MEDIA RELEASE

19 October 2015

OCEANAGOLD FILES NI43-101 TECHNICAL REPORT AND JORC TABLE 1 FOR HAILE GOLD MINE

(MELBOURNE) Following the completion of the Romarco acquisition, OceanaGold Corporation **(TSX/ASX/NZX: OGC)** ("OceanaGold" or the "Company") has released the Haile Gold Mine National Instrument 43-101 Technical Report ("Technical Report") under OceanaGold ownership. The Technical Report restates the resources and reserves of Haile as previously reported by Romarco Minerals in November 2014.

The Haile combined open pit and underground Resources were calculated with a US$1,200 / oz price assumption and are inclusive of reserves. As at 21 November 2014, the Measured and Indicated Resources were estimated as 71.2 Mt at 1.77 g/t Au, containing 4,039 koz of gold. Inferred Resources total 20.1 Mt at 1.24 g/t Au, containing 801 koz of gold.

The open pit reserves were calculated with a US$950 / oz price assumption. Proved and Probable Reserves, as at 21 November 2014, were estimated as 30.5 Mt at 2.06 g/t Au containing 2,018 koz of gold. There are currently no underground reserves.

Note that the resources and reserves have been converted to metric tonnes and grades.

In addition, the Company has released the JORC Table 1 pursuant to its obligations under the ASX listing rules. The Technical Report and JORC Table 1 have been filed with the stock exchanges and are also available on the Company's website www.oceanagold.com

Mick Wilkes, Managing Director and CEO for OceanaGold stated, "Over the course of the next several months, OceanaGold will undertake a comprehensive exploration program that will feed into an optimisation study which will further demonstrate the robustness of the Haile asset." He added, " There are currently two drills operating on regional targets and we are now assembling an exploration plan and budget for the next 15 months for the Haile deposit particularly testing and infilling depth extensions in addition to other regional targets within the Carolina Slate belt."

He went on to say, "Earthworks at Haile are advancing well and the team at site continues to ramp up activities. The Company expects to pour the concrete foundations for the Ball and SAG mills prior to the end of the year."

- ENDS -

Technical Disclosure

The estimates of Mineral Resources and Reserves were prepared in accordance with the standards set out in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and

Ore Reserves' ("JORC Code") and in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators ("NI 43-101"). The JORC Code is the accepted reporting standard for the Australian Stock Exchange Limited ("ASX") and the New Zealand Stock Exchange Limited ("NZX").

Information relating to Mineral Resources and Ore Reserves in this document is based on information compiled by or prepared under the supervision of John Marek, a Competent Person who is a Registered Member of the Society for Mining Metallurgy & Exploration. John Marek is an employee (and President) of Independent Mining Consultants, Inc. (IMC). John Marek and IMC are independent of OceanaGold Corporation, Romarco Minerals Inc. and/or the Haile Gold Mine Project. John Marek has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves' ("JORC Code"). John Marek consents to the inclusion in the report of the matter based on his information in the form and context in which it is reported.

The Technical Report was also prepared under the supervision of Josh Snider, Erin L. Patterson, Lee "Pat" Gochnour, John Marek and Carl Burkhalter. Each is a Qualified Person for the purposes of the NI 43-101. Messrs Snider and Patterson are registered professional engineers and full time employees of M3 Engineering & Technology Corporation. Lee "Pat" Gochnour is a member of Mining and Metallurgical Society of America and a full time employee of Gochnour & Associates, Inc. Carl Burkhalter is a registered member of SME and a full time employee of NewFields Mining Design & Technical Services LLC. Carl is also a Registered Professional Civil Engineer. The Qualified Persons have reviewed the information contained in this press release and consent to the inclusion in the report of the matter based in the form and context in which they are reported.

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About OceanaGold

OceanaGold Corporation is a significant multinational gold producer with assets located in New Zealand, the Philippines and the United States. The Company's assets encompass its flagship operation, the Didipio Gold-Copper Mine located on the island of Luzon in the Philippines. On the south island of New Zealand, the Company operates the largest gold mine in the country at the Macraes Goldfield which is made up of the Coronation open pit and the Frasers underground mine. On the west coast of the South Island, the Company operates the Reefton Gold Mine. On the north island of New Zealand, the Company acquired the

Waihi Gold Mine; the transaction has now received regulatory approval and is expected to close on 30 October 2015. In South Carolina, United States, the Company acquired the Haile Gold Mine through its acquisition of Romarco Minerals Inc. in September 2015. The Haile Gold Mine is a top-tier asset currently in construction and is expected to operate commercially early in 2017. OceanaGold has a pipeline of organic growth and exploration opportunities in the Australasia and Americas regions.

OceanaGold has operated sustainably over the past 25 years with a proven track record for environmental management and community and social engagement. The Company has a strong social license to operate and works collaboratively with its valued stakeholders to identify and invest in social programs that are designed to build capacity and not dependency.

In 2015, the Company expects to produce and attribute 380,000 to 410,000 ounces of gold from the combined New Zealand and Didipio operations and produce 22,000 to 23,500 tonnes of copper from the Didipio operation.

Cautionary Statement for Public Release

Certain information contained in this public release may be deemed “forward-looking” within the meaning of applicable securities laws. Forward-looking statements and information relate to future performance and reflect the Company’s expectations regarding the generation of free cash flow, execution of business strategy, future growth, future production, estimated costs, results of operations, business prospects and opportunities of OceanaGold Corporation and its related subsidiaries. Any statements that express or involve discussions with respect to predictions, expectations, beliefs, plans, projections, objectives, assumptions or future events or performance (often, but not always, using words or phrases such as "expects" or "does not expect", "is expected", "anticipates" or "does not anticipate", "plans", "estimates" or "intends", or stating that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved) are not statements of historical fact and may be forward-looking statements. Forward-looking statements are subject to a variety of risks and uncertainties which could cause actual events or results to differ materially from those expressed in the forward-looking statements and information. They include, among others, the accuracy of mineral reserve and resource estimates and related assumptions, inherent operating risks and those risk factors identified in the Company’s most recent Annual Information Form prepared and filed with securities regulators which is available on SEDAR at www.sedar.com under the Company’s name. There are no assurances the Company can fulfil forward-looking statements and information. Such forward-looking statements and information are only predictions based on current information available to management as of the date that such predictions are made; actual events or results may differ materially as a result of risks facing the Company, some of which are beyond the Company's control. Although the Company believes that any forward-looking statements and information contained in this press release is based on reasonable assumptions, readers cannot be assured that actual outcomes or results will be consistent with such statements. Accordingly, readers should not place undue reliance on forward-looking statements and information. The Company expressly disclaims any intention or obligation to update or revise any forward-looking statements and information, whether as a result of new information, events or otherwise, except as required by applicable securities laws. The information contained in this release is not investment or financial product advice.

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Material Summary - Haile Gold Mine Project

The Haile project property site is located 4.8 km northeast of the town of Kershaw in southern Lancaster County, South Carolina Lancaster County lies in the north-central part of the state. The HGM property site is approximately 27.4 km southeast of the city of Lancaster, the county seat, which is approximately 48.3 km south of Charlotte, North Carolina. It is also approximately 80.5 km northeast of Columbia, South Carolina.

The Haile Gold Mine will utilize conventional open pit hard rock mining methods to deliver 6,350 tonnes per day (tpd) of sulphide ore to a process facility consisting of grinding, flotation, cyanide leach, carbon handling, and refining.

The project consists of mine development, overburden storage areas, surface water management, process facilities, ancillary buildings, infrastructure and a tailing storage facility. 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 being bordered by US Highway 601 to the west and the Haile Gold Mine Road to the south, both of which are paved. Natural gas, sanitary sewer, and potable water lines run along Highway 601. Power for the Haile property may be provided from Duke Energy and/or Lynches River Electric Cooperative. The power transmission infrastructure is well established, a new 69 kV (Lynches River) services will be required.

The 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.

Following a Plan of Arrangement completed on October 1st, 2015 between Romarco Minerals Inc and OceanaGold Corporation, 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.

Romarco Minerals Inc acquired the Haile property from Kinross and another private party in October of 2007. HGM, a wholly owned subsidiary of Oceana Gold 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.

1.0 Haile Mineral Resources and Ore Reserves

The Mineral Resources at HGM are based on a block model and resource estimate by John Marek of IMC that was published March 13, 2012. The Ore Reserves are based on a block model and mine plan by John Marek of IMC that was published February 10, 2011. That February 2011 Ore Reserve was republished without change within the March 13, 2012 Technical Report. The HGM feasibility study was updated to reflect more current costs and progress toward mine permits on November 21, 2014. The November 21, 2014 report presented the same Mineral Resources and Ore Reserves as the March 13, 2012 report. The impact of the later March 13, 2012 model was evaluated within the reserve volume and the net change to the Ore Reserve was minor. It is the opinion of John Marek (Competent Person) that the impact of the March 13, 2012 on the Ore Reserve was not material.

The Mineral Resources and Ore Reserves within this document and Table 1 have not been changed since the November 21, 2014 Canadian Technical Report. For consistency with that disclosure, the effective date of this Mineral Resource and Ore Reserve is November 21, 2014.

The Mineral Resource is comprised of both open pit and underground ores. The open pit component was developed using conventional block model procedures and a floating cone pit geometry to determine the component of the deposit that has “reasonable prospects of economic extraction” for open pit mining. The underground resource is contained within estimated stope geometries based on open stoping with delayed back fill.

The cut-off grade for open pit resources is 0.41 gm/tonne Au (0.012 oz/ton) and the underground cut-off is 2.74 gm/t Au (0.080 oz/ton)

The November 2014 Haile Mineral Resources and Ore Reserves are presented on Tables 1, and 2 respectively. The Mineral Resources are reported inclusive of the Ore Reserve.

Table 1: Haile Open Pit and Underground Mineral Resources as of 21 November 2014

Open Pit Mineral Resources

Category	Cut-Off	Tonnes (kt)	Grade g/t Au	Contained Gold (koz)
Measured	0.41	36,767	1.78	2,107
Indicated	0.41	33,561	1.68	1,813
Measured + Indicated	**0.41**	**70,329**	**1.73**	**3,920**
Inferred Resource	0.41	19,424	1.13	707

Underground Mineral Resources

Category	Cut-Off	Tonnes (kt)	Grade g/t Au	Contained Gold koz
Measured	2.74	127	4.41	18
Indicated	2.74	716	4.39	101
Measured + Indicated	**2.74**	**843**	**4.39**	**119**
Inferred Resource	2.74	701	4.17	94

Combined Open Pit and Underground Mineral Resources

Category	Tonnes (kt)	Grade g/t Au	Contained Gold koz
Measured	36,894	1.79	2,125
Indicated	34,277	1.74	1,914
Measured + Indicated	**71,171**	**1.77**	**4,039**
Inferred Resource	20,125	1.24	801

A gold price of $1,200 per troy ounce was applied
Mineral Resources are inclusive of Ore Reserves

Ore Reserves for HGM will be produced from open pits and were developed from the block model dated February 10, 2011 and the feasibility mine plan. The Ore Reserve is the total of all Proven and Probable category mineralisation planned for processing during the course of the feasibility mine plan. The block model and determination of the Ore Reserves were completed by IMC, with John Marek, P.E. acting as the Competent Person for the calculation. The Ore Reserves are summarized in Table 2. The Mineral Resources are reported inclusive of Ore Reserves.

Table 2: Haile Ore Reserves as of 21 November 2014

Category	Cut-Off	Tonnes (kt)	Grade g/t Au	Contained Gold (koz)	Recovered Grade g/t Au	Recovered Gold (koz)
Proven	0.48	19,592	2.19	1,382	1.85	1,166
Probable	0.48	10,917	1.81	636	1.47	515
Proven + Probable	**0.48**	**30,509**	**2.06**	**2,018**	**1.71**	**1,682**

Ore Reserve based on gold price of $950 per troy ounce
Mineral Resources are inclusive of Ore Reserves

1.1 Haile Mineral Resources

1.1.1 Material Assumptions for Mineral Resources

The gold mineralisation at the Haile property occurs along a trend of moderate 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 1 km wide (NNW to SSE) and is over 3.4 km long (WSW to ENE). Most of the mineralisation at Haile is restricted to the laminated meta-siltstone of the Richtex Formation. The gold mineralized zones within the laminated metasediments can appear at different stratigraphic levels within the metasediments.

Mineralised zones at Haile can strike (trend) northeast to southwest and east to west. The interpreted dips of the mineralised zones range from 20° at the western end of the property to steeply southeast at the eastern end of the known trend and locally can exceed 100m in width. In several areas, multiple mineralised zones exist.

Mineralisation is hosted within a folded sequence of meta-sediments / meta-volcanics. The majority of mineralisation is hosted within the meta-sediments, so the geometry of folding is an important control. Barren dolerite (diabase) dykes cut across the sequence. This has been addressed in the interpretation and domaining for resource estimation.

The gold mineralisation 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.

1.1.2 Drilling Techniques

Approximately 72% of the drilling used for resource estimation was reverse circulation (RC) with 16 cm bits. The remaining 28% of the drilling was diamond core (DDH) with both HQ and NQ core sizes.

Drill holes were drilled on a variable grid pattern and down hole deviation has resulted in variable spacing within the mineralization. Typical drill spacings are in the range of 30m x 30m to 50m to 50m for Measured and Indicated respectively.

1.1.3 Sampling

Core is split longitudinally either by diamond saw or by putty knife if soft. Core sample intervals are generally 1.52m (5 ft) in length. Sample lengths are typically established irrespective of geologic contacts.

The reverse circulation drilling at Haile typically uses 16 cm drill bits. The RC rigs are equipped with a cyclone and a rotary splitter. Most RC drilling at Haile is in wet conditions. Procedures including addition of flocculent and careful draining of samples were put in place to control the sampling process. RC samples are dried and then split with a riffle splitter to obtain roughly 2.7 kg of material that is sent directly to the sample preparation and assay laboratory.

Drilling completed prior to Romarco's ownership was not well documented, so paired sample analysis between historic data and Romarco's data was used to verify the pre-Romarco data. Paired sample analysis was also used to verify the quality of RC sampling, by comparing against nearby diamond drill sampling.

1.1.4 Estimation Methodology

Block grade estimation used Ordinary Kriging (OK) of 6.1m composites into 7.62mE x 7.62mN x 6.1mRL bench height (25 ft x25 ft and 20 ft bench). The small block size in plan was selected in order to provide a reasonable method of modelling the interpreted geology with particular emphasis on the late barren dykes that cross the Haile deposit. 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. Grade capping was applied by domain. Most rock type boundaries were modelled as hard grade boundaries. Density was assigned by lithology.

1.1.5 Resource Classification

Blocks were coded as Measured, Indicated or Inferred based on the gold grade estimate, the kriged standard deviation and the number of composites used to estimate the block. The classification was completed with two kriging passes. The Mineral Resource estimate appropriately reflects the view of the Competent Person and classification is assigned in accordance with the JORC 2012 guideline.

1.1.6 Cut-off Grade

Open pit Mineral Resources are reported using a cut-off grade of 0.41 g/t Au. Underground Mineral Resources are reported using a cut-off of 2.74 gm/tonne Au (0.080 oz/ton). Cut-off grades reflect the cost and price assumptions.

1.1.7 Mining, Metallurgical, and Other Modifying Factors

Mining and process costs and recoveries for the mineral resource were similar to the parameters established for the Ore Reserve. See sections 1.2.3 and 1.2.4 that follow.

1.2 Haile Ore Reserves

1.2.1 Material Assumptions for Ore Reserves

The Haile open pit Ore Reserve estimate is defined within a pit design which is based on detailed geotechnical design parameters and practical mining considerations. The Ore Reserve is the total of all Proven and Probable class material that is planned for processing within the feasibility study.

There are no underground reserves defined or reported for the project at this time.

1.2.2 Ore Reserve Classification

All of the in-situ Ore Reserves are derived from Measured and Indicated Resources. Inferred mineral resources are not included within the Ore Reserve.

1.2.3 Mining Method

The Haile Gold Mine is planned to be mined on 6.1m benches 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.

1.2.4 Processing method

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 plant will consist of 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.

1.2.5 Cut-off Grade

The cut-off grade used to report the Ore Reserves is derived from the cost of processing ore (including site general and administration costs), metallurgical recoveries, and gold price. A variable cut-off grade equation has been applied to each block in the model. The resulting “recoverable gold grade” is actually used for determination of cut-off grade for the Ore Reserves.

Conversion of the recoverable cut-off grade to in-situ gold grade results in an in-situ cut-off grade for Ore Reserves of 0.48 gm/tonne.

1.2.6 Estimation Methodology

See section 1.1.4 above.

1.2.7 Material Modifying Factors for Ore Reserves

The resource model was assembled so that block grades reflect the production inclusive of dilution and selective mining criteria. No additional factors for mining dilution or recovery need to be applied to the block model.

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.

1.3 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 404 Dredge and Fill Permit, 401 Water Quality Certification, air quality permit. NPDES Permits (wastewater discharge, wastewater treatment system construction, and stormwater). The last remaining permit, the Mine Operating Permit, became final in January of 2015.

1.4 Economic Analysis

The Haile Gold Project economics were analysed 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.

JORC Code, 2012 Edition – Table 1, Haile Gold Mine Project

Section 1 Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* • *Aspects of the determination of mineralisation that are Material to the Public Report.* • *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	Both Reverse Circulation (RC) and Diamond Drilling (DDH) have been used for the resource estimates at Haile. This section describes the sampling procedures applied to both data collection techniques. Historical drilling accounts for approximately 30% of the data. The sample procedures applied to the historic drilling (i.e. drilling prior to Romarco Minerals Inc.) at Haile were not well documented. However, 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 competent 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 Ore Reserves and Mineral Resources. Reverse Circulation Drilling The reverse circulation drilling at Haile typically uses 16 cm 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 15 to 19 ltr/min above the water table and decreases to 4 ltr/min when groundwater is encountered. Sample sizes are generally between 9 and 14 kg dry mass. The standard size reflects a 15 to 20% split of the total drilled volume. Drill intervals are predominately of 1.5m length. The following paragraphs describe sample procedures as reported by Romarco personnel. Observations during a site visit confirmed the application of these techniques. For each 1.52m interval (5 feet), a sample container is placed on top of the splitter table to catch the flow from the sample splitter. Labelled, sample bags measuring 17.5 by 61 cm are placed in 19 to 26 ltr. plastic buckets. Multiple 0.65 cm holes are pre-drilled 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. Flocculent is added to each sample bag as it is placed on

Criteria	JORC Code Explanation	Commentary
		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 1.5m drill intervals. Sampling during advancement of each 6.1m drill 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 6.1m 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 1.5m 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 1.5m 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 labelled with the drill hole number and depth intervals in permanent marker.

Criteria	JORC Code Explanation	Commentary
		Diamond Drilling Diamond core drilling is by wireline methods and generally utilizes HQ and NQ size core 6.35cm and 4.8cm core. Core is transferred from the core barrels to plastic core boxes at the drill rig by the driller. Core orientation is not utilized other than for specific geotechnical programs. Core is broken as required to completely fill the boxes. Drill intervals are marked on the core boxes and interval marker blocks are labelled and placed in the core box. Whole core is transported to the sample preparation area by Romarco personnel. 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 riffle splitter to reduce the size to roughly 2.7 kg 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.

Criteria	JORC Code Explanation	Commentary
		Off Site Sample Preparation The AHK and KML sample preparation and assay facilities that are discussed in this section are independent of HGM. <u>AHK Geochem (AHK)</u> 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 65.5 degrees C 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 gm sample to 90% passing 150 mesh (0.106 mm) 8) Clean the pulveriser between samples with sand and compressed air 9) Ship about 125 gm 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. <u>Kershaw Mineral Laboratory (KML)</u> 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 93 degrees C 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 gm sample (+/- 50 gm) to 85% passing 140 mesh (0.106 mm) 8) Clean the pulveriser between samples with sand and compressed air 9) Approximately 225 gm 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 analysed at KML. Check assays in for mineralized intervals were sent to ALS Minerals in Reno for third party check assays.

Criteria	JORC Code Explanation	Commentary
Drilling techniques	- *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	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 date there was a total of 3,747 drill holes in the data base totalling 460,830m of drilling (1,511,912 ft). However, not all of this drilling was used for resource estimation within the block model. The reverse circulation drilling at Haile typically uses 16 cm drill bits. Diamond core is HQ and NQ. Some additional drilling has been completed on the property since the November 2011 time period. That information has not been incorporated into the resource model or into the determination of Mineral Resources or Ore Reserves. Property, permit, and other constraints are such that the additional drilling would not constitute a material change to the Mineral Resources or Ore Reserves. 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.
Drill sample recovery	- *Method of recording and assessing core and chip sample recoveries and results assessed.* - *Measures taken to maximise sample recovery and ensure representative nature of the samples.* - *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	Core recoveries were measured at the core shed by the logging geologist. Core recoveries average 97%. There is no observed relationship between core recovery and grade. RC samples are collected from a rotary splitter at the drill rig. Splitter ratios range from 8 to 17% and rock density ranges from 2.60 to 2.77 spg. Consequently, sample weights can range from 6.3 to 14.2 kg. 75% of the measured sample weights fall within this range. The dry weight of each sample is recorded at the Haile sample preparation facility. Because of the variable split proportions used, there is no direct relationship between the sample weight and sample recovery and no specific calculation of sample recovery. After drying and weighing, the samples are subsequently further split down to yield a 2.7 kg sample for dispatch to the laboratory. There RC samples have been checked by paired sample comparison to closely spaced DDH samples.
Logging	- *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* - *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.*	Core logging is completed on site by staff geologists at Haile Gold Mine. 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. All core intervals are photographed. RC chips are logged by the staff geologists from samples that are sieved from the coarse rejects. All logging, which is qualitative, is recorded in Excel files with a separate file for each drill hole. The

Criteria	JORC Code Explanation	Commentary
	- *The total length and percentage of the relevant intersections logged.*	logged information is stored on site and backed up periodically. All drilled intervals are logged.
Sub-sampling techniques and sample preparation	- *If core, whether cut or sawn and whether quarter, half or all core taken.* - *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* - *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* - *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* - *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* - *Whether sample sizes are appropriate to the grain size of the material being sampled.*	- Refer to sampling techniques section or the Quality of Assay data section for more detail. - Half core samples are taken, either by saw, or if too soft, cut by knife. - RC samples are rotary split under water injection - It is believed that preparation for both the diamond core and RC samples is appropriate. - The RC drilling procedures as discussed in sampling techniques were put in place to optimise sampling during water injection (flocculent, careful draining of samples, rotary splitting etc) - Romarco has consistently been sending pulps and duplicates to an outside third party laboratory. It is believed that the sample sizes are adequate. Although coarse gold has been observed in drill core, it is not common and not representative of the mineralization that will be processed. John Marek is of the opinion that the sample sizes and procedures are standard methods for deposits of this type.
Quality of assay data and laboratory tests	- *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* - *For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.* - *Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	The Mineral Resources and Ore Reserves at Haile are based on fire assay of a 30 gm aliquot for gold with Atomic Absorption finish. For additional detail refer to the sample preparation section. Blanks and standards, are inserted by Haile, and check assays are submitted to a second lab on a regular basis. The Haile drill hole data base was verified by IMC in late 2010 and the results published in the Technical Report titled “Haile Gold Mine Project, NI43-101 Technical Report Feasibility Study” dated 10 February 2011. The following discussion 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

Criteria	JORC Code Explanation	Commentary
		assay laboratory. 3) A comparison of Romarco drilling and assay information versus closely spaced historic information. (see the Verification Section) 4) A comparison of diamond drilling versus reverse circulation drilling (see the Verification Section) The approach presented above is to verify that the Romarco data is reliable based on the quality control information that is collected with the data. Once that is established, the applicability of the historic information is established by a nearest neighbour statistical analysis of old versus Romarco drilling (See the Verification Section). **Romarco Data Verification** The verification of the Romarco drill hole data was completed in two iterations: a) Drilling completed by Romarco during the period of 2007 through October 2010 of which 94% was collected during 2009 and 2010 (referred to as 2010 Data Verification). b) Drilling completed by Romarco between October 2010, and November 2011 (referred to as 2011 Data Verification). The two periods reflect the support work behind public statements of Mineral Resources and Ore Reserves that were published during late 2010 and then updated in late 2011. Summaries of the two verification exercises are included here. 2010 Romarco Data Verification The following checks were applied to the 2007 – Oct 2010 Romarco data by John Marek of IMC. References to work by IMC were completed by or under the responsible charge of John Marek. 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 internal duplicates that are selected by the Alaska Laboratory assay lab as an internal check. 5) Statistical analysis of the check samples that are submitted by Romarco to a third party laboratory.

Criteria	JORC Code Explanation	Commentary
		No material problems in data entry, Romarco inserted QC samples, internal duplicates selected by Alaska Laboratories, or the external third party laboratory check sampling were detected. John Marek and IMC conclude that the Haile data base at that date was reliable for the determination of Mineral Resources and Ore Reserves. 2011 Romarco Data Verification The following checks were applied to the October 2010 to November 2011 Romarco data by John Marek of 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. No material problems in data entry, Romarco inserted QC samples, or the external third party laboratory check samples were detected. John Marek and IMC conclude that the Haile data base at that date was reliable for the determination of Mineral Resources and Ore Reserves. 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 Ore Reserves. John Marek of Independent Mining Consultants, Inc. (IMC) acted as the Competent 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 Ore Reserves.
Verification of	• *The verification of significant intersections by*	IMC completed a nearest neighbour comparison of old drilling versus new drilling on a 6.1 m

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr>
<td>sampling and assaying</td>
<td>*either independent or alternative company personnel.*
• *The use of twinned holes.*
• *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.*
• *Discuss any adjustment to assay data.*</td>
<td>composite basis In order to gain comfort with the historic drilling at Haile. 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 6.1m 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 15.2m (50 ft) apart or less. The distance represents 2 model blocks.

Table 1: Old Drilling versus New Drilling, Statistical Comparison
<table>
<tr><th colspan="4"></th><th colspan="4">Hypothesis Tests</th></tr>
<tr><th>Sample Separation (m)</th><th>Number of Pairs</th><th>New Mean gm/t</th><th>Old Mean Gm/t</th><th>T Test</th><th>Paired-T</th><th>Binomial</th><th>KS</th></tr>
<tr><td>15.2m</td><td>878</td><td>0.686</td><td>0.823</td><td>Pass</td><td>Pass</td><td>Pass</td><td>Pass</td></tr>
</table>

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 below.

<u>Diamond Drilling vs RC Drilling</u>

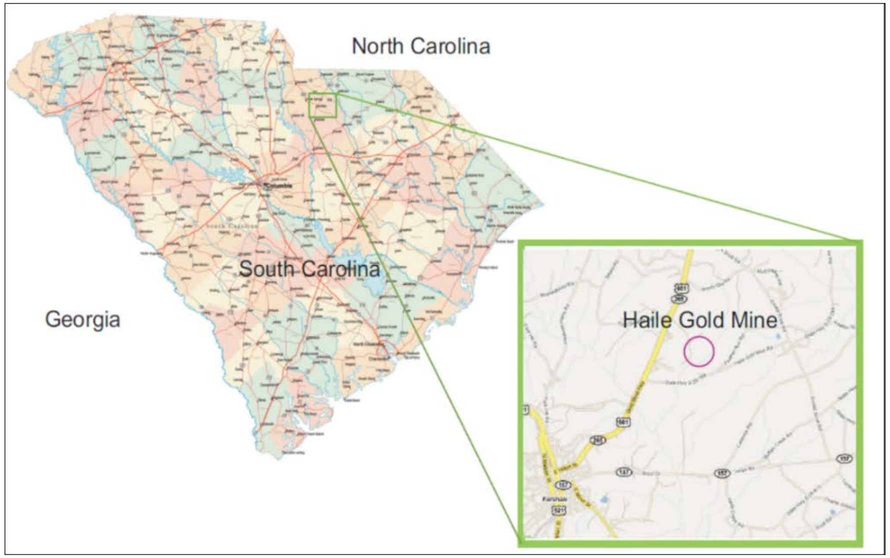
The data base at Haile consists of a mix of diamond drilling (DDH) and reverse circulation drilling (RC). IMC has compared the results of these two drill methods to ascertain whether or not they are biased relative to one another.</td>
</tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>A similar procedure was applied as outlined in the previous section. The 6.1m 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 neighbour analysis was completed with sample spacing's of 7.6m and 15.2m . Table 2summarizes the results.
Table 2: DDH Drilling versus RC Drilling, Statistical Comparison
<table>
<tr><td colspan="4"></td><td colspan="4">Hypothesis Tests</td></tr>
<tr><td>Sample Separation (m)</td><td>Number of Pairs</td><td>DDH Mean Gm/t</td><td>RC Mean Gm/t</td><td>T Test</td><td>Paired-T</td><td>Binomial</td><td>KS</td></tr>
<tr><td>7.6m</td><td>504</td><td>.891</td><td>.857</td><td>Pass</td><td>Pass</td><td>Fail</td><td>Pass</td></tr>
<tr><td>15.2m</td><td>1277</td><td>.891</td><td>.857</td><td>Pass</td><td>Pass</td><td>Fail</td><td>Pass</td></tr>
</table>
The results indicate that there is no observed bias between the two sampling methods.</td></tr>
<tr><td>Location of data points</td><td>• <i>Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.</i>
• <i>Specification of the grid system used.</i>
• <i>Quality and adequacy of topographic control.</i></td><td>Drill hole collars are currently surveyed with differential GPS with sub-centimetre accuracy. The historic Amax and early Romarco holes were surveyed by a South Carolina licensed surveyor using conventional ground methods. Frequent check surveys have been completed during the project.
The drill hole locations and the project coordinate system are South Caroline State Plane Coordinates NAD 27 North.
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.
Topographic control has been established to a high level of precision. Resource estimation and mine planning relied on contour maps with 0.6m contour intervals.</td></tr>
<tr><td>Data spacing</td><td>• <i>Data spacing for reporting of Exploration</i></td><td>Drill hole spacing is not a simple calculation at Haile because many holes are angle holes and down</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
and distribution	*Results.* - *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* - *Whether sample compositing has been applied.*	hole deflections occur during the drilling process. Several angle holes were often drilled from a single position. Estimation of block confidence was based on statistical parameters and the number of drill hole composites within the search radius. The drill hole spacing is variable, but typically 30m x 30m to 50m x 50m excluding Inferred Resources. Average Drill Spacing for Inferred = 82m. Drill hole data was composited to 6.1m (20ft) composites prior to block grade estimation. Additional discussion is included within the Section on Mineral Resources.
Orientation of data in relation to geological structure	- *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* - *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	The Haile gold mineralisation is not a vein deposit. The orientation of the mineralisation generally parallels the foliation of the host metasediments. The metasediments have variable dip that ranges between 20 degrees to the north-northwest and vertical. The majority of mineralisation dips 30 degrees to the north-northwest. Drilling orientation ranges from vertical to SE bearing angles to intercept mineralisation. All drill holes deviate and self-align perpendicular to the north-northwest dipping foliation and mineralisation. There is no evidence of orientation-related sample bias at Haile.
Sample security	- *The measures taken to ensure sample security.*	All drill hole samples are transported from the drill rigs to the Romarco sample prep facility by Romarco personnel. Access to the property is limited and controlled. When samples are shipped, to the lab, the sample manifests are checked by the lab and the receipt of all samples are confirmed. During off-shift hours, a Deputy Sherriff is on site providing security for the site and sample storage facility.
Audits or reviews	- *The results of any audits or reviews of sampling techniques and data.*	John Marek understands that the Haile data base and resource model have been reviewed by 3rd parties in several iterations. Those reviews were proprietary and IMC does not have access to the results. IMC is not aware of any suggested changes that may have resulted from the 3rd party reviews.

Section 2 Reporting of Exploration Results

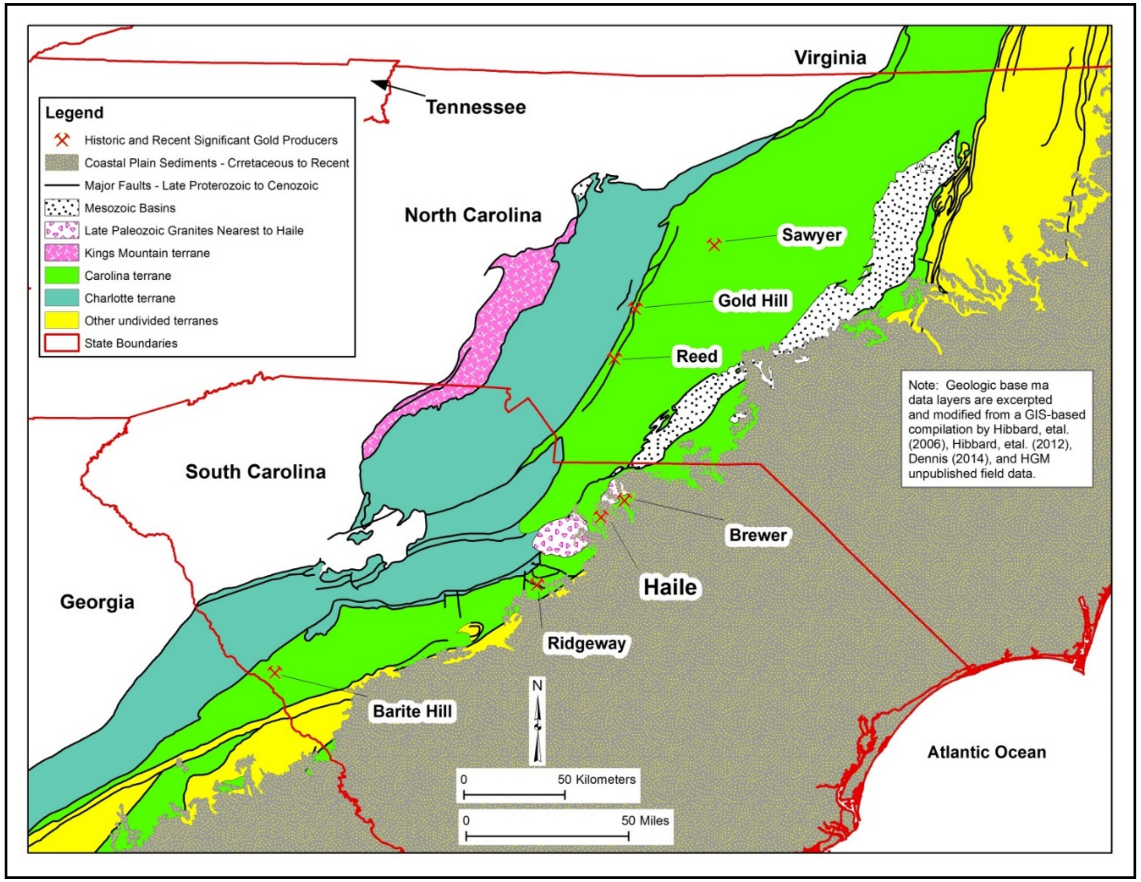
(Criteria listed in the preceding section also apply to this section.)

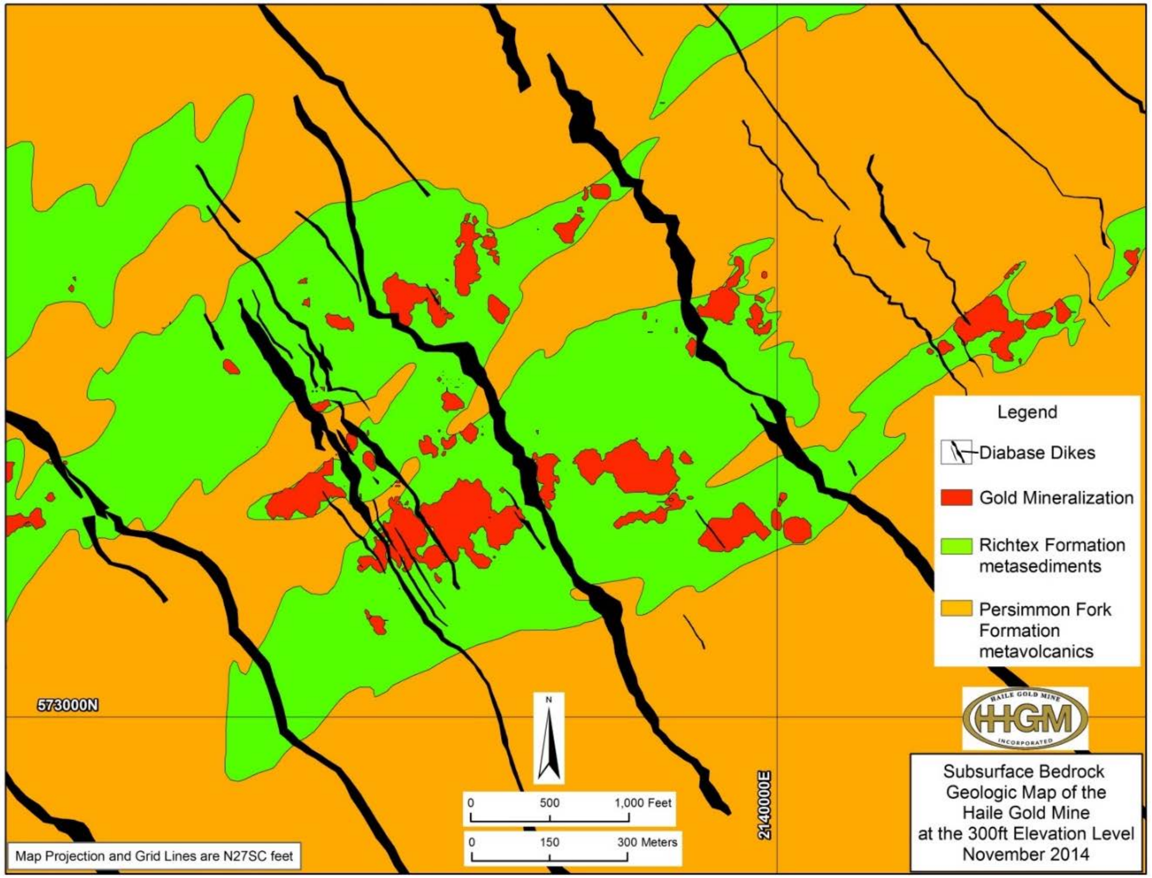
Criteria	JORC Code Explanation	Commentary
Mineral tenement and land tenure status	- *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* - *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	**Property Location** The Haile property site is located 4.8km (3mi) 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 27.4 km (17 mi) southeast of the city of Lancaster, the county seat, which is approximately 48.3 km (30 mi) south of Charlotte, North Carolina. The approximate geographic centre 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 1: General Location Map of the Haile Gold Mine **Ownership** Following a Plan of Arrangement completed on October 1st, 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 provided an inventory of property that is owned both within the project boundary and as

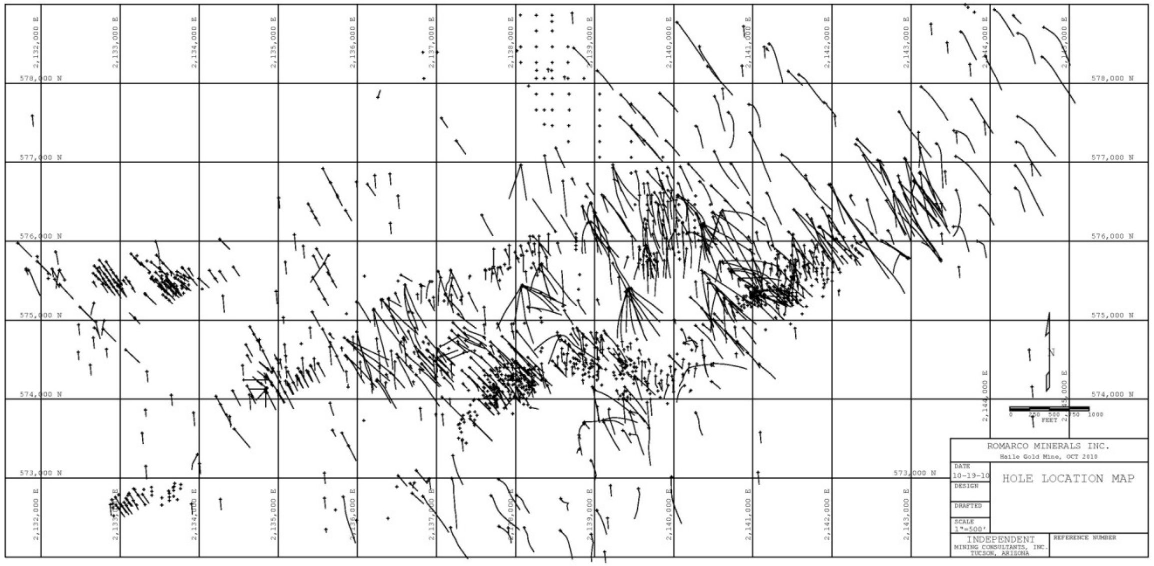
Criteria	JORC Code Explanation	Commentary
		buffer land outside the project boundary. After transferring approximately 4,388 acres of land into mitigation and conservancy projects, HGM owns approximately 5,719 acres of land in total. HGM owns additional land that is not associated with the project.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	Historic exploration was completed prior to acquisition of the Haile Gold Mine by Romarco. That work has been superseded by the drilling completed at Haile.
Geology	• *Deposit type, geological setting and style of mineralisation.*	This section was originally 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 Competent Person. 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 mineralisation styles indicate that this is a complex metalogenic province. Brewer has many features of an acid-sulfate mineralisation system such as the presence of aluminosilicates, topaz, and enargite. Gold mineralisation 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 mineralisation is hosted within silicified siltstones. Both deposits contain molybdenite and the mineralisation 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 mineralisation 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 mineralisation dissipates away from the source. Alternatively, several workers have proposed the mineralisation is structurally controlled and was caused by deformation. Tomkinson (1990) proposed that shearing was responsible for the mineralisation 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

Criteria	JORC Code Explanation	Commentary
		controlled the gold mineralisation. 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 mineralisation 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 mineralisation is pre-tectonic and rules out that the mineralisation 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 mineralisation is post depositional and invalidate the submarine hot springs or exhalative model.
Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* o *easting and northing of the drill hole collar* o *elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar* o *dip and azimuth of the hole* o *down hole length and interception depth* o *hole length.* • *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	• No Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed
Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* • *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the*	No Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed.

Criteria	JORC Code Explanation	Commentary
	procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. • *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	
Relationship between mineralisation widths and intercept lengths	• *These relationships are particularly important in the reporting of Exploration Results.* • *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* • *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').*	Drill intercepts are typically reported in down hole length from the drill collar. Most are 1.5m long assay intervals. No Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed.

Criteria	JORC Code Explanation	Commentary
Diagrams	- *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	 Figure 1: Gold Deposit Locations within the Carolina Terrane

Criteria	JORC Code Explanation	Commentary
		 Figure 2: Schematic Geologic Map of Haile Property, November 2014

Criteria	JORC Code Explanation	Commentary
		 Figure 3: Drill Hole Collar Locations
Balanced reporting	- *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	No Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed.
Other substantive exploration data	- *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	OceanaGold Corporation (OGC) continues to drill in the district surround the Haile Gold Mine. However, no Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed.

Criteria	JORC Code Explanation	Commentary
Further work	- *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* - *Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.*	OGC continues to drill in the district surrounding the Haile Gold Mine. However, no Exploration Results are being presented in this document. This report is focused on an advanced project that has well defined geological models and associated resource estimates completed.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Database integrity	- *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.* - *Data validation procedures used.*	IMC and John Marek as Competent Person has completed checks on roughly 10% of the drill hole data base by comparing the electronic data base against the Certificates of Assay. Transcription or data entry errors are minor and would not impact the stated Mineral Resources or Ore Reserves. See the section on Data QAQC.
Site visits	- *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* - *If no site visits have been undertaken indicate why this is the case.*	John Marek of IMC is the Competent Person for Mineral Resources and Ore Reserves. Mr. Marek has made visits to Haile on the following dates: November 4 – 6, 2009 and June 20, 2012 during which times I reviewed the drill core, core handling procedures, sample preparation, core logging and site conditions. June 2-3, 2015 for mine plan reviews, strategic discussions, and property progress updates. Other site visits can be seen in the Summary section.
Geological interpretation	- *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* - *Nature of the data used and of any assumptions made.* - *The effect, if any, of alternative interpretations on Mineral Resource estimation.*	Geologic surfaces were interpreted from drill logs 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 by IMC and John Marek for completeness. The geological interpretation is believed to appropriate for purposes of estimation. No alternative interpretations have been made. The rock type codes that are assigned to the mode are:

Criteria	JORC Code Explanation	Commentary
	- *The use of geology in guiding and controlling Mineral Resource estimation.* - *The factors affecting continuity both of grade and geology.*	Code 100 = Meta-Sediments 200 = Meta-Volcanics 400 = Dolerite (localled called Diabase at Haile) Dykes 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. Mineralisation is hosted within a folded sequence of meta-sediments / meta-volcanics. The majority of mineralisation is hosted within the meta-sediments, so the geometry of folding is an important control. Barren dolerite dykes cut across the sequence.
Dimensions	- *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	The gold mineralisation 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 1 km wide (NNW to SSE) and over 3.4 km long (WSW to ENE). Most of the mineralisation at Haile is restricted to the laminated metasiltstone of the Richtex Formation. The gold mineralised zones within the laminated metasediments can vary in distance from the metavolcanic contact, and can appear at different stratigraphic levels within the metasediments. Mineralised zones at Haile can strike (trend) northeast to southwest and east to west. The mineralised 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 mineralised zones exist.

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr>
<td>Estimation and modelling techniques</td>
<td>• <i>The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.</i>
• <i>The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.</i>
• <i>The assumptions made regarding recovery of by-products.</i>
• <i>Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).</i>
• <i>In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.</i>
• <i>Any assumptions behind modelling of selective mining units.</i>
• <i>Any assumptions about correlation between variables.</i>
• <i>Description of how the geological interpretation was used to control the resource estimates.</i>
• <i>Discussion of basis for using or not using grade cutting or capping.</i>
• <i>The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.</i></td>
<td>The model method was selected to provide sound estimates for mine planning and the corresponding estimation of Mineral Resources and Ore Reserves. The application of an indicator grade boundary was selected to minimize the “grade smearing” often associated with ordinary linear kriging. High grade values have been capped and additional limitations placed on the search radius of high grade composites. Those procedures are summarizes in the following paragraphs.
The resource estimates are depleted for previous small scale open pit mining, but not for the minor underground mining that historically occurred.
No by-products were estimated.
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 6.1 m 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:
<table>
<tr><th>Boundary</th><th>Boundary Type</th></tr>
<tr><td>Meta-Sediments vs Saprolite</td><td>Soft-Transitional over about 15.2m vertically</td></tr>
<tr><td>Meta-Seds vs Meta-Volcanics</td><td>Hard boundary</td></tr>
<tr><td>Sand to Saprolite</td><td>Hard boundary</td></tr>
<tr><td>Dolerite</td><td>Barren and not estimated</td></tr>
<tr><td>All other rock boundaries</td><td>Hard Boundaries</td></tr>
</table>
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 mineralisation 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 bedding in the Meta-Sediments. Since the mineralisation pre-dated the deformation, the orientation of the bedding appears to be indicative of the primary access of the mineralisation. Consequently, the new zones represent a structural overlay that is superimposed on the rock type information.
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 as follows:</td>
</tr>
</table>

Criteria	JORC Code Explanation	Commentary

Assay Cap Values

Meta-Sediments: Rock = 100

VarioZone	Description	Cap
45	Haile- South	24.00 g/t
303	Ledbetter	34.29 g/t
60	Snake	34.29 g/t
602	Horseshoe	102.86 g/t
Everywhere Else		13.71 g/t

Meta-Volcanics, Sand, Saprolite All Zones 10.97 g/t

Haile is a greenfield project which is currently under construction. Consequently, reconciliation against production history is not possible at this time.

IMC and John Marek have completed the following types of checks on the block model:

a) Visual comparisons of drill hole sections and plans versus the block model.
b) Comparisons of block mean grade versus contained composite grade within each estimation population and domain. This work has not indicated any bias between the drill data and the model.
c) Comparisons of the block mean grades and the contained composites at a range of cut-off grades. These comparisons indicate the appropriate trends with regard to the volume variance difference between the block model and the contained composite data. There is no indication of block model overestimation on a domain by domain basis.

Composites

The assay information was composited to 6.1m (20 ft) down hole composite intervals. The lithology 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 3.05m (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.

Table 1 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 Table 1. They were however, kept in place for block grade estimation.

Criteria	JORC Code Explanation	Commentary

Table 1: Basic Statistics of 6.1m Composites, By Rock Type and Variogram Zone

Zero Values Removed

Cross Tabulation of 6.1m Composites

Number of Composites and Mean Fire Gold Grade of Composites (gm/tonne) 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 0.411	3,699 0.789	1,033 0.137	387 0.171	206 0.171	3,917 0.514	1,011 0.343	281 0.309	1,303 1.166	27,272 0.494
Meta-Volcanics	1,841 0.069	914 0.069	400 0.034	752 0.000	113 0.034	1,640 0.034	659 0.034	101 0.034	280 0.034	6,700 0.045
Diabase	749 0.069	101 0.103	22 0.137	33 0.000		175 0.034	17 0.034	13 0.000	27 0.206	1,137 0.068
Saprolite	2,216 0.274	437 0.411	139 0.034	196 0.206	25 0.000	357 0.000	125 0.000	21 0.000	37 0.000	3,553 0.234
CPS	171 0.137	26 0.103	16 0.000	10 0.034	2 0.000	49 0.000	4 0.000	4 0.000	23 0.000	305 0.087
Column Totals	20,412 0.351	5,177 0.613	1,610 0.101	1,378 0.078	346 0.113	6,138 0.338	1,816 0.204	420 0.215	1,670 0.919	38,967 0.378

The drill hole spacing is variable, but typically between 30m x 30m and 50m x 50m. The block model was developed using blocks sized 7.62m x 7.62m on plan and 6.1m bench height (25 ft x25 ft and 6.1 m (20 ft) bench). The small block size in plan was selected in order to provide a reasonable method of modelling the interpreted geology with particular emphasis on the late barren dykes that cross the Haile deposit.

The bench height of 6.1m 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 6.1m (20 ft) selection. A bench height of 6.1m 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 2 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.

Criteria	JORC Code Explanation	Commentary
		Table 2: January 2012 Haile Model Area – Block Corners in South Carolina State Plane Coordinates (ft)

	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		

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 6.1m 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 0.14 and 0.34 gm/tonne . IMC opted to apply a 0.34 g/t discriminator to minimize over estimation of tonnage in that grade range.

The frequency plots also indicated changes in population at high grade values of 3.43 to 5.14 g/t (0.100 to 0.150 oz/ton). Within the Horseshoe zone (Variogram zone 602) the population break was interpreted at 17.14 g/t. 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 0.34 gm/tonne and gold grade variograms were run and interpreted.

As a result of this work, a single stage indicator kriging approach with a 0.34 gm/tonne discriminator was used in each of the population zones. A kriged indicator threshold of 0.50 was used to differentiate low from high grade domain volumes. Composites were back-flagged from the indicator domains to ensure consistency between data and estimates.
Table 3 summarises the grade estimation parameters that applied ordinary linear kriging.

Criteria	JORC Code Explanation	Commentary
		Table 3, 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 have been converted to meters
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	45.7	45.7	34.1	0.1	0.9	0.010	0.100	30.5
202	345	20	45.7	45.7	34.1	0.1	0.9	0.010	0.100	30.5
204	345	20	45.7	45.7	34.1	0.1	0.9	0.010	0.100	30.5
301	330	30	45.7	45.7	34.1	0.1	0.9	0.010	0.100	30.5
303	315	30	45.7	45.7	34.1	0.1	0.9	0.010	0.150	30.5
304	315	30	45.7	45.7	34.1	0.1	0.9	0.010	0.100	30.5
45	335	45	50.3	36.6	50.3	0.1	0.9	0.010	0.100	38.1
60	315	60	54.9	45.7	34.1	0.1	0.9	0.010	0.150	30.5
452	315	45	50.3	36.6	50.3	0.1	0.9	0.010	0.100	30.5
602 HS	325	60	54.9	45.7	34.1	0.1	0.9	0.010	0.500	15.2

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	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
202	345	20	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
204	345	20	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
301	330	30	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
303	315	30	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
304	315	30	45.7	45.7	7.6	0.1	0.9	0.010	0.100	30.5
45	335	45	50.3	36.6	7.6	0.1	0.9	0.010	0.100	30.5

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><td>60</td><td>315</td><td>60</td><td>54.9</td><td>45.7</td><td>7.6</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>452</td><td>315</td><td>45</td><td>50.3</td><td>36.6</td><td>7.6</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>602 HS</td><td>325</td><td>60</td><td>54.9</td><td>45.7</td><td>7.6</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td colspan="11">Saprolite</td></tr>
<tr><td>Vario</td><td>Bearing</td><td>Plunge</td><td colspan="3">Range and Search</td><td colspan="2">Variogram</td><td>Discrim</td><td colspan="2">High Grade Limit</td></tr>
<tr><td>Zone</td><td>Degrees</td><td>Degrees</td><td>Plunge</td><td>Strike</td><td>Cross</td><td>Nugget</td><td>Sill</td><td>oz/ton</td><td>Grd oz/ton</td><td>Max Srch</td></tr>
<tr><td>201</td><td>345</td><td>20</td><td>45.7</td><td>45.7</td><td>34.1</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>202</td><td>345</td><td>20</td><td>45.7</td><td>45.7</td><td>34.1</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>204</td><td>345</td><td>20</td><td>45.7</td><td>45.7</td><td>34.1</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>301</td><td>330</td><td>30</td><td>39.6</td><td>45.7</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>303</td><td>315</td><td>30</td><td>39.6</td><td>45.7</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.150</td><td>30.5</td></tr>
<tr><td>304</td><td>315</td><td>30</td><td>39.6</td><td>45.7</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>45</td><td>335</td><td>45</td><td>35.7</td><td>36.6</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>60</td><td>315</td><td>60</td><td>27.4</td><td>45.7</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.150</td><td>30.5</td></tr>
<tr><td>452</td><td>315</td><td>45</td><td>35.7</td><td>36.6</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.100</td><td>30.5</td></tr>
<tr><td>602 HS</td><td>325</td><td>60</td><td>27.4</td><td>45.7</td><td>15.2</td><td>0.1</td><td>0.9</td><td>0.010</td><td>0.150</td><td>30.5</td></tr>
</table>
<p>The table illustrates that the high grade search was typically limited to 30.5 to 38.1m or about 2/3 of the total search radius applied to all other mineralisation. The purpose is to limit the smearing of high grade over low grade that occurs with most grade estimation techniques.</p>
<p>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.</p>
<p>The dolerite dykes were not estimated because they are essentially barren.</p>
</td></tr>
<tr><td>Moisture</td><td><i>Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.</i></td><td>Estimates of tonnage are prepared on a dry basis.</td></tr>
</table>

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr>
<td>Cut-off parameters</td>
<td>• The basis of the adopted cut-off grade(s) or quality parameters applied.</td>
<td>
<p>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 mineralisation that has reasonable prospects of economic extraction. Table 1 summarizes the economic parameters that were applied to the resource floating cone.</p>
<p>Table 1: Floating Cone Input Parameters for Resource</p>
<table>
<tr><td colspan="2">Mining Cost</td><td colspan="3">$1.31/tonne material</td></tr>
<tr><td colspan="2">Incremental Haul Cost</td><td colspan="3">$0.011/ bench below 440</td></tr>
<tr><td colspan="2"></td><td colspan="3"></td></tr>
<tr><td colspan="2">Process Cost</td><td colspan="3">$7.96 /tonne ore</td></tr>
<tr><td colspan="2">G&A</td><td colspan="3">$2.06 /tonne ore</td></tr>
<tr><td colspan="2">Total=</td><td colspan="3">$10.02 /tonne ore</td></tr>
<tr><td colspan="5">Process Recovery</td></tr>
<tr><td colspan="5">100 x (1-(0.0583 x Grd ^ -0.3696))   Based on Imperial Grade Units of troy oz / short ton</td></tr>
<tr><td colspan="2">Refining Cost</td><td>$3.00 / ounce</td><td></td><td></td></tr>
<tr><td colspan="5">Slope Angles Overall Angles to Include Roads</td></tr>
<tr><td></td><td>North Wall=</td><td>48 Degrees</td><td></td><td></td></tr>
<tr><td colspan="2">South Wall=</td><td>40 Degrees</td><td></td><td></td></tr>
<tr><td colspan="2">Saprolite=</td><td>40 Degrees</td><td></td><td></td></tr>
<tr><td colspan="2">Sand=</td><td>27 Degrees</td><td></td><td></td></tr>
<tr><td colspan="5">Calculated Cutoff Grades</td></tr>
<tr><td colspan="5">Open Pit Mineral Resources are tabulated at Internal Cutoff</td></tr>
<tr><td>Price $/oz</td><td colspan="2">Recovered Au g/tonne</td><td colspan="2">In-Place Au tm/tone</td></tr>
<tr><td></td><td>Breakeven</td><td>Internal</td><td>Breakeven</td><td>Internal</td></tr>
<tr><td>1200</td><td>0.297</td><td>0.260</td><td>0.446</td><td>0.411</td></tr>
</table>
<p>There are additional Mineral Resources that have reasonable expectation of production by underground open stoping methods. The underground resource estimate was completed by Anthony Finch P.Eng of Snowden Mining Industry Consultants. IMC and John Marek have sufficiently reviewed this work to take responsibility as the Competent Person for the underground resources. Mine operating costs for underground mining were estimated at $88.28/tonne. The resulting cut-off grade is 2.74 gm/tonne in-situ gold grade.</p>
<p>A variable recovery function was applied to model the process response as shown on Table 1. 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 1 for the open pit resource cut-off.</p>
</td>
</tr>
</table>

Criteria	JORC Code Explanation	Commentary
		Slope angles on Table 1 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 1 are based on the assumption that the mill production rate would be increased from the 6,350 tpd feasibility rate to 9,072 tpd for the open pit resource due to the increased tonnage in the resource pit. The underground resource calculations are based on 6,350 tpd. The block model and the determination of the open pit Mineral Resources were completed by IMC with John Marek, P.E. acting as the competent person for the calculation. Mr. Marek is independent of Haile Gold, Romarco and OGC and has been working on Mineral Resource and Ore Reserve estimates for precious metals projects for over 39 years. A component of the Horseshoe mineralisation is included within the resource pit. The Horseshoe mineralisation 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 Ore Reserves. There is no certainty that the Mineral Resources will be realized or that they will convert to Ore Reserves.

Criteria	JORC Code Explanation	Commentary
Mining factors or assumptions	- *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	There are two components of the Mineral Resource at the Haile project. The open pit Mineral Resource is the material that is contained within a computer generated floating cone using the economic assumptions of the previous Table item. No additional mining dilution is applied to the open pit resource estimate because the block model methods contain dilution built in to the block grades. Underground stope resources are based on the judgement that open stoping with paste back fill would be an appropriate mine method. Once stope targets were developed for the resource, mining dilution of 12.5% was added to the stope model tonnage at a grade of 0.0 g/ton.
Metallurgical factors or assumptions	- *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	Extensive processing testing was completed for detailed design and engineering of the process plant. The following equation of process recovery is the result of that work and has been applied to the Mineral Resource and Ore Reserve. Process Recovery in % = 100 x (1-(0.0583 x Grd ^ -0.3696)) The equation is based on head grades in terms of imperial units in troy oz/short ton. The equation has not been converted to metric units within this Table. The details of process testing and process plant design are the same for the resource and the Ore Reserve. That information is provided in the Table 1 Section on Ore Reserves.
Environmental factors or assumptions	- *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well*	Waste storage was studied in detail for the Ore Reserve and that information will be presented in the Table 1 Section on Ore Reserves. In Summary, acid generating rock is stored on lined storage facilities and the seepage from the facility treated before release. Potentially acid generating material is typically stored in the depleted pits below the water table after mixing with lime. Non-acid generating rock is stored adjacent to the pit at convenient haul distances.

Criteria	JORC Code Explanation	Commentary
	advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.* • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit.* • *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	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 Meta-Seds 2.77 Meta-Volcanic 2.60 Dolerite Dykes 2.91 Saprolite 2.14 Sand 1.89 Fill 2.14 assumed Old Heaps 1.89 assumed same as Sand
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* • *Whether the result appropriately reflects the*	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: The first utilized the techniques and parameters as outlined in Tables 8 and 9 presented earlier. Then, A second indicator and grade kriging run was completed where an additional 15.2m (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

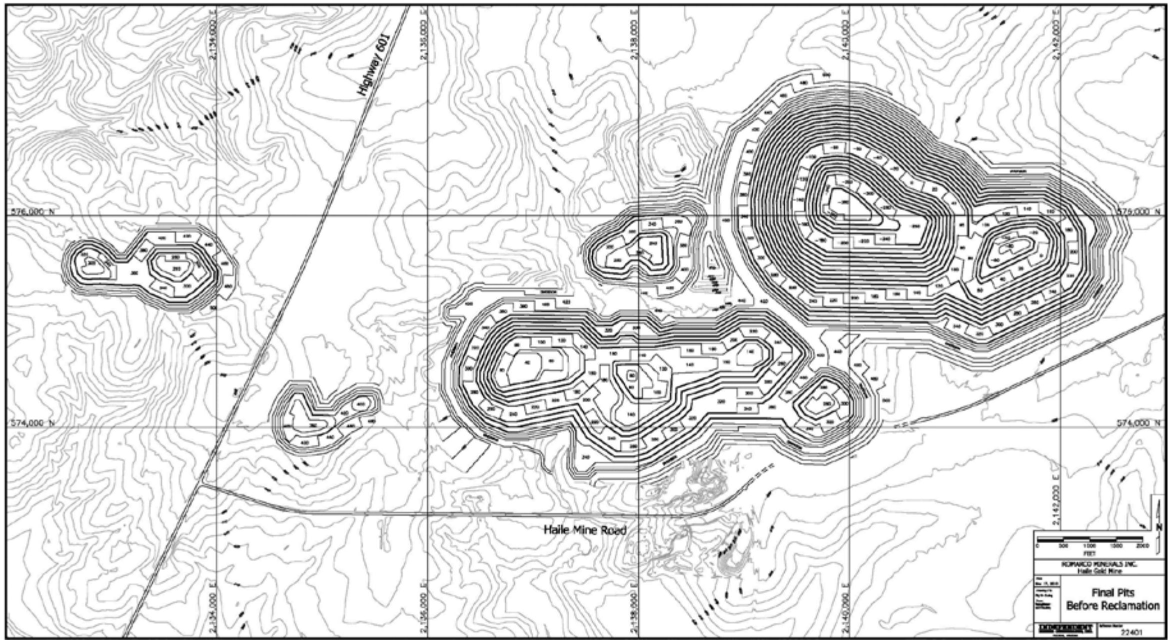
Criteria	JORC Code Explanation	Commentary
	Competent Person's view of the deposit.	Indicated: Kriged Standard Deviation < = 1.00 Minimum Number of composites = 5 (Two Holes) Inferred: If not assigned a code above but received a gold grade from the first pass. Second Pass Inferred A second kriging was completed with 15.2m (50 ft) additional search. Any block assigned a grade in the second kriging run that was not already assigned becomes an additional inferred block
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	Runge Pincock Minarco completed a review of the Haile project in July 2014. They considered the Feasibility Study and proposed development plans supplemented by a subsequent resource estimate, updated cost structure, and EIS to be acceptable for the proposed production goals and future output from the Haile Gold Project.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	Haile is a greenfield project which is currently under construction. Consequently, reconciliation against production history is not possible at this time. Comparisons of the block mean grades and the contained composites at a range of cut-off grades have been completed. These comparisons indicate the appropriate trends with regard to the volume variance difference between the block model and the contained composite data. There is no indication of block model overestimation on a domain by domain basis. The Haile estimate is expected to provide acceptable estimates in global terms. Given however that most of the Haile resource is based on moderately broad drilling, the performance of local estimates is likely to be more variable.

Section 4 Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code Explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• *Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve.* • *Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.*	The Mineral Resources at HGM are based on a block model and resource estimate by John Marek of IMC that was published March 13, 2012. The Ore Reserves are based on a block model and mine plan by John Marek of IMC that was published February 10, 2011. That February 2011 Ore Reserve was republished without change within the March 13, 2012 Technical Report. The HGM feasibility study was updated to reflect more current costs and progress toward mine permits on November 21, 2014. The November 21, 2014 report presented the same Mineral Resources and Ore Reserves as the March 13, 2012 report. The impact of the later March 13, 2012 model was evaluated within the reserve volume and the net change to the Ore Reserve was minor. It is the opinion of John Marek (competent person) the impact of the March 13, 2012 on the Ore Reserve was not material. The Mineral Resources and Ore Reserves within this Table 1 document have not been changed since the November 21, 2014 Canadian Technical Report. For consistency with that disclosure, the effective date of this Mineral Resource and Ore Reserve is November 21, 2014. The model procedures applied to the February 10, 2011 model and the March 13, 2012 model were identical. The amount of available drilling was the only difference between the two models.. All Ore Reserves are contained within a detailed open pit mine plan. There are no underground reserves stated at this time. The Mineral Resources are reported inclusive of Ore Reserves.
	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* • *If no site visits have been undertaken indicate why this is the case.*	John Marek of IMC is the Competent Person for Mineral Resources and Ore Reserves. Mr Marek has made visits to Haile on the following dates: November 4 – 6, 2009 and June 20, 2012 during which times I reviewed the drill core, core handling procedures, sample preparation, core logging and site conditions. June 2-3, 2015 for mine plan reviews, strategic discussions, and property progress updates.
Study status	• *The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves.* • *The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine*	The mine plan and process plant design that support the statement of Ore Reserves is more detailed than a final feasibility study. 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. A mine plan has been developed that is technically achievable and economically viable, and that

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td><i>plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.</i></td><td>material Modifying Factors have been considered.</td></tr>
<tr><td>Cut-off parameters</td><td><i>The basis of the cut-off grade(s) or quality parameters applied.</i></td><td>
<p>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 1 summarizes the input economics and recoveries that were applied to the development of Ore Reserves. Slope angles on Table 1 reflect estimated overall angles that would include mine haul roads.</p>
<p>Table 1: Cost Parameters for Ore Reserve Cut-off in $USD and Metric Tonnes</p>
<table>
<tr><td>Mining Cost</td><td colspan="2">Adjust Fuel and Lime</td><td colspan="2">$1.42/ tonne material</td></tr>
<tr><td colspan="3">Add Sustaining Capex</td><td colspan="2">$0.18/ tonne material</td></tr>
<tr><td colspan="3">Mining Total</td><td colspan="2">$1.60 tonne material</td></tr>
<tr><td colspan="3">Process Cost</td><td colspan="2">$8.32/ tonne ore</td></tr>
<tr><td>G&A</td><td colspan="2">$5,629 k$/yr</td><td colspan="2">$2.43/ tonne ore</td></tr>
<tr><td>Process Recovery</td><td colspan="2">100x(1-(0.0583xGrd^-0.3696))</td><td colspan="2"></td></tr>
<tr><td colspan="3">Refining Cost</td><td colspan="2">$3.00/ ounce</td></tr>
<tr><td colspan="3">Incremental Haul Cost</td><td colspan="2">$0.011/ bench below 440</td></tr>
<tr><td colspan="3">Bench Discount Rate</td><td colspan="2">1.00% /bench</td></tr>
<tr><td colspan="3">Slope Angles Overall Angles to Include Roads</td><td colspan="2"></td></tr>
<tr><td></td><td colspan="2">North Wall=</td><td colspan="2">41 Degrees</td></tr>
<tr><td></td><td colspan="2">Deep South Wall=</td><td colspan="2">35 Degrees</td></tr>
<tr><td></td><td colspan="2">Shallow South Wall=</td><td colspan="2">32 Degrees</td></tr>
<tr><td></td><td colspan="2">Saprolite=</td><td colspan="2">40 Degrees</td></tr>
<tr><td></td><td colspan="2">Sand=</td><td colspan="2">27 Degrees</td></tr>
<tr><td colspan="5">Calculated Cutoff Grades</td></tr>
<tr><td rowspan="2">Price
$/oz</td><td colspan="2">Recovered Au gm/tonne</td><td colspan="2">In-Place Au gm/tonne</td></tr>
<tr><td>Breakeven</td><td>Internal</td><td>Breakeven</td><td>Internal</td></tr>
<tr><td>950</td><td>0.411</td><td>0.343</td><td>0.578</td><td>0.480</td></tr>
</table>
</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
		 Figure 1: Final Pit Design for Orel Reserve
Mining factors or assumptions	- *The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design).* - *The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc.* - *The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.* - *The major assumptions made and Mineral*	The Ore Reserve is based on the application of conventional open pit hard rock mining methods with 6.1m (20 ft) bench heights. A detailed mine plan was prepared to deliver 6,350 tpd to the flotation process mill. Sufficient waste material is moved to assure continued ore release of the required ore. The feasibility mine plan for the Haile Gold Mine was developed by Independent Mining Consultants, Inc. (IMC). John Marek of IMC acted as the Competent Person for the development of the feasibility mine plan. The mine plan produces 2,317 kt of gold bearing ore per year to the process plant (6,350 tpd for 365 days/year). After a one year preproduction period, total material movement ramps up to 20,045 kt/year (54,874 tpd) for the first three years followed by 31,745 kt/year (86,981 tpd) for four years. No Inferred Resources were included in the mine plan. Mining will utilize 6.1m (20 ft) benches. Drilling and blasting will be required for the hard rock units

Criteria

JORC Code Explanation

Resource model used for pit and stope optimisation (if appropriate).

- *The mining dilution factors used.*
- *The mining recovery factors used.*
- *Any minimum mining widths used.*
- *The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.*
- *The infrastructure requirements of the selected mining methods.*

Commentary

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 below represents the final selection by the Haile mine engineering staff.

Unit	Initial Fleet for 3 Years	Fleet, Year 4 and Beyond
16.5 cm Blast Hole Drills	3	4
11.5 Cu Meter Front Loader	1	2
13 Cu Meter Front Loader	1	1
11 Cu Meter Hyd Shovel	1	1
91 tonne Trucks	12	24

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

The mine production schedule is summarized of Table1. Quarterly mine plans were developed for the preproduction period and the first 2 years of the mine plan.

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	6.1m
Road Width with Ditch and Berm	29.0m
Maximum Road Gradient	10%
Typical Pushback Width	91.4m

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

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>
<table>
<tr><th>Zone</th><th>Interramp Angle Degrees</th></tr>
<tr><td>Sand</td><td>27</td></tr>
<tr><td>Saprolite</td><td>40</td></tr>
<tr><td>South Pit</td><td></td></tr>
<tr><td>North Side</td><td>49</td></tr>
<tr><td>South Side</td><td>38</td></tr>
<tr><td>Ledbetter Pit</td><td></td></tr>
<tr><td>North Side</td><td>49</td></tr>
<tr><td>South Side</td><td>42</td></tr>
<tr><td>Snake Pit</td><td></td></tr>
<tr><td>North Side</td><td>49</td></tr>
<tr><td>South Side</td><td>45</td></tr>
</table>
<p>Dilution</p>
<p>The block model as described in Section 3 was assembled so that block grades reflect the production inclusive of dilution and selective mining criteria. The competent person has compared the block grade distribution against the supporting data and finds that no additional factors for mining dilution or recovery need to be applied to the block model.</p>
</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary

Mine Production Schedule for Ore Reserve in Metric Units

Table 1: Mine Production Schedule

Mine Schedule									
Year	Recov Cutoff gm/tonne	Ore Ktonnes	Head Grade gm/t	Recov Grade gm/t	LG Stkp Ktonnes	Head Grade gm/t	Recov Grade gm/t	Waste Ktonnes	Total Material Ktonnes
ppQ1								136	136
ppQ2								600	600
ppQ3	0.583	7	0.857	0.019	16	0.651	0.480	1,047	1,070
ppQ4	0.583	26	0.926	0.720	24	0.651	0.514	2,571	2,622
ppQ5	0.583	34	1.200	0.960	24	0.617	0.446	4,953	5,012
ppQ6	0.583	72	3.154	2.743	24	0.617	0.480	4,916	5,012
yr1Q1	0.583	295	3.120	2.709	50	0.617	0.446	4,667	5,012
yr1Q2	0.583	579	3.189	2.743	88	0.617	0.446	4,345	5,012
yr1Q3	0.583	579	2.914	2.503	73	0.617	0.446	4,361	5,012
yr1Q4	0.583	580	2.606	2.229	83	0.617	0.480	4,350	5,012
yr2Q1	0.651	580	2.606	2.229	93	0.686	0.514	4,340	5,012
yr2Q2	0.651	580	2.194	1.851	96	0.651	0.480	4,336	5,012
yr2Q3	0.651	580	1.886	1.577	166	0.651	0.480	4,266	5,012
yr2Q4	0.651	579	1.851	1.543	168	0.686	0.514	4,266	5,012
3	0.411	2,318	2.571	2.194	80	0.514	0.377	17,742	20,140
4	0.583	2,318	2.434	2.091	601	0.617	0.480	27,926	30,844
5	0.754	2,318	2.091	1.783	1,239	0.720	0.549	28,194	31,752
6	0.480	2,318	2.126	1.817	190	0.549	0.411	29,244	31,752
7	0.754	2,318	2.331	1.954	1,385	0.720	0.549	27,141	30,844
8	0.343	2,318	2.160	1.851				23,507	25,825
9	0.343	2,318	2.537	2.194				5,954	8,272
10	0.343	2,318	2.503	2.126				4,726	7,043
11	0.343	2,318	1.749	1.440				4,384	6,701
12	0.343	758	0.789	0.617				1,023	1,782
Total		26,109	2.272	1.933	4,400	0.680	0.516	218,996	249,505

Tonnages are Dry Metric Tonnes x 1000
All Mill Feed Ore is Combined Proven and Probable

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>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 the environmental and acid rock contractors to HGM. 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.</td></tr>
<tr><td>**Metallurgical factors or assumptions**</td><td>• *The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.*
• *Whether the metallurgical process is well-tested technology or novel in nature.*
• *The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.*
• *Any assumptions or allowances made for deleterious elements.*
• *The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.*
• *For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?*</td><td>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 metallurgical test results were used to develop process design criteria and the flow sheet for processing the ore.

A detailed description of the flowsheet development and associated test programs has been previously published in the “NI 43-101 Technical Report Project Update” dated November 21, 2014 for the Haile Gold Mine Project.

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 1.

Table 1: Comminution Parameters

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

Criteria	JORC Code Explanation	Commentary

tested to determine the 100 and 200 mesh bond ball mill work index. The results for this work are presented in Table 2.

Table 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

RDi performed flotation test work to investigate the recovery of gold and silver to a sulphide 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 3 and Table 4.

Table 3: 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

Criteria	JORC Code Explanation	Commentary

Table 4: Flotation Test Results

Composite No.	Grind Size (P80, 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 5.

Table 5: Flotation Tailing Leach Test Results

Composite No.	Grind Size (P80, 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

Criteria	JORC Code Explanation	Commentary

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.

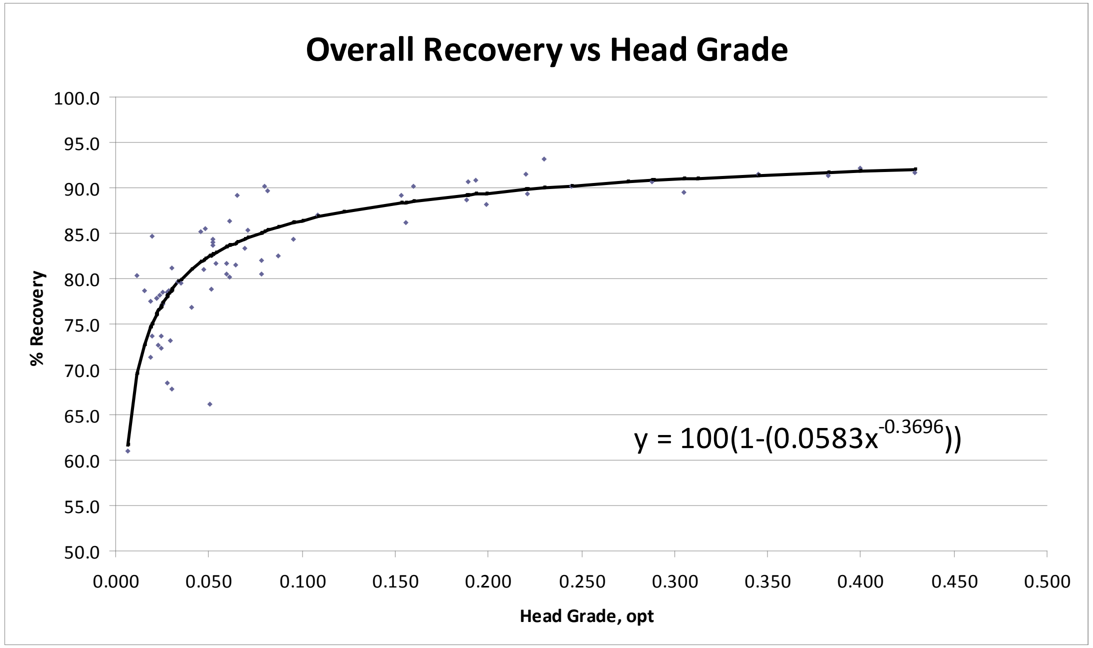
Cyanide destruction tests were run on process slurry samples from the master composite tests. The SO_2/air process (with sodium meta-bisulfite addition as the SO_2 source) was successful in destroying cyanide in the concentrate leach slurry samples. A test performed on a flotation tailing slurry sample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

Thickening tests indicate leached tailings could be dewatered to 60% solids (w/w) in thickeners using a unit area factor of1.2 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

Criteria	JORC Code Explanation	Commentary
		SO_2/air technology (sodium sulphite 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 -4.80 kg $CaCO_3$ /tonne (-9.6 lbs $CaCO_3$/t) for leached flotation tailing to -1,130 kg $CaCO_3$ /tonne (-2,260 lbs $CaCO_3$/t) for flotation concentrate leach tailing. The cyanide destruction test results indicate that the SO2/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. 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 mineralisation 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 favourably 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.

Criteria	JORC Code Explanation	Commentary
		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 results of grade and recovery data analysis is shown in Figure 1.  Figure 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 6. Grinding media consumption rates are presented in Table 7.

Criteria	JORC Code Explanation	Commentary

Table 6: 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
Flocculent	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

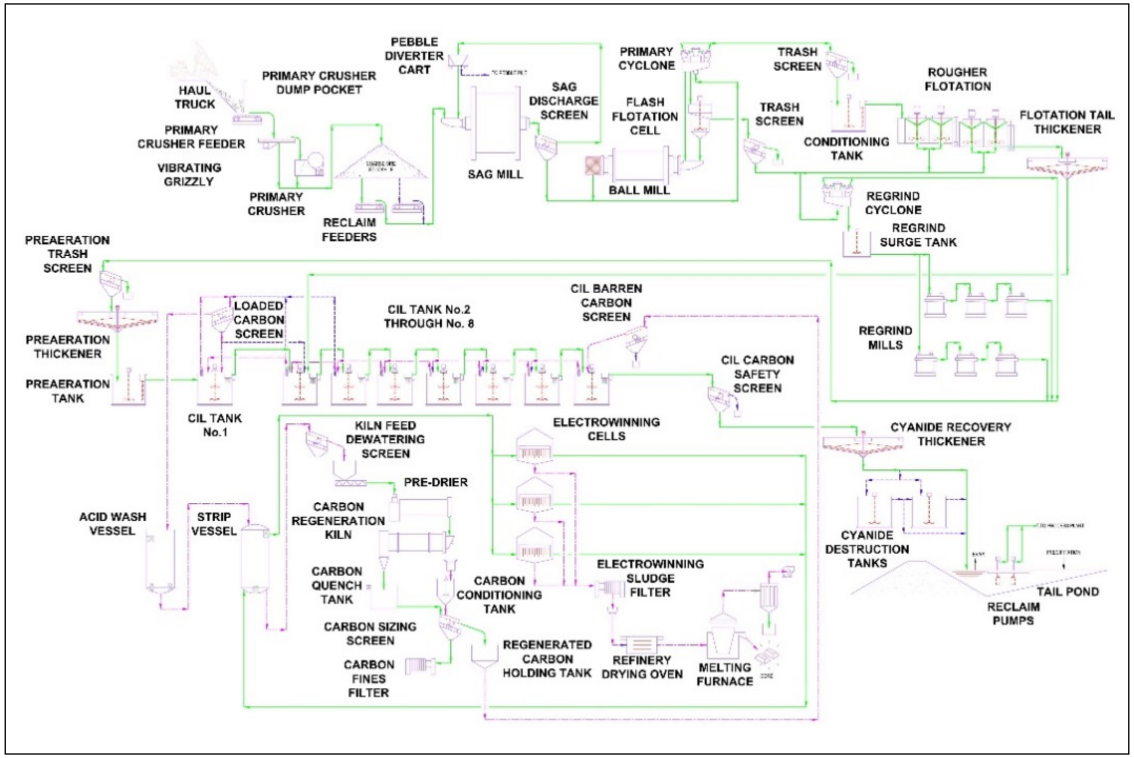
Table 7: Grinding Media

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

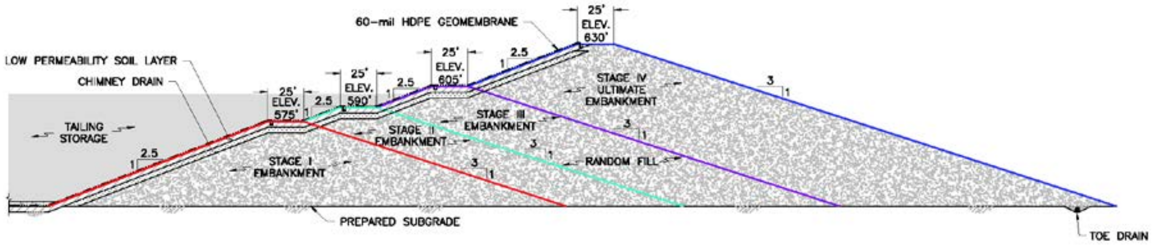
The following items summarize the process operations required to extract gold and silver from the Haile ore. The plant was designed to process 6,350 tpd.

- Size reduction of the ore by a primary jaw crusher to reduce the ore size from run-of-mine (ROM) to minus 15 cm (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.

Criteria	JORC Code Explanation	Commentary
		- 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 sulphur and oxygen, with copper sulphate 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, flocculent, copper sulphate, ammonium bisulfite, hydrochloric acid, lime, antiscalant, UNR 811 A, sulphuric acid, Aero 404, potassium amyl xanthate (PAX), MIBC, and lead nitrate. The overall process flow sheet is shown in Figure 2.

Criteria	JORC Code Explanation	Commentary
		 (Source: M3, 2014) Figure 2: Overall Process Flowsheet

Criteria	JORC Code Explanation	Commentary
	- *The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.*	Haile Gold Mine holds all of the main permits necessary to construct and operate the project. Construction is now in progress. Discussions with regulatory agency personnel over more than six (6) years and the successful completion of the Environmental Impact Statement by the Corps of Engineers have resulted in obtaining all of the major permits necessary to construct, operate and close a new operation at Haile. In addition, regulatory discussions do not reveal any new legislation or regulations that are being contemplated that could have an adverse impact on mine construction schedules, operations or anticipated costs. The regulatory agencies have also acknowledged that they are encouraged that successful reclamation has been completed previously at the site (documented through successful partial bond release) and that successful reclamation can be performed again in the future. The project is unique in that it occurs wholly on private land owned or controlled by HGM and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies. In addition, there is no potential for the federal government to impose a royalty by an amendment to the 1872 Mining Law. Since the property has been mined in the past, a significant amount of background and environmental baseline data existed while additional data was collected through the Environmental Impact Statement (EIS) process. This data continues to be collected. Major permits/certifications obtained include 404 Dredge and Fill Permit, 401 Water Quality Certification, air quality permit, and NPDES Permits (wastewater discharge, wastewater treatment system construction, and stormwater). The last remaining permit, the Mine Operating Permit, was issued in January of 2015. The operation of the mine, tailing, and process facilities are subject to permits that impose various operational limits. All of those items have been addressed and all are accounted for in the estimated project costs and production schedules. The permits currently held by the Haile Mine may be kept, modified, terminated, or replaced during the life of the mine.
Infrastructure	- *The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can*	The Haile Project is located in an area that is highly populated, therefore a good infrastructure exists. The project is adjacent to a state highway and there is a very large, skilled workforce nearby. 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 6.5 and 37.1 million tonnes

Criteria	JORC Code Explanation	Commentary
	be provided, or accessed.	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. The TSF is a zoned earth fill 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 1: Tailing Storage Facility Typical Section Stability analyses were conducted under both static and seismic loading conditions. **Overburden Storage** During the mine life, seven different overburden storage areas (OSAs) will be utilized for the storage of approximately 219 million tonnes 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. All of the OSA's will be developed with a final reclaimed overall 3(H):1(V) slope. Stability analyses

Criteria	JORC Code Explanation	Commentary
		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. **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. 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. 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. **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. **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

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>- Truck Wash
- Regional Geology Building (not part of this project, but available to Haile Gold Mine)
- Laboratory (established already)

Power Supply

Power for the Haile property will be provided from Duke Energy, Central Electric Power Cooperative and/or Lynches River Electric Cooperative. The power transmission infrastructure is well established. A new 69 kV (Lynches River) service will be required.

The cost for the construction of the power line is included in the power rate schedule as shown in Table 1.

Table 1: Power Rate Schedule
<table>
<tr><th>Year</th><th>Power Rate ($/kW-H)</th></tr>
<tr><td>1</td><td>0.0477</td></tr>
<tr><td>2</td><td>0.0492</td></tr>
<tr><td>3</td><td>0.0519</td></tr>
<tr><td>4</td><td>0.0555</td></tr>
<tr><td>5</td><td>0.0577</td></tr>
<tr><td>6</td><td>0.0699</td></tr>
</table></td></tr>
<tr><td>**Costs**</td><td>- *The derivation of, or assumptions made, regarding projected capital costs in the study.*
- *The methodology used to estimate operating costs.*
- *Allowances made for the content of deleterious elements.*
- *The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co- products.*
- *The source of exchange rates used in the study.*
- *Derivation of transportation charges.*
- *The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.*
- *The allowances made for royalties payable, both Government and private.*</td><td>**Capital Cost Estimate**

The project was estimated in Q4 2014 US 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 labour rates gathered from Means, and local contractors.

Capital costs include all infrastructure costs, owner's costs and contingency. No specific deleterious elements have been found with the Haile project. The management of acid rock drainage as discussed in the mine plan and geotechnical sections have been address in the project costs.

Exchange rates do not apply to this project because it was designed and is under construction in the United States, based on U.S. Dollars.

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</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
		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 1: Summary of Initial Capital Costs in US Dollars

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**

Sustaining Capital Cost Estimate

Sustaining capital costs were also evaluated for the project. Costs were estimated for future sustaining costs as shown in Table 2.

Table 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**

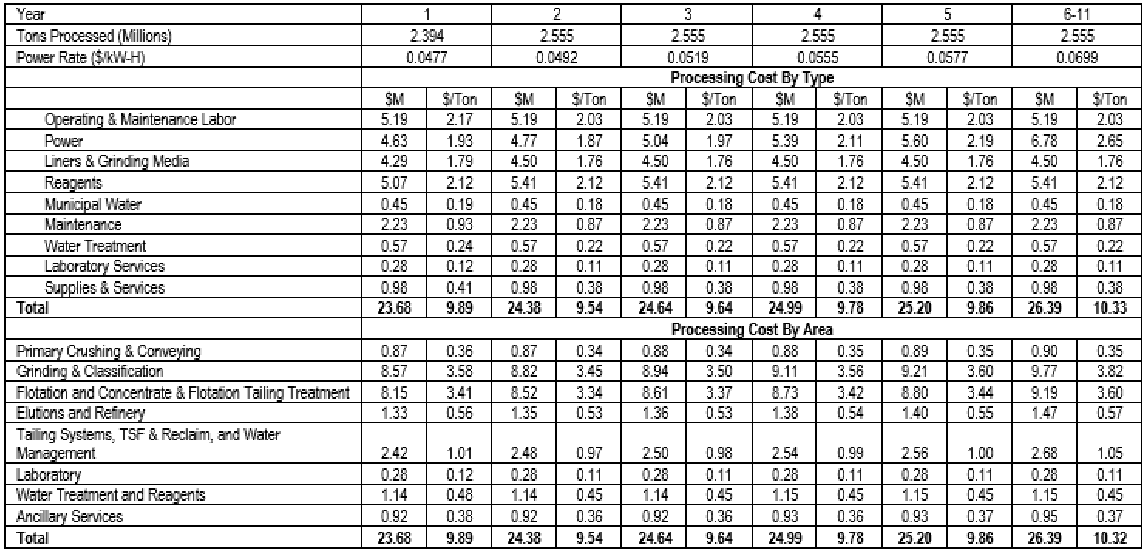
Sustaining capital costs include indirect costs, but do not include contingency. All sustaining costs are in Q4, 2014 US dollars.

Criteria	JORC Code Explanation	Commentary
		Operating Cost Estimate

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 3 below. Labour costs were developed based on a staffing plan and rate schedule.

Table 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**

No allowance was made for royalties, government or private.

Gold pricing, refining, and transport cots are discussed in the Market Assessment section.

Table 4 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.

Criteria	JORC Code Explanation	Commentary

Table 4: Summary of Mine Capital and Operating Costs

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

	Mine Equipment			(1)		
Year	Initial Capital Cost	Sustaining Capital Cost	Mine Preprod. Development	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

The mine assay cost was calculated using a unit rate of $7.46 per sample

<table>
<tr><th>Criteria</th><th>JORC Code Explanation</th><th>Commentary</th></tr>
<tr><td></td><td></td><td>

G&A Costs

HGM provided an estimate for the G&A cost for the project of $9.03 Million per year. These costs include labour, property costs, utilities, external assays, legal fees, outside services, insurance and other general costs. Table 5 shows a summary of these costs.

Table 5: General and Administrative Costs

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

</td></tr>
<tr><td>Revenue factors</td><td>

- *The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc.*
- *The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.*

</td><td>

Gold and silver are readily traded and the cost structure is well known. The basis of the financial analysis within this study was $1250/ troy ounce gold. Transportation, and refining cost have been included in the Financial analysis based on current terms at precious metal refineries.

The mine plan as presented in previous sections has been directly utilized into the financial analysis of the project. Appropriate dilutions and mining recoveries are reflected in the mine schedule.

Process recoveries are as presented previously.

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

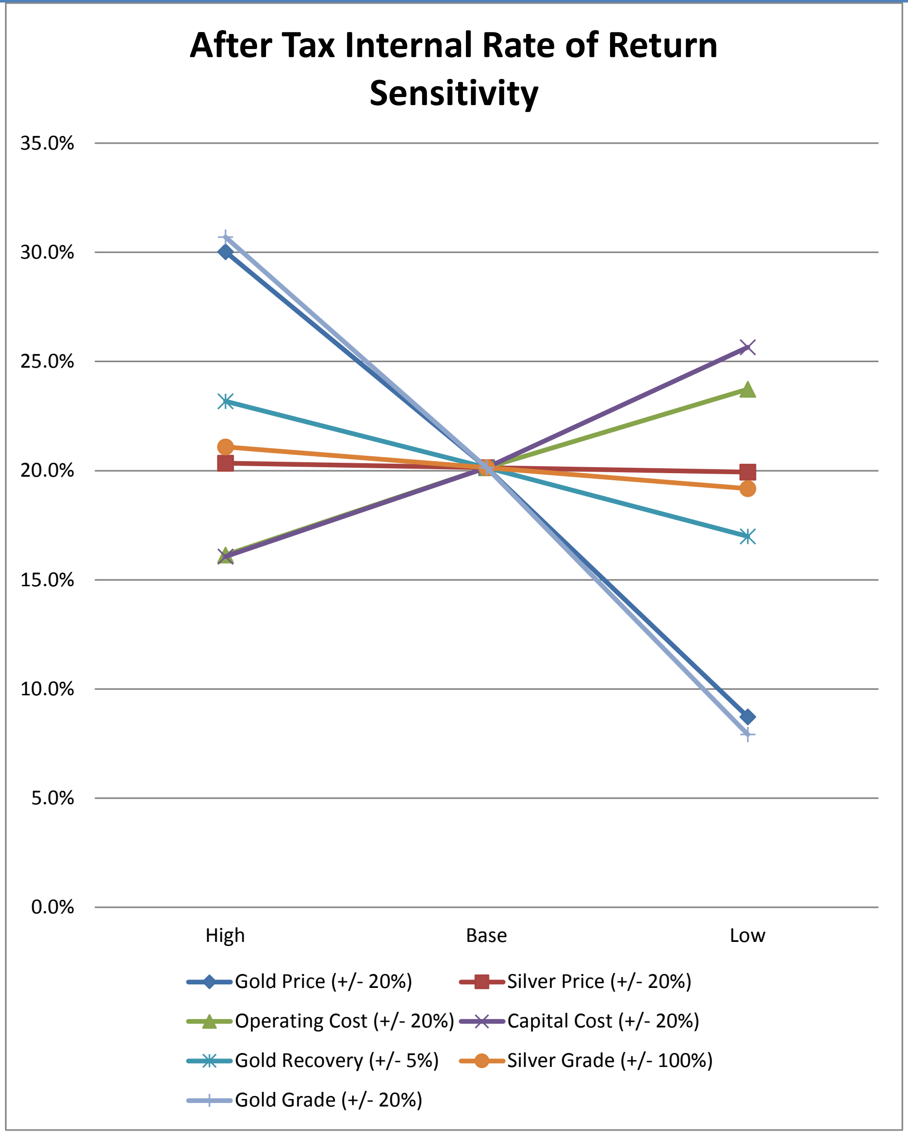
Silver is a by-product for this project and it has been assumed to have a grade of 1.5 x the grade of gold. This assumption is based on the assays of metallurgical samples. The overall contribution of silver to revenue is small. A silver price of $20 / troy ounce was used within the financial analysis.

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

</td></tr>
</table>

Criteria	JORC Code Explanation	Commentary
		- 1,000kg $0.155 per oz Also, it was assumed that the doré bars were 95% pure with minimal or no deleterious elements. The payable metal percentages after refining are: gold 99.95% and silver 99.00%. All economic analysis are in 4th Quarter 2014 USD.
Market assessment	- *The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future.* - *A customer and competitor analysis along with the identification of likely market windows for the product.* - *Price and volume forecasts and the basis for these forecasts.* - *For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.*	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 The market for gold dore is well-established. Market predictions and discussions for gold are beyond the scope of this document. The impacts of gold price volatility on the mine plan and process operation are are well understood. The Competent Persons are not aware of an forward sales or hedging contracts for Haile metal production.
Economic	- *The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc.* - *NPV ranges and sensitivity to variations in the significant assumptions and inputs.*	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 presented on Figure 1

Criteria	JORC Code Explanation	Commentary
		 Figure 1: Sensitivity Analysis

Criteria	JORC Code Explanation	Commentary
Social	- *The status of agreements with key stakeholders and matters leading to social licence to operate.*	The project endured rigorous permitting reviews on the federal, state, and local levels. At each step of this process, the public was afforded the opportunity to participate in the review. The main permit to construct and operate was the Environmental Impact Statement (EIS) and Record of Decision (ROD). During this review and evaluation, Nation to Nation consultation with Native Americans occurred. All required permits from Lancaster County, the State of South Carolina, and the U.S. Government have been obtained and the project is legally proceeding with construction at this time.
Other	- *To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves:* - *Any identified material naturally occurring risks.* - *The status of material legal agreements and marketing arrangements.* - *The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.*	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. The overall contribution of silver to revenue however is small. - *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 angles, 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 Mineralisation* - There is known inferred mineralisation within the bounds of the reserve that is not included as reserves. If this mineralisation is converted to reserves the

Criteria	JORC Code Explanation	Commentary
		available ore tonnage may go up and the amount of waste (overburden) that will need to be handled will be reduced.
Classification	• *The basis for the classification of the Ore Reserves into varying confidence categories.* • *Whether the result appropriately reflects the Competent Person's view of the deposit.* • *The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).*	The Proved Ore Reserve is a sub-set of Measured Mineral Resources, and the Probable Ore Reserve is derived from Indicated Mineral Resources. Inferred Mineral Resource material has not been included in the Ore Reserves. Inferred Resource is treated as waste in the mine plan and Ore Reserve. It is the opinion of the Competent Person for Ore Reserve estimation that the Mineral Resource classification adequately represents the degree of confidence in the orebody.
Audits or reviews	• *The results of any audits or reviews of Ore Reserve estimates.*	Runge Pincock Minarco completed a review of the Haile project in July 2014. They considered the Feasibility Study and proposed development plans supplemented by a subsequent resource estimate, updated cost structure, and EIS to be acceptable for the proposed production goals and future output from the Haile Gold Project.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* • *Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material*	The accuracy and corresponding confidence in the mineralisation is addressed based on both qualitative and quantitative means. The ore reserve estimate is sufficiently accurate such that 2/3 of the Ore Reserve is considered to be "proven" ore. Drill hole spacing is sufficiently close that continuity between sampling points is assured. Gold deposits have higher levels of grade uncertainty than other metals deposits due to the high coefficients of variation. Short term variability will be an ongoing condition in the mine operation. However, mine plans open sufficient room so that the probability of finding sufficient ore in any time frame is maximized. One would expect monthly variations in predicted versus actual grade to be in the range of 10 to 15% based on the experience of other gold deposits. Annual production predictions versus actual performance will be much closer with variances less than roughly 5%. The key to accuracy in the ore reserve is the high number of sample assays, with closely spaced and well distributed locations. Mine Planning and subsequent engineering cost estimation has been done to a level of accuracy that supports a 15% contingency. That contingency is indicative of the high level of confidence in the Ore Reserve and the project in general.

Criteria	JORC Code Explanation	Commentary
	impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • *It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	

Section 5 Estimation and Reporting of Diamonds and Other Gemstones

(Criteria listed in other relevant sections also apply to this section. Additional guidelines are available in the ‘Guidelines for the Reporting of Diamond Exploration Results’ issued by the Diamond Exploration Best Practices Committee established by the Canadian Institute of Mining, Metallurgy and Petroleum.)

[Section 5 is not applicable to the Haile Gold Mine].