NI 43-101 Technical Report
Haile Gold Mine
Lancaster County, South Carolina

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Report Prepared for

OceanaGold Corporation

Report Prepared by

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Table of Contents

List of Tables

List of Figures

Appendices

1 Summary

This National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) was prepared for OceanaGold Corporation (OceanaGold) to a feasibility study (FS) level by SRK Consulting (U.S.), Inc. (SRK) on the Haile Gold Mine (Haile or Project). This report includes both open pit and underground mining components and a single economic analysis based on open pit and underground reserves as of December 31, 2021.

1.1 Property Description and Ownership

The Haile Gold Mine is 100% owned and operated by OceanaGold. Haile is located 5 km northeast of Kershaw in southern Lancaster County, South Carolina. The Haile property site is 30 km southeast of Lancaster, the county seat, and 80 km northeast of Columbia, the state capital of South Carolina. Haile Gold Mine Inc. (HGM) is a wholly owned subsidiary of OceanaGold Corporation (OceanaGold). As of December 31, 2021, HGM owns a total of 11,003 acres in South Carolina with no royalties. Of this total, 4,522 acres are within the mine permit boundary. Proposed expansion in the Supplemental Environmental Impact Statement (SEIS) will increase the mine property to 5,469 acres.

1.2 Geology and Mineralization

Haile is situated within the northeast-trending Carolina Terrane, also known as the Carolina Slate Belt, which hosts the past-producing Ridgeway, Brewer and Barite Hill gold mines in South Carolina. Haile is the largest gold deposit in the eastern USA. The Haile district consists of nine gold deposits within a 3.5 km by 1 km area. Haile occurs within a variably deformed ENE-trending structural zone at or near the contact between metamorphosed Neoproterozoic volcanic and sedimentary rocks. Haile is hosted in laminated siltstones and minor volcanic rocks of the Upper Persimmon Fork Formation and is dissected by barren NNW-striking diabase dikes. Deformation includes brittle and ductile styles with ENE-trending foliation, faults, brecciation, and isoclinal folds. Sedimentary rocks are folded within an ENE-trending anticlinorium with a steep SE limb and a gentle NW limb. Gold mineralization is assumed at ~549 Ma based on closely associated molybdenite dated using Re-Os (Mobley et al., 2014), which postdates peak volcanism and predates the Alleghanian greenschist facies metamorphic deformation. Pressure shadows around pyrite grains, stretched pyrite and pyrrhotite grains, and flattened hydrothermal breccia clasts indicate that there has been deformation subsequent to sulphide mineralization. These observations are consistent with either pre- or syn-tectonic gold mineralization. Interpreted timing of gold mineralization coincides with a major tectonostratigraphic change from intermediate volcanism and tuffaceous to epiclastic sedimentation to basinal turbiditic sedimentation. Quartz-sericite-pyrite alteration is overprinted by regional greenschist facies metamorphism with carbonate-chlorite-pyrite alteration. Haile is currently interpreted as a low-sulfidation, sediment-hosted, disseminated, epithermal gold deposit.

1.3 Status of Exploration; Development and Operations

Resource definition drilling at Haile by Romarco Minerals and OceanaGold has increased the resources more than five-fold since 2007. Reserve growth has resulted from exploration, conceptual 3D modeling, deep drilling of a previously underexplored gold system and higher gold prices. This is exemplified by pre-development of the Horseshoe underground Reserve in 2017 (0.44 Moz) and announcement of an initial Inferred Resource at Palomino in 2020 (0.6 Moz). In-house core drilling continues by OceanaGold at about 15 km per year, focused on pit design margins and high-grade underground targets proximal to the sedimentary-volcanic contact. Underground development of the Horseshoe deposit in 2022 will provide better located underground drill stations for converting Horseshoe Inferred resource and other targets along the prospective one km long Horseshoe-Palomino trend.

1.4 Mineral Processing and Metallurgical Testing

Samples of ore were collected by Haile Gold Mine, Inc. (HGM) for metallurgical testing, which indicate that the ore will respond to flotation and direct agitated cyanide leaching technology to extract gold.

Comminution test work on mineralized samples was performed by RDi, and ALS Limited (ALS). Tests included Bond work indices, Sag Mill Comminution (SMC), and JK Drop Weight impact testing. The results of the test work were used for additional power modeling to predict circuit throughputs with a modified SAG–Ball Mill-Pebble Crusher (SABC) grinding circuit that was subsequently commissioned in 2018.

Laboratory testing on ore composite samples demonstrated that the mineralization was readily amenable to flotation and cyanide leaching process treatment. A conventional flotation and cyanide leaching flow sheet can be used as the basis of process design.

The relative low variability of test work indicates that the different mineralized zones are similar in terms of ore grindability, mineral composition, and flotation and cyanide leaching response. Overall gold recovery will be in the range of 65% to 92% dependent primarily on head grade to the mill and less related to which zone the ore is mined from.

The data developed in the test programs has been used to establish a relationship between overall gold recovery and head grade. Operating experience and metallurgical development programs have indicated this relationship is valid and is expected to be slightly exceeded over the LoM.

1.5 Mineral Resource Estimate

The Mineral Resources at Haile comprise both open pit and underground resources. Separate block models were generated for the open pit and underground areas. Mineral Resources are reported using a gold price of US$1,700/oz and are inclusive of Mineral Reserves.

At this time, there are no unique situations in relation to environmental, socio-economic or other relevant conditions that would put the Haile Mineral Resource at a higher level of risk than any other developing resource within the United States, or that would materially affect the Mineral Resource estimates. Whilst there may be delays in receiving the permits, such delays are not expected to materially impact the reported resource estimates.

1.5.1 Open Pit Mineral Resource Estimate

Drillhole data available as of October 2020 were included in the HA1220OLM open pit resource estimate. The assay coverage for gold covers all core and RC drilling. However, the collection of silver, carbon and sulfur assay data has largely been retrospective and is significantly sparser than for gold. Sulfur and carbon data are primarily used for the prediction of waste classification types. Sulfur grades are also used for mill feed sulfur estimates. Silver grade estimates are provided for metallurgical considerations (carbon stripping and electro-winning) as well as for revenue estimation, albeit silver is a minor contributor to revenue.

Gold estimation was constrained within implicitly modeled grade shells, approximating a 0.065 g/t gold indicator. Metasediment/metavolcanic contacts were not used to constrain gold estimation. This approach is supported by relationships between mineralization and bedding observed in open pit grade control sample data. Post-mineralization dikes were assigned zero grade after gold estimation.

Grades were estimated into 10mE x 10mN x 5mRl blocks using 2.5 m bench composites. Grade estimation was done in Vulcan software, using Multiple Indicator Kriging (MIK) to produce E-Type estimates for gold. MIK is well suited to estimating positively skewed grade distributions. Top caps of 50 g/t Au were used to temper mean grades for the top indicator class threshold.

Ordinary kriging was used for silver, sulfur and carbon estimates, given the lower number of data. The Mineral Resources are classified as Measured, Indicated and Inferred Mineral Resources, based primarily on drillhole spacing but guided also by kriging variance and slope regression.

The open pit Mineral Resources are shown in Table 1-1 and open pit stockpiles in Table 1-2.

Table 1-1: Open Pit Mineral Resources as of December 31, 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Measured	2.68	1.30	0.11	2.54	0.22
Indicated	43.0	1.55	2.14	2.41	3.33
Measured & Indicated	**45.6**	**1.54**	**2.25**	**2.42**	**3.55**
Inferred	5.7	1.0	0.2	1.3	0.2

Source: OceanaGold, 2022

- Cut-off grade for primary mineralization is 0.45 g/t and 0.55 g/t Au for oxide, based on a gold price of US$1,700/oz.
- Open pit resource is reported within a US$1,700/oz optimized shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The open pit Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

Table 1-2: Open Pit Stockpiles as of December 31, 2021

Type	Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Stockpiles	Measured	1.83	1.10	0.06	1.10	0.06
	Indicated	0.0	0.00	0.0		
	Measured & Indicated	**1.83**	**1.10**	**0.06**	**1.10**	**0.06**
	Inferred	0.0	0.0	0.0		

Source: OceanaGold, 2022

- Mineral Resources include Mineral Reserves

1.5.2 Underground Mineral Resource Estimate

Horseshoe

The Horseshoe resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimate is supported by drilling and sampling current to April 22, 2020.

Gold estimation was constrained within implicitly modeled grade shells, approximating a 1.0 g/t gold indicator. The grade shells were constructed using interpreted trend planes of mineralization. Two dominant zones of mineralization were identified, an upper moderately northwest dipping zone and a lower near vertical zone.

Post-mineralization dikes were assigned zero grade. Metasediment/metavolcanic contacts were not used to constrain gold estimation.

Gold grades were estimated with Vulcan™ modeling software into 10mE x 10mN x 10mRl blocks (sub-blocked to 5mE x 5mN x 5mRl) using Ordinary Kriging with 3 m composites. In situ dry bulk densities, based upon core analyses, were assigned by rock type.

The Mineral Resources reported for the Horseshoe deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drillhole spacing but also considering geological complexity.

The Horseshoe Mineral Resource statement is based on the Ordinary Kriging (OK) model as presented in Table 1-3. A CoG of 1.35 g/t Au has been applied without mine design constraint because the Measured and Indicated resources broadly correspond with the mine design. The CoG assumes underground mining methods and is based on a gold price of US$1,700/oz, and a gold recovery of 88%. No mining dilution has been applied.

Table 1-3: Horseshoe Underground Mineral Resource Statement as of December 31, 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Oz (Moz)
Measured	0.0	0.00	0.00
Indicated	3.2	5.05	0.52
Measured & Indicated	**3.2**	**5.05**	**0.52**
Inferred	2.0	4.6	0.3

Source: OceanaGold, 2022

- Cut-off grade 1.35 g/t Au based on a gold price of US$1,700/oz.
- No mining dilution applied.
- Spatially constrained by a 1 g/t Au indicator shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

Palomino

The Palomino resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling completed in 2021 although final assays were received on January 17, 2022. The effective reporting date is recorded as 31 December 2021 to match the site-wide Haile mining depletion date, no mining has occurred at Palomino.

Gold estimation was constrained within implicitly modeled grade shells, approximating a 0.8 g/t gold indicator. Post-mineralization dikes were assigned zero grade. Metasediment / metavolcanic contacts were not used to constrain gold estimation.

Gold grades were estimated into 10 m E x 10 m N x 10 m RL parent blocks with Vulcan™ modeling software using Ordinary Kriging on 3 m composites. Sub-blocking was to 2.5 m E x 2.5 m N x 2.5 m RL for better volumetric determination, estimation was into the parent block. A probability Kriging approach was used within the 0.8 g/t grade shell, with a 0.3 g/t Au indicator employed to account for a lower bimodal population. On a block-by-block basis, the grade and probability of the higher and lower grade indicators were estimated by Ordinary Kriging and weight-averaged to generate a final block grade. Densities, based upon core analyses, were assigned by rock type.

The Mineral Resources reported for the Palomino deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drillhole spacing and geological understanding but guided also by kriging variance and slope regression.

The Palomino Mineral Resource statement is presented in Table 1 4. The reported resource is constrained within a conceptual stope design based on a gold price of US$1,700/oz, approximating a 1.39 g/t cut-off. Due to the diffuse nature of the grade boundaries, all unclassified material within the conceptual design was assigned zero grade for the purposes of reporting.

Table 1-4: Palomino Underground Mineral Resource Statement as of December 31, 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Oz (Moz)
Measured	0.0	0.00	0.00
Indicated	2.3	2.79	0.20
Measured & Indicated	**2.3**	**2.79**	**0.20**
Inferred	3.6	2.3	0.3

Source: OceanaGold, 2022
- Cut-off grade 1.39 g/t Au based on a gold price of US$1,700/oz.
- Mineral Resources are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

1.5.3 Combined Open Pit and Underground Resource Estimate

Table 1 5 presents the combined open pit, stockpiles, and underground resource statement for the Haile Property.

Table 1-5: Haile Combined Open Pit, Stockpiles and Underground Resource Statement as of December 31, 2021

Type	Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Open Pit	Measured	2.68	1.30	0.11	2.54	0.22
	Indicated	43.0	1.55	2.14	2.41	3.33
	Measured & Indicated	**45.6**	**1.54**	**2.25**	**2.42**	**3.55**
	Inferred	5.7	1.0	0.2	1.3	0.2
Stockpiles	Measured	1.83	1.10	0.06	1.10	0.06
	Indicated	0.0	0.00	0.0		
	Measured & Indicated	**1.83**	**1.10**	**0.06**	**1.10**	**0.06**
	Inferred	0.0	0.0	0.0		
Underground	Measured	0.0	0.00	0.00		
	Indicated	5.48	4.12	0.73		
	Measured & Indicated	**5.48**	**4.12**	**0.73**		
	Inferred	5.6	3.1	0.6		
Combined	Measured	4.51	1.22	0.18		0.28
	Indicated	48.5	1.84	2.87		3.33
	Measured & Indicated	**52.9**	**1.79**	**3.04**		**3.61**
	Inferred	11	2.0	0.7		0.2

Source: OceanaGold, 2022

- Cut-off grades for the open pit, Horseshoe underground and Palomino underground are 0.45 g/t / 0.55 g/t (primary / oxide), 1.35 g/t and 1.39 g/t Au respectively, based on a gold price of US$1,700/oz.
- No cut-off applied to reported mined stockpiles.
- Open pit resource is reported within a US$1,700/oz optimized shell. Palomino is constrained within a conceptual stope design and Horseshoe underground is spatially constrained by a 1 g/t Au indicator shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

1.6 Mineral Reserve Estimate

1.6.1 Open Pit Mineral Reserves Estimate

The geological model was used without modification for open-pit optimization, as the block size in the model matched the SMU size of 10 m x 10 m x 5 m. This block size is currently considered appropriate for the backhoe loading units operating at Haile.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results. The optimization included Measured, Indicated and Inferred Mineral Resource categories with a gold price of US$1,500/oz Au. Silver was not assigned economic value in the optimization. Subsequent to pit optimization, Inferred material (approximately 10%) within the reserve pit was treated as waste and given a zero-gold grade. Dilution and ore recovery have been applied during the mine scheduling process to account for mineralized material mined by Face Shovel excavators. This has limited impact on the Mineral Reserve, with an effective global dilution of 1% and ore recovery of 98.9%.

The overall pit slopes (interramp angle slopes) used for the design are based on operational level geotechnical studies and range from 32°to 45°. This includes a 5° allowance for ramps and geotechnical catch benches.

Measured Mineral Resources were converted to Proven Mineral Reserves and Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining pit shapes created during the mine design process.

The open pit mine design process results in open pit mining reserves, including stockpiles, of 42.0 Mt with an average grade of 1.58 g/t. The Mineral Reserve statement, as of December 31, 2021, for the Haile Open Pit is presented in Table 1-6.

Table 1-6: Haile Open Pit Mineral Reserves Estimate as of December 31, 2021

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	4.4	1.26	1.98	0.18	0.28
Probable*	37.6	1.62	2.44	1.96	2.95
Proven + Probable	42.0	1.58	2.39	2.14	3.23

* Includes 1.8 Mt of stockpile material grading 1.1 g/t Au and 1.1 g/t Ag
Source: OceanaGold, 2022

- Reserves are based on a US$1,500/oz Au gold price.
- Open pit reserves are stated using a 0.5 Au g/t cut-off for primary and 0.6 g/t Au cut-off for oxide material.
- Open pit reserves include variable dilution and mining recovery that has been applied in the mine schedule to the upper benches of each pit stage to account for assumed mining by face shovel excavator in these areas.
- Metallurgical recoveries are based on a recovery curve for primary material of (1-(0.2152*Au grade^-0.3696)). with +2.5% uplift applied to material > 1.7 g/t Au. Recovery for oxide material is applied at 67%. This equates to an overall average recovery of 81%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- The open pit Mineral Reserves were estimated by Gregory Hollett P.Eng (EGBC) of OceanaGold, a Qualified Person.

OceanaGold knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the open pit Mineral Reserve estimate. While there may be delays in receiving the permits, such delays are not expected to materially impact the reported reserve estimates.

1.6.2 Underground Mineral Reserves Estimate

The Project is currently being mined as an open pit mine. Mineralization extends down at depth and outside of the pit extents. This mineralization, not mined by the open pit, is assessed as an underground mine and is referred to as the Horseshoe deposit.

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with ramp access is used. The stopes will be 20 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. Cemented rock fill (CRF) will be used to backfill the stopes. There will be an opportunity for some non-cemented waste rock to be used in select stopes based on the mining sequence. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes.

The deposit has been divided into three production areas. The uppermost block extends from approximately the 930 m elevation to 1,030 m elevation and includes four stoping levels that will be mined bottom up. The mid-block extends from approximately the 850 m elevation to the 930 m elevation and includes three stoping levels that will be mined bottom up. The uppermost level in this block includes mining out of the sill. The lowest block extends from approximately the 780 m elevation to the 930 m elevation. It includes three stoping levels that are also mined bottom up. The uppermost level in this block also includes mining out of the sill. A lower extraction has been used for sill levels.

The underground mine design process resulted in underground mining reserves of 3.4 Mt (diluted) with an average grade of 3.78 g/t Au. The Mineral Reserve statement, as of December 31, 2021, for the Haile Horseshoe Underground is presented in Table 1-7.

This estimate is based on a mine design cut-off of 1.53 g/t Au. The numbers include a 94% to 100% mining recovery based on type of opening (e.g., stope, development) to the designed wireframes in addition to a 0% to 10% unplanned dilution using zero grade for dilution.

Table 1-7: Haile Horseshoe Underground Reserves Estimate as of December 31, 2021

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	-	-	-	-	-
Probable	3.42	3.78	-	0.42	-
Proven + Probable	3.42	3.78		0.42	-

Source: SRK, 2022

- Reserves are based on a gold price of US$ 1,500/oz. Metallurgical recoveries are based on a recovery (1-(0.2152*Au grade^-0.3696)) + 0.025 that equates to an overall recovery of 88%.
- Underground reserves are stated using a 1.53 g/t Au cut-off. The reserve estimate is based on a mine design using an elevated cut-off grade of 1.67 Au g/t, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.37 g/t Au and an incremental development cut-off grade of 0.46 g/t Au.
- Mining recovery ranges from 94% to 100% depending on activity type. Sill levels use a 75% recovery. Mining dilution is applied using zero grade. The dilution ranges from 2% to 10% depending on activity type.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP of SRK, a Qualified Person.

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the underground Mineral Reserve estimate. While there may be delays in receiving the permits, such delays are not expected to materially impact the reported reserve estimates.

1.6.3 Combined Open Pit and Underground Reserves Estimate

Table 1-8 presents the combined open pit and underground resource statement for the Haile Property.

Table 1-8: Reserve Statement for OceanaGold's Haile Gold Mine as of December 31, 2021

Type	Category	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au Contained (Moz)	Ag Contained (Moz)
OP	Proven	4.4	1.26	1.98	0.18	0.28
	Probable*	37.6	1.62	2.44	1.96	2.95
	Proven + Probable	42.0	1.58	2.39	2.14	3.23
UG	Proven		-	-		-
	Probable	3.42	3.78	-	0.42	-
	Proven + Probable	3.42	3.78	-	0.42	-
OP + UG	Proven	4.4	1.3	2.0	0.2	0.3
	Probable	41.0	1.8	2.2	2.4	2.9
	Proven + Probable	45.4	1.8	2.2	2.6	3.2

*Includes1.8 Mt of stockpile material grading 1.1 g/t Au and 1.1 g/t Ag
Source: OceanaGold/SRK, 2022

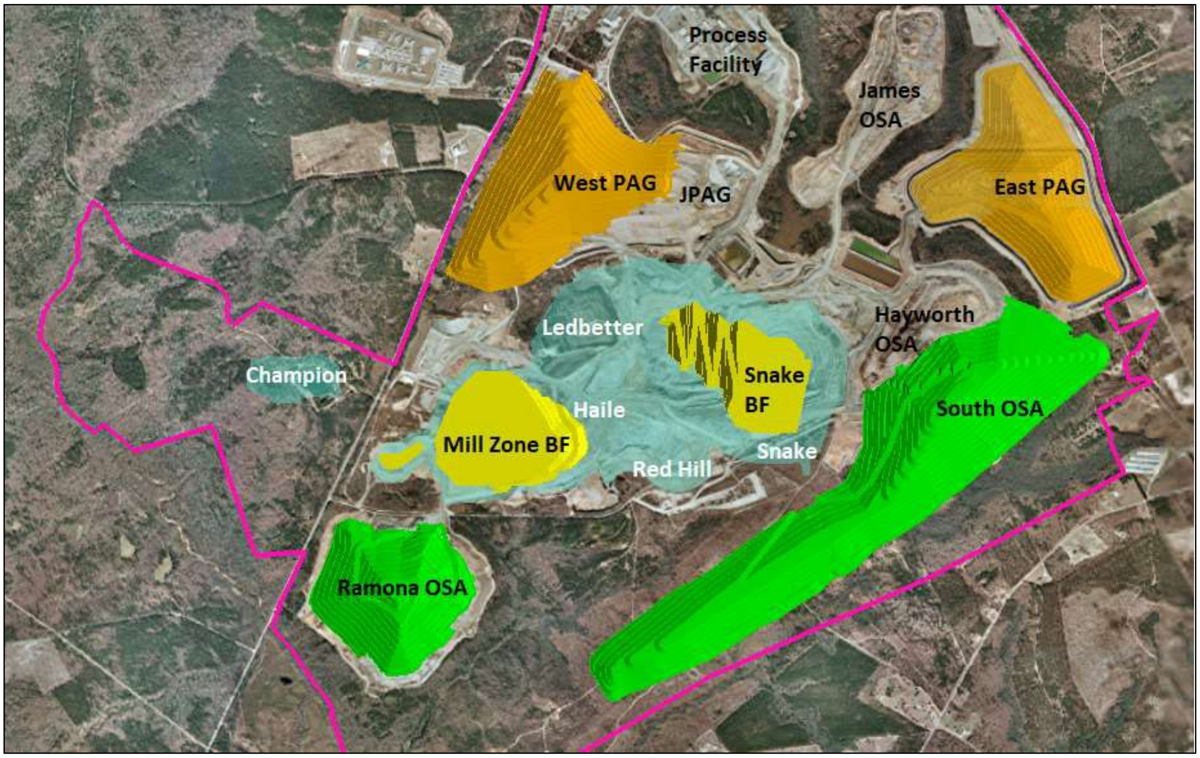
- Mineral Reserves are based on a gold price of US$ 1,500/oz.
- Metallurgical recoveries are based on a recovery curve for primary material of (1-(0.2152*Au grade^-0.3696)) with +0.025 uplift applied to material > 1.7 g/t Au. Recovery for oxide material is applied at 67%. This equates to an overall recovery of 81% for the open pit material and 88% for the underground material.
- Open pit reserves are stated using a 0.5 g/t Au cut-off for primary and 0.6 g/t Au cut-off for oxide material. Open pit reserves include variable dilution and mining recovery that has been applied in the mine schedule to the upper benches of each pit stage to account for assumed mining by face shovel excavator in these areas.
- Open pit reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Underground reserves are stated using a 1.53 g/t Au cut-off. The reserve estimate is based on a mine design using an elevated cut-off grade of 1.67 Au g/t, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.37 g/t Au and an incremental development cut-off grade of 0.46 g/t Au. Mining recovery ranges from 94% to 100% depending on activity type. Sill levels use a 75% recovery. Mining dilution is applied using zero grade. The dilution ranges from 2% to 10% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The open pit Mineral Reserves were estimated by Gregory Hollett P.Eng (EGBC) of OceanaGold, a Qualified Person. The underground Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP of SRK, a Qualified Person.

1.7 Mining Methods

1.7.1 Open Pit Mining Methods

Haile is currently being mined using conventional open pit methods. The open pit is located in gently undulating topography intersecting natural drainages that will require diversion to withstand high-volume rainfall events during the summer months. Overburden type is calculated from blasthole sulfur and carbon assays and informs the routing and placement of materials. Red potentially acid generating (PAG) material is sent to geomembrane-lined facilities where the material is placed in lifts and compacted by haulage trucks. Yellow PAG can be stored in a lined facility or below a prescribed water table within pits. Yellow material in-pit will be mixed with lime before placement in the pit void.

The open pit that forms the basis of open pit reserves and life of mine (LoM) production schedule is approximately 2.5 km from east to west, 1.25 km north to south with a maximum depth of 370 m. The design consists of multiple pushbacks with ramp locations targeting saddle points between the pit bottoms and also acting as catch benches for geotechnical purposes. Each bench has at least one ramp for scheduling. Generally, the number of benches mined within a pit phase within a given year fall below the one bench per month target bench sinking rate. Figure 1-1 illustrates the site layout and final pit design.

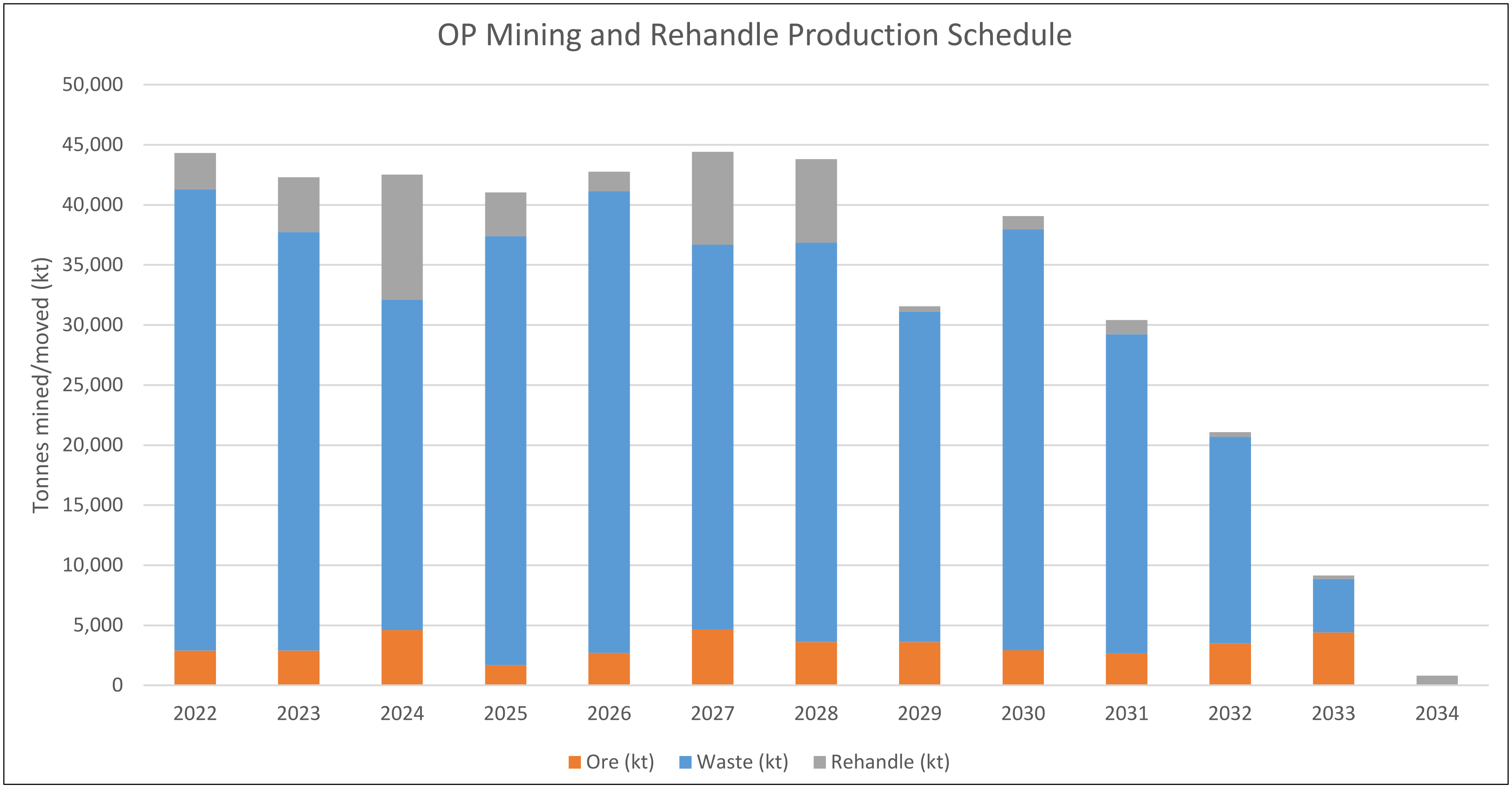


Yellow inside pits is backfill material.
Source: OceanaGold, 2022

Figure 1-1: Final Pit Design and Site Layout

Open-pit ore production rates are targeted to balance processing requirements with underground production and stockpile balance. Open pit ore processed averages approximately 3 Mtpa while the underground is operating, increasing to 3.8 Mtpa once underground mining has been completed. Total open-pit fleet material movement, including rehandle, averages between 40 Mtpa and 45 Mtpa, reducing toward the end of the mine life.

The mine production schedule (Mined + Rehandle) is summarized in Figure 1-2.



Source: OceanaGold, 2022

Figure 1-2: LoM Production Schedule

The open pit loading and hauling equipment fleet consists of hydraulic excavators (Komatsu PC3000 and PC4000 models) and rigid frame haul trucks (CAT 785 and Komatsu 730). Blasthole drilling and wall control drilling is performed with a fleet of Sandvik DR410i and Epiroc D65 drills. Typical ancillary equipment, including track dozers, wheel dozers, motor graders and water trucks support the mining operation. Table 1-9 shows the major equipment required annually to achieve the mine schedule.

Table 1-9: Major Equipment Required to Achieve the Mine Schedule

Fleet	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
PC4000 Excavator	2	2	2	2	2	2	2	2	2	2	1	1	0
PC3000 Excavator	1	1	1	1	1	1	1	1	1	1	1	1	0
Cat 993 Loader	2	2	2	2	1	1	1	1	1	1	1	1	1
Cat 785 Trucks	3	1	3	3	1	2	1	0	1	1	1	1	1
Komatsu 730 Trucks	19	15	17	18	15	16	15	11	14	12	10	6	0
Sandvik DR410i Drill	4	4	4	4	4	4	4	4	4	4	4	3	0
Epiroc D65 Drill	4	3	2	3	4	4	4	4	4	3	3	2	0

Source: OceanaGold, 2022

1.7.2 Underground Mining Methods

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality of data for feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that longhole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented rockfill.

The design has been laid out using empirical design methods based on similar case histories. The stability of the design has been checked with 3D numerical stress-strain models of the workings which included consideration for mine-scale faulting. The modeling results confirm that stopes and access drifts are predicted to remain stable during active mining.

The 2020 updated underground design has conducted more detailed numerical modeling of the revised mine plan. These analyses used rock mass properties consistent with the geotechnical database, considering that no new data has been collected since 2017. Additional details of sill pillar extraction have been considered and a series of detailed ground support drawings have been developed for the underground workings and infrastructure. A detailed ground support design has been developed for the first portal and can be adopted for the second and third portal based on experience with ground conditions from the first portal. The updated design includes the main decline between the footwall of the Snake Pit and the Horseshoe orebody and the ground support design has considered mining induced stresses as well as the presence of diabase dikes, especially at locations where the main decline crosses over ventilation declines with a sill in between.

Although the base case design considers cemented waste rock fill material for backfilling stopes, the option for using pastefill is still under consideration. There are operational and geotechnical advantages for using pastefill. However, until questions of environmental impact of cement encapsulated tailings waste can be fully addressed, the backfill design remains the use of cemented waste rock backfill of a specified 1 MPa strength. Pastefill would have a similar strength requirement, however, requires additional testing for cement contents.

Mine Design

Stope optimization was completed on prior versions of the model. Results from those prior runs were used for comparison during the mine design process. For the design, vertical slices were created through the orebody along 2 m strike length intervals. The model was then interrogated, filtered on cut-off, and then the slices were combined to create minable stopes. The mining method is transverse sublevel open stoping with cemented rockfill (CRF). Stope sizes, used in the optimization were 25 m high, 20 m wide, and varying stope length based on geotechnical considerations.

Each stope has a 4.5 m x 5 m access located at the bottom of the stope. Top accesses (also 4.5 m x 5 m) are designed to give access to stopes on the next level and to allow for backfilling. The stopes are drilled from the top and rings are blasted from the end of a stope toward the footwall access. The blasted material is remotely mucked from the stope access. A primary/secondary stoping sequence will be used. The stope accesses are connected to a level access located in waste material. The level accesses connect to the main ramp, which is located in the footwall. Each level access is connected to the ventilation system. Ore will be remotely mucked from the bottom stope access using a 14.9-t LHD and loaded into 51-t trucks for haulage to surface.

The underground mine is accessed via a decline from the surface. The decline portal is located on an open pit bench approximately 80 m below the natural surface. Two ventilation drift portals are also located on an open pit bench.

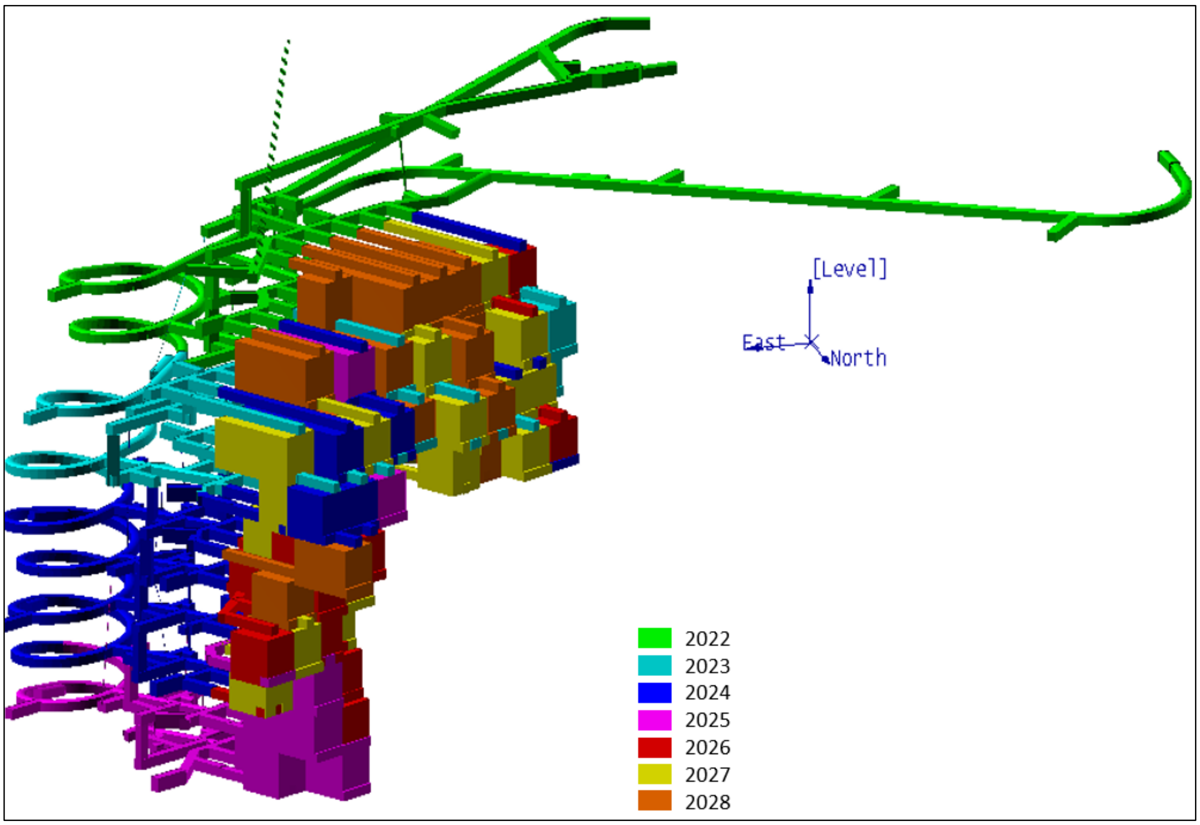
The Horseshoe underground mine production schedule is based on the productivity rates developed from first principles which were adjusted based on benchmarking and the experience of OceanaGold personnel. The schedule was completed using Deswik scheduling software and is based on mining operations occurring 365 days/year, 7 days/week, with two 12 hour shifts each day. A production rate of approximately 2,000 t/d was targeted with ramp-up to full production as quickly as possible.

The underground permitting process is expected to be complete in early 2022.Underground portal development is scheduled to begin in April 2022. The Snake Pit has progressed sufficiently to allow access to the portal location. First production from the stopes is scheduled to occur in September 2023 and will last through May 2028 based on the current Mineral Reserves. Table 1-10 shows the yearly production schedule and Figure 1-3 shows the mine production schedule colored by year.

Table 1-10: Horseshoe Mine Production Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Au (g/t)	Waste Tonnes (kt)	Backfill Volume (m^3)
2022			180.0	
2023	190.9	4.07	309.4	44,412
2024	735.9	4.31	181.6	253,804
2025	748.6	3.48	133.1	268,636
2026	738.8	4.64	13.0	276,982
2027	739.2	3.18	3.4	295,760
2028	262.8	2.23	1.3	122,738
Total	3,416.3	3.78	821.8	1,262,333

Source: SRK, 2022



Source: SRK, 2022

Figure 1-3: Mine Production Schedule Colored by Year

1.7.3 Combined OP and UG Production Schedule

Table 1-11 shows the combined open pit and underground production schedule annually.

Table 1-11: Combined OP and UG Production Schedule

	Year	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	LoM Total
Mining - Open-pit															
Total Material Moved	kt	44,321	42,300	42,516	41,038	42,775	44,408	43,881	31,556	39,081	30,423	21,090	9,154	821	433,362
Total Material Mined	kt	41,278	37,732	32,117	37,393	41,140	36,697	36,859	31,120	37,971	29,220	20,705	8,844	-	391,076
Waste Mined	kt	38,380	34,844	27,524	35,718	38,430	32,019	33,259	27,520	35,051	26,550	17,186	4,425	-	350,907
Ore Mined	kt	2,898	2,888	4,593	1,675	2,710	4,678	3,600	3,600	2,919	2,670	3,519	4,419	-	40,170
Gold Grade Mined	Au g/t	1.7	1.9	1.7	1.1	2.6	1.6	1.8	1.4	0.9	1.0	1.4	1.9	-	1.6
Silver Grade Mined	Ag g/t	2.3	2.3	2.7	2.2	2.6	2.1	2.0	2.5	2.4	2.8	2.6	2.7	-	2.4
Contained Au Mined	Au koz	157	173	250	59	223	243	211	166	80	87	158	266	-	2,073
Contained Ag Mined	Ag koz	216	215	395	121	228	322	234	293	228	238	291	382	-	3,163
Mining - Underground															
Total Material Mined	kt	180	497	918	882	752	745	265	-	-	-	-	-	-	4,238
Waste Mined	kt	180	306	182	133	13	6	2	-	-	-	-	-	-	822
Ore Mined	kt	-	191	736	749	739	739	263	-	-	-	-	-	-	3,416
Gold Grade Mined	Au g/t	-	4.1	4.3	3.5	4.6	3.2	2.6	-	-	-	-	-	-	3.8
Silver Grade Mined	Ag g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Au Mined	Au koz	-	25	102	84	110	76	22	-	-	-	-	-	-	418
Contained Ag Mined	Ag koz	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL Mining															
Total Material Moved	kt	44,501	42,797	43,434	41,919	43,526	45,153	44,145	31,556	39,081	30,423	21,090	9,154	821	437,600
Total Material Mined	kt	41,458	38,229	33,035	38,275	41,892	37,442	37,124	31,120	37,971	29,220	20,705	8,844	-	395,314
Waste Mined	kt	38,560	35,150	27,706	35,851	38,443	32,025	33,261	27,520	35,051	26,550	17,186	4,425	-	351,728
Ore Mined	kt	2,898	3,079	5,329	2,424	3,449	5,417	3,863	3,600	2,919	2,670	3,519	4,419	-	43,586
Gold Grade Mined	Au g/t	1.7	2.0	2.1	1.8	3.0	1.8	1.9	1.4	0.9	1.0	1.4	1.9	-	1.8
Silver Grade Mined	Ag g/t	2.3	2.2	2.3	1.6	2.1	1.9	1.9	2.5	2.4	2.8	2.6	2.7	-	2.3
Contained Au Mined	Au koz	157	198	352	143	333	319	233	166	80	87	158	266	-	2,491
Contained Ag Mined	Ag koz	216	215	395	121	228	322	234	293	228	238	291	382	-	3,163
Processing															
Total Ore Processed	kt	3,507	3,770	3,694	3,682	3,683	3,683	3,697	3,800	3,800	3,800	3,677	3,800	821	45,414
Gold Grade Processed	Au g/t	1.7	1.8	2.6	1.5	2.6	2.5	1.9	1.4	0.9	0.9	1.4	1.7	2.3	1.7
Silver Grade Processed	Ag g/t	2.1	1.9	2.3	1.8	2.0	1.9	1.9	2.5	2.3	2.5	2.5	2.7	2.8	2.2
Gold Recovery	%	83%	84%	87%	82%	88%	86%	85%	82%	76%	76%	81%	84%	82%	84%
Silver Recovery	%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%	70%
Gold Produced/Sold	Au koz	160	181	272	146	273	255	195	142	81	83	130	176	49	2,141
Silver Produces/Sold	Ag koz	168	165	187	147	162	158	160	218	199	210	208	227	51	2,259
Operating Costs															
OP Mining Cost	$/t mined	2.73	2.70	2.89	2.37	2.18	2.41	2.35	2.37	2.18	2.14	2.68	3.59	-	2.49
UG Mining Cost	$/t ore mined	-	155.85	61.48	53.63	45.58	42.01	51.52	-	-	-	-	-	-	56.62
Processing Cost	$/t milled	12.82	12.44	11.75	11.23	11.54	11.34	11.42	11.32	11.24	11.27	11.41	11.11	16.53	11.66
G&A Cost	$/t milled	5.65	5.51	5.43	5.35	4.99	4.87	4.73	4.58	4.48	4.25	4.02	3.39	12.02	4.89

	Year	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	LoM Total
Indirect Costs															
Indirect Costs	$/t milled	7.25	6.80	7.58	6.56	2.92	1.15	1.08	0.25	0.20	0.20	0.20	0.20	0.91	2.79

Source: SRK, OceanaGold 2022

1.8 Recovery Methods

The processing plant continues to utilize the conventional flowsheet developed in the feasibility study comprising:

- Primary Jaw Crushing
- Conventional SABC grinding circuit incorporating flash flotation on the cyclone underflow
- Rougher flotation
- Two stage concentrate regrind with a tower mill followed by an Isamill
- CIL leaching of reground concentrate and flotation tailings
- Carbon stripping, electrowinning and smelting of bullion
- Cyanide destruction

Additional equipment was installed in some areas of the plant between 2018-2020 to achieve the expanded capacity of up to 4.0 Mtpa. The production plan sees annual milling rates between 3.5 and 3.8 Mtpa being processed.

1.9 Project Infrastructure

The permitted Duckwood Tailings Storage Facility (TSF) will be expanded to store plant tailings by raising the crest height. The existing permitted potentially acid generating (PAG) facilities will be expanded to store additional PAG material. Johnny's PAG (JPAG) Overburden Storage Area (OSA) will be expanded to the west (West PAG). The upper Haile Gold Mine Creek (HGMC) is reporting to the existing Fresh Water Storage Dam (FWSD) and is pumped around the proposed pits in a pipe and released into an unnamed tributary downstream of the pits.

The underground infrastructure required to support underground mining will include general buildings, upgrade and extension of the power lines and water supply. An underground run-of-mine (RoM) pad area will contain the stockpiles, CRF plant, as well as a truck shop and laydown area.

1.10 Environmental Studies and Permitting

In late 2014, HGM received all of the major permits required to begin construction and operation of the mineincluding: Clean Water Act 404 Dredge and Fill Permit, Mine Operation Permit, 401 Water Quality Certification, TSF Dam Permit, North Fork Dam Permit, Air Permit (to construct), and various National Pollutant Discharge Elimination System (NPDES) Permits (wastewater treatment and storm water). Additional permits are in progress to support the planned underground and expansions as presented in this report.

Haile is unique in that it occurs wholly on private land owned by HGM and does not impact federal/public (United States Department of the Interior Bureau of Land Management (BLM) or United States Forest Service (USFS)) lands that would be subject to projected modifications from these surface management agencies

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20.3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the expansion process. OceanaGold will work closely with all key stakeholders to ensure that the permitting of the mine expansion meets all of the federal and state requirements.

1.11 Capital and Operating Costs

Capital Cost

The total LoM capital cost is US$833 million as summarized in Table 1-12. Non-sustaining capital is US$155 million. Sustaining capital cost is US$749 million, the majority of which includes open pit capitalized pre-strip. The remaining balance carries primarily for surface infrastructure, open pit mine equipment replacements and underground development. Capital costs have been estimated with reference to actual experience in undertaking the same or similar works at Haile, quotations from suppliers and estimates provided by consultants with appropriate expertise.

Capital cost estimation is consistent with proposed development programs and ongoing requirements and undertaken to an appropriate level of estimation accuracy. It is likely though that over the life of the mine that actual expenditures will vary, due to modifications, upgrades, introduction of new technology and other unforeseen factors.

Table 1-12: Total Capital Cost Summary (US$000)

Description	Non-sustaining Capex	Sustaining Capex	Total
Land Acquisitions	-	1,887	1,887
Permitting	2,040		2,040
On-Site Exploration Drilling	1,933	1,767	3,700
Op Capitalized Pre-Strip	-	427,660	427,660
OP Mining PP&E	-	99,586	99,586
OP Tech Services PP&E	-	975	975
Pit Dewatering	-	18,550	18,550
Site Works	-	6,200	6,200
PAG Cell Development	-	44,206	44,206
TSF Lift Design	-	108,019	108,019
Underground Mining: Non-Sustaining	79,096	-	79,096
Mill PP&E	-	34,962	34,962
Total Net Capex	**$83,982**	**$748,945**	**832,927$**
Reclamation/Closure(1)	71,072		71,072
Total LoM Net Capex	**$155,054**	**$748,945**	**$903,999**

Source: OceanaGold/SRK, 2022
(1) Captured as Capex in Cashflow

Operating Cost

The total LoM operating cost (excluding capitalized operating cost) is US$1,496 million. Operating costs have been estimated based on historical performance at Haile, supplier quotations, estimates from consultants with appropriate expertise and otherwise estimated internally by appropriately credentialled OceanaGold people. Operating cost estimates include allowance related to performance improvement opportunities identified by site management.

Total life of mine operating costs, and the total RoM operating cost unit rate of US$32.94/t processed are summarized in Table 1-13.

Table 1-13: RoM Operating Cost Summary

Description	US$000's	US$/t mined
OP Mining ($/t rock mined) - Operational Material	545,064	2.49
UG Mining ($/t rock mined)	193,432	44.98
Description	**US$000's**	**US$/t Ore Processed**
Subtotal Mining (Operational Material Only)	738,496	16.24
Processing	529,367	11.64
G&A Cost	222,613	4.90
Refining/Freight Costs	5,732	0.13
Total Operating Costs	**$1,495,827**	**$32.94**

Source: OceanaGold/SRK, 2022

There are several important cost items excluded from the operating cost which are detailed in Table 1-14 that OceanaGold does not consider to be direct operating costs but which the operation does incur. These include payments related to leasing arrangements of the open pit and underground mobile equipment fleets.

Table 1-14: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond	14,800	0.33
Interest Expense - Capital Leases	5,495	0.12
Principal Payment - Capital Leases: Sustaining	49,158	1.08
Principal Payment - Capital Leases: Non-Sustaining	53,832	1.18
Total Non-Operating Costs	**$126,885**	**$2.71**

Source: OceanaGold/SRK, 2022

1.12 Economic Analysis

The Project consists of an operating mine with a mill. The mill is mainly fed by open pit ore. The mill feed will be supplemented with ore from a five-year UG 2,150 t/d max annual capacity high grade operation that begins operations in 2023.

The Project is expected to produce 2.14 million ounces of payable gold over a 13-year mine life at an average rate of 174 koz Au per year during full production years with a run-of-mine (RoM) all-in sustaining cost (AISC) of US$1,055/oz.

The Project is expected to incur sustaining capital in the amount of US$749 million over the modeled life and a non-sustaining capital spend, including rehabilitation costs, of US$155 million for total capital expenditure of US$904 million.

The project cash flow results using the reserve price of US$1,500 / oz gold flat over the life of mine and a 5% discount rate include a pre-tax and after-tax NPV of US$529 million. As a result of significant depreciation pools, loss carryforwards and depletion, the operation is not expected to incur an income tax liability at the reserve price.

OceanaGold provided a consensus view of forward metal pricing (refer section 22.4.2) which averages US$1,630/oz gold over the life of the operation. At these prices and a 5% discount rate the Project is estimated to produce pre-tax and after-tax NPV values of US$764 million and US$740 million, respectively. The modeled prices presented here are substantially below the current market spot prices.

As summary of the model results for both the reserve case and the OceanaGold price case is presented in Table 1-15.

Table 1-15: Indicative Economic Results

Description	**US$000's**	US$000's
Scenario	**Reserve Case Price**	OceanaGold Price
Market Prices		
Gold (US$/oz) Silver (US$/oz	**$1,500** **$18**	~$1,630 ~$21
Payable Gold (koz)	**2,141**	2,141
Revenue		
Gross Gold Revenue Silver By-Product Credit	**3,211,934** **40,260**	3,489,638 45,912
Total Gross Revenue	**$3,252,195**	$3,535,549
Operating Costs		
Total Operating Costs	**($1,603,712)**	($1,603,712)
Operating Margin (EBITDA)	**$1,648,483**	$1,931,838
Taxes		
Income Tax	-	(30,448)
Operating Cash Flow	**$1,648,483**	$1,901,390
Capital		
Total Capital	**($903,999)**	($903,999)
Metrics		
Pre-Tax Free Cash Flow After-Tax Free Cash Flow Pre-Tax NPV @ 5% After-Tax NPV @ 5%	**$744,484** **$744,484** **$529,211** **$529,211**	$1,027,839 $997,391 $764,318 $740,387

Source: SRK

Because the project is operational and is valued on a total project basis and not by an incremental analysis of the UG start up, an IRR value is not relevant in this analysis. In terms of sensitivity, the Project is most sensitive to gold grade and price, followed by operating costs and capital costs.

1.13 Conclusions and Recommendations

1.13.1 Geology and Mineral Resources

Open Pit

Haile 3D geologic models are updated twice per year using Maptek's Vulcan and Seequent's Leapfrog software, based largely on core drilling and pit mapping to reflect controls to mineralization. Geologic domains are used to guide gold grade interpolation. Exploration drilling has been accompanied by an industry standard QA/QC program showing good quality analytical results in terms of precision and accuracy. OceanaGold has conducted extensive core logging resulting in a high-quality geologic model. The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation. Model validation, model peer or external review processes, and production reconciliation processes are regularly undertaken. New lithologic and structural interpretations aided by pXRF alteration zonation and geochemistry are being used to support geological interpretations and new drill targets. Lithology, alteration and mineralogical data are being integrated with geotechnical, overburden storage and metallurgical evaluations to optimize reserve growth, mill recoveries and throughput.

Underground

OceanaGold completed an industry standard exploration drilling program to delineate the Horseshoe mineralization sufficient to support the current mineral resource estimation. The average drill spacing is approximately 25 m x 25 m within the Indicated Mineral Resource and 50 m x 50 m in the Inferred Mineral Resource. Exploration drilling has been accompanied by an industry standard QA/QC program showing good quality analytical results in terms of precision and accuracy. OceanaGold has conducted extensive core logging resulting in a high-quality geologic model. The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation. Model validation, model peer or external review processes, and production reconciliation processes are regularly undertaken. The results of the resource estimation at Horseshoe define an Indicated Mineral Resource of 3.3 Mt at an average Au grade of 4.95 g/t containing 0.53 Moz of gold and an additional Inferred Mineral Resource of 2.1 Mt at an average Au grade of 4.4 g/t containing 0.3 Moz of gold at a 1.35 g/t Au cut-off. The Palomino underground resource is located about 650 m southwest of the Horseshoe reserve. The Palomino Indicated Mineral Resource includes 2.3 Mt at 2.79 g/t Au containing 0.20 Moz of gold and an Inferred Mineral Resource including 3.6 Mt at 2.30 g/t Au for 0.26 Moz of gold at a 1.39 g/t Au cut-off.

1.13.2 Status of Exploration; Development and Operations

OceanaGold will continue to expand resources adjacent to open pit and underground reserves with development drilling. Systematic target generation supported by mapping, drilling, geochemistry and geophysics is expected to yield new discoveries at Haile during the next five to ten years. Interpretation of geophysical data and integration with geology and geochemistry will play an important role. An ‘exploration toolkit’ of diagnostic criteria for Haile-like deposits has been developed to drive exploration for potentially mineable deposits.

1.13.3 Mining and Reserves

Open Pit

The mine block model, geotechnical stability, pit design, phase design, dump design, production schedule and reserve estimation have been completed to a feasibility study standard. The Project confirms a positive cash flow using only Measured and Indicated Resources for the conversion of reserves using a US$1,500/oz gold price. The mine design supports the style and size of equipment selected for operations. While subject to continual improvement, the mine plan implementation will require qualified staff and the integration of all mining and related disciplines for the successful execution of the Project.

The mine operating and capital costs have been estimated from first principles and operational knowledge from current mine operations. The equipment is sized to meet minimum SMU requirements that support the dilution and mine recovery factors while providing bulk earthwork capability for the expected production rates.

The LoM production schedule includes provision for careful control of potentially acid generating overburden and appropriate material handling costs have been included in the mining cost estimate. It is recommended that initiatives as identified in this report relating to open pit mining productivities and costs, grade control, ore loss and dilution and PAG waste definition and classification be pursued, including the development and implementation of appropriate project plans.

Underground

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize cost and grades are not overly diluted. Mine planning work considered revenue for Au and a cut-off grade (CoG) of 1.53 g/t Au was used. A detailed 3D mine design was completed around economically minable areas above cut-off grade.

The underground mine is accessed via a decline from the surface. The decline portal is located on an open pit bench approximately 80 m below the natural surface. Two ventilation drift portals are also located on an open pit bench. And emergency egress system is included in the design.

Tonnage and grades presented in the reserve include dilution and recovery and are benchmarked to other similar operations. Productivities were generated from first principles with inputs from mining contractors, blasting suppliers, and equipment vendors where appropriate. The productivities were also benchmarked to similar operations. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

A production schedule was generated using Deswik software. The schedule targeted 2,000 t/d.

1.13.4 Mineral Processing and Metallurgical Testing

A significant portion of equipment installed at the Haile process plant was designed conservatively enough (that is, with sufficient additional capacity) to readily accommodate expansion. A targeted debottlenecking project has increased plant capacity to a new target capacity of 3.8 Mtpa over the last three years.

No novel, experimental or unproven technologies are used for the Haile process plant. Gold production from conventional grinding, sulfide flotation and fine grinding followed by CIL treatment is able to achieve gold recoveries in excess of 82% for the majority of the sulfide ore treated.

1.13.5 Recovery Methods

There is no effective change to the existing plant recovery methods originally envisaged for the constructed plant or from the expansion activities undertaken. The processing plant has been successfully operating since 2017 and is achieving targeted throughput rates, equipment utilization and gold production. Progressive ramp up has been achieved since mid-2018 to current levels. Ongoing improvement projects and process monitoring will continue to develop operating knowledge to maintain recovery and provide the basis for a first principals cost model to predict unit costs. The existing Haile facilities have an ample site footprint to achieve a capacity of 3.8 Mtpa.

1.13.6 Project Infrastructure

The surface infrastructure required for the increase in open pit production as per the mine plan is straightforward as it is aligned with the current surface infrastructure requirements. The primary changes are increases in size to the main waste storage facilities such as the TSF and PAG OSAs. Major site water management facilities are in place with upgrades in progress.

The surface infrastructure required for the development of the underground Horseshoe deposit is straightforward and relatively minor. Consideration should be given into whether a crusher is required to support the CRF plant.

1.13.7 Environmental Studies and Permitting

Section 26.1.7 summarizes the environmental studies. The mine is currently operating as permitted.

Permits currently held by HGM may be kept, modified, terminated, or replaced during the expansion process. OceanaGold will work closely with all key stakeholders to ensure that the permitting of the mine expansion meets all of the federal and state requirements.

1.13.8 Economic Analysis

The current metal price environment is extremely strong. If prices are forecast to remain elevated for long periods, the project reserves and resources should be updated and fed into an economic model at a revised price deck reflective of the long-term price forecasts.

2 Introduction

2.1 Terms of Reference and Purpose of the Report

This National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) was prepared for OceanaGold Corporation (OceanaGold) to a feasibility level by SRK Consulting (U.S.), Inc. (SRK) on the Haile Gold Mine (Haile or Project). This report includes both open pit and underground mining components and a single economic analysis based on open pit and underground reserves.

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

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014).

2.2 Qualifications of Consultants

The consultants preparing this technical report are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, underground mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

SRK consultants employed in the preparation of this Technical Report have no beneficial interest in OceanaGold. The SRK consultants are not insiders, associates, or affiliates of OceanaGold. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between OceanaGold and SRK. The SRK consultants are being paid a fee for their work in accordance with normal professional consulting practice.

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

- David Carr, BEng Metallurgical (Hons), MAusIMM(CP), OceanaGold Chief Metallurgist is the QP responsible for mineral processing, all of Sections 13 and 17, Section 18.10, the process plant capital and operating costs of section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

- Michael Kirby, PE, MEng Civil, (OceanaGold Environmental Superintendent), is the QP responsible for Environmental and Permitting, Section 20 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, SME-RM (OceanaGold Executive General Manager) is the QP of chapter 19, the open pit and tailings/overburden capital costs portion of section 21, the other/G&A portions of the operating costs in section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Gregory Hollett, BEng Mining Engineering, P.Eng (EGBC) (OceanaGold Group Mining Engineer) is the QP responsible for open pit Mineral Reserves, the open pit portions of Section 15 and 16.3, Sections 16.1, 16.1.1, 16.1.3, 16.1.4, 16.1.6, 16.1.7, 16.1.8, 18.6, the open pit operating cost portions of section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Jonathan Moore, BSc Geology (Hons), MAusIMM(CP), (OceanaGold Chief Geologist), is the QP responsible for open pit and underground Mineral Resources, Sections 4 through 12, 14, 23 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Brianna Drury, BEng Mining, SME-RM (OceanaGold Underground Project Manager), is the QP responsible for underground capital and operating costs portion of Section 21, and portion of Sections 1, 25, and 26 summarized therefrom of this Technical Report.
- Larry Standridge, PE, MSE Geotechnical, (Call & Nicholas Principal Engineer, Geotechnical Engineer) is the QP responsible for open pit geotechnical work, the geotechnical portion of Section 16.1.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Jay Newton Janney-Moore, PE, (NewFields Project Manager II), is the QP responsible for tailing and overburden storage, Sections 18.1, 18.2, 18.3, 18.4, 18.5, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- William Lucas Kingston, P.G. Hydrogeology and Groundwater Management, (NewFields Senior Hydrogeologist) is the QP responsible for hydrogeology, Sections 16.1.9, 16.2.3, the hydrogeological portion of section of 16.1.2, 18.7 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Joanna Poeck, BEng Mining, SME-RM, MMSAQP, (SRK Principal Consultant, Mining Engineer), is the QP responsible for underground Mineral Reserves, all of Sections 2, 3, 24, 27, 28, the underground portions of Section 15 and 16.3, Section 16 opening statements, Sections 16.2, 16.2.1, 16.2.5, 16.2.6, 16.2.7, 16.2.8, 16.2.9, 16.2.11, 18.8, 18.9 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- Matt Sullivan, BEng, RM-SME (SRK Principal Consultant, Mineral Economics), is the QP responsible for technical-economics Sections 22, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- David Bird, MSc., PG, RM-SME, (SRK Principal Consultant, Geochemistry), is the QP responsible for geochemistry, Section 16.1.5, 16.2.4, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- John Tinucci, PhD, PE, (SRK Principal Consultant, Geotechnical Engineer), is the QP responsible for underground geotechnical information, Section 16.2.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

- Brian S. Prosser, PE (SRK Principal Consultant), is the QP responsible for ventilation, Section 16.2.10 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.

2.3 Details of Inspection

Site visits conducted by QPs are summarized in Table 2-1.

Table 2-1: Site Visit Participants

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
Larry Standridge	Call & Nicholas	Geotechnical	November 29 and 30 2017 October 10 and 11 2018 January 14 to 16 2020 May 18-20 2021	Review of existing pit slopes and examination of geotechnical drill core
David Carr	OceanaGold	Metallurgy	Various visits in 2017, 2018 and 2019 March 8 to March 25 2020	Plant Commissioning support and plant investigations Plant Commissioning and ramp-up of operations to 4 Mtpa
Gregory Hollett	OceanaGold	OP Mining	Various visits 2018, 2019 and 2020. January 3 to January 14, 2022	OP mine planning and reserves estimation review, including site tours of OP mining areas, mine waste facilities, and TSF
Jay Newton Janney-Moore	NewFields	Geotechnical/ Infrastructure	Sept. 24-26, 2019 Aug 31-Sept 2 2021	Inspection of the Duckwood TSF, PAG OSA, and geomembrane lined ponds
Joanna Poeck	SRK	UG Mining/ Infrastructure	February 6 and 7, 2017 March 12 to14, 2019	Site tour of current mining and infrastructure
John Tinucci	SRK	Geotechnical	November 29 to December 1, 2016	Site tour of current mining, geotechnical drilling activities, and examination of core
William Lucas Kingston	NewFields	Hydrogeology and Groundwater Management	August 31 to September 2, 2021	General site inspection, including dewatering system
Jonathan Moore	OceanaGold	Geology/Resources	January 13 to 28, 2020	Review of Resource and Reconciliation. Visited open pit.

Personnel	Company	Expertise	Date(s) of Visit	Details of Inspection
David Bird	SRK	Geochemistry	November 18, 2021	General site inspection with focus on waste rock managment

Michael Kirby, Brianna Drury, and David Londono are based in South Carolina and are on-site regularly.

2.4 Sources of Information

This report is based in part on internal Company technical reports, previous feasibility studies, maps, published government reports, Company letters and memoranda, and public information as cited throughout this report and listed in the References Section 27.

2.5 Effective Date

The effective date of this report is December 31, 2021.

2.6 Units of Measure

The Metric System for weights and units has been used throughout this report. Tonnes are reported in metric tonnes of 1,000 kg. Gold is reported in grams and troy ounces, where applicable (1 Troy ounce = 31.1035 grams). All currency is in U.S. dollars (US$) unless otherwise stated.

3 Reliance on Other Experts

The Consultant's opinion contained herein is based on information provided to the Consultants by OceanaGold throughout the course of the investigations. SRK has relied upon OceanaGold and the work of other consultants in the project areas in support of this Technical Report.

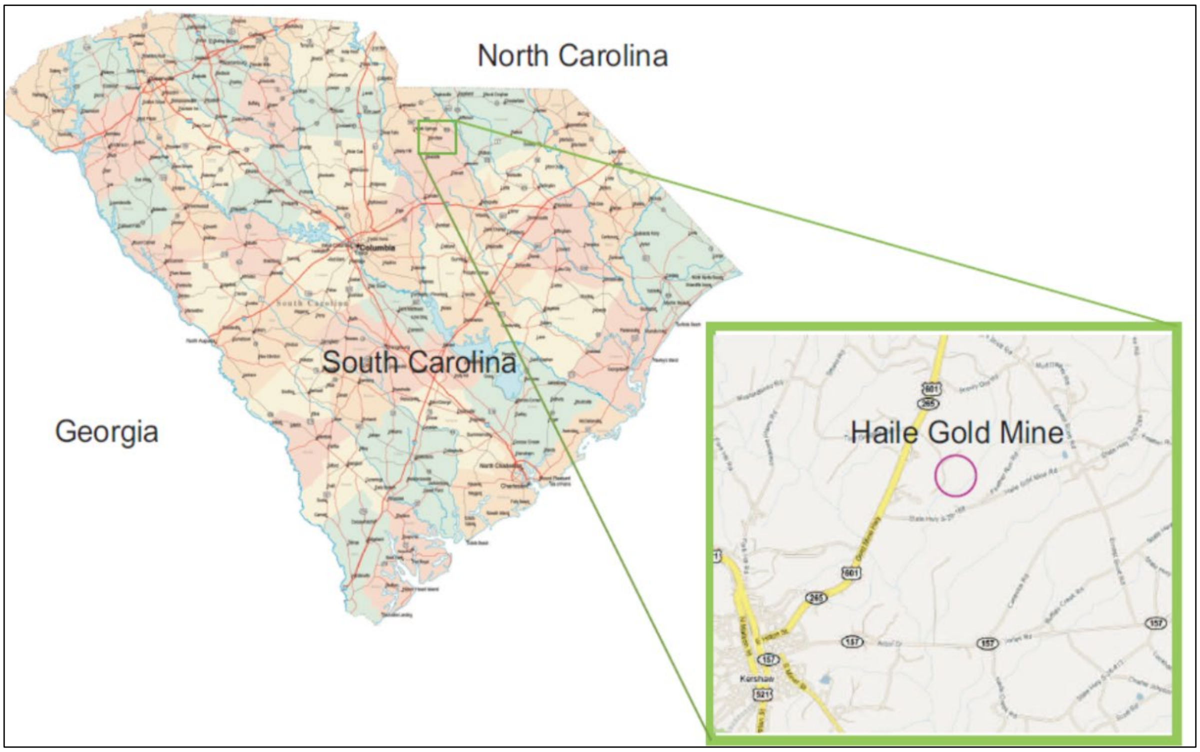
The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

SRK has relied upon OceanaGold for information regarding the surface land ownership/agreements as well as the mineral titles and their validity. Land titles and mineral rights for the project have not been independently reviewed by SRK and SRK did not seek an independent legal opinion for these items.

4 Property Description and Location

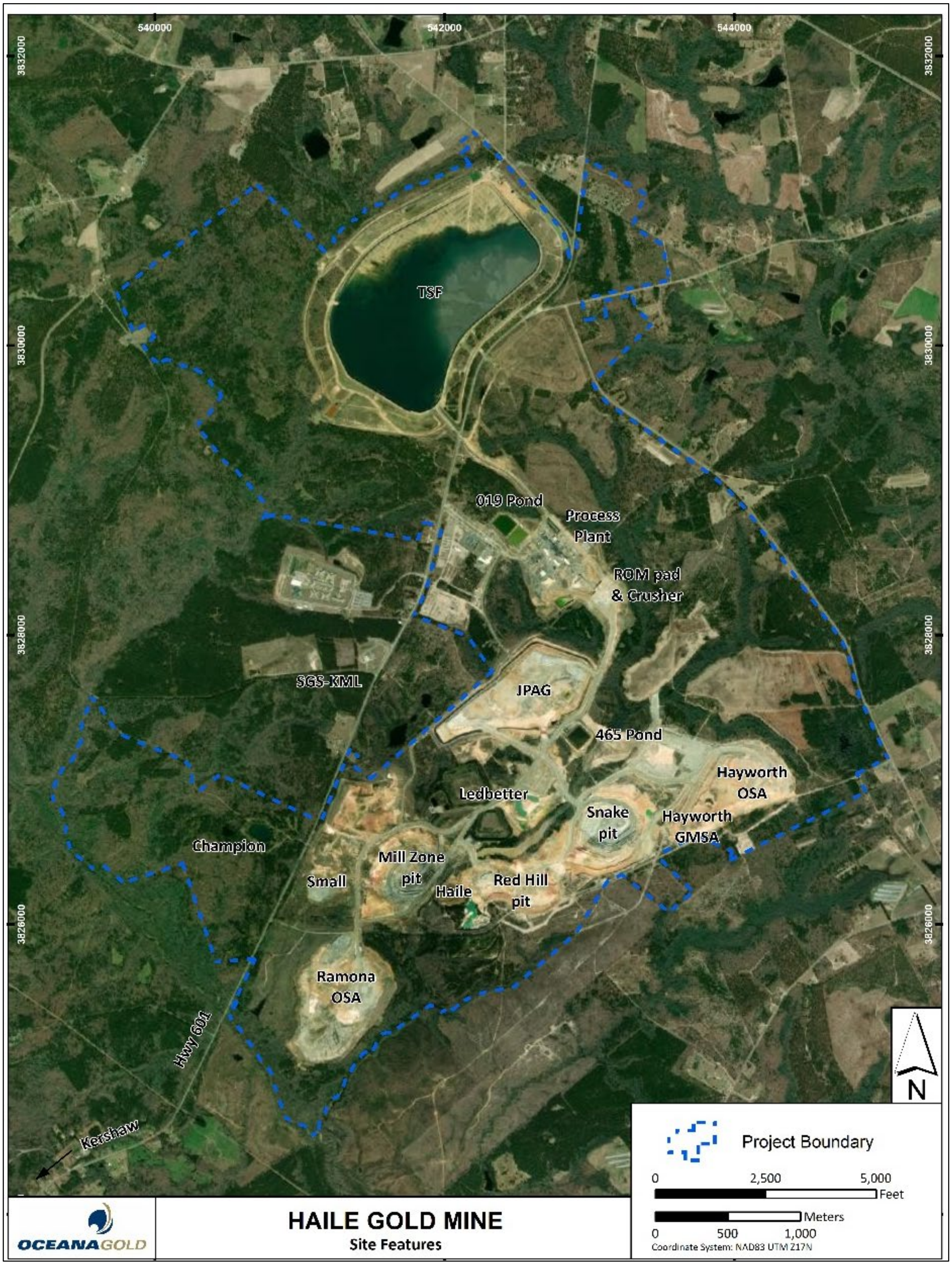
4.1 Property Location

The Haile gold mine is located 5 km northeast of Kershaw in southern Lancaster County, South Carolina, USA, in the north-central part of the state, as shown in Figure 4-1. Haile is 27 km southeast of Lancaster, the county seat, and is 80 km northeast of Columbia, the state capital. The geographic center of the mine is at 34° 34' 46" N latitude and 80° 32' 37" W longitude. Mineralized zones at Haile lie within an area extending from UTM NAD83 zone 17N coordinates 540000E to 544000E and 3825500N to 3827500N. Figure 4-2 shows a site map of the Haile Gold Mine.



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

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

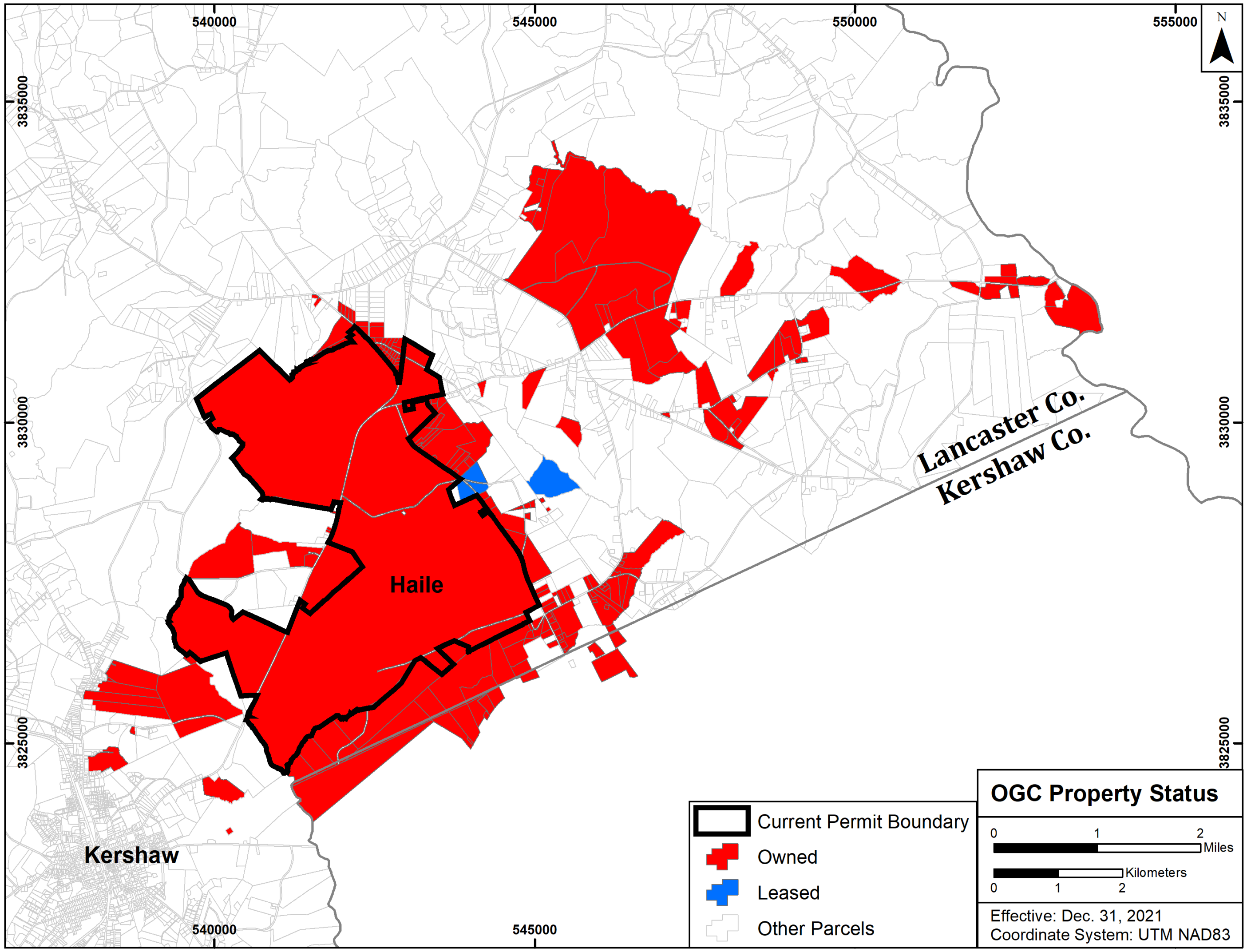


Source: OceanaGold, 2019

Figure 4-2: Site Map of the Haile Gold Mine. Background Imagery from March 22, 2019

4.2 Ownership

Haile Gold Mine Inc. (HGM) is a wholly owned subsidiary of OceanaGold Corporation (OceanaGold). References in this document to OceanaGold refer to the parent company together with its subsidiaries, including HGM and Romarco Minerals Inc. As of December 31, 2021, HGM owns a total of 11,003 acres in South Carolina. Of this total, 4,522 acres are within the mine permit boundary. Proposed expansion in the Supplemental Environmental Impact Statement (SEIS) will increase the mine property to 5,469 acres. Figure 4-3 shows the Land Tenure map as of December 31, 3031, with Fee Simple (OGC owned) and leased properties, almost entirely in Lancaster County.



Source: OceanaGold, 2021

Figure 4-3: Land Tenure Map

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Haile property is easily accessible on paved roads and highways from U.S. Highway 601 to the mine entrance on Snowy Owl Road, located 5 km northeast of Kershaw, South Carolina. The major international airport at Charlotte, North Carolina, is an 80-minute drive from the mine.

5.2 Climate

The Haile area of South Carolina has a sub-tropical climate. Summers are hot and humid with daytime temperatures averaging 29°C to 35°C. Winters are mild and temperatures range from 0°C to 15°C. Average annual precipitation is 1,270 mm while annual evaporation is estimated at 760 mm. Rain is abundant throughout the year with January, March, and July being the wettest months. Snowfall is insignificant and averages less than 80 mm per year. South Carolina averages 50 days of thunderstorm activity and 14 tornadoes per year. The mine operating season is year-round.

5.3 Local Resources and Infrastructure

Local resources (labor force, manufacturing, supplies, housing, utilities, emergency services, etc.) and infrastructure are in place and are widely utilized at Haile. Numerous small communities exist around the Haile mine with populations ranging from 700 to 10,000 people. Power is available in the area via an existing 44 kV transmission grid with Duke Energy and a 69 kV transmission grid with Lynches River. The company utilizes both grids. Surrounding nearby land use is dominantly for agriculture and timber.

5.4 Physiography

The Haile Gold Mine and its surroundings occur within the Sand Hills sub-province of the Piedmont physiographic province of the southeastern United States. This province trends from southwest to northeast and is bound by the Coastal Plain to the southeast and the southern Appalachian Mountains to the northwest. Gentle topography and rolling hills, dense networks of stream drainages, and white sand to red brown saprolitic soils characterize the province. The mine elevation ranges from 120 m to 170 m above mean sea level. Topography is dissected by the perennial, southwest-flowing Haile Gold Mine Creek and by its intermittent tributaries. Haile Gold Mine Creek (HGMC) enters the southeast-flowing Little Lynches River 1.6 km southwest of the mine site. Gradients within the drainages are gentle to moderate (9% to 13%) and slopes above the drainages are gentle to nearly flat (less than 1%). The property is heavily wooded by pine and hardwood forests. Pine timber harvesting occurs frequently in and around the property area as each harvestable tract matures.

5.5 Infrastructure Availability and Sources

There are large industrial centers near the mine. Equipment and sources of logistical and professional expertise can be obtained from the major cities of Charlotte, N.C., and Columbia, S.C., which are both within one-hour travel of the mine. Multiple contractors provide skilled workers for the project. There is adequate labor for operations.

6 History

Gold was discovered in 1827 near Haile by Colonel Benjamin Haile, Jr. in gravels of Ledbetter Creek (now HGMC). This led to placer mining and prospecting until 1829, when lode deposits at the Haile-Bumalo pit site were found. Surface pit and underground work continued at the Haile-Bumalo site for many years. In 1837, a five-stamp mill was built (Newton et al., 1940). Gold production and pyrite-sulfur mining for gunpowder continued through the Civil War from 1861 to 1865. General Sherman's Union troops invaded the area and burned down the operations near the war's end.

In 1882 a sixty-five-stamp mill was constructed by E.G. Spilsbury and operated continuously until a fatal boiler explosion killed the mine manager in 1908. During that time, Adolph Thies developed the Thies barrel chlorination extraction process and improved gold recovery from Haile sulfide ores (Pardee and Park, 1948). During the 26-year operation period, mining grew to include the Blauvelt, Bequelin, New Bequelin, and Chase Hill areas. From 1907 to 1913, an attempt to operate a cyanide plant to extract gold from mine tailings was unsuccessful. Pyrite used to produce sulfuric acid was mined at Haile from 1914 to 1918 (Newton et al., 1940).

From mid-1937 to 1942, larger-scale mining was undertaken by the Haile Gold Mines Company. The property then consisted of owned or leased ground totaling about 1,335 hectares (ha) (Newton et al., 1940). Most of the main pits were mined to the 46-m level with some underground operations at Haile-Bumalo reaching the 106-m level (Pardee and Park, 1948). The Red Hill Deposit was discovered by crude induced polarization techniques next to the Friday pyrite diggings (Newton et al., 1940). This fairly large operation was shut down by presidential decree in 1942 because of World War II. By this time, Haile had produced over US$6.4 million worth of gold (in 1940 dollars) (Newton et al., 1940).

Starting in 1951, the Mineral Mining Company (Kershaw, South Carolina) mined Mineralite® from sericite-rich pits around Haile. This industrial product is a mixture of sericite, kaolinite, quartz, and feldspar and is used in manufacturing insulators and paint base. Mineralite mining ended in 1991.

In 1966, Earl Jones conducted exploration work in the area and eventually interested Cyprus Exploration Company (Cyprus) in the project. Cyprus worked Haile from 1973 to 1977. Numerous companies explored the Haile regional area in the 1970s and 1980s, including Amselco, Amax, Nicor, Callaghan Mining, Westmont, Asarco, Newmont, Superior Oil, Corona, Cominco, American Copper and Nickel, Kennecott, and Hemlo.

The 1980s heralded the first successful modern exploration and production at Haile. Piedmont Land and Exploration Company (later Piedmont Mining Company) explored Haile and surrounding properties from 1981 to 1985. Piedmont drilled 67 core holes and 1,215 reverse circulation holes on the property and greatly expanded the footprint of the Haile deposits. Piedmont mined the Haile deposits from 1985 to 1992 and produced 85,000 ounces of gold from open pit heap leach operations in oxide and transitional ores. New areas mined by Piedmont included the Gault Pit (next to Blauvelt), the 601 pits (by the US 601 highway), and the Champion Pit. Piedmont expanded the Chase Hill and Red Hill pits and combined the Haile-Bumalo zone into one pit. Piedmont also discovered the large Snake deposit sulfide gold resource and mined its small oxide cap. Piedmont extracted gold ores from a mineralized trend 1.6 km long, from east to west. Historical gold production at Haile is estimated at 360,000 ounces (Speer and Madry, 1993, Maddry and Kilbey, 1995).

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

From 1992 to 1994, HMC completed a program of exploration and development drilling, property evaluation, Mineral Resource estimation, and technical report preparation. During this period, the large Ledbetter resource zone was discovered under a mine haul road. At the end of the HMC program in 1994, the gold reserve was stated as 780,000 ounces of gold contained within 7.9 million tonnes at an average gold grade of 3.05 g/t. A qualified person has not done sufficient work to classify the historical estimate as Mineral Resources or Mineral Reserves. HGM is not treating the historical estimate as Mineral Reserves. Because of unfavorable economic conditions at the time, Amax did not proceed with mining, and began a reclamation program to mitigate acid rock drainage (ARD) conditions at the site.

Kinross acquired Amax in 1998, assumed Amax's portion of the Haile joint venture, and later purchased Piedmont's interest. Because Haile was a low priority compared to larger and more profitable projects, Kinross decided not to reopen the mine and continued the reclamation and closure program. Reclamation and closure proceeded through to 2015 when Haile operations commenced again under Romarco Minerals Inc.

Romarco acquired Haile from Kinross in October 2007 and began a confirmation drilling program in late 2007. Romarco completed the confirmation drill program in early 2008 and began infill and exploration drilling focused around the Ledbetter resource. Drilling accelerated in early 2009 with a major reverse circulation infill drilling program that continued through 2012. Condemnation drilling by Romarco for mine facilities commenced in September 2009. Drilling east of the Snake deposit discovered the high-grade Horseshoe deposit in 2010 and required the planned tails storage facility to be relocated 3 to 4 km northwest of the mine. Geotechnical drilling was initiated in September 2009 for pit slope designs. The final hole at Ledbetter discovered a deeper northwest extension in 2010 that was named Mustang. Drilling between the Red Hill and Horseshoe areas had identified large zones of lower grade material that led to the late 2011 discovery of the deep Palomino prospect. Due to low gold prices and mine permitting, Haile exploration drilling was suspended during 2013 and 2014.

Romarco submitted a feasibility study for Haile in February 2011. Drillhole data available as of November 17, 2011, were used in the March 2012 Mineral Resource estimate. Romarco completed a large portion of detailed engineering and permitting for the project in 2011 and 2012. In November 2014, an updated feasibility study was completed after receiving the necessary permits. In April 2015, construction of the project began by Romarco and mining commenced in the Mill Zone pit.

OceanaGold Corporation acquired Romarco Minerals Inc. in October 2015 and became owner and operator of Haile. Project construction during 2015 and 2016 included a new Carbon-In-Leach (CIL)-flotation process plant, power upgrades, a lined PAG overburden storage area, and a tails storage facility. The first gold pour at the new process plant was in January 2017.

7 Geological Setting and Mineralization

7.1 Regional Geology

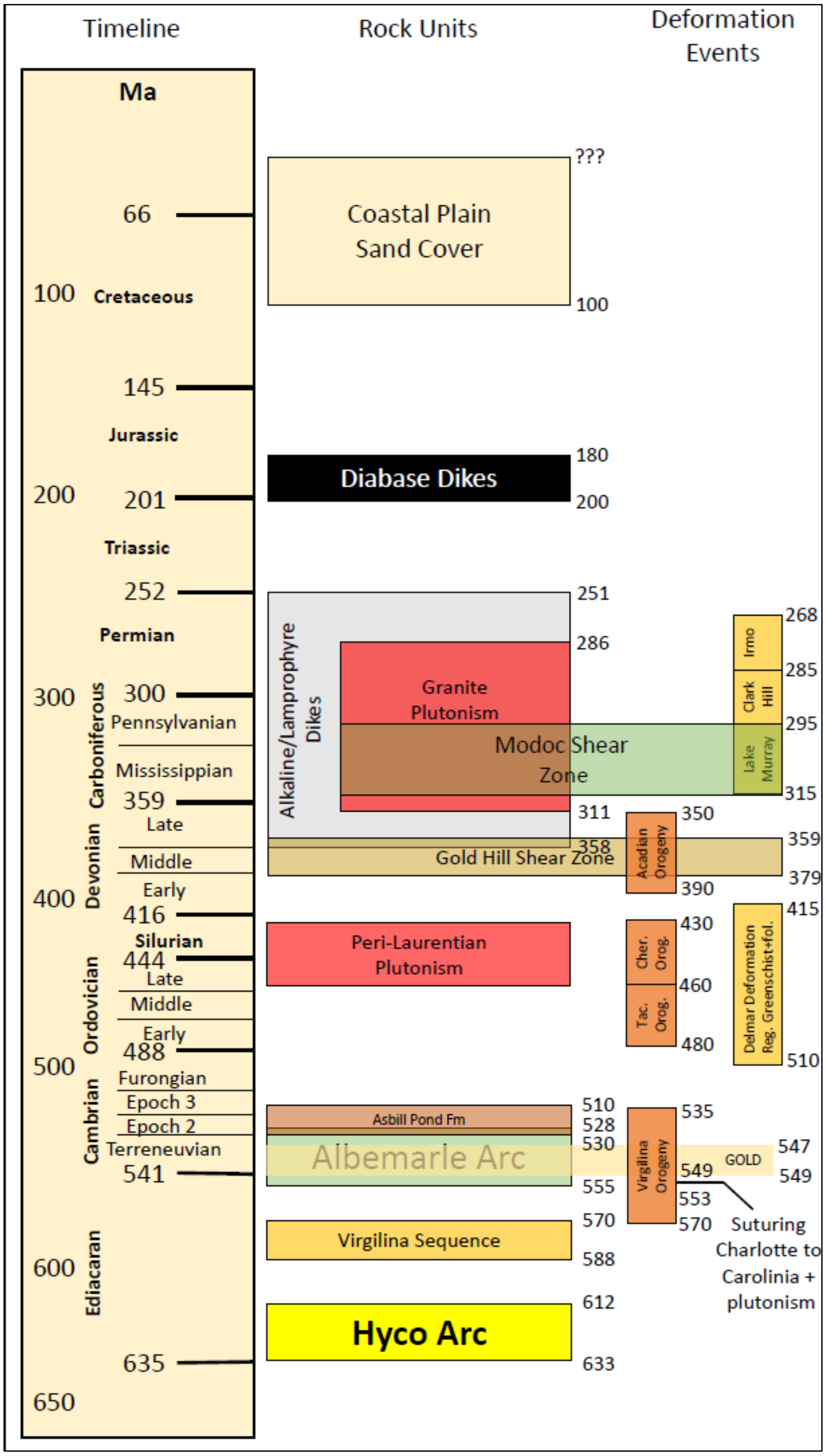
Gold endowment in the southern Appalachian piedmont is predominantly from the *Carolina Slate Belt* (CSB), also known as the *Carolina Terrane* (Hibbard et al., 2010). The 700 km long belt is characterized by a strong northeast structural grain (stratigraphy, faults, foliation, fold axes) extending from Alabama to Virginia that is up to 140 km wide in North Carolina. Volcanic arcs formed adjacent to the African continent and were accreted to the North American craton during the Late Proterozoic to Silurian (Hibbard et al., 2010).

The CSB is a northeast-trending, late Proterozoic to early Cambrian belt of felsic volcanic flows and pyroclastic rocks admixed with fine-grained epiclastic and turbiditic sediments. Volcanic rocks of the belt are both fragmental tuffs and cohesive lavas that generally define a bimodal suite of calc-alkaline to felsic rocks based on SiO_2 content and Ti/Al ratios (Feiss, 1982). Sedimentary rocks were deposited under calm, subaqueous anoxic slopes conditions as evidenced by laminated, laterally extensive siltstones. Volcanic and sedimentary facies are interfingered in the Haile region, and are cut by post-metamorphic mafic dikes.

The CSB has a prominent flexure in central South Carolina near Haile. Structural trends southwest of this area are east–northeasterly, whereas trends northeast of the flexure are mostly northeasterly (Hibbard et al., 2002). The CSB was intruded by dominantly granite plutons approximately 595 to 520 Ma and by post-metamorphic Carboniferous granite plutons approximately 300 Ma (Fullagar and Butler, 1979). Hydrothermal activity prior to Alleghanian regional metamorphism is indicated by folded and recrystallized quartz veins and by pressure shadows with fringes of chlorite and quartz on pyrite. Four tectonothermal periods are recorded in the Carolina Terrane (Hibbard et al., 2002), including:

- Late Neoproterozoic to Early Cambrian Virgilina events (617 to 544 Ma): folding, foliation and faulting with granite plutonism.
- Late Ordovician–Silurian Taconian events (457 to 425 Ma): upright folding with a penetrative axial planar cleavage accompanied by greenschist facies metamorphism.
- Devonian events of the Gold Hill-Silver Hill dextral shear zone (393 to 381 Ma) that juxtaposes the Carolina and Charlotte Terranes.
- Late Paleozoic Alleghanian events (333 to 286 Ma): ductile mylonitic shear zones (e.g., Hyco, Modoc shears) (Hibbard et al., 1998) with orogenic quartz veins, greenschist facies metamorphism and granite plutonism. This event is strongly superimposed on the rocks at Haile.

Gold mineralization at Haile (approximately 549 Ma, Mobley et al., 2014) occurred during suturing of the Carolina and Charlotte arcs coincident with emplacement of the Longtown and Little Mountain stitching plutons (Barker et al., 1998) (Figure 7-1). The Alleghanian orogeny formed the Appalachian Mountains with the emplacement of granite plutons and mafic dikes approximately 250 million years after gold mineralization at Haile (Hibbard et al., 2002, Hatcher et al., 2007). Importantly, the Alleghanian Orogeny superimposed a pervasive regional structural fabric on all pre-Alleghanian rocks. Greenschist facies metamorphism converted micas and clay minerals to chlorite and introduced carbonate and pyrite. Northwest-trending magnetic diabase dikes were intruded about 100 million years later during the Triassic period. Erosion and weathering with saprolite and strong sericite formation up to 40 m deep occurred in a sub-tropical, humid paleo-environment. A southeastward-thickening apron of clayey sands with basal quartz-rich gravels was deposited in the Cretaceous period. Continental uplift and Atlantic Ocean regression have caused continued erosion and incision of the region by dominantly southeast-flowing drainage systems.



Source: OceanaGold, 2021

Figure 7-1: Time Distribution of Major Geological Events in the Carolinas

Volcaniclastic sedimentation and slope sedimentation are characteristic of convergent plate margins in marine forearcs and back arcs. The forearc region between the volcanic arc and the down-going subducting crustal slab, above which lies the trench, can be up to 300 km wide and can form a large forearc basin (Fisher and Schmincke, 1994). This tectonic setting is interpreted during formation of Haile host rocks in the Carolina Terrane. The volcaniclastic component depends on intensity and proximity of volcanic activity and the volume of debris that enters the sedimentary environment. Sediments in the forearc basin are largely turbidite-dominated, and clastics are epiclastic volcanic and reworked pyroclastics and hyaloclastites. Pyroclastics as lapilli tuffs and ash flow tuffs dominated during episodes of aerial volcanism, whereas epiclastic deposition dominated between volcanic episodes. This overlap of volcanic and sedimentary rocks produces frequent facies changes and rheological contrasts in the Haile region.

Stratigraphic nomenclature for Carolina Terrane units has not been standardized between North and South Carolina. Generally, the Uwharrie and Tillery Formations (the latter being the lowermost formation in the Albemarle Group) in North Carolina are equivalent to the Persimmon Fork and Richtex Formations, respectively, in South Carolina. The Persimmon Fork and Richtex Formations are respectively about 3,000 m and 5,000 m thick (Whitney et al., 1978, Secor and Snoke, 1986). Variability in composition, grain size, and proximity to hydrothermal centers during subaqueous lithification produced highly variable alteration geochemistry and textures. Cohesive dacite lavas that experienced alkali loss, silica addition, quenching and variable cooling rates produced mottled textures that can be mistaken for volcaniclastic rocks.

The largest known gold deposits in the southeastern US are in the north-central portion of South Carolina. They are oriented SW-NE and occur at or near the contact between metamorphosed volcanic and sedimentary rocks of Neoproterozoic to Early Cambrian age. Gold is present in quartz veins and as fine-grained disseminations in sedimentary and volcanic rocks with silicified, argillic, and propylitic alteration zones. The largest gold deposits in South Carolina are the Haile (3.8 Moz resource plus 1.3 Moz production), Ridgeway (1.44 Moz) and Brewer (0.26 Moz) deposits (Foley and Ayuso, 2012) as shown in Table 7-1. Haile is the largest and is currently the only active gold mine in the region. The inactive Brewer and Ridgeway mines are respectively located 12 km northeast and 50 km southwest of Haile. Haile and Brewer are hosted in sedimentary and volcanic rocks of the Upper Persimmon Fork Formation. Ridgeway is hosted in sheared metasediments along the Persimmon Fork – Richtex contact. Gold in sediments is often stratiform and occurs along lithotectonic boundaries between interfingered volcanic and sedimentary rocks of the Persimmon Fork Formation. Haile is classified as a low sulfidation, sediment-hosted, disseminated, epithermal gold deposit with proximal quartz-sericite-pyrite alteration and distal carbonate-chlorite alteration (Robert et al., 2007). Ridgeway is geologically similar to Haile in that it is dominantly sediment-hosted with mineralized volcanic rocks. East-west to ENE structural controls and local folding characterize the Haile and Ridgeway deposits. Brewer is a high sulfidation, pyrite-enargite-chalcopyrite-topaz-rich, volcanic-hosted, breccia pipe overprinted by advanced argillic alteration (pyrophyllite-andalusite).

Table 7-1: Geological Summary of Major Gold Deposits of SE USA

Deposit	Type	Host Rocks	Alteration	Inventory (Moz Au)	Au Age (Ma)
Haile	Sediment-hosted epithermal	Persimmon Fork metasediments	quartz-pyrite-sericite	5.0*	549
Ridgeway	Sediment-hosted epithermal	Persimmon Fork metasediments	quartz-pyrite-sericite	1.44	553
Brewer	High sulfidation	Persimmon Fork metavolcanics	pyrite-enargite-chalcopyrite	0.26	550
Barite Hill	VMS	Persimmon Fork metasediments	quartz-barite-sericite	0.06	566

Source: OceanaGold & Foley and Ayuso, 2012
*Remaining resources of 3.8Moz, 0.9 Moz of 2015 to 2021 gold production and an estimated 0.36 Moz of historical production

7.2 Local Geology

Haile geological history includes several major events, as listed below from oldest to youngest. Regional and local geologic maps are presented in Figure 7-2 and Figure 7-3. A schematic stratigraphic column is presented in Figure 7-4. A map of drillholes and two geologic cross sections are presented in Figure 7-5 and Figure 7-6.

Late Pre-Cambrian to Early Cambrian (580 to 530 Ma)

- Carolina Terrane formed as part of a subduction zone-oceanic island arc complex
- Persimmon Fork Formation deposited with metavolcanic and metasedimentary rocks - foliated laminated siltstone unit is the primary host rock for Haile mineralization
- Gold mineralization at Haile assumed at ~549 Ma (Mobley, et al., 2014) by close association with molybdenite dated using Re-Os
- Transition from volcanism to basinal sedimentation ~548 Ma (Persimmon Fork-Richtex boundary)
- Richtex Formation deposited (mudstones and siltstones) conformably on the Persimmon Fork

Carboniferous (320 to 290 Ma)

- ENE structural fabric developed during NW-SE compression and emplaced granite plutons and lamprophyre dikes within 5 km of Haile

Triassic - Early Jurassic (250 to 200 Ma)

- Diabase dikes intrude the Carolina Terrane as magnetic NW-SE-trending anomalies

Cretaceous to Present (100 Ma)

- Coastal Plain clayey sands deposited over all units, thickens southeastward

7.2.1 Lithology

The following rock units are described in chronostratigraphic order from oldest to youngest. Haile stratigraphy is described from mapping and core drilling over a thickness of about 1 km.

Neoproterozoic Rocks

Volcanic and interbedded epiclastic rocks are assigned to the approximately 3 km thick Persimmon Fork Formation that formed about 555 to 551 Ma (Hibbard et al., 2002). Richtex Formation siltstones conformably overlie the Persimmon Fork Formation approximately 0.5 km southeast of the Haile district. The Persimmon Fork-Richtex boundary marks the ~550 Ma change from volcanic-dominated arc terrane to basinal sedimentary facies. Persimmon Fork and Richtex units are pervasively metamorphosed to lower greenschist facies with chlorite, carbonate, and pyrite. Neoproterozoic orthoclase-rich pink granites, such as the Longtown granite near Ridgeway, do not outcrop at Haile; the nearest equivalent outcrop is 10 km north of Haile.

Persimmon Fork Formation

The Neoproterozoic Persimmon Fork Formation at Haile consists of laminated siltstone with minor sandstone and conglomerate overlain and interfingered with lapilli and ash flow tuffs. Grey laminated, pyritic siltstones are the dominant host rocks the middle and lower portions of the mine stratigraphy. Siltstones dominantly strike N40^{0}E to N75^{0}E and dip 30-60° NW in north, west and central mine areas and dip 60-80^{0}SE along the southern flank of the Haile district. Clastic rocks coarsen westward as sandstones around the Champion deposit. Sedimentary units are intruded by dacite, rhyodacite and rhyolite dikes and sills that range in thickness from 1 to 150 m. Contacts are often gradational, and all units are foliated and sheared. High-strain features such as shearing are frequently observed within 5 to 10 m of lithologic contacts. Dacite is the most common volcanic rock at Haile with blocky, massive textures that contrast with the platy, foliated, more pyritic (1% to 2%) metasediments. Light grey dacite has porphyritic textures with 1 to 2 mm long euhedral plagioclase phenocrysts (3% to 10%) in an aphanitic groundmass with <1% pyrite. U-Pb ages from zircons in dacite yielded crystallization ages of 553 ± 2 Ma (Ayuso et al., 2005). The 550 Ma Haile gold system has been preserved from erosion because it is largely capped by resistant dacite flows and by rotation into its current sub-vertical position. Dacite occurs in three main areas:

- Along the southeast edge of the Haile system where dacite constrains mineralization at Horseshoe, Palomino and Red Hill
- As a 10 to 60 m thick ENE-trending, 30-40° NW-dipping sill that cuts and underplates orebodies at Red Hill, Haile and Snake
- Overlying and capping ore zones along the north edge of the district at Mill Zone and Ledbetter

Dacite is conformably overlain and interbedded with pale grey, pyrite-poor lapilli and ash flow tuffs with subangular siltstone clasts in a fine-grained tuffaceous matrix with chloritized pumice fragments. Tuffaceous rocks mostly occur in north-central areas of the Haile district at Ledbetter and Snake. They have irregular, hackly joints in contrast to the harder, well-jointed dacites. Quartz feldspar porphyry (QFP) dikes with stockwork quartz-pyrite veins are locally mineralized at Ledbetter, Horseshoe, Snake and Palomino. QFP dikes may be an important component of the hydrothermal system and cause local upgrading.

Richtex Formation

The Richtex Formation conformably overlies the Persimmon Fork Formation along the southeast edge of Haile. The Richtex consists of ENE-striking, 40 to 60° SE-dipping, thin-bedded siltstone and mudstone with sandstone. The lower portion of the Richtex Formation contains mafic tuff and amygdaloidal basalt flows near Ridgeway (Secor and Wagener, 1968). Thickness near Haile is unknown but the Richtex is >3 km thick near Ridgeway. The Richtex Formation is unconformably overlain by Cretaceous Coastal Plain Sands southeast of Haile.

Lamprophyre Dikes

Lamprophyre dikes intrude rocks of the Persimmon Fork and Richtex Formations. Lamprophyres are dark green and fine-grained with spherulitic textures. These dark green dikes contain biotite, hornblende and plagioclase with chlorite and calcite. The near-vertical dikes are strike N-S and E-W and range in thickness from 1 cm to 2 m. Lamprohypre volume at Haile is estimated at about 1%. Lamporophyres are not foliated or pyritic and were likely emplaced during waning stages of the Alleghanian Orogeny. $^{40}Ar/^{39}Ar$ dates in biotite yielded Pennsylvanian ages at approximately 311 Ma, coincident with the Dutchman Creek Gabbro (Fullagar and Butler, 1979).

Granites

The northeast-elongated Liberty Hill and Pageland plutons are exposed 8 km west and 5 km north of the Haile mine. These fresh, medium-grained granites have less than 5% biotite + hornblende and are weakly foliated. The Liberty Hill granite (30 x 20 km) is dated at 293 ± 15 Ma. The Pageland granite (25 x 10 km) is dated at 296 ± 5 Ma (Fullagar and Butler, 1979). Granite has not been observed in drillholes at Haile. Metamorphic aureoles around the plutons are <0.5 km wide and do not impact rocks at Haile.

Diabase Dikes

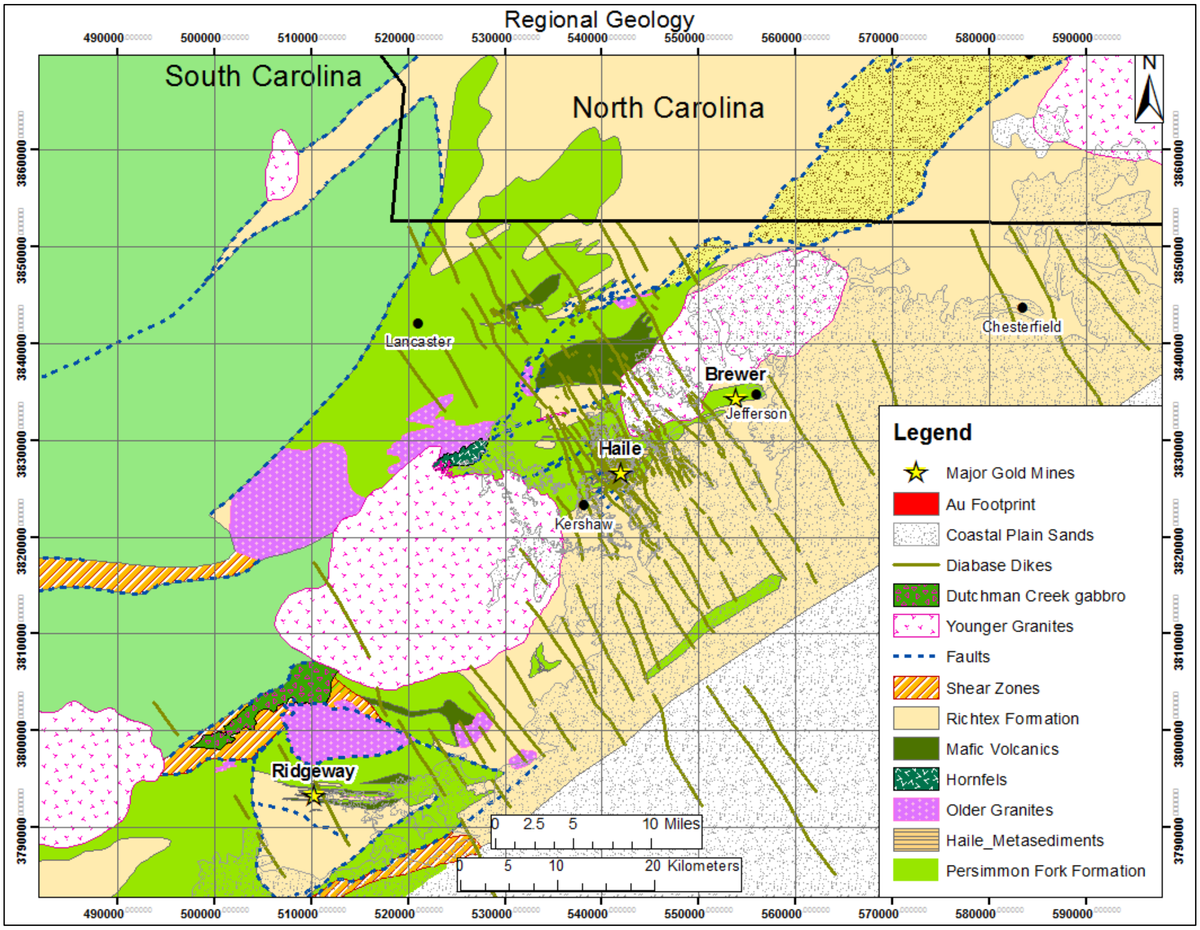
Diabase dikes are dark gray, dense, medium-grained, ophitic and magnetic. These dikes conspicuously cut all other units except the Coastal Plain Sands. Dominant minerals are plagioclase and pyroxene with minor biotite, olivine, chlorite and calcite. The diabase dikes often have chilled margins that are sometimes faulted with apparent dextral displacement. The dikes strike N20^{0}W to N30^{0}W with near-vertical dips and range in thickness from 1 to 30 m. Diabase dikes at Haile occur as both discrete dikes and as swarms (at Mill Zone and Horseshoe) tens of meters wide with a spacing of 300 to 400 m. Diabase dikes have horsetail, anastomosing and sigmoidal geometries. The dikes weather to dark brown, earthy colors. The Triassic diabase dikes postdate gold mineralization and range in age from 222 to 220 Ma (Dooley and Smith, 1982). They account for about 5% by volume of rocks at Haile and often truncate ore zones.

Saprolite

Most of South Carolina is covered by saprolite. Saprolite is a thick, structureless, unconsolidated, kaolin-rich, red orange to white residuum derived from intense acidic bedrock weathering in sub-tropical climates. Saprolite thickness ranges from 10 to 40 m at Haile and is thickest in Fe-Mg-rich metavolcanic rocks and along faults. The base of saprolite is irregular and grades downward into partially weathered bedrock. Saprolite is rarely mineralized at Haile.

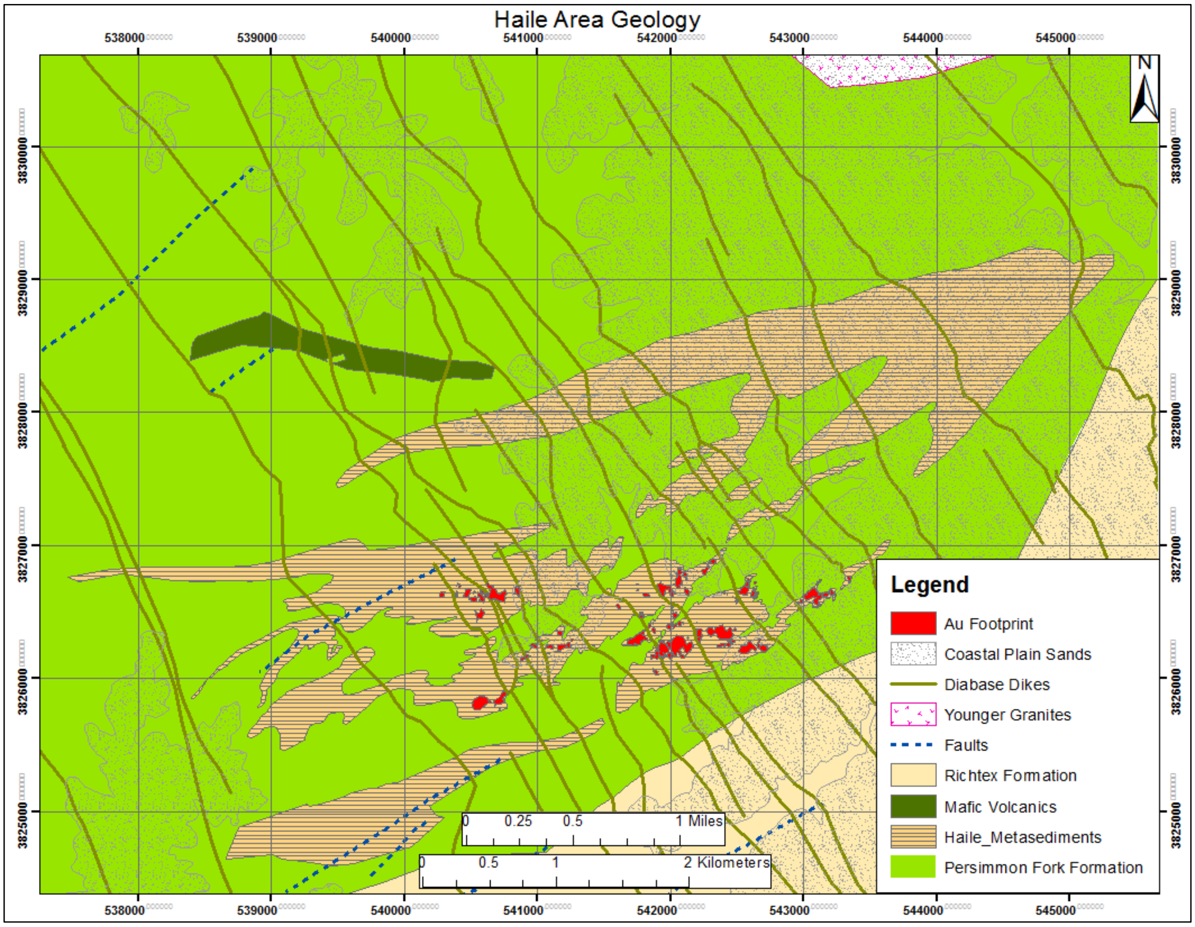
Coastal Plain Sand

The Cretaceous Middendorf Formation (Nystrom et al., 1991) is a southeastward-thickening apron of unconsolidated sand. Its northwest limit conceals much of the Haile property. The sands postdate gold mineralization and is the youngest unit in the region. The sands are up to 30 m thick at Haile and hundreds of meters thick south of Haile. The basal portion contains 10 to 60 cm thick layers of red brown ferricrete and quartz pebbles in a sandy matrix. The middle unit is white to red sand with a kaolinite matrix with frequent cross bedding. The upper unit is a clean tan to white quartz sand.



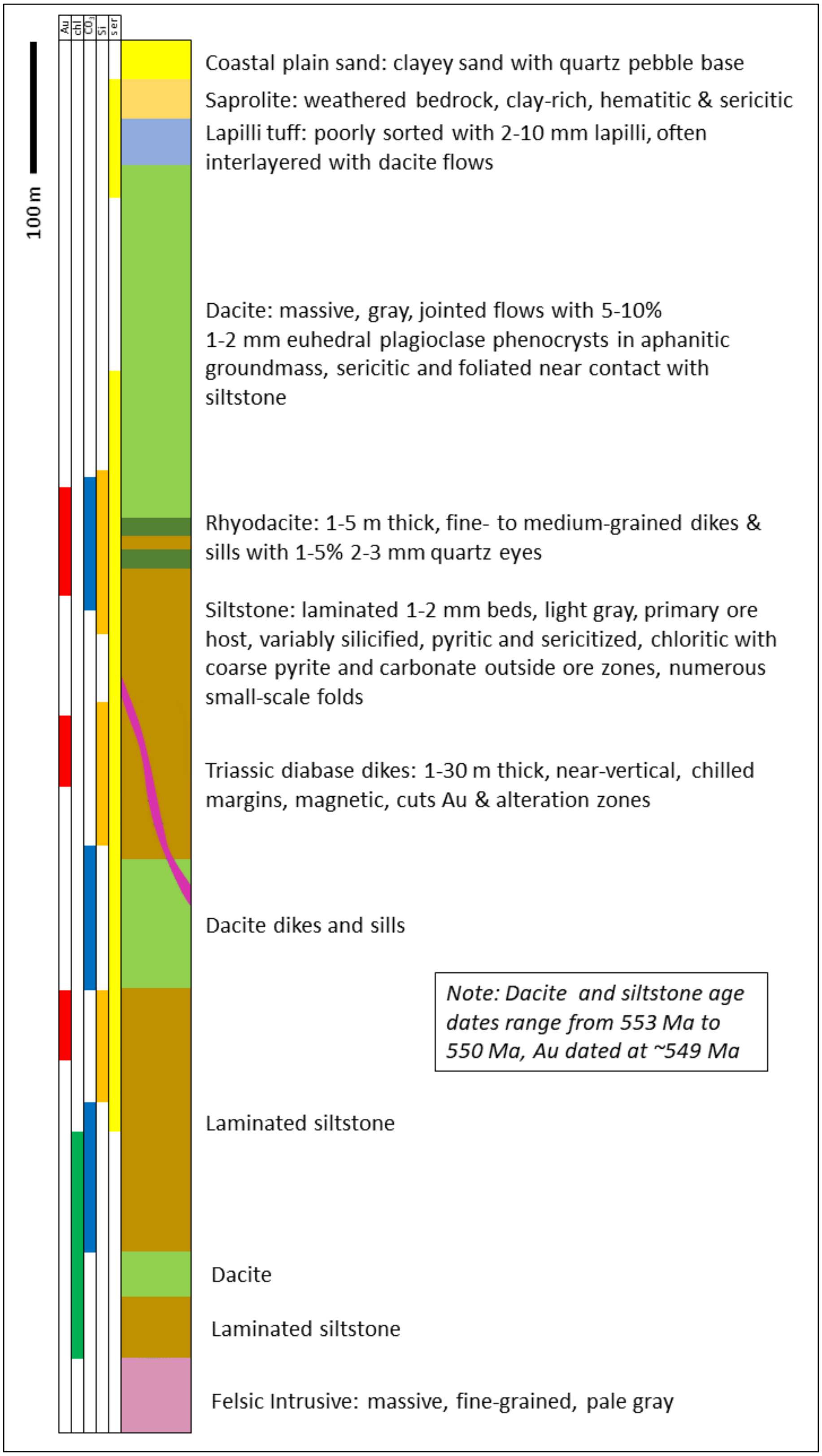
Source: OceanaGold, 2021

Figure 7-2: District Geology of North-Central South Carolina (UTM NAD83 Z17N)



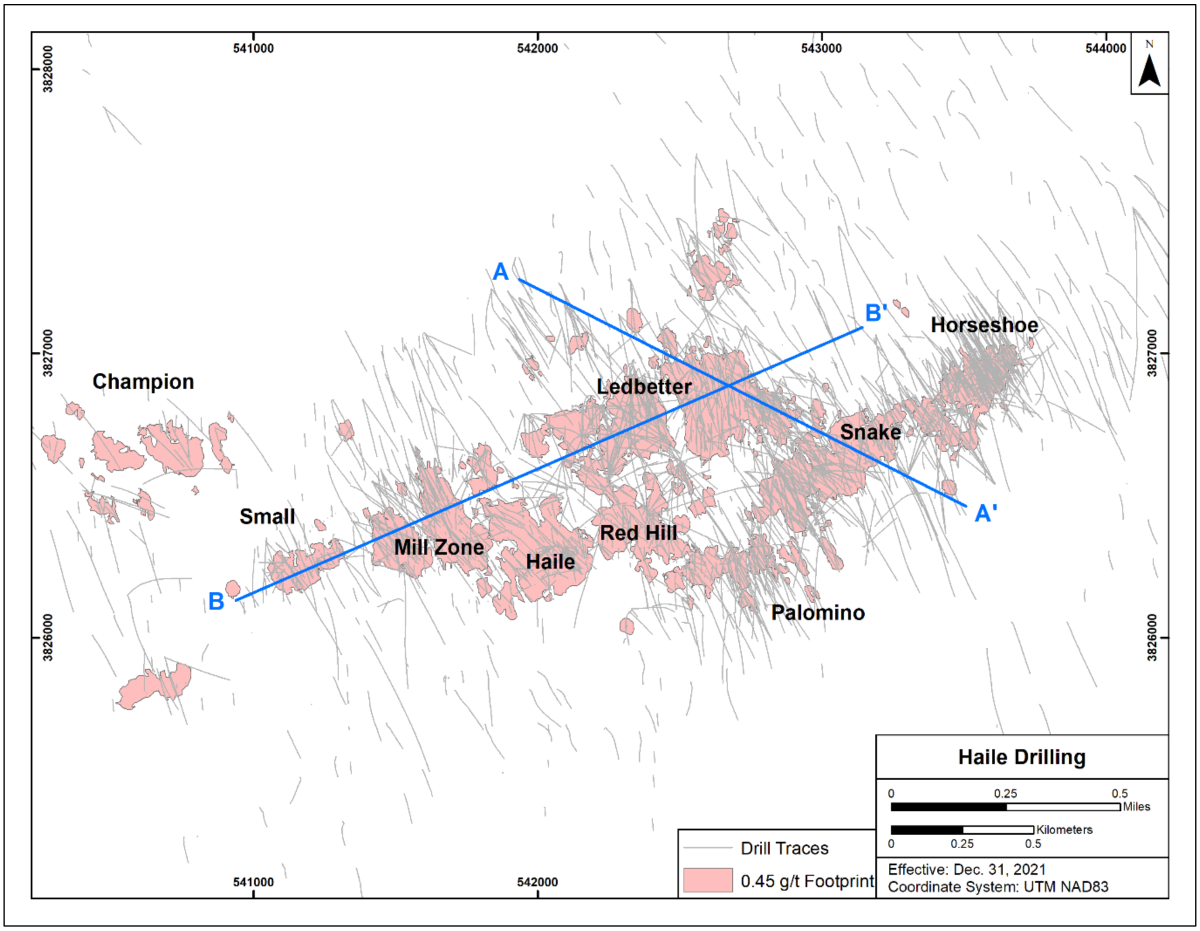
Source: OceanaGold, 2021

Figure 7-3: Geologic Map of the Haile Area with Gold Zones (UTM NAD83 Z17N)



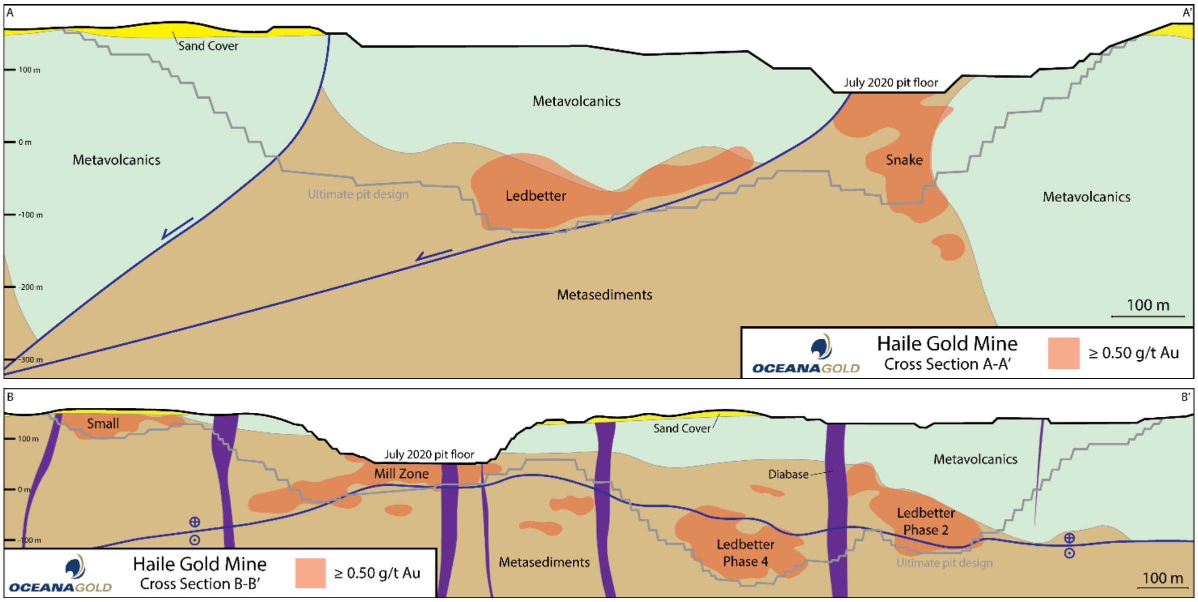
Source: OceanaGold, 2021

Figure 7-4: Haile Stratigraphic Column



Source: OceanaGold, 2021

Figure 7-5: Plan View of Drillholes used for Resource Estimation, Gold Mineralization, and EOY 2019 Reserve Pit Limit (Sections A-A' and B-B' are Presented in Figure 7-6)



Source: OceanaGold, 2021

Figure 7-6: Cross-sections A-A' (NE-looking) and B-B' (NW-looking) with Geology, Gold Mineralization, and Pit Design

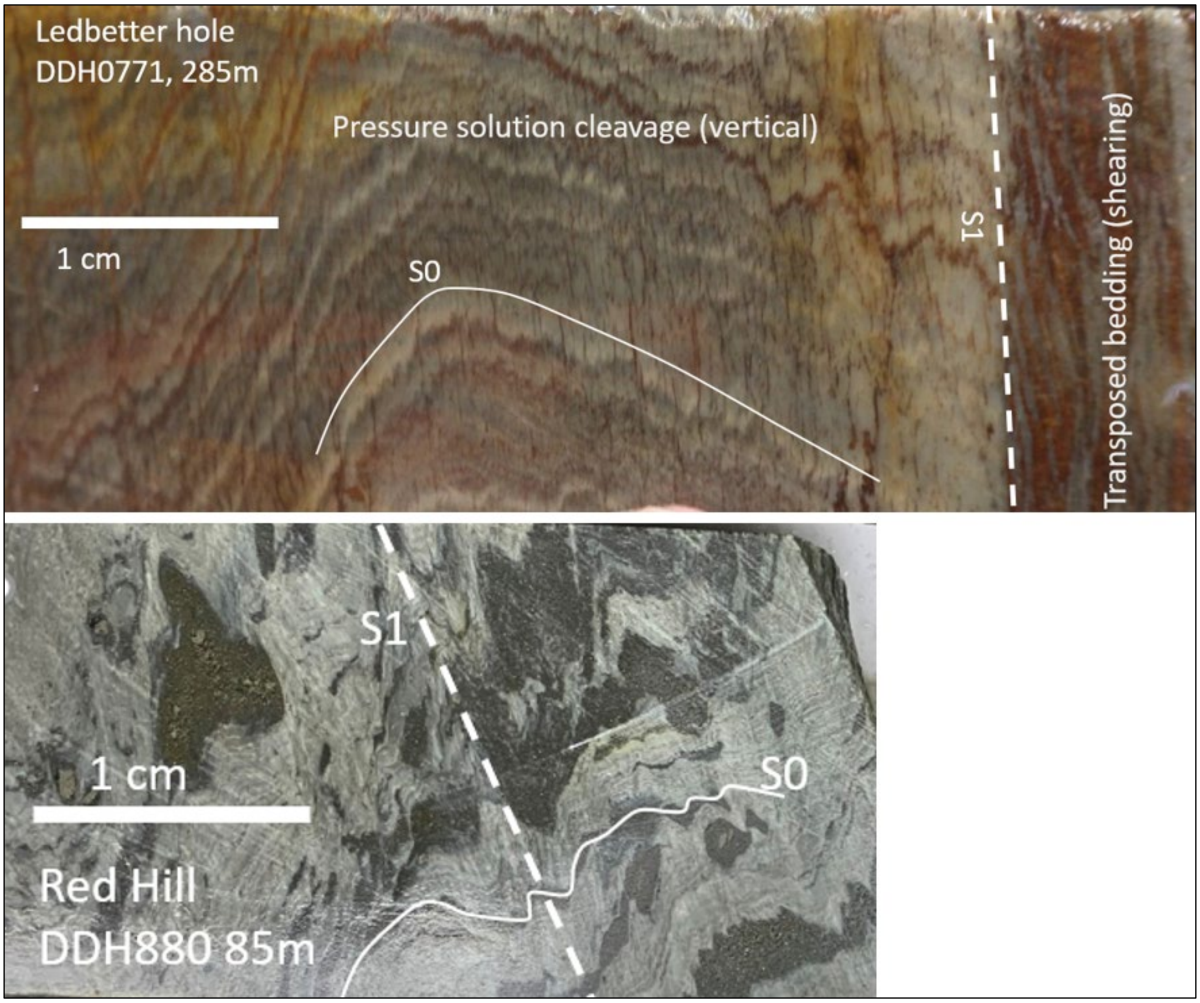
7.2.2 Structure

The structural history of Haile is complex and is affected by four orogenic episodes, as summarized in Section 7.1 and depicted in Figure 7-1.

Four deformation/geological events are observed at Haile; the D_1 and D_2 events provide the dominant geometric constraints on the Haile ore deposits.

- **D_1 syn-mineral hydrothermal replacement** of sedimentary rocks by silica and pyrite with Au, Ag, As and Mo precipitation along ENE faults and foliation; rocks were likely in a nearly flat position and were intruded by dacite flows and capped by pyroclastic rocks.
- **D_2 regional Alleghanian metamorphism and deformation** (folding, shearing, pervasive foliation, quartz veins) and greenschist facies metamorphism (chlorite, carbonate, coarse pyrite); this is the dominant event that overprints the Haile region. Pervasive foliation strikes east-northeast and dips 40-60^0 northwest. Foliation intensity increases along sedimentary-volcanic contacts and is strongly developed in laminated siltstones where gold mineralization is hosted. Rocks were folded into an asymmetric anticlinorium with a steep southeast limb and moderate northwest limb.
- **D_3 minor brittle reactivation** of foliation along NW-dipping normal faults; displaces some ore zones.
- **D_4 diabase dike intrusion** and minor dextral faults down step district from west to east (Figure 7-6).

Figure 7-7 highlights the intense ductile and brittle strain fabrics that overprint and deform ore zones at Haile. The top photo shows pressure solution cleavage S1 imposed on folded S0 bedding flanked by a shear zone of completely transposed bedding. The lower photo shows axial planar cleavage S1 in folded and bedded S0 siltstone with folded pyrite lenses. At ore deposit scale, the pyrite lenses are similar to irregular ore zone shapes. Compositional layering sub-parallel to foliation is generally transposed bedding and represents a S_1 or S_2 fabrics (Hayward, 1992). S_1 foliation is formed by both pressure solution parallel to axial planes of folds and ductile shear strain.



Source: OceanaGold, 2021

Figure 7-7: High Strain Fabrics in Core Samples

Many contacts between volcanic and sedimentary rocks have undergone grain size reduction and thus represent high strain zones recording ductile slip. Greenschist facies metamorphism produced significant volume loss due to dehydration reactions at temperatures of 250 to 300°C and developed pressure solution cleavage. Metamorphism converted micas and feldspars to chlorite and clays.

The Haile gold deposits are exposed within a metasediment (Ms) window flanked and overlain by metavolcanic rocks (Mv). Eastward tilting and erosion have exposed an oblique section with older rocks exposed in the west near the Champion deposit. The ENE-trending window is about 4 km long by 0.5 to 1 km wide (Figure 7-3). Sedimentary rocks are folded within an ENE-trending anticlinorium with a steep SE limb and a moderate NW limb. The Mv/Ms contact is conformable with bedding. High strain fabrics overprint all Mv and Ms units in the mine area. The ENE-striking Mv/Ms contact dips 60-80^0SE along the southeast margin of the district and generally constrains gold mineralization at Red Hill East, Palomino, Deep Snake and Horseshoe. The Mv/Ms contact dips 30-50^0NW in north-central portions of district and caps gold mineralization at Ledbetter, Upper Snake, Mill Zone, and Red Hill West.

The district is dissected by several ENE-striking, 30 to 60˚ NW-dipping dip- and oblique-slip normal faults that appear to both focus gold mineralization and displace mineralized zones. The eastern portions of Ledbetter are interpreted to represent the fault-displaced and sheared upper part of the Snake orebody (Figure 7-6). The Mill Zone orebody is faulted into two segments with about 100 m of normal displacement and is likely a normal displaced portion of the Haile deposit. Brittle deformation is characterized by anastomosing fault zones with discrete but thin slip planes, commonly filled with ribbon quartz or gouge. Fold axes observed in drilling and mapping mostly occur in the southeast portion of the district in the Red Hill, Palomino and Snake deposits. Portions of the Ledbetter deposit contain chaotic folds with variable orientations. Fold axes strike N40˚ E to N70˚ E and plunge gently east.

Quartz (± pyrite + calcite + ankerite) veins are locally common at Haile, notably in the Mill Zone, Haile, Red Hill and Horseshoe deposits. Near-vertical quartz veins typically strike N-S to N15^0E, are 1 to 60 cm thick, and are rarely mineralized. Crustiform textures indicative of boiling are only observed in high-grade portions of the Horseshoe and Ledbetter deposits. Two phases of quartz veins are recognized at Haile. The earlier phase is pale gray with finely disseminated pyrite, has diffuse contacts, and is very fine-grained. In rare mineralized examples, pyrite adjoins or is interlocked with visible gold up to 0.2 mm in size at Horseshoe. The more common quartz vein type consists of massive, cross-cutting, white bull quartz veins that are barren of gold.

7.2.3 Mineralization and Alteration

Haile gold mineralization occurs as en echelon clusters of moderately to steeply dipping ore lenses within a 4 km x 1 km area. Nine named gold deposits are recognized at Haile. From west to east, these deposits include Champion, Small, Mill Zone, Haile, Ledbetter, Red Hill, Palomino, Snake and Horseshoe that often show ‘pearls on a string’ alignment (Figure 7-5). Ledbetter is by far the largest orebody (approximately 1 Moz) and includes the shallow Chase and deep Mustang deposits. Orebody geometry, depth, size, grade, mineralogy, and alteration are variable. The orientation of gold mineralization generally parallels the regional NW dipping foliation but is concentrated along the metavolcanic-metasediment contact. Orebody geometry is partly controlled by orientation of volcanic-sediment contacts and location of barren dacite sills. Ore lenses are typically 50 to 300 m long, 20 to 100 m wide, and 5 to 30 m thick. Ore zones are separated by barren siltstone, dacite sills and diabase dikes. The Mv/Ms contact and gold mineralization gradually deepen from west to east across the Haile district (Figure 7-6). The Mv/Ms contact at Champion has been partly removed by erosion in the west portion of the district and is over 500 m deep at the Horseshoe deposit, 4 km east of Champion. Depth and position of the contact are complicated by faulting and folding. Drilling in southeast areas around Palomino has encountered gold mineralization up to 1 km deep.

Gold mineralization at Haile is mostly hosted by laminated siltstone in the Upper Persimmon Fork Formation and is capped by less permeable coherent dacite. Mineralization is typically within 100 m of the dacite-siltstone contacts. Gold mineralization is disseminated in silicified and fine-grained, pyritic siltstone with local K-feldspar and molybdenite. Small, mineralized zones at Ledbetter, Red Hill and Snake are hosted in dacite along fault zones within 15 m of the Mv/Ms contact. Gold grades in mineralized dacite are typically weaker than in the underlying siltstone and sericite alteration is stronger in the dacite. Hydrothermal brecciation is common in portions of the Ledbetter, Horseshoe, Small and Champion deposits where milled, silicified siltstone clasts occur in a fine-grained quartz-pyrite matrix intruded by fingers of quartz feldspar porphyry with quartz stockwork veinlets.

Mineral zonation grades outward from quartz-pyrite ± K-feldspar + gold (QS) → quartz-sericite-pyrite ± gold (QSP) → sericite + pyrite ± pyrrhotite → chlorite-calcite ± epidote (propylitic). QS and QSP mineralized zones are tens of meters thick. Sericite envelopes range in thickness from tens to hundreds of meters and are controlled by protolith, permeability, and weathering. Within the mineralized zones, quartz is dominant (60% to 80%), pyrite is moderate (1% to 10%), and sericite is variable at 5% to 40%. Semi-massive pyrite zones are locally observed over thicknesses of 0.5 to 5 m, especially in the Mill Zone, Red Hill and Haile pits.

Two silicification events are observed in the mineralized zones. Early massive silicification is finely disseminated to diffuse with less than 1% pyrite. This pyrite-poor silicification correlates roughly with the 0.1 g/t Au shell. Phase two silicification is manifested as matrix fill in tectonic and hydrothermal breccias with 1% to 5% pyrite and is associated with grades more than 0.5 g/t Au. High-grade gold zones more than 3 g/t are characterized by intense silicification and more than 1% fine-grained disseminated pyrite. Pyrite grain size is typically less than 20 microns in ore zones. Pre-ore silica-pyrite zones are frequently stretched, boudined and sheared (Figure 7-7). A second phase of barren, coarse, cubic, undeformed pyrite overprints the fine-grained pyrite that formed during regional greenschist metamorphism. Pyrite cubes in chloritic metamorphosed rocks are 0.5 to 1 mm in size but can be as large as 1 to 2 cm. Pyrrhotite occurs in 5 to 25 m thick halos around and on the edges of ore zones. Its ductile nature produces length: width ratios more than 5:1 in foliated rocks. Pyrrhotite formation is interpreted to be coeval with early, fine-grained pyrite precipitation.

Supergene sericite-kaolinite alteration forms large bleached, cream to white halos around the ore zones with little to no pyrite that was removed during intense acidic leaching. Strong supergene alteration caps and flanks most of the district. Strong sericite alteration is rarely observed deeper than 40 to 50 m. Numerous shallow sericite-kaolinite bodies were mined historically for paint filler.

Propylitic alteration is characterized by increased chlorite (25% to 50%) and a mottled texture with blebs of 1 to 5 mm calcite/ankerite aggregates (2% to 10%) and stockwork. Late calcite ± quartz veining is often focused along fault zones and along shear zones within strongly deformed rocks. Sigmoidal pods of strained quartz are often observed. Oxidation at Haile extends to depths of 20 to 60 m and is deepest along faults and in volcanic rocks. Hematite and goethite are strongest near surface in the saprolite and decrease at depth as weak joint stains.

Gold spatially correlates with silver, arsenic, molybdenum, and tellurium. Base metals are rare at Haile. Thin section petrography and scanning electron microscopy show that the gold occurs as native gold, gold-pyrite and gold-pyrite-pyrrhotite clusters in fine-grained silicified zones. Smeared molybdenite occurs primarily on foliation surfaces and as fine-grained aggregates in silicified zones. Molybdenite at Haile has been dated by Re-Os isotopes at 553.8 ± 9 Ma (Stein et al., 1997), which is coeval with the zircon crystallization age of 553 ± 2 Ma reported by Ayuso et al. (2005). This age correlation indicates that molybdenite mineralization was concurrent with Persimmon Fork volcanism. Seven Re-

Os molybdenite ages from Haile (Mobley et al., 2014) yielded ages ranging from 529 to 564 Ma. Four of these samples produced an average age date of 548.7 ± 2 Ma (Mobley et al., 2014).

8 Deposit Type

Hundreds of gold deposits in the southeast USA are located along a 700 km long SW-NE trend that extends from Alabama to Virginia (McCauley and Butler, 1966, Butler and Secor, 1991). Most of these deposits are small prospects worked and explored along narrow quartz veins. The larger gold deposits are located at or near the contact between volcanic and sedimentary rocks, including the Haile, Brewer, Barite Hill and Ridgeway mines. Brewer is unique in the region and is classified as a high-sulfidation epithermal gold system with volcanic and breccia-hosted gold accompanied by quartz, pyrite, topaz, enargite and chalcopyrite. Gold mineralization at Barite Hill contains the assemblage of pyrite-chalcopyrite-galena-sphalerite and is characteristic of a submarine, high-sulfidation volcanogenic massive sulfide deposit. Haile and Ridgeway are similar in that sediment-hosted gold mineralization is hosted by silicified, sheared and foliated siltstone.

8.1 Haile Genetic Model

The origin of the Carolina slate belt deposits is one of the more controversial types of gold deposits. Genetic models include:

- Worthington and Kiff (1970) concluded that a genetic link must exist between ore genesis and volcanism in the Carolina Terrane due to their volcanic host rocks.
- Spence et al. (1980) found a genetic link between gold mineralization hosted within siliceous and pyritic zones and intense alumina alteration which produced kaolinite and sericite-rich zones stratigraphically above mineralized zones and interpreted these as analogous to features observed in modern hot springs based on geochemical signatures, stratiform nature, stratigraphic position, and geochronology.
- Feiss et al. (1993) built on the model of Spence et al. (1980) by proposing that hot spring-type mineralization must have occurred under extension in a back-arc setting based on oxygen isotope data, which they interpreted to mark a shift from a subaerial to submarine environment. Feiss et al. classified Haile as a syngenetic hot spring system formed as the volcano-sedimentary pile accumulated with subsequent metamorphic overprints.
- Maddry and Speer (1993) proposed an exhalative model for mineralization at Haile whereby gold deposition resulted from hydrothermal fluids venting to the seafloor to produce stratabound ore bodies in marine volcaniclastic rocks. They interpreted intense alumina alteration noted by Spence et al. (1980) to be the effects of saprolitic weathering in warm, humid climates.
- Strong ductile deformation and structural dismemberment, mineral paragenesis, and mineral textures suggest ore deposition within shear zones or fold axes and that the fluid source was from metamorphic devolatilization reactions and pressure solutions related to Precambrian collisional events (e.g., Tomkinson, 1988; Hayward, 1992). Hayward (1992) emphasized the importance of folds in controlling the location of ore formation in anticlinal fold hinges. Hayward also noted that alteration zones at Haile are generally discordant to bedding and commonly display symmetrical patterns around ore bodies.
- Tomkinson (1988) proposed that Haile was an orogenic deposit based on textural and structural connections between gold mineralization and shear zones.

- Bierlein & Crowe (2000) discussed evidence for epigenetic (i.e., orogenic) vs. syngenetic gold mineralization for CSB gold deposits and concluded the evidence strongly favored syngenetic mineralization with local gold remobilization.
- Hardy (1989) and Worthington (1993) interpreted the features observed by Hayward (1992) and Tomkinson (1988) as evidence for the remobilization of pre-existing mineralized horizons, causing gold enrichment along structurally controlled pathways during deformation which postdated the primary phase of mineralization at Haile. Hardy also concluded that fluids deposited silica, K-feldspar, pyrite, and gold in breccia zones.
- Gillon et al. (1995) proposed a model at Ridgeway that invoked early gold mineralization and remobilization during Neoproterozoic deformation.
- O'Brien et al. (1998) proposed that the gold deposits were formed during arc-related volcanic activity in a hydrothermal system.
- Foley et al. (2001) observed multiple generations of pyrite in Haile ores and concluded that disseminated pyrite and gold mineralization were contemporaneous with host volcanic rocks and volcaniclastic sediments.

Pressure shadows around pyrite grains, stretched pyrite and pyrrhotite grains, and flattened hydrothermal breccia clasts indicate that there has been deformation subsequent to sulphide mineralization. These observations are consistent with either pre- or syn-tectonic gold mineralization. Mineralized zones were subsequently foliated and sheared accompanied by regional greenschist facies metamorphism. Similar timing for gold mineralization and peak magmatism in the Haile and Ridgeway areas suggests that the hydrothermal systems that produced these deposits were related to magmatism. The geological understanding of the Haile deposit is increasing as exploration and mining continue. Haile is currently interpreted as a low sulfidation, disseminated, sediment-hosted, structurally-controlled gold system based on tectonic setting, structural control of shear zone-hosted mineralization, low sulfide content, host rock lithology, and geochemistry.

8.2 Haile Geological Model

The Haile geological model was constructed using Maptek's Vulcan and Seequent's Leapfrog software. The model box is approximately 4 km EW by 2 km NS by 800 m deep. The geological understanding of the Haile mineralization continues to evolve and is documented by mapping and drilling in 3,443 drillholes including 1,378 core holes for 422,829 m (59% m) and 2,065 holes for 297,823 m (41% m). Mineralization is typically continuous, albeit exhibits local complexity. The 3D geological interpretations provide a good basis for 3D modeling and gold estimation (e.g., Coastal Plain Sands and dikes). The Haile geological models are updated twice per year.

The model consists of 3D solids for the following five geological units, from youngest to oldest: Coastal Plain Sands, saprolite, diabase dikes, metasedimentary rocks, and metavolcanic rocks. Surfaces that represent oblique-normal faults, the base of oxidation, and the base of clay have also been modeled. Consistency of the geologic model has been improved by relogging of several hundred core holes and incorporation of pXRF and other geochemical data.

9 Exploration

9.1 Pre-Romarco

Modern exploration, development, and mining activity on the Haile property began with mapping in 1970 (Worthington and Kiff, 1970). Between 1973 and 1977, Cyprus Exploration Company (Cyprus) conducted an extensive exploration program consisting of surface geophysical surveys, trenching, geologic mapping, auger drilling, core drilling, air-track drilling, and metallurgical testing. Cyprus calculated the Haile resources at 186,000 oz (5,785 kg) of gold with an average grade of 2.13 g/t. Resources reported in this section do not conform to the standards of NI 43-101 and are included only as part of the historic record.

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

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

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

9.2 Romarco

Romarco completed the Haile property acquisition in October 2007. By February 2008 Romarco had reviewed the quality of historical drilling and assay data and turned their effort to exploration and resource expansion drilling. During its ownership, Romarco expanded the resource and reserve of the property by five-fold. This report documents the results of the drill program achieved to date with Romarco assay data available through November 17, 2011 (i.e., data cut-off for previous IMC15 resource estimate), and subsequently by OceanaGold, as described below.

9.3 OceanaGold Exploration

OceanaGold purchased the Haile property from Romarco in October 2015 and continued the drilling programs to expand and de-risk resources and reserves at Haile. Mine development drilling is ongoing.

9.3.1 Geologic Mapping and Surface Sampling

Numerous workers have performed geologic mapping and surface sampling in and around the Haile Mine area. Mapping is challenged by poor bedrock exposure due to extensive saprolitic weathering, Coastal Plain Sand cover, and dense vegetation. Outcrop is estimated at only 1% to 2% in the Haile area. Detailed mapping is generally restricted to mining excavations. The USGS published a geologic map for the Kershaw quadrangle in 1980 (Bell, 1980). More detailed mapping was conducted at Haile by Spence, Kiff, and Maye, who constructed a detailed geologic map for the mine site in 1975. Subsequent detailed geologic mapping was done by Taylor in 1985 and Cochrane in 1986. Ph. D. dissertations by Tomkinson (1985) and by Hayward (1988) included detailed geologic mapping in open pits. Geologic mapping at the Mill Zone pit resumed with mining in 2016 by OceanaGold geologists.

Historical mapping has been scanned and loaded into the Vulcan software for structural interpretation, exploration planning, and geologic modeling. The use of the structural dataset in conjunction with the drilling dataset has provided the foundation for a 3D digital geologic model. This model continues to be used successfully to expand the resources and reserves at the Haile property. Surface samples have been compiled into an Access database and evaluated by OceanaGold. Over 5,000 samples have been compiled based on location, sample type (rock chip, saprolite, soil, stream sediment), rock type, alteration and assay. QA/QC data are generally lacking for these surface samples, and most were assayed only for gold.

9.3.2 Geophysics

Numerous geophysical surveys have been conducted at Haile since the 1970s. The following geophysical methods have been applied at Haile:

- Gravity
- Airborne Magnetics
- Airborne Electromagnetics
- Ground-based Induced Polarization
- Ground-based Electrical Resistivity
- Self-Potential

Numerous IP/Resistivity surveys have been conducted at Haile, including surveys by Piedmont in 1975 and 1989, by Romarco in 2015 at Champion, Mill Zone, Ledbetter, and Horseshoe, and by OceanaGold in 2016 adjacent to Haile. Geophysical surveys conducted by Piedmont in the late 1980s include ground magnetics and dipole-dipole IP/resistivity methods that led to discovery of the Snake deposit (Larson and Worthington, 1989). The ground magnetic data were acquired in a patchwork fashion and were not corrected for diurnal changes. The dipole-dipole IP/resistivity data were reprocessed by OceanaGold in 2016 (Weis, 2016).

Regional gravity survey and aeromagnetic data have been downloaded from the South Carolina data repository (Daniels, 2005). These were supplemented by more detailed gravity stations in 2009 and 2010 by Romarco along roads in the Haile area and as transects over Haile deposits.

Airborne EM and magnetic surveys were flown by Aeroquest for Romarco in 2010 over the Haile-Brewer area on 50 m and 100 m spaced flight lines with a bearing of 150°-330°. The magnetic data can map the diabase dikes and granite plutons but do not differentiate the older units. Proprietary 3D inversion modeling was conducted by OceanaGold in 2016 to depths of 1,500 m using airborne magnetic and EM data.

10 Drilling

During 2016, the Romarco Minerals drilling database was translated to OceanaGold's standard acQuire database platform. Where available, original source assay and survey data were used for the acQuire translation and database validation. There was a further internal database review in late 2018 / early 2019. No material errors were identified.

10.1 Type and Extent

Drilling at the Haile property commenced in the 1970s and has continued intermittently to the present by several companies. The database used for this resource estimate was extracted from the acQuire database on 31 December 2021. It contains 3,443 drillholes including 1,378 core holes for 422,829 m (59% m) and 2,065 reverse-circulation (RC) holes for 297,822 m (41% m). Some of the historical drilling (i.e., shallow exploration auger or air-track drilling) was judged insufficiently reliable and was excluded from the resource estimation database. RC and core drilling by Romarco continued from 2008 to 2012 and then resumed in 2015 after a two-year hiatus due to permitting and lower gold prices. Drilling at Haile since early 2015, has almost entirely been as core drilling. Drill campaigns by company and year are summarized in Table 10-1.

Table 10-1: Haile Drilling Campaigns by Year, Owner and Lab

Start Hole ID	End Hole ID	Hole Type	Start Yr.	End Yr.	Owner	Lab
DDH0001	DDH0031	core	1975	1977	Cyprus	CMS, Cyprus, Union
DDH0032	DDH0098	core	1985	1990	Piedmont	Piedmont, NE Geochemical
DDH0099	DDH0288	core	1991		AMAX	Bondar Clegg
DDH0289	DDH0341	core	2008	Aug 2015	Romarco	Inspectorate
DDH0342	DDH0431	core	2008	2009	Romarco	Alaska
DDH0432	DDH511	core	2009	Sep-11	Romarco	Kershaw Mineral Lab (KML)
DDH512	DDH596	core	Oct-11	Jun-17	OceanaGold	KML
DDH0597	DDH1120	core	Jul-17	ongoing	OceanaGold	ALS
NDH0001	NDH0037	core	1985	1988	Nicor	Cone Geochemical
NRH0001	NRH0054	RC	1987	1988	Nicor	Cone Geochemical
RC0001	RC0031	RC	1985	1986	Piedmont	Union
RC0032	RC0183	RC	1986	1987	Piedmont	NE Geochemical
RC0184	RC1230	RC	1987	1990	Piedmont	Bondar Clegg, NE Geochemical
RC1231	RC1303	RC	1990	1992	Piedmont	Bondar Clegg
RC1304	RC1501	RC	1992	1994	AMAX	Bondar Clegg
RC1502	RC1527	RC	2008	2009	Romarco	Inspectorate
RC1528	RC2083	RC	Jan-10	Jan 2011	Romarco	Alaska
RC2084	RC2122	RC	Jan-11	Sep-11	Romarco	Acme, Chemex, KML
RC2123	RC2205	RC	Oct-11	Jun-15	Romarco	KML
RC2219	RC2235	RC	Feb-20	Mar-20	OceanaGold	ALS
RCT0001	RCT0157	RC/core	Apr-10	Jan-11	Romarco	Alaska
RCT0158	RCT0178	RC/core	Jan-11	Sep-11	Romarco	Acme
RCT0179	RCT0211	RC/core	Oct-11	Dec-12	Romarco	KML
WW0600	WW0673	RC	1975	1990	Piedmont	NE Geochemical

Source: OceanaGold, 2021

10.2 Sample Collection

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 prior to Romarco Minerals (pre-2007) accounts for approximately 18% of the data. The sample procedures applied to the historic drilling (i.e., drilling prior to Romarco Minerals Inc.) at Haile are not well documented. Having said this, approximately five years of mining has tested the veracity of the resource estimates which are based on this data. No material flaws have been identified.

The techniques described in this section reflect the procedures applied by Romarco and OceanaGold during the period 2007 to July 2020.

Reverse Circulation (RC) Drilling

RC drills are equipped with a cyclone and a rotary splitter. Most RC drilling at Haile was under wet conditions. Sample sizes were between 3 to 7 kg (20 and 30 lbs) with a minimum requirement of 3 kg (15 lbs). The standard size reflected a 15% to 20% split of the total drilled volume. Drill intervals were generally 1.5 m (5 ft) intervals and were collected in bags, to which which flocculant was added to settle fine particles. Sampling during advancement of each twenty-foot (6.1 m) rod was a continuous process. Chip samples were collected from the waste discharge and stored in plastic chip trays for geologic logging. The samples were stored on the ground or in the bed of a pickup truck to begin water drainage. Sample bags were collected at the end of each shift and transferred to the Haile sample storage area for initial drying.

Diamond Drilling

Diamond core drilling is by wireline methods and generally utilizes HQ and NQ size core with 63.5 mm and 48.3 mm diameters, respectively. Drill rods are 10 ft (3.1 m) long. Core is transferred from the core barrels into plastic core boxes at the drill rig by the driller.

Each core box can hold up to 10 ft (3.1 m) of cored stored in five rows each 2 ft (0.6 m) long. Core is gently broken by hammer as required to completely fill the boxes and marked on core as a mechanical break. Hole numbers and drill depths are marked on the outside of the core boxes and interval marker blocks are labeled and placed in the core box. Boxed whole core is covered with plastic lids and is transported to the core shed for logging and sampling by OceanaGold personnel.

Sample Recovery

Reverse Circulation: No primary RC sample weights were recorded for RC drilling, so RC recoveries cannot be directly calculated. However, 34,000 rotary split RC sub-samples were weighed by Romarco. Splitter ratio settings ranged from 8% to 17% and on the basis of back-calculating the range of likely total sample weights. RC recoveries appear to have been largely acceptable. As a precautionary measure, grade factors have been applied to gold grades where RC recoveries are estimated to be low on the basis of sub-sample mass and sampled at depths >200 m. These factors will remain until verified and/or replaced by diamond samples. Sensitivity analysis shows the impact on the resource estimates to be low (a few per cent globally). Given this mitigation, the residual risk is considered to be low.

Diamond Core: Core recoveries average 97% and are rarely less than 90%. There is no observed grade relationship between core recovery and grade. Core recovery in saprolite ranges from 10% to 50%. Minimal ore has been identified in saprolite.

10.3 Collar Locations and Downhole Surveys

Drillhole numbers are assigned and maintained by company geologists via an Excel tracking spreadsheet that records location, depth, azimuth, dip, start and end dates. Historical drillhole collar surveys by Piedmont and Cyprus were surveyed by theodolite and recorded on paper. Drill set ups in the field were by traditional Brunton compass methods to establish azimuths within 2° accuracy. Since February 2019, the Reflex Aziliner tool has been used for drillhole set ups within 0.3° accuracy.

Haile drillhole collars from 2007 to October 2017 were surveyed by Romarco and OceanaGold surveyors using digital GPS methods. Survey equipment included a SPS985 Rover receiver and a TSC3 SCS900 data receiver. Surveys utilized an on-site base station with sub cm accuracy. Since October 2017, collar surveys have been collected by the geologist overseeing drilling using a Juniper Geode GPS Receiver with accuracy to 2 cm.

Collar surveys are named by hole number, downloaded as .csv files and saved on a network drive. Collar coordinates are verified against planned coordinates by the geologist overseeing the drilling, and then imported into the acQuire database. Drillhole collars are marked with 10 cm PVC pipe and a metal tag. After verification of collar coordinates, drill sites are rehabilitated within 60 days as mandated by the operating permit.

Historical drillholes prior to Romarco in 2007 were not surveyed for downhole deviation. The majority of these holes intersected mineralization less than 75 m down-hole, so the locational uncertainty is unlikely to be large. A significant number of pre-Romarco holes in Snake Pit intersected mineralization at depths of 125 m or more but the majority of these have since been mined out.

Since 2007, all angle holes have been surveyed using the Reflex Sprint-IQ and EZ-Gyro survey tools. The tools are stored in a case at the Haile OceanaGold office and are calibrated at the Reflex office in Tucson, Arizona once a year. Multishot surveys are recorded down hole for azimuth and dip every 6.1 m (20 ft). Survey data are downloaded and verified using Imdex HUB-IQ software by the geologist overseeing the drilling. Upon verification, the data are imported into the acQuire database.

Original paper or printed survey records are located in the OceanaGold exploration office and are filed by hole number with other relevant drill data. Survey data are also stored digitally in the acQuire database. As part of a company-wide metrification process, OceanaGold transformed all surface and drillhole data from the South Carolina NAD27 coordinate system to the UTM NAD83 zone 17N system in November 2016. Coordinate transfer was verified by geologists and engineers and no issues were identified. Coordinate transformation was supported and utilized by SRK personnel during the Horseshoe underground evaluation.

10.4 Significant Results and Interpretation

Haile has a relatively large percentage (18% of drill meters) of historical holes drilled before 2007 for which sampling methods have not been well documented. Drilling since that time is summarized in this report. For the historical drilling, collar coordinates were written on paper and therefore some field note errors may have occurred during surveys or copying into databases. Historical holes were also not surveyed for downhole deviation and their location in 3D space cannot be verified. QA/QC records are sparse for the historical drillholes and QA/QC methodology was not documented.

As summarized earlier, in 2016, during the translation of the Romarco Minerals drilling database to OceanaGold's standard acQuire database platform, where available, original source assay and survey data were used for the acQuire translation and database validation. There was a further internal database review in late 2018 / early 2019. No material errors were identified and OceanaGold believe that the database is sufficient for the purposes of resource estimation. Five years of resource model to mine to mill reconciliation performance have identified no material data flaws.

The database used for this technical report includes 3,443 holes in the Haile district. Drillhole data are securely stored in OceanaGold's acQuire database. Drillhole collar locations, downhole surveys, geological logs, geotechnical logs and assays have been verified and used to build 3D geological models and in grade interpolations. Geologic interpretation is based on structure, lithology and alteration as logged in the drillholes.

Significant mineralization has been recorded in drillholes at nine named gold deposits at Haile within a 3.5 km by 1 km area since drilling commenced in the 1970's. Hole depths used for resource estimates typically range from 120 to 500 m and are dependent on depth of mineralization. Drillhole spacing typically ranges from 30 to 60 m. Resource drilling at Haile has predominantly been conducted by core holes angled to the southeast at dips of -45° to -75°, roughly perpendicular to regional foliation and generally also bedding. Intersection angles between drillholes and mineralized zones are typically in the range of 40° to 70° depending on drill rig access. Significant drillhole intercepts therefore range from 70% to 100% of true mineralized widths. Many holes have been angled to the north, northwest and west to infill drill gaps where collar access is restricted by infrastructure (leach pads, pits, haul roads, lakes, wetlands). Despite these challenges, the orebody sampling is believed to provide an acceptable basis for resource estimation.

Sample interval lengths are typically 1.5 m (5 ft) and increase to 3.0 m in potentially barren rocks in relatively homogenous rock units. Variable sample interval lengths are selected by the geologists based on core logging to reflect changes in rock type, alteration, mineralogy and structure.

11 Sample Preparation, Analysis and Security

11.1 Sample Preparation for Analysis

Reverse Circulation Drilling

The reverse circulation drilling at Haile typically used 16 cm drill bits. Sample intervals were predominately 1.5 m. The RC rigs were equipped with a cyclone and a rotary splitter. Most RC sampling at Haile was under wet conditions. Water injection was typically 15 to 19 L/min above the water table and decreases to 4 L/min when groundwater is encountered. Wet samples were bagged, drained and allowed to settle (aided by flocculent) before being transported to a storage facility for initial drying. Sample sizes were generally between 9 and 14 kg dry mass, representing a 11% to 17% split of the total sample mass.

The reverse circulation (RC) sample bags were transferred by company personnel to the Haile sample handling facility where they are prepared for shipment to a lab. RC samples were prepared at either the Kershaw Mineral Lab (KML) in Kershaw, South Carolina, the AHK Geochem (AHK) preparation facility in Spartanburg, South Carolina, or the ALS preparation facility in Tucson Arizona.

Lithological chip samples are retained in chip trays, labelled with the drillhole number and depth intervals in permanent marker.

Diamond Drilling

Diamond core drilling is by wireline methods and generally utilizes HQ and NQ size core 6.35 cm and 4.8 cm 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 company personnel

RC Samples

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

Samples follow one of two paths:

- Some samples are weighed, and sample number tags added to the bags. The samples are poured through a Jones splitter to reduce the size to roughly 2.7 kg (6 lbs) for shipment to the sample lab. Coarse rejects are kept in their original sample bags and stored on site on pallets.
- Alternatively, samples are staged at Haile and placed in containers for direct shipment to KML, AHK, or ALS.

Core Samples

At the core logging facility, the core is cleaned, measured, and photographed. Geotechnical and geologic logging are completed on the whole core. All logging and sampling handling are conducted by OceanaGold personnel. Data collecting during core logging include structure, rock type, alteration, mineralogy, Rock Quality Designation (RQD), core recovery, hardness and joint condition. Alteration is logged as relative intensity and includes weak, moderate and strong categories. Mineralogy is visually estimated to the nearest 0.1%. Standardized templates are used for all logging with drop down menus. Geologists routinely review core together and compare notes to ensure accuracy and consistency. Density samples are collected every 10 m (33 ft) and use the water immersion method to measure specific gravity. Competent core at Haile does not require plastic or wax coatings for density measurements. Paper logs are entered into an Excel spreadsheet and then imported in the acQuire database by the admin assistant. Logs are periodically checked by the geologists for accuracy and completeness. Tablet-based geology logging in Excel was initiated in 2017 and enables logs to be directly uploaded into acQuire.

Logging is conducted in the Imperial system using feet due to the 10 ft drill rods. Data are converted to metric units after being imported into the acQuire database. The logging geologist assigns the sample intervals and sample numbers prior to core sawing. Sample ID tags are placed in the core boxes. Sample lengths are typically 5 ft (1.5 m) and can range in length from 1 ft (0.3 m) to 10 ft (3.1 m). Geologic sample breaks may be selected by the geologists based on contacts or structural boundaries. 'No sample' intervals are marked by orange flagging tape in surficial fill or rubble zones and in massive barren diabase dikes exceeding 50 ft (15 m) in thickness.

Core is sawed in half along the core axis using circular masonry blades and then placed into sample bags labelled with the sample ID. Paper ID tags are also placed into the bags. Saprolite zones are manually cut with a putty knife. The saw or knife are cleaned between each sample. A brick or barren rock sample is sawed between intervals to minimize cross-contamination. The cooling water for the saw is not recycled and is discharged into a permitted pond.

Core samples are delivered to the sample preparation facilities. Core is prepared primarily at the ALS facility in Tucson, Arizona but has also been prepared at the company-owned Kershaw Mineral Lab (KML) facility in Kershaw, South Carolina and the AHK preparation facility in Spartanburg, South Carolina.

11.1.1 Off-Site Sample Preparation

AHK, Spartanburg, South Carolina (ISO/IEC 17025 accredited)

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

- Sample Preparation
- Inventory and log samples into the laboratory LIMS tracking system
- Print worksheets and envelope labels
- Dry samples at 65°C (150°F)
- Jaw crush samples to 80% passing 2 mm
- Clean the crusher between samples with barren rock and compressed air
- Split sample with a riffle splitter to prepare the sample for pulverizing
- Pulverize a 250 g sample to 90% passing 150-mesh (0.106 mm)
- Clean the pulverizer between samples with sand and compressed air
- Ship about 125 g of sample pulp for assay
- Coarse rejects are returned to Haile for storage

- The 125 g 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, Alaska for analysis.

Kershaw Mineral Laboratory (KML), Kershaw, South Carolina (ISO/IEC 17025 accredited)

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

- Sample Preparation
- Inventory and log samples into the laboratory LIMS tracking system
- Print worksheets and envelope labels
- Dry samples at 93°C (200°F)
- Jaw crush samples to 70% passing 10-mesh (2 mm)
- Clean the crusher between samples with barren rock and compressed air
- Split sample with a riffle splitter to prepare the sample for pulverizing
- Pulverize a 450 g sample (± 50 g) to 85% passing 140-mesh (0.106 mm)
- Clean the pulverizer between samples with sand and compressed air
- Approximately 225 g of pulp sample is sent for fire assay
- Coarse rejects and reserve pulps are returned to Haile for storage.

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

ALS, Tucson, Arizona (ISO/IEC 17025 accredited)

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

- Weigh Samples
- Inventory and log samples into the laboratory tracking system
- Dry samples, if excessively wet, at up to 120°C (248°F)
- Jaw crush samples to 70% passing 10-mesh (2 mm)
- Clean the crusher between samples with compressed air
- Split sample with a Boyd rotary splitter to prepare the sample for pulverizing
- Pulverize a 250 g sample to 85% passing 75 microns
- Clean the pulverizer between samples with compressed air
- Prepared pulps are shipped to an internal ALS laboratory for analysis
- Approximately 225 g of pulp sample is sent for fire assay
- Fire assay performed on a 30g sample with an Atomic Absorbtion Spectroscopy finish
- NaCN leach test on a 30 g sample with an Atomic Absorbtion Spectroscopy finish
- For samples over 10 g/t Au, fire assay is performed on an additional 30 g sample with a gravimetric finish
- Coarse rejects and reserve pulps are returned to Haile for storage.

11.2 Sample Analysis

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

- Inventory the samples and create worksheets
- Insert Quality Control samples of two duplicates, one certified standard, and one blank in each batch of 40 samples.

- Fire assay a 30 g aliquot for gold with 4-acid digestions and Atomic Absorption finish.
- Analyze 0.50 g samples for Multi-Element by inductively coupled plasma mass spectrometry (ICP-MS).
- Review the internal QC results and check as required.
- Review and sign off on final values including the internal check assays.
- Issue the final report and certificate of assay.
- Deliver the certificate to the client.

AHK is ISO/IEC 17025 accredited for all facilities that handle Haile samples.

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

- Inventory the samples and create worksheets.
- Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 20 samples.
- Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
- If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is cyanide leached for gold using Atomic Absorbtion finish
- If the gold assay result from step 3 is greater than or equal to 10 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish.
- Multi-Element ICP analysis is performed as requested.
- Carbon and sulfur determinations are performed as requested.
- Review the internal QC results and perform check assays as required.
- Review and sign off on final values including the internal check assays.
- Issue the final report and certificate of assay.
- Deliver the certificate to the client.

ALS is ISO 9001 certified and ISO/IEC 17025 accredited. Coarse rejects and returned samples are stored at Haile under the control of company personnel. During off-shift hours, a Deputy Sherriff is on site providing security for the site and sample storage facility.

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

- Inventory the samples and create worksheets.
- Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 24 samples.
- Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
- If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish, and 0.5 g of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
- Multi-Element ICP analysis is performed as requested.
- Carbon and sulfur determinations are performed as requested.
- Review the internal QC results and perform check assays as required.
- Review and sign off on final values including the internal check assays.
- Issue the final report and certificate of assay.
- Deliver the certificate to the client.

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

11.3 Check Assays

Early in the Romarco drill program, samples were sent to the Inspectorate Lab in Reno, Nevada for preparation and assay. Inspectorate is an ISO 9001 certified laboratory. Check assays were sent to ALS-Chemex in Reno, Nevada. Sample analysis procedures at ALS are as follows:

- Inventory the samples and create worksheets.
- Insert Quality Control samples of one duplicate, one certified standard, and one blank in each batch of 20 samples.
- Fire assay 30 g of pulp sample for gold, with Atomic Absorption finish.
- If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is cyanide leached for gold using Atomic Absorbtion finish
- If the gold assay result from step 3 is greater than or equal to 10 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish
- If the gold assay result from step 3 is greater than or equal to 3 g/t Au, an additional 30 g of pulp sample is fire assayed for gold using gravimetric finish, and 0.5 g of pulp sample is analyzed for silver using a 4-acid digestion with Atomic Absorption finish.
- Multi-Element ICP analysis is performed as requested.
- Carbon and sulfur determinations are performed as requested.
- Review the internal QC results and perform check assays as required.
- Review and sign off on final values including the internal check assays.
- Issue the final report and certificate of assay.
- Deliver the certificate to the client.

11.4 Quality Assurance/Quality Control Procedures

11.4.1 Standards

Certified standards are routinely inserted at a rate of one in twenty samples (5%) per industry guidelines. Standards used by Romarco and OceanaGold are purchased from and certified by Rocklabs and include six standards of various grades. Five are oxide standards and one is sulfidic.

11.4.2 Blanks

Blanks are routinely inserted at a rate of one in twenty samples (5%). Blanks used by Romarco and OceanaGold include commercially available marble, sand, quartz pebble.

11.4.3 Duplicates

No duplicates samples were collected or analyzed.

11.4.4 Actions and Results

QA/QC data and graphs are generated from the acQuire database. Standards and blanks greater than 20% of the expected value are re-assayed for failed batches (20 samples). Reruns have been acceptable, and those values were imported into the acQuire database.

Security Measures

RC coarse rejects and returned samples are stored and secured at Haile where they are under the control of OceanaGold personnel. Pulps are stored in converted turkey barns at Haile with the coarse rejects. RC chip trays are stored in the exploration office. Boxed core is palletized and stored in a sand lot on the south side of the mine property. Pallets are covered by tarps and aluminum tags with hole IDs are attached to each pallet.

11.5 Opinion on Adequacy (Security, Sample Preparation, Analysis)

Haile has a relatively large percentage (18% of drill meters) of historical holes drilled before 2007 for which sampling methods have not been well documented. There is no evidence of material problems with the pre-2007 drilling, sampling and analyses. Furthermore, over four years of mining has tested the veracity of the resource estimates which are based on this data. No material flaws have been identified. Drilling, sampling and analyses since 2007 are summarized in this section.

Sample collection, preparation and analysis are according to industry standards. All labs used by Romarco and OceanaGold are certified to ISO-9001 standard or 17025 accredited for gold and silver through the Standards Council of Canada. The primary external lab used for check assays at ALS Reno is both ISO-9001 certified and 17025 accredited.

Core, pulp and RC sample storage are considered secure. Sample transport is by company personnel between secure facilities and by approved couriers to external labs. No significant risks have been identified for sample contamination or sample exchange. No samples have been reported as missing during transport or as tampered with upon receipt at the lab.

All Haile drillhole data (assays, logs, surveys) are stored in the secure acQuire database, which is managed by the senior database specialist in New Zealand. Assay data can only be imported by the database specialist. The database specialist has no direct reporting relationships to the Haile geologists or to the Director of Exploration. The acQuire database is an industry certified database. Database changes are tracked and verified. Strict data importing and verification protocols must be followed to avoid, for example, overlapping or missing intervals, mismatched hole depths in different fields, duplicate hole IDs or sample numbers, and invalid logging codes.

In the opinion of the qualified person, the sample security, sample preparation and analyses are adequate for the purposes of resource estimation.

12 Data Verification

Haile has a relatively large percentage (18% of drill meters) of historical holes drilled before 2007 for which sampling and analytical methods have not been well documented. There is no evidence of material problems with the pre-2007 drilling, sampling and analyses. Furthermore, over four years of mining has tested the veracity of the resource estimates which are based on this data. No material flaws have been identified.

During 2016, the Romarco Minerals drilling database was translated to OceanaGold's standard acQuire database platform. Where available, original source assay and survey data were used for the acQuire translation and database validation. There was a further internal database review in late 2018 early 2019. No material errors were identified.

Verification of drilling, sampling and analyses is discussed as Pre-Romarco, Romarco and OceanaGold data groups in the sections below.

12.1 Data Validation of Pre-Romarco Holes

Data validation was conducted by OGC Exploration geologists in 2019 for pre-Romarco (pre-2008) RC and core holes. Haile has been drilled by multiple companies since 1975 (Table 10-1 in section 10.1). A total of 1,775 holes representing 54% of the resource database were validated using AuBest values stored in the acQuire database. This includes drillholes RC0001-1501 (n=1403), DDH0001-0288 (n=288), NDH0001-0037 and NRH0001-0054 (n=84) drilled between 1975 and 1994. Key projects were to compile, sort, file and record source data using original logs and assay certificates from dozens of binders and hard copy files for each drillhole. Differences between assays, depths, dips, azimuths, downhole surveys and collar coordinates were recorded and evaluated in spreadsheets. All paper files and logs are securely stored in OceanaGold's exploration office at Haile.

No major or systematic errors were identified and there is no material impact to reserves or resources based on validation of pre-2008 drillhole data (Jory, 2019). Legacy drillhole data from 1975 to 1994 stored in acQuire are regarded as reliable and accurate. Romarco and OGC data are also reliable and accurate. Minor data corrections were made for some legacy gold assays, collar coordinates, hole depths and interval depths. Data validation showed that 0.77% of DDH holes and 5.4% of RC holes required corrections based on differences >0.007 ppm Au between acQuire AuBest values and assay certificates or assay sheets. Many of the RC holes had negligible errors <1% of the acQuire assay vs. original assay. Most of the suspect holes are from the early Cyprus and Piedmont drill campaigns. AMAX holes are of high confidence and include certified assays by Bondar Clegg with fire and gravimetric assays.

As a precautionary measure, diamond core drilling targeted within reserve pit designs has been completed in areas with large numbers of legacy (pre-Romarco) RC holes, including Snake, Red Hill and Haile. The Mill Zone, Ledbetter, Horseshoe and Small pits are largely drilled with core holes and have no RC grade bias risk.

12.2 Verification of Romarco and OceanaGold Data

21,732 standards are recorded in the OceanaGold AcQuire database. There are an additional 16,783 blanks. The number of standards submitted by year (Table 12-1) varies and largely reflects the drilling activity in that year. The performance of these standards revealed no material problems with laboratory performance. QA/QC inserts decreased in 2018, 2019 and 2020 proportional to decreased drill meters; blanks and standards are inserted at industry normal frequencies of 1 in 20 samples.

Table 12-1: Number of Standards Inserted by Year

Year	No. of Standards
2008	242
2009	2,640
2010	4,981
2011	7,318
2012	2,653
2013	475
2014	52
2015	233
2016	926
2017	907
2018	459
2019	457
2020	223
2021	168
Total	**21,734**

Source: OceanaGold, 2021

12.3 Romarco Data Verification

In addition to the checks done by OceanaGold during database translation and the 2018/2019 review, the following checks have been made by IMC for drilling completed by Romarco (2008 to 2014).

- A comparison of certificates of assay from the laboratory versus the Romarco computerized data base to check the reliability of data entry
- Statistical analysis of the standards reference materials that were inserted by Romarco for analysis by the assay lab
- Statistical analysis of the blank samples that were inserted by Romarco for analysis by the assay lab
- Statistical analysis of the check samples that were submitted by Romarco to a third-party laboratory

The qualified person has reviewed the checks and believes that the data are of acceptable quality for the purposes of resource estimation. In 2016, OceanaGold undertook a program of database verification for drilling at the Horseshoe Underground deposit:

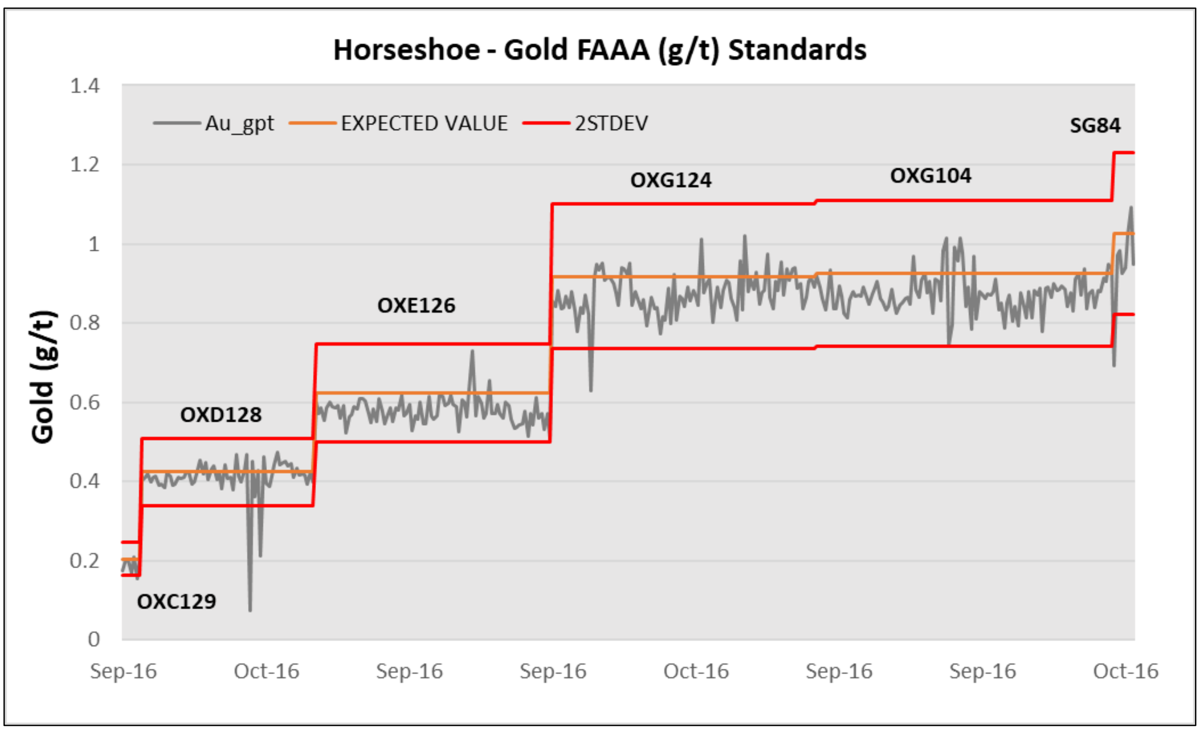
- Assay Verification – 5% check of assay values
- Collar Verification – 100% check of collar locations
- Downhole Survey Verification – 100% check on downhole surveys
- Standard and Blank QA/QC
- KML vs. ALS Horseshoe assay comparison

The review identified no material flaws. The Horseshoe data is considered of acceptable quality for the purposes of resource estimation. The KML vs. ALS check assay comparison study for Horseshoe concluded that “statistical variance from these studies for AuAA vs. AuFA comparisons (n=512) between KML and ALS Tucson indicate that the KML lab is 5% to 10% low, or conversely that the ALS lab is 5% to 10% high. KML adjusted their AA dilution process in late October to achieve better fit with expected values". The KML assay data were validated and used in the resource model.

12.4 Horseshoe Data Verification 2016

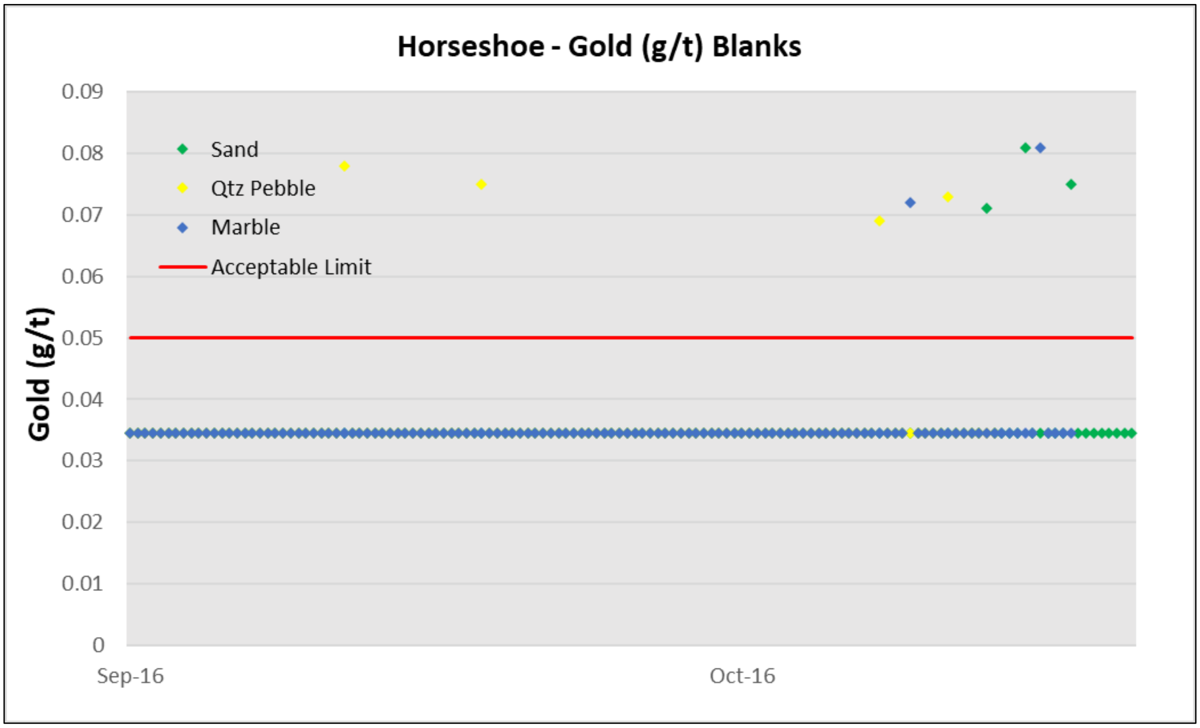
Horseshoe drillhole data were validated by OceanaGold in late 2016. Validation included assays, collar locations, and downhole surveys in the acQuire database for Romarco and OceanaGold drillholes. This includes a 5% check of assay values and a 100% check of collar coordinates and downhole surveys. An analysis of standards and blanks from the 2015 to 2016 Horseshoe drilling program was also conducted. All Horseshoe drilling was core using OceanaGold LF90 drills and company drillers. Sample preparation and assays were conducted by OceanaGold's Kershaw Mineral Lab (KML) at Haile. Check assays on sample pulps were performed by ALS in Tucson, AZ. No significant errors were identified by the study.

Standards and blanks greater than 20% of the expected value were scrutinized and were re-assayed in some cases. Of the 1,699 controls submitted for FA-atomic absorption (AA) during the Horseshoe drilling campaign, 39 standards and blanks (2.3%) are outside acceptable limits. Most failed controls were in unmineralized zones. Validated blanks were used for assays up to 0.080 g/t. Box and whiskers plots for standards and blanks were generated and evaluated as shown in Figure 12-1 and Figure 12-2 respectively. Batches with failed standards in mineralized zones were re-assayed at KML and passed the second time.



Source: OceanaGold, 2016

Figure 12-1: Horseshoe 2016 Standards



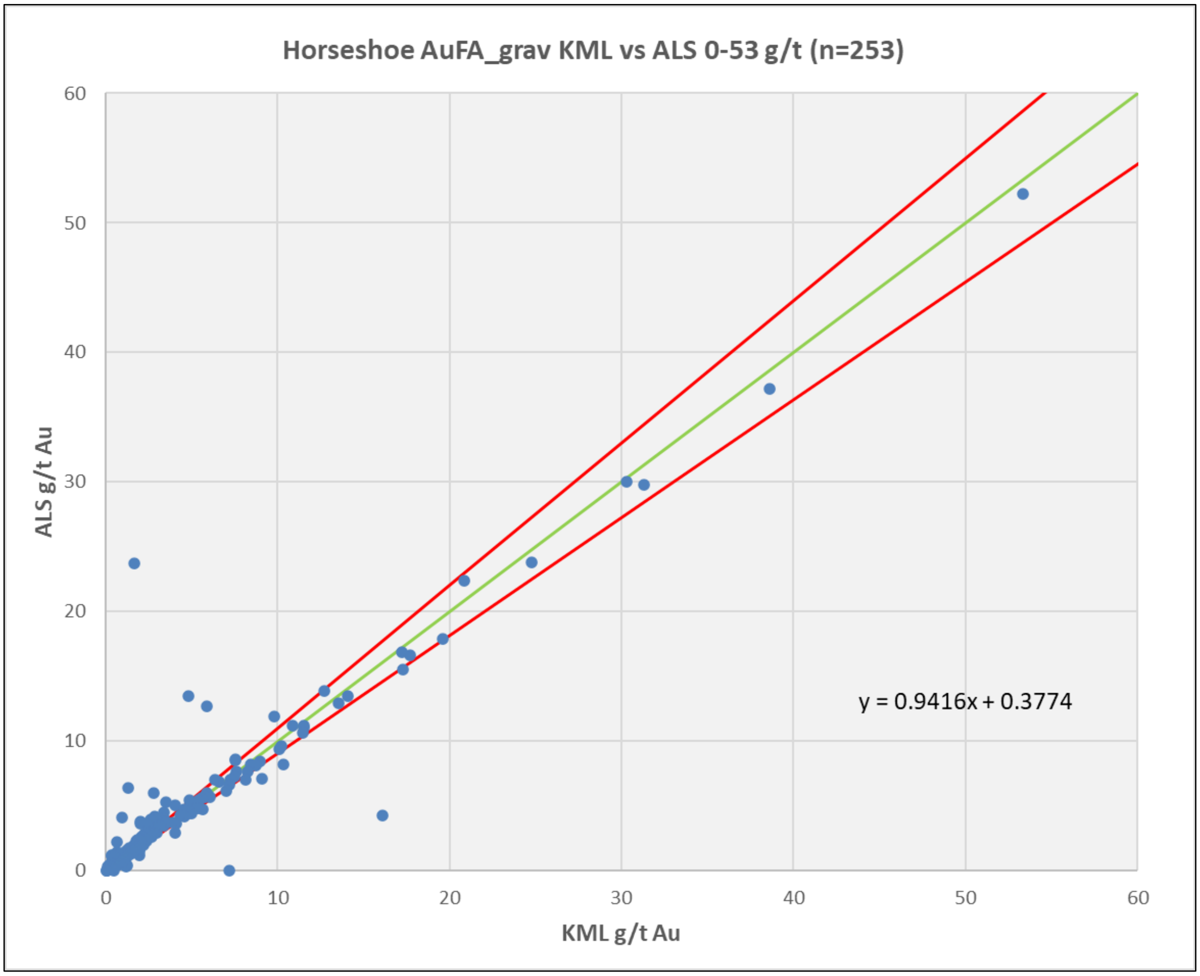
Source: OceanaGold, 2016

Figure 12-2: Horseshoe Blanks

A check assay program was conducted for the Horseshoe drilling at ALS Tucson, AZ. Conclusions were:

- KML is 10% lower for AuAA values than ALS for 259 Horseshoe sample pairs in the 0.5 to 3 g/t range (R2=0.73).
- KML is 5% lower for AuFA values than ALS for 253 Horseshoe samples pairs in the 0 to 53 g/t range (R2=0.90) shown in Figure 12-3 Assays >100 g/t Au (n=18) show poor correlation when comparing KML to ALS results, likely due to the presence of coarse gold and difficulty in achieving assay precision.

Statistical variance from these studies for AuAA vs. AuFA comparisons (n=512) between KML and ALS Tucson indicate that the KML lab is 5% to 10% low, or conversely that the ALS lab is 5% to 10% high (Figure 12-3). KML adjusted their AA dilution process in October 2016 to achieve better fit with expected values. There were no changes to fire assay procedures at KML. It is suggested that gold assay upgrades of 5% can be applied for internal planning, but that no upgrade be used for external reporting.



Source: OceanaGold, 2021

Figure 12-3: Horseshoe AuFA_grav KML vs. ALS 0-53 g/t (n=253)

This section focuses on verification of the drilling, sampling, and assaying completed for the data included in the Horseshoe block model completed by OceanaGold Exploration on November 20, 2016.

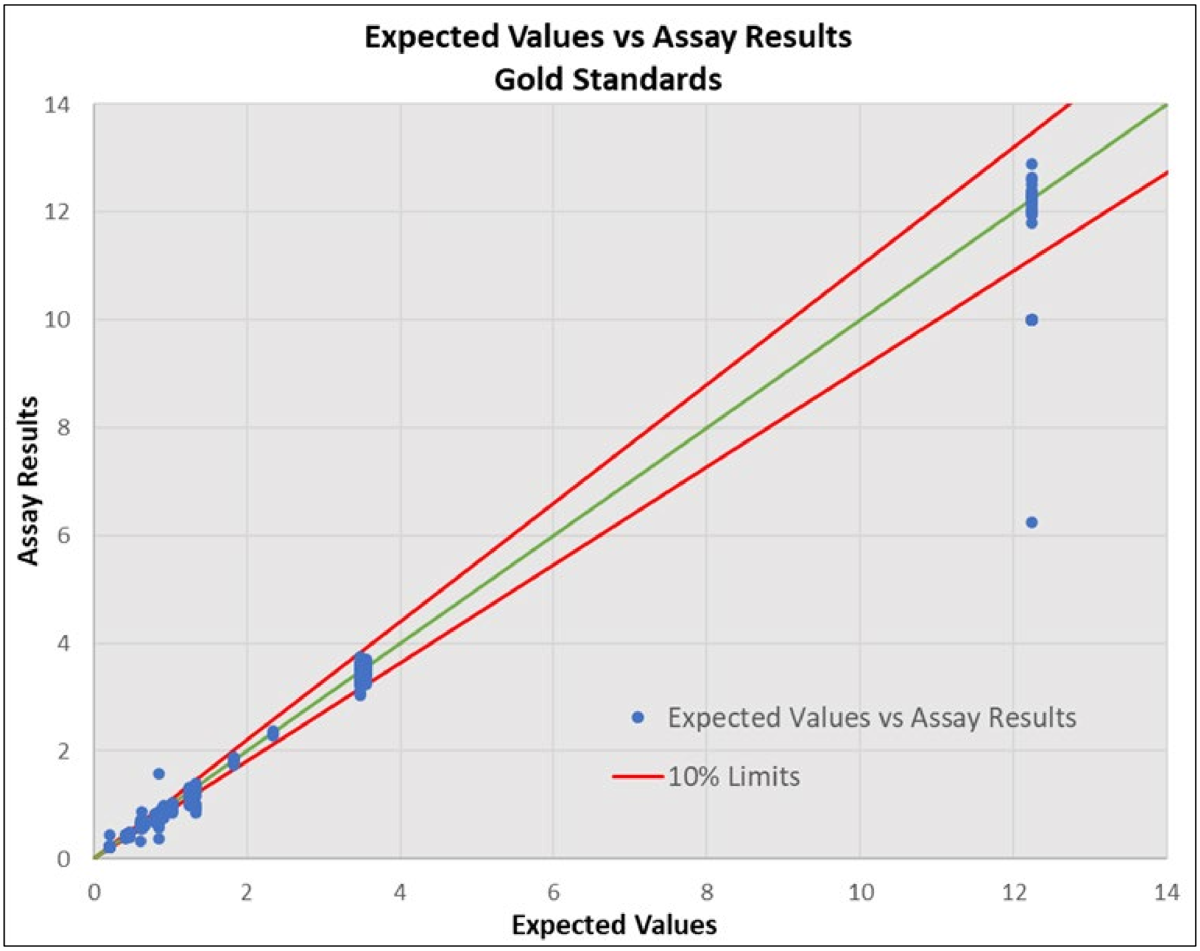
12.5 Haile QA/QC ALS July 2017-January 2022

During the period July 2017-January 2022, exploration and resource definition samples were submitted to ALS laboratories. Samples were prepared at the Tucson, AZ lab and certified assays were done in Reno, NV. Analyses for Au were by Fire Assay, and any assay returning over 10 g/t Au was re-assayed with a gravimetric finish. Analyses have been reported (and are stored) in g/t in the acQuire database. All historic data is converted to g/t for resource and reserve purposes.

12.5.1 July 2017- January 2022 CRM Performance

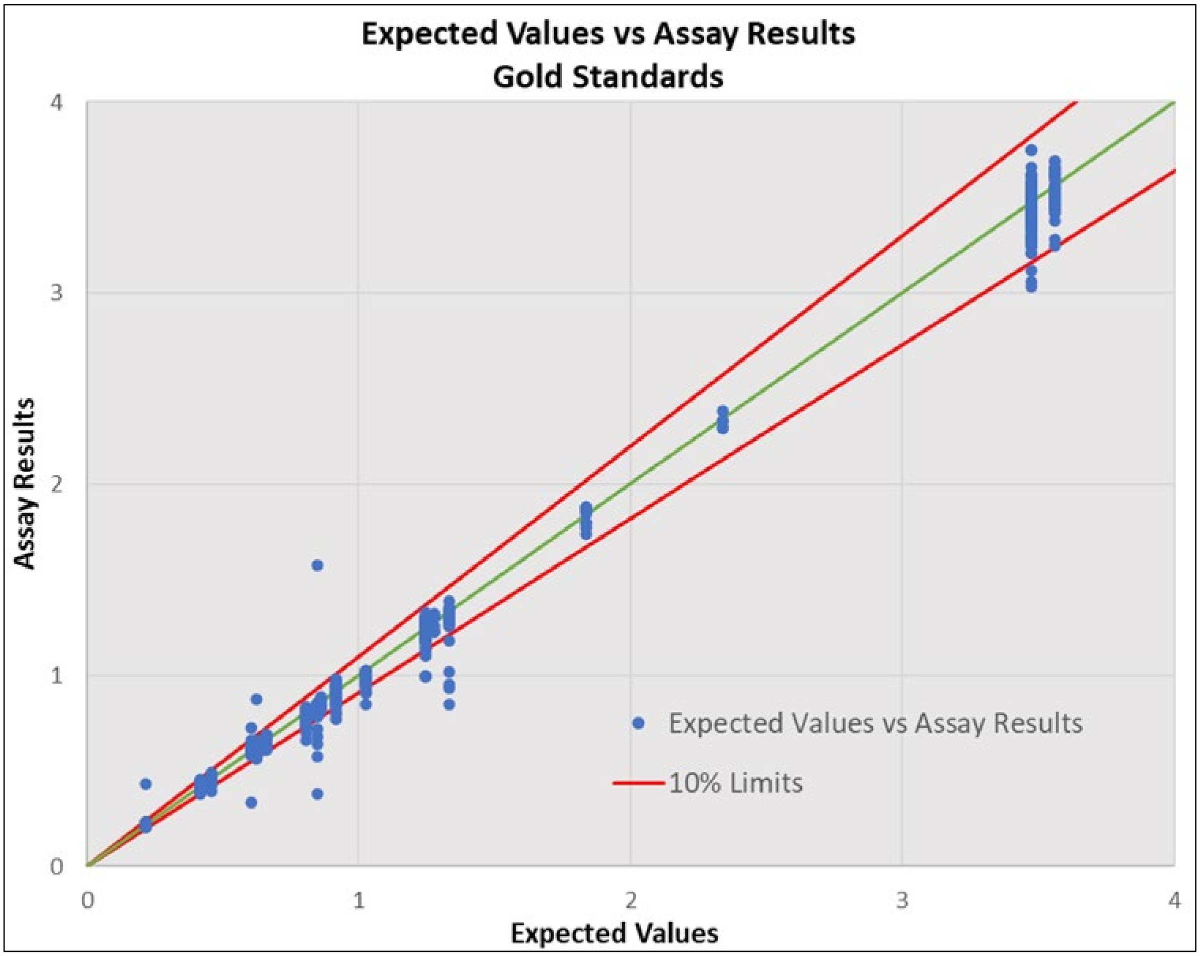
ALS Reno laboratory accuracy was monitored by insertion of commercially Certified Reference Materials (CRMs) into the sample stream. CRMs were inserted at a rate of one per twenty samples (5%) and alternated with blank insertions at one per twenty samples (5%). Hence, alternating blanks and CRMs were inserted at a rate of one in 10 (10%) per industry guidelines. A total of 21 different

CRMs sourced from Rocklabs were submitted in sample batches for a total of 1,725 CRM analyses during the four and a half-year period. Fourteen of the 21 CRMs had more than 30 insertions into the sample stream. CRMs include oxide, sulfide, and high silica standards. Figure 12-4 shows CRM values plotted against the expected values. Purple triangles show the averaged results for a CRM, while greyed boxes show the number of submissions for each CRM. Figure 12-5 shows a zoomed in view of the most commonly inserted CRMs. No obvious bias was observed within the CRM expected versus actual data. Relative standard deviation of all CRMs was very good at 2% to 4% of expected values. Results confirm excellent precision and accuracy of assays provided by ALS Reno for Haile resource calculations that are within industry guidelines. Results from blanks showed no contamination of samples used for resource calculations.



Source: OceanaGold, 2021

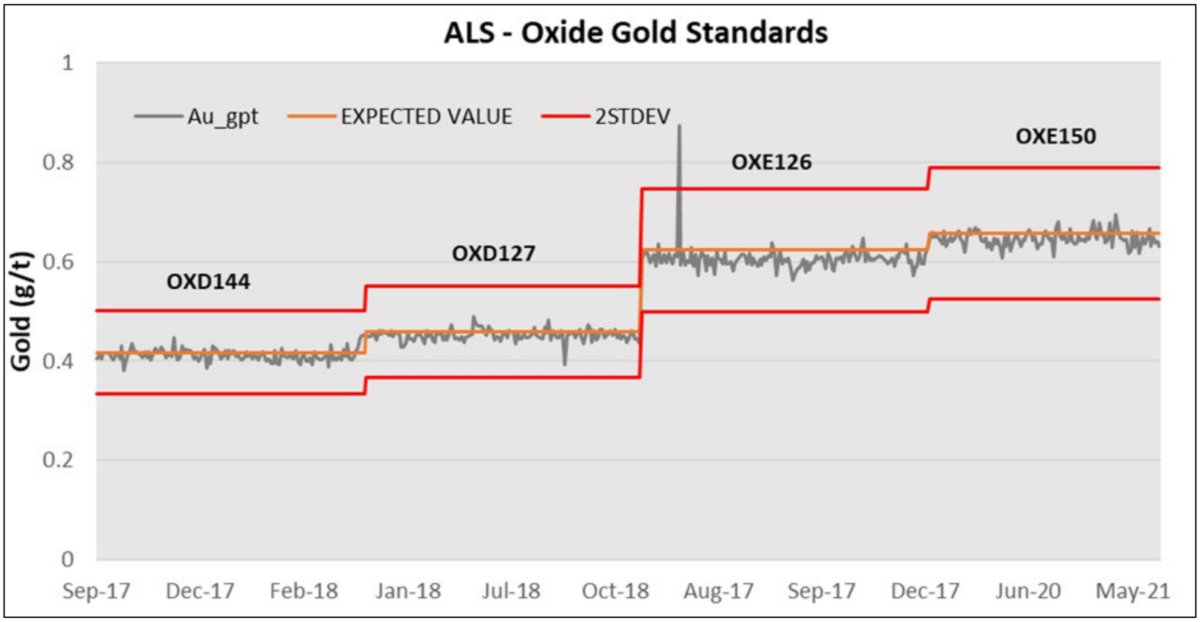
Figure 12-4: July 2017-Current CRM Analyses versus Expected Value (n=1,725)



Source: OceanaGold, 2021

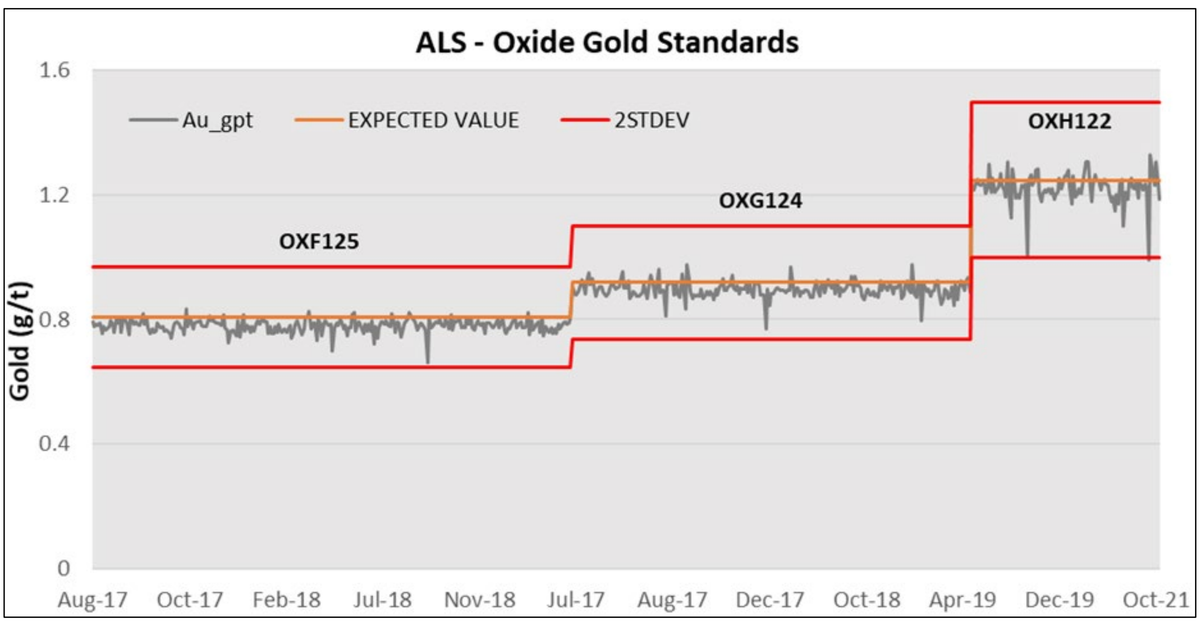
Figure 12-5: July 2017-Current CRM Analyses versus Expected Value <4 g/t (n=1,701)

Standard control charts were plotted for commonly used CRMs. Examples of oxide (OxF125, OxG124), sulfide (SG84), and high silica (HiSilK2) CRMs are show in Figure 12-6 through Figure 12-9. No obvious trends in the process mean are apparent in the longer run CRM results, and there are only minor variations in precision. If a CRM returned a value greater than 20% above or below the expected value, then all intervals within the failed batch and the nearest passing CRM or Blank were successfully rerun and uploaded into the database.



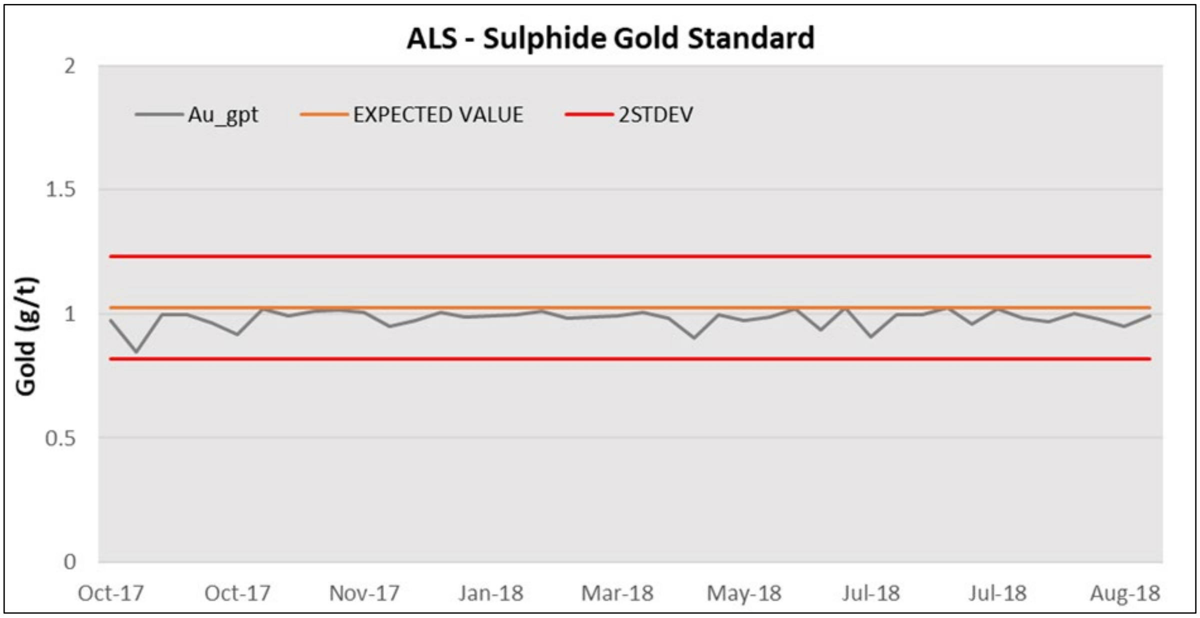
Source: OceanaGold, 2021

Figure 12-6: July 2017-Current, Control Chart, OxD144 (n=130), OxD127 (n=133), OxE126 (n=139), and OxE150 (n=112)



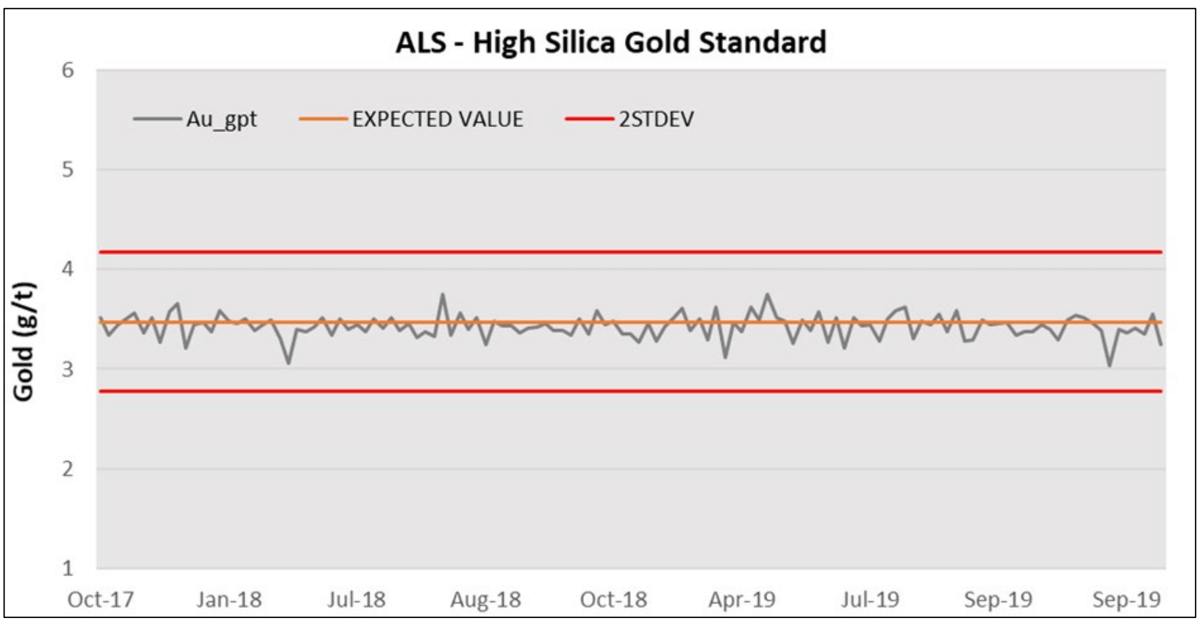
Source: OceanaGold, 2021

Figure 12-7: July 2017-Current, OxF125 (n=268), OxG124 (n=218), and OxH122 (n=106)



Source: OceanaGold, 2021

Figure 12-8: July 2017-Current, Control Chart, SG84 (n=42)

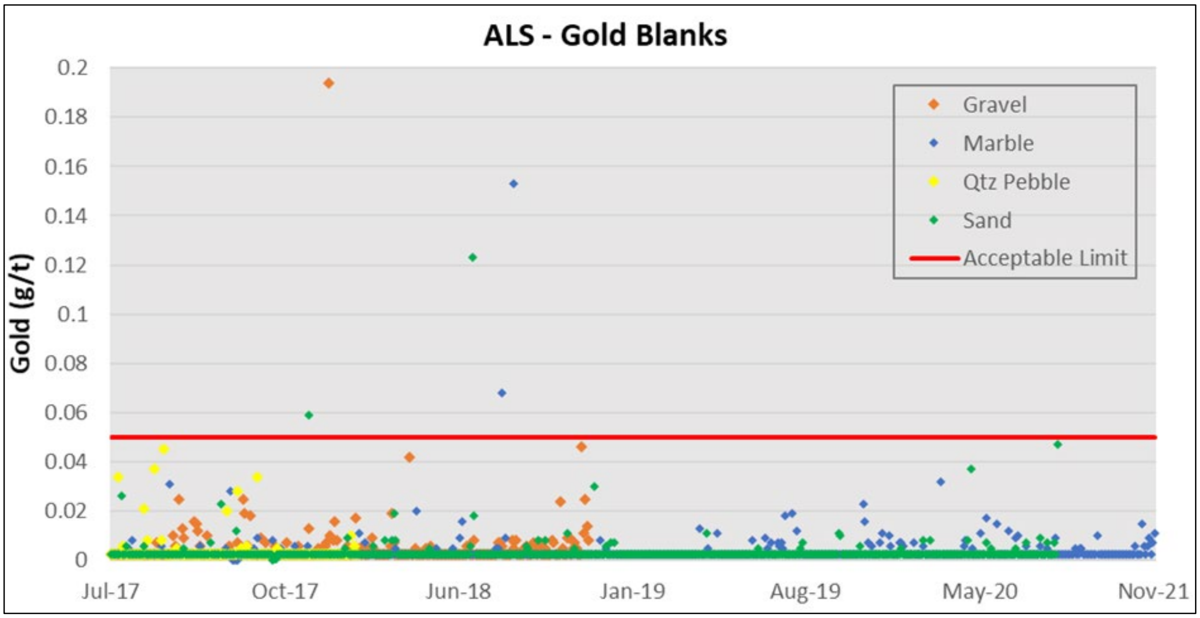


Source: OceanaGold, 2021

Figure 12-9: July 2017-Current, Control Chart, HiSiIK2 (n=63)

12.5.2 Contamination Monitoring

Contamination is monitored by insertion of blank materials. From July 2017-January 2022 a total of 1,622 blank samples of four different materials were inserted: Marble, Sand, Gravel, and Quartz Pebble. Results from the ALS Reno laboratory are shown in Figure 12-10. Two minor non-compliances were observed in the reporting period. Lab detection limit (LDL) is 0.005 ppm Au and control limit used is 10 times the LDL (i.e., 0.050 ppm Au). If a blank returned a value greater than 0.050 ppm, then all intervals between the failed Blank and the nearest passing Blank or CRM were rerun. Overall, there is no indication of contamination during sample preparation.



Source: OceanaGold, 2021

Figure 12-10: July 2017-Current Blank Insertions (n=1,622)

The qualified person has reviewed the July 2017 to current data and believes it to be of acceptable quality for the purposes of resource estimation.

12.5.3 Statement of Data Adequacy

The qualified person believes that the data reviewed above, including drilling prior to 2007 and subsequent drilling by Romarco and OceanaGold, is adequate for the purposes of resource estimation.

13 Mineral Processing and Metallurgical Testing

Sample preparation and characterization, grinding studies, gravity concentration tests, whole ore leach tests, flotation tests and leaching of flotation tailings and flotation concentrate tests were completed to determine the metallurgical response of the ore.

Samples of ore were collected by HGM for metallurgical testing. A series of metallurgical testing programs have been completed by independent commercial metallurgical laboratories. The test work indicated that the ore will respond to flotation and direct agitated cyanide leaching technology to extract gold. The results of the test programs are available in the following reports:

- Phillips Enterprises, LLC (Phillips) 17 September 2008, Progress Report #2 Process and Metallurgical Testing on Haile Gold Mine Ore Project No. 082003
- Pocock Industrial Inc. (Pocock) Salt Lake City, Utah, May 2009, Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Romarco Minerals Haile Gold Project
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, September 16, 2009, Romarco Minerals, Inc. Haile Gold Project, Metallurgical Report
- Metso Minerals Industries Inc. (Metso), York, Pennsylvania, December 7, 2009, Test Plant Report No. 20000134-135
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, March 31, 2010, Romarco Minerals, Inc. Work Index Data for Haile Composite Sample
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, March 31, 2010, Romarco Minerals, Inc. Metallurgical Testing of Ledbetter Extension Samples
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, May 27, 2010, Romarco Minerals, Inc. Flash Flotation, Cyanide Destruction & Leaching of Concentrate and Tailing for Haile Composites
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, September 27, 2010, Romarco Minerals, Inc. Optimization of Leaching of Flotation Concentrate
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, August 2010, Metallurgical Testing of Horseshoe Zone Samples
- Metso Minerals Industries, Inc. (Metso), York, Pennsylvania, February 2011, Stirred Media Detritor and Jar Mill Grindability Test on Bulk Flotation Concentrate T11-04
- KML Metallurgical Services, (KML), Kershaw, South Carolina, December 27, 2012, HGM Years 1 – 3 Silver Characterization Project Test Report
- Resource Development Inc. (RDi), Wheat Ridge, Colorado, June 6, 2011, Production of Flotation Concentrate and Confirmation Testing of Flowsheet
- G&T Metallurgical Services Ltd (G&T), Kamloops, Canada November 24, 2011, Flotation & Cyanidation Testing on Samples from the Horseshoe Deposit, Haile Gold Mine KM3076;
- Gekko Global Cyanide Detox Group (Gekko), Ballarat, Australia, July 18, 2016, OceanaGold Haile Gold Mine Cyanide Detox Test Work DTXSC021
- ALS Metallurgy Kamloops, BC, Canada, December 2016, Comminution Testing on Samples from the Haile Gold Mine KM 5180
- ALS Metallurgy Kamloops, BC, Canada, Comminution and Thickening Testing for Haile Gold Mine KM 5293

The metallurgical test results were used to develop process design criteria and the flow sheet for processing the ore in both the existing and for the upgraded plant.

The following sections contain some information in short tons (st) and others in metric tonnes (t).

13.1 Testing and Procedures

13.1.1 Comminution

Comminution test work on mineralized samples was performed by RDi (using Phillips Enterprises, LLC) and by ALS Kamloops.

Bond ball mill (BM) work indices were determined by RDi for various Haile samples. Bond impact and abrasion tests were also completed. The BM work index results for selected composites from this work are presented in Table 13-1.

Table 13-1: Bond Ball Mill Work Indices (Wi) for Haile Samples

Composite Number	Area	BM Wi @ 100-mesh (kWh/st)
1	Mill Zone	8.42
2	Mill Zone	8.07
3	Mill Zone	7.95
4	Mill Zone	8.03
5	Mill Zone	7.88
6	Haile	8.55
7	Haile	9.78
8	Ledbetter	7.49
26	Snake	10.34
27	Snake	10.39
31	Snake	5.13

Source: OceanaGold, 2022

Further testing, including Bond rod mill (RM) index testing as completed on Mill Zone, Haile, Ledbetter and Red Hill ore zone samples. The Bond ball mill work index for each composite was also determined at both 100- and 200-mesh for these samples.

The results for selected composites from this work are presented in Table 13-2.

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

Composite Number	Sample Description	RM Wi (kWh/st)	BM Wi @ 100-mesh (kWh/st)	BM Wi @ 200-mesh (kWh/st)
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

Source: OceanaGold, 2022

RDi also performed comminution tests on samples from the Ledbetter Extension zone. The Bond rod and ball mill indices and an abrasion index for an ore composite (83) was determined. The results of this work are presented in Table 13-3.

Table 13-3: Rod and Ball Mill Work Indices for Ledbetter Extension Samples

Abrasion Index	Value (kWh/st)
Rod Mill Work Index	12.71
Ball Mill Work Index at 100-mesh	10.21
Ball Mill Work Index at 200-mesh	9.81

Source: OceanaGold, 2022

RDi also performed comminution studies on samples from Horseshoe. The Bond rod, ball mill and abrasion indices for four different composites were determined. The samples were relatively abrasive and moderately hard. The results are presented in Table 13-4.

Table 13-4: Rod and Ball Mill Work and Abrasion Indices for Horseshoe Samples

Composite Number	Sample Description (Hole ID / intercepts / lithology)	RM Wi (kWh/st)	BM Wi @ 200-mesh (kWh/st)	Abrasion Index
83	RCT-03 / 1412 to 1460 ft / Silicified Metasediment	-	12.29	0.2167
84	RCT-04 / 1460 to 1510 ft / Silicified Metasediment	-	11.29	0.2691
85	RCT-04 / 1510 to 1585 ft / Silicified Breccia	14.93	12.95	0.3786
86	RCT-04 / 1585 to 1655 ft / Silicified Breccia	13.56	13.77	0.8330

Source: OceanaGold, 2022

ALS performed comminution tests on samples from Horseshoe and Ledbetter. The SMC and Bond ball mill indices for composites were determined. The results of this work are presented in Table 13-5.

Table 13-5: ALS Comminution Tests on Horseshoe Samples

Composite Number	A x b	SCSE (kWh/st)	BM Wi @ 200-mesh (kWh/st)
Horseshoe 1	28.9	11.4	13.5
Horseshoe 2	29.9	11.3	13.6
Horseshoe 3	30.7	11.2	9.3
Horseshoe 4	29.4	11.6	10.9
Horseshoe 5	28.0	11.6	14.4
Horseshoe 6	27.1	12.4	10.6
Ledbetter 1	27.3	12.0	11.6
Ledbetter 2	25.6	12.2	13.5
Ledbetter 3	27.8	11.9	11.8
Ledbetter 4	30.8	11.3	8.9

Source: OceanaGold, 2022

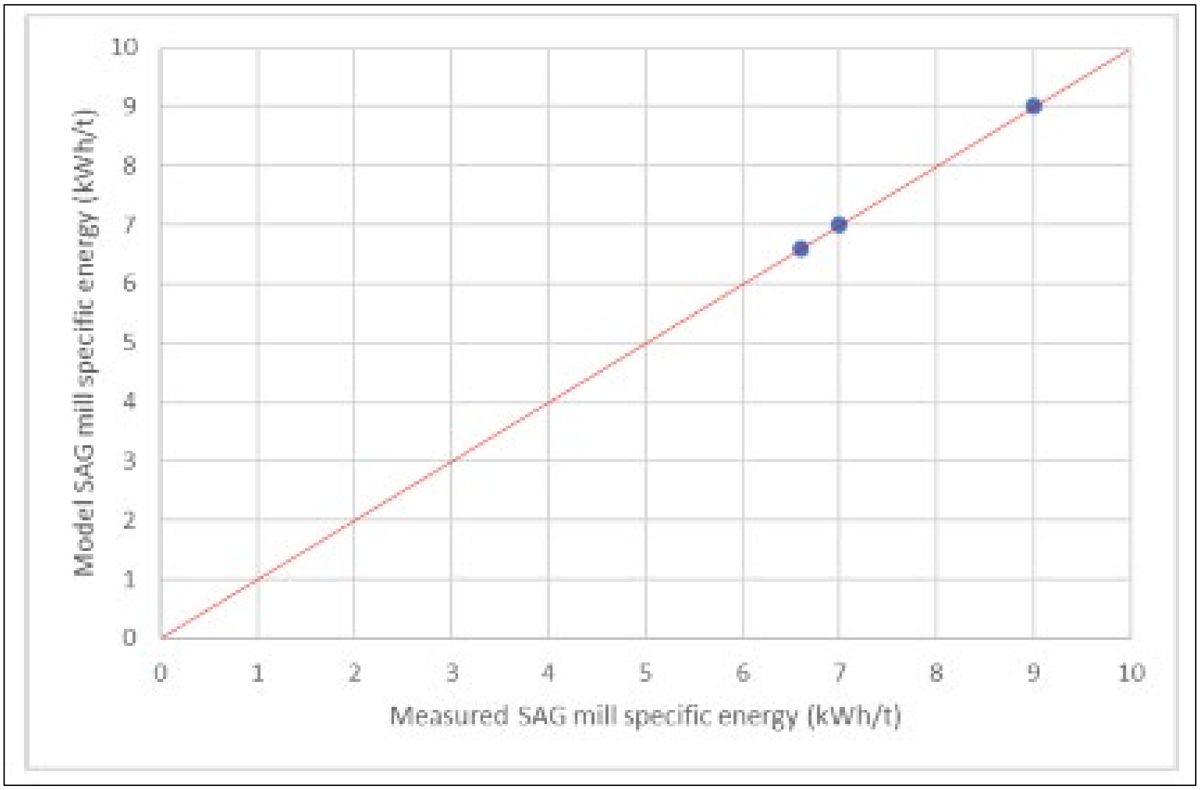
ALS performed JK Drop Weight and Bond ball mill index tests on samples from mineralized material exposed in Mill Zone Pit. The results of this work are presented in Table 13-6.

Table 13-6: ALS Comminution Tests on Mill Zone Pit Samples

Composite Number	A x b	SCSE (kWh/st)	BM Wi @ 200-mesh (kWh/st)
1a	93.6	7.15	9.4
1b			9.1
2a	52.8	8.85	6.8
2b			6.6

Source: OceanaGold, 2022

The comminution circuit design developed for the expansion project incorporated the additional competency test work and power modeling for the overall circuit was developed with the assistance of external consultants. A series of plant grinding circuit surveys were completed in 2017 and 2018 to validate the predictions of the modeling work. The survey data indicated the Haile SAG specific energy requirement was significantly lower than that predicted from the SMC test results on all surveys. Additional modeling work developed a Haile site specific model for SAG specific energy as a function of the drop weight index (DWI) from the SMC test and Bond ball mill work index. The results of the modeled against actual measured SAG specific energy are shown in Figure 13-1.



Source: OceanaGold, 2022

Figure 13-1 Modeled vs. Measured SAG Specific Energy Values

Updated throughput modeling, using the site-specific model, indicated an increase in throughput averaging 70 tph higher than the original work. Based on the outcome of the power modeling work, the confidence of achieving 3.5 to 4.0 Mtpa throughput rates for the majority of the ore sources was sufficient to proceed with the installation of the pebble crushing circuit but not to proceed with the detailed design of the secondary crushing circuit.

ALS performed SMC, Bond rod mill and Bond ball mill index tests on a further 17 composite samples taken from the Ledbettter, Snake, Haile and Red Hill pits from infill drilling in 2018. These allowed additional variability analysis on expected ore competency across these pits based on the power modeling work that represented mill feed from 2019 to 2024. The results of the program are summarized in Table 13-7 and indicate similar values to the previous programs.

Table 13-7: ALS Comminution test results for 2018 Infill Sample Program

Sample ID (DDH/Deposit)	A x b	SCSE (kWh/tonne)	BM Wi @ 200-Mesh (kWh/tonne)
672A Ledbetter	25.4	12.6	11.0
693A Snake	39.8	10.1	8.6
698A Snake West	27.9	11.8	11.3
726A Snake West	24.8	10.6	10.9
726B Snake West	28.9	11.7	8.9
746A Snake	40.5	10.0	9.8
746B Snake	31.0	11.3	11.6
746C Snake	36.2	10.5	10.0
752A Ledbetter	28.3	11.9	9.9
752B Ledbetter	33.0	10.9	10.2
773A Ledbetter	28.9	11.8	10.1
802A Haile	31.7	11.2	8.8
802C Haile	35.7	10.5	10.1
802C Haile	31.1	11.2	9.4
802D Haile	38.4	10.1	9.1
803A Red Hill	31.8	11.2	10.5
806A Red Hill	46.6	9.4	7.5

Source: OceanaGold, 2018

13.1.2 Throughput Estimate Assumptions

Following the treatment of deeper sourced ore from the Snake Stage 1 and 2 pits and associated mill performance on the expected very competent material a series of plant surveys were instigated to further refine the site specific energy model developed previously. Mill throughput on parcels of the lower portions of each stage, characterized from SMC testing as being in the most competent of the material expected in the deposit, was restricted by the SAG mill to 400-420tph (equivalent to 3.2 to 3.4 Mtpa rate). This has led to a broader geo-metallurgy program to improve confidence in expected ore competency and related throughput over the life of mine. This project is expected to carry on over the next 12-18 months with the assistance of external parties along with blast optimization trials and further debottlenecking options.

With the introduction of Horseshoe ore expected from 2024 and prior core testing results on deeper portions of the Ledbetter stage 3 and 4 pits, a variable throughput model based on source has been adopted for production planning incorporating the following rates:

- Undergorund Horseshoe ore a milling rate of 3.2 Mtpa.
- Open pit ore has a milling rate of 3.8 Mtpa

These assumptions are utilized in the scheduling of mill feed to the plant in the production forecasting process. The ongoing geomet program seeks to develop improved relationships that can be incorporated directly into the block model in the future.

13.1.3 Flotation and Cyanidation

The Philips test work described in the September 2008 report was performed on composite ore samples of average grade material from the Haile and Mill Zone pit areas.

The testing was conducted to substantiate metal recoveries from sulfide flotation and cyanide leaching of flotation tailings and investigate oxidation methods for enhancing gold extraction from sulfide concentrate. Additional work was executed on tailings samples to assess thickening and filtration response, neutralization requirements, and provide material for environmental and tailing disposal engineering studies by others.

The work confirmed the sulfides carry the majority of the metal values in Haile deposits and this allows their concentration into a smaller fraction for processing. Previous operations at the site recognized this and sulfide concentration was practiced. However, the sulfides do not easily release the metal values and limited extraction was experienced by simple cyanidation. The sulfides contained in the ore composites tested by Phillips showed the same characteristics.

Flotation tests on the Haile composite indicated 66% of the gold was separated into a concentrate that represented 6.7% of the flotation feed mass. Tests on the Mill Zone composite indicated 89% of the gold was separated into a flotation concentrate that represented 13.6% of the feed. The Mill Zone composite test had a finer flotation feed particle size distribution and extended residence time which may explain the difference in recovery.

Leach tests on flotation tail indicated 82% (stage) extraction for gold for both composites. Leach tests on a blend of Haile and Mill Zone flotation concentrate revealed gold extraction of only 67% of the gold with the as-floated particle size. Applying a test procedure entailing a regrind in cyanide solution to 80% passing 15μm, followed by an agitated cyanidation step, raised extraction to 80%.

The subsequent phase of work from 2009 was carried out by RDi on samples from the five areas within the Haile Gold mineralized zone; Mill Zone, Haile, Red Hill, Ledbetter and Snake. These discrete areas were provisionally derived from initial distinct open pits, but later design work and optimization may make the designation merely a legacy naming convention.

The methodology of compositing samples was to prepare composites from each hole's intervals based on their assays as follows:

- Less than 0.5 g/t Au was considered waste;
- Less than 1 g/t but greater than 0.5 g/t Au were combined as low-grade composites;
- Between 1 g/t Au and 4 g/t Au were combined as average grade; and
- Over 4 g/t Au were combined as high-grade.

Almost all the samples assayed over 0.3% sulfur and sulfide sulfur accounted for over 95% of the total sulfur.

RDi performed gravity concentration testing using a laboratory centrifugal concentrator with cleaner gravity concentration using a shaking Gemini table. The results indicate that the cleaner stage recovered about 20% of the feed gold but into a concentrate with a mass pull of 1% to 2% of the feed, assaying 11 to 75 g/t Au. The concentrate grade was too low grade to treat separately and there appears to be no coarse gold in the deposit, thus a gravity circuit was not considered to be applicable.

RDi performed whole-ore cyanide leach tests to examine the effect of ore grind size and leach time on gold recovery. The test work indicated that direct leaching gold extraction from the samples was generally poor and variable, ranging from 40% to 79%.

Most of the gold that leached was in the initial six hours of leach time and extraction generally increased with increasing fineness of grind. The refractoriness of the gold is partially due to size dependence but predominantly due to gold association with sulfides. A summary of the test work is presented in Table 13-8.

Table 13-8: RDi Whole-Ore Leach Test Results

Composite Number	Grind Size (P_{80}, mesh)	% Gold Extraction, Leach Time			NaCN Consumption at 48 hr. (lbs/st)
		6 hr.	24 hr.	48 hr.	
Mill Zone Average	100	57.0	65.0	64.7	0.50
Mill Zone Average	200	64.7	65.7	65.9	0.42
Mill Zone Average	325	68.0	69.2	68.4	0.84
Haile Average	200	67.5	71.3	71.5	0.52
Haile Average	325	69.0	73.7	75.3	0.96
Ledbetter Average	200	72.2	75.60	75.8	0.24
Ledbetter Average	325	70.4	80.3	79.1	1.40

Source: OceanaGold, 2022

RDi performed flotation test work to investigate the recovery of gold and silver to a sulfide mineral concentrate. The tests indicated that a reagent suite of potassium amyl xanthate (PAX), AERO 404 (or equivalent), and methyl isobutyl carbinol (MIBC) frother, 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 RDi flotation test work is presented in Table 13-9 and Table 13-10.

Table 13-9: Flotation Test Results – Averages by Grind

Sample Description	Primary Grind (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (oz/st)	
		% 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

Source: OceanaGold, 2022

Table 13-10: Flotation Test Results Average by Grade and Grind

Sample Description	Primary Grind (P_{80}, mesh)	Flotation Concentrate 6-minute Flotation Time Recovery %			Concentrate Grade (oz/st)	
		% 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

Source: OceanaGold, 2022

RDi performed flotation tailing cyanide leach tests to investigate the extraction of gold from the flotation tailing. The test results indicate that gold can be extracted from the flotation tails. A summary of the test work is presented in Table 13-11.

Table 13-11: Flotation Tailing Leach Test Results Average by Grade and Grind

Sample Description	Primary Grind (P_{80}, mesh)	Gold Extraction Leach Time – 24 hr. (%)	NaCN Consumption (lbs/st)	Lime Addition $Ca(OH)_2$ (lbs/st)
Mill Zone Average-Grade	200	52.9	0.14	3.08
Mill Zone Average-Grade	325	63.0	0.50	3.08
Mill Zone High-Grade	200	71.7	0.16	3.08
Mill Zone High-Grade	325	71.9	0.44	3.08
Red Hill Average-Grade	200	68.5	0.74	13.19
Red Hill Average-Grade	325	67.5	1.22	12.83
Red Hill High-Grade	200	74.1	2.56	15.76
Red Hill High-Grade	325	81.1	1.40	15.30
Ledbetter Average-Grade	200	68.6	0.44	6.35
Ledbetter Average-Grade	325	70.7	0.24	5.65
Ledbetter High-Grade	200	72.0	0.20	n.r.-
Ledbetter High-Grade	325	76.5	0.16	n.r.
Haile Average-Grade	200	62.7	0.16	13.68
Haile Average-Grade	325	62.2	0.26	13.70
Haile High-Grade	200	75.6	0.22	6.71
Haile High-Grade	325	77.1	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

Source: OceanaGold, 2022

Larger scale flotation test results achieved 91% gold recovery into a concentrate representing 8.8% weight of the flotation feed in 13.5 minutes of flotation time. Subsequent leach tests of flotation tail gave results that indicated 50% gold extraction in 16-hours of leaching.

Regrind test work on concentrate samples generated was performed by Metso Minerals Industries, Inc. (Metso) to predict specific energy requirements for concentrate regrind.

RDi performed flotation test work on 23 drill core composite samples from the Ledbetter Extension zone. The methodology for compositing samples by grade was the same as used earlier.

Gold recovery by flotation averaged 86% for the 100-mesh grind samples, averaged 87% for the 150-mesh grind samples, and ranged from 81% to 95% but averaged 89% for the 200-mesh grind samples.

The flotation tailing samples were leached for 24 hours at 40% solids and gold extractions averaged 66% for 100-mesh grind samples, from 52% to 85% and averaged 68% for 150-mesh grind samples, and from 44% to 87% and averaged 69% for 200-mesh grind samples.

In 2010, RDi performed additional metallurgical testing on duplicate ore samples from the earlier testing. Additional composite samples were made to evaluate carbon loading, cyanide destruction, flash flotation, conventional flotation time, and leaching of concentrate and tailing samples.

A procedure was developed and used to evaluate “flash flotation”. Flash flotation was shown to recover 62% to 66% of the gold in two 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 sample.

Fifteen samples were selected for the generation of flotation concentrate in one cubic foot flotation cell tests. The fifteen samples were low, average and high grade from different ore zones (Red Hill, Snake, Ledbetter, and Mill Zone).

Five samples were selected for the generation of flotation concentrate in small-scale laboratory flotation cell tests. The five samples were identified as average grade material from the different ore zones.

The flotation tests were followed by leaching tests conducted on the flotation concentrates and flotation tailings. The results of these tests are presented in Table 13-12.

Table 13-12: Test Results for Flotation and Leaching

Test No.	Zone	Grade	Comp. No.	Flotation			Conc. Leaching			Tails Leaching			Total Recovery Au (%)
				Head Grade Au (oz/st)		Au Recovery (%)	Head Grade Au (oz/st)		Au Extraction (%)	Head Grade Au (oz/st)		Au Extraction (%)	
				Assay	Calc		Assay	Calc		Assay	Calc		
1/2	RH	L	49	0.027	0.033	91.5	0.172	0.140	62.7	0.003	0.005	83.8	64.5
7/8	H	L	47	0.010	0.011	64.7	0.093	0.190	82.6	0.004	0.006	85.9	83.8
17/18	S	L	51	0.015	0.015	84.0	0.230	0.245	79.8	0.003	0.003	66.0	77.6
19/20	L	L	43	0.021	0.020	86.7	0.248	0.207	71.9	0.003	0.005	61.3	70.5
25/26	MZ	L	H290	0.024	0.035	95.4	0.152	0.190	77.4	0.002	0.004	72.5	77.2
15/16	RH	A	34	0.080	0.095	92.0	0.589	0.513	83.3	0.009	0.010	67.2	82.0
11/12	H	A	8	0.085	0.064	85.5	0.455	0.467	74.8	0.010	0.012	60.3	72.7
9/10	S	A	39	0.056	0.052	89.6	0.735	0.583	64.2	0.006	0.006	77.7	65.8
3/4	L	A	23	0.059	0.073	89.6	1.009	0.752	80.4	0.008	0.013	71.8	79.5
13/14	MZ	A	2	0.057	0.059	92.6	0.423	0.382	69.3	0.005	0.006	69.2	69.3
C34	RH	A	-	0.073	0.072	86.0	-	0.370	80.0	0.012	0.012	80.2	80.0
C28	H	A	-	0.086	0.085	68.1	-	0.580	59.7	0.030	0.029	79.6	66.0
C31	S	A	-	0.051	0.056	93.7	-	0.166	58.5	0.005	0.005	45.1	57.7
C61	L	A	-	0.048	0.047	86.1	-	0.341	80.7	0.007	0.008	81.4	80.4
C5	MZ	A	-	0.073	0.078	92.2	-	0.292	69.5	0.008	0.008	67.0	69.3
27	RH	H	35	-	0.429	94.1	2.601	2.094	73.6	0.030	0.038	77.5	73.8
28	H	H	9	0.180	0.194	90.5	1.394	1.321	88.5	0.021	0.024	64.5	86.2
5/6	S	H	53	0.304	0.312	95.2	2.365	1.875	75.2	0.017	0.020	68.3	74.9
23/24	L	H	71	0.240	0.274	94.7	2.622	2.222	74.0	0.015	0.034	81.5	74.4
21/22	MZ	H	12/3	0.168	0.199	96.0	1.563	1.155	79.7	0.009	0.020	73.3	79.4

Source: OceanaGold, 2022

The overall extraction, sorted by sampled zones, is presented in Table 13-13.

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

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

Source: OceanaGold, 2022

RDi performed additional leach tests on flotation concentrates to ascertain if better results could be obtained. The results of performing leach tests on larger concentrate samples (i.e., twice the size used in previous tests) demonstrated a significant improvement in gold and silver extraction. Concentrate samples were ground to a size distribution of 80% passing 15 to 18 microns. The slurry was then pre-aerated for four hours and lead nitrate was added for the final three hours of pre-aeration and then leached for 48 hours with carbon present. A summary of the larger scale leach test results is presented in Table 13-14.

Table 13-14: CIL Test Results for Fine Ground Flotation Concentrate

Test No.	Pit	Grade	Composite Number	Grind Size (P_{80}, µm)	48 hr. Leach Time % Extraction		NaCN Consumption (lbs/st)
					Au	Ag	
37	Red Hill	L	49	17	80.9	71.1	2.00
36	Haile	L	47	14	77.2	49.5	4.99
38	Snake	L	51	16	81.0	94.4	10.83
35	Ledbetter	L	43	16	88.3	91.9	5.09
21	Mill Zone	L	Hole 290	-	79.8	91.0	5.59
26	Mill Zone	L	Hole 290	-	85.0	82.3	4.75
33	Red Hill	A	34	16	85.8	77.2	4.60
31	Haile	A	28	18	95.6	97.4	4.36
22	Haile	A	8	-	81.6	93.2	3.62
32	Snake	A	31	18	58.8	18.2	4.26
24	Snake	A	39	-	84.7	96.4	5.25
40	Ledbetter Ext	A	61	14	89.8	98.3	1.96
27	Mill Zone	A	2	16	81.5	96.2	4.77
28	Mill Zone	A	5	17	79.2	50.0	4.72
41	Ledbetter Ext	A	73	16	83.7	93.4	3.30
23	Ledbetter	A	23	-	88.3	79.9	4.72
34	Red Hill	H	35	16	92.6	95.9	3.66
29	Haile	H	9	20	93.7	97.7	3.46
39	Snake	H	53	16	83.4	97.4	5.03
30	Mill Zone	H	12/3	19	88.7	95.9	4.00
25	Ledbetter Ext	H	71	-	94.9	95.6	12.3

Source: OceanaGold, 2022

KML was commissioned to perform additional flotation and leach tests on 29 composites from Mill Zone and Snake areas. The samples selected were chosen to represent the initial three-years of the operation's mine schedule anticipated at the time.

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% and overall silver recoveries ranged from 32.9% to 81.9%. A summary of the results is presented in Table 13-15.

Table 13-15: Tests Results for Composites from Mill Zone and Snake areas

Composite	Au Head Grade (oz/st)	Au Recovery (%)	Ag Head Grade (oz/st)	Ag Recovery (%)
1	0.224	90.6	0.07	75.8
2	0.028	74.7	0.05	74.2
3	0.127	89.8	0.06	73.1
4	0.101	82.6	0.05	73.8
5	0.128	88.6	0.06	81.4
6	0.037	71.6	0.04	68.4
7	0.320	90.7	0.08	77.7
8	0.071	83.7	0.06	77.5
9	0.077	87.9	0.13	81.1
10	0.142	88.7	0.17	81.9
12	0.038	77.5	0.05	64.5
13	0.064	81.3	0.06	78.5
14	0.047	84.0	0.11	80.6
15	0.079	85.1	0.11	75.0
16	0.114	82.6	0.15	75.3
17	0.056	82.9	0.06	78.7
18	0.054	76.0	0.09	75.5
19	0.061	77.7	0.03	70.7
20	0.036	75.8	0.04	76.7
21	0.065	76.6	0.08	71.8
22	0.148	87.5	0.12	71.6
23	0.245	91.0	0.10	74.4
24	0.055	86.5	0.03	52.8
25	0.029	87.6	0.02	43.6
26	0.013	76.3	0.02	62.3
27	0.010	80.8	0.01	58.0
28	0.016	76.8	0.02	35.5
29	0.027	80.9	0.01	32.9
30	0.107	88.8	0.04	46.0

Source: OceanaGold, 2022

RDi undertook additional test work on the composite samples from the Horseshoe Zone to determine the response to the process flowsheet selected. Visible gold was reported in some core intercepts used for Horseshoe test work. Test work included comminution (described above) and flotation and leaching of concentrate and flotation tailing.

The flotation process utilizing a simple reagent suite (PAX, AP404 and MIBC) developed for the deposit in earlier studies recovered 85% to 90% of the gold into the concentrate for most of the composites. Cyanide leaching consistently extracted about 70% of the gold in the flotation tailings. The composite concentrate samples were reground and subjected to a preparation step and a carbon in leach (CIL) test, which showed gold and silver extractions of over 90% for most composites. Lower recoveries were achieved for the low-grade composite 83.

A summary of the test results is presented in Table 13-16.

Table 13-16: Test Results for Horseshoe Samples

Composite Number	Assay Head Au (g/t)	Primary Grind Size (P_{80}, mesh)	Flotation Recovery (%)		Tailing Leach (%) Extraction		Concentrate Leach Extraction (%)		Overall Extraction (%)	
			Au	Ag	Au	Ag	Au	Ag	Au	Ag
83	1.8	100	86.5	63.0	70.2	3.7	69.3	64.9	69.4	42.3
83		150	89.0	57.1	70.4	9.6			69.4	41.2
83		200	87.6	54.1	73.9	5.5			69.9	37.6
84	9.1	100	88.4	74.4	69.4	5.3	96.2	94.6	93.1	71.7
84		150	90.2	77.3	71.3	22.0			93.8	78.1
84		200	90.1	77.7	71.0	7.1			93.7	75.1
85	10.4	100	83.3	70.7	66.5	39.4	96.0	91.9	91.1	76.5
85		150	89.2	81.4	71.7	19.1			93.4	78.4
85		200	87.4	80.7	76.3	37.5			93.5	81.4
86	12.1	100	86.4	74.5	67.2	8.9	94.9	92.1	91.1	70.9
86		150	85.8	73.8	72.5	45.8			91.7	80.0
86		200	90.4	75.2	76.6	40.7			93.1	79.4
87	10.6	100	69.7	57.7	59.5	53.4	95.2	95.0	84.4	77.4
87		150	77.8	64.8	66.1	54.1			88.7	80.6
87		200	75.9	69.0	70.8	54.1			89.3	82.3

Source: OceanaGold, 2022

In late 2011, G&T Metallurgical Services was selected to perform the metallurgical test program on additional Horseshoe samples. The metallurgical test program involved testing of twelve variability samples to evaluate recoveries by flotation and cyanidation of concentrate and flotation tails.

The Horseshoe samples responded very well to the Haile flowsheet. A summary of the test results compiled by HGM personnel is provided in Table 13-17.

Table 13-17: Test Results for Horseshoe Samples

Composite Number	Gold Head Grade (oz/st)	Flotation P_{80} (µm)	Kinetic Flotation Recovery (%)	Bulk Flotation Recovery (%)	Regrind P_{80} (µm)	Concentrate Leach Extraction (%)	Tailings Leach Extraction (%)	Overall Gold Recovery (%)
1	0.199	81	93.9	86.5	15	90.4	81.1	89.2
2	0.339	70	92.5	90.7	13	99.5	67.3	96.6
3	0.043	78	96.5	94.0	14	86.1	87.6	89.8
4	0.060	73	93.3	97.0	16	98.5	74.1	86.1
5	0.076	81	90.2	83.4	10	94.9	85.5	94.4
6	0.168	84	88.4	86.0	11	93.0	90.2	94.9
7	0.082	82	85.2	88.0	15	94.9	75.6	93.8
8	0.094	66	89.0	84.4	12	98.6	83.3	92.6
9	0.057	69	86.2	89.2	11	97.1	81.7	96.0
10	0.121	75	75.1	77.6	15	90.3	82.1	90.6
11	0.349	88	86.2	87.5	14	97.4	88.4	95.8
12	0.129	77	85.3	83.6	10	97.4	84.2	96.1

Source: OceanaGold, 2022

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

The samples tested responded favorably at a moderately fine feed size range of 80% passing 200-mesh (74 microns). Therefore, a primary grind size of 80% passing 200-mesh was recommended for process circuit design development. Operational experience and minerology may allow this criterion to be relaxed reducing comminution requirements and increasing plant capacity.

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

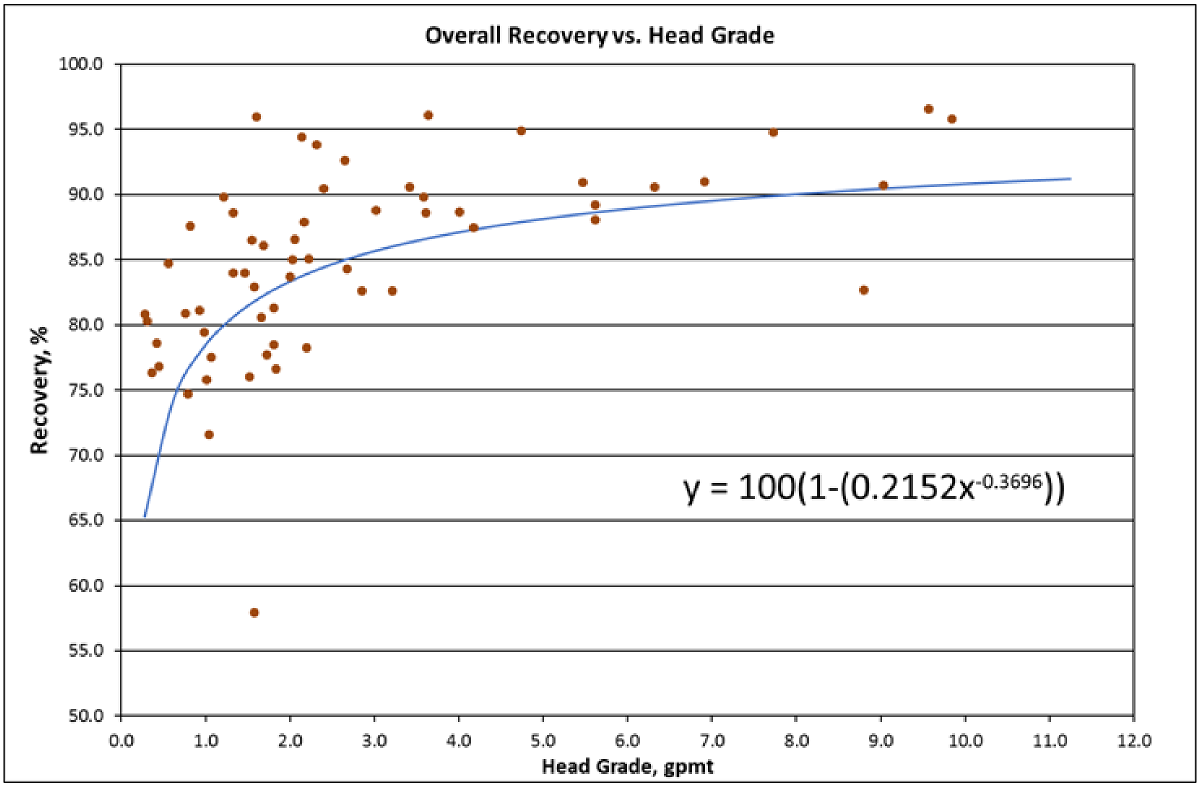
The test work indicated that the circuit should include regrinding of the flotation concentrate before slurry pre-aeration, and a leach time of 24 hours. A regrind circuit product size of 80% passing 15 microns is an appropriate target for regrind circuit design. Operational experience may allow this criterion to be relaxed reducing comminution requirements and increasing plant capacity.

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 from which zone the ore is mined.

The unit operations that determine the amount of gold extraction are flotation, flotation concentrate regrind and leaching, and flotation tailing leaching.

The data developed in the test programs has been used to establish a relationship between overall gold recovery and head grade as shown in the graph in Figure 13-2. The algorithm for the “best-fit” line that describes the head grade to recovery relationship can be used to estimate gold recovery from a predicted mill head grade for example, at a mill head grade of 2.3 g/t, the recovery equation graph predicts a gold recovery of 84.2%.

The results of grade and recovery data analysis are shown in Figure 13-2.



Source: OceanaGold, 2022

Figure 13-2: Overall Percent Recovery vs. Head Grade

Overall, the results from Horseshoe tests were in-line with or exceeded the recovery model. The cyanide destruction test results indicated that the sulfur dioxide (SO_2)/air cyanide destruction process destroys weak acid dissociable (WAD) cyanide very effectively, as well as free cyanide. Operations to date have achieved target levels of cyanide removal from the TSF and reagent usage is being optimized to further reduce costs.

13.1.4 Sample Representativeness

Samples were collected from a range of locations across the main area of the resources that is planned to be fed to the processing plant over the LoM. The minimal variability of test work indicates that the different mineralized zones are relatively similar in terms of ore grindability, chemical and mineral compositions, and flotation and cyanide leaching response. Given the uniform geological setting and mineralization, this is not surprising.

It is evident from the comminution testing that ore competency is increased in the deeper resources (e.g., Horseshoe underground). This is typified by the trend of increased Bond Ball Milling Work indices for samples sourced from lower levels.

13.2 Recovery Estimate Assumptions

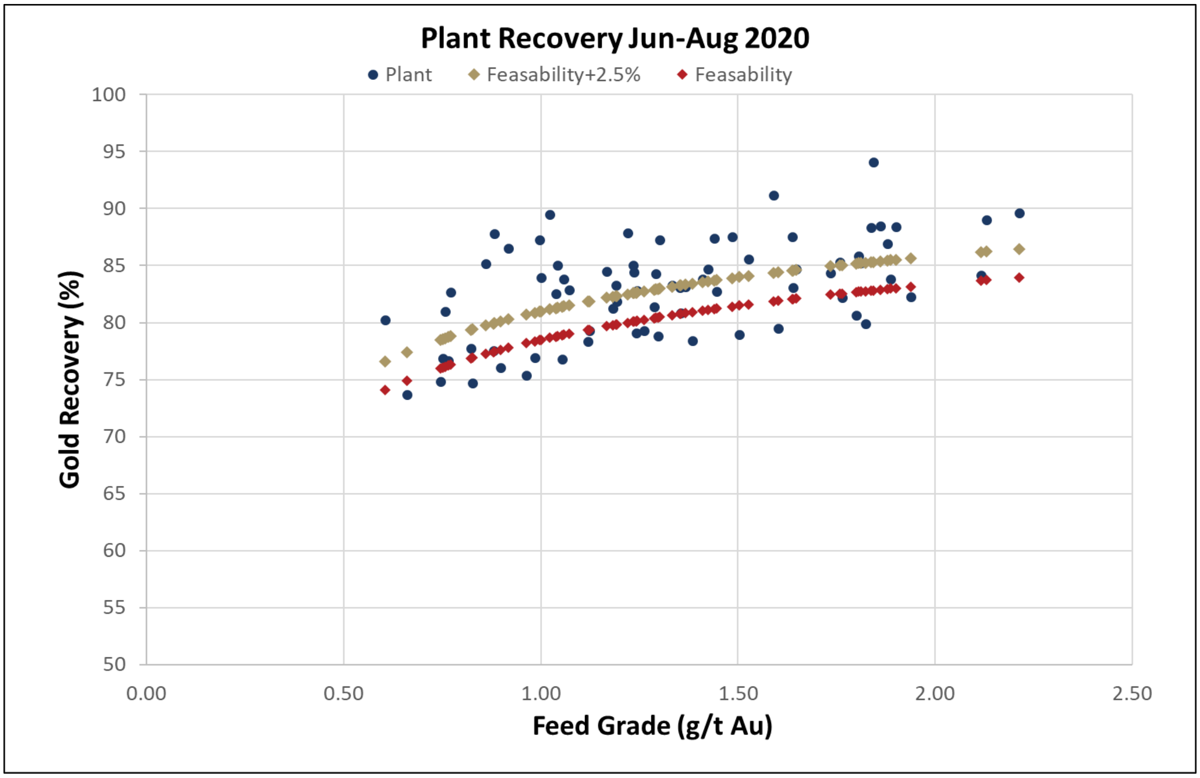
The recovery estimate previously was based on the interpretation of test work results from a range of samples and test work campaigns and shown in Figure 13-2 as a function of gold grade. This was based on the original flowsheet incorporating sulfide flotation and open circuit regrind of concentrate in the SMD circuit to a target P80 of 13 um.

Operating experience since plant commissioning had been below target due to the inability of the original regrind circuit to achieve the planned 13 um grind size and, with the staged increase in mill throughput, the concentrate feed rate to the regrind section continued to increase above the original design further exacerbating the issue. The completion of the replacement of the regrind circuit in 2019 with a two stage Towermill/Isamill circuit operating in closed circuit has achieved the targeted regrinding product size. In addition, a number of plant modifications have been implemented to address mechanical issues in the CIL circuit leading to short circuiting and issues with carbon management related to interstage screen cleaning schedules.

With the plant ramping up tonnage ahead of plan in 2019, mill feed has been supplemented with oxide material from stockpiles in the feed blend. While the flotation concentrates and tailings are leached allowing the oxide material to be blended, there has been a detrimental impact on flotation performance impacting the recovery of sulfide to the regrind circuit and subsequent gold loss to leach tailings. Since June 2020, the mill feed strategy has been altered to campaign the oxide feed on its own direct to CIL to minimize the impact on sulfide processing.

Additional flowsheet development has been ongoing since 2019 with changes to process control, CIL circuit operation and concentrate pretreatment. The overall effect has been to achieve gold recoveries in excess of that predicted by the original feasibility study model when feeding the plant predominantly fresh sulfide ore. The performance of the plant from June to August 2020 is shown in Figure 13-3 with daily reconciled data and the recovery predicted by the feasibility relationship resulting in a 2.5% increase on this model used during the 2020 planning process.

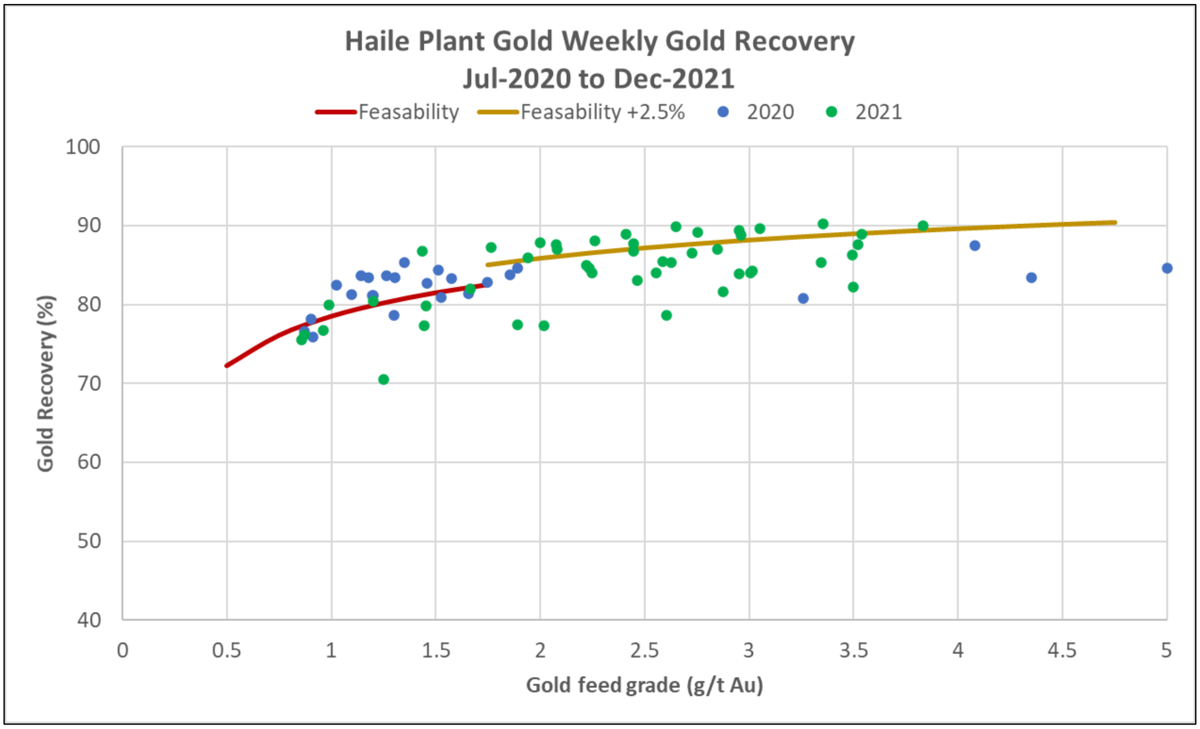
Based on the metallurgical development projects completed and planned in the process plant and performance on fresh sulfide feed, the recovery model used post-2022 has adopted the 2.5% increase above the original feasibility relationship for fresh ore fed to the mill in the previous technical report.



Source: OceanaGold, 2020

Figure 13-3: Plant Gold Recovery Performance Jun-Aug 2020

Extending the data set to incorporate production from June 2020 to the end of 2021 the ability of the plant to achieve the feasibility plus 2.5% recovery was demonstrated consistently above 1.7g/t gold feed grade. For head grades below the 1.7 g/t level the data supports the use of the original recovery regression without the aditional uplift. Figure 13-4 shows the weekly recovery data distribution over the last 18 months and the two recovery relationships. This data incorporates varying feed blends of both fresh ore and oxide material and has not been filtered on feed blend.



Source: OceanaGold, 2022

Figure 13-4: Weekly Plant Gold Recovery Performance Jun 2020-Dec 2021

The operating strategy for ROM management is based on segregating significant quantities of oxidized ore and campaign milling when possible to minimize the impact on the flotation circuit performance on fresh ore. This is used to create maintenance windows for the regrind circuit without mill downtime.

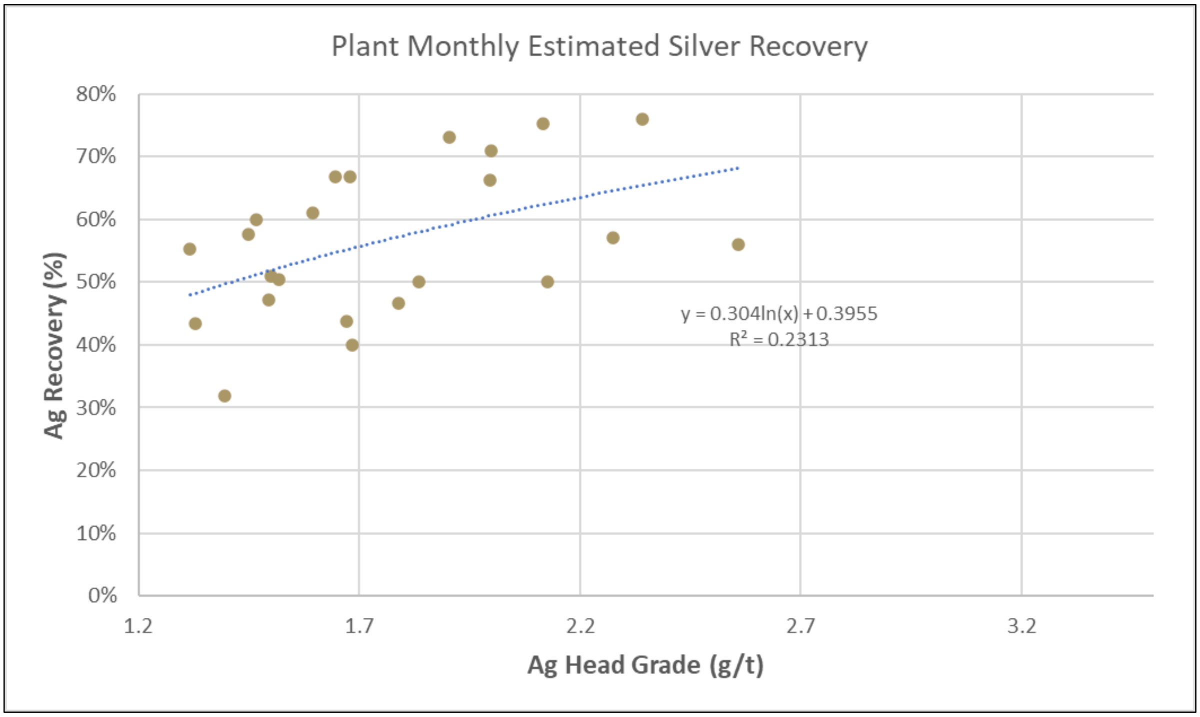
Given the mill feed schedule over the life of mine tracks the major sources the following criteria have been applied to estimate overall gold recovery for production forecasting:

- Fresh sulfide ore delivered to the ROM above 1.7 g/t Au from open pit and Horeshoe UG utilse the feasibility recovery plus 2.5%
- Fresh sulfide ore delivered to the ROM below 1.7 g/t Au from open pit and Horseshoe UG utilize the feasibility recovery
- Oxide ore assumes a flatline 68% gold recovery based on direct leach flowsheet
- Sulfide ore that is stockpiled and then rehandled to the ROM will attact a 5% recovery penalty compared to the fresh mined recovery assumption

This creates a slight reduction in overall recovery compared to previous methods recognizing the influence of periods of mill feed at lower grades below 1.7 g/t in the mine plan.

Silver recovery was estimated based on work completed at the KML laboratory on an additional 30 composites in 2012, representing the first three years of the mine plan looking at the performance of silver recovery. This program looked at composites over a wide range of silver to gold ratios from 0.25:1 to 2.23:1 and generated a recovery model based on silver recovery to the flotation concentrate and subsequent leach extraction from the reground concentrate and flotation tailings.

Monthly reconciled silver recovery data has been collected since the beginning of 2020 to allow for an improved assessment of expected silver recovery for forecasting. For head grades above 1.8 g/t Ag, the monthly achieved recovery ranged from 50% to 76% averaging 66%. The LoM schedule indicates silver feed grades to the plant of 1.14 to 3.52 g/t and averaging 2.22 and based on these expected grades a flat line assumption of 70% silver recovery has been used in metal production forecasting. Actual achieved silver recovery to bullion is shown in Figure 13-5.



Source: OceanaGold, 2022

Figure 13-5: Plant Reconciled Silver Recovery from 2020-21

Silver revenue accounts for approximately 1.4% of overall total sales and as such the the impact of changes to the silver recovery have a marginal impact on overall economic outcomes. The product from Haile is gold containing moderate amounts of silver. The doré bars are 95% pure with minimal or no deleterious elements.

14 Mineral Resource Estimate

This section describes the Mineral Resource estimation methodology and summarizes the key assumptions adopted by OceanaGold. In the opinion of OceanaGold, the Mineral Resource estimates reported herein are a reasonable representation of the Mineral Resources at Haile. The Mineral Resources and a classification of resources were prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014). Mineral Resources are reported in accordance with NI 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

The Mineral Resource estimates were prepared under the supervision of Mr. Jonathan Moore, BSc (Hons) Geology, DipGrad Physics, member and Chartered Professional of the Australasian Institute of Mining and Metallurgy. Mr Moore is OceanaGold Chief Geologist, a Qualified Person as this term is defined pursuant to NI 43-101 for Mineral Resources. The effective date of the resource statement is December 31, 2021.

Separate block models were generated for the open pit and underground areas given the differing mining selectivity and cut-off grade assumptions. The open pit resource model is “HA1220OLM”, built under the supervision of Mr. Moore. The geologic model was provided by the OceanaGold Haile Exploration department. Implicit gold grade shell generation, grade estimation and block model construction were completed within Vulcan software. Resource pit shells were generated using Whittle software.

The Horseshoe underground model is “HA0420UG_R” and was built under the supervision of Mr. Moore. The implicit gold grade shell was generated in Leapfrog software whilst grade estimation and block model construction were completed in Vulcan software.

The Palomino underground model is “PA0122U” and was built under the supervision of Mr. Moore. The implicit gold grade shell was generated in Leapfrog software whilst grade estimation and block model construction were completed in Vulcan software.

14.1 Open Pit Mineral Resources Estimate

Grade estimation was completed with Vulcan software, using Multiple Indicator Kriging (MIK) based on 2.5 m bench composites to produce E-Type gold estimates into a 10 m E x 10 m N x 5 m Rl model blocks. Gold estimation was constrained within implicitly modeled grade shells, approximating a 0.065 g/t gold indicator. Metasediment/metavolcanic contacts were not used as hard boundary constraints for gold estimation. Modeled post-mineralization dikes were assigned zero grade.

MIK is well suited to estimating positively skewed grade distributions. Top caps of 50 g/t Au were used to temper mean grades above the top-class indicator threshold. Acceptable long-term model to mill-adjusted mine reconciliation suggests that this approach is reasonable. Ordinary kriging (OK) was used for silver, sulfur and carbon estimates. The general workflow for model generation was as follows:

- Extraction of drillhole data from acQuire database
- Data validation, exclusion of drillholes from early drilling campaigns of unknown quality
- Composite drillholes to 2.5 m for Au, total carbon, total sulfur and silver

- Update grade shell in Vulcan using Implicit modeling tool
- Flag drillholes for lithology, domain area and grade shell
- Code model with lithological wireframes, domain area and grade shell
- Run MIK estimation for Au
- Set block grades to zero for blocks coded as post-mineralization dike
- Run OK estimation for Ag (with simulated values), total carbon % and total sulfur %
- Deplete resource with historical open pit and underground workings
- Assign model densities
- Classify model

14.1.1 Drillhole Database

During 2016, the Romarco Minerals drilling database was translated to OceanaGold's standard AcQuire database platform. Where available, original source assay and survey data were used for the AcQuire translation and database validation. There was a further internal database review in late 2018/early 2019. No material errors were identified.

Historical drilling (i.e., prior to 2007) accounts for approximately 18% of the data. The sample procedures applied to the historical drilling (i.e., drilling prior to Romarco Minerals Inc.) at Haile were not well documented. Having said this, approximately five years of mining at Haile has tested the veracity of the resource estimates which are based on this data. No material flaws have been identified.

Drillhole data available in October 2020, was included in the HA1220OLM estimate (Table 14-1). Approximately 1,700 historical drillholes were excluded based on unknown quality.

The assay coverage for gold covers all core and RC drilling. However, the collection of carbon, sulfur and silver assay data has largely been retrospective and is significantly sparser than for gold. Sulfur and carbon data are primarily used for the prediction of waste classification types. Sulfur grades are also used for mill feed sulfur estimates. Silver grades are provided for metallurgical considerations (carbon stripping and electro-winning) as well as for revenue estimation, albeit silver is a minor contribution relative to total revenue

Table 14-1: Sample Numbers for Gold, Sulfur and Carbon

	Au	S	C	Ag
Count	384,109	70,876	33,156	20,294

Source: OceanaGold, 2022

14.1.2 Geologic Model Concepts

A detailed 3D lithological model, including weathering, has been constructed. This model, which has evolved over time, has been used to assign variable densities to the block model (Table 14-5). Faults have also been modeled. However, other than post-mineralization dikes, and post-mineralization erosion/deposition, there are few geological features that define mineralization boundaries at the economic cut-off grade.

14.1.3 Lithology

Lithologic codes used at Haile capture many geologic attributes including the primary rock type, presence of brecciation, silicification, lamination and numerous variations on the general rock unit.

The majority of mineralization is hosted within the metasediments and the lithological units are as follows:

- S - sand
- Sap - saprolite
- MV - metavolcanics
- DB - intrusive dikes
- Fill - back-fill from historical mining
- Ml - laminated metasediments
- Ms - silicified metasediments
- Breccia - brecciated rocks

14.1.4 Silicification

The progression of “silicification” increases from 0 (non-existent) to 3 (main component) and is logged visually by site geologists. The minor silicification (1) population has an average grade of about 0.5 g/t. The average grade of moderately silicified (2) rocks is 1.0 g/t and the very silicified (3) average grade increases to approximately 2.0 g/t. There is a clear gold to silicification relationship. However, the subjectivity of the logging code is probablematic. Previous attempts have been made to combine logged silicification intensity with gold grade to generate a mineralization indicator for domaining gold estimation. The outcomes have been less successful than using a gold-only indicator. There are also some areas of gold mineralization not associated with intense silicification.

14.1.5 Pyrite

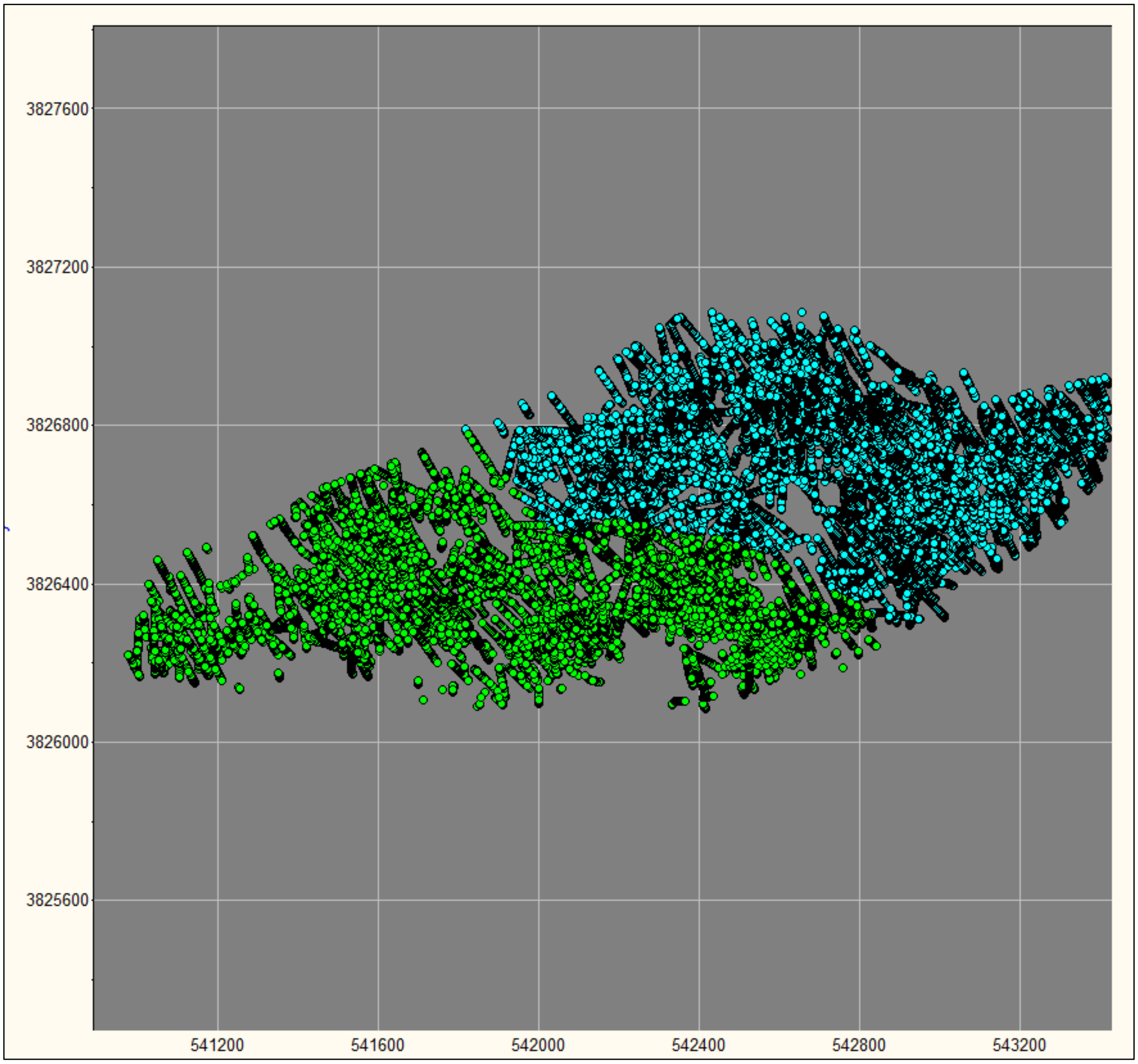
Multiple morphologies of pyrite have been identified at Haile, ranging from fine to coarse cubic pyrite. Based on logging, it has been established that the fine-grained pyrite is commonly associated with mineralization.

14.1.6 Grade Domain Construction

Although both silicification and pyrite occurrence are qualitatively associated with gold mineralization, their relationships are not used for quantitative gold domain definition. Implicit modeling in Vulcan software was used to create a grade shell at a 0.065 g/t gold threshold. The grade threshold selection was optimized, using sensitivity estimate comparisons against production data.

The grade shell was then sub-divided into two domains based upon gold distribution and orientation, albeit the differences were not large between the two domains.

The mineralized zones within domain 1 (blue in Figure 14-1), approximate a dip direction of -40 to 330, while in domain 2 (green), they approximate a dip direction of -30 to 335.



Source: OceanaGold, 2022

Figure 14-1: Estimation Domains

14.1.7 Compositing

2.5 m bench composites for carbon, gold and sulfur were calculated for estimation. Due to the paucity and uneven distribution of data, carbon and sulfur were estimated directly from the sample grades which were collected intermittently down the hole, typically only one sample per 6.1 m of drilling.

14.1.8 Assay Cap Values

Multiple Indicator Kriging (non-linear estimation) has been used for gold estimation which is better suited to positively skewed grade distributions than linear estimation methods. 2.5 m composited gold grades were top capped to 50 g/t Au based on reconciliation to production data. This lowered the mean grade of the top indicator class.

14.1.9 Multiple Indicator Gold Class Thresholds and Means

Gold Indicator thresholds were set at cumulative frequencies of 10^{th}, 20^{th}, 30^{th}, 40^{th}, 50^{th}, 60^{th}, 70^{th}, 80^{th}, 85^{th}, 90^{th}, 95^{th}, 97.5^{th} and 99^{th}. Table 14-2 summarizes the indicator threshold grades and class means used for gold estimation.

Table 14-2: Indicator Gold Class Thresholds and Means

Cumulative Frequency	Domain 1		Domain 2	
	Threshold	Mean	Threshold	Mean
10	0.05	0.03	0.03	0.02
20	0.07	0.06	0.06	0.05
30	0.10	0.09	0.09	0.07
40	0.13	0.12	0.13	0.11
50	0.18	0.16	0.19	0.16
60	0.25	0.21	0.28	0.23
70	0.37	0.30	0.43	0.35
75	0.46	0.41	0.55	0.48
80	0.59	0.52	0.71	0.62
85	0.79	0.68	0.97	0.82
90	1.17	0.96	1.48	1.18
95	2.07	1.55	2.71	1.89
97.5	3.05	2.51	4.16	3.32
99	6.00	4.14	9.20	5.64
Max	50.00	11.34	50.00	12.90

Source: OceanaGold, 2022

14.1.10 Variogram Analysis and Modeling

Variograms were estimated for gold for each indicator threshold. Variograms for carbon and sulfur were estimated globally.

14.1.11 Block Model

The HA1220OLM resource block model was constructed in Vulcan and the parameters below in Table 14-3 are based on a Parent block size of 10 m x 10 m x 5 m in x, y, z respectively and is not sub-blocked or rotated.

Table 14-3: HA1220OLM Block Model Dimensions

Variable	East	North	RL
Minimum	539,810	3,825,575	200
Maximum	544,510	3,827,725	1200
Block Size (Parent)	10	10	5
No. of Blocks (Parent)	470	215	200

Source: OceanaGold, 2022

14.1.12 Estimation Methodology Gold, Sulfur and Carbon

Gold estimation was completed using Multiple Indicator Kriging, while carbon and sulfur were estimated using ordinary kriging. Carbon and sulfur values are used for classification of waste material.

For gold estimation, two domains were used:

- Domain 1 (Mill Zone, Haile/Red Hill, Champion)
- Domain 2 (Snake, Ledbetter)

Each domain area was estimated in three passes, with each subsequent search ellipse larger than the previous. Each of the main two open pit domain areas has unique search parameters based upon indicator variogram models for 14 different Au cut-offs. Table 14-4 shows the block search parameters.

Table 14-4: Block Search Parameters

Domain	Area	Search Orientation			Search Radius				Sample Thresholds		
		Bearing	Plunge	Dip	Pass	Major	Semi	Minor	Min	Max	Max/Hole
1	Mill Zone, Champion, Red Hill/Haile	335	-30	0	1	30	30	10	4	16	3
1	Mill Zone, Champion, Red Hill/Haile	335	-30	0	2	60	60	15	4	16	3
1	Mill Zone, Champion, Red Hill/Haile	335	-30	0	3	90	90	20	1	12	3
2	Snake, Ledbetter	330	-40	0	1	30	30	10	4	16	3
2	Snake, Ledbetter	330	-40	0	2	60	60	15	4	16	3
2	Snake, Ledbetter	330	-40	0	3	90	90	20	1	12	3

Source: OceanaGold, 2022

14.1.13 Estimation Methodology Silver

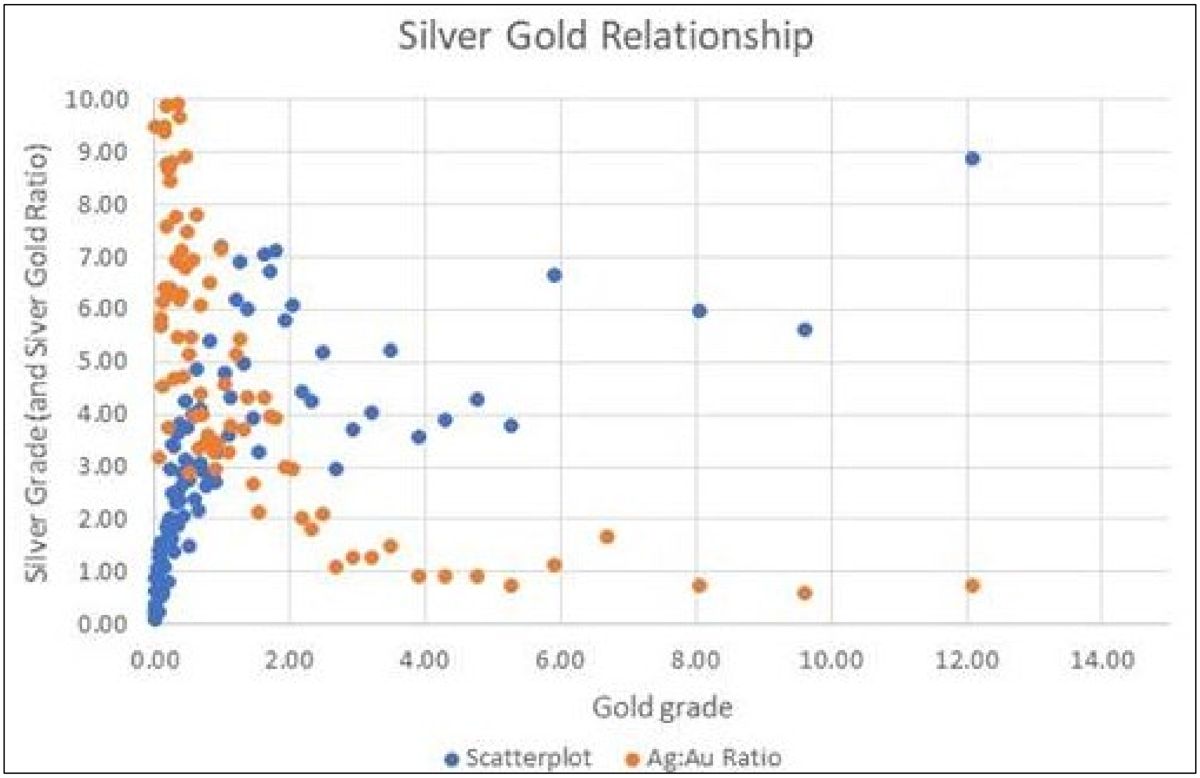
Silver estimates have not been completed for the underground. For the open pit, silver grade estimates are provided for metallurgical considerations (carbon stripping and electro-winning) as well as for revenue estimation, albeit silver contributes only about 1.5% of total revenue and so is not particularly material. Silver content is not used as a gold-equivalent input for cut-off calculation nor to guide mine design decisions.

The sample support basis for the open pit silver estimates is approximately 10% of that for gold (54,100 x 3 m composites for gold versus 5,551 x 3 m composites for silver). While the paucity of data reduces the local accuracy of silver estimates, it does not preclude providing silver estimates for revenue modeling given that silver is mined as a by-product and not used for ore delineation.

The selection of samples for silver assaying was undertaken retrospectively, based upon previously assayed gold grades. Sample selection for silver assaying tended to favor more strongly gold-mineralized intervals, leaving less intensely mineralized intervals, on the flanks of the mineralization under-represented. In order to mitigate the impacts of the selection bias, bivariate simulation was implemented (using the “simulate missing data” program in GS3 proprietary software). This non-spatial simulation assigned silver grades to locations with gold assays, but no silver assays, based upon relationships between silver and gold in the assay database. Figure 14-2 shows gold-ranked silver and gold percentile means to highlight the underlying silver gold relationship. The silver-gold relationship changes with gold grade; lower gold grades show a significantly higher silver-gold ratio. Overall, the assayed population showed a rank Spearman correlation coefficient of 0.47, reflecting a moderate correlation between silver and gold. This relationship was captured in the simulation process.

From the original 5,551 assayed samples with a mean silver grade of 2.36 g/t, a combined population of 54,100 assayed and simulated values (i.e., simulated silver values based on gold assay grades) with a mean silver grade of 1.84 g/t resulted. The result confirmed that a selection bias was present and provided justification for the downward silver grade adjustment via simulation.

A 95th percentile top cap value of 9.9 g/t Ag was applied to both domains, the 95th percentile was selected to err on the side of conservatism.



Source: OceanaGold, 2022

Figure 14-2: Silver Gold Relationship for Gold Ranked Percentile Averaged Grades

The domaining and search orientations as used for gold, were used for silver. Given the low number of original silver assays, ordinary kriging (rather than MIK) was used. The silver estimation methodology described above is considered to be appropriate for the purposes of silver byproduct estimation.

14.1.14 Bulk Density

In situ dry bulk density (BD) determinations have been carried out at regular intervals on drill core. The immersion/displacement method involved weighing the sample both in air and in water. Average measurements were used for each lithology. BD values were assigned to model blocks based on geological coding rather than estimated as a continuous variable (Table 14-5). Over 44,000 density measurements have been collected from core samples.

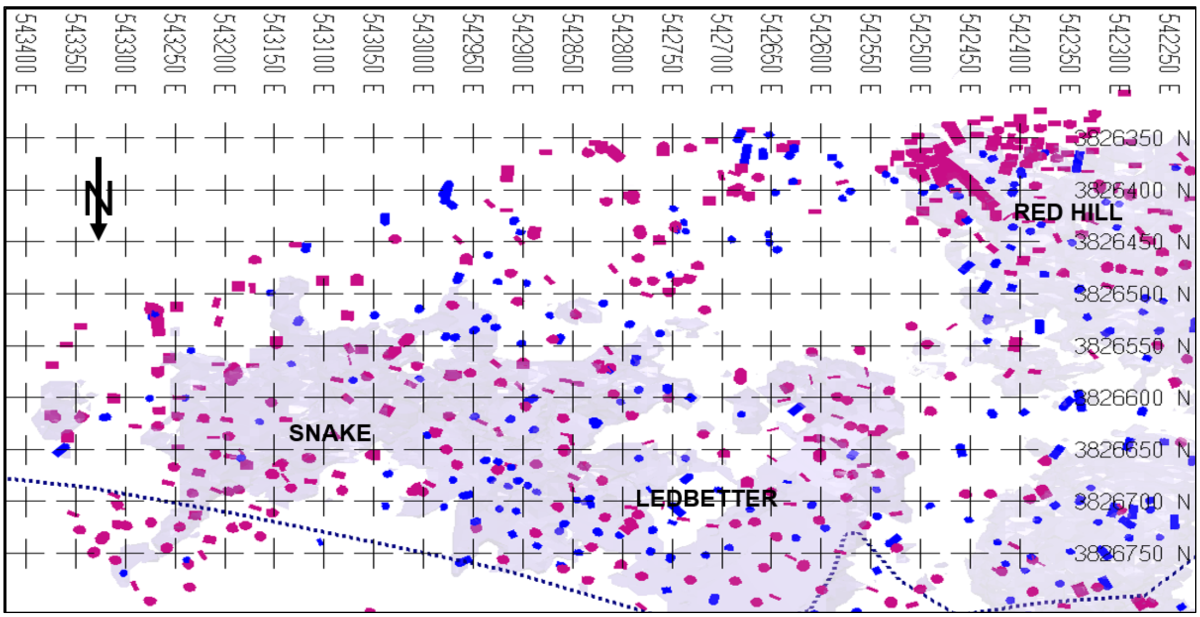
Table 14-5: HA1220OLM BD Assignment

BD Assignment Criteria								Criteria for Ore vs. Waste			
								Ore = Inside Gold Shell			
								Waste = Outside Summary Shell			
BD Assignment Criteria for HA1220OLM Model											
Sand	Saprolite	Dike	Meta Volcanics		Meta Sediments				Pag Fill	Tails	Heap
			Oxidized	Fresh	Ore		Waste				
					Oxidized	Fresh	Oxidized	Fresh			
2.06	2.18	2.88	2.52	2.7	2.57	2.78	2.49	2.76	1.89	2.14	1.7

Source: OceanaGold, 2022

14.1.15 Resource Classification

The location of wetlands, historical dumps and open pit mining activity has at times restricted drill rig access. In many cases, this has meant that mutliple holes have been collared off single drill pads, resulting in fanned and variable drillhole orientations, oblique intersection angles and irregular drillhole spacings. Figure 14-3 below provides an example in the Snake, Ledbetter and Red Hill areas. The View is -45° to South, showing drillhole pierce points (+/- 5 m) projected onto the mineralization planes for Ledbetter, Snake and Red Hill areas. A 50m grid is shown. Purple shading represents Measured and Indicated Resources (mined and unmined). Maroon dots represents pre-OceanaGold drillholes and blue dots represent OceanaGold drillholes. Given the erratic 3D spatial distribution of drillhole samples, the approach to resource classification for the Haile open pit resources relies on a 3D search methodology. This approach compares well against broad and consistent regions of kriging variance and slope regression. Although variable, Indicated Resources are typically based on drill spacings of 40 m or less.



Source: OceanaGold, 2022

Figure 14-3: View -45° to South of Drillhole Pierce Points

Resource classification was assigned by Vulcan script according to the search criteria for each domain in Table 14-6 and Table 14-4.

Table 14-6: Resource Classification Parameters

Confidence Category	Criteria
Measured	Blocks estimated within first pass, A minimum of 4 drillholes
Indicated	Blocks estimated within first and second pass, A minimum of 2 drillholes
Inferred	All blocks estimated using first and second pass.

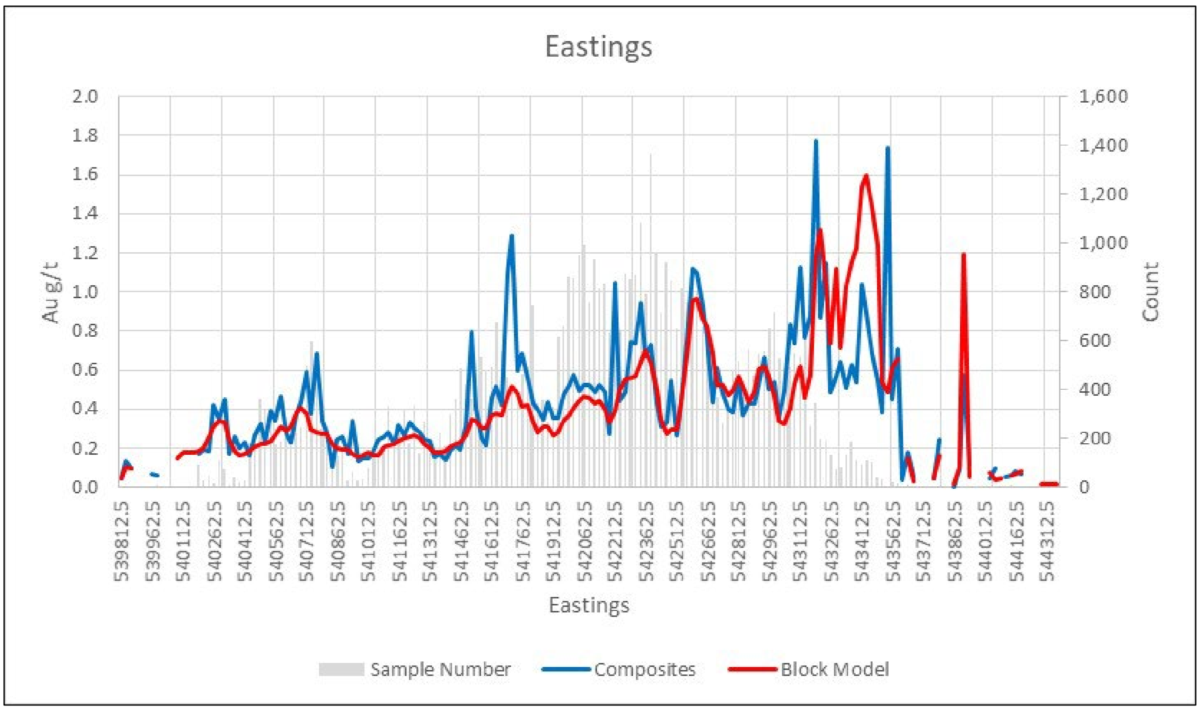
Source: OceanaGold, 2022

14.1.16 Model Validation

Numerous methods have been used to validate the HA1220OLM resource model, such as:

- Cross-sectional checks on composite file and block model coding from lithological wireframes, domain area and grade shell
- Visual checks of estimated block grade on sections, plan and in 3D to ensure good correlation with composite data
- Swath plots comparing the gold estimates with the underlying composite grades
- Detailed comparisons to previous model at global and local scales
- Review of the methodology and validation of the scripts used

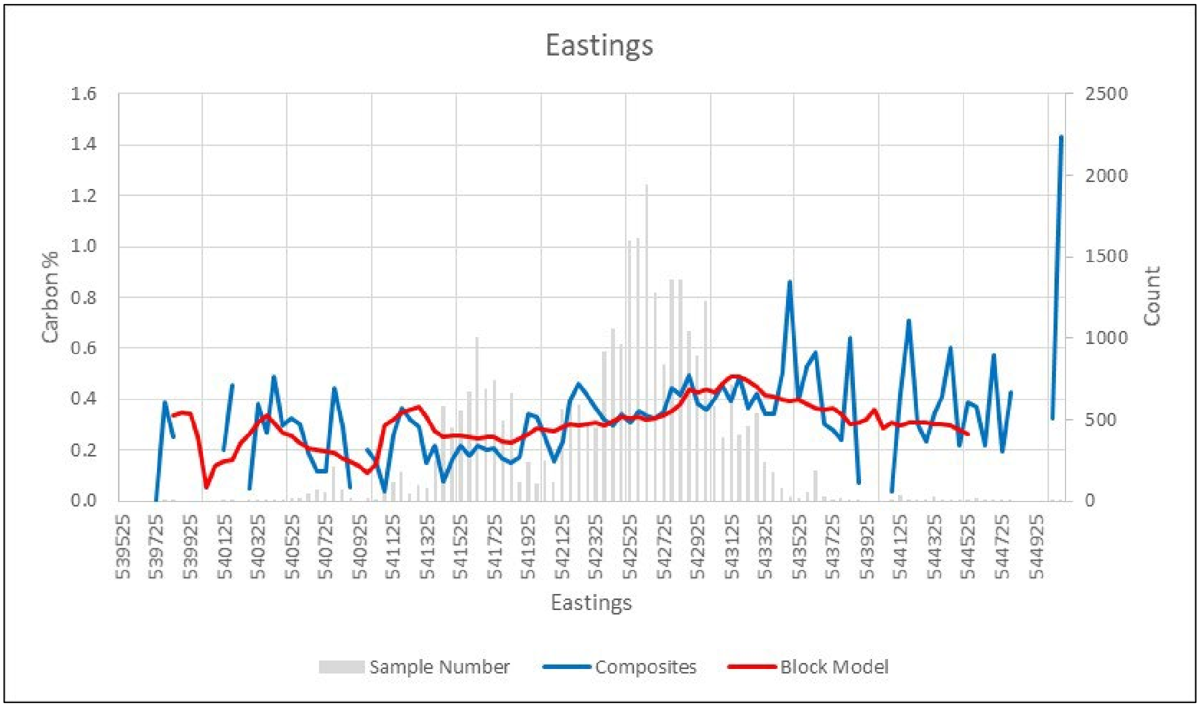
The swath plots in Figure 14-4 and Figure 14-5 compare 2.5 m bench composite grades to the estimated block grades for gold, carbon, sulfur and silver respectively.



Source: OceanaGold, 2022

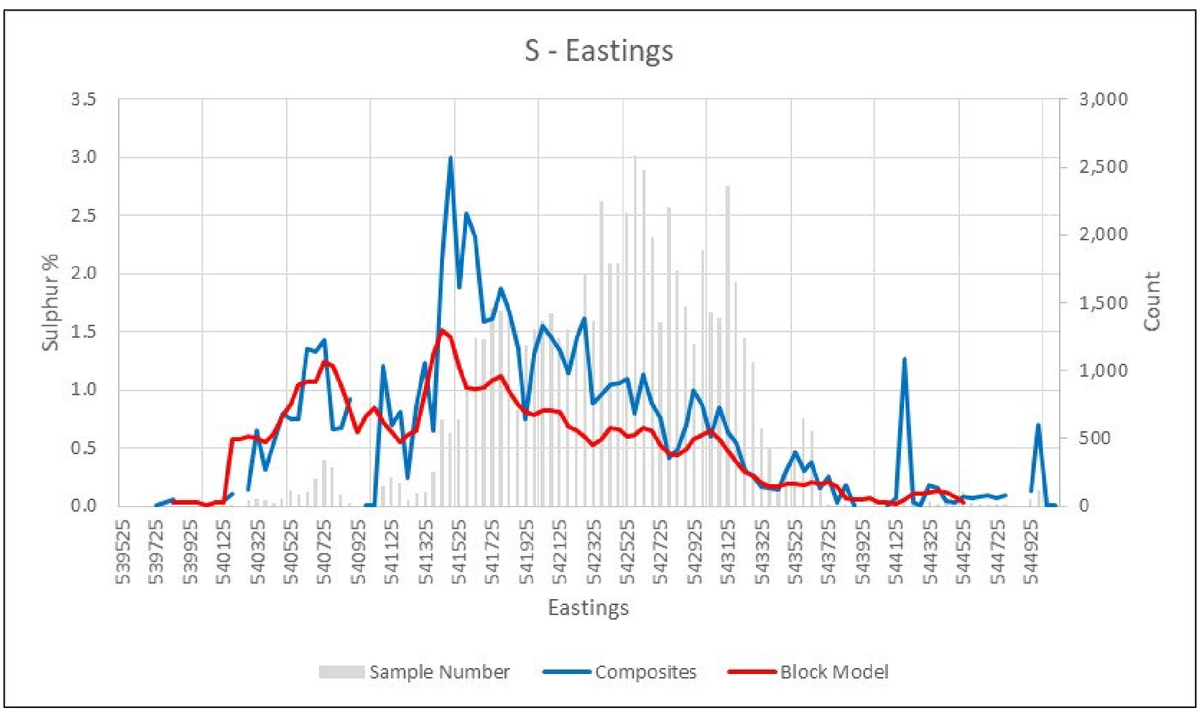
Figure 14-4: Swath Plot with Mean Composite Gold Grade vs. HA1220OLM Block Values

Carbon and sulfur were estimated from raw drillhole data because of the lack of data relative to the gold and silver data sets. Swath plots for carbon, sulfur, and silver are shown in Figure 14-5 through Figure 14-7.



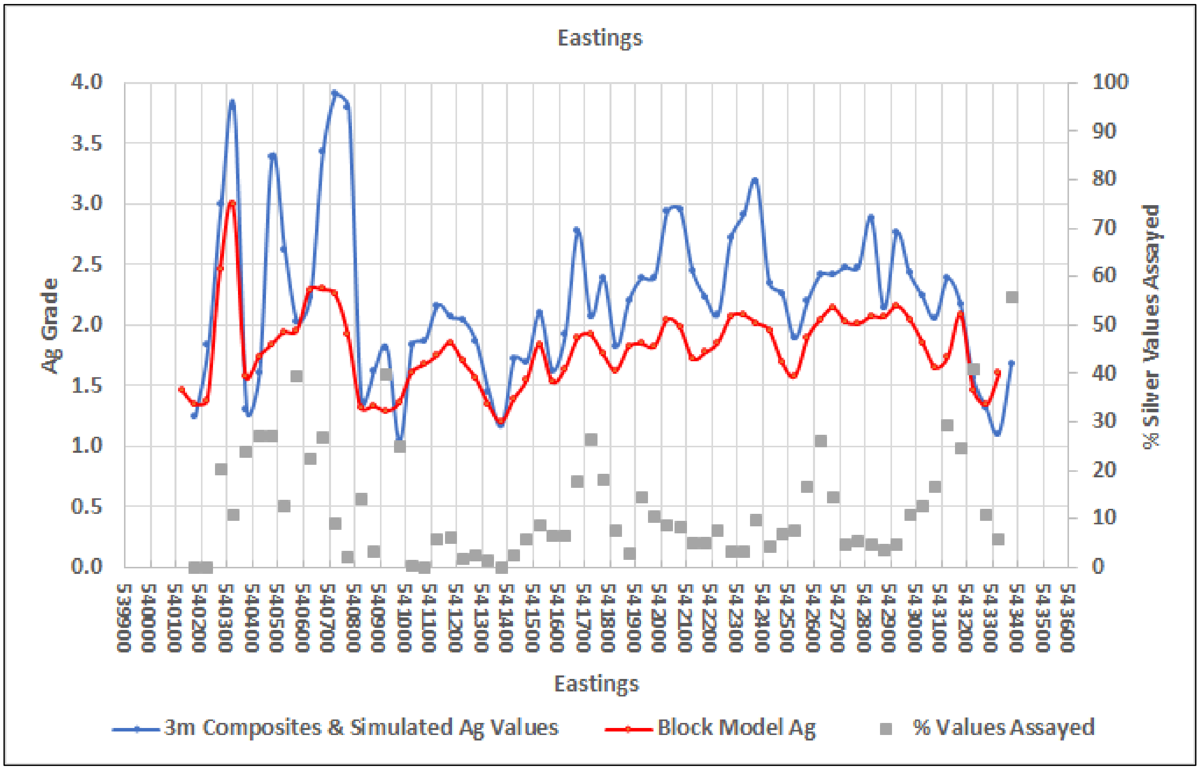
Source: OceanaGold, 2022

Figure 14-5: Swath Plot with Mean Composite Carbon Grade vs. HA0520OLM Block Values



Source: OceanaGold, 2022

Figure 14-6: Swath Plot with Mean Composite Sulfur Grade vs. HA1220OLM Block Values



Source: OceanaGold, 2022

Figure 14-7: Swath Plot with Mean Composite Silver Grade vs. HA1220OLM Block Values

As a gold estimation methodology check, an independent large panel recoverable resource estimate using MIK was constructed and compared to HA1220OLM on a stage-by-stage and easting swath basis. The check estimate did not require constraint by an implicitly derived grade shell for gold estimation, so provided a useful parallel estimate via alternative modeling assumptions. Globally, the two estimates are within 2% of each other in terms of tonnes, grade and contained gold. There were some local differences, most pronounced in Haile Pit and Red Hill Pit areas where the distribution of mineralization is locally more complex.

Model Reviews

External

- "The Haile Gold Mine, South Carolina, United States, Review of the 2020 Mineral Resource Estimate", July 20, 2021 by Ginto Consulting Inc. Summary of findings:
  - Gold grade estimates have no bias overall and considered appropriate.
  - Drill spacing is adequate for the gold grade continuity and classification used.
  - Recommendation to consider simplifying process by using ordinary kriging rather than Multiple Indicator Kriging. Initial trials have not proven successful. However, it may be that by reconsidering the grade domaining strategy, ordinary kriging may eventually yield success.
  - Recommendation to look at ways to increase geological input. Oceana will continue to explore the potential to integrate alteration logging and/or structure data. To-date the use of logged alteration (silica and pyrite) to improve estimation domains has not been successful, despite an observable relationship with gold mineralization.

- Recommendation was to review the degree of smoothing in the estimate. Kriging Neighbourhood Analysis was completed by OceanaGold in response to the recommendation and the search parameters were found to be appropriate. A sensitivity modeling analysis, summarised below, clarifies the cause of local estimation uncertainty.

Grade control data reveal that high grade lozenges with approximate 10mE x 10mN to 20m x 20m dimensions, account for a large proportion of the in-pit contained gold. This lumpy grade distribution introduces a hit and miss element into the resource drilling which supports the resource estimates. During 2021, Oceana generated approximately 80 open pit sensitivity resource estimates using a range of resource drilling patterns sampled by filtering the exhaustive grade control data. Although a number of drill spacings were tested, the 35 m x 35 m spacing is the most representative of typical Haile open pit resource drillhole spacing:

- Numerous 35 m x 35 m drilling sets were generated by moving the origin of each drilling pattern at 5 m E and 5 m N increments and propagating each pattern at 35 m x 35 m spacings from it's origin.
- Resource estimates using identical parameters were then generated from each of the 35 m x 35 m drilling sets.
- The variation across the 35m x 35m sensitivity estimates was significant and would explain the annual variation range for Haile open pit model to mine reconciliation (see section below).

Constructing large numbers of sensitivity estimates is time-consuming. Nonetheless, this study should be extended to achieve a larger number of sensitivities on which expected monthly, quarterly and annual model performance ranges can be approximated.

14.1.17 Resource Model Reconciliation

Table 14-7 summarizes the resource model reconciliations from 2018 to-date (31 December 2021). The reconciliation is expressed as a ratio of mill-adjusted mining / resource estimate.

The resource model to mill-adjusted mine reconciliation data the 4-years to 2021 show variable performance from year to year. The long-term average performance for this period shows +12% for tonnes, -4% for grade and +8% for contained gold. The four-year grade reconciliation is however negatively skewed by low mining selectivity during 2020 which resulted in excessive mining dilution during that year. More selective mining practices re-introduced during 2021 have resolved this.

Whilst annual reconciliation fluctuations are expected to continue, the open pit resource estimates are believed to provide an acceptable basis for medium to long term mine planning purposes.

Table 14-7: Resource Model Reconciliation

Year	HA1220OLM (MEAS&IND)			Mine (Mill-Reconciled)			Reconciliation Ratios		
	Tonnes	Grade	Au Oz	Tonnes	Grade	Au Oz	Tonnes	Grade	Au Oz
2021	3,162,000	1.98	201,695	3,273,607	2.17	228,179	1.04	1.09	1.13
2020	2,567,953	2.08	171,375	3,329,027	1.59	170,031	1.30	0.76	0.99
2019	2,867,894	1.96	180,599	3,179,597	1.78	182,394	1.11	0.91	1.01
2018	2,850,606	1.67	153,353	2,570,522	1.93	159,172	0.90	1.16	1.04
Total	**11,040,947**	**1.94**	**687,481**	**12,352,753**	**1.86**	**739,777**	**1.12**	**0.96**	**1.08**

Source: OceanaGold, 2022

14.1.18 Mineral Resource Statement

Table 14-8 summarizes the resulting open pit resources. A US$1,700/oz shell was used with a cut-off of 0.45 g/t.

Table 14-8: Open Pit Total Mineral Resources as of 31 December 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Measured	2.68	1.30	0.11	2.54	0.22
Indicated	43.0	1.55	2.14	2.41	3.33
Measured & Indicated	**45.6**	**1.54**	**2.25**	**2.42**	**3.55**
Inferred	5.7	1.0	0.2	1.3	0.2

Source: OceanaGold

- Cut-off grade 0.45 g/t Au for primary mineralization and 0.55 g/t Au for oxide based on a gold price of US$1,700/oz.
- Open pit resource is reported within a US$1,700/oz optimized shell.
- Stockpiles not included (see Table 14 26)
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The open pit Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

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

14.2 Underground Mineral Resource Estimate

14.2.1 Horseshoe Mineral Resource Estimate

The Horseshoe resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling current to April 22, 2020.

Gold estimation was constrained within implicitly modeled grade shells, approximating a 1 g/t gold indicator. The 1 g/t Au threshold was selected to be sufficiently below the reserve reporting cut-off grade of 1.44 g/t Au to minimize conditional bias. Model blocks with centroids outside the 1 g/t shell were coded as unclassified and not reported (i.e., assigned zero grade). Modeled post-mineralization dikes were assigned zero grade. Metasediment/metavolcanic contacts were not used to constrain gold estimation.

Gold grades were estimated with Vulcan™ modeling software into 10 m E x 10 m N x 10 m Rl blocks (sub-blocked to 5 m E x 5 m N x 5 m Rl) using Ordinary Kriging with 3 m composites. In situ dry bulk densities based upon core analyses were assigned by rock type.

The Mineral Resources reported for the Horseshoe deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drillhole spacing, but also considering geological complexity.

<u>Horseshoe General Geology and Geologic Model</u>

The Horseshoe deposit is the highest grade and easternmost known gold deposit in the Haile district. Mineralization extends over a vertical distance of 350 m, length of 200 m and width of 120 m. The top of the deposit is about 120 m below surface and is one of several meta-siltstone-hosted deposits located near the steeply SE-dipping contact with metamorphosed volcanic rocks of the Upper Persimmon Fork Formation. The Horseshoe gold deposit is characterized by strong silicification, 1% to 5% pyrite, and a halo of 0.5% to 1% pyrrhotite. The rocks have been deformed by high strain, isoclinal folding and shearing with a pervasive foliation striking 060° E and dipping 40° to 60° NW. All units are cut by post mineralization diabase dikes striking 330° with near-vertical dips. OceanaGold has constructed a geologic model which includes the metasiltstone, metavolcanics, diabase dikes, saprolite and sand. The metasiltstones are the main host of mineralization. These five rock types constitute the lithologies coded in the block model. This has resulted in a detailed, 3D geologic model created with Leapfrog® software.

Horseshoe Geologic Model and Controls on Gold Mineralization

Mineralization is concentrated in two main zones based on vertical position which form a "horseshoe" geometry over a vertical distance of 350 m. Both zones strike NE adjacent to the siltstone-dacite contact, however, the upper zone dips about 40° NW and the bulk geometry of the lower zone is near-vertical. The upper zone NW-dipping high-grade lenses of mineralization are focused along bedding-parallel foliation with intense silicification. The Horseshoe fault (NE strike, 40° NW dip) juxtaposes the hanging wall of upper Horseshoe against barren dacite with a sill-like geometry. This geometry extends southwestward into the nearby Snake pit. The steeply dipping Lower Zone is adjacent to the sub-vertical contact with barren dacite. Compared to the upper zone, gold grades are lower, silicification is less intense, and pyrite contents are lower. Extents of economic mineralization in lower Horseshoe have not been fully delineated by drilling.

Horseshoe Bulk Density

Model in situ dry bulk densities (BD) are based on domain averages, as shown in Table 14-9. The model densities were assigned through a combination of lithology, oxidation, alteration, and an ore/waste threshold. The BD was assigned for each lithology type. In situ density determinations have been carried out at regular intervals on drill core samples. The method involved weighing the sample both in air and in water. The measurements were then averaged for each lithology.

Table 14-9: Densities Assigned in the Block Model

<table>
<tr><th colspan="8" rowspan="3">BD Assignment Criteria</th><th colspan="4">Criteria for Ore vs. Waste</th></tr>
<tr><th colspan="4">Ore = Inside Gold Shell</th></tr>
<tr><th colspan="4">Waste = Outside Summary Shell</th></tr>
<tr><th colspan="12">BD Assignment Criteria for HA0619OLM Model</th></tr>
<tr><th rowspan="3">Sand</th><th rowspan="3">Saprolite</th><th rowspan="3">Dike</th><th colspan="2">Meta Volcanics</th><th colspan="4">Meta Sediments</th><th rowspan="3">Pag Fill</th><th rowspan="3">Tails</th><th rowspan="3">Heap</th></tr>
<tr><th rowspan="2">Oxidized</th><th rowspan="2">Fresh</th><th colspan="2">Ore</th><th colspan="2">Waste</th></tr>
<tr><th>Oxidized</th><th>Fresh</th><th>Oxidized</th><th>Fresh</th></tr>
<tr><td>2.06</td><td>2.18</td><td>2.88</td><td>2.52</td><td>2.7</td><td>2.57</td><td>2.78</td><td>2.49</td><td>2.76</td><td>1.89</td><td>2.14</td><td>1.7</td></tr>
</table>

Source: OceanaGold, 2022

Horseshoe Sample Database

Previous owners drilled the majority of the holes over the entire area of mineralization between 1981 and 2015. OceanaGold have continued with focused diamond core drilling at Horseshoe since 2015. 90 drillholes for 31,873 m were used within the Horseshoe resource area. Of these 74 drillholes intersected the 1 g/t grade shell, the remainder were outside this volume.

Horseshoe Compositing and Top Capping

Compositing was completed in Vulcan software to 3 m downhole lengths with no breaks at lithologic contacts. The 3 m length was chosen to reflect the low degree of mining selectivity and the absence of any visual features that coincide with the 1 g/t cut-off. It also reduced noise in the data which was resulting in irregular implicit shell geometries and smoothed assay values across two 1.5 samples.

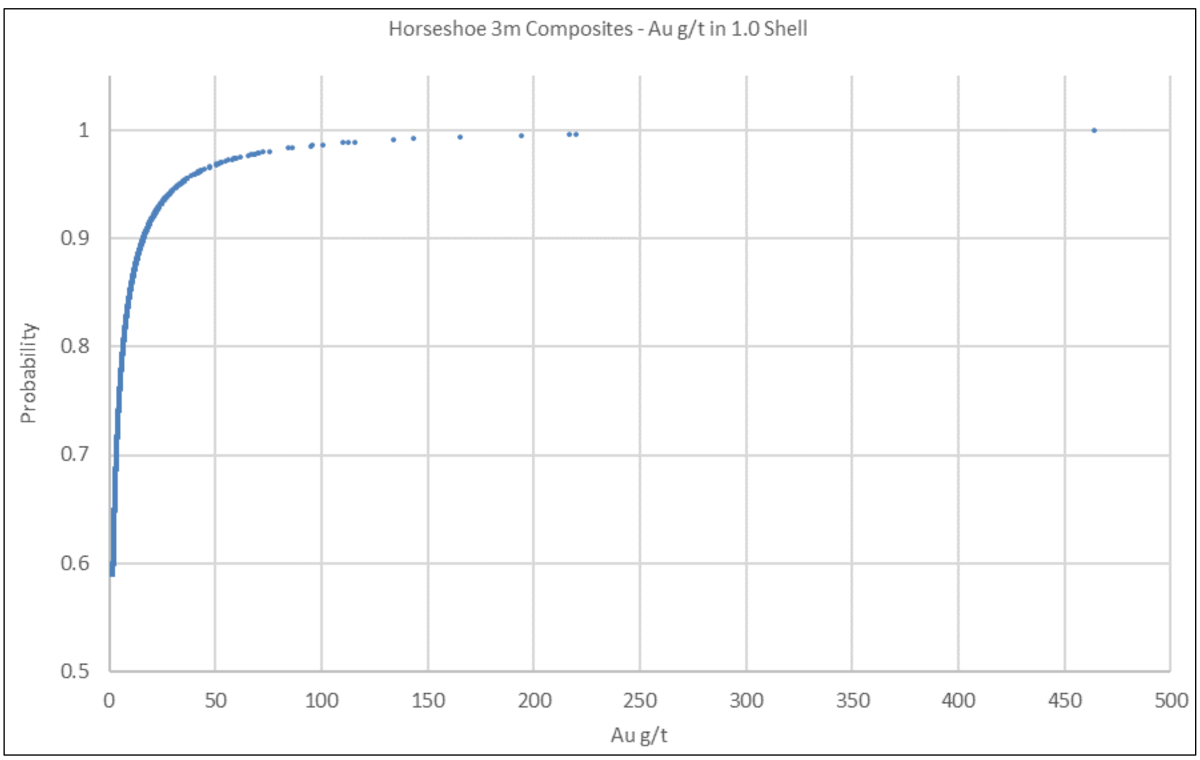
Table 14-10 summarizes the statistics of 3 m composites within the 1 g/t Au indicator shell. Sample localities without gold assays were assigned 0.0 grades unless belonging to drillholes with pending assays results.

Table 14-10 Basic Statistics for 3 m Composites Within 1 g/t Au Indicator Shell

	Sample Stats
	Au
Count	1,194
Min	0.005
Max	149
Mean	4.63
CV	2.13

Source: OceanaGold, 2022

Statistical analysis of the original drillhole sample data has resulted in a capping value of 100 g/t for the composites used in the estimation. The results of the cumulative distribution plot are presented in Figure 14-8.



Source: OceanaGold, 2022

Figure 14-8: Log Normal Cumulative Distribution Plot of Gold Assays above 2 g/t

Horseshoe Block Model

In July 2019, an 030°-rotated local grid was created for Horseshoe. This rotated grid facilitated Mine Design due to alignment of the primary 060° mineralization direction with the long axis. Details of the rotation, and block model limits of the OceanaGold resource model are listed below. The block model coordinates are referenced to a rotation about the UTM NAD83 coordinate system and are based on a compromise between the average drillhole spacing, a typical underground stope selective mining unit and the variability of the mineralization.

The HUG local grid is based on a 30° clockwise rotation around 3,824,000 m N and 541,000 m E, with a 1,000 m adjustment to elevations as shown in Table 14-11. Elevations were increased by 1,000 m relative to sea level to remove negative values. The origin, limits and block sizes are listed in Table 14-12.

Table 14-11: HUG Grid Transformation Details

	Haile Surface Grid			Horseshoe UG Local Grid		
	Easting	Northing	RL	Easting	Northing	RL
Origin	541,000	3,824,000	0	0	0	1,000
Point 1	543,431	3,826,789	0	3,500	1,200	1,000
Point 2	542,732	3,825,000	0	2,000	0	1,000

Source: OceanaGold, 2022

Table 14-12: Block Model Dimensions and Origin

Variable	X	Y	Z
Minimum	3,300	900	620
Maximum	3,920	1,600	1,200
Block Size (Parent)	10	10	10
Sub-block size	5	5	5
No. of Blocks (Parent)	62	70	58

Source: OceanaGold, 2022

Horseshoe Estimation

Gold estimation was constrained within implicitly modeled grade shells, which were implemented as hard grade boundaries. The shells were generated in Leapfrog® software at a 1 g/t Au threshold guided by interpreted trend planes of mineralization. The trend planes were developed by digitizing section profiles of gold continuity which were then triangulated into 3D planes of gold continuity. The upper zone of mineralization utilized two trend planes which essentially represent the hanging wall and footwall of mineralization. The lower zone utilized two additional trend planes.

As described above, the gold mineralization is hosted in two domains each with a unique orientation. For each zone, the trend planes used to guide the grade shells construction were translated outward to capture all model blocks enclosed by the grades shell. These translated planes were then used to guide a dynamic search orientation utilized in the ordinary kriging gold grade estimation.

The grade estimations utilize a two-pass sample search strategy with each pass searching longer distances than the previous. The search distances and variogram parameters are listed in Table 14-13 and Table 14-14, respectively. For all estimations, the following criteria were used:

- Dynamic search orientation essentially parallel to the plane of gold continuity for each zone
- Minimum of two composites and maximum of twelve composites to estimate grade

- Sample length weighting to account for any short composites located at the ends of drillholes
- Composites from a minimum of two drillholes
- Composites from a minimum of two octants

Table 14-13: Au Grade Estimation Search Distances

Estimation Domain	Estimation Pass	Search Range (m) (X, Y, Z)
Upper	1	60,20,10
	2	300,100,50
Lower	1	60,60,60
	2	210,210,210

Source: OceanaGold, 2022

Table 14-14: Ordinary Kriging Parameters

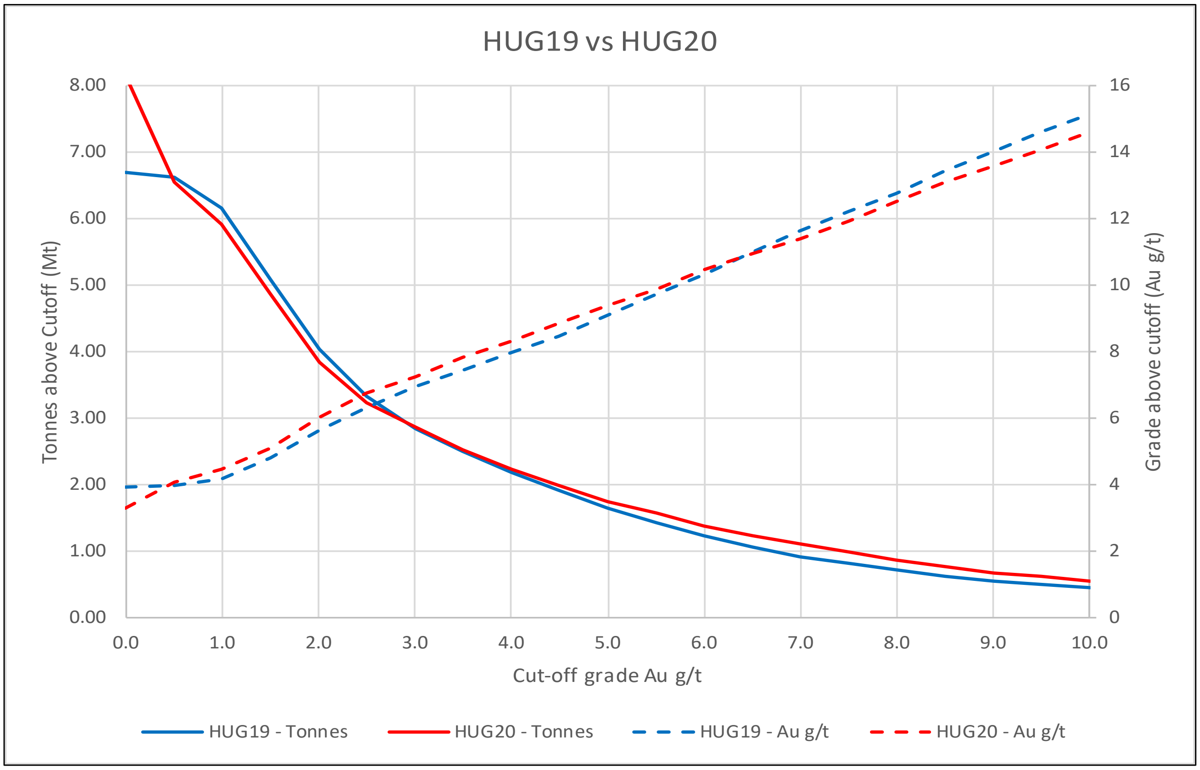
Estimation Domain	Variogram Structure	Nugget	Sill Differential	Rotations (Vulcan, X, Y, Z)	Ranges (m) (X, Y, Z)
Upper	1st Spherical 2nd Exponential	0.3	0.3 0.7	20°, 0°, 75° 20°, 0°, 75°	15,12,9 65,30,15
Lower	1st Spherical 2nd Exponential	0.3	0.42 0.32	Isotropic	12,12,12 105,105,105

Source: OceanaGold, 2022

Horseshoe Model Validation

Several techniques were used to evaluate the validity of the block model. All new drilling and lithological mapping data were visually validated. QA/QC was performed on drilling data as per database procedures, and visual validation of lithological logging was performed on new drillholes. A visual review of all generated wireframes, including implicitly modeled mineralization wireframes, was performed, to compare with previous versions and to drillhole lithological logging. The methodology used for the resource modeling was reviewed, to ensure industry standard processes and assumptions were used. A review of all macros used in the estimation process was performed, to ensure all appropriate files were used, and correct naming conventions were followed. Model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data. This included the number of composites used, number of drillholes used, average distance to samples used, and the number of blocks estimated in each pass. Comparisons were made to the previous 2019 resource model (HUG19), in terms of grade, tonnages and contained metal.

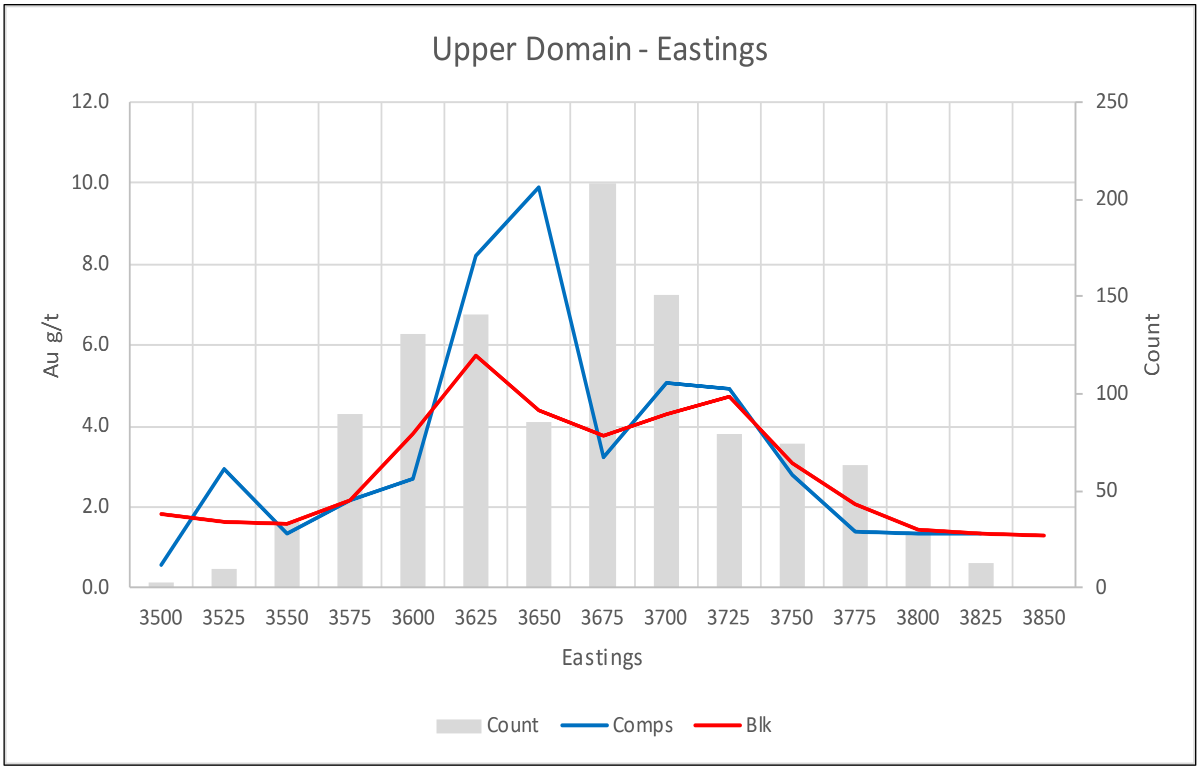
Figure 14-9 compares estimates and reveals very little change.



Source: OceanaGold, 2022

Figure 14-9: Global Grade Tonnage Comparison between 2019 and 2020 Models

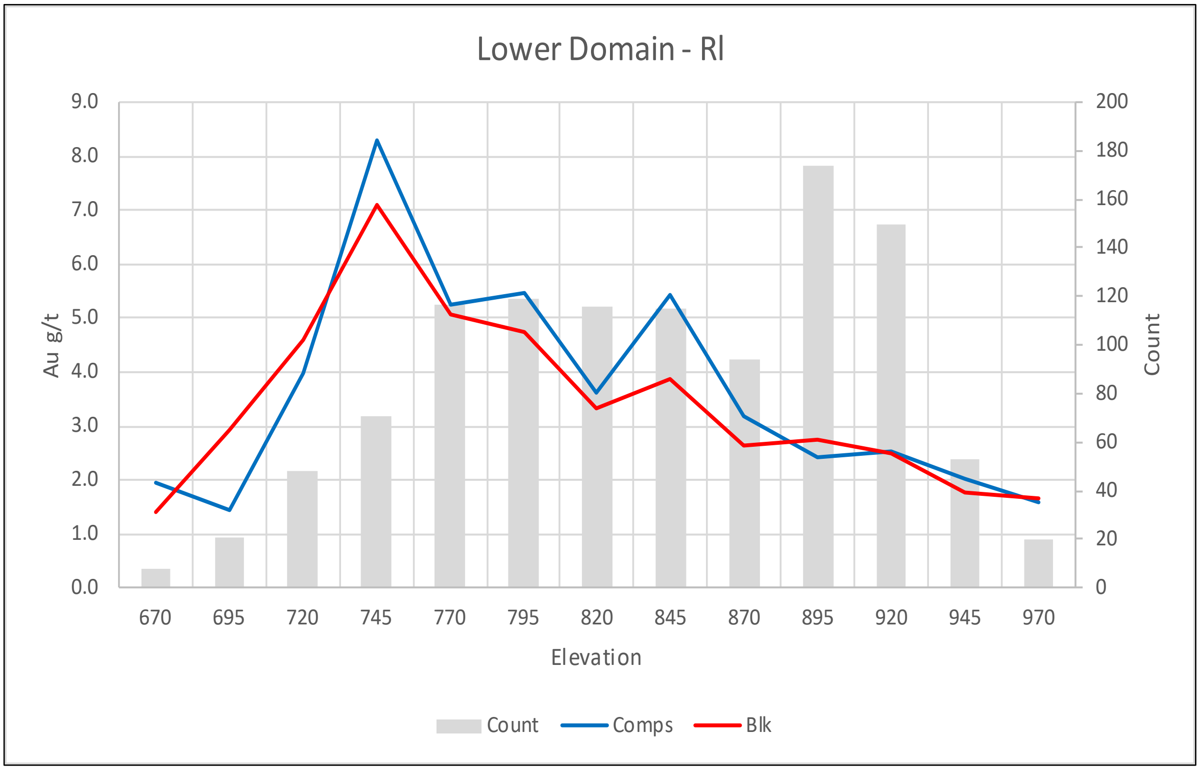
Swath plots were used to compare the estimation with underlying composite grades for each domain. Figure 14-10 shows an acceptable correlation between the composites (Comps) and the block estimation grade (Blk) for the Upper Domain. The deterioration at 3650 m E is related to the relatively low number of composites at that Easting.



Source: OceanaGold, 2022

Figure 14-10: Upper Domain Easting Swath Plot

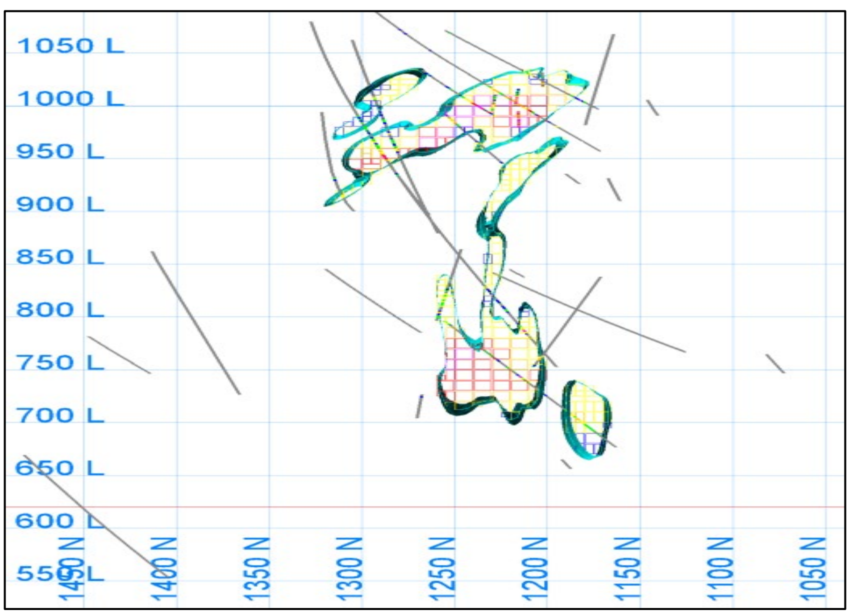
Figure 14-11 shows the Lower Domain and shows an acceptable correlation between the composites and the block estimation grades. The last validation involved a visual validation of the final block model to both domain limits and composite grade comparison.



Source: OceanaGold, 2022

Figure 14-11: Lower Domain RI Swath Plot

Figure 14-12 and Figure 14-13 show representative cross sections of the gold estimation results.



Source: OceanaGold, 2022

Figure 14-12: Representative Cross Section with Estimated Au Grades (Viewing E90°)

Based on the results of the various model validations, the OK estimate was chosen as the final Mineral Resource estimation. All resource reporting tables are based on this estimation.

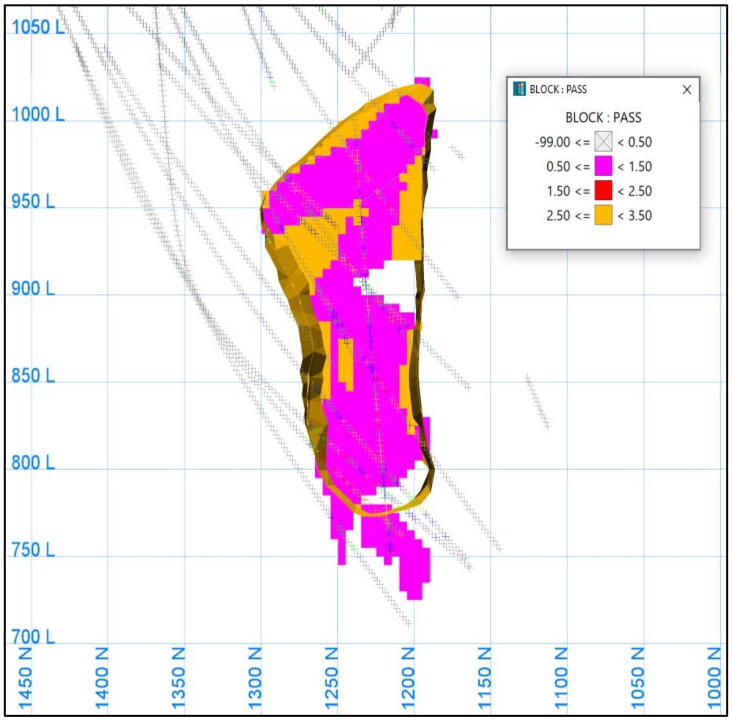
An external audit of the Horseshoe underground estimate was completed by SD2 Pty Ltd in October 2021. The key findings and recommendations for the Horseshoe underground estimate were:

- The Horseshoe mineral resource estimate is consistent with good industry practice
- The greatest risk associated with the Horseshoe estimate is the interpreted geometry and extent of the mineralisation. Multiple plausible interpretations are possible and, while these alternatives do not materially affect the global estimate, there are potential differences in the local orientation and distribution of the mineralisation. More sensitivities recommended.
- Continuously monitor the spatial and statistical distribution of very-high grade samples. Investigate potential geological controls for extreme grades to ensure they are appropriately modelled and estimated.

Horseshoe Resource Classification

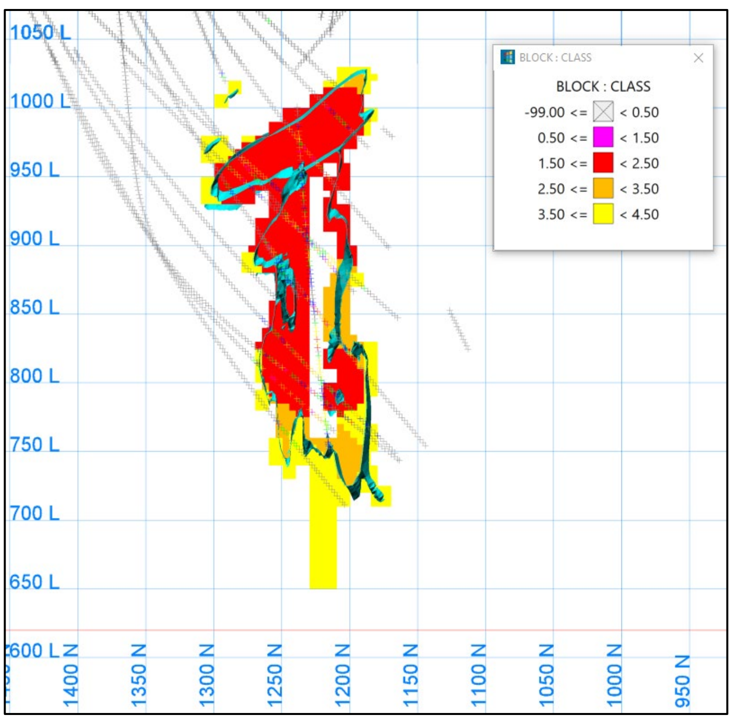
Mineral Resources are classified as Indicated and Inferred in accordance with CIM guidelines. There are no Measured Resources. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based primarily on the sample spacing and geological complexity. No single factor controls the Mineral Resource classification, rather each factor influences the end result.

A wireframe solid was constructed around the areas where the majority of the blocks were estimated in the first pass of the estimation, Figure 14-13. These wireframe solids were used to assign the Indicated Mineral Resource classification. All blocks outside of the Indicated wireframes were classified as Inferred Mineral Resources. No resource was estimated outside of the 1.0 g/t implicit grade shell. Figure 14-14 shows a representative cross section of the final resource classification.



Source: OceanaGold, 2022

Figure 14-13: Representative Cross Section Showing Indicated Wireframe around Pass 1 Estimated Blocks (Viewing N90°E)



Source: OceanaGold, 2022

Figure 14-14: Representative Cross Section Showing Resource Classification (Viewing N90°E)

Horseshoe Mineral Resource Statement

The Horseshoe Mineral Resource statement is based on the OK model as presented in Table 14-15. A CoG of 1.35 g/t Au has been applied without mine design constraint because the Measured and Indicated resources correspond broadly with the mine design. The CoG assumes underground mining methods and is based on a gold price of US$1,700/oz, and a gold recovery of 88%. No mining dilution has been applied.

Table 14-15: Horseshoe Underground Mineral Resource Statement as of December 31, 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)
Measured	0.0	0.00	0.00
Indicated	3.2	5.05	0.52
Measured & Indicated	**3.2**	**5.05**	**0.52**
Inferred	2.0	4.6	0.3

Source: OceanaGold, 2022

- Cut-off grade 1.35 g/t Au based on a gold price of US$1,700/oz
- No Mining dilution applied
- Spatially constrained by a 1 g/t Au indicator shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

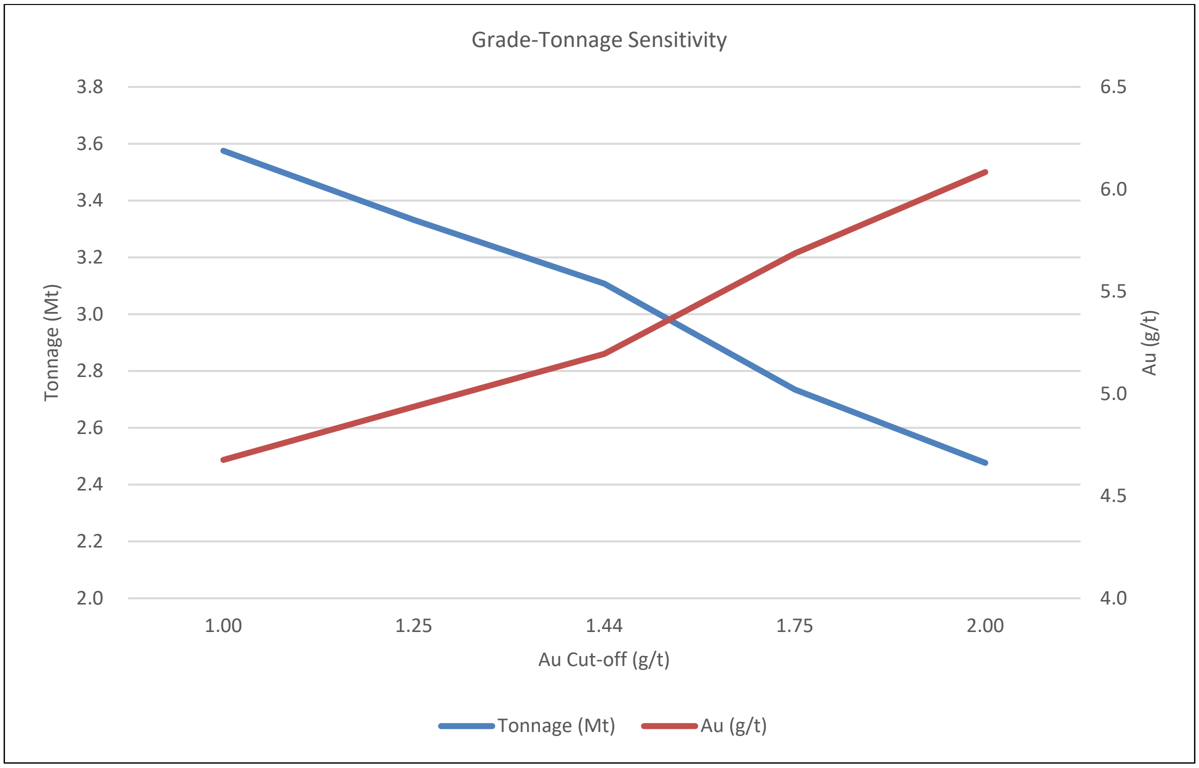
Horseshoe Mineral Resource Sensitivity

The Indicated Mineral Resources shown in Table 14-16 are presented at a range of CoG's, subdivided. Graphical representations of the grade and tonnage sensitivities of the Indicated resources are presented in Figure 14-15. Resources are not confined within any conceptual stope design.

Table 14-16: Mineral Resource Sensitivity

	Indicated		
Cut-off	Au (g/t)	Tonnes (Mt)	Au (koz)
1	4.68	3.58	538
1.26	4.95	3.32	529
1.35	5.05	3.23	524
1.5	5.28	3.04	516
1.75	5.69	2.73	500
2	6.08	2.48	484

Source: OceanaGold, 2022

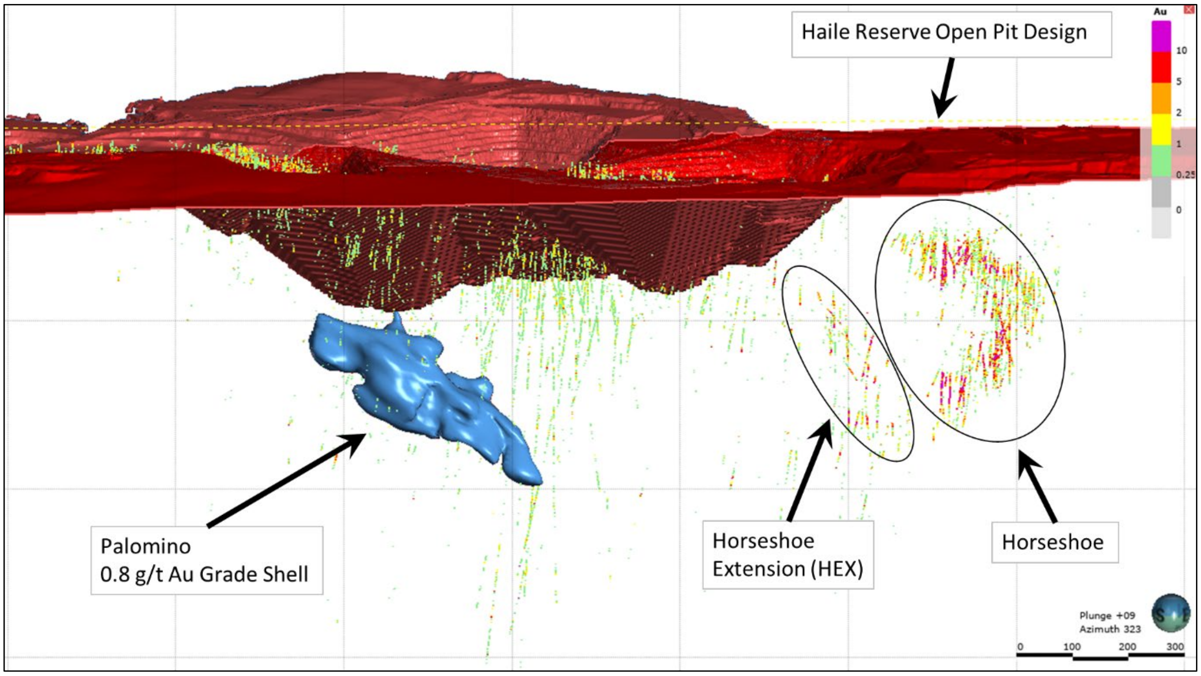


Source: OceanaGold, 2022

Figure 14-15 Sensitivity of Indicated Resource Tonnes and Grade to Cut-off

14.2.2 Palomino Mineral Resource Estimate

The Palomino deposit is a medium grade deposit, located approximately 650 m southwest of the Horseshoe, and 300 m below surface (Figure 14-16). The HEX deposit, which is located immediately to the southwest of Horeshoe, will be estimated in H1 2022, no formal estimate of this area exists to date. The Palomino resource estimation is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The resource estimation is supported by drilling and sampling completed in 2021 although final assays were received on January 17, 2022. The effective reporting date is recorded as 31 December 2021 to match the site-wide Haile mining depletion date, no mining has occurred at Palomino.



Source: OceanaGold, 2022

Figure 14-16: Long-Section looking NNW, showing Palomino Mineralization Relative to Horseshoe, HEX and, entire Haile drilling intercept dataset shown (colored by Au g/t)

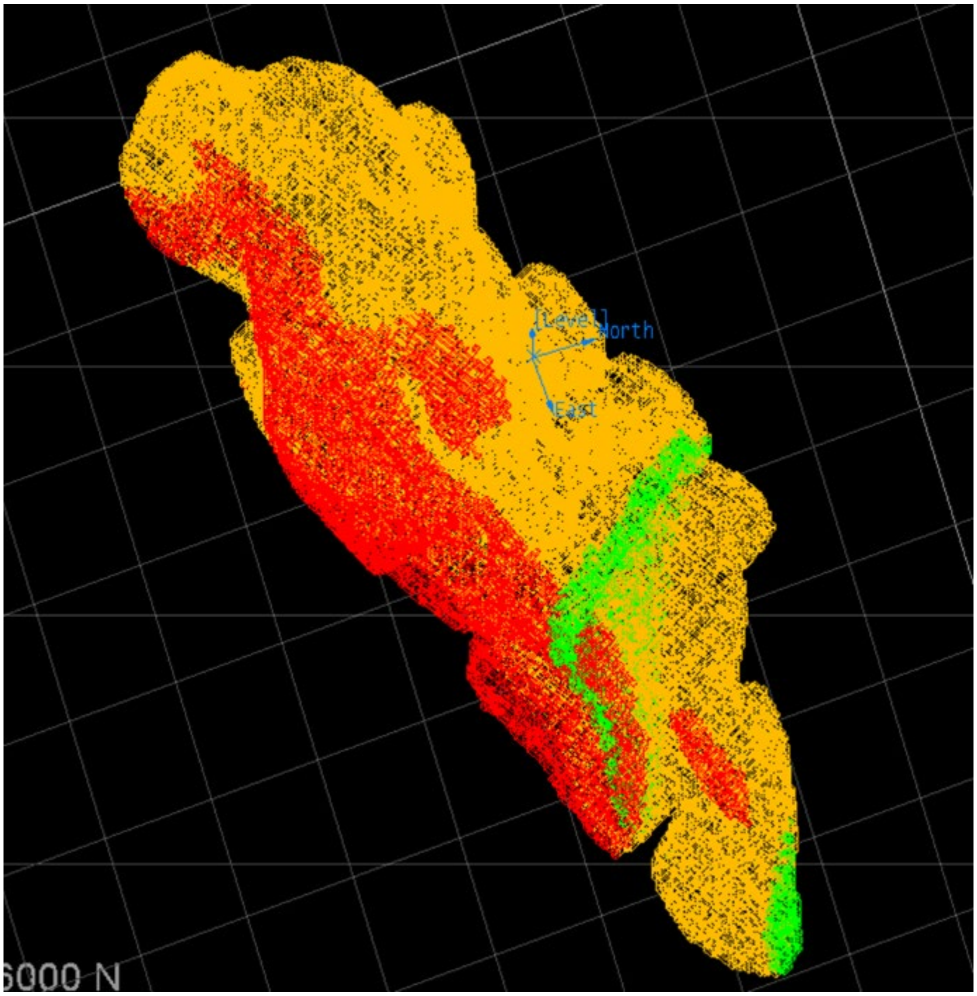
<u>**Palomino Geologic Model and Controls on Gold Mineralization**</u>

The deposit dimensions are approximately 400 m long x 70 m high x 90 m wide. Lozenge-shaped mineralized zones strike ENE, dip northwest and plunge gently northeast. Diamond drillhole spacing ranges from 20 to 70 m. Fine-grained gold is hosted in pyritic and silicified siltstone and intrusives along a steeply SE-dipping, ENE-striking contact with barren dacite flows. Mineralization is truncated by several NNW-striking, sub-vertical, 1 to 25 m thick diabase dikes.

OceanaGold has constructed a geologic model which includes the metasiltstone, metavolcanics and diabase dikes. These three rock types constitute the lithologies coded in the block model (Figure 14-17).

Red Hill which was mined in 2018 and 2019, is considered to be a good analogue for Palomino; Red Hill is located on a steep SE dipping limb in close proximity to Palomino (the majority of mineralization at Haile is located on the NW dipping limb). Review of grade control data at Red Hill confirms that there is a structural component to the control of mineralization at Haile. The metasediment is the preferential host to mineralization, although there are many instances (e.g., Mill Zone and Red Hill) where significant swaths of mineralization in the open pits have been mined in what is interpreted as metavolcanics. Figure 14-18 shows a cross-section through both Red Hill and Palomino, which shows a shallow north-west dipping structural component to the control on mineralization.

Gold estimation was constrained within an implicitly modeled grade shells created using Leapfrog® software, approximating a 0.8 g/t gold indicator, with appropriate trend surfaces to represent the controls on mineralization at Palomino.



Source: OceanaGold, 2022

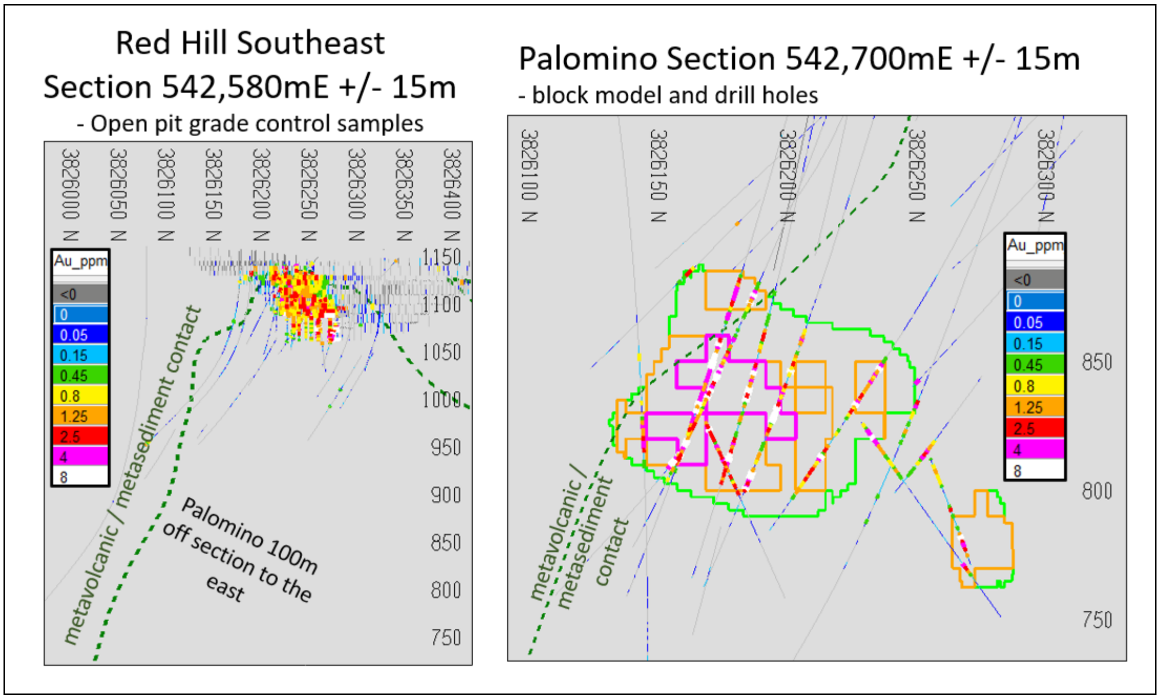
Figure 14-17 Perspective View looking WNW, showing Palomino Lithology within 0.8 g/t Au Grade Shell (orange – Meta_Sediments / red – Meta-Volcanics / green – Diabase Dike)

In 2018, prior to being mined in the open pit, Red Hill SE was identified as a good analogue for Palomino:

- Like Palomino, Red Hill SE is located on a steep SE dipping limb, and
- Is in close proximity to Palomino.

Note that a large proportion of mineralization at Haile mined up until 2018 was located on the moderately dipping, NW dipping limb. Red Hill SE was mined during 2019 and 2020 and the grade control data at confirmed that there is a structural component to the control of mineralization at Haile. The metasediment is the preferential host to mineralization, although there are instances (e.g., Mill Zone and Red Hill) where significant swaths of mineralization in the open pits have been mined in what is interpreted as metavolcanics. Figure 14-18 shows a cross-section through both Red Hill and Palomino, which shows a shallow north-west dipping structural component to the control on mineralization.

Gold estimation was constrained within an implicitly modeled grade shells created using Leapfrog® software, approximating a 0.8 g/t gold indicator, with appropriate trend surfaces to represent the controls on mineralization at Palomino.



Source: OceanaGold, 2022

Figure 14-18 Cross-Section of Red Hill Grade Control Drilling (left) and Palomino Resource Drilling (right), relative to the Metavolcanic / Metasediment contact

Palomino Bulk Density

Model in situ dry bulk densities (BD) are based on domain averages, as shown in Table 14-17. The BD was assigned for each lithology type (all lithologies within model area are fresh). In situ density determinations have been carried out at regular intervals on drill core samples. The method involved weighing the sample both in air and in water. The measurements were then averaged for each lithology.

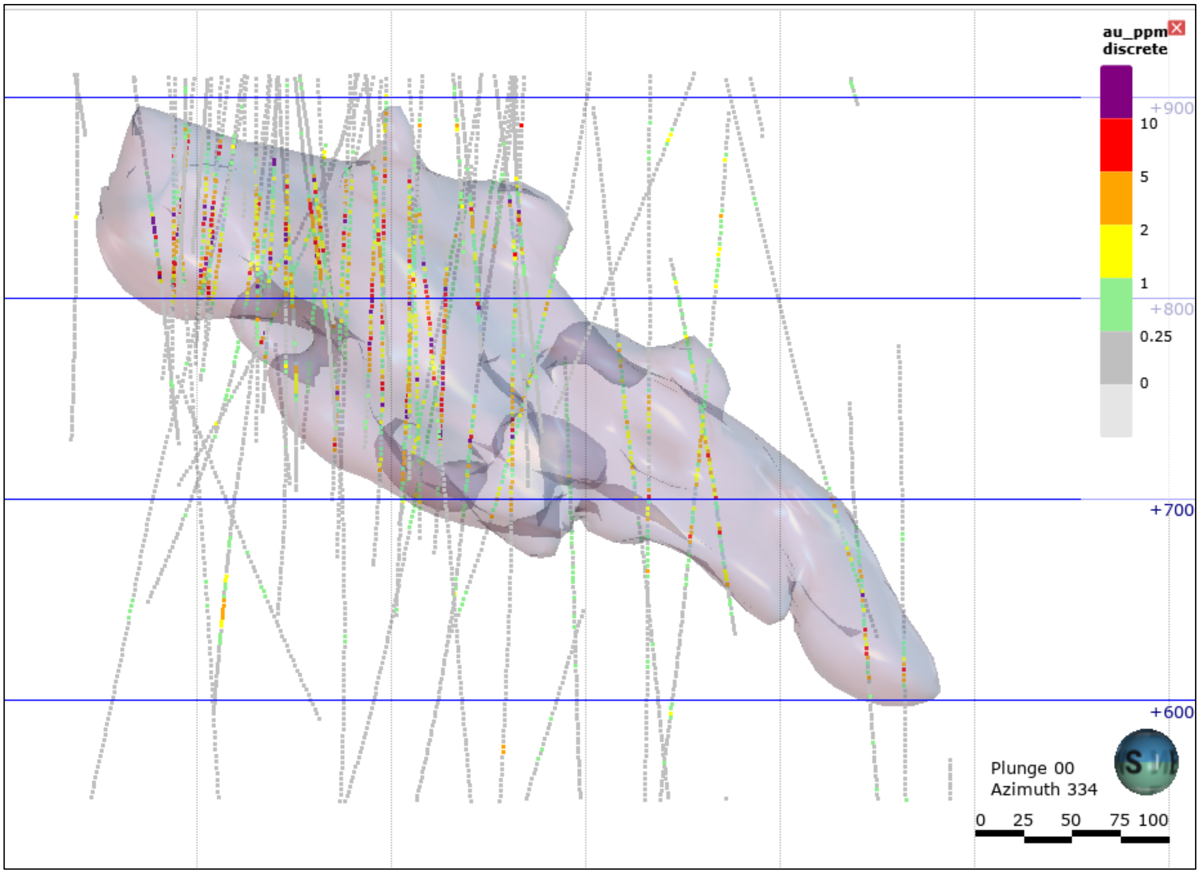
Table 14-17: Densities Assigned in the Block Model

BD Assignment Criteria		
Dike	Meta Volcanics	Meta Sediments
2.88	2.7	2.78

Source: OceanaGold, 2022

Palomino Compositing and Top Capping

Compositing was completed in Vulcan software to 3 m downhole lengths with no breaks at lithologic contacts. The 3 m length was chosen to reflect the low degree of mining selectivity and the absence of any visual features that coincide with the 0.8 g/t cut-off. It also reduced noise in the data which was resulting in irregular implicit shell geometries and smoothed assay values across two 1.5 samples. Figure 14-19 shows the 3 m composite dataset used for the estimation and shows the spatial distribution of grade within the 0.8 g/t Au grade shell.



Source: OceanaGold, 2022

Figure 14-19 Long-Section Looking NNW, Showing Palomino Mineralization with 3m Composited Data used for Estimation, 0.8 g/t Au Grade Shell shown

Statistical analysis of the original drillhole sample data has resulted in a capping value of 25 g/t for the composites used in the estimation. Table 14-18 summarizes the statistics of 3 m composites within the 0.8 g/t Au indicator shell (pug_0p8=1) and outside (pug 0p8=0). Sample localities without gold assays were assigned 0.0 grades unless belonging to drillholes with pending assays results.

Table 14-18 Basic Statistics for 3 m Composites Within and Outside the 0.8 g/t Au Indicator Shell

Variable	Domain	Count	Minimum	Maximum	Mean	Variance	Variat.Coef.
AU_PPM	pug_0p8=0	3599	0.002	7.74	0.13	0.11	2.59
AU_PPM	pug_0p8=1	1307	0.003	37.09	2.46	15.08	1.58
AU_CUT	pug_0p8=1	1307	0.003	25.00	2.42	12.79	1.48

Source: OceanaGold, 2022

Palomino Block Model

In June 2021, all Haile grids were aligned to cater for open pit and underground operations. Elevations from the Haile grid were increased by 1,000 m relative to sea level to remove negative values in underground operations. The block model is rotated to align with the primary 060° mineralization direction with the long axis. The block model parameters are listed in Table 14-19.

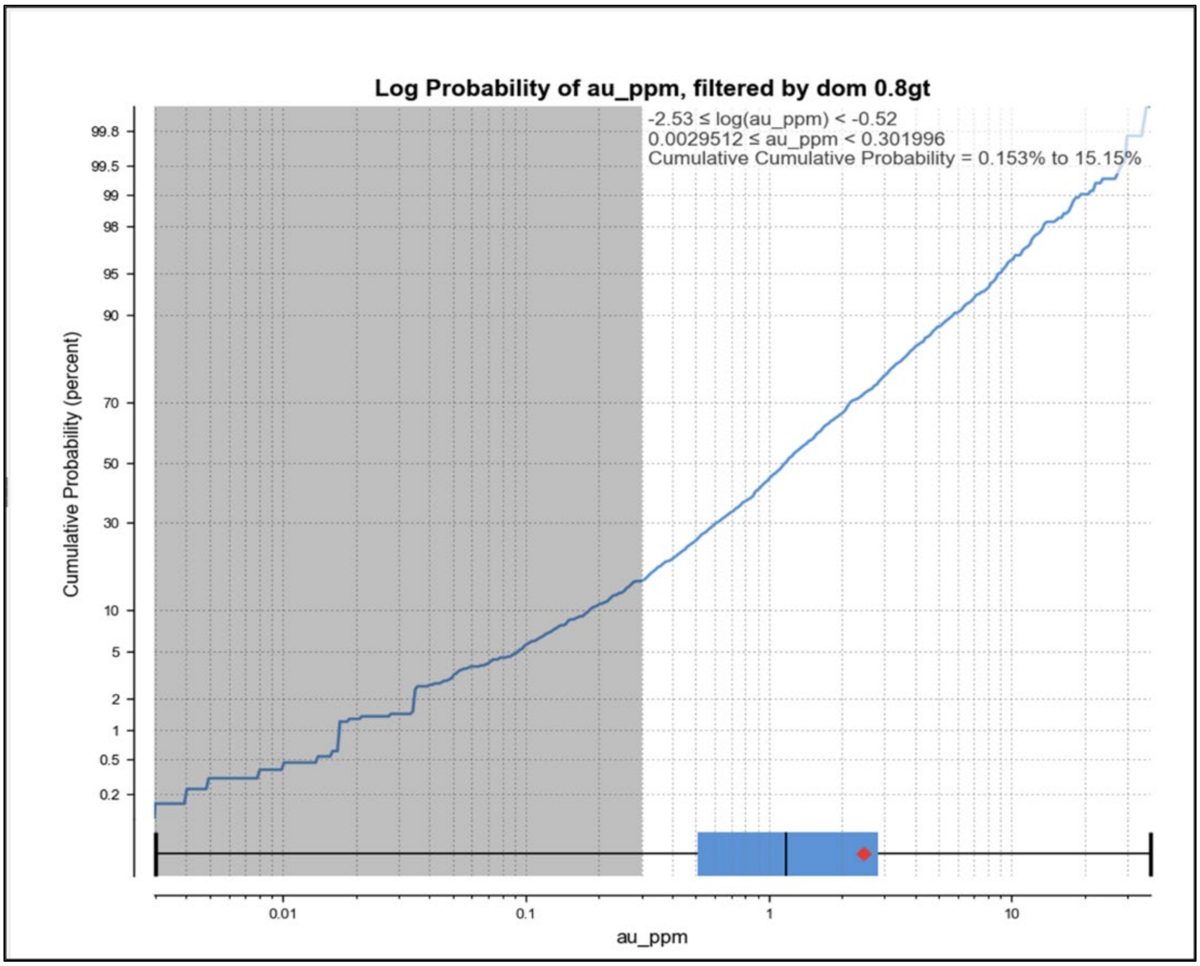
Table 14-19: Palomino Block Model Dimensions and Origin

Variable	X	Y	Z
Rotation	060°		
Origin	542,500	3,825,800	250
Length	920	520	750
Block Size (Parent)	10	10	10
Sub-block size	2.5	2.5	2.5
No. of Blocks (Parent)	92	52	75

Source: OceanaGold, 2022

Palomino Estimation Methodology

Examination of the data suggested a broad bimodal distribution of grade within the 0.8 g/t Au indicator shell, (see Figure 14-20), the log-probability plot shows the two grade populations, above and below a 0.3 g/t Au indicator (grey area shows population below 0.3 g/t Au).



Source: OceanaGold, 2022

Figure 14-20 Log-probability Plot of 3 m Composite Data within 0.8 g/t Indicator Shell

A probabilty kriging methodology was used to separate the higher (HG) and lower grade (LG) portions (grade and probability) for estimation into the parent block. The two estimates were then weight-averaged for whole block grade estimates. This was based on the probability (proportion) of the high- and low-grade domains within the block, where:

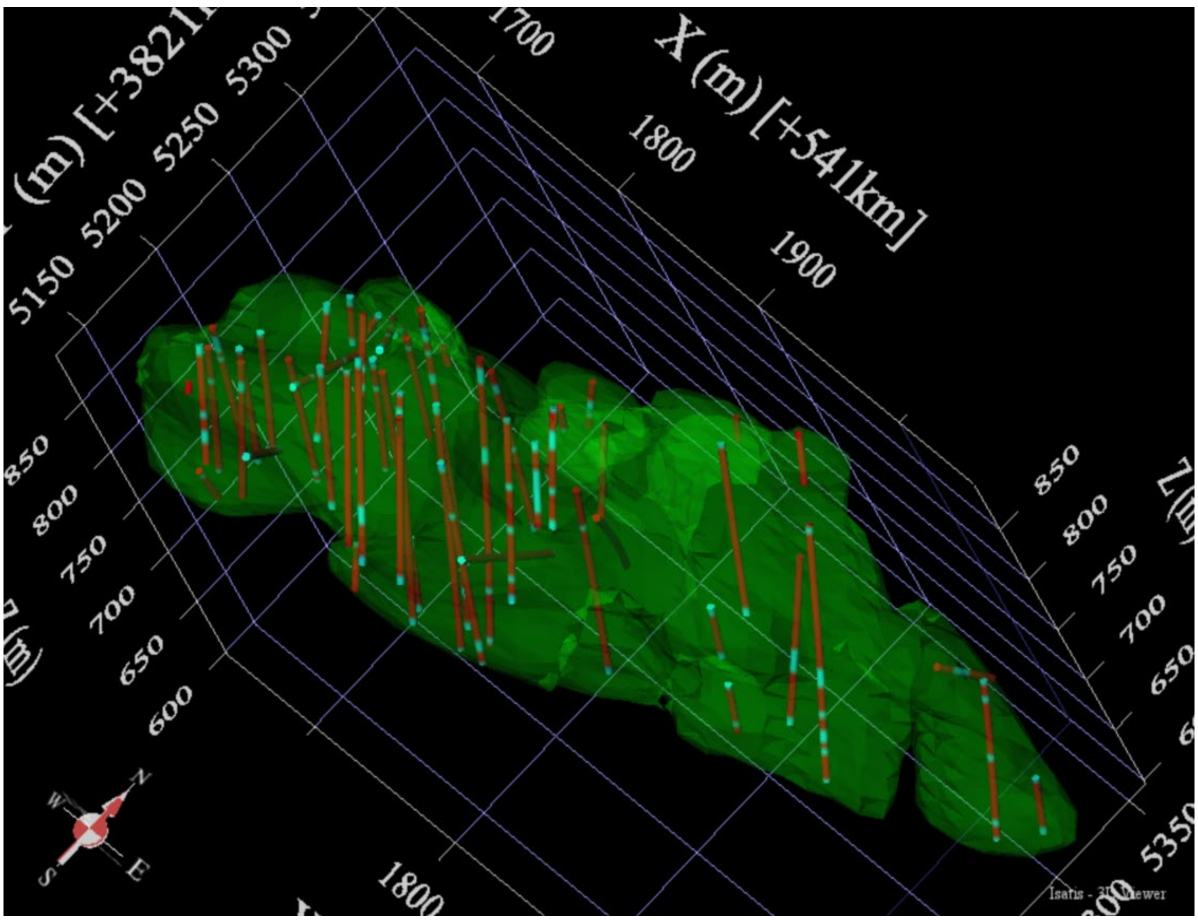
Block grade = ((Proportion HG x Grade HG) + (Proportion LG x Grade LG))

Statistics of the data within the sub-set HG and LG domains, based on the 0.3 g/t Au indicator, is shown in Table 14-20. The distribution of the HG and LG data is shown spatially in Figure 14-21.

Table 14-20: Statistics of Top-Cut 3 m Composite Data within 0.8g/t Au Domain – High / Low Grade Domains

Variable	Domain	Probability (% data)	Count	Minimum	Maximum	Mean	Variance	Variat.Coef.
AU_CUT	LG_ind_0.3=0	0.1515	198	0.003	0.29	0.14	0.01	0.60
AU_CUT	HG_ind_0.3=1	0.8485	1109	0.302	25.00	2.83	13.98	1.32

Source: OceanaGold, 2022



Source: OceanaGold, 2022

Figure 14-21 Distribution of High / Low Domain (0.3 g/t Indicator) Data within 0.8 g/t Indicator Shell (HG domain – red / LG domain – blue) – Perspective View Looking NW

Gold grades were estimated into 10 m E x 10 m N x 10 m RL parent blocks with Vulcan™ modeling software using Ordinary Kriging on 3 m composites. Sub-blocking was to 2.5 m E x 2.5 m N x 2.5 m RL for better volumetric determination, estimation was into the parent block.

Metasediment/metavolcanic contacts were not used to constrain gold estimation. Post-mineralization dykes were assigned zero grade.

The following methodology was used:

- Build a variogram for the 0.3 g/t Au Indicator for data within the 0.8g/t mineralized shell
- Search orientation essentially parallel to the plane of gold continuity
- Estimate LG indicator probability (LG_ind_pr). Calculate HG indicator probability (HG_ind_pr), where HG_ind_pr = 1 – LG_ind_pr
- Build a variogram for the Au grade in the LG and HG domains
- Estimate Au grade for HG and LG Domain
- Limit data to five samples per DH
- No octant restriction applied
- Post estimation – calculate final block grade where, Block grade = ((Proportion HG x Grade HG) + (Proportion LG x Grade LG))

The Ordinary Kriging parameters used and estimation search strategy used are summarized in Table 14-21 and Table 14-22 respectively.

Table 14-21: Palomino Ordinary Kriging Parameters

Estimation Domain	Variogram Structure	Nugget	Sill Differential	Rotations (Vulcan, Bearing, Plunge, Dip)	Ranges (m) (M, SM, Min)
LG_ind_Pr	1st Spherical 2nd Spherical 3rd Spherical	0.266	0.359 0.180 0.195	69,8°, -33.8°, -53° 69,8°, -33.8°, -53° 69,8°, -33.8°, -53°	12,7,6 70,65,11 260,70,22
HG Au Grade	1st Spherical 2nd Spherical 3rd Spherical	0.313	0.284 0.323 0.079	69,8°, -33.8°, -53° 69,8°, -33.8°, -53° 69,8°, -33.8°, -53°	8,5,5 30,20,15 100,60,35
LG Au Grade	1st Spherical 2nd Spherical	0.462	0.066 0.472	Isotropic	18,18,18 40,40,40

Source: OceanaGold, 2022

Table 14-22: LG Domain Probability and Au Grade Estimation Search Distances

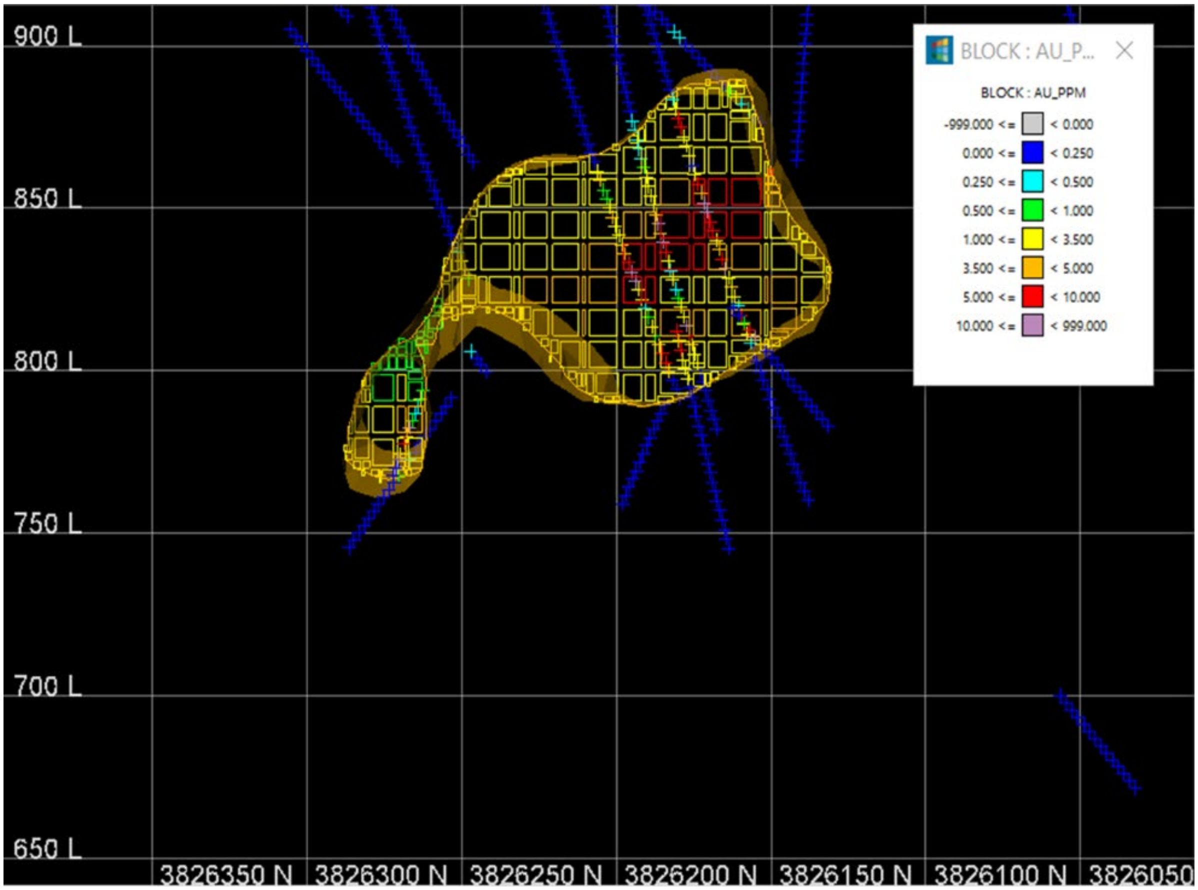
Estimation Domain	Estimation Pass	Search Range (m) (M, SM, Min)
LG_ind_Pr	1	250,250,250
HG Au Grade	1	100,100,100
	2	150,150,150
LG Au Grade	1	40,40,40
	2	120,120,120

Source: OceanaGold, 2022

Palomino Model Validation

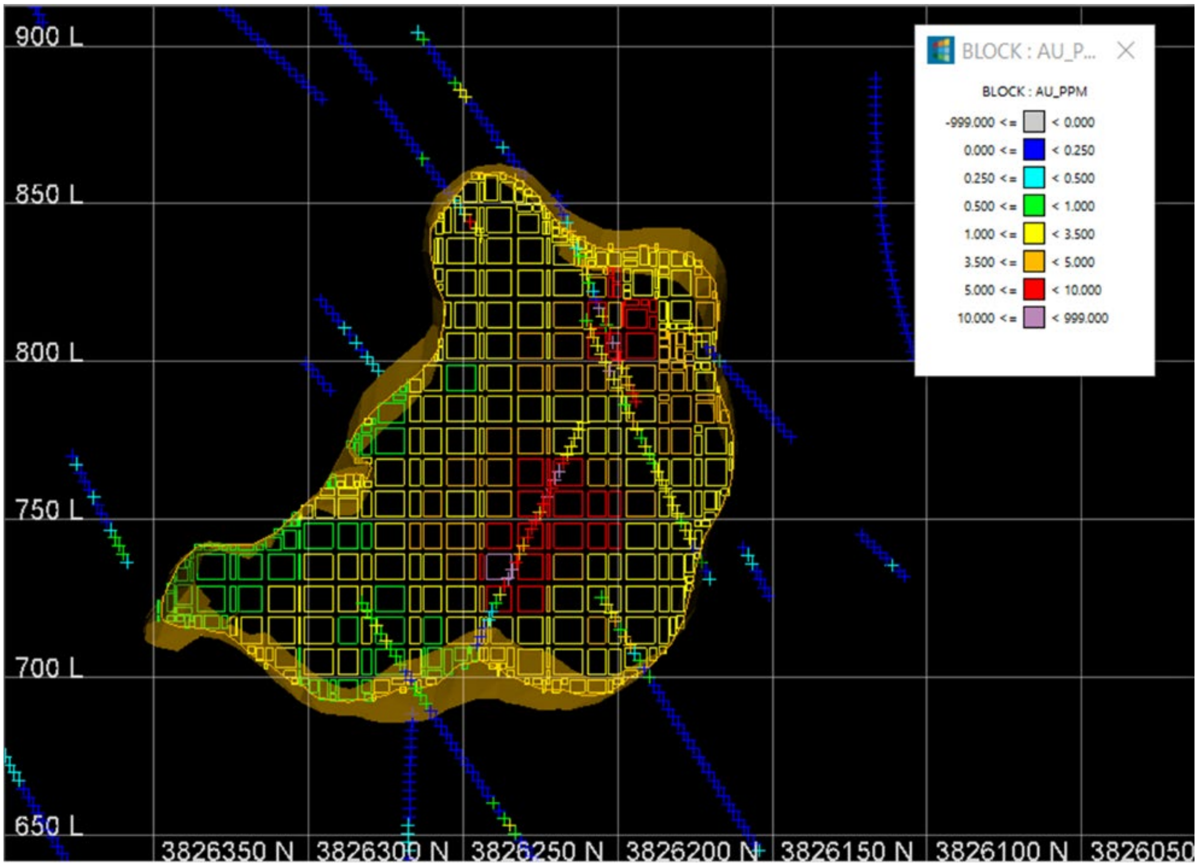
Several techniques were used to evaluate the validity of the block model. QA/QC analysis was performed on drilling data as per database procedures, and visual validation of the supporting drill data Au grades and estimated final block Au grades was performed, (Figure 14-22 and Figure 14-23). The methodology used for the resource modeling was reviewed, to ensure industry standard processes and assumptions were used. A review of all macros used in the estimation process was performed, to ensure all appropriate files were used, and correct naming conventions were followed.

Model estimation parameters were reviewed to evaluate the performance of the model with respect to supporting data. This included Kriging Neighbourhood Analysis, the number of composites used, number of drillholes used, average distance to samples used, and the number of blocks estimated in each pass.



Source: OceanaGold, 2022

Figure 14-22 Section at 542,690 m E (+/- 10m) showing Final Block Au Grade, with 3 m Topcut Composite Drillhole Data



Source: OceanaGold, 2022

Figure 14-23 Section at 542,810mE (+/- 10m) showing Final Block Au Grade, with 3 m Topcut Composite Drillhole Data

Global comparison of the 3 m topcut drill data (with an appropriate declustering weighting applied of 30 m x 30 m x 30 m), was compared to the final calculated block grade (block volume weighted) within the mineralized 0.8 g/t grade shell. This shows good correlation as shown in Table 14-23.

Table 14-23 Comparison Global Statistics – Block Model (vol weighted) vs. 3 m topcut DH (30mx30mx30m declustered weighting) within 0.8 g/t Grade Shell

Variable	Count	Minimum	Maximum	Mean	Variance	Variat.Coef.
Final BM Grade (vol Weighted)	14,657	0	10.53	**2.17**	1.83	0.62
DH 3m Topcut (declustered weighted)	1,307	0.003	25.00	**2.14**	10.98	1.55

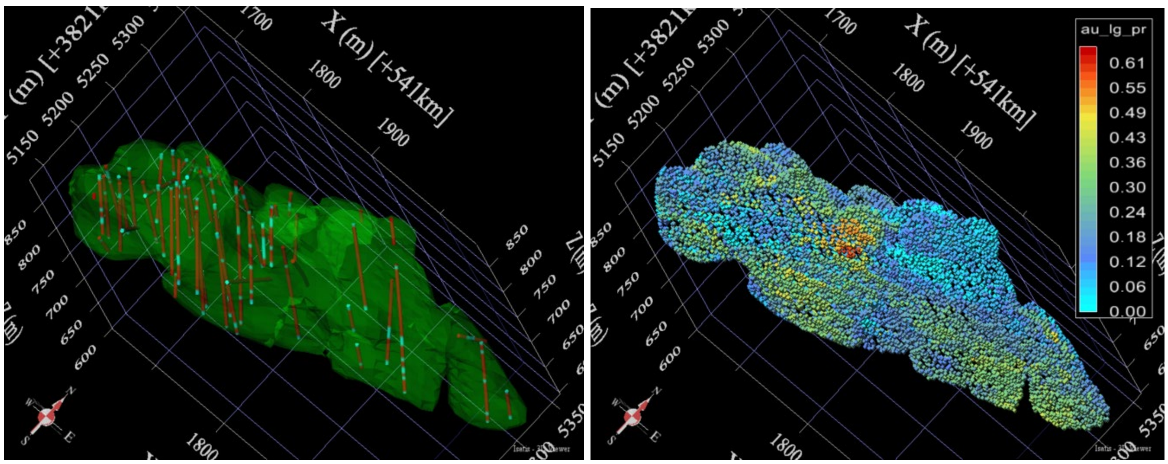
Source: OceanaGold, 2022

Validation was also completed on the estimated LG_ind_PR and calculated HG_ind_PR within the final blockmodel. Global comparison of the block model probabilities estimated (Table 14-24) showed a close correlation to the input drillhole data (Table 14-20). The spatial distribution of the estimated LG_ind_PR values were also compared (Figure 14-24). Areas of consistent HG indicators within the drill data, show up as cooler colours in the estimated model, which confirm a correlation with the data inputs.

Table 14-24 Global Statistics – LG Indicator Probability and HG Indicator Probability Estimated within Block Model

Variable	Count	Minimum	Maximum	Mean	Variance	Variat.Coef.
HG_ind_Pr	14,657	0.380	1.000	**0.8493**	0.121	0.143
LG_ind_Pr	14,657	0.000	0.620	**0.1507**	0.121	0.804

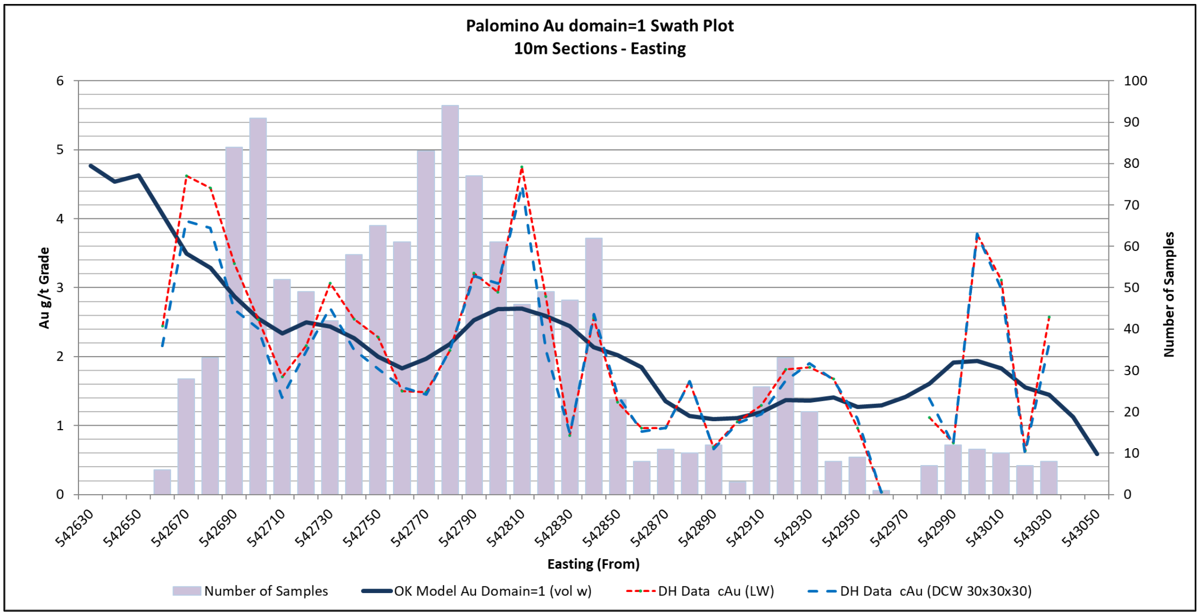
Source: OceanaGold, 2022



Source: OceanaGold, 2022

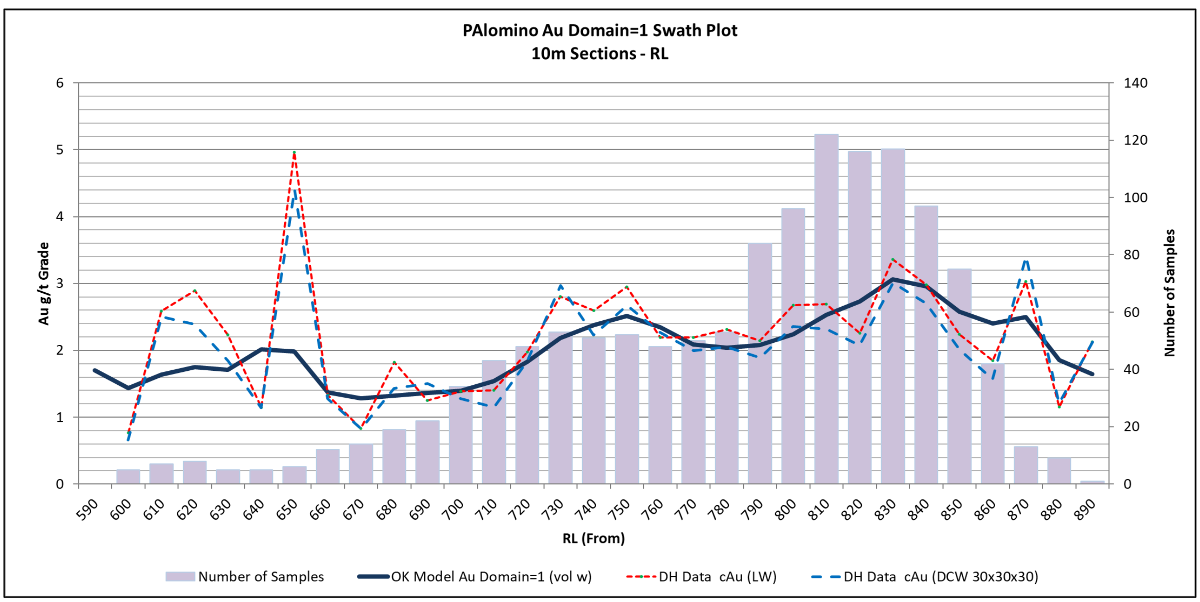
Figure 14-24 Distribution of LG Indicators (blue) within Drillhole Data – LEFT / Spatial Distribution of LG_ind_Pr Estimated within Block Model – RIGHT, Perspective View Looking NW

Swath plots were used to compare the estimation with underlying topcut composite grades for each domain. Figure 14-25 and Figure 14-26 show an acceptable local correlation between the composites and the block estimation grade for the mineralized domain.



Source: OceanaGold, 2022

Figure 14-25: 0.8 g/t Domain - Easting Swath Plot - Final Block Grade vs. DH 3m Topcut (length and declustered weighted)



Source: OceanaGold, 2022

Figure 14-26: 0.8 g/t Domain - RL Swath Plot - Final Block Grade vs. DH 3m Topcut (length and declustered weighted)

Model Reviews

Internal

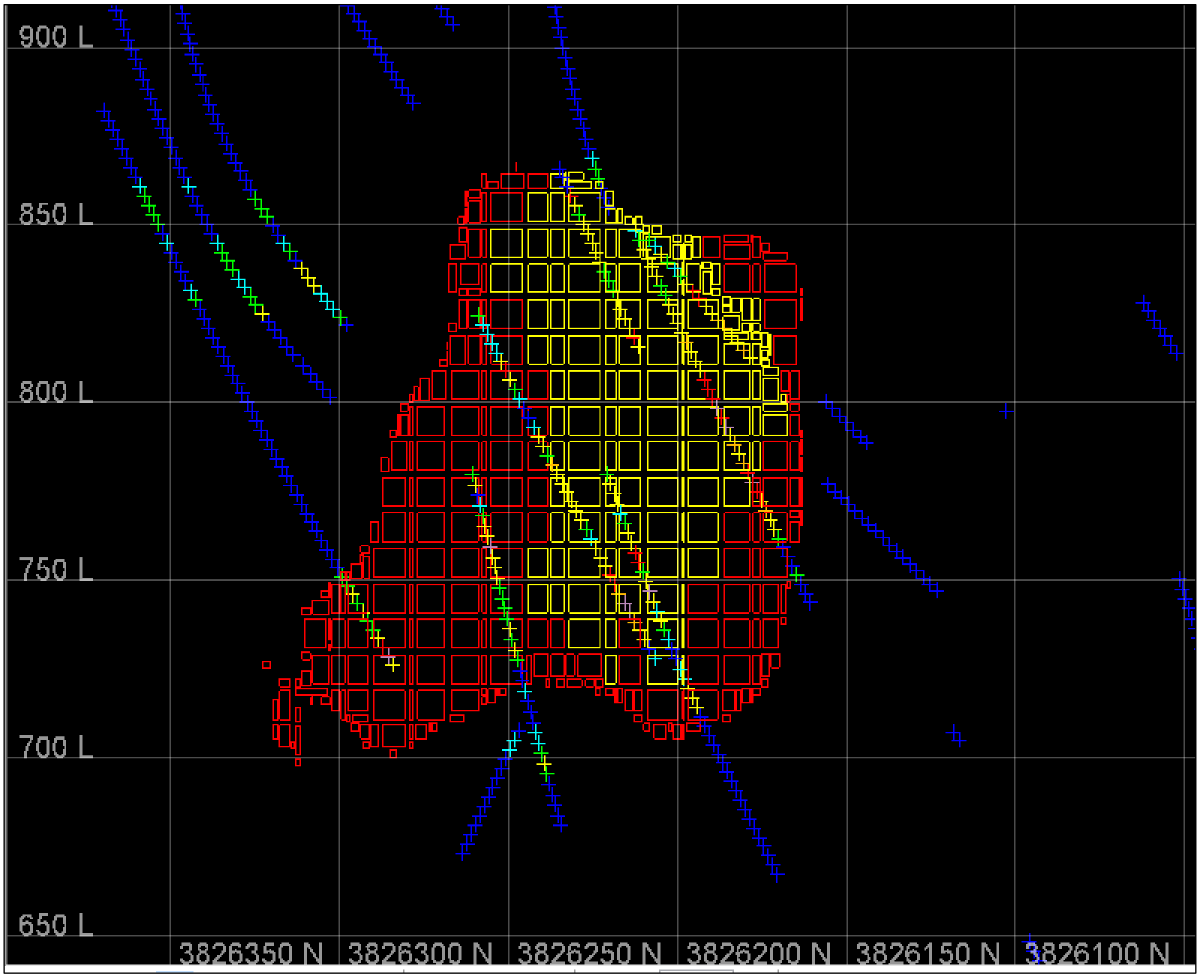
- Original model version was reviewed independently within the corporate team. Summary of findings:
    - The original version had a stronger anisotropy along the direction of the mineralized domain, giving a higher continuity in this direction. It was decided to make the sample search isotropic in the final estimate to account for other possible controls on mineralization.

The results of the model validations support a robust estimation.

Palomino Resource Classification

Mineral Resources are classified as Indicated and Inferred in accordance with CIM guidelines. There are no Measured Resources. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based primarily on the sample spacing and geological complexity. No single factor controls the Mineral Resource classification, rather each factor influences the result.

A wireframe solid was constructed around the areas where the majority of the blocks were estimated in the first pass of the estimation, and the mean distance of the samples used to estimate the blocks was 25 m or less. This was was also tested on other estimation parameters such as slope of regression, to confirm a consistent approach. This wireframe solid was used to assign the Indicated Mineral Resource classification. All blocks outside of the Indicated wireframes were classified as Inferred Mineral Resources. No resource was estimated outside of the 0.8 g/t implicit grade shell. 14-27 shows a representative cross section of the final resource classification.



Source: OceanaGold, 2022Figure 14-27: Representative Cross Section 543,790mE looking E - Showing Resource Classification (Yellow -Indicated / Red - Inferred)

Palomino Mineral Resource Statement

The Palomino Mineral Resource statement is presented in Table 14-25. The reported resource is constrained within a conceptual stope design based on a gold price of US$1,700/oz, approximating a 1.39 g/t cut-off. All unclassified material within the conceptual design was assigned zero grade for the purposes of reporting.

- The Mineral Resources reported for the Palomino deposit are classified as Indicated and Inferred Mineral Resources, based primarily on drillhole spacing and geological understanding. All blocks outside of the 0.8 g/t Implicit grade shell remain unclassified.

The Palomino Mineral Resource statement is based on the Ordinary Kriging (OK) model. It is constrained within a conceptual stope design using a CoG of 1.39 g/t Au, which assumes underground mining methods and is based on a mining cost of US$45/t, milling cost of US$13.94/t, administration cost of US$5.70/t, based on a gold price of US$1,700/oz, and a gold recovery of 85%.

Table 14-25: Palomino Underground Mineral Resource Statement as of December 31, 2021

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)
Measured	0.0	0.0	0.0
Indicated	2.3	2.79	0.2
Measured & Indicated	**2.3**	**2.79**	**0.2**
Inferred	3.6	2.3	0.3

Source: OceanaGold, 2022

- Cut-off grade 1.39 g/t Au based on a gold price of US$1,700/oz.
- Constrained within a conceptual stope design.
- Dilution is included due to the difusse grade boundaries.
- Mineral Resources are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

14.3 Open Pit, Stockpile and Underground Combined Mineral Resource Statement

Table 14-26 presents the combined open pit, stockpile, and underground resource statement for Haile.

Table 14-26: Haile Combined Open Pit, Stockpile and Underground Resource Statement December 31, 2021

Type	Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Open Pit	Measured	2.68	1.30	0.11	2.54	0.22
	Indicated	43.0	1.55	2.14	2.41	3.33
	Measured & Indicated	**45.6**	**1.54**	**2.25**	**2.42**	**3.55**
	Inferred	5.7	1.0	0.2	1.3	0.2
Stockpiles	Measured	1.83	1.10	0.06	1.10	0.06
	Indicated	0.0	0.00	0.0		
	Measured & Indicated	**1.83**	**1.10**	**0.06**	**1.10**	**0.06**
	Inferred	0.0	0.0	0.0		
Underground	Measured	0.0	0.0	0.0		
	Indicated	5.48	4.12	0.73		
	Measured & Indicated	**5.48**	**4.12**	**0.73**		
	Inferred	5.6	3.1	0.6		
Combined	Measured	4.51	1.22	0.18		0.28
	Indicated	48.5	1.84	2.87		3.33
	Measured & Indicated	**52.9**	**1.79**	**3.04**		**3.61**
	Inferred	11	2.0	0.7		0.2

Source: OceanaGold, 2022

- Cut-off grades for the open pit, Horseshoe underground, and Palomino underground are 0.45 g/t / 0.55 (primary / oxide), 1.35 g/t and 1.39 g/t Au respectively, based on a gold price of US$1,700/oz.
- No cut-off applied to reported mined stockpiles
- Open pit resource is reported within a US$1,700/oz optimized shell. Palomino is constrained within a conceptual stope design and Horseshoe underground is spatially constrained within the 1 g/t Indicator shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- There is no certainty that Mineral Resources that are not Mineral Reserves will be converted to Mineral Reserves.
- All figures are rounded to reflect the relative accuracy and confidence of the estimates and totals may not add correctly.
- The underground Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

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

14.4 Relevant Factors

At this time, there are no unique situations in relation to environmental, socio-economic or other relevant conditions that would put the Haile Mineral Resource at a higher level of risk than any other developing resource within the United States, or that would materially affect the Mineral Resource estimates. Whilst there may be delays in receiving the permits, such delays are not expected to materially impact the reported resource estimates.

15 Mineral Reserve Estimate

Separate Mineral Reserve estimates were generated for the open pit and underground mines. A combined Mineral Reserve statement is provided in Section 15.3. The open pit and underground mining areas are located entirely on land owned by HGM. There are no royalties.

The open pit and underground work was completed using the site coordinate system. This is based on UTM NAD83 with a plus 1,000 m adjustment to elevation.

15.1 Open Pit Mineral Reserve Estimate

Open pit LoM plans and resulting open pit Mineral Reserves are determined based on a gold price of US$1,500/oz Au. Reserves stated in this report are dated effective as of December 31, 2021.

The ore material is converted from Mineral Resource to Mineral Reserve based primarily on positive cash flow pit optimization results, pit design, and geological classification of Measured and Indicated resources. The in situ value is derived from the estimated grade and certain modifying factors.

The open pit reserve consists of several pit areas. Material is truck hauled from the pits to an existing crusher/processing facility. Waste material is categorized, and truck hauled to the appropriate waste rock stockpile location.

15.1.1 Introduction

The geological model was used for open-pit optimization without modification, as the block size in the model matched the SMU size of 10 m x 10 m x 5 m. This block size is currently considered appropriate for the backhoe excavator loading units operating at Haile.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results (Lerch-Grossman algorithm). The optimization included Measured, Indicated and Inferred Mineral Resource categories with a gold price of US$1,500/oz Au. Silver was not assigned economic value in the optimization. Subsequent to pit optimization, inferred material (approximately 10% by volume) within the reserve pit was treated as waste and given a zero-gold grade. Whittle optimization parameters were derived by OceanaGold and are shown in

Dilution and ore recovery have been applied during the mine scheduling process to account for mineralized material mined by Face Shovel excavators. This has limited impact on the Mineral Reserve, with an effective global dilution of 1% and ore recovery of 98.9%. This is discussed further in Section 16.1.6.

Table 15-1.

The overall pit slopes (inter-ramp angle slopes) used for the design are based on operational level geotechnical studies and range from 32° to 45°. This includes a 5° allowance for ramps and geotechnical catch benches.

Dilution and ore recovery have been applied during the mine scheduling process to account for mineralized material mined by Face Shovel excavators. This has limited impact on the Mineral Reserve, with an effective global dilution of 1% and ore recovery of 98.9%. This is discussed further in Section 16.1.6.

Table 15-1: Pit Optimization Parameters

Parameter	Unit	Value
Base Mining Cost	US$/t	1.81
Incremental Mining Cost	US$/t / 5 m bench	0.01
PAG Rehabilitation Cost	US$/t PAG waste	0.65
Processing Cost	US$/t ore	11.50
G&A Cost	US$/t ore	5.00
Ore Rehandle Cost	US$/t ore	0.70
TSF Expansion	US $/t ore	2.44
Gold Recovery	%	(1-(0.2152*Au grade^-0.3696))+0.025
Mill Throughput	Mtpa	3.8[1]
Gold Price	US$/oz	1,500
Gold Refining & Selling Cost	US$/oz	3.00
Calculated Au Cut-off Grade	US $/t	0.5
Royalties	%	0.0
Discount Rate	%	5.0

[1] Actual throughput is prorated at the average of 3.8 Mtpa for open-pit ore and 3.2 Mtpa for underground ore
Source: OceanaGold, 2022

A 3D mine design, based on the selected Whittle pit, was completed using Vulcan software and is the basis for the open pit reserves.

15.1.2 Conversion Assumptions, Parameters and Methods

The conversion of Mineral Resource to Mineral Reserve entails the evaluation of modifying factors that should be considered in stating a Mineral Reserve. Table 15-2 shows a reserve checklist and associated commentary on the risk factors involved for the Haile Open Pit Reserve statement.

Table 15-2: Haile Open Pit Reserve Checklist

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Mining				
Mining Width	X			Minimum 100 m
Open Pit and/or Underground	X			Open pit and underground
Density and Bulk handling	X			Density and swell considered
Dilution	X			SMU 10 m x 10 m x 5 m
Mine Recovery	X			Full mine recovery assumed
Waste Rock	X			NAG/PAG waste dumps
Grade Control	X			Operating mine – blast chips
Processing	X			Operating mine
Representative Sample	X			Previous feasibility study and operating mine
Product Recoveries	X			Feasibility study - operating
Hardness (Grindability)	X			Feasibility study - operating
Bulk Density	X			Feasibility study - operating
Deleterious Elements	X			Feasibility study - operating
Process Selection	X			Feasibility study - operating
Geotechnical/Hydrological	X			Feasibility study – operating
Slope Stability (Open Pit)	X			Feasibility study, periodic operational reviews.
Water Balance	X			Full site water balance
Area Hydrology	X			Hydrology considered
Seismic Risk	X			Low
Environmental				
Baseline Studies	X			Operating Mine
Tailing Management	X			Operating Mine
Waste Rock Management	X			Operating Mine – Overburden Management Plan (OMP).
ARD Issues	X			Lined waste facilities
Closure and Reclamation Plan	X			2020 Reclamation Plan.
Permitting Schedule	X			Ongoing – reasonable expectation of success
Location and Infrastructure				
Climate	X			High rainfall events
Supply Logistics	X			Operating Mine
Power Source(S)	X			Operating Mine
Existing Infrastructure	X			Operating Mine
Labor Supply and Skill Level	X			Operating Mine
Marketing Elements or Factors				
Product Specification and Demand	X			Gold Market
Off-site Treatment Terms and Costs	X			
Transportation Costs	X			Low
Legal Elements or Factors		X		
Security of Tenure	X			Operating Mine
Ownership Rights and Interests	X			Operating Mine
Environmental Liability	X			ARD potential
Political Risk (e.g., land claims, sovereign risk)	X			Low political risk - USA
Negotiated Fiscal Regime		X		
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			Operating Mine

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
Commodity Price Forecasts	X			US$1,500/oz Au
Foreign Exchange Forecasts			X	
Inflation	X			Small
Royalty Commitments	X			No royalty
Taxes	X			Operating Mine
Corporative Investment Criteria		X		
Social Issues				
Sustainable Development Strategy	X			Environmental Impact Statement (EIS)
Impact Assessment and Mitigation	X			EIS
Negotiated Cost/Benefit Agreement	X			EIS
Cultural and Social Influences	X			EIS

Source: SRK, 2020, Updated OceanaGold 2022

15.1.3 Reserve Estimate

Mineral Reserves were classified using the 2014 CIM Definition standards. Measured Mineral Resources were converted to Proven Mineral Reserves, and Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining pit shapes created during the mine design process.

The open pit mine design process results in open pit mining reserves of 42.0 Mt with an average gold grade of 1.58 g/t. The Mineral Reserve statement, as of December 31, 2021 for the Haile Open Pit is presented in Table 15-3.

Table 15-3: Haile Open Pit Mineral Reserves Estimate as of December 31, 2021

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	4.4	1.26	1.98	0.18	0.28
Probable*	37.6	1.62	2.44	1.96	2.95
Proven + Probable	42.0	1.58	2.39	2.14	3.23

* Includes 1.8 Mt of stockpile material grading 1.1 g/t Au and 1.1 g/t Ag
Source: OceanaGold, 2022

- Reserves are based on a US$1,500/oz Au gold price.
- Open pit reserves are stated using a 0.5 g/t Au cut-off for primary and 0.6 g/t Au cut-off for oxide material.
- Open pit reserves include variable dilution and mining recovery that has been applied in the mine schedule to the upper benches of each pit stage to account for assumed mining by face shovel excavator in these areas.
- Metallurgical recoveries are based on a recovery curve for primary material of (1-(0.2152*Au grade^-0.3696)). with +2.5% uplift applied to material > 1.7 g/t Au. Recovery for oxide material is applied at 67%. This equates to an overall average recovery of 81%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- The open pit Mineral Reserves were estimated by Gregory Hollett P.Eng (EGBC) of OceanaGold, a Qualified Person.

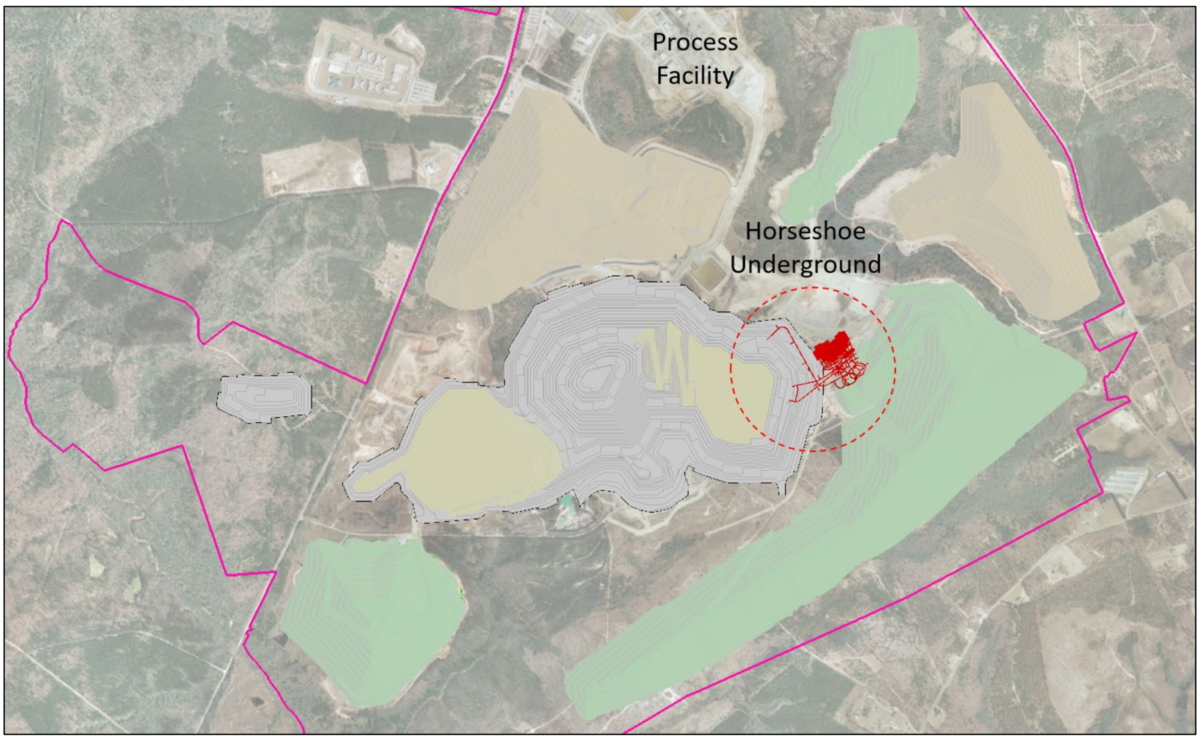
Relevant Factors

OceanaGold knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the Mineral Reserve estimate. OceanaGold knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the open pit Mineral Reserve estimate. While there may be delays in receiving the permits, such delays are not expected to materially impact the reported reserve estimates."

15.2 Underground Mineral Reserve Estimate

15.2.1 Introduction

Mineral Resources extend below and outside of the existing open pit mine. A portion of these Mineral Resources will not be mined by the ultimate pit shell that is described in this feasibility study and therefore have been evaluated for potential underground mining. The Mineral Resource area evaluated for underground mining is referred to as "Horseshoe". Horseshoe is located to the northeast of the Snake Pit, as shown in Figure 15-1. Note that the underground reserve does not consider the Palomino underground resource at this time.



Source: OceanaGold, 2022

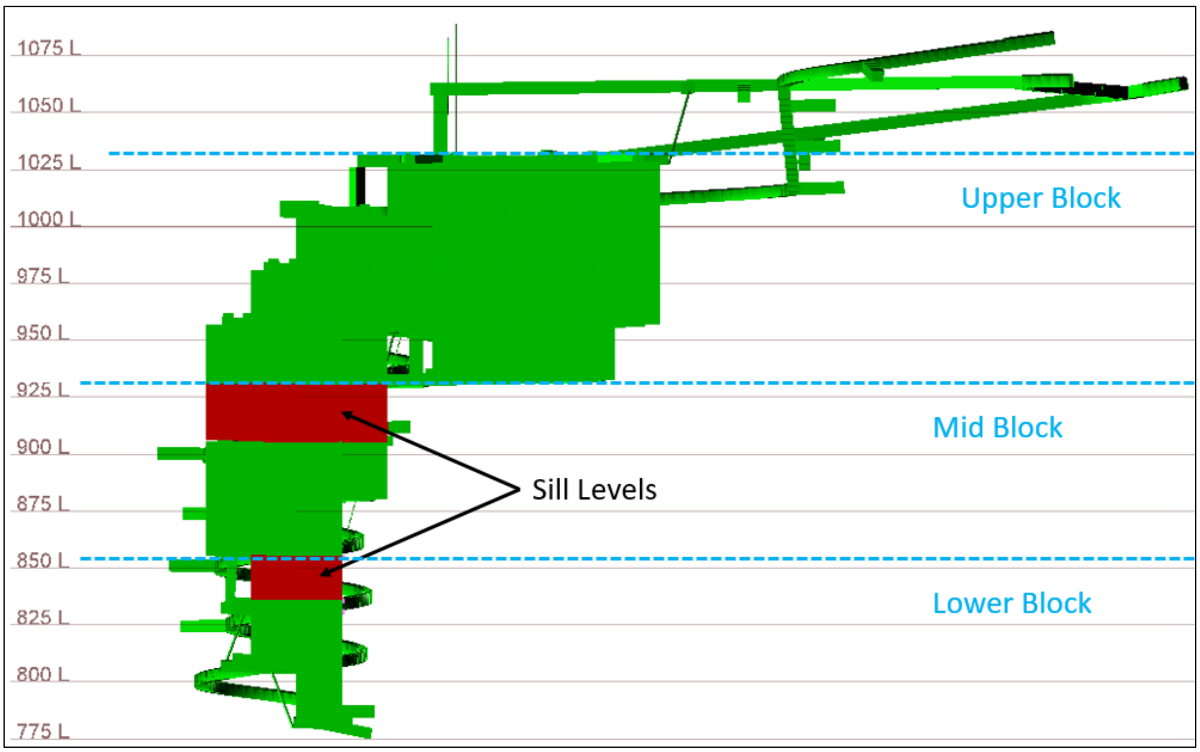
Figure 15-1: General Site Layout and Location of the UG Reserve Area (in Red)

15.2.2 Conversion Assumptions, Parameters and Methods

Measured and Indicated Mineral Resources were converted to Proven and Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining block shapes created during the mine design process.

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with ramp access is used. The stopes will be 20 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. Cemented rock fill (CRF) will be used to backfill the stopes. There will be an opportunity for some non-cemented waste rock to be used in select stopes based on the mining sequence. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes.

The deposit has been divided into three production areas as shown in Figure 15-2. Using the local grid, the uppermost block extends from approximately the 930 m elevation to 1,030 m elevation and includes four stoping levels that will be mined bottom up. The mid-block extends from approximately the 850 m elevation to the 930 m elevation and includes three stoping levels that will be mined bottom up. The uppermost level in this block includes mining out of the sill. The lowest block extends from approximately the 780 m elevation to the 850 m elevation. It includes three stoping levels that are also mined bottom up. The uppermost level in this block also includes mining out of the sill. A lower extraction has been used for sill levels.



Source: SRK, 2022

Figure 15-2: UG Design Sill Levels (Local Grid)

A detailed design was completed including re-mucks, passing bays, etc. All Mineral Reserve tonnages are expressed as "dry" tonnes (i.e., no moisture) and are based on the density values stored in the block model. Inferred Mineral Resources are not included in the mine plan. Mining dilution and recovery have been applied to the reserves using the methodologies described in the following sections.

<u>Dilution</u>

The mining dilution estimate is based on the equivalent linear overbreak/slough methodology (ELOS; Clark, 1997). ELOS is an empirical design method that is used to estimate the amount of overbreak/slough that will occur in an underground opening based on rock quality and the hydraulic radius of the opening.

Dilution estimates were applied differently for primary and secondary stopes as follows:

- For a typical primary stope, the sources of dilution are in the floor (CRF backfill) and in the front endwall of the stope (CRF backfill). Dilution from the sidewalls and the back endwall is not included, as this material is typically ore and is already accounted for within the volumes of adjacent secondary stopes.
- For a typical secondary stope, the sources of dilution are in the floor (CRF backfill), in the front endwall of the stope (CRF backfill), and in the sidewalls of the stope (CRF backfill).
- For the sill pillar primary and secondary stopes, an additional source of dilution is applied from the crown (CRF backfill). To account for more difficult expected mining conditions in the sill stopes, an additional dilution allowance has been included.

ELOS assumptions are shown in Table 15-4.

Table 15-4: Dilution ELOS Assumptions

Type	ELOS Value (m)
Sidewalls (backfill)	0.40
Endwall (backfill)	0.15
Floor (backfill)	0.10
Crown (backfill)	0.50

Source: OceanaGold, 2022

The rock sidewall/endwall dilution material will contain low-grade mineralization. However, a conservative approach was adopted by applying zero grade to all rock dilution. Zero grade was applied to CRF backfill dilution. The ELOS and additional dilution factor for the sill stopes results in the dilution factors shown in Table 15-5. These factors were conservatively applied uniformly across each stope type.

Table 15-5: Mine Design Dilution Factors

Stope Type	Dilution Applied (at Zero Grade) (%)
Primary Stopes	2
Secondary Stopes	6
Sill Pillar Primary Stopes	6
Sill Pillar Secondary Stopes	10

Source: OceanaGold, 2022

For all horizontal development, dilution of 10% was applied at zero grade.

Recovery

A stope recovery factor of 94% was used. For sill levels, a 75% recovery factor was used to account for material left in situ in the sill. The following items were used to calculate this factor:

- Material loss into backfill (floor) of 0.15 m
- Material loss to side and endwalls (under blast) of 0.15 m
- Material loss from leaving wing-shaped pillars in stope crowns (for stope stability and to enable tight-filling of stopes)
- Material loss to mucking along the sides and in blind corners of the stopes
- Additional loss factor due to rockfalls, misdirected loads, and other geotechnical reasons

A development recovery factor of 100% was used for all horizontal development. Recoveries of the temporary sill levels have been reduced by 25%, to reflect room and pillar mining of the sill pillars.

15.2.3 Reserve Estimate

Mineral Reserves were classified using the 2014 CIM Definition standards. Indicated Mineral Resources were converted to Probable Mineral Reserves by applying the appropriate modifying factors, as described herein, to potential mining shapes created during the mine design process.

The underground mine design process resulted in underground mining reserves of 3.4 Mt (diluted) with an average grade of 3.78 g/t Au. The Mineral Reserve statement, as of December 31, 2021, for the Haile Horseshoe Underground is presented in Table 15-6.

Table 15-6: Haile Horseshoe Underground Reserves Estimate as of December 31, 2021

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	-	-	-	-	-
Probable	3.42	3.78	-	0.42	-
Proven + Probable	3.42	3.78	-	0.42	-

Source: SRK

- Reserves are based on a gold price of US$1,500/oz. Metallurgical recoveries are based on a recovery (1-(0.2152*Au grade^-0.3696)) + 0.025 that equates to an overall recovery of 88%.
- Underground reserves are stated using a 1.53 g/t Au cut-off. The reserve estimate is based on a mine design using an elevated cut-off grade of 1.67 Au g/t, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.37 g/t Au and an incremental development cut-off grade of 0.46 g/t Au.
- Mining recovery ranges from 94% to 100% depending on activity type. Sill levels use a 75% recovery. Mining dilution is applied using zero grade. The dilution ranges from 2% to 10% depending on activity type.
- Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP of SRK, a Qualified Person.

<u>Relevant Factors</u>

SRK knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors which could materially affect the underground Mineral Reserve estimate. While there may be delays in receiving the permits, such delays are not expected to materially impact the reported reserve estimates.”

15.3 Open Pit and Underground Combined Reserves Statement

Table 15-7 presents the combined open pit and underground Mineral Reserves statement for Haile.

Table 15-7: Reserve Statement for OceanaGold's Haile Gold Mine as of December 31, 2021

Type	Category	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au Contained (Moz)	Ag Contained (Moz)
OP	Proven	4.4	1.26	1.98	0.18	0.28
	Probable*	37.6	1.62	2.44	1.96	2.95
	Proven + Probable	42.0	1.58	2.39	2.14	3.23
UG	Proven		-	-		-
	Probable	3.42	3.78	-	0.42	-
	Proven + Probable	3.42	3.78	-	0.42	-
OP + UG	Proven	4.4	1.3	2.0	0.2	0.3
	Probable	41.0	1.8	2.2	2.4	2.9
	Proven + Probable	45.4	1.8	2.2	2.6	3.2

*Includes1.8 Mt of stockpile material grading 1.1 g/t Au and 1.1 g/t Ag
Source: OceanaGold

- Mineral Reserves are based on a gold price of US$ 1,500/oz.
- Metallurgical recoveries are based on a recovery curve for primary material of (1-(0.2152*Au grade^-0.3696)) with +0.025 uplift applied to material > 1.7 g/t Au. Recovery for oxide material is applied at 67%. This equates to an overall recovery of 81% for the open pit material and 88% for the underground material.
- Open pit reserves are stated using a 0.5 g/t Au cut-off for primary and 0.6 g/t Au cut-off for oxide material. Open pit reserves include variable dilution and mining recovery that has been applied in the mine schedule to the upper benches of each pit stage to account for assumed mining by face shovel excavator in these areas.
- Open pit reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- Underground reserves are stated using a 1.53 g/t Au cut-off. The reserve estimate is based on a mine design using an elevated cut-off grade of 1.67 Au g/t, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.37 g/t Au and an incremental development cut-off grade of 0.46 g/t Au. Mining recovery ranges from 94% to 100% depending on activity type. Sill levels use a 75% recovery. Mining dilution is applied using zero grade. The dilution ranges from 2% to 10% depending on activity type.
- Mineral Reserves are inclusive of Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimates. Totals may not sum due to rounding.
- Mineral Reserves have been stated on the basis of a mine design, mine plan, and cash-flow model.
- The open pit Mineral Reserves were estimated by Gregory Hollett P.Eng (EGBC) of OceanaGold, a Qualified Person. The underground Mineral Reserves were estimated by Joanna Poeck, BEng Mining, SME-RM, MMSAQP #01387QP of SRK, a Qualified Person.

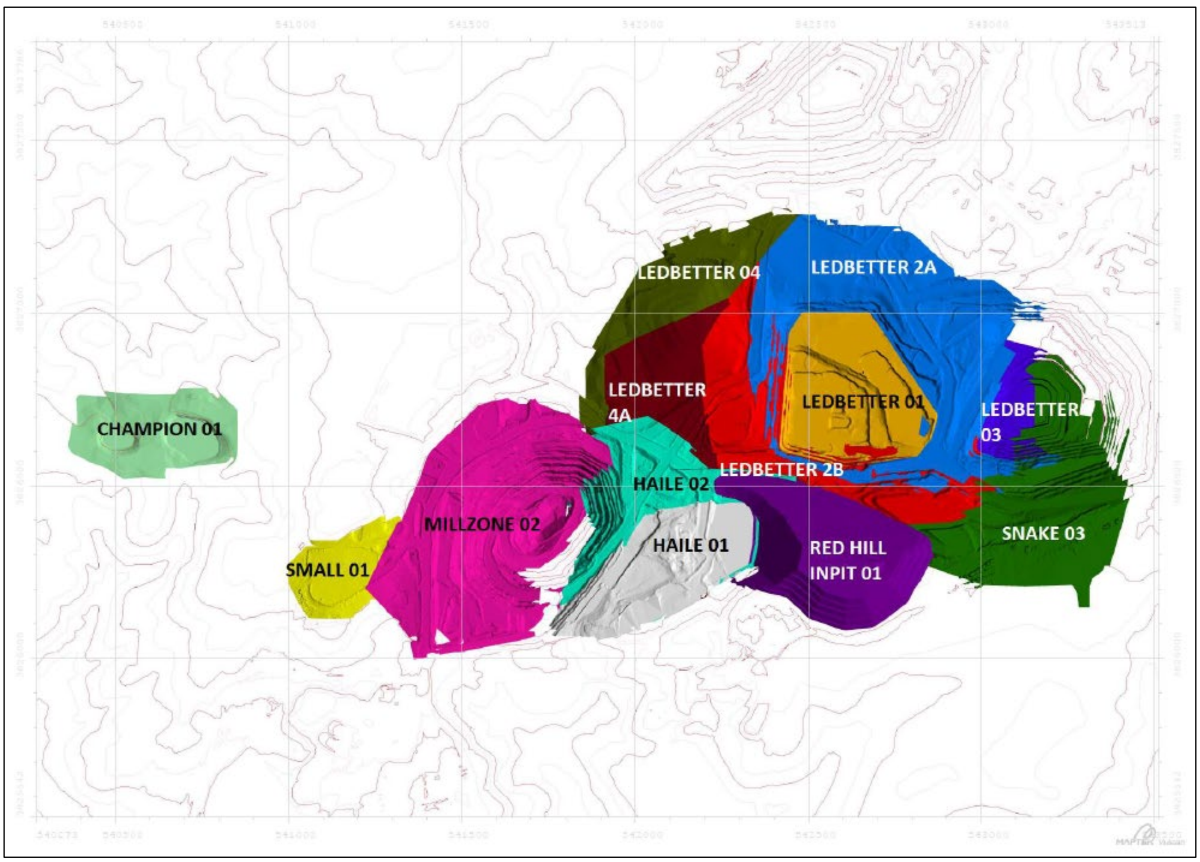
16 Mining Methods

Both open pit and underground mining methods will be used at Haile. As such, the following sections describe the open pit and underground mining methods separately. A combined open pit and underground production schedule is provided in Section 16.3.

16.1 Open Pit Mining Methods

OceanaGold staff completed the mine planning for the Haile open pit operations. The mine plans are valid from December 31, 2021 and are based on a surveyed topographical surface of the same date. The mine plan is intended to provide a practical approach to extracting the potential reserves from open pit operations and integrating the Horseshoe underground mine into the Haile LoM plan and reserve.

The primary pit names referenced in the sections to follow are based on historical naming conventions when many of the smaller gold pits did not merge into the large Haile open pit described in this report. As such, phases have been named to replicate the historical pit areas. Figure 16-1 illustrates the pit area names.



Source: OceanaGold, 2022

Figure 16-1: Haile Open Pit Naming Convention

16.1.1 Current or Proposed Mining Methods

Haile is currently being mined using conventional truck-and-excavator open-pit methods and OceanaGold plans to continue with current mining methods. The material encountered at Haile is a combination of soft (Costal Plains Sands [CPS] and saprolite) and hard (metavolcanics and metasediments) rock units.

CPS is loosely consolidated sand which can be mined without the need for drilling and blasting. Mineralization is not present in CPS thus drilling for the purposes of ore control and waste classification is not necessary. Saprolite is mined without blasting where possible. Saprolite is sampled for waste classification to meet the requirements in Haile's Overburden Management Plan (OMP).

Drilling and blasting are required in all hard rock. Drilling and blasting are performed on 10 m benches. Multiple bit sizes (115 mm, 171 mm, 200 mm) are used depending on material type and application. Blasthole depth is 10 m plus subdrill; subdrill ranges from 1.4 to 1.9 m.

The number of samples taken per blasthole is material-type dependent. Blastholes in waste are typically sampled once on a 10 m interval for NAG/PAG definition. Blastholes in ore are typically sampled three times at 3.3 m sample intervals.

Flitch height is variable. Waste is typically mined on a 10 m flitch and ore is typically mined on a 3.3 m flitch. Ore is usually mined with hydraulic backhoe excavators, while the majority of waste is mined with hydraulic shovels. Front-end loaders may be used in either application in back-up capacity. The haul truck fleet is a mix of 175 t and 140 t payload units.

16.1.2 Parameters Relevant to Mine or Pit Designs and Plans

Geotechnical

The Interramp Slope Angle (ISA) recommendations presented in Table 16-1 through Table 16-5 are based on a slope stability study performed by Call & Nicholas, Inc. for Haile Gold Mine. The design criteria for the ISA recommendations are a minimum Factor of Safety (FoS) of 1.20 for overall slopes, an 80% catch bench width reliability for 10 m high single benches, or a 90% catch bench width reliability for 20 m high double benches. Catch bench scale structural evaluations were performed using CNI's probabilistic bench-scale analytical method (backbreak), while the FoS for overall slopes were based on two-dimensional limit equilibrium analyses. The ISA recommendations are the highest achievable angles that meet all design criteria.

Data used for this study included the following:

- 2021 LoM Pit design (LTP21A_13_CP_01_V03_TOPO.00t)
- 3D geology block model based on drilling and pit mapping (developed by HGM)
- 3D geotechnical rock type (GTRCK) block model developed by CNI
- RQD Data from 967 drillholes (330,000 m)
- 3D RQD block model developed by CNI
- Structure data from 52 cells mapped within the Mill Zone pit between the 1070 and 1125 mine levels
- Structure data from 11 televiewer drillholes
- Structure data from eight oriented core holes
- A total of 20 small scale direct shear test performed on four different rock types

- 91 disc tension test performed on three different rock types
- 20 uniaxial compression tests performed on three different rock types
- 44 triaxial compression tests performed on 3 different rock types (Section 4.2.2)
- Slope Angle Evaluation for the Haile Gold Mine (CNI 2018)
- Design Angle Recommendations for the Haile Gold Mine (CNI 2022)
- 2017 pit slope study performed by BGC Engineering Inc. (BGC)
- An interpreted 3D phreatic surface (weathered Water Levels_EOM_Feb2018_linear.dxf) for the project area by HGM

The GTRCK block model consists of twelve geomechanical groups, based on geology, RQD, and material properties:

- Coastal Plain Sands (CPS)
- Sericite
- Saprolite (SAP)
- Metasediments
    - RQD ≤ 30%
    - 30% < RQD ≤ 60%
    - 60% < RQD
- Metavolcanics
    - RQD ≤ 30%
    - 30% < RQD ≤ 60%
    - 60% < RQD
- Diabase dikes
    - RQD ≤ 30%
    - 30% < RQD ≤ 60%
    - 60% < RQD

Material properties used in this analysis were either derived from a combination of laboratory testing, statistical regression, and RQD data, or were previously reported during earlier studies. Without additional laboratory testing available for the CPS and saprolite units, CNI used strength properties reported in the *Haile Gold Mine Optimization Study – Open Pit Slope Designs* report by BGC Engineering Inc. from July 2017. The material properties for the sericite unit (actually, a low plasticity silt), were derived from small scale direct shear test.

Linear rock mass properties were calculated for the metasediment, metavolcanic, and diabase rock types based on three RQD ranges: RQD ≤ 30%, 30% < RQD ≤ 60%, and 60% < RQD. The RQD ≤30% unit roughly correlates to the "Weathered" category from earlier studies, while the 30% <RQD ≤60% unit represents a transition zone between the "Weathered" and the higher RQD (60%<) "Fresh" materiel from the previous studies.

The shear strength of a rock mass is weakest along discontinuities. The orientation of discontinuities therefore defines the critical direction of shear strength anisotropy. At Haile, this direction is parallel to foliation and to a lesser extent, parallel to cross joint orientations. Anisotropic rock mass strengths were used for both the metasediment and metavolcanic rock types in the slope stability analysis. The rock mass properties are presented in Table 16-6.

The main geologic structures identified in the project area are:

- Regional northwest-dipping foliation, best developed in the metasedimentary rocks
- Southeast and southwest-dipping joints
- Sub-vertical or steeply dipping joints parallel to the north-northwest-striking diabase dikes
- Regional faults dipping northwest

For the probabilistic backbreak analysis, the property was separated into four geologic domains based on rock strength and structural orientation data. Due to spatial variations in the structure data, the Mill Zone Pit was separated from the other pits. These two spatial domains were each divided into two additional domains based on rock type. Within each of the four geologic domains, twelve design sectors were defined based on wall orientations and locations of ramps. All design sectors in each domain were evaluated for both single and double catch bench performance to identify the optimal bench design parameters that meet the reliability criteria. The backbreak analysis is based on the use of controlled blasting. If controlled blasting is not possible, the ISA design parameters may need to be adjusted.

Two-dimensional limit-equilibrium analyses were performed on 11 critical sections by CNI, 3 in the Mill Zone pit, five in the Ledbetter pit, two in the Snake pit, and one in the Haile pit (sample section shown in Figure 16-2). Rocscience's Slide© limit equilibrium software (LEQ) was utilized to calculate the lowest overall slip surface FoS for each analysis section

FEFLOW (v. 7.4) was used to simulate pore pressure distributions for input into the eight limit-equilibrium cross sections analyzed in the Haile, Snake, and Ledbetter pits (example shown in Figure 16-3 To constrain the pore pressure distributions, the 2018 phreatic surface provided by Haile was used. For the Mill Zone pit, depressurization of the pit slopes was conservatively estimated by constructing a phreatic surface 10 m horizontally behind the pit slope face from the pit bottom up to the elevation of the regional phreatic surface. In some areas where the regional water level is high and significant slope heights of saprolite and CPS exist, additional depressurization is needed. The ISA recommendations require depressurizing the saprolite and CPS portions of the pits to 25 m horizontally behind the pit slope for all areas where the saprolite slope height is 50 m or less. Depressurization requirements increase to 40 m behind the pit slopes for saprolite slope heights greater than 50 m and less than 110 m. If any future design options expose saprolite slope heights in excess of 110 m, additional depressurization will be required. Assuming all depressurization is achieved, all sections analyzed meet or exceed the minimum design criteria of FoS ≥1.2.

Although all areas of the design meet or exceed the FoS design criteria, part of the Haile pit design is currently 4° steeper than design recommendations. In this area (Figure 16-4), pit slope dip direction between 315° and 345° is designed with an ISA of 42° while the recommendations are 37°. This area of non-compliance does not present a risk to overall slope stability. The issue is related to rockfall protection as it is expected that catch-benches will likely fail through the foliation orientation. Remedial measures are currently being evaluated and it is expected that the rockfall risk can be managed, practically and economically, through minor design modifications or by installation of ground control measures such as rock-fences or roll mesh.

CNI recommends the following future work as HGM continues to optimize their mine plans:

- Perform overall stability analysis on future mine plans to verify changes in slope geometry, geology, and wall orientations still meet design criteria.

- Additional cell mapping to expand the rock fabric database – this data is required to optimize the bench designs and to determine if areas that do not meet the design reliabilities are caused by structural conditions or by non-optimal blasting and excavation practices.
- Continued geologic mapping is required to identify major fault structures that could impact the Haile Gold Mine design. A geologic model of the major fault structures should be continually updated for the project area. Modifications to the design may be required if adverse fault structures are identified.
- Additional laboratory testing is recommended to further refine the material properties used in stability analyses.
- Continued geotechnical drilling is recommended to continue design optimizations. When possible oriented core or televiewer logging data should also be collected.
- Continue auditing constructed benches to determine if the design is being achieved satisfactorily – Lidar scans or aerial drone surveys of the excavated benches can be used to provide the data needed to perform the audit.
- A 3D model of foliation orientations should be developed. The orientation of the foliation will dictate the achievable bench face angle in some areas. A good understanding of the foliation will enable mine planners to minimize the impact of unfavorable orientations.

Table 16-1: ISA Recommendations for Near Surface Materials

Material Type	ISA (°)	Height (m)	BFA (°)	CBW (m)	Maximum Slope Height (m)	Comment
CPS and “Sericite”	30	5	50	4.5	15	
Saprolite - 1	35	5	63	4.6	50	Requires depressurization a minimum of 25 m behind face
Saprolite - 2	32	5	63	5.5	110	Requires depressurization a minimum of 40 m behind face

Source: Call & Nicholas Inc., 2022

Table 16-2. Mill Zone ISA Recommendations for Metasediments and Diabase Dikes

Range of Wall DDR (°)	Metasediments / Diabase Dikes - Bench Design							
	ISA (°)	Height (m)	BFA (°)	CBW (m)	ISA (°)	Height (m)	BFA (°)	CBW (m)
355 - 025	42	10	78	9	20 m Bench Heights not Recommended			
025 - 055	48	10	78	6.9	50	20	78	12.5
055 - 085	48	10	78	6.9	50	20	78	12.5
085 - 115	48	10	78	6.9	50	20	78	12.5
115 - 145	46	10	78	7.5	48*	20	78	13.8
145 - 175	45	10	78	7.9	48*	20	78	13.8
175 - 205	46	10	78	7.5	50	20	78	12.5
205 - 235	48	10	78	6.9	50	20	78	12.5
235 - 265	47	10	78	7.2	50	20	78	12.5
265 - 295	48	10	78	6.9	50	20	78	12.5
295 - 325	40	10	78	9.8	20 m Bench Heights not Recommended			
325 - 355	37	10	78	11.1	20 m Bench Heights not Recommended			

* ISA design controlled by overall slope stability
Source: Call & Nicholas Inc., 2018

Table 16-3: Mill Zone ISA Recommendations for Metavolcanics

Range of Wall DDR (°)	Metavolcanics - Bench Design							
	ISA (°)	Height (m)	BFA (°)	CBW (m)	ISA (°)	Height (m)	BFA (°)	CBW (m)
355 - 025	43	10	78	8.6	20 m Bench Heights not Recommended			
025 - 055	48	10	78	6.9	50	20	78	12.5
055 - 085	48	10	78	6.9	50	20	78	12.5
085 - 115	48	10	78	6.9	50	20	78	12.5
115 - 145	47	10	78	7.2	48*	20	78	13.8
145 - 175	46	10	78	7.5	48*	20	78	13.8
175 - 205	46	10	78	7.5	50	20	78	12.5
205 - 235	48	10	78	6.9	50	20	78	12.5
235 - 265	47	10	78	7.2	50	20	78	12.5
265 - 295	48	10	78	6.9	50	20	78	12.5
295 - 325	40	10	78	9.8	20 m Bench Heights not Recommended			
325 - 355	38	10	78	10.7	20 m Bench Heights not Recommended			

* ISA design controlled by overall slope stability
Source: Call & Nicholas Inc., 2018

Table 16-4: ISA Recommendations for Metasediments and Diabase Dikes – Snake, Haile, and Ledbetter Pits

Range of Wall DDR (°)	Metasediments / Diabase Dikes - Bench Design							
	ISA (°)	Height (m)	BFA (°)	CBW (m)	ISA (°)	Height (m)	BFA (°)	CBW (m)
345 - 015	42	10	78	9.0	20 m Bench Heights not Recommended			
015 - 045	48	10	78	6.9	50	20	78	12.5
045 - 075	48	10	78	6.9	50	20	78	12.5
075 - 105	47	10	78	7.2	50	20	78	12.5
105 - 135	47	10	78	7.2	50	20	78	12.5
135 - 165	47	10	78	7.2	50	20	78	12.5
165 - 195	47	10	78	7.2	50	20	78	12.5
195 - 225	48	10	78	6.9	50	20	78	12.5
225 - 255	47	10	78	7.2	50	20	78	12.5
255 - 285	44	10	78	8.2	50	20	78	12.5
285 - 315	43	10	78	8.6	20 m Bench Heights not Recommended			
315 - 345	37	10	78	11.1	20 m Bench Heights not Recommended			

Source: Call & Nicholas Inc., 2022

Table 16-5: ISA Recommendations for Metavolcanics – Snake, Haile, and Ledbetter Pits

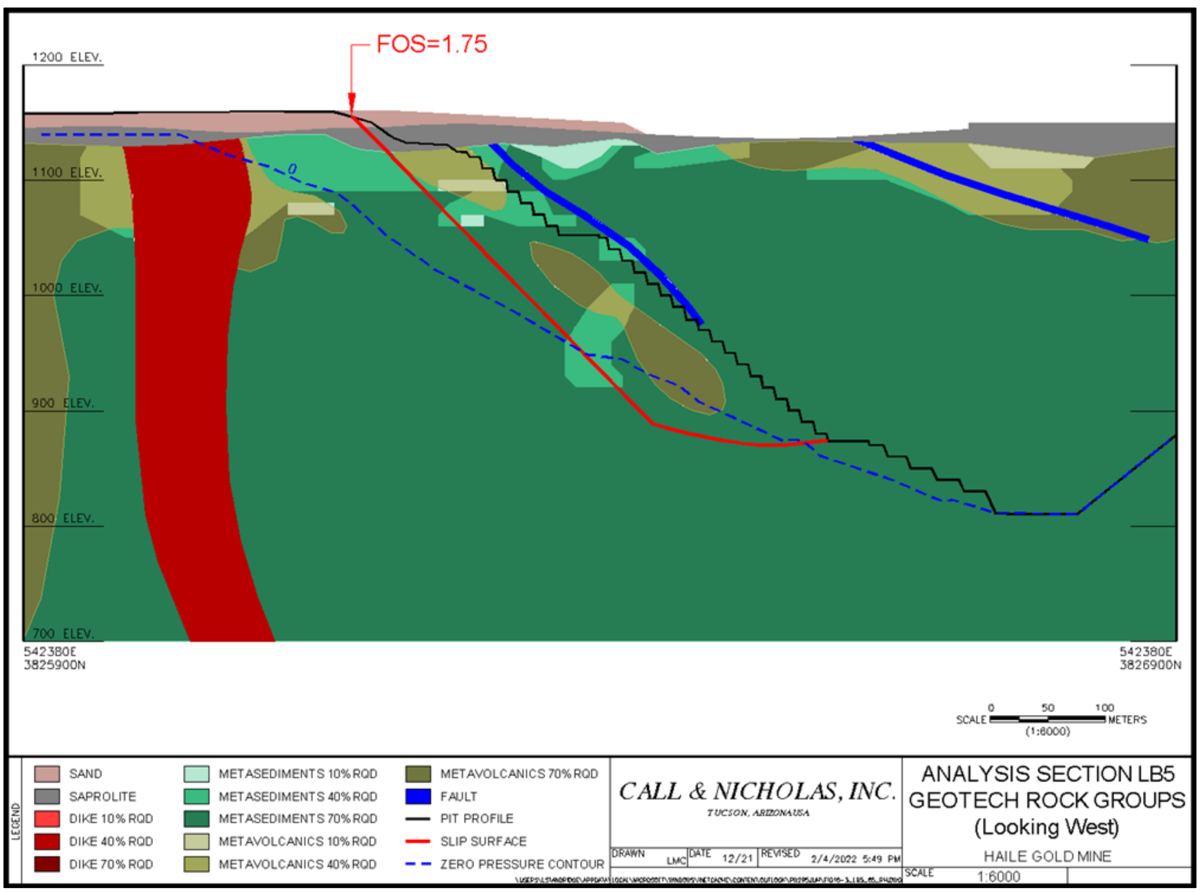
Range of Wall DDR (°)	Metavolcanics - Bench Design							
	ISA (°)	Height (m)	BFA (°)	CBW (m)	ISA (°)	Height (m)	BFA (°)	CBW (m)
345 - 015	42	10	78	9.0	20 m Bench Heights not Recommended			
015 - 045	48	10	78	6.9	50	20	78	12.5
045 - 075	49	10	78	6.6	50	20	78	12.5
075 - 105	47	10	78	7.2	50	20	78	12.5
105 - 135	47	10	78	7.2	50	20	78	12.5
135 - 165	47	10	78	7.2	50	20	78	12.5
165 - 195	47	10	78	7.2	50	20	78	12.5
195 - 225	48	10	78	6.9	50	20	78	12.5
225 - 255	47	10	78	7.2	50	20	78	12.5
255 - 285	45	10	78	7.9	50	20	78	12.5
285 - 315	44	10	78	8.2	20 m Bench Heights not Recommended			
315 - 345	37	10	78	11.1	20 m Bench Heights not Recommended			

Source: Call & Nicholas Inc., 2022

Table 16-6: Summary of Rock Mass Properties

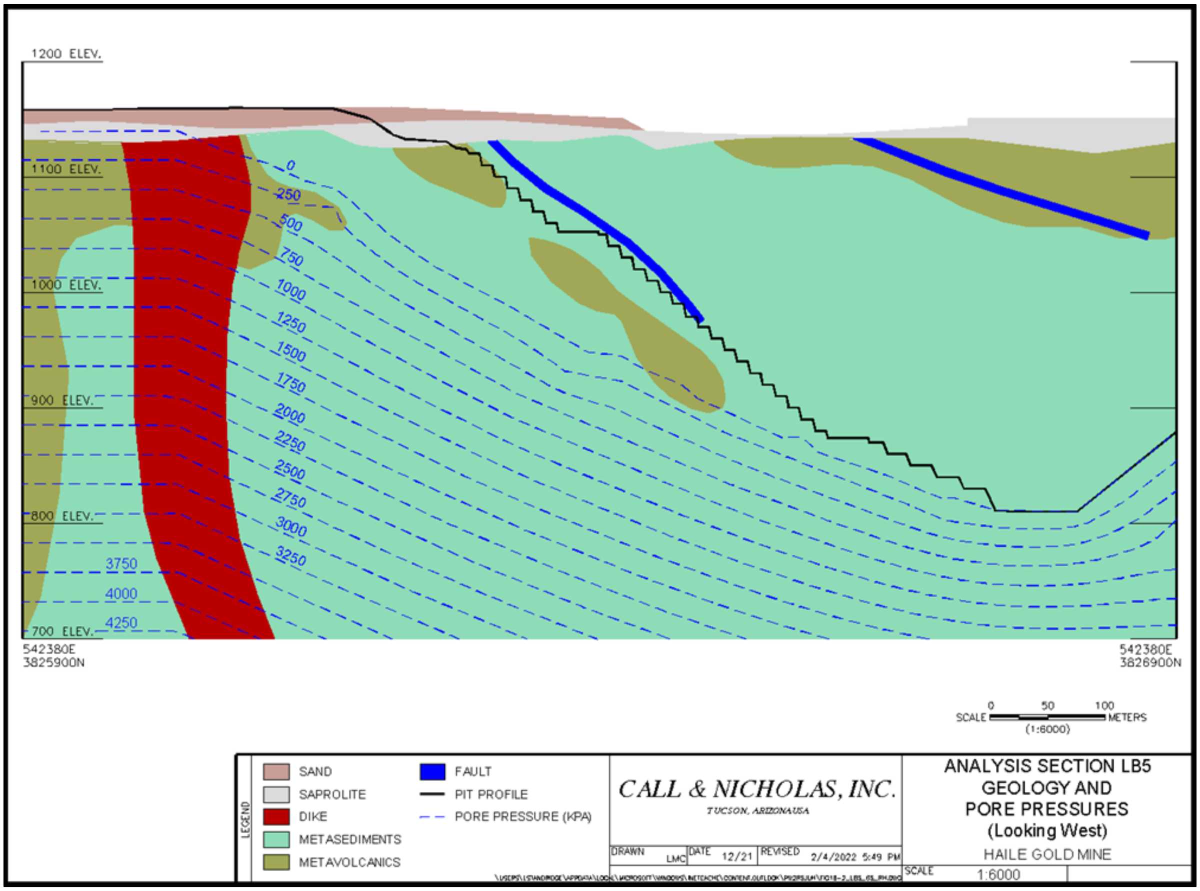
Rock Type	Strength Type	Density (kN/m³)	Cohesion (kPa)	Phi (°)
Meta Sediments	10% RQD	28.0	397.6	25.2
	40% RQD		859.7	27.5
	70% RQD		1867.9	30.9
	Foliation Anisotropy		251.2	21.5
	Cross Joint Anisotropy		489.5	22.0
Meta Volcanics	10% RQD	25.8	465.4	28.6
	40% RQD		1008.3	30.7
	70% RQD		2192.7	33.7
	Foliation Anisotropy		291.7	25.4
	Cross Joint Anisotropy		571.6	25.8
Diabase Dike	10% RQD	25.9	424.7	28.3
	40% RQD		913.6	31.3
	70% RQD		1980.9	35.8
CPS	Rock-mass	19.0	2.0	30
Saprolite	Rock-mass	22.0	20.0	32

Source: Call & Nicholas Inc., 2022



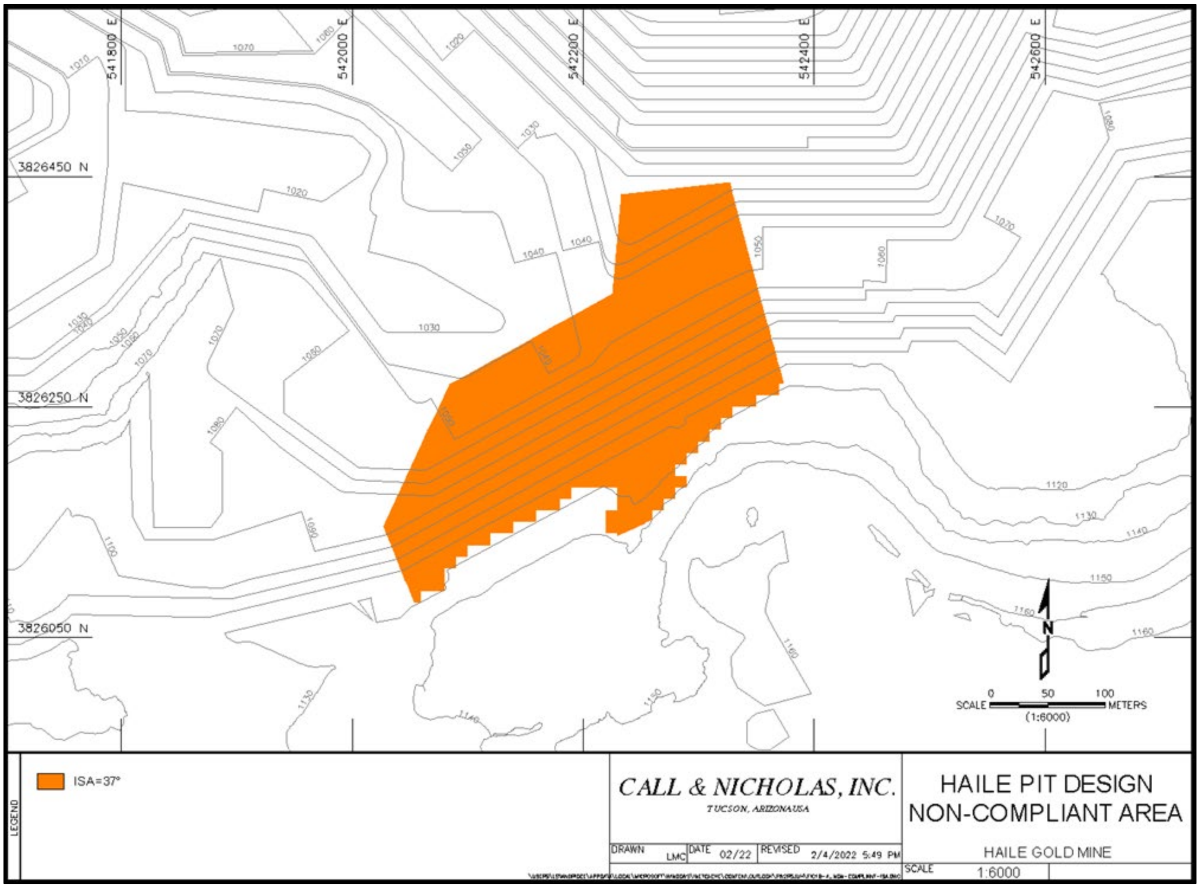
Source: Call & Nicholas Inc., 2022

Figure 16-2: Example Overall Analysis: Cross-Section LB5 for the South Wall of Ledbetter Pit



Source: Call & Nicholas Inc., 2022

Figure 16-3: Example of Steady State Pore Pressure Estimates: Cross-Section LB5 for the South Wall of the Ledbetter Pit



Source: Call & Nicholas Inc., 2022

Figure 16-4: Portion of Haile Pit Design Currently Exceeding Design Recommendations

<u>Hydrogeological</u>

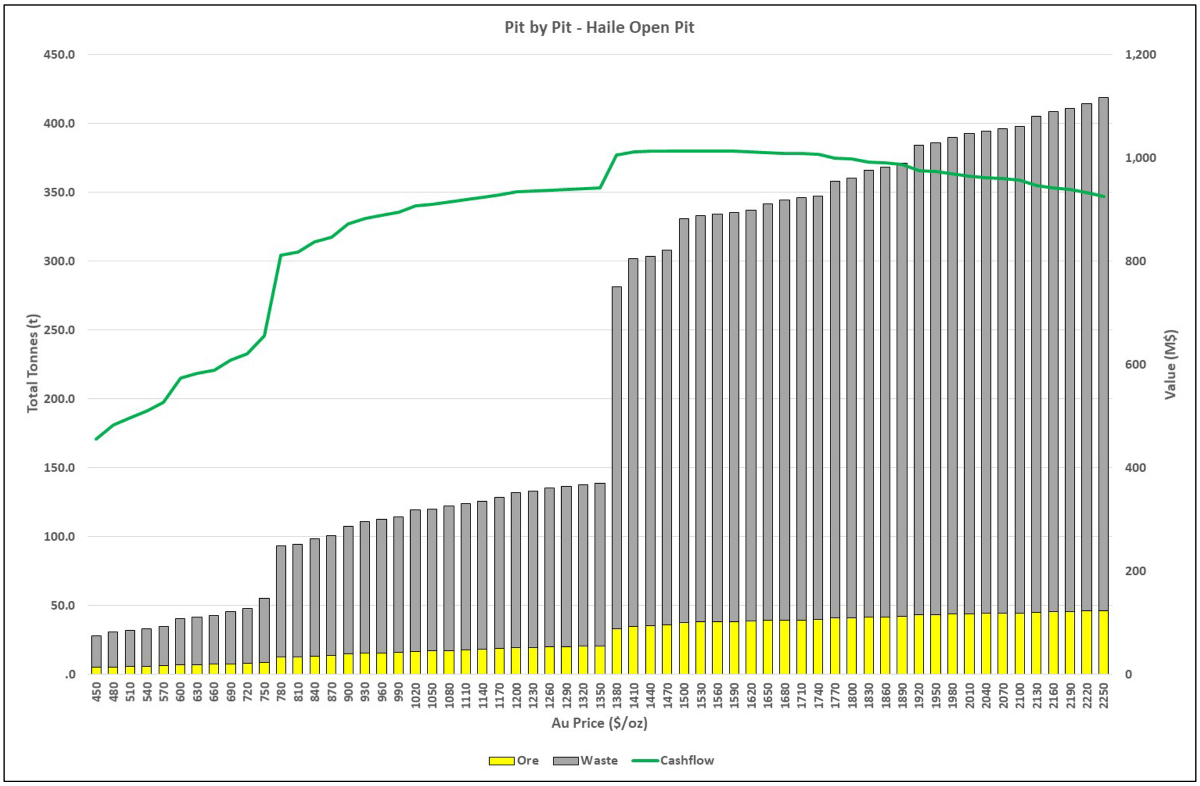
Dewatering and depressurization of highwalls is on-going. Ground water levels continue to decrease, and the drawdown trend appears to be reaching steady state in active mining areas. There are currently seven dewatering wells in service at Haile. Additional wells are planned to facilitate efficient and safe mining in accordance with the mine plan.

16.1.3 Optimization

The geological model has a block size of 10 m x 10 m x 5 m. This block size is considered appropriate for the loading units and mining practices currently is use at Haile. No further ore losses or ore dilution were applied at the optimization stage.

The open pit Ore Reserves are reported within a pit design based on open pit optimization results shown in Figure 16-5 and Table 16-7. The optimization included Measured and Indicated Mineral Resource categories with a gold price of US$1,500/oz Au. The optimization results using inferred material are very similar (within 3%) and therefore is considered immaterial. Starting topography for optimization work was a forecast end of period surface for 31 December 2021 produced in November

2021. There are no material differences between the forecast and actual 31 December 2021 surfaces.



Source: OceanaGold, 2022

Figure 16-5: Pit Optimization Tonnes by Revenue Factor

Table 16-7: Optimization Results for Selected Shell on US$75 Gold Price Increments

Gold Price	Rev Factor	Total	Waste	Strip Ratio	Ore		Contained metal	Recovered metal	Avg Met Recovery	UCF	Mining Cost	Process Cost	Selling Cost	Avg Cash Cost
$/oz	#	Mt	Mt	Wt:Ot	Mt	Au (g/t)	Moz	Moz	%	$M	$/t	$/t	$/oz	$/oz
750	0.50	55.0	46.3	5.34	8.7	2.65	0.74	0.65	87.5%	655.5	2.38	19.64	3.00	486
780	0.52	93.1	80.7	6.53	12.4	2.49	0.99	0.86	87.3%	811.6	2.39	19.64	3.00	561
810	0.54	94.1	81.5	6.47	12.6	2.47	1.00	0.87	87.2%	817.7	2.39	19.64	3.00	563
840	0.56	98.6	85.3	6.41	13.3	2.42	1.04	0.90	87.2%	836.7	2.40	19.64	3.00	573
870	0.58	100.9	87.1	6.35	13.7	2.39	1.05	0.92	87.1%	846.8	2.40	19.64	3.00	578
900	0.60	107.2	92.4	6.26	14.8	2.33	1.10	0.96	87.0%	871.3	2.39	19.64	3.00	592
930	0.62	110.6	95.3	6.26	15.2	2.30	1.13	0.98	86.9%	882.4	2.39	19.64	3.00	599
960	0.64	112.3	96.7	6.21	15.6	2.28	1.14	0.99	86.9%	888.8	2.39	19.64	3.00	603
990	0.66	114.1	98.2	6.18	15.9	2.26	1.15	1.00	86.8%	894.5	2.39	19.64	3.00	607
1020	0.68	119.2	102.6	6.17	16.6	2.22	1.19	1.03	86.7%	907.1	2.39	19.64	3.00	618
1050	0.70	120.2	103.2	6.10	16.9	2.20	1.20	1.04	86.7%	910.4	2.39	19.64	3.00	622
1080	0.72	121.9	104.6	6.06	17.3	2.18	1.21	1.05	86.6%	914.5	2.39	19.64	3.00	626
1110	0.74	124.1	106.4	6.01	17.7	2.15	1.22	1.06	86.6%	919.5	2.38	19.64	3.00	632
1140	0.76	125.8	107.7	5.97	18.1	2.13	1.24	1.07	86.5%	923.0	2.38	19.64	3.00	637
1170	0.78	128.2	109.7	5.90	18.6	2.10	1.25	1.08	86.4%	927.7	2.38	19.64	3.00	644
1200	0.80	132.0	112.7	5.84	19.3	2.06	1.28	1.10	86.3%	933.6	2.38	19.64	3.00	654
1230	0.82	133.2	113.6	5.80	19.6	2.05	1.29	1.11	86.3%	935.6	2.38	19.64	3.00	658
1260	0.84	135.3	115.4	5.81	19.9	2.03	1.30	1.12	86.3%	937.9	2.38	19.64	3.00	663
1290	0.86	136.3	116.2	5.77	20.1	2.02	1.31	1.13	86.2%	939.3	2.38	19.64	3.00	666
1320	0.88	137.4	117.1	5.76	20.3	2.01	1.31	1.13	86.2%	940.2	2.38	19.64	3.00	669
1350	0.90	138.8	118.3	5.75	20.6	2.00	1.32	1.14	86.2%	941.1	2.38	19.64	3.00	673
1380	0.92	281.5	248.6	7.58	32.8	1.78	1.88	1.61	85.6%	1004.6	2.40	19.64	3.00	875
1410	0.94	301.6	266.7	7.65	34.9	1.74	1.96	1.67	85.5%	1011.5	2.39	19.64	3.00	895
1440	0.96	303.4	267.9	7.56	35.5	1.73	1.97	1.68	85.5%	1012.3	2.39	19.64	3.00	899
1470	0.98	307.8	271.9	7.58	35.9	1.72	1.99	1.70	85.4%	1013.0	2.39	19.64	3.00	903
1500	1.00	330.8	293.3	7.83	37.5	1.71	2.06	1.76	85.4%	1013.5	2.40	19.64	3.00	924
1530	1.02	332.9	295.1	7.81	37.8	1.70	2.07	1.77	85.4%	1013.4	2.40	19.64	3.00	927
1560	1.04	334.0	296.0	7.79	38.0	1.70	2.08	1.77	85.4%	1013.2	2.40	19.64	3.00	928
1590	1.06	335.2	296.9	7.77	38.2	1.69	2.08	1.78	85.4%	1012.8	2.40	19.64	3.00	930
1620	1.08	337.1	298.6	7.75	38.5	1.69	2.09	1.78	85.3%	1011.9	2.40	19.64	3.00	933
1650	1.10	341.6	302.6	7.76	39.0	1.68	2.11	1.80	85.3%	1010.2	2.40	19.64	3.00	938
1680	1.12	344.1	304.9	7.78	39.2	1.68	2.11	1.80	85.3%	1009.0	2.40	19.64	3.00	940
1710	1.14	346.2	306.8	7.78	39.4	1.67	2.12	1.81	85.3%	1007.9	2.40	19.64	3.00	943
1740	1.16	347.3	307.8	7.78	39.6	1.67	2.13	1.81	85.3%	1007.0	2.40	19.64	3.00	945
1770	1.18	358.1	317.4	7.80	40.7	1.65	2.16	1.84	85.2%	999.1	2.40	19.64	3.00	958
1800	1.20	360.5	319.6	7.82	40.9	1.65	2.17	1.85	85.2%	997.3	2.40	19.64	3.00	960
1830	1.22	365.9	324.4	7.82	41.5	1.64	2.19	1.86	85.2%	992.1	2.40	19.64	3.00	967
1860	1.24	368.0	326.3	7.82	41.7	1.64	2.19	1.87	85.2%	989.9	2.40	19.64	3.00	970
1890	1.26	371.3	329.2	7.83	42.0	1.63	2.20	1.88	85.2%	986.8	2.40	19.64	3.00	974
1920	1.28	384.2	341.2	7.94	43.0	1.62	2.24	1.90	85.1%	975.2	2.40	19.64	3.00	988
1950	1.30	385.7	342.5	7.93	43.2	1.61	2.24	1.91	85.1%	973.2	2.40	19.64	3.00	990

Blue – Revenue Factor 1
Source: OceanaGold, 2022

Whittle optimization parameters are summarized in Table 16-8.

Table 16-8: Pit Optimization Parameters

Parameter	Unit	Value
Base Mining Cost	US$/t	1.81
Incremental Mining Cost	US$/t / 5 m bench	0.01
PAG Rehabilitation Cost	US$/t PAG waste	0.65
Processing Cost	US$/t ore	11.50
G&A Cost	US$/t ore	5.00
Ore Rehandle Cost	US$/t ore	0.70
TSF Expansion	US $/t ore	2.44
Gold Recovery	%	(1-(0.2152*Au grade^-0.3696))+0.025
Mill Throughput	Mtpa	3.8
Gold Price	US$/oz	1,500
Gold Refining & Selling Cost	US$/oz	3.00
Calculated Au Cut-off Grade	US $/t	0.5
Royalties	%	0.0
Discount Rate	%	5.0

Source: OceanaGold, 2022

Subsequent to the optimization being completed, gold recovery assumptions were changed in the life-of-mine plan to only include the +2.5% factor at grades greater than 1.7 g/t Au. This does not materially impact the optimization results.

The base mining cost was applied to all blocks and an incremental cost was added to blocks below the elevation where the haulage ramp exits the pit. The incremental cost was not added to blocks above the pit exit. As the Whittle optimization was completed prior to the pit design, pit exit elevations were taken from previous pit design work.

Rehabilitation costs have been added to potentially acid generating (PAG) blocks that will not be processed as this material will require permanent storage within the lined PAG facilities. The cost associated with permanent storage of processed blocks within the Tailing Storage Facility (TSF) is included in the processing cost. The determination of whether a block is processed and, by extension rehabilitation cost, is made by Whittle during optimization.

16.1.4 Mine Design

Mine Block Model

The OceanaGold Resource block model has been modified for mine planning purposes to include:

- Geotechnical variables for berm width, batter angle and bench height
- Ore and waste classifications based on calculated cut-off grades and Measured, Indicated, and Inferred material
- NAG/PAG determination

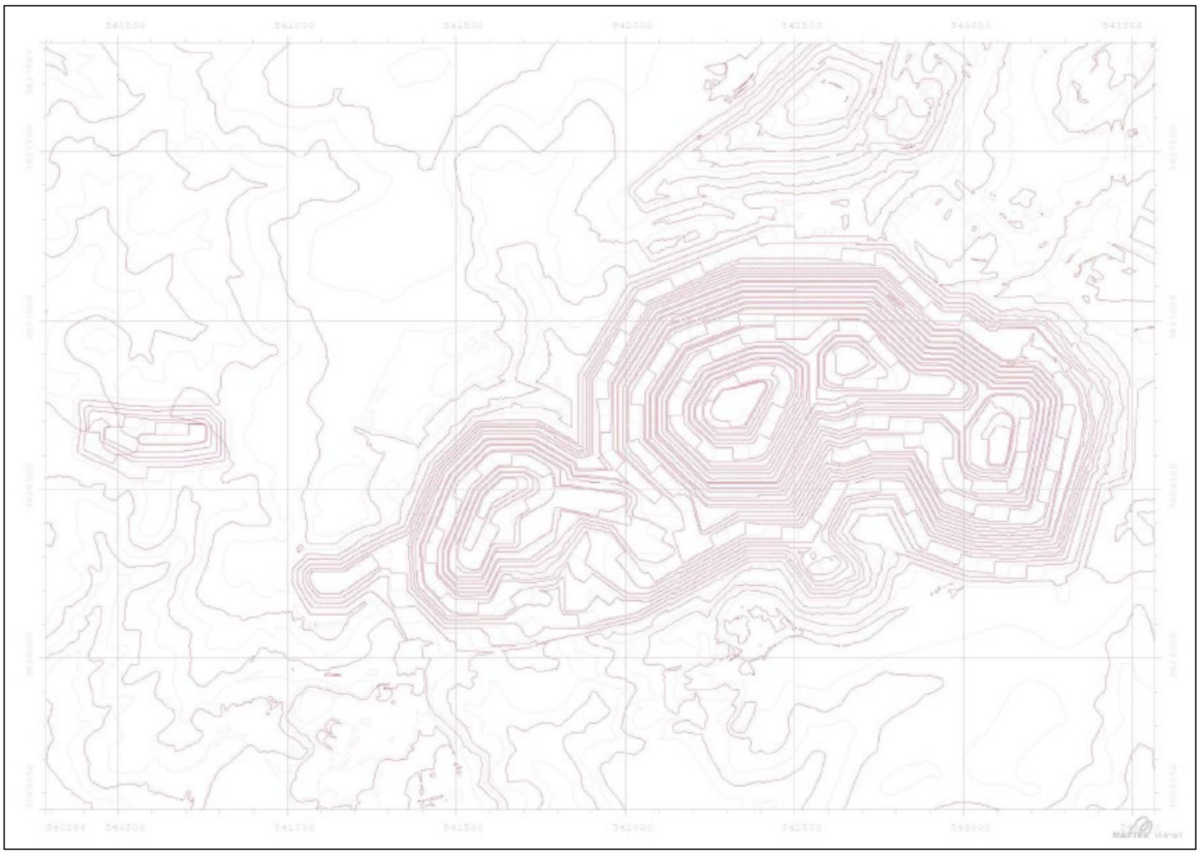
The NAG/PAG determinations govern the routing of waste material to either lined PAG waste dumps, in-pit PAG waste dumps (yellow only) or unlined NAG waste dumps.

Pit Design

OceanaGold used the Whittle pits as a guide for practical phase design. The major design parameters used are as follows:

- Ramp grade = 10%
- Full ramp width = 32 m (3x operating width for 730E)
- Single ramp width = 20 m for up to 60 m vertical or six benches
- Minimum mining width = 40 m but targets between 150 to 300 m
- Flat switchbacks
- Bench heights, berm widths and bench face angles in accordance with current site-specific design criteria

Figure 16-6 illustrates the final open pit design and associated ramp system. Ramp locations targeted saddle points between the various pit bottoms with ramps also acting as catch benches for geotechnical purposes. Each bench has at least one ramp for scheduling purposes.



Source: OceanaGold, 2022

Figure 16-6: LoM Pit Design

As noted in Section 16.1.2 and highlighted in Figure 16-4, one area of the final pit design has been identified as not complying with the geotechnical requirements detailed in that section. This non-compliance does not present a risk to overall slope stability. The issue related to rockfall protection as it is expected that catch-benches will fail through the foliation orientation. Remedial measures are under investigation and it is expected that the rockfall risk can be managed, practically and economically, through a minor design adjustment or installation of ground control measures such as rock-fences or roll mesh.

16.1.5 Overburden/Geochemical

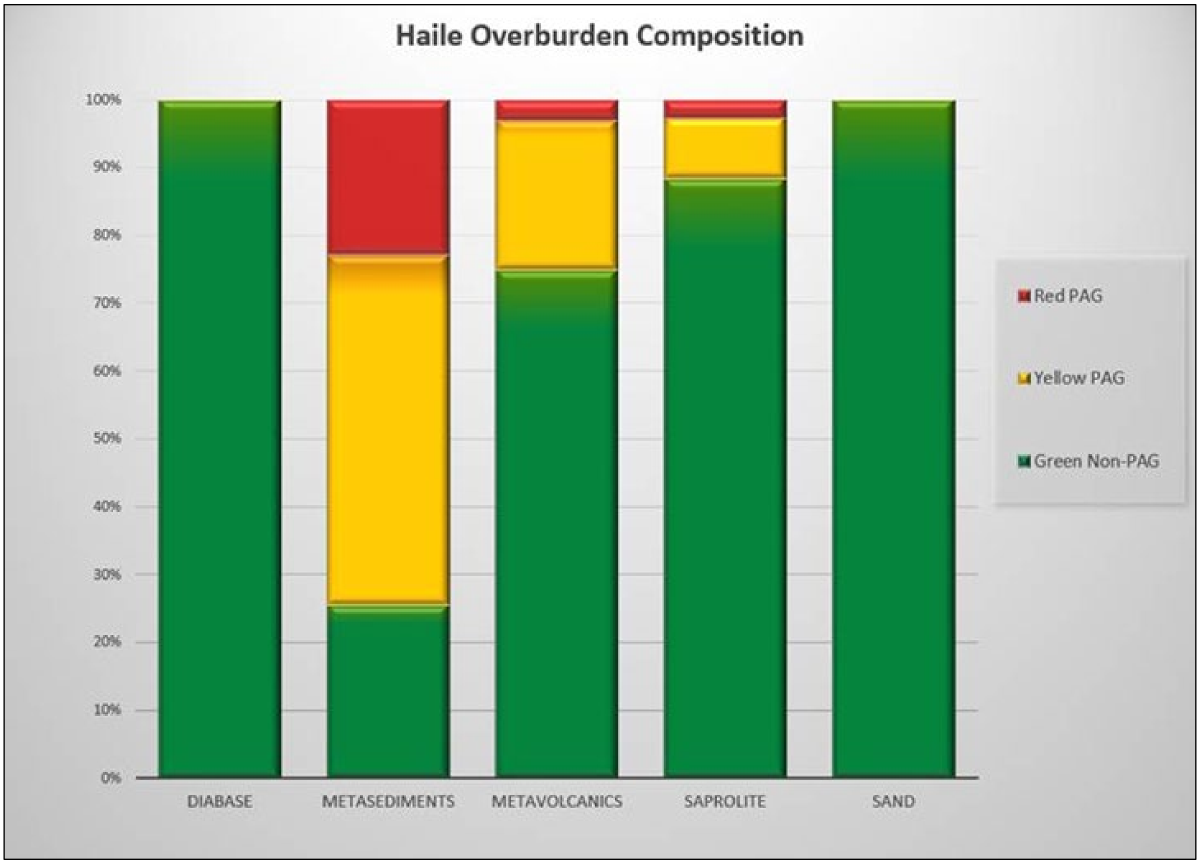
Overburden mined at Haile consists of oxide and sulfide material. Oxidation generally extends 20 to 40 m deep with no sulfide minerals. Unweathered rock below the base of oxidation contains sulfides with potential to generate sulfuric acid when exposed to air and water. The most common sulfide mineral at Haile is pyrite, FeS_2, typically at levels of 0.1% to 2%. Minor pyrrhotite and molybdenite are also observed in drillholes and pit exposures. Schafer (2015) performed an extensive geochemical characterization program of existing and future mine development rocks to identify, manage and mitigate geochemical risks at Haile as part of the open pit plan. The characterization program included static testing of 4,911 samples as well as kinetic testing of nine samples of overburden and one tailings sample. The current development rock management plan employs three categories based on combinations of the total sulfur content (S_T) and net neutralization potential (NNP). The NNP is a measure of overall acid generation potential calculated as the difference between the neutralization potential (NP) and AP. The categories of potentially acid generating (PAG) development rock are:

- Red PAG – Strongly acid generating:
  - NNP < -31.25 kg $CaCO_3$/t
- Yellow PAG – Moderately acid generating:
  - S_T > 0.2 % or NNP between 0 and -31.25 kg $CaCO_3$/t
- Green – Not acid generating, oxidized rock:
  - S_T < 0.2 % and NNP > 0 kg $CaCO_3$/t

The geochemical categories, and their associated management approaches, of development rock and saprolite within 15 m (50 ft) of the bedrock contact are summarized in Table 16-9. This is identical to categories in the underground plan described in section 16.2.4. A summary of the distribution of the three PAG categories in overburden rock is presented in Figure 16-7.

Kinetic testing of “Red” and “Yellow” material confirmed the applicability of the development rock classification scheme based on results of static tests, which included:

- Test work over 140 weeks on Red PAG samples developed low pH values (1.6 to 2.2) and released sulfate corresponding to one-third to one-half of the pyritic sulfur originally contained in the sample.
- Yellow PAG samples also had acidic pH values, though generally higher than Red PAG. High concentrations of metals were also released from these samples. Cumulative sulfate leached was much lower than for Red PAG samples owing to the lower levels of pyritic sulfur.
- Green overburden samples maintained near neutral pH and maintained low to non-detectable sulfate levels.



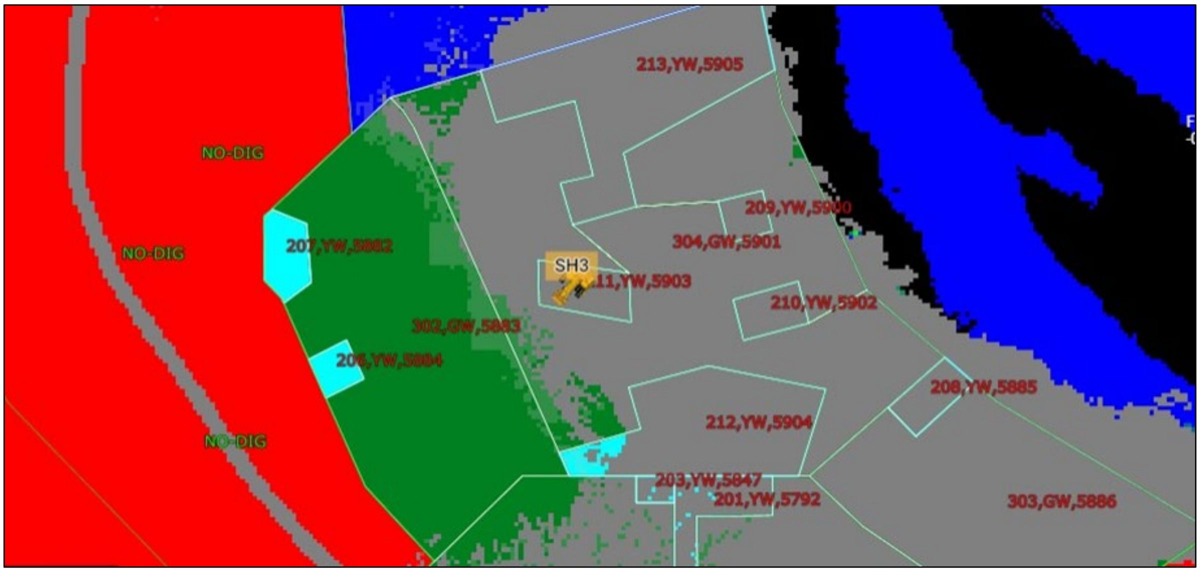
Source: Schafer, 2015

Figure 16-7-: Lithological Distribution of PAG Categories in Overburden

Data from historical sulfur and carbon data, used to originally populate the waste rock block model, are used together with operational sulfur and carbon assays from blastholes to calculate overburden type and inform the placement of materials as follows:

- All Red PAG is sent to the East or West OSAs. These are lined facilities, with the East OSA already in use. Material will be placed in lifts and compacted by haulage fleet traffic. For closure, the top of these facilities will be covered with saprolite, followed by an HDPE geomembrane liner and then a layer of growth media. The growth media will be seeded to establish vegetation.
- Yellow PAG can be stored in the East or West OSAs, or below a prescribed water table within pits. Yellow material in-pit will be mixed with lime before placement in the pit void. Material will be placed in lifts and compacted by haulage fleet traffic. For closure, the top surface of these facilities will be covered with green waste. Yellow PAG is able to be utilized in-pit for road construction.
- Green material will be stored in unlined facilities or backfilled into the pits. Material placed on unlined facilities will be placed in lifts and compacted by haulage fleet traffic. The reclaimed slopes will be seeded to establish vegetation.

Haile utilizes a state-of-the-practice overburden management system. The PAG / non-PAG block model categorizations have been incorporated into the Caterpillar MineStar™ operations and equipment management software. Equipment operators can see in real time on their vehicle screen the PAG category of the pit bench on which they are working (Figure 16-8). The screen display is also simultaneously transmitted to department managers, allowing the entire team to see the PAG / non-PAG category of overburden as it is being excavated. New PAG / non-PAG categorizations from blasthole analyses are populated to the software as they become available. The system is an efficient means of maintaining up-to-date overburden categorization and informing material handling to guide each haul truck to the appropriate overburden stockpile for disposal.



Source: Schafer, 2015

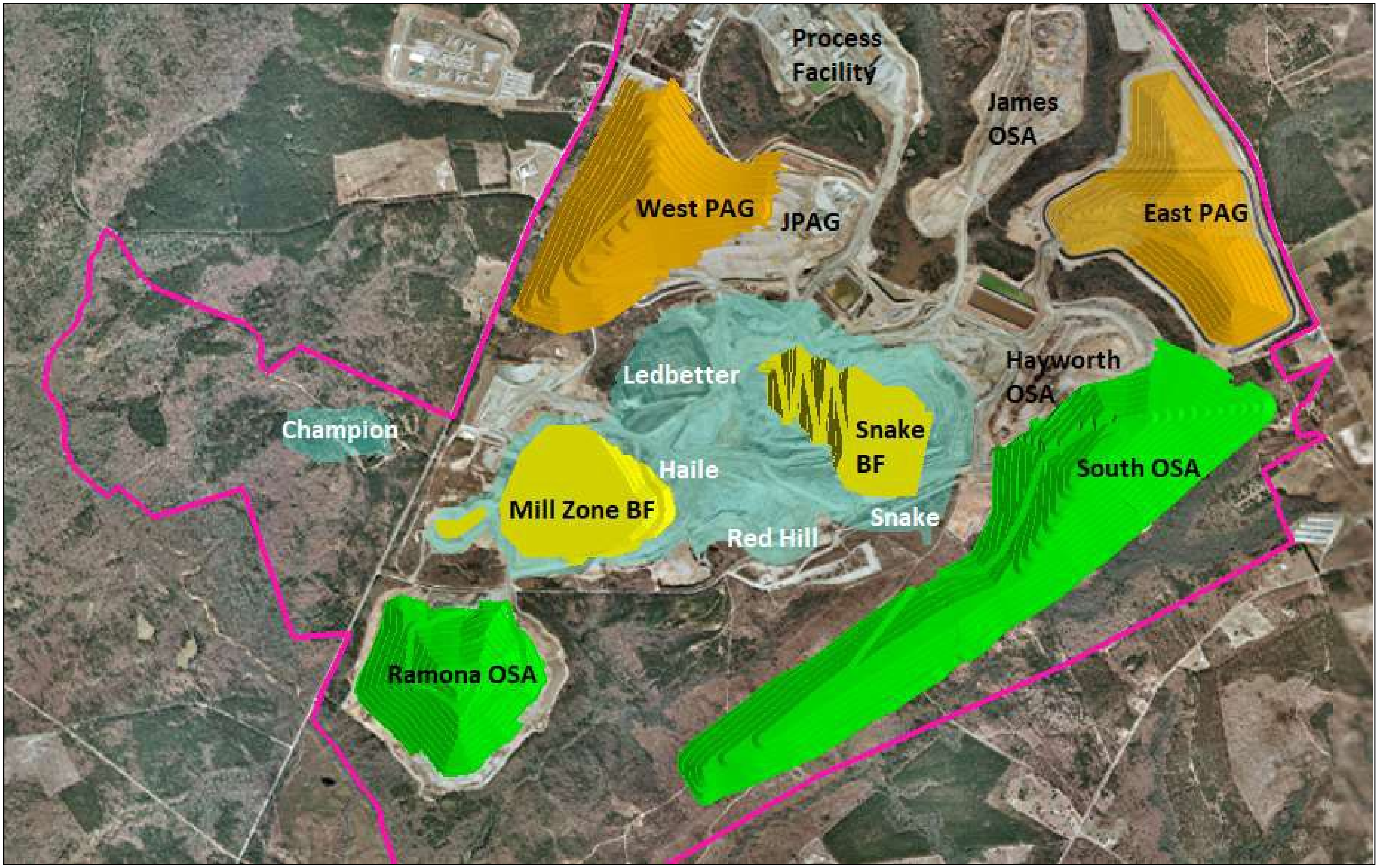
Figure 16-8: Vehicle Screen Display in MineStar™ Software showing Real-Time PAG Categories (YW = yellow waste, GW = green waste)

Table 16-9: Current Open Pit and Adopted UG Overburden Classification at Haile

Operational Testing Criterion	Overburden Abundance	Characteristics	Proposed Management
Red PAG - Strongly Acid Generating Overburden			
Found in Metasediment Unit. For Metasediment, NNP < -31.25 kg/t as $CaCO_3$	About 38% of Metasediment unit	When oxidized, contact water will have low pH (< 3.0) and very high metals, sulfate and acidity (>5,000 mg/L)	Stored in geomembrane encapsulated PAG cell, placed in lifts, compacted and Saprolite-lined outside perimeter to reduce oxygen supply
Yellow PAG - Moderately Acid Generating Overburden			
Found in Metasediment and Metavolcanic Bedrock Units and Saprolite. For bedrock, Total S > 0.2% or NNP <0 and NNP> -31.25 kg/t as $CaCO_3$. For Saprolite within 50 ft of bedrock contact, Total S > 0.2% and NNP> -31.25 kg/t as $CaCO_3$	About 22% of Metasediment unit, 6% of Metavolcanic unit, and 5% Saprolite	If allowed to oxidize, contact water will have low pH (3.0 to 4.0) and low to moderate metals (mostly Fe and Al)	Managed as above then may be placed in lifts as subaqueous pit backfill (2 lbs/t lime added as needed and 5-ft saprolite cover)
Green Overburden - Not Acid Generating			
Found in Metasediment and Metavolcanic Bedrock Units, Saprolite and Coastal Plain Sands. For Bedrock, Total S < 0.2% and NNP > 0 kg/t as $CaCO_3$. For Saprolite within 50 ft of bedrock contact, Total S < 0.2%. All Saprolite more than 50 ft above bedrock and all Coastal Plain Sand is Green Overburden.	40% Metasediment Unit, 94% Metavolcanics, 95% Saprolite and all Coastal Plain Sand	Contact water may have moderately acidic to alkaline pH (4.0 to 8.0), sulfate low (<1,000 mg/L) and metals non-detectable.	Placed in unlined overburden piles. Runoff will not require treatment assuming it meets storm water requirements as expected

Source: Schafer, 2015

The overburden storage is discussed in more detail in Section 18.2, with the final year site plan shown in Figure 16-9.



Yellow inside pits is backfill material.
Source: OceanaGold, 2022

Figure 16-9: Final Pit Design and Ultimate Overburden Storage Site Plan

16.1.6 Mine Production Schedule

Cut-Off Grade

OceanaGold have used a mine breakeven CoG for the determination of ore and waste. The base assumptions for the calculation are detailed in Table 16-10. Primary and Oxide ores have different processing recovery responses, and separate CoG values have been applied. The CoG grade applied to Primary Measured and Indicated Resources is 0.5 g/t Au and to Oxide is 0.6 g/t.

Table 16-10: Cut-off Grade Calculation

Description	Units	CoG Primary	CoG Oxide
Assumptions			
Gold Price	US$/oz	1,500	1,500
Smelting and Refining	US$/oz	3.00	3.00
Au Recovery*	%	73.0	68.0
Operating Costs			
G&A	US$/t ore	5.00	5.00
Tailings	US$/t ore	2.44	2.44
Rehandle (ROM)	US$/t ore	0.70	0.70
Processing	US$/t ore	11.50	11.50
Subtotal	**US$/t**	**19.64**	**19.64**
CoG - Head Grade	g/t	0.5	0.6

*Recovery at Primary CoG based on recovery formula: (1-(0.2152*Au grade^-0.3696))
Source: OceanaGold, 2022

The processing recovery equation changes at 1.7 g/t Au, above this grade a 2.5% uplift is applied based on operating history at HGM, as described in Section 13.

Dilution and Mining Recovery

Reserves are based on the Haile resource block model that uses a 10 m x 10 m x 5 m block dimension. No further mining dilution or ore recovery is applied to the resource block model during optimization on the assumption that ore is mined on 3.3 m benches using a backhoe excavator.

Scheduling studies completed in 2021 highlighted that with the current equipment fleet at Haile, some mining of ore by Face Shovel excavators will be unavoidable. Subsequently, an SMU study was completed to estimate the impacts on mining dilution and ore recovery at bench heights suitable for mining by Face Shovel excavator.

The results of the SMU study indicated that different areas of the mineralized zone react with variable magnitude to different bench heights. These results are shown in Table 16-11. Note that the values in the table are multiplier adjustments to the 10 m x 10 m x 5 m block model tonnes and grade estimates that include diluting grades appropriate to 3.3 m benches.

Table 16-11: Tonnage and Grade Multipliers for Application of Mining Dilution and Ore Recovery

Multipliers	3.3 m Flitch			5 m Flitch			10 m Bench		
Phase	**tonnes**	**g/t**	**oz**	**tonnes**	**g/t**	**oz**	**tonnes**	**g/t**	**oz**
Mill Zone: Phase 1	1.00	1.00	1.00	1.08	0.96	1.04	1.15	0.85	0.98
Mill Zone: Phase 2	1.00	1.00	1.00	1.12	0.94	1.06	1.18	0.83	0.98
Snake: Phase 1	1.00	1.00	1.00	1.03	0.94	0.97	1.06	0.76	0.81
Snake: Phase 2	1.00	1.00	1.00	1.03	0.94	0.97	1.06	0.76	0.81
Snake: Phase 3	1.00	1.00	1.00	1.10	0.95	1.05	1.18	0.87	1.02
Ledbetter: Phase 1	1.00	1.00	1.00	1.01	0.90	0.91	1.13	0.73	0.83
Ledbetter: Phase 2A	1.00	1.00	1.00	1.01	0.96	0.97	1.07	0.93	0.99
Ledbetter: Phase 2B	1.00	1.00	1.00	1.01	0.96	0.97	1.07	0.93	0.99
Ledbetter: Phase 3	1.00	1.00	1.00	1.05	0.92	0.96	1.17	0.82	0.95
Ledbetter: Phase 4	1.00	1.00	1.00	1.05	0.97	1.02	1.05	0.94	0.99
Small: Phase 1 (MZ1)	1.00	1.00	1.00	1.08	0.96	1.04	1.15	0.85	0.98
Haile: Phase 1 (RH1)	1.00	1.00	1.00	1.09	0.88	0.96	1.17	0.84	0.98
Haile: Phase 2 (RH1)	1.00	1.00	1.00	1.09	0.88	0.96	1.17	0.84	0.98
Champion: Phase 1 (MZ1)	1.00	1.00	1.00	1.08	0.96	1.04	1.15	0.85	0.98

Note: Values are multipliers applied directly to tonnes and grade, with impact shown to contained gold ounces.
Phase names in parentheses phase results used for areas that weren't directly part of the SMU study.
Source: OceanaGold, 2022

Given the variability between pit phases, a single, blanket approach to applying mining dilution and ore recovery is considered unsuitable. To account for these differences, tonnes and grade multipliers identified in Table 16-11 have been applied directly to the resource block model within each pit phase, to the first 30% of ore mined in each phase. The selection of the first 30% of ore is based on previous scheduling exercises that identified the general crossover point between bulk mining with Face Shovels and selective mining with Backhoe Excavators. The global impact of this is a mining dilution factor of approximately 1%, and ore recovery of 98.9%. While the adjustments have limited impact on the overall mine plan, it accounts for variability on a period-by-period basis.

Phase Design Inventory

The ultimate pit design has been broken into 12 mine phases for sequenced extraction in the mine production schedule. The design parameters for each phase are the same as those used for the final pit design including assumed ramp widths. Phase designs were constructed by splitting up the final pit into smaller and more manageable pieces, while still ensuring each bench within each phase has ramp access. The phases have been developed by balancing mining constraints with the extraction sequence suggested by pit optimization results presented previously.

The phases and direction of extraction allow for multiple benches on multiple elevations with a sump always available for pit dewatering. This means that during periods of heavy rainfall, perched benches will be available for extraction.

Once the phases have been designed, solid triangulations are created for each phase as they cut into topography from previous phases. These solid phases are then imported to RPM Global OPMS mine scheduling software and inventories reported using the resource block model. Scheduling is completed using 5 m bench heights to match the resource block model.

Table 16-12 details the phase inventory that forms the basis of the production schedule.

Table 16-12: Phase Inventory (1/1/2022 to End of Mine Life)

Phases	Ore (Measured + Indicated)			Waste	TOTAL	
	Tonnes (kt)	Au g/t	Contained Au (koz)	Tonnes (kt)	SR	Tonnes (kt)
Mill Zone: Phase 2	3,240	1.88	195	19,979	6.2	23,218
Snake: Phase 2	70	2.31	5	77	1.1	147
Snake: Phase 3	4,238	1.43	195	41,344	9.8	45,582
Ledbetter: Phase 1	412	2.94	39	714	1.7	1,126
Ledbetter: Phase 2A	2,334	2.53	190	47,682	20.4	50,016
Ledbetter: Phase 2B	3,684	2.50	296	46,009	12.5	49,693
Ledbetter: Phase 3	5,355	1.82	314	24,817	4.6	30,172
Ledbetter: Phase 4	14,037	1.34	603	139,349	9.9	153,387
Small: Phase 1	840	0.71	19	2,939	3.5	3,779
Haile: Phase 1	1,473	1.43	68	4,475	3.0	5,948
Haile: Phase 2	3,458	1.08	120	18,511	5.4	21,969
Champion: Phase 1	1,029	0.89	29	5,011	4.9	6,040
Grand Total	**40,170**	**1.60**	**2,073**	**350,907**	**8.7**	**391,076**

Source: OceanaGold, 2022

Production Schedule Targets

The production schedule has a start date of 1 January 2022 and is based on the end of year survey dated 31 December 2021. Scheduling is completed using RPM Global's OPMS scheduling software. Scheduling outputs are reported monthly for the first two years, quarterly for the next two years, and annually thereafter, however are summarized into annual periods for the following sections.

Production scheduling uses an activity-based approach using productivity rates for excavators and trucks, and targets maintaining a balance between ore supply to the processing plant and pre-stripping on subsequent phases. Bench vertical advance rates are generally limited to two 5 m benches per month. However, mining is usually limited by total movement constraints rather than vertical advance.

Production Schedule Results

The results of the production schedule are detailed in Table 16-13. A more detailed breakdown of the production schedule is found in Table 16-14. Note that stockpile material is not included in this summary and therefore numbers here do not match the reserve table and economic models (which do include stockpile material).

Table 16-13: Open Pit Production Summary

Year	Ore (kt)	Au Grade (g/t)	Ag Grade (g/t)	Waste (kt)	Total Mined (kt)
2022	2,898	1.7	2.3	38,380	41,278
2023	2,888	1.9	2.3	34,844	37,732
2024	4,593	1.7	2.7	27,524	32,117
2025	1,675	1.1	2.2	35,718	37,393
2026	2,710	2.6	2.6	38,430	41,140
2027	4,678	1.6	2.1	32,019	36,697
2028	3,600	1.8	2.0	33,259	36,859
2029	3,600	1.4	2.5	27,520	31,120
2030	2,919	0.9	2.4	35,051	37,971
2031	2,670	1.0	2.8	26,550	29,220
2032	3,519	1.4	2.6	17,186	20,705
2033	4,419	1.9	2.7	4,425	8,844
2034	0	0	0	0	0
Total	**40,170**	**1.6**	**2.4**	**350,907**	**391,076**

Source: OceanaGold, 2022

Points of note on the schedule:

- Low ore production in 2025 relates to completion of Ledbetter 2A and a focus on waste stripping in Ledbetter 2B.
- Excess Yellow Waste stored in in-pit dumps during 2022 and 2023. Rehandle of this material is scheduled on an as-required basis to expose the footprint of subsequent pit phases and is spread across 2023 to 2029.
- Total mining movement rates reduce in the second half of the mine schedule to maintain a maximum low-grade stockpile size of approximately 3 Mt.
- Low ore production and grade in 2030 and 2031 relate to mining of Champion Phase 1 for ore supply, waste stripping in Ledbetter 4.

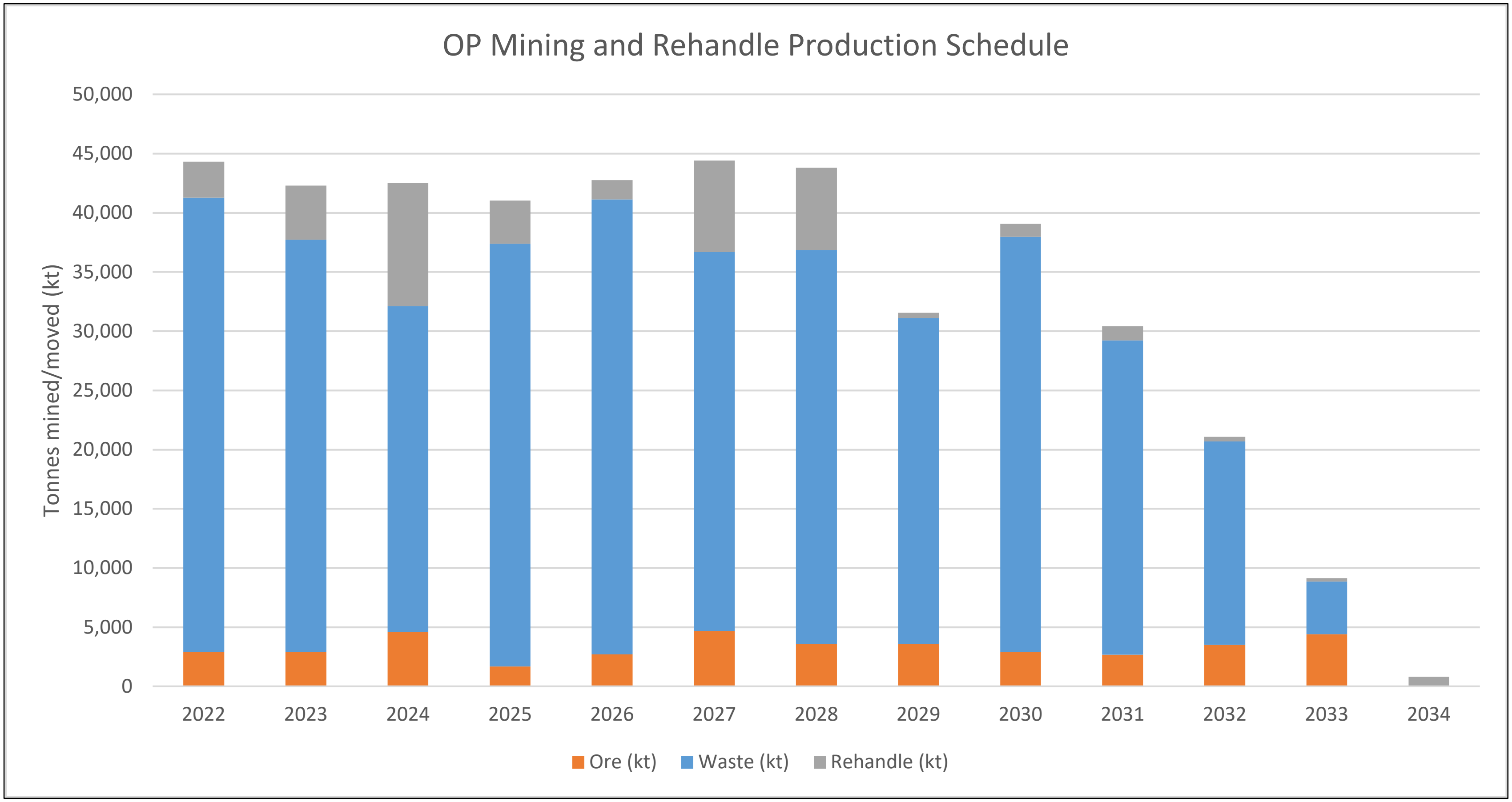
Figure 16-10 illustrates the annual production schedule for ore and waste tonnes, plus rehandle completed using the OP mining fleet.

Figure 16-11 breaks down the PAG (Red and Yellow) waste, NAG (Green) waste, and Ore tonnes for the LoM. The proportion of Red/Yellow to Green waste generally increases with depth, highlighted by the decreasing quantities of Green waste produced toward the end of the OP mine life.

Table 16-14: Mine (UG + OP) and Mill Production Schedule

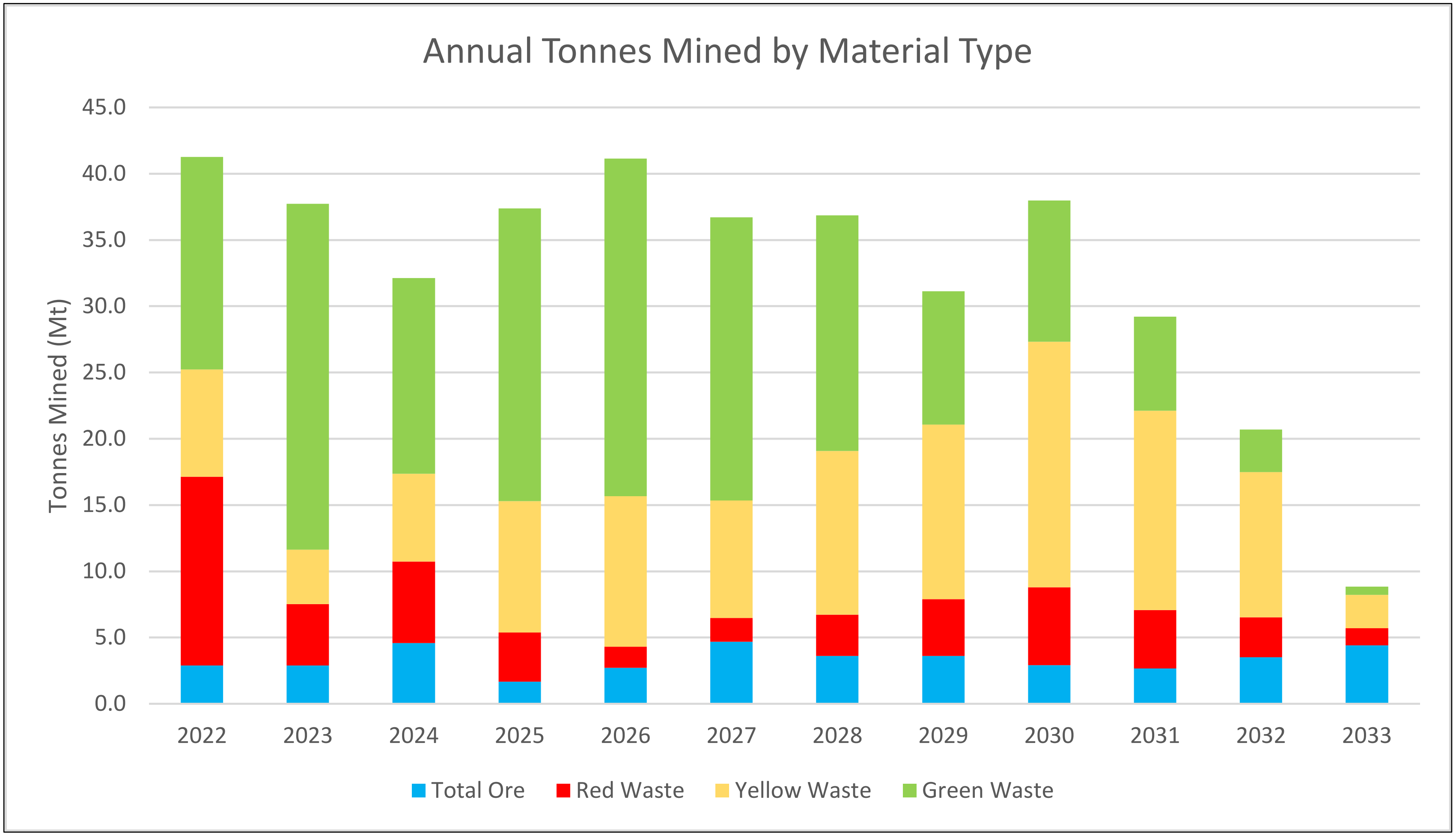
Description	**Description**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**LoM Total**
UG Ore	RoM (kt)		191	736	749	739	739	263							3,416
	RoM Au (g/t)		4.1	4.3	3.5	4.6	3.2	2.2							3.8
	RoM Ag (g/t)		0.0	0.0	0.0	0.0	0.0	0.0							0.0
Total OP Ore	**RoM (kt)**	**2,898**	**2,888**	**4,593**	**1,675**	**2,710**	**4,678**	**3,600**	**3,600**	**2,919**	**2,670**	**3,519**	**4,419**		**40,170**
	RoM Au (g/t)	**1.7**	**1.9**	**1.7**	**1.1**	**2.6**	**1.6**	**1.8**	**1.4**	**0.9**	**1.0**	**1.4**	**1.9**		**1.6**
	RoM Ag (g/t)	**2.3**	**2.3**	**2.7**	**2.2**	**2.6**	**2.1**	**2.0**	**2.5**	**2.4**	**2.8**	**2.6**	**2.7**		**2.4**
	EOP Stock (kt)	**1,219**	**528**	**2,163**	**905**	**671**	**2,405**	**2,571**	**2,371**	**1,491**	**360**	**202**	**821**		
	RoM Au (g/t)	**0.7**	**0.7**	**0.8**	**0.6**	**1.9**	**0.8**	**0.8**	**0.7**	**0.6**	**0.6**	**0.6**	**2.3**		
	RoM Ag (g/t)	**1.0**	**1.2**	**2.1**	**2.0**	**2.5**	**2.0**	**1.9**	**1.8**	**1.7**	**1.8**	**2.2**	**2.8**		
UG Mill Feed	Ore Feed (kt)		191	736	749	739	739	263							3,416
	Ore Feed Au (g/t)		4.1	4.3	3.5	4.6	3.2	2.2							3.8
	Ore Feed Ag (g/t)		0.0	0.0	0.0	0.0	0.0	0.0							0.0
	Ore Feed Contained Au (koz)		25.0	101.9	83.7	110.3	75.5	18.8							415
	Ore Feed Contained Ag (koz)		0.0	0.0	0.0	0.0	0.0	0.0							0
OP Mill Feed	Ore Feed (kt)	3,507	3,579	2,958	2,933	2,945	2,944	3,434	3,800	3,800	3,800	3,677	3,800	821	41,998
	Ore Feed Au (g/t)	1.7	1.6	2.2	1.0	2.1	2.3	1.9	1.4	0.9	0.9	1.4	1.7	2.3	1.6
	Ore Feed Ag (g/t)	2.1	2.1	2.8	2.2	2.4	2.4	2.1	2.5	2.3	2.5	2.5	2.7	2.8	2.4
	Ore Feed Contained Au (koz)	193	189	210	93	201	220	211	174	107	109	161	210	60	2,137
	Ore Feed Contained Ag (koz)	240	236	268	210	232	225	228	311	285	300	297	324	73	3,228
Total Mill Feed	**Total Feed (kt)**	**3,507**	**3,770**	**3,694**	**3,682**	**3,683**	**3,683**	**3,697**	**3,800**	**3,800**	**3,800**	**3,677**	**3,800**	**821**	**45,414**
	Total Feed Au (g/t)	**1.7**	**1.8**	**2.6**	**1.5**	**2.6**	**2.5**	**1.9**	**1.4**	**0.9**	**0.9**	**1.4**	**1.7**	**2.3**	**1.7**
	Total Feed Ag (g/t)	**2.1**	**1.9**	**2.3**	**1.8**	**2.0**	**1.9**	**1.9**	**2.5**	**2.3**	**2.5**	**2.5**	**2.7**	**2.8**	**2.2**
	Total Feed Contained Au (koz)	**193**	**214**	**312**	**176**	**311**	**295**	**230**	**174**	**107**	**109**	**161**	**210**	**60**	**2,552**
	Total Feed Contained Ag (koz)	**240**	**236**	**268**	**210**	**232**	**225**	**228**	**311**	**285**	**300**	**297**	**324**	**73**	**3,228**

Source OceanaGold, 2022



Source: OceanaGold, 2022

Figure 16-10: Annual Production Schedule



Source: OceanaGold , 2022

Figure 16-11: Material Type Annual Schedule

Bench Sinking Rate

Table 16-15 shows the benches mined from each pit/phase on an annual basis converted to 10 m equivalents. Sinking rates are reasonable in each year.

Table 16-15: LoM Yearly Bench Sinking Rates

Phase	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
SP_03	# 10 m Benches	-	-	1.5	3.5	6.0	5.6	2.9	3.9	1.3	-	-	-
MZ_02	# 10 m Benches	8.0	5.5	-	-	-	-	-	-	-	-	-	-
SM_01	# 10 m Benches	-	2.2	3.3	-	-	-	-	-	-	-	-	-
HL_01	# 10 m Benches	5.0	-	-	-	-	-	-	-	-	-	-	-
HL_02	# 10 m Benches	-	-	9.4	4.6	-	-	-	-	-	-	-	-
LB_01	# 10 m Benches	3.0	-	-	-	-	-	-	-	-	-	-	-
LB_2A	# 10 m Benches	6.4	10.3	4.8	-	-	-	-	-	-	-	-	-
LB_2B	# 10 m Benches	-	-	3.5	8.2	10.8	2.4	-	-	-	-	-	-
LB_03	# 10 m Benches	-	-	-	0.5	5.9	6.0	4.0	4.1	1.1	-	-	-
LB_04	# 10 m Benches	-	-	-	-	-	3.4	4.7	3.7	5.2	4.7	5.8	6.0
LB_4A	# 10 m Benches	-	-	-	-	2.3	2.6	4.4	3.1	6.1	-	-	-
CP_01	# 10 m Benches	-	-	-	-	-	-	-	-	1.9	4.6	-	-
LP_02	# 10 m Benches	-	2.5	0.5	-	-	-	-	-	-	-	-	-

Source SRK, 2020

16.1.7 Mining Fleet and Requirements

The open pit loading and hauling equipment fleet consists of hydraulic excavators (Komatsu PC3000 and PC4000 models) and rigid frame haul trucks (CAT 785 and Komatsu 730E). Blasthole drilling and wall control drilling is performed with a fleet of Sandvik DR410i and Epiroc D65 drills. Typical ancillary equipment, including track dozers, wheel dozers, motor graders and water trucks support the mining operation.

Table 16-16 shows the estimation of the scheduled hours per year after deducting for weather delays. Table 16-17 shows the assumed mechanical availabilities and use of availabilities for the loading and hauling equipment to estimate the potential operating hours per year.

Table 16-16: Estimation of Scheduled Hours per Year for a Typical Year

Maximum Days Per Year	365
Weather Days/year	30
Operating Days/year	335
Scheduled hours/shift	12
Shifts/day	2
Scheduled hours/year	8,040

Source: OceanaGold, 2022

Table 16-17: Factors in Estimation of Potential Operating Hours for a Typical Year

	140 t Truck	175 t Truck	PC3000 Backhoe	PC4000 Face Shovel
Mechanical Availability	85%	88%	85%	88%
Use of Availability	78%	82%	80%	80%
Operating Hours per Year	5,808	6,341	5,985	6,139

Source: OceanaGold, 2022

Waste is drilled with either 171 mm or 200 mm diameter holes and ore is drilled with 171 mm diameter holes. Productivity rates vary depending on drill type and rock type, but the average LoM drilling productivity is approximately 26.5 m/hr across the drill fleet.

Five to seven passes of the primary digging units will be used to load the matching trucks. Annual productivity rates have been estimated from equipment specifications, material characteristics, spot and loading times, truck presentation and primary digging unit propel factors, scheduled hours per year, mechanical availability and use of availability. For PC3000 excavators, the estimated productivity is 1,700 tonnes per hour (t/h). For PC4000 excavators, the estimated productivity is 2,500 t/h.

Truck number estimates are outputs from the mine schedule based on various source/destination combinations, which include:

- Green waste (to waste rock dumps and TSF)
- Red waste (to dedicated PAG cells)
- Yellow waste (to dedicated PAG cells or in-pit dumps)
- Inferred (to relevant waste rock dump or PAG facility)
- Ore (to ROM or to stockpile)
- Yellow waste rehandle (from temporary in-pit dump to permanent in-pit dumps or PAG cells)
- OP ore rehandle (Stockpile to ROM)
- Rehandle underground ore (to ROM)
- Rehandle underground waste (to relevant waste rock dump or PAG facility)

Table 16-18 shows the major mining equipment fleet required to achieve the mining schedule. The open pit mining fleet is relatively new and, accordingly, no replacement of mining equipment will be required. Rebuilds are schedule at appropriate intervals over the LoM timeframe. A down the hole service will continue to be provided by an explosive's supplier.

Ancillary equipment to support the load and haul and drilling fleets includes six tracked and two rubber-tired dozers. In addition, three motor graders cover the pit, dump and surface roads. A FEL provides stockpile and extra loading capability, and two others are assigned to project work or as backup production tools. Three water trucks are assigned to roads and servicing drills. Other equipment includes lighting plants, sumps pumps, fuel truck compactor and light vehicles.

Table 16-18: Major Equipment Required to Achieve the Mine Schedule

Fleet	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
PC4000 Excavator	2	2	2	2	2	2	2	2	2	2	1	1	0
PC3000 Excavator	1	1	1	1	1	1	1	1	1	1	1	1	0
Cat 993 Loader	2	2	2	2	1	1	1	1	1	1	1	1	1
Cat 785 Trucks	3	1	3	3	1	2	1	0	1	1	1	1	1
Komatsu 730 Trucks	19	15	17	18	15	16	15	11	14	12	10	6	0
Sandvik DR410i Drill	4	4	4	4	4	4	4	4	4	4	4	3	0
Epiroc D65 Drill	4	3	2	3	4	4	4	4	4	3	3	2	0

Source: OceanaGold, 2022

16.1.8 Labor

Labor costs on an annual basis have been built up with reference to equipment unit numbers, the roster and requirements for management and technical staff. The roster is a continuous four panel crew roster of 12-hour shifts with two shifts per day. Rostered days off, leave, training, sick leave and absenteeism allowances are accounted for. Operator numbers are based on allocating specific numbers to each equipment item. Staff numbers assume an efficient workforce operating with high level of multi-skilling and flexibility. Table 16-19 shows the required workforce for the year 2022.

Table 16-19: Labor Levels in 2022

Labor Category	Year 2022
Management and Technical Services	
Mine Operations Management	3
Mine Engineering/Geology/Survey	43
Total	**46**
Operations Labor	
Mine Foreman/Supervisors	29
Operators	225
Blast Crew/ Explosives Ops	0
Holiday/Training/Sickness, etc	22
Total	276
Maintenance Labor	
Maintenance Super/Shift Foreman	11
Fitter/Welder/Electrician/Serviceman	82
Maintenance Planner	3
Maintenance Clerk	1
Total	**97**
Grand Total	**419**

Source: OceanaGold, 2022

The staff numbers are then applied against the annual labor cost by job type. These costs have been supplied by Haile operations and are inclusive of on-costs.

16.1.9 Mine Dewatering

The Project area is located within the Carolina Terrane, also known as the Carolina Slate Belt, which is composed predominantly of metavolcanic and metasedimentary rocks. The Site occurs within a strongly deformed ENE-trending structural zone of the Carolina Terrane at or near the contact between metamorphosed volcanic and sedimentary rocks of Neoproterozoic to Early Cambrian age. The relevant stratigraphy of the Project area includes (from youngest to oldest) the Coastal Plain Sands (CPS), saprolite, metasedimentary and metavolcanic bedrock, along with lamprophyre and diabase dikes.

The CPS unit was deposited within the last 100 million years and is discontinuous across the Site. It is present in the highland areas with a thickness of up to approximately 120 ft (37 m), has been eroded from drainages, and thins towards the west. The upper layer is a clean, tan to light brownish yellow, fine- to medium-grained quartz sand. The lower portion of the CPS consists of white to red sand with some clay and silt and the base of the lower layer is characterized as oxide-cemented coarse gravel and sand.

A thick layer of saprolite covers the majority of the Site, underlying the CPS or at the surface where CPS has been eroded. It is a red brown to white, dense, unconsolidated, kaolin-rich clay resulting from intense chemical weathering of bedrock. The saprolite does not exhibit any significant structural features. Saprolite ranges in thickness from approximately 33 to 130 ft (10 to 40 m) and is thickest above metavolcanic rocks and along faults but is absent in isolated areas.

The metasedimentary rocks on Site are part of the Richtex Formation. The metasediments consist of bedded, calcareous, chloritic mudstone and silty mudstone that were deposited conformably on the underlying metavolcanics. The metavolcanics consist of andesitic metavolcanic and tuffaceous rocks and are part of the Persimmon Fork Formation that was formed 530 to 580 million years ago when the

Carolina Terrane was formed as part of a subduction zone, oceanic island arc complex. The metavolcanics are strongly foliated and are metamorphosed to greenschist facies.

Mafic diabase dikes intruded the Carolina Terrane in the Triassic – Early Jurassic about 200 to 250 million years ago. The dikes strike northwest, dip steeply from 60° to 90° and are dark gray, dense and medium grained. They range in thickness from approximately 3 to 33 ft (1 to 10 m) but can be as thick as 100 ft (30 m). Dike contacts can be faulted with tens of meters of dextral displacement. Lamprophyre dikes are also present, trend ENE, and range in thickness from less than 3 to 6 ft (1 to 2 m).

During mining, groundwater flow directions in the CPS and bedrock hydrostratigraphic units generally reflect topography, except in the immediate vicinity of the pits where depressurization pumping has influenced flow direction and increased hydraulic gradients. Hydraulic testing results and geotechnical assessments suggest declining hydraulic conductivity (K) with depth and the division of bedrock into higher K, weathered, fractured unit and underlying lower K, unweathered, more competent unit. The tests completed at Snake Pit area suggest that metavolcanic rocks are slightly more permeable than metasediments. High yielding water strikes in metavolcanics in Snake Pit depressurization borings seem to support slightly higher K in metavolcanics, at least in that area.

Based on monitoring of open pit mining operations data around Mill Zone and Snake Pit, weathered and fractured bedrock transmits the majority of the bedrock groundwater flux across the site. Although the underlying, unweathered/competent bedrock is of relatively low K, the unit can be expected to produce water, particularly as more saturated thickness is intercepted as mine development advances in accordance with the mine plan.

A numerical groundwater flow model was developed for the project. The model represents the identified hydrostratigraphic units as 11 model layers:

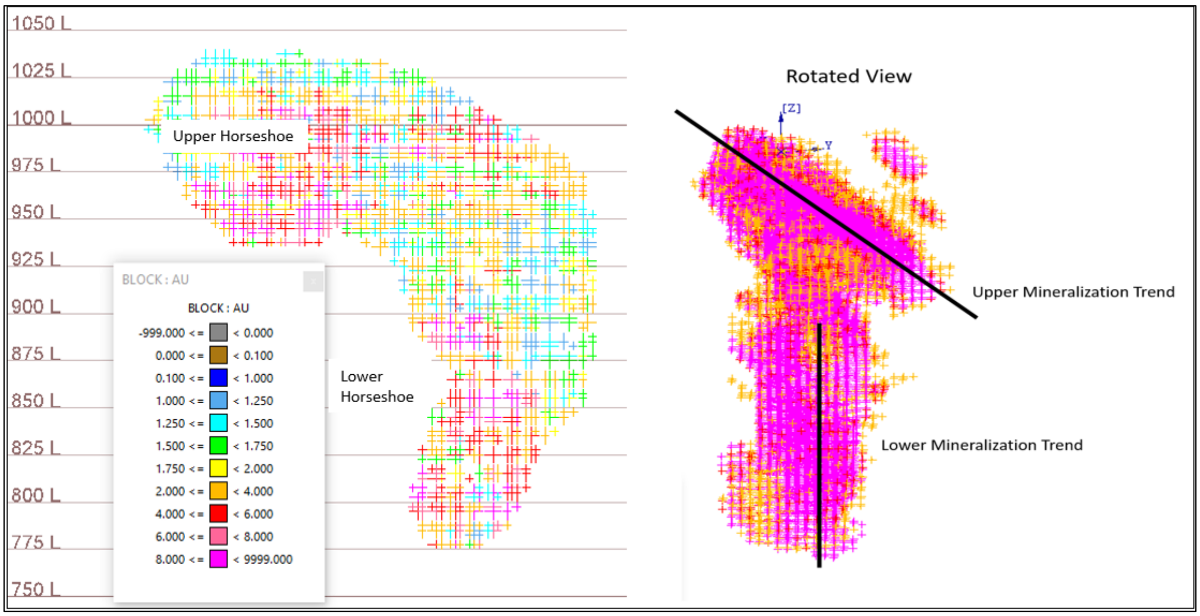
- Layers 1 - 2: CPS
- Layers 3 - 4: Saprolite
- Layers 5 - 6: Weathered, fractured bedrock
- Layers 7 - 11: Unweathered/competent bedrock with a gradational decrease in K with depth

Site stratigraphic data, meteorological data and groundwater pressure/level data from numerous monitoring wells and vibrating wire piezometers installed in each of the hydrostratigraphic units, and groundwater production data, were used to develop the conceptual groundwater model and calibrate the numerical model.

After the model was calibrated under both steady-state and transient conditions, it was used to predict dewatering rates required to facilitate safe and efficient open pit and underground mining. The calibrated model predicts that total annual dewatering rates from the in-pit dewatering sumps and dewatering wells will range from approximately 400 to 1,100 gpm (25 to 72 L/sec) and average around 700 gpm (43 L/sec). On average, the in-pits sumps are estimated to produce approximately 45 percent of the total estimated dewatering volume from the open pits. The timing and volume of extracted groundwater from the open pits is expected to be manageable.

16.2 Underground Mining Methods

The Project is currently being mined as an open pit mine. Economic mineralization extends below and outside of the pit extents. Mineralization is concentrated in two main zones based on vertical position which form a “horseshoe” geometry over a vertical distance of 350 m. Both zones strike NE adjacent to the siltstone-dacite contact, however, the upper zone dips about 40^{0}NW and the lower zone is near-vertical. The upper zone NW-dipping high-grade lenses of mineralization are focused along bedding-parallel foliation with intense silicification. The Horseshoe fault (NE strike, 40^{0}NW dip) juxtaposes the hanging wall of upper Horseshoe against barren dacite with a sill-like geometry. This geometry extends southwestward into the nearby Snake pit. The steeply dipping Lower Zone is adjacent to the sub-vertical contact with barren dacite. This mineralization will be mined as an underground mine and is referred to as Horseshoe. Figure 16-12 shows the Horseshoe upper and lower mineralized zones.



Source: SRK, 2022

Figure 16-12: Horseshoe Upper and Lower Mineralized Zones

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with has been selected. The stopes will be 20 m wide and stope length will vary based on mineralization grade and geotechnical considerations. A spacing of 25 m between levels is used. Cemented rock fill (CRF) will be used to backfill the stopes. There will be an opportunity for some non-cemented waste rock to be used in select stopes based on the mining sequence. The CRF will have sufficient strength to allow for mining adjacent to backfilled stopes. Paste backfill could be used instead of CRF and there is ongoing work investigating this possible change.

16.2.1 Cut-off Grade Calculations

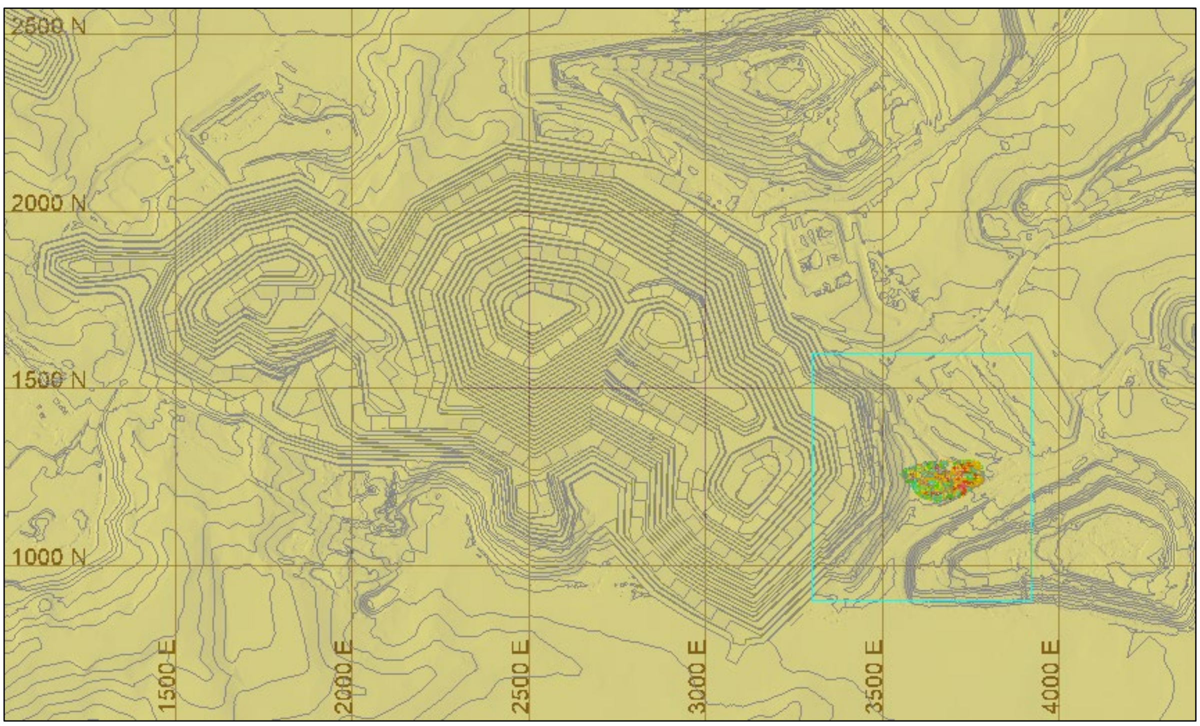
Current estimated project costs and the calculated Au CoG are shown in Table 16-20. For mine design, an elevated cut-off grade of 1.67 g/t Au was used, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.37 g/t Au and an incremental development cut-off grade of 0.46 g/t Au.

Table 16-20: Underground Cut-off Grade Calculation

Parameter	Operating CoG	Incremental Stoping CoG	Marginal Development CoG	Unit
Mining cost (1)	44.98	38.39	-	US$/t
Process cost	11.50	11.50	11.50	US$/t
Tailings	2.44	2.44	2.44	
G&A	5.70	5.70	5.70	US$/t
Total Cost	**$64.75**	**$58.15**	**$19.68**	**US$/t**
Gold price	1,500.00	1,500.00	1,500.00	US$/oz
Average Au mill Recovery (2)	88%	88%	88%	
Smelting & Refining	3.00	3.00	3.00	US$/oz
CoG	**1.53**	**1.37**	**0.46**	**g/t**

Source: SRK, OceanaGold, 2022
(1) Includes backfill
(2) Average stated. Variable recovery is expected based on head grade based on the following equation: (1-(0.2152*Au grade^-0.3696)) + 0.025

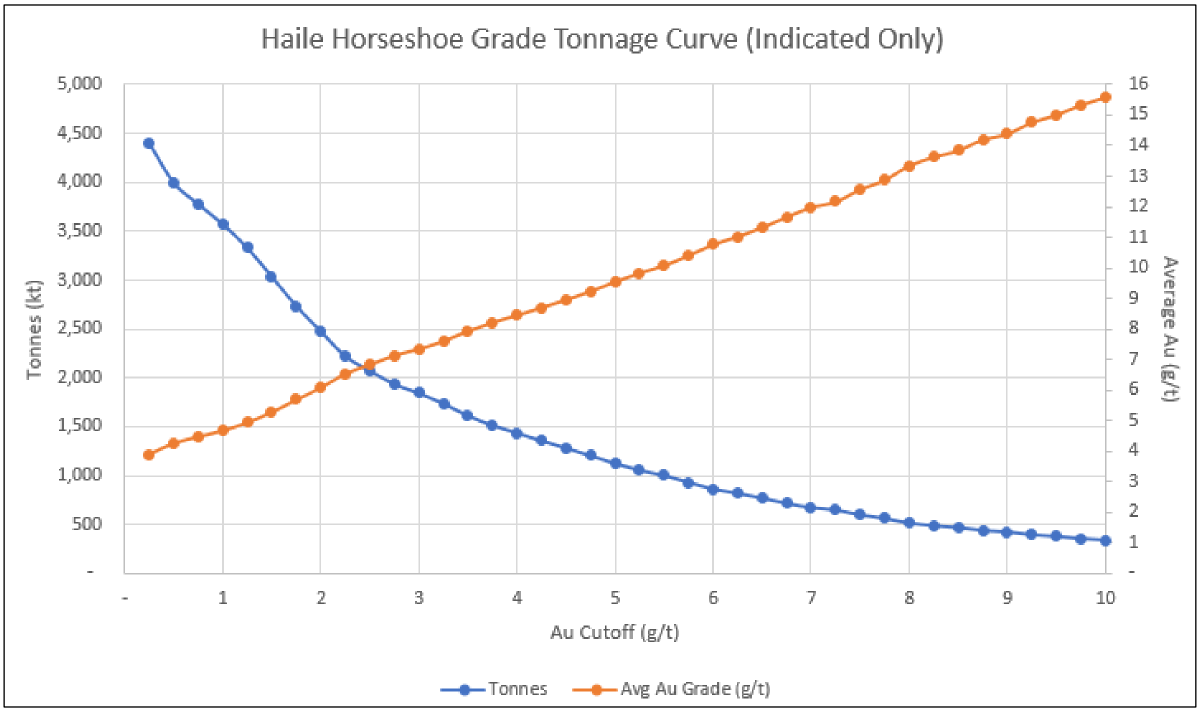
The basis for the FS mine design work is the underground resource model described in Section 15.2. The model is rotated 30° when in site (UTM NAD 83) coordinates to fit the general mineralization/foliation trends. Figure 16-13 shows the block model orientation and mineralized blocks in the underground coordinate system.



Source: SRK, 2022

Figure 16-13: Haile Underground Block Model and Mineralization Extents

Figure 16-14 shows a grade-tonnage curve for the Horseshoe deposit.



Source: SRK, 2022

Figure 16-14: Horseshoe Underground Model Grade/Tonne Curve Based on Au Cut-off

16.2.2 Geotechnical

The 2017 geotechnical field investigation consisted of 14 cored holes designed to examine rock mass fabric and structural features in and around the mineralized zone at different depths and orientations. Since that time, an additional three holes have been drilled/logged geotechnically (DDH-612, DDH-613, DDH-614) totally an additional 456 m.

Drillholes were drilled at varying orientations into the hangingwall, footwall, and mineralized rock. The field investigation included drilling of core, geophysical televiewer borehole logging of structural features, geotechnical core logging, core sample collection for laboratory strength testing. A total of 4,217 m of core was characterized. Two previous geotechnical characterization campaigns were conducted (Golder, 2010; SRK, 2016).

Data from all three campaigns have been combined into a single database. Table 16-21 is a summary of the 7,345 m of core that has been logged for geotechnical characterization. Since that time an additional 3 holes have been drilled/logged geotechnically (DDH-612, DDH-613, DDH-614) totally an additional 456 m. Data from these three holes have not been included in the current geotechnical data analysis. Since the coverage of data from the other 19 holes is adequate for FS design purposes, the geotechnical design parameters based on the database are still considered valid. The field program included in situ stress measurements. A summary of the laboratory testing program for rock strength is provided on Table 16-22.

Figure 16-15 shows a 3D perspective view of the 2017 underground workings with the as-drilled characterization core holes from each drilling campaign. The hole diameters have been enlarged to 40 m to simulate the representative coverage volume for each hole. This figure illustrates that most of the planned mining area and underground infrastructure has been covered by characterization. The updated workings are similar in spatial extents and the geotechnical drillholes have a similar coverage.

Table 16-21: Discontinuity Orientation Data for 2014 Geotechnical*

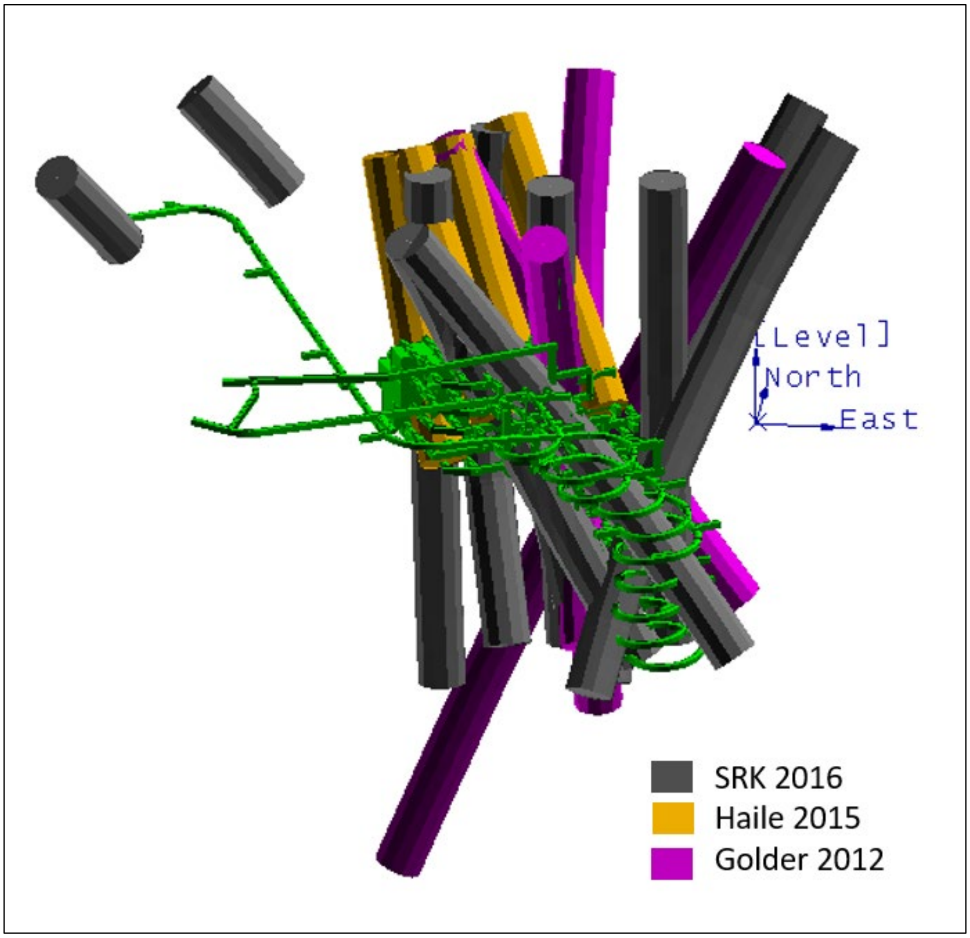
Drillhole ID	Drillhole Length (m)	Total Discontinuities		Total Discontinuities Logged
		Logged	ATV	
RCT-120	475.8	469	-	469
RCT-122	535.8	273	-	273
RCT-123	693.6	277	-	277
RCT-124	461.7	365	-	365
DDH-517	264.0	215	-	215
DDH-522	258.3	381	-	381
DDH-524	179.8	436	-	436
DDH-533	259.1	315	-	315
DDH-601	121.9	179	425	604
DDH-602	406.9	134	648	782
DDH-603	450.5	364	1,347	1,711
DDH-604	519.1	481	717	1,198
DDH-605	504.9	577	-	577
DDH-606	432.2	625	1,021	1,646
DDH-607	113.4	153	300	453
DDH-608	399.9	226	767	993
DDH-609	403.9	137	805	942
DDH-610	438.3	674	1,404	2,078
DDH-611	426.4	651	1,024	1,675
Total	**7,345.5**	**6,932**	**8,458**	**15,390**

*An additional 3 holes have been drilled/logged since totaling 456 m.
Source: SRK, 2017

Table 16-22: Summary of the Laboratory Tests

Year	Laboratory	Rock Type	UCS	UCSM	TCS	BT	DSS	Total
2009 (Golder OP)	ATT	Metasediments	1	1	4		2	13
		Dike	1	1			-	
		Metavolcanics	2	1			-	
2011 (Golder UG)	ATT	Metasediments	12					12
		Dike						
		Metavolcanics						
2016 (SRK UG)	Agapito	Metasediments	7	8	5	1	3	37
		Dike	1	1			1	
		Metavolcanics	2	2	3	1	2	
2017 (SRK UG)	Agapito	Metasediments	11	9	10	10	6	125
		Dike	3	4		2	1	
		Metavolcanics	21	16	15	9	8	
Total			**61**	**43**	**37**	**23**	**23**	**187**

Source: SRK, 2017



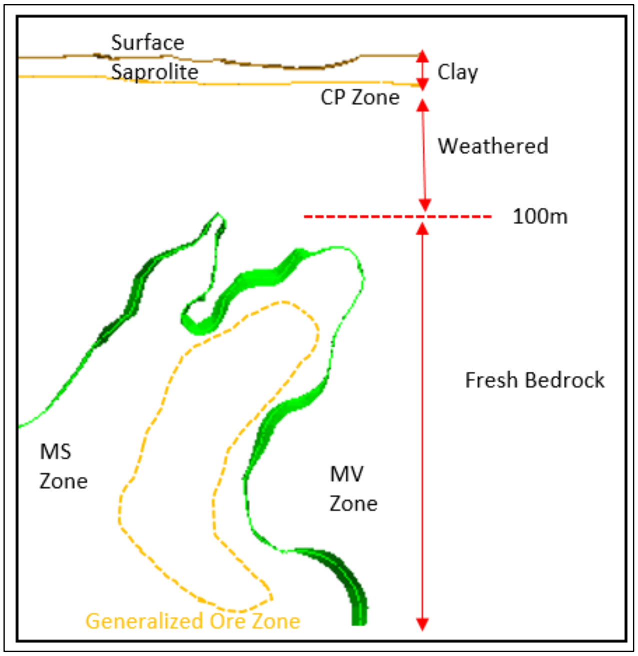
Note: Slight adjustments in the declines (geotechnically insignificant) have been made since 2020
Source: SRK, 2020

Figure 16-15: Horseshoe 2020 Underground Workings with As-Drilled Characterization Core Holes

Rock Mass Characterization

Most of the rock quality designation (RQD) values indicate fair to good rock quality throughout the drillholes (i.e., RQD = 80 to 100). A total of 4,220.5 m of core length were logged with the majority being located within mineralized zone. Areas with lower RQD (i.e., RQD = 10 to 60) were mainly associated with weathered or altered rock and minor geological intrusions (e.g., weathered and fractured metasediments). Fracture frequency in the crown pillar ranged between 0 to 15 fractures per m with an average of 6.5 fractures per m. The fracture frequency in the orebody ranged between 0 to 8.3 fractures per m with an average of 2 fractures per m.

A northwest-southeast cross section through the geotechnical model is illustrated in Figure 16-16. The crown pillar (CP Zone) is the area above the mineralized zone in the fractured metasediments (MS) and metavolcanics (MV). The mineralized zone is in the metasediments. The hangingwall is in the metasediments and the footwall is in the metavolcanics.



Source: SRK, 2017

Figure 16-16: Geotechnical Model Cross Section (Looking NE)

Drill core is fresh in areas within and adjacent to the Horseshoe orebody. Surficial rocks penetrated by the planned decline within 40 m of the surface are variably weathered and clay-altered in saprolite. Field strength tests indicate that the fresh rock is, on average, strong (R4). Fresh rock breaks along pre-existing planes of weakness such as veins, bedding, foliation and dike contacts.

Laboratory uniaxial compression strength test (UCS) results show that the strength of the weathered metavolcanic rocks are in the medium-strong range (UCS = 40 to 78 MPa), and fresh metavolcanic strength range is very strong (UCS = 110 to 195 MPa). The strength of the foliated metasediments range is medium strong (UCS = 32 to 70 MPa), and the non-foliated metasediments strength is considered very strong (UCS = 107 to 150 MPa). The dikes are very strong (UCS = 100 to 190 MPa). Table 16-23 through Table 16-25 provides a summary of the laboratory testing results.

Table 16-23: Summary of Point Load (Is_{50}) Test Results

Geotechnical Unit		Mine Zone	UCS Lab (MPa)	K Value	UCS PLT (MPa)			Field Estimated Strength Category (ISRM)
Lithology	**Weathering**				**Mean**	**Lower Boundary (Px<20%)**	**Upper Boundary (Px<80%)**	
Metavolcanics	Weathered	Crown Pillar	60	18	50	13	85	R2-R3
	Fresh	Hanging Wall	153	27	106	70	145	R4-R5
		Orebody						
		Footwall						
Metasediments	Fresh	Hanging Wall	105	21	70	34	105	R3-R4
		Orebody						
		Footwall						

Source: SRK, 2017

Table 16-24: Summary of Strength Properties (mi and σ_{ci})

Geotechnical Unit		Density (t/m³)	Laboratory Test		Intact Rock Properties from TCS		Elastic Constants	
Lithology	Weathered		σ_t (MPa)	UCS (MPa)	σ_{ci} (MPa)	mi	Ei (GPa)	ν
Metavolcanics	Weathered	2.60	4.4	60	61	14	21.9	0.28
	Fresh	2.70	12.7	153	158	13	73.4	0.24
Metasediments Foliated	Fresh	2.76	6.2	50.5	59	10	41.6	0.24
Metasediments Not Foliated		2.75	11.8	127.8	138	12	58.9	0.24
Dike	Fresh	2.92	12.94	146	-	-	78.7	0.22

Source: SRK, 2017

Table 16-25: Summary of Elastic Properties (E_i and ν)

Geotechnical Unit		Mine Zone	UCS (MPa)			E_i (GPa)	ν
Lithology	Weathering		Mean	Low Boundary (Px<20%)	Upper Boundary (Px<80%)		
Metavolcanics	Weathered	Crow Pillar	57	35	80	25.8	0.21
	Fresh	Hanging Wall	148	105	192	73.6	0.23
		Orebody					
		Footwall					
Metasediments	Fresh	Hanging Wall	92	65	120	62.6	0.21
		Orebody					
		Footwall					
Dike	Fresh		146	100	190	78.7	0.22

Source: SRK, 2017

The Rock Mass Rating (RMR_{76}) values ranged from 49 to 68 in the hangingwall rock, with Barton Q' values varying from 4.3 to 19, with most of the rock mass being of fair to good quality. In the footwall metavolcanics RMR_{76} values were in the 64 to 88 range, with Q' values varying from 7.7 to 30, with most of the rock mass being of Good quality. The mineralized rock had the greatest RMR_{76} variations, with values between 56 and 79, while Q' values ranged from 5.8 to 24. Table 16-26 summarizes the rock mass characterization data. Overall, the rocks are Fair to Good quality. The table also indicates the quantity of core in each domain.

Table 16-26: Summary of Rock Mass Quality by Domain

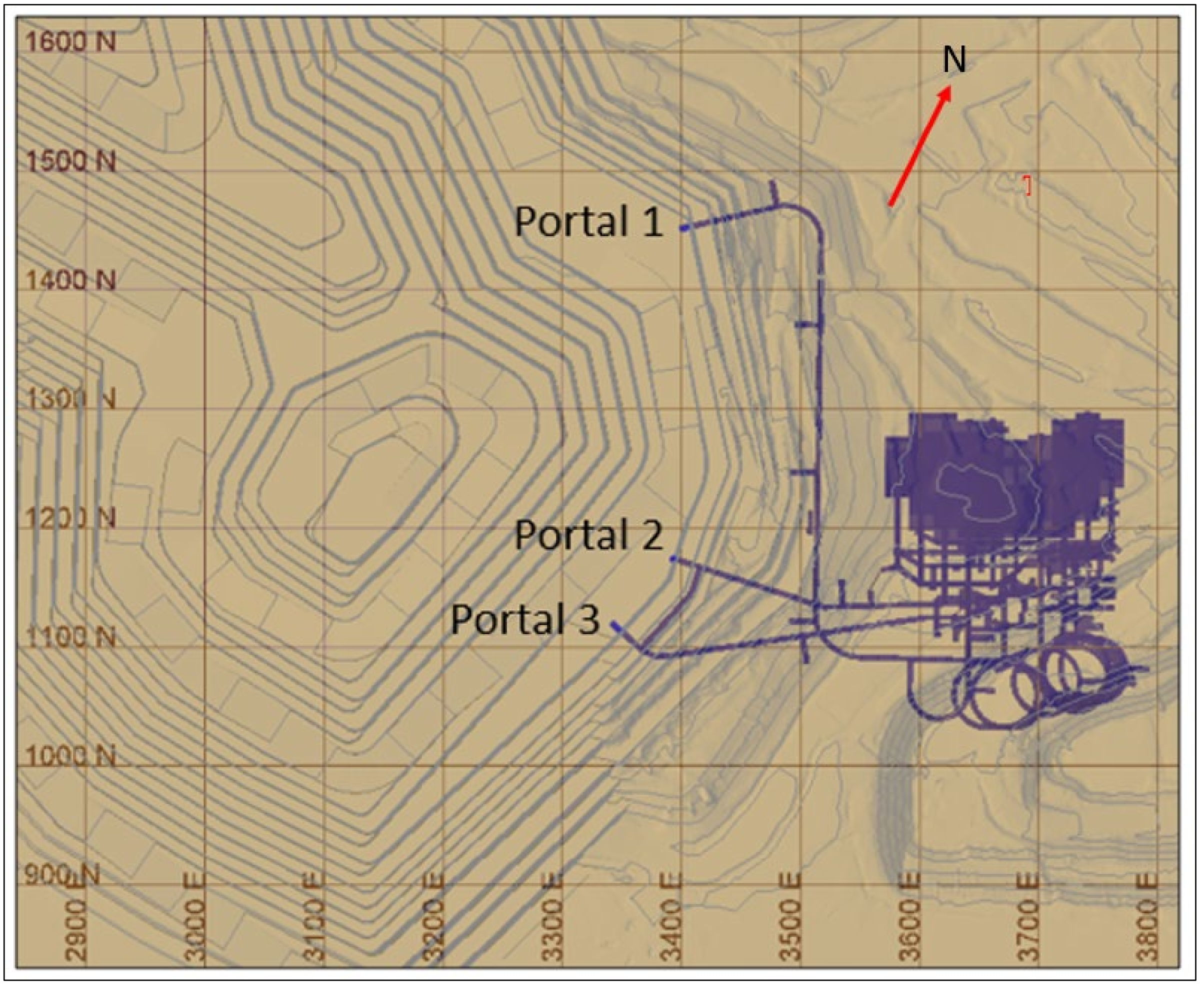
Geotechnical Domain	Lithology	Density (t/m^3)	IRS (MPa)	RQD (%)	Fracture Frequency (FF/m)	RMR76/ GSI	Q’
Crown Pillar (13%)	MV	2.60	13 to 86	61 to 100	0 to 15.0	45 to 69	2.8 to 13.5
	MS	2.63	26 to 64	41 to 91	1.2 to 19.2	43 to 62	2.3 to 8.3
	DB	2.91	22 to 140	72 to 100	0 to 11.8	47 to 68	2.2 to 11
Hanging Wall (22%)	MV	2.69	41 to 219	88 to 100	0 to 3.7	63 to 82	10.7 to 26
	MS	2.76	22 to 97	70 to 100	0 to 10.6	49 to 68	4.3 to 19
	DB	2.96	42 to 173	45 to 100	0 to 26.6	39 to 72	0.9 to 9.6
Orebody (50%)	MV	2.70	86 to 205	85 to 100	0 to 5.8	64 to 89	8.3 to 34
	MS	2.75	37 to 125	75 to 100	0 to 8.3	56 to 79	5.8 to 24
Footwall (15%)	MV	2.69	86 to 197	83 to 100	0 to 6.5	64 to 88	7.7 to 30
Decline	DCL<100m MV	2.60	13 to 121	64 to 100	0 to 15.7	46 to 74	1.7 to 18
	DCL>100m MV	2.70	33 to 159	69 to 100	0 to 9.8	52 to 83	5.2 to 27

Source: SRK, 2017

Mine Design Parameters

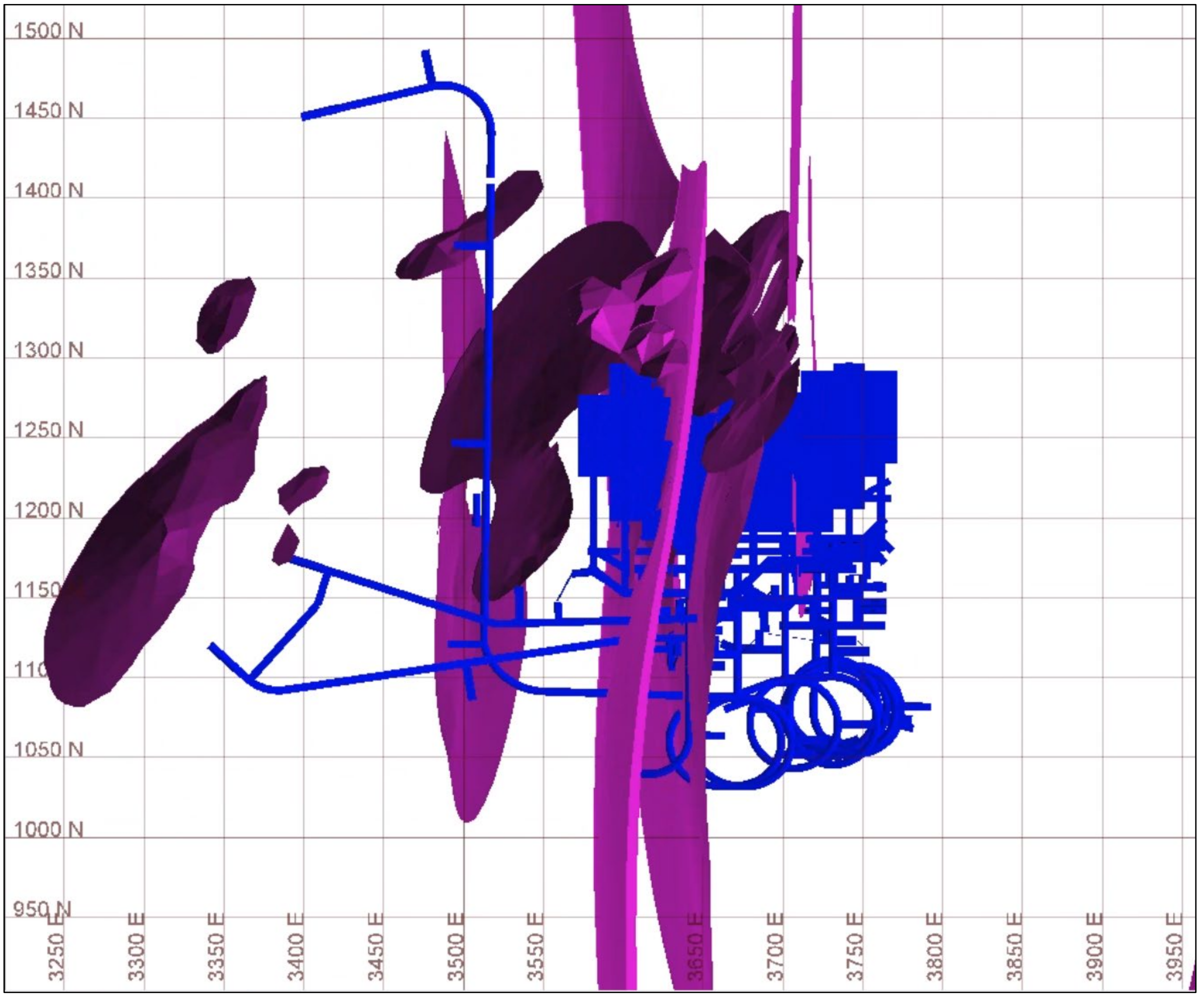
Transverse longhole mining has been selected as the mining method for the entire orebody. This method requires upper and lower access drifts to be developed from the footwall haulages with a proposed 25 m vertical spacing between levels. The total open stope height in 25 m with the overdrive drift above the open stope. The overall open stope dimensions are 20 m wide x 25 m high x 25 m long (maximum length).

Figure 16-17 is a plan view of the ultimate pit outline with the Horseshoe underground workings. The mine coordinate system has been rotated by 30° from true north to simplify the underground workings such that Horseshoe Underground grid north (HUG grid north) is the hangingwall of the deposit and HUG grid south is the footwall of the deposit. The orientation of the stopes is the same as the 2017 design, so the same geotechnical domains apply to the new design. The mining infrastructure (i.e., ramps, declines, ventilation raises, shops, etc.) are in the footwall offset from the stopes to minimize mining-induces stress damage. Figure 16-18 shows the same plan view, but without the pit outline and with the identified dikes as they traverse the orebody and planned mining. The ventilation system raises are in the low-stress stress shadow of stope mining to minimize potential of mining-induced damage.



Note: Slight adjustments in the declines (geotechnically insignificant) have been made since 2020
Source: OGC, 2020

Figure 16-17: Snake Pit Showing Slopes Above Proposed Horseshoe UG Portal Locations



Note: Slight adjustments in the declines (geotechnically insignificant) have been made since 2020
Source: SRK, 2020

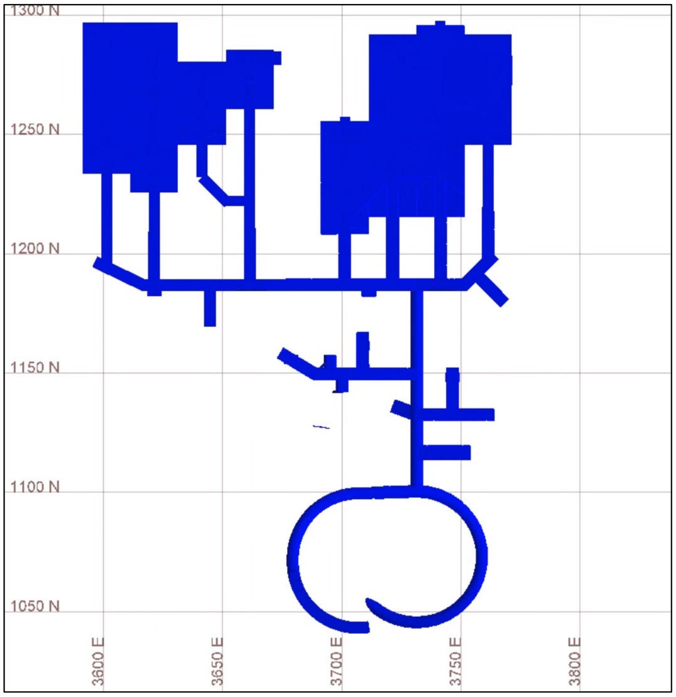
Figure 16-18: Plan View of Planned Workings Showing Dikes (local grid)

Figure 16-19 is a plan section through the 930 Level showing proximity of footwall access drifts to planned stopes. To minimize mining-induced damage to long-term access drifts, the setback distances used in the design include:

- Haulage setback of 18 m to 22 m from stopes
- Main ramp setback of 64 m to 70 m from stopes

These values were also confirmed by the results from the numerical model by evaluating induced shear strain locations.

The MAP3D model results (OceanaGold, 2020b) indicate that the stand-off distance between stopes and footwall drives is adequate, but some footwall drives are likely to experience some stress damage as the stoping front retreats toward the south. Fibercrete is recommended for surface support in the footwall drives. The model result also indicates that the stand-off distance between the stopes and main decline is far enough that the ground conditions in the decline should not be affected by stoping. Welded wire mesh surface support will be adequate in the main decline.



Note: Slight adjustments in the declines (geotechnically insignificant) have been made since 2020
Source: SRK, 2020

Figure 16-19: Plan View of 930 Level Showing Proximity of Footwall Access Drifts to Planned Stopes

Empirical methods of stope design have been employed to evaluate stability conditions. The Stability Graph Method (Mathews et al., 1981), as modified by Potvin et al. (2001), is based on more than 480 case histories worldwide and has been used to size stopes so they remain stable during mining. The stability graph method plots the stability number (N') on the vertical axis against the hydraulic radius (wall area divided by wall perimeter) of the stope wall or back on the x axis. The stability number is calculated based on the rock mass quality (Q' system), geologic structure and induced stress conditions. The stability of the stope walls has been checked using Matthew's method to ensure that remote mining equipment can extract all the ore in an unsupported stope without being buried by falling ground.

Dilution was estimated using the method developed by Clark and Pakalnis (1997) based on an extensive set of case histories for open stopes. The method predicts the quantity of unstable wall rock for a given rock mass quality from a given stope size. The thickness of external dilution is estimated as equivalent linear overbreak/slough (ELOS). Table 16-27 summarizes the ELOS dilution assumptions used for mine planning purposes. ELOS estimates were made for open stopes in each geotechnical domain and at three stress conditions (shallow, medium and deep levels).

Table 16-27: Summary of Dilution ELOS Estimates

Type	ELOS Value (m)
Sidewalls (rock)	0.50
Sidewalls (backfill)	0.25
Endwalls (rock and backfill)	0.15
Bottom (backfill)	0.05

Source: SRK, 2020

Stability of the stopes for the updated underground workings have been checked using a mine-scale numerical model. The MAP3D numerical program was used to simulate the mining sequence. Results of the analysis confirm that the stopes are predicted to remain stable during active mining. The focus of this modeling was to assess the potential for mining double stack primary and secondary stopes. Conclusions from this stability analysis include the following.

- Stress damage to primary stope sidewalls could result in secondary stopes having backfill dilution when secondary stopes are mined, and this problem could be more serious in double-lift primaries since damage is predicted deeper in the sidewalls. If this is a major problem, then the sequence can be adjusted to either single-lift primaries and secondaries, or continuous retreat sequence where slender ore pillars are avoided.
- The model also indicates stress damage in secondary pillars which might result in drilling difficulties in the secondaries. This can be overcome by either using up-hopes or going to "just-in-time" drilling to avoid re-drilling.
- Although modeled conditions need to be verified with experience from early stoping, the experience can be used to modify alternative ground support, stope design and stoping sequence to mitigate stress damage issues depending on actual conditions.

Backfill

The mining method employs primary/secondary stoping sequence. The primary stopes will be backfilled prior to mining the adjacent secondary stopes.

The required strength of cemented rockfill (CRF) placed in primary stopes has been checked for stability under the critical conditions of mining the adjacent secondary stopes. The secondary stopes must remain stable so remote equipment can safely muck the ore without being buried by ground fall. A numerical model was used to assess the minimum backfill strength requirements. The model results indicate that a minimum UCS strength of 0.7 MPa is required after 14 days of curing to accommodate the mining cycle. Laboratory tests results on batches of cemented rockfill indicate that this required strength can be achieved using 4% cement by mass and a water-to-cement ratio of 2.0 with aggregate from waste rock currently available. The CRF mix designs at Haile will be based on two cement contents: 3.8% and 4.8% GP cement by dry mass. The aggregate will consist of NAF waste that is crushed and screened to -80 mm. It will also have a sufficient amount of fines to eliminate vuggy backfill and create a relatively low rill/beach angle for CFR (i.e., less than 20° will be required to achieve 'tight-fill' conditions in the 'stope wings' located below the upper ore drive. Technically, this is cut and fill (CAF) backfill. The 4.8% CRF might have a 28-day UCS ranging from about 1.7 to 3 MPa if the placed mix is relatively free of vugs. For the sill pillar recovery, the curing time will be much longer than 28-days and will likely be even stronger than the 28-day UCS.

Recent testing is being conducted by Outotec (2020) on pastefill material as an alternative to CRF backfill. CRF is assumed as the base case backfill material for the FS design. Should testing indicate that the required 0.7 MPa UCS strength can be developed in 14-day curing, then the backfilling concepts will be advanced. A tradeoff study between CRF and pastefill has been conducted by AMC (2019). That study concluded that the improvement in safety of backfill operations, improvement in fill quality with reduced dilution and the ability to achieve tight filling for improved ground stability make the paste fill option the preferred choice for Haile underground mining options. Some credits from underground tailings disposal and extensions to the underground mine operations will further improve the economics of that choice. However, for this study, CRF is assumed as the base case backfill material.

Ground Support and Drift Stability

For Haile, fibercrete and welded wire mesh was only considered for surface support. Galvanized wire mesh was selected for the main decline and access drives. Black wire mesh was selected for ore drives and slot drives. These four types of drives account for most lateral development meters. Fibercrete was selected for all other types of excavations, e.g., stockpiles, decline intersections, sub stations, pump stations, backfill cuddies, escapeways, RAR/FAR breakthroughs, RAR drives, and footwall drives. These types of drives require more robust surface support for several reasons. For example, footwall drives will have truck loading bays and will also experience stress changes. RAR's have a high potential for steel corrosion. Pump stations and sub-stations cannot be easily rehabilitated once the infrastructure is installed.

In all lateral development, the surface support will extend down to the gradeline (nominally 1.5 m above the design floor elevation). Ground support requirements were estimated using empirical support charts developed by Barton (1974). The method relates the rock mass quality (Q) to the equivalent dimension of the excavation (*De*). *De* is the ratio of the excavation width (D) to the excavation support ratio (ESR) index, which relates the use of the excavation to the degree of safety required. ESR parameters were assigned to each type of excavation within the framework of the empirical method. Table 16-28 is a summary of the drift dimensions for which ground support has been provided.

Table 16-28: Design Parameters for Different Drift Types

Excavation	Type Excavation	Opening Dimension W x H (m)	ESR	D	De
Decline <150 m depth	Long Term (LoM)	5.0 x 5.5	1.6	5.5	3.4
Main Ramp >150 m depth	Long Term (LoM)	5.0 x 5.5	1.6	5.5	3.4
Footwall Accesses	Medium Term (0.5 to 1 year)	5.0 x 5.0	2	5.0	2.5
Stope Accesses	Short Term (1 to 6 months)	5.0 x 5.0	2.5	5.0	2.0

Source: Barton, 1974

Various levels of support were specified for each range of rock quality. Table 16-29 is a summary of rock mass quality for each ground support type. The long-term ground support for access drifts in the footwall area have been specified to maintain safe access to infrastructure and ore haulage from the stopes back to the hoisting shaft while those drifts are still required.

Table 16-29: Summary of Rock Quality for Each Ground Support Type

Drift Type	Ground Condition Type	Rock Zone	Q Class	Q Range	Q Portion of Drifting	Support Category
Decline	1	Metavolcanic Weathered	Very Poor	0.1 to 1	19%	IV
	2		Poor	1 to 4	47%	III2
	3		Fair	4 to 10	24%	II2
	4		Good	>10	10%	I2
Main Ramp	2	Metavolcanic Fresh	Poor	1 to 4	19%	III2
	3		Fair	4 to 10	39%	II2
	4		Good	>10	42%	I2
Footwall Access	5	Metavolcanic Fresh	Poor	1 to 4	19%	III1
	6		Fair	4 to 10	39%	II1
	7		Good	>10	42%	I1
Stope Access	8	Metavolcanics/ Metasediments Fresh	Poor	1 to 4	19%	III1
	9		Fair	4 to 10	39%	II1
	10		Good	>10	42%	I1

Source: SRK, 2017

Specifications for each classification of ground support utilized in distinct types of ground have been revised since the 2017 FS design. The backs and upper sidewalls of capital development will use 2.4 m long by 25 mm diameter fully encapsulated resin bolts. The lower sidewalls of capital development will be supported with 1.8 m long by 46 mm diameter galvanized friction bolts (Split Sets). The resin bolts should have some type of corrosion protection, e.g., epoxy or galvanizing. These types of drives are expected to have a service life of between 2 to 10 years.

The backs and sidewalls of ore drives, and slot drives will be supported with 1.8 m long by 46 mm diameter black friction bolts. These drives are not expected to have a service life greater than two years, thus no need to consider corrosion protection.

Ground support recommendations for the 2020 design considers as “best practice” to install cable bolts at all lateral development intersections, spans wider than 7.5 m, stope brows wider than 4 m, and open stope backs (depending on the span and ground conditions) in accordance with Australian mining industry practices. The ground support design includes specific cable bolt designs using 2 to 15 mm diameter strands of bulbed black cable (standard twin-strand cable bolts). The cable bulbs are 25 mm diameter and spaced every meter along the length of the cable. The cable bolt lengths will be standardized as either 6.3 m or 8.3 m twin-strand units. The standard borehole lengths will either be 6.0 m or 8.0 m long. A 0.3 m tail is required for tensioning one of the two strands with a steel plate, barrel and wedge.

The 8.3 m cable bolts are only used for supporting stope crowns. The 6.3 m cable bolts are used for stope brows, or in lateral development with spans exceeding 7.5 m, e.g. decline passing bays or 3-way intersections.

The cable bolt patterns for lateral development are assume a hypothetical volume of material above the backs. The volume is based on the geometric shape of a parabolic dome with a height equal to one-third of the span. The cable bolts are intended to provide long-term support for deep wedges that may loosen-up due to degradation over time, adjacent blasting, or mining-induced stress changes. The “parabolic dome” design method has been used described by geotechnical consultants in Australia (Rosengren), and in Canada (Pakalnis).

A series of drift layout and ground support drawings have been developed by HGM for the FS Underground project using the parameters in Table 16-28 (OceanaGold, 2020c). Figure 16-20 illustrates the ground support layout for level assesses and decline drifts (5W x 5.5H) with arched back. The following additional design drawings have been developed:

- Face Support (typical)
- Decline Passing Bay (10W x 5.5H)
- Decline Stockpile, Footwall Drive, Ore Drive (first 20 m), Pastefill Cuddy and FAD/RAD Drives (5W x 5.5H)
- Ore Drive beyond 20 m and Cross Cut (4.5W x 5H)
- Pump station Drive (5.5W x 5.5H)
- Escapeway Drive (4.5W x 5H)
- Loading Bay ((5W x 7H)
- Level Sump (5.5W x 5H)
- Slot Drive (5.5W x 5H)
- FAD RAD BT (6W x 5.5H)
- Substation (6W x 5H)
- CRF Stockpile (5W x 7.5H)
- Jumbo Starter Cuddy (6W x 4H)

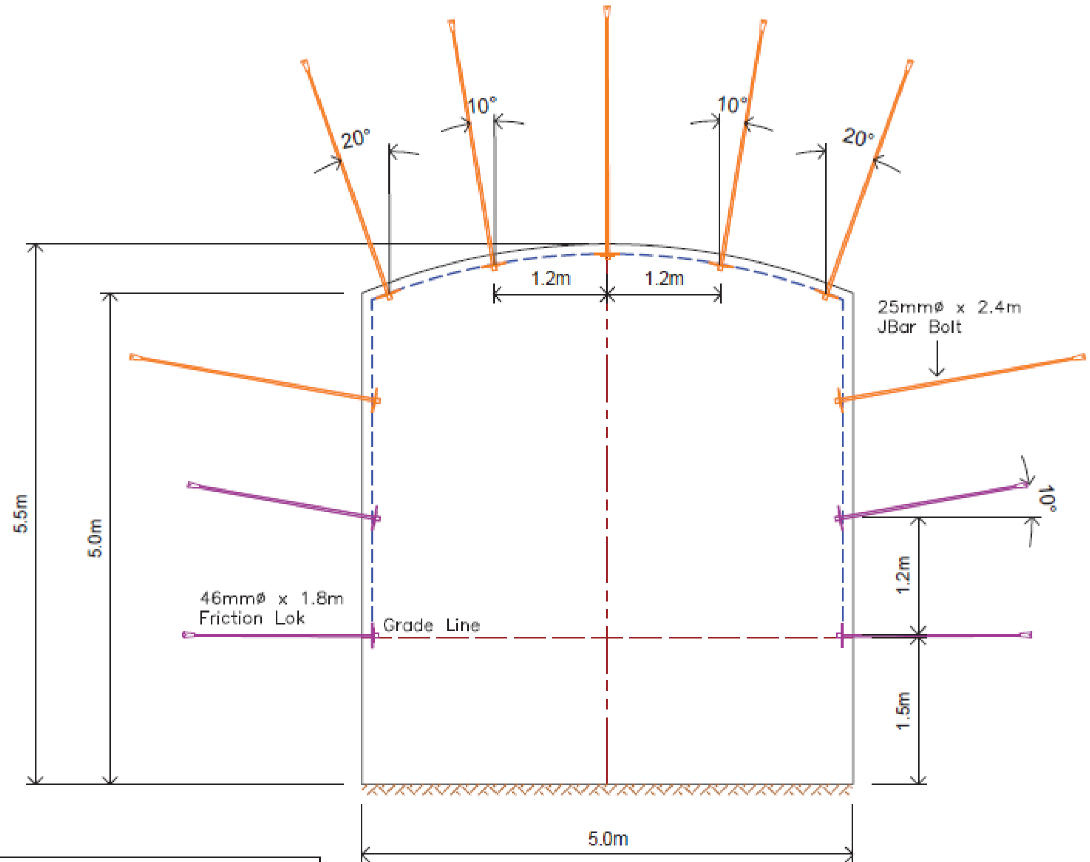
GROUND SUPPORT QUANTITY AND SPECIFICATION

Nominal Cut Length = 4.3 m

JBAR BOLT	Diameter Bolt Length Backs Bolt Spacing Walls Bolt Spacing Ring Spacing	25 mm 2.4 m 1.2 m 1.2 m 1.2 m
RESIN	Size	30mmØ x 2.4m
FRICTION LOK	Diameter Bolt Length Bolt Spacing Ring Spacing	46 mm 1.8 m 1.2 m 1.2 m
WIREMESH	Size Wire Opening	2.8mW x 4.8mL x 5.3 mm Thickness 100mm x 100mm
FIXTURES	Combi Volcano Plate Combi Dome Plate	(300mm x 300mm; 150mm x 150mm) (300mm x 300mm; 150mm x 150mm)
MINIMUM QUANTITIES PER CUT	JBar Bolt Resin Cartridge 1.8m Friction Bolt 0.9m Friction Bolt Wiremesh Combi Volcano Plate Combi Dome Plate	21 pcs. 21 pcs. 12 pcs. 14 pcs. 5 pcs. 21 pcs. 26 pcs.

INSTRUCTIONS:

- Install wiremesh even to the Grade Line.
- Wiremesh overlap must be at least 300mm or 3 mesh squares
- Install additional ground support when required.
- Face Support Standard is in a separate drawing



NOTE:
This profile will be used for Level Access and Decline drives.

DRIVE DESIGN (FRONT VIEW)

GEOLOGY SUPERINTENDENT.	OPERATIONS SUPERINTENDENT.	PROJECT MANAGER	PROJECT DIRECTOR
DATE :	DATE :	DATE :	DATE :

GSS_A_WM
Level Access and Decline
5.0mW x 5.5mH Arched Back

OCEANAGOLD

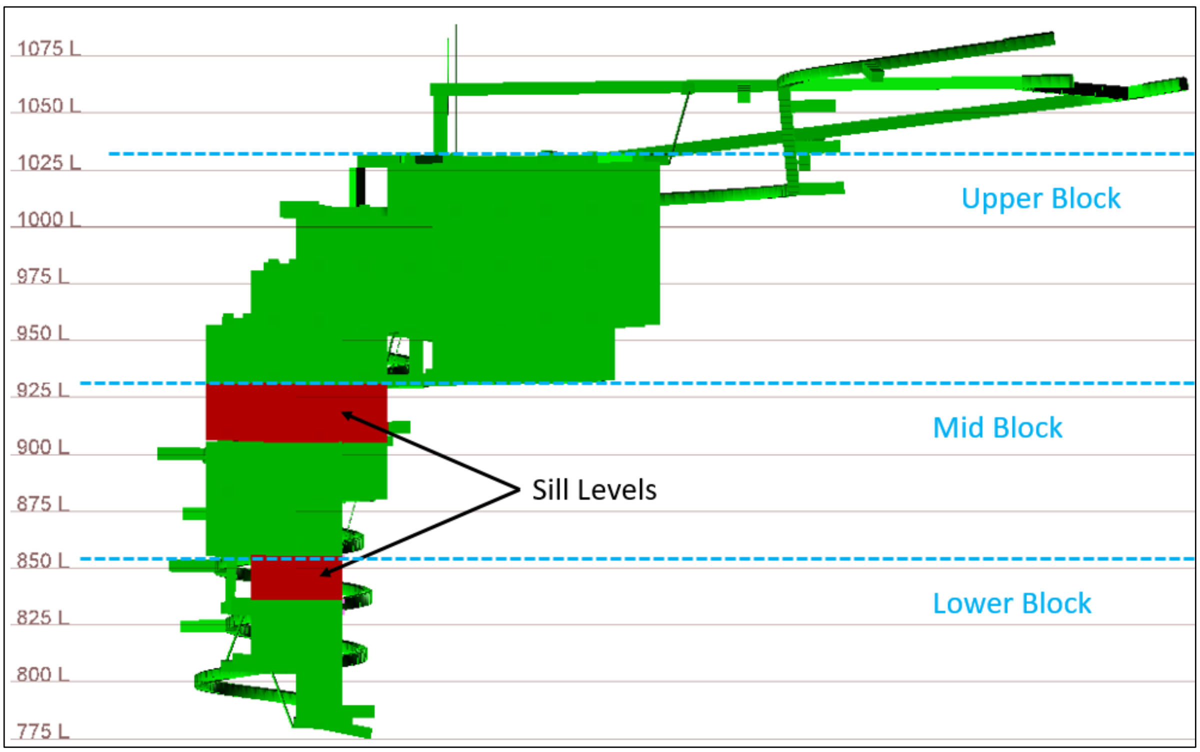
Source: OceanaGold, 2020

Figure 16-20: General Ground Support Drawing for Level Assesses and Decline Drifts with Arched Back

MAP3D numerical modeling results predict that altered dike intersections with lateral development should anticipate stress damage along ore drive sidewalls, backs, and footwall /over-drive bullnose pillars. The altered dike intersections with lateral development is only expected about once on each level. The stress damaged areas can be managed by using fibercrete for surface support.

Sill Pillar Stability

The updated design does not include leaving a 5 m permanent sill in the design. HGM has added to the updated design two temporary sills, shown on Figure 16-21, which will be extracted late in the life of the mine. This allows mining to spread out across three panels. For the CRF scenario, the recoveries of the temporary sill levels have been reduced by 25% because the sill will be mined by a room-and-pillar method (i.e., no filling) with island pillars on backfill or by small modified open stope method. This is due to not having top access and not being able to fill these stopes. If pastefilling is adopted, it will be easier to backfill these stopes since leaving unbackfilled areas is unfavorable for stability. Once mining experience is obtained, sill mining strategy will be reassessed.



Source: SRK, 2020

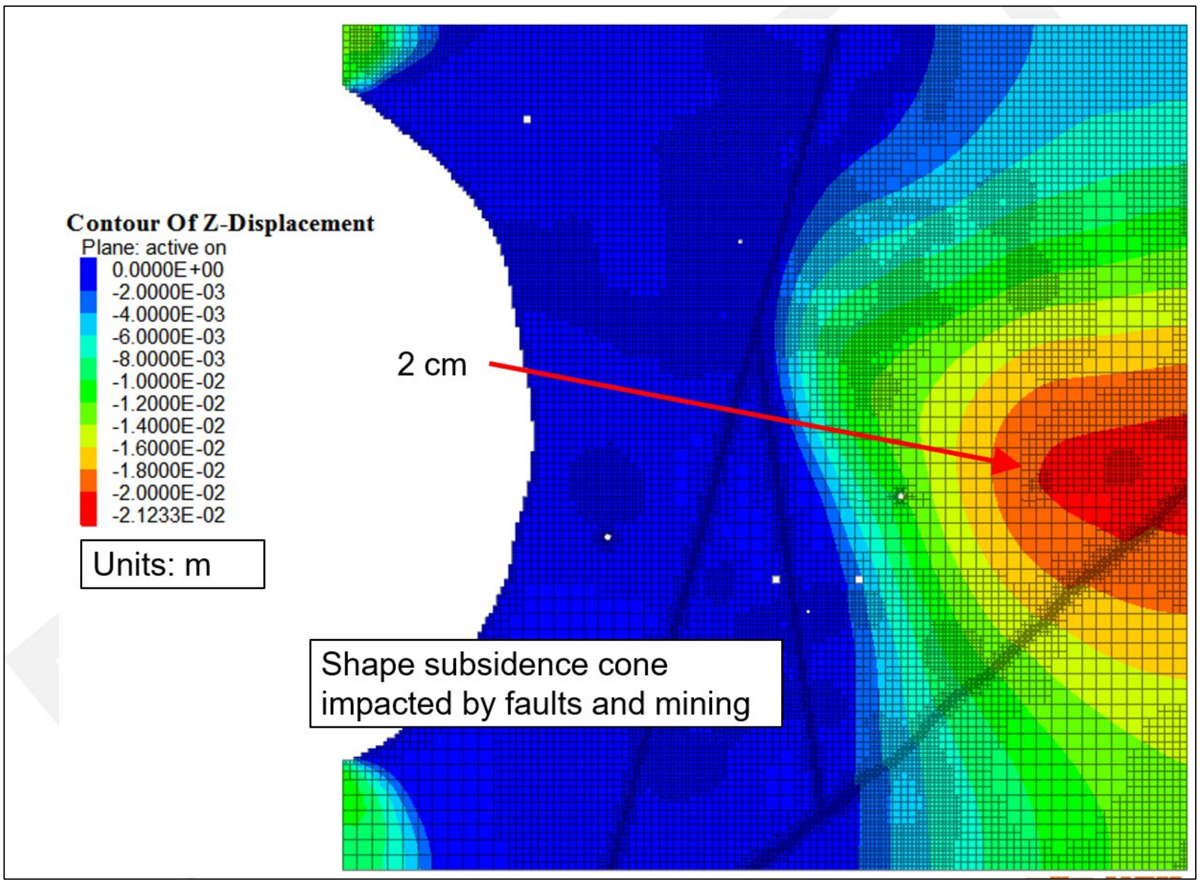
Figure 16-21: Sill Pillars (Red Stopes) to be Extracted Late in Mining Sequence

Crown Pillar Stability

The crown pillar above the mine has been checked for stability against ground collapse back to the surface. The estimated factor of safety is estimated using empirical crown pillar stability methods (Carter, 1980) to be significantly greater than 1.5, especially considering the top stopes will be tightly backfilled. The stability was confirmed by numerical modeling, which predicts no rock mass failure in the crown pillar area above mining and the stress arch above stopes is maintained.

Surface subsidence has been analyzed in the FLAC3D numerical model to assess the potential for damage to the infrastructure above the mine or inflow of surface waters. Figure 16-22 shows the predicted subsidence above the underground mine. This analysis evaluated the 2017 stope layouts but results for the updated 2020 layout are expected to be very similar since stoping areas are similar, although the stoping sequence is slightly different. The extents of the nearby Snake pits are seen on the left-hand side of the figure as are the faults simulated in the model. The maximum predicted vertical subsidence above the mine will be less than 2 cm. The location of the maximum subsidence is influenced by movement along faults that intersect mining.

The recent MAP3D numerical modeling results indicate that relatively small zones of stress-damaged rock (up to 10 m) should be anticipated in the crown pillar, sill pillars and along abutments. These zones can be managed with cable bolts and backfill. Tight-fill in the stopes does not need to be perfect, but it is required to minimize dilution from the backs while extracting secondary stopes.



Source: SRK, 2020

Figure 16-22: Plan View of Vertical Displacements Representing Surface Subsidence

Portal Stability

Three access portals are planned, as shown on Figure 16-23. The most northern portal will be the first to be driven from the pit wall area. Batter face ground support design for Portal 1 includes the following (OceanaGold, 2020d):

- Shotcrete-sprayed safety bund (windrow) along the 1,100 m berm to provide additional catchment capacity and prevent falling rocks from rolling down the portal. The windrow will have built-in drainage holes to manage running water during rainfall events. The windrow has a nominal height of 1.5 m, and 20 m long. This can be extended if required.
- There are four rows of triple-strand 8.3 m cable bolts along 1,097 m, 1,093 m, 1,089 m, 1,085 m.
- The cable bolts are spaced 4 m vertical, and 2 m horizontal. The cable bolts will stabilize the largest wedges and can provide an FoS = >2 for all underground portals.
- Rock bolts and friction bolts will be installed along 1,087 m, 1,083.5 m and 1,082 m to provide ground support for smaller scale structural failures and to pin the drape mesh at the toe of the batter.
- The entire batter face, which has a dimension of 20.5 m slope distance and 20 m wide will be sprayed with 100 mm shotcrete. This will become an effective surface support.
- The entire batter face will be covered with drape mesh. The mesh will be pinned by cable bolt plates, rock bolt plates, and friction bolt plates.
- The batter of the portal will have effective drain holes. The drain holes are 12 m deep, to drain water at the back of the slope. The spacing of the drain holes is 4 m vertical and 4 m horizontal. The first row of drain holes is at 1,094 m and the second row is at 1,090 m. The friction bolts installed at the toe of the batter will serve as drain holes.

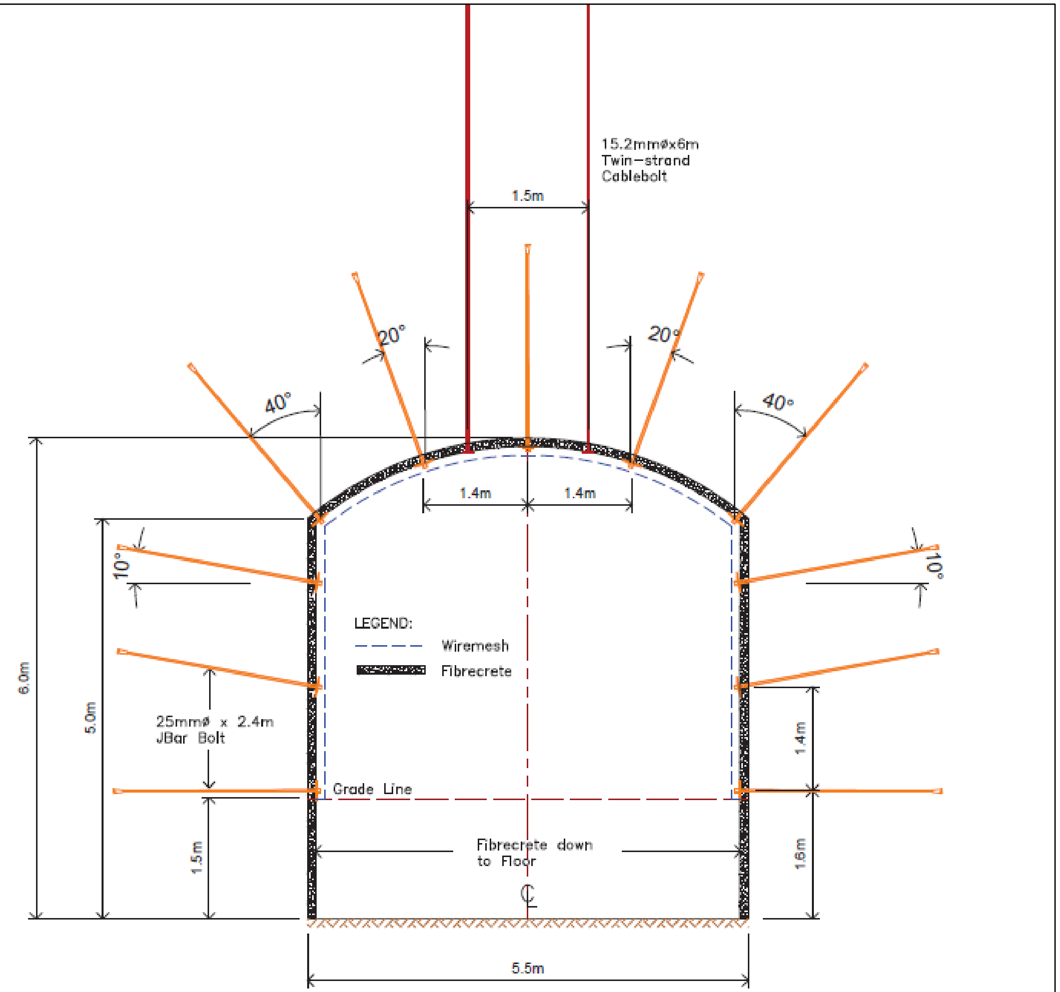
The ground support design for the decline (5.5W x 6H) at the Portal is illustrated on Figure 16-23.

GROUND SUPPORT QUANTITY AND SPECIFICATION

Nominal Cut Length = 2.5 m

JBAR BOLT	Diameter Bolt Length Backs Bolt Spacing Walls Bolt Spacing Ring Spacing	25 mm 2.4 m 1.4 m 1.4 m 1.2 m
RESIN	Size	30mmØ x 2.4m
TWIN-STRAND CABLE BOLT	Diameter Cable Length Cable Bolt Spacing Cable Ring Spacing	15.2 mm 6.3 m x 2 1.5 m 2.5 m
WIREMESH	Size Wire Opening	2.8mW x 4.8mL 100mm x 100mm
FIXTURES	Combi Volcano Plate Combi Dome Plate Flat Plate	(300mm x 300mm; 150mm x 150mm) (300mm x 300mm; 150mm x 150mm) 200mm x 200mm x 12mm Thick
FIBRECRETE	Minimum Thickness Minimum Shotcrete Volume	**100 mm** **14 m3**
MINIMUM QUANTITIES PER CUT	JBar Bolt Resin Cartridge 0.9m Friction Bolt Cablebolt Wiremesh Combi Volcano Plate Combi Dome Plate Flat Plate	33 pcs. 33 pcs 15 pcs 2 pcs. 5 pcs. 33 pcs. 15 pcs. 2 pcs.

INSTRUCTIONS:
- Spray Fibrecrete to the floor
- Install Jbar Bolt 1.6m from the floor.
- Install additional ground support when required.
- Face Support Standard is in a separate drawing



GEOLOGY SUPERINTENDENT.	OPERATIONS SUPERINTENDENT.	PROJECT MANAGER	PROJECT DIRECTOR	GSS_Z_FC Portal 5.5mW x 6.0mH Arched Back	OCEANAGOLD
DATE :	DATE :	DATE :	DATE :		

Source: OceanaGold, 2020

Figure 16-23: Ground Support Drawing for Decline Drift at Portal 1

Similar design and approach will be implemented in Portal 2 and Portal 3. It is planned that all the ground support requirements of the batter be carried out during pit mining when actual ground conditions are known.

16.2.3 Hydrogeology and Mine Dewatering

The hydrogeology of the Site is summarized in Section 16.1.9. The planned underground mine workings will extend to a depth of approximately 1,150 ft (350 m) below ground surface and will be accessed by a decline from Snake Pit. The planned mine development will intersect the weathered, fractured bedrock and the underlying unweathered/competent bedrock hydrostratigraphic units. As described in Section 16.1.9, weathered, fractured bedrock will likely be the predominant source of groundwater inflows to the underground workings. Annual dewatering of the underground workings will be accomplished by capturing water entering the tunnels and pumping the water to the surface of the open pit where the water will be evacuated. Dewatering rates from the underground workings have been estimated using the groundwater numerical model (Section 16.1.9) and range from approximately 200 to 330 gpm (13 to 21 L/sec), with an average rate of 290 gpm (18 L/sec). The timing and volume of extracted groundwater from the underground workings is expected to be manageable.

16.2.4 Geochemical

The current mine design calls for 822 kt of waste rock to be mined from the underground (Table 16-34), which is approximately 19% of the total combined ore plus waste rock that will be mined. The waste rock classification scheme to determine acid rock drainage and metal-leaching (ARDML) properties for the underground mine waste rock will be the same as that used for the open pit. PAG / non-PAG categories will be classified as either green, yellow, or red as described in Table 16-10, and the material will be handled and deposited following the same protocol as open pit waste, as shown in Table 16-10. Initial indications are that most of the underground development rock could be classified as green. However, until more detailed geochemical characterization testing is completed, underground waste rock will be assumed to be yellow or red PAG (B. Schafer, personal commun., 2021).

Some of the waste rock removed is expected to be returned to the underground as backfill. CRF is assumed as the base case backfill material for the FS design, but cemented paste backfill is also being evaluated as an alternative. Geochemical testing of paste backfill is ongoing to determine if the material is favorable environmentally (B. Schafer, personal commun., 2021).

Previous Work

Schafer Limited LLC (Schafer) has been assisting HGM with the overburden management program since 2010 and has released several reports describing the research and application of the program (Schafer, 2010a, 2010b, 2012, 2013, 2015). The program has proven successful in containing waste rock discharge, and there have been no reports of release of acidic drainage from any of the PAG waste rock facilities. The underground testing program has not yet progressed to the point where the PAG properties of underground waste rock can be considered characterized.

In support of the 2016 PEA (SRK, 2016a), SRK coordinated a geochemical characterization program of sampling and analyses with the objective of determining the ARDML potential of underground development rock, tailings, and CRF. The program consisted of collection and analyses of 24 samples, including eight that were representative of the Horseshoe deposit (SRK, 2016b).

SRK coordinated additional geochemical characterization work in support of the FS, consisting of collection and analyses of 76 drill core samples (SRK, 2017). This work was a continuation of the site wide geochemical characterization by previous investigators and the PEA level characterization of the three underground targets (Horseshoe, Mustang and Mill Zone Deep) performed by SRK in 2016 and expanded the scope to provide detailed geochemical characterization data for development rock from the Horseshoe underground area. Samples were selected to provide the full range of the principal lithologies and to cover the vertical range of each area. All proposed samples were within approximately 50 m of the anticipated mining infrastructure under the proposed mine plan in place at the time.

Results of SRK's geochemical characterization program of underground development rock were consistent with those of HGC's overburden waste rock characterization program and are summarized below.

- Total sulfur in development rock ranged from less than 0.005% to 2.45%. Average concentrations varied significantly by lithologies with values of 0.10%, 0.7%, 0.43% and 0.06% for the Diabase dike (DB), Metasediments (L and LC.LS), and Metavolcanics (MV), respectively.
- Almost all sulfur occurs as sulfide, especially at higher total sulfur concentrations (SRK, 2017). Schafer (2015) showed in the overburden characterization that sulfide S levels average 94% of total S values.
- Carbonate minerals provide NP, but silicate mineralization is considered the dominant source of acid neutralization, especially in the low ANP metavolcanic unit (B. Schafer, personal commun., 2022).
- Carbon is mostly inorganic (TIC). Although organic carbon is neglibible, local graphite introduces a slight complication in that graphite will report in the laboratory analyses as organic carbon since it is not removed by the weak hydrochloric acid of the method (B. Schafer, personal commun., 2022).
- The majority of MV rock (approximately 75%) is non-PAG (Figure 16-8).
- Approximately two-thirds (64%) of the L rock are PAG.
- The majority of the coarse / clastic metasediments (LC) are non-PAG.
- The metasediments with secondary silicification are predominantly PAG (Red) or potentially PAG (Yellow) (66%).
- Most Red PAG samples are located in the Upper Persimmon / lower Richtex formations.
- Access and vent workings proposed in the updated mine plan will be driven in the metasediments, which was extensively characterized by geochemical testing and is not believed to present a significantly greater risk than the same material in the earlier mine plan. However, sampling is recommended prior to and during development to obtain confirmatory data on the ML/ARD properties of the new areas.
- The CRF rock samples exhibited significantly greater total sulfur and sulfide concentrations than FS development rock, with sulfide in CRF samples ranging from 0.01% to 4.42% and an average value of 1.94%. Further investigation determined that overburden materials collected for CRF backfill testing were more representative of Red PAG rock than Green non-PAG rock.
- Overburden materials collected for CRF backfill testing were more representative of Red PAG rock than Green non-PAG rock as originally intended. Controls will need to be implemented to ensure that only Green non-PAG rock is used for CRF backfill.

16.2.5 Stope Optimization

Stope optimization was completed on prior versions of the model. Results from those prior runs were used for comparison during the mine design process.

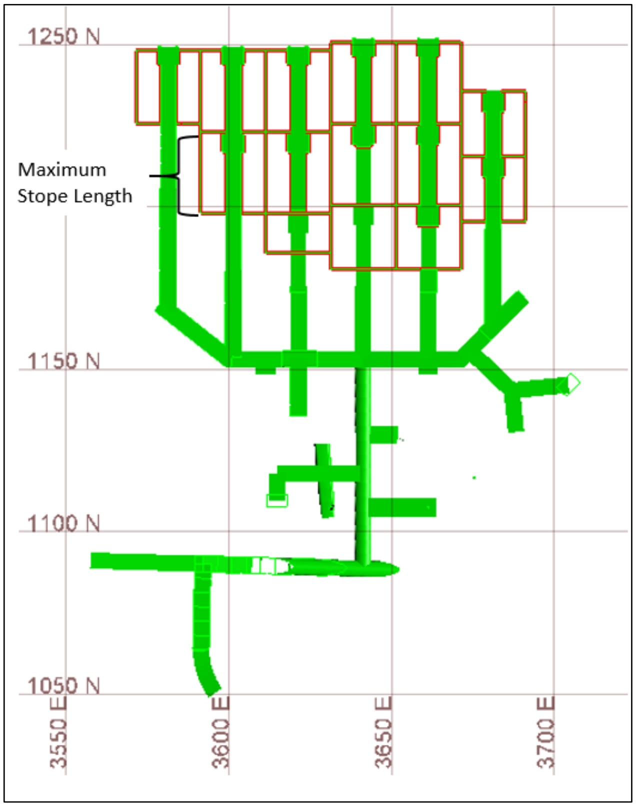
16.2.6 Mine Design

Stope Design

For the design, vertical slices were created through the orebody along 2 m strike length intervals. The model was then interrogated, filtered on cut-off, and then the slices were combined to create minable stopes.

Both top and bottom stope access are designed, as mucking will occur from the lower access and drilling/backfilling will occur from the upper access. Slashing of drifts in necessary areas has also been included in the design. Stope accesses are expected to be in waste until they intercept the stoping block, but grade control will be used to determine the exact ore/waste boundary during mining.

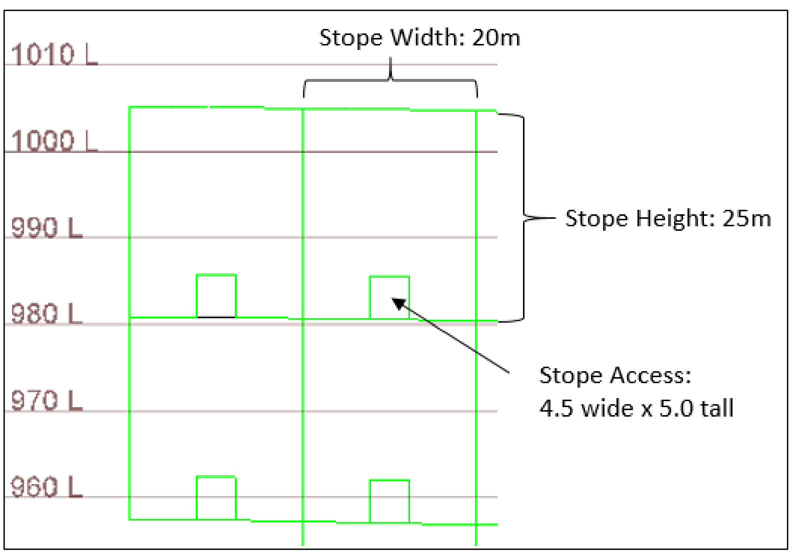
A typical level in the upper zone is made up of approximately eight stopes across, while the lower zone has approximately two to four stopes across. The length of stopes is limited by geotechnical stability and often several stope cuts are taken as shown in Figure 16-24. A primary/secondary stoping sequence will be used where, on any given level, primary stopes must be separated by a secondary stope. Extraction of the secondary stope can only occur after the two immediately adjacent primary stopes have been mined, backfilled and have had time to cure. Backfilling will be an integral part of the mining cycle as there is a limited quantity of stopes available on each level.



Source: SRK, 2020

Figure 16-24: Level Cross Section of Stopes

Figure 16-25 shows a cross section of the stopes.



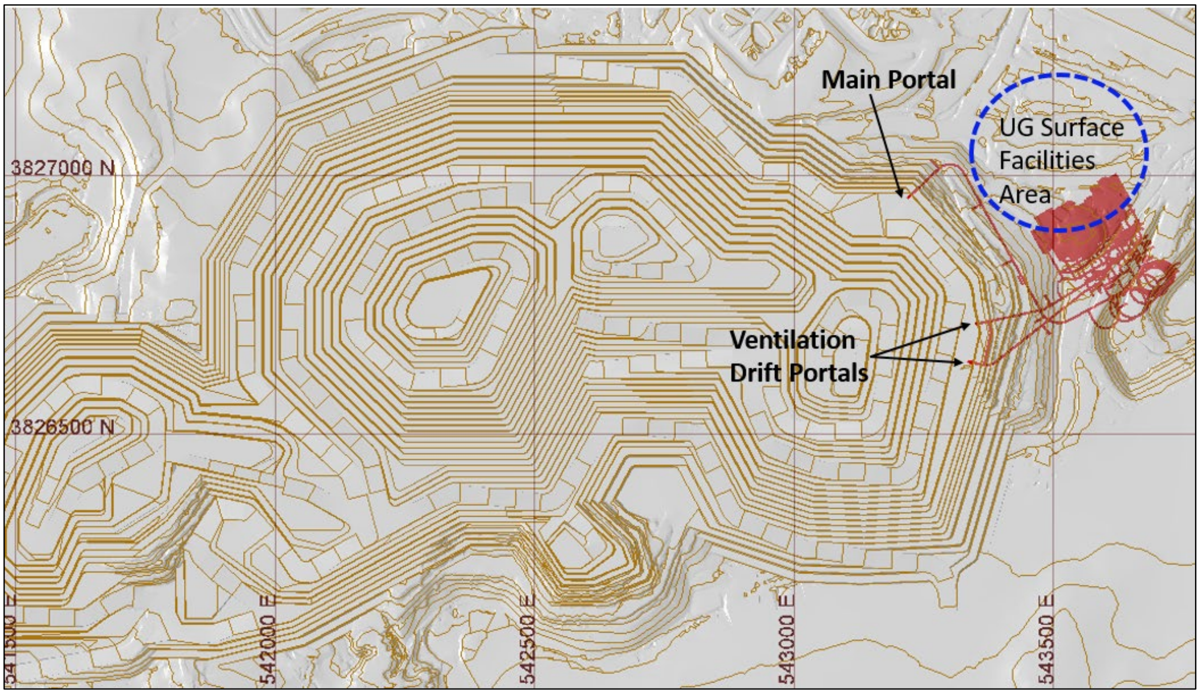
Source: SRK, 2020

Figure 16-25: Stope Cross Section

Stopes are developed using a slot as discussed in Section 16.2.9. Separate slot triangulations were not constructed for each stope, but the slot tonnage of each stope is separated out and a slot activity is used for scheduling.

Development Design

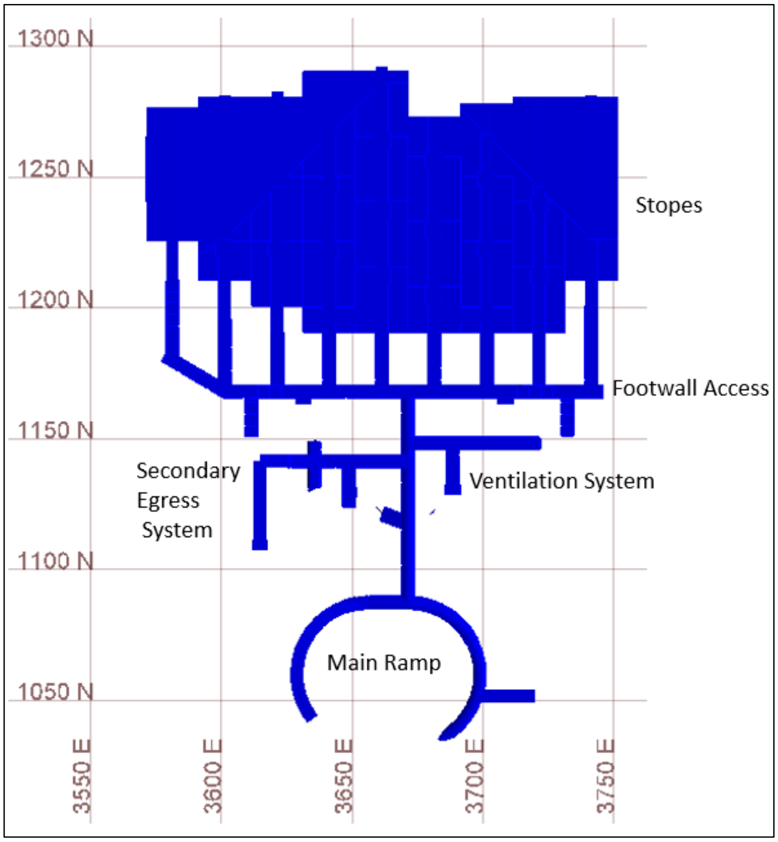
The underground mine is accessed via a decline from the surface. The decline portal is located on an open pit bench approximately 80 m below the natural surface as shown in Figure 16-26. Two ventilation drift portals are also located on an open pit bench.



Source: SRK, 2022

Figure 16-26: Underground Portal Location (shown in the OP coordinate system)

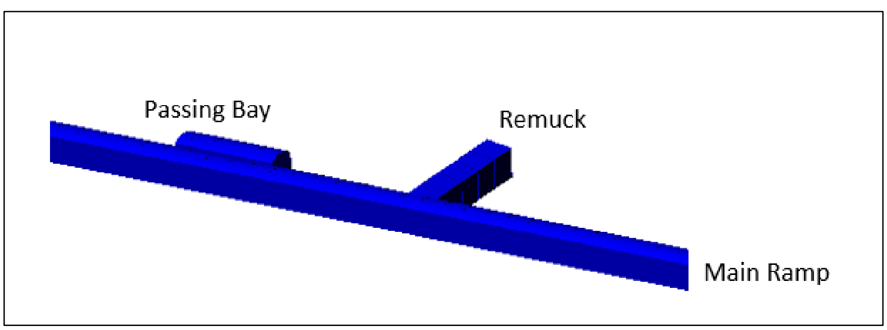
The stope accesses are connected to a level access located in the footwall in waste material. The level accesses are offset a minimum of 25 m from the stopes. The level accesses connect to the interlevel ramp system which is located in the footwall and is offset approximately 75 m from the footwall accesses. Each level is connected to the ventilation system and emergency egress system. Figure 16-27 shows a typical level section.



Source: SRK, 2022

Figure 16-27: Typical Level Section

The design completed is detailed and includes remucks, passing bays, sumps, etc. 16-28 shows a portion of the main ramp with these items designed. As such, no additional allowances were applied to the design.

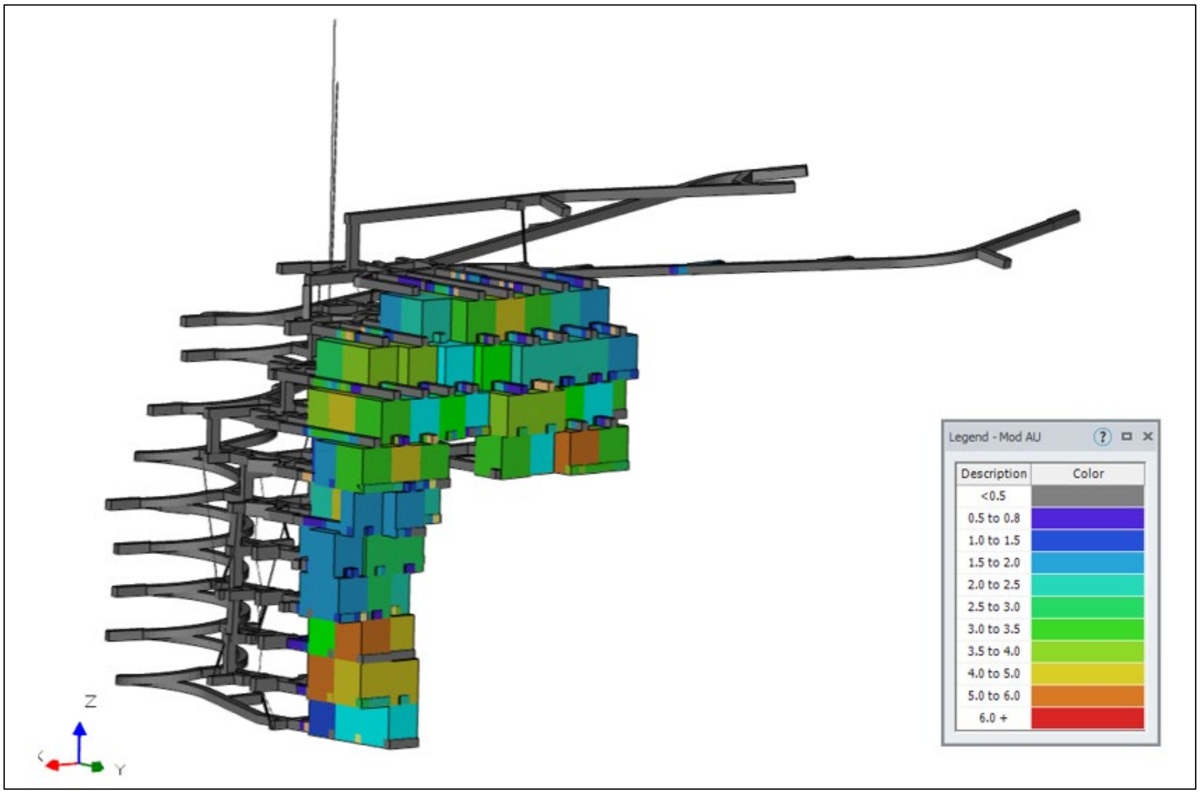


Source: SRK, 2022Figure 16-28: Main Ramp Design Detail

All planned maintenance will be on the surface and underground shop facilities are not included in the design. The CRF facilities are also located on the surface and no additional infrastructure is required underground.

Where possible, accesses/ramps have been designed to be located in the metavolcanics and away from known dikes. Where ramps must cross a fault/dike, the crossing is designed perpendicular to the structure to minimize the length of development through these structures.

Figure 16-29 shows the completed mine design colored by Au.



Source: SRK, 2022

Figure 16-29: Horseshoe Completed Mine Design, colored by activity type (looking South)

Table 16-30 summarizes the mine design by activity type.

Table 16-30: Mine Design Summary – by Activity Type

General Summary	Value
Ore Tonnes (kt)	3,416
Ore Au (g/t)	3.78
Waste Tonnes (kt)	822
Total Tonnes Moved (kt)	4,238
Ore Summary	
Development Ore Tonnes (kt)	262.8
Stope Production Tonnes (kt)	3,153.6
Horizontal Development Summary	
ACC - Level Access	697.1
CRF - CRF Stockpile	234.7
DEC - Decline	2,661.9
DSP - Decline Stockpile	261.4
EWD - Escapeway Drive	220.9
FAD - Fresh Air Drive	734.9
FWD - Footwall Drive	1,014.8
LSM - Level Sump	223.9
OD - Ore Drive	5,374.5
PSN - Pump Station Drive	36.0
RAD - Return Air Drive	798.8
SLD - Slot Drive	903.6
SUB - Substation	133.4
XCT - Crosscut	78.4
Total Horizontal Development Length (m)	**13,374**
Vertical Development Summary	
DH_ Drainhole	236.5
EWR - Escapeway Rise	292.5
FAR - Fresh Air Rise	106.3
FH_ Fill Hole	0.0
HV_ Cable Hole	0.0
OP_ Orepass	0.0
RAR - Return Air Rise	228.0
RMH_ Rising Main	306.6
SH_ Service Hole	235.9

Source: SRK, 2022

Waste material within the design was not characterized geochemically in detail, but rather a conservative approach was taken assuming 75% of the underground waste rock will be hauled to a PAG rock location on surface and that only 25% would go to the ROM pad and be used back underground as backfill. Geochemical samples taken near underground workings locations in a 2016/2017 study indicate many areas would be characterized as Green, non-PAG material. Further classification work of the waste rock should continue as the project moves forward.

16.2.7 Productivities

Productivities were developed from first principles. Input from mining contractors, blasting suppliers and equipment vendors was considered for key parameters such as drilling penetration rates, blasthole size and spacing, explosives loading time, bolt and mesh installation time, etc. The rates developed from first principles were adjusted based on benchmarking and the experience and judgment of OceanaGold.

The productivity rates used for mine scheduling are shown in Table 16-31.

Table 16-31: Productivity Rates

Activity Type	Dimensions	Rate (1)
Level Access	5 m x 5.5 m	120 m/mo
Main Decline	5 m x 5.5 m	120 m/mo
Sumps, Remucks, Ventilation Drifts, Footwall and Stope Accesses	5 m x 5.5 m	120 m/mo
Escapeway Drifts	5 m x 5 m	120 m/mo
Stope Production Longhole Drilling		250 m/d
Stope Slot Raiseboring		4 m/d
Longhole Stoping		1,500 t/d
Ventilation Raises	6 mx 4 m	100 m/mo
Escapeway Raises	1 m dia	90 m/mo
Backfilling (CRF)		1,000 m^3/d

Source: OceanaGold, 2022

(1) All rates are per face. Multiple areas/faces are mined together to generate the production schedule.

General Parameters

General schedule parameters applicable to all underground mining activities are presented in Table 16-32.

Table 16-32: Schedule Parameters for Underground Mining

Schedule Parameters	Units	Value
Annual mining days	days/year	365
Mining days per week	days/week	7
Shifts per day	shifts/day	2
Scheduled shift length	hrs/shift	12
Maximum productive hours for drills (60% utilization	hrs/shift	7.2
Maximum productive hours for LHDs (55% utilization)	hrs/shift	6.6
Maximum productive hours for trucks (60% utilization)	hrs/shift	7.2

Source: OceanaGold, 2022

Key assumptions regarding ore and waste material characteristics are detailed in Table 16-33.

Table 16-33: Material Characteristics

Characteristic	Units	Value
Ore in situ density	t/m^3	2.80
Ore swell	%	40
Ore loose density	t/m^3	2.00
Waste in situ density	t/m^3	2.80
Waste swell	%	40
Waste loose density	t/m^3	2.00

Source: OceanaGold, 2022

For the purposes of developing productivity estimates and costs, the ground support requirements are as follows:

- The backs and upper sidewalls of capital development (decline, level access) will use 2.4 m x 25 mm fully encapsulated resin bolts. The lower sidewalls will be supported with 1.8 m x 46 mm galvanized friction bolts (split sets). The resin bolts will have corrosion protection, e.g. epoxy or galvanizing. Capital drives are expected to have a service life of between 2 to 10 years.

- The backs and sidewalls of ore drives, and slot drives will be supported with 1.8 m x 46 mm black friction bolts. These drives are not expected to have a service life greater than two years, thus no need to consider corrosion protection.
- The rock bolt spacing required for openings supported with wire mesh (level access, decline, slot drive, ore drive) is dependent on the dimensions of the mesh sheet, thus the rock bolt spacing is approximately 1.2 m to 1.4 m. The rock bolt spacing in fibercrete has been designed at about 1.4 m. With both the wire mesh and the fibercrete, the design assumes that rock bolts will be installed at the gradeline so that the surface support terminates with a rock bolt.
- The ground support design includes specific cable bolt designs using 2 x 15 mm diameter strands of bulbed black cable (standard twin-strand cable bolts). The cable bulbs will be 25 mm diameter and spaced every meter along the length of the cable. The cable bolt lengths will be standardized as either 6.3 m or 8.3 m twin-strand units. The standard borehole lengths will either be 6 m or 8 m. A 0.3 m tail is required for tensioning one of the two strands with a steel plate, barrel and wedge.
- The 8.3 m cable bolts are only used for supporting stope crowns. The 6.3 m cable bolts are used for stope brows, or in lateral development with spans exceeding 7.5 m (e.g., decline passing bays or 3-way intersections).
- The ground support design includes face support. A temporary face in lateral development will be supported down to the gradeline with sacrificial sheets of black wire mesh and 0.9 m x 46 mm friction bolts. A permanent face in lateral development must be supported to the same ground support standard as the adjacent sidewalls.

The following is a description of the general and activity-specific parameters upon which the productivity rates are based.

Main Ramp Development (long-term development openings)

The main ramp is 5 m wide x 5.5 m high with an arched back. It will be developed with a twin-boom jumbo drilling 45 mm diameter blastholes. All jumbo holes will be drilled 4.9 m in length, which allows for an effective advance rate of 4.2 m per round. The drill pattern provides for 56 charged blastholes and four uncharged relief holes. Drilling times were calculated based on average penetration rates of 100 m/hour (hr). A 10% re-drill factor was assumed.

Use of a bulk emulsion explosive was assumed at 108.6 kg/m of advance. The blasting time considered tramming, setup and rig down, and loading holes.

Loading will be performed with an 8.4 m^3 (14.9 t) load-haul-dump unit (LHD). Load, maneuver, dump and tramming times were considered in the overall cycle time calculation. It was assumed that 70% of the waste rock will be placed in muck bays and then will be rehandled into trucks and hauled to the surface (or to an empty secondary stope when available). For the remaining 30% of the waste rock, it was assumed that the LHD will load the truck directly (i.e., no rehandling from a muck bay).

For ground support installation, time allowances have been included for mobilization and setup, bolting/meshing/shotcreting as required, and demobilization. Cable bolts will be installed at intersections. Utility installation includes water line, drain line, ventilation tube, and electrical reticulation. The mining schedule is based on an average decline advance rate of 120 m/month.

Level Access Drifts (medium-term development openings)

The level access drifts will be 5 m wide x 5.5 m high with flat backs. They will be developed with a twin-boom jumbo drilling 45 mm diameter blastholes. All jumbo holes will be drilled 4.9 m in length, which allows for an effective advance rate of 4.2 m per round. The drill pattern provides for 58 charged blastholes and four uncharged relief holes. Drilling times were calculated based on average penetration rates of 100 m/hr. A 10% re-drill factor was assumed.

Use of a bulk emulsion explosive was assumed at 112.5 kg/m of advance. The blasting time considered tramming, setup and rig down, and loading holes.

Loading will be performed with an 8.4 m^3 (14.9 t) load-haul-dump unit (LHD). Load, maneuver, dump and tramming times were considered in the overall cycle time calculation. It was assumed that 70% of the waste rock will be placed in muck bays and then will be rehandled into trucks and hauled to the surface (or to an empty secondary stope when available). For the remaining 30% of the waste rock, it was assumed that the LHD will load the truck directly (i.e., no rehandling from a muck bay).

For ground support installation, time allowances have been included for mobilization and setup, bolting/meshing/shotcreting as required, and demobilization. Cable bolts will be installed at intersections. Utility installation includes water line, drain line, ventilation tube, and electrical reticulation. The mining schedule is based on an average lateral development advance rate of 120 m/month.

Through the Stope Development Drifts (short-term development openings)

The stope development drifts will be 4.5 m wide x 5 m high with flat backs. They will be developed with a twin-boom jumbo drilling 45 mm diameter blastholes. All jumbo holes will be drilled 4.9 m in length, which allows for an effective advance rate of 4.2 m per round. The drill pattern provides for 59 charged blastholes and four uncharged relief holes. Drilling times were calculated based on average penetration rates of 100 m/hr. A 10% re-drill factor was assumed.

Use of a bulk emulsion explosive was assumed at 98.9 kg/m of advance. The blasting time considered tramming, setup and rig down, and loading holes.

Loading will be performed with an 8.4 m^3 (14.9 t) load-haul-dump unit (LHD). Load, maneuver, dump and tramming times were considered in the overall cycle time calculation. It was assumed that 70% of the ore will be placed in muck bays and then will be rehandled into trucks and hauled to the surface. For the remaining 30% of the ore, it was assumed that the LHD will load the truck directly (i.e., no rehandling from a muck bay).

For ground support installation, time allowances have been included for mobilization and setup, bolting and meshing as required, and demobilization. Utility installation includes water line and ventilation tube. The mining schedule is based on an average lateral development advance rate of 120 m/month.

Stoping

Stopes will be 25 m in height x 20 m in width and will have varying lengths. A longhole production drill will be used to ring drill the stope. Blastholes will be 89 mm in diameter and the estimated drilling rate is 30 m/hr. Twenty blastholes will be required per ring. The total drilling requirement is 220 m per ring (including 10% re-drill) and the ore blasted per ring is 2,100 t.

A bulk emulsion product will be used to blast the stopes and the powder factor will be 0.40 kg/t.

Loading will be performed with an 8.4 m^3 (14.9 t) load-haul-dump unit (LHD). Load, maneuver, dump and tramming times were considered in the overall cycle time calculation. It was assumed that 90% of the ore will be placed in muck bays and then will be rehandled into trucks and hauled to the surface. For the remaining 10% of the ore, it was assumed that the LHD will load the truck directly (i.e., no rehandling from a muck bay). The mining schedule is based on an average stope production rate of 1,500 t/day.

Ventilation Connections Between Levels

Ventilation rises will be 6.0 m wide x 4.0 m long and 25 m high. The ventilation rise will be drilled with 89 mm diameter holes and will break to a 1.1 m diameter raisebored hole. A total of 18 charged holes will be required for each ventilation rise.

16.2.8 Mine Production Schedule

The Horseshoe underground mine production schedule is based on the productivity assumptions shown in Table 16-34. The schedule was completed using Deswik scheduling software and is based on mining operations occurring 365 days/year, 7 days/week, with two 12-hr shifts each day. A production rate of approximately 2,000 t/d was targeted with ramp-up to full production as quickly as possible. Resource levelling was used on a monthly basis for ore tonnage and lateral development.

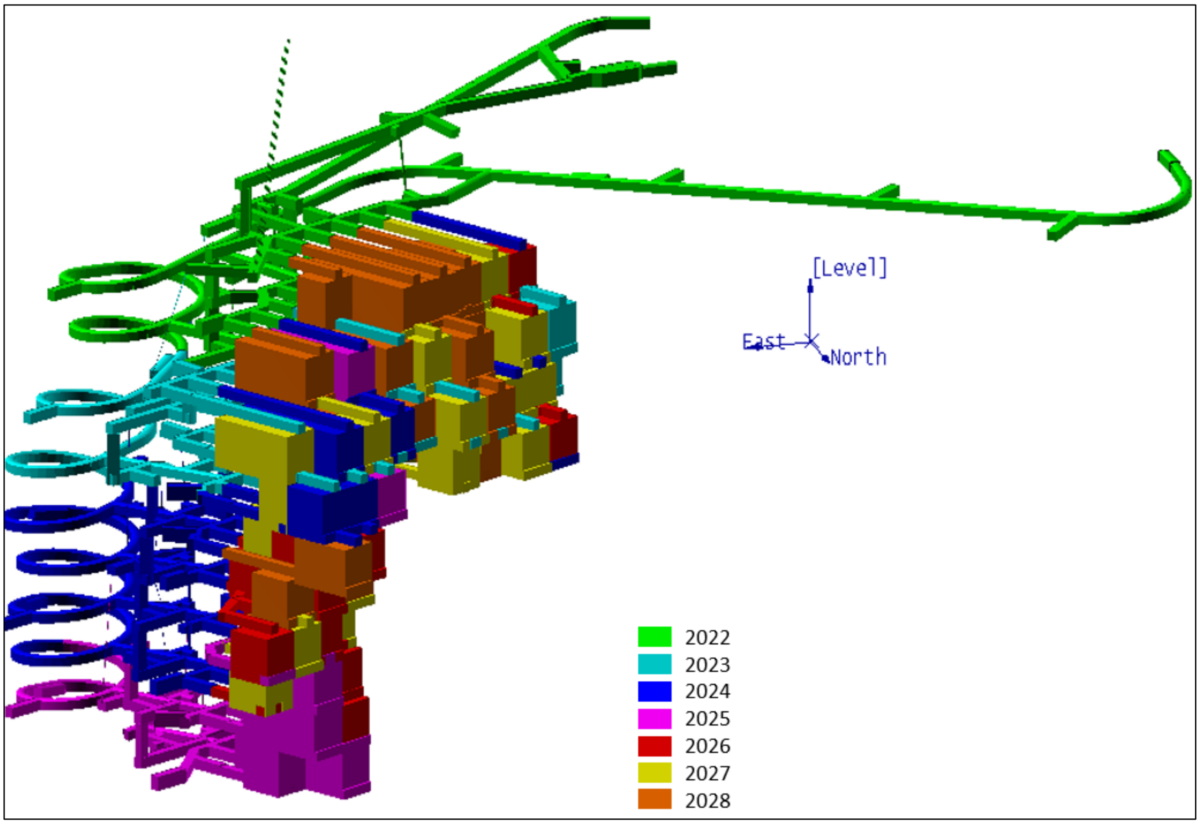
The scheduling work includes placing CRF in the mined-out stopes. Allowances were included for the time that will be required to cure the cement binder in the CRF. Specifically, a 14-day delay was assumed prior to mining adjacent or driving on top of a CRF filled stope.

Figure 16-30 presents the annual mining schedule based on the aforementioned scheduling assumptions.

Table 16-34: Horseshoe Mine Production Annual Mining Schedule

Year	Mineralized Tonnes (kt)	Au (g/t)	Waste Tonnes (kt)	Backfill Volume (m^3)
2022			180.0	
2023	190.9	4.07	309.4	44,412
2024	735.9	4.31	181.6	253,804
2025	748.6	3.48	133.1	268,636
2026	738.8	4.64	13.0	276,982
2027	739.2	3.18	3.4	295,760
2028	262.8	2.23	1.3	122,738
Total	3,416.3	3.78	821.8	1,262,333

Source: SRK, 2022

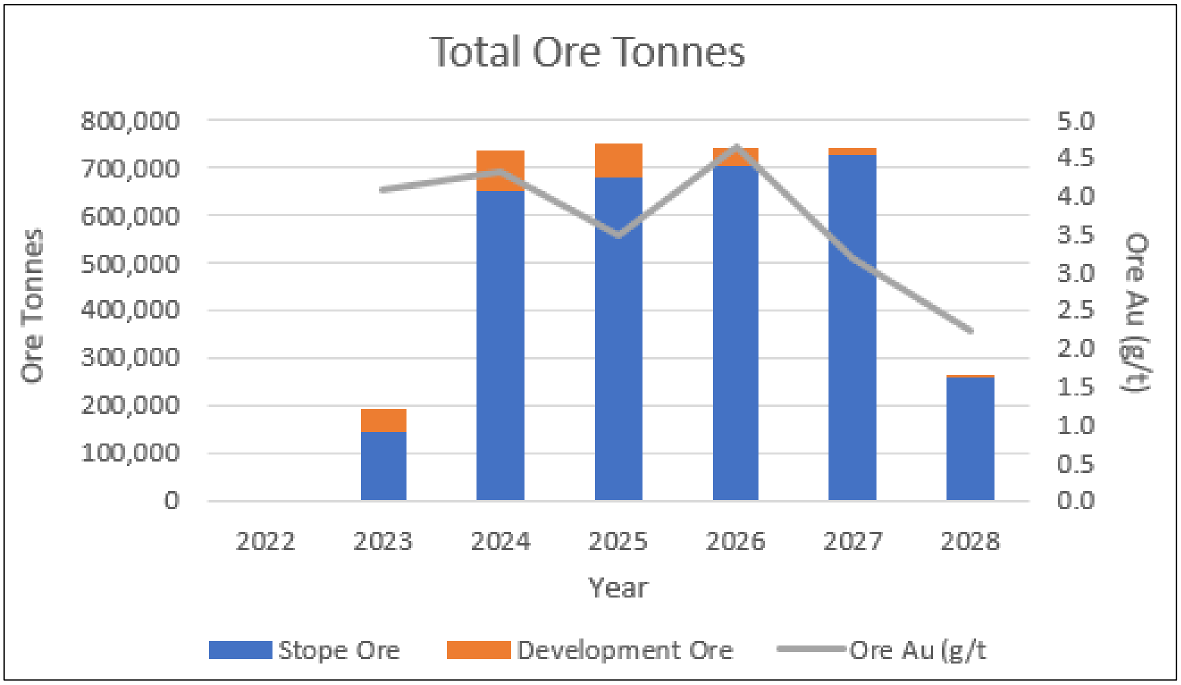


Source: SRK, 2022

Figure 16-30: Mine Production Schedule Colored by Year

The underground permitting process is expected to be complete in early 2022.Underground portal development is scheduled to begin in April 2022. The Snake Pit has progressed sufficiently to allow access to the portal location. First production from the stopes is scheduled to occur in September 2023 and will last through May 2028 based on the current Mineral Reserves. Figure 16-31 and Figure 16-32 present additional summary information for the mine production schedule. Table 16-35 shows additional items in the production schedule summarized annually.

Haulage distances from each level underground to portal were calculated. Surface haulage distances for ore and waste were also calculated. The tonne-kilometer (TKM) is calculated and flagged in the production schedule to give hauling TKM's for each period for ore and waste material.



Source: SRK, 2022

Figure 16-31: Ore Tonnes by Activity Type

Source: SRK, 2022

Figure 16-32: All Material Tonnes

Table 16-35: Detailed Mine Production Schedule

	Totals	2022	2023	2024	2025	2026	2027	2028
General Summary								
Total Ore (t)	**3,416,327**		190,929	735,942	748,635	738,839	739,206	262,777
Total Waste (t)	**821,800**	179,984	309,419	181,642	133,058	13,022	3,408	1,267
Total Tonnes (t)	**4,238,127**	179,984	500,348	917,584	881,693	751,861	742,614	264,044
Total Lateral Development (m)	**13,374**	2,185	4,354	3,272	2,558	657	264	83
Total Metal - Au (oz)	**415,144**		24,992	101,910	83,659	110,282	75,496	18,805
Total Grade - Au (g/t)	**3.8**		4.07	4.31	3.48	4.64	3.18	2.23
Ore Breakout								
Stoping Ore (t)	3,153,566		142,401	647,667	675,942	702,731	726,131	258,693
Stoping Ore Grade - Au (g/t)	3.8		4.32	4.42	3.48	4.72	3.19	2.23
Development Ore (t)	262,761		48,528	88,275	72,693	36,107	13,075	4,083
Development Ore Grade - Au (g/t)	3.3		3.35	3.49	3.41	3.23	2.64	1.83
Development Summary								
Total Lateral Development (m)	**13,374**	**2,185**	**4,354**	**3,272**	**2,558**	**657**	**264**	**83**
Total Vertical Development (m)	**1,430**	**353**	**335**	**360**	**354**	**28**		
Lateral Development Ore (m)	3,647		692	1,243	999	484	177	52
Lateral Development Waste (m)	9,727	2,185	3,662	2,029	1,559	174	87	31
Lateral Development Breakout								
ACC - Level Access	697	172	192	168	165			
CRF - CRF Stockpile	235	17.5	140.0	47.2	30.0			
DEC - Decline	2,618	1,079	575	428	536			
DSP - Decline Stockpile	261	100	60	42	59			
EWD - Escapeway Drive	221	22	82	48	69			
FAD - Fresh Air Drive	757	89	416	251				
FWD - Footwall Drive	1,015	136	655	136	83	1		4
LSM - Level Sump	224	101	38	47	38			
OD - Ore Drive	5,375	65	1,777	1,635	1,184	469	200	44
PSN - Pump Station Drive	36	12	12		12			
RAD - Return Air Drive	821	363	224	140	95			
SLD - Slot Drive	904		93	273	251	187	64	35
SUB - Substation	133	27	67	30	10			
XCT - Crosscut	78		24	28	26			
Vertical Development Breakout								
DH_ Drainhole	236	52	29	78	51	28		
EWR - Escapeway Rise	292		140	52	101			
FAR - Fresh Air Rise	106			106				
RAR - Return Air Rise	252	30	98	49	74			
RMH_ Rising Main	307	111	68		128			
SH_ Service Hole	236	161		75				

Source: SRK, 2022

16.2.9 Mining Operations

Mine Access

A mine access and material handling tradeoff study were completed for the Horseshoe deposit. Options considered included shaft access with ore hoisting, decline access with conveyor haulage, and decline access with truck haulage. The tradeoff study demonstrated that, given the deposit depth, production rate, and anticipated mine life, the economically superior option is decline access with truck haulage.

The upper portion of the 5 m wide by 5.5 m high access decline is expected to be in weathered rock and therefore will require an increased level of ground support. After the decline has passed through the weathered rock, a less intensive level of ground support will be required. The decline is designed at a maximum gradient of 14%. A minimum turning radius of 27.5 m was used, which is suitable for the underground haul trucks contemplated for the operation.

The portal for the access decline will be located on an open pit bench approximately 80 m below the natural surface. The first 20 m of each portal will be mined 0.5 m wider and higher to accommodate greater ground support. An all-weather gravel surface will be established at the portal and portal bench area and drainage will be maintained away from the portal entrance to minimize water entering the portal and decline from the bench area.

Ventilation, power, water discharge, supply water, and communications will be installed at the portal and carried down the decline to support the initial development operation. Secondary egress will be via 1.1 m diameter raisebored escapeway raises fitted with ladderways. The escapeway path begins at the fresh air portal, then across a link drive to the exhaust drive, then follows the fresh-air system to the 900mRL, then finally follows the return air system to the bottom of the mine.

Stoping

Stopes will be mined using the sublevel open stoping method. Individual stope blocks are designed to be 20 m wide, up to 25 m long, and will have a transverse orientation. Levels are spaced 25 m apart and each stope block will have a top and bottom access (5 m high x 5.5 m wide flat back drifts).

Stopes will be drilled using 89 mm diameter holes (stope slots will include 1.1 m diameter raise bored holes and, when the stopes will be situated on top of in situ rock, a 15.5 m slot drive will be mined). In most instances, a bottom up, primary/secondary extraction sequence will be followed. Primary stopes will be backfilled with CRF and secondary stopes will be backfilled with RoM waste from the underground and open pit operations.

Stope extraction will occur in two steps. During the first step, a slot will be mined at the far end of the stope. The slot is required to create sufficient void space for the remainder of the stope to be blasted. During the second step, production rings will be blasted three rows at a time until the stope is completely extracted. The number of three-row blasts in each stope will depend on the length of the stope. All blasting with be with bulk emulsion.

Ore will be remotely mucked from the bottom stope access using a 14.9-t LHD. Cable bolts will be installed at the stope brow to ensure stability. The LHD will transport 90% of the ore to a muck bay to maximize the efficiency of the stope mucking operations. The remaining 10% of the ore will be mucked directly into a truck. A fleet of 51-t haul trucks will be used to transport ore to the surface. Multiple muck bays will be used on each level to avoid interference between the stope loader and the haul trucks.

At the surface, the haul trucks will dump onto a RoM ore stockpile and will then travel to an adjacently located backfill plant to be loaded with CRF. After being loaded, the haul trucks will return to the underground mine and will dump the CRF into a muck bay near the top of an empty primary stope. After dumping the load of CRF at the muck bay, the haul truck will return to the producing level to once again be loaded with ore. An LHD will be used to transport the CRF from the muck bay to a dumping point at the top access of the empty stope.

Lateral Development

Lateral development includes interlevel ramps, level accesses, stope accesses, and short connecting drifts for ventilation. The interlevel ramp system will be a continuation of the main access decline and will have the same maximum gradient (14%). Level accesses will be 5 m wide by 5.5 m high with a flat back and will be mined higher at the muck bays to allow the haul trucks to be loaded by the LHD.

Interlevel ramps and levels accesses will be located in the footwall and have been designed to avoid crossing fault zones to the maximum extent possible. Stope accesses are oriented perpendicular to the strike of the orebody.

The lateral development is sized for the operation of the mining equipment fleet that has been selected for the operation. The development profiles include allowances for ventilation ducting and services.

Vertical Development

Ventilation rises will be 6 m wide x 4 m long and 25 m high. Each ventilation rise will be conventionally drilled and blasted but will break to a 1.1 m diameter raisebored hole. Escapeways between levels will be sub-vertical and raisebored at a diameter of 1.1 m. Rising main holes, electrical holes, service holes, pastefill holes, and drain holes will be drilled either by longhole or raisebore at variable diameters and lengths as required.

Truck Haulage

The mine plan assumes that LHDs will load 51-t haul trucks from muck bays that will be strategically located throughout the development workings. A small percentage of the ore and waste will be loaded directly into trucks (i.e., not rehandled from muck bays). Ore and waste haulage distances were accounted for in the productivity and cost calculations and are based on the mine production schedule. Ore haulage distances are approximately to 2 km to 3 km over the LoM timeframe. Waste haulage distances vary considerably depending on the time period and where the waste is being deposited. At the peak, six haul trucks are required to transport the ore, waste and CRF.

Backfilling

The CRF will use aggregate that is crushed and screened to -80 mm. The CRF crushing and screening sequence will work as follows:

- Contractors will load and haul material from a waste dump and deliver it to the Crushing and Screening Plant (CSP) located near the UG RoM area. Alternatively, waste rock can be delivered to the CSP directly from OP or UG sources.
- The CSP will be designed for 90 t/h with an availability of 90%.
- The average CSP output will be about 1,600 t/d with two shifts.
- The CSP will produce a coarse aggregate and a fine aggregate product for the CRF.
- This CSP will require one Cat 962G loader and one operator per shift.
- A typical annual CRF requirement of 264,000 m^3 will require about 470,000 tpa of aggregate production, i.e., 294 days at 1,600 t/d.

The CRF batch plant will be located next to the CSP. The production sequence will work as follows:

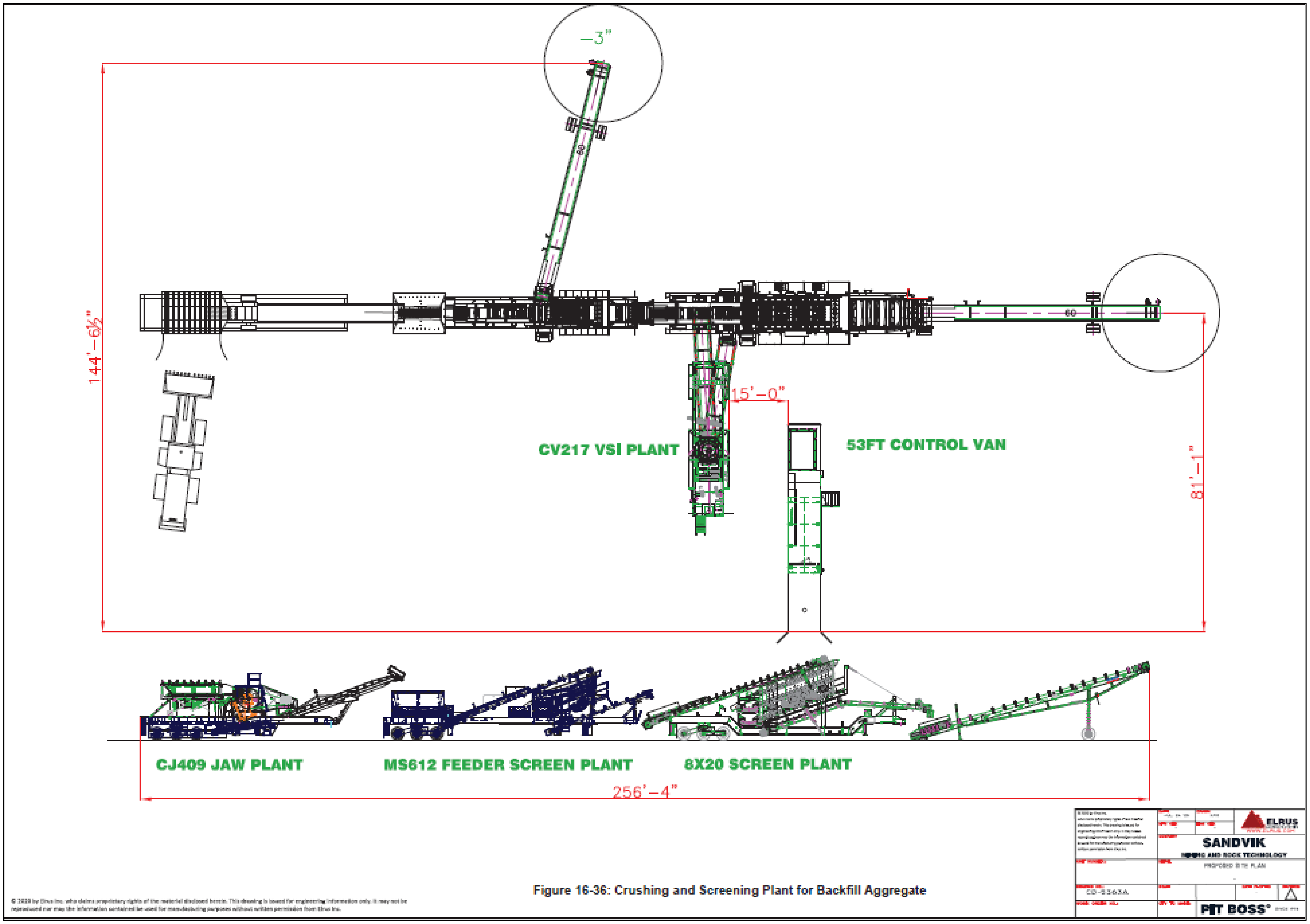
- A second Cat 962G loader and plant operator will keep the aggregate bins topped-up.
- The automated batch plant mixer will make 5 x 5 m^3 CRF batches per 50-t haul truck.
- Bulk density of the mix will be about 2 t/m^3, thus 50-t loads per truck.
- The batching time will be about 10 minutes to make the 5 batches.
- The typical cycle time will be about 1 hour per truck between batch plant and the underground mine.
- The batch plant would typically produce 50 m^3/hr x 20 hours per day, i.e., 1,000 m^3/day.
- Typical annual requirement of 264,000 m^3 would require 264 CRF days per year.

The underground CRF production sequence will work as follows:

- The 50-t haul trucks will tip their loads into a CRF stockpile bay located on each footwall drive.
- The CRF stockpile will be equipped with an automated water spray to wash-down the tray after tipping each load.
- An LHD will move the CRF from the stockpile to the stope.

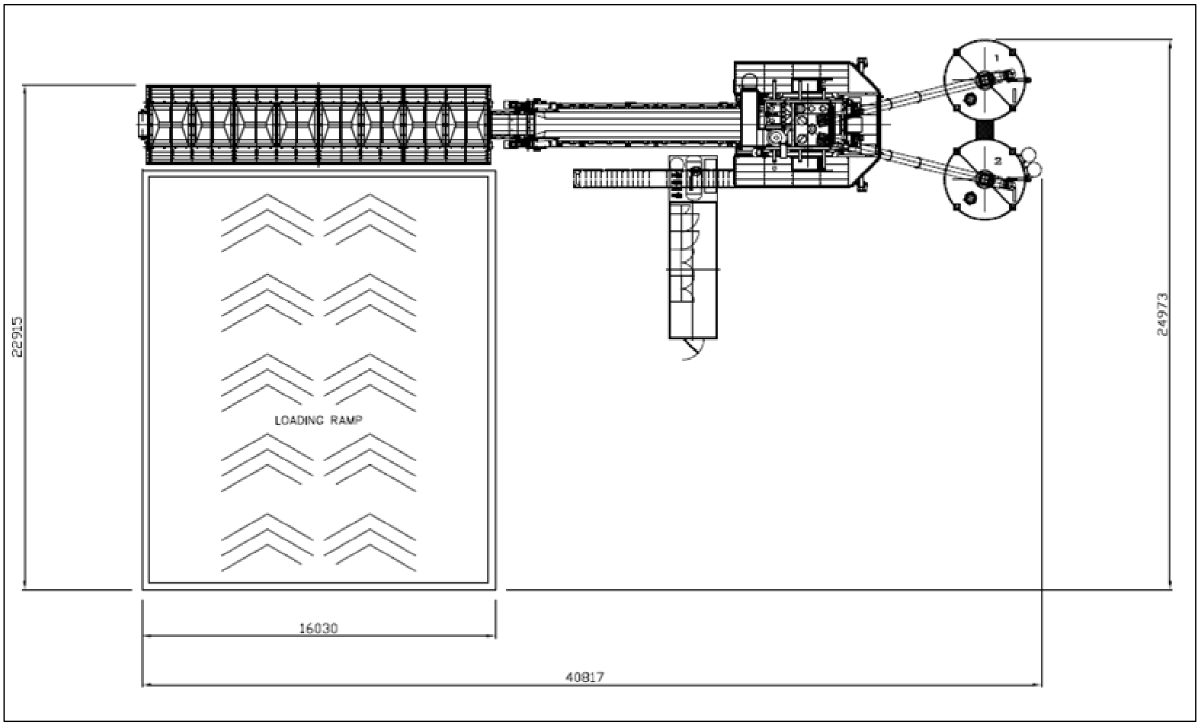
Two different CRF mix designs will be required: (1) low strength, and (2) high strength. The low strength mix will require about 3.8% cement by dry mass. The high strength mix will require about 4.8% by dry mass. The expected 28-day UCS of the low strength mix could range from 750 to 3,200 kPa with an average of 1,500 kPa. The high strength mix could range from 1,900 to 4,500 kPa with average of 3,000 kPa (Saw et al, 2017).

Figure 16-33 shows the crushing and screening plant and Figure 16-34 shows the CRF plant.



Source: Sandvik, 2020

Figure 16-33: Crushing and Screening Plant for Backfill Aggregate



Source: OceanaGold, 2020

Figure 16-34: CRF Backfill Plant

Table 16-36 shows the total LoM volume breakdown of the rock fill (low strength) and CRF (high strength).

Table 16-36: LoM Backfill Quantities

Backfill Summary	**(m^3)**
Total Backfill Volume	**1,131,988**
CRF High Strength (4.8% cement)	152,219
CRF Low Strength (3.8% cement)	979,768

Source: OceanaGold, 2022

Waste Rock

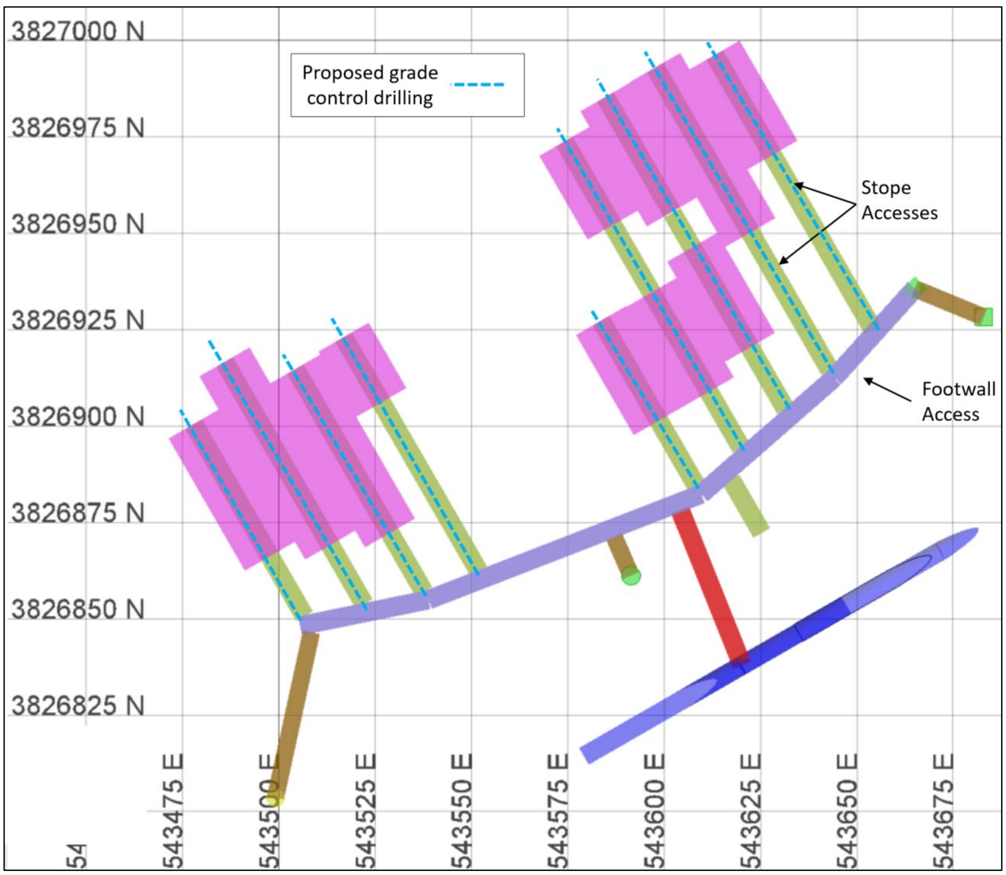
Waste rock from the underground will be hauled to the surface to be used in the production of CRF. Although there is scope to haul some waste to secondary stopes, it is not included in this study.

Grade Control

The characterization of ore versus waste and further geochemical waste classification will be completed through diamond core drilling of the stope accesses prior to mining as shown in Figure 16-35. Once the footwall level access is established, a mobile drill rig will drill an uphole and a downhole from each ore-drive access. Drilling will occur on every second level (the downhole from each collar location removes the need to drill from the level below). Each stope will have a drillhole pierce both the hangingwall and the footwall. The drilling in 2023 totals approximately 10,000 m and follows the development schedule. This will drill out the deposit to an adequate drill density for accurate grade control estimation.

The core will be logged, sampled and analyzed to provide grade control and geochemical waste classification information. Geologic and block models will be updated with this information and ore/waste grade boundaries will be pre-determined prior to mining the stope accesses. Areas considered to be waste will be characterized geochemically to determine which stockpile the material should be sent to. Geochemical sampling techniques for the underground mine will follow existing open pit sampling techniques. Initially, all stope accesses will be drilled and sampled to ensure adequate definition for each stope.

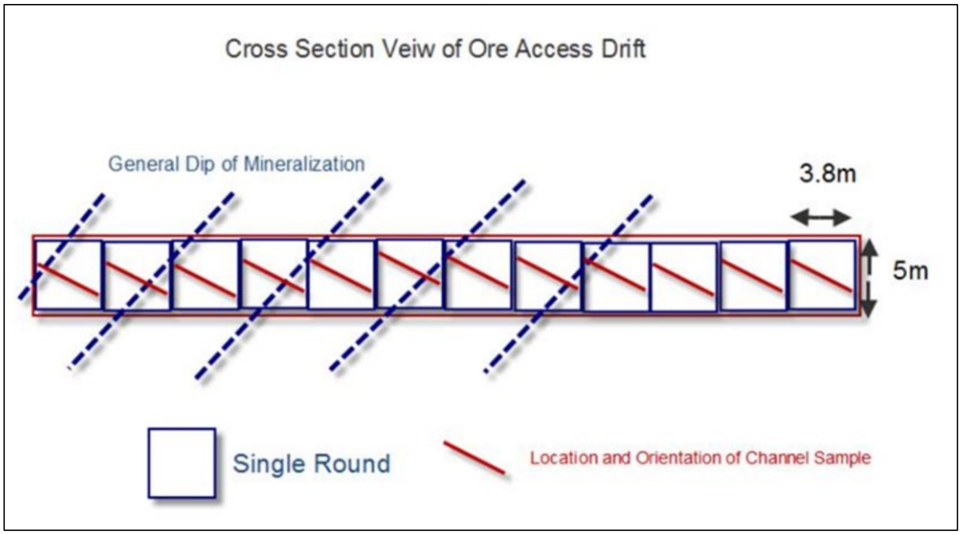
The existing onsite lab, Kershaw Minerals Lab (KML) analyzes Au, Ag, total carbon, and total sulfur. LECO is used to determine total carbon/sulfur on a percentage of production samples for detailed carbon and sulfur speciation. Lab turnaround time is expected to be 18 to 26 hours.



Source: SRK, 2017

Figure 16-35: Level Section Showing Planned Diamond Drilling from Footwall Access Drifts

Additionally, as the stope accesses are being mined, a mine geologist will observe, map and channel sample the exposed rock on each rib. Typically, the predominant fabric of the mineralization will display an inclined orientation on the ribs and the channel samples will need to be oriented normal to this fabric as shown in Figure 16-36.



Source: SRK, 2017

Figure 16-36: Cross Section View of Development Rounds and Proposed Channel Sampling Locations

The mine geologist will identify the presence of silicification or sulfidation and utilize these alterations to determine where sample breaks will occur. Using the inclined channel sample orientation, samples lengths for each cut will be 4 to 5 m in length. Samples will be collected as continuous channels at nominal 1 to 2 m lengths depending on the observations of the geologist. Channels can be sawn or chipped depending on the nature of the rock being collected. Sample breaks will be made at lithologic or alteration contacts.

Once mining has progressed to a point where sufficient testing has shown that estimated grades based on core samples reconcile to channel sampling during actual mining, a portion of the diamond drilling or channel sampling may be eliminated.

16.2.10 Ventilation

The ventilation system has been designed to support the development and production activities for the underground mine. The total life-of-mine analysis includes the predicted distribution of airflow and pressure. The design is detailed in a life of mine development model. A 3D ventilation model was created using Ventsim Design 5.3.

Input Parameters

The location of the mine is in a very temperate area with the average low temperature in January of 1.3°C (average high 11.8°C), and an average high temperature in July of 32.6°C (average low 19.8°C). Combined with the shallow depth and apparent lack of geothermal activity, this mine does not appear to present thermal stress related issues, so studies in these areas were not developed.

No harmful strata gases are expected to be encountered at this site. No crushers or fixed ore/waste conveyances (continuous acute dust sources) are currently designed underground. Strategies for controlling dust while loading and hauling ore/waste are an important operational consideration with little impact at this design stage. The configuration of the system is as an exhausting ventilation system, which minimizes the blast clearance time/possibility of exposure to blast-generated gases by maintaining the ramp clear of blasting fumes.

Airway dimensions are as per the mine design, with the main ramp being 5 m x 5.5 m and the raises to surface at 4 m x 6 m (blasthole design with offsets on each level). Oval equivalent duct of 1.65 m equivalent was modeled for the main decline development, with 1.4 m duct used for the fresh air and exhaust access developments. Model friction factors, resistances, shock losses, etc. were used based on available data and standard best practice as shown in Table 16-37.

Table 16-37: General Ventilation Modeling Parameters

Design Point	Friction Factor (kg/m^3)	Resistance (Ns^2/m^8)	Notes
Level Access Drive	0.012		Standard base value for general horizontal development
Ramp	0.014		Slightly elevated friction factor to represent the additional losses associated with the spiral development
Drop Raise	0.016		Elevated friction factor associated with the construction methodology, accurate and controlled drilling will be required, shock loss of 0.3 added for every 90°transition
Bulkhead		500	The isolation bulkheads for the fresh air raise will be sprayed with shotcrete to reduce leakage, however, access will not be limited
Bulkhead with Closed Regulator/Personnel Door		100	The exhaust raise will be accessed through personnel doors installed in the system isolation bulkheads. These personnel doors will be latched and gasketed to reduce leakage

Source: SRK, 2022

Airflow Requirements

The expected equipment load was calculated based on the equipment list. A generic airflow dilution value of 0.04 m^3/s per kW power for diesel engines has been used for this study because of the decision to use Tier 4 diesel equipment, and the assumption that all equipment listed in the fleet summary would be used in the mine and listed with a 100% utilization value. The high utilization value provides for some conservativeness in the evaluation. A general leakage value of 20% was assumed to provide the minimum airflow requirement. Airflow for individual pieces of equipment in the mine will need to meet the requirements of CFR57.5067 which refers to the nameplate dilution values determined by MSHA/NIOSH testing, or meet the EPA requirements. Overall, the minimum airflow requirement for the mine is approximately 250 m^3/s for the CRF design Table 16-38.

Table 16-38: Cemented Rock Fill Equipment Load and Minimum Airflow Calculation

Equipment (Main Equipment Only)	Rated Power (kW)	Measured and Indicated Reserve Schedule								
		2022	2023	2024	2025	2026	2027	2028	2029	2030
LHD (LH517i) - Tier 4	315	2	4	4	4	4	4	4	4	4
Haul Truck (TH 551i) - Tier 4i	515	2	5	6	6	6	6	6	6	6
Spray Mech - Tier 3	110	1	1	1	1	1	1	1	1	1
Agi Truck - Tier 3	170	1	1	1	1	1	1	1	1	1
Large IT - Tier 4	180	1	1	1	1	1	1	1	1	1
Small IT - Tier 4	130	1	1	1	1	1	1	1	1	1
12k Grader - Tier 4	140	1	1	1	1	1	1	1	1	1
Charge-Up - Tier 3	120	1	1	1	1	1	1	1	1	1
Scissor Lift - Tier 3	120	1	1	1	1	1	1	1	1	1
Multimec- Tier 3	120	1	2	2	2	2	2	2	2	2
Diesel Dilution @ 0.04 m^3/s/kW (m^3/s)			104	196	216	216	216	216	216	216
General Leakage Allowance			10%	10%	20%	20%	20%	20%	20%	20%
Total Minimum Airflow Requirement (m3/s)			**110**	**220**	**260**	**260**	**260**	**260**	**260**	**260**

Source: OceanaGold, 2022

Auxiliary Ventilation Systems - Development

The decline and main access development auxiliary ventilation systems are sized to ventilate the simultaneous operation of a truck and LHD.

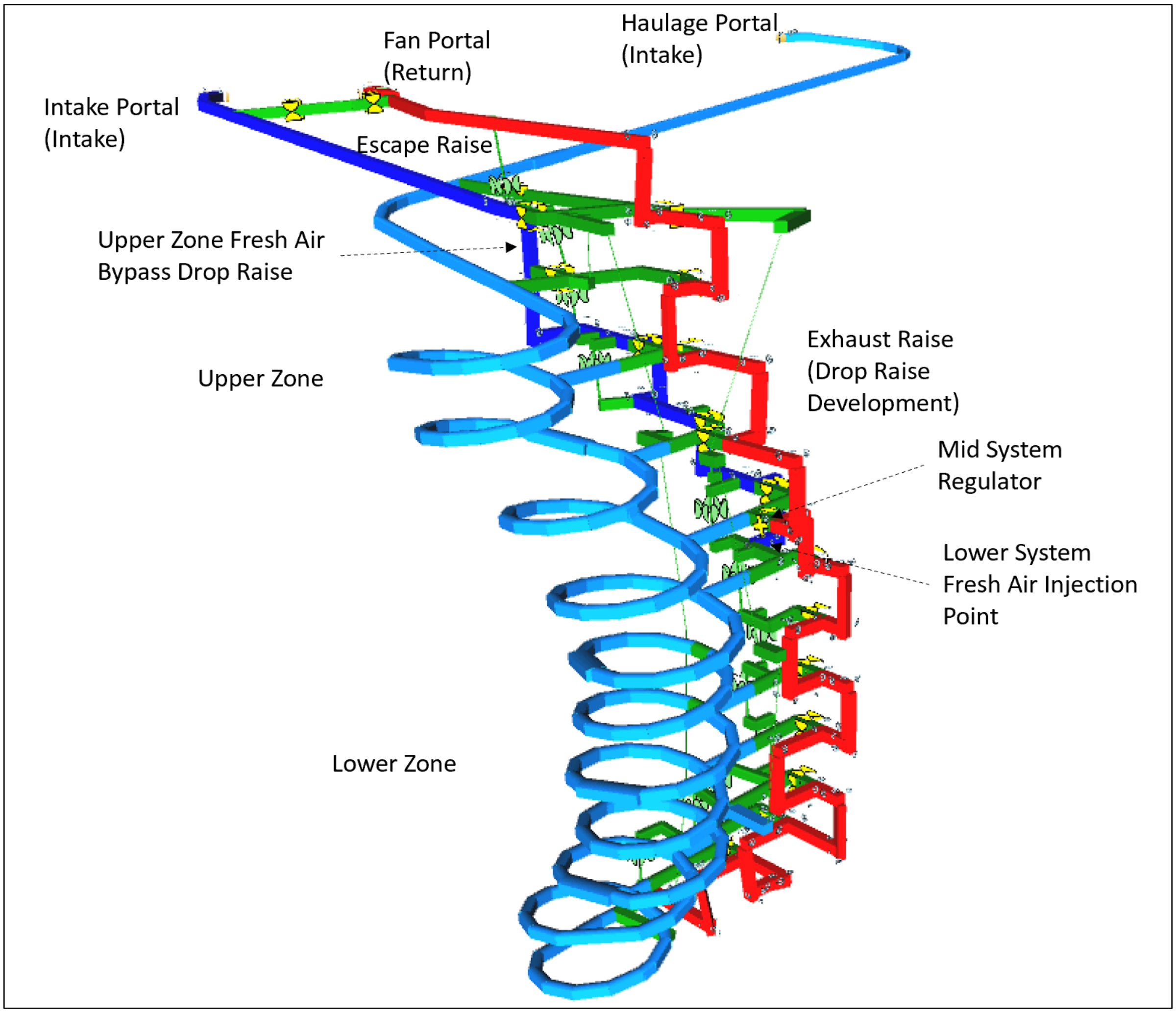
- The fresh air and exhaust accesses will be developed with single 1.5 m, 150 kW fan and will be able to support the operation of one truck and one LHD. The development length will be approximately 300 m. A single 1.4 m diameter duct will be required.
- The main access decline will be approximately 600 m and will require a slightly different ventilation system for development. Two 1.5 m, 150 kW fans mounted outside of the portal in series blowing into a 1.65 m oval equivalent duct will be able to provide ventilation for one LHD and a single truck. The single 1.65 m oval equivalent duct can be used to develop the initial decline. In the immediate development face area, a length of supplemental duct can be attached to the main duct to provide ventilation directly at the face during blasting, and during loading. The supplemental duct is advanced with the face and is not left incorporated within the long-term duct installation.

Stope Ventilation

For the stope auxiliary ventilation system, auxiliary fans will be located in “cutouts” in the ramp and will then draw the airflow to the active stope areas. The initial trunk line duct will be approximately 100 m (1.4 m diameter), with two approximate 160 m (1.2 m diameter) ducts delivering the airflow to the stope faces. The level ventilation fan will be required to operate at approximately 40 m3/s at 1.5 kPa, 80kW (allowing for approximately 15% leakage).

Ventilation Model

The life of mine ventilation model was developed to represent two options: measured/indicated reserves and measured/indicated/inferred reserves. Two different levels of airflow were examined; CRF equipment load and paste fill equipment load. The basic ventilation circuit remains the same for all scenarios. The mine is split into two operating zones: upper and lower. The upper zone is ventilated by fresh air drawn in from the ramp from the surface and exhausted into the exhaust raise system through a connection at the ramp midpoint, and the lower zone is ventilated by airflow that is drawn into the ramp through a fresh air raise system and exhausts into the main exhaust raise system at the bottom of the mine. This provides two distinct mining areas that are split and provides a degree of compartmentalization. The basic ventilation layout is shown in Figure 16-37.



Source: SRK, 2022

Figure 16-37: Ventilation System Layout

Each of the connections between the levels and the fresh air raise system and exhaust raise system is isolated by a set of manual doors so that infrequent mucking of the transfer levels can be accomplished. This results in a possible increase in leakage if the doors are not maintained, but will be necessary to allow for access. Secondary egress will be by way of a separate raise system.

The required main fan operating points for the surface exhaust fans are identified in Table 16-39. A parallel exhaust fan system (2 fans) should be considered for this system. This will allow for some amount of redundancy in the event of a fan outage (when one fan of a parallel fan installation is in operation generally approximately 70% of the original airflow can be achieved). The fans should also be installed with variable frequency drives so that the rotational speed can be modified which will increase the efficiency and adaptability of the ventilation system. This will allow the system to be easily modified as it grows.

Table 16-39: Main Fan Requirements

Scenario	Total Fan Requirement					
	Airflow (System) (m^3/s)	Bulkhead Pressure (kPa)	Inlet/Outlet Losses (kPa)	Total Pressure (kPa)	Efficiency (%)	Power (kW)
Stage 2 (full system – steady state)	270	1.43	0.25	1.68	75%	605

Note. System efficiency presented in this table is a generalized value, manufacturers efficiency will likely be different
Source: SRK, 2022

The parallel Zitron ZVN 1-24-286/6 286 kW fans selected by Oceana for the steady state ventilation should be adiquate for the long-term ventilation of the mine, however there will be very little room for additional capacity. Every effort will need to be made to limit leakage and reduce airway resistances.

16.2.11 Mine Services

Dewatering

The dewatering system will be built in phases as the mine develops and consists of a portable development system and permanent level pump stations. The system is designed to handle a capacity of 31.5 L/sec.

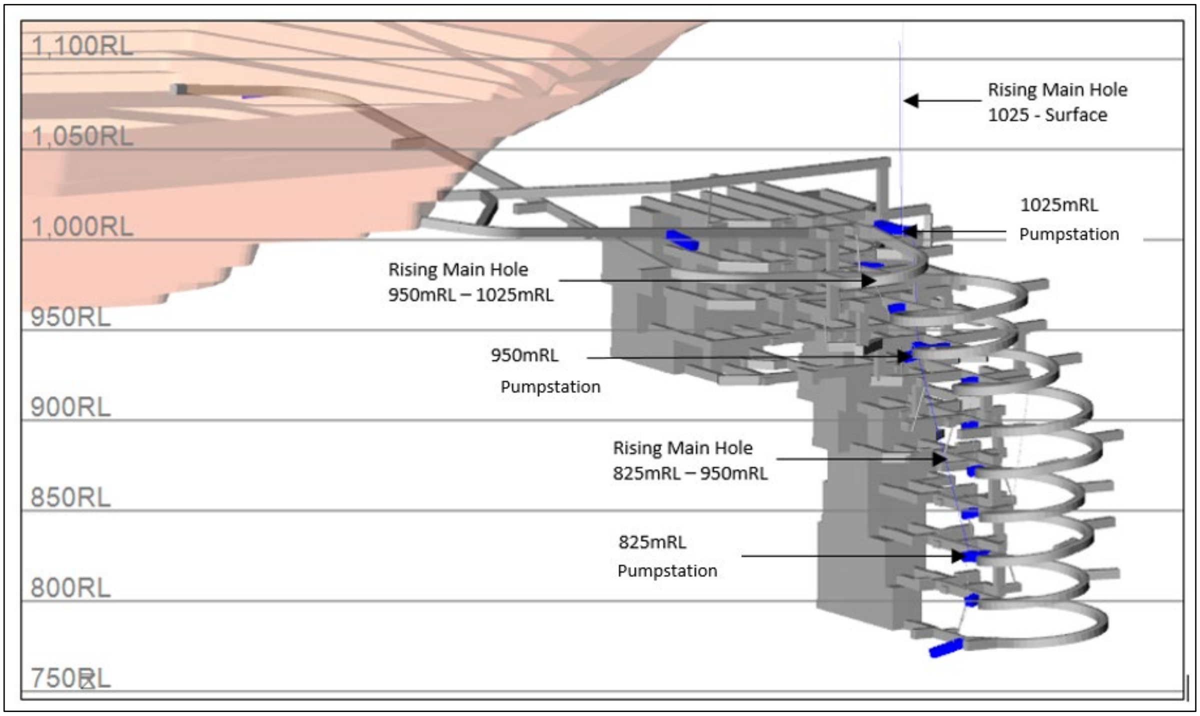
The system will pump from the underground workings through steel and HDPE pipes to a surface pond where the water will be returned through an existing system to the existing water treatment plant at the processing plant.

The portable development system will be used from the portal to the 1025 level where a permanent pump station will be constructed. A rising main hole will be drilled from surface to the 1025 pump station. Once commissioned, the portable pumping system will pump to the 1025 pump station, and from there to surface. The option also exists to pump from the 1025 pump station through the fresh air intake portal to a sump in the Snake Pit, where existing diesel pumps and infrastructure will pump to the surface pond.

As the mine progresses deeper, second and third permanent pump stations will be established on the 950 and 825 levels, respectively. Rising main holes will be drilled to connect each pump station directly.

Level sumps will be established on each level to catch water generated from mining activities on the levels and pump water to the nearest pump station in stages. Drain holes will be drilled between level sumps to help prevent overflowing of water onto the decline if pump failure occurs.

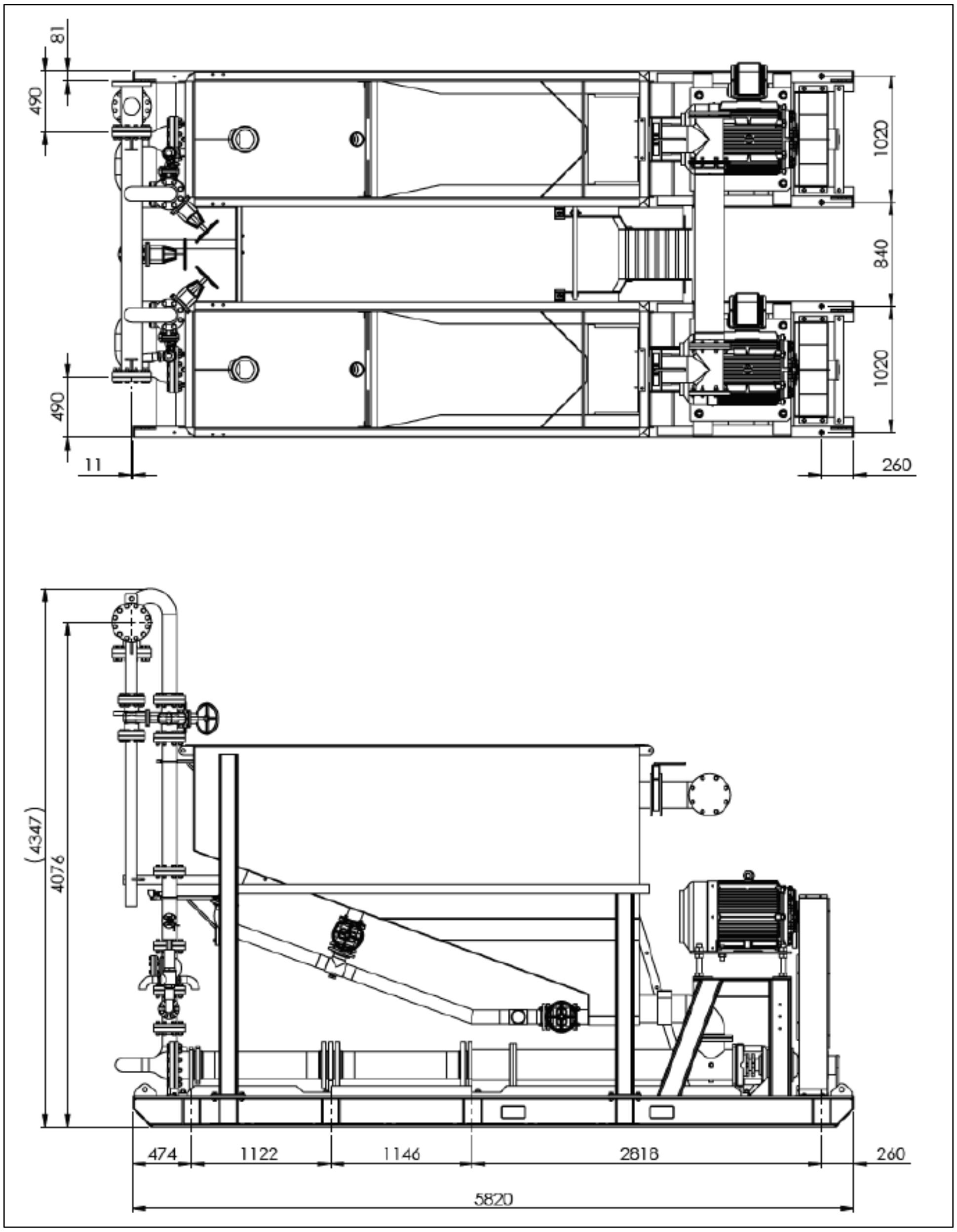
The location of the underground dewatering infrastructure is shown in Figure 16-38.



Source: OceanaGold, 2020

Figure 16-38: Dewatering System Layout

The primary pumping system will consist of pump skid stations as shown in Figure 16-39. Each skid station has two pumps, with a nominal flow capacity of 20 L/s per pump. The pumps are powered by 75 kW motors and are capable of 240 m of maximum head.



Source: Challenge Pumps, 2020 (units shown in millimeters)

Figure 16-39: Portable Pumping Skids

Surface dewatering will be ongoing during the time of underground production and a surface dewatering well is anticipated to be located near the main decline to minimize water encountered during development.

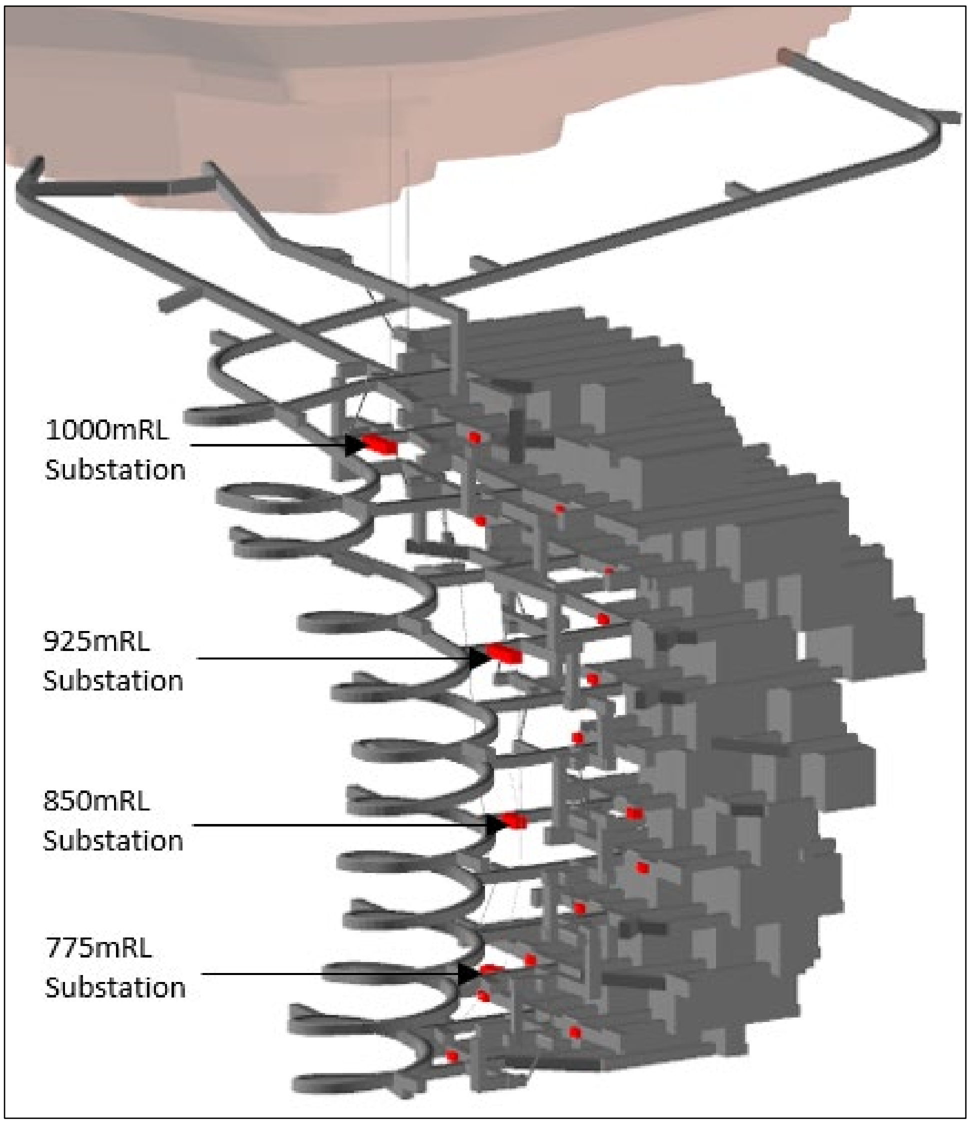
Electrical

Initially, power will be provided by a generator located at the portal location for the portal fans, mine pump system, and mine equipment required during the initial development of the mine.

Once mine development has progressed to the 1000mRL and the first substation cuddy is developed, the mine electrical system will be supplied from the existing power line from the main substation at the processing plant through the 24.9 kV power line. Electrical holes will be drilled from surface to the 1000mRL substation, establishing high voltage reticulation underground.

Power will be carried deeper as the mine progresses in power cables fed through additional electrical holes between each substation in order to keep high-voltage cables off the decline. The substations will step the power down to distribution boards and jumbo boxes located in cuddies on the footwall drives of each level to feed the underground mobile equipment, fans and pumps.

Locations of electrical underground infrastructure is shown in Figure 16-40.



Source: OceanaGold, 2020

Figure 16-40: Electrical System Layout

Once stoping commences in 2023, the average power usage is 1.65 million kWhr per month with an average operating load of 2.32 MW.

Health and Safety

The mine design incorporates MSHA safety standards for secondary egress, which will be via 1.1 m diameter raisebored escapeway raises fitted with ladderways. Additionally, three 12-person and one six-person mobile refuge chambers are included and will be located in active working areas over the LoM.

The mine will have a communications system that has both mine phones and wireless communication. A mine rescue team will support the operation. The mine safety program will integrate with local providers in case of any mine emergency. A stench gas emergency warning system will be installed in the mine's intake ventilation system. This system can be activated to warn underground employees of a fire situation or other emergency whereupon emergency procedures will be followed.

Manpower

Manpower levels are estimated based on the production schedule and associated equipment operating requirements. The estimate is based on a mine operating schedule consisting of 12 hours per shift, two shifts per day, and seven days/week. Each 12 hr shift is supported by a four-crew rotation.

The management and technical team will work five eight-hour days per week.

Table 16-40 shows the required workforce by year.

Table 16-40: Mine Labor by Year

Category	2022	2023	2024	2025	2026	2027	2028
Supervision	8	8	9	9	8	8	6
Tech Services	17	21	18	21	21	17	12
Maintenance	25	49	55	53	49	45	33
Lateral Dev.	45	45	43	34	24	16	12
Vertical Dev.	0	0	0	0	0	0	0
Production	0	34	34	34	38	38	34
Material Handling	0	0	0	0	0	0	0
Backfill	0	0	18	26	26	26	17
Mine Services	1	17	17	17	17	11	11
Total	**96**	**174**	**194**	**194**	**183**	**161**	**125**

Source: OceanaGold, 2022

The distribution of owner and contractor labor is shown in Table 16-41.

Table 16-41: Owner and Contractor Workforce Distribution

Category	2022	2023	2024	2025	2026	2027	2028
Contactors	51	53	39	6	0	0	0
OGC Employees	45	121	155	188	183	161	125
Total	**96**	**174**	**194**	**194**	**183**	**161**	**125**

Source: OceanaGold, 2022

Mine Mobile Equipment

The mine mobile equipment requirements as summarized in Table 16-42 are based on the production schedule.

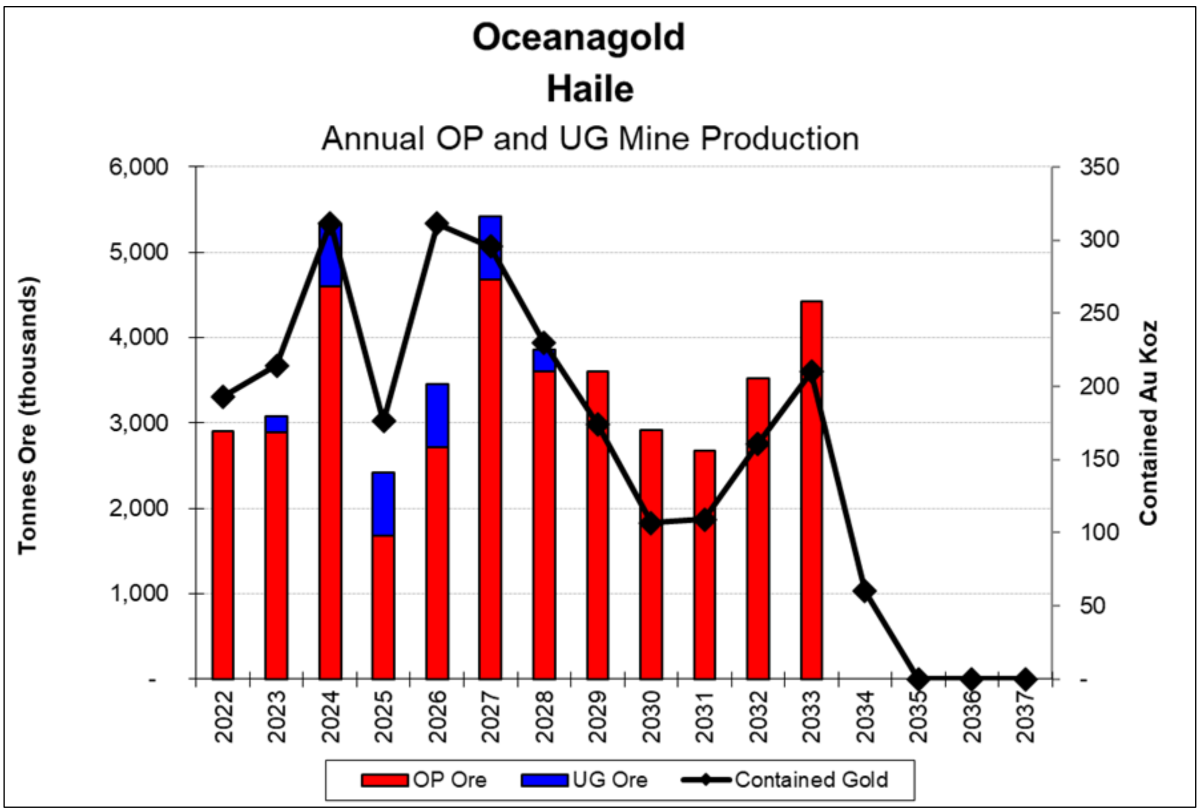
Table 16-42: Mobile Equipment Fleet

Category	Make	Model	Max Units
Development Drill	Sandvik	DD421-60C	2
Bolter	Sandvik	DS421C Bolter	1
Longhole Drill	Sandvik	DL421-7C	2
LHD	Sandvik	LH517	4
Truck	Sandvik	TH551	6
Shotcreter	Spraymec	6050WP	1
Transmixer	UTIMEC	MF 500	1
Wheel Loader	Caterpillar	962H	3
Wheel Loader	Caterpillar	938K	1
Grader	12K	12K	1
Hilux	Hilux	Hilux	1
Charge-up	Charmec	MC 605	2
Scissor Lift	UTIMEC	Scissor	1

Source: OceanaGold, 2022

16.3 Combined Open Pit and Underground Production Schedule

Figure 16-41 and Table 16-43 and show the combined open pit and underground production schedule annually.



Source: SRK, 2022

Figure 16-41: Combined Open Pit and Underground Production

Table 16-43: Combined Open Pit and Underground Production Schedule*

Year		2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	LoM Total
Total Production														
Ore	kt	2,898	3,079	5,329	2,424	3,449	5,417	3,863	3,600	2,919	2,670	3,519	4,419	43,586
Ore	t/d	7,939	8,435	14,601	6,641	9,450	14,842	10,583	9,863	7,998	7,315	9,640	12,106	9,951
Au Grade	g/t	1.68	2.00	2.05	1.83	3.00	1.83	1.88	1.43	0.85	1.01	1.40	1.87	1.78
Contained Au	koz	157	198	352	143	333	319	233	166	80	87	158	266	2,491
Ag Grade	g/t	2.32	2.17	2.30	1.55	2.06	1.85	1.88	2.53	2.43	2.77	2.57	2.69	2.26
Contained Ag	koz	216	215	395	121	228	322	234	293	228	238	291	382	3,163
Underground														
Ore	kt	-	191	736	749	739	739	263	-	-	-	-	-	3,416
Ore	t/d	-	523	2,016	2,051	2,024	2,025	720	-	-	-	-	-	1,560
Au Grade	g/t	-	4.1	4.3	3.5	4.6	3.2	2.6	-	-	-	-	-	3.8
Contained Au	koz	-	25	102	84	110	76	22	-	-	-	-	-	418
Ag Grade	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Ag	koz	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste	kt	180	306	182	133	13	6	2	-	-	-	-	-	822
Open Pit														
Ore	kt	2,898	2,888	4,593	1,675	2,710	4,678	3,600	3,600	2,919	2,670	3,519	4,419	40,170
Ore	t/d	7,939	7,912	12,584	4,590	7,425	12,817	9,863	9,863	7,998	7,315	9,640	12,106	9,171
Au Grade	g/t	1.68	1.86	1.69	1.10	2.56	1.62	1.83	1.43	0.85	1.01	1.40	1.87	1.60
Contained Au	koz	157	173	250	59	223	243	211	166	80	87	158	266	2,072.51
Ag Grade	g/t	2.32	2.32	2.67	2.24	2.62	2.14	2.02	2.53	2.43	2.77	2.57	2.69	2.45
Contained Ag	koz	216	215	395	121	228	322	234	293	228	238	291	382	3,162.74
Strip Ratio	t/t	13.2	12.1	6.0	21.3	14.2	6.8	9.2	7.6	12.0	9.9	4.9	1.0	8.7
Waste	kt	38,380	34,844	27,524	35,718	38,430	32,019	33,259	27,520	35,051	26,550	17,186	4,425	350,907

*Does not include stockpile material
Source: OceanaGold/SRK, 2022

17 Recovery Methods

A conventional flotation and cyanide leaching flow sheet is used at HGM. The process commenced commercial operation in 2017 with a nameplate capacity of 2,300,000 tpa, with a progressive debottlenecking process undertaken to upgrade the plant to the current capacity of up to 3,800,000 tpa dependent on ore competency.

In general, the response of the ore treated to the plant flowsheet has been within expectations with gold deportment and leach extractions observed to be in accordance with that predicted through the original feasibility program. Mill throughput has exceeded predictions with mill specific energy requirements lower than that originally forecast and in line with updated competency-based power modeling predictions.

Leach recovery has been observed to be affected primarily by the concentrate regrind size achieved. Flotation recovery has been impacted from blending oxidized rehandled ore with fresh sulfide ore reducing the effective recovery of sulfides in the floation circuit. Improved blend control and segregation of feed has led to improved control of flotation and overall recovery.

17.1 Processing Methods

Progressive debottlenecking and upgrades to the processing plant proceeded following successful commissioning of the process plant. The flowsheet and unit operations did not change as part of the upgrades with the target of up to 3,800,000 tpa capacity increase achieved with a reduced scope than that expected during the optimization study completed in 2016.

Benchmarking surveys of the grinding circuit were completed in 2017/18 along with additional competency testing of future ore sources. Power modeling of the circuit with several external consultants identified the Haile ore requires approximately 30% less energy than that predicted from power modeling and SMC test results. A site-specific comminution model was developed with a strong correlation between SMC test parameters and Bond ball mill work index with SAG specific energy requirements. The work provided confidence to proceed with installation of the pebble crusher and did not need secondary crushing of SAG feed for the majority of the ore types in the mine plan tested to achieve desired annualized throughput of 3.8 Mtpa. Additional surveys conducted during 2021 on ores regarded from prior core testing as being amongst the most competent in the deposit have lead to a reduction in the assumed milling rate for these types with plant throughput budgeted between 3.5 and 3.8 Mtpa depending on the feed blend expected as shown in Table 16-14.

The key additions to the process plant for the expanded capacity included:

- Upgrade to apron feeder motors for the crusher and emergency feeder
- Speed upgrade to the SAG feed conveyor to achieve 600 t/h rate
- Pebble crushing installation on existing SAG mill scats recycle and grate redesign
- Installation of a Nippon Eirich ETM-1500 tower mill (1.2MW) and 10-inch cyclone pack
- Installation of an M10000 Isamill (3MW) and six-inch cyclone pack
- Installation of a larger 14 m diameter high-rate pre-aeration thickener
- Replacement of the flotation tailings thickener feed well and rakes with an Outotec Vane Feedwell system

- Replacement of the cyanide recovery tailings thickener feed well and rakes with an Outotec Vane Feedwell system
- Installation of a second parallel interstage screen in each CIL tank and second carbon safety screen
- Installation of a third cyanide destruction tank and upgrades to the agitators of the existing two tanks
- Modifications to the strip circuit automation, barren tank management and cyanide strength to reduce cycle time to under 10 hours
- Motor upgrades to the flotation tailings and cyanide recovery thickener underflow pumps
- Upgrade to the final tailings pumps in the plant
- Installation of a tailings pump booster station at the tailings storage facility to accommodate the raising of the dam wall and discharge around the entire dam perimeter at higher tonnage
- Implementation of the Andritz expert system to cover SAG mill, ball mill, cyclone, thickening and cyanide destruction circuits to maximize throughput

The process plant consists of the following major components:

- Crushing and conveying
- Storage and stockpiling of ore and reclaim
- Grinding
- Flotation
- Fine grinding of concentrate
- Carbon in leach (CIL) recovery of precious metal values from reground flotation concentrate and flotation tailings
- Acid washing and elution of precious metal values from CIL loaded carbon
- Electrowinning and refining of precious metal value
- Thermal regeneration of eluted carbon and recycle to CIL
- CIL tailing thickening, cyanide recovery, detoxification and pumping of slurry to storage

The following section describes the plant operation currently in operation at Haile. A relatively compact Run of Mine (ROM) area is provided for storage and re-handling of ore allowing blending to minimize variation of head grade (sulfur and gold) and rock type, into the crusher. Ore is rehandled into the crusher dump pocket by Front End Loader (FEL).

Ore is reclaimed by an apron feeder onto a vibrating grizzly that delivers scalped oversize to the primary jaw crusher to reduce the ore size from RoM to minus 100 mm. Crushed ore is conveyed for surge and storage of the recombined grizzly undersize fines and primary crushed ore in a coarse ore surge bin or diverted on to an open conical emergency stockpile for later reclaim by Front End Loader (FEL) into a reclaim bin.

Ore is reclaimed from either the surge or reclaim bins, separately or simultaneously, using apron feeders onto a SAG mill feed conveyor belt delivering into the SAG mill feed chute.

Ore is milled in the SAG–Ball Mill-Pebble Crusher (SABC) circuit. The SAG mill operates in closed circuit with a vibrating discharge screen and a pebble return circuit incorporating a surge bin and Sandvik CH-440 cone crusher. The ball mill operates in closed circuit with hydrocyclones to produce the desired grinding product size of 75 microns.

Selected flotation reagents are added in the grinding circuit. A portion of the grinding circuit ball mill circulating load is treated in a flash flotation cell with the concentrate going to the regrind circuit.

The grinding circuit product passes to a bank of bulk rougher flotation cells to recover the balance of the sulfide mineralization. Thereafter, the combined flash and rougher flotation concentrates are reground in a two-stage circuit utilizing an ETM-1500 tower mill in closed circuit with cyclones to a P_{80} under 40 microns and then followed by an M10000 Isamill in closed circuit with cyclones to a target P_{80} of 13 microns.

The reground concentrate slurry is dewatered in a high-rate thickener prior to transfer to a tank for the pre-aeration step followed by cyanide leaching in a Carbon-in-Leach (CIL) circuit to dissolve gold and silver and adsorb the precious metal values from the solution onto activated carbon.

The flotation tailings slurry is thickened to recycle process water to the grinding circuit. The thickened tails slurry will be combined with the leached concentrate stream and processed in an extension of the carbon in leach circuit to recover any leachable gold and silver contained in the float tail.

The loaded carbon is removed via screens from the CIL circuit and after further treatment by acid washing to reduce calcium scaling, precious metals are stripped with hot caustic-cyanide solution. The gold and silver are recovered by electrowinning from this solution and the stripped carbon is heated in a kiln under a reducing atmosphere for thermal reactivation of its adsorption capability before being returned to the CIL tanks for reuse. The precious metal sludge from the electrowinning cells is dried and blended with fluxes and smelted to produce gold-silver doré bars, which are the final product of the ore processing facility.

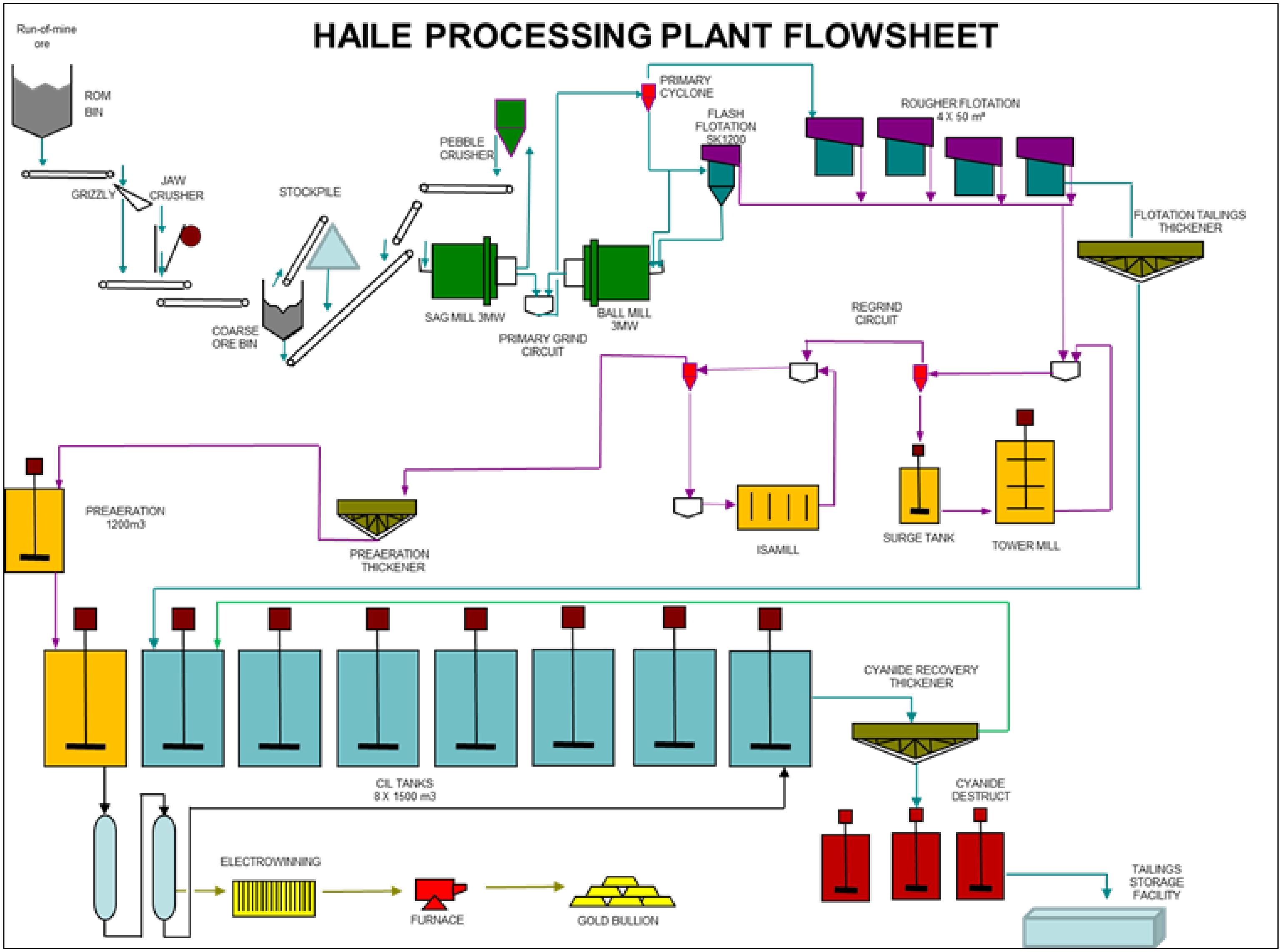
The tailing slurry exiting the CIL is dewatered in a similar thickening stage to the flotation tailings for recovery of the cyanide solution and reduction of the volume of slurry needing to be treated by oxidation of the residual cyanide. The detoxified tailing slurry is pumped for long-term storage in a lined Tailings Storage Facility (TSF) and supernatant water in the pond is recycled for reuse back at the plant.

The plant has facilities for the storage, preparation, and distribution of reagents to be used in the process. Reagents include flotation reagents i.e., Sodium isobutyl xanthate (SIBX) and frother, as well as sodium cyanide, caustic soda, flocculant, copper sulfate, ammonium bisulfite, hydrochloric acid, lime and anti-scalants. Small amounts of fresh and potable water make-up are required in the process, but the main water requirements are satisfied by internal recycle from the thickeners and tailings decant water returned from the TSF.

The process plant also operates the water treatment plant (WTP) treating contact water from the mine active pits, seepage from the PAG cells and surface water runoff from the active PAG cells. A two-stage process is utilized with pH raised to a target of 9.4 to precipitate out metals followed by clarification for solids removal, then the addition of a metals precipitant to precipitate out other heavy metals. This is followed by clarification, microfiltration, pH adjustment back to a 7 to 8 range and is then discharged to the environment. The WTP utilizes lime from the main plant ring main for pH control along with local mixing of flocculant, coagulants and metals precipitants as required.

17.2 Processing Flowsheet

The overall simplified process flow sheet for the proposed expanded plant is shown in Figure 17-1.

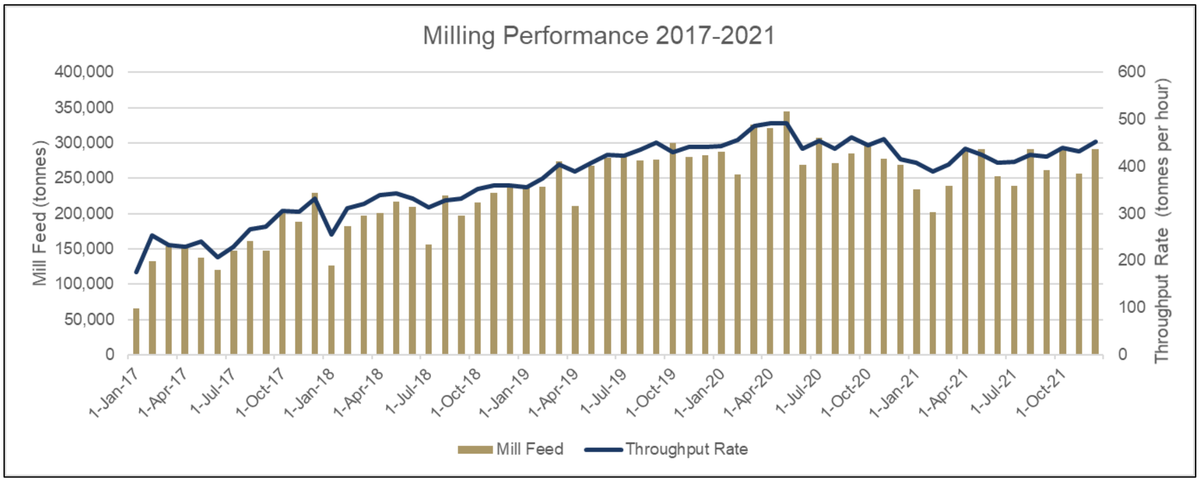


Source: OceanaGold, 2022

Figure 17-1: Haile Process Flow Sheet

17.3 Operational Results

Mill throughput ramped up following plant commissioning with nameplate of 285 t/h achieved within approximately six months. Following identification of key bottlenecks and circuit modeling with external consultants, a pebble crusher was commissioned in July 2018 along with an upgrade to the flotation tailings thickener. Mill throughput was then progressively increased as downstream restrictions were addressed. Production history and monthly throughput are shown below in Figure 17-2. Monthly milling rates achieved in the 2020-21 period have varied as a function of ore competency and oxidized clay content in feed between 389 t/h and 492 t/h.

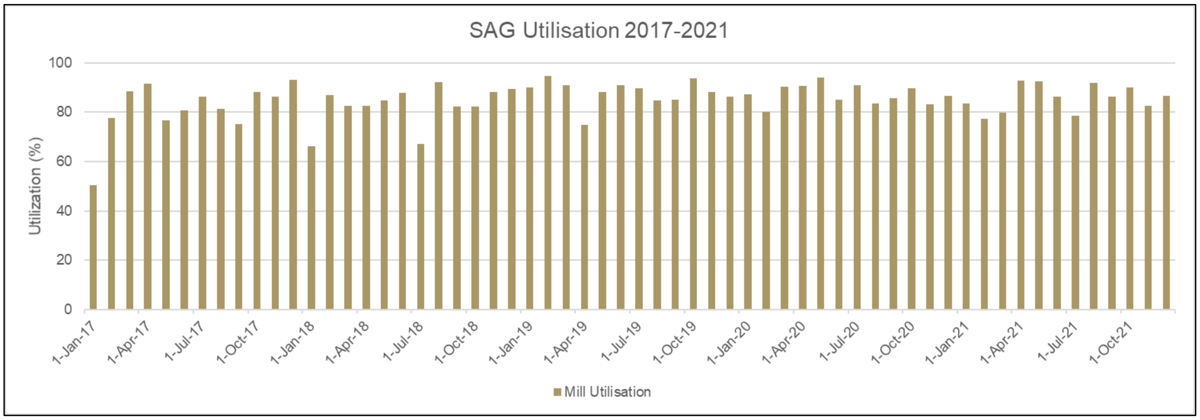


Source: OceanaGold, 2022

Figure 17-2: Mill Throughput Performance Since Startup

Mill throughput restrictions are tracked by operations and are characterized by cause. The main drivers of reduced throughput are upstream constraints in the crushing circuit during high rainfall events increasing ore moisture, ore competenecy affecting SAG mill feed rate, pumping capacity constraints in the thickening/final tailings systems or equipment failure within the plant that leads to redced throughput but not plant stoppage. No single area is the primary constraint on throughput and downtime and restriction data is used to prioritize capital investment such as the upgrade to the final tailings pumps during 2021.

SAG mill utilization has progressively improved since startup with unplanned downtime reducing as rectification of circuit design has taken place to address high wear issues in the plant. Overall mill utilization has been tracking in the 85% to 90% range over 2021 and was affected significantly in the first 5 months of the year from very high rainfall and subsequent high moisture levels in the ore supply affecting the primary crusher utilization. Long term budget assumptions for utilization target a 92% overall utilization. Historical utilization data since start up is shown in Figure 17-3.

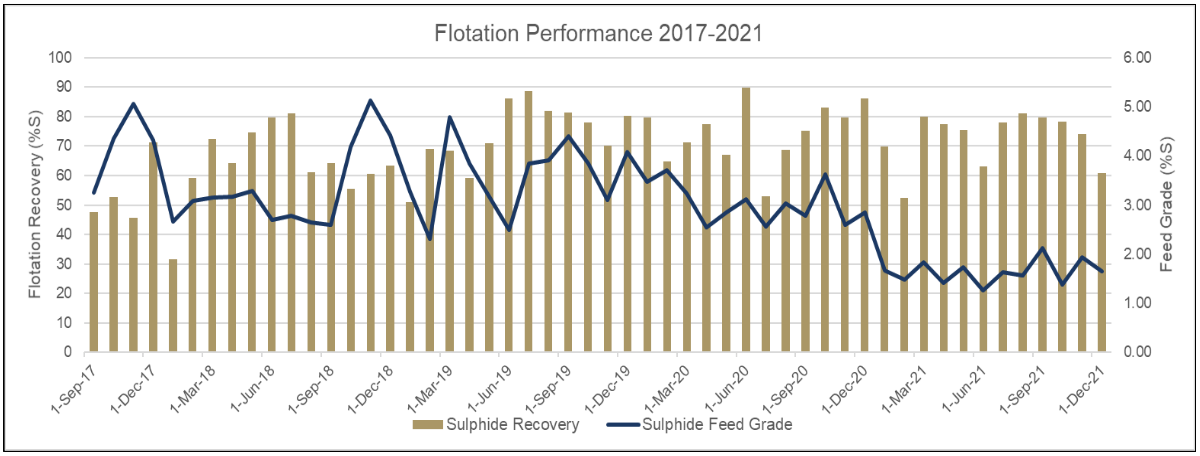


Source: OceanaGold, 2022

Figure 17-3: SAG Mill Utilization Since Startup

Flotation performance was affected post commissioning due to the bottleneck of the regrind circuit. The original circuit with two trains of three SMD mills proved unable to achieve the target P80 of 13 microns in open circuit reducing the mass pull that could be achieved. The SMD circuit was converted to closed circuit in late 2017 and changes to the maintenance strategy and media feed systems were made to maximize usable power. The sulfur recovery of the flotation circuit is shown in Figure 17-4 with regrind capacity constraining mass pull and recovery of pyrite to concentrate to around 60% until the regrind circuit upgrade was completed in May 2019 when it then approached the target 80%.

Flotation recovery of pyrite is largely affected by the proportion of oxide material in the plant feed particularly when higher proportions are required during high rainfall periods. Operating strategy is now aligned with planned campaigns of oxide material to allow maintenance windows for the regrind circuit whilst the plant is operating. In the mill feed schedule, the average sulfur feed grade is 1.6% with a maximum monthly grade of 4.2%. The average of the highest 12 periods in the mine plan is 3.7%. The design basis of the regrind circuit is based on a 5% sulfur feed grade at a 4 Mtpa feed rate ensuring this circuit will not be a restriction in the future.



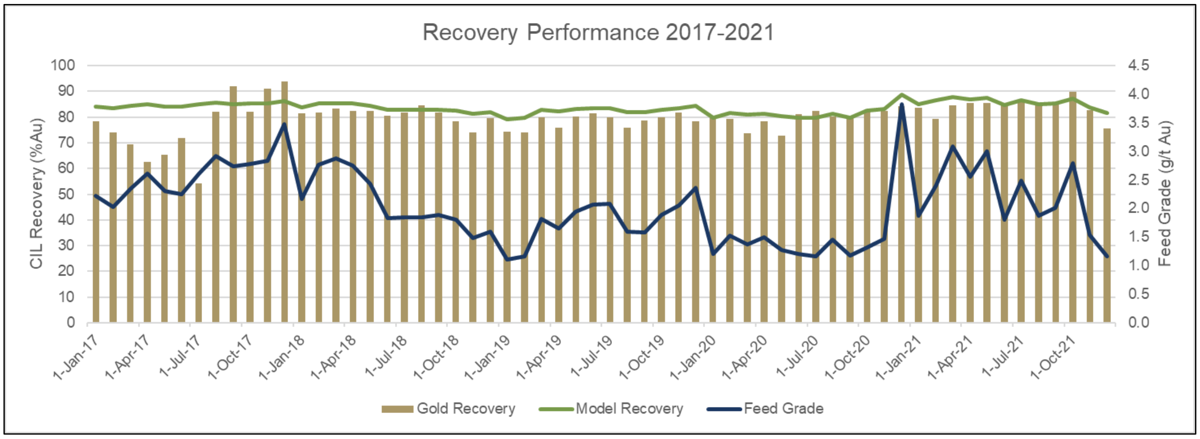
Source: OceanaGold, 2022

Figure 17-4: Flotation Circuit Recovery

Gold recovery trended at or below the feasibility study model as a function of feed grade in the first years of operation. A number of key drivers to poor recovery were identified and progressively rectified including:

- Short circuiting in the CIL tank connecting launder requiring modification and installation of downcommers completed in 2019.
- Reduced flotation recovery due to regrind capacity allowing pyrite coarser than target to reach the CIL circuit in the flotation tailings stream with poor gold liberation and reduced leach residence time. This has been progressively addressed through 2019 as regrind capacity increased.
- Coarser than design regrind product reaching the CIL circuit reducing liberation of gold, optimization of the new regrind circuit has been progressing over the last 12 months with the target 13 microns now regularly achieved.
- Issues with the regeneration kiln effective operation and utilization have had a significant impact on carbon activity leading to low levels of solution loss. Changes to equipment in the circuit has increased utilization of this circuit.
- Blending of oxide ore sources with the plant feed as plant throughput ramped up faster than mine output impacting the flotation recovery. Since May 2020 the feed strategy has changed to campaign processing any oxide material on its own straight to CIL
- The operation of the pre-aeration stage ahead of concentrate leach has been examined both internally and with external laboratories under a range of conditions and in practice does not achieve the savings in reagents envisaged. The tank was bypassed in June 2020 leading to lower leach residue grades and a subsequent reduction in cyanide destruction reagent costs from the change in process chemistry. Initial data in the 10 weeks since the change indicates an improvement in leach recovery.

Improvements implemented following the rectification of many of the mentioned issues and completion of the regrind circuit upgrade has led to the overall gold recovery improving relative to the feasibility model with the model re-adjusted upwards by 2.5% in 2020. Plant operation in 2021 in months with predominantly fresh sulfide ore feed have continued to meet this prediction, particularly with feed grades above 1.7 g/t Au. For budgeting purposes, oxide/transitional material is assigned a gold recovery of 68% gold and the modified fresh ore relationship is used for fresh ore. The leach recovery since startup is shown in Figure 17-5 along with the feasibility model and plant head grade.



Source: OceanaGold, 2022

Figure 17-5: Gold Recovery Performance Since Startup

A number of ongoing process improvement projects have been instigated to further improve recovery based on plant and laboratory test work. These include:

- Implementation of expert system control to maintain target cyanide concentration in the first three CIL tanks
- Installation of froth cameras on the flotation rougher cells for improved control of concentrate mass pull
- Monthly composite analysis by MLA and diagnostic leach tests of key process streams to understand circuit performance and gold loss opportunities
- Ongoing monthly mineralogical analysis and diagnostic leach test work on composites of key process streams to quantify circuit loss causes and allow rectification planning
- Daily lab leach testing of key process stream to evaluate daily plant performance versus lab performance to trigger investigation and/or rectification if needed

17.4 Process Unit Costs

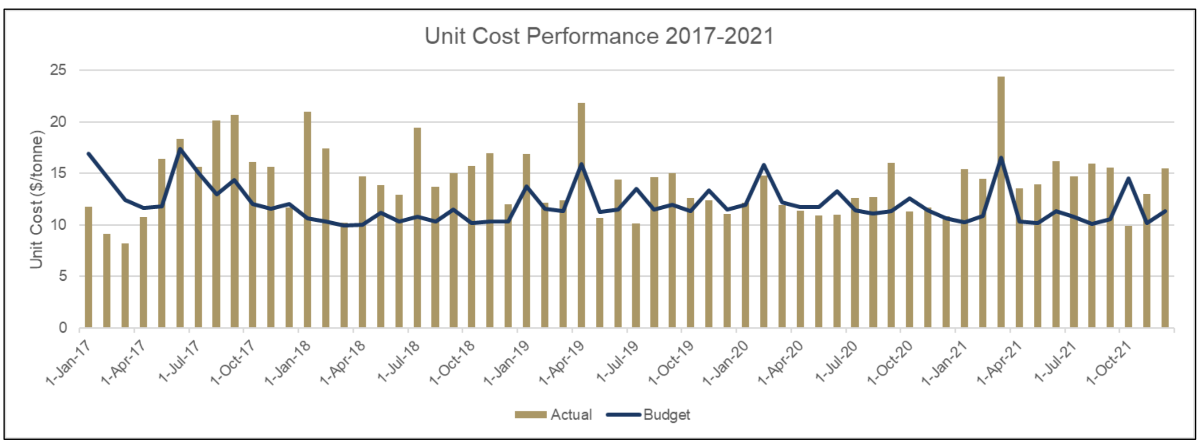
Process unit cost history is shown in Figure 17-6 below since operations commenced. Unit cost compliance to budget has improved in the last 18 months from a combination of increased throughput on the fixed cost base and improvements in maintenance planning processes and increasing knowledge of component service life. During 2017 and 2018, modifications and replacement of components occurred to address high wear or material compatibility issues inflating unit costs. In 2021, significant costs associated with repairs and modification to chutes in the crushing area to accommodate higher ore moisture without blocking lead to significant overruns on budget along with the lower throughputs experienced on a $/tonne basis. The impact of this type of expenditure is now significantly decreased.

Forward budgeting is based on a zero-order buildup based on drivers of feed tonnage or operating time. Operating experience over the last three years has allowed benchmarking of key consumable rates going forward. Key consumable consumption rates are shown in Table 17-1, the production driver metric being mill feed tonnage or concentrate tonnage depending on the area of the circuit. Power usage models for the process plant are established to calculate electrical power demand. Long-term maintenance schedules are used to identify reline activities, major overhauls and expected contractor costs.

Table 17-1: Key Consumable Consumption Rates

Consumable		Consumption Rate
2" Ball Mill Balls	kg/tonne ore	0.35
5 SAG Balls	kg/tonne ore	0.320
.75" Tower Mill Balls	kg/tonne concentrate	2.22
2.5mm ISA Mill Media	kg/tonne concentrate	1.35
Carbon	kg/tonne ore	0.040
Promoter	kg/tonne ore	0.013
Frother	kg/tonne ore	0.004
SIBX	kg/tonne ore	0.070
CuSO4	kg/tonne ore	0.150
Flocculant- Float Tails Thickener	kg/tonne ore	0.010
Flocculant- CN Recovery Thickener	kg/tonne ore	0.011
Flocculant- Concentrate Thickener	kg/tonne concenrate	0.055
NaCN	kg/tonne ore	0.60
Lime-Conditioner andPreaeration	kg/tonne ore	0.8561
Lime-CIL	kg/tonne ore	0.8641
Ammonium Bisulfate	kg/tonne ore	1.06
HCl	kg/tonne ore	1250
Caustic	kg/strip	800
Natural Gas	Kg/strip	3600

Source: OceanaGold, 2022



Source: OceanaGold, 2022

Figure 17-6: Process Unit Cost History Since Startup

The completion of the cyanide destruction circuit upgrade in Q1 2020 and implementation of expert control on the CIL and cyanide destruct process is seeing further reduction in reagent usage and costs.

At the current forecast mill throughput rates of 3.8 Mtpa, process costs are budgeted at an average of US$11.570/tonne of ore milled over the LoM.

Some increased costs have been assumed/identified in the current budget preparation round from:

- Impact of plant throughput achieving a consistent 3.8 Mtpa rate has slightly increased the fixed cost component on the overall unit cost.
- Additional maintenance repair costs around the crushing/conveying section from increased wear associated with the higher levels of ore moisture encountered
- Increased labaour rates for HGM personnel over the last 18 months
- Increased costs for key reagents from inflationary pressures on suppliers
- Slightly increased meda consumption rates on more comptetent ores

Unit costs have allowed for a progressive increase in water treatment plant capacity from the current 1,100 gpm to 2,640 gpm by the end of 2022 and the associated increased chemical consumption.

17.5 Water Treatment Plant

The processing department also operates and maintains the contact water treatment plant on site. Contact water from the mining areas (pit dewatering, PAG storage areas) is collected in a series of lined ponds and treated via a conventrional two stage pH adjustment water treatment process using lime and coagulents followed by filtration to remove dissolved metals and suspended solids. The current facility is capable of treating and discharging up to 1,100 gpm of water to the environment.

Design, permitting and procurement work is underway to expand the facility to treat up to 2,640 gpm of contact water by the addition of two additional trains of multiflo clarifiers and a new three train microfiltration facility to allow for a single stage pH adjustment followed by polishing filtration.

18 Project Infrastructure

18.1 Tailing Storage Facility

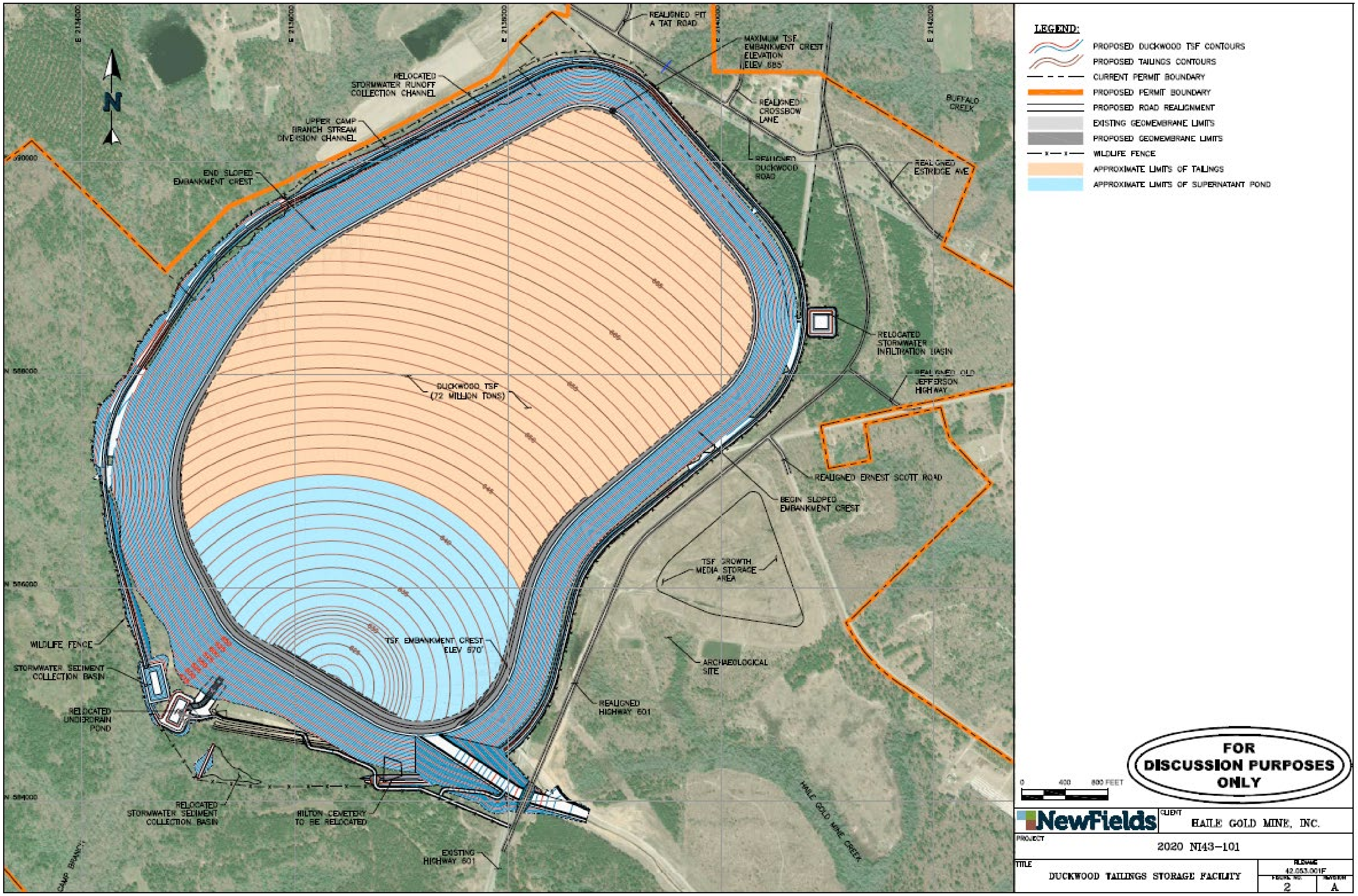
Currently, the Duckwood TSF has been permitted to be constructed in four stages to a crest elevation of 192 m with an ultimate capacity of 36 Mt. The first three stages of the embankment construction have been completed, and the crest elevation is currently at 183 m. As part of the Supplemental Environmental Impact Statement (SEIS) that has been submitted, the Duckwood TSF’s capacity will be increased to 63 Mt. The increased storge capacity is accomplished by raising the embankment crest to 204 m and sloping the embankment crest in the north corner of the facility, which is outside the predicted limits of the maximum storm water pond. The ultimate embankment will be constructed in two additional stages, bringing the total number of stages to six. All other design concepts, which include basin lining, embankment section and materials, tailings deposition scheme and freeboard requirements, will remain the same as the permitted facility. The revised Duckwood TSF layout with ultimate footprint (Stage 6) is presented in Figure 18-1. Figure 18-2 shows a tailing storage facility typical section.

In addition to raising the embankment, the footprint of the revised ultimate facility will require the following improvements that will be constructed during Stage 5:

- Realign portions of the following local roads and highways:
  - US Highway 601
  - Duckwood Road
  - Old Jefferson Highway
  - Estridge Avenue
  - Pit a Tat Road
  - Crossbow Lane
- Potentially a new haul truck overpass over the realigned Highway 601 pending final designs
- Reconstruct all perimeter runoff collection channels
- Relocate and construct a new channel for upper Camp Branch creek.
- Remove existing stormwater sediment collection basin P2 and the current TSF Underdrain Collection Pond
- Reconstruct perimeter stormwater sediment collection basins: P1, P3 and P4
- Repurposing the existing P4 stormwater sediment collection basin into the new TSF Underdrain Collection Pond
- Reshape the Existing TSF Growth Media Stockpile
- Relocate the Hilton Archaeological Site

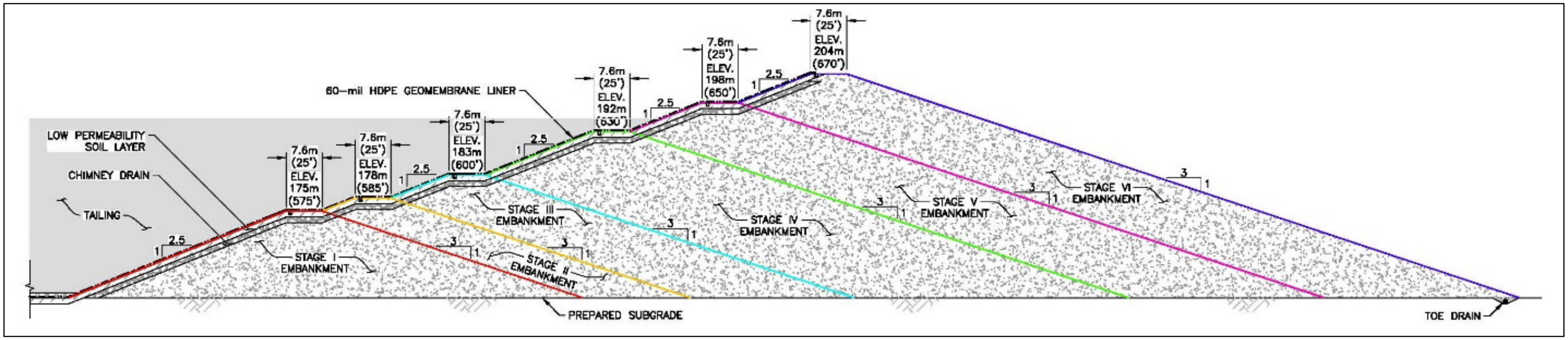
The deposition of tailings into the TSF is via a HDPE pipeline located around the perimeter of the embankment crest. Deposition will occur from multiple spigots inserted along the tailings distribution line. The deposition locations will be moved progressively along the distribution line on each stage crest, as required, to maintain slightly graded deposition of tailings towards the decant pond that is located in the southeast corner of the facility. Water from the decant pond will be recycled back to the mill for make-up water and will be reclaimed by utilizing skid mounted pumps to be located on the ramp within the southern end of the decant pond.

The TSF is designed as a zero-discharge facility. In addition to the anticipated tailing storage and operating pool requirements, the facility is designed to contain the Probable Maximum Precipitation (PMP) storm event and an additional 4 ft of freeboard at all times. Note, the existing large supernatant pond has been accounted for in the TSF design. The goal is to reduce this via enhanced evaporation and additional mill use as much as possible before construction of Stage 5.



Source: OceanaGold, 2022

Figure 18-1: Tailing Storage Facility Layout



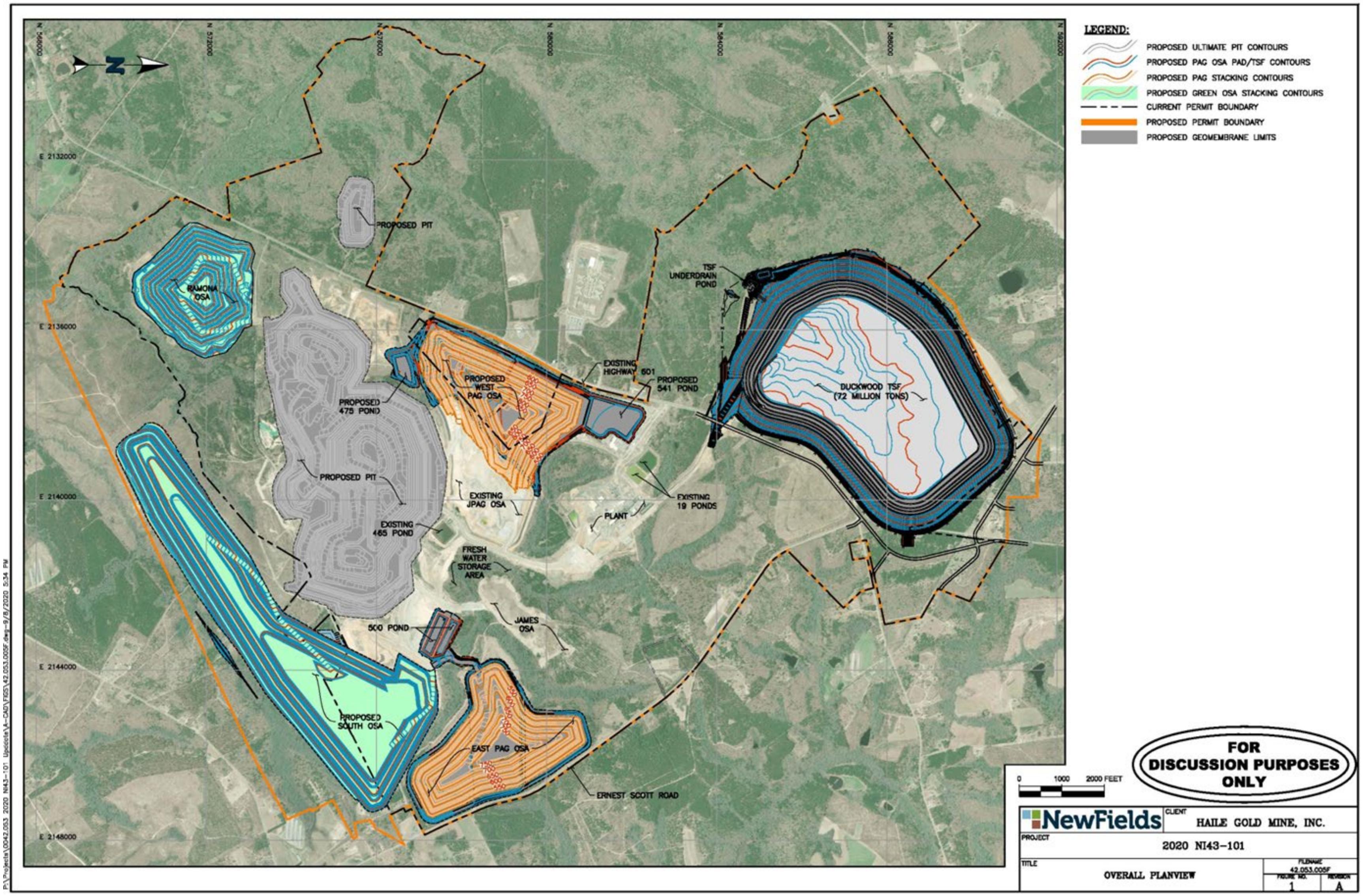
Source: OceanaGold, 2022

Figure 18-2: Tailing Storage Facility Typical Section

18.2 Overburden Storage

During the mine life from the present, five different Overburden Storage Areas (OSAs) and pit backfill will be utilized for the storage of approximately 383 million tonnes of additional material generated from the pit development. The material generated from the development of the pits will be classified as either potentially acid generating (PAG) or non-acid generating overburden material. 199 million tonnes of PAG material will be stored in either East or West PAG OSAs, or in mined out pits. A new non-acid generating OSA, South OSA, along with the existing Ramona and James OSAs, are designated for storing the remaining 184 million tonnes of non-PAG material. The OSAs will be developed according to the pit progression and the final footprint of the OSAs is presented in Figure 18-3.

Prior to construction of the non-acid generating OSAs, the footprints will be timbered. Grass lined sediment collection control channels will be constructed around the footprint of each OSA. Sediment control structures will be constructed at the outfall of the sediment control channels for each facility. Water retained within the ponds is routed through a low-level riser pipe to an adjacent drainage. All of the OSAs will be developed with a final reclaimed overall 3(H):1(V) slope.



Source: OceanaGold, 2022

Figure 18-3: Overburden Storage Areas Plan

18.3 Potential Acid Generating (PAG) Overburden Storage Areas

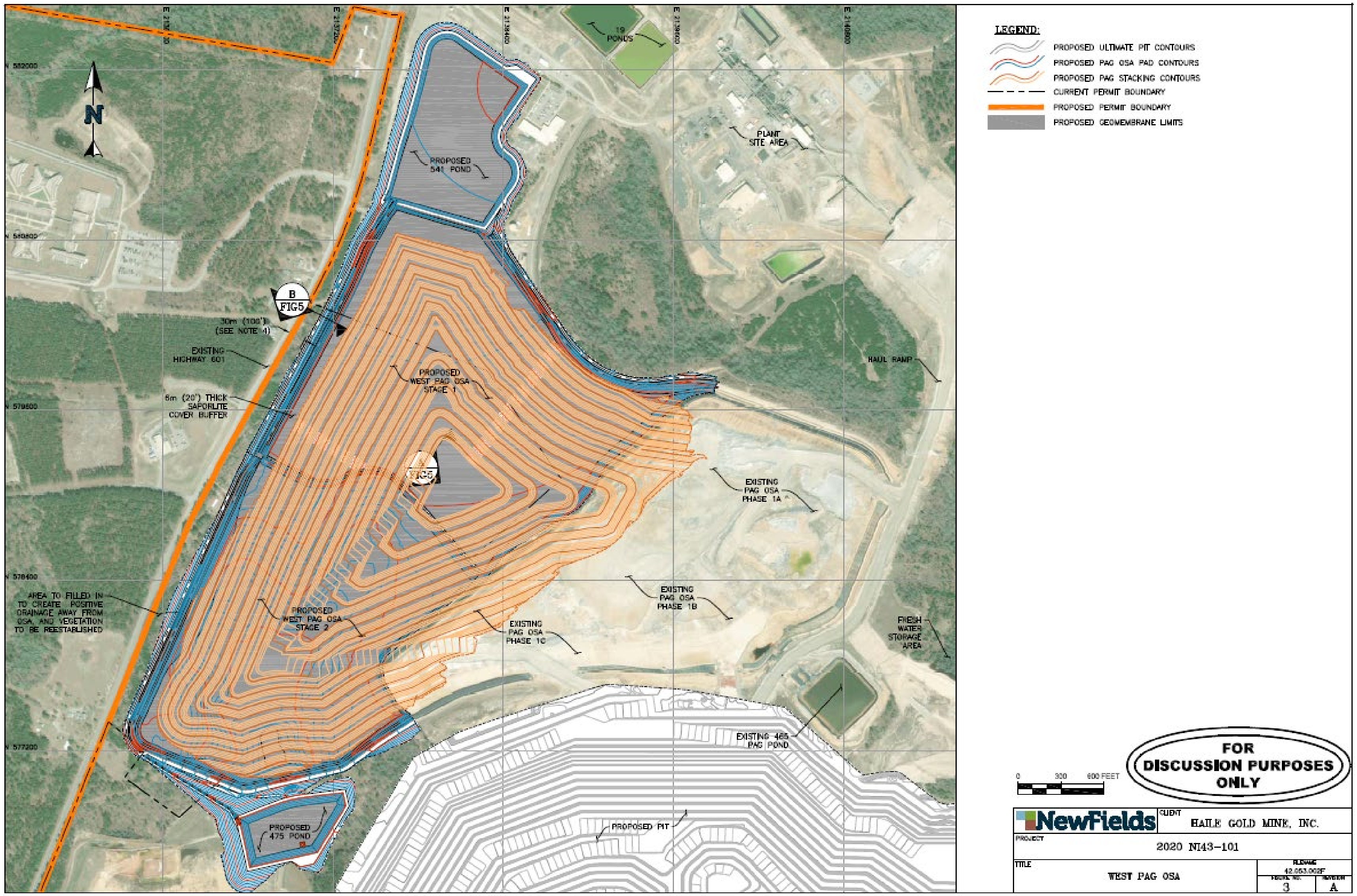
The expansion of the open pit is expected to increase the amount of PAG material that is required to be stored in fully geomembrane-lined facilities. Under the current mining permit, Johnny's PAG (JPAG) OSA and East PAG are designated as the dedicated facilities for storing the current permitted mine plan PAG material. It is estimated that the revised mine plan will generate an additional 199 Mt of which 92 Mt will need to be stored above ground. This will require the westward expanded area of JPAG with the new expanded facility designated as West PAG OSA to meet the future storage requirements.

The West PAG OSA, presented in Figure 18-4, has a total capacity of approximately 73 Mt. Similar to the permitted PAG facilities, the expanded base pad of West PAG will be lined with a composite lining system utilizing a low permeability soil layer overlain by a geomembrane. The geomembrane will be covered with a 600 mm drainage layer. A pipe network will be installed within the drainage layer to collect and transmit infiltration through the PAG material and direct flow into the contact water collection ponds.

The West PAG OSA requires two new contact water collection ponds. The pond included in the first north expansion is referred to as the 541 Pond and is sized to contain 132 million litres (ML), including an extra 37 ML for additional contact water storage for staging to the water treatment plant (WTP). The remaining 95 ML is sized to contain the predicted runoff from the 100 year/24-hour storm event on the first phase. This pond will replace the proposed 29 Pond, presented in the SEIS, which was sized to handle the additional contact water staging before the WTP. The second south expansion will drain to the 475 Pond that will replace the 455 Pond. The 475 pond is sized to hold 95 ML which equates to the predicted runoff from the 100 year/24-hour storm event for the second phase. When complete, the perimeter runoff collection channels for the full West PAG build out will drain to either the 465 (existing), 541 or 475 Pond. The PAG solution and storm water collected in the 465 and 475 Ponds will be pumped to the 541 Pond, and from there to the existing 19 Ponds for treatment and release, or for use in the milling process.

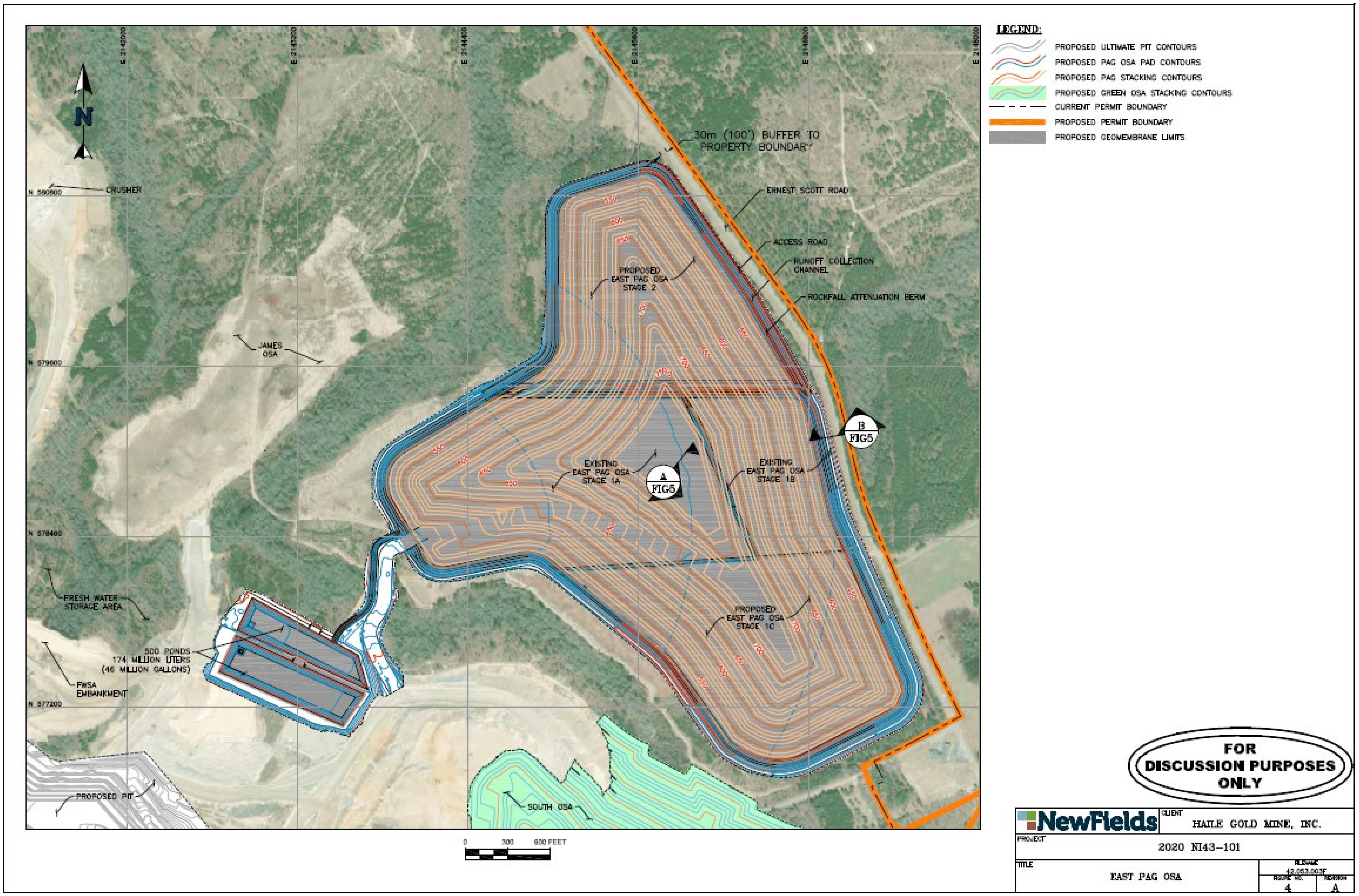
The ultimate footprint of West PAG OSA will have an overall footprint of approximately 126 ha and the PAG material will be loaded with an overall slope of 3(H):1(V).

Figure 18-5 shows the East PAG Overburden Storage Area Expansion. The remaining capacity in East PAG (19Mt), plus the West PAG (73 Mt, and pit backfill ((107 Mt) will account for all of the predicted life-of-mine PAG storage requirements.



Source: OceanaGold, 2022

Figure 18-4: West PAG Overburden Storage Area

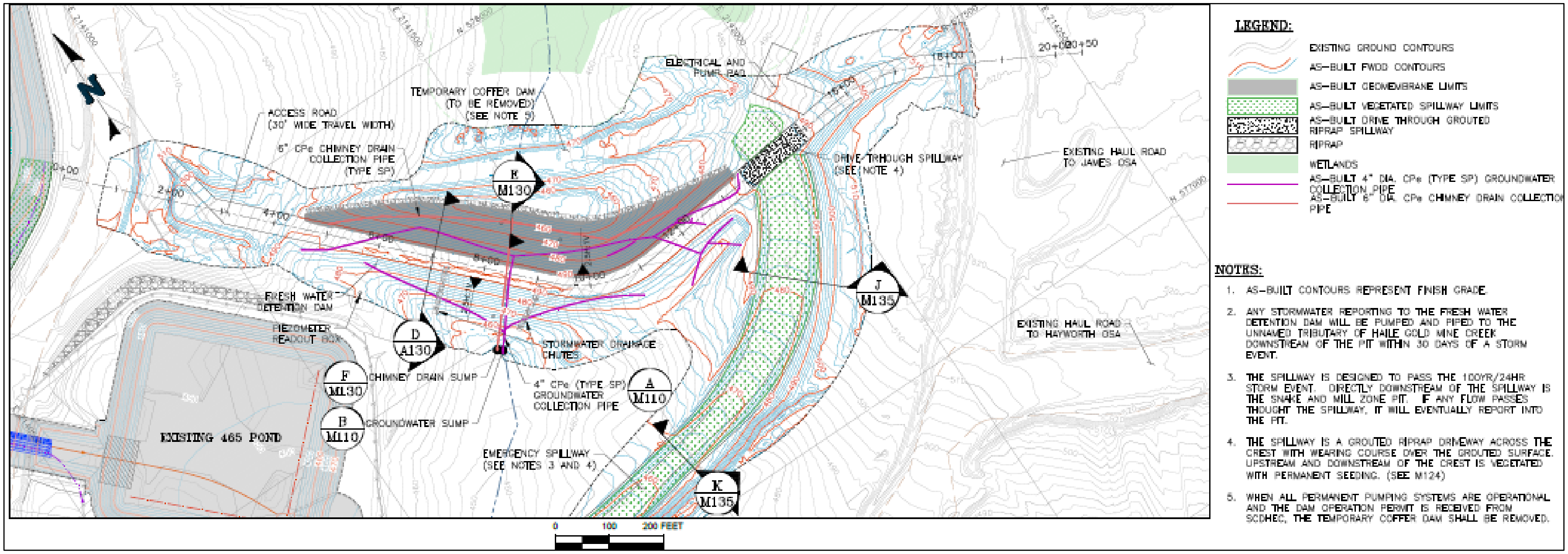


Source: OceanaGold, 2022

Figure 18-5: East PAG Overburden Storage Area Expansion

18.4 Site Wide Water Management

The existing site wide water management plan will be updated to reflect changes to the mine plan. With the revised mine plan, the incoming water from Upper Haile Gold Mine Creek will be detained by the existing Fresh Water Storage Dam (FWSD) at the upper reaches of the watershed (currently permitted as a detention dam only). The as-built dam is shown in Figure 18-6. Water from the FWSD is pumped around the pits using the existing pumps and pipelines installed with the embankment. The pumps are sized to maintain the water storage elevation below 143 m to allow for sufficient freeboard for the 100-year storm event within the upper Haile Gold Mine Creek watershed. Low flow pumps are included to maintain the minimum flow of 15.9 L/s in Haile Gold Mine Creek, per the existing mining permit. The FWSD has capacity to store approximately 590 ML of fresh water. Should the FWSD level reach elevation 148 m, an emergency spillway is sized to pass the ½ PMP event safely into the pits to allow ample time for evacuation.



Source: OceanaGold, 2022

Figure 18-6: As Built Fresh Water Storage Dam

18.5 Site Wide Water Balance

A GoldSim site wide water balance model was developed to evaluate operations associated with the Mill, TSF, contact water treatment plant (CWTP), fresh water storage dam (FWSD) and associated water management facilities. Analyses looked at multiple possible scenarios covering a range of potential occurrences. Results from the study provide a variety of potential outcomes allowing risk-based decision making. The balance includes all major facilities that are expected to add water to the system, facilities that store water, facilities that use water and facilities for water treatment/release.

Sources of water can be considered to fall into three different categories: process water, contact water and non-contact water. Contact water requires treatment before it can be released but can be used in the process. Process water includes water in the mill process or TSF which cannot be released; process water is recycled to minimize the amount of water required at the mine.

Process water comes from:

- Free water in the TSF including direct precipation on the TSF and runoff into the TSF
- Underdrain from the TSF
- Any water in the Mill process stream
- Natural moisture in the processed ore after it enters the process circuit

Contact water comes from:

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

Contact water can be used in the process as make up water or be treated in the CWTP and discharged. Non-contact water includes water that does not require treatment (beyond sediment control, as required that can be released to the environment.

Sources of non-contact water include:

- Groundwater from pit depressurization
- Municipal water
- Runoff from Topsoil Stockpiles
- Runoff from Non-PAG Overburden Storage Areas
- Runoff from Undisturbed Ground
- Runoff from TSF Outer Perimeter
- Runoff from the Plant Site (process water is contained within the process)

The results of the site wide water balance analysis indicate that under the full range of meteorological conditions evaluated, there is expected to be adequate storage in the TSF to contain process and anticipated meteorological water. Municipal supplies and non-contact water generated on site are expected to be sufficient to meet water demands. Contact water accumulation will require treatment throughout operations and can be accomplished by expanding the current CWTP capacity.

18.6 Surface Roads and Bridges

The existing Highway 601 overpass provides both a traffic crossing and a means of carrying the tailings delivery line across Highway 601 from the Process Plant. The existing overpass has been upgraded to allow fully loaded 175 tonne haul trucks to deliver random fill from the Mine to the TSF for the construction of TSF Stage 4. TSF Stage 5 may require both Highway 601, and the existing overpass, to be relocated pending final design of the TSF and road expansions.

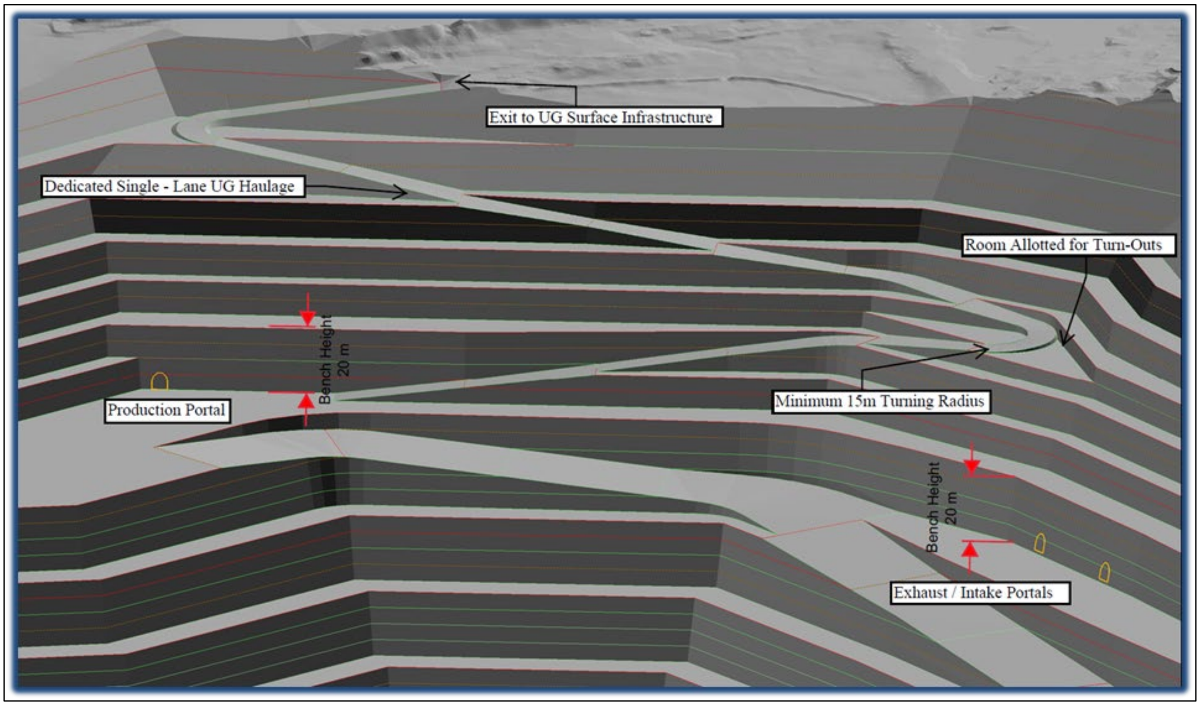
18.7 Water Supply

Fresh water required for dust suppression and the process plant will be supplied by the pit depressurization wells and meteoric water intercepted prior to running into the pits. Any excess water from the depressurization wells will be pumped to the FWSD and / or the FWST (Fresh Water Storage Tank) before releasing to the environment. Storage in the FWSD provides a source of water as required. Mill make up water will be supplied by the return water from the TSF. Surface runoff into the pits and PAG OSAs can also be used as mill make up water if necessary.. For the underground operation, a fresh water supply line (HDPE) will be constructed from the water treatment plant to the underground yard. A water storage tank will then provide clean water down the decline for underground mining equipment and water use at the underground yard. The site is connected to the town of Kershaw municipal water system for potable water supply.

18.8 Underground Access

The underground will be accessed through the highwall in the Snake pit with a total of three portals. The haul road accessing the portals will be from an open pit ramp that will tie into the haul road from the plant to the Snake pit. The haul road and bench from the open pit will create enough width to allow the installation of required fans and portable substations during the construction of the portals and declines. Figure 18-7 shows the underground access route, with the locations of the production portal and intake/exhaust portals annotated. The main fans are planned to be located underground.

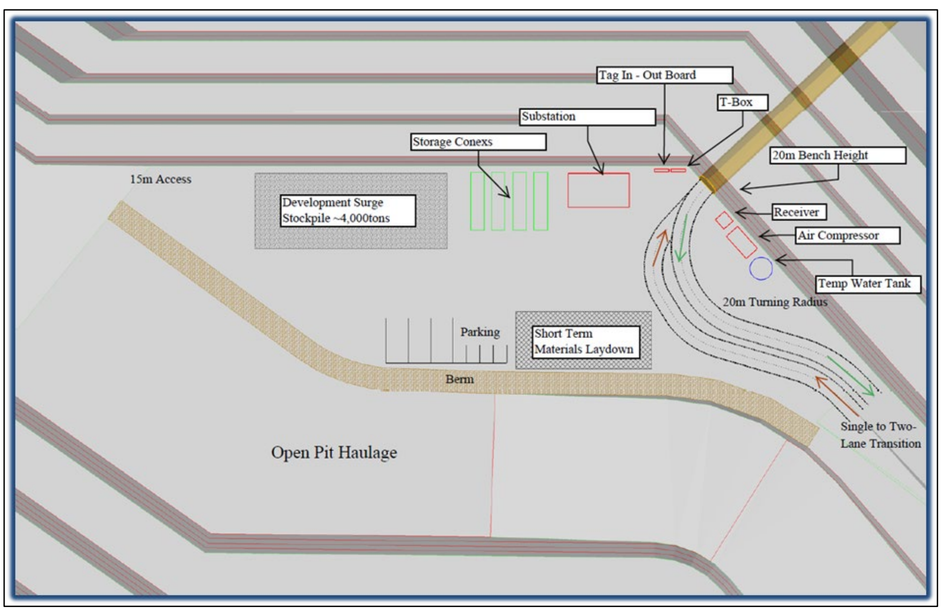
The haul road will be used to deliver underground waste to the CRF plant located above Snake Pit. The haul road and bench at the 80 RL production portal location will house a portable electrical substation, ventilation fan, compressor, laydown area, parking and stockpiling facilities to provide support for the portal and underground decline development. These portable units will be removed once additional mining has been completed for permanent underground infrastructure to be utilized.



Source: OceanaGold, 2020

Figure 18-7: Portal Access (Looking North)

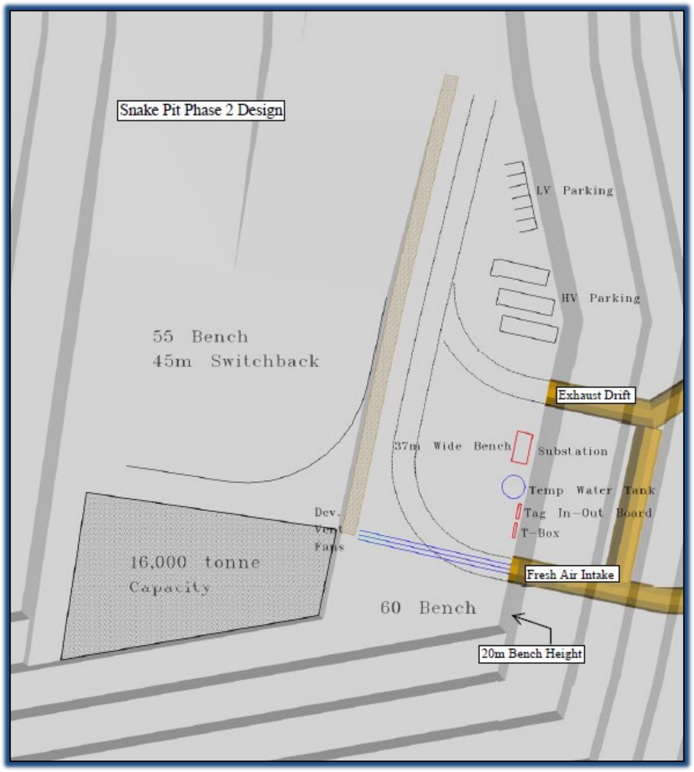
Figure 18-8 shows the layout of the planned facilities outside the production portal.



Source: OceanaGold, 2020

Figure 18-8: Production Portal Layout

Figure 18-9 shows the layout outside of the ventilation portals.

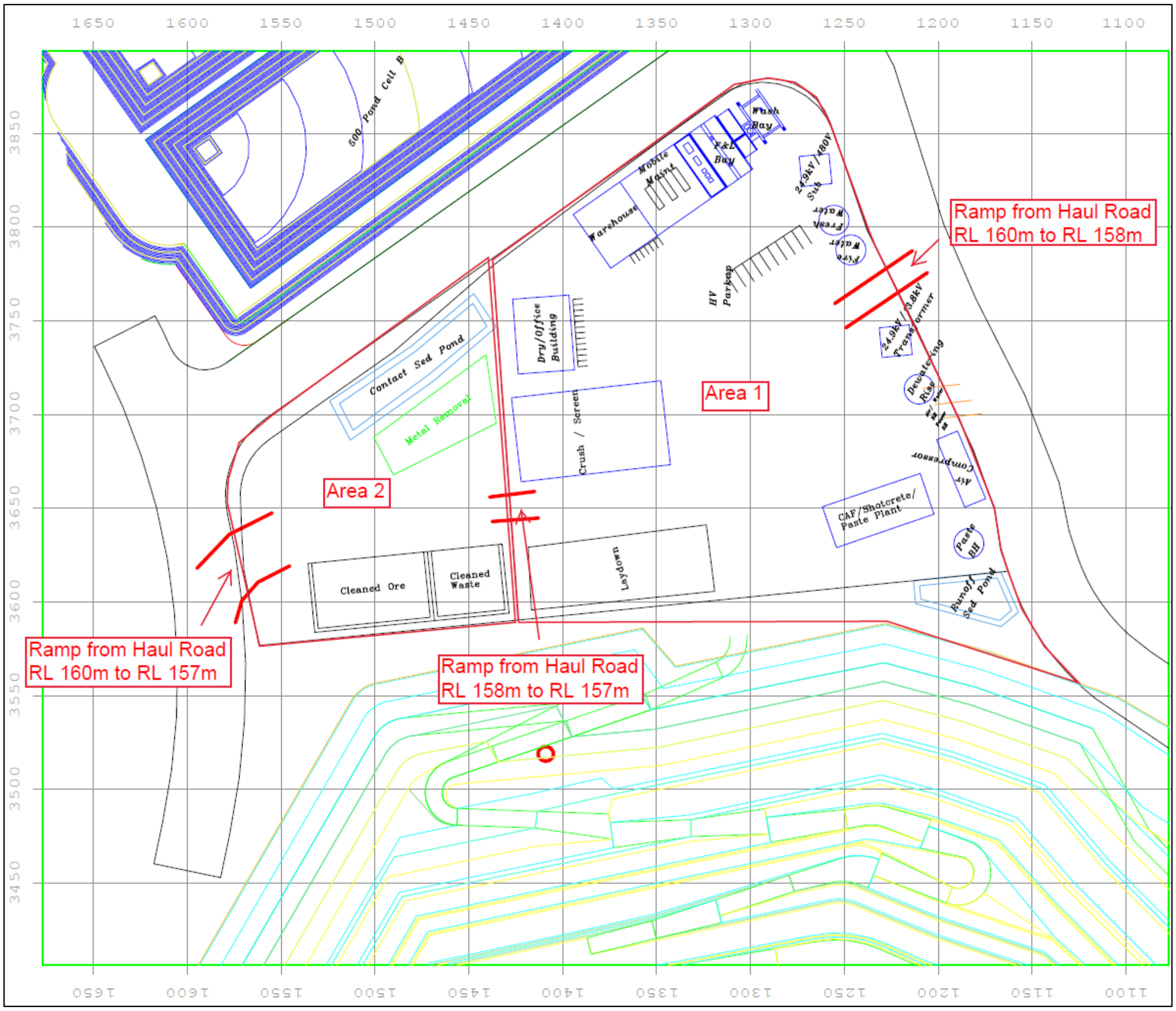


Source: OceanaGold, 2020

Figure 18-9: Layout for Ventilation Portals Area

18.9 Ancillary Facilities

The project will utilize existing facilities such as maintenance workshop, truck shop and offices to support open pit and process plant operation. The underground operation will be supported by a main laydown and infrastructure yard that will contain the underground ore stockpile, underground waste stockpile, crushing/screening plant, CRF plant, shotcrete plant, wash bay, fuel bay, mobile maintenance shop and office, warehouse, contact sediment pond, contractor office, electrical transformers, and an underground maintenance shop as shown in Figure 18-10. The design has been developed to minimize interaction between the underground mobile equipment and the surface haul road traffic.



Source: OceanaGold, 2020

Figure 18-10: Underground Surface Infrastructure Detail

The ROM Pad (Area 4) has been designated for stockpiling all underground ore, PAG and Green non-PAG waste rock material. Ore and PAG material will be rehandled into surface trucks and transported to either the processing plant or surface PAG dumps. Green waste rock from the underground will be either processed into CRF or hauled to the appropriate waste storage area. The facility also has an area allocated for extraction of waste metal, will be clay lined, and contain a HDPE lined sediment pond to control drainage. Water contained within the ROM Pad area will be pumped from the sediment pond to the 500 Pond.

The fuel and lube bay and wash bay facilities are located opposite the surface workshops. The refueling facility will consist of a double bunded tank, electric fuel pumps, oil water separator and concrete foundation.

18.10 Power Supply

The total power demand for the site (including Horseshoe underground operation) is estimated to be 23 MVA. The study undertaken by Lynches River Electric Cooperative confirmed the availability of power to site with some minor upgrades to the existing 69 kV substation and transmission line (costs associated with this is +/- 300K).

For the underground operation, the existing power line from the main substation to Pond 465 will be extended to the underground yard and a 24.9 kV to 13.8 KV transformer will serve the CRF plant. Two additional transformers (24.9kV to 480V) will provide power to the aboveground facilities on the surface. A separate take off point will feed a 6 MVA (24.9 to 13.8 kV) transformer to provide power to the underground operation. This transformer will be located on the surface and feed the underground distribution system.

19 Market Studies and Contracts

General

Haile Gold Mine has been operating continuously since 2016 and has current contracts and purchase arrangements in place for doré refining and other goods and services required for the operation.

Bullion Production and Sales

The market for gold doré 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 well understood.

- A contract is in place with Metalor USA Refining Corporation, located in North Attleboro, Massachusetts for the refining of doré bullion (Metalor). This company is a subsidiary of Metalor Technologies SA which is a well-known and established precious metal refiner. Metalor Technologies SA is a subsidiary of Japan's Tanaka Kikinzoku Group and is headquartered in Marin, Switzerland.

The contract has a two-year term starting with an Effective Date of January 31, 2020, with a current extension to 2023, subject to termination by either party. This contract also sets a range of prices and surcharges for refining the doré under terms and conditions which generally comply with industry norms. It is assumed that these contract terms will be renewed through the LoM operation without changes or will be negotiated with a new refiner, if necessary.

Hedging and Forward Sales Contracts

Haile currently has no forward sales, hedge, gold loans, offtake or similar type agreements. OceanaGold does from time to time enter into group wide pre-sales agreements. Currently, these represent a relatively minor component of total gold production. OceanaGold has four operating mines across which these can be spread.

Key contracts and status include:

- Komatsu Financial Limited:
  - Open pit mining equipment
  - Registered Address: Rolling Meadows, Illinois
  - End Date of Contract: April 19, 2024
- Caterpillar Financial Service Corporation:
  - Open pit mining equipment
  - Registered Address: Nashville, Tennessee
  - Master Service Agreement, with no expiry date
- Dyno Nobel Inc.:
  - Explosives Products
  - Registered Address: Salt Lake City, Utah
  - End Date of Contract: August 31, 2024
- Lynches River Electric Cooperative Inc:
  - Electrical power
  - Registered Address: Kershaw, South Carolina
  - End Date of Contract: December 31, 2030

- Draslovka (previously The Chemours):
  - Sodium cyanide
  - Registered Address: Wilmington, Delaware
  - End Date of Contract: December 31, 2024
- Bridgestone Americas Tire Operations:
  - Heavy equipment tires
  - End Date of Contract: December 31, 2025
- Grinding Media is bought through Molycop and Media without a contract, and through Keramos, through a master services agreement

Fuel also represents a major consumable. Fuel is purchased from Roberts Shell, Kershaw, South Carolina. No formal contract is in place.

20 Environmental Studies, Permitting and Social or Community Impact

Haile's current mine plan is based on construction, mining operation, closure, and reclamation of eight open pits, with three of those pits being left as pit lakes (Champion, Small and Ledbetter) and one as a partial pit lake (Snake). On May 24, 2018, Haile applied to the US Army Corp of Engineers (USACE) to initiate the National Environmental Policy Act (NEPA) process and launch a Supplemental Environmental Impact Statement (SEIS). United States Army Corp of Engineers (USACE) has jurisdictional responsibility for all Waters of the United States and works cooperatively with US Environmental Protection Agency (US EPA), and South Carolina Department of Health and Environmental Control (SC DHEC) for modifications such as this that have impacts to wetlands, groundwater and surface water conditions and air emissions. Since that time, Haile has submitted a Project Description, Alternatives Analysis, and 127 additional technical reports in support of this application. These technical reports cover a wide range of topics including impact assessments to the wetlands, air, land, vegetation, groundwater, surface water, flora and fauna, cultural heritage sites, socioeconomic conditions, and reclamation plans.

In order to adjust current and supplemental mine plans, a modified application to the 404 Permit - Clean Water Act has been submitted. The USACE is the lead federal Agency overseeing the NEPA process. The level of anticipated impacts to the environment from the adjusted mine plan determines the appropriate level of review under NEPA. Various permitting approvals/certifications are also required from SC DHEC, including modification of Haile's Mine Operating Permit, Reclamation Plan, TSF Dam Safety Permit, Air Permit, NPDES permits and 401 Water Quality certification. DHEC has indicated that these permits will be based on data from the approved SEIS and therefore no issues or delays are expected from them.

In addition, approvals are required from other federal and state agencies, including: United States Environmental Protection Agency (EPA), United States Fish and Wildlife Service (US FWS), South Carolina Department of Natural Resources (SC DNR), South Carolina State Historic Preservation Office (SC SHPO), South Carolina Department of Transportation (SC DOT) and Catawba Indian Nation. NEPA also allows non-governmental organizations (NGOs) and other interested parties an opportunity for review and comment on the anticipated impacts.

Haile is unique in that it occurs wholly on private land owned by HGM and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies.

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20-3) are of technical interest to the federal and state agencies in evaluating the request to expand the current mining operation.

Table 20-1 is a summary of the current HGM permits as issued under the 2014 EIS process.

Table 20-1: Mine Permits

Agency	Permit/Authorization Number	Date Received	Description
US Army Corps of Engineers	404 Permit – SAC-1992-24122-4IA	27 Oct, 2014	Permit to affect wetlands and streams per the approved Mine Plan.
U.S. Army Corps of Engineers	Permit 2004-1G-157	16 Oct, 2007	Permit to fill a portion of the old North Fork Creek
Mine Safety and Health Administration (MSHA)	MSHA ID: 38-00600	5 Feb, 2010	Operate mine within MSHA standards
Federal Communications Commission	Call Sign: WQJB814	18 Jul, 2008	Base station frequency, ten local frequencies
South Carolina Department of Health and Environmental Control (SCDHEC), Bureau of Water	401 Water Quality Certification	23 Oct, 2014	Water Quality certification to construction and operate a gold mine on HGMC, Camp Branch Creek, unnamed tributaries and adjacent wetlands.
SCDHEC, Division of Mining and Solid Waste Management	Mining/Operating Permit No. I-000601	5 Nov, 2014	Mine Operating Permit – Regulation of closure and reclamation.
SCDHEC, Bureau of Solid and Hazardous Waste Management	Permit No. SCD987596806	12 Apr, 1993	Conditionally large quantity generator
SCDHEC, Industrial Wastewater (IW) Permitting Section	National Pollutant Discharge Elimination System Discharge Permit No. SC0040479	7 Oct, 2013	Permit to discharge treated water from the mine operation / reclamation areas from Outfall 003.
SCDHEC, Industrial Wastewater Permitting Section	WTR-Wastewater Construction Permit Permit No. 19852-IW	30 Jan, 2015	Permit to construct sewer lines
SCDHEC, Bureau of Water, Industrial, Agricultural, and Storm Water Permitting Division	Dams and Reservoirs Safety Permit 29-0007 (Issued October 7, 2013)	7 Oct, 2013	Dam Safety Permit – Significant Hazard (Construction). Stability during earthquake-induced ground motion was evaluated by SCDHEC prior to issuance of the TSF construction permit. SC DHEC completed evaluation of the seismic stability study pursuant to the International Commission of Large Dam (ICOLD) design and performance standards.
SCDHEC, National Pollutant Discharge Elimination System (NPDES) Program, Water Facilities Permitting Division	General Permit for Stormwater Discharges for Small and Large Construction (Activities Permit) SCR100000	1 Jan, 2013	Discharge of stormwater in connection with construction of structures not covered under the Industrial General Permit – requires submittal of Storm Water Pollution Prevention Plan (SWPPP) and public notice prior to construction
SCDHEC, NPDES Program, Water Facilities Permitting Division	Stormwater discharges associated with industrial activity SCR000000, Permit No. SCR004763	1 Jan, 2011	Discharge of stormwater in connection with industrial activities, Industrial General Permit

Agency	Permit/Authorization Number	Date Received	Description
SCDHEC, Office of Environmental Quality, Bureau of Air Quality	Bureau of Air Quality, State Construction Permit No. 1460-0070-CA	4 Oct, 2013	Authorizes construction of the proposed facility and equipment specified in HGM's application for a Department of Army permit; a permit to operate also is required.
Lancaster County Council	Floodplain Development Permit June 27, 2013	27 Jan, 2013	Floodplain Administrator oversees and implements the provisions of the Flood Damage Prevention Ordinance.
Lancaster County Council	Ordinance 2013-1207	1 Jan, 2015	Rezoned the Haile property within the permit boundary to the M, Mining District designation.
SCDOT	177006	14 Jan, 2015	Encroachment Permit

Source: OceanaGold, 2022

20.1 Required Permits and Status

Table 20-2 is a summary of the permits necessary to start modification of the Open Pit Mine Plan, adjust the Underground Mine Plan, and installation of additional equipment for the Process Plant optimization. HGM will submit permit applications at the conclusion of the required Environmental Studies.

Table 20-2: Environmental Permits

Application	Permit Required	Regulatory Agency
Open Pit Mining Modification	Mining Operating Permit No. I-000601	SCDHEC, Division of Mining and Solid Waste Management
	Fill or modify wetlands	US Army Corps of Engineers
	Surface water impacts to Haile Gold Mine Creek, Camp Branch Creek, and Little Lynches River	SCDHEC, Bureau of Water
	Modification HGMC diversion dam	SCDHEC, Bureau of Water, Dams and Reservoirs Safety
	Expand current Mine Permit Boundaries	SCDHEC, Division of Mining and Solid Waste Management with cooperation from US EPA, US FWS, SC SHPO, and SC DNR, and Catawba Indian Nation
	New and modified existing Potentially Acid Generating (PAG) Storage Facilities and Collection Ponds	SCDHEC, Division of Mining and Solid Waste Management with cooperation from US EPA; and SCDHEC, Bureau of Water
	Additional Mine Equipment Incl. Haul Trucks, Excavators, Water Carts, Graders, Light Trees, and Generators	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Fuel Storage and Distribution	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Division of Mining and Solid Waste Management
	Explosive storage, loading and distribution	SCDHEC, Office of Environmental Quality, Bureau of Air Quality; SCDHEC, Division of Mining and Solid Waste Management; US BATF
Underground Mine	Mining Operating Permit	SCDHEC, Division of Mining and Solid Waste Management
	Ventilation Emissions, Stockpiles,	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Cement Plant Permit	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Division of Mining and Solid Waste Management
	Groundwater disturbance	SCDHEC, Bureau of Water
	Underground Depressurization Water Wells	SCDHEC, Division of Mining and Solid Waste Management and SCDHEC, Bureau of Water
	Additional Mine Equipment Incl. Haul Trucks, Excavators, Water Carts, Graders, Light Trees, and Generators	SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Explosive storage, loading and distribution	SCDHEC, Office of Environmental Quality, Bureau of Air Quality; SCDHEC, Division of Mining and Solid Waste Management; US BATF
Process Plant Optimization	Additional Process Plant Equipment, Modified ROM Pad, and Modified Stockpile storage	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Bureau of Water
	Modified Equipment Run Rate	SCDHEC, Office of Environmental Quality, Bureau of Air Quality and SCDHEC, Division of Mining and Solid Waste Management
	Expand Tailing Storage Facility surface area	And SCDHEC, Division of Mining and Solid Waste Management and SCDHEC, Office of Environmental Quality, Bureau of Air Quality
	Modified Tailing Storage Facility Dam Height	SCDHEC, Bureau of Water, Dams and Reservoirs Safety and SCDHEC, Division of Mining and Solid Waste Management
	Re-route Highway 601	SC DOT

Application	Permit Required	Regulatory Agency
	Modification to Earnest Scott and Duckwood Roads	Lancaster County DOT
Extended Support Services	Extended Operation of Water Treatment Plant	SCDHEC, Bureau of Water
	Extended Site-wide Monitoring	SCDHEC, Division of Mining and Solid Waste Management

Source: OceanaGold, 2022

20.2 Environmental Studies

Table 20-3 is a summary of the environmental studies initiated by HGM to support the approval process.

Table 20-3: Environmental Studies

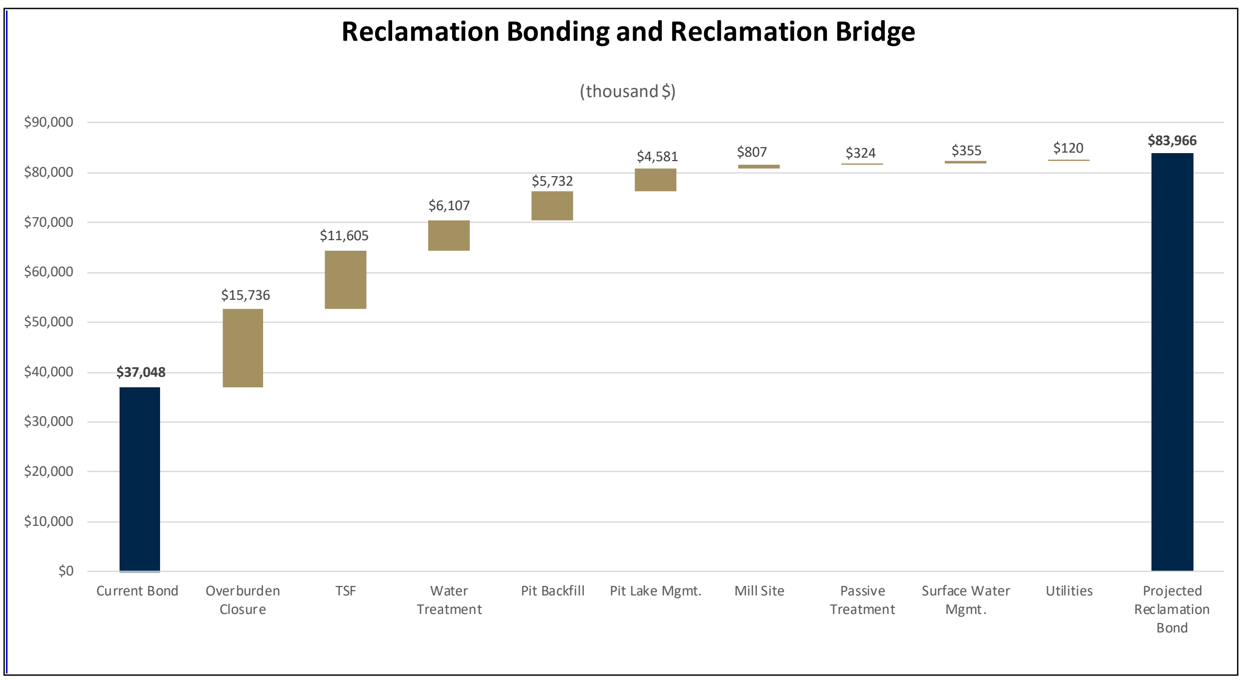
Study	Scope of Work
Air Emissions	Assess impact to air pollution loading based on additional operating conditions and new equipment on active point sources – engine exhausts, conveyor drop points, discharge stacks, ventilation shafts, dust controls, blast gasses, cement plant, etc.
Aquatic Resources – plants and animals	Review and assess Carolina Heel Splitter mussel fish and macroinvertebrate studies to quantify impacts to aquatic species over LoM.
Cultural and Historical Resources	Review and assess potentially impacted cultural and historical sites from surface disturbances. Relocate any potential gravesites.
Economic Impact to Local Community	Social and economic effect on the state and local economy – effect on local businesses, wages, local resources, emergency services, and external jobs.
Emergency Response Plans	Assessment of coordination with available local emergency support services.
Floodplain Assessment	Assess surface water impacts (flows and water quality) based unanticipated concurrent failures of the TSF, storage ponds, and Process Plant containment areas.
Geochemistry Analysis	Update OMP for PAG material placement, ARD control, and underground cut and fill practices.
Geology and Soils Assessment	Update an assessment of suitable materials for future reclamation actions.
Groundwater Modeling	Assess potential impacts to neighborhood wells, ponds and springs.
Hazardous Materials and Waste Inventory	Review of the chemicals, reagents, fuels, and hydrocarbon products transportation, storage, distribution, and disposal.
Health and Safety Assessment	Review of industrial hygiene monitoring and potential impacts to employee health.
Hydrology Assessment	Identification of direct and indirect impacts on local wetlands, wells, and streams.
Impacted Wetland Assessment	Assess potentially impacted wetland areas – vegetation and stream flow.
Land Use	Assess alternatives to potentially impacted land masses and identify future potentially higher value applications.
Noise and Vibration Study	Identify potential impacts from noise and vibration sources, including blasting activities, mobile equipment operating at elevated surfaces, crushing and grinding equipment, TSF evaporators, and mine equipment (haul trucks, dozers, and excavators)
Reclamation Plan	Pit closure and remediation plans, surface controls, revegetation plans, stormwater control plans, surface run-off plans, timelines, and sequence for surface reclamation activities.
Socioeconomic Impacts	Assess the socioeconomic impacts to state and local communities.
Stormwater Plans	Create Stormwater plans for sediment ponds, borrow pits, location for BMP[1] devices, assessment locations, and site controls.
Surface Water Impacts	Assess impact to surface water flows – volume, dissolved oxygen, chemistry, pH and conductivity. Assess drainage patterns and develop recommendations for additional monitoring and measurement stations, if required.
Terrestrial Resources – Plant Life	Perform terrestrial plant evaluations, specifically in impacted areas and areas of significant disturbance.
Terrestrial Resources – Wildlife	Perform seasonal terrestrial studies on migratory endangered wildlife, such as species of bats and raptors.
Transportation and Traffic / Road Impacts	Perform a traffic study and predict road patterns and potential impacts based on employment and support service usage.
Vibration Analysis	Develop vibration predictions based on underground blasting and changes in surface contours and geological morphology.
Visual Impact Studies	Assess visual impacts along major thoroughfares: Highways, County Roads, Neighborhoods and Public spaces
Water Quality Assessment	Assess current and future impacts to changes in water quality based on volume changes, available precipitation and evaporation, and geochemical leach studies.
Wetland Delineation	Create surface maps with jurisdictional delineations. These will include necessary 50-ft (15.25 m) offset boundaries.

Study	Scope of Work
Wetland Delineation and Marking	Set the visual markers around the 50-ft offset. Measure the final size of wetland delineation.
Wildlife Assessment	Biological assessment of primary and secondary wildlife species and identify impacts and changes to migratory patterns, mortality changes and secondary food sources.

Source: OceanaGold, 2022
[1] Best Management Practice Devices – 1) Erosion Prevention – slope surfaces, seeding, and erosion controls; and 2) Sediment Control - check dams, sediment dams, sediment ponds, and silt fencing.

20.3 Environmental Issues

As required by Haile's Mine Operating Permit, a progressive US$55 million (M) Bond and a US$10 million Reclamation Trust Agreement is in place between HGM and SCDHEC. Currently, US$44.4 million has been paid under the agreed upon schedule with an additional US$9.7 million paid to cover two minor modifications. Under the SEIS, the proposed bond is increased to US$78 million. It is to provide financial assurance to the State of South Carolina that funds will be available (in the event of default by HGM) to implement and complete the Reclamation Plan and for implementing, maintaining, repairing, or enhancing any aspect of reclamation, closure, and post closure activities. The financial assurance is in the form of surety bonds and an interest-bearing trust account. Under the changes outlined in the NI43-101, the bond requirements are anticipated to increase to $84 million, as shown in Figure 20-1, predominately associated with reclamation of the larger PAG facilities.



Source: OceanaGold, 2022

Figure 20-1: Reclamation Bond and Reclamation Bridge

21 Capital and Operating Costs

21.1 Capital Cost Estimates

A summary of the total capital cost is provided in Table 21-1; the basis of the capital cost estimate is discussed below.

Table 21-1: Total Capital Cost Summary (US$000)

Description	Sustaining Capital	Non-Sustaining Capital	Total
General Operations Expenditure	28,456	-	28,456
Operations Based Mining Projects	292,829	-	292,829
Capitalised Pre-Strip / UG Mine Development	427,660	-	427,660
Greenfields Exploration	-	1,933	1,933
Stand-alone Mining Projects	-	82,049	82,049
Rehabilitation (Non-Sustaining)[1]	-	71,072	71,072
Total LoM Net Capex	**$748,945**	**$155,054**	**$903,999**

Source: OceanaGold, 2022
(1) Captured as Capex in Cashflow

21.1.1 Basis for Capital Cost Estimates

The capital cost estimates throughout this section has a base or effective date of December 31, 2021. All values are in United States dollars (US$), and no foreign currencies have been considered in the estimates. Contingencies applied to capital estimates are considered appropriate and have been variably applied to reflect the source of the estimate.

Open Pit

All major mining equipment is currently under finance lease arrangements. For NI 43-101 financial reporting purposes, these leases are treated as Indirect costs. All LoM equipment costs under lease arrangements in place and assumed as provided in Table 21-2.

Table 21-2: Open Pit Mining Equipment Capital Leasing – Indirect Costs

Item	Units	LoM Total
Principal Payment - Capital Leases: Non-Sustaining	$000’s	53,832
Principal Payment - Capital Leases: Sustaining	$000’s	49,158
Interest Expense - Capital Leases	$000’s	5,495
Total – Capital Leases	**$000’s**	**108,485**

Source: OceanaGold, 2022

Main open pit capital items include capitalized pre-stripping, major rebuild capital, and supporting equipment and infrastructure not covered by capital lease arrangements, as provided in Table 21-3. Pit dewatering and site work include items to support the highwall depressurization and overall site wide water management.

Table 21-3: Open Pit Mining Capital

Item	Units	LoM Total
OP Capitalized Pre-Strip	$000’s	427,660
OP Mining PP&E	$000’s	99,586
OP Tech Services PP&E	$000’s	975
Pit Dewatering	$000’s	18,550
Site Works	$000’s	6,200
Total	**$000’s**	**552,971**

Source: OceanaGold, 2022

Underground

All major mining equipment is supplied, and sustaining maintenance carried out, under operating lease arrangements. For financial reporting purposes, leases are treated as Indirect costs. LoM equipment costs under lease arrangements in place and assumed are as provided in Table 21-4.

Table 21-4: Underground Mining Equipment Capital Leasing – Indirect Costs

Item	Units	LoM Total
Principal Payment – Capital Leases: Non-Sustaining	US$000’s	20,984
Interest Expense – Capital Leases	US$000’s	874
Total – Capital Leases	**$000’s**	**$21,858**

Source: OceanaGold, 2022

Projected capital costs for underground mining are summarized in Table 21-5.

Total remaining initial capital costs are estimated to be US$59.0 million. Total sustaining capital costs are estimated to be US$20.1 million.

The underground capital estimate includes capitalized development, mine infrastructure and equipment not sourced under capital leasing arrangements.

Table 21-5: Underground Capital Cost Summary

Description	Project Phase	Growth	Total Cost
	(US$000’s)	(US$000’s)	(US$000’s)
Supervision and Control	13,089	4,222	17,311
Lateral Development	18,954	3,609	22,563
Vertical Development	1,140	2,943	4,083
Production	0	0	0
Material Handling	1,402	1,328	2,730
Mine Services	2,305	1,378	3,683
Diamond Drilling	1,330	0	1,330
Rebuilds	0	3,461	3,461
Infrastructure - Underground	5,365	1,856	7,221
Infrastructure - Surface	14,891	90	14,981
Light Vehicles	555	1,179	1,734
Total	59,030	20,066	79,096

All capital costs incurred after production commences in August 2023 are considered sustaining capex.
Source: OceanaGold, 2022

Process Plant

The Haile process plant has been progressively upgraded from 2017 to 2019 to increase milling rates up to a 4.0 Mtpa rate and improve gold recovery to the design rates. Work is substantially completed on the process upgrades over the last three years. Process plant sustaining capital over the remaining life of mine totaling US$13.8 million is principally related to:

- Progressive infrastructure upgrades to support tailings dam lifts and the associated impact on tailings pumping.
- Expansion of the contact water treatment plant to 2,640 gpm capacity.
- Debottlenecking and minor pumping upgrades within the plant to maintain milling rates.
- Upgrades to the stripping circuit to increase rates with higher gold production >200 koz pa.

Cost estimates are based on preliminary scopes or vendor estimates for upgrades to the water treatment plant. A progressive tailing pumping study was completed in 2019 to confirm pumping requirements for each stage of the dam raising at the higher 4.0 Mtpa design milling rate and provided required changes to the pumping systems.

Infrastructure – Tailings/Overburden

Infrastructure capital associated with tailings/overburden is estimated to be US$152 million for the LoM. Infrastructure capital has been estimated internally by OceanaGold or provided by external consultants. The major capital items are PAG storage, tailings storage facility expansion, and Non-PAG OSA and minor water management facilities.

The remaining capital items have been sourced from the 2021 Haile LoM plan. The major infrastructure items, such as the original JPAG and EPAG first phase, tailings storage facility Stage 1 2, and 3, East PAG and FWSD and other water management and open pit dewatering, which were budgeted are excluded as these have been completed.

The site works category in the open pit capital cost carries the Green OSA storm water pollution prevention plan controls (SWPPP).

Other Capital

Other capital required for the life of mine plan is as shown in Table 21-6 and covers land acquisitions, permitting and surface drilling.

Table 21-6: Other Capital

Item	Units	LoM Total
Land Acquisitions	$000’s	1,887
Permitting	$000’s	2,040
On-Site Exploration Drilling	$000’s	3,700
Total	**$000’s**	**7,627**

Source: OceanaGold, 2022

21.2 Operating Cost Estimates

The total RoM operating cost unit rate of US$32.94/t processed is summarized in Table 21-7.

Table 21-7: RoM Operating Cost Summary

Description	US$000's	US$/t mined
OP Mining ($/t rock mined) - All Material	545,064	2.49
UG Mining ($/t rock mined)	193,432	44.98
Description	**US$000's**	**US$/t Ore Processed**
Subtotal Mining (Operational Material Only)	738,496	16.24
Processing	529,387	11.64
G&A Cost	222,613	4.90
Refining/Freight Costs	5,732	0.13
Total Operating Costs	**$1,495,827**	**$32.94**

Source: OnceanaGold, 2022

There are several important cost items excluded from the operating cost, as detailed in Table 21-8, which OceanaGold does not consider to be direct operating costs but the operation does incur.

Table 21-8: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond (Expenditure Only)	14,800	0.33
Interest Expense - Capital Leases	5,495	0.12
Principal Payment - Capital Leases: Sustaining	49,158	1.08
Principal Payment - Capital Leases: Non-Sustaining	53,832	1.18
Total Non-Operating Costs	**123,285**	**$2.71**

Source: OceanaGold, 2022

21.2.1 Basis for Operating Cost Estimates

The operating cost estimates throughout this section has a base or effective date of December 31, 2021. All values are in United States dollars (US$), and no foreign currencies have been considered in the estimates. No contingency has been applied to operating cost estimates for open pit mining, processing or G&A. Contingencies have been variably applied as appropriate to operating cost items for underground mining.

Open Pit

Projected operating costs for mining have been developed based on mine production and equipment schedules over the life of the mine, with reference to current actual costs and assumed productivity improvements from 2023 onwards.

Productivity improvements have been assumed from 2023 that will result in further unit cost reductions over the LoM. Key initiatives will include:

- Construction and/or maintenance of all-weather surface haul roads to reduce lost time due to weather events, increase haulage fleet travel speeds and improve haulage fleet tire life.
- In-pit water management addressing pre-mining dewatering bores, bench drainage and on-bench road construction to reduce lost time due to groundwater and weather events.
- Increased equipment utilization realized through practices to reduce standing time during shift breaks and at shift change.
- Utilization of remote and autonomous operation capabilities of the blasthole drill fleet, reducing labor costs and increasing operating hours.
- Utilization of real-time payload monitoring technology on the primary excavator fleet to consistently meet target haulage fleet payloads.

- Drill and blast optimization and supplier contract negotiations.
- Increased excavator productivities, through upskilling of operators, reduction in hang and queue times, double-side loading, and improved material fragmentation.

The average mining cost over the life of the mine is US$2.49 per tonne of material mined. The cost by activity is presented in Table 21-9.

Table 21-9: Open Pit Mining Cost Summary

Description	$/t mined [(1)]	Total $000's
Drill and Blast	0.48	189,067
Load and Haul	0.71	278,966
Ancillary	0.62	241,137
Maintenance	0.30	117,480
Management	0.10	39,366
Technical Services	0.27	106,710
Total	2.49	972,725

Source: OceanaGold, 2022
(1) Ore + Waste

Underground

Projected operating costs for underground mining have been developed based on the LoM production schedule. The average cost of underground ore mining is US$48.67/t and the cost by activity is presented in Table 21-10.

Table 21-10: Underground Mining Cost Summary

Description	US$/t-ore Mined	US$000's
Diamond Drilling	0.00	$
Supervision and Control	13.33	$45,530
Lateral Development	7.02	$23,988
Vertical Development	0.00	$
Production	8.34	$28,493
Material Handling	4.27	$14,589
Backfill	11.03	$37,690
Mine Services	4.67	$15,969
LoM UG Mining Costs	**48.67**	**$166,259**

Unit costs are based on 3,416,327 t of ore mined, which includes 15,101 t of pre-production ore mined during Jul and Aug 2023.
Source: OceanaGold, 2022

Process Plant

The power cost component of the estimate is based on current power consumption for each area of the plant with allowance for increased loads from planned equipment upgrades. The current unit energy cost rates in the existing power supply agreement with the power supplier to the current operation (Lynches River Authority) were used.

Labor costs were developed based on the current staffing plan for the plant reflecting the four-panel operations roster and maintenance support roles with a total head count of 90 people, using the current labor rate schedules.

The reagent and grinding media consumption estimates are based on forecasts used in the current Haile LoM plan, adjusted for concentrate mass pull predictions from head grades and expected improvements from improved control of the CIL and cyanide destruct circuits.

Crusher and mill liner replacement costs are based on vendor pricing for current supply of components and a long-term reline schedule for the life of mine based on life predicted on tonnage treated developed over the last three years of operation.

Maintenance costs are based on forecast consumable rates for each area of the plant from operating experience since startup. Contractor costs are based on expected usage based on recent experience to support shutdown and rebuild activities.

Miscellaneous costs cover assay laboratory charges assigned to the process plant and other minor ad-hoc expenses such as software license and lease fees, technical consultancy services, development test work and advisors fees, etc.

The breakdown of the processing operating cost estimate is summarized in Table 21-11.

Table 21-11: Summary of Process Operating Costs

Description	$/t milled	US$000's Per Annum
Power	1.40	5,201
Labor	2.88	10,695
Grinding Media	1.20	4,449
Reagents	3.30	12,265
Mill Liners	0.48	1,777
Maintenance Materials	1.41	5,229
Miscellaneous	0.58	2,141
Assay	0.33	1,224
Total	**11.57**	**42,983**

Source: OceanaGold, 2022

Selling and Refining

Sales refining charges are charges incurred in the sale and transport of material to the refiner and are listed below. These total US$5.7 milion over the LoM or US$0.13/t processed.

Gold

- 99.996% payable Au
- US$0.70/troy oz Au treatment charge
- US$0.24/troy oz Au + US$785/shipment freight costs

Silver

- 99.0% payable Ag
- US$0.70/troy oz Ag treatment charge
- US$0.24/troy oz Ag freight cost

General and Administration

General and administration (G&A) costs are using Haile 2021 actual expenditures as a baseline, of which a high-level summary is shown in Table 21-12. G&A costs adjusted early in the LoM for activities related to permitting and CI establishment. Then, costs decrease per annum from 2025 thereafter based on improvements in site efficiencies, as well as reductions in corporate overheads.

The total G&A will reduce year-on-year from 2030 as mining rates reduce toward the end of mine life in 2033, decreasing significantly in 2034 when the plant facility is processing material from low grade stockpile in alignment with the mine plan.

The overall life of mine total is US$223 million, or US$4.90/t processed.

Table 21-12: General and Administration Cost Summary (US$000's)

Description	2021	2022 to 2030 p.a. OP + UG
General Administration	6,160	5,048
Commercial	5,416	4,294
Human Resources	1,261	1,385
Information Technology	916	1,113
Health, Safety and Envitonmental	5,317	4,967
Supply Chain	1,810	1,964
Total	**20,880**	**18,771**

Source: OceanaGold, 2022

Indirect Costs

Indirect costs are estimated to be US$126.9 million for the LoM or $2.71/t processed. These costs have been sourced from the 2020 actual expenditures. Indirect costs include rehabilitation, closure, mitigation and finance lease payments and are summarized in Table 21-13.

Table 21-13: Indirect Cost Summary

Description	Total Cost (US$000's)
Environmental Bond	14,800
Principal/Interest Payment – Capital Leases	108,485
Total	**123,285**

Source: OceanaGold, 2022

22 Economic Analysis

22.1 Principal Assumptions and Input Parameters

The indicative economic results summarized in this section are based upon work performed by SRK and OceanaGold in 2022. The metrics reported in this volume are based on the annual cash flow model results. The metrics are on both a pre-tax and after-tax basis, on a 100% equity basis with no Project financing inputs and are in Q4 2021 U.S. constant dollars. Non-site costs have been excluded from this analysis.

Key criteria used in the analysis are discussed in detail throughout this section. Principal Project assumptions used are summarized in Table 22-1.

Table 22-1: Basic Model Parameters

Description	Value
TEM Time Zero Start Date	January 1, 2022
OP Operations Commercial Production Start	January 1, 2022
UG Capital Spend Start	January 2022
UG Mine Production Start (estimate)	July 2023
OP Mine Life	12 years
UG Mine Life	6 years
Discount Rate	5%

Source: SRK, 2022

All costs incurred prior to January 2022 are considered sunk with respect to this analysis. The selected Project discount rate is 5% as directed by OceanaGold. A sensitivity analysis of the discount rate is discussed later in this section. Foreign exchange impacts were deemed negligible as most, if not all, costs and revenues are denominated in US dollars.

22.2 Cashflow Forecasts and Annual Production Forecasts

TEM inputs/results for the Cashflow Forecasts are summarized on a LoM basis in this section while a full LoM annual cash flow forecast is presented in Appendix B.

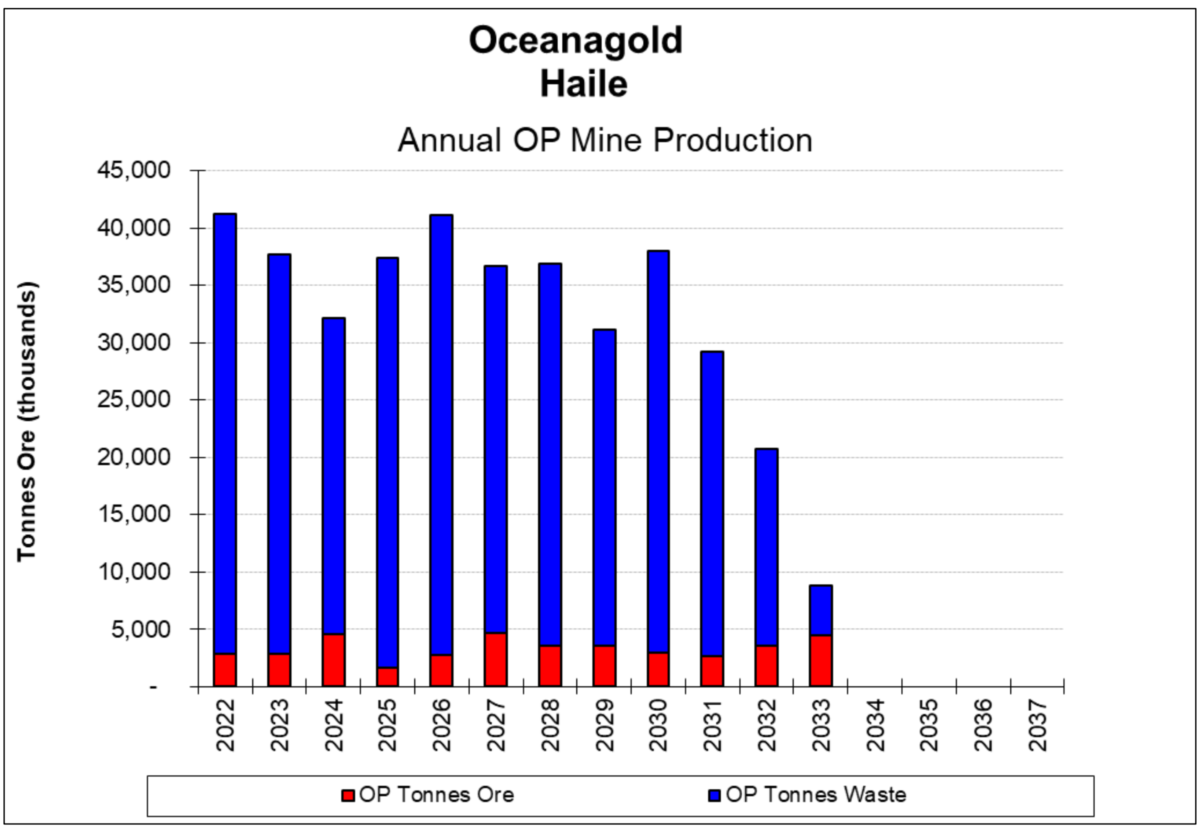
22.2.1 Mine Production

Table 22-2 is a summary of the estimated mine production over a 13-year mine life for the combined open pit/underground operations. Ore mined refers to Proven and Probable Mineral Reserves. Figure 22-1 and Figure 22-2 show LoM production by year for OP and UG operations.

Table 22-2: Life-of-Mine Production Summary

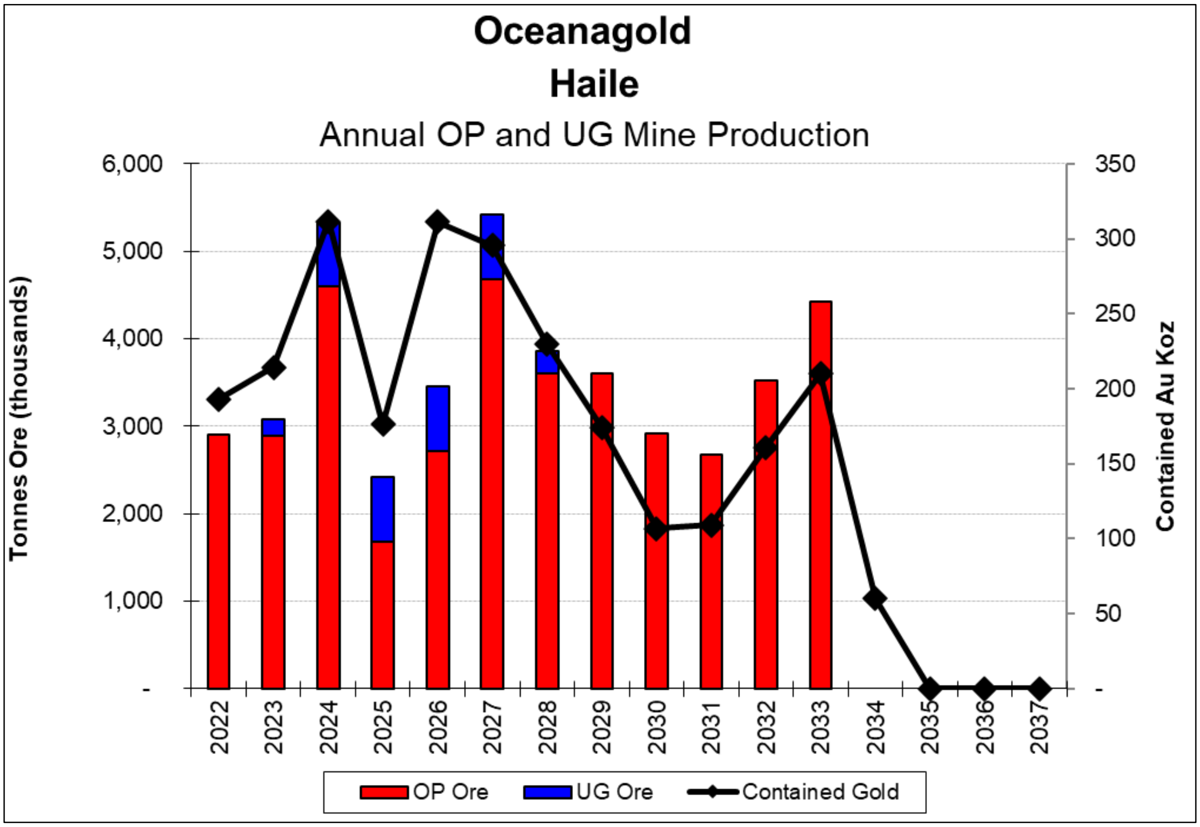
Description	Units	Value
OP Ore Mined	kt	40,170
OP Waste Mined	kt	350,907
OP Total Material Mined	kt	391,076
OP Mined Gold Grade	g/t	1.60
OP Mined Silver Grade	g/t	2.45
OP Contained Gold	koz	2,073
OP Contained Silver	koz	3,163
UG Ore Mined	kt	3,416
UG Waste Mined	kt	821.76
UG Mined Gold Grade	g/t	3.81
UG Mined Silver Grade	g/t	0.00
UG Contained Gold	koz	418
UG Contained Silver	koz	0
Total Ore Mined	**kt**	**43,586**
Waste Mined	kt	351,728
Total Material Mined	**kt**	**395,314**
Mined Gold Grade	g/t	1.75
Mined Silver Grade	g/t	2.21
Total Contained Gold	**koz**	**2,552**
Total Contained Silver	**koz**	**3,228**

Source: SRK, 2022



Source: SRK, 2022

Figure 22-1: Annual Open Pit Mine Production



Source: SRK, 2022

Figure 22-2: Annual Open Pit and Underground Ore Production

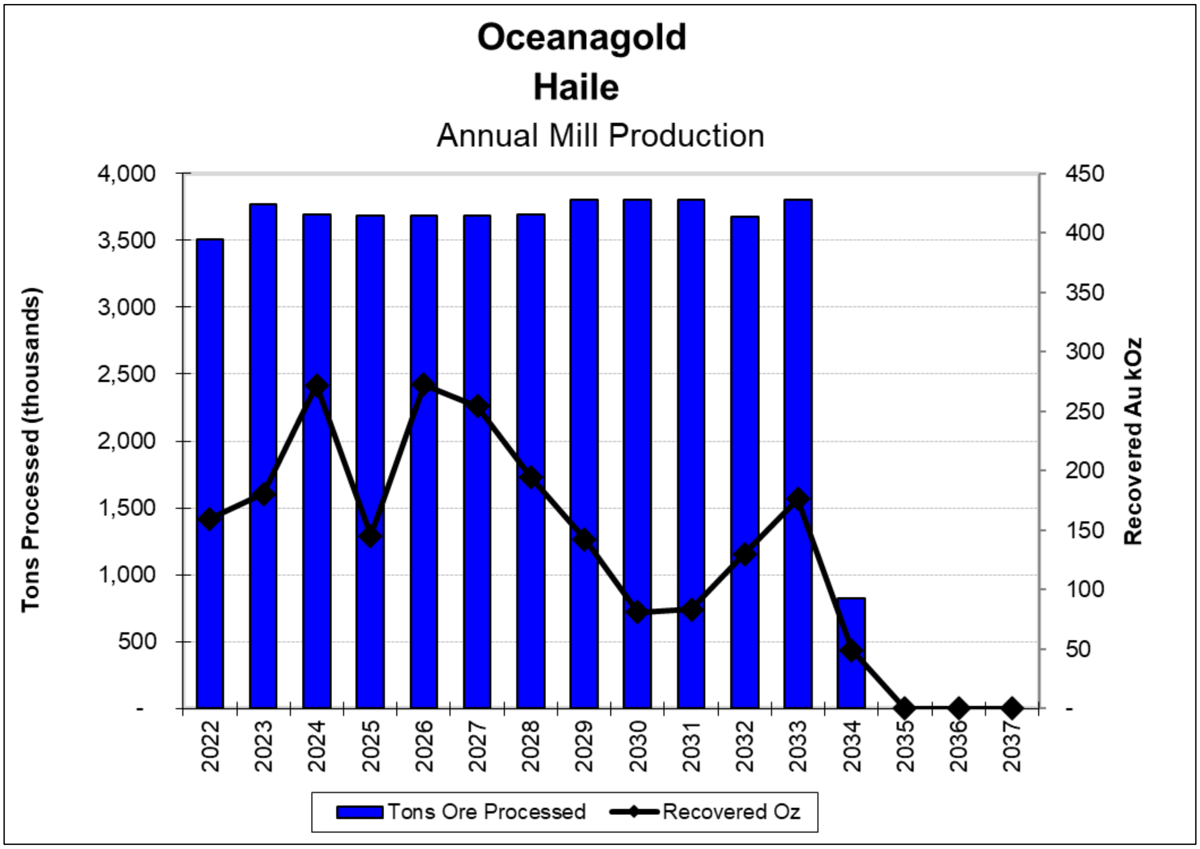
22.2.2 Mill Production

A summary of the estimated process plant production for the Project is contained in Table 22-3 for a 13 year operating life. Figure 22-3 shows LoM production by year. Ore processed refers to Proven and Probable Mineral Reserves.

Table 22-3: Life-of-Mine Process Production Summary

Description	Units	Value
Total Ore Processed	**kt**	**45,414**
Daily Process Capacity	t/d	10,118
Processed Grade	g/t	1.75
Contained Gold	koz	2,552
Contained Silver	koz	3,228
Gold Recovery	%	83.9%
Silver Recovery	%	70.0%
RoM Recovered Gold	koz	2,141
RoM Recovered Silver	koz	2,259

Source: SRK, 2022



Source: SRK, 2022

Figure 22-3: Annual Process Plant Production

22.2.3 Revenue

Gold pricing assumptions used in the economic analysis include a constant LoM gold price of US$1,500/troy oz and a LoM silver price of US$18.00/troy oz.

Doré refining/freight costs are modeled as follows:

- 99.996% payable Au
- US$0.70/troy oz Au treatment charge
- US$0.24/troy oz Au + US$785/shipment freight costs.

Silver is also included in current the Mineral Resource or Reserve estimates.

The silver by-product credit in the TEM is calculated by using a constant silver price of US$18/troy oz and an average recovery of 70%. The additional silver related doré refining costs are as follows:

- 99.0% payable Ag
- US$0.70/troy oz Ag treatment charge
- US$0.24/troy oz Ag freight cost.

The silver by-product credit of US$40 million over LoM represents of 1.4% of revenue over LoM for the Project.

22.2.4 Operating and Capital Costs

No contingency was applied to capital and operating costs within the economic model. The total RoM operating cost unit rate of US$32.94/t processed is summarized in Table 22-4.

Table 22-4: RoM Operating Cost Summary

Description	US$000's	US$/t Mined
OP Mining ($/t rock mined) - All Material	972,725	2.49
OP Mining ($/t rock mined) - (excl. capitalized cost)	545,064	1.39
UG Mining ($/t ore mined)	193,432	56.62
	US$000's	**US$/tOreProcessed**
Subtotal Mining (Operational Material Only)	**738,496**	**16.26**
Processing	529,387	11.66
G&A Cost	222,212	4.89
Refining/Freight Costs	5,732	0.13
Total Operating Costs	**$1,495,827**	**$32.94**

Source: SRK, 2022

There are several important cost items excluded from the operating cost, which are detailed in Table 22-5, that OceanaGold does not consider to be direct operating costs but which the operation does incur.

Table 22-5: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond	14,800	0.33
Interest Expense - Capital Leases	3,600	0.08
Principal Payment - Capital Leases: Sustaining	51,259	1.13
Principal Payment - Capital Leases: Non-Sustaining	57,226	1.26
Total Non-Operating Costs	**107,885**	**$2.79**

Source: SRK, 2022

Total LoM capital costs totaling US$904 million including reclamation and closure costs are summarized in Table 22-6. The capital expenditure items have been separated into sustaining and non-sustaining categories per guidance from OceanaGold. Sustaining capital is primarily related to the development of the underground mine and associated infrastructure and totals US$749 million over life of mine.

Table 22-6: Life-of-Mine Capital Costs (000's)

Description	Sustaining Capital	Non-Sustaining Capital	Total
General Operations Expenditure	28,456	-	28,456
Operations Based Mining Projects	292,829	-	292,829
Capitalised Pre-Strip / UG Mine Development	427,660	-	427,660
Greenfields Exploration	-	1,933	1,933
Stand-alone Mining Projects	-	82,049	82,049
Rehabilitation (Non-Sustaining)1		71,072	71,072
Total LoM Net Capex	**$748,945**	**$155,054**	**$903,999**

Source: SRK, 2022
(1) Captured as Capex in Cashflow

The assumptions used for working capital for this estimate are as follows:

- Accounts Receivable (A/R): 5-day delay
- Accounts Payable (A/P): 30-day delay

- Zero opening balance for A/P and A/R

Annual adjustments to working capital levels are made in the TEM with all working capital recaptured by the end of the mine life resulting in a LoM net free cash flow (FCF) impact of US$0.

22.2.5 Economic Results

The TEM metrics are prepared on an annual after-tax basis, the results of which are summarized in Table 22-7. A full LoM annual cash flow forecast is presented in Appendix B. The results indicate that at a flat US$1,500/oz gold price and a 5% discount rate the Project returns an after-tax NPV of US$529 million.

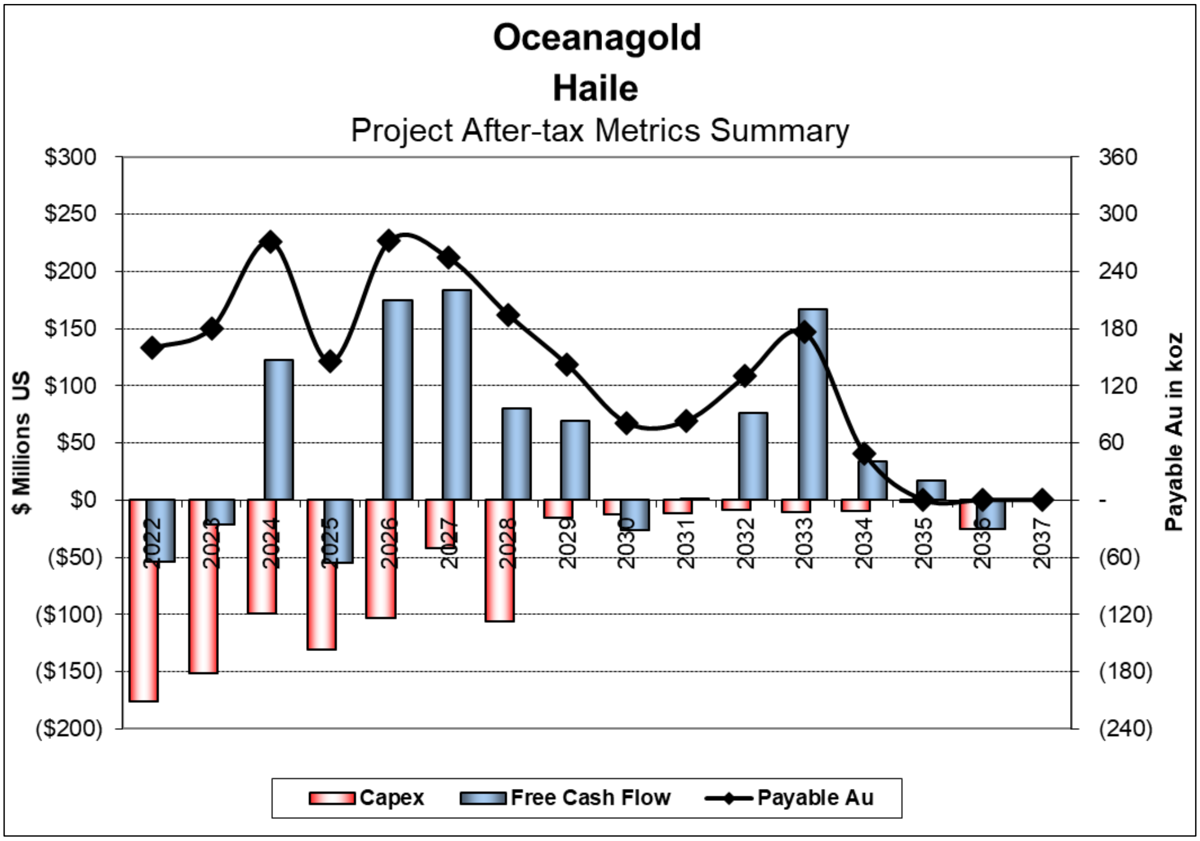
Note that because the project is operating and is valued on a total project basis with prior capital treated as sunk, and not by an incremental analysis of the UG and mill expansions, an IRR value is not relevant in this analysis. Source: SRK, 2022

Figure 22-4 presents annual cash flow metrics versus recovered gold production and shows that the project does not generate positive free cash flow in 2022 and 2023 and 2025 due to the level of capital expenditures being made during that time. The operation also does not generate positive cashflow in 2030 due to higher mining costs and lower processed grades.

Table 22-7: Indicative Economic Results

Description	**US$000's**
Market Prices	
Gold (US$/oz)	$1,500
Payable Gold (koz)	2,141
Revenue	
Gross Gold Revenue	3,211,934
Silver By-Product Credit (@ US$18 / oz Ag)	40,260
Total Gross Revenue	$3,252,195
Operating Costs	
OP Mining	(545,064)
UG Mining	(193,432)
Processing	(529,387)
Site G&A	(222,212)
Selling/Refining	(5,732)
Non-Operating Costs	(107,885)
Total Operating Costs	**($1,603,712)**
Operating Margin (EBITDA)	$1,648,483
Taxes	
Income Tax	-
Total Taxes	**-**
Working Capital	-
Operating Cash Flow	**$1,648,483**
Capital	
Sustaining Capital	(748,945)
Non-Sustaining Capital	(155,054)
Total Capital	**($903,999)**
Metrics	
Pre-Tax Free Cash Flow	$744,484
After-Tax Free Cash Flow	$744,484
Pre-Tax NPV @ 5%	$529,211
After-Tax NPV @ 5%	$529,211

Source: SRK, 2022



Source: SRK, 2022

Figure 22-4: Project After-Tax Metrics Summary

Early in the operational life, negative cashflow years are incurred due to high capital spend. In 2030, the mill processes lower grade material results in negative cashflow. Overall, the operation generates positive LoM cashflow.

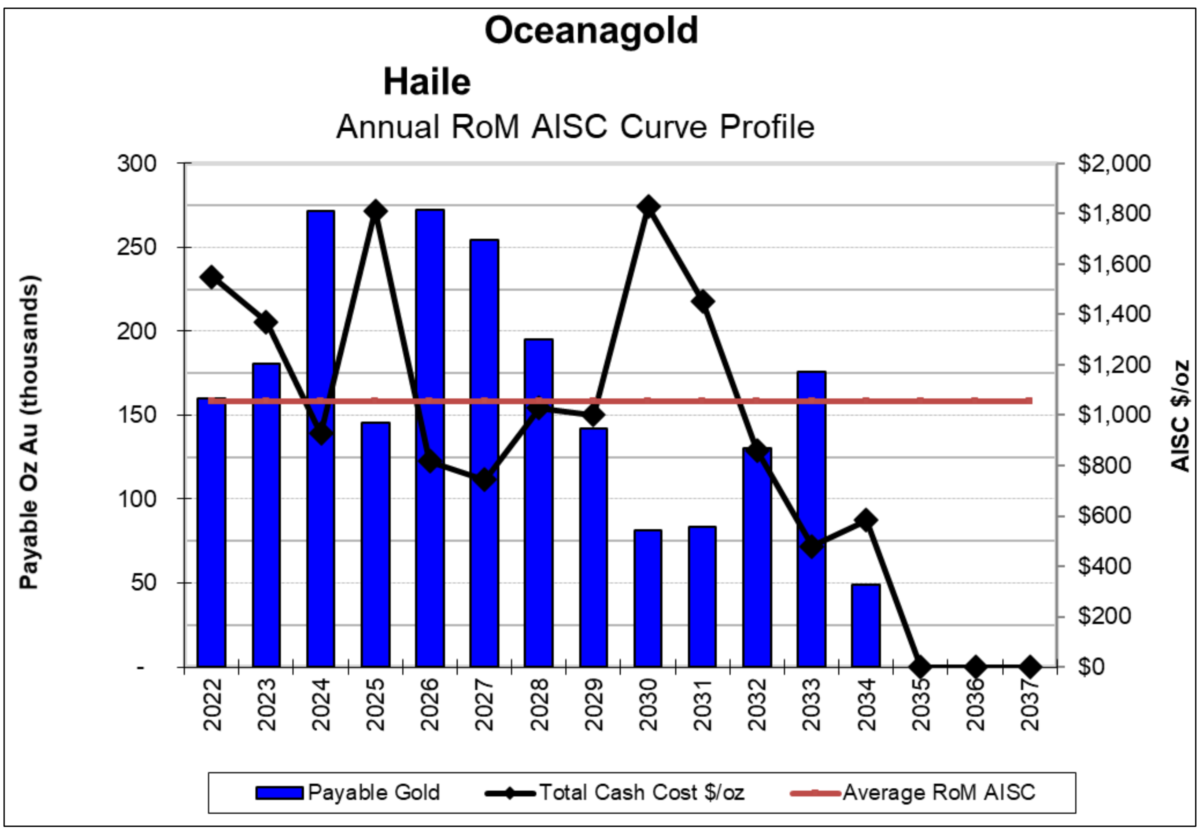
Table 22-8 shows the build-up of a RoM AISC of US$1,055/oz Au, net of a US$19/oz silver by-product credit, over the 14-year life of the Project.

Table 22-8: RoM AISC Contribution

Total RoM Payable Gold Sales in koz		2,470
Description	**US$000's**	**US$/oz**
OP Mining	545,064	255
UG Mining	193,432	90
Processing	529,387	247
Site G&A	222,212	104
Selling/Refining/Freight	5,732	3
Direct Cash Costs Before By-Product Credit	**$1,495,827**	**$699**
Silver By-Product Credit	(40,260)	(19)
Direct Cash Costs After By-Product Credit	$1,455,566	$680
Endowment Liability - State of SC (Removed)	-	-
Environmental Bond (Removed)	-	-
Reclamation Fund Payments (Removed)	-	-
Interest Expense - Capital Leases	3,600	2
Principal Payment - Capital Leases: Sustaining	51,259	24
Non-Operating Cash Costs	**54,859**	**26**
Sustaining Capex	748,945	350
Total RoM AISC	**$2,259,370**	**$1,055**

Source: SRK, 2022

Figure 22-5 shows the annual RoM AISC trend during the mine operations against an overall average RoM AISC of US$1,055/payable oz over the 13-year LoM at an annual average production rate of 195 koz Au per year during full years of operation. The AISC variations are mainly driven by annual gold production levels and can range from US$480 to US$1,812 per oz in a given year.



Source: SRK, 2022

Figure 22-5: Annual AISC Curve Profile

22.3 Taxes, Royalties and Other Interests

As the project is currently in operations, a comprehensive tax analysis for the remainder of the mine life was provided by OceanaGold for incorporation into the TEM on a whole year basis. As the TEM begins in January and contained variable monthly, quarterly and annual periods, some minor adjustment was required for inclusion in the TEM. The project after tax forecast includes benefits associated with net operating losses (NOL) carried forward which are available. The main taxation assumptions utilized in this analysis are as follows:

- Corporate Income Tax (CIT) rates are 21% for Federal and 5% for South Carolina
- Net Operating Losses (NOL) are considered
- Federal Depletion allowance is estimated for each period by applying a depletion rate of US$244/oz .
- A Tax Depreciation allowance schedule was provided by OceanaGold. The tax depreciation schedule provided yields US$1,858 million of depreciation from the whole year 2022 through life of mine.
  - The total tax deductions calculated for the operation from 2022 through the life of mine are presented in Table 22-9.

Table 22-9: Haile Tax Deductions (US$000's)

Tax Deductions	LoM US$(000's)
Depreciation	1,858
Depletion	480
Net Operating Losses (utilized)	220
Total Tax Deductions	**2,558**

Source: OceanaGold

As a result of these deductions, the operation, as modeled is not expected to pay income tax over the life of the operation. As this project is being modeled in isolation at fixed prices and costs, actual results may differ.

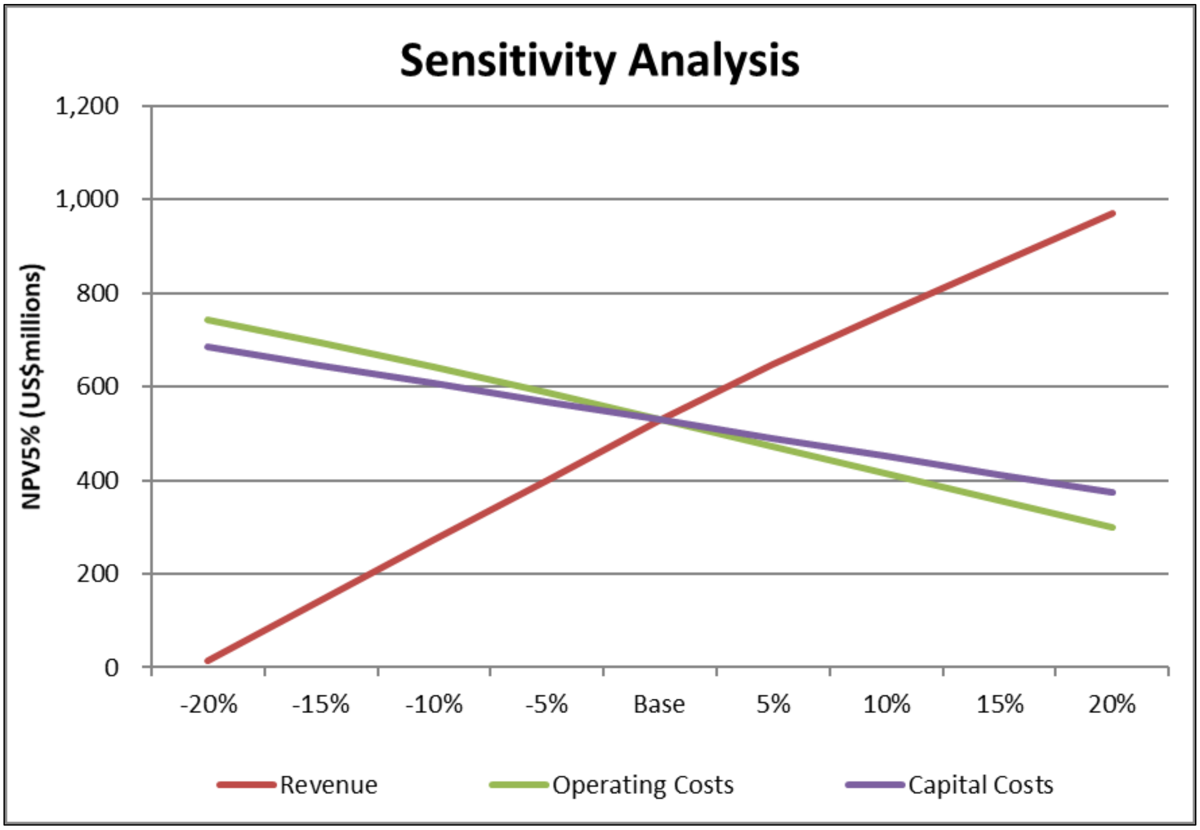
There are no third-party government or private royalties or government severance taxes due on the Project during LoM. The TEM was created on a project level basis and no fractional ownership, if applicable, was considered in the result.

22.4 Sensitivity Analysis

22.4.1 Operational Sensitivity

After-tax sensitivity analyses for key operational parameters are shown in Source: SRK, 2022

Figure 22-6. The Project is nominally most sensitive to revenue. The Project's sensitivities to capital and operating costs are similar but slightly more susceptible to variations in operating costs.



Source: SRK, 2022

Figure 22-6: Operational Sensitivity Analysis

22.4.2 Gold Price Sensitivity

Additional gold price sensitivity analyses are shown with after-tax Project NPV 5% at constant "Robust" prices of US$1,750/oz and constant "Distressed" price of US$1,250/oz , and a price deck provided by Oceana Gold that reflects current consensus pricing (Oceana Gold Price Profile), which show a five year forecast with gold prices reaching US$1,600/oz in 2025 and constant from 2026 onward as presented in Table 22-10.

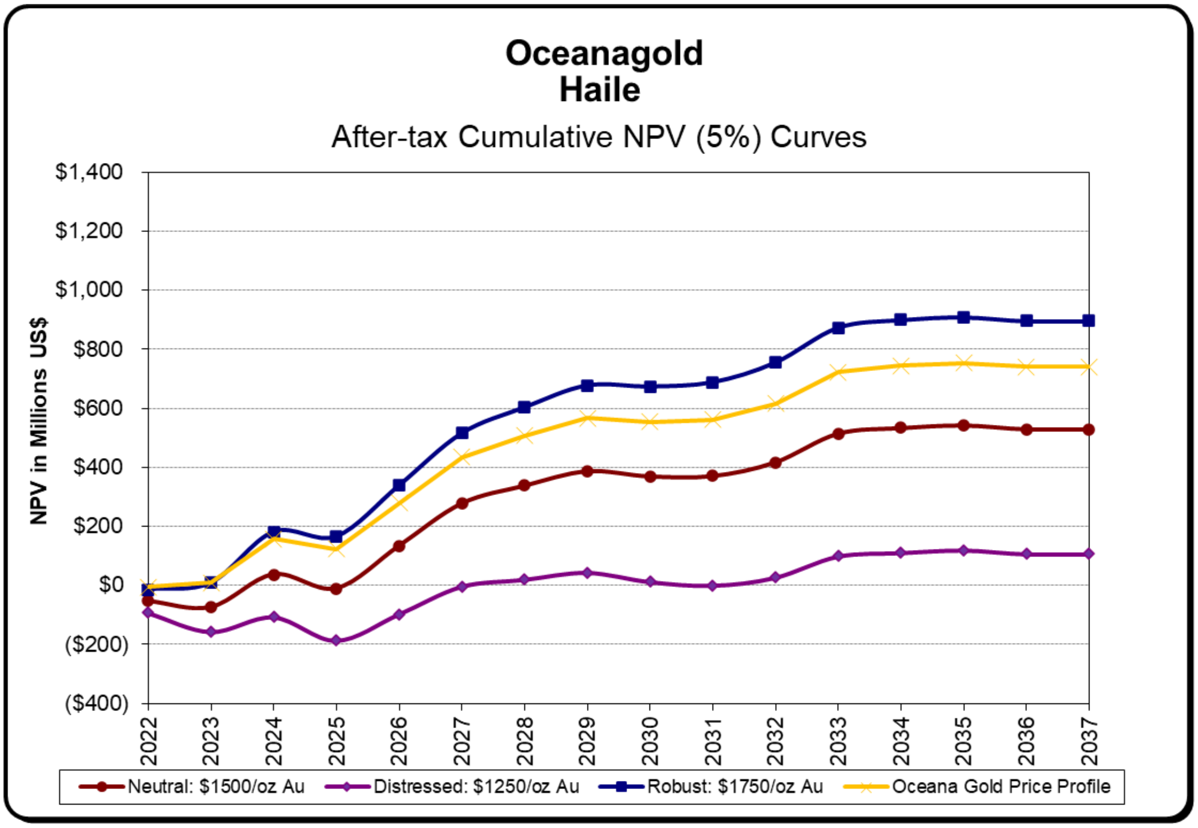
Table 22-10: Oceana Gold Price Profile

Metal	Unit	2022	2023	2024	2025	2026+
Gold	US$/oz	1,800	1,700	1,650	1,600	1,600
Silver	US$/oz	24.00	22.00	21.00	20.00	20.00

Source: Oceana Gold

Source: SRK, 2022

Figure 22-7 shows the gold price sensitivity analysis.



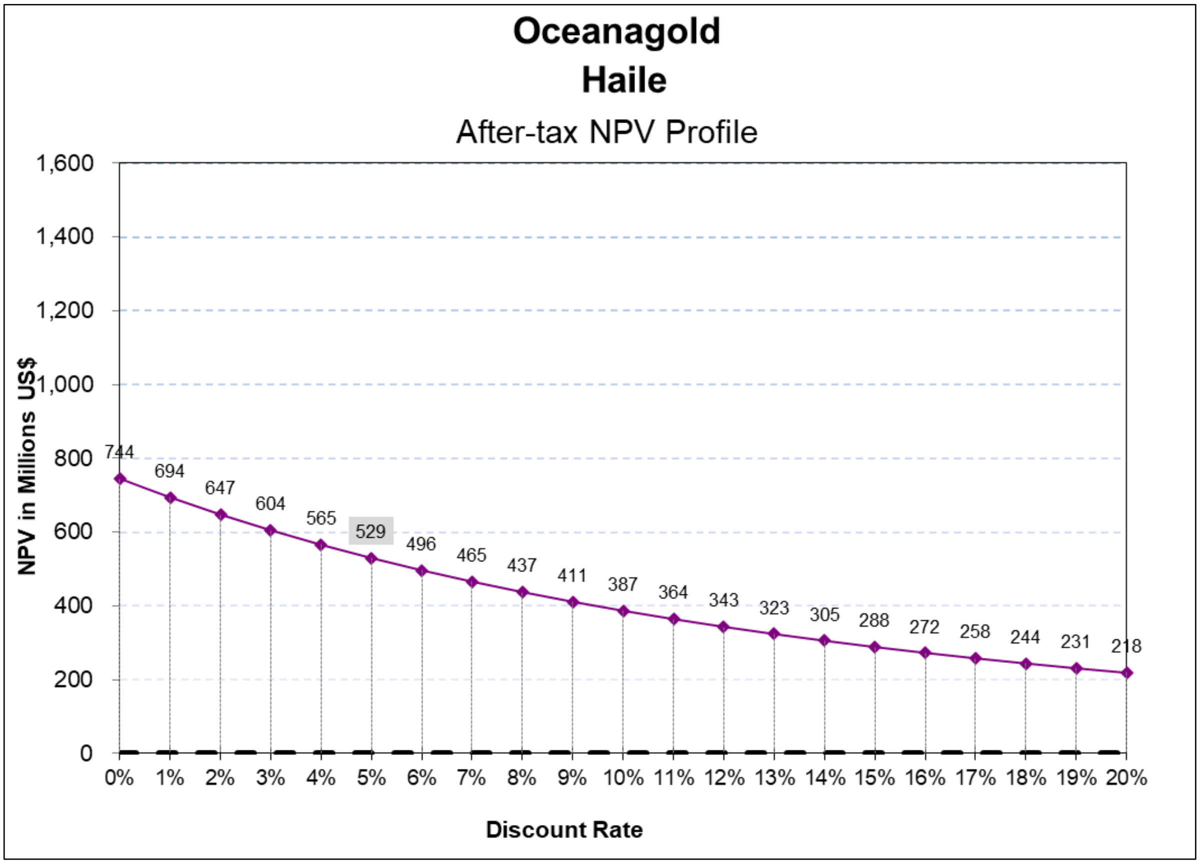
Source: SRK, 2022

Figure 22-7: Gold Price Sensitivity Analysis

22.4.3 Discount Rate Sensitivity

A sensitivity analysis of discount rates presented in Source: SRK, 2022

Figure 22-8 shows that the Project as currently modeled would be NPV positive through a 20% discount rate.



Source: SRK, 2022

Figure 22-8: Discount Rate Sensitivity Analysis

22.5 Oceana Gold Pricing Model Result

As the current gold price environment differs from the pricing used to evaluate the reserves at the Project and in recognition that this may potentially penalize the value of the project, the modeled indicative economic results are presented in Table 22-11 at the Oceana Gold Price Profile price profile. This price profile remains substantially lower than the current gold spot price.

As is evident in the table below, the economics of the project improve at higher metal prices while holding other assumptions constant. At time of publication, the spot metal prices are substantially higher than both the reserve case price and the OceanGold price profile.

Table 22-11: Indicative Economic Results at Ocean Gold Price Profile

Description	US$000's
Market Prices	
Gold (US$/oz)	$1,630
Payable Gold (koz)	2,141
Revenue	
Gross Gold Revenue	3,489,638
Silver By-Product Credit (@ ~ US$21 / oz Ag)	45,912
Total Gross Revenue	$3,535,549
Operating Costs	
OP Mining	(545,064)
UG Mining	(193,432)
Processing	(529,387)
Site G&A	(222,212)
Selling/Refining	(5,732)
Non-Operating Costs	(107,885)
Total Operating Costs	**($1,603,712)**
Operating Margin (EBITDA)	$1,931,838
Taxes	
Income Tax	(30,448)
Total Taxes	($30,448)
Working Capital	-
Operating Cash Flow	$1,901,390
Capital	
Sustaining Capital	(748,945)
Non-Sustaining Capital	(155,054)
Total Capital	**($903,999)**
Metrics	
Pre-Tax Free Cash Flow	$1,027,839
After-Tax Free Cash Flow	$997,391
Pre-Tax NPV @ 5%	$764,318
After-Tax NPV @ 5%	$740,387

Source: SRK

23 Adjacent Properties

The Carolina terrane contains numerous historical gold mines and mining districts. Over 1,500 gold prospects have been documented. Most of these deposits were discovered in the 1800's. Significant gold deposits in South Carolina include the Haile, Ridgeway, Brewer, and Barite Hill Mines. Numerous quartz vein-hosted mines of the Gold Hill and Cid Mining Districts occur in neighboring North Carolina. Some gold deposits have similar geologic and mineralization features to Haile, and several are polymetallic with Cu, Ag, Pb and Zn.

23.1 Ridgeway Mine

The Ridgeway Mine is located 8 km east of Ridgeway, South Carolina and 40 km north of Columbia, South Carolina. Kennecott produced 1.5 Moz (46,655 kg) of gold from 1988 to 1999 from two open pits in low-grade oxide and sulfide ore from siliceous deposits in the Richtex Formation. The Ridgeway deposit has strong geological similarities to Haile (Gillon et al., 1995, 1998). The saprolite, volcanic and metasedimentary rocks are quartz-sericite-pyrite altered in mineralized areas. Post-mineral mafic and diabase dikes crosscut the deposit and are often accompanied by shearing and/or faulting. Gold grade is related to lithology, cleavage development, pyrite grain size and abundance, and silica content. Molybdenite is also associated with the mineralization.

The mine and mill had a production capacity of 13,608 tpd. Ore was milled to minus 200-mesh, then fed into a modified carbon-in-leach circuit. Carbon was stripped of gold, electroplated onto steel wool cathodes, and then transferred to electro-refining cells where gold was plated onto stainless steel plates. Mine closure and reclamation were successfully completed in the early 2000's.

23.2 Brewer Mine

The Brewer gold mine is located 12 km northeast of Haile in Jefferson County. Brewer rocks include schist, volcanics, and granite overlain by 40 to 60 ft of saprolite and sand. Gold mineralization is associated with quartz-sericite-pyrite altered schist, strong silicification and brecciation, and >2% pyrite. Gold ore was produced from a breccia body of hydrothermal origin and a related smaller body of fault-controlled ore. Pyrite content is generally 2% to 5%, unevenly distributed as aggregates and individual crystals in quartz veins. Gold grades were reported in the 1.41 g/t to 4.06 g/t range with associated silver, copper, tin, and bismuth. Brewer is classified as a high sulfidation breccia pipe hosted in the Persimmon Fork volcanics and may have deep porphyry roots.

Like Haile and other mines in the region, the mine produced gold intermittently, first as a placer, then as a surface and underground mine, and finally as a low-grade, heap leach operation in the 1980s. In 1987, Westmont Mining estimated a non-NI 43-101 compliant reserve for Brewer of 4.6 Mt grading 1.4 g/t gold (188,000 oz) (Scheetz, et al. 1991). The reserve does not conform to NI 43-101 standards and is reported for historical purposes only. The most recent production was from 1987 to 1995 by Westmont Mining/Costain Ltd Group. Ore was mined using conventional truck and loader open pit methods and ore was processed using cyanide leaching.

Brewer has been managed by the EPA as an active Superfund site since 1999 due to Acid Rock Drainage (ARD). Westmont mined and heap leached 12 Mt of ore with dilute cyanide solutions from 1987 to 1995. Heavy rains in April 1990 broke the tails dam; over 10 Mt of cyanide solution flowed into Little Fork Creek and downstream into Lynches Creek. The tails dam was repaired in 1991 and mining continued until 1995 when reserves were mined out. Mine reclamation commenced in 1995 with SCDHEC guidance. Barite Hill Mine

The Barite Hill Mine is located 4 km southwest of McCormick, South Carolina. It is within the Lincolnton-McCormick Mining District, which includes other small mines and prospects of gold, silver, copper, zinc, lead, kyanite, and manganese.

23.3 Barite Hill Mine

The Barite Hill Mine is located 4 km southwest of McCormick, South Carolina. It is within the Lincolnton-McCormick Mining District, which includes other small mines and prospects of gold, silver, copper, zinc, lead, kyanite, and manganese.

The Barite Hill deposit was mined from 1989 to 1994 by Nevada Goldfields, Inc. The mine produced 59,000 oz of gold (1.8 million grams) and 109,000 oz (3.4 million grams) of silver, mainly from oxidized ore in the 20-acre (8 ha) Main Pit and the 4-acre (1.6 ha) Rainsford Pit. The mine used conventional open pit mining methods and an on/off pad heap leach process.

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

The Barite Hill deposit is hosted by sericitized felsic metavolcanic and metasedimentary rock of the Persimmon Fork Formation. The deposit occurs along the contact between upper and lower pyroclastic units. Mafic to intermediate post-mineralization dikes and sills cross-cut NE-trending mineralized zones. Multiple Main Pit ore zones are associated with lenses of siliceous barite rock and pyrite-quartz altered breccias, some of which are offset by normal faulting. Rainsford Pit ore zones are associated with silicified rock and chert. The Barite Hill deposit is interpreted to be the result of a Kuroko-type submarine volcanogenic base-metal sulfide system followed by epithermal precious metal deposition (Clark, 1999).

24 Other Relevant Data and Information

SRK knows of no other relevant data or information available at this time, other than what has been presented, to make the technical report understandable and not misleading.

25 Interpretation and Conclusions

The understanding of the geological controls for Haile mineralization continues to evolve and is documented by mapping and drilling in 3,295 drillholes including 1,255 core holes for 394,771 m (57% m) and 2,040 RC holes for 294,216 m (43% m).

Mineralization is relatively continuous, albeit with local grade complexity, and allows reliable long term resource estimation. The interpreted geology provides a good basis for 3D geological modeling and grade estimation (sand, dikes, volcanic-sediment contacts).

En echelon mineralized zones strike ENE as mostly stratiform lenses within a 3.5 km long x 1 km wide area. Stratigraphic controls in the Upper Persimmon Fork Formation comprise mostly unmineralized tuffs and dacite flows adjacent to mineralized metasiltstone that dips 30° to 60° NW. Younger unmineralized units include crosscutting Triassic diabase dikes and Cretaceous sand cover. Dominant structural controls reflected by foliation, faults, and shear zones strike ENE and dip 30° to 60° NW sub-parallel to bedding. Alteration is characterized by strong quartz-sericite-pyrite alteration in mineralized areas flanked by barren sericite and propylitic carbonate-chlorite alteration. Saprolite and oxidation are 10 to 40 m deep. Gold mineralization is poorly developed in saprolite. The Haile mine area is partly covered by an unmineralized southeast-thickening apron of Coastal Plain Sands.

25.1 Geology and Mineralization

Similar timing for gold mineralization and peak magmatism in the Haile and Ridgeway deposits indicates that the hydrothermal systems that produced these deposits were driven by magmatism and were not a product of North American accretion or metamorphism. Timing of gold mineralization is interpreted to coincide with a major tectonostratigraphic change from intermediate volcanism and tuffaceous sedimentation to basinal turbiditic sedimentation. Haile is interpreteded as a low-sulfidation epithermal, sediment-hosted, disseminated gold deposit with proximal quartz-sericite-pyrite (QSP) alteration and distal carbonate-chlorite alteration overprinted by regional greenschist facies metamorphism. Haile is hosted by reduced siliciclastic rocks with less permeable volcanic caprocks and is structurally dismembered.

25.2 Resource Estimation

25.2.1 Open Pit

The drillhole database and resource estimation methodology are appropriate for the purposes of estimating the open pit gold resources. This is supported by reasonable long term resource model to mine-to-mill reconciliation performance. The local grade variability remains a challenge and open pit reconciliation analysis together with sensitivity modeling using simulated resource drilling sets (see sections **Error! Reference source not found.** and **Error! Reference source not found.**) suggest that the annual reconciliation performance previously experienced will remain a feature of the resource estimates.When underground mining commences at Horseshoe, there will potentially be three confluent mill feed sources; open pit, underground and stockpiles (putting aside individual pit stages or underground development areas). Likely continuation of the open pit reconciliation variability observed project to-date implies that multi-source mine to mill reconciliation may be a challenge, certainly for periods less than 3 months. Having said this, long term performance is expected to be acceptable.

OceanaGold will continue to optimize gold estimation but the focus will move to capturing geometallurgical characteristics in the block model. This is likely to include integrating drillhole information such as intensity of silicification, clay content, silver grade, sulfur grade and lithology.

25.2.2 Underground

The drillhole database and resource estimation methodology are appropriate for the purposes of estimating the underground gold resources. OceanaGold has completed an industry standard resource definition drilling at both Horseshoe and Palomino underground deposits to support the current Mineral Resource estimation. The work has been accompanied by an industry standard QA/QC program showing good quality analytical results. OceanaGold has conducted extensive core logging resulting in a high-quality geologic model. The results of the drilling, sampling, analytical testing, core logging and geologic interpretation provide good support for an industry standard resource estimation.

25.3 Status of Exploration, Development and Operations

Systematic target generation and rationalization supported by mapping, drilling, geochemistry, and geophysics is expected to yield new discoveries during the next five years, particularly for underground deposits. Regional exploration is ongoing.

Reserve growth has been enabled by 3D geologic interpretation, higher gold prices, and deeper drilling of a previously underexplored gold system. This has been recently exemplified by pre-development of the Horseshoe underground reserve (0.44 Moz) in 2017 and announcement of an initial Inferred resource at Palomino in 2020 (0.6 Moz). Subsequent infill drilling or the upper parts of the Palomino Inferred Resource has produced an Indicated Resource of 0.2 Moz to-date. Further drilling is in progress.

In-house core drilling continues by OceanaGold at focused on high-grade underground targets proximal to the sedimentary-volcanic contact. Underground development of the Horseshoe deposit in 2022 to 2023 will provide access for underground drill stations along the prospective 2 km long Horseshoe-Palomino trend.

25.4 Mining and Reserves

25.4.1 Open Pit

The Project confirms a positive cash flow using only Measured and Indicated resources for the conversion of reserves using a US$1,500/oz gold price. The mine design supports the style and size of equipment selected for operations.

The mine operating and capital costs have been estimated from first principles and operational knowledge from current mine operations. The equipment is sized to meet minimum SMU requirements that support the dilution and mine recovery factors while providing bulk earthwork capability for the expected production rates.

The LoM production schedule includes provision for careful control of potentially acid generating overburden and appropriate material handling costs have been included in the mining cost estimate. The mine plan is based on a specific set of assumptions and therefore the results of this Technical Report are subject to various risks including, but not limited to:

- Commodity prices and foreign exchange assumptions (particularly relative movement of gold and oil prices)
- Unanticipated inflation of capital or operating costs
- Significant changes in equipment productivities
- Geotechnical assumptions in pit designs
- Ore dilution or loss
- Throughput and recovery rate assumptions

OceanaGold knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the Mineral Reserve estimate.

25.4.2 Underground

Geotechnical

A geotechnical field characterization program has been undertaken to assess the expected rock quality. This program included logging core, laboratory strength testing, in situ stress measurements and oriented core logging of jointing. The results of this program have provided adequate quantity and quality data for feasibility-level design of the underground workings.

A geotechnical assessment of the orebody shape and ground conditions has determined that longhole open stoping mining is an appropriate mining method. Stopes have been sized to maintain stability once mucked empty. A primary/secondary extraction sequence with tight backfilling allows optimization of ore recovery while maintaining ground stability. Primary stopes will be backfilled with cemented rockfill, while secondary stopes will be backfilled with uncemented waste rock.

The design has been laid out using empirical design methods based on similar case histories. The stability of the design has been checked with 3D numerical stress-strain models of the workings which included consideration for mine-scale faulting. The modeling results confirm that stopes and access drifts are predicted to remain stable during active mining.

Mining

Longhole stoping is seen as the appropriate mining method for the deposit geometry. The large stope sizes minimize cost and grades are not overly diluted. Mine planning work considered revenue for Au and an elevated CoG of 1.67 g/t Au was used for design purposes. A 3D detailed mine design was completed.

The underground mine is accessed via ramp, with the ramp portal is located on an open pit bench approximately 80 m below natural surface. Two ventilation drift portals are also located on an open pit bench.

Productivities were developed from first principles. Input from mining contractors, blasting suppliers and equipment vendors was considered for key parameters such as drilling penetration rates, blasthole size and spacing, explosives loading time, bolt and mesh installation time, etc. The rates developed from first principles were adjusted based on benchmarking and the experience and judgment of OceanaGold. Equipment used in this study is standard equipment used worldwide with only standard package/automation features.

The UG production schedule was completed using Deswik scheduling software and is based on mining operations occurring 365 days/year, seven days/week, with two 12-hr shifts each day. A production rate of approximately 2,000 t/d was targeted with ramp-up to full production as quickly as possible. Resource levelling was used on a monthly basis for ore tonnage and lateral development

25.5 Mineral Processing and Metallurgical Testing

The Haile process plant has been in operation for approximately 5 years and has been progressively upgraded from its original nameplate capacity of 2.3 Mtpa to approximately 3.8 Mtpa through utilizing existing inherent capacity and addition of targeted equipment. The plant is now approaching a steady state of operations following upgrading meeting throughput, utilization and recovery expectations.

No novel, experimental or unproven technologies are used for the Haile process plant.

25.6 Recovery Methods

There has been no effective change to the existing plant recovery method for the plant following its expansion compared to the original circuit configuration. Ongoing metallurgical development will continue to target improvements in gold recovery and focusing on controlling unit costs.

25.7 Project Infrastructure

25.7.1 Underground Support Infrastructure

The underground support infrastructure is relatively straightforward including construction of a new dry/meeting building, new underground yard, an upgrade and addition to the power distribution system, water supply system, movement of an existing shop on site to the new underground yard, and installation of a portable crusher/screen plant and CRF plant at the underground yard.

25.8 Environmental Studies and Permitting

There is a significant amount of existing background and environmental baseline data available for the Project. This data continues to be collected and reported to the regulators as part of operational controls. Additional data and environmental studies (Section 20-3) will be of technical interest to the federal and state agencies in evaluating a request to expand the current mining operation.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the expansion process. OceanaGold will work closely with all key stakeholders to ensure that the permitting of the mine expansion meets all of the federal and state requirements.

25.9 Economic Analysis

The Project consists of an operating mine with a mill. The milling facility is mainly fed by the OP mine. The mill feed is also supplemented with ore from a five-year UG 2,150 t/d max annual capacity operation that begins operations in 2023.

The Project is expected to produce 2.14 Moz of payable gold over a 13-year mine life at a rate of 174 koz Au per year during full production years with a LoM AISC of US$1,055/oz Au.

The Project is expected to incur sustaining capital in the amount of US$748.9 million over the modeled life and a non-sustaining capital spend, including rehabilitation costs, of US$155.0 million for total capital expenditure of US$904.0 million.

Project metrics using a constant US$1,500/oz gold price include pre-tax and after-tax NPV 5% values ofUS$529 million. As a result of significant depreciation pools, loss carryforwards and depletion, the operation is not expected to incur income tax liability as modeled. This result would change at higher metal prices. Because the project is operational and is valued on a total project basis and not by an incremental analysis of the UG start up, an IRR value is not relevant in this analysis. In terms of sensitivity, the Project is not surprisingly most sensitive to gold grade and price following by operating costs and capital costs.

26 Recommendations

26.1 Recommended Work Programs

26.1.1 Exploration

OceanaGold will continue to expand resources and reserves in the Haile district through core drilling aligned with LoM plans. Systematic target generation and rationalization supported by mapping, drilling, geochemistry, and geophysics is expected to yield new discoveries during the next five years, particularly for underground deposits.

Haile 3D geologic models continue to be integrated with metallurgical data to facilitate geometallurgical modeling. Evaluate if portable XRF testing or/other technologies can be used to further refine the geology interpretation (in tandem with in-pit studies).

26.1.2 Resource Estimation

When underground mining commences at Horseshoe, there will potentially be three confluent mill feed sources; open pit, underground and stockpiles (putting aside individual pit stages or underground development areas). Likely continuation of the open pit reconciliation variability observed project to-date implies that multi-source mine to mill reconciliation may be a challenge, certainly for periods less than 3 months. Any measures that might minimize the reconciliation uncertainty need to be considered, discussed and implemented to ensure that reconciliation remains a business improvement tool.

Open Pit

OceanaGold continue to optimize the gold estimation methodology via reconciliation analysis and geological review. Also, the Company continues to focus on capturing geometallurgical characteristics in the block model. This is likely to include integrating drillhole information such as silicification intensity, clay content, silver grade, sulfur grade and lithology.

Local grade variability remains a challenge and open pit reconciliation analysis together with sensitivity modeling using simulated resource drilling sets (see sections **Error! Reference source not found.** and **Error! Reference source not found.**) have provided some insight into this. The sensitivity modeling study needs to be leveraged to quantify and communicate likely monthly, quarterly and annual variability as well as to provide practical reconciliation trigger thresholds.

In terms of risk mitigation, OceanaGold should continue to develop interim open pit scheduling estimates using combinations of resource and grade control data.

Underground

Pre-production infill drilling at 15 m x 15 m spacing will be conducted from underground development to provide additional confidence in the tonnes and grade to support a Measured Mineral Resource and refine the mine design. A small number of longitudinal holes will better define cross-cutting barren diabase dike swarms sub-parallel to existing drilling. Future capital development and resource infill drilling will further improve the geological interpretation.

- Study grade control strategy by investigating the potential for underground, reverse circulation, grade control drilling.

- Close spaced, short, ore-drive parallel diamond drilling has been chosen as the preferred option for grade control drilling.
- Additional carbon-sulfur Leco analyses will be conducted to refine the waste model and to improve sulfur grade schedules for the process plant.

26.1.3 Status of Exploration; Development and Operations

OceanaGold continues to expand resources adjacent to open pit and underground reserves in the Haile district through core drilling aligned with LoM plans.

26.1.4 Mining and Reserves

Open Pit

Equipment productivities, drill and blast efficiency and costs are all part of internal continuous improvement processes.

Implementation of the mining plan will include:

- Continuous review of mine designs at an operational level, including further optimization of the material movements and the ultimate ramp systems
- Further geotechnical analysis of pit design criteria as operating experience is gained and additional drilling information becomes available
- Continuing pursuit of productivity and cost reduction initiatives, including:
  - Maintaining in-pit and surface haul roads to all-weather trafficability to reduce time lost to wet weather and increasing tire-life.
  - In-pit water management practices to reduce time lost to wet weather
  - Training of operators with industry best-practice processes
  - Review of blasting techniques and powder factors to improve fragmentation
  - Other initiatives to increase excavator productivities
- Pursuit of initiatives relating to grade control, dilution and ore loss, including:
  - Trial of reverse circulation (RC) grade control to improve orebody definition, particularly in higher-grade areas
  - Modifications where appropriate to primary excavator configurations over the life of the mine to better suit bench height, material and grade control applications

Underground

- Infill drilling at 15 m x 15 m spacing will provide additional confidence in the tonnes and grade of the production schedule to support a Measured Mineral Resource.
- Close spaced Grade Control drilling has been designed and scheduled ahead of stope production.
- A small number of longitudinal holes angled NE or SW will better define the locations of cross-cutting barren diabase dike swarms that are sub-parallel to existing drilling.

The key recommendations relating to the underground project include:

- Categorization of waste rock material as data is available.
- Grade control drilling to delineate ore/waste and guide material destination.
- Investigate UG RC drilling

26.1.5 Mineral Processing and Metallurgical Testing

The process plant flowsheet is effectively fixed and established on site. Ongoing mineralogical and diagnostic leach work on monthly composites should continue to track gold deportment and losses as each pit stage is processed to track variability and understand the impact of regrind size impact for each stage.

Infill drilling presents the opportunity to continue test work on available core samples to confirm recovery estimates for any new reserves that are defined. This should occur as material becomes available to de-risk the use of the current recovery model.

A structured geomet program should continue to focus on understanding expected ore competency to allow improved scheduling and blending to maximise throughput opportunities.

26.1.6 Project Infrastructure

Open Pit Infrastructure

A significant portion of the required open pit infrastructure is in place as part of the existing operation. Infrastructure, such as expit haul roads and water management infrastructure, will be added as necessary. The remaining requirements are waste storage facilities and the costs for these are included as sustaining capital.

Underground Infrastructure

Although the base case design considers cemented waste rock fill material for backfilling stopes, the option for using pastefill is still under consideration. If the pastefill solution is chosen, the necessary underground infrastructure should be re-evaluated.

Tailings and Overburden Infrastructure

The tailings infrastructure will require several road relocations that will need to be coordinated with the State of South Carolina. The Underdrain Collection Pond and run-off collection channels will need to be relocated. Expansion of the West PAG OSA will require the relocation of utility and site infrastructure. Additional geotechnical information may be needed.

26.1.7 Environmental Study Results

Table 26-1 is a summary of the environmental studies initiated by HGM to support the approval process.

Table 26-1: Environmental Studies

Study	Scope of Work
Air Emissions	Assess impact to air pollution loading based on additional operating conditions and new equipment on active point sources – engine exhausts, conveyor drop points, discharge stacks, ventilation shafts, dust controls, blast gasses, cement plant, etc.
Aquatic Resources – plants and animals	Review and assess Carolina Heel Splitter mussel fish and macroinvertebrate studies to quantify impacts to aquatic species over LoM.
Cultural and Historical Resources	Review and assess potentially impacted cultural and historical sites from surface disturbances. Relocate any potential gravesites.
Economic Impact to Local Community	Social and economic effect on the state and local economy – effect on local businesses, wages, local resources, emergency services, and external jobs.
Emergency Response Plans	Assessment of coordination with available local emergency support services.

Study	Scope of Work
Floodplain Assessment	Assess surface water impacts (flows and water quality) based unanticipated concurrent failures of the TSF, storage ponds, and Process Plant containment areas.
Geochemistry Analysis	Update OMP for PAG material placement, ARD control, and underground cut and fill practices.
Geology and Soils Assessment	Update an assessment of suitable materials for future reclamation actions.
Groundwater Modeling	Assess potential impacts to neighborhood wells, ponds and springs.
Hazardous Materials and Waste Inventory	Review of the chemicals, reagents, fuels, and hydrocarbon products transportation, storage, distribution, and disposal.
Health and Safety Assessment	Review of industrial hygiene monitoring and potential impacts to employee health.
Hydrology Assessment	Identification of direct and indirect impacts on local wetlands, wells, and streams.
Impacted Wetland Assessment	Assess potentially impacted wetland areas – vegetation and stream flow.
Land Use	Assess alternatives to potentially impacted land masses and identify future potentially higher value applications.
Noise and Vibration Study	Identify potential impacts from noise and vibration sources, including blasting activities, mobile equipment operating at elevated surfaces, crushing and grinding equipment, TSF evaporators, and mine equipment (haul trucks, dozers, and excavators)
Reclamation Plan	Pit closure and remediation plans, surface controls, revegetation plans, stormwater control plans, surface run-off plans, timelines, and sequence for surface reclamation activities.
Socioeconomic Impacts	Assess the socioeconomic impacts to state and local communities.
Stormwater Plans	Create Stormwater plans for sediment ponds, borrow pits, location for BMP[1] devices, assessment locations, and site controls.
Surface Water Impacts	Assess impact to surface water flows – volume, dissolved oxygen, chemistry, pH and conductivity. Assess drainage patterns and develop recommendations for additional monitoring and measurement stations, if required.
Terrestrial Resources – Plant Life	Perform terrestrial plant evaluations, specifically in impacted areas and areas of significant disturbance.
Terrestrial Resources – Wildlife	Perform seasonal terrestrial studies on migratory endangered wildlife, such as species of bats and raptors.
Transportation and Traffic / Road Impacts	Perform a traffic study and predict road patterns and potential impacts based on employment and support service usage.
Vibration Analysis	Develop vibration predictions based on underground blasting and changes in surface contours and geological morphology.
Visual Impact Studies	Assess visual impacts along major thoroughfares: Highways, County Roads, Neighborhoods and Public spaces
Water Quality Assessment	Assess current and future impacts to changes in water quality based on volume changes, available precipitation and evaporation, and geochemical leach studies.
Wetland Delineation and Marking	Set the visual markers around the 50-ft offset. Measure the final size of wetland delineation.
Wildlife Assessment	Biological assessment of primary and secondary wildlife species and identify impacts and changes to migratory patterns, mortality changes and secondary food sources.

Source: OceanaGold, 2022
[1] Best Management Practice Devices – 1) Erosion Prevention – slope surfaces, seeding, and erosion controls; and 2) Sediment Control - check dams, sediment dams, sediment ponds, and silt fencing.

26.1.8 Economic Analysis

The current metal price environment is extremely strong. If prices are forecast to remain elevated for long periods, the project reserves and resources should be updated and fed into an economic model at a revised price deck reflective of the long-term price forecasts.

26.2 Recommended Work Programs Costs

Table 26-2 lists the estimated costs for the recommended work described in section 26. Note that these costs are included in the cost schedules presented in Section 21.

Table 26-2: Summary of Costs for Recommended Work

Discipline	Program Description	Cost (US$)	No Further Work is Recommended Reason:
Geology and Mineralization	External reviews	30,000	
Status of Exploration, Development and Operations	OceanaGold exploration programs and development drilling. External assay laboratories used.	1,500,000	
Mineral Processing and Metallurgical Testing	External laboratory testing – future ore sources	800,000	
pXRF analyses to augment geological interpretation	Collect more data and analyze		Salaries only
Mineral Resource Estimate – Open Pit	Develop Geometallurgical model		Salaries only
Open Pit Mining	Geotechnical drilling and analysis	2,275,000	
Open Pit Mining	RC grade control trial	3,150,000	
Open Pit Mining	PAG Waste definition and classification	100,000	
Project Infrastructure	Detailed design of bridge, road relocations tailings and waste storage facilities expansions	200,000	
Environmental Studies and Permitting	Additional studies and legal support	2,100,000	
Total US$		**$10,155,000**	

Source: OceanaGold 2022

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28 Glossary

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

28.1 Mineral Resources

A **Mineral Resource** is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.

An **Inferred Mineral Resource** is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

An **Indicated Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

A **Measured Mineral Resource** is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.

28.2 Mineral Reserves

A **Mineral Reserve** is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.

The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported. The public disclosure of a Mineral Reserve must be demonstrated by a Pre-Feasibility Study or Feasibility Study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated Mineral Resource, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve.

A **Proven Mineral Reserve** is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors.

28.3 Definition of Terms

The following general mining terms may be used in this report.

Table 28-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with ore.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of an orebody or stope.
Gangue	Non-valuable components of the ore.
Grade	The measure of concentration of gold within mineralized rock.
Hangingwall	The overlying side of an orebody or slope.
Haulage	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserve.

Term	Definition
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

28.4 Abbreviations

The following abbreviations may be used in this report.

Table 28-2: Abbreviations

Abbreviation	Unit or Term
%	percent
~	approximately
°	degree
°C	temperature in degrees Celsius
°F	temperature in degrees Fahrenheit
2D	two-dimensional
3D	three-dimensional
AA	atomic absorption
AHK	AHK Geochem
AISC	All-In Sustaining Cost
ALS	ALS Limited
AP	acid generating potential
Ar	argon
ARD	acid rock drainage
ASTM	American Society for Testing and Materials
Au	gold
Bhp	brake horsepower
BLM	United States Department of the Interior Bureau of Land Management
BM	Bond ball mill
Breccia	brecciated rocks
$CaCO_3$	Calcium carbonate
Cat	Caterpillar
Cfm	cubic feet per minute
CIL	Carbon-In-Leach
CIT	Corporate Income Tax
cm^3	cubic centimeter
CoG	cut-off grade
CPLEX	IBM ILOG CPLEX Optimizer

Abbreviation	Unit or Term
CPS	coastal plain sands
CRF	cemented rock fill
CRM	Certified Reference Materials
CV	coefficient of variation
Cyprus	Cyprus Exploration Company
DB	intrusive dikes
DDH	diamond drilling
DHEC	Department of Health and Environmental Control
DSS	direct shear strength
EIS	Environmental Impact Statement
ELOS	equivalent linear overbreak/slough
EPCM	Engineering, Procurement and Construction Management
Excel	Microsoft
FA	fire assay
FDD	Fresh dikes
FF	Fracture frequency
FF/m	frequency fracture per meter
Fill	back-fill from historical mining
FMS	Fresh metasediment
FMV	Fresh metavolcanic
FoS	factor of safety
FS	feasibility study
Ft	foot (feet)
G	gram
g/t	grams per tonne
g/t	grams per tonne
geology data	GL
GPa	gigapascal
Gpm	gallons per minute
Ha	hectares
Haile	Haile Gold Mine
HDPE	height density polyethylene
HGM	Haile Gold Mine, Inc.
HMC	Haile Mining Company
HMV	
Hp	horsepower
Hr	hour
ICP-MS	inductively coupled plasma mass spectrometry
IDW^2	Inverse Distance Weighting Squared
IMC	Independent Mining Consultants
In	inch
IP	Induced polarization
IRR	initial rate of return
IRS	intact rock strength
ISO/IEC	International Electrotechnical Commission
ISO-9001	International Organization for Standardization
ISRM	International Society of Rock Mechanics
Ja	joint alteration
Jn	joint number
JPAG	Johnny’s PAG
Jr	joint roughness
Kg	kilogram
Km	kilometer
KML	Kershaw Mineral Lab
KML	Kershaw Mineral Lab
kN	kilonewton
kN/m^3	kilonewton per cubic meter
Koz	thousand troy ounce
KS	Kolmogorov-Smirnoff

Abbreviation	Unit or Term
Kt	thousand tonnes
kV	kilovolt
kW	kilowatt
L	liter
Lb	pound
Leapfrog	ARANZ Leapfrog® Geo software
LECO	LECO Corporation
LHD	long-haul-dump
LoM	life-of-mine
M	meter
m/d	meter per day
m^3	cubic meter
Ma	mega-annum
Ml	laminated metasediments
MIBC	methyl isobutyl carbinol
Min	minute
ML	metal leaching
Mm	millimeter
MOA	Memorandum of Agreement
MPa	megapascal
MS	metasediments
Ms	silicified metasediments
Mst	million short tons
Mtpa	million tonnes per annum
MV	metavolcanics
MW	million watts
N’	stability number
N	north
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NN	nearest neighbor
NNP	net neutralization potential
NOL	Net Operating Losses
NP	acid neutralization potential
NPDES	National Pollutant Discharge Elimination System
NPR	neutralization potential ratio
NPV	net present value
OC	oriented core data
OceanaGold	OceanaGold Corporation
OK	Ordinary Kriging
OMP	Overburden Management Plan
OP	open pit
OSA	overburden storage area
Oz	ounce
PAG	potential acid generating
PAX	potassium amyl xanthate
PEA	preliminary economic assessment
Piedmont	Piedmont Mining Company
PLT	point load test
PMP	Probable Maximum Precipitation
Project	Haile Gold Mine
POF	probability of failure
Ppb	parts per billion
Ppm	parts per million
QA	Quality Assurance
QC	Quality Control
QP	Qualified Persons
QSP	quartz-sericite-pyrite
RC	reverse circulation

Abbreviation	Unit or Term
rd	rounds
RDi	Resource Development Inc.
RM	rock mass
RMI	Romarco Minerals, Inc.
RMR	Rock Mass Rating
RoM	run-of-mine
RQD	rock quality designation
S	Sulfur
SAG	Semi-Autogenous Grinding
SAP	sampling and analysis plan
SAP	Saprolite
SCDHEC	South Carolina Department of Health and Environmental Control
Sec	second
SMC	Sag Mill Comminution
SMU	selective mining unit
SO_2	Sulfur dioxide
SRF	stress reduction factor
SRK	SRK Consulting (U.S.), Inc.
St	short ton (2,000 pounds)
S_T	total sulfur
st/d	short tons per day
STD	standard deviation
t/d	metric tonnes per day
TCC	total cash costs
TCR	total core recovery
TCS	triaxial compressive strength
TIC	total inorganic carbon
TSF	tailings storage facility
TV	televiewer data
UCS	uniaxial compressive strength
UG	underground
U-Pb	Uranium–lead
US$	United States Dollar
USA	United States of America
USFS	United States Forest Service
V	volts
VFD	variable frequency drive
W	watt
W:O	waste: ore
WAD	weak acid dissociable
WDD	weathered dikes
Wi	work indices
WMS	Weathered metasediment
WMV	weathered metavolcanic
Y	year
Mm	microns

Appendices

Appendix A: Certificates of Qualified Persons

CERTIFICATE OF QUALIFIED PERSON

I, David Read Carr, MAusIMM CP (Met) do hereby certify that:

1. I am Chief Metallurgist of OceanaGold Corporation, 99 Melbourne Street, South Brisbane, Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2022 (the "Technical Report").
3. I graduated with a degree in Bachelor of Engineering in Metallurgical Engineering (Hons) from the University of South Australia in 1993. I am a Member and Chartered Professional of the Australasian Institute of Mining and Metallurgy. I have worked as a metallurgist for a total of 29 years since my graduation from university. My relevant experience includes base metal flotation, flotation and leaching of gold ores, pressure oxidation of refractory sulphide ores, ultrafine grinding, process plant design, project evaluation and plant commissioning
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile property in 2017, 2018, 2019 and on March 8, 2020 for 17 days.
6. I am responsible for mineral processing, all of Sections 13 and 17, Section 18.10, the process plant capital and operating costs of section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101 because I am an employee of OceanaGold Management Pty Limited.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is the preparation of the "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated August 9, 2017, "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" dated September 30, 2020 and in my role as Chief Metallurgist for OceanaGold with involvement in the site commissioning and development since acquisition in 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

David Read Carr, MAusIMM CP (Met)

CERTIFICATE OF QUALIFIED PERSON

I, Michael Kirby, PE, MEng Civil do hereby certify that:

1. I am Environmental Superintendent of OceanaGold Corporation, 6911 Snowy Owl Road, Kershaw, SC.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a Masters degree in Civil Engineering from the University of Louisville in 1998. In addition, I have obtained a Professional Engineering license in Kentucky, Alabama, Georgia, North Carolina and South Carolina. I am a member of the Society for Mining, Metallurgy & Exploration. I have worked as an Engineer for a total of 27 years since my graduation from university. My relevant experience includes 20 years as an engineering consultant and 7 years of mining.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am based at the Haile property.
6. I am responsible for Environmental and Permitting, Section 20 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101. I am an employee of OceanaGold
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is that I have been working at this site for 7 years as an employee of OceanaGold.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March 2022.

SOUTH CAROLINA LICENSED PROFESSIONAL ENGINEER No. 23673 MICHAEL W. KIRBY

30 March 2022

____

Michael Kirby, PE, MEng Civil

OceanaGold
Haile Gold Mine

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OCEANAGOLD

CERTIFICATE OF QUALIFIED PERSON

I, David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, SME-RM do hereby certify that:

1. I am the Executive General Manager of OceanaGold, 6911 Snowy Owl Road, Kershaw, South Carolina, 29067.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Universidad Nacional de Colombia in 1984. In addition, I have obtained a master's degree in Earth and Systems Engineering from Colorado School of Mines. I am a registered member of the Society of Mine Engineers (SME). I have worked as a Mine Engineer for a total of 37 years since my graduation from university. My relevant experience includes:
    a. Production supervisor and mine planner for Cerrejon Coal Mine in Colombia
    b. Mine Planning consultant for Exploration Computer Services in Denver, Colorado,
    c. Corporate Support for Operations and Mine Planning for Ymeris (USA, Mexico, France, Spain, Peru, China, Chile).
    d. Technical Services Manager for San Cristobal Mine (Lead, Silver and Zinc) in Bolivia and for South Gobi Sands coal mine in Mongolia
    e. Mine Manager for Calenturitas Coal Mine (Glencore-Colombia)
    f. General Manager for Lumwana Copper Mine (Zambia) Detour Gold Mine (Canada), Haile Gold Mine (USA)
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have been employed by the Haile Gold Mine since June 15, 2021, and still employed as Executive General Manager.
6. I am responsible for Sections 19, the open pit, underground, and other costs of section 21 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101. as I work directly for Haile Gold Mine.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is and continue to be as the Executive General Manager.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March 2022.

SME Registered Engineer # 04038617

David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, SME-RM

OCEANAGOLD

CERTIFICATE OF QUALIFIED PERSON

I, Gregory Hollett, BEng Mining Engineering, P.Eng (EGBC) do hereby certify that:

1. I am Group Mining Engineer of OceanaGold Corporation, Level 3, 99 Melbourne St, South Brisbane, Queensland, 4101, Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Curtin University in 2000. In addition, I have obtained a Master of Business Administration from Torrens University Australia in 2017. I am a Professional Engineer (P.Eng) registered with Engineers and Geoscientists of British Columbia (EGBC, #157783). I have worked as a mining engineer for a total of 21 years since my graduation from university. My relevant experience includes open-pit operational management, mine design and implementation, short- and long-term scheduling, haulage analysis, equipment selection, and cost estimation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I last visited the Haile property on January 3, 2022 for 10 days.
6. I am responsible for open pit portions of Section 15 and 16.3, Sections 16.1, 16.1.1, 16.1.3, 16.1.4, 16.1.6, 16.1.7, 16.1.8, 18.6, the open pit operating cost portions of section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101. I am a full-time employee of OceanaGold Corporation.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is in technical studies and technical reviews.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022

PROFESSIONAL PROVINCE OF G. R. HOLLETT # 157783 BRITISH COLUMBIA ENGINEER

Gregory Hollett, BEng Mining Engineering, P.Eng (EGBC)

OCEANAGOLD

CERTIFICATE OF QUALIFIED PERSON

I, Jonathan Moore, BSc Geology (Hons), MAusIMM(CP) do hereby certify that:

1. I am Chief Geologist of OceanaGold Corporation, Level 3, 99 Melbourne St, South Brisbane Qld 4101, Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in BSc (Hons) Geology from University of Otago in 1985. In addition, I have obtained a Graduate Diploma in Physics from the University of Otago in 1993. I am a member and Chartered Professional of the AusIMM. I have worked as a Geologist for a total of 31 years since my graduation from university; in open pit and underground resource estimation throughout Australia, New Zealand, Philippines, South America and the USA. Commodities include gold, silver, tungsten, lead and zinc.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I last visited the Haile property on January 13, 2020 for 16 days.
6. I am responsible open pit and underground Mineral Resources, Sections 4 through 12, 14, 23 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. Being an OceanaGold employee, I am not independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been ongoing assistance as Chief Geologist with the project since OceanaGold acquired the Haile Gold Mine in 2015.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

Jonathan Moore, BSc Geology (Hons), MAusIMM(CP)

OCEANAGOLD
Haile Operation

CERTIFICATE OF QUALIFIED PERSON

I, Brianna Drury, BEng Mining, SME-RM do hereby certify that:

1. I am the Underground Project Manager for OceanaGold, Haile Gold Mine 6911 Snowy Owl Road, Kershaw, SC 29067.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Missouri S&T in 2010. I am a Registered Member of the Society of Mining, Metallurgy & Exploration. I have worked as a Mining Engineer for a total of 12 years since my graduation from university. My relevant experience includes 10.5 years of underground & surface hard rock gold mining in Nevada's Carlin trend and the last 1.5 years with OceanaGold as the underground project manager.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am currently located on site.
6. I am responsible for underground operating costs Section 21.2.1, and portion of Sections 1, 25, and 26 summarized therefrom of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March 2022.

Brianna Drury

Brianna Drury, BEng Mining, SME-RM

SME
Society for Mining, Metallurgy & Exploration
Brianna Drury
SME Registered Member No. 4151277
Signature Brianna Drury
Date Signed 3-30-2022
Expiration date ——
Area of Competency UG Mining

CALL & NICHOLAS

CERTIFICATE OF QUALIFIED PERSON

I, Larry Standridge, PE, MSE, do hereby certify that:

1. I am a Principal Engineer of Call & Nicholas, Inc., 2475 North Coyote Drive, Tucson, AZ, USA, 85750.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2022 (the "Technical Report").
3. I graduated with a MS in Mining, Geological, and Geotechnical Engineering from The University of Arizona in 2001. In addition, I have obtained a BA in Geology from The University of North Carolina in 1996. I am a registered Professional Engineer (Geological) in the State of Arizona (No. 64435). I have worked as an engineer for a total of 20 years since my graduation from university. My relevant experience includes 16 years of pit slope and waste dump design, prefeasibility and feasibility studies, field mapping, 3D modeling, stability analyses, debris flow analyses, and numerous other geotechnical activities in support of the mining industry.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on November 29th, 2017 for 2 days, on October 10th, 2018 for 2 days, January 14th, 2020 for 3 days, and May 18, 2021 for 2 days.
6. I am responsible for open pit geotechnical work, co-authoring Section 16.1.2 and portions of Sections 1, 25 and 26 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is providing slope design recommendations for the open pits and waste dumps, develop mitigation options for several geotechnical hazards, and assisting with geotechnical data collection.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

Larry Standridge

Larry Standridge, PE

REGISTERED PROFESSIONAL ENGINEER (GEOLOGICAL)
CERTIFICATE NO.
64435
LARRY
STANDRIDGE
DATE SIGNED 3-30-22
ARIZONA, U.S.A.
Larry Standridge
Expiration Date 9-30-23

CERTIFICATE OF QUALIFIED PERSON

I, Jay Newton Janney-Moore, PE do hereby certify that:

1. I am employed as an engineer at NewFields Mining & Technical Services LLC NewFields, 9400 Station Street, Suite 300, Lone Tree, Colorado 80124, USA.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report Feasibility Study Haile Gold Mine Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Bachelor of Science in Civil Engineering from University of Colorado Denver in 1998. I am a registered professional engineer in the State of South Carolina (No. 28306) and in the State of Colorado (No. 37571). I have worked as an engineer a total of 24 years since my graduation from university. My relevant includes designing heap leach pads, tailings storage facilities, surface water diversions, and other supporting infrastructure.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I last visited the Haile Gold Mine property 31st of August 2021 for 3 days.
6. I am responsible for tailing and overburden storage, Sections 18.1, 18.2, 18.3, 18.4, 18.5, the Tailings/Overburden section of and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is the engineer of record for the Potentially Acid Overburden Storage Areas and Contact Water Ponds, Fresh Water Storage Dam, and Duckwood Tailings Storage Facility. I have also been a QP on reports titled "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated August 9, 2017 and June 30, 2020.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

____*"Signed"*____________________________ *"Stamped"*

Jay Newton Janney-Moore

NewFields
Perspective. Vision. Solutions.

CERTIFICATE OF QUALIFIED PERSON

I, William Lucas Kingston (MSc, PG). do hereby certify that:

1. I am an Associate Hydrogeologist of NewFields Mining & Technical Services LLC ("NewFields"), 9400 Station Street, Suite 300, Lone Tree, Colorado 80124, USA .
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Geology from Tulane University in 1984. In addition, I have obtained a Master of Science degree in Hydrogeology and Hydrology from the University of Nevada – Reno in 1989. I am a professional geologist in the State of South Carolina (No. 2666), in the State of Wyoming (No. 3645), and in the State of California (No. 8679). I have worked as a Hydrogeologist for a total of 32 years since my graduation from university. My relevant experience includes water supply, pit dewatering, and mine water management studies that often include elements of data collection, development of conceptual groundwater flow models, and numerical predictive models.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile Gold Mine property on December 4, 2019 for two days and August 31, 2021 for 3 days.
6. I am responsible for hydrogeology Sections 16.1.9, 16.2.3, the hydrogeological portion of section of 16.1.2, 18.7 and portions of Sections 1, 25 and 26 summarized therefrom of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is managing prior pit dewatering design and preparation of quarterly groundwater and surface water monitoring reports. I have also been a QP on a report titled "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated June 30, 2020.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022

WILLIAM L. KINGSTON
REGISTERED
No. 2666
SOUTH CAROLINA
PROFESSIONAL GEOLOGIST

3/30/2022

William Lucas Kingston, MSc, PG.

SRK Consulting (U.S.), Inc.
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1125 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Joanna Poeck, BEng Mining, SME-RM, MMSAQP, do hereby certify that:

1. I am a Principal Mining Engineer of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Colorado School of Mines in 2003. I am a Registered Member of the Society of Mining, Metallurgy & Exploration Geology. I am a QP member of the Mining & Metallurgical Society of America. I have worked as a Mining Engineer for a total of 18 years since my graduation from university. My relevant experience includes open pit and underground design, mine scheduling, pit optimization and truck productivity analysis.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile property on February 6, 2017 for 2 days and March 12, 2019 for 3 days.
6. I am responsible for underground Mineral Reserves, all of Sections 2, 3, 24, 27, 28, the underground portions of Section 15 and 16.3, Section 16 opening statements, Sections 16.2, 16.2.1, 16.2.5, 16.2.6, 16.2.7, 16.2.8, 16.2.9, 16.2.11, 18.6, 18.7, 18.8, 18.9 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is serving as QP on reports titled "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated August 9, 2017 and June 30, 2020.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

"Signed" *"Sealed"*

Joanna Poeck, BEng Mining, SME-RM[4131289RM], MMSAQP[01387QP]
Principal Consultant (Mining Engineer)

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www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Matthew Sullivan, B.Sc., RM-SME do hereby certify that:

1. I am a Senior Consultant of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2022 (the "Technical Report").
3. I graduated with a degree in Metallurgical and Materials Engineering from Colorado School of Mines in 2009. In addition, I also obtained a degree in Economics and Business from the Colorado School of Mines in 2009. I am an SME Registered Member. I have worked evaluating mining projects for a total of 9 years since my graduation from university. My relevant experience includes economic evaluation of mining projects supporting mining sector investment and economic analysis supporting mining project studies.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile property.
6. I am responsible for technical-economics Sections 22, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

"Signed" *"Sealed"*

Matthew Sullivan, BEng, RM-SME

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CERTIFICATE OF QUALIFIED PERSON

I, David Bird, MSc., PG, RM-SME, do hereby certify that:

1. I am an Associate Principal Consultant (Geochemistry) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with Bachelor's Degrees in Geology and Business Administration Management from Oregon State University in 1983. In addition, I obtained a Master's Degree in Geochemistry/Hydrogeology from the University of Nevada-Reno in 1993. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration (SME). I am a certified Professional Geologist in the State of Oregon (G1438). I have worked full time as a Geologist and Geochemist for a total of 34 years. My relevant experience includes design, execution, and interpretation of mine waste geochemical characterization programs in support of open pit and underground mine planning and environmental impact assessments, design and supervision of water quality sampling and monitoring programs, geochemical modeling, and management of the geochemistry portion of numerous PEA, PFS and FS-level mine projects in the US and abroad.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile property on November 18, 2021 for 1 day.
6. I am responsible for underground geochemistry, Section 16.1.5, 16.2.4, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is serving as QP on report titled "NI 43-101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" dated August 9, 2017.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

"Signed"

"Sealed"

David Bird, MSc., PG, RM-SME [4186987RM]

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denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, John Tinucci, PhD, PE, do hereby certify that:

1. I am Principal Consultant (Geotechnical Engineer) of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in B.S. in Civil Engineering from Colorado State University, in 1980. In addition, I have obtained a M.S. in Geotechnical Engineering from University of California, Berkeley, in 1983 and I have obtained a Ph.D. in Geotechnical Engineering, Rock Mechanics from the University of California, Berkeley in 1985. I am member of the American Rock Mechanics Association, a member of the International Society of Rock Mechanics, and a Professional Member of the Society for Mining, Metallurgy & Exploration. I have worked as a Mining and Geotechnical Engineer for a total of 37 years since my graduation from university. My relevant experience includes 34 years of professional experience. I have 20 years managerial experience leading project teams, managing P&L operations for 120 staff, and directed own company of 8 staff for 8 years. I have technical experience in mine design, prefeasibility studies, feasibility studies, geomechanical assessments, rock mass characterization, project management, numerical analyses, underground mine stability, subsidence, tunneling, ground support, slope design and stabilization, excavation remediation, induced seismicity and dynamic ground motion. My industry commodities experience includes salt, potash, coal, platinum/palladium, iron, molybdenum, gold, silver, zinc, diamonds, and copper. My mine design experience includes open pit, room and pillar, (single and multi-level), conventional drill-and-blast and mechanized cutting, longwall, steep narrow vein, cut and fill, block caving, sublevel caving and cut and fill longhole stoping and paste backfilling.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Haile property on November 29, 2016 for 3 days and November 26, 2018 for 3 days.
6. I am responsible for underground geotechnical information, Section 16.2.2 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is serving as QP on reports titled "NI 43-101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" dated August 9, 2017 and June 30, 2020.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

"Signed" *"Sealed"*

John Tinucci, PhD, PE

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1125 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Brian Prosser, PE do hereby certify that:

1. I am Principal Consultant of SRK Consulting (U.S.), Inc., 1125 Seventeenth Street, Suite 600, Denver, CO, USA, 80202.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report, Haile Gold Mine, Lancaster County, South Carolina" with an Effective Date of December 31, 2021 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Virginia Polytechnic Institute and State University in 1994. I am a licensed engineer in the states of Virginia, West Virginia, Kentucky, and Nevada. I am a Registered Member of the Society for Mining, Metallurgy, and Exploration and a member of the Mine Ventilation Society of South Africa. I have worked as a mining engineer for a total of 25 years since my graduation from university. My relevant experience includes the development of ventilation systems, network modeling, system troubleshooting, and design evaluation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Haile property.
6. I am responsible for ventilation, Section 16.2.10 and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is serving as QP on report titled "NI 43-101 Technical Report, Feasibility Study, Haile Gold Mine, Lancaster County, South Carolina" and dated August 9th, 2017.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th Day of March, 2022.

"Signed"

"Sealed"

Brian Prosser, PE
Principal Consultant

Appendix B: LoM Annual Cash Flow Forecast

Technical-Economic Model Summary

srk consulting

Company: **Oceanagold**

Haile

YEAR	US$ & Metric Units	LoM Total or Avg.	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Time Before Mine Closure			16	15	14	13	12	11	10	9	8	7	6	5	4	3	2
Production Timeline			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Discount Factors 5%			1.0000	0.9524	0.9070	0.8638	0.8227	0.7835	0.7462	0.7107	0.6768	0.6446	0.6139	0.5847	0.5568	0.5303	0.5051
Market Prices																	
Gold	US$/oz	**$1,500**	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500	$1,500
Silver	US$/oz	**$18.00**	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Physicals																	
Total OP Ore Mined	kt	40,170	2,898	2,888	4,593	1,675	2,710	4,678	3,600	3,600	2,919	2,670	3,519	4,419	-	-	-
Total OP Waste Mined	kt	350,907	38,380	34,844	27,524	35,718	38,430	32,019	33,259	27,520	35,051	26,550	17,186	4,425	-	-	-
Total OP Material Mined	kt	391,076	41,278	37,732	32,117	37,393	41,140	36,697	36,859	31,120	37,971	29,220	20,705	8,844	-	-	-
Strip Ratio	w:o	8.7	13.2	12.1	6.0	21.3	14.2	6.8	9.2	7.6	12.0	9.9	4.9	1.0	-	-	-
Total UG Ore Mined	kt	3,416	-	191	736	749	739	739	263	-	-	-	-	-	-	-	-
Total Ore Tonnes, Mined	kt	43,586	2,898	3,079	5,329	2,424	3,449	5,417	3,863	3,600	2,919	2,670	3,519	4,419	-	-	-
Total Ore Tonnes, Processed	kt	45,414	3,507	3,770	3,694	3,682	3,683	3,683	3,697	3,800	3,800	3,800	3,677	3,800	821	-	-
Gold Grade, Processed	g/t	1.75	1.71	1.77	2.62	1.49	2.63	2.49	1.93	1.43	0.87	0.89	1.36	1.72	2.27	-	-
Silver Grade, Processed	g/t	2.21	2.13	1.95	2.25	1.78	1.96	1.90	1.92	2.54	2.33	2.45	2.51	2.65	2.75	-	-
Contained Gold, Processed	koz	2,552	193	214	312	176	311	295	230	174	107	109	161	210	60	-	-
Contained Silver, Processed	koz	3,228	240	236	268	210	232	225	228	311	285	300	297	324	73	-	-
Average Recovery, Gold	%	83.9%	82.7%	84.2%	87.2%	82.5%	87.5%	86.2%	84.9%	81.7%	76.0%	76.3%	81.0%	83.9%	81.7%	--	--
Average Recovery, Silver	%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	70.0%	--	--
Recovered Gold, Dore	koz	2,141	160	181	272	146	273	255	195	142	81	83	130	176	49	-	-
Recovered Silver, Dore	koz	2,259	168	165	187	147	162	158	160	218	199	210	208	227	51	-	-
Payable Gold, Dore	koz	2,141	160	181	272	146	273	255	195	142	81	83	130	176	49	-	-
Payable Silver, Dore	koz	2,237	166	164	185	146	160	156	158	215	197	208	206	224	50	-	-
Cash Flow																	
Gross Revenue	**$000s**	**3,211,934**	**239,412**	**270,966**	**407,655**	**218,256**	**408,800**	**381,771**	**292,390**	**213,448**	**121,608**	**124,774**	**195,300**	**264,071**	**73,484**	**-**	**-**
OP Mining Cost	$000s	(545,064)	(31,330)	(25,274)	(49,080)	(15,772)	(28,451)	(65,937)	(28,798)	(73,687)	(79,210)	(54,380)	(55,565)	(31,710)	(5,870)	-	-
UG Mining Cost	$000s	(193,432)	-	(29,757)	(45,248)	(40,152)	(33,678)	(31,057)	(13,539)	-	-	-	-	-	-	-	-
Process Cost	$000s	(529,387)	(44,961)	(46,898)	(43,402)	(41,338)	(42,510)	(41,772)	(42,233)	(43,008)	(42,712)	(42,821)	(41,965)	(42,206)	(13,561)	-	-
Site G&A Cost	$000s	(222,212)	(19,815)	(20,756)	(20,052)	(19,712)	(18,362)	(17,933)	(17,484)	(17,403)	(17,023)	(16,143)	(14,782)	(12,884)	(9,865)	-	-
Dore Refining/Freight Costs	$000s	(5,732)	(411)	(470)	(649)	(388)	(597)	(559)	(465)	(455)	(320)	(335)	(420)	(533)	(130)	-	-
By-Product Credit	$000s	40,260	2,993	2,946	3,338	2,625	2,889	2,809	2,847	3,877	3,549	3,736	3,706	4,039	906	-	-
Direct Cash Costs	**$000s**	**(1,455,566)**	**(93,524)**	**(120,209)**	**(155,092)**	**(114,738)**	**(120,709)**	**(154,449)**	**(99,672)**	**(130,676)**	**(135,716)**	**(109,943)**	**(109,025)**	**(83,294)**	**(28,520)**	**-**	**-**

Royalties	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Environmental Bond	$000s	(14,800)	(800)	(1,550)	(1,550)	(1,550)	(1,550)	(1,550)	(1,550)	(950)	(750)	(750)	(750)	(750)	(750)	19,000	-
Interest Expense - Capital Leases	$000s	(3,600)	(3,600)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Principal Payment - Capital Leases: Sustaining	$000s	(51,259)	(8,540)	(8,437)	(12,064)	(18,548)	(3,669)	-	-	-	-	-	-	-	-	-	-
Principal Payment - Capital Leases: Non-Sustaining	$000s	(57,226)	(12,483)	(15,645)	(14,389)	(4,039)	(5,541)	(2,691)	(2,438)	-	-	-	-	-	-	-	-
Non-Operating Cash Costs	**$000s**	**(107,885)**	**(25,423)**	**(25,632)**	**(28,004)**	**(24,137)**	**(10,760)**	**(4,241)**	**(3,988)**	**(950)**	**(750)**	**(750)**	**(750)**	**(750)**	**(750)**	**19,000**	**-**
Total Project Costs	**$000s**	**(1,563,452)**	**(118,947)**	**(145,841)**	**(183,096)**	**(138,875)**	**(131,469)**	**(158,690)**	**(103,660)**	**(131,626)**	**(136,466)**	**(110,693)**	**(109,775)**	**(84,044)**	**(29,270)**	**19,000**	**-**
Operating Margin	**$000s**	**1,648,483**	**120,465**	**125,124**	**224,560**	**79,381**	**277,332**	**223,080**	**188,730**	**81,822**	**(14,858)**	**14,081**	**85,524**	**180,027**	**44,214**	**19,000**	**-**
Earnings Before Interest, Taxes & Depreciation	**$000s**	**1,648,483**	**120,465**	**125,124**	**224,560**	**79,381**	**277,332**	**223,080**	**188,730**	**81,822**	**(14,858)**	**14,081**	**85,524**	**180,027**	**44,214**	**19,000**	**-**
Depreciation Allowance	$000s	(1,858,417)	(128,000)	(131,987)	(142,130)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	(145,630)	-	-
Depletion Allowance	$000s	(479,832)	(35,766)	(40,480)	(60,900)	(32,605)	(61,071)	(57,033)	(43,680)	(31,887)	(18,167)	(18,640)	(29,176)	(39,450)	(10,978)	-	-
Earnings Before Taxes	**$000s**	**(689,766)**	**(43,301)**	**(47,342)**	**21,530**	**(98,854)**	**70,631**	**20,418**	**(580)**	**(95,695)**	**(178,655)**	**(150,189)**	**(89,282)**	**(5,052)**	**(112,394)**	**19,000**	**-**
Income Tax @ Eff. Rate of 0.0%	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Income	**$000s**	**(689,766)**	**(43,301)**	**(47,342)**	**21,530**	**(98,854)**	**70,631**	**20,418**	**(580)**	**(95,695)**	**(178,655)**	**(150,189)**	**(89,282)**	**(5,052)**	**(112,394)**	**19,000**	**-**
Non-Cash Add Back - Depreciation	$000s	1,858,417	128,000	131,987	142,130	145,630	145,630	145,630	145,630	145,630	145,630	145,630	145,630	145,630	145,630	-	-
Non-Cash Add Back - Depletion	$000s	479,832	35,766	40,480	60,900	32,605	61,071	57,033	43,680	31,887	18,167	18,640	29,176	39,450	10,978	-	-
Working Capital	$000s	(0)	1,813	4,795	(2,347)	(3,356)	589	2,298	(1,912)	2,925	1,533	(1,529)	(1,010)	(2,407)	(709)	(682)	-
Operating Cash Flow	**$000s**	**1,648,483**	**122,277**	**129,919**	**222,213**	**76,024**	**277,921**	**225,378**	**186,818**	**84,747**	**(13,324)**	**12,552**	**84,514**	**177,621**	**43,505**	**18,318**	**-**
Operations Information Technology	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Operations Expenditure	$000s	(28,456)	-	(9,756)	(3,700)	(3,000)	(3,000)	(3,000)	(3,000)	(3,000)	-	-	-	-	-	-	-
Operations Land and Building (Acquisitions)	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Brownfields Exploration	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operations Based Mining Projects	$000s	(292,829)	(60,058)	(32,765)	(37,582)	(54,401)	(34,022)	(9,026)	(40,276)	(8,626)	(8,816)	(2,984)	(2,984)	(1,290)	-	-	-
Capitalised Pre-Strip / UG Mine Development	$000s	(427,660)	(81,464)	(76,581)	(43,677)	(73,008)	(61,045)	(22,503)	(57,702)	-	(3,675)	(8,006)	-	-	-	-	-
Rehabilitation (Sustaining)	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Sustaining Capital Spend	**$000s**	**(748,945)**	**(141,522)**	**(119,102)**	**(84,959)**	**(130,409)**	**(98,067)**	**(34,528)**	**(100,978)**	**(11,626)**	**(12,490)**	**(10,990)**	**(2,984)**	**(1,290)**	**-**	**-**	**-**
Growth Information Technology	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Growth Expenditure	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Land and Building (Growth)	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Greenfields Exploration	$000s	(1,933)	-	(625)	(535)	(501)	(91)	(182)	-	-	-	-	-	-	-	-	-
Stand-alone Mining Projects	$000s	(82,049)	(34,581)	(31,960)	(5,551)	-	-	(6,033)	(3,924)	-	-	-	-	-	-	-	-
Capitalised Pre-Strip / UG Mine Development	$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Rehabilitation (Non-Sustaining)		$000s	(71,072)	-	-	(8,545)	-	(4,541)	(1,006)	(1,374)	(3,806)	-	-	(5,576)	(9,517)	(9,587)	(1,567)	(25,553)
Total Non-Sustaining Capital		**$000s**	**(155,054)**	**(34,581)**	**(32,585)**	**(14,631)**	**(501)**	**(4,632)**	**(7,220)**	**(5,298)**	**(3,806)**	**-**	**-**	**(5,576)**	**(9,517)**	**(9,587)**	**(1,567)**	**(25,553)**
Total Capital		**$000s**	**(903,999)**	**(176,103)**	**(151,688)**	**(99,591)**	**(130,910)**	**(102,699)**	**(41,749)**	**(106,275)**	**(15,432)**	**(12,490)**	**(10,990)**	**(8,560)**	**(10,807)**	**(9,587)**	**(1,567)**	**(25,553)**
Metrics																		
Economic Metrics																		
a) Pre-Tax																		
Free Cash Flow		$000s	**744,484**	(53,826)	(21,768)	122,622	(54,886)	175,222	183,630	80,543	69,315	(25,815)	1,563	75,954	166,814	33,918	16,751	(25,553)
Cumulative Free Cash Flow		$000s		(53,826)	(75,595)	47,028	(7,858)	167,364	350,994	431,537	500,852	475,037	476,600	552,555	719,368	753,286	770,037	744,484
NPV @	5.00%	$000s	**529,211**	(53,826)	(20,732)	111,222	(47,412)	144,156	143,879	60,102	49,261	(17,472)	1,007	46,629	97,533	18,887	8,883	(12,906)
Cumulative NPV		$000s		(53,826)	(74,558)	36,664	(10,748)	133,407	277,286	337,388	386,650	369,177	370,185	416,814	514,347	533,233	542,116	529,211
IRR		%	**73.3%**															
b) After-Tax																		
Free Cash Flow		$000s	**744,484**	(53,826)	(21,768)	122,622	(54,886)	175,222	183,630	80,543	69,315	(25,815)	1,563	75,954	166,814	33,918	16,751	(25,553)
Cumulative Free Cash Flow		$000s		(53,826)	(75,595)	47,028	(7,858)	167,364	350,994	431,537	500,852	475,037	476,600	552,555	719,368	753,286	770,037	744,484
NPV @	5.00%	$000s	**529,211**	(53,826)	(20,732)	111,222	(47,412)	144,156	143,879	60,102	49,261	(17,472)	1,007	46,629	97,533	18,887	8,883	(12,906)
Cumulative NPV		$000s		(53,826)	(74,558)	36,664	(10,748)	133,407	277,286	337,388	386,650	369,177	370,185	416,814	514,347	533,233	542,116	529,211
IRR		%	**73.3%**															
Operating Metrics (RoM)																		
Mine Life		Years	13															
Maximum Open Pit Mining Rate (Ore + Waste)		MTPA	41,278															
Maximum Ore Processing Rate		MTPA	3,800															
OP Mining Cost		$ / t rock	1.39	0.76	0.67	1.53	0.42	0.69	1.80	0.78	2.37	2.09	1.86	2.68	3.59	-	-	-
UG Mining Cost		$ / t UG ore mined	56.62	-	155.85	61.48	53.63	45.58	42.01	51.52	-	-	-	-	-	-	-	-
Total Mining Cost		$ / t ore milled	16.26	8.93	14.60	25.53	15.19	16.87	26.33	11.45	19.39	20.84	14.31	15.11	8.34	7.15	-	-
Processing Cost		$ / t ore milled	11.66	12.82	12.44	11.75	11.23	11.54	11.34	11.42	11.32	11.24	11.27	11.41	11.11	16.53	-	-
G&A Cost	18%	$ / t ore milled	4.89	5.65	5.51	5.43	5.35	4.99	4.87	4.73	4.58	4.48	4.25	4.02	3.39	12.02	-	-
Dore Refining/Freight Costs		$ / t ore milled	0.13	0.12	0.12	0.18	0.11	0.16	0.15	0.13	0.12	0.08	0.09	0.11	0.14	0.16	-	-
Total Operating Costs		$ / t ore milled	$32.94	27.52	32.67	42.89	31.88	33.56	42.70	27.73	35.41	36.65	29.92	30.66	22.98	35.86	-	-
Non-Operating Costs		$ / t ore milled	$2.79	7.25	6.80	7.58	6.56	2.92	1.15	1.08	0.25	0.20	0.20	0.20	0.20	0.91	-	-
Sales Metrics																		
LoM Dore Sales		koz Au	2,141	160	181	272	146	273	255	195	142	81	83	130	176	49	-	-
LoM All-In Sustaining Costs (AISC)		$000s	2,259,370	247,186	247,748	252,116	263,695	222,444	188,978	200,650	142,301	148,206	120,932	112,009	84,583	28,520	-	-
LoM AISC Metric		$ / oz Au	$1,055	$1,549	$1,371	$928	$1,812	$816	$743	$1,029	$1,000	$1,828	$1,454	$860	$480	$582	$0	$0