M3-PN150129
Issue Date:
October 13, 2015
Revision 0

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 October 13, 2015. The issue date of this report is October 13, 2015. 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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- Certificate of Qualified Person ("QP")

Responsibility	Qualified Person	Registration	Company
Process Plant Cost & Principal Author	Joshua Snider	PE	M3
Process & Metallurgy	Erin L. Patterson	PE	M3
Environmental & Permitting	Lee "Pat" Gochnour	MMSA	G&A
Resources Modeling, Mine Planning, Reserves, Geology & Description of Potential Mineral Deposits	John Marek	PE	IMC
Tailing, Overburden & Site Water Management	Carl Burkhalter	PE	NewFields

Companies Listed Above:

M3 Engineering & Technology Corporation, Tucson AZ (M3)
Independent Mining Consultants, Inc., Tucson AZ (IMC)
Gochnour & Associates, Inc., Parker CO (G&A)
NewFields Mining Design & Technical Services LLC, Denver CO (NewFields)

1 SUMMARY

1.1 IMPORTANT NOTE

This report is based on the *November 21, 2014 Haile Gold Mine Technical Report* that was issued for Romarco Minerals, Inc. In October of 2015, Romarco Minerals was acquired by OceanaGold Corporation. This is a re-issue of the Technical report with minor modifications to the history, land ownership and permitting sections. The authors of this report have not updated any resource, reserve, metallurgical data, capital or operating cost data for this issuance of the Technical Report. All estimates and designs that were the basis for this report are effective November of 2014.

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 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 $329.2 million at a 5% discount rate, an IRR of 20.1% and a payback period of 3.9 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: 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)	$333.1 (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	<u>Gold</u>		
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	<u>Silver</u>		
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:	<u>Base Case</u>	<u>Low Case</u>	<u>High Case</u>
Gold (price per troy ounce)	$1250	$950	$1550
Pre-Tax Project Internal Rate of Return (IRR)	22.6%	7.8%	35.3%
Pre-Tax NPV at 5% Discount Rate ($ Millions)	$416.7	$59.0	$774.3
Benefit Cost Ratio at 5% Discount Rate	2.4	1.2	3.6
Pre-Tax Payback (years)	3.6	8.3	2.5
After Tax Project Internal Rate of Return (IRR)	20.1%	6.2%	31.9%
After Tax NPV at 5% Discount Rate ($ Millions)	$329.2	$23.8	$620.9
Benefit Cost Ratio at 5% Discount Rate	2.1	1.1	3.1
After Tax Payback (years)	3.9	8.5	2.6

1.4 SCOPE

M3 prepared this feasibility study update on behalf of HGM. The purpose and scope of this 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

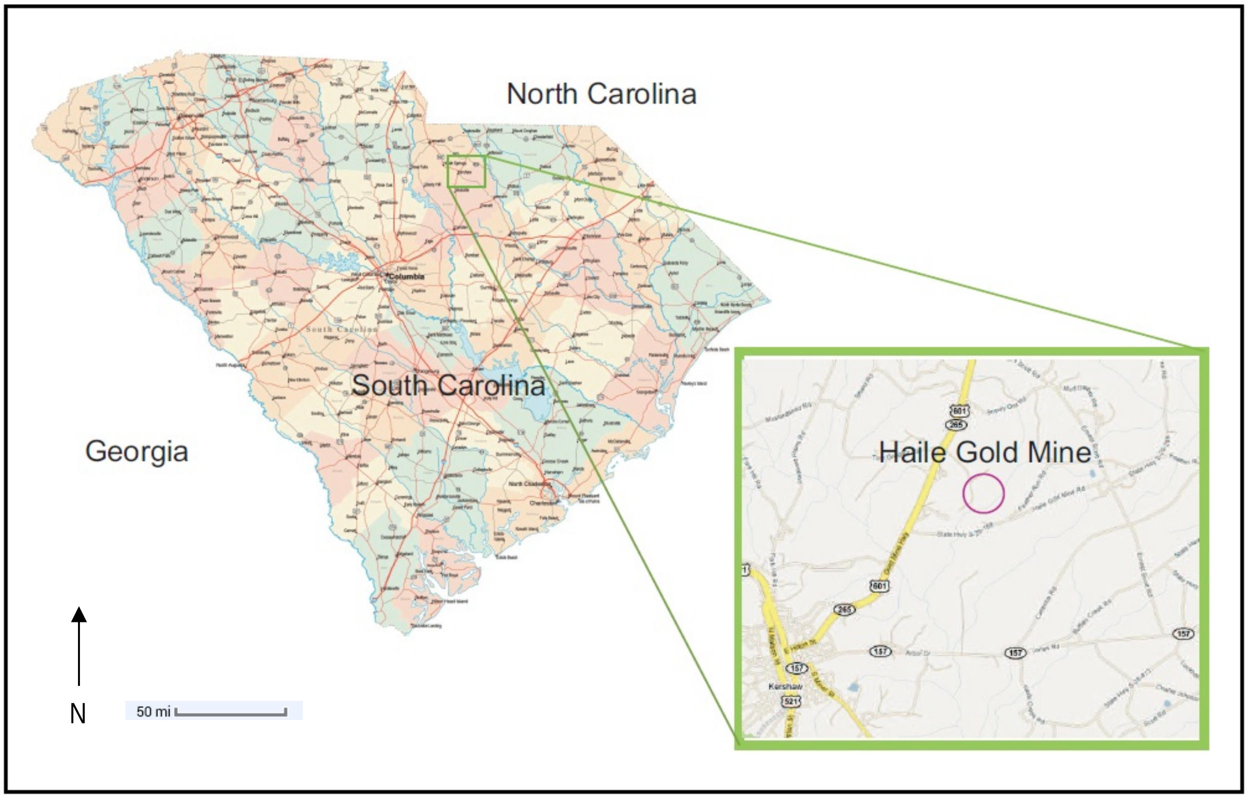
- Detailed level engineering design including equipment specifications and procurement
- Development of drawings to describe the project and support the equipment and material takeoffs
- Solicitation of firm equipment and material costs from vendors
- Preparation of capital estimates and the master capital 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.

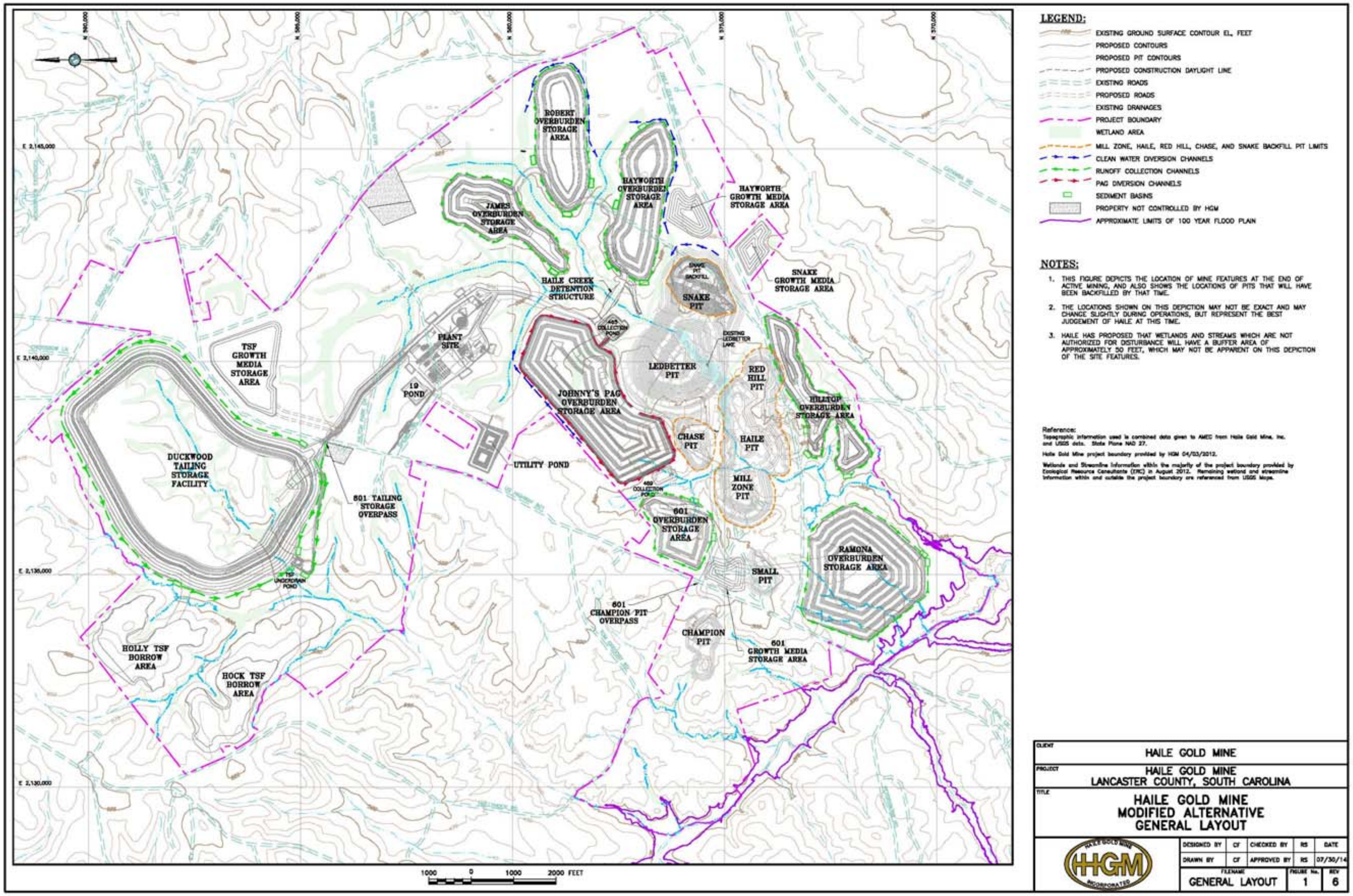


Figure 1-2: Overall Site Layout

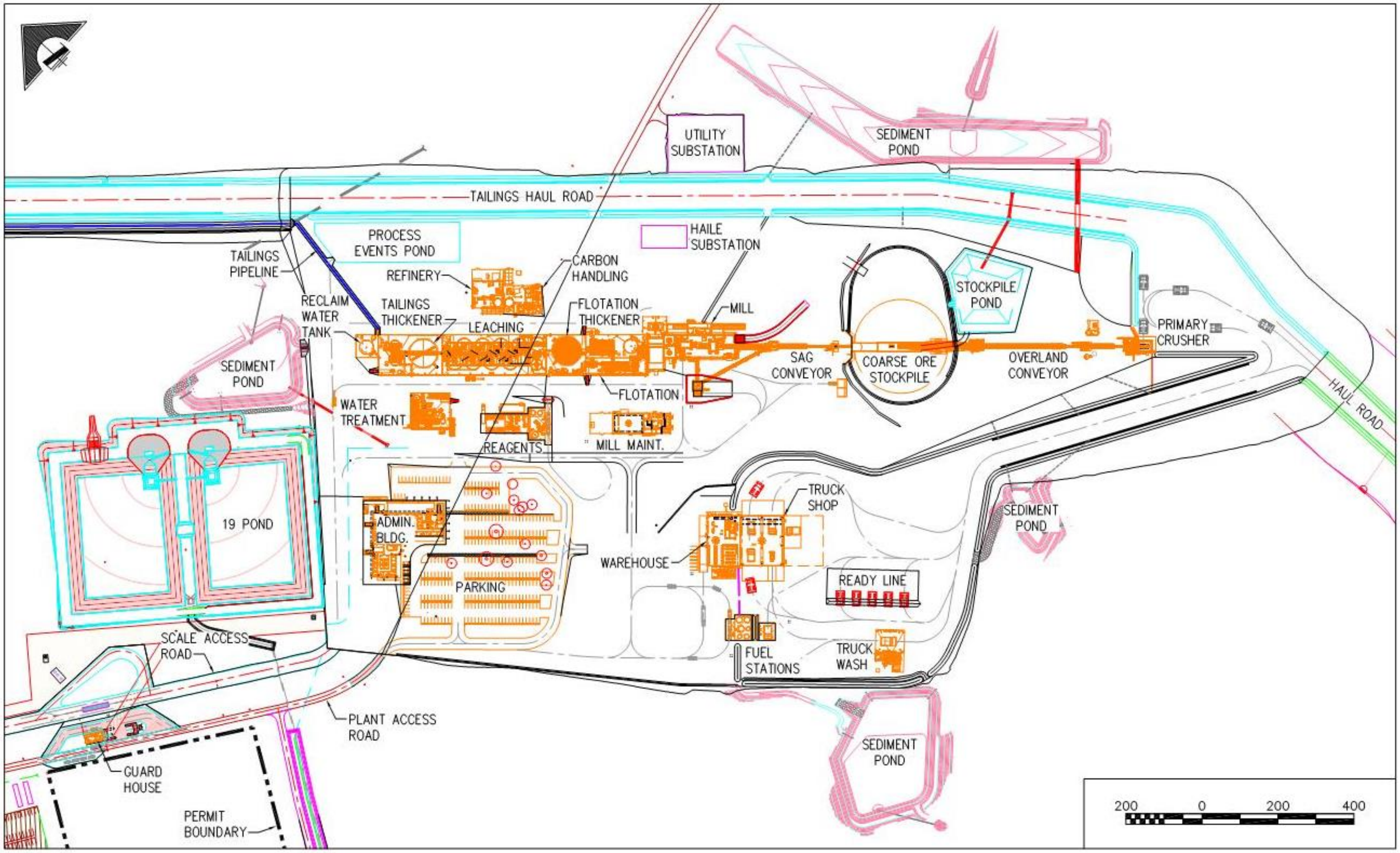


Figure 1-3: Process Area Site Plan

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 north east 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 US 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 fee simple 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 Ordivician-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 ± 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 529 to 564 Ma. Four of these samples give a weighted age of 548.7±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 IMC block model was used by Snowden Mining Industry Consultants (Snowden) to determine the component of the mineralization that had reasonable prospects of economic extraction for underground mining. Anthony Finch P.Eng (APEGBC) at Snowden acted as the Qualified Person for the development of the underground mineral resource.

The open pit mineral resource is contained in a computer generated open pit (floating cone) to assure reasonable prospects of economic extraction. The underground resource is contained within practical stope geometries at economic cutoff grades that can be accessed and potentially produced. The table below combines the open pit and underground resources. The open pit cutoff was 0.012 oz/ton and the underground cutoff was 0.080 oz/ton.

Table 1-2: Haile Mineral Resources as of January 1, 2012 and 1 November 2014
Combined Open Pit Plus Underground Material

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000
Measured	0.012- 0.080	40,669	0.052	2,125
Indicated	0.012- 0.080	37,784	0.051	1,914
Measured + Indicated	0.012- 0.080	78,453	0.051	4,039
Inferred Resource	0.012- 0.080	22,184	0.036	801
Notes: Cutoff grades are 0.012 oz/ton open pit, and 0.080 oz/ton underground 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				

Qualified persons for the mineral resources are John Marek, P.E. of IMC and Anthony Finch, P. Eng of Snowden.

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
6 ½" Blast Hole Drills	3	4
15 Cubic Yd Front Loader	1	2
17 Cubic Yd Front Loader	1	1
14.4 Cubic Yd Hyd Shovel	1	1
100 ton Trucks	12	24

Appropriate mine auxiliary 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

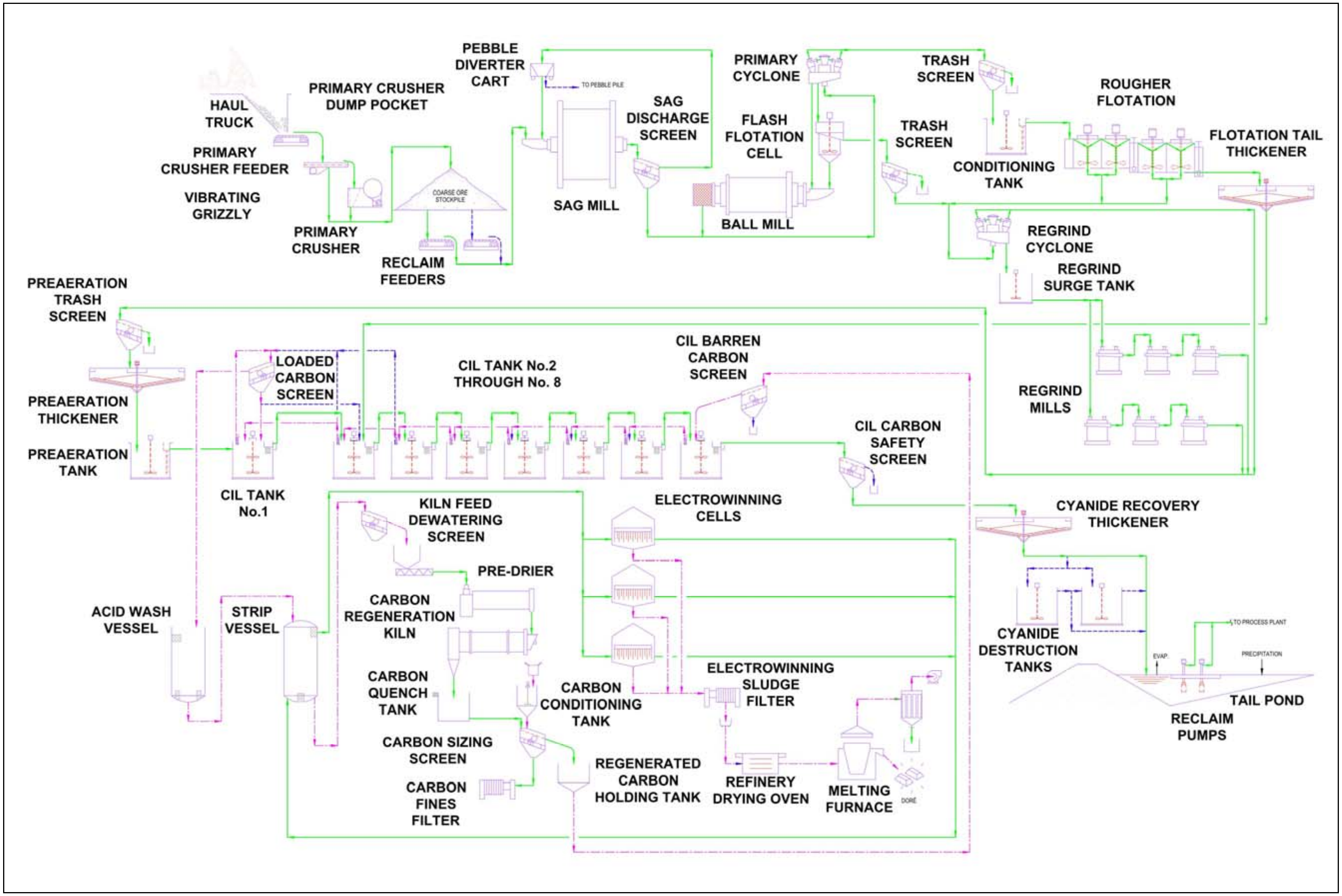
Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet has been used as the basis of process design. The relative low variability of flotation test 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 south east 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 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).

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.

Table 1-7: Initial Capital Costs

Description	($ Millions)
Direct Costs	257.2
Indirect Costs	40.8
Owners Costs*	20.1
Contingency	15.0
Escalation (2014 Dollars)	0
Total Project Capital	**333.1**

*Includes $1.8 million of contingency within Owner's Cost

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.

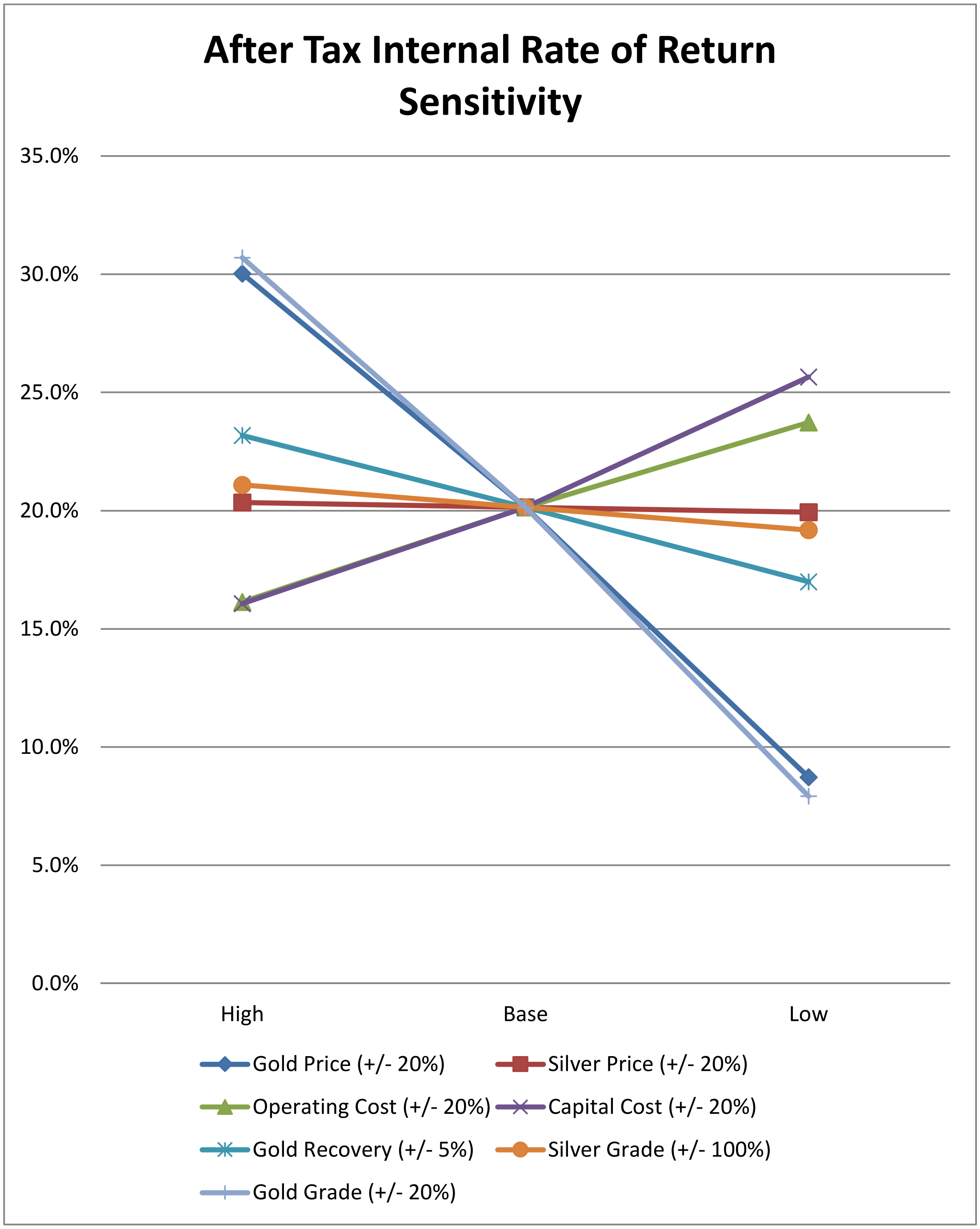
As of the November 2014 Technical Report, HGM has spent $30.8 Million of capital on the project. Those costs are considered “sunk” in the economic model.

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

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

	Before Tax	After Tax
IRR	22.6%	20.1%
NPV @ 0%	$732.0 Million	$596.6 Million
NPV @ 5%	$416.7 Million	$329.2 Million
NVP @ 10%	$229.2 Million	$170.6 Million
Payback Period	3.6 Years	3.9 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, 2014)

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 AND TERMS OF REFERENCE

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
Joshua Snider, PE	September 2015	Sections 1, 2, 3, 4, 6, 18, 19, 21, 22, 23, 25, 26, and 27.
Erin L. Patterson, PE	July 2015	Sections 1, 13, 17, 25, 26 and 27.
Lee "Pat" Gochnour, MMSA	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, 25, 26 and 27.
Carl Burkhalter, PE	May 2014	Section 1, 18, 25, 26 and 27.

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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This report is current as of October 13, 2015.

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

Full Name	Abbreviation
Acid Rock Drainage	ARD
Carbon-In-Leach	CIL
Coastal Plain Sand	CPS
Cubic feet	ft^3
Department of Health and Environmental Control	DHEC
Engineering, Procurement and Construction Management	EPCM
Feet	ft
Haile Gold Mine	HGM
High Density Polyethylene	HDPE
Inches	in
Independent Mining Consultants	IMC
M3 Engineering & Technology Corporation	M3
Mega Watt	MW
Memorandum of Agreement	MOA
Net Present Value	NPV
New Fields LLC, Denver CO	New Fields
Overburden Storage Area	OSA
Potentially Acid Generating	PAG
Probable Maximum Precipitation	PMP
Romarco Minerals, Inc.	RMI
Specific gravity	S.G.
Tailing Storage Facility	TSF
Temperature in Degrees Fahrenheit	°F
Troy ounce	oz
Troy ounces per short ton	opt
Short tons per year	t/y

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.

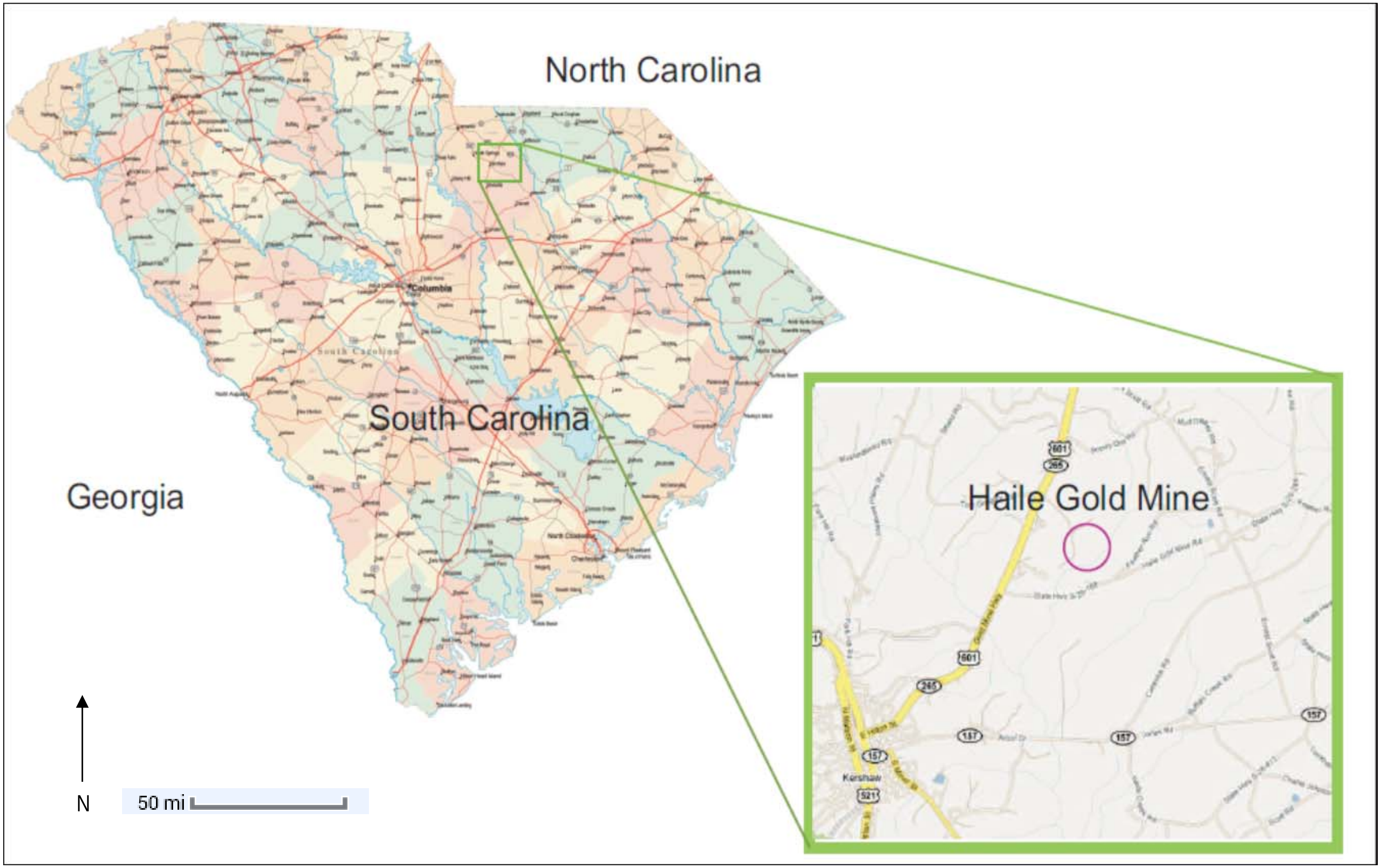
An independent verification of land title and tenure was not performed. M3 has not verified the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties. Likewise, HGM has provided data for and verified water rights, land ownership, and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

7.1 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 Ordovician–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.

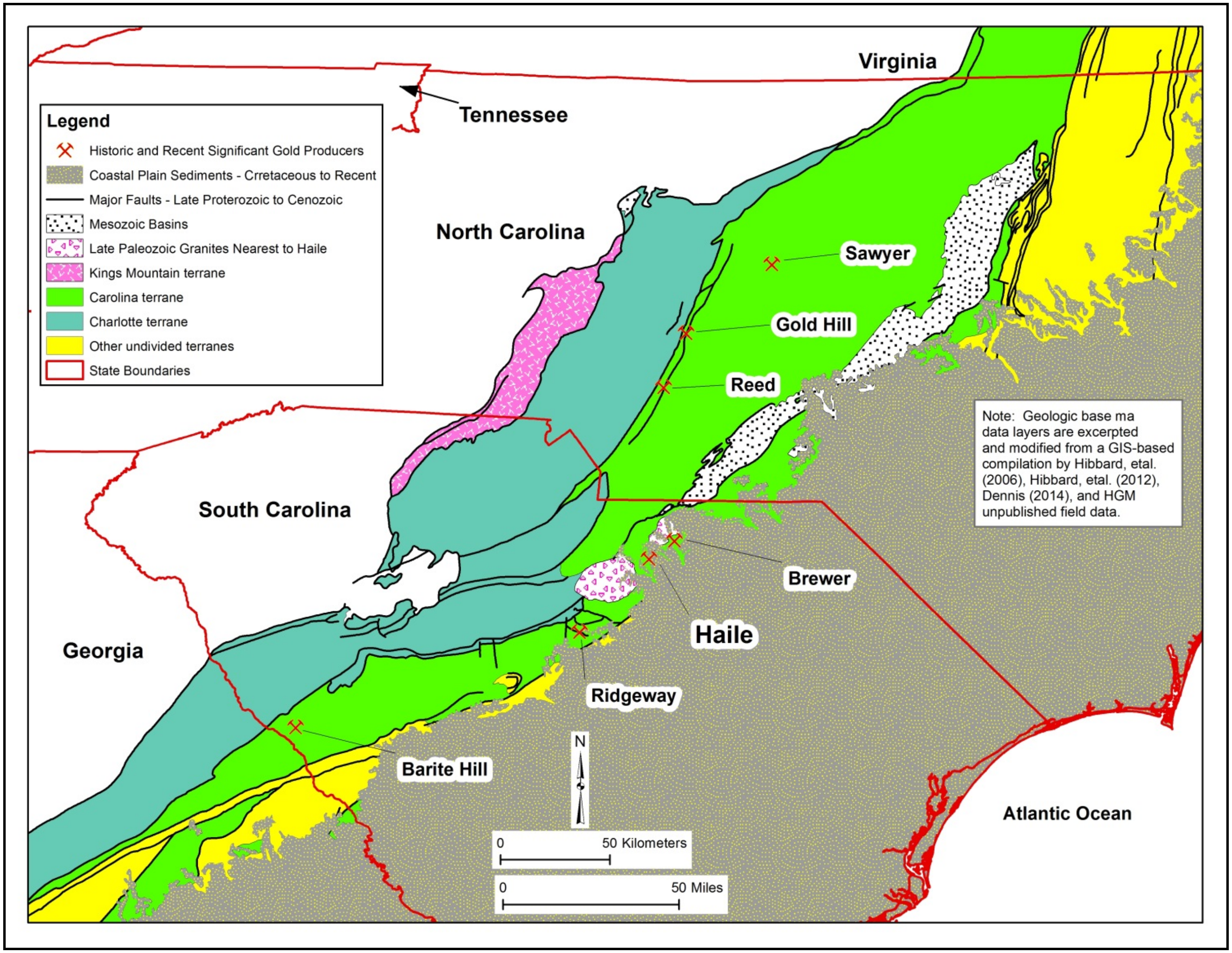


Figure 7-1: Gold Deposit Locations within the Carolina Terrane

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 Neoproterozoic 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 thicknes 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.

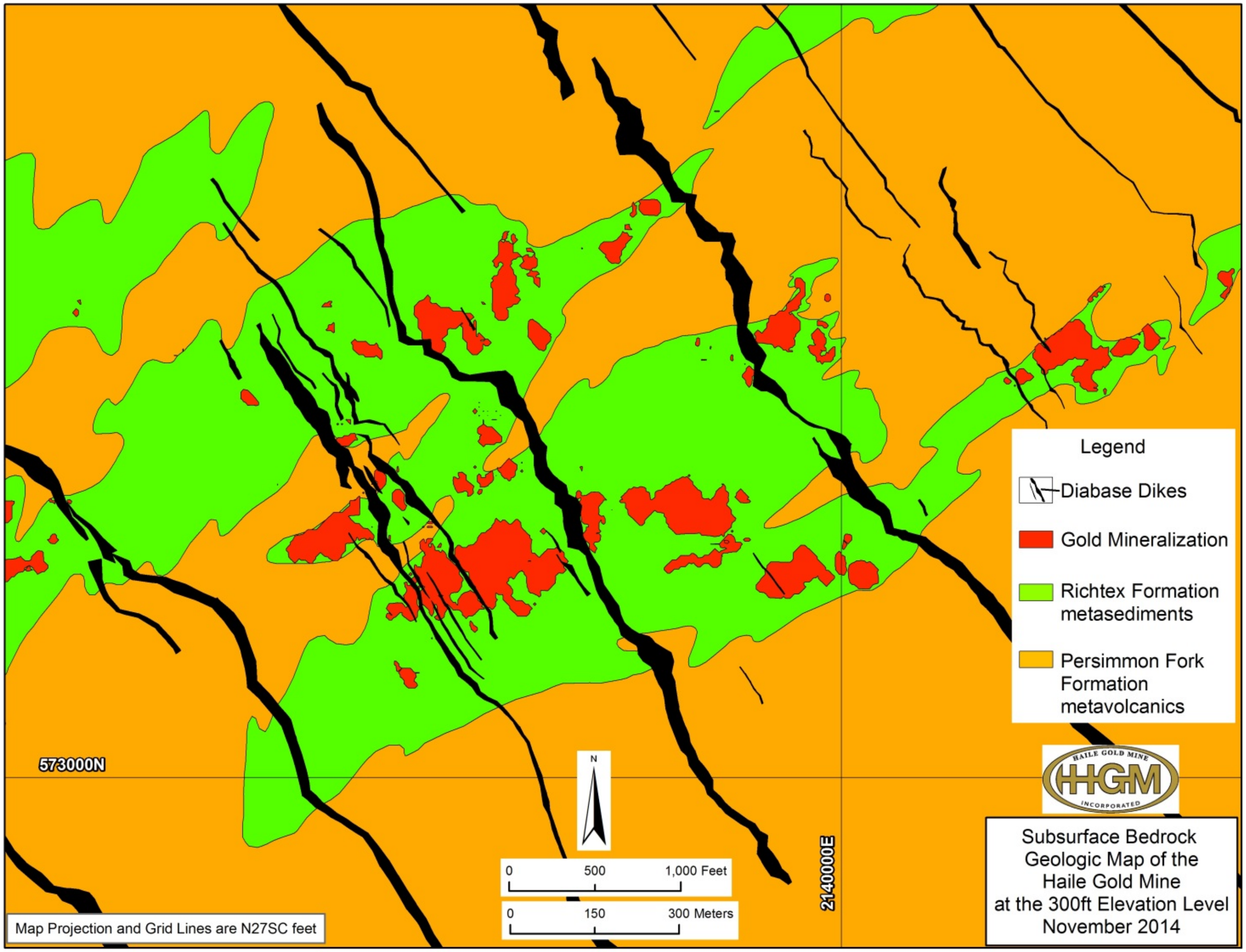


Figure 7-2: Schematic Geologic Map of Haile Property, November 2014

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 $\pm$ 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 from 529 to 564 Ma. Four of these samples give a weighted age of 548.7 $\pm$ 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 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.

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 volcanics 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 *November 21, 2014 Haile Gold Mine Technical Report* that was issued for Romarco Minerals, Inc. In October of 2015, Romarco Minerals was acquired 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 *November 21, 2014 Haile Gold Mine Technical Report* that was issued for Romarco Minerals, Inc. In October of 2015, Romarco Minerals was acquired 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.

Drill holes that fire assay above a grade of zero amounted to 2,039 drill holes containing 254,681 assay intervals amounting to 1,372,473 ft of drilling information. The presence of fire assay is indicative of the amount of drilling that was used for development of the block model and mineral resource.

As of November 17, 2011, Romarco had drilled 1,001,594 ft of the fire assayed drilling out of the total 1,372,473 ft on the property. The historical 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 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.

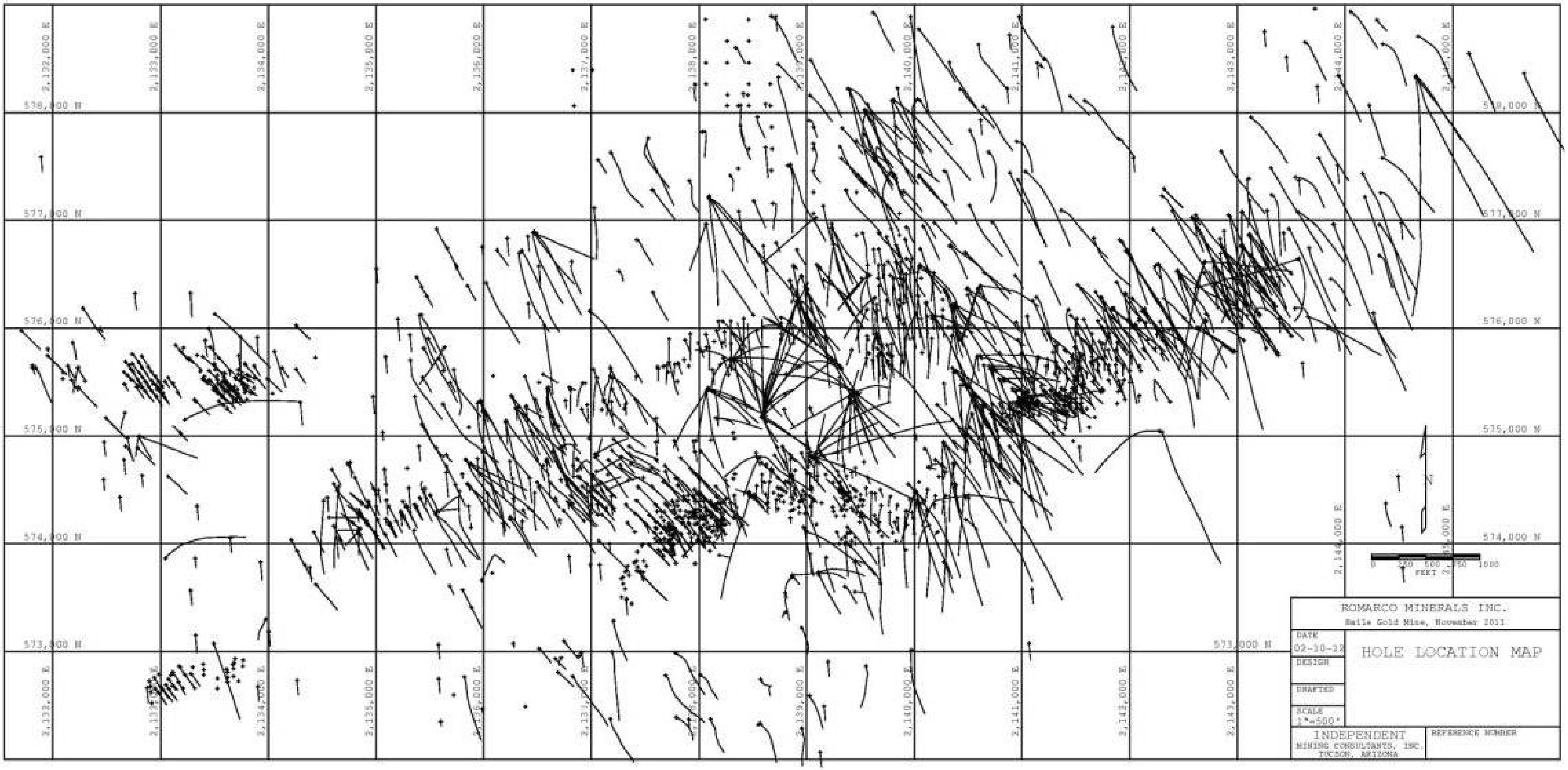
IMC has completed a comparison of historic drilling to Haile-Romarco drilling and has found that the old and new data can be commingled if it has been fire assayed.

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.



(Source: IMC, 2012)

Figure 10-1: Drill Hole Location Map – Holes with Fire Assay as of November 2011

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

This section is based on the *November 21, 2014 Haile Gold Mine Technical Report* that was issued for Romarco Minerals, Inc. 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.

Reverse Circulation Drilling

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 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 digestion and Atomic Absortion 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-Chemx 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 may be used from KML 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 *November 21, 2014 Haile Gold Mine Technical Report* that was issued for Romarco Minerals, Inc. In October of 2015, Romarco Minerals was acquired by OceanaGold Corporation.

The Haile drill hole data base was verified by IMC in late 2011 and the results published in the Technical Report titled "Haile Gold Mine Project, NI43-101 Technical Report Feasibility Study" dated 10 February 2011. This chapter 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.

In summary, the 2011 IMC standards data set 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 33 individual standards with the highest grade standard (SN50) being 0.2533 oz/ton.

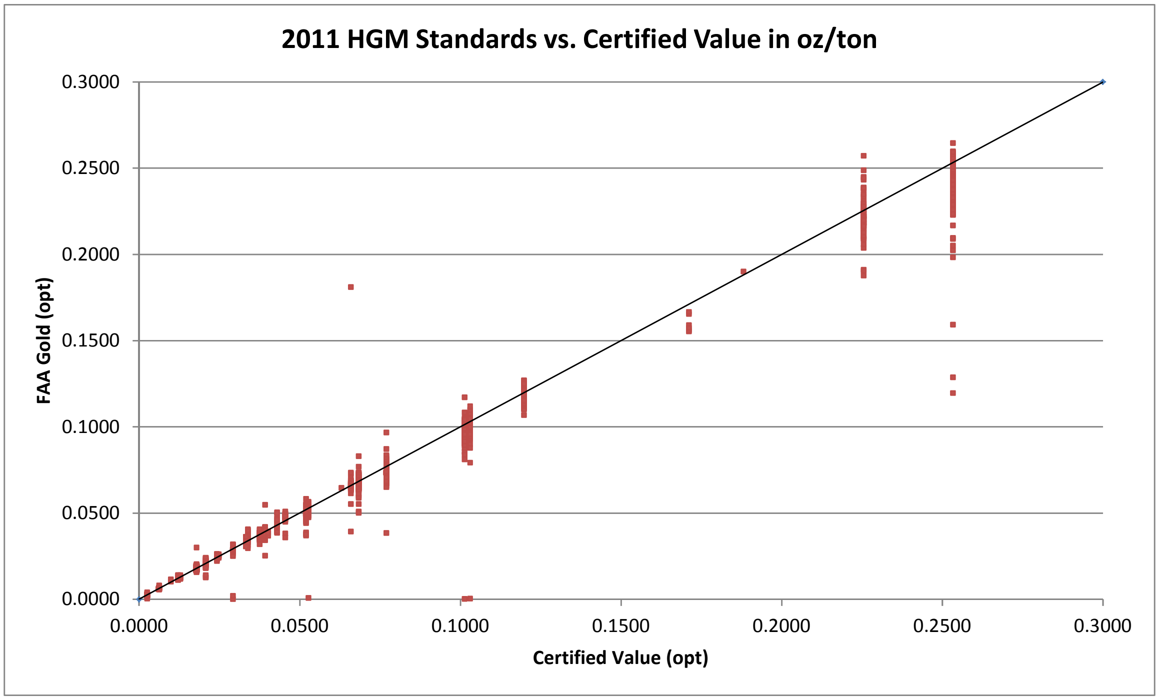


Figure 12-1: 2011 HGM Standards vs. Certified Value in oz/ton

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.

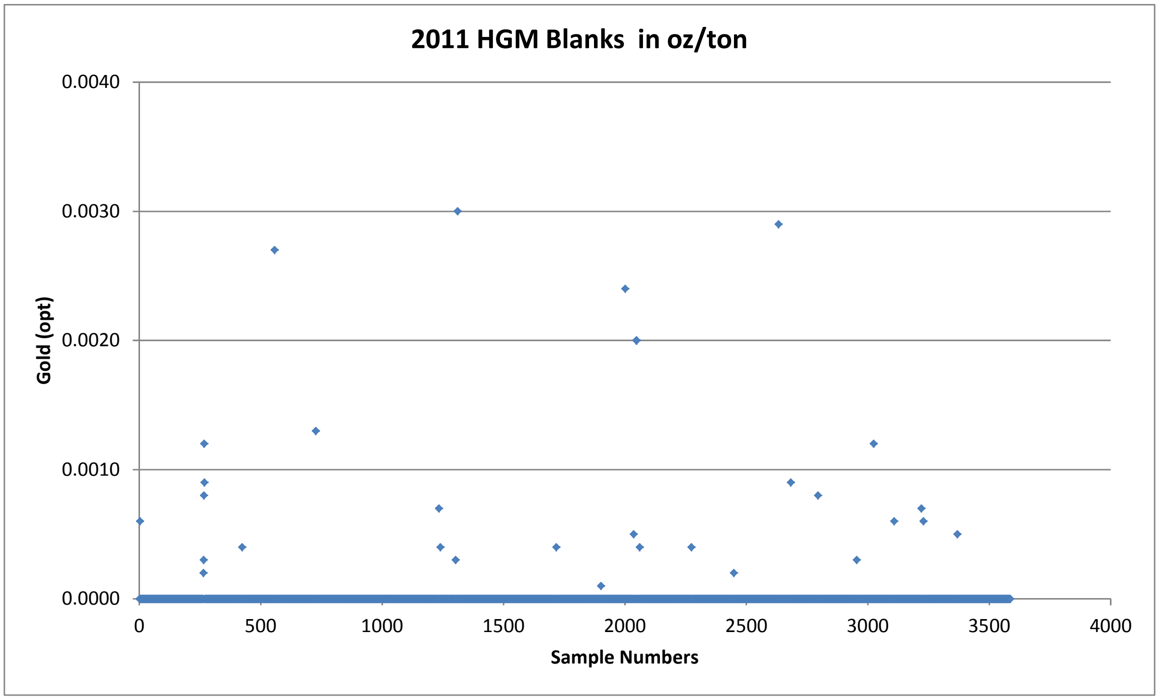


Figure 12-2: 2011 HGM Blanks in oz/ton

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 14.1.

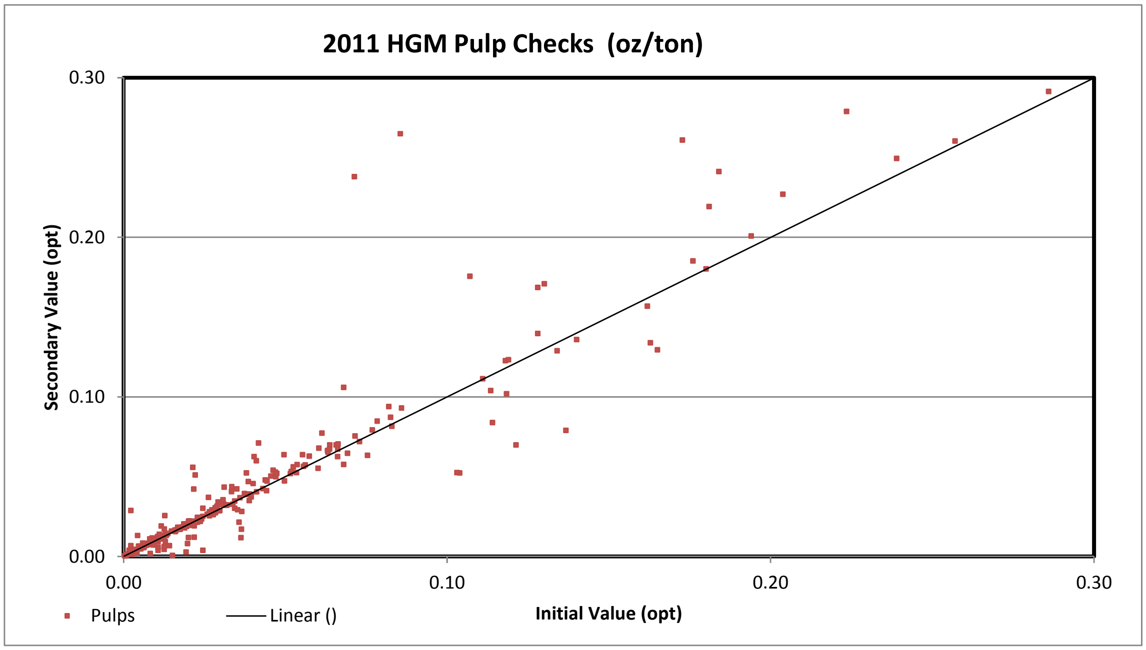


Figure 12-3: AHK/KML Gold Assays versus Chemex Pulp Assays

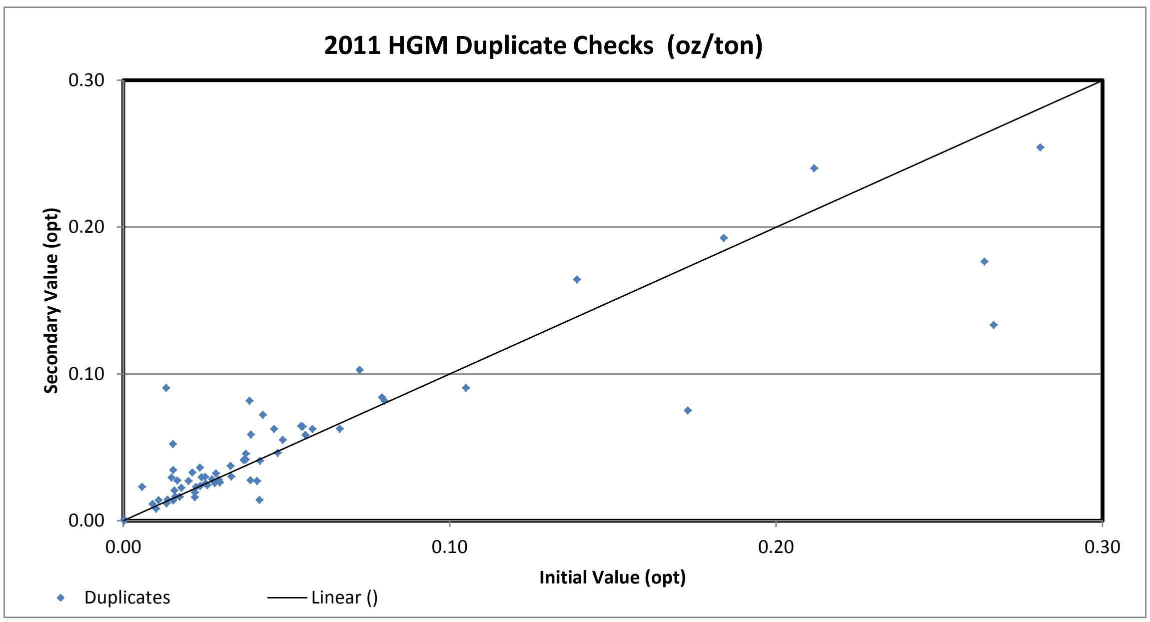


Figure 12-4: AHK/KML Gold Assays versus Chemex Duplicate Preparation and Assay

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 communition 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, 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

Ore Zone	% Au Recovery – Combined				Average % Au Recovery
	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		NaCN Consumption lbs/t
				Au	Ag	
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.

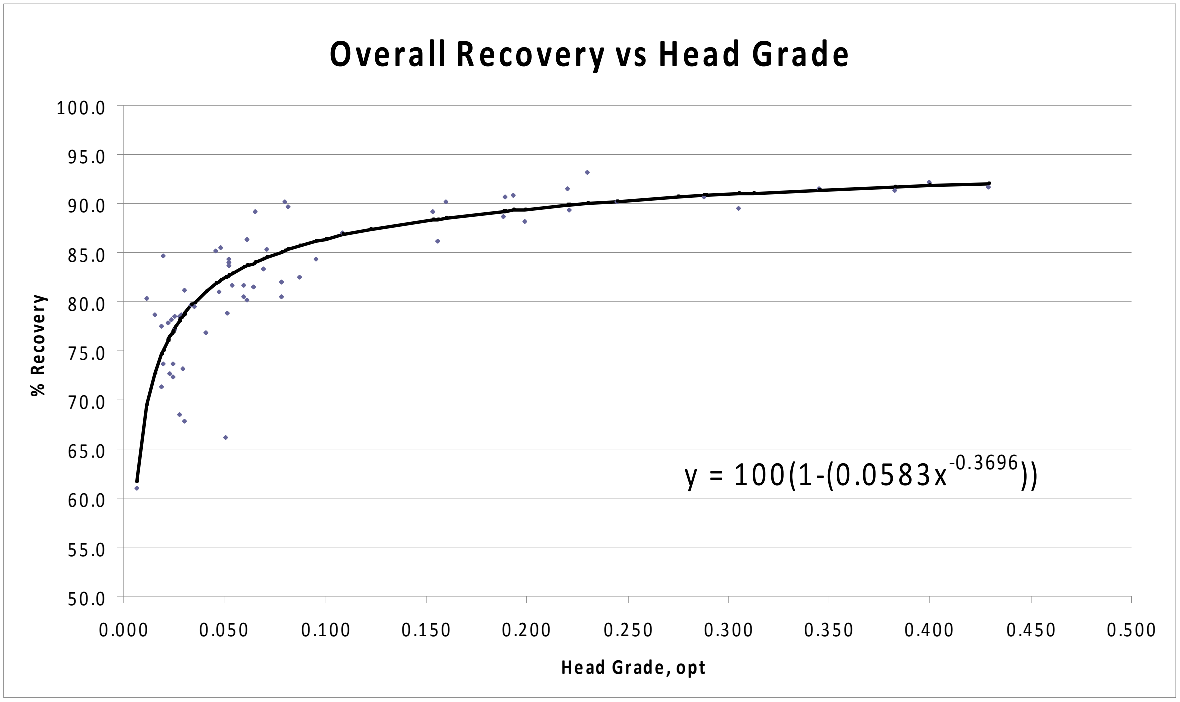


Figure 13-1: Overall Percent Recovery vs. Head Grade

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

This 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. It is presented in its entirety without change since that date. The mineral resources at Haile include material with potential for economic extraction by both open pit and underground methods. Both targets are included within a single block model of the Haile mineralization that was assembled by Independent Mining Consultants, Inc. (IMC). The potential open pit component was estimated by IMC. The potential underground component was estimated by Snowden Mining Industry Consultants, Vancouver BC (Snowden).

This section will first describe the block model and follow with a discussion of both open pit and underground resource targets. John Marek, P.E. of IMC acted as the Qualified Person for the development of the model and the open pit mineral resource estimate. Anthony Finch, P. Eng. of Snowden acted as the Qualified Person for the underground 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	
Model		578 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.

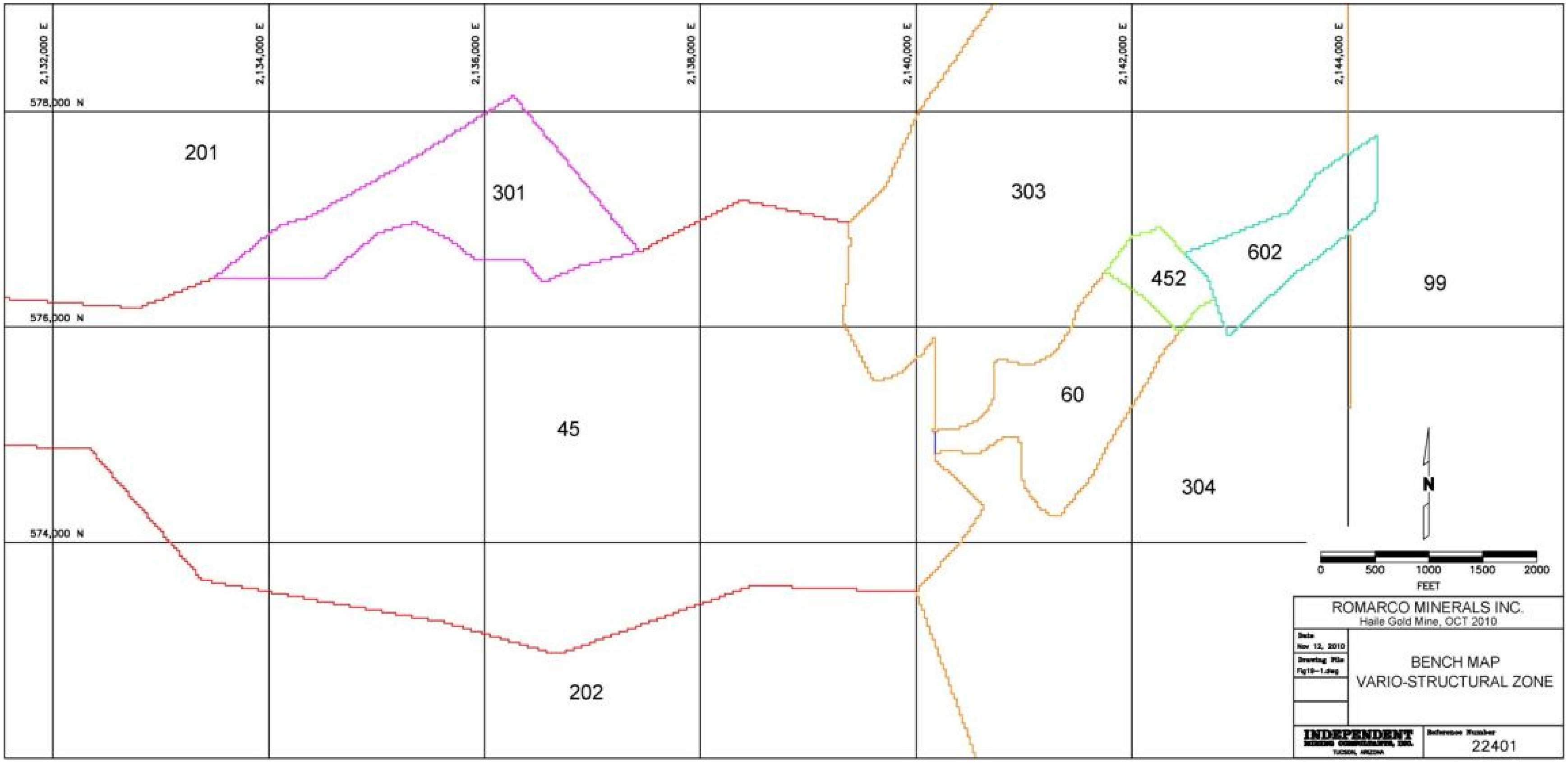
Population statistics were developed for each of the zones on the map. Cumulative frequency plots were developed to understand the populations. The cumulative frequency plots were also used to set levels to cut individual assays. The cap values for assays are summarized below:

Assay Cap Values

Meta-Sediments: Rock = 100

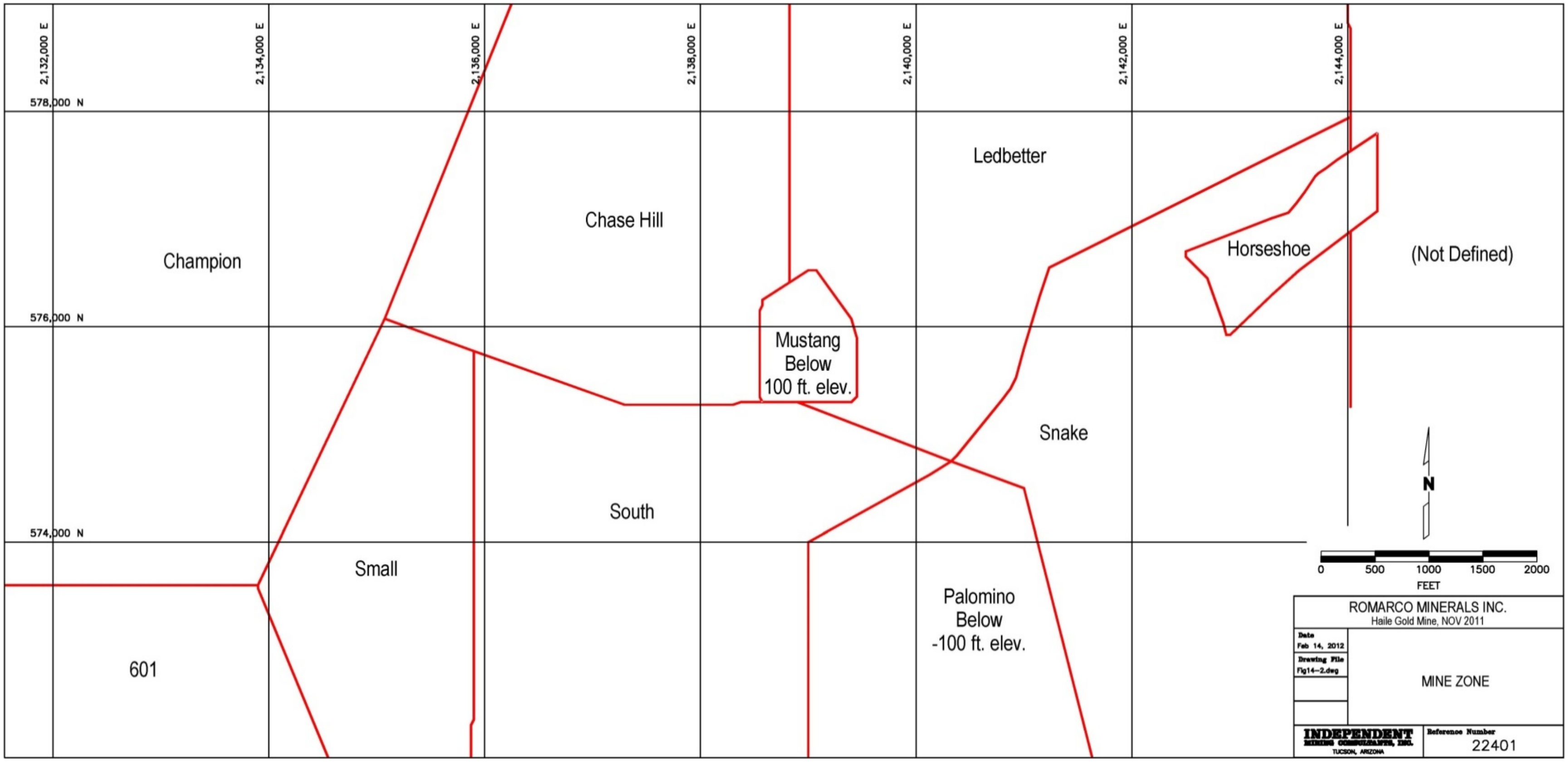
VarioZone	Description	Cap
45	Haile- South	0.70 oz/ton
303	Ledbetter	1.00 oz/ton
60	Snake	1.00 oz/ton
602	Horseshoe	3.00 oz/ton
Everywhere Else		0.40

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									Row Total
	45	60	201	202	301	303	304	452	602	
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			Variogram		Discrim oz/ton
			Plunge	Strike	Cross	Nugget	Sill	
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			Variogram		Discrim oz/ton
			Plunge	Strike	Cross	Nugget	Sill	
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			Variogram		Discrim oz/ton
			Plunge	Strike	Cross	Nugget	Sill	
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 seach 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 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	0.036	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

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.

14.2.2 Underground Mineral Resource

The component of the resource model with reasonable prospects of economic underground extraction were developed by Snowden Mining Industry Consultants, Vancouver BC (Snowden). Anthony Finch P. Eng (APEGBC) M AusMM (CP Min) acted as the Qualified Person for the underground resource shape estimation.

Snowden generated underground resource shapes that had the potential to be extracted using underground mining methods. These shapes lie below the US$1200 per ounce gold open pit resource shell which was provided by IMC to Snowden. These shapes were developed by Snowden using the resource model prepared by IMC in December 2011. Background technical information and general site information was based upon the feasibility study filed on SEDAR on February 22, 2011.

Open stoping with paste back-fill was selected as the mining method for the purpose of estimating the underground resource inventory. The underground resource shape dimensions were based upon parameters determined as a consequence of a geotechnical review of relevant drill hole data and reports by Snowden. Stopes were a nominal 100 feet wide, 100 feet long, and 60 feet high, although variations to these shapes were applied in some cases due to partial stopes, and the influence of the open pit shells.

The identified inventory was assumed to be accessed by decline from either the surface, or from the lower parts of the open pits as they are excavated. Fill for the stopes could be delivered by either truck or surface bore hole. Ventilation would be provided by surface mounted primary fans, with a system of return air raises progressively installed as the mine is developed.

An underground resource gold cutoff grade of 0.08 oz/ton was used for the calculation of the inventory. The cut-off grade was based on a total production cost of $72.61/ton, which was inclusive of operating mining costs, processing and site G&A. The production cost did not include capital development to access the stoping blocks, however the ability of the inventory to carry capital access costs and associated infrastructure was considered during the estimation process as a test of the potential practicality of the inventories. Mining dilution was 12.5% (by mass) which was an average based on different stope dilution responses in primary and secondary stoping operations. Dilution was assigned no grade.

Isolated zones beneath the pit shell that were deemed not amenable to practical mine access and generally less than 100,000 tons were not reported. These isolated blocks were tested by applying typical development costs to access them, and determine whether there was enough contained value to cover basic access expense.

Metallurgical recovery was applied using a variable recovery algorithm provided by Romarco. The algorithm was developed from test work undertaken as part of the feasibility study (2011). The algorithm is represented in Table 14-4 above and is consistent with the open pit metallurgical recovery assumptions.

The spatial location of the shapes comprising the resource estimate, relative to the $1200 per ounce gold pit shell, are shown in plan, long-section and cross-section in Figure 14-4, Figure 14-5 and Figure 14-6, respectively.

All calculations were undertaken using US standard imperial measurements (feet, pounds, tons) due to the site locale, and the format of the resource model.

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.

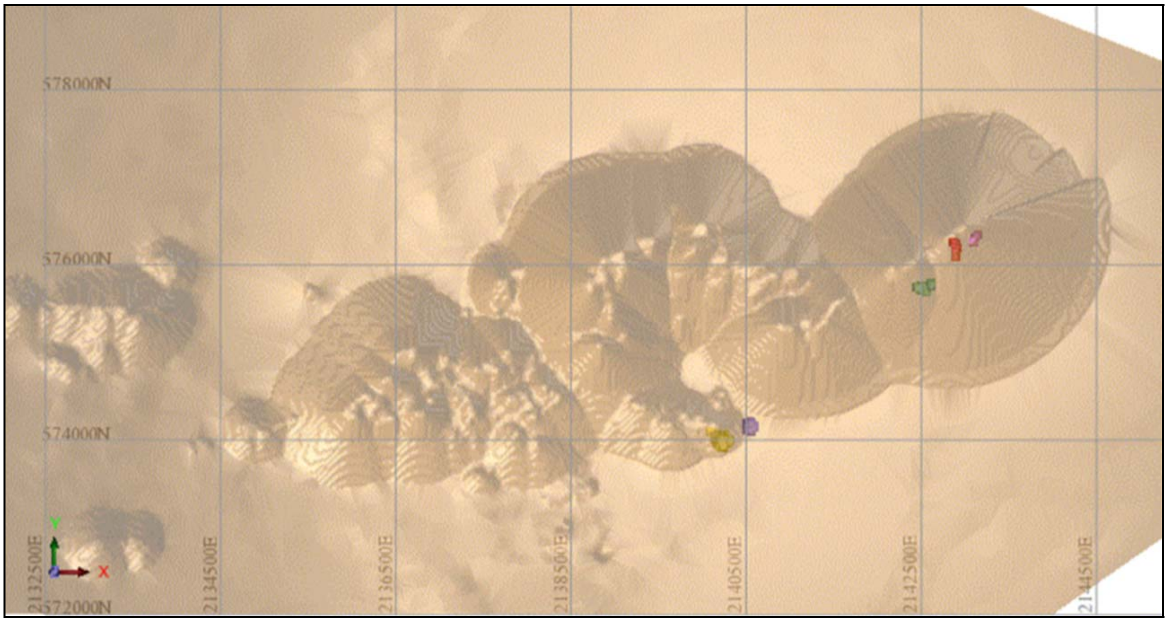


Figure 14-4: Plan view of $1200 pit shell showing the relative position of the underground inventory at 0.080 oz/ton cut-off grade

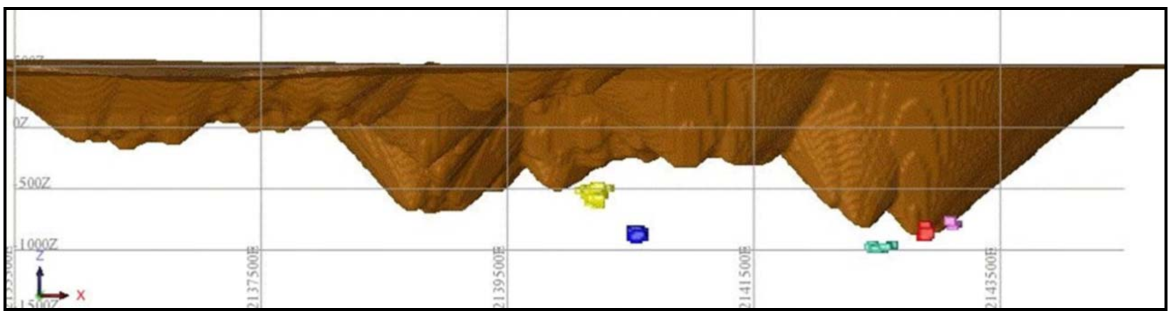


Figure 14-5: Long Section (W to E) showing location of underground stope shapes at 0.080 oz/t cut-off grade relative to $1200/oz pit shell

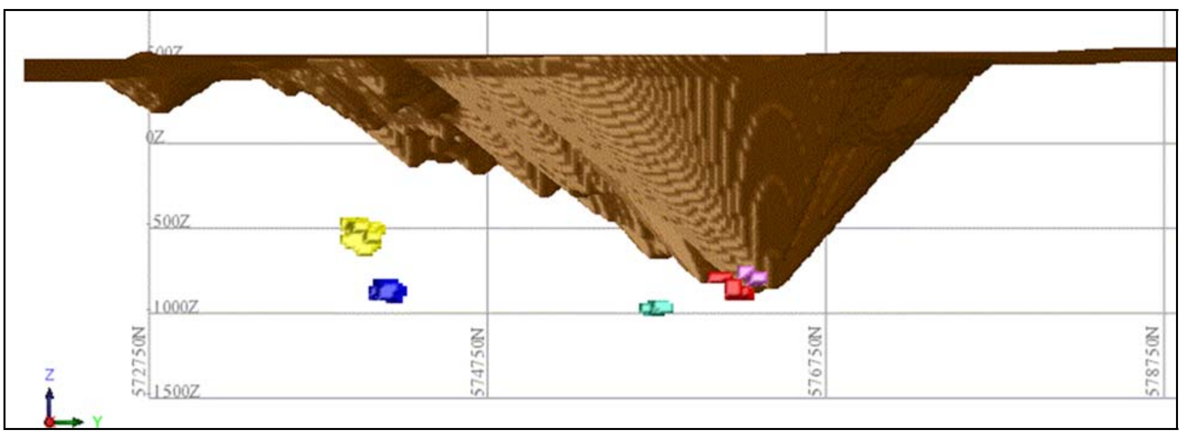


Figure 14-6: Cross Section (S to N) showing location of underground stope shapes at 0.080 oz/t cut-off grade relative to $1200/oz pit shell

Table 14-8: Haile Gold Mine Inc. Underground Mineral Resources as of January 1, 2012 and 1 November 2014

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000
Measured	0.080	140	0.128	18.0
Indicated	0.080	789	0.128	101.0
Measured + Indicated	0.080	929	0.128	119.0
Inferred Resource	0.080	773	0.122	94.0

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 are outside of the mineral reserve
Mineral Resources in this table are outside of the open pit resource

The underground resources currently exist in three named zones. The breakout of the resources by zone is shown in Table 14-9.

Table 14-9: Underground Mineral Resource by Zone, $1200 Gold, 0.80 oz/ton Cutoff

	Measured		Indicated		Measured + Indicated		Inferred	
Zone	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton	Ktons	Oz/ton
Palomino			373	0.114	373	0.114	672	0.127
Snake	140	0.128	116	0.107	256	0.118	24	0.120
Horseshoe			300	0.154	300	0.154	77	0.081
Totals	140	0.128	789	0.128	929	0.128	773	0.122

14.2.3 Total Mineral Resource

The total mineral resource with the open pit and underground material combined is summarized on Table 14-10.

Table 14-10: Haile Mine Inc. Total Mineral Resources as of January 1, 2012 and 1 November 2014
Combined Open Pit Plus Underground Material

Category	Gold Cutoff oz/t	Tons x 1000	Grade Troy Oz/ton	Contained Oz x 1000
Measured	0.012- 0.080	40,669	0.052	2,125
Indicated	0.012- 0.080	37,784	0.051	1,914
Measured + Indicated	0.012- 0.080	78,453	0.051	4,039
Inferred Resource	0.012- 0.080	22,184	0.036	801

Notes:
Cutoff grades are 0.012 oz/ton open pit, and 0.080 oz/ton underground
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 on this table include the mineral reserve

Qualified persons for the mineral resources are John Marek, P.E. of IMC and Anthony Finch, P.Eng of Snowden.

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.

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 increased	10.1%	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.

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 Section16.

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
6 ½" Blast Hole Drills	3	4
15 Cubic Yd Front Loader	1	2
17 Cubic Yd Front Loader	1	1
14.4 Cubic Yd Hyd Shovel	1	1
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 are presented in this section for brevity.

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.

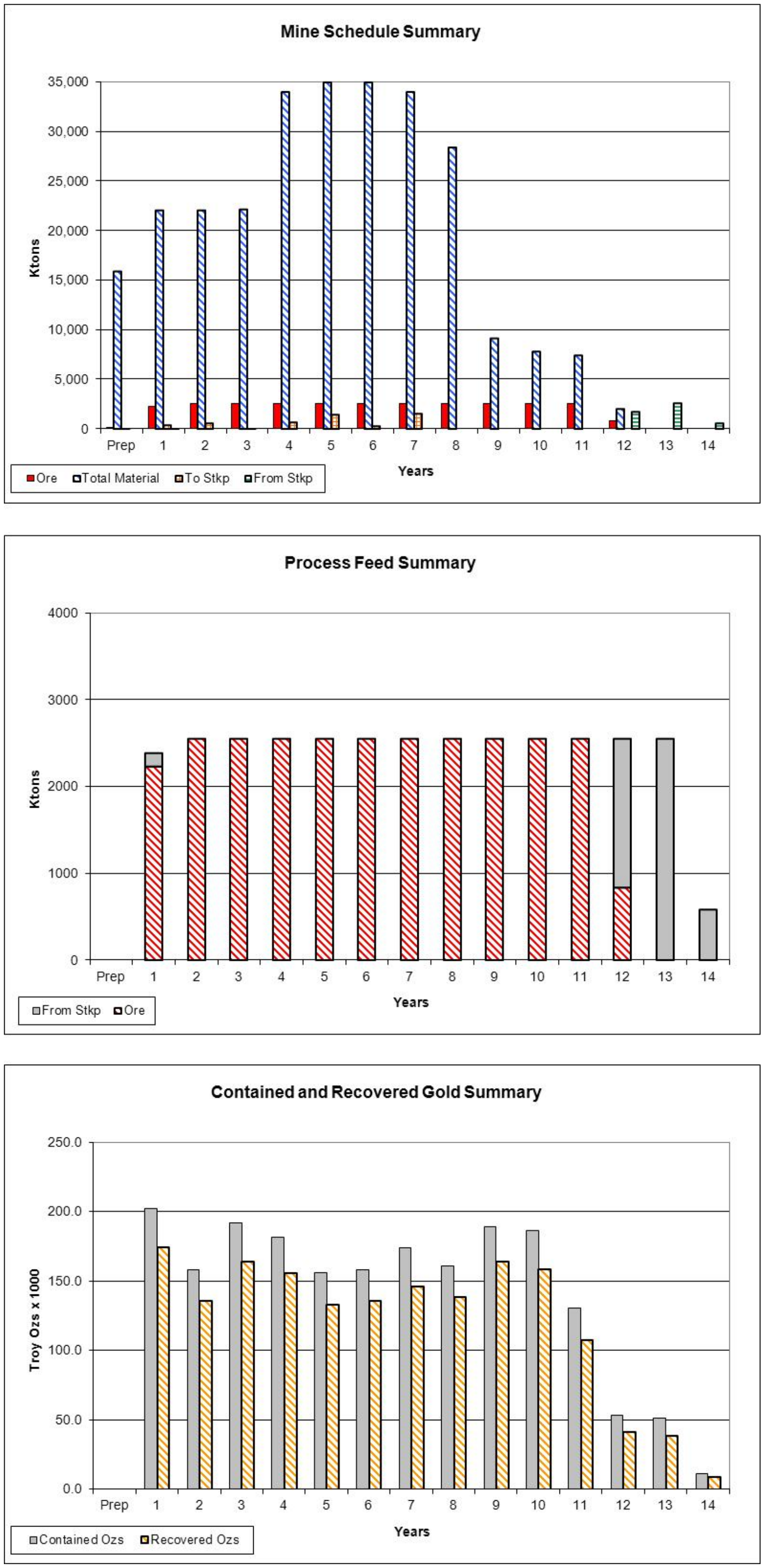


Figure 16-1: Mine Production Schedule, Graphical Summary

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.

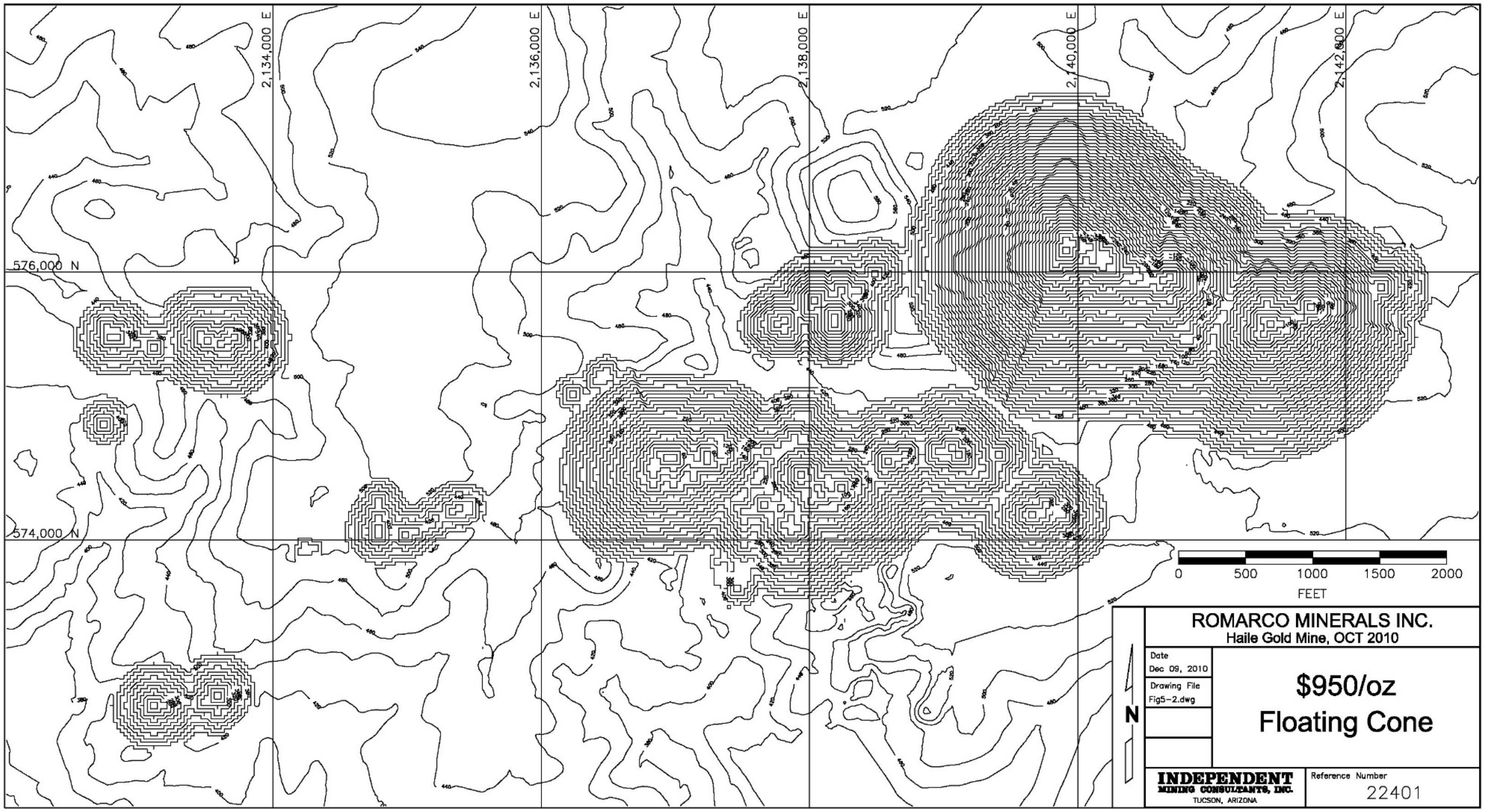
Slope angles for the floating cones and later phase designs were recommended by Golder Associates, Inc., in their report titled "Feasibility Level Pit Slope Evaluation", March 2010. The interramp angles recommended by Golder were reduced to reflect overall angles for input to the floating cone program. 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.

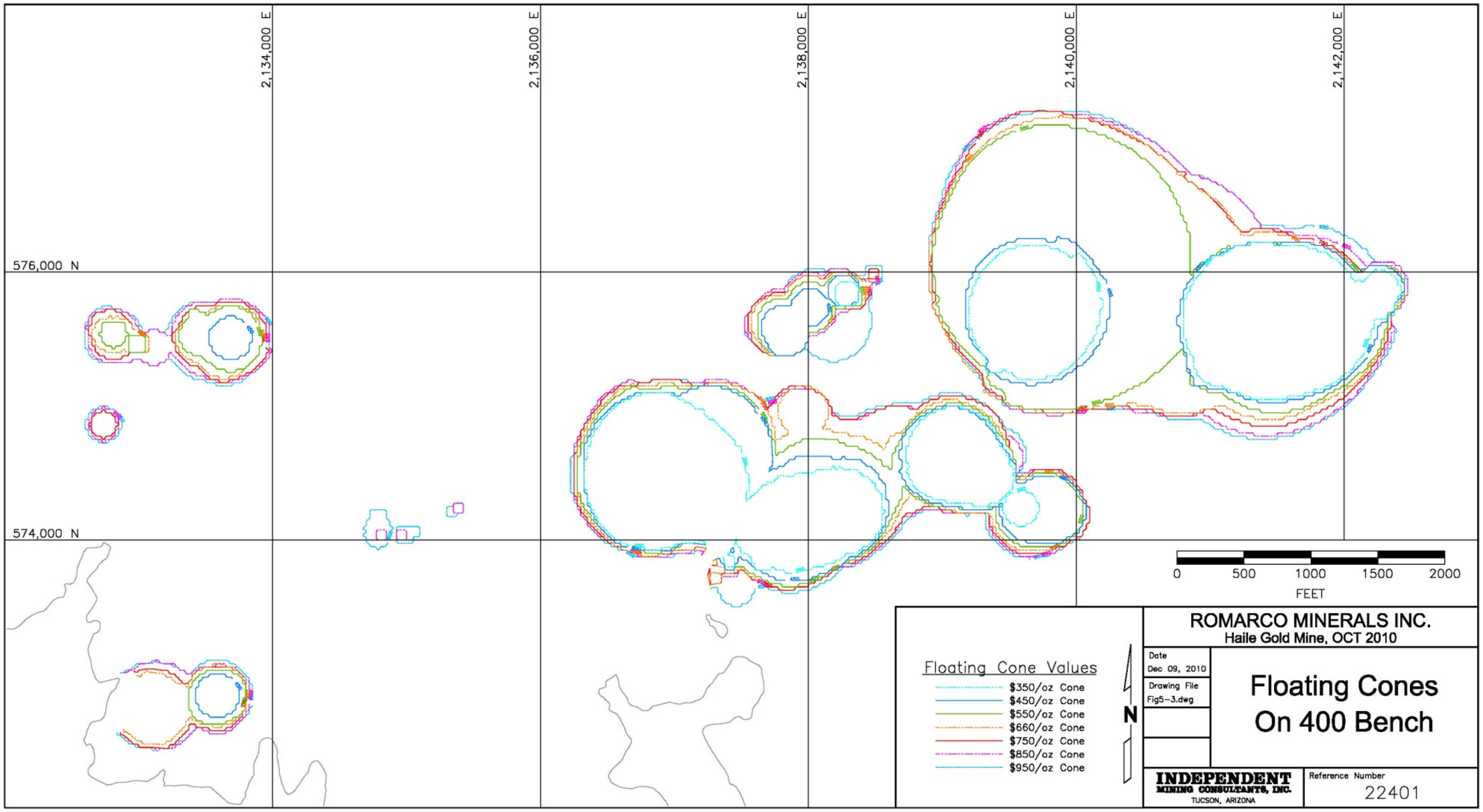
In addition to the $950/oz cone, IMC completed multiple floating cones at metal prices ranging from $350/oz to $950/oz. These high value floating cones were used as guidance for the development of pushback designs at Haile.

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.

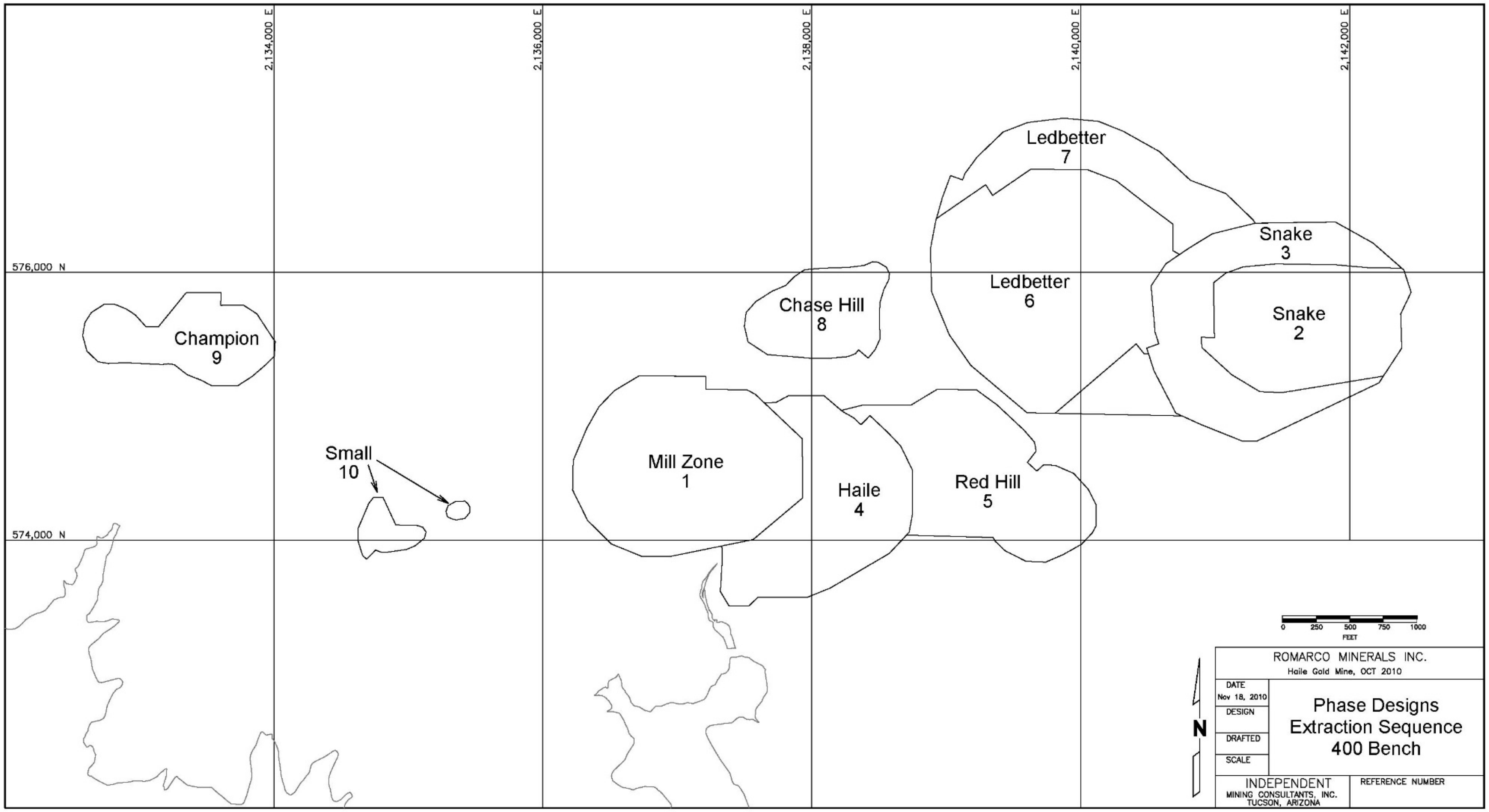
The following design criteria were incorporated into the phase designs:

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.

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.

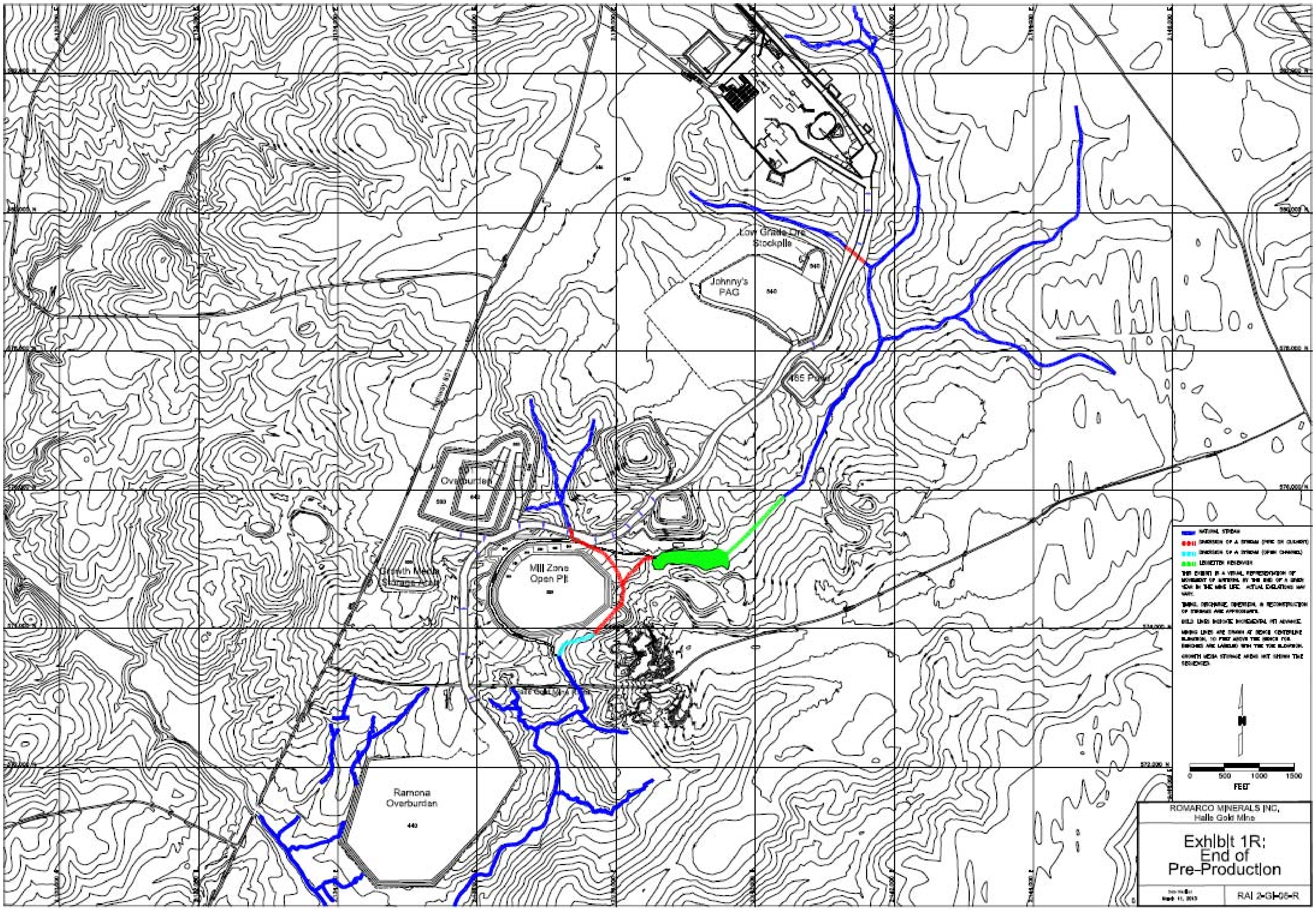


Figure 16-5: End of Preproduction

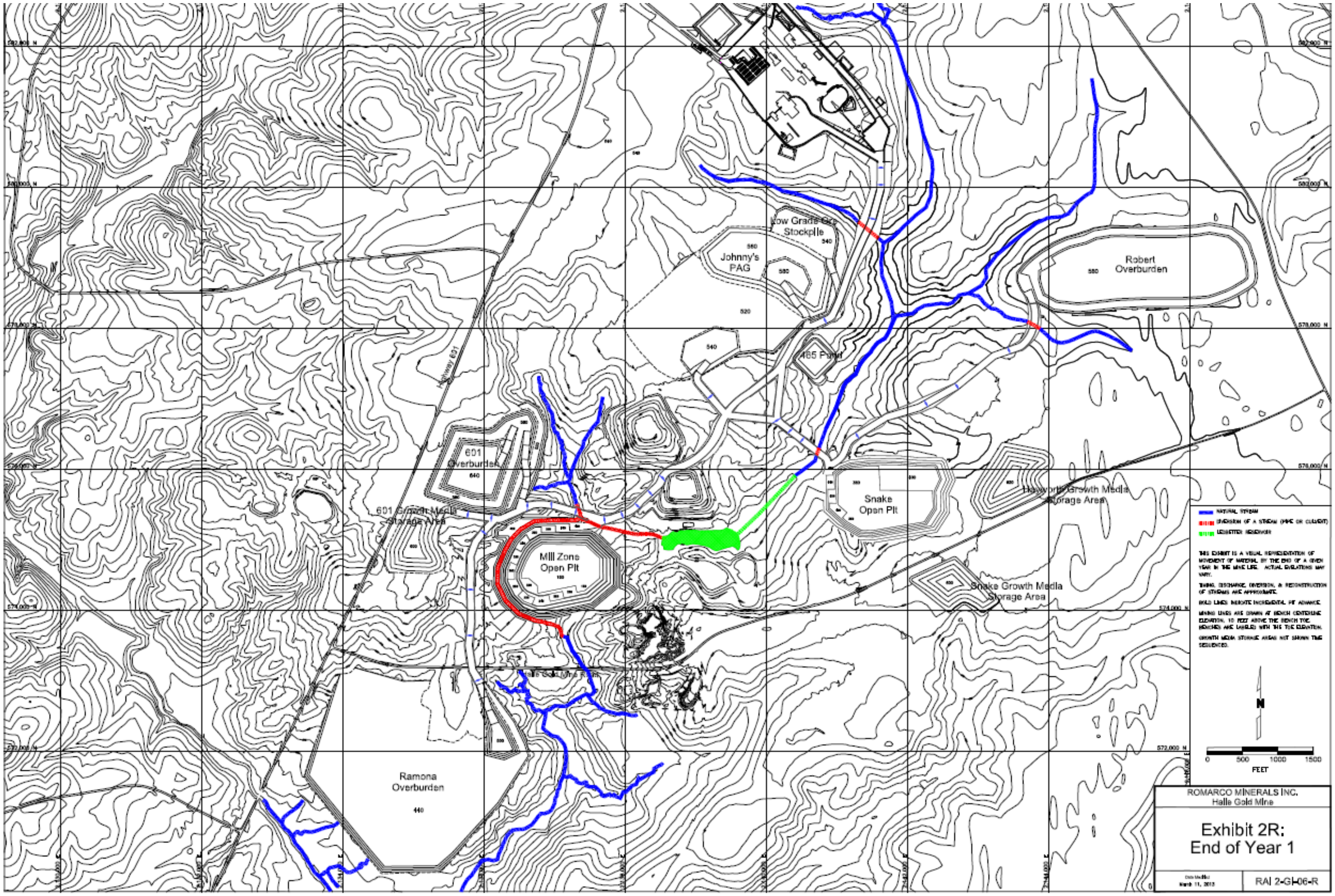


Figure 16-6: End of Year 1

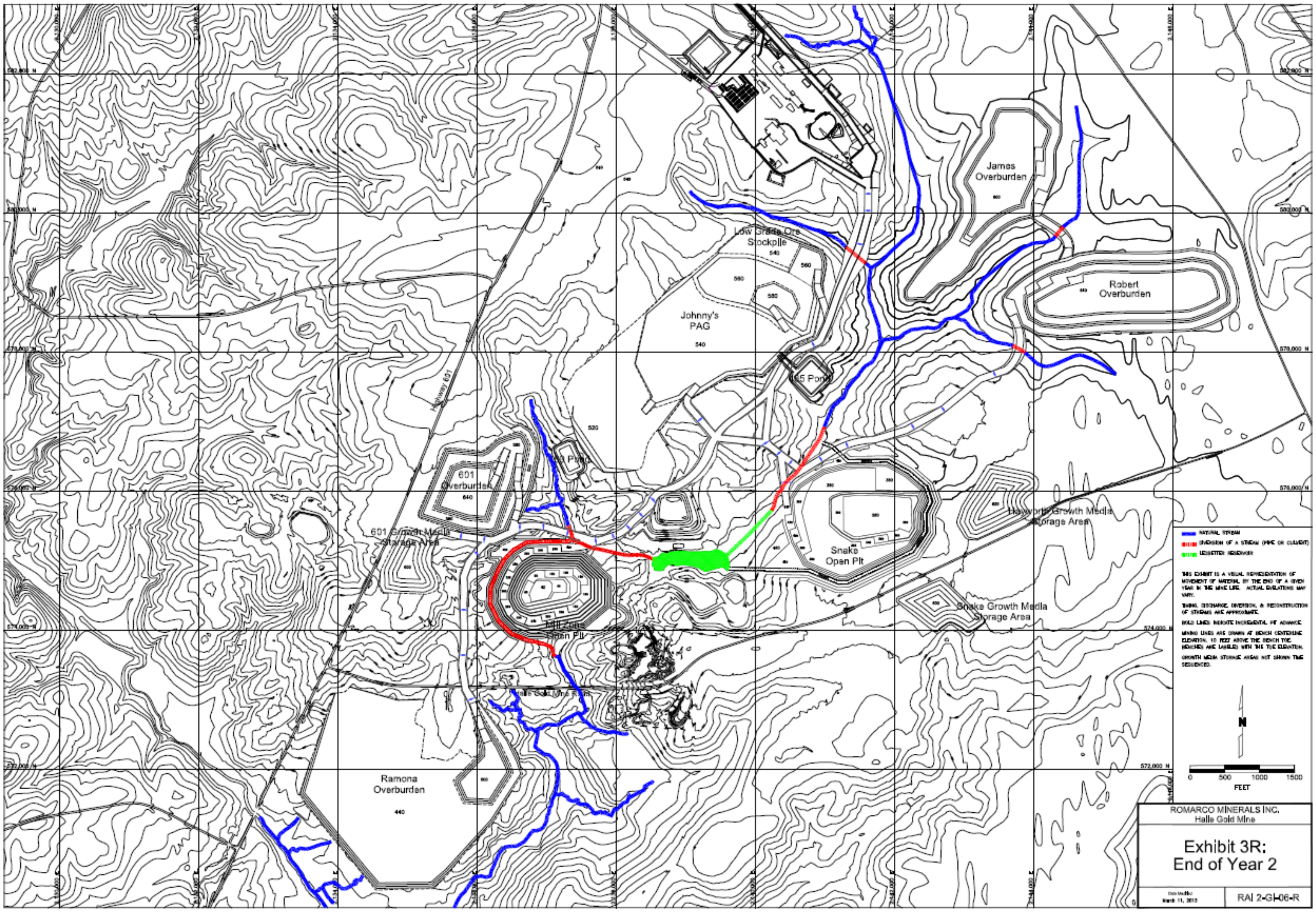


Figure 16-7: End of Year 2

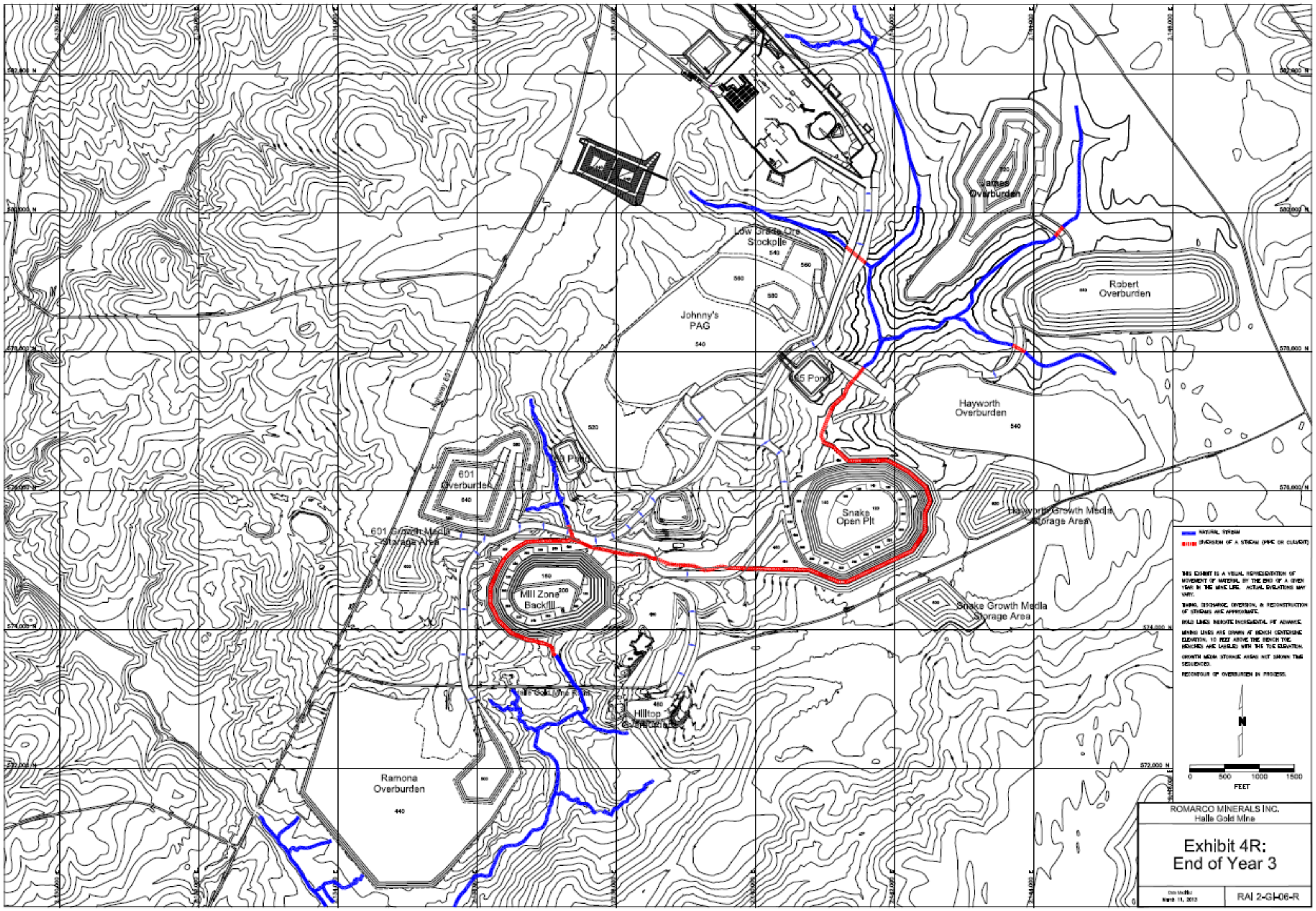


Figure 16-8: End of Year 3

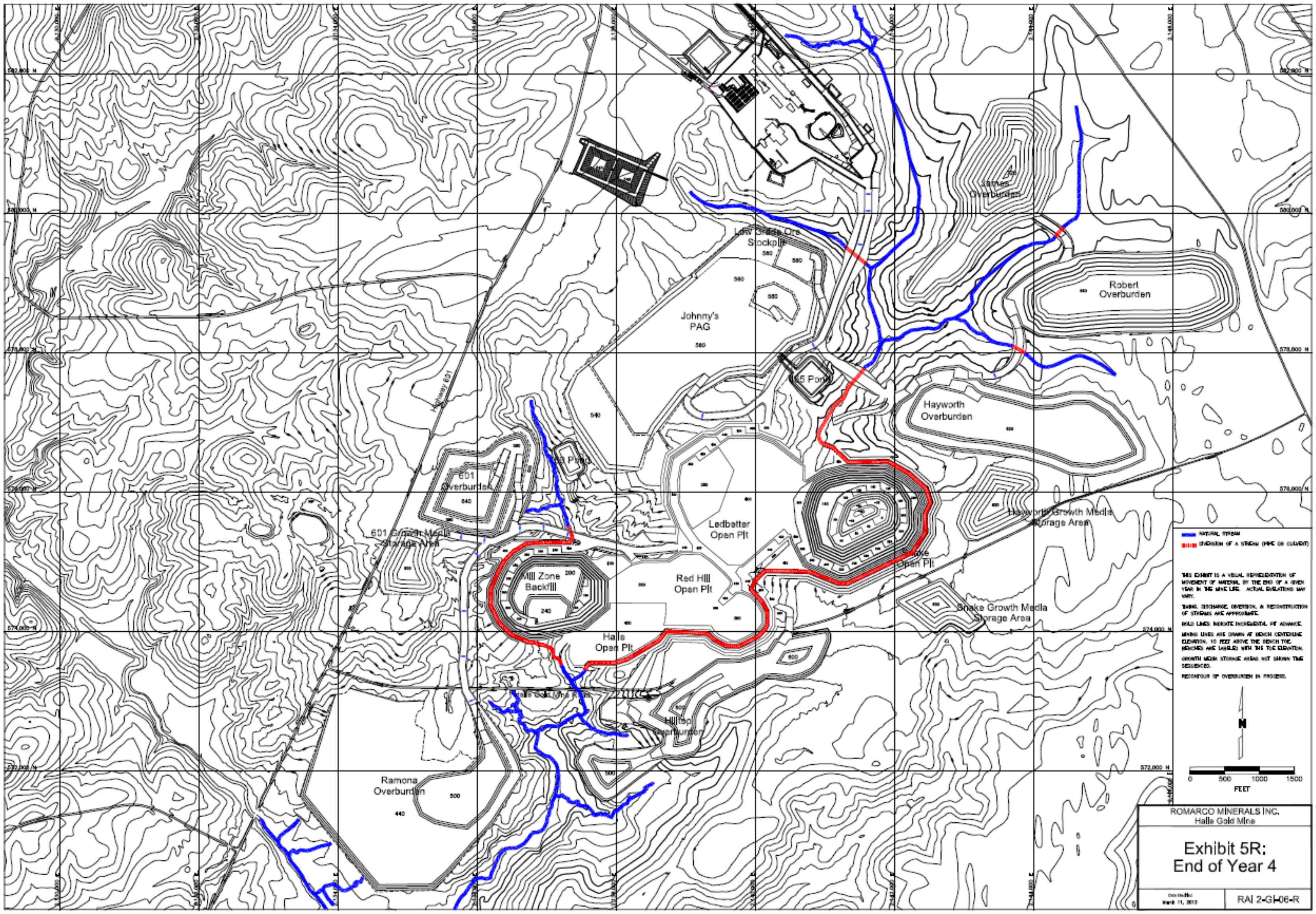


Figure 16-9: End of Year 4

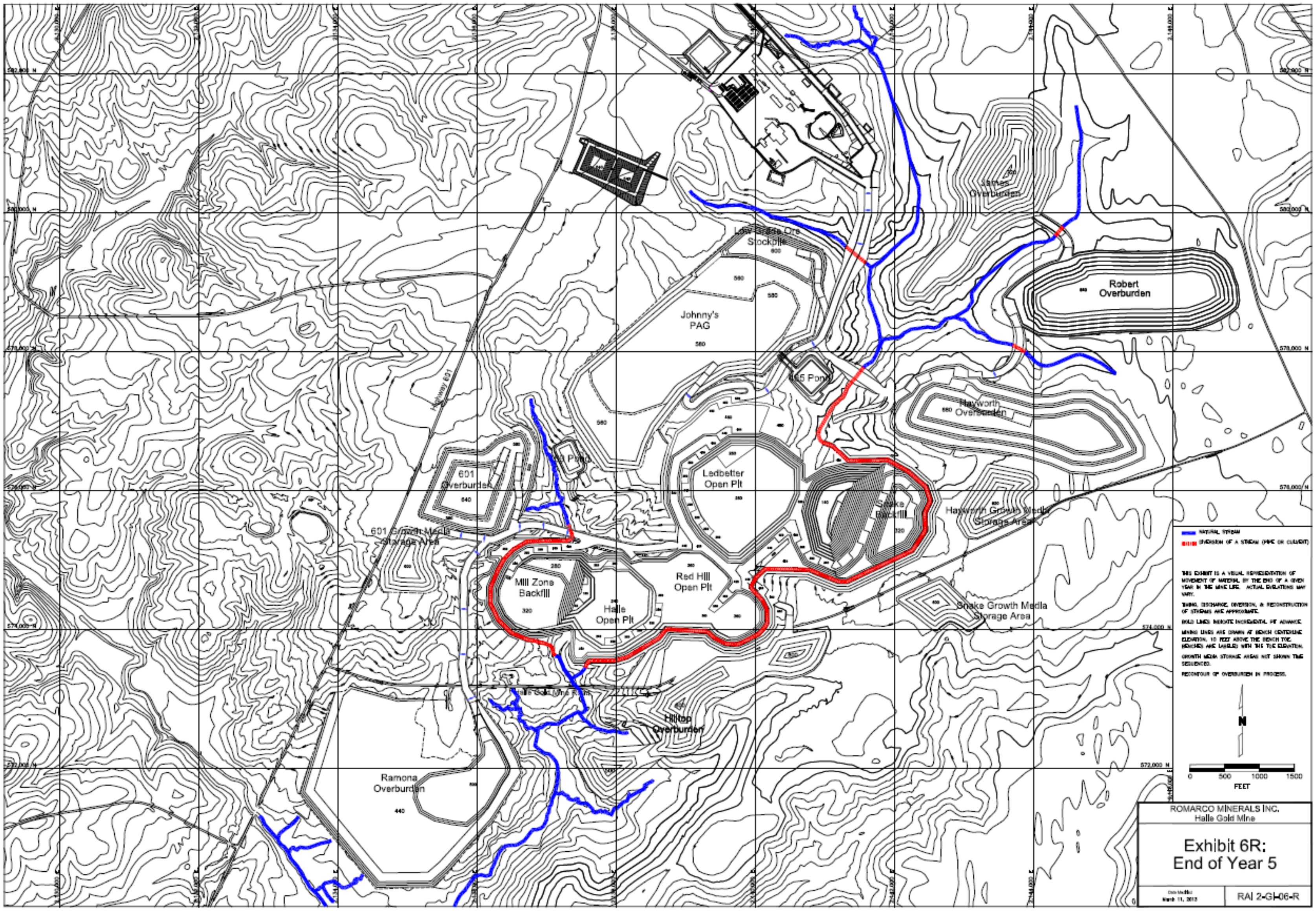


Figure 16-10: End of Year 5

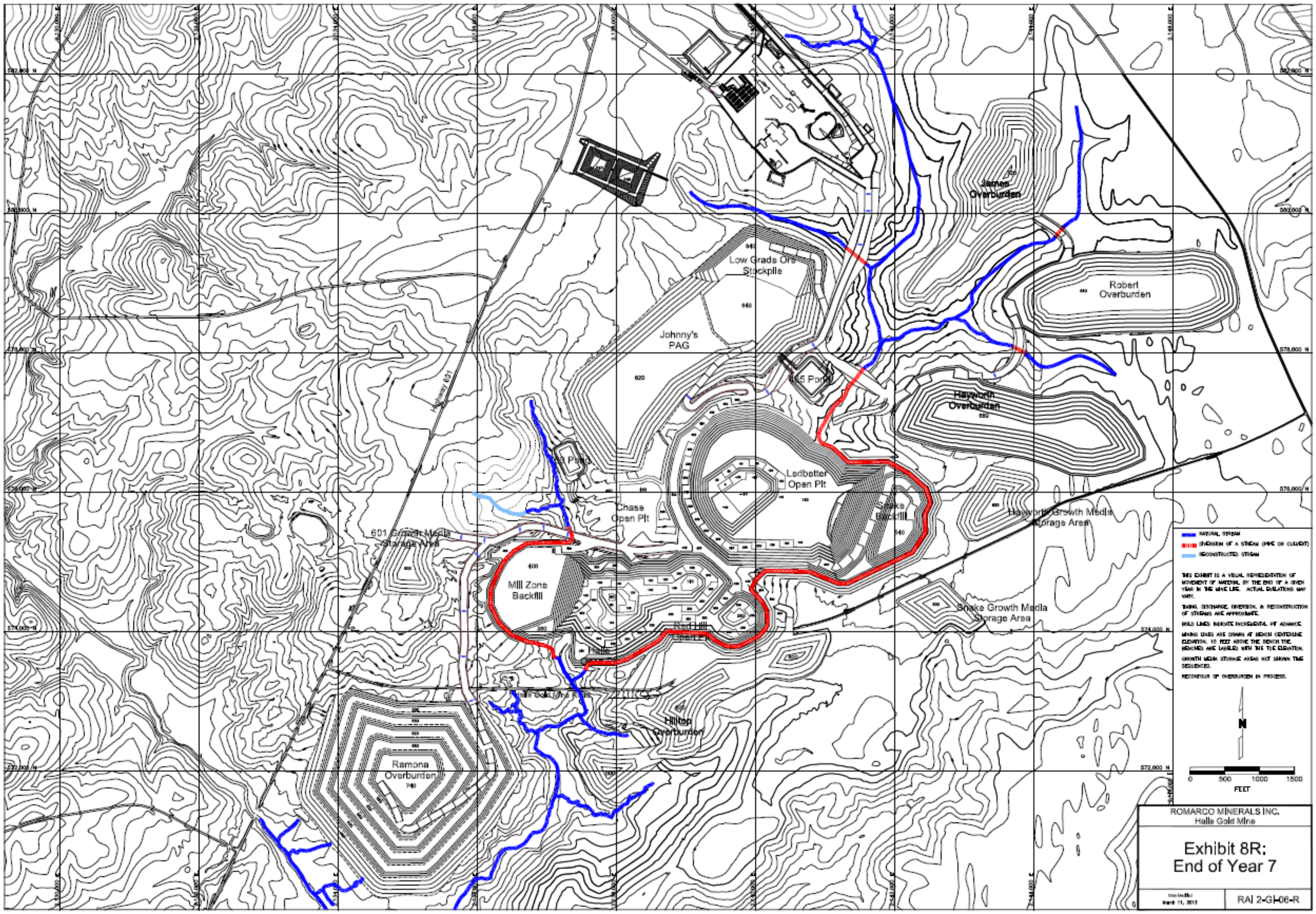


Figure 16-11: End of Year 7

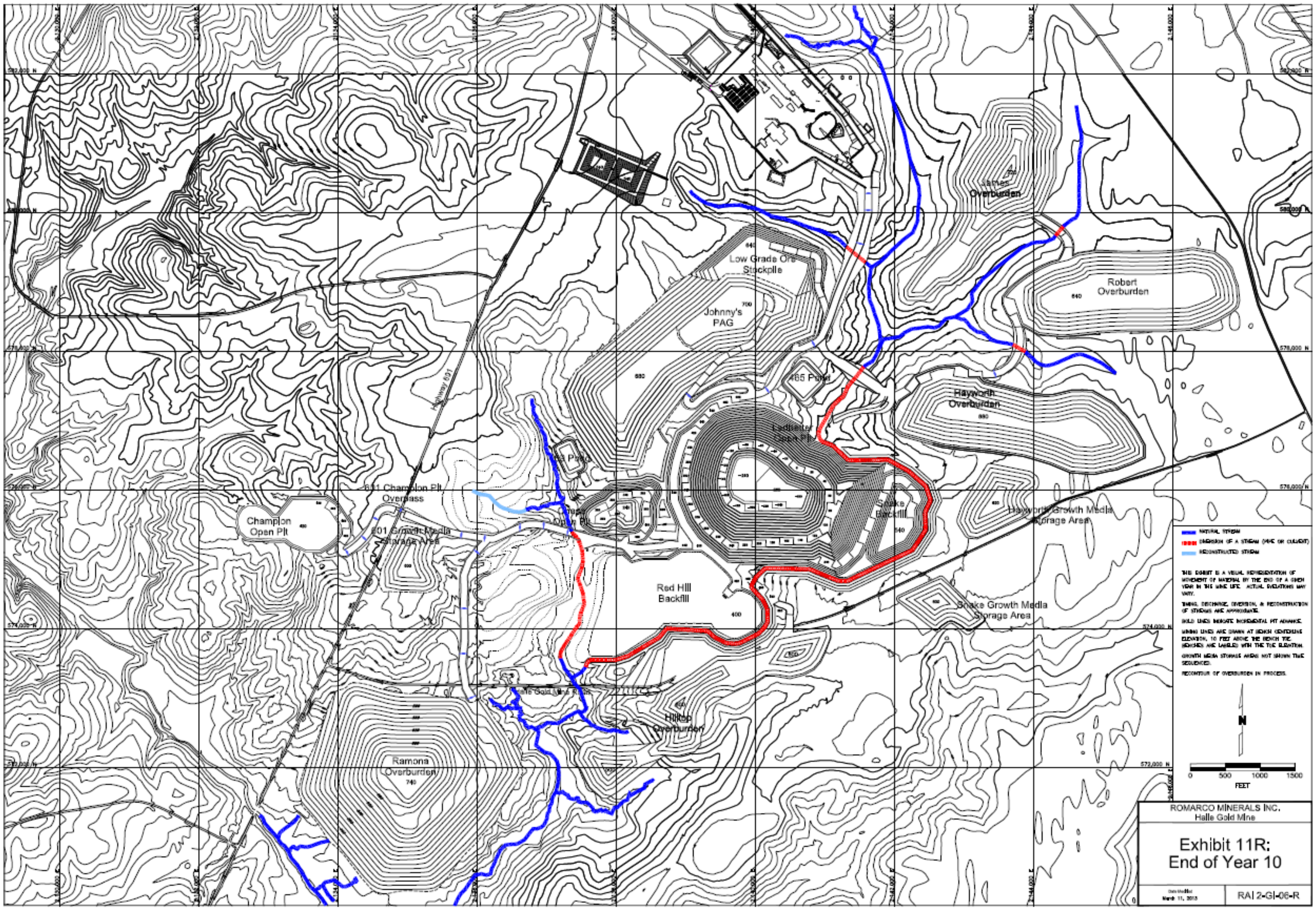


Figure 16-12: End of Year 10

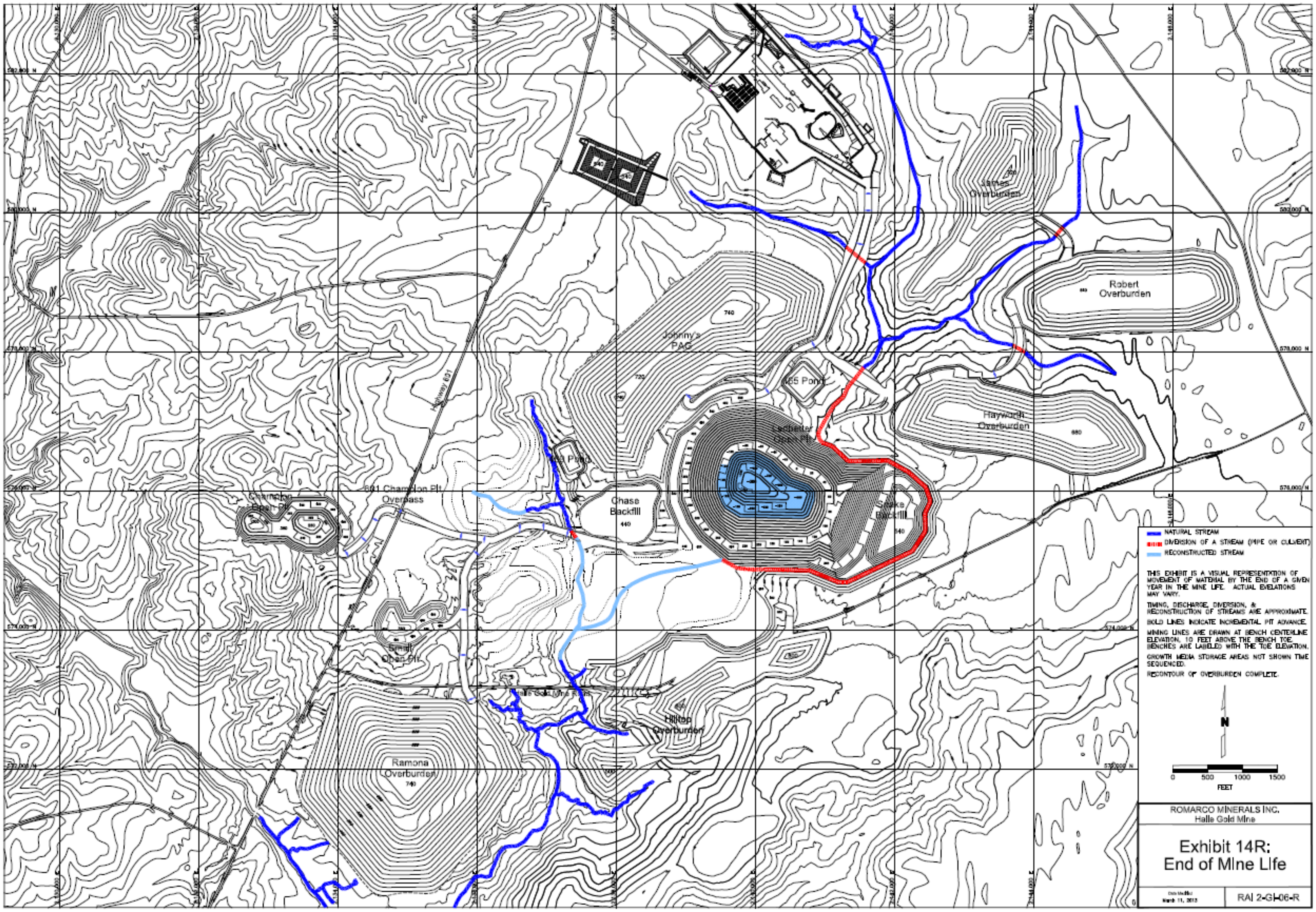


Figure 16-13: End of Mine Life

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 will be lined during the first quarter of preproduction stripping. The mine will deliver 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 is planned with rotary down hole hammer drills equipped to drill 6.5 inch holes. Three drills will be needed initially followed by a total of 4 later in the mine life.

During the first three years of production, loading equipment will be a mixed fleet of one hydraulic shovel and 2 front end loaders. The hydraulic shovel is a 14.4 cubic yard unit. The front end loaders are 15 cubic yard and 17 cubic yard units respectively. Front end loaders are selected due to versatility and high mobility. The hydraulic shovel is 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 shovel has a better break-out force for digging material that has not been blasted. Consequently, a shovel is provided as a contingent approach to the front end loaders.

A third loader 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
Atlas Copco DM45 Blasthole Drill			1	1	3	3	3	3	3	3	3	3	3	3	3	3	5	5	5	5	5	5	5	5	5	5
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 992K & 993K Wheel Loaders	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	2
Cat 777F Haul Truck	12	12	12	12	12	12	12	12	12	12	12	12	12	12	14	19	19	24	24	24	24	24	18	12	12	12
Cat D9 Track Dozers	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	1	1
Cat D10 Track Dozers	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 824H Wheel Dozer	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 14M Motor Grader	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
Cat 773 Water Truck	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Cat 988 Loader	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 773 Haul Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Cat 740 Fuel Truck	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 336DL Excavator	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
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
TOTAL	27	27	29	29	31	31	32	32	32	32	33	33	33	33	35	41	43	48	48	48	47	47	41	35	34	32

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

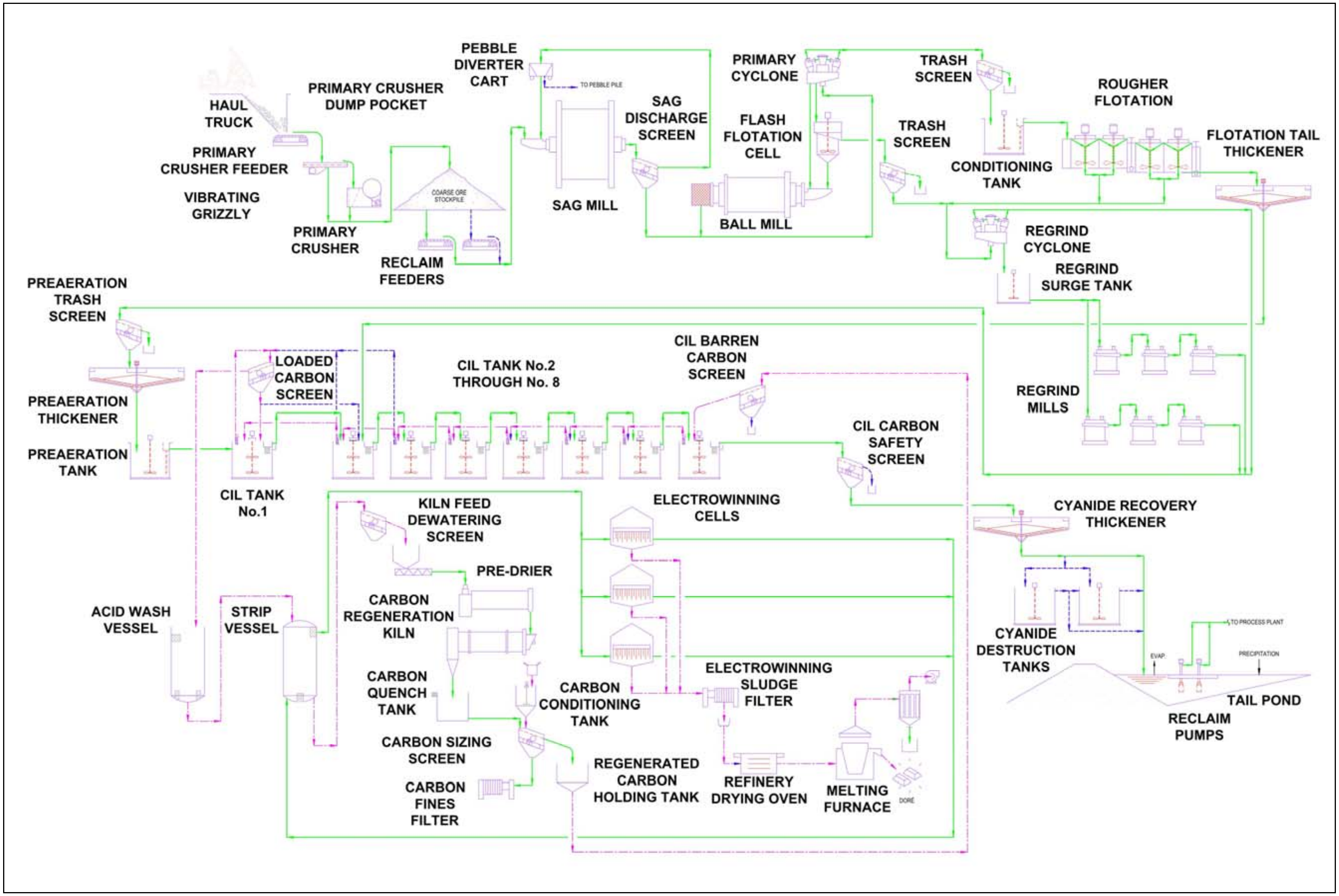
Mine Salaried and Supervisory Staff Labor Requirements																										
JOB TITLE	PPQ1	PPQ2	PPQ3	PPQ4	PPQ5	PPQ6	Y1Q1	Y1Q2	Y1Q3	Y1Q4	Y2Q1	Y2Q2	Y2Q3	Y2Q4	3	4	5	6	7	8	9	10	11	12	13	14
Mine Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Total	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	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.

- 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 a coarse-ore 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. Reagents which require storage and distribution include: sodium cyanide, caustic soda, flocculant, copper sulfate, ammonium bisulfite, hydrochloric acid, lime, antiscalant, UNR 811 A, sulfuric acid, Aero 404, potassium amyl xanthate (PAX), MIBC, and lead nitrate.

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, the smaller flash flotation cell could be replaced with a larger cell, 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.

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 self-priming centrifugal pumps placed on an access ramp in the south east corner of the facility. Process water will recycled back to the Mill for makeup water.

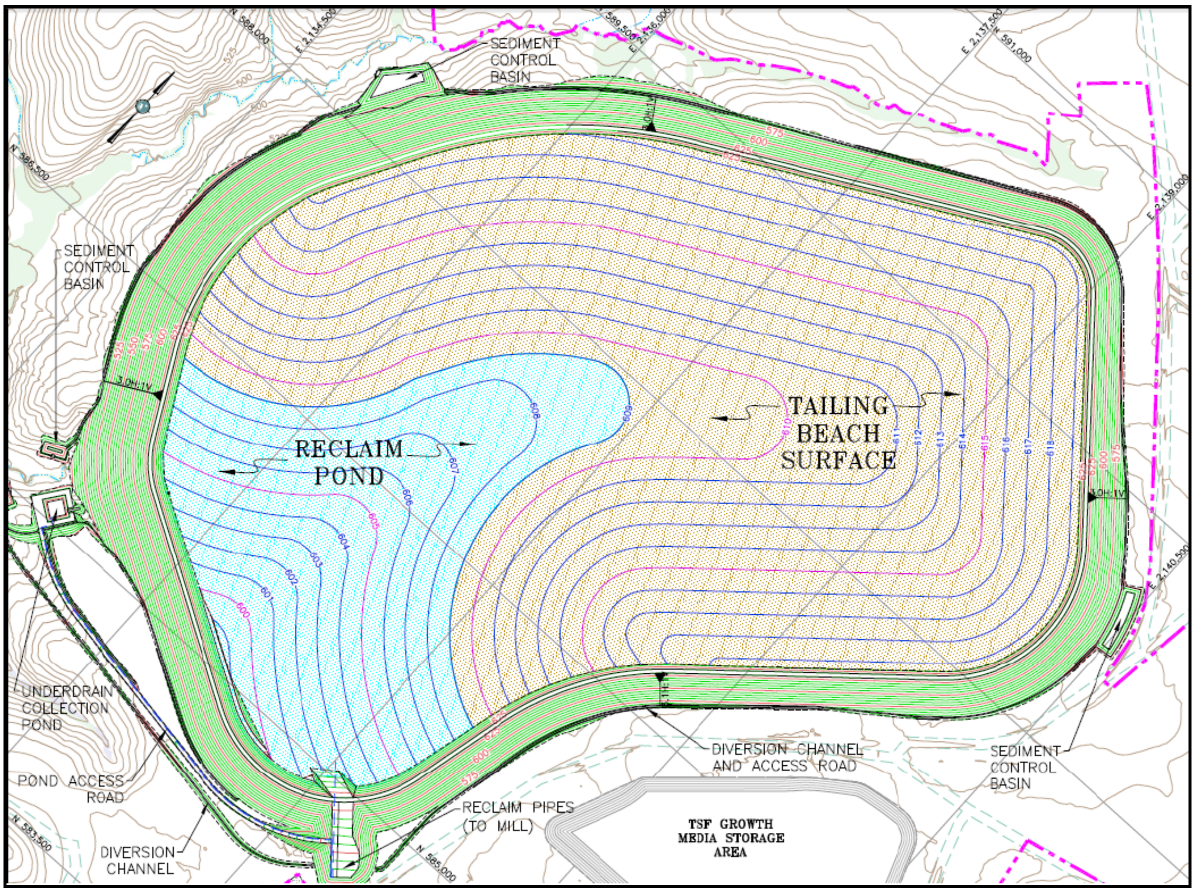


Figure 18-1: Tailing Storage Facility Layout

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.

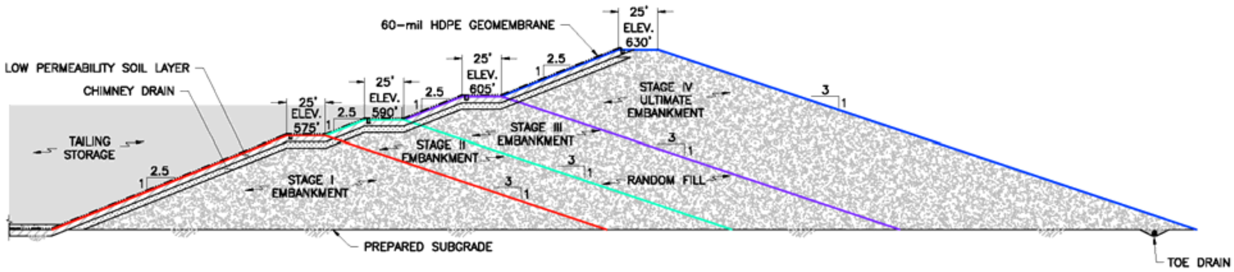


Figure 18-2: Tailing Storage Facility Typical Section

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.

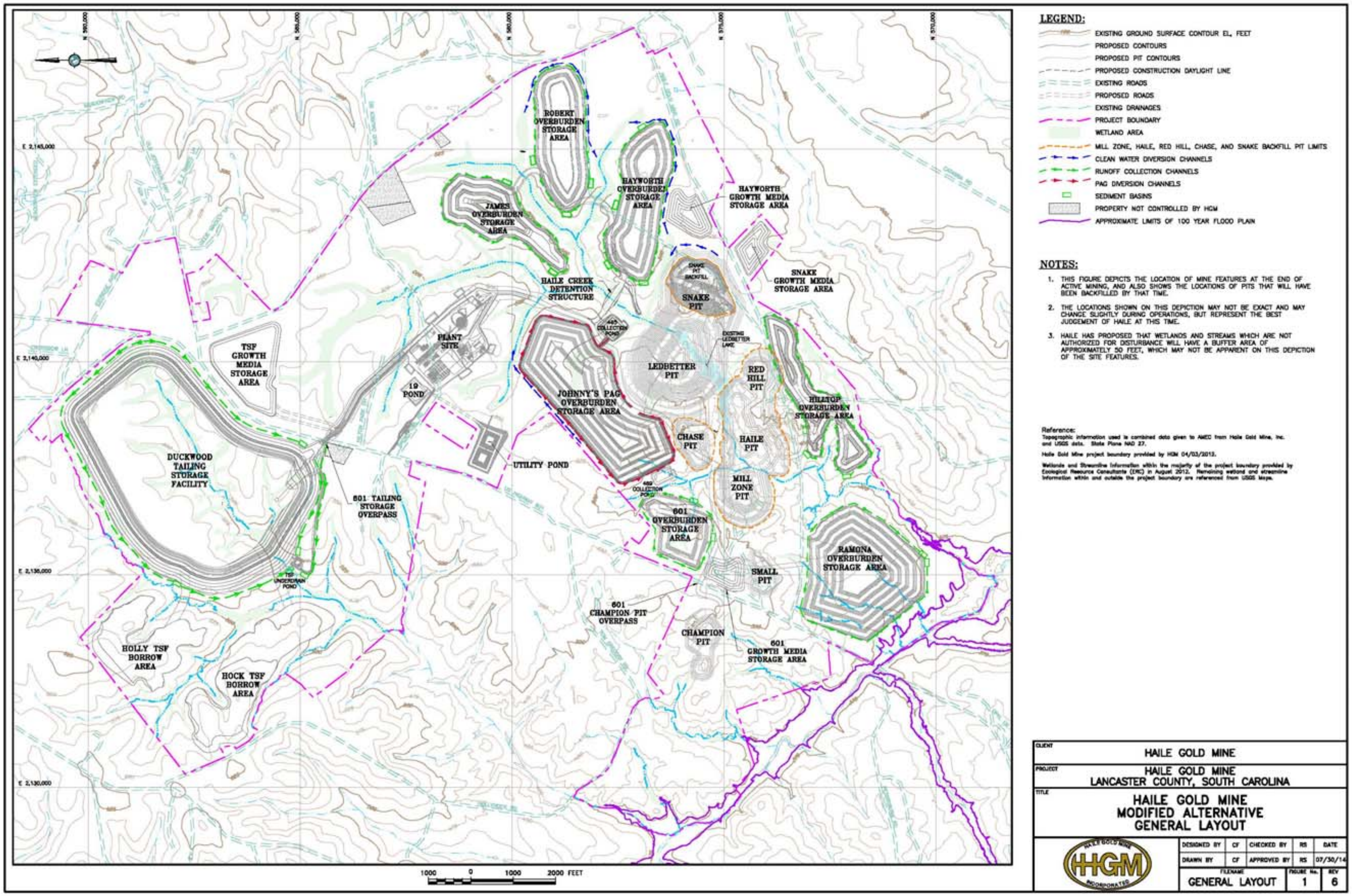


Figure 18-3: Overburden Storage Areas Plan

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.

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.

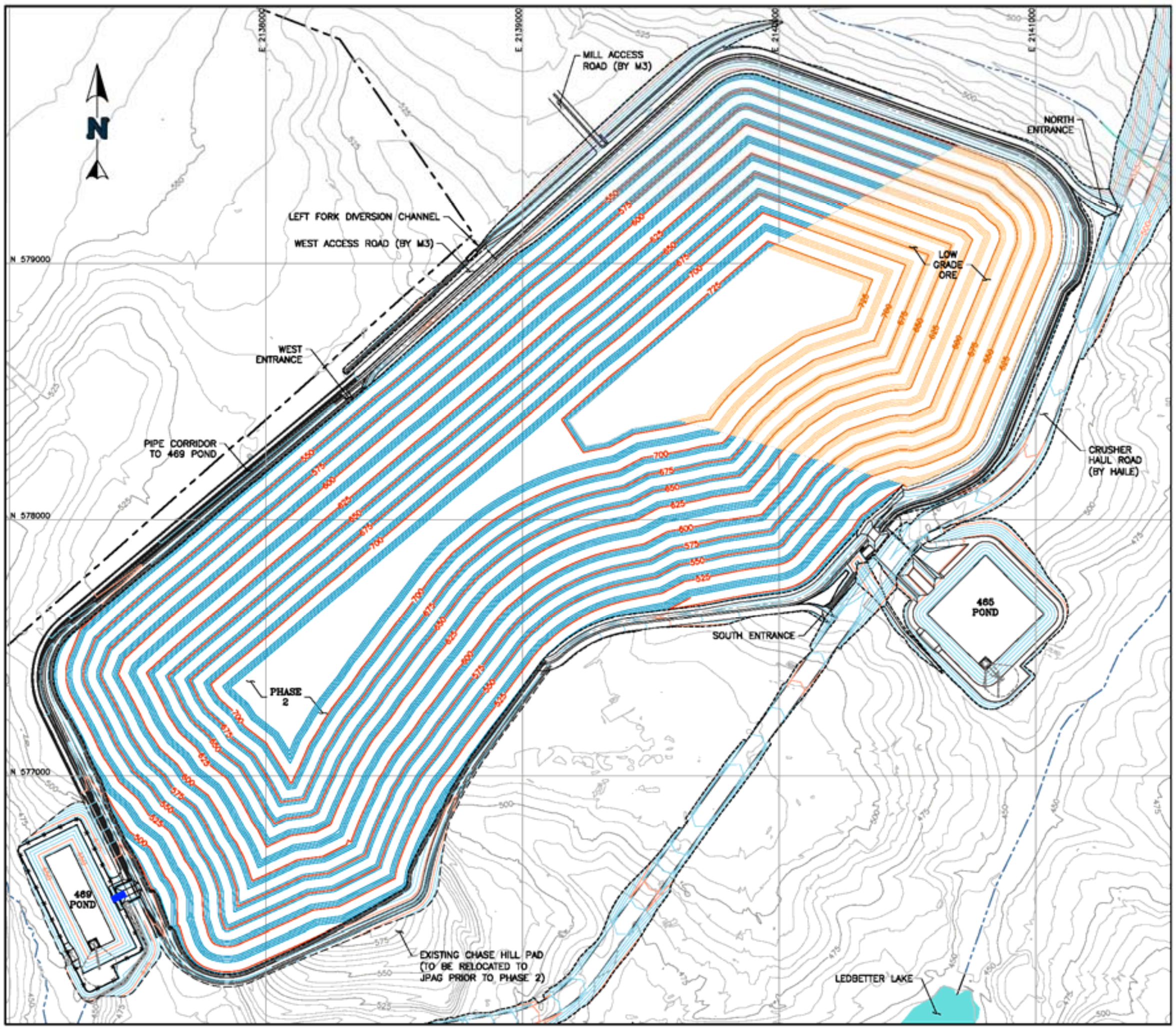


Figure 18-4: Johnny's PAG OSA Area Plan

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.

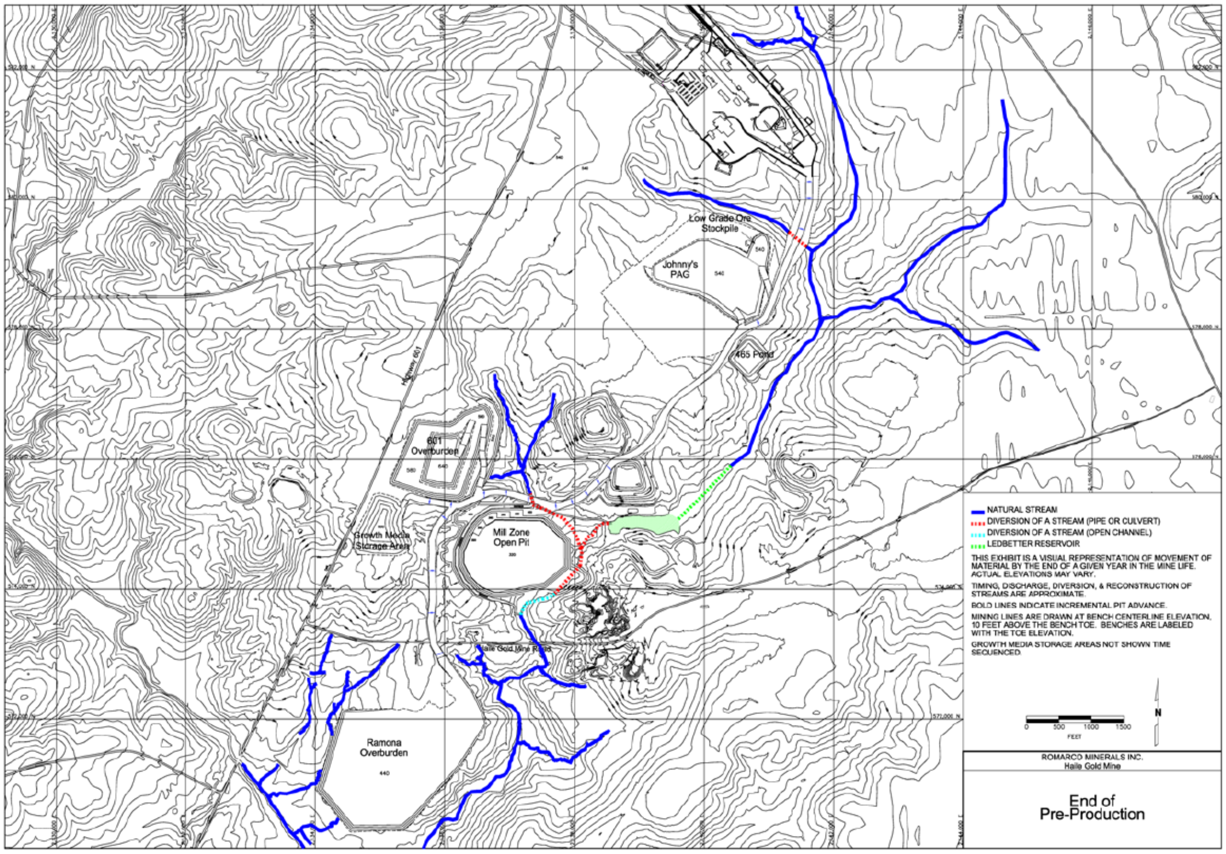


Figure 18-5: Preproduction Site Wide Water Management Plan

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 Pit 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 six (6) years and the successful completion of an Environmental Impact Statement by the Corps of Engineers have resulted in obtaining all but one 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 stormwater). 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.

Agency	Permit/Authorization Number	Description
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).
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 Stormwater 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

Agency	Permit/Authorization Number	Description
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
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.

All permits necessary to start construction have been received. Remaining permits to be obtained include:

Lancaster County

- Lancaster County Building Permit(s)
- Road Closure Permit(s)

SC Department of Transportation

- Highway Encroachment or Re-Routing Permit(s)

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	37.7
Mine	Mine Equipment and Preproduction, Mine Dewatering, Overburden Stockpiles	91.4
Process Facilities	Primary Crushing, Grinding, Flotation, Cyanide Leach, Carbon Handling and Refinery	77.8
Tailing	Tailing Thickening, Detox, Tailing Starter Dam Civil, Highway Overpass	50.3
Indirect Costs	Freight, Mobilization, EPCM, Vendor Commissioning and Spare Parts	40.8
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	20.1
Contingency	Calculated based on each sub area of the project.	15.0
Escalation	Not included in this estimate	-
Total		333.1

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 21-2.

Table 21-2: 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 21-3 below. Labor costs were developed based on a staffing plan and rate schedule.

Table 21-3: 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 21-4).

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 21-4 are payments for equipment that has already been delivered. The mine capital cost table does not include contingency. 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 21-6 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 21-4: 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	32,197																						
Blast Hole Drill	910	60,000					1	910			2	1,820														
Hitachi 14.4 cu m Hyd. Shovel	1,400	80,000	1	2,662																						
Hitachi Bucket wear Package	20		1	20																						
Cat 992K Wheel Loader	2,268	30,000																								
Cat 777G Haul Truck	1,819	55,000																								
Cat D9T Track Dozer	1,081	30,000																								
Cat D10T Track Dozer	1,453	30,000																								
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	143	20,000					1	143																		
Subtotal Major Equipment				34,879				1,053				1,820				557								1,148		
MINE SUPPORT EQUIPMENT:		Years																								
Fuel/Lube Truck (4,000 gal)	887	6																								
Flatbed Truck (8 - 10 ton)	110	6	1	110																						
Crane Truck (8 - 10 ton)	65	6	1	65																						
Rough Terrain Crane (40 ton)	366	12							1	366																
Mechanics Truck	110	6	1	110																						
Welding Truck	168	6							1	168																
Mechanics Truck and Shop Equip.	448	6							1	448																
Tractor & Lowboy (75 T)	1,152	12															1	1,152								
Shop Forklift (Hyster H100XM)	40	6	1	40																						
RT Forklift (Sellick SD-100)	120	6							1	120																
Fire Suppresion systems mobile				82		73		94				33														
Man Van	38	6	2	76	1	38	1	38																		
Pickup Truck (4x4)	38	4	9	342	1	38			1	38																
Light Plants	20	4	2	40	1	20																				
Mine Communications Network	50	12	1	50																						
Mine Radios	1	12	30	30																						
Mine Dewatering	0	12		465		465		488		534		713														
Spare Shovel Bucket	365																						1	365		
Lime Silo & Dispensing System	200	12																					1	200		
Temporary Maintenance Shop	200		1	200																						
Temporary Fueling Facility	150		1	150																						
Subtotal Mine Support Equipment				1,760		634		620		1,674		746						1,152						565		
Engineering/Geology Equipment	150	6	1	150																						
Operator Training Program	396						1	396																		
Shop Tools				525						1069																
Contingency (0%)																										
TOTAL EQUIPMENT/FACILITIES CAPITAL				37,314		634		2,069		2,743		2,566				557		1,152						1,713		

Table 21-5: 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																										32,197
Blast Hole Drill	910	60,000									2	1,820															4,550
Hitachi 14.4 cu m Hyd. Shovel	1,400	80,000																									2,662
Hitachi Bucket wear Package	20																										20
Cat 992K Wheel Loader	2,268	30,000							1	2,268					2	4,536											6,804
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																	1,453
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																									143
Subtotal Major Equipment								3,638		12,816		2,985		9,386		5,093		2,719									76,094
MINE SUPPORT EQUIPMENT:		Years																									
Fuel/Lube Truck (4,000 gal)	887	6											1	887													887
Flatbed Truck (8 - 10 ton)	110	6											2	220													330
Crane Truck (8 - 10 ton)	65	6											1	65													130
Rough Terrain Crane (40 ton)	366	12																									366
Mechanics Truck	110	6											2	220													330
Welding Truck	168	6											1	168													336
Mechanics Truck and Shop Equip.	448	6																									448
Tractor & Lowboy (75 T)	1,152	12																									1,152
Shop Forklift (Hyster H100XM)	40	6											1	40													80
RT Forklift (Sellick SD-100)	120	6											1	120													240
Fire Suppresion systems mobile																											282
Man Van	38	6							4	152							4	152									456
Pickup Truck (4x4)	38	4							8	304							8	304							6	228	1,254
Light Plants	20	4					1	20																			80
Mine Communications Network	50	12																									50
Mine Radios	1	12							30	30							30	30									90
Mine Dewatering	0	12						408		3,399		117		897		215		31		31		486		35			8,284
Spare Shovel Bucket	365																										365
Lime Silo & Dispensing System	200	12																									200
Temporary Maintenance Shop	200																										200
Temporary Fueling Facility	150																										150
Subtotal Mine Support Equipment								428		3,885		117		2,617		215		517		31		486		35		228	15,710
Engineering/Geology Equipment	150	6											1	150													300
Operator Training Program	396																										396
Shop Tools																											1,594
Contingency (0%)																											
TOTAL EQUIPMENT/FACILITIES CAPITAL								4,066		16,701		3,102		12,153		5,308		3,236		31		486		35		228	94,094

Table 21-6: 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	37,314		1,436	37,314	1,436	38,750
PP Q2	634		1,493	634	1,493	2,127
PP Q3	2,069		3,334	2,069	3,334	5,403
PP Q4	2,743		4,585	2,743	4,585	7,328
PP Q5	2,566		7,102	2,566	7,102	9,668
PP Q6			8,140		8,140	8,140
Yr1 Q1		557		557	7,754	8,311
Yr1 Q2		1,152		1,152	7,190	8,342
Yr1 Q3					7,183	7,183
Yr1 Q4					7,157	7,157
Yr2 Q1		1,713		1,713	7,089	8,802
Yr2 Q2					7,455	7,455
Yr2 Q3					7,500	7,500
Yr2 Q4					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					6,523	6,523
14					1,938	1,938
TOTAL	45,326	48,768	26,090	94,094	425,479	519,573

(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 21-7.

Table 21-7: 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 21-8 shows a summary of these costs.

Table 21-8: 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 $329.2 million at a discount rate of 5%. This results in an IRR of 20.1% and a payback period of 3.9 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, but not included in the economic analysis.

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% ($000)	NPV @ 5% ($000)	IRR	Payback
Base Case		$596,585	$329,223	20.1%	3.9
Gold Price	+20%	$935,425	$572,776	30.0%	2.7
	-20%	$242,733	$74,899	8.7%	7.7
Operating Cost	+20%	$457,072	$231,462	16.1%	4.6
	-20%	$728,857	$421,885	23.7%	3.3
Gold Recovery	+5%	$698,942	$402,607	23.2%	3.4
	-5%	$492,917	$254,952	17.0%	4.5
Gold Grade	+20%	$960,594	$590,553	30.7%	2.6
	-20%	$219,353	$58,372	7.9%	8.1
Silver Price	+20%	$603,456	$334,147	20.3%	3.8
	-20%	$589,680	$324,275	19.9%	3.9
Capital Cost	+20%	$541,035	$274,363	16.1%	4.7
	-20%	$651,041	$383,315	25.7%	3.0
Silver Grade	+100%	$628,132	$351,887	21.1%	3.7
	-100%	$564,696	$306,326	19.2%	4.1

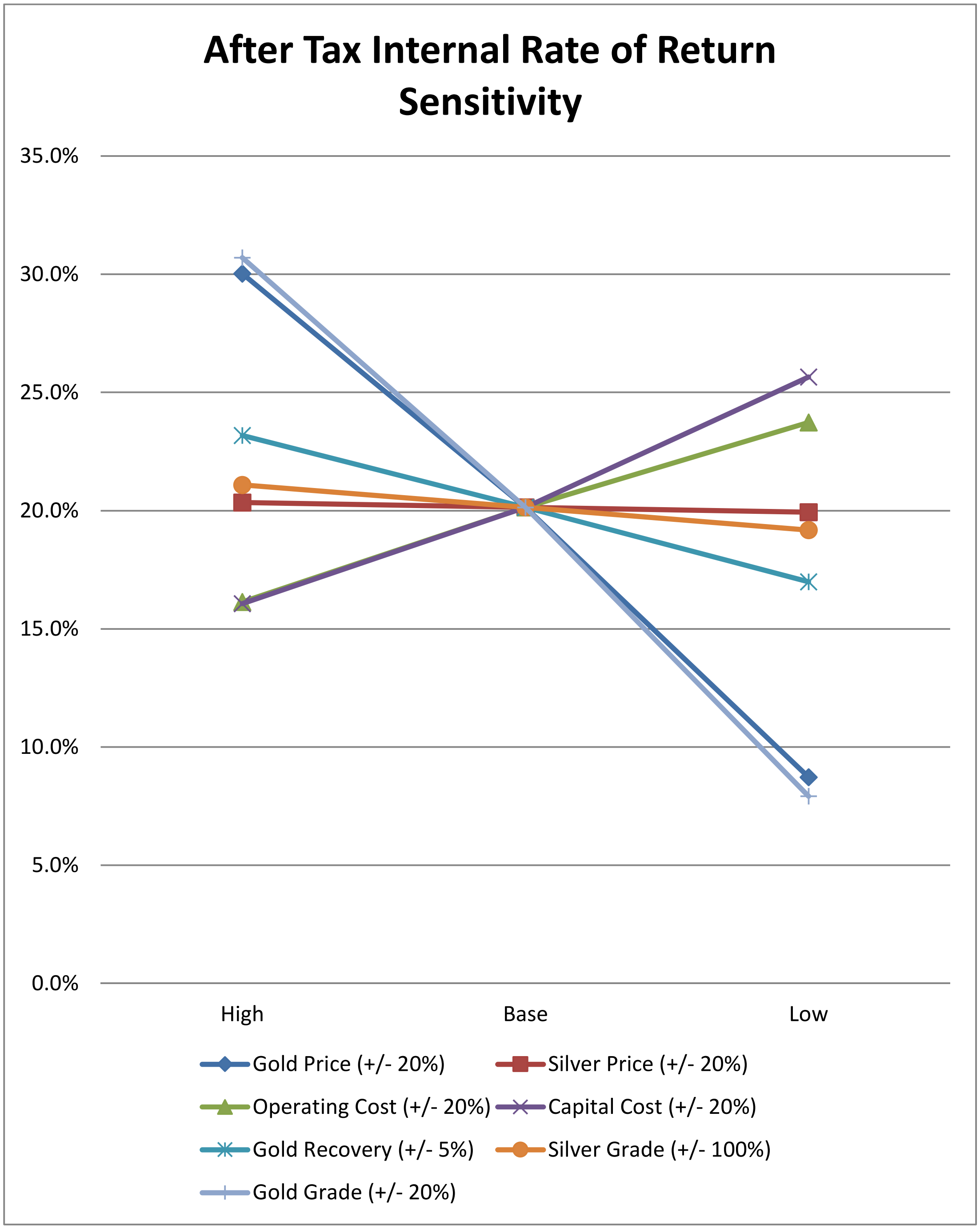


Figure 22-1: Sensitivity Analysis

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

	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
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	-	-	-	-	-
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%	0.00%	0.00%	0.00%	0.00%	0.00%
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%	0.00%	0.00%	0.00%	0.00%	0.00%
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	$1,250.00	$1,250.00	$1,250.00	$1,250.00	$0.00
Silver ($/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	$20.00	$20.00	$20.00	$20.00	$0.00
Revenues																						
Gold Revenue ($ 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	$0	$0	$0	$0	$0
Silver Revenue ($ 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	$0	$0	$0	$0	$0
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	$0	$0	$0	$0	$0

	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
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	$0	$0	$0	$0	$0
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	$0	$0	$0	$0	$0
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	$0	$0	$0	$0	$0
Treatment & Refining Charges																						
Dore'																						
Treatment Charges	$2,581			$262	$205	$248	$235	$201	$204	$224	$207	$245	$241	$167	$66	$63	$14	$0	$0	$0	$0	$0
Gold Refining Charges	$1,049			$107	$83	$102	$96	$81	$83	$91	$84	$100	$99	$67	$25	$24	$5	$0	$0	$0	$0	$0
Silver Refining Charges	$421			$42	$33	$40	$38	$33	$33	$36	$34	$40	$39	$27	$11	$11	$2	$0	$0	$0	$0	$0
Transportation	$2,075			$210	$165	$200	$189	$161	$164	$180	$167	$197	$194	$134	$53	$51	$11	$0	$0	$0	$0	$0
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	$0	$0	$0	$0	$0
Mine Development	$18,263	$2,050	$16,213	$0	$0	$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	$0	$0	$0	$0	$0
Salvage Value	-$4,575				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$3,431	$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	$9,145	$1,135	$3,861	-$2,018	$457
Total Production Cost	$980,630	$30,797	$57,193	$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	$5,714	$1,135	$3,861	-$2,018	$457
Operating Income	$1,157,942	$(30,797)	$ (57,193)	$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	$ (5,714)	$ (1,135)	$ (3,861)	$2,018	$ (457)
Initial Capital Depreciation	$307,314			$43,915	$75,261	$53,749	$38,384	$27,443	$27,412	$27,443	$13,706	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine Development	$7,827			$1,565	$1,565	$1,565	$1,565	$1,565	$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	$1,374	$191	$106	$52	$10
Total Depreciation	$453,682	$0	$0	$46,578	$81,775	$63,076	$51,289	$43,303	$41,116	$42,278	$31,040	$16,519	$12,341	$8,694	$5,872	$4,739	$3,326	$1,374	$191	$106	$52	$10
Net Income After Depreciation	$704,260	-$30,797	-$57,193	$104,720	$18,009	$66,397	$60,926	$41,290	$39,707	$52,046	$57,514	$130,678	$133,044	$74,699	-$549	$566	-$2,930	-$7,088	-$1,327	-$3,967	$1,966	-$467
Income Taxes	$135,431		$0	$1,771	$511	$6,898	$11,859	$6,293	$5,975	$9,747	$9,689	$27,852	$36,087	$18,750	$0	$0	$0	$0	$0	$0	$0	$0
Net Income After Taxes	$568,829	-$30,797	-$57,193	$102,949	$17,499	$59,499	$49,067	$34,997	$33,732	$42,299	$47,825	$102,826	$96,957	$55,949	-$549	$566	-$2,930	-$7,088	-$1,327	-$3,967	$1,966	-$467
Cash Flow																						
Operating Income	$1,157,942	-$30,797	-$57,193	$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	-$5,714	-$1,135	-$3,861	$2,018	-$457
Add Back Cost Depletion	$0			$0	$0	$0	$0	$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	$908	$0	$0	$0	$0
Accounts Payable (30 days)	$0	$0	$0	$5,016	$179	$364	$254	$557	$548	$85	-$603	-$1,923	-$131	-$90	-$601	-$271	-$2,585	-$799	$0	$0	$0	$0
Inventory - Parts, Supplies	$0	$0	-$2,400	-$6,000	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$8,400	$0	$0	$0	$0	$0
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	$109	$0	$0	$0	$0
Capital Expenditures																						
Initial Capital																						
Mine	$45,326	$37,948	$7,378	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Process Plant	$241,884	$12,094	$217,696	$12,094	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owners Cost	$20,104	$1,005	$18,094	$1,005	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Land Acquisition	$3,045	$3,045	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Mine Development	$7,827	$878	$6,949	$0	$0	$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	$0	$0	$0	$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	$0	$0	$0	$0	$0
Total Capital Expenditures	$425,926	$24,170	$250,116	$20,782	$21,458	$8,137	$32,436	$10,927	$12,153	$17,981	$24,905	$1,431	$486	$716	$228	$0	$0	$0	$0	$0	$0	$0
Cash Flow before Taxes	$732,016	-$54,968	-$309,709	$111,529	$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	-$5,605	-$1,135	-$3,861	$2,018	-$457
Cumulative Cash Flow before Taxes		-$54,968	-$364,676	-$253,147	-$170,586	-$51,955	$29,044	$105,671	$174,649	$249,636	$313,884	$455,079	$600,088	$687,949	$699,447	$704,702	$714,040	$708,436	$707,301	$703,440	$705,458	$705,001
				1.0	1.0	1.0	0.6	-	-	-	-	-	-	-	-	-	-					
Taxes																						
Income Taxes	$135,431	$0	$0	$1,771	$511	$6,898	$11,859	$6,293	$5,975	$9,747	$9,689	$27,852	$36,087	$18,750	$0	$0	$0	$0	$0	$0	$0	$0
Cash Flow after Taxes	$596,585	-$54,968	-$309,709	$109,758	$82,050	$111,733	$69,140	$70,334	$63,003	$65,240	$54,559	$113,343	$108,923	$69,111	$11,497	$5,255	$9,338	-$5,605	-$1,135	-$3,861	$2,018	-$457
Cumulative Cash Flow after Taxes		-$54,968	-$364,676	-$254,919	-$172,868	-$61,135	$8,005	$78,339	$141,342	$206,582	$261,141	$374,485	$483,408	$552,519	$564,016	$569,271	$578,610	$573,005	$571,870	$568,009	$570,027	$569,571
				1.0	1.0	1.0	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Economic Indicators before Taxes																						
NPV @ 0%	0%	$732,016																				
NPV @ 5%	5%	$416,660																				
NPV @ 10%	10%	$229,192																				
IRR		22.6%																				
Payback	Years	3.6																				
		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
	-$294,961	-$54,968	-$294,961	$101,160	$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	-$2,568	-$495	-$1,604	$799	-$172
Benefit Cost Ratio @ 5%	2.4																					
Economic Indicators after Taxes																						
NPV @ 0%	0%	$596,585																				
NPV @ 5%	5%	$329,223																				
NPV @ 10%	10%	$170,634																				
IRR		20.1%																				
Payback	Years	3.9																				
		0	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Maximum Risk	-$294,961	-$54,968	-$294,961	$99,553	$70,878	$91,923	$54,173	$52,485	$44,775	$44,157	$35,169	$69,583	$63,685	$38,484	$6,097	$2,654	$4,492	-$2,568	-$495	-$1,604	$799	-$172
Benefit Cost Ratio @ 5%	2.1																					

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

There is no other relevant data supplied in this Report.

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 and most 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

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Tailing and Process Water Management Bankable Feasibility Study, AMEC Earth and Environmental, November 19, 2010.

APPENDIX A – FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

I, Joshua W. Snider, PE, do hereby certify that:

1. I am employed as an engineer at M3 Engineering & Technology Corporation ("M3 Engineering"), 2051 W Sunset Rd, Suite 101, Tucson, AZ 85704, USA.
2. I am a graduate of the University of Arizona and received a Bachelor of Civil Engineering in 1996.
3. I am a registered professional engineer in the State of Arizona (No. 41971) and the State of South Carolina (No. 30439).
4. I have practiced engineering and project management at M3 Engineering for 18 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, 22, 23, 25, 26 and 27 of the technical report entitled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina," dated effective as of October 13, 2015 (the "Technical Report"). I visited the property in September of 2015.
7. I have had prior involvement with the property that is the subject of the Technical Report. I participated in the preparation of the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Feasibility Study, Lancaster County, South Carolina", dated February 10, 2011.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of OceanaGold Corporation (the "Company") as defined by Section 1.5 of NI 43-101 and do not own any shares or stocks of the Company.
10. I have read NI 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 for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 13th day of October 2015.

(Signed) "*Joshua W. Snider*"
Signature of Qualified Person

Joshua W. Snider, PE
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 practiced design and engineering at M3 Engineering & Technology Corporation for six 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,” dated effective 13 October 2015 (the "Technical Report").

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.

8. I last visited the property in July 2015.

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 13th day of October, 2015.

/s/ Erin L. Patterson
Signature of Qualified Person

Erin L. Patterson, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Lee P. Gochnour, do hereby certify that:

1. I am the principal of:

   Gochnour & Associates, Inc.
   P.O. Box 4430
   Parker, CO 80134

2. I graduated with a degree in Park Administration and Environmental Land Use Planning from the Eastern Washington University in 1981.

3. I am recognized as a Qualified Professional (QP) Member with special expertise in Environmental, Permitting and Compliance with the Mining and Metallurgical Society of America (Member # 01166QP), a Member of the Society of Mining, Metallurgy and Exploration and Past President and Life Member of the Northwest Mining Association.

4. I have worked as an environmental professional in the mining industry for a total of 34 since my graduation from university.

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 Environmental and Permitting section of the technical report titled "NI 43-101, Technical Report", dated October 13, 2015 relating to the Haile Gold Mine Project property. I have visited the Haile Gold Mine property on numerous occasions with my most recent visit occurring on 11/30/2012.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is to perform due diligence and advise on environmental and permitting matters associated with the property.

8. To the best of my knowledge, information and belief, the technical report contains all scientific and technical information required to be disclosed to make the report not misleading.

9. I am independent of the issuer applying all of the tests in section 1.4 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 for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 13th day of October, 2015.

Lee P. Gochnour

Signature of Qualified Person

Lee P. 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 38 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 and I contributed to sections 1, 21, 25, 26, 27 of the Technical Report titled "Haile Gold Mine Project, NI43-101 Technical Report, Project Update, Lancaster County, South Carolina", dated effective October 13, 2015.

7. I visited the Haile Gold property on November 4 – 6, 2009, June 20, 2012, and June 2-3 2015 during which times I reviewed the drill core, core handling procedures, sample preparation, core logging mining plans 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 between late 2009 and mid-

2015. I was a co-author of four technical reports regarding the Haile project dated 7 December 2010, 10 February 2011, and 13 March 2012, 21 November 2014.

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 1.4 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: 13 October 2015.

SME
Society for
Mining, Metallurgy
& Exploration
John M. Marek
SME Registered Member No. 2021600
Signature
Date Signed 13 Oct 2015
Expiration date 31 Dec 2015

John M. Marek
Registered Member of the SME

CERTIFICATE OF QUALIFIED PERSON

I, Carl J. Burkhalter, PE, do hereby certify that:

1. I am employed as an engineer at NewFields Mining & Technical Services LLC ("NewFields"), 9400 Station Street, Suite 300, Lone Tree, CO 80124, USA.
2. I am a graduate of the University of Wisconsin- Madison and received a Bachelor of Science in Mining Engineering in 1984 and a Master of Science in Civil and Environmental (Geotechnical) Engineering in 1987.
3. I am a registered professional engineer in the State of Colorado (No. 29447), State of Arizona (No. 34925), State of Nevada (No. 21178) and the State of South Carolina (No. 31243).
4. I have practiced engineering and project management at several engineering consulting firms and the State of Arizona for 29 years, the last year at NewFields.
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 portions of Sections 1, 18, 25, 26 and 27 of the technical report entitled "Haile Gold Mine Project NI 43-101 Technical Report, Project Update, Lancaster County, South Carolina," dated effective as of October 13, 2015 (the "Technical Report"). I visited the property in May of 2014.
7. I have had prior involvement under my previous firm with the property that is the subject of the Technical Report. I participated in the preparation of the technical report titled "Haile Gold Mine Project, NI 43-101 Technical Report, Feasibility Study, Lancaster County, South Carolina", dated February 10, 2011.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of Haile Gold Mine, Inc. (the "Company") as defined by Section 1.5 of NI 43-101 and do not own any shares or stocks of the Company.
10. I have read NI 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 for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 13th day of October, 2015.

(Signed)
Signature of Qualified Person

Carl J. Burkhalter, PE
Print Name of Qualified Person