost Profile

Sensitivity Analysis

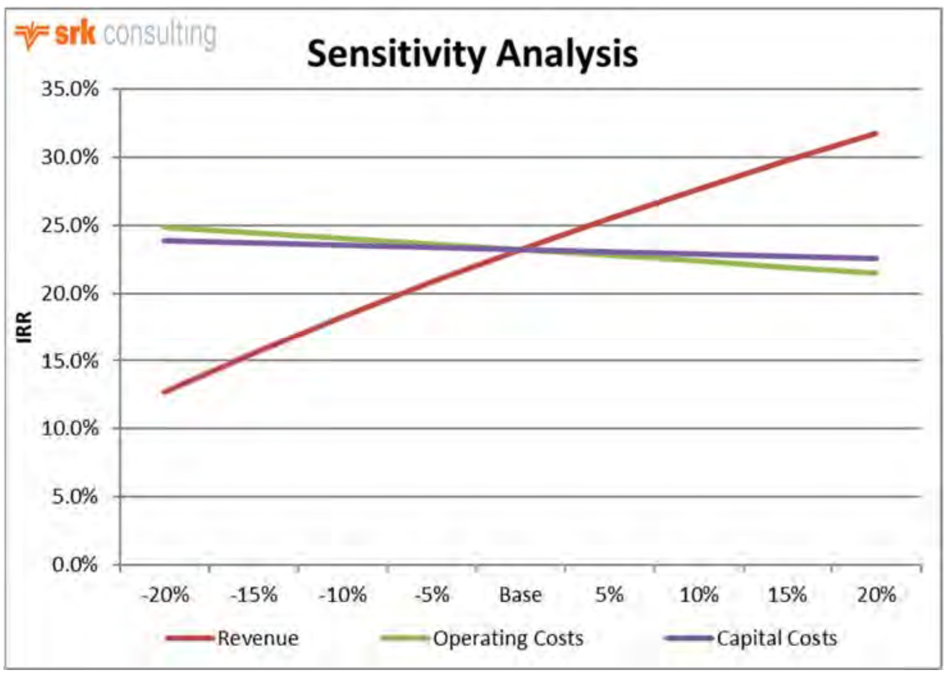
Figure 24-53 and Figure 24-54 show spider diagrams reflecting impacts to Project After-Tax NPV (5%) and IRR with incremental changes in gold price plus underground related operating and capital costs (mining, processing and G&A). The open pit related costs were not included in the sensitivity analysis in order to show the impact of the underground production.

Not surprisingly as with most gold projects, gold price has the highest impact on project economics but, in this case, changes in underground operating and capital costs have very little impact on project economics reflecting again a solid business case for the combined open pit and underground operation.



Source: SRK, 2016

Figure 24-53: Deterministic After-Tax NPV (15%) Sensitivity



Source: SRK, 2016

Figure 24-54: Deterministic After-Tax IRR Sensitivity

24.23 ADJACENT PROPERTIES

The adjacent properties information remains the same as that stated in Section 23.

24.24 INTERPRETATION AND CONCLUSIONS

24.24.1 Mineral Resource Estimate

The largest uncertainty in the resource estimates lies in defining the geometry of mineralization at the adopted underground mining cut-off. As yet, the controls on mineralization are not fully understood. Where there is sufficient drilling, such as in the areas classified as Inferred, the gross volume of mineralized material is considered to be reasonably well defined. Even if subsequent drilling results in a substantial change in interpretation of mineralization continuity, the estimates of tonnage and metal are likely to remain robust, to the level of confidence implied by the classification. The total resource is expected to be within +/- 30% on tonnes and metal.

The broad mineralization envelopes constrain the bulk of anomalous Au mineralization, but still contain a high proportion of grades that are below economic cut off grades (CoGs). Around 66% of these grades are in contiguous intersections of >6 m (19.7ft) length.

The model attempts to define the likely geometry of mineralization for a range of likely mining CoGs. The approach that has been used is to create indicator grade shells within the broad 0.25 g/t (0.073 oz/set) Au mineralization envelope at a range of different CoGs. Grades are then estimated within these shells using the data constrained within them (i.e. the samples used to estimate grades are identified by back-flagging the samples from the wireframes). This takes account of the smoothing implicit in the indicator geometry modelling.

In the Horseshoe area the general geometry of mineralization is reasonably well constrained by drilling. While additional drilling will likely reveal greater geometric complexity, it is unlikely to change the interpretation and consequent mine-designs substantially.

For the remainder of the potential underground resources, the generally lower grade means that interpretation of the geometry of mineralization at likely mining CoGs is less well defined. The resource as defined is highly sensitive to choice of CoG. An iterative approach to mine design will be required, based on preliminary analysis of a best-guess first model followed by progressive refinement.

It is important that mapping, drilling and geochemistry be gathered early from the open pits in order to improve geological understanding so that this can be used to better inform underground mining studies. In particular, close spaced grade control drilling should be obtained in order to understand short scale continuity of mineralization. It must be born in mind that the majority of mineralization targeted in the open pits is from the shallow dipping north-western limb of the deposit, while Horseshoe mineralization lies in the steeply dipping south-eastern part of the deposit.

In planning for underground mining, close spaced definition drilling should be included in schedules prior to development. It is better to start by obtaining too much drilling, then relaxing the drilling requirement, rather than attempting development with insufficient drilling and risking compromise of mine design and mineralization extraction.

In addition, a sufficient amount of drilling to define the geometry and location of diabase dykes should be planned for. At present, the majority of resource drilling is sub-parallel to the dyke orientations.

24.24.2 Mining and Mineral Reserve Estimate

Geotechnical

The rock mass characterization for the geotechnical investigation was based on geotechnical core logging, laboratory tests and rock mass parameters and are summarized as follows:

- A total of 62 laboratory tests were used to estimate the properties of intact rock. The majority of these tests were completed as part of the Oceana 2016 drill program;
- A good correlation between RQD and FF/m was obtained as a means of verifying the consistency of the field logging;
- Three geotechnical domains were defined using the RQD parameters and Q' rock mass classification from the core logging data. The domains include: a saprolite zone (25< RQD to about 100ft depth), a fracture zone (25< RQD < 75 to about 330 ft depth), and a non-fracture zone (RQD> 75 deeper than about 330 ft; and
- The underground stope geotechnical design parameters have been selected using empirical design methods based on published state-of-practice design charts created from case histories in similar ground conditions.

Geochemical

Results from the initial geochemical characterization of waste rock that would be generated by underground development of the Horseshoe, Mustang, and Mill Zone Deep areas indicate the following:

- Samples from the Horseshoe target are classified as not acid generating, based on both the Haile and industry classification criteria;
- All samples from the Mill Zone Deep area are classified as potentially acid generating or uncertain acid generation potential based on industry classification criteria. Based on the Haile classification criteria, the samples are strongly (red) and moderately (yellow) acid generating;
- Five of the eight samples from the Mustang area are classified as potentially acid generating or uncertain acid generation potential based on industry classification criteria;
- Based on the Haile classification criteria, three of the samples from Mustang would be classified as moderately (yellow) acid generating, but classified as PAG using industry criteria;
- Many of the metasedimentary samples are enriched in As, Sb, Co, Hg and Se, and might have environmental impacts if the waste rock is not properly managed; and
- The pyritic metasedimentary rock should only account for a small percentage of the total waste rock, but has to be given careful consideration during mine operations due to its very high sulfide content (15%) and high concentrations of As, Ag, Hg and Mo.

The results of the PEA level geochemical characterization should be considered preliminary, as they are based on a very small number of samples.

Mining

No Mineral Reserves have been estimated for the Project. The available data indicate that underground operations using longhole stoping methods with cemented backfill are viable for the Project.

- The current air permit allows for mining 9,120 st/d. The open pit plan mines 7,000 st/d and therefore the underground operation has been sized to produce 2,120 st/d making up the balance of the permitted capacity.
- For mine design purposes a minimum cut-off of 0.05 oz/st was used. The underground mine maintains a production of 2,120 st/d for seven years including ramp up and ramp down. The underground mine design process results in mine plan resources of 4.78 Mst (diluted) with an average grade of 0.14 oz/st Au. Three areas are included in the mine plan - Horseshoe, Mustang, and Mill Zone Deep.

- Access and infrastructure development underground was designed to support the mining method and sized based on mining equipment and production rate requirements. Surface infrastructure is shared with the open pit when possible.

24.24.3 Metallurgy and Processing

Process recoveries are anticipated to follow the same recovery curve and sufficient metallurgical test work exists across the property to support these assumptions at a PEA level. Mill feed will be a blend of the open pit and underground material.

A preliminary review of the plant design criteria and equipment sizing has been undertaken envisaging a process plant capable of treating up to 9,120 st/d of material. The main items identified to date that are likely to be required at the higher capacity include:

- ROM Pad upgrade;
- Recycle Crusher;
- Flash Flotation Cleaner Cell installation;
- Rougher Cell installation;
- Regrind Tower Mill installation;
- Flotation Tailings Thickener replacement;
- Leach Tank Installation;
- Cyanide Recovery Thickener replacement;
- Tailings Pump upgrade;
- Tailings Line replacement; and
- Slurry Pump Motor upgrades.

24.24.4 Environmental and Social

Underground mining operations would require a modification of Haile's State Mine Operating Permit (Mine Permit) issued by the South Carolina Department of Health & Environmental Control (DHEC), and Haile's associated Mine Plan and Reclamation Plan. Due to the cost of reclaiming the underground operations, Haile would likely be required to increase its current USD$65 million reclamation bond. DHEC would also determine whether a modification of any of Haile's other state-issued permits was required. If the underground mining operation would impact Waters of the United States not currently authorized to be impacted by Haile's federal 404 Clean Water Act Permit (404 Permit), issued by the U.S. Army Corps of Engineers (the Corps), then Haile would be required to obtain another 404 Permit. If a 404 Permit were required, the Corps would determine what level of the National Environmental Policy Act (NEPA) of 1970 review was applicable. Other state and federal agencies, nongovernmental organizations, and the public would be afforded an opportunity to participate in the permit modification process.

The permitting modification time frame would decrease/increase depending on the level of stakeholder participation, including participation by other state and federal agencies, as well as the nature of the anticipated impacts to the physical and human environment. Haile's application would require a high degree of technical confidence. The timeframe to complete the 404 Permit application process would be primarily dependent upon a determination as to what level of NEPA review applied.

24.24.5 Projected Economic Outcomes

Given the herein presented underground scenario, Oceana has a robust business case to having a 2,120 t/d underground mining operation run concurrently with the proposed open pit mine at the Haile project. With the combined

LoM operation producing an average of 200 koz/y of gold at a TCC of US$554/oz, SRK estimates that the operation would be comfortably within the second cash cost quartile of gold producers.

24.25 RECOMMENDATIONS

24.25.1 Geology & Recourses

The geological recommendations are as follows:

- To construct a 3D form line model based upon bedding measurements from orientated core;
- To update the 3D geological wireframe interpretations (primarily metasediment/metavolcanic contact and post-mineralization dykes) with all additional drilling;
- Use the form line model, geological interpretation and foliation data (from orientated core) to determine the extent to which these influence grade distributions;
- Update the resource model with recent drilling and revised geological understanding;
- Collate pit mapping and multi-element geochemistry (from grade control samples) to better understand the short scale controls on mineralization at Haile;
- Investigate whether underground reverse circulation drilling could be used to supplement diamond drilling to improve grade control definition for stoping;
- Assess the potential for multi-element geochemistry (perhaps pXRF) to define lithological or alteration features that might assist resource modelling; and
- Assess the infill drilling requirements for Mill Zone Deep, Mustang and Palomino.

24.25.2 Geochemical

Geochemical recommendations are as follows:

- To move to a prefeasibility/feasibility study level of certainty additional data are required in the infrastructure zones to develop a statistically robust data set that would allow for modeling of the waste rock zones into geochemical categories.
- Additional borings are required to address data gaps for geochemical (and likely geotechnical) characterization in the following areas:
  - The Horseshoe portal,
  - The Horseshoe spiral ramp,
  - The spiral ramp and access development for Mustang, and
  - The Mill Zone Deep portal and development.
- Underground disposal of the waste rock as cemented backfill will require additional studies, including:
  - Tests of cemented backfill to determine acid rock potential of the cemented material as well as geotechnical characteristics. Use of cement and/or other sources of alkalinity might neutralize the acid generating potential of PAG waste rock, and
  - Grain size distribution of the material that will be backfilled. Fine grained material tends to be more reactive.

- Develop alteration block models in the infrastructure zones. This is required to extrapolate the results of the geochemical samples to the waste rock, since acid generating potential is strongly correlated to alteration. The waste block model should be used to stage, schedule and/or blend waste rock.
- A detailed hydrogeologic study should be performed for the underground targets to determine the potential for groundwater inflow and to develop groundwater management/treatment strategies;
- Kinetic tests of PAG rock to measure release rates and time of initiation. The rates would be used in parallel with the mine plan to determine how long the stopes can remain open before the reactive surfaces start to release acidity, metals and sulfate. Ideally, each stope would be backfilled before any reactive surfaces start to generate acid drainage.

24.25.3 Geotechnical

Geotechnical recommendations are as follows:

- Additional laboratory testing should be conducted, especially in the metasediments of the crown pillar, hangingwall metasediments and dykes in the Horseshoe area and all rock types of the Mustang and Mill Zone Deep areas with the aim of defining strength limits.
- Numerical analyses should be conducted to assess stability of the crown pillar and any anticipated surface subsidence. This 3D stress analyses should evaluate stope/backfill and footwall infrastructure stability.
- Once the 3D lithology and alteration models are updated, the geomechanical domains should be reviewed to determine if the alteration and lithology can provide further refinement to the distribution of rock mass quality, including statistically valid variations.
- Specific rock mass characteristics of the infrastructure locations such as shops and powder magazines should be more closely evaluated at the detailed design level of the project. Initial pilot core holes should be drilled in advance of ramps, and raises to confirm rock mass quality and determine if remedial measures or relocation are necessary.
- Detailed stope sequencing plans should be reviewed from a geomechanical or stress perspective.
- Cemented rockfill testing should be carried out to verify that recommended UCS strengths can be achieved within 14 day curing time allotted for in the mine plan. Test backfill stopes should also be conducted to confirm that the backfill performs as intended.

24.25.4 Mining

Mining recommendations are as follows:

- Refine underground mine plan with updated drilling;
- Optimization of underground/open pit interface to maximize value and meet required company and permit objectives;
- Refine the backfill quantities, material characteristics, and placement scheme to optimize the system and confirm PEA assumptions. This should include testwork to determine cemented fill make up and strength;
- Perform geotechnical analysis on the sill pillar at Horseshoe to confirm PEA assumptions;
- Perform geotechnical analysis on the portal and raisebore locations to confirm material characteristics and provide basis to further develop the costing and locations;

- Optimization of the production schedule by delaying development to an as-needed basis, however ensuring development is completed early enough for planned exploration drilling where necessary;
- Optimize the production fleet to meet required production with consideration of larger trucks and LHDs;
- Optimize the labor required based on any revision to the equipment sizing;
- Further refine the ground support methods and develop a more detailed ground support plan including review of appropriate equipment to match plan;
- Update, refine, and optimize ventilation and power system designs;
- Confirm cost and impact of utilizing modified open pit blasting to supply material for backfill and perform tradeoff with separate crushing and sizing facility; and
- Refine and update administrative, dry and change house facility design to match labor optimization.

24.25.5 Hydrogeology

Hydrogeology recommendations are as follows:

- Incorporate additional aquifer testing (such as airlift and slug testing, and water quality sampling and analysis) within mineral resource drilling programs to provide ongoing data by which to further develop the conceptual hydrogeological model of the project area.
- Integrate geotechnical engineering studies with groundwater studies.
- Undertake a detailed review of groundwater system response to open pit dewatering and depressurization to contribute to further development of the conceptual hydrogeological model of the project area.
- Undertake eco-hydrological studies to further understand interactions between groundwater and surface water resources, and groundwater and ecological receptors so that the outcomes of mine water management practices can be considered in context.
- Update existing hydrological baseline studies with information arising from eco-hydrological studies, as well as with the available climate record so that climate-groundwater related trends are understood.
- Revise / refine / update the numerical groundwater model to ensure it is fit for purpose and representative of climate-mine-groundwater-surface water interactions.

24.25.6 Mineral Processing

Mineral processing recommendations are as follows:

- The underground estimated mill feed is of a higher grade than the samples tested (~0.14oz Au/t). To progress to the next level of study additional higher grade samples should be tested to confirm the recovery assumptions.
- Continue process plant expansion design and costing to a prefeasibility/feasibility study level of accuracy.

24.25.7 Environmental & Social

Environmental and social recommendations are as follows:

- Utilize existing and additional studies and other information developed pursuant to Section 24-25 to further refine anticipated impacts to the physical and human environment from underground mining.

- Conduct baseline environmental studies of all impacted areas, not otherwise impacted by Haile's current Mine Plan.
- Identify location of above ground OSA storage, and volume changes over LoM.
- Identify any point source air emissions, and calculate any changes to current emissions levels, and assess requirements of Haile's current air permit.
- More fully described water management at TSF in later years to accommodate additional tailings, and assess requirements of Haile's current TSF/Dam permit.
- Finalize ROM pad design to understand potential impacts to wetlands.
- Calculate reclamation costs and update Haile's Reclamation Plan.
- Update economic impact study to account for underground mining operations.
- Update traffic study to accommodate additional workforce.
- Update Haile's Mine Plan and Site Wide Water Balance to include underground mining.

24.25.8 Costs

Table 24-62 summarizes the recommended work program costs.

Table 24-62: Summary of Costs for Recommended Work

Recommended Work	Cost Estimate (US$)
Geology – Drilling	in progress
Geochemical – Sampling program, modeling, and analysis	150,000
Geotechnical – Drilling/logging/sampling program, modeling, and analysis	1,150,000
Mining - Optimization of underground/open pit interface	200,000
Mining – Backfill testing program	65,000
Mining – Study updates to new information, optimization and detail, reporting	300,000
Hydrogeology – Modeling and Site Wide Water Balance	130,000
Hydrogeology – Monitoring and Baseline Studies	120,000
Hydrogeology – Water Management Plans	40,000
Mineral Processing	200,000
Environmental & Social	75,000
Feasibility level analysis reporting	100,000
Total	2,315,000

Note: A portion of the work included in the table is ongoing at the time of writing with the geology drilling program nearing completion.

24.26 REFERENCES

The Qualified Persons have used the allowance under Instruction (4) to the Form NI43-101F1 whereby disclosure included under one heading is not required to be repeated under another heading, and have compiled all references used in collating this Report in Section 27.

25 INTERPRETATION AND CONCLUSIONS

The project is located in a socially and economically stable part of the world. The climate is moderate and local infrastructure is present. A state highway runs adjacent to the site, eliminating many logistical problems typically associated with mining projects. The project is somewhat unique because it is located on private and previously mined land. A significant amount of time and effort was devoted to the permitting process on the project. The major permits have been received and construction has begun.

The ultimate degree of success will be linked to gold prices. The project is favorable at all evaluated price sensitivities.

25.1 MINING AND GEOLOGICAL CONCLUSIONS

IMC reviewed the exploration program, drill program and core logs provided by HGM. A substantial effort went into the development of an economical mine plan. A grade recovery curve was used to optimize the plan.

25.2 METALLURGICAL CONCLUSIONS

M3 reviewed metallurgical data and test work provided by HGM. This data was used to develop the project flow sheets and design criteria. No unproven technologies are planned for the Haile project. Many process plants of this size have been constructed in the past and this project can be constructed to meet the schedule.

25.3 WATER BALANCE AND WATER SOURCE CONCLUSIONS

A site wide water balance was completed to ensure that adequate storage is available in the Duckwood TSF for both mill process and meteorological water. An additional objective was to estimate the available fresh water supply versus demand for mine operations. Results of the analysis indicate that there is always an excess of capacity in the TSF over and above the volume of free water and PMP inflows predicted in the system. Results also indicate that fresh water demands for the mill process will be met based on the predicted model assumptions for the current production rate.

Water management structures such as permanent diversion channels were designed for the 100 year, 24-hour storm event. Sediment control channels were designed for the 10 year 24-hour storm event. Seepage and stability analyses were completed in support of the detailed design of the TSF and feasibility-level design of the Haile Gold Mine Creek Detention Structure.

The results of the seepage analyses indicate that both the Duckwood TSF and Haile Gold Mine Creek Detention Structure can properly route seepage through the embankment under normal operating and seismic loading conditions without adversely affecting the stability of each facility.

25.4 TAILING, OSA AND WATER DIVERSION CONCLUSIONS

Stability analyses were conducted and indicate that the TSF, OSAs and Haile Gold Mine Creek Detention Structure are stable under the static and seismic loads evaluated.

25.5 ENVIRONMENTAL AND PERMITTING CONCLUSIONS

Environmental baseline information for the Haile project has been established, all of the major permits have been received and construction has begun. HGM has proposed and agreed to required mitigation to offset projected impacts. This includes reclamation/closure and the funding/posting of financial assurance (Reclamation Bond) to guarantee that this occurs.

Construction, operational and post-closure monitoring is required to ensure compliance with issued permits and regulations.

25.6 RISKS AND OPPORTUNITIES

During the course of the feasibility study, several potential risks and opportunities were identified.

- *Metal Prices* – The base case gold price is $1250/ounce. At the completion of this study, gold was trading at over $1160/ounce.
- *Silver Grade* – Silver is a byproduct for this project and is assumed to have a grade of 1.5 x the grade of gold. This assumption is based on assays of metallurgical samples. There is a potential that silver grade may differ from that assumption.
- *Silver Recovery* – Based on metallurgical test work, it was assumed that there will be a 70% recovery of silver in the project economic model.
- *Pit slope Angles* – There may be opportunities to increase the inter-ramp slope angels, especially in the hanging wall, or northern portions of the pit. The result of steeper slopes would be reduced waste (overburden) stripping and better potential economics.
- *Saprolite Mining* – It was assumed that no drilling and blasting was required for mining of Saprolite overburden material. If drilling and blasting is required, mine operating and capital costs could increase.
- *Existing Mining Facilities and Underground Workings* – Due to the historic mining in the area, there is a chance that underground mining and other facilities will be found. This could potentially reduce mining efficiency.
- *Reclamation/Closure* – Interim reclamation is a part of the overall mine. Opportunity(s) may present themselves to include additional/more expedient reclamation/closure activities as part of mining, thus reducing final closure obligations and financial assurance costs.
- *Fresh Water Makeup Risks and Opportunities* – The results of the site wide water balance indicate that sufficient water is expected to be available. Because the water balance is run on a monthly time step, instantaneous water demand shortages can be handled with the addition of water storage once Haile moves into operations. Water is available from the local municipal source if there is a shortage.
- *Inferred Mineralization* - There is known inferred mineralization within the bounds of the reserve that is not included as reserves. If this mineralization is converted to reserves the available ore tonnage may go up and the amount of waste (overburden) that will need to be handled will be reduced by that number of tons.

26 RECOMMENDATIONS

M3 recommends that the project complete construction as soon as practicable.

27 REFERENCES

AMEC Earth and Environmental. (2010). Tailing and Process Water Management Bankable Feasibility Study. November 19, 2010.

AMIRA. (2002). ARD Test Handbook. Project P387A: Prediction and Kinetic Control of Acid Mine Drainage.

Barton, N.R. (2000). TBM Tunneling in Jointed and Faulted Rock.

Barton, N.R. (1988). Rock Mass Classification and Tunnel Reinforcement Selection using the Q-system.

Barton, N. (1974). Estimating the shear strength of rock joints. Proc. of 3rd Congr. Int. Soc. Rock Mech. Advances in rock mechanics, Denver, Vol. IIA: 219-220.

Barton, N. (1974). Rock slope performance as revealed by a physical joint model. Proc. of 3rd Congr. Int. Soc. Rock Mech. Advances in rock mechanics. Denver, Vol. IIB: 765-773.

CDM Smith Australia Pty Ltd. (2016). Haile Gold Mine Inc., Scoping Study for Haile Underground Mine – Groundwater Model. May 27, 2016.

CIM. (2014). Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014.

Clark, L. (1998). Minimizing Dilution in Open Stope Mining with a Focus on Stope Design and Narrow Vein Longhole Blasting. M.Sc. thesis, Department of Mining and Mineral Process Engineering, The University of British Columbia, 316 p.

Golder Associates. (2012). Preliminary Geotechnical Investigation, Horseshoe Deposit, Haile Gold Mine, Lancaster County, South Carolina, Appendix A: Rock Core Logging Data, Project No. 11390150, January 5, 2012.

Golder Associates. (2010). Feasibility Level Pit Slope Evaluation. October 2010.

Golder Associates. (2010). Haile Gold Mine, Lancaster County, South Carolina, Report on Feasibility Level Pit Slope Evaluation, Project No. 093-91767.300, October 2010.

Haile Gold Mine, Inc. (2010). Haile Power Supply Recommendations for Feasibility Study. December 2010.

Haile Gold Mine Project. (2014). Final Environmental Impact Statement. July 2014.

Haile Gold Mine Project. (2014). Record of Decision. October 27, 2014.

Hibbard, J.P., van Staal, C.R., and Millar, B.V. (2007). Links between Carolinia, Avalonia, and Ganderia in the Appalachian per-Gondwanan Realm; in Sears, J., Harms, T., and Evenchick, C., eds., From Whence the Mountains?: Inquiries into the Evolution of Orogenic Systems: A Volume in Honor of Raymond A. Price: GSA Special Paper 433, p. 291-311.

Hibbard, J.P., van Staal, C.R., and Rankin, D.W. (2010). Comparative analysis of the geological evolution of the northern and southern Appalachian orogen: Late Ordivician-Permian: in Tollo, R.P., Bartholomew, M.J., Hibbard, J.P., and Karabinos, P.M., eds., From Rodinia to Pangea: The Lithotectonic Record of the Appalachian Region: Geological Society of America Memoir 206, p. 51-69.

Ecological Services and Consulting. (2010). Intermediate List of Vascular Plant Species Noted During Surveys of the Haile Gold Mine Properties May, 2008 to August, 2010. September 2010.

Keferl, E. P., Dr. (2010). Memorandum on the Report on Survey for mollusks on Camp Branch of the Little Lynches River, Lancaster County, South Carolina. September 2010.

KML Metallurgical Services (KML). (2012). HGM Years 1 – 3 Silver Characterization Project Test Report, Kershaw, South Carolina. December 27, 2012.

M3 Engineering & Technology Corporation. (2015). Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina. October 13, 2015.

M3 Engineering & Technology Corporation. (2012). Haile Gold Mine Project Resource Estimate NI 43-101 Technical Report Lancaster County, South Carolina. March 13, 2012.

M3 Engineering & Technology Corporation. (2011). Haile Gold Mine Project NI 43-101 Technical Report Feasibility Study Lancaster County, South Carolina. February 10, 2011.

Mathews, K.E., Hoek, E., Wyllie, D.C. and Stewart, S.B.V. (1981). Prediction of stable excavations for mining at depth below 1000 metres in hard rock. CANMET Report DSS Serial No. OSQ80-00081, DSS File No. 17SQ.23440-0-9020. Ottawa: Dept. Energy, Mines and Resources.

Metso Minerals Industries, Inc., (Metso). (2009). Test Plant Report No. 20000134-135. York, Pennsylvania, December 7, 2009.

Mobley, R.M., Yogodzinski, G.M., Creaser, R.A., and Berry, J.M. (2014). Geologic History and Timing of Mineralization at the Haile Gold Mine, South Carolina, Economic Geology, v. 109, p. 1863-1881.

Needham Environmental Inc. (2010). Search for Federal Threatened and Endangered Species and State Species of Concern on the Duckwood Tract. October 2010.

Potvin Y. & J. Hadjigeorgiou. (2001). The Stability Graph Method for Open Stope Design. Chapter 60 in Underground Mining Methods. Hustrulid & Bullock Eds. Society of Mining Engineers, pp. 513-520.

Price, W.A. (1997). Guidelines and Recommended Methods for the Prediction of Metal Leaching and Acid Rock Drainage at Minesites in British Columbia—DRAFT. British Columbia Ministry of Employment and Investment. Smithers, BC. April 1997.

Price, W.A. (2009). Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials. CANMET Mining and Mineral Sciences Laboratories. December 2009.

Rohde, F.C. (2010). Fishes in Camp Branch. August 2010.

Romarco Minerals, Inc. (2010). Flash Flotation, Cyanide Destruction & Leaching Of Concentrate and Tailing for Haile Composites. Resource Development Inc., (RDi). Wheat Ridge, Colorado. May 27, 2010.

Romarco Minerals, Inc. (2009). Haile Gold Project, Metallurgical Report, Resource Development Inc., (RDi), Wheat Ridge, Colorado. September 16, 2009.

Romarco Minerals, Inc. (2010). Metallurgical Testing Of Ledbetter Extension Samples, Resource Development Inc., (RDi), Wheat Ridge, Colorado, March 31, 2010.

Romarco Minerals, Inc. (2010). Optimization Of Leaching Of Flotation Concentrate, Resource Development Inc., (RDi), Wheat Ridge, Colorado, September 27, 2010.

Romarco Minerals, Inc. (2010). Work Index Data For Haile Composite Sample, Resource Development Inc., (RDi), Wheat Ridge, Colorado. March 31, 2010.

Schafer Limited LLC. (2010). Haile Gold Mine Addendum to Baseline Geochemistry Report. November 2010.

Schafer Limited LLC. (2015). Revised Haile Gold Mine Overburden Management Plan. For Romarco Minerals Inc.

Schlumberger Water Services. (2010). Haile Gold Mine Baseline Hydrologic Characterization Report. November 2010.

Schlumberger Water Services. (2010). Haile Gold Mine Depressurization and Dewatering Feasibility Study. December 2010.

Schlumberger Water Services. (2010). Mine Water Treatment Plant Preliminary Design Report. December 2010.

Schlumberger Water Services. (2010). Reclamation Plan Overview, prepared for Haile Gold Mine. December 2010.

SRK Consulting. (2016). Sample Selection and Analysis for Geochemical Testing, Haile Underground Scoping Study.

APPENDIX A – FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

I, Daniel H. Neff, P.E., do hereby certify that:

1. I am currently employed as President by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of the University of Arizona and received a Bachelor of Science degree in Civil Engineering in 1973 and a Master of Science degree in Civil Engineering in 1981.

3. I am a:

   Registered Professional Engineer in the State of Arizona (No. 11804 and 13848)

4. I have practiced civil and structural engineering and project management for 42 years. I have worked for engineering consulting companies for 12 years and for M3 Engineering & Technology Corporation for 30 years.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Sections 1, 2, 3, 4, 5, 6, 18, 19, 21 (except 21.4, 21.5), 22, 23, 25, 26, and 27 of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation.

7. I have had prior involvement with the property that is the subject of the Technical Report as President of M3 Engineering & Technology Corporation. M3 is the EPCM firm responsible for design and construction of the Haile Project as of the effective date of this Technical Report.

8. I last visited the Haile Site on September 1, 2016.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October, 2016.

(Signed) Daniel H. Neff
Signature of Qualified Person

Daniel H. Neff, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Erin L. Patterson, P.E. do hereby certify that:

1. I am currently employed as an Engineer by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of the University of Arizona and received a Bachelor of Science in Chemical Engineering in 2005.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 54243)

4. I have worked as a process engineer for a total of 9 years.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 13 and 17, as well as corresponding items of Sections 1, 25, 26 and 27 of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation.

7. I have had prior involvement with the property that is the subject of the Technical Report. I have performed design and engineering in support of the overall Engineering, Procurement, and Construction Management (EPCM) effort since 2011. I was contributing author of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina," dated effective 13 October 2015.

8. I last visited the property in October 5, 2016.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October, 2016.

(Signed) (Sealed) Erin L. Patterson
Signature of Qualified Person

Erin L. Patterson, P.E.
Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Lee "Pat" Gochnour

I, Lee "Pat" Gochnour, do hereby certify that:

1. I am President of:

   Gochnour & Associates, Inc.
   915 Fairway Lane, Aberdeen WA 98520

2. I graduated with a bachelors' degree in Park Administration and Environmental Land Use Planning from Eastern Washington University in 1981.

3. I am a Mining and Metallurgical Society of America, Qualified Professional (#1166) in good standing in the areas of Environmental, Permitting and Compliance.

4. I have worked as an environmental, permitting and compliance professional in the mining business for a total of 35 years.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for Sections 1, 20, 25, 26 and 27. I last visited the project site in November, 2012.

7. I have prior involvement with the property that is the subject of the Technical Report. This previous involvement was as a contributing author for previous versions of the Technical Reports for the Project.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October, 2016.

"signed"
Signature of Qualified Person

Lee "Pat" Gochnour
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

1. I am currently employed as the President and a Senior Mining Engineer by:

   Independent Mining Consultants, Inc.
   3560 E. Gas Road
   Tucson, Arizona, USA 85714

2. I graduated with the following degrees from the Colorado School of Mines
   Bachelors of Science, Mineral Engineering – Physics 1974
   Masters of Science, Mining Engineering 1976

3. I am a Registered Professional Mining Engineer in the State of Arizona USA
   Registration # 12772
   I am a Registered Professional Engineer in the State of Colorado USA
   Registration # 16191
   I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers, Registration # 2021600

4. I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 40 years. I have managed drill programs, overseen sampling programs, and interpreted geologic occurrences in both precious metals and base metals for numerous projects over that time frame. My advanced training at the university included geostatistics and I have built upon that initial training as a resource modeler and reserve estimation specialist in base and precious metals for my entire career. I have acted as the Qualified Person on these topics for numerous Technical Reports.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I am responsible for sections 7, 8, 9, 10, 11, 12, 14, 15, 16, 21.4, 21.5, and I contributed to Section 1 of the Technical Report titled "Haile Gold Mine Project, NI43-101 Technical Report, Project Update, Lancaster County, South Carolina", dated effective July 31, 2016.

7. I visited the Haile Gold property on the following occasions: November 4 - 6, 2009, June 20, 2012, and again from June 2-3, 2015 during which times I reviewed the drill core, core handling procedures, sample preparation, core logging and site conditions.

8. Independent Mining Consultants, Inc., and this author worked the Haile project prior to this study. That work included block modeling and mine planning and was completed during late 2009 and early 2010. I was a co-author of previous technical reports regarding the Haile project dated: 7 December 2010, 10 February 2011, 13 March 2012, 21 November 2014, and 13 October 2015

9. As of the date hereof, to the best of my knowledge, information, and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

11. I am independent of the issuer applying the tests in Section 1.5 of NI 43-101.

12. I have read national Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated: 14 October 2016.

SME
Society for Mining, Metallurgy & Exploration
John M. Marek
SME Registered Member No. 2021600
Signature
Date Signed 22 Sep 2016
Expiration date 31 Dec 2016

John M. Marek
Registered Member of the SME

CERTIFICATE OF QUALIFIED PERSON

Carl John Burkhalter

I, Carl John Burkhalter, PE, do hereby certify that:

1. I am partner of:

   NewFields Mining & Technical Services LLC ("NewFields")
   9400 Station Street, Suite 300, Lone Tree, CO 80124, USA

2. I graduated with a Bachelor of Science in Mining Engineering in 1984 and a Master of Science in Civil and Environmental (Geotechnical) Engineering in 1987 from the University of Wisconsin- Madison.

3. I am a registered professional engineer in good standing in Colorado in the area of Civil Engineering, P.E. No. 29447. I am also registered as a professional civil engineer in State of Arizona (No. 34925), State of Nevada (No. 21178) and the State of South Carolina (No. 31243).

4. I have worked as civil engineer for a total of 29 years. My experience includes the design of tailings and waste storage facilities.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for portions of Sections 1, 18, 25, 26, and 27. I have visited the project site on several occasions, the latest on August 16, 2016.

7. I have prior involvement with the property that is the subject of the Technical Report as a contributing author of the 2012, 2014 and 2015 versions of the Haile Gold Mine NI 43-101 Reports.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October 2016.

"signed" "sealed"
Signature of Qualified Person

Carl John Burkhalter, P.E.
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

Jonathan Godfrey Moore

I, Jonathan Godfrey Moore, BSc(Hons), Geology, do hereby certify that:

1. I am Chief Geologist of:

   OceanaGold New Zealand Limited
   22 MacLaggan Street, Dunedin, New Zealand

2. I graduated with a BSc (Hons) geology 1985 and DipGrad (physics), 1993.

3. I am a Member and Chartered Professional in good standing of the AusIMM.

4. I have worked as geologist for a total of 27 years. My experience includes 17 years working as a resource geologist.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for Sections 24.7, 24.8, 24.9, 24.10, 24.11, 24.12, 24.14, and corresponding items in 24.1, 24.24, 24.25. I have visited the project site on October 11, 2016.

7. I have been involved with the property that is the subject of the Technical Report, since November 2015 as an employee of Oceanagold.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101 but am able to act as a Qualified Person because OceanaGold Corporation is a producing issuer.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14 day of October, 2016.

Signature of Qualified Person

Jonathan Godfrey Moore
Print Name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

David Read Carr

I, David Read Carr, do hereby certify that:

1. I am Chief Metallurgist of:

   Oceanagold Corporation, 22 MacLaggan Street, Dunedin, New Zealand

2. I graduated with a Bachelor of Engineering in Metallurgical Engineering in 1993 from the University of South Australia.

3. I am a member and Chartered Professional (Metallurgy) in good standing with the AusIMM.

4. I have worked as a metallurgist for a total of 23 years since graduation.

5. I have read the definition of “Qualified Person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled “Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina”, (the “Technical Report”), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for Sections 24.13, 24.17, 24.19, 24.21, and corresponding items in 24.1, 24.24, 24.25. I have visited the project site in October 2016

7. I have no prior involvement with the property that is the subject of the Technical Report

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101 because I am an employee of OceanaGold (New Zealand) Limited.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October, 2016.

(signed) David Read Carr
Signature of Qualified Person

David Read Carr MAusIMM CP (Met)
Print Name of Qualified Person

srk consulting

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1125 Seventeenth Street
Denver, CO 80202

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F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, Ph.D., P.E., ISRM, do hereby certify that:

1. I am a Principal Geotechnical Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina" with an Effective Date of July 31, 2016 (the "Technical Report").
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, a member of the ASCE GeoInstitute, and a Registered Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 31 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 15 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile property.
6. I am responsible for the preparation of for geotechnical Sections 24.16.2, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of October, 2016.

"signed"

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John Tinucci, Ph.D., P.E.

CERTIFICATE OF QUALIFIED PERSON

Robert P. Schreiber

I, Robert P. Schreiber, P.E., D.WRE, BCEE, do hereby certify that:

1. I am Vice President of:

   CDM Smith Inc.
   75 State Street, Suite 701, Boston, MA 02109

2. I graduated with a Bachelor of Science degree in Civil Engineering from the Massachusetts Institute of Technology (MIT) in 1975 and with a Master of Science degree in Civil Engineering from MIT in 1976.

3. I am a Professional Engineer in good standing in Maine (3925), Massachusetts (35612), New Jersey (24GE03100300), and Florida (44465) in the area of Civil Engineering. I am also registered as a Diplomate, Water Resources Engineer (D.WRE) by the American Academy of Water Resources Engineers (AAWRE), and as a Board Certified Environmental Engineer (BCEE) by the American Academy of Environmental Engineers and Scientists (AAEES).

4. I have worked as Civil Engineer focusing on water resources and specializing in groundwater hydrology for a total of 40 years. My experience includes simulation modeling of groundwater flow, including application of 3D groundwater flow simulation modeling to mine dewatering design and operations, the management of impacts to surrounding resources, and the meeting of regulatory requirements related to groundwater.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a senior technical reviewer for the preparation of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for technical review of Sections 24.16.3 and corresponding items in 24.1, 24.24, 24.25 that describe the simulation modeling of mine dewatering activities in regard to impacts from and on ambient groundwater at and immediately surrounding the mining described. I have not visited the project site because it is not necessary for conducting adequate technical review per my role.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have not had any additional involvement with the project or collaboration with the Client.

9. As of the date of this certificate, to the best of my knowledge, information and belief, and as limited by my role as defined in items #4 and #6 above, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 14th day of October, 2016.

Robert P. Schreiber Digitally signed by Robert P. Schreiber
DN: C=US, E=schreiberrp@cdmsmith.com, O=CDM Smith, OU=TSU/WSO/WR Group, CN=Robert P. Schreiber
Date: 2016.10.14 20:41:54-04'00'

Signature of Qualified Person

Robert P. Schreiber

Print Name of Qualified Person

srk consulting

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Denver, CO 80202

T: 303.985.1333
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denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Patrick Williamson, MSc Geology, do hereby certify that:

1. I am Principal Consultant of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina" with an Effective Date of July 31, 2016 (the "Technical Report").
3. I graduated with a degree in Geology from the Colorado College in 1982. In addition, I have obtained a Masters of Science in Geology in 1987 from the University of Colorado. I have worked as a Geologist for a total of 31 years since my graduation from university. My relevant experience includes environmental geochemistry of rock, soil, sediments and water for mining industry and industrial complexes.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I did not visit the Haile property.
6. I am responsible for the preparation of economic Sections 24.16.4, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of October, 2016.

"signed"

____

Patrick Williamson, MSc Geology, MMSAQP
SRK Principal Consultant, Geochemistry and Hydrogeology

U.S. Offices:		Canadian Offices:		Group Offices:
Anchorage	907.677.3520	Saskatoon	306.955.4778	Africa
Clovis	559.452.0182	Sudbury	705.682.3270	Asia
Denver	303.985.1333	Toronto	416.601.1445	Australia
Elko	775.753.4151	Vancouver	604.681.4196	Europe
Fort Collins	970.407.8302	Yellowknife	867.873.8670	North America
Reno	775.828.6800			South America
Tucson	520.544.3688			

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denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, do hereby certify that:

1. I am a Senior Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina" with an Effective Date of July 31, 2016 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Geology. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 12 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling, pit optimization and truck productivity analysis.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile property.
6. I am responsible for the preparation of mining planning Sections 24.2 through 24.6, 24.15, 24.16.1, 24.16.5 through 24.16.9 (co-authored), 24.23, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of October, 2016.

"signed"

Joanna Poeck, BEng Mining, SME-RM [4131289RM], MMSAQP [01387QP]
Senior Consultant (Mining Engineer)

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CERTIFICATE OF QUALIFIED PERSON

I, Jeff Osborn, BEng Mining, MMSAQP do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina" with an Effective Date of July 31, 2016 (the "Technical Report").
3. I graduated with a Bachelor of Science Mining Engineering degree from the Colorado School of Mines in 1986. I am a Qualified Professional (QP) Member of the Mining and Metallurgical Society of America. I have worked as a Mining Engineer for a total of 29 years since my graduation from university. My relevant experience includes responsibilities in operations, maintenance, engineering, management, and construction activities.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile property on April 13, 2016 for 2 days.
6. I am responsible for the preparation of mining and infrastructure Sections 24.16.5 through 24.16.9 (co-authored), 24.18, 24.21 (co-authored), and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101. I have not had prior involvement with the property that is the subject of the Technical Report.
8. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of October, 2016.

"signed"

Jeff Osborn, BEng Mining, MMSAQP [01458QP]
Principal Consultant (Mining Engineer)

CERTIFICATE OF QUALIFIED PERSON

William Scott McDaniel

I, William Scott McDaniel, do hereby certify that:

1. I am Environmental Manager of:
   OceanaGold – Haile Operation
   6911 Snowy Owl Road
   Kershaw, South Carolina 29067

2. I graduated with a Bachelor of Science in Metallurgical Engineering – Colorado School of Mines, Golden Colorado 1978.

3. I have worked as a Metallurgical Engineer / Environmental Manager for a total of 38 years. My experience includes Exploration Driller, Mining Engineer, Mine Safety Manager, Environmental Specialist, Environmental Health and Safety Manager, Technical Director, Process Superintendent, and Mine Manager. I am a member of Society for Mining, Metallurgy and Exploration (SME), # 04185359.

4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

5. I am a contributing author for the preparation of the technical report titled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina", (the "Technical Report"), dated effective July 31, 2016, prepared for OceanaGold Corporation; and am responsible for Sections 24.20 and corresponding items in 24.1, 24.24, 24.25. I have been active on the project site since March 9, 2012.

6. I have had no prior involvement with the property that is the subject of the Technical Report.

7. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

8. I am not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101. I work for Oceana Gold as the Environmental Manager of Haile Operation.

9. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 17 day of October, 2016.

Signature

William Scott McDaniel
Print Name of Qualified Person

SRK Consulting (U.S.), Inc.
Suite 600
1125 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Grant Malensek, MEng, PEng/PGeo, do hereby certify that:

1. I am Principal Consultant (Mineral Project Evaluation) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina" with an Effective Date of July 31, 2016 (the "Technical Report").
3. I graduated with a degree in B.S. Geological Sciences from University of British Columbia in 1987. In addition, I have obtained a M.E. in Geological Engineering (Colorado School of Mines, 1997). I am a Professional Engineer of the Association of Professional Engineers & Geoscientists of British Columbia. I have worked as an Engineer for a total of over 20 years since my graduation from university. My relevant experience includes business experience in financial analysis, project management and business development.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile property.
6. I am responsible for the preparation of economic Sections 24.21 (co-authored), 24.22, and portions of Sections 24.1, 24.24 and 24.25 summarized therefrom, of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 14th Day of October, 2016.

"signed"

Grant Malensek, MEng, PEng/PGeo [APEGBC 23905]
Principal Consultant (Mineral Project Evaluation)