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

Effective Date: December 31, 2023
Report Date: March 28, 2024

Report Prepared for

OceanaGold Haile Gold Mine

Report Prepared by

SRK Consulting (U.S.), Inc.
999 Seventeenth Street, Suite 400
Denver, CO 80202

SRK Project Number: USPR001819

Signed by Qualified Persons:

David Carr, BEng Metallurgical (Hons), MAusIMM (CP) (OceanaGold Group Manager Metallurgy)
David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, RM-SME (OceanaGold Executive Vice President, Chief Operating Officer Americas)
Jonathan Moore, BSc Geology (Hons), Mausam (CP) (OceanaGold Group Manager Resource Development)
Brianna Drury, BEng Mining, RM-SME (OceanaGold Underground Engineering Superintendent)
Larry Standridge, PE, MSE Geotechnical (Call & Nicholas Principal Engineer, Geotechnical Engineer)
Robert Cook, PE, RM-SME (Call & Nicholas Principal I Geological Engineer)
Jay Newton Janney-Moore, PE, RM-SME (NewFields Senior Project Manager I)
William Lucas Kingston, MSc, P.G., RM-SME, Hydrogeology and Groundwater Management (NewFields Senior Hydrogeologist)
Matt Sullivan, BEng, RM-SME (SRK Principal Consultant, Mineral Economics)
Brooke Miller Clarkson, MSc., CPG (SRK Principal Consultant, Geology)

Reviewed by:

Joanna Poeck, BEng Mining, SME-RM, MMSAQP (SRK Principal Consultant, Mining Engineer)

Cautionary Note Regarding Forward-Looking Statements

This report contains certain "forward-looking statements" and "forward-looking information" (collectively, "forward-looking statements") within the meaning of applicable Canadian securities laws. All statements, other than statements of historical fact regarding OceanaGold Corporation ("OceanaGold") or the Haile Gold Mine, are forward-looking statements. The words "believe", "expect", "anticipate", "contemplate", "target", "plan", "intend", "project", "continue", "budget", "estimate", "potential", "may", "will", "can", "could" and similar expressions identify forward-looking statements. In particular, this report contains forward-looking statements with respect to cash flow forecasts, projected capital, operating and exploration expenditures, targeted cost reductions, mine life and production rates, potential mineralization and metal or mineral recoveries, and information pertaining to potential improvements to financial and operating performance and mine life at the Haile Gold Mine. All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to such assumptions, the forward-looking statements are inherently subject to significant business, economic and competitive uncertainties and contingencies. Known and unknown factors could cause actual results to differ materially from those projected in the forward-looking statements. Such factors include, but are not limited to: fluctuations in the spot and forward price of metals and commodities (including gold, diesel fuel, natural gas and electricity); the speculative nature of mineral exploration and development; changes in mineral production performance, exploitation and exploration successes; diminishing quantities or grades of reserves; increased costs, delays, suspensions, and technical challenges associated with the construction of capital projects; operating or technical difficulties in connection with mining or development activities, including disruptions in the maintenance or provision of required infrastructure and information technology systems; damage to OceanaGold's or the Haile Gold Mine's reputation due to the actual or perceived occurrence of any number of events, including: negative publicity with respect to the handling of environmental matters or dealings with community groups, whether true or not; risk of loss due to acts of war, terrorism, sabotage and civil disturbances; uncertainty whether the Haile Gold Mine will meet OceanaGold's capital allocation objectives; the impact of global liquidity and credit availability on the timing of cash flows and the values of assets and liabilities based on projected future cash flows; the impact of inflation; fluctuations in the currency markets; changes in interest rates; changes in national and local government legislation, taxation, controls or regulations and/or changes in the administration of laws, policies and practices; expropriation or nationalization of property and political or economic developments; failure to comply with environmental and health and safety laws and regulations; timing of receipt of, or failure to comply with, necessary permits and approvals; litigation; contests over title to properties or over access to water, power and other required infrastructure; increased costs and physical risks including extreme weather events and resource shortages, related to climate change; and availability and increased costs associated with mining inputs and labor. In addition, there are risks and hazards associated with the business of mineral exploration, development, and mining, including environmental hazards, industrial accidents, unusual or unexpected formations, pressures, cave-ins, flooding and gold bullion, copper cathode or gold or copper concentrate losses (and the risk of inadequate insurance, or inability to obtain insurance, to cover these risks).

Many of these uncertainties and contingencies can affect OceanaGold's actual results and could cause actual results to differ materially from those expressed or implied in any forward-looking statements made by, or on behalf of, OceanaGold. All of the forward-looking statements made in this report are qualified by these cautionary statements and OceanaGold and the qualified persons who authored this report undertake no obligation to update publicly or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.

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 pre-feasibility study (PFS) 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, 2023. The underground includes both the previously reported and currently operating Horseshoe area and the new development Palomino area. The Open Pit is currently operating. Operating components of this study are developed to a Feasibility study (FS) level and development components are to a PFS level.

1.1 Property Description and Ownership

The Haile Gold Mine is 100% owned and operated by OceanaGold and 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. As of December 31, 2023, HGM owns a total of 10,978 acres in South Carolina with no royalties. Of this total, 5,469 acres are within the mine permit boundary.

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 x 1 km area. The deposits occur within a variably deformed ENE-trending structural zone at or near the contact between metamorphosed Neoproterozoic volcanic and sedimentary rocks. The deposits are hosted in laminated siltstones and volcanic rocks of the Upper Persimmon Fork Formation and are 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. The age of gold mineralization is assumed at ~549 Ma, based on closely associated molybdenite dated using Re-Os (Mobley et al., 2014), which postdates peak volcanism. Pressure shadows around pyrite grains, stretched pyrite and pyrrhotite grains, and flattened hydrothermal breccia clasts indicate that some deformation has occurred subsequent to sulfide mineralization. The bulk geometry and orientation of the deposits implies emplacement subsequent to folding. The Re-Os date 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 structurally controlled, low-sulfidation, disseminated gold deposit.

1.3 Status of Exploration; Development and Operations

Resource definition drilling at Haile by Romarco Minerals and OceanaGold has significantly increased the resources since 2007. Reserve growth has resulted from exploration, conceptual 3D modeling, deep drilling of a previously underexplored gold system and higher gold prices. This has been recently exemplified by continued growth of the Horseshoe underground reserve (0.52 Moz) in 2023 and announcement of an initial reserve at Palomino in 2024 (0.38 Moz). In-house core drilling continues by OceanaGold at about 20 km per year, focused on resource conversion at high-grade underground targets proximal to the sedimentary-volcanic contact. Underground development of the Horseshoe deposit in 2023 provided underground drill stations for converting Horseshoe Inferred resource and other targets along the prospective one km long Horseshoe-Palomino trend. Underground core drilling is conducted by Major Drilling Group International.

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 over the life of mine (LoM).

Testing of core samples from the Horseshoe and Palomino deposits has been undertaken using the same laboratory flowsheet that correlates well with plant performance. Overall results suggest these deposits will respond well to processing in the existing process plant without modification.

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.

Model validation, peer review, team-based training and production reconciliation processes are regularly undertaken. Where appropriate, independent model estimates are completed and compared. Where appropriate, external model reviews are undertaken.

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 operating gold mine within the United States, or that would materially affect the Mineral Resource estimates.

1.5.1 Open Pit Mineral Resource Estimate

Drillhole data available as of April 2022 were included in the HA0922OLM_V6 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 overburden 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. Model block estimates for post-mineralization dikes were assigned zero gold grades post-estimation. These approaches are supported by relationships between mineralization, bedding and dikes observed in open pit grade control sample data.

Grades were estimated into 10 m E x 10 m N x 5 m RL blocks using 2.5 m bench composites. Grade estimation was completed 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 (OK) was used for silver, sulfur and carbon estimates, given the lower amount 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.

In situ dry densities are based upon lithologically grouped immersion determinations from core samples.

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

Table 1-1: Open Pit Mineral Resources as of December 31, 2023

Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Measured	3.8	1.02	0.12	1.1	0.1
Indicated	34.4	1.58	1.75	2.5	2.8
Measured & Indicated	38.1	1.53	1.87	2.4	2.9
Inferred	2.8	0.90	0.10	-	-

Source: OceanaGold, 2023

- Cut-off grade for primary mineralization is 0.50 g/t and 0.60 g/t Au for oxide, based on a gold price of US$1,700/oz.
- Open pit resource is reported within the open pit reserve design.
- Includes stockpiles of 2.1 million tonnes (Mt) at 0.89 g/t Au for 0.06 Moz gold.
- 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.

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 November 8, 2023.

Gold estimation was constrained within wireframes implicitly modeled using Leapfrog® software at a 1.0 g/t Au threshold[1] with a 25% probability. These 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 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 OK with 3 m composites.

In situ dry densities are based upon lithologically grouped immersion determinations from core samples.

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 OK model as presented in Table 1-2. A cut-off grade (CoG) of 1.55 g/t Au has been applied without mine design constraint. However, the reported resources broadly correspond with a conceptual mine stope optimized volume. The CoG assumes underground mining methods and is based on a gold price of US$1,700/oz. No mining dilution has been applied.

Table 1-2: Horseshoe Underground Mineral Resource Statement as of December 31, 2023

Class	Tonnes (Mt)	Gold (g/t)	Gold (Moz)	Silver (g/t)	Silver (Moz)
Measured	0.1	5.04	0.02	2.0	0.01
Indicated	3.7	5.49	0.66	2.3	0.3
Measured & Indicated	3.8	5.48	0.67	2.3	0.3
Inferred	1.8	4.1	0.2	2.1	0.1

Source: OceanaGold, 2023

- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- Cut-off grade 1.55 g/t Au based on a gold price of US$1,700/oz.
- No mining dilution applied.
- Spatially constrained by a 1 g/t threshold shell.
- 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 2023, with final assays received on October 11, 2023. The effective reporting date is recorded as December 31, 2023 to match the site-wide Haile mining depletion date, no mining has occurred at Palomino.

[1] This approximates a 0.4 g/t Au Indicator at a 50% probability.

Gold estimation was constrained within wireframes implicitly modeled using Leapfrog® software at a 0.8 g/t Au threshold[2] with a 25% probability. The lower grade threshold reflects a lower inflection than Horseshoe on the cumulative histogram. Post-mineralization dikes were assigned zero grade. Metasediment / metavolcanic contacts and the rhyodacite were not used to constrain gold or silver 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 an approximate 0.8 g/t threshold shell, with a lower grade indicator employed to account for grades outside the volume. On a block-by-block basis, the grade and probability of the higher and lower grade indicators were estimated by OK and weight-averaged to generate a final block grade.

In situ dry densities are based upon lithologically grouped immersion determinations from core samples.

Silver was estimated via co-kriging with gold, within the 0.8 g/t gold threshold shell.

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 1-3. The reported resource is constrained within a conceptual stope design based on a gold price of US$1,700/oz, approximating a 1.55 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 1-3: Palomino Underground Mineral Resource Statement as of December 31, 2023

Class	Tonnes (Mt)	Gold (g/t)	Gold (Moz)	Silver (g/t)	Silver (Moz)
Measured	0.0	0.00	0.00	0.0	0.0
Indicated	4.5	3.10	0.45	2.8	0.4
Measured & Indicated	4.5	3.10	0.45	2.8	0.4
Inferred	0.8	2.2	0.1	2.0	0.1

Source: OceanaGold, 2023

- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- Cut-off grade 1.55 g/t Au based on a gold price of US$1,700/oz.
- No mining dilution applied.
- Spatially constrained by a 0.8 g/t threshold shell.
- 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.

[2] This approximates a 0.4 g/t Au Indicator at 50% probability.

1.5.3 Combined Open Pit and Underground Resource Estimate

Table 1-4 presents the combined open pit, stockpiles, and underground resource statement for the Haile Property.

Table 1-4: Haile Combined Open Pit and Underground Resource Statement as of December 31, 2023

Type	Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Open Pit	Measured	3.8	1.02	0.12	1.1	0.1
	Indicated	34.4	1.58	1.75	2.4	2.7
	Measured & Indicated	38.1	1.53	1.87	2.4	2.9
	Inferred	2.8	0.90	0.08	-	-
Underground	Measured	0.1	5.04	0.02	2.0	0.0
	Indicated	8.2	4.18	1.10	2.6	0.7
	Measured & Indicated	8.3	4.19	1.12	2.6	0.7
	Inferred	2.5	3.7	0.30	2.1	0.2
Combined	Measured	3.9	1.12	0.14	1.1	0.1
	Indicated	42.6	2.08	2.85	2.4	3.3
	Measured & Indicated	46.4	2.01	2.99	2.4	3.6
	Inferred	5.3	2.22	0.38	1.0	0.2

Source: OceanaGold, 2023

- Cut-off grades for the open pit are 0.50 g/t / 0.60 g/t (primary / oxide), and for Horseshoe underground and Palomino underground are 1.55 g/t Au based on a gold price of US$1,700/oz.
- Open pit resource is reported within the open pit reserve design. Palomino is constrained within a conceptual stope design and Horseshoe underground is spatially constrained by a 1 g/t Au indicator shell.
- Open Pit estimate includes stockpiles of 2.1 Mt at 0.89 g/t Au for 0.06 Moz gold, classified as Measured.
- 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 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

Dilution and ore recovery have been applied to the resource block model to account for a portion of mineralized material expected to be mined by face shovel excavators. The resource block model was then used for open-pit optimization without further modification, as the block size in the model matched the SMU size of 10 m x 10 m x 5 m considered appropriate for the backhoe excavator loading units operating at Haile. This has limited impact on the Mineral Reserve, with an effective global dilution of 2.8% and mining recovery of 99.8%.

The open-pit Mineral 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 and silver price of US$18/oz Ag. Subsequent to pit optimization, inferred material (approximately 10% by volume) within the reserve pit was treated as waste and given a zero-gold grade. The overall pit slopes (inter-ramp angle slopes) used for the design are based on operational level geotechnical studies and range from 30° 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 36.4 Mt with an average grade of 1.56 g/t. The Mineral Reserve statement, as of December 31, 2023, for the Haile Open Pit is presented in Table 1-5.

Table 1-5: Haile Open Pit Mineral Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven (1)	3.6	1.03	1.6	0.12	0.2
Probable	32.8	1.62	2.4	1.71	2.5
Proven + Probable	36.4	1.56	2.3	1.82	2.7

Source: OceanaGold, 2024
(1) Includes 2.0 Mt of stockpile material grading 0.9 g/t Au and 0.9 g/t Ag

- Reserves are based on a US$1,500/oz Au gold price and US$18/oz Ag silver 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 for gold 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%.
- Metallurgical recovery for silver is applied at 70%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- 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 under the supervision of David Londono 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.

1.6.2 Underground Mineral Reserves Estimate

The current underground reserves consist of two deposits: Horseshoe Underground (HUG) and Palomino Underground (PUG). These deposits are separated by ~1 km of development and encompass mineralization that extends down at depth and outside the pit extents.

Based on the orientation, depth, and geotechnical characteristics of the mineralization, a transverse sublevel open stoping method (longhole) with ramp access is used for both deposits. The stopes will be 20 m wide at HUG and 15 m wide for PUG 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 underground mine design process resulted in combined underground mining reserves of 7.9 Mt (diluted) with an average grade of 3.56 g/t Au for both Horseshoe & Palomino. The Mineral Reserve statement, as of December 31, 2023, for the Haile Horseshoe and Palomino Undergrounds are presented in Table 1-6.

This estimate is based on a mine design cut-off of 1.74 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 1-6: Haile Underground Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	0.1	4.53	2.0	0.01	0.01
Probable	7.7	3.54	2.0	0.88	0.5
Proven + Probable	7.8	3.56	2.0	0.89	0.5

Source: OceanaGold 2024

- 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 ~85%.
- The reserve estimate is based on a mine design using an elevated cut-off grade of 1.87 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 1.74 g/t Au and an incremental development cut-off grade of 0.59 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.
- 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 Brianna Drury of OceanaGold, a Qualified Person.

The authors know of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the underground Mineral Reserve estimate.

1.6.3 Combined Open Pit and Underground Reserves Estimate

Table 1-7 presents the combined open pit and underground Mineral Reserves statement for Haile.

Table 1-7: Combined OP and UG Reserve Statement for OceanaGold's Haile Gold Mine as of December 31, 2023

Type	Category	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au Contained (Moz)	Ag Contained (Moz)
OP	Proven (1)	3.6	1.03	1.6	0.12	0.2
	Probable	32.8	1.62	2.4	1.71	2.5
	Proven + Probable	36.4	1.56	2.3	1.82	2.7
UG	Proven	0.1	4.53	2.0	0.01	0.0
	Probable	7.7	3.54	2.0	0.88	0.5
	Proven + Probable	7.8	3.56	2.0	0.89	0.5
OP + UG	Proven	3.7	1.13	1.6	0.13	0.2
	Probable	40.6	1.98	2.4	2.58	3.1
	Proven + Probable	44.3	1.91	2.3	2.72	3.3

Source: OceanaGold, 2024

(1) Includes 2.0 Mt of stockpile material grading 0.9 g/t Au and 0.9 g/t Ag

- Mineral Reserves are based on a gold price of US$ 1,500/oz Au and silver price of US$18/oz Ag.
- 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 85% for the underground material.
- Metallurgical recovery for silver is applied at 70%.
- 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.
- The Underground reserve estimate is based on a mine design using an elevated cut-off grade of 1.87 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 1.74 g/t Au and an incremental development cut-off grade of 0.59 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.
- 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 supported by a positive cash-flow model.
- The open pit Mineral Reserves were estimated under the supervision of David Londono of OceanaGold, a Qualified Person. The underground Mineral Reserves were estimated by Brianna Drury of OceanaGold, a Qualified Person.

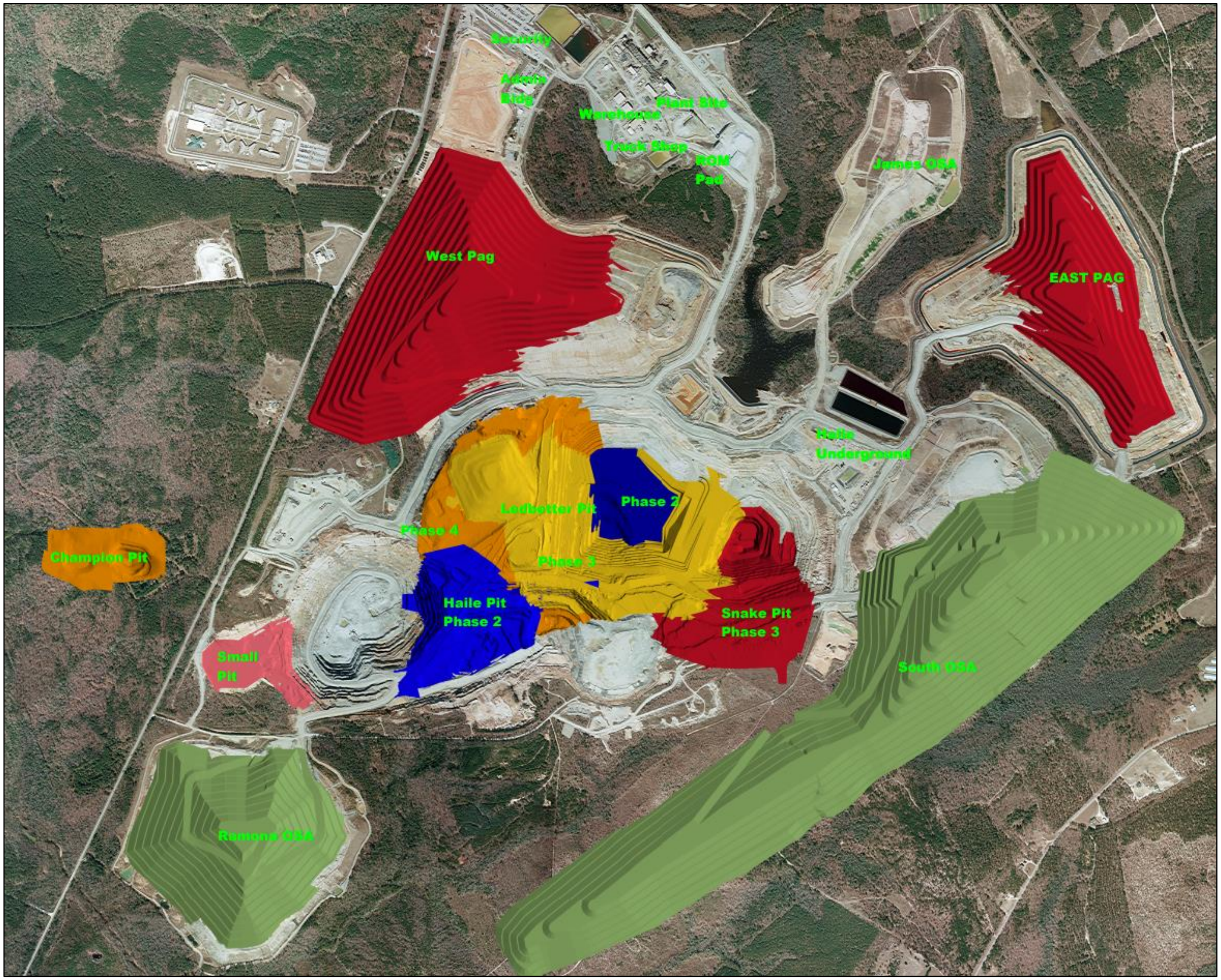
An underground vs. open-pit trade-off study has been commenced specifically targeting the mineralization within and in close proximity to Ledbetter Phase 4. This study is expected to be completed in March 2025. If the underground option is proven to be economically preferable, this will improve the overall project value, however with a potential reduction in overall Mineral Reserves.

1.7 Mining Methods

1.7.1 Open Pit Mining Methods

Haile is currently being mined using conventional open pit methods. Overburden and waste material are classified using blasthole sulfur and carbon assays that inform 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. "Green" Non-PAG material can be placed in unlined facilities or used for construction.

The open pit that forms the basis of open pit reserves and 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.



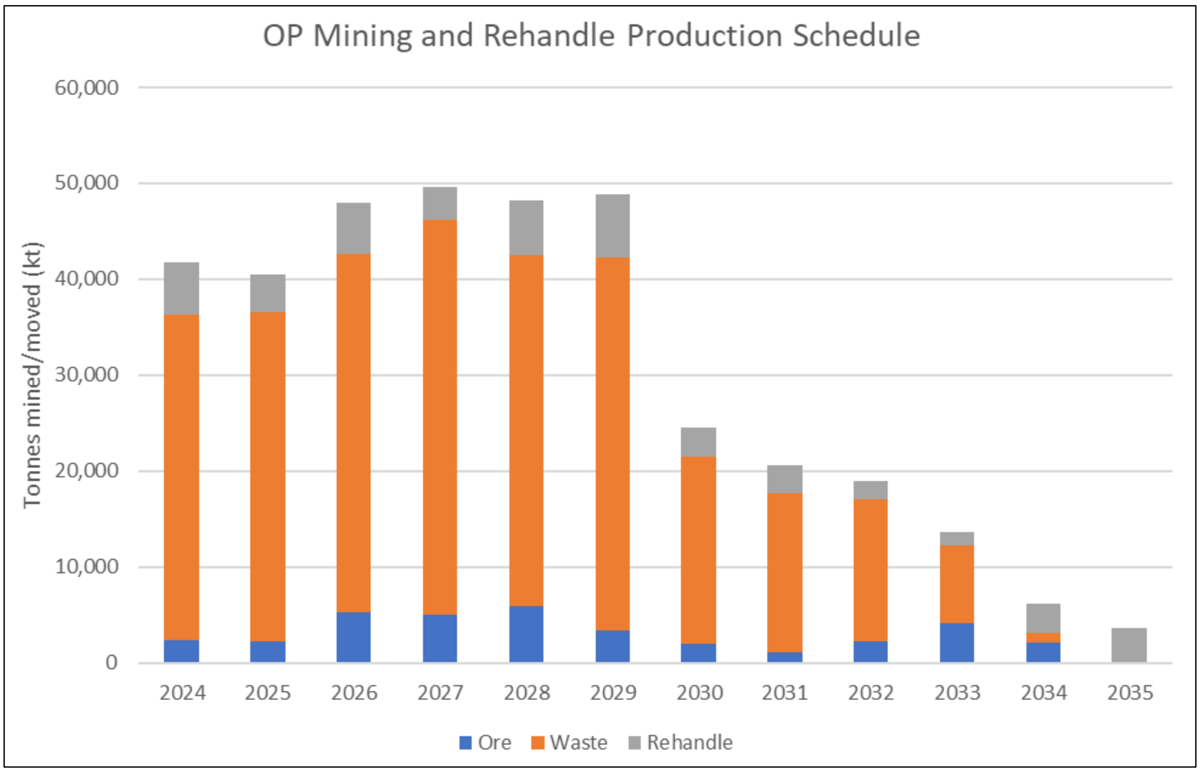
Source: OceanaGold, 2024
Current Phases and Final LOM Ex-Pit Facilities

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. Processed tonnes average approximately 3.7 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 between 2024 and 2029.

At the end of the mine life, a low-grade stockpile inventory of 1.3 Mt at 0.6 g/t Au and 2.9 g/t Ag remains unprocessed due to limitations on current TSF capacity. This material has been excluded from the Mineral Reserve. This material will be stored within the PAG cell footprint and is currently assumed to be rehabilitated with that facility. Further investigation of TSF capacity requirements and constraints will be completed in conjunction with the Trade-off Study noted in Section 1.6.3.

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



Source: OceanaGold, 2024

- Low ore production in 2024 and 2025 relates to completion of Ledbetter 2A in 1Q 2025 and Ledbetter Phase 3 reaching consistent ore in 3Q 2025
- Total mining movement rates reduce from 2030 on to control low-grade stockpile size given the availability of underground ore for processing feed.
- Low ore production and grade in 2030 and 2031 relates waste stripping in Ledbetter Phase 4

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 730E). Blasthole drilling and wall control drilling is performed with a fleet of Sandvik DR410i, Sandvik Leopard DI650i, and Epiroc D65 drills. Typical ancillary equipment, including track dozers, wheel dozers, motor graders and water trucks support the mining operation. Table 1-8 shows the major equipment required annually to achieve the mine schedule.

Table 1-8: Major Equipment Required to Achieve the Mine Schedule

Fleet	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
PC4000 Front Shovel	2	2	2	2	2	2	2	2	2	1	1
PC3000 Excavator	1	1	1	1	1	1	1	1	1	1	1
CAT 6020B Excavator	1	0	1	1	1	1	1	1	1	1	0
Cat 785 Trucks	1	1	1	1	1	1	1	1	2	2	1
Komatsu 730 Trucks	16	18	19	19	17	17	9	8	8	8	8
Sandvik DR410i Drill	4	4	4	4	4	4	4	4	4	2	2
Sandvik Leopard DI650i & Epiroc D65 Drill	3	4	4	4	4	4	2	2	2	3	2

Source: OceanaGold, 2024

1.7.2 Underground Mining Methods

Geotechnical

Geotechnical field characterization programs have been undertaken to assess the expected rock quality at the underground targets. These programs included geotechnical core logging, laboratory strength testing, in situ stress measurements and oriented core logging of discontinuities. The results of these programs have provided adequate quantity and quality of data for feasibility-level design of Horseshoe and prefeasibility-level design of Palomino.

Geotechnical assessments of the Horseshoe and Palomino orebody shapes and ground conditions has determined that longhole open stoping mining is an appropriate mining method. The design has been laid out using empirical design methods based on similar case histories. 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.

Horseshoe has been successfully put into production and is serviced by 3 portals (production decline, exhaust air decline, and fresh air decline) located in the Snake Pit. Palomino is planned for access via drift development from the Horseshoe main decline and then ventilated using a shaft proximal to the Palomino orebody. A preliminary shaft stability and liner design have been completed in support of the prefeasibility study.

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). Aggregate for CRF is sourced from Green Non-PAG overburden produced from the open pit mine. Stope sizes used in the optimization were 25 m high, 20 m wide for Horseshoe and 15 m wide for Palomino, and varying stope length based on geotechnical considerations.

Each stope has a 5 m x 5 m access located at the bottom of the stope. Top accesses (also 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. Development rock that is brought to surface is stockpiled on a clay liner until it is moved to a lined overburden storage facility by the open pit mine operations department. All development rock from the underground mine is handled as “Red” PAG for efficiency given the small amount of waste material produced during development.

The Horseshoe and Palomino underground mine production schedules are 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 safely possible.

Table 1-9 shows the yearly underground production schedule and includes both Horseshoe and Palomino material.

Table 1-9: Haile Underground Annual Production

Year	Ore Tonnes (kt)	Au (g/t)	Ounces (koz)	Waste Tonnes (kt)	Backfill Volume (m^3)
2024	561.7	3.99	72.1	292.9	197,853
2025	624.5	4.05	81.2	276.3	221,018
2026	778.4	3.40	85.1	348.2	286,414
2027	761.4	4.64	113.6	303.2	279,642
2028	1,078.9	3.40	117.8	198.4	377,636
2029	968.5	3.65	113.6	352.6	383,041
2030	866.9	3.68	102.6	186.2	279,005
2031	746.7	2.91	69.9	1.3	310,851
2032	748.8	3.01	72.5	12.8	286,982
2033	642.6	2.95	61.0	-	307,671
2034	80.4	4.05	10.5	-	72,355
Total	**7,858.9**	**3.56**	**899.9**	**1,972.3**	**3,002,467**

Source: OceanaGold, 2024

1.7.3 Combined OP and UG Production Schedule

Table 1-10 shows the combined open pit and underground production schedule annually. A schedule optimization process is planned to be completed in 2024 that will aim to identify the ideal start date for the Palomino Underground, open-pit production rate, and maximum stockpile size to maximize cashflow and value for the overall project.

Table 1-10: Combined OP and UG Production Schedule [(1)]

Year		2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	LoM Total
Total (OP+UG) Production													
Ore	kt	2,921	2,863	6,059	5,808	6,955	4,366	2,850	1,839	2,975	4,773	2,151	43,559
Ore	t/d	8,003	7,844	16,600	15,912	19,055	11,962	7,808	5,038	8,151	13,077	5,893	10,849
Au Grade	g/t	2.41	2.29	2.30	1.94	1.63	1.56	1.75	1.84	1.75	1.69	2.40	1.92
Contained Au	koz	226	210	448	362	365	219	160	109	167	260	166	2,692
Ag Grade	g/t	2.37	2.13	2.32	2.37	2.12	2.34	2.63	2.22	2.70	2.53	2.93	2.38
Contained Ag	koz	223	196	451	442	474	328	241	131	259	389	202	3,335
Underground													
Ore	kt	562	624	778	761	1,079	969	867	747	749	643	80	7,859
Ore	t/d	1,539	1,711	2,133	2,086	2,956	2,654	2,375	2,046	2,052	1,761	220	1,957
Au Grade	g/t	3.99	4.05	3.40	4.64	3.40	3.65	3.68	2.91	3.01	2.95	4.05	3.56
Contained Au	koz	72	81	85	114	118	114	103	70	72	61	10	900
Ag Grade	g/t	1.75	1.70	1.68	1.78	2.33	2.47	2.10	2.01	3.03	2.57	3.49	2.18
Contained Ag	koz	32	34	42	43	81	77	58	48	73	53	9	551
Waste	kt	293	276	348	303	198	353	187	1	13	-	-	1,972
Open Pit													
Ore	kt	2,359	2,239	5,281	5,047	5,876	3,397	1,983	1,092	2,226	4,130	2,071	35,700
Ore	t/d	6,446	6,134	14,468	13,827	16,056	9,307	5,433	2,993	6,081	11,314	5,673	8,885
Au Grade	g/t	2.03	1.79	2.14	1.53	1.31	0.97	0.90	1.10	1.32	1.50	2.34	1.56
Contained Au	koz	154	129	363	248	247	105	57	39	95	199	156	1,792
Ag Grade	g/t	2.52	2.24	2.41	2.46	2.08	2.30	2.86	2.37	2.59	2.53	2.90	2.43
Contained Ag	koz	191	162	409	398	393	251	182	83	186	336	193	2,785
Strip Ratio	t/t	14.4	15.4	7.1	8.2	6.2	11.4	9.9	15.2	6.7	2.0	0.5	7.9
Waste	kt	33,992	34,367	37,372	41,170	36,634	38,892	19,550	16,551	14,860	8,163	1,073	282,624

Source: OceanaGold, 2024

[(1)] Does not include stockpile material

- Open Pit ore and Open Pit contained Au ounce profile from 2024 – 2026 driven by transition between phases Ledbetter 2A and Ledbetter Phase 3
- Open Pit ore production and grade in 2030 and 2031 relates to waste stripping in Ledbetter Phase 4

1.8 Recovery Methods

The processing plant continues to utilize the conventional flowsheet developed 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 operating pits in a pipe and released into an unnamed tributary downstream of the pits.

The underground infrastructure required to support underground mining includes upgrade and extension of the ventilation, dewatering, power lines, air and water supply. The underground surface area includes run-of-mine (RoM) pad area which contain the stockpiles, CRF plant, as well as a truck shop, offices, and laydown area. The current underground surface infrastructure utilized for Horseshoe will support Palomino with nominal upgrades required. Palomino will require an upgrade and extension of high voltage power line run along with ventilation shaft.

1.10 Environmental Studies and Permitting

In late 2022, HGM received all of the major permits required to begin construction and operation of the mine including: 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). Haile is unique in that the Project 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.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the mining process.

In 1Q 2024, South Carolina Department of Health and Environmental Control approved mine operating permit modifications for additional Horseshoe underground levels and Palomino.

1.11 Capital and Operating Costs

Capital Cost

The total LoM capital is US$1,015 million as summarized in Table 1-11. LoM Non-sustaining capital is US$282 million. LoM Sustaining capital is US$733 million, the majority of which includes open pit capitalized pre-strip, TSF expansion, and underground operating capital development. The remaining balance carries primarily for surface infrastructure, open pit mine equipment replacements and underground development. Capital expenditures have been estimated referencing same or similar works at Haile, quotations from suppliers and estimates provided by consultants with appropriate expertise.

Capital expenditure estimation is consistent with proposed development programs and ongoing requirements and undertaken to an appropriate level of estimation accuracy. Actual expenditures are likely to vary over the LoM due to modifications, upgrades, introduction of new technology and other unforeseen factors.

Table 1-11: Total Capital Expenditure Summary (US$000's)

Description	Non-sustaining Capex	Sustaining Capex	Total
Land Acquisitions	-	-	-
Permitting	1,016	-	1,016
On-Site Exploration Drilling	15,523	372	15,895
OP Capitalized Pre-Strip	-	408,813	408,813
OP Mining PP&E	-	125,276	125,276
OP Tech Services PP&E	-	892	892
Pit Dewatering	-	4,900	4,900
Site Works	3,950	2,725	6.675
PAG Cell Development	-	40,121	40,121
TSF Lift Design	-	93,411	93,411
Underground Mining	129,212	30,361	159,573
Mill PP&E	16,536	25,828	42,364
Total Net Capex	**$166,237**	**$732,699**	**$898,936**
Reclamation/Closure (1)	115,599		115,599
Total LoM Net Capex	**$281,836**	**$732,699**	**$1,014,535**

Source: OceanaGold, 2024
(1) Captured as Capex in Cashflow

Operating Cost

The total LoM operating cost (excluding capitalized operating cost) is US$2,045 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 LoM operating costs, and the total RoM operating cost unit rate of US$46.21/t processed are summarized in Table 1-12.

Table 1-12: RoM Operating Cost Summary

Description	US$000's	US$/t Mined
OP Mining ($/t rock mined (ore and waste)) - All Material	1,070,195	3.36
OP Mining ($/t rock mined (ore and waste)) - (excl. capitalized cost)	661,382	2.08
UG Mining ($/t rock mined (ore and waste)) – All Material	533,669	54.28
UG Mining ($/t rock mined (ore and waste)) - (excl. capitalized cost)	466,019	47.40
	US$000's	**US$/t Ore Processed**
Subtotal Mining (Operational Material Only)	1,127,402	25.47
Processing	668,051	15.09
G&A Cost	242,173	5.47
Refining/Freight Costs	7,385	0.17
Total Operating Costs	**$2,045,011**	**$46.21**

Source: OceanaGold, 2024

There are several important cost items incurred but excluded from the operating cost which are detailed in Table 1-13 because OceanaGold does not consider them to be direct operating. These include payments related to leasing arrangements of the open pit and underground mobile equipment fleets.

Table 1-13: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond	12,200	0.27
Environmental Bond Release	-20,000	-0.45
Interest Expense - Capital Leases	3,382	0.07
Principal Payment - Capital Leases: Sustaining	32,452	0.71
Principal Payment - Capital Leases: Non-Sustaining	27,546	0.61
Total Non-Operating Costs	**$55,580**	**$1.26**

Source: OceanaGold, 2024

1.12 Economic Analysis

The Project consists of an operating surface and underground mine with a mill. The milling facility is mainly fed by the OP mine. The mill feed is supplemented with ore from underground at a 1.1 million tpa max annual capacity operation.

The Project is expected to produce 2.3 Moz of payable gold over a 12-year mine life at an average rate of 192 koz Au per year during full production years with a LoM AISC of US$1,200/oz Au.

The Project is expected to incur sustaining capital in the amount of US$732.7 million over the modeled life and a non-sustaining capital spend, including rehabilitation costs, of US$281.8 million for total capital expenditure of US$1,014.5 million.

The project cash flow results using the reserve price of US$1,500 / oz gold flat over the LoM and a 5% discount rate include a pre-tax NPV of US$261 million and after-tax NPV of US$256 million. As a result of significant depreciation and depletion, the operation is expected to incur minimal income tax liability at the reserve price. Existing loss carryforwards have not been included in the economic model. Inclusion of these items may further reduce the income tax liability of the operation.

OceanaGold provided an alternative price profile (refer section 22.4.2) which consists of a flat US$2,000/oz gold price 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$1,177 million and US$1,044 million, respectively.

As summary of the model results for both the reserve case and the OceanaGold price case is presented in Table 1-14.

Table 1-14: Indicative Economic Results

Description	US$000's	US$000's
Scenario	**Reserve Case Price**	**Alternative Price**
Market Prices		
Gold (US$/oz)	1,500	2,000
Silver (US$/oz	18	24
Payable Gold (koz)	2,303	2,303
Revenue		
Gross Gold Revenue	3,454,270	4,605,693
Silver By-Product Credit	40,810	54,414
Total Gross Revenue	**$3,495,080**	**$4,660,107**
Operating Costs		
Total Operating Costs	**($2,100,591)**	**($2,100,591)**
Operating Margin (EBITDA)	$1,394,490	2,559,516
Taxes		
Income Tax	(4,699)	(153,526)
Operating Cash Flow	$1,389,790	2,405,990
Capital		
Total Capital	**($1,014,535)**	**($1,014,535)**
Metrics		
Pre-Tax Free Cash Flow	$379,954	1,544,981
After-Tax Free Cash Flow	$375,255	1,391,455
Pre-Tax NPV at 5%	$260,540	1,177,167
After-Tax NPV at 5%	$256,340	1,043,576

Source: SRK, 2024

Because the Project is operational and is valued on a total project basis and not by an incremental analysis, 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

Haile 3D geologic models are updated using Maptek's Vulcan™ and Seequent's Leapfrog software, based largely on core logging and pit mapping to reflect controls to mineralization. Geological / grade-based 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 a resource estimation. The drillhole database and resource estimation methodology are appropriate for the purposes of estimating the open pit gold resources.

Model validation, peer review, and production reconciliation processes are regularly undertaken. Where appropriate, independent model estimates are completed and compared. Where appropriate, external model reviews are undertaken.

Haile 3D geologic models continue to be integrated with metallurgical data to facilitate geometallurgical modeling. Continue using portable XRF testing or other technologies to further refine the geology interpretation (in tandem with in-pit studies).

1.13.1 Geology and Mineral Resources

Open Pit

Open pit mining commenced in 2016. Reasonable long term resource model to mine-to-mill reconciliation performance suggests that the data, interpretation, estimation and classification methodologies are appropriate. The drill spacing for Indicated Mineral Resources is approximately 35 m x 35 m but variable and locally is up to 42 m.

Local grade variability characterizes the resource and open pit reconciliation analysis together with sensitivity modeling using simulated resource drilling sets suggest that the annual reconciliation performance previously experienced will remain a feature of the resource estimates.

The commencement of underground mining at Horseshoe means that there will be three confluent mill feed sources (open pit, underground and stockpiles - putting aside individual pit stages or underground development areas). Short-term multi-source mine to mill reconciliation may be a challenge. However, long term performance is expected to be acceptable.

Moving point of origin analysis using closely spaced production sample data will be used to quantify likely quarterly and annual estimation uncertainty and therefore offer a tool for optimizing infill drilling.

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 will continue to be integrated with geotechnical, overburden storage and metallurgical evaluations to optimize reserve growth, mill recoveries and throughput.

Underground

OceanaGold has undertaken 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 resource estimation. Model validation, peer review, and production reconciliation processes are regularly undertaken. From time to time, external independent model reviews are undertaken.

To-date the results from grade control drilling during 2023 for underground mining at Horseshoe are in line with the resource estimate. Underground mapping and model to mine reconciliation will continue be used to test and improve the Horseshoe estimate.

1.13.2 Status of Exploration; Development and Operations

OceanaGold will continue to expand resources and reserves in the Haile district through core drilling aligned with LoM plans. Systematic target generation supported by mapping, drilling, geochemistry, and geophysics will continue to guide exploration over the next five years, particularly in the search for underground deposits. 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.

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 1.74 g/t Au was used for both Horseshoe and Palomino. A detailed 3D mine design was completed around economically minable areas above cut-off grade.

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.

Metallurgical test work of underground ore samples performed to date for both Horseshoe and Palomino deposits supports the existing processing plant design will effectively be able to treat the material produced.

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

Future Studies and Recommendations

Three future studies are either in progress or are recommendations from this report, including:

- Ledbetter Phase 04 open-pit vs. underground trade-off study
- Palomino Underground Feasibility Study
- Site-wide schedule optimization, aimed at maximizing NPV by:
    - Optimizing timing of underground development capital
    - Optimizing open-pit production rate and size of low-grade stockpile

Furthermore, an internal continuous improvement program and resources are in place to progress existing initiatives, and identify additional initiatives, focusing on drill / blast practices, equipment performance, technician and operator training, and other cost reduction opportunities.

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 meet the requirement of the LoM plan maximum of 3.8 Mtpa.

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.

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 for the Palomino deposit.

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 required.

1.13.6 Project Infrastructure

The primary infrastructure changes required to support the open pit operation are increases in size of the main waste storage facilities such as the TSF and PAG OSAs. Major site water management facilities are in place.

The surface infrastructure required for the development of the underground Horseshoe and Palomino deposits are straightforward and relatively minor.

1.13.7 Environmental Studies and Permitting

The mine is currently operating as permitted. In 1Q 2024, South Carolina Department of Health and Environmental Control approved two modifications to Haile's Mine Operating Permit. An expansion of the Horseshoe underground operation was approved on February 21, 2024, and the Palomino underground operation was approved on March 15, 2024.

Permits currently held by HGM may be kept, modified, terminated, or replaced during any expansion process.

1.13.8 Economic Analysis

The current metal price environment is 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 pre-feasibility Study (PFS) 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 underground includes both the previously reported and currently operating Horseshoe area and the new development Palomino area. The Open Pit is currently operating. Operating components of this study are developed to a Feasibility study (FS) level and development components are to a PFS level.

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 are 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).

This report includes technical data from the Pre-feasibility Study to support the initial Mineral Reserve for the Palomino Underground, first reported in the OceanaGold Mineral Reserve Statement for End of Year 2023.

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 Group Manager Metallurgy) 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.
- David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, RM-SME (OceanaGold Executive Vice President, Chief Operating Officer Americas) is the QP responsible for environmental and open pit Mineral Reserves, the open pit portions of Section 15 and 16.3, Section 16 opening statements, Sections 16.1, 16.1.1, 16.1.3, 16.1.4, 16.1.6, 16.1.7, 16.1.8, 18.7, 19, 20, the open pit capital and operating costs portion of section 21, the other/G&A portions of the operating costs in section 21, the tailings/overburden capital and 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 Group Manager, Resource Development), 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, RM-SME (OceanaGold Underground Engineering Superintendent), is the QP responsible for underground Mineral Reserves, the underground portions of Section 15 and 16.3, Sections 16.2, 16.2.1, 16.2.5 - 16.2.11, 18.8, 18.9, the underground mining 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.
- Robert Cook, PE, RM-SME, (Call & Nicholas Principal I Geological 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.
- Jay Newton Janney-Moore, PE, RM-SME, (NewFields Senior Project Manager I), is the QP responsible for tailing and overburden storage, Sections 18.1, 18.2, 18.3, 18.4, 18.5, the tailings/overburden capital and operating cost portions of section 21, and portions of Sections 1, 25 and 26 summarized therefrom, of this Technical Report.
- William Lucas Kingston, MSc, P.G., RM-SME, 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, 18.6 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.
- Brooke Clarkson, MSc, C.P.G. (SRK Principal Consultant, Geology), is the QP responsible for geochemistry, Section 2, 3, 16.1.5, 16.2.4, 24, 27, 28 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
Robert Cook	Call & Nicholas	Geotechnical	November 29-30, 2017 July 27-28, 2021 January 16-19, 2023 January 22- 24, 2024	Inspect open pit and underground mining conditions and portal stabilization work.
Larry Standridge	Call & Nicholas	Geotechnical	November 29-30, 2017 October 10-11, 2018 January 14-16, 2020 May 18-20, 2021 January 24, 2024	Review of existing pit slopes and examination of geotechnical drill core.
David Carr	OceanaGold	Metallurgy	Various visits in 2017, 2018, 2019, 2020 and 2022, August 28-September 23 and November 6-December 14, 2023	Plant Commissioning support and plant investigations. Plant Commissioning and ramp-up of operations to 4 Mtpa debottlenecking investigations and study support.
Jay Newton Janney-Moore	NewFields	Geotechnical/ Infrastructure	Sept. 24-26, 2019 Aug 31-Sept 2, 2021 Oct 18, 2022 Sept. 26-27, 2023	Inspection of the Duckwood TSF, PAG OSA, and geomembrane lined ponds.
Brooke Clarkson	SRK	Geochemistry	November 14, 2023	Visited Overburden Storage Areas and open pit mine, tour of surface facilities on site.
William Lucas Kingston	NewFields	Hydrogeology and Groundwater Management	December 4-5, 2019 August 31 - September 2, 2021	General site inspection, including dewatering system.
Jonathan Moore	OceanaGold	Geology/ Resources	Annually since 2015. Most recently, November 20 to December 8, 2023	Review of Resource and Reconciliation. Drill core, open pit and underground visits.

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, 2023.

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 various project areas in support of this Technical Report.

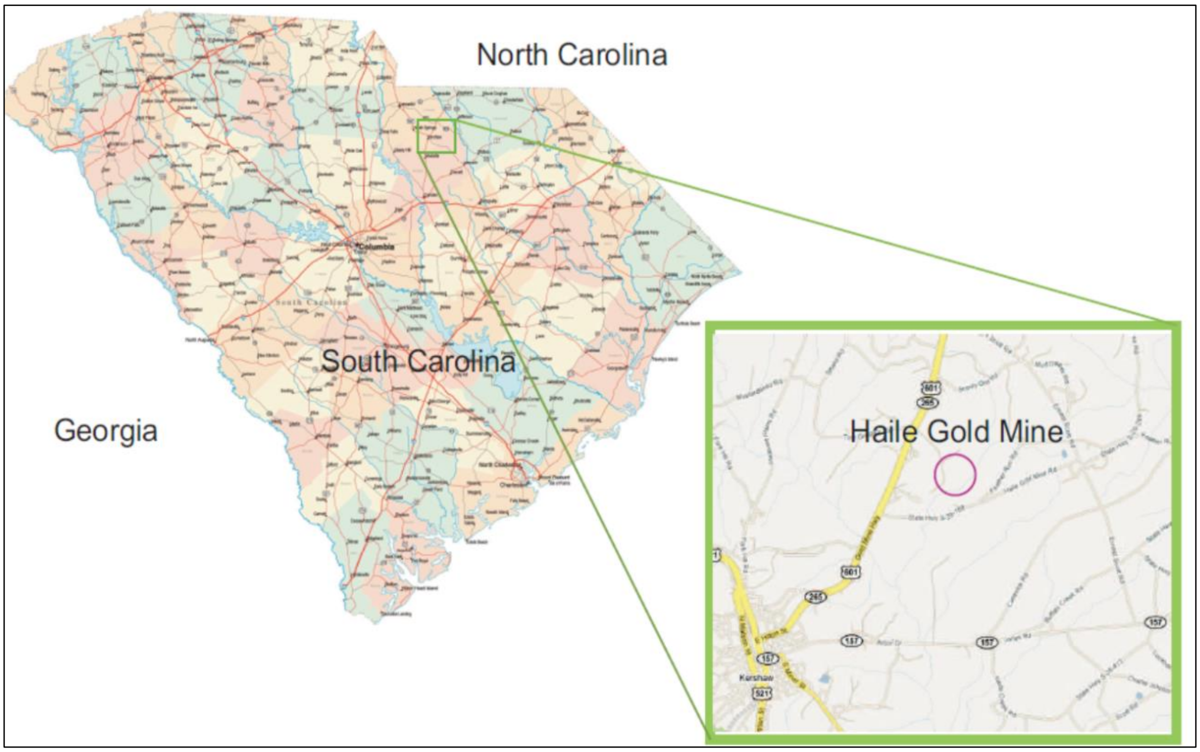
The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals 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

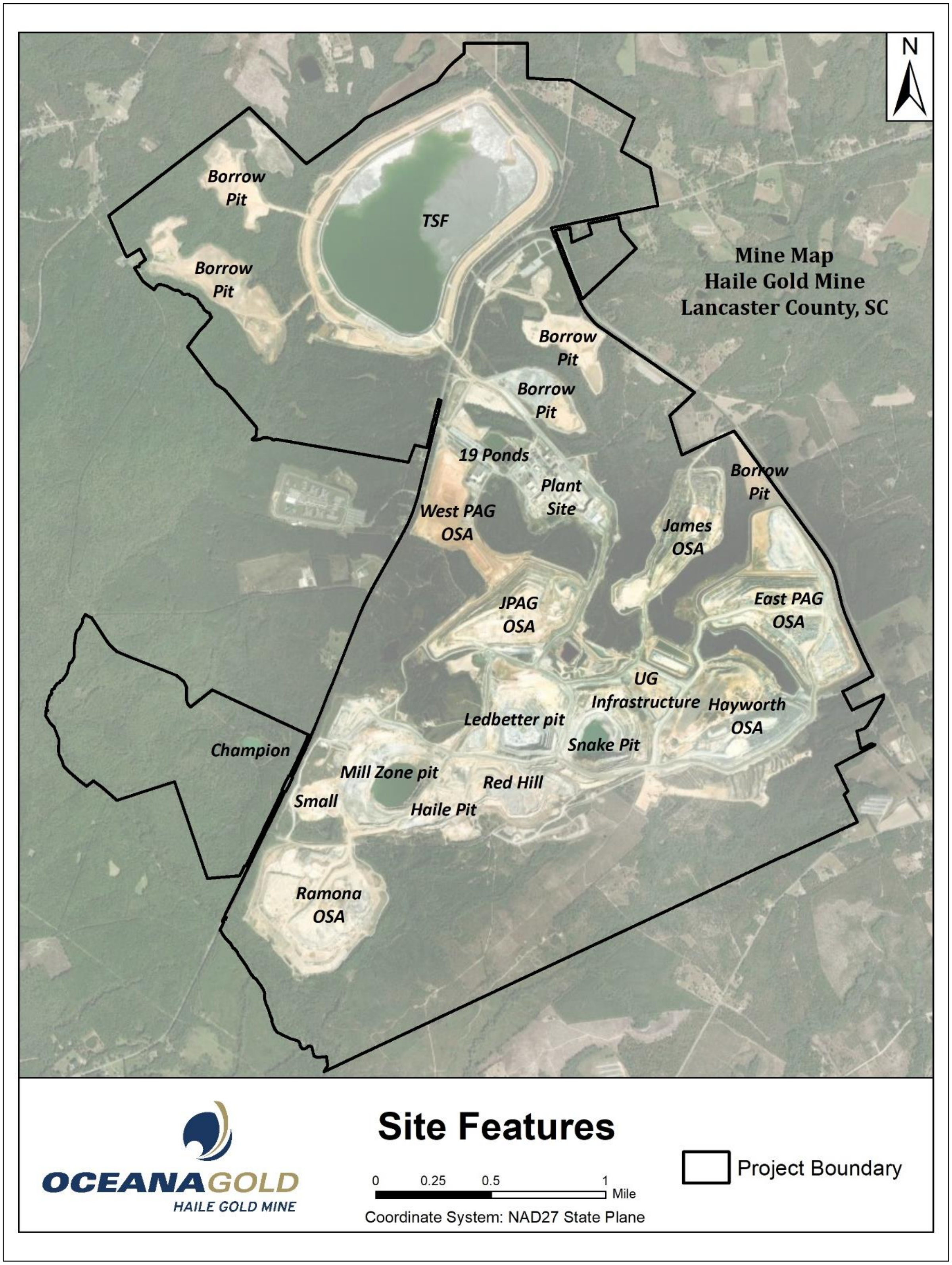
4.1 Property Location

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

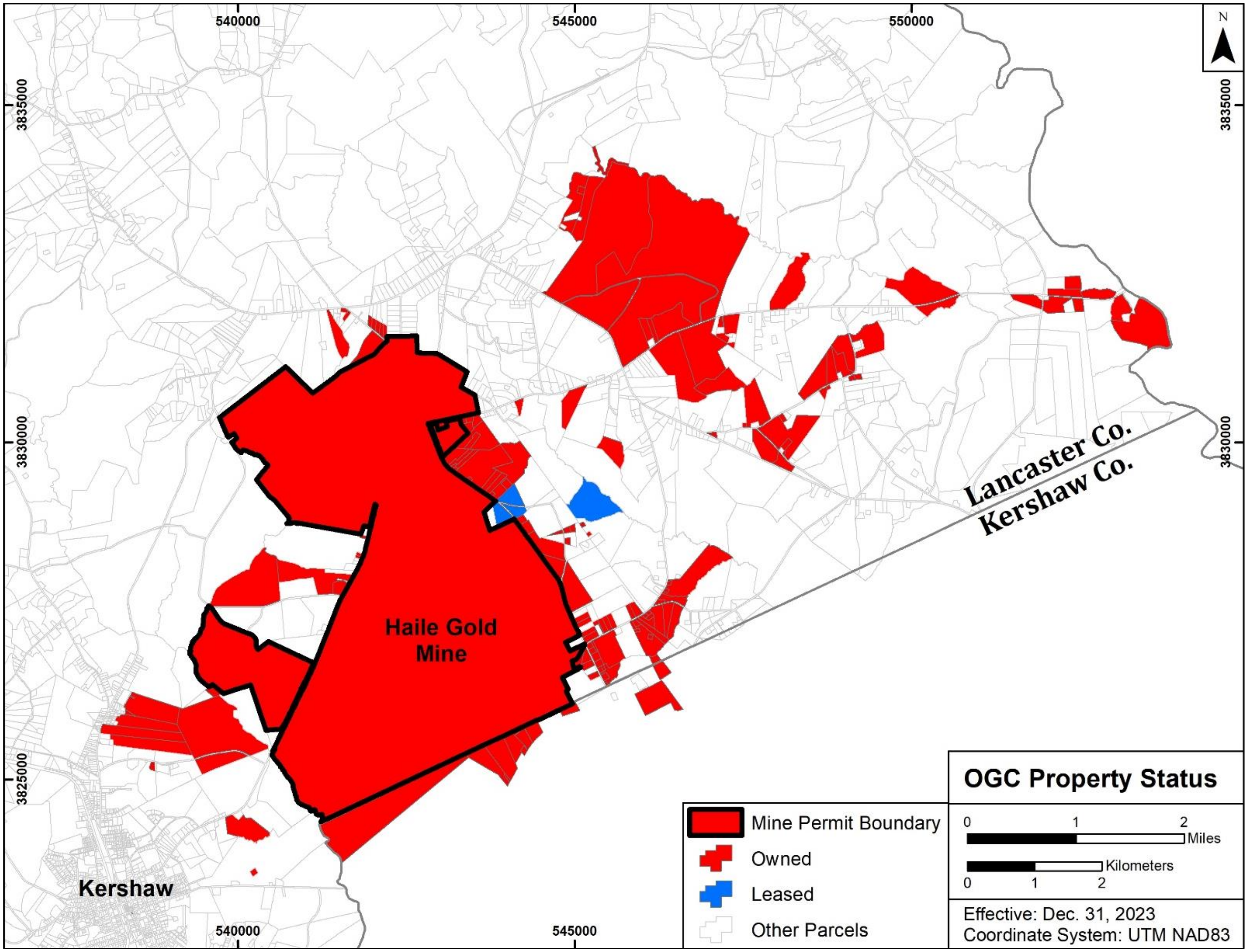


Source: OceanaGold, 2023

Figure 4-2: Site Map of the Haile Gold Mine.

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, 2023, HGM owns a total of 10,978 acres in South Carolina. Of this total, 5,469 acres are within the mine permit boundary. Figure 4-3 shows the Land Tenure map as of December 31, 2023, with Fee Simple (OceanaGold owned) and leased properties, almost entirely in Lancaster County.



Source: OceanaGold, 2024

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.

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 by Colonel Benjamin Haile, Jr. in gravels of Ledbetter Creek (now Haile Gold Mine Creek). This led to placer mining and prospecting until 1829, when lode deposits at the Haile-Bumalo pit site were found. Surface pit and underground work continued at the Haile-Bumalo site for many years. In 1837, a five-stamp mill was built (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 Mt 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

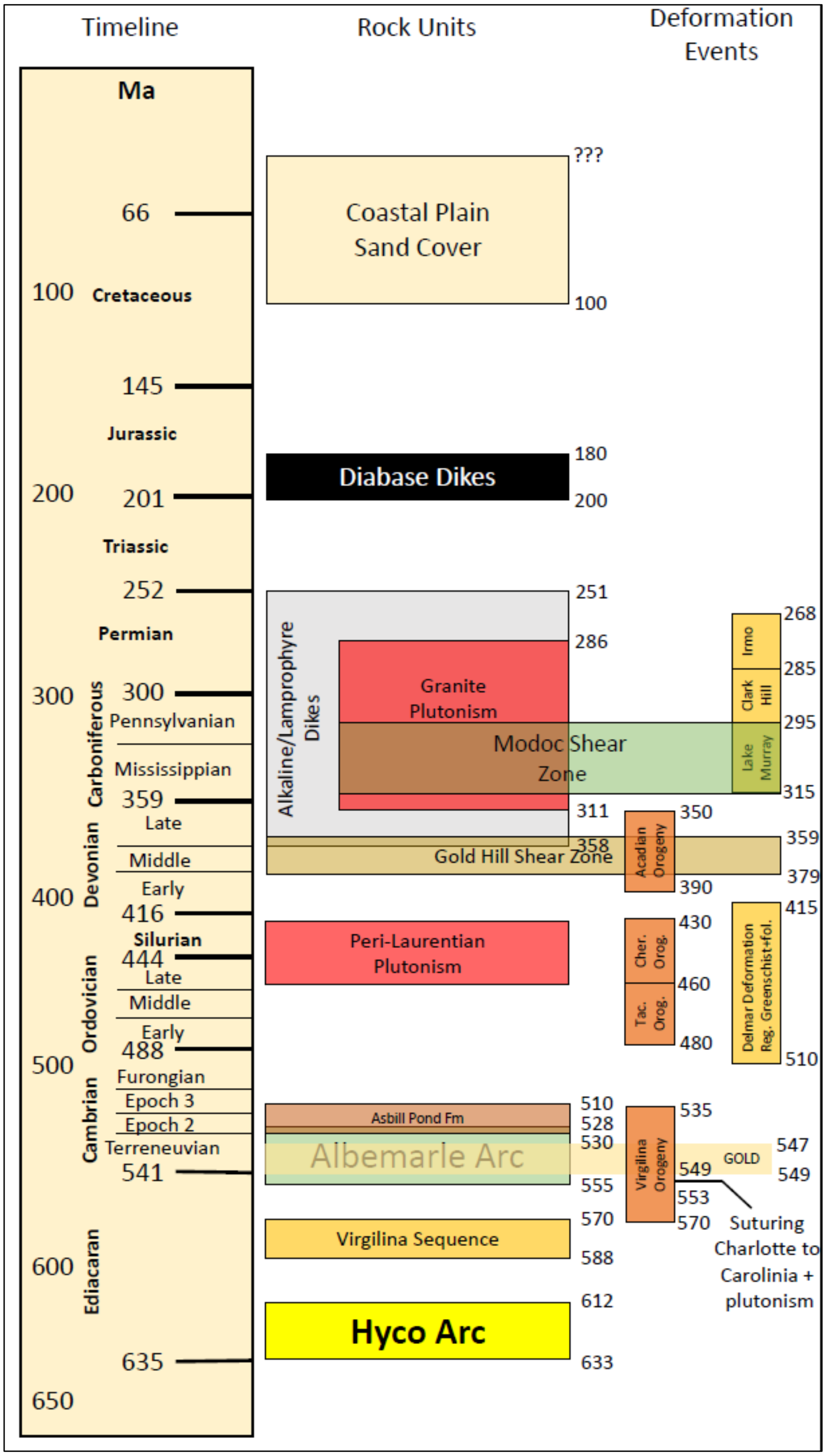
Gold endowment in the southern Appalachian piedmont is predominantly from the Carolina Slate Belt (CSB), also known as the Carolina Terrane (Hibbard et al., 2010). The 700 km long belt is characterized by a strong northeast structural grain (stratigraphy, faults, foliation, fold axes) extending from 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 intermediate to felsic volcanic flows and pyroclastic rocks mixed with fine-grained epiclastic and turbiditic sediments. At Haile, sedimentary rocks were deposited under calm, subaqueous anoxic slopes conditions as evidenced by laminated, laterally extensive siltstones and subordinate turbidite flows. 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 regional metamorphism is indicated by folded and recrystallized quartz veins and by pressure shadows with fringes of chlorite and quartz on pyrite. At least four tectonothermal periods are recorded in the Carolina Terrane (Hibbard et al., 2002), including:

- Late Neoproterozoic to Early Cambrian Virgilina events (578 to 535 Ma): folding, foliation and faulting with granite plutonism.
- Late Ordovician to Silurian Cherokee orogeny (457 to 425 Ma): greenschist facies metamorphism accompanied by a steep, generally northwest-dipping slaty cleavage that is axial planar to regional-scale folds that are commonly overturned to the southeast
- Devonian events of the Gold Hill-Silver Hill dextral shear zone (393 to 381 Ma) reactivation of Cherokee structures that juxtaposes the Carolina and Charlotte Terranes (west over east reverse motion).
- Late Paleozoic Alleghanian events (333 to 260 Ma): ductile, often mylonitic (e.g., Hyco and Modoc shears) (Hibbard et al., 1998) reactivation of older structures. Deformation is generally constrained to areas immediately proximal to such structures and focused along the southeast margin of the CSB.

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 Early Phanerozoic Cherokee orogeny records the stitching of the volcanic arcs on the eastern margin of the Laurentian continent. Northwest-trending magnetic diabase dikes intruded during the Late Triassic to Early Jurassic period coeval with the initial rifting of the supercontinent Pangea. Erosion and weathering with saprolite formation up to 40 m deep occurred in a sub-tropical, humid paleo-environment. A southeast thickening apron of clayey sands with basal quartz-rich gravels was deposited in the Cretaceous period associated with the initial opening of what is now the Atlantic Ocean. 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. 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 arc basins 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 (3.7 Moz resource plus 1.33 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, 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 predominantly sediment-hosted and lies proximal to a major volcanic-sedimentary transition. 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 characterized 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/volcanic-hosted Low Sulfidation	Persimmon Fork	quartz-pyrite-sericite	5.0 [(1)]	549
Ridgeway	Sediment / volcanic-hosted Low Sulfidation	Persimmon Fork	quartz-pyrite-sericite	1.44	553
Brewer	Breccia Pipe High Sulfidation	Persimmon Fork	pyrite-enargite-chalcopyrite	0.26	550
Barite Hill	VMS	Persimmon Fork	quartz-barite-sericite	0.06	566

Source: OceanaGold and Foley and Ayuso, 2012
(1) Remaining resources, plus 2015 to 2023 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)

- Units that will later comprise the Carolina Terrane formed as part of a subduction related oceanic island arc complex off the margin of Gondwana. This includes the Persimmon Fork Formation consisting of interbedded volcanics and sediments which is the primary host for Haile mineralization.
- Richtex Formation deposited (mudstones and siltstones) conformably on the Persimmon Fork
- Virgilina deformation results in folding, shearing, and foliation development related to the accretion of the Charlotte and Albemarle (including the Persimmon Fork Formation) island arcs.

Gold mineralization at Haile assumed at ~549 Ma (Mobley, et al., 2014) by close association with molybdenite dated using Re-Os.

Ordovician to Silurian (451 to 425 Ma)

- ENE structural fabric developed axial planar to regional NE trending, SE vergent folds and shears during NW-SE compression

Carboniferous (320 to 290 Ma)

- Emplaced granite plutons and lamprophyre dikes within 5 km of Haile

Triassic to Early Jurassic (250 to 200 Ma)

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

Cretaceous to Present (100 Ma)

- Coastal Plain clayey sands deposited over all units, thickens south-eastward

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

Igneous 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 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. 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. 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). Dacite occurs in three main areas:

- Along the southeast edge of the Haile system where dacite dips steeply to the SE and 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 where the contact dips gently to the NW

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. A quartz-phyric unit of rhyodacitic composition occurs locally at the metavolcanic-metasediment contact across site and can be highly mineralized. This unit has recently been recognized as a significant ore-host, especially in the Palomino deposit.

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.

Paleozoic Rocks

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 to moderately dipping dikes commonly strike NE-SW or 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 km x 20 km) is dated at 293 ± 15 Ma. The Pageland granite (25 km 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.

Mesozoic Rocks

Diabase Dikes

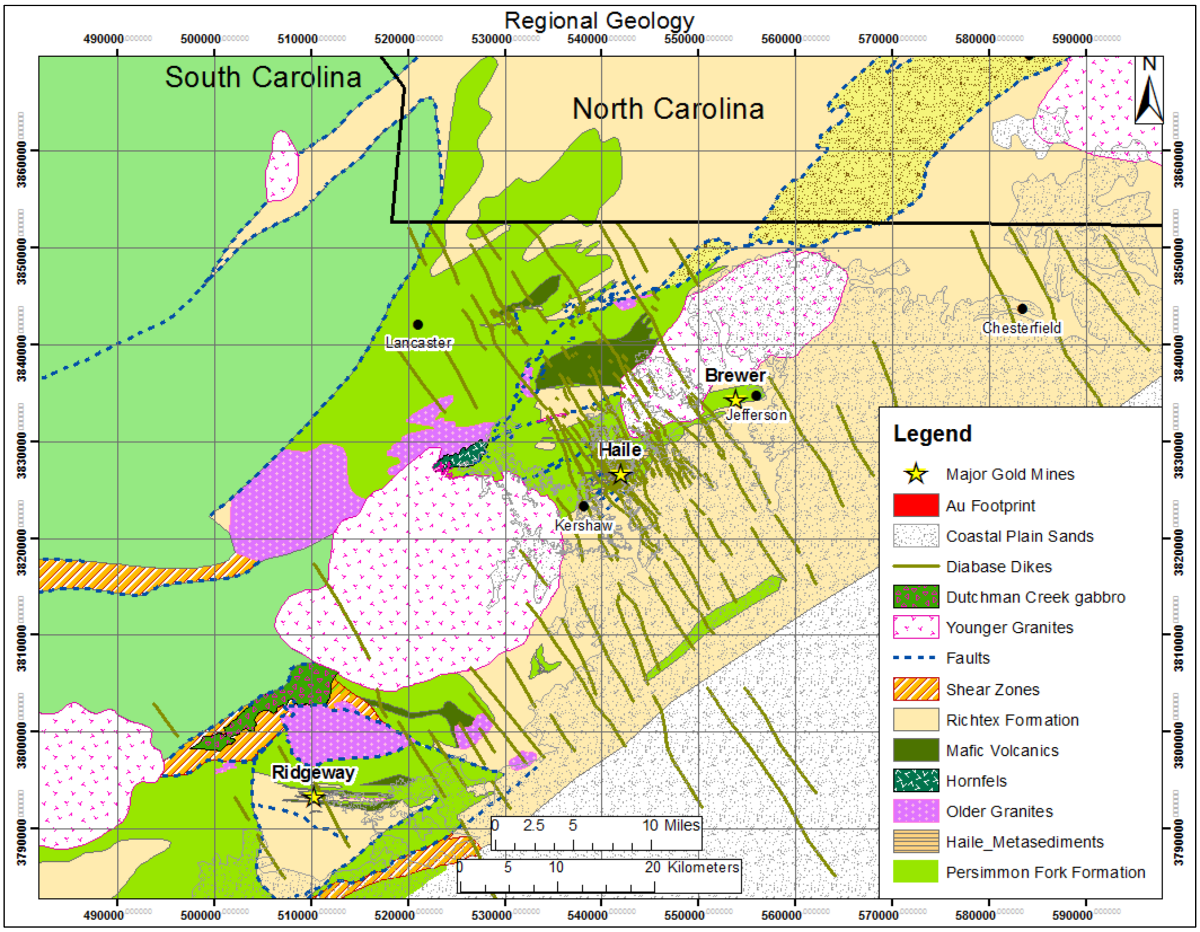
Diabase dikes are dark gray, dense, medium-grained, sub-ophitic and magnetic. These dikes conspicuously cut all other units except the Coastal Plain Sands. Dominant minerals are plagioclase and pyroxene with minor olivine. The diabase dikes margins are often chilled and/or spherulitic. The dikes strike 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 with chlorite and serpentine commonly observed along fractures. Dike emplacement postdates gold mineralization and occurred during the Late Triassic to Early Jurassic. 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 metavolcanic rocks and along faults. The base of saprolite is irregular and grades downward into partially weathered bedrock. Saprolite is rarely mineralized at Haile.

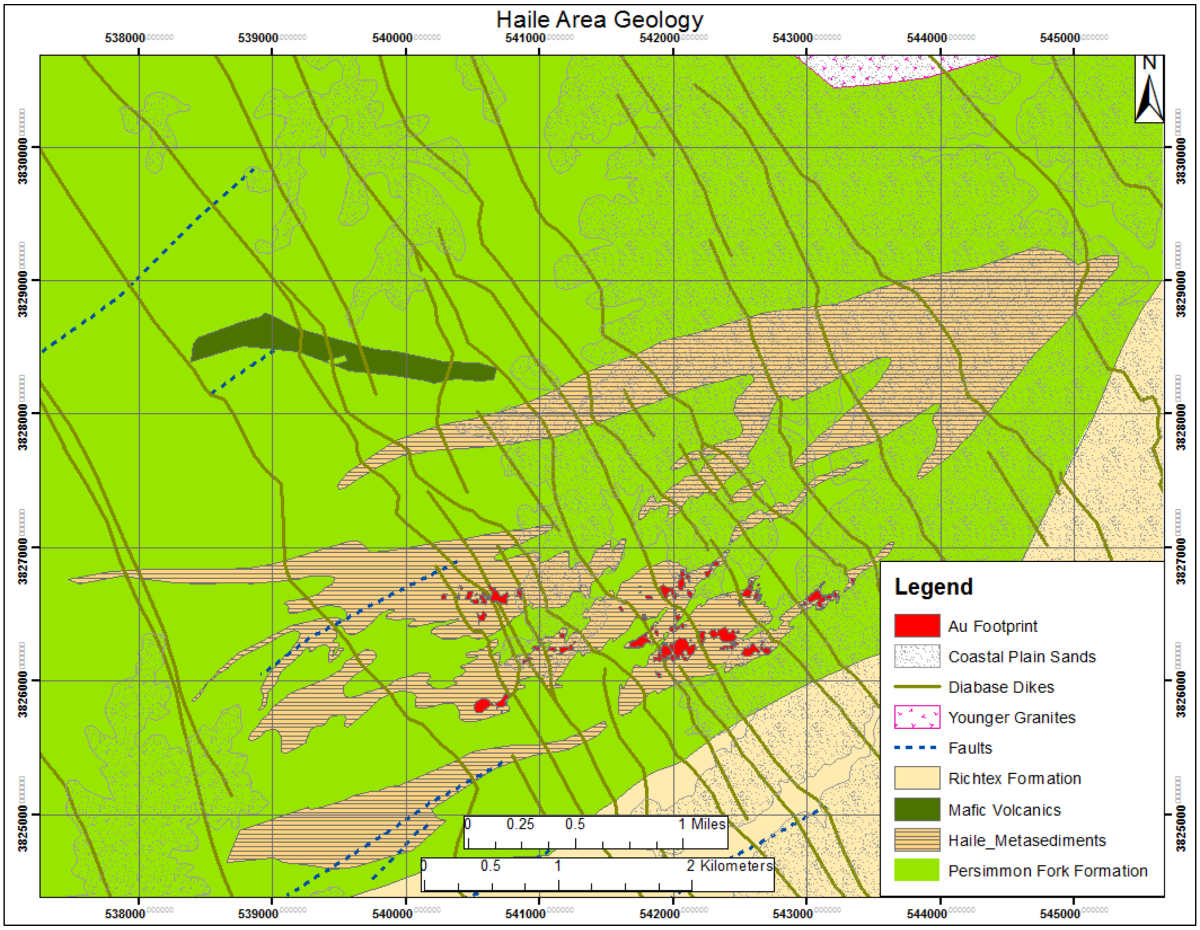
Coastal Plain Sand

The Cretaceous Middendorf Formation (Nystrom et al., 1991) is a south-eastward-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.



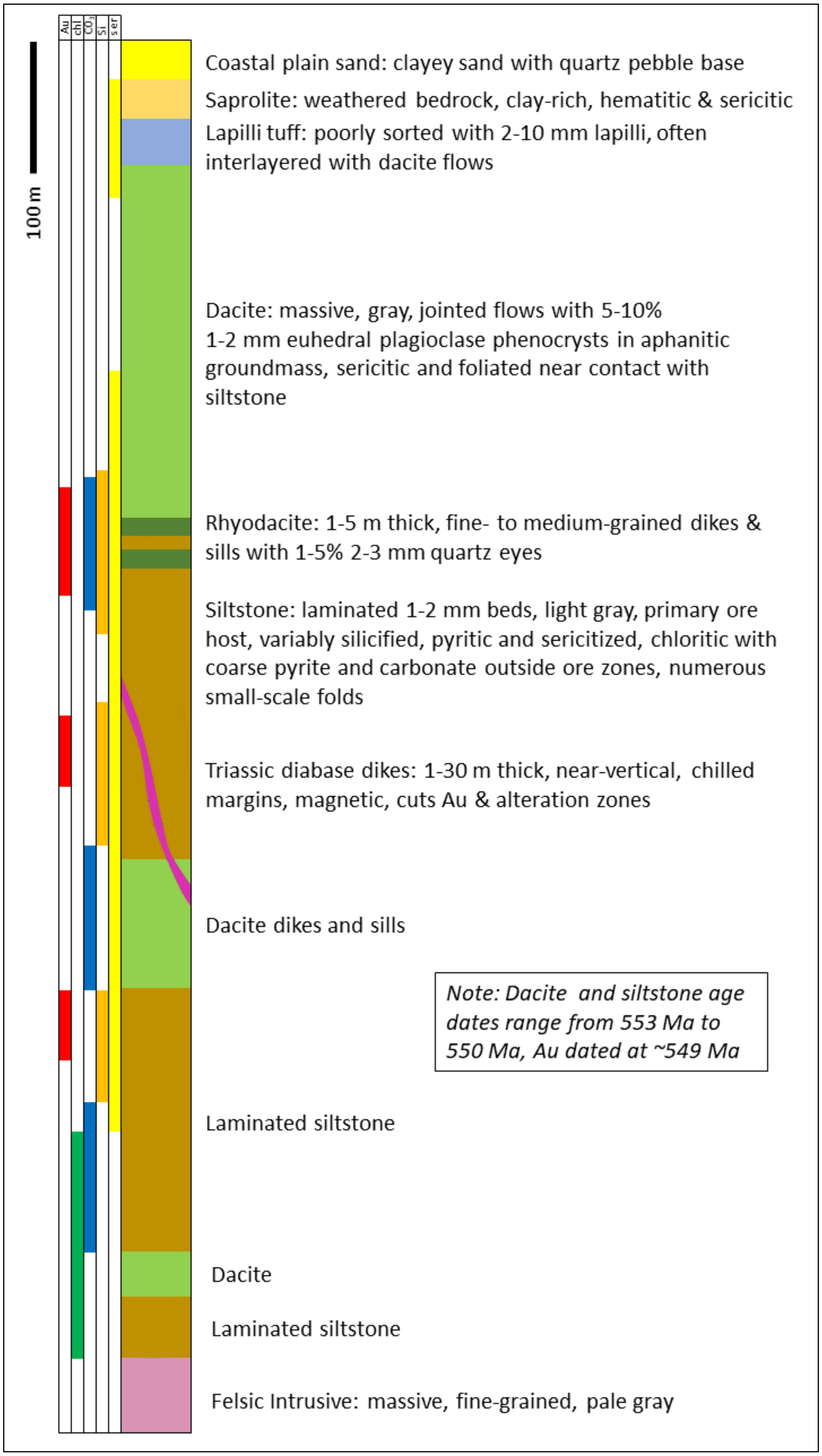
Source: OceanaGold, 2021

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



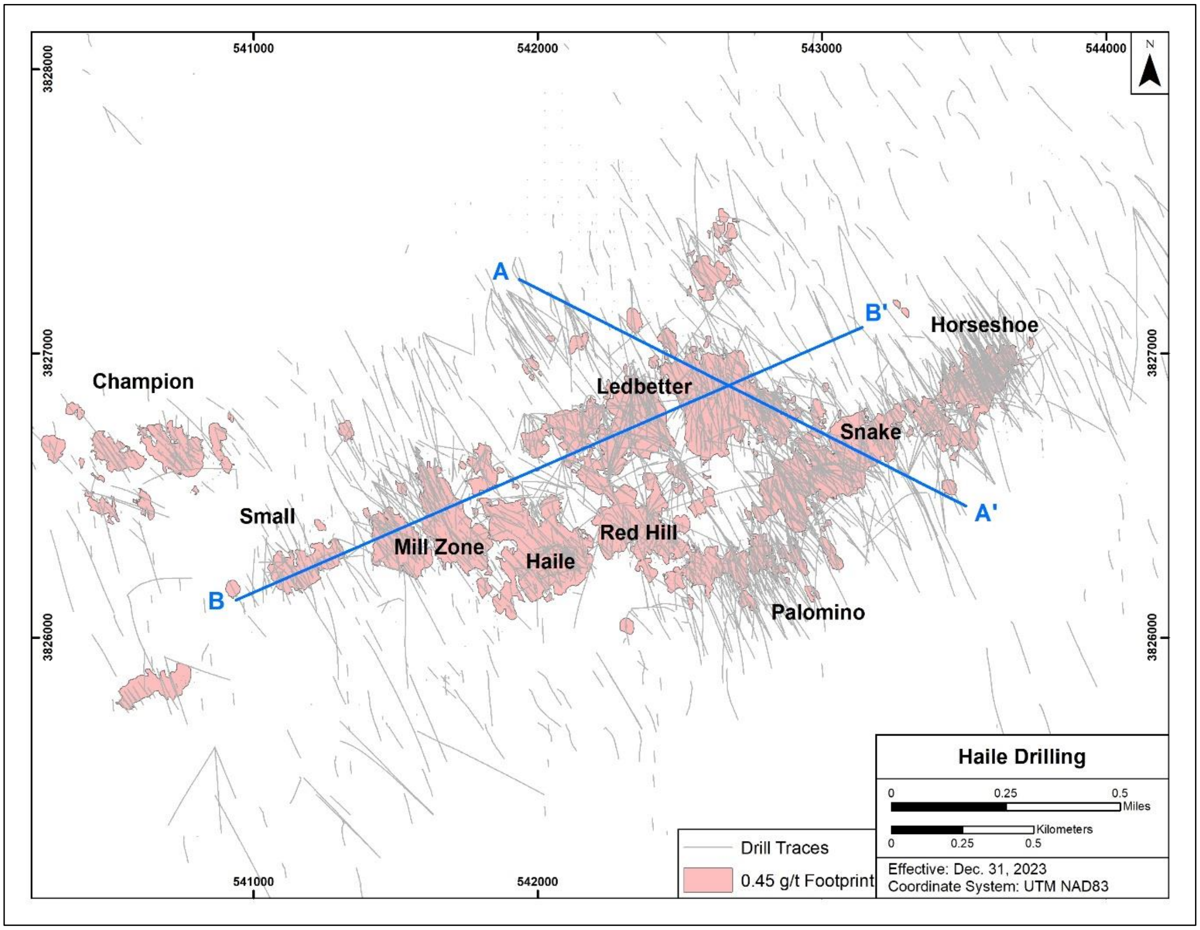
Source: OceanaGold, 2021

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



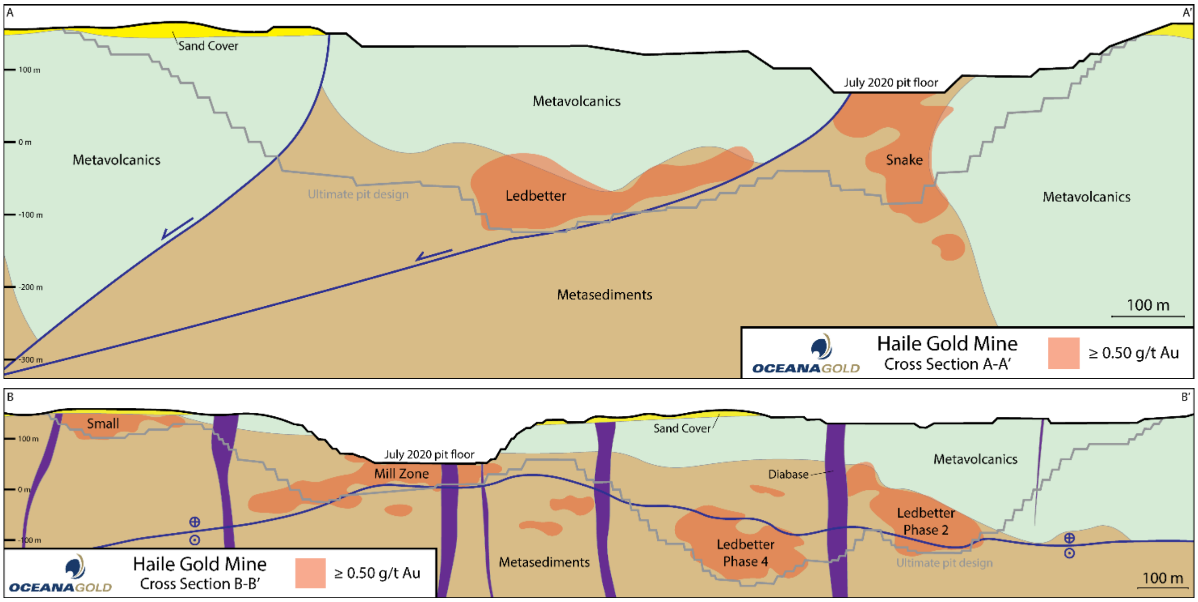
Source: OceanaGold, 2021

Figure 7-4: Haile Stratigraphic Column



Source: OceanaGold, 2023

Figure 7-5: Plan View of Drillholes used for Resource Estimation, Gold Mineralization, and Deposits (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

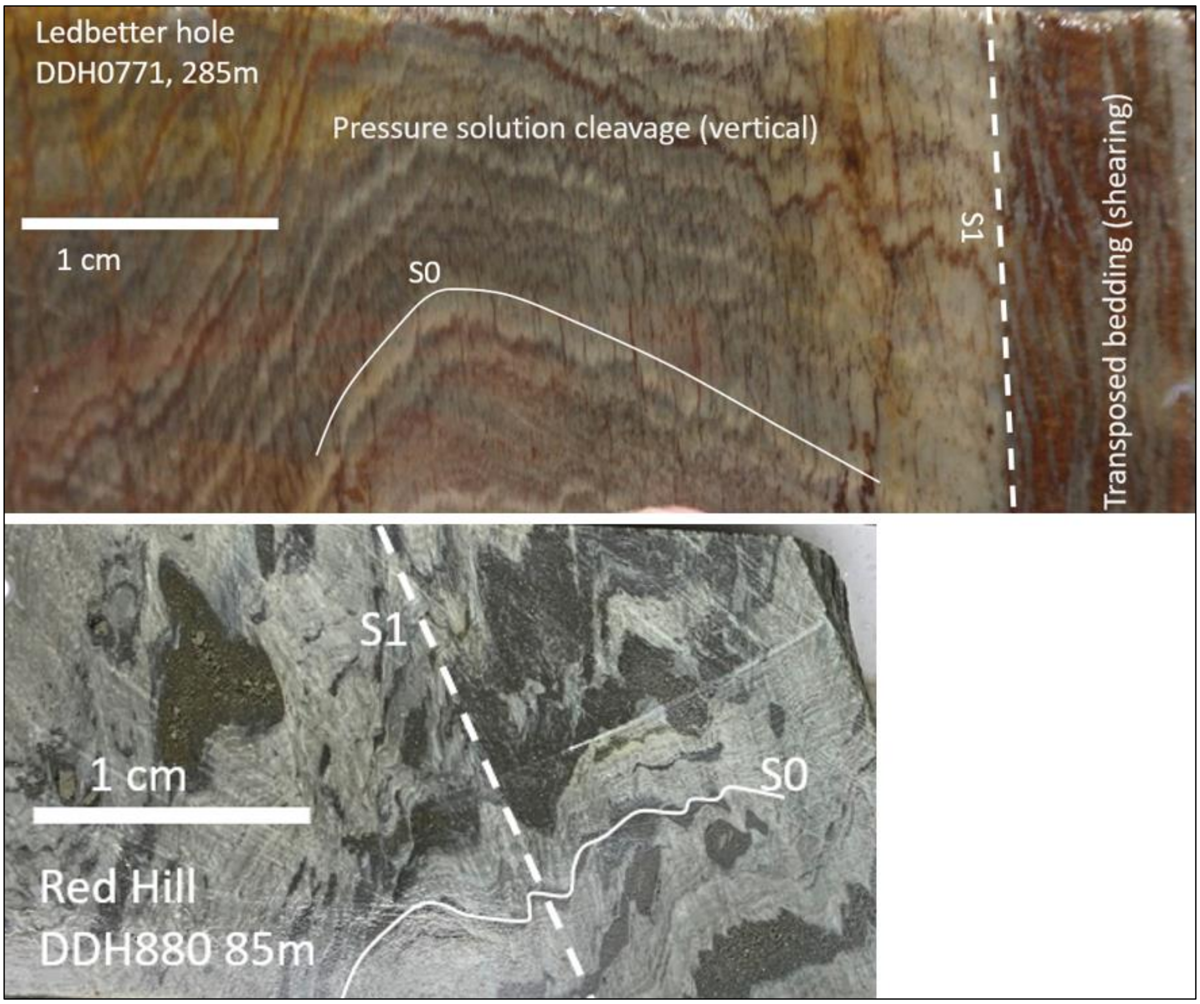
7.2.2 Structure

The structural history of Haile is complex and is affected by four deformational/hydrothermal episodes, as summarized in Section 7.1 and depicted in Figure 7-1. The timing of mineralization in relation to major orogenic events is also unclear but has traditionally been interpreted as pre-deformation. Recent structural analysis incorporating pit mapping data and high-resolution blast hole sampling suggests syn or post-deformational mineralization and investigations are ongoing. Consequently, the relative timing of the D1 and D2 events listed below is currently unclear.

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; mineralization was focused at the volcanic-sedimentary contact with the overlying volcanics acting as a cap
- **D_2 regional metamorphism and deformation** (folding, shearing, pervasive foliation, quartz veins and greenschist facies mineral assemblage); 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 the volcanic-sedimentary contact and is more strongly developed in the weaker laminated siltstones. 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 (Tosdal, 2020).



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. Primary feldspars and biotite have been converted to sericite and chlorite.

The Haile gold deposits are exposed within a metasediment (Ms) window flanked and overlain by metavolcanic rocks (Mv). 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^{0}SE 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^{0}NW 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 shears that appear to both focus gold mineralization and displace mineralized zones. The Mill Zone orebody is faulted into two segments with about 100 m of normal displacement. 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^{0}E, are 1 to 60 cm thick, and are rarely mineralized. Banded veins with crustiform textures are only observed in high-grade portions of the Ledbetter Phase 4 deposit. 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 hosted by laminated siltstone and felsic volcanics in the Upper Persimmon Fork Formation and is capped by less permeable coherent dacite flows. Mineralization is typically within 100 m of the dacite-siltstone contacts. Gold mineralization is disseminated in silicified, pyritic rocks with local K-feldspar and molybdenite. Small, mineralized zones at Ledbetter, Red Hill, Mill Zone, and Snake are hosted in the overlying dacite along fault zones within 15 m of the Mv/Ms contact. Gold grades in mineralized dacite are typically weaker than in the underlying rocks 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.

Early pervasive, replacement style sulfidation and silicification is overprinted locally by hydrothermal brecciation, quartz stockwork veining, and cm-scale quartz-pyrite veining. These secondary features generally define the high-grade zones within an ore body. Pyritized and sericitized envelopes extend beyond the silicified ore zones, are elongated parallel to foliation, and broadly define the 0.1 g/t Au shell. Pyrite grain size is typically less than 20 microns in ore zones. A late phase of barren, coarse, cubic, undeformed pyrite that formed during regional greenschist metamorphism is present outside of mineralized zones. 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 commonly occurs in 5 to 25 m thick halos around and on the edges of ore zones but is sometimes present within the deeper, underground deposits. 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 surface 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 quartz ± calcite 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 occurrences 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 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.
- 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 sulfide 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, gold system based on tectonic setting, 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 3,666 drillholes including 1,473 core holes for 466,907 m (60% m) and 2,193 reverse-circulation (RC) holes for 307,372 m (40% 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: dacitic metavolcanics, rhyodacitic metavolcanics, metasediments, basement undifferentiated rocks, and dikes. Surfaces that represent faults and shears, the base of Coastal Plain Sands, base of surface clay alteration, base of saprolite, and the base of surface oxidation 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 significantly expanded the resource and reserve of the property. 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

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. Both brownfields exploration and mine development drilling is ongoing, with particular focus on underground growth opportunities.

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
- Down-hole Induced Polarization

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).

In 2023, OceanaGold contracted Zonge International Geophysical Services and Equipment to reprocess previous surface IP/resistivity data and to perform additional downhole IP surveys. Downhole IP/resistivity was able to successfully identify known mineralization in a test hole, but five additional holes did not reveal any priority targets. Re-processed surface IP/resistivity data yielded new potential target areas.

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 2023. It contains 3,666 drillholes including 1,473 core holes for 466,907 m (60% m) and 2,193 reverse-circulation (RC) holes for 307,372 m (40% 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	DDH1213	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	RC2288	RC	Feb-20	Oct-22	OceanaGold	ALS, SGS
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
UGD0001	UGD0044	core	2023	ongoing	OceanaGold	ALS, SGS

Source: OceanaGold, 2023

10.2 Sample Collection

Both Reverse Circulation (RC) and Diamond Drilling (DDH, UGD) 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 17% 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 Dec. 31, 2023.

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 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) or 5 ft (1.5 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 or contractors.

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 percent globally). Given this mitigation, the residual risk is considered to be low.

Diamond Core: Core recoveries average 92% 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 an SPS985 Rover receiver and a TSC3 SCS900 data receiver. Surveys utilized an on-site base station with sub cm accuracy. Since October 2017, surface collar surveys have predominantly been collected by the geologist overseeing drilling using a Juniper Geode GPS Receiver with accuracy to 2 cm. Underground collars are surveyed by OceanaGold surveyors.

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. After verification of collar coordinates, surface drill sites are rehabilitated 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 surface 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 saved to a local drive and 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. Additionally, collar coordinate elevations have been adjusted by +1000 meters to avoid negative RL values.

All underground holes are surveyed using Stockholm Precision Tool's GYROLOGIC or Boart Longyear's TRUCORE. Survey data is downloaded from the tool and verified by OceanaGold personnel, stored on a local network, and uploaded to the acquire database.

10.4 Significant Results and Interpretation

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 3,666 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. 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. 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. Pre 2017, paper logs were 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. Since 2018, KML has been operated and managed by SGS.

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 either analyzed at KML or 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 30 g sample with an Atomic Absorption Spectroscopy finish
- NaCN leach test on a 30 g sample with an Atomic Absorption 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 Absorption 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.

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 Absorption 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 duplicate samples were collected or analyzed.

11.4.4 Actions and Results

QA/QC data and graphs are generated from the acQuire database. Standards returning values greater than 20% from the expected value are re-assayed for failed batches. Blanks returning values greater than 0.05 ppm are also re-assayed. 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 and coarse rejects are stored at the exploration office. RC chip trays are stored in the exploration office. Boxed core is palletized and stored in a grass lot on the east side of the mine property. Pallets are covered by tarps and aluminum tags with hole IDs attached to each pallet.

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

Historical holes drilled before 2007 comprise 17% of total drill meters and were not documented to the same standard as current OceanaGold practices. There is no evidence of material problems with the pre-2007 drilling, sampling and analyses. Furthermore, over eight years of mining has tested the veracity of the resource estimates which are based on this data. No material flaws have been identified.

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. 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

Historical holes drilled before 2007 comprise 17% of total drill meters and were not documented to the same standard as current OceanaGold practices. There is no evidence of material problems with the pre-2007 drilling, sampling and analyses. Furthermore, over eight 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 OceanaGold 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 OceanaGold 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

Twenty-two thousand six hundred eight one (22,681) standards are recorded in the OceanaGold acQuire database. There are an additional 17,722 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. 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
2022	410
2023	537
Total	**22,681**

Source: OceanaGold, 2023

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 vs. 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:

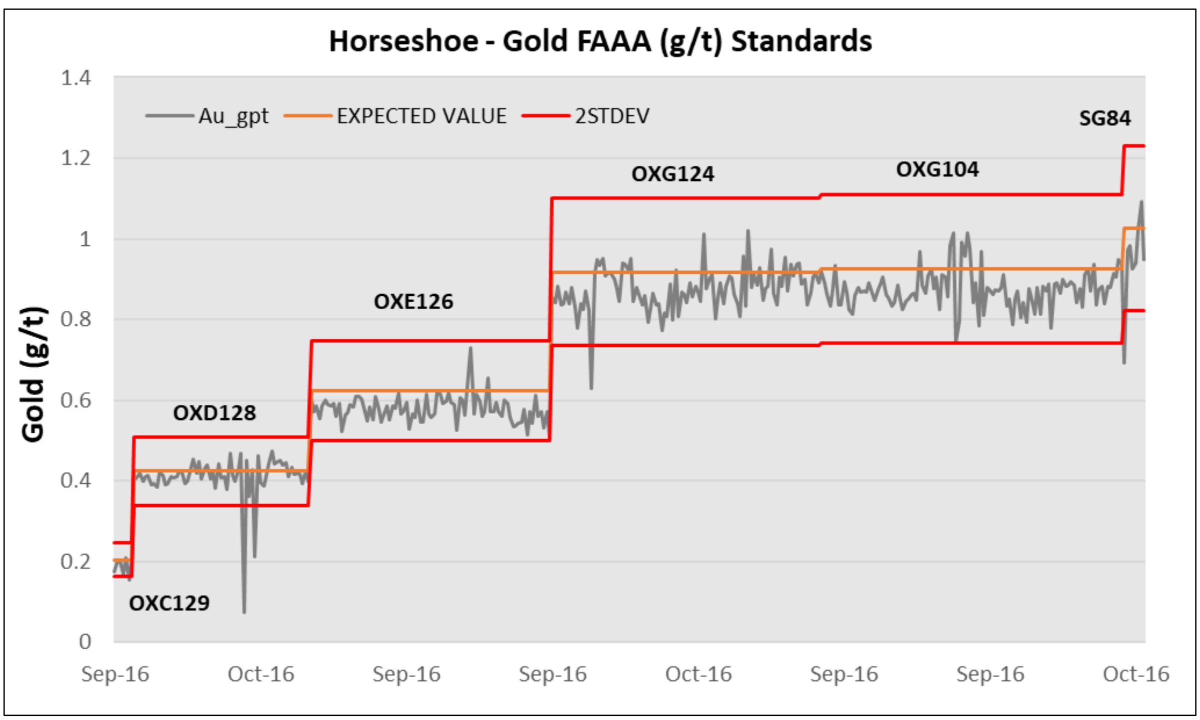
- Assay Verification – 5% check of assay values
- Collar Verification – 100% check of collar locations
- Downhole Survey Verification – 100% check on downhole surveys
- Standard and Blank QA/QC
- KML vs. ALS Horseshoe assay comparison

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

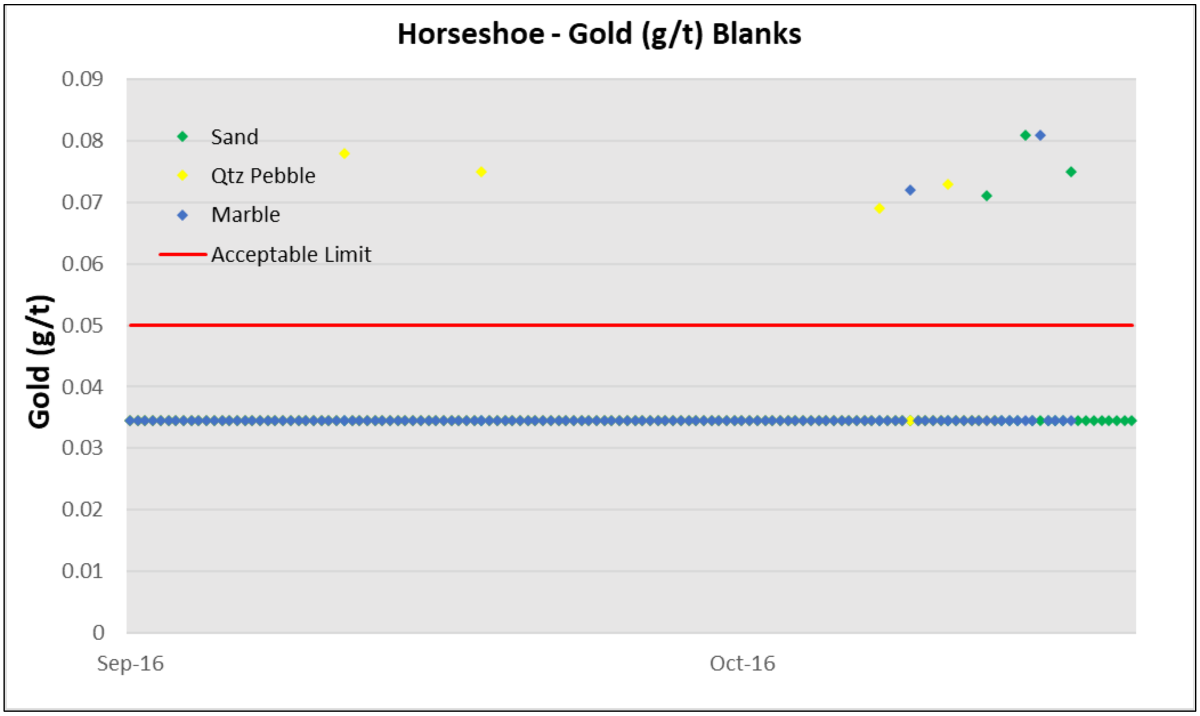
Horseshoe drillhole data were validated by OceanaGold in late 2016. Validation included assays, collar locations, and downhole surveys in the acQuire database for Romarco and OceanaGold drillholes. This includes a 5% check of assay values and a 100% check of collar coordinates and downhole surveys. An 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



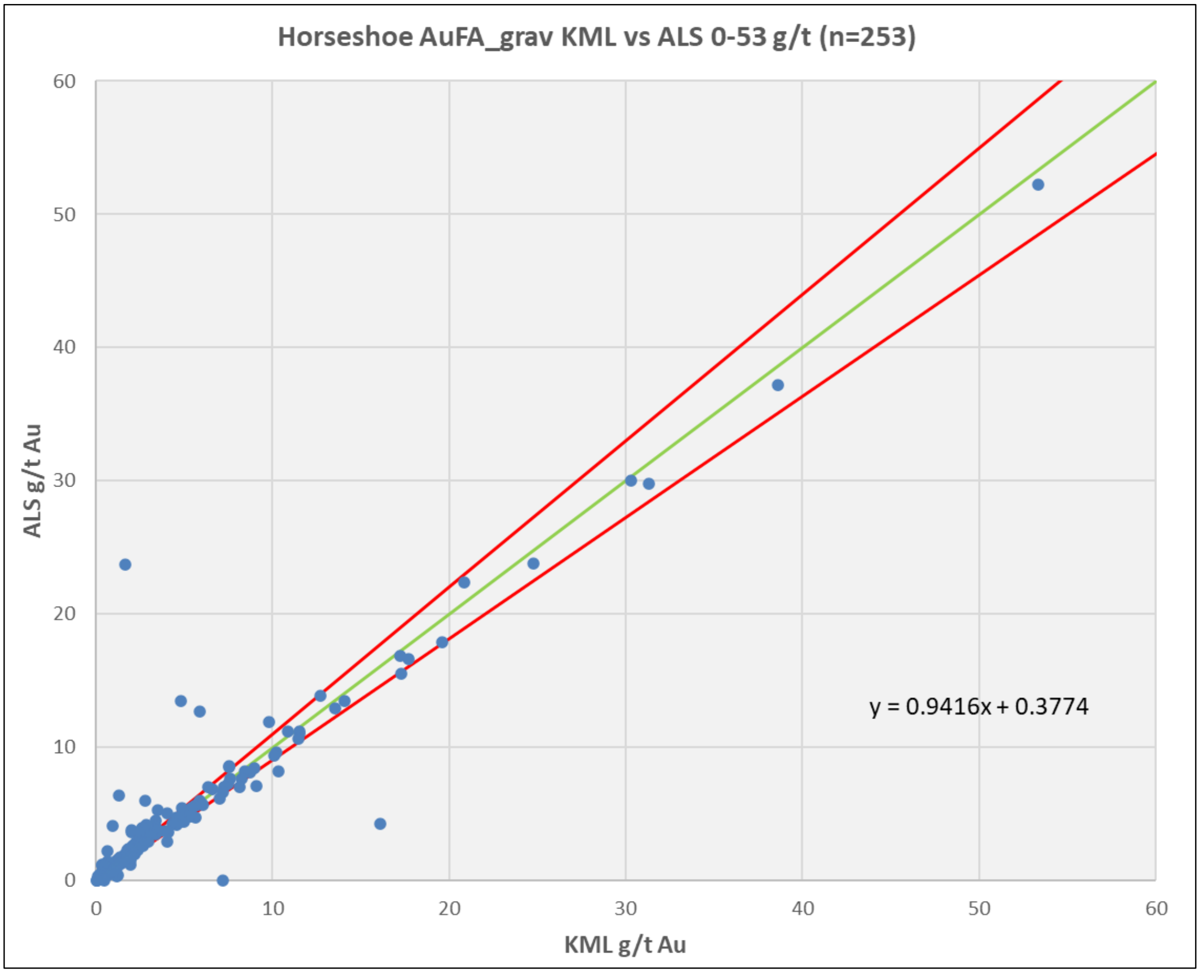
Source: OceanaGold, 2016

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 on November 20, 2016.

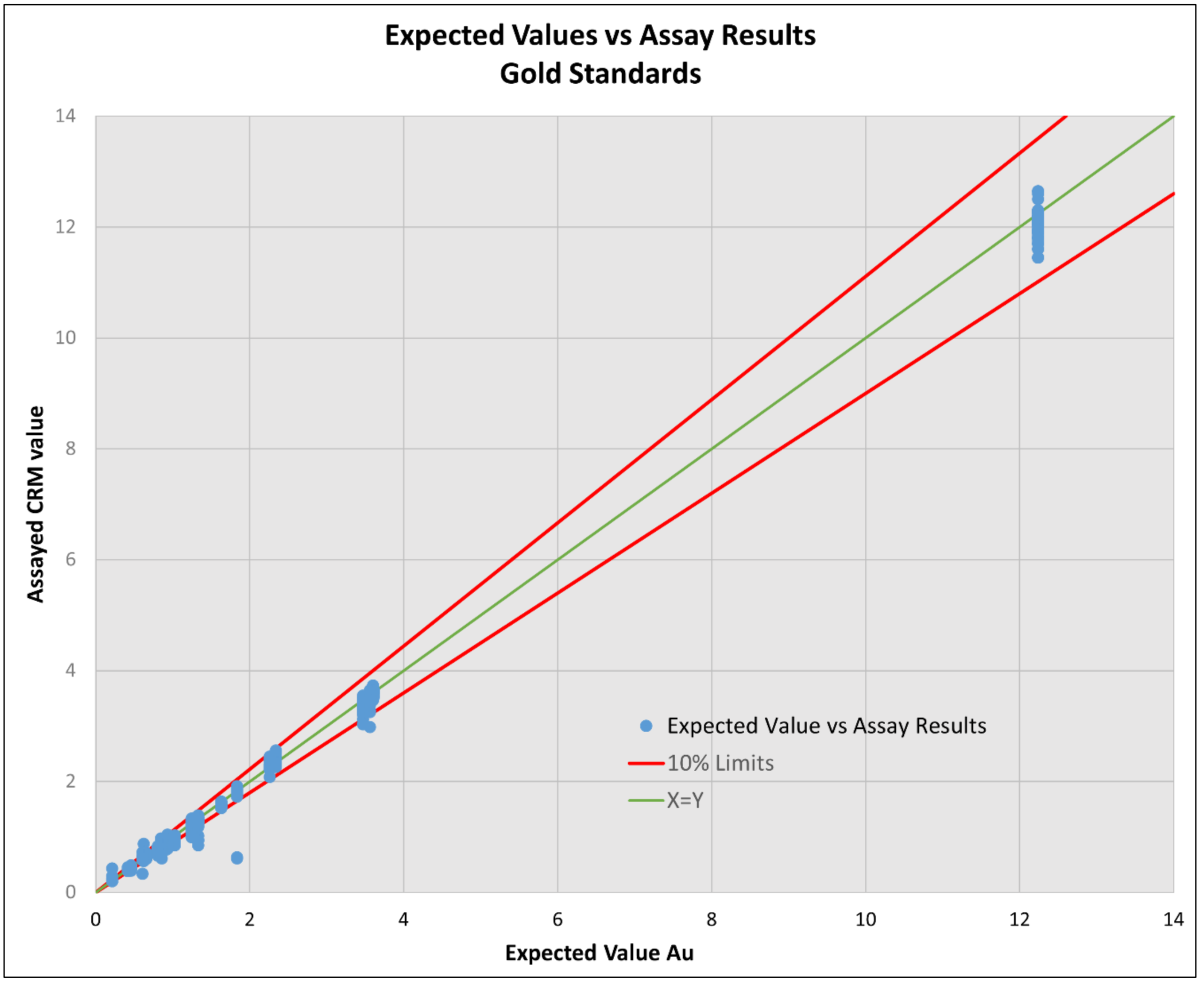
12.5 Haile QA/QC ALS July 2017-January 2022

During the period July 2017-January 2024, nearly all 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

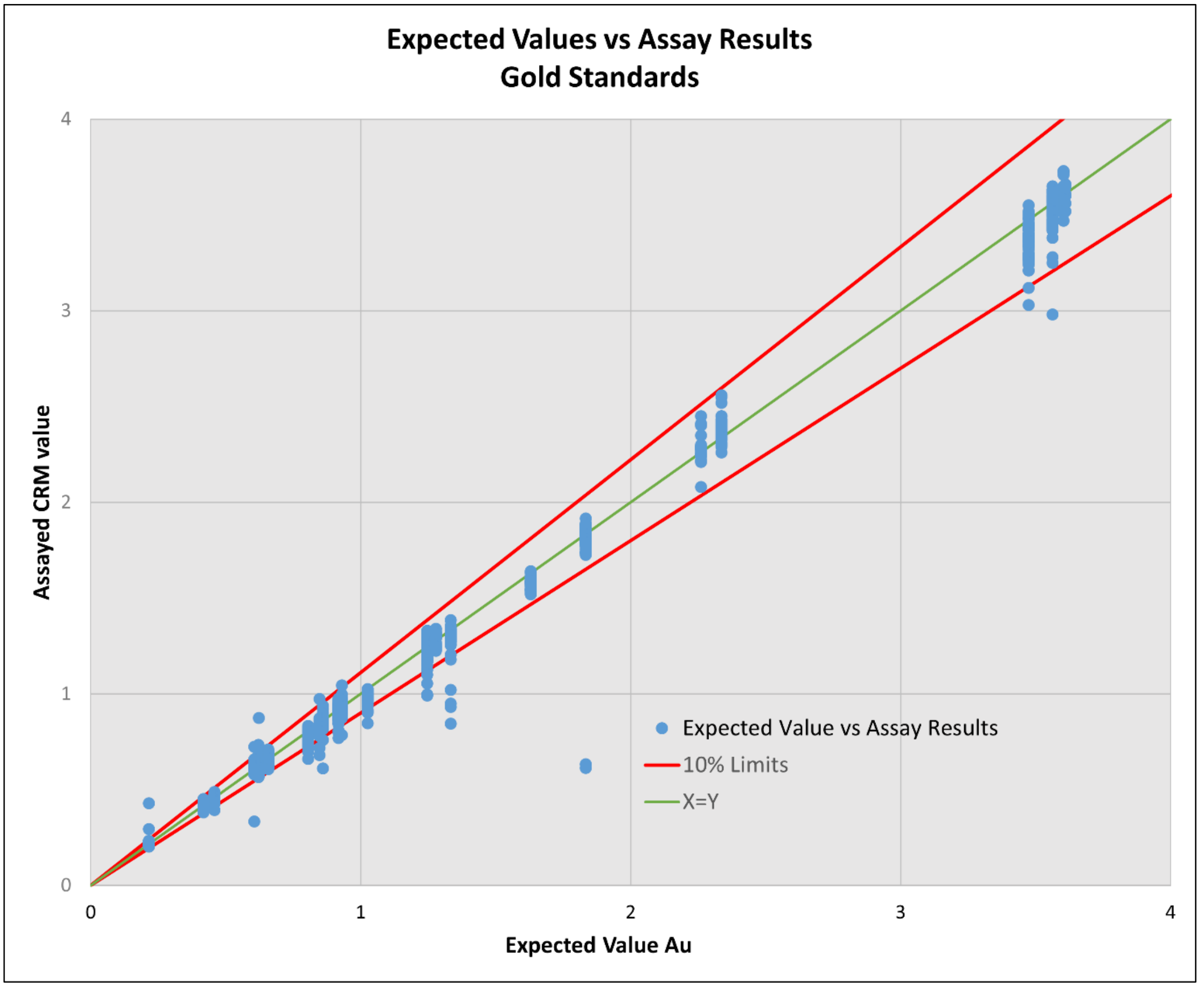
ALS Reno laboratory accuracy was monitored by insertion of commercially Certified Reference Materials (CRMs) into the sample stream. CRMs were inserted at a rate of one per twenty samples (5%) and alternated with blank insertions at one per twenty samples (5%). Hence, alternating blanks and CRMs were inserted at a rate of one in 10 (10%) per industry guidelines. A total of 27 different

CRMs sourced from Rocklabs and OREAS were submitted in sample batches for a total of 2,563 CRM analyses during the four and a half-year period. Twenty of the 27 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. Figure 12-5 shows a zoomed in view of the most commonly inserted CRMs. No obvious bias was observed within the CRM expected vs. actual data. Relative standard deviation of all CRMs was very good, averaging 2.5% of expected values over the period. 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, 2023

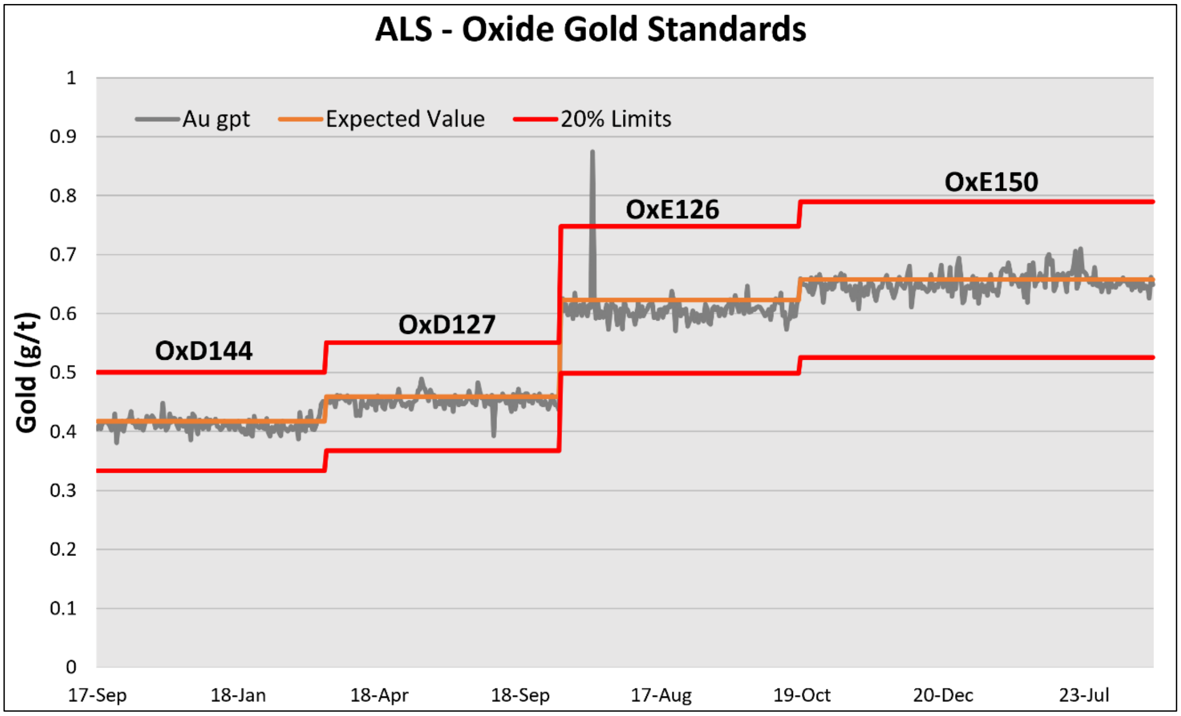
Figure 12-4: July 2017-Dec. 31, 2023 CRM Analyses vs. Expected Value (n=2,563)



Source: OceanaGold, 2023

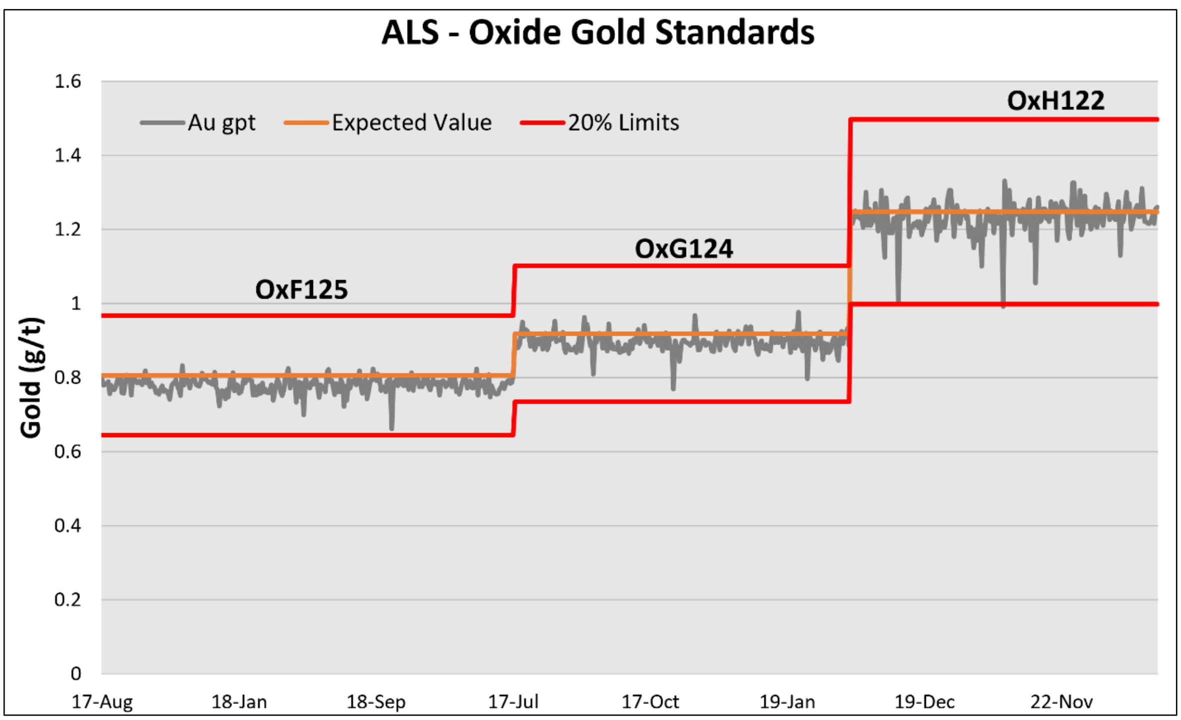
Figure 12-5: July 2017-Dec. 31, 2023 CRM Analyses vs. Expected Value <4 g/t (n=2,511)

Standard control charts were plotted for commonly used CRMs. Examples of oxide (OxD144, OxD127, OxE126, OxE150, OxF125, OxG124, and OXH122), sulfide (SG84), and high silica (HiSilK2) CRMs are shown 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, and no sample swap was evident, all intervals within the failed batch and the nearest passing CRM or Blank were rerun.



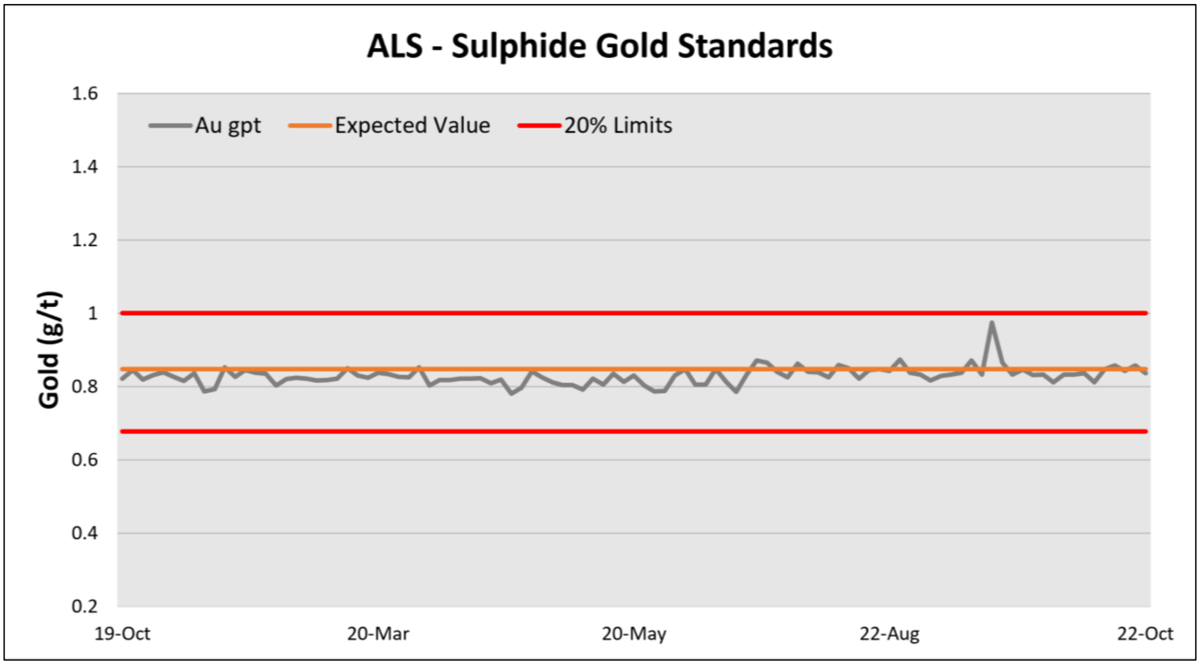
Source: OceanaGold, 2023

Figure 12-6: July 2017-Dec. 31, 2023, Control Chart, OxD144 (n=130), OxD127 (n=134), OxE126 (n=137), and OxE150 (n=202)



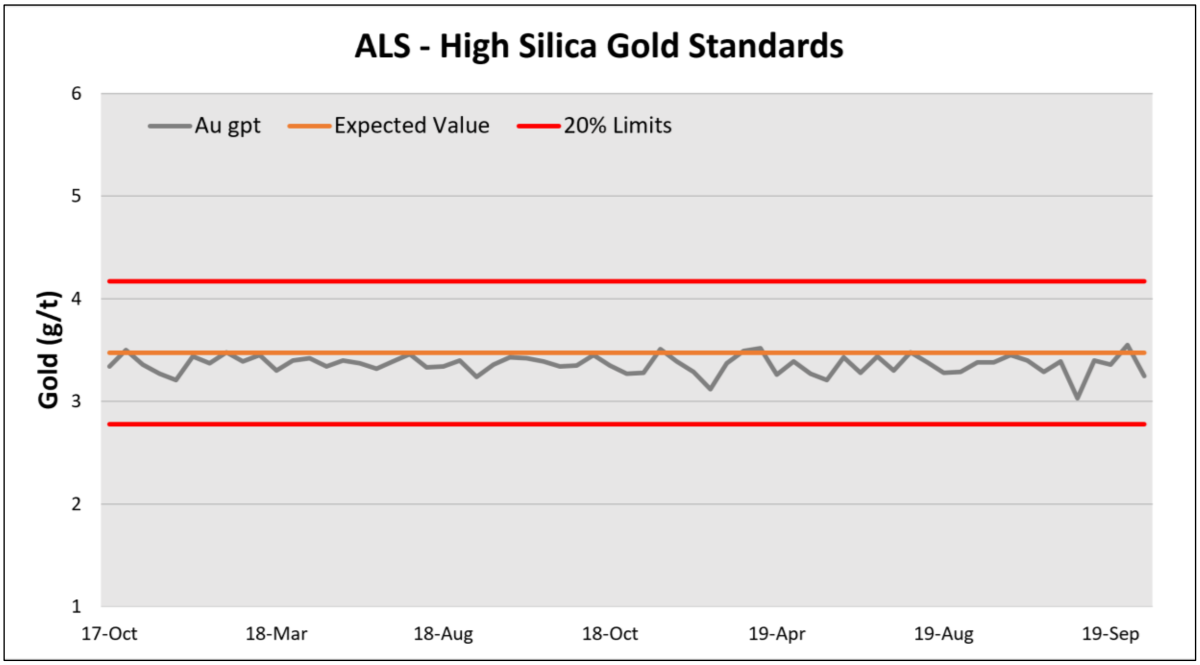
Source: OceanaGold, 2023

Figure 12-7: July 2017-Dec. 31, 2023, OxF125 (n=268), OxG124 (n=218), and OxH122 (n=193)



Source: OceanaGold, 2023

Figure 12-8: July 2017-Dec. 31, 2023, Control Chart, SG84 (n=42)

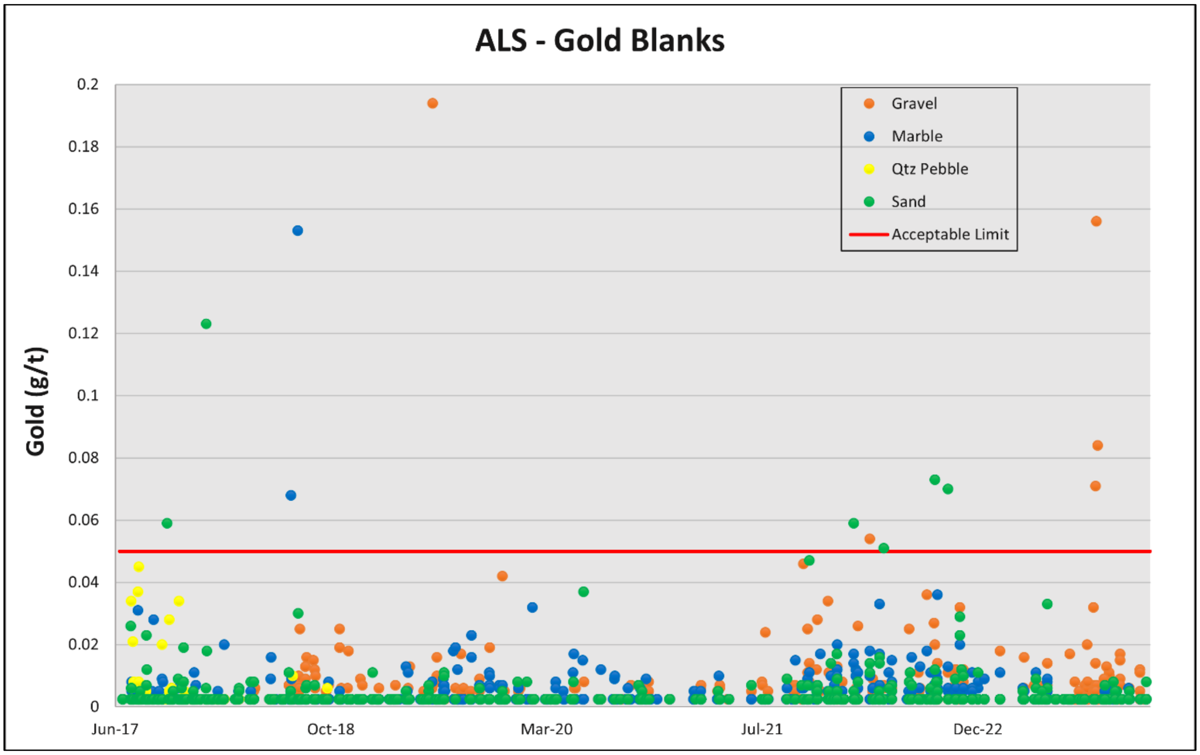


Source: OceanaGold, 2023

Figure 12-9: July 2017-Dec. 31, 2023, Control Chart, HiSilK2 (n=63)

12.5.2 Contamination Monitoring

Contamination is monitored by insertion of blank materials. From July 2017-January 2024 a total of 2,493 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. 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 significant contamination during sample preparation.



Source: OceanaGold, 2023

Figure 12-10: July 2017-Dec. 31, 2023 Blank Insertions (n=2,493)

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 at 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 at 100-mesh (kWh/st)	BM Wi at 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 at 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 at 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

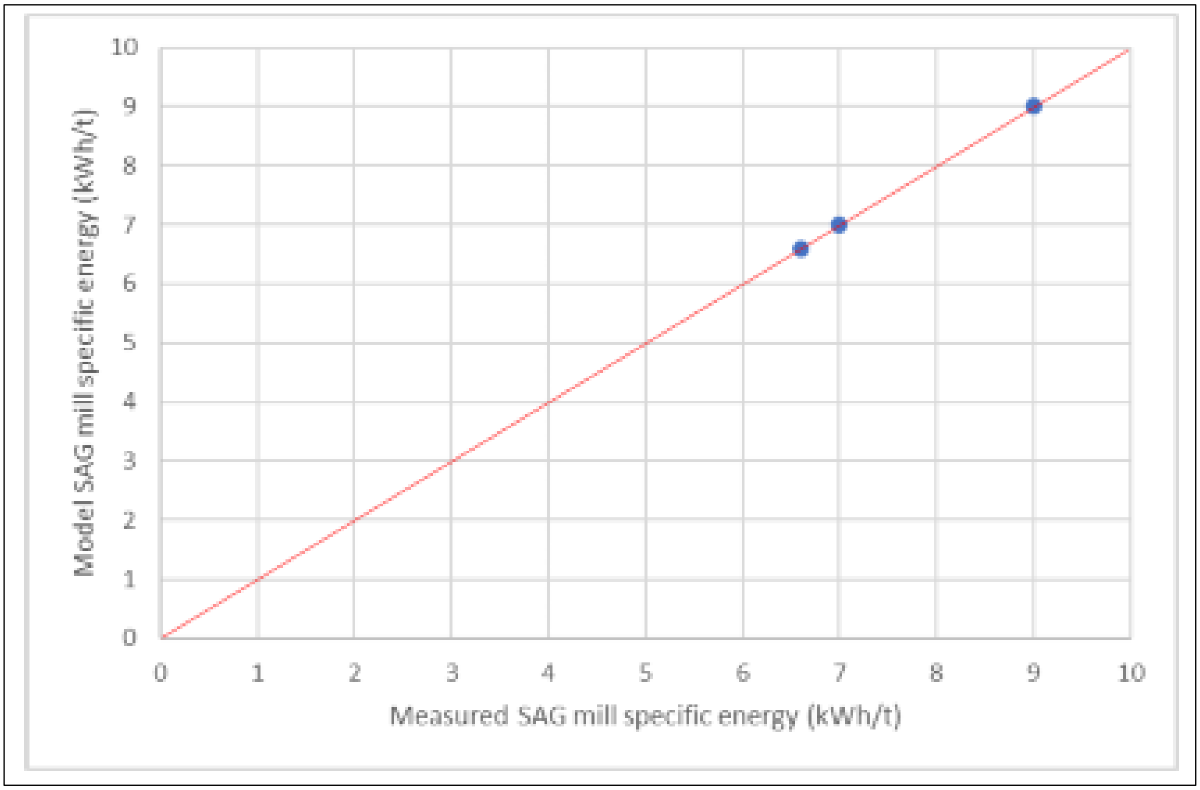
ALS performed JK Drop Weight and Bond ball mill index tests on samples from mineralized material exposed in Mill Zone Pit. The results of this work are presented in Table 13-6.

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

Composite Number	A x b	SCSE (kWh/st)	BM Wi at 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 at 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

In 2022 SGS Burnaby was sent 30 core composite samples from ore zones from the Haile, Ledbetter and Millzone pits as part of a competency variability program to quantify variability expectations in the LOM plan from these pit stages. SMC, Bond rod mill and Bond ball mill index tests were conducted on these samples providing a larger data set confirming the increased hardness expected in the Ledbetter pit compared to the upper areas of the Haile pit and Mill Zone pits. The results of the program are summarized in Table 13-8 and has increased the data set of competency data to allow development of hardness proxy measurements for the block model.

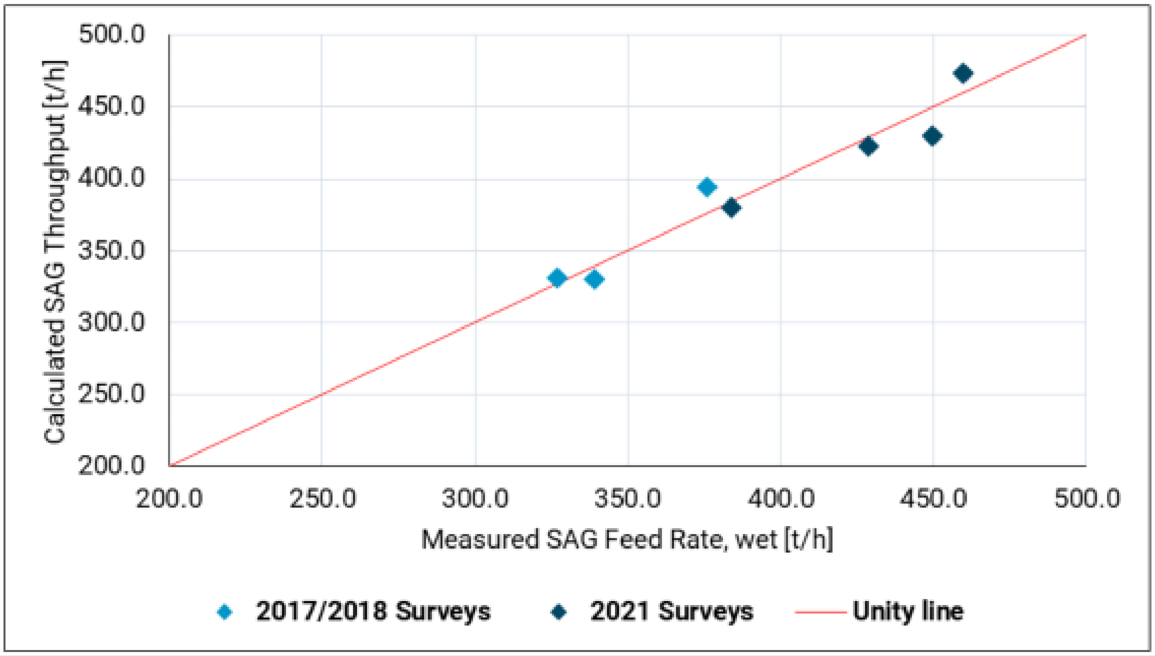
Table 13-8 SGS Comminution Test Results for 2022 Variability Program

Sample ID (DDH/Deposit)	A x b	SCSE (kWh/tonne)	BM Wi at 200-Mesh (kWh/tonne)
MET-MZ-88	128.5	6.31	4.7
MET-MZ-89	37.1	10.47	8.9
MET-MZ-90	59.0	8.34	4.9
MET-MZ-91	94.8	6.97	5.3
MET-HL-92	49.8	8.99	9.1
MET-HL-93	39.7	9.92	9.7
MET-HL-94	58.5	8.34	7.5
MET-HL-95	45.1	9.33	8.3
MET-HL-96	37.0	10.23	8.7
MET-HL-97	40.0	9.91	8.6
MET-HL-98	39.2	10.20	8.7
MET-HL-99	44.8	9.45	9.2
MET-HL-100	40.8	9.92	9.4
MET-LB-101	30.3	11.28	11.9
MET-LB-102	44.3	9.45	9.7
MET-LB-103	30.0	11.49	11.1
MET-LB-104	39.9	9.96	10.0
MET-LB-105	34.6	10.61	11.4
MET-LB-106	38.0	10.33	9.3
MET-LB-107	31.8	10.98	10.2
MET-MZ-108	32.9	10.98	11.5
MET-LB-109	41.8	9.79	10.8
MET-LB-110	37.8	10.26	9.4
MET-LB-111	30.8	11.77	9.6
MET-LB-112	47.9	9.22	10.3
MET-LB-113	37.0	10.26	8.9
MET-MZ-114	46.8	9.40	6.9
MET-MZ-115	48.4	9.37	9.0
MET-MZ-116	38.3	10.17	7.9
MET-MZ-117	53.7	8.80	8.7

Source: OceanaGold, 2024

In 2022 as part of a program of blast optimization trials to evaluate the impact on mill throughput a series of full grinding surveys were undertaken to validate the previously developed site specific comminution model to allow conversion of competency parameters into throughput rates for the current circuit. 4 additional surveys were completed and provided to Ausenco to balance and model fit with the additional points displayed below in Figure 13-2 with a good correlation between the measured and predicted SAG specific energy.

$$SAG\ E_{CS}\left(\frac{kWh}{t}\right) = 2.2427 - 0.0297 * DWI + 0.4975 * BBWi$$ **(2022 Regression Model)**



Source: OceanaGold, 2024

Figure 13-2: Updated Modeled vs. Measured SAG Specific Energy Values

The updated model has been used to validate the throughput estimates used for the mill feed types in the LOM planning process.

13.1.2 Throughput Estimate Assumptions

Following the treatment of deeper sourced ore from the Snake Phase 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 phase, 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 LoM. 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.

In 2023 several grinding trials were conducted feeding 100% Horseshoe development ore to validate the throughput expectations of this material.

With the introduction of Horseshoe ore from 2023 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:

- Underground 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
- 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 13-9.

Table 13-9: 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 13-10 and Table 13-11.

Table 13-10: 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 13-11: 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 13-12.

Table 13-12: 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 13-13.

Table 13-13: 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 13-14.

Table 13-14: 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 13-15.

Table 13-15: 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 13-16.

Table 13-16: 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 13-17.

Table 13-17: 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 13-18.

Table 13-18: 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.

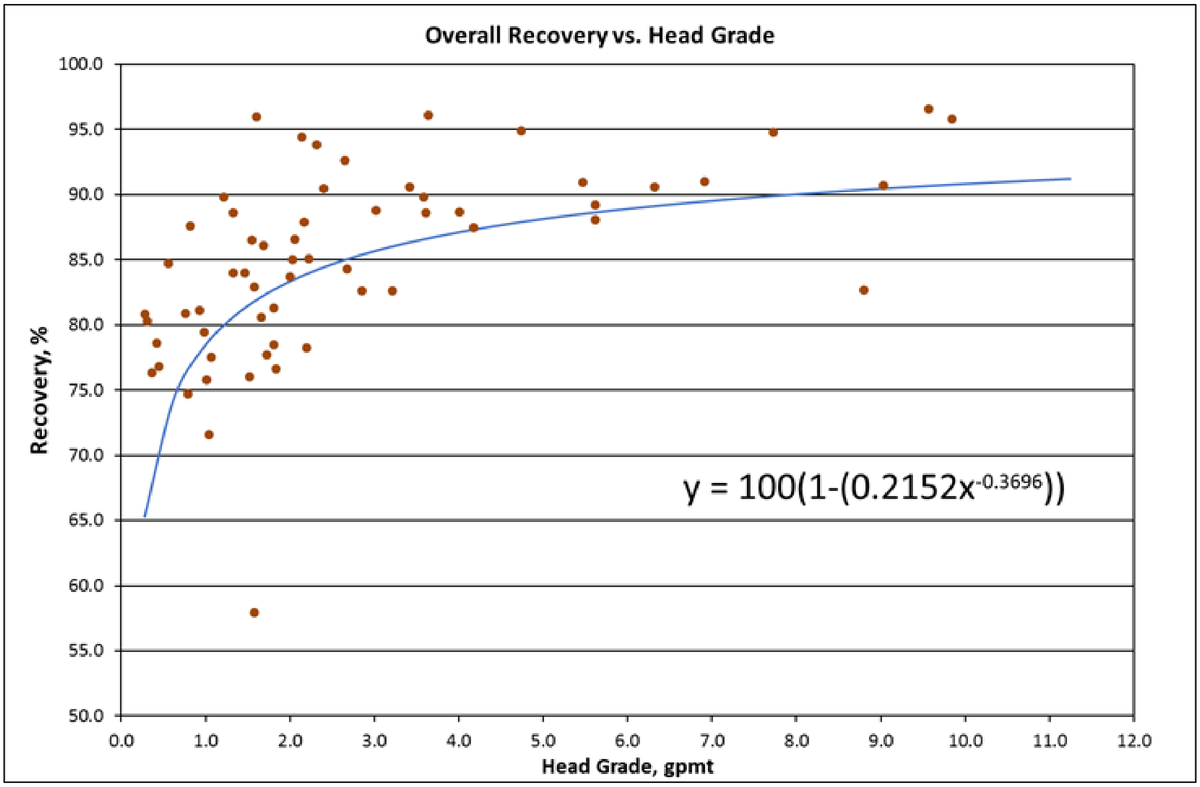
The test work indicated that the circuit should include regrinding of the flotation concentrate before slurry pre-aeration, and a leach time of 24 hours. A regrind circuit product size of 80% passing 15 microns is an appropriate target for regrind circuit design. Operational experience may allow this 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 13-3. 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 13-3.



Source: OceanaGold, 2022

Figure 13-3: 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 Palomino Deposit

The Palomino deposit at Haile is expected to be similar in nature to the other deposits at Haile that have been encountered to date. Gold is present both in silicates and as fine inclusions within pyrite requiring a fine grind to achieve economic liberation. Within the Palomino deposit two key lithologies have been identified as carrying the majority of the gold values, classified as Rhyodacite and Siltstone, the latter the equivalent of the metasediment domain previously used to describe the highly silicified mineralization hoisting much of the gold mineralization. Metavolcanics occur but are a much more minor abundance.

Based on the geological and lithological model a bingo chart was constructed for the deposit based on the total Measured, Indicated and Inferred resource (MI&I) available at the conclusion of the scoping study encompassing the largest expected mineral inventory and allowing for any further conversion drilling success to be accommodated. The bingo chart considered nine bins based on the three core lithologies and the following three grade bins:

- 1.4 to 1.7 g/t Au to cover development ore below cut-off grade
- 1.7 to 3 g/t Au for medium grade ore
- >3 g/t Au for high grade ore

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 13-3 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.

From the bingo chart the distribution of composite samples targeted for testing was reviewed and discussed with the exploration team to locate mineralized intercepts targeting a single lithology and grade bin. The distribution of composites required is shown below in Figure 13-4 with a total of 22 composites required. Intercepts of individual drillholes were then identified from the infill drilling program to match the lithology and target grade bin across the deposit.

Section #12. Samples/Tests NEEDED - (Difference Between Starting # in Section #9 and Balanced Representivity in Section #3)							
		Difference between Prior Stage and Balanced Next Stage Table (GO COLLECT THESE!)					
		GeoMet Ore classification					
Ore Class	*>g/t Au*	***LG Au (1.3-1.7)***	***MG Au (1.7-3)***	***HG Au (>3)***	***0***	***0***	***Total Needed***
Rhyodacite	**3.000**	1	5	4	0	0	10
Siltstone	**3.000**	2	6	3	0	0	11
MV	**3.000**	0	1	0	0	0	1
0	**3.000**	0	0	0	0	0	0
0	**3.000**	0	0	0	0	0	0
0	**3.000**	0	0	0	0	0	0
NEW Samples/Tests Needed		3	12	7	0	0	22

Source: OceanaGold, 2024

Figure 13-4: Palomino Metallurgical Composite Plan

From the 22 composites 6 of these were selected to be from quarter core to allow for competency and ore hardness characterization with the remaining composites sourced from coarse crushed rejects from the initial exploration core submitted for assay to preserve core for future testing. The samples were shipped to SGS Burnaby to conduct the test program designed to characterize the hardness of future Palomino ores and to estimate overall gold and silver recoveries by replicating the existing process flowsheet in the laboratory.

The program consisted of two phases. Ore competency testing was carried out on six composites (three Rhyodacite and three Siltstone) sourced from intact quarter core incorporating SMC Testing for competency and Bond ball mill work index and Abrasion index for a measure of hardness.

The second phase on all 21 composites involved:

- Stage crushing and homogenization to -10 mesh followed by head assay analysis
- Batch flotation testing to produce a bulk concentrate on standard flotation conditions provided by OceanaGold based on prior programs to assess flotation response for sulfur and gold recovery
- Fine grinding of the flotation concentrates to a P80 of 13 microns
- Cyanide leach tests on both flotation tailings and reground flotation concentrate streams to assess gold recovery in the leach circuit
- Mass balance of results to estimate overall gold and silver recovery for each composite

The comminution testing results are summarized in Table 13-19 and are comparable to that observed in the Haile and Millzone pits rather than the harder deep deposits in Horseshoe and Ledbetter Stage 3 and 4 and should not present an issue with the current grinding circuit.

Table 13-19 Palomino Competency Testing Results

Sample ID (DDH/Deposit)	Domain	A x b	t_a	BM Wi at 200-Mesh (kWh/tonne)
MET_PAL_119	Rhyodacite	51.9	49	9.3
MET_PAL_124	Rhyodacite	42.6	38	11.2
MET_PAL_127	Rhyodacite	42.7	39	10.1
MET_PAL_137	Siltstone	46.4	43	10.1
MET_PAL_138	Siltstone	47.2	44	9.7
MAT_PAL_139	Siltstone	40.6	39	11.3

Source: OceanaGold, 2024

All samples were subjected to a suite of chemical analysis as per Figure 13-5. At 11.1 g/t, the gold grade of MET_PAL_123 was much higher than the other samples. The gold grades of the remainder ranged from 1.39 to 6.80 g/t, with an average gold grade of 3.16 g/t. silver grades varied from 1.0 to 14.0 g/t, with an average of 5.0 g/t.

The total sulfur grade ranged from 0.27% to 13.1%, with an average of 4.67%; sulfur was present mainly as sulfide (S2-) with a low sulfate (SO4) content. This indicates low sample oxidation which gives confidence that laboratory test results will mirror plant performance.

All other analytes were at low levels and not considered to be a risk for processing.

Sample ID	Au g/t	Ag g/t	Cu %	Fe %	As %	Hg g/t	S %	S-S2-* %	S-SO4* %	C %	TIC** %	TOC** %
MET_PAL_119	5.90	1.0	< 0.01	1.84	0.011	< 0.3	1.16	1.06	<0.1	0.18	0.19	< 0.05
MET_PAL_120	1.83	4.0	< 0.01	11.3	0.016	< 0.3	11.4	11.30	0.07	0.13	0.13	< 0.05
MET_PAL_121	2.66	5.0	< 0.01	3.70	0.017	< 0.3	3.31	3.19	0.07	0.07	0.03	< 0.05
MET_PAL_122	5.36	11.0	0.01	11.7	0.015	< 0.3	10.7	10.40	0.13	0.08	0.08	< 0.05
MET_PAL_123	11.1	14.0	< 0.01	5.19	0.014	< 0.3	3.21	3.09	0.07	0.14	0.13	< 0.05
MET_PAL_124	2.16	3.2	< 0.01	6.50	0.012	< 0.3	6.20	5.84	0.03	0.12	0.11	< 0.05
MET_PAL_125	2.21	3.5	< 0.01	3.76	0.013	< 0.3	3.45	3.25	0.07	0.07	0.06	< 0.05
MET_PAL_126	3.00	6.0	< 0.01	12.3	0.016	< 0.3	13.1	12.90	0.17	0.06	0.03	< 0.05
MET_PAL_127	2.86	3.0	< 0.01	5.84	0.008	< 0.3	4.89	4.36	<0.1	0.38	0.39	< 0.05
MET_PAL_128	4.71	3.0	< 0.01	7.41	0.021	< 0.3	6.11	5.87	0.07	0.07	0.05	< 0.05
MET_PAL_129	5.27	3.0	< 0.01	7.21	0.020	< 0.3	1.51	1.47	< 0.1	0.07	0.05	< 0.05
MET_PAL_130	1.39	4.5	< 0.01	2.62	0.006	< 0.3	1.57	1.53	<0.1	0.11	0.11	< 0.05
MET_PAL_131	2.06	4.0	< 0.01	4.38	0.018	< 0.3	3.92	3.92	0.03	0.16	0.15	< 0.05
MET_PAL_132	2.31	5.0	< 0.01	9.05	0.028	< 0.3	9.99	9.75	0.13	0.07	0.06	< 0.05
MET_PAL_133	1.65	3.0	< 0.01	6.71	0.006	< 0.3	5.52	5.34	0.03	0.23	0.22	< 0.05
MET_PAL_134	2.68	<2	< 0.01	1.84	< 0.001	< 0.3	0.27	0.26	<0.1	0.10	0.10	< 0.05
MET_PAL_135	2.88	12.5	< 0.01	4.25	0.017	< 0.3	3.71	3.46	<0.1	0.15	0.14	< 0.05
MET_PAL_136	1.65	4.0	< 0.01	3.42	0.012	< 0.3	3.24	3.08	0.03	0.22	0.22	< 0.05
MET_PAL_137	2.22	1.5	< 0.01	3.07	0.004	< 0.3	1.15	1.11	<0.1	0.46	0.46	< 0.05
MET_PAL_138	6.80	4.3	< 0.01	3.21	0.012	< 0.3	2.42	2.07	0.03	0.21	0.20	< 0.05
MET_PAL_139	3.63	3.7	< 0.01	2.23	0.005	< 0.3	1.14	1.16	<0.1	0.21	0.20	< 0.05

**S-S2-: sulphide sulphur; S-SO4: sulphate sulphur*
*** TIC: total inorganic carbon; TOC: total organic carbon*

Source: OceanaGold, 2024

Figure 13-5: Geochemical Analysis of Palomino Composites

All of the composites responded well to flotation under the standard conditions for the bulk rougher test with 82.8 to 96.9% of the gold present reporting to the flotation concentrate. Mass pull was in line with the expectations from the given sulfur grade, with the expected strategy of Palomino representing 20% of mill feed it is not expected to be an issue for the regrind circuit capacity. The mass pull, grade and recovery results are presented below in Table 13-20.

Table 13-20: Palomino Flotation Test Results

Composite	Mass Pull %	Grade g/t Au	%S	Au Recovery
MET_PAL_120	31.3	5.03	33	96
MET_PAL_121	20.1	11.8	13.9	93.7
MET_PAL_122	28.9	16	32.2	93
MET_PAL_123	15.0	54.9	15.2	89
MET_PAL_125	16.8	11.3	16.7	93.8
MET_PAL_126	35	8.74	38.6	96.7
MET_PAL_128	18	19	29	82.7
MET_PAL_129	9.6	43.3	14.6	98.4
MET_PAL_130	9.6	14	15.8	91.1
MET_PAL_131	13.8	12.6	28.6	96.9
MET_PAL_132	30.4	6.96	29.9	95.6
MET_PAL_133	21.1	7.47	26.7	94.3
MET_PAL_134	6.9	32.1	3.23	93.2
MET_PAL_135	14.7	16.6	20.3	91.7
MET_PAL_136	13.8	11	21.8	94.9
MET_PAL_119	9.8	40.6	8.04	82.8
MET_PAL_124	17.5	17.8	24.2	95.8
MET_PAL_127	16	17.9	30	93.2
MET_PAL_137	11.4	20.9	10.8	92.2
MET_PAL_138	14.1	30.0	15.2	89.3
MET_PAL_139	9.9	28.2	10.7	87.8

Source: OceanaGold, 2024

The overall gold extractions from the flotation products (concentrates and tailings) corresponding to gold extractions by cyanidation from the head samples varied from 69.9% to 94.7%, with an average value of 84.7%. Full results are summarized in Table 13-21.

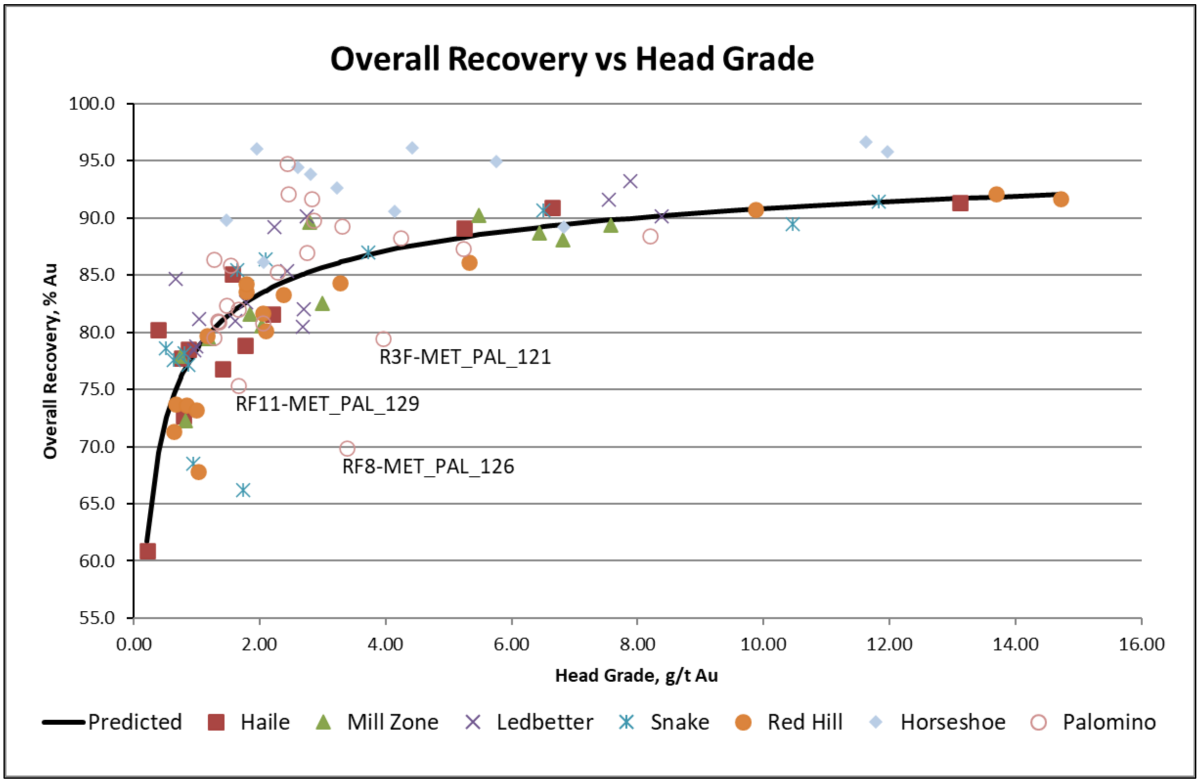
Table 13-21: Palomino Leach Test Results

Composite	Au Recovery	Float Con %	Float Tail %	Overall %
MET_PAL_120	96	82.3	51	81.0
MET_PAL_121	93.7	82.5	55.4	80.8
MET_PAL_122	93	80.5	65.8	79.5
MET_PAL_123	89	90.6	71.1	88.4
MET_PAL_125	93.8	83.8	55.3	82.0
MET_PAL_126	96.7	87.5	71.3	86.9
MET_PAL_128	82.7	90.8	70	89.3
MET_PAL_129	98.4	70.1	58.1	69.9
MET_PAL_130	91.1	88	69.4	86.4
MET_PAL_131	96.9	82.8	68.8	82.3
MET_PAL_132	95.6	76.6	471	75.3
MET_PAL_133	94.3	82.5	54.9	80.9
MET_PAL_134	93.2	97.6	64.7	92.1
MET_PAL_135	91.7	86.3	74.7	85.3
MET_PAL_136	94.9	80.5	62.3	79.6
MET_PAL_119	82.8	92.4	68.2	88.3
MET_PAL_124	95.8	87.3	69	86.5
MET_PAL_127	93.2	93.3	68.7	91.7
MET_PAL_137	92.2	97	68.2	94.7
MET_PAL_138	89.3	89.7	66.9	87.3
MET_PAL_139	87.8	92.6	69.3	89.8

Source: OceanaGold, 2024

Four samples demonstrated low recoveries in comparison with the Haile recovery database as can be seen graphically in Figure 13-6) which is based on Figure 13-3 but with the point identified by ore source. Three of the low recovery samples were from the siltstone geomet domain (MET_PAL_129 (RF8), MET_PAL_132 (RF11), MET_PAL_136 (RF15)) and one was from the rhyodacite geomet domain (MET_PAL_122 (RF3)). The low recoveries are primarily due to poor leach performance of the flotation concentrate, and tails residues from each of these bottle roll tests have been submitted for diagnostic leach test work. This test work will establish gold deportment in the leach residues, giving insight into potential opportunities for process/recovery improvements.

Further review is recommended in a future feasibility study on the source of MET_PAL_126 to identify the nearest composites and their performance to assess what portion of the resource may be considered as at risk of a significant underperformance from the model and also what other mineralized drill core may be available around the location of this sample to include in a future test program to further characterize the risk.



Source: OceanaGold, 2024

Figure 13-6: Haile Global Recovery Database

13.3 Recovery Estimate Assumptions

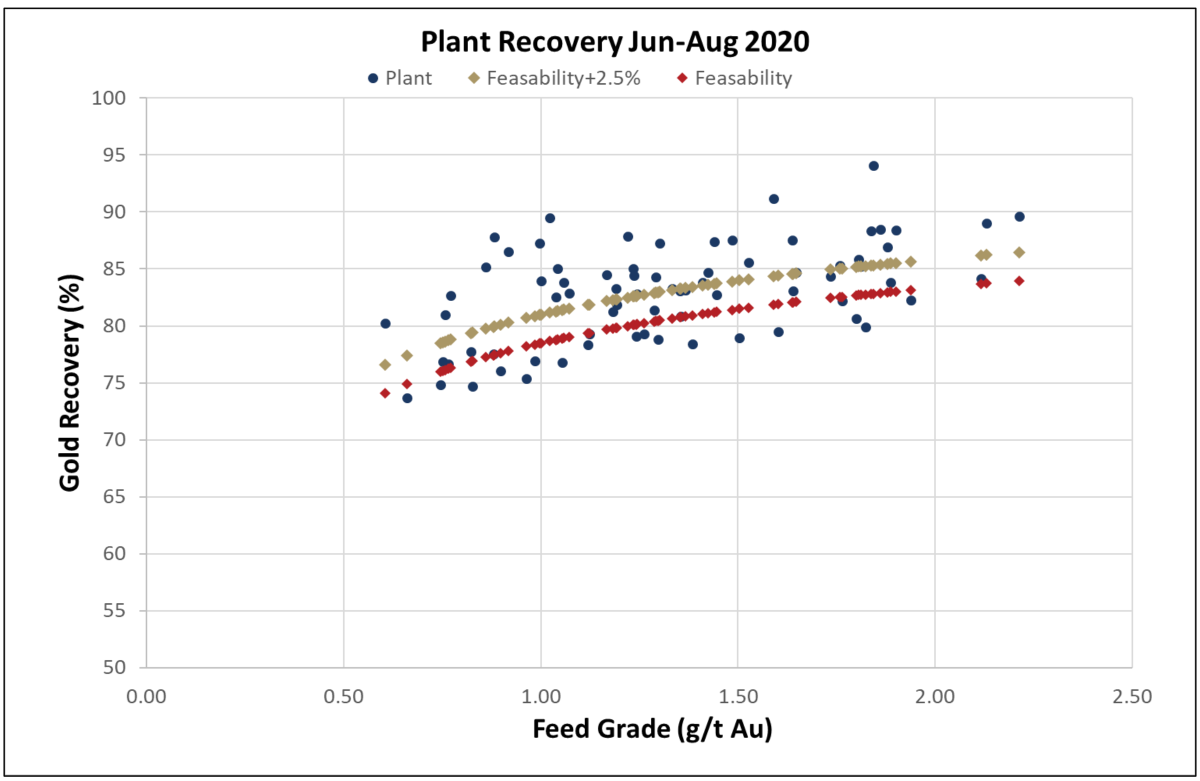
The recovery estimate previously was based on the interpretation of test work results from a range of samples and test work campaigns and shown in Figure 13-3 as a function of gold grade. This was based on the original flowsheet incorporating sulfide flotation and open circuit regrind of concentrate 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 13-7 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 13-7: Plant Gold Recovery Performance Jun-Aug 2020

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.

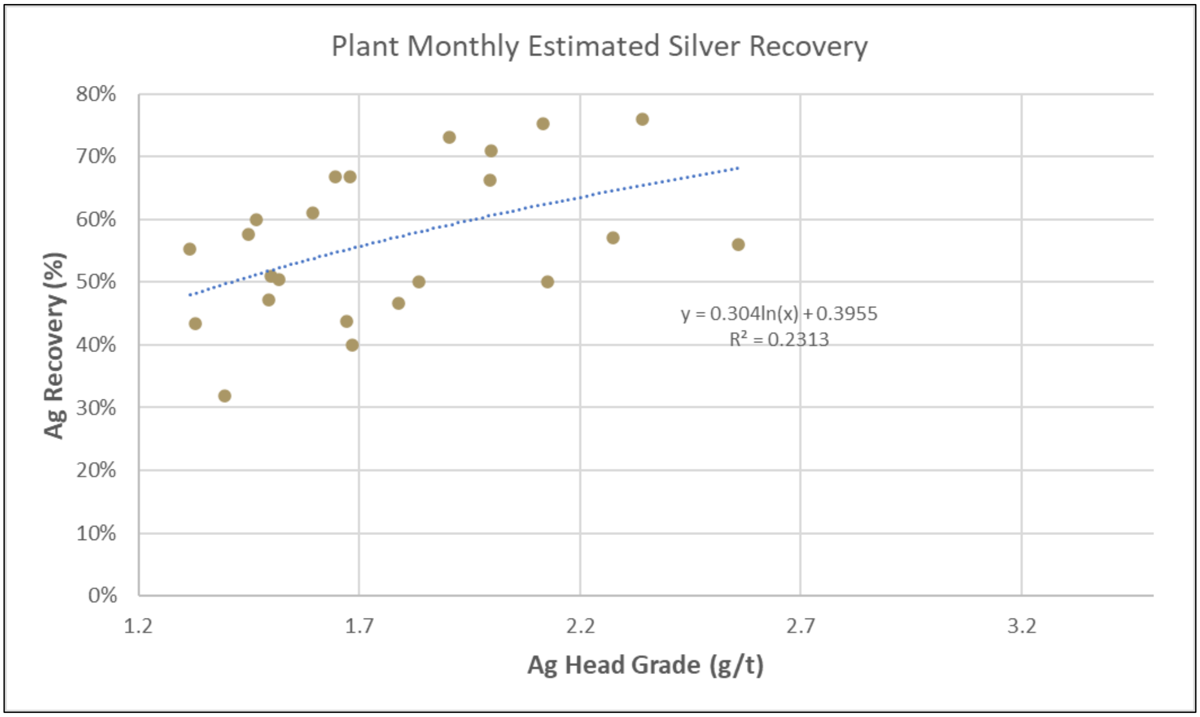
Given the mill feed schedule over the LoM tracks the major sources the following criteria have been applied to estimate overall gold recovery for production forecasting:

- Fresh sulfide ore delivered to the ROM above 1.7 g/t Au from open pit and Horseshoe UG utilize 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 attract 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 2.09 to 2.92 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 13-8.



Source: OceanaGold, 2022

Figure 13-8: Plant Reconciled Silver Recovery from 2020-21

Silver revenue accounts for approximately 1.4% of overall total sales and as such 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 have been 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 Group Manager Resource Development, 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, 2023.

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 “HA0922OLM_V6”, 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 “HA1123URR_V3” 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 “PA_1023_URR” 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 downhole composites to produce block gold grade estimates into a 10 m E x 10 m N x 5 m RL model blocks. Gold estimation was constrained within implicitly modeled 0.065 g/t Au indicator grade shell. Metasediment/metavolcanic contacts were not used to constrain gold estimation. Model block grade estimates for post-mineralization dikes were reset to zero gold grades subsequent to estimation. The above approaches are supported by relationships between mineralization, bedding and dikes observed in open pit grade control sample data.

MIK is well suited to estimating positively skewed grade distributions. Top caps of 50 g/t Au were applied 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.

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 with poor documentation
- Drillhole downhole composite to 2.5 m for Au.
- No composite (straight) for total carbon, total sulfur and silver due discontinuous sampling
- Composites duplicates removal
- Grade shell generation in Vulcan™ using Implicit modeling tool using 0.065 g/t Au threshold
- Composite flagging including domain area and au grade shell
- Block model creation in Vulcan™ including lithology, domain area and grade shell
- Run multipass MIK estimation for Au
- Set block grades to zero for blocks coded as post-mineralization dike
- Run OK estimation for Ag (using stepwise simulated Ag values for intervals with gold assays but no silver assay), total carbon and total sulfur
- Deplete resource with historical open pit and underground workings
- Assign model densities based on lithology
- Model classification

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 an additional internal database review in late 2018/early 2019. No material errors were identified.

Historical drilling (i.e., prior to 2007) accounts for approximately 17% of the data, although much of the resource associated with this data has now been mined out. The sample procedures applied to the historical drilling (i.e., drilling prior to Romarco Minerals Inc.) at Haile were not well documented. Pre-Romarco RC holes drilled by Piedmont were downgraded by 5%. (RC0039 – RC1303).The following analysis of rotary split sub-sample mass for Romarco RC holes (RC1502 to RC2216) was undertaken. Rotary 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.

Drillhole data available in September 2022, was included in the HA0922OLM_V6 estimate (Table 14-1). The acQuire database extraction excludes numerous shallow 3-10 m deep auger and conventional rotary holes. Blastholes are also excluded from the extraction.

The assay coverage for gold includes 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 overburden 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, Silver, Sulfur and Carbon

	Sample Stats			
	Au	**Ag**	**S**	**C**
Count	399,585	16178	71,203	34,077
Min	0.001	0.005	0.001	0.001
Max	1281.92	5211.47	49.000	12.70
Mean	0.33	2.49	1.19	0.31
Stdev	4.63	42.37	2.15	0.37
CV	14.21	17.02	1.82	1.19

Source: OceanaGold, 2024

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 14-3). 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 intensity of “silicification” increases from 0 (absent) to 3 (pervasive) 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. Whilst there is a broad gold to silicification relationship, there is an element of subjectivity in the logging of silicification. 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 estimates based upon a gold-only indicator. There are also some areas of gold mineralization not associated with intense silicification.

14.1.5 Pyrite

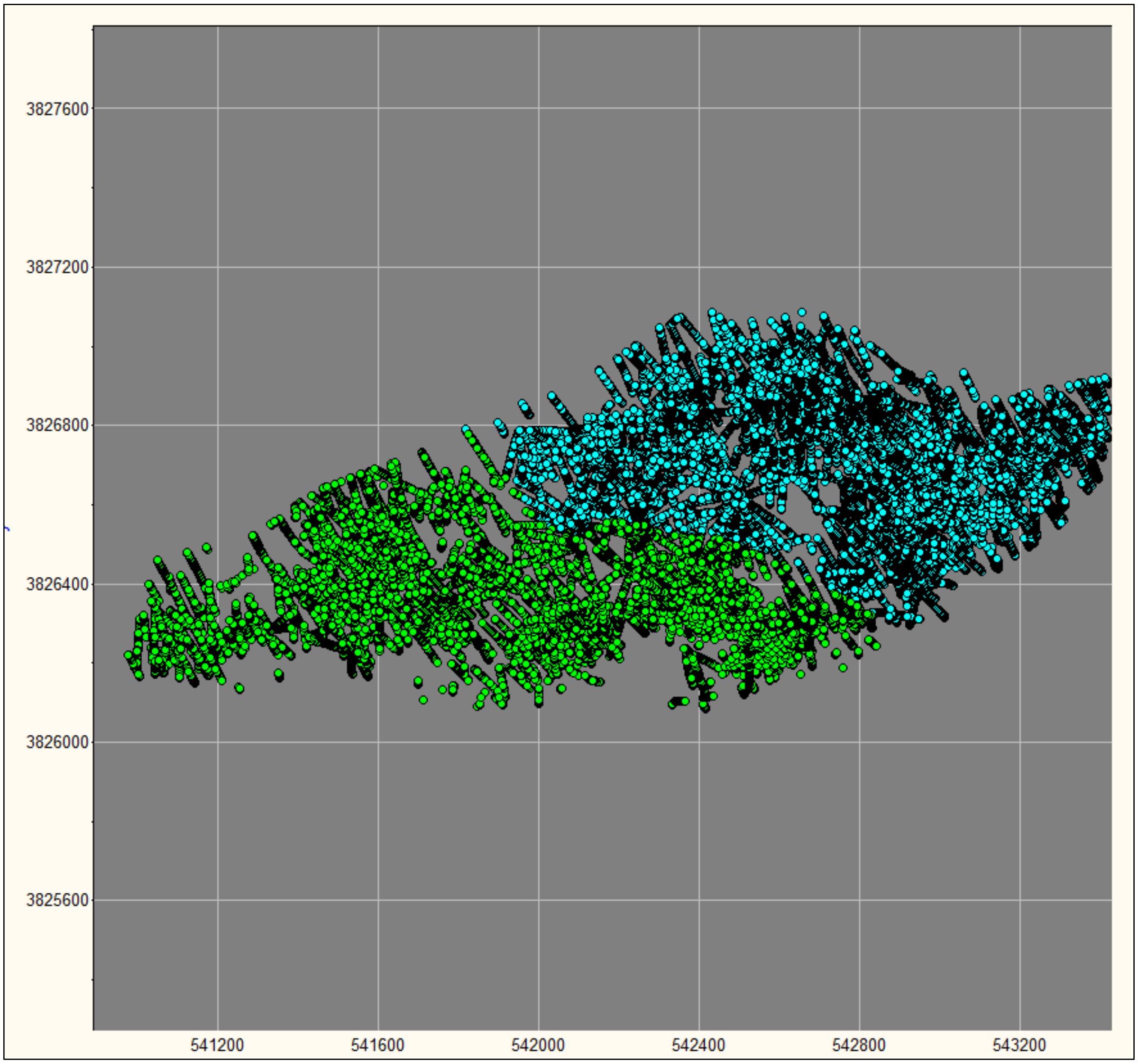
Multiple morphologies of pyrite have been identified at Haile, ranging from fine to coarse cubic pyrite. Based on logging, it has been established that the fine-grained pyrite is commonly associated with mineralization.

14.1.6 Grade Domain Construction

Although both silicification and pyrite occurrence are qualitatively associated with gold mineralization, these broad relationships are not used for quantitative gold domain definition. Implicit modeling in Vulcan^{TM} 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, 2024

Figure 14-1: Estimation Domains

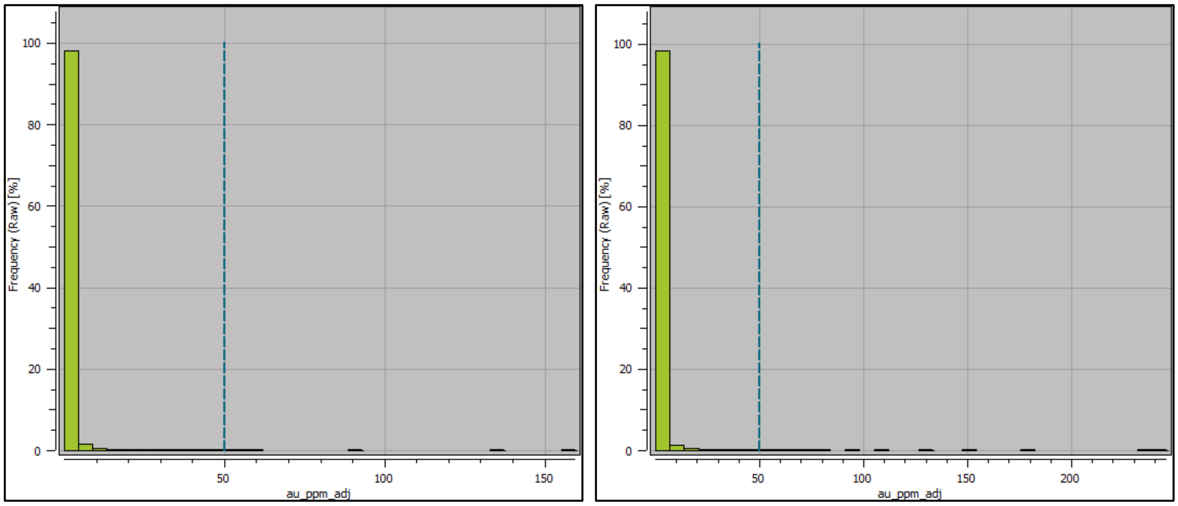
14.1.7 Compositing

Compositing for gold was completed in Vulcan™ software to 2.5 m downhole lengths within au implicit wireframe. The merge function was used, where intervals less than or equal to 1.25 m are merged with the adjacent sample, resulting in lengths ranging from 1.25 to 3.75 m with a mean of 2.5 m. A straight composite (no composite) chosen for total sulfur, total carbon and silver due to the paucity and discontinuous distribution of data.

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. Additionally, 2.5 m composited gold grades were top capped to 50 g/t Au to temper mean grades above the top-class indicator threshold (Figure 14-2). Only nine and 15 composites were capped from domain 1 and 2 respectively.

Total sulfur was top capped to 33% for extreme value (one composite only). No top capped was applied for total carbon.



Source: OceanaGold, 2024

Figure 14-2: Top Capped to 50 g/t Au for Domain 1 (left) and Domain 2 (right)

14.1.9 Multiple Indicator Gold Class Thresholds and Means

Gold grade thresholds were determined for each domain based on their histograms and cumulative distribution functions (CDF). Thresholds were set at 10th, 20th, 30th, 40th, 50th, 60th, 70th, 75th, 80th, 85th, 90th, 95th, 97.5th and 99th percentiles. Figure 4-3 summarizes the indicator threshold grades and class means used for gold estimation. A top cap of 50 g/t were used to temper mean grades above the top indicator threshold in the MIK estimation process. This process was completed using GS3 software.

Summary of Grade Thresholds and Class Mean Values for MIK Run									
Domains with Threshold, Bin Ranges and Bin Values.									
DOMAIN 1					DOMAIN 2				
Bin #	Class Width	Class Mean	Class Median	No. of Samples	Bin #	Class Width	Class Mean	Class Median	No. of Samples
1	>=0.00, <0.05	0.026	0.03	3054	1	>=0.00, <0.03	0.016	0.02	2432
2	>=0.05, <0.07	0.06	0.06	3054	2	>=0.03, <0.06	0.046	0.05	2433
3	>=0.07, <0.10	0.085	0.08	3054	3	>=0.06, <0.09	0.073	0.07	2432
4	>=0.10, <0.13	0.116	0.12	3054	4	>=0.09, <0.13	0.11	0.11	2433
5	>=0.13, <0.18	0.155	0.15	3054	5	>=0.13, <0.19	0.161	0.16	2432
6	>=0.18, <0.25	0.212	0.21	3054	6	>=0.19, <0.28	0.236	0.23	2433
7	>=0.25, <0.37	0.303	0.3	3054	7	>=0.28, <0.43	0.351	0.35	2432
8	>=0.37, <0.46	0.409	0.41	1527	8	>=0.43, <0.55	0.486	0.48	1216
9	>=0.46, <0.59	0.516	0.51	1527	9	>=0.55, <0.71	0.624	0.62	1217
10	>=0.59, <0.79	0.679	0.68	1527	10	>=0.71, <0.97	0.829	0.82	1216
11	>=0.79, <1.17	0.96	0.95	1527	11	>=0.97, <1.48	1.196	1.18	1216
12	>=1.17, <2.07	1.55	1.52	1527	12	>=1.48, <2.71	1.957	1.89	1216
13	>=2.07, <3.05	2.513	2.5	611	13	>=2.71, <4.16	3.352	3.32	487
14	>=3.05, <6.00	4.139	3.94	611	14	>=4.16, <9.2	5.932	5.64	486
15	>=6.00, <max = 50	11.34	8.46	306	15	>=9.2, < max = 50	16.6	12.9	244

Source: OceanaGold, 2024

Figure 14-3: Indicator Gold Class Thresholds and Means

14.1.10 Variogram Analysis and Modeling

Indicator variograms were produced for all grade thresholds in both domains. Variograms were generated in GS3 software and exported in Vulcan™ format. Figure 14-4 to Figure 14-9 shows indicator variogram details. Variograms for carbon and sulfur were estimated globally.

Variography for Domain 1 - *Structure 1*									
Cutoff/Thresholds	Nugget	Rotation			Range Structure 1		Structure 1 axis radius		
Cutoff / Threshold	Nugget	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 1 Model Type	Sill 1	r_{1Major}	$r_{1Semi\text{-}Major}$	r_{1Minor}
0.050	0.010	27.0	-17.0	7.0	Exponential	0.92	39.5	22.5	5
0.070	0.007	37.0	-43.0	19.0	Exponential	0.85	7.5	87.5	7.5
0.100	0.010	64.0	-39.0	36.0	Exponential	0.73	44	10	5.5
0.130	0.038	38.0	-20.0	53.0	Exponential	0.61	5.5	10.5	8
0.180	0.150	9.0	53.0	-60.0	Exponential	0.48	12	11.5	9
0.250	0.091	86.0	5.0	41.0	Exponential	0.5	29.5	12.5	9
0.370	0.023	11.0	-33.0	9.0	Exponential	0.5	9.5	36.5	7.5
0.460	0.073	103.0	8.0	35.0	Exponential	0.47	14.5	13.5	8
0.590	0.280	35.0	-36.0	-68.0	Exponential	0.49	40.5	49	10
0.790	0.140	101.0	7.0	-55.0	Exponential	0.41	14.5	52	6
1.170	0.180	23.0	-31.0	12.0	Exponential	0.54	8.5	9.5	32.5
2.070	0.090	12.0	47.0	-53.0	Exponential	0.62	33.5	6	28
3.050	0.030	68.0	-13.0	49.0	Exponential	0.74	8	7	14
6.000	0.007	62.0	14.0	25.0	Exponential	0.71	6.5	7	5.5

Source: OceanaGold, 2024

Figure 14-4: Domain 1 Variography – Structure 1

Variography for Domain 1 - *Structure 2*								
Cutoff/Thresholds	Rotation			Range Structure 2		Strucutre 2 axis radius		
Cutoff / Threshold	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 2 Model Type	Sill 2	r_{2Major}	$r_{2Semi\text{-}Major}$	r_{2Minor}
0.050	27.0	-17.0	7.0	Exponential	0.051	40	260	62
0.070	37.0	-43.0	19.0	Exponential	0.016	8	103	27
0.100	64.0	-39.0	36.0	Exponential	0.053	91	16	56
0.130	38.0	-20.0	53.0	Exponential	0.18	20	85	118
0.180	9.0	53.0	-60.0	Exponential	0.18	96	52	51
0.250	86.0	5.0	41.0	Exponential	0.19	30	52	88
0.370	11.0	-33.0	9.0	Exponential	0.28	52	37	73
0.460	103.0	8.0	35.0	Exponential	0.24	15	84	75
0.590	35.0	-36.0	-68.0	Exponential	0.063	236	64	173
0.790	101.0	7.0	-55.0	Exponential	0.22	20	53	189
1.170	23.0	-31.0	12.0	Exponential	0.091	189	23	43
2.070	12.0	47.0	-53.0	Exponential	0.27	44	643	66
3.050	68.0	-13.0	49.0	Exponential	0.21	80	709	48
6.000	62.0	14.0	25.0	Exponential	0.016	47	36	6

Source: OceanaGold, 2024

Figure 14-5: Domain 1 Variography – Structure 2

Variography for Domain 1 - *Structure 3*								
Cutoff/Thresholds	Rotation			Range Structure 3		Strucutre 3 axis radius		
Cutoff / Threshold	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 3 Model Type	Sill 3	r_{3Major}	$r3_{Semi\text{-}Major}$	$r3_{Minor}$
0.050	27.0	-17.0	7.0	Spherical	0.02	2850	288	375
0.070	37.0	-43.0	19.0	Spherical	0.12	1092	166	98
0.100	64.0	-39.0	36.0	Spherical	0.21	976	137	65
0.130	38.0	-20.0	53.0	Spherical	0.17	554	86	548
0.180	9.0	53.0	-60.0	Spherical	0.18	97	1423	260
0.250	86.0	5.0	41.0	Spherical	0.22	321	926	89
0.370	11.0	-33.0	9.0	Spherical	0.19	1028	369	74
0.460	103.0	8.0	35.0	Spherical	0.21	353	1092	78
0.590	35.0	-36.0	-68.0	Spherical	0.17	333	76	500
0.790	101.0	7.0	-55.0	Spherical	0.22	401	61	218
1.170	23.0	-31.0	12.0	Spherical	0.19	190	315	68
2.070	12.0	47.0	-53.0	Spherical	0.02	68	844	984
3.050	68.0	-13.0	49.0	Spherical	0.02	766	764	51
6.000	62.0	14.0	25.0	Spherical	0.27	99	53	19

Source: OceanaGold, 2024

Figure 14-6: Domain 1 Variography – Structure 3

Variography for Domain 2 - *Structure 1*									
Cutoff/Thresholds	Nugget	Rotation			Range Structure 1		Structure 1 axis radius		
Cutoff / Threshold	Nugget	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 1 Model Type	Sill 1	r_{1Major}	$r_{1Semi\text{-}Major}$	r_{1Minor}
0.030	0.014	940.0	24.0	16.0	Exponential	0.87	6	61.5	5
0.060	0.008	30.0	-27.0	10.0	Exponential	0.88	26.5	13	6
0.090	0.006	51.0	-35.0	-7.0	Exponential	0.82	43.5	10	7
0.130	0.009	67.0	40.0	40.0	Exponential	0.27	5	8.5	46.5
0.190	0.007	171.0	-41.0	42.0	Exponential	0.63	9	14	6
0.280	0.170	168.0	-37.0	-39.0	Exponential	0.53	9.5	44.5	22.5
0.430	0.190	48.0	-23.0	11.0	Exponential	0.44	11	29.5	15.5
0.550	0.170	31.0	-26.0	-1.0	Exponential	0.33	16.5	9.5	10
0.710	0.220	19.0	67.0	29.0	Exponential	0.53	23	73	11
0.970	0.270	50.0	-13.0	9.0	Exponential	0.45	5.5	46	32
1.480	0.280	57.0	-8.0	11.0	Exponential	0.33	8	34	14
2.710	0.010	79.0	2.0	55.0	Exponential	0.55	6.5	5.5	7
4.160	0.040	89.0	-8.0	59.0	Exponential	0.53	6	5	5
9.200	0.130	81.0	12.0	-27.0	Exponential	0.58	42.5	30.5	55

Source: OceanaGold, 2024

Figure 14-7: Domain 2 Variography – Structure 1

Variography for Domain 2 - *Structure 2*								
Cutoff/Thresholds	Rotation			Range Structure 2		Strucutre 2 axis radius		
Cutoff / Threshold	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 2 Model Type	Sill 2	r_{2Major}	$r_{2Semi\text{-}Major}$	r_{2Minor}
0.030	940.0	24.0	16.0	Exponential	0.094	33	40.1	28
0.060	30.0	-27.0	10.0	Exponential	0.082	199	99	132
0.090	51.0	-35.0	-7.0	Exponential	0.014	120	38	9
0.130	67.0	40.0	40.0	Exponential	0.48	6	10	72
0.190	171.0	-41.0	42.0	Exponential	0.35	62	443	81
0.280	168.0	-37.0	-39.0	Exponential	0.07	21	46	288
0.430	48.0	-23.0	11.0	Exponential	0.14	25	108	71
0.550	31.0	-26.0	-1.0	Exponential	0.47	149	142	51
0.710	19.0	67.0	29.0	Exponential	0.22	82	150	1194
0.970	50.0	-13.0	9.0	Exponential	0.045	49	147	53
1.480	57.0	-8.0	11.0	Exponential	0.16	12	166	61
2.710	79.0	2.0	55.0	Exponential	0.17	7	78	38
4.160	89.0	-8.0	59.0	Exponential	0.1	9	133	33
9.200	81.0	12.0	-27.0	Exponential	0.025	46	31	8

Source: OceanaGold, 2024

Figure 14-8: Domain 2 Variography – Structure 2

Variography for Domain 2 - *Structure 3*								
Cutoff/Thresholds	Rotation			Range Structure 3		Strucutre 3 axis radius		
Cutoff / Threshold	Bearing (Z rot)	Plunge (Y rot)	Dip (X rot)	Structure 3 Model Type	Sill 3	r_{3Major}	$r3_{Semi-Major}$	$r3_{Minor}$
0.030	940.0	24.0	16.0	Spherical	0.02	136	1677	2042
0.060	30.0	-27.0	10.0	Spherical	0.029	2028	198	135
0.090	51.0	-35.0	-7.0	Spherical	0.16	1575	116	105
0.130	67.0	40.0	40.0	Spherical	0.25	85	100	1245
0.190	171.0	-41.0	42.0	Spherical	0.02	128	1923	1921
0.280	168.0	-37.0	-39.0	Spherical	0.23	106	123	902
0.430	48.0	-23.0	11.0	Spherical	0.23	1084	138	72
0.550	31.0	-26.0	-1.0	Spherical	0.032	1061	192	71
0.710	19.0	67.0	29.0	Spherical	0.033	84	264	1261
0.970	50.0	-13.0	9.0	Spherical	0.24	716	169	61
1.480	57.0	-8.0	11.0	Spherical	0.23	1048	204	70
2.710	79.0	2.0	55.0	Spherical	0.27	271	121	63
4.160	89.0	-8.0	59.0	Spherical	0.32	433	152	34
9.200	81.0	12.0	-27.0	Spherical	0.26	472	32	33

Source: OceanaGold, 2024

Figure 14-9: Domain 2 Variography – Structure 3

14.1.11 Block Model

The HA0922OLM_V6 resource block model was constructed in Vulcan™ and the parameters below in Table 14-2 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 14-2: HA0922OLM_V6 Block Model Dimensions

Variable	X	Y	Z
Minimum	539810	3825575	200
Maximum	544510	3827725	1200
Block Size (Parent)	10	10	5
No. of Blocks (Parent)	470	215	200

Source: OceanaGold, 2024

14.1.12 Estimation Methodology Gold, Sulfur and Carbon

Gold estimation was completed using Multiple Indicator Kriging, while carbon and sulfur were estimated using OK. Carbon and sulfur values are used for classification of overburden 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. Figure 14-10 shows the block search parameters and Figure 14-11 shows visual orientation of the pass 1 ellipsoid search.

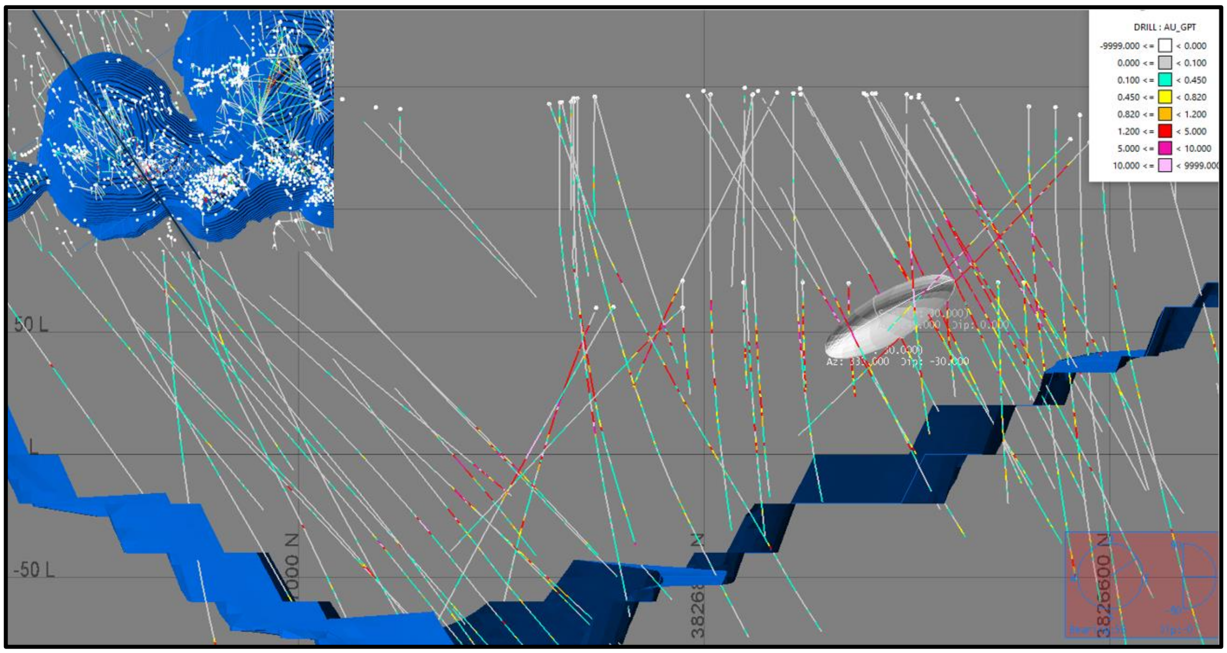
Block Au Estimation Search Parameters

HA0619OLM Model

	Search Orientation				Pass		Search Radius			Sample Counts		
Domain	**Area**	**Bearing**	**Plunge**	**Dip**	**Pass Name**	**Pass #**	**Major**	**Semi-Major**	**Minor**	**Min**	**Max**	**Max per Hole**
1	Mill Zone, Champion, Red Hill/ Haile	335	-30	0	MC_P1	1	30	30	10	4	16	3
					MC_P2	2	60	60	15	4	16	3
					MC_P3	3	90	90	20	1	12	3
2	Snake and Ledbetter	330	-40	0	SL_P1	1	30	30	10	4	16	3
					SL_P2	2	60	60	15	4	16	3
					SL_P3	3	90	90	20	1	12	3

Source: OceanaGold, 2024

Figure 14-10: Block Search Parameters



Source: OceanaGold, 2024

Figure 14-11: Cross Section View Of Domain 1 Search Ellipsoid – 1st Pass

14.1.13 Estimation Methodology Silver

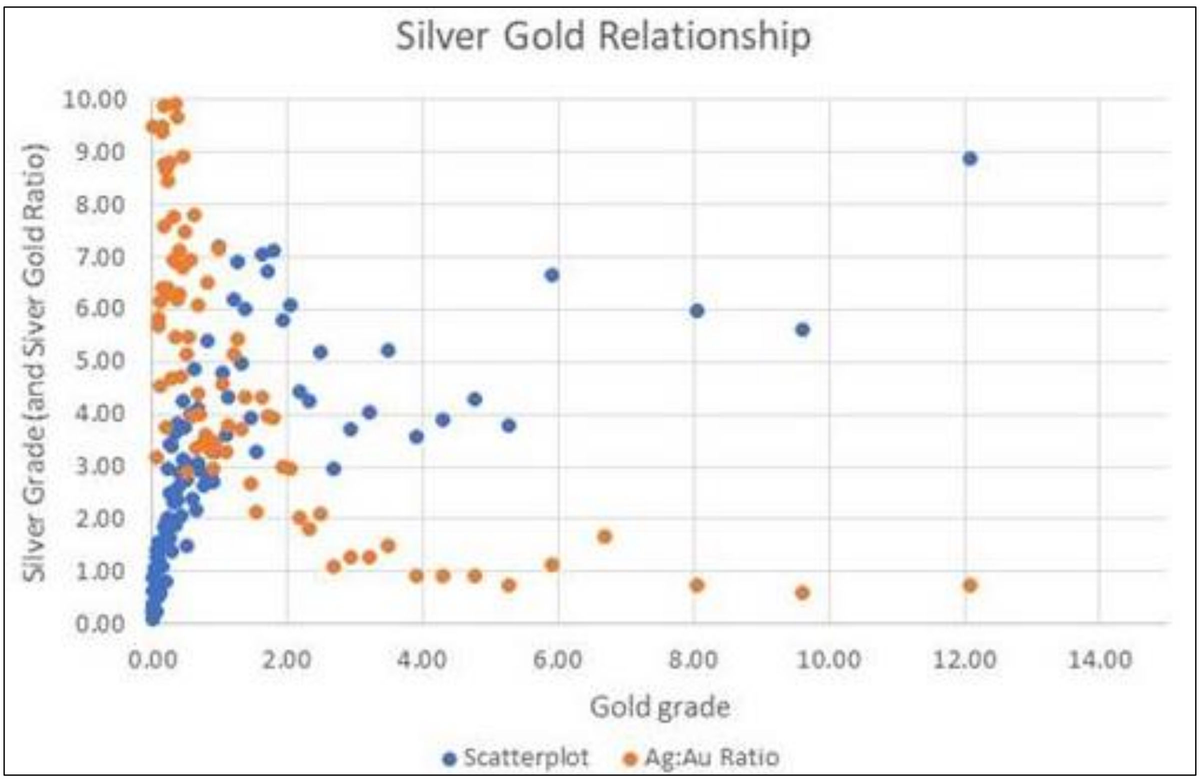
For the open pit, silver grade estimates are provided for metallurgical considerations (carbon stripping and electro-winning) as well as for revenue estimation, albeit silver contributes only about 1.5% of total revenue. Silver content is not used as a gold-equivalent input for cut-off calculation nor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

The sample support basis for the open pit silver estimates is approximately 4% of that for gold (Table 14-1). 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 stepwise 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 14-12 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 composite (3m) 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 14-12: 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, OK 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 14-3). Over 46,000 density measurements have been collected from core samples.

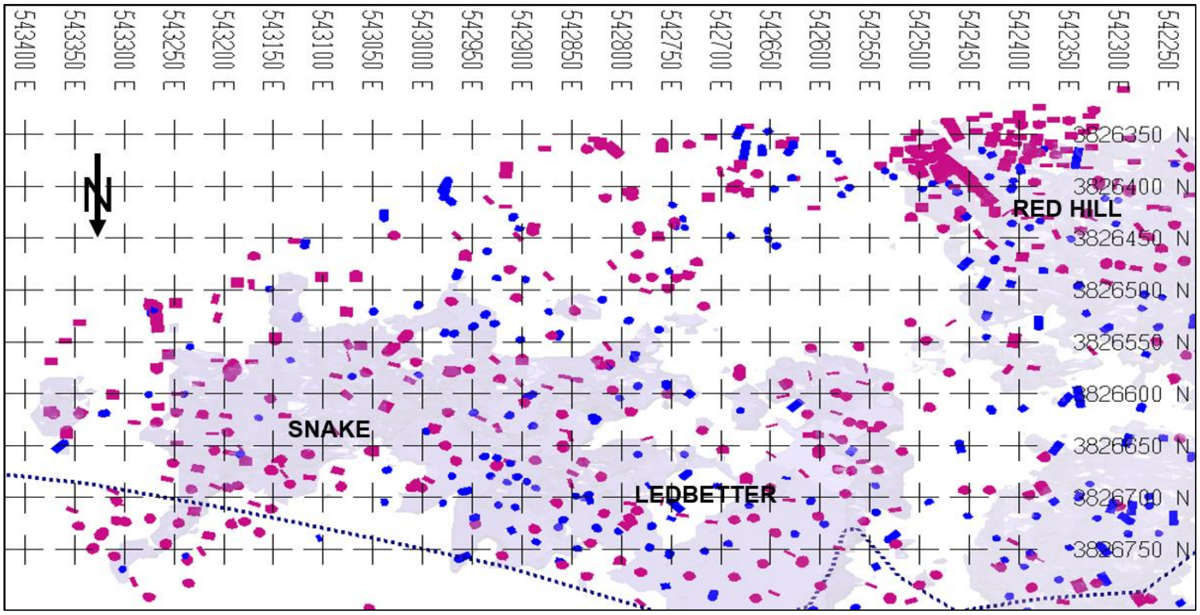
Table 14-3: HA0922OLM_V6 BD Assignment

<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 HA1220OLM Model</th></tr>
<tr><th rowspan="4">Sand</th><th rowspan="4">Saprolite</th><th rowspan="4">Dike</th><th colspan="2">Meta Volcanics</th><th colspan="4">Meta Sediments</th><th rowspan="4">Pag Fill</th><th rowspan="4">Tails</th><th rowspan="4">Heap</th></tr>
<tr><th>Oxidized</th><th>Fresh</th><th colspan="2">Ore</th><th colspan="2">Waste</th></tr>
<tr><td rowspan="2">2.52</td><td rowspan="2">2.7</td><th>Oxidized</th><th>Fresh</th><th>Oxidized</th><th>Fresh</th></tr>
<tr><td>2.57</td><td>2.78</td><td>2.49</td><td>2.76</td></tr>
<tr><td>2.06</td><td>2.18</td><td>2.88</td><td colspan="6"></td><td>1.89</td><td>2.14</td><td>1.7</td></tr>
</table>

Source: OceanaGold, 2024

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 multiple holes have been collared off single drill pads, resulting in fanned and variable drillhole orientations, oblique intersection angles and irregular drillhole spacings. Figure 14-13 below provides an example in the Snake, Ledbetter, and Red Hill areas. The View is -45° to the South, showing drillhole pierce points (+/- 5 m) projected onto the mineralization planes for Ledbetter, Snake and Red Hill areas. A 50 m grid is shown. Purple shading represents Measured and Indicated Resources (mined and unmined). Maroon dots represent 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. The drill spacing for Indicated Mineral Resources is approximately 35 m x 35 m but variable and locally is up to 42 m.



Source: OceanaGold, 2024

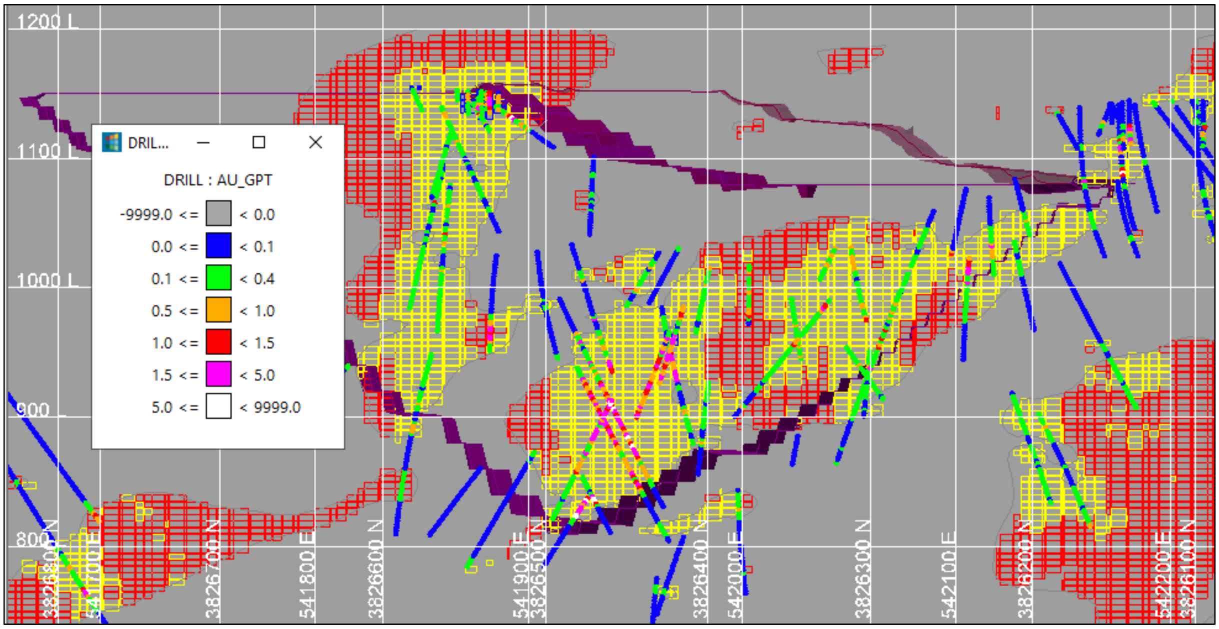
Figure 14-13: View -45° to South of Drillhole Pierce Points

Resource classification was assigned by Vulcan™ script according to the search criteria for each domain in Table 14-4 and Figure 14-14. These approximate a 35 m x 35 m drillhole spacing for Indicated Resources.

Table 14-4: Resource Classification Parameters

Confidence Category	Criteria
Measured	Blocks estimated within first pass A minimum of 4 drillholes Average distance < 14 m
Indicated	Blocks estimated within first and second pass A minimum of 2 drillholes Average distance < 36.8 m
Inferred	Blocks estimated within first and second pass Average distance >= 36.8 m Blocks estimated within third pass

Source: OceanaGold, 2024



Source: OceanaGold, 2024

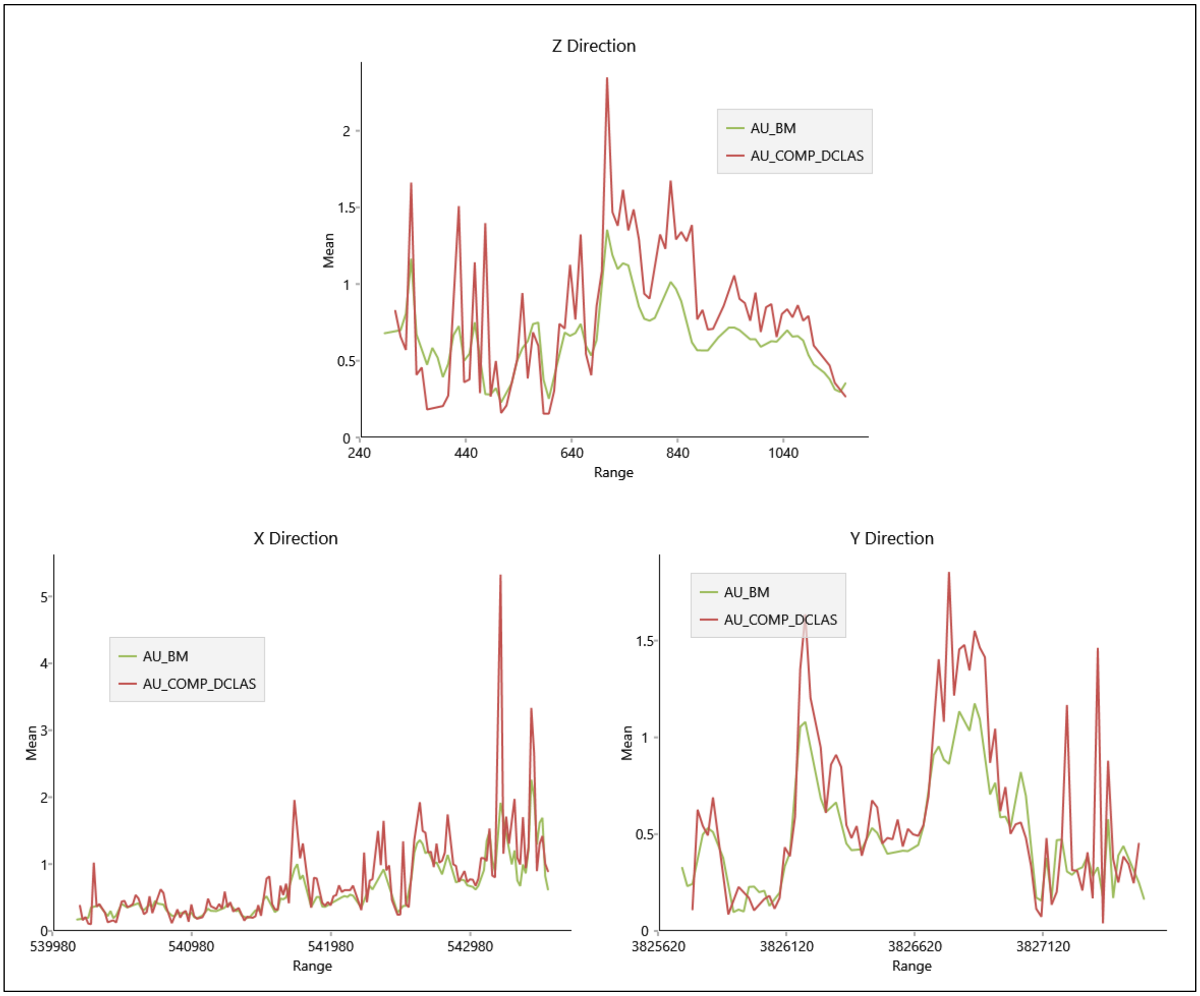
Figure 14-14: Oblique Cross-Section Looking NE showing Resource Classification within Ledbetter Pits (yellow: Indicated, red: Inferred)

14.1.16 Model Validation Prior and Post Estimation

Numerous methods have been used to validate the HA0922OLM_V6 resource model:

- Data validation
- Review of historical RC holes factoring and exclusion
- Cross-sectional checks on composite file and block model coding from lithological wireframes, domain area and grade shell
- Checked the execution of multi-pass search process
- Visual checks of estimated block grade on sections, plan and in 3D to ensure good correlation with underlying composite data
- Visual validation of resource classification
- Swath plots X, Y and Z comparing the gold estimates with the underlying composite grades.
- Detailed comparisons to previous model at global and local scales (i.e., grade tonnage and curve, contained gold by benches)
- Review of the methodology and validation of the scripts used

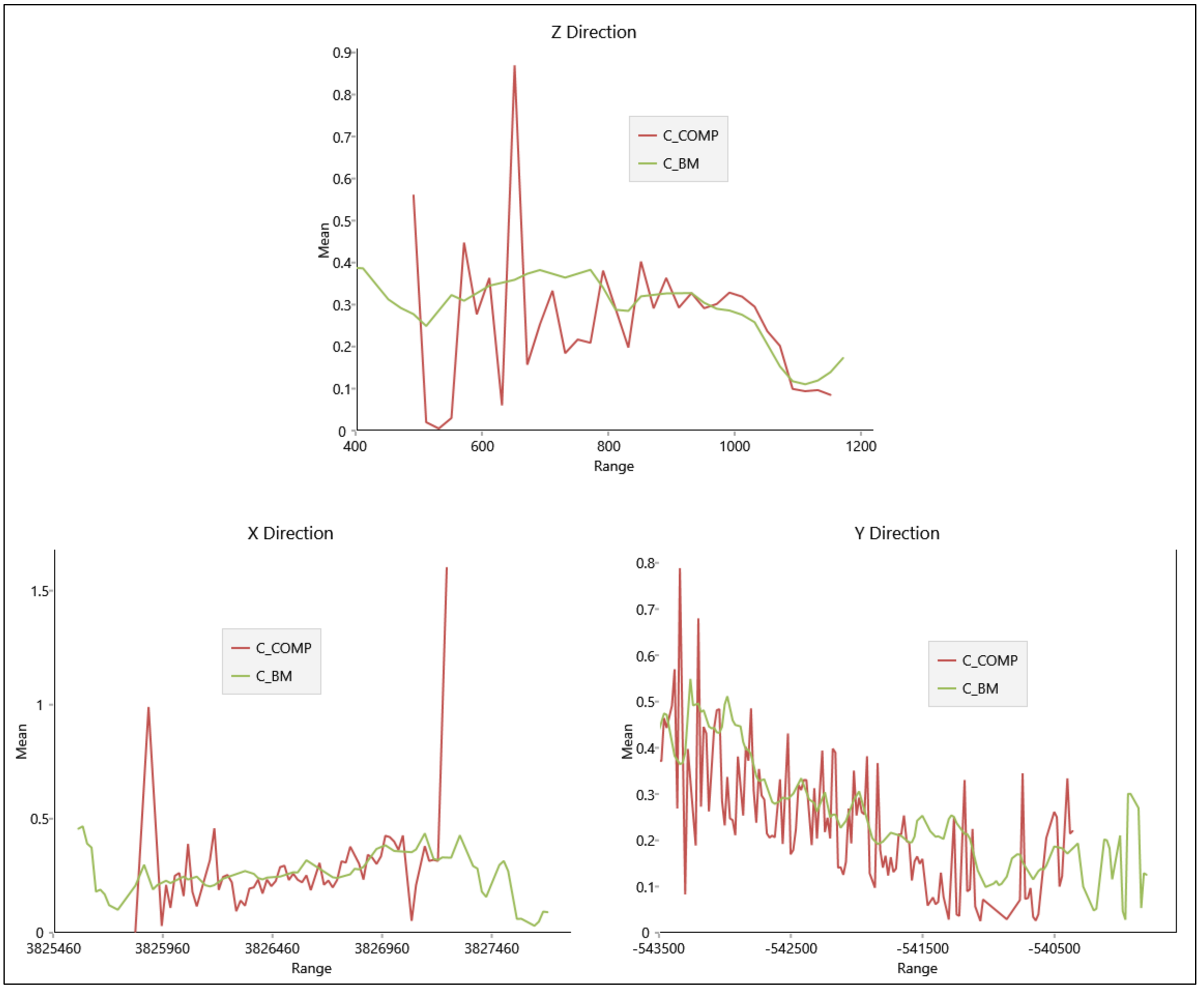
The swath plots in Figure 14-15 to Figure 14-17 compare 2.5 m bench composite grades to the estimated block grades for gold, carbon, and sulfur respectively.



Source: OceanaGold, 2022

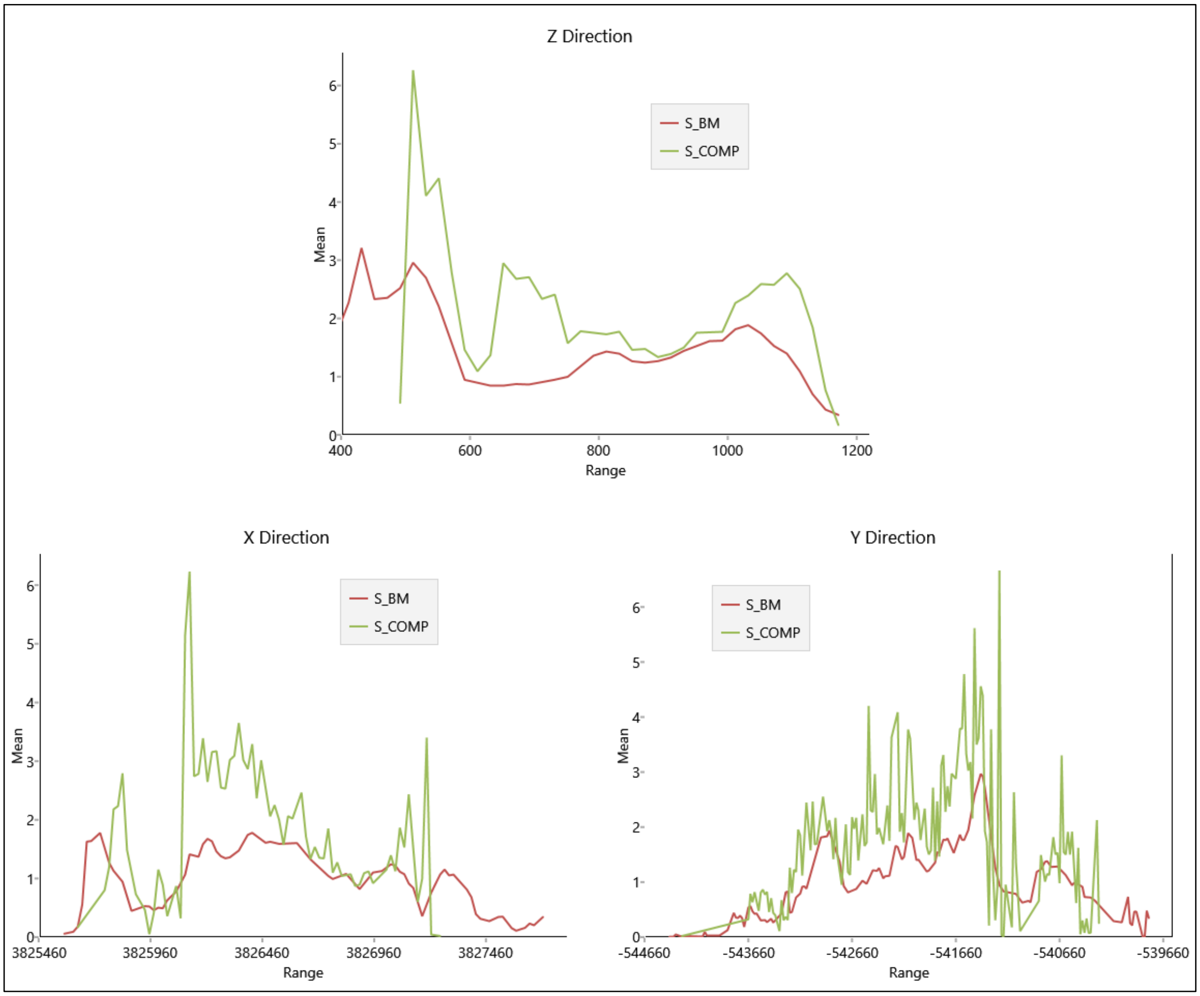
Figure 14-15: Swath Plot with Mean Composite Gold (Red) vs. Block Model Gold (Green)

Carbon and sulfur were estimated from raw drillhole data because of the lack of data relative to the gold and silver data sets.



Source: OceanaGold, 2024

Figure 14-16: Swath Plot with Mean Composite Carbon (Red) vs. Block Model Carbon (Green)



Source: OceanaGold, 2024

Figure 14-17: Swath Plot with Mean Composite Sulfur (Red) vs. Block Model Sulfur (Green)

As a gold estimation methodology check, an independent large panel recoverable resource estimate using MIK was constructed and compared to HA0922OLM_V6 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.

<u>Model Reviews</u>

<u>External</u>

- "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
    - No fatal flaws

Grade control data reveal that high grade lozenges with approximate 10 m E x 10 m N to 20 m x 20 m 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. The results suggested that an approximate 35m x 35m spacing provided acceptable annual precision.

14.1.17 Resource Model Reconciliation

Table 14-5 summarizes the Haile open pit resource model reconciliations from 2018 to 2023. Q4 2023 saw first production from the Horseshoe Underground, which contributed 82.8 kt at 5.08 g/t Au for 13.5 koz gold. This is included in the 2023 reconciliation in Table 14-5.

Mill-adjusted, the Horseshoe Underground resource model reconciled positively with +5% tonnes, +2% grade for +7% contained gold. The overall negative reconciliation for 2023 however (+8% tonnes, -12% grade for -5% contained gold), was due to open pit performance, where short-scale disruption of mineralization was encountered at the base of the Mill Zone Phase 2 open pit. The long term six-year average performance is reasonable; +10% for tonnes, -2% for grade and +9% for contained gold.

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 14-5: Resource Model Reconciliation

	Resource Model			Mine (Mill-Reconciled)			Mine / Model Factor (%)		
	Mt	Au g/t	Moz	Mt	Au g/t	Moz	Mt	Au g/t	Moz
2023	2.81	2.00	0.18	3.03	1.76	0.17	108%	88%	95%
2022	3.39	1.59	0.17	4.13	1.76	0.23	122%	110%	134%
2021	3.16	1.98	0.20	3.27	2.17	0.23	103%	110%	115%
2020	2.57	2.08	0.17	3.33	1.59	0.17	130%	76%	100%
2019	2.87	1.96	0.18	3.18	1.78	0.18	111%	91%	100%
2018	2.85	1.67	0.15	2.57	1.93	0.16	90%	116%	107%
Total	**17.7**	**1.86**	**1.05**	**19.5**	**1.83**	**1.14**	**110%**	**98%**	**109%**

Source: OceanaGold, 2024
2020 mining selectivity was poor and led to excessive mining dilution

14.1.18 Mineral Resource Statement

Table 14-6 summarizes the resulting open pit resources. A US$1,700/oz shell was used for reporting.

Table 14-6: Open Pit Total Mineral Resources as of 31 December 2023

Class	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (Moz)	Contained Ag (Moz)
Measured	3.8	1.02	1.1	0.12	0.1
Indicated	34.4	1.58	2.5	1.75	2.8
Measured & Indicated	**38.1**	**1.53**	**2.4**	**1.87**	**2.9**
Inferred	2.8	0.90	1.1	0.10	0.1

Source: OceanaGold, 2024

- Cut-off grade 0.50 g/t Au for primary mineralization and 0.60 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. It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

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 November 08, 2023.

Gold estimation was constrained within a wireframe implicitly modeled using Leapfrog® software at a 1.0 g/t Au threshold[3] with a 25% probability. The threshold was selected to be sufficiently below the reserve reporting cut-off grade of 1.55 g/t Au to minimize conditional bias. Sensitivity estimates using a range of threshold grade were tested. Model blocks with centroids outside the 1.0 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.

Since 2016, Haile's open pit grade control sampling has accumulated >450,000 closely spaced grade control samples, including gold, sulfur and carbon assays. A comprehensive structural geology review was completed by external consultant Jun Cowan in September 2023 using this and other available geological data. The review featured stereographic analysis of the relationships between bedding, dykes, foliation, and grade (Au, S and C) trends to establish a nested hierarchy of controls on mineralization. The Jun Cowan review implicates dikes not only as being preclusive to mineralization, but also potentially as controls on the distribution of high-grade mineralization. Over time, underground mapping and core logging will test this concept (Figure 14-27).

[3] This approximates a 0.4 g/t Au Indicator.

Following the review, OceanaGold evaluated a range of estimation domaining strategies by varying the structural trend of mineralization controls for lower part of the Horseshoe. The implicit models in Leapfrog were generated, imported, and estimated in Vulcan™ for sensitivity testing. The sensitivity estimates were compared statistically and visually. The estimates showed consistency with less than 4% variation, an acceptable range. As grade control drilling is completed and drilling density is increased, the variation will reduce considerably.

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 OK 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 Measured, Indicated and Inferred Mineral Resources, based primarily on drillhole spacing, but also considering geological complexity.

Horseshoe General Geology and Geologic Model

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, albeit the geometry is complex. 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 pervasive 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 dikes (diabase and lamprophyre) striking 330˚ with near-vertical dips. OceanaGold has constructed 3D wireframes which include the metasiltstone, metavolcanics, dikes, saprolite and sand. The metasiltstones are the main host of mineralization. This has resulted in a detailed, 3D geologic block model created with Leapfrog® software.

Horseshoe Geologic Model and Controls on Gold Mineralization

Horseshoe mineralization zone extends over a vertical distance of 350 m. Previously modeled as two separate upper and lower zones; these have now been combined in this model update. The mineralization has been interpreted to strike 55° (NE) with a dip of 40° NW, with the bulk of the mineralization in upper zone. This trend, parallel to the regional foliation, is also observed in the south-east of the Red Hill Open Pit (Figure 14-18). The extents of economic mineralization in the lower Horseshoe area have not been fully delineated by drilling. The relative location of Horseshoe to the Haile open pits is shown in Figure 14-19.

Although the silicification occurrence is qualitatively associated with gold mineralization, attempts to exploit the relationship for domain definition have been unsuccessful, possibly due to inconsistent logging (scratch test) of silicification over time.

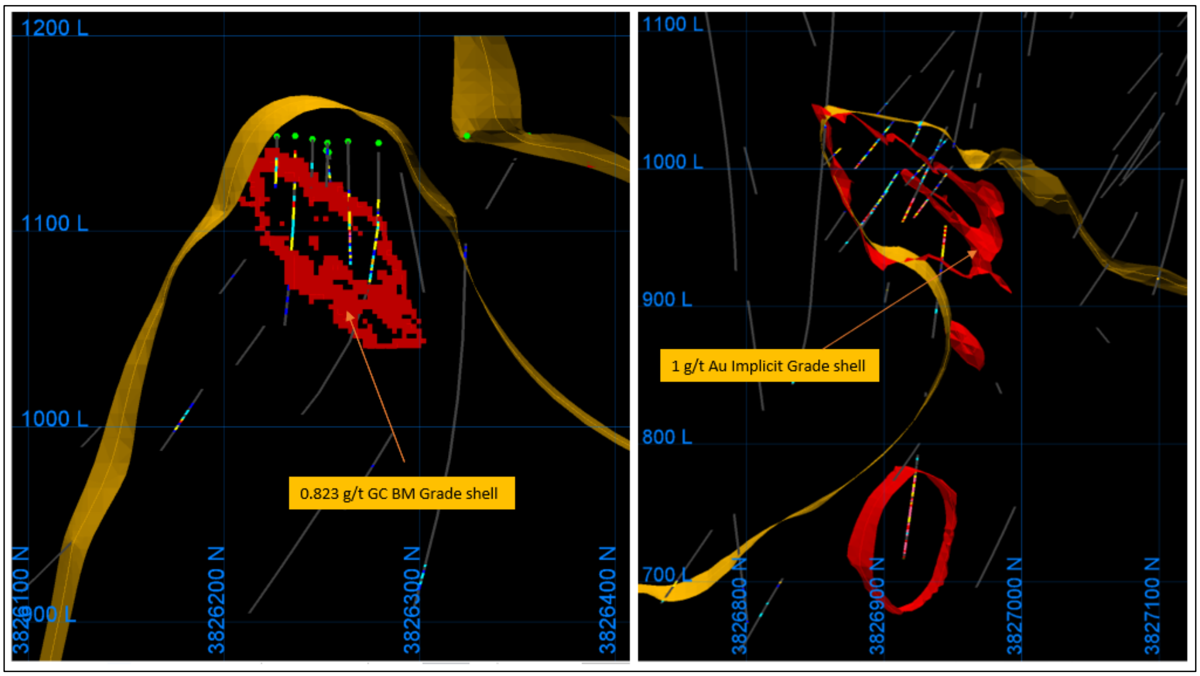
Based on the observed relationships between 3D interpreted lithology and grade distribution of high resolution (135 approximately 4 m x 4 m) open pit blast hole samples, lithology is not used to constrain the Au g/t grade shell (host lithology is however a control on mineralization, and in some cases considered for resource classification):

- Mineralization is commonly discordant to bedding (particularly in fold hinges and

- Bulk mineralization geometry does not reflect interpreted folding of the metavolcanic / metasediment contact (Figure 14-20).

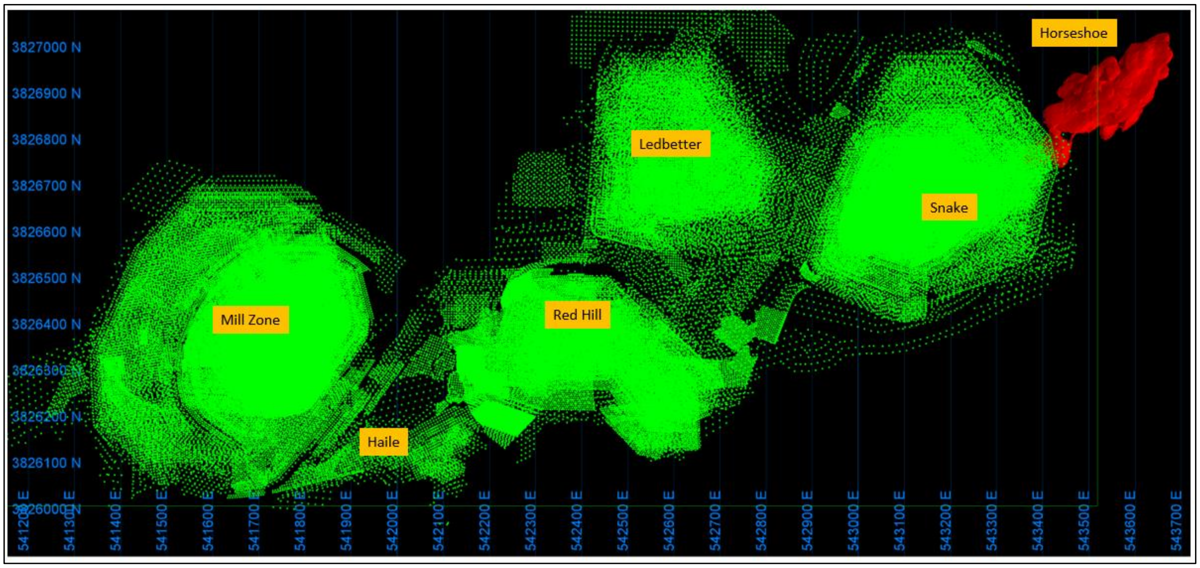
The gold grade shell used to constrain the gold estimation was generated in Leapfrog using resource drillhole data. Various Indicator RBFs were tested by varying the composite length, structural trend, grade thresholds and “iso” value (degree of confidence). The final grade shell is shown in Figure 14-21, and was generated using a 1 g/t Au threshold with 0.25 iso domain, 3 m length downhole composites, and small volumes discarded. Due to lack of data for Ag g/t, Au g/t grade shell was used to constrain the Ag g/t estimate.

An additional domain was generated by expanding the primary 1 g/t Au grade shell by 15 m to allow for dilution grade estimation.



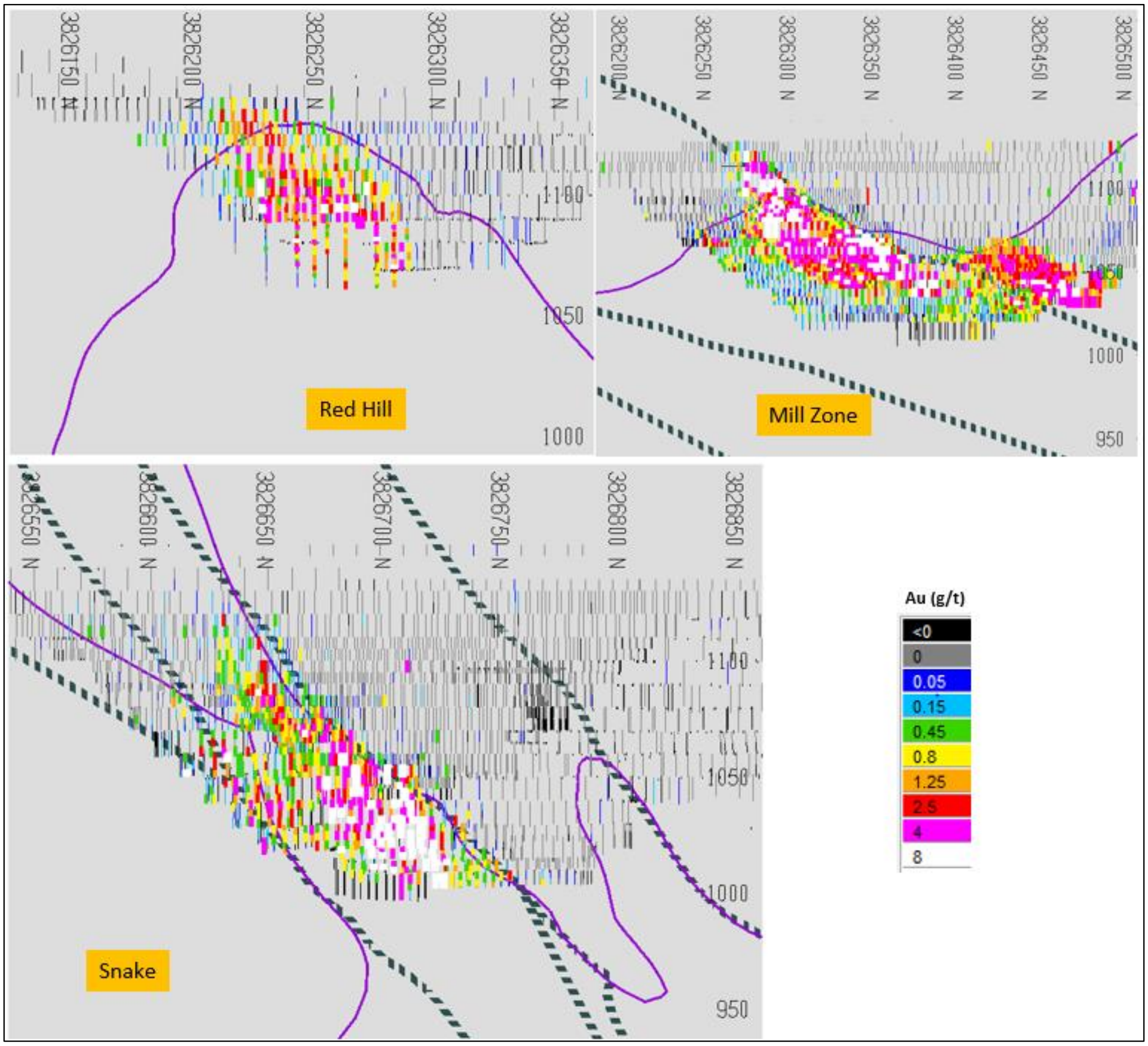
Source: OceanaGold, 2024

Figure 14-18: Cross Section Looking West, Red Hill at 542,600 mE (left) vs. Horseshoe at 543,480 mE (Right)



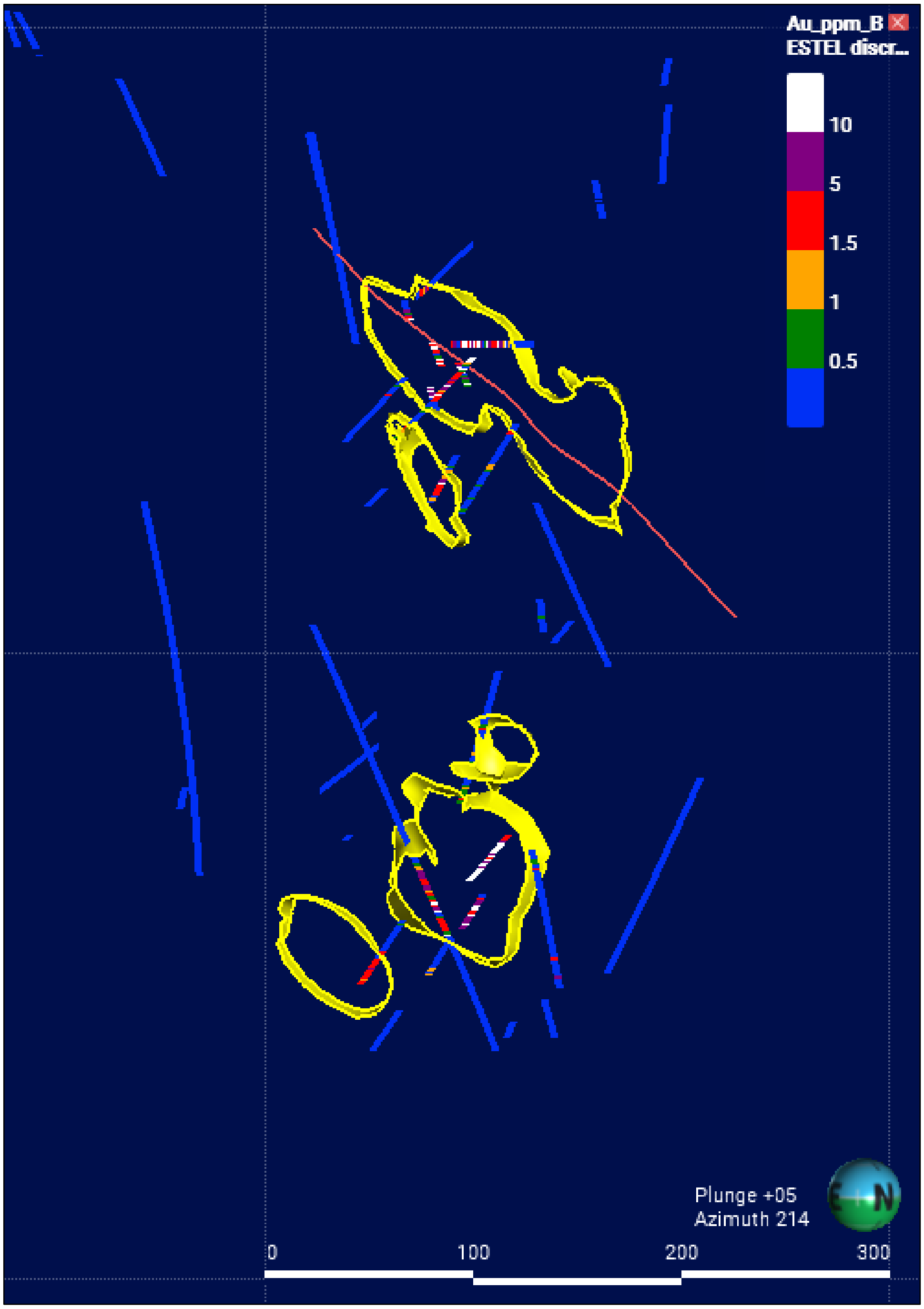
Source: OceanaGold, 2024

Figure 14-19: Plan View showing Horseshoe (Red) Relative to Open Pit Areas



Source: OceanaGold, 2024
Cross section looking west from several Open Pit locations showing the interpreted contact (purple) between metasediment (below) and metavolcanic (above) vs. grade control samples colored by Au g/t. Faults as black dashes.

Figure 14-20: Cross Section Looking West from Several Open Pit



Source: OceanaGold, 2024
Cross section at 54,3560mE showing 0.4 g/t grade shell interpretation in yellow, structure trend in maroon for upper part only. the resource drilling showing Au g/t grade.

Figure 14-21: Horseshoe Cross Section at 54,3560mE

Horseshoe Bulk Density

Model in situ dry bulk densities (BD) are based on domain averages, as shown in Table 14-7. 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 immersion method involved weighing the sample both in air and in water. The measurements were then averaged for each lithology.

Table 14-7: 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="4">Sand</th><th rowspan="4">Saprolite</th><th rowspan="4">Dike</th><th colspan="2">Meta Volcanics</th><th colspan="4">Meta Sediments</th><th rowspan="4">Pag Fill</th><th rowspan="4">Tails</th><th rowspan="4">Heap</th></tr>
<tr><th>Oxidized</th><th>Fresh</th><th colspan="2">Ore</th><th colspan="2">Waste</th></tr>
<tr><td rowspan="2">2.52</td><td rowspan="2">2.7</td><th>Oxidized</th><th>Fresh</th><th>Oxidized</th><th>Fresh</th></tr>
<tr><td>2.57</td><td>2.78</td><td>2.49</td><td>2.76</td></tr>
<tr><td>2.06</td><td>2.18</td><td>2.88</td><td></td><td></td><td></td><td></td><td></td><td></td><td>1.89</td><td>2.14</td><td>1.7</td></tr>
</table>

Source: OceanaGold, 2024

Horseshoe Compositing and Top Capping

The Horseshoe resource area includes 90 drillholes for 31,873 m. Of these 74 drillholes intersected the 0.4 g/t grade shell, the remainder were outside this volume.

The composite statistics shown in Table 14-8 for Au g/t and Ag g/t. A 100 g/t Au top cap and 26 g/t Ag top cap were applied. Based on coefficient of variation (CV), there is not a compelling case for top capping, but a 100 g/t Au top cap was applied in line with previous estimates. Six Au g/t composites and three Ag g/t composites are affected by top capping. Furthermore, domain 3 composite (Au and Ag) severely capped to account the high-grade outliers.

Table 14-8: Statistics Comparison

DOMAIN	FUNCTIONS	Raw		3 m Composite		Top Capping	
		Au	Ag	Au	Ag	Au	Ag
1	No. of Samples	5649	603	2404		2404	603
	Minimum	0.003	0.005	0.004		0.004	0.005
	Maximum	947	83.0	211		**100**	**26.0**
	Range	946.997	82.995	211.043		99.996	25.995
	Mean	**4.60**	**2.26**	**4.60**		**4.51**	**2.22**
	Std. dev	17.44	3.69	10.25		8.97	3.22
	Variance	304.15	13.63	105.02		80.44	10.35
	Coef. of Variance	3.79	1.63	2.23		1.99	1.45
	Skewness	27.47	7.89	8.75		6.04	4.55
	Median	1.50	1.50	1.77		1.77	1.50
3	No. of Samples	6126	192	3167		3167	192
	Minimum	0.003	0.005	0.002		0.002	0.005
	Maximum	66.4	4.1	33.7		**0.4**	**2.2**
	Range	66.375	4.109	33.712		0.438	2.195
	Mean	**0.16**	**0.43**	**0.16**		**0.10**	**0.42**
	Std. dev	0.95	0.48	0.68		0.14	0.42
	Variance	0.90	0.23	0.46		0.02	0.18
	Coef. of Variance	6.07	1.12	4.35		1.34	1.02
	Skewness	56.36	3.27	39.11		1.54	2.18
	Median	0.03	0.25	0.04		0.04	0.25

Source: OceanaGold, 2024

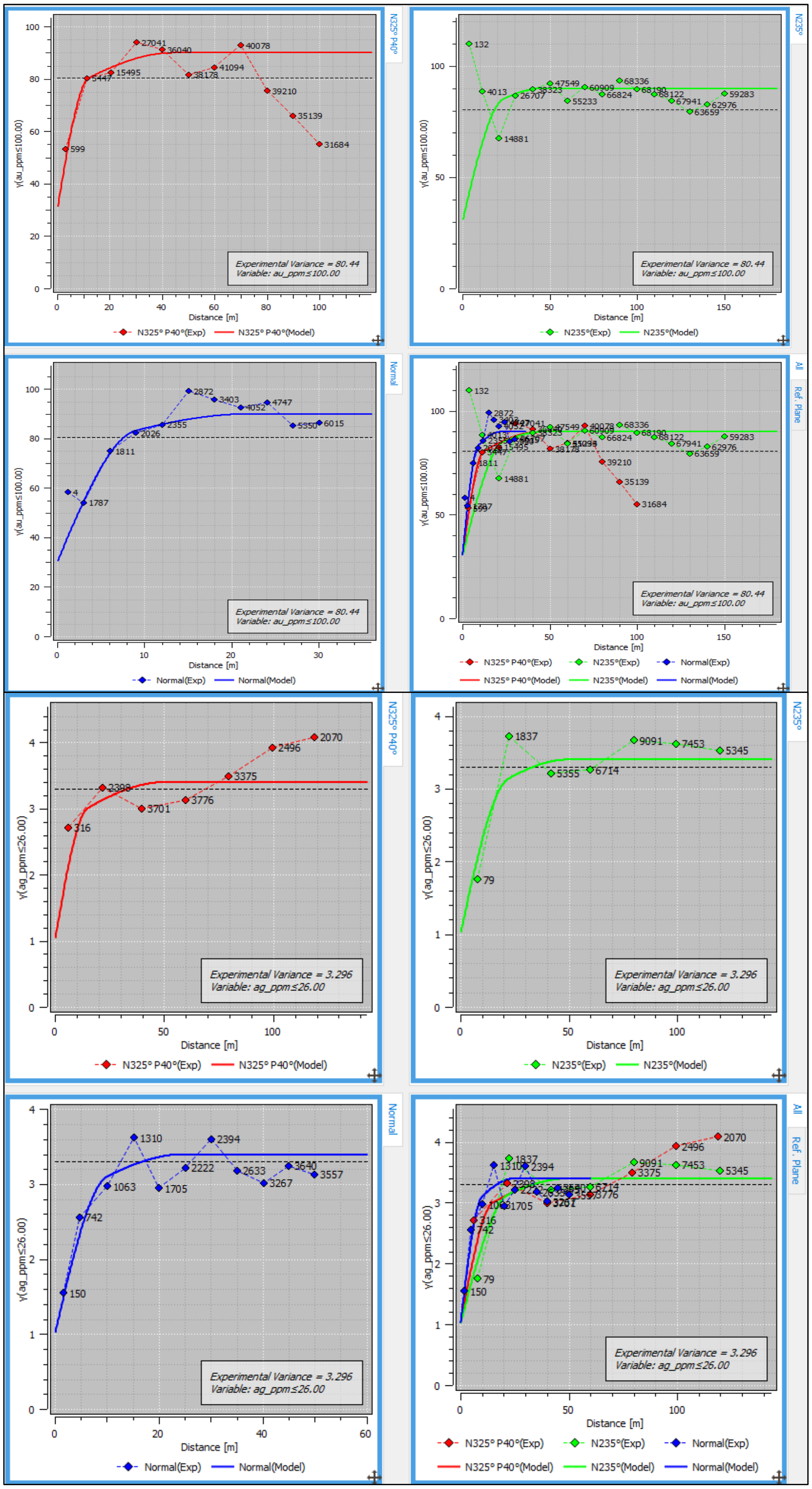
Experimental Variogram and Modeling

The experimental variogram was calculated in Isatis Neo software using 3 m downhole composite in Vulcan^{TM} within 0.4 g/t Au grade shell (Domain 1). A nugget and two spherical structures variogram were modeled for Au and Ag (Table 14-9 and Figure 14-22). The variogram model then used in Vulcan™ for Quantitative Kriging Neighborhood Analysis (QKNA) and Estimation.

Table 14-9: D Variogram Model, with Standardized Variogram Sill

Domain	Variables	Nugget	Spherical 1				Spherical 2			
			Sill	Major	Semi	Minor	Sill	Major	Semi	Minor
1	Au g/t	0.33	0.49	12	25	10	0.18	44	47	25
1	Ag g/t	0.29	0.59	15	22	10	0.21	50	50	25

Source: OceanaGold, 2024



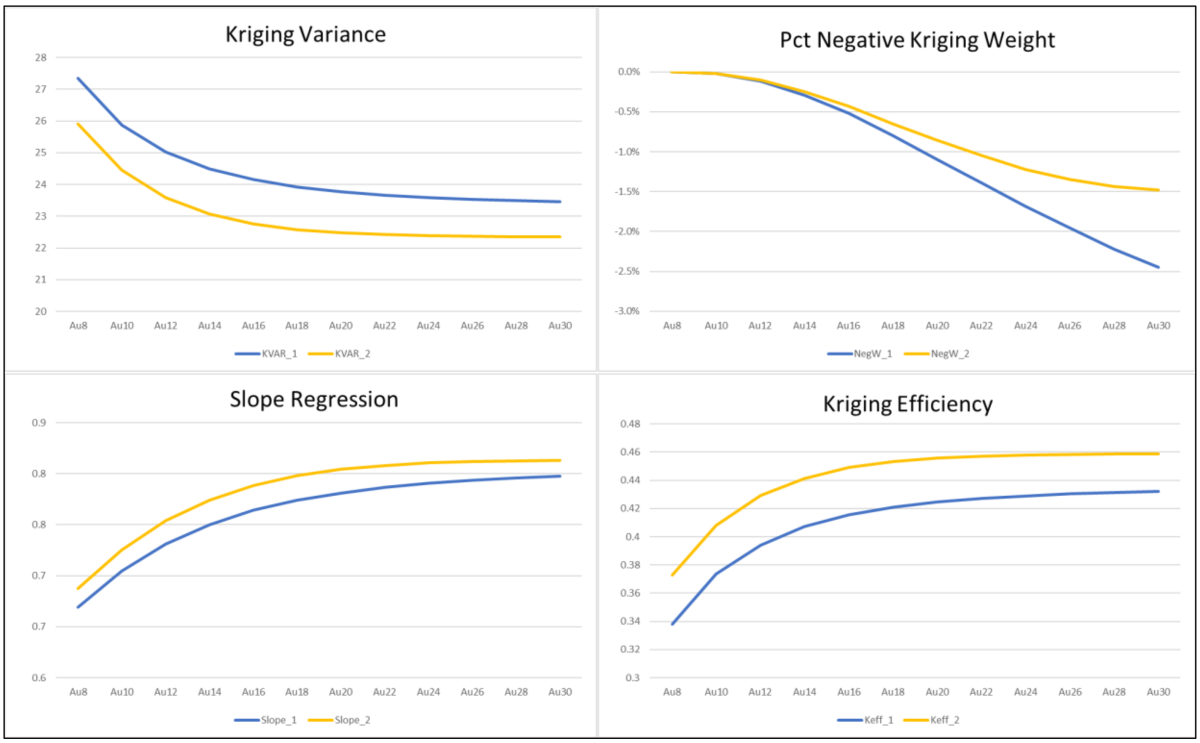
Source: OceanaGold, 2024

Figure 14-22: Variogram Model for Au (top) and Ag (bottom) within Domain 1

Horseshoe Quantitative Kriging Neighborhood Analysis (QKNA)

A quantitative kriging neighborhood analysis (QKNA) was completed in Vulcan™, the test included optimizing the maximum number of samples with / without octants. The maximum number of samples from 8 to 30 samples with increment of 2 samples was tested. A minimum number of 6 samples, with a maximum of 4 samples per drillhole and the 60 m x 30 m x 10 m search remain unchanged; a discretization of 5 x 5 x 5 was used.

The result from the QKNA suggests that using more than 16 composites will no longer improve the quality of estimate using OK; as well the use of octants is preferable (Figure 14-23. No QKNA was completed for Ag g/t due to lack of data and uncertainly of the variogram.



Source: OceanaGold, 2024

Figure 14-23: QKNA Analysis showing Octant (Yellow) vs. without Octant (Blue)

Horseshoe Block Model

The block model is rotated in Vulcan™ with a 60° bearing with 0° plunge and dip. The dimension, origin and cell size are provided in Table 14-10.

Table 14-10: Block Model Dimensions and Origin

Variable	X	Y	Z
Origin	542900	3826100	600
Minimum	0	0	0
Maximum	1250	700	580
Block Size (Parent)	10	10	10
Sub-block size	5	5	5

Source: OceanaGold, 2024

Horseshoe Estimation

A two-pass strategy was used for estimation of Au g/t and Ag g/t, with each subsequent pass search ellipse larger than the previous. A larger search was required for Ag g/t to satisfy total number of blocks estimated. Table 14-11 shows the block estimation parameters.

Table 14-11: Estimation Parameters for Au and Ag

Passes	Variables	Bearing/ Plunge/ Dip	Major	Semi	Minor	Min/ Max Samples per Estimate	Min/Max Holes per Estimate	Max Samples per Drillhole	Discretization	Sample per Octant
Au g/t	Dom 1 Pass 1	55/0/40	60	40	10	6/16	2/10	4	5x5x5	6
	Dom 1 Pass 2	55/0/40	180	160	30	4/16	N/A	4	5x5x5	6
	Dom 3 Pass 1	55/0/40	180	120	30	6/16	N/A	N/A	5x5x5	6
Ag g/t	Dom 1 Pass 1	55/0/40	180	120	30	6/20	N/A	4	5x5x2	N/A
	Dom 1 Pass 2	55/0/40	220	150	40	2/20	N/A	4	5x5x2	N/A
	Dom 3 Pass 1	55/0/40	180	120	30	6/20	N/A	N/A	5x5x2	N/A
	Dom 3 Pass 2	55/0/40	220	150	40	2/20	N/A	N/A	5x5x2	N/A

Source: OceanaGold, 2024

Horseshoe Model Validation

A variety of methods have been used to validate the final model (HA1123URR_V3.bmf), which included:

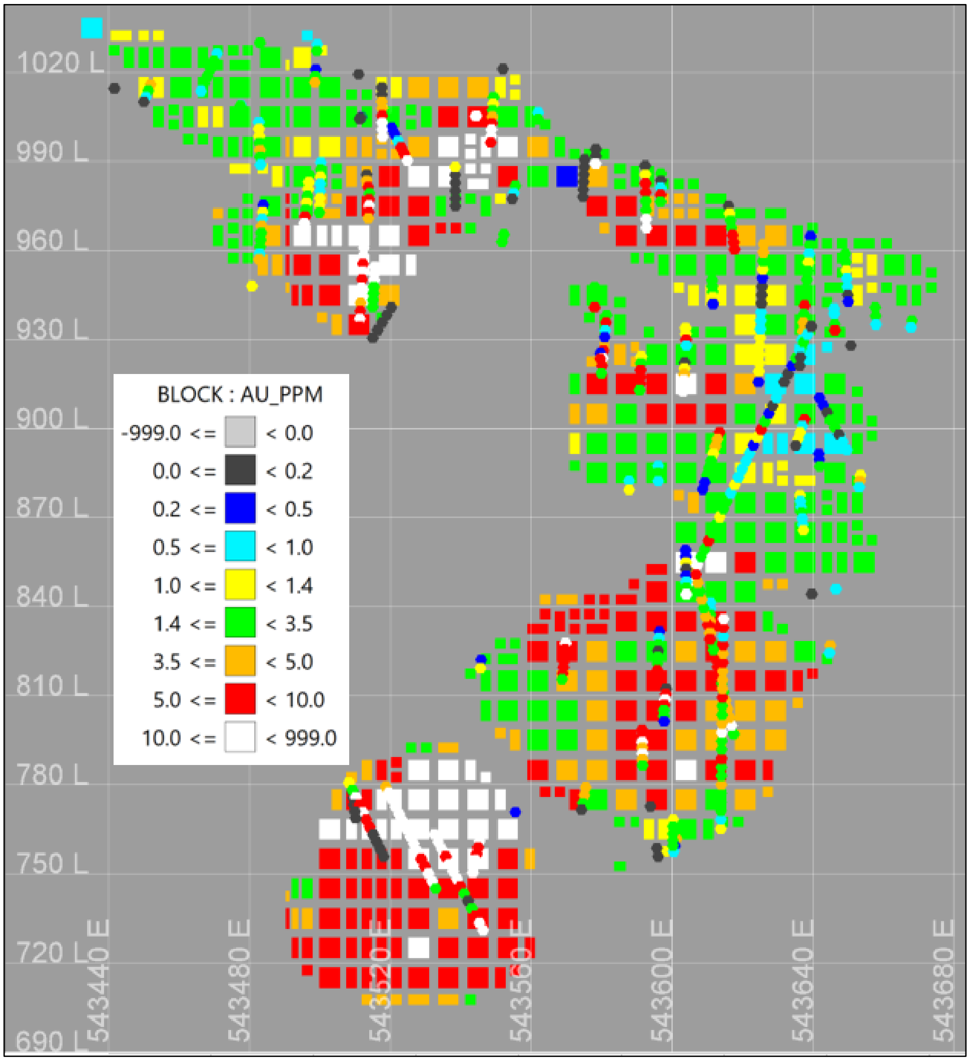
- Estimation passes statistics and visual validation (Figure 14-24)
- Statistical comparison (Table 14-12)
- A visual sectional validation of the block model with interpretation and drilling (Figure 14-24)
- Swath plots comparing the estimation with the underlying composite grades (Figure 14-25)
- Global grade and tonnage comparisons with the previous model

On this basis, the estimate is reasonable for the purposes of medium to long term estimation.

Table 14-12: Statistics Comparison between Composite vs. Block Model

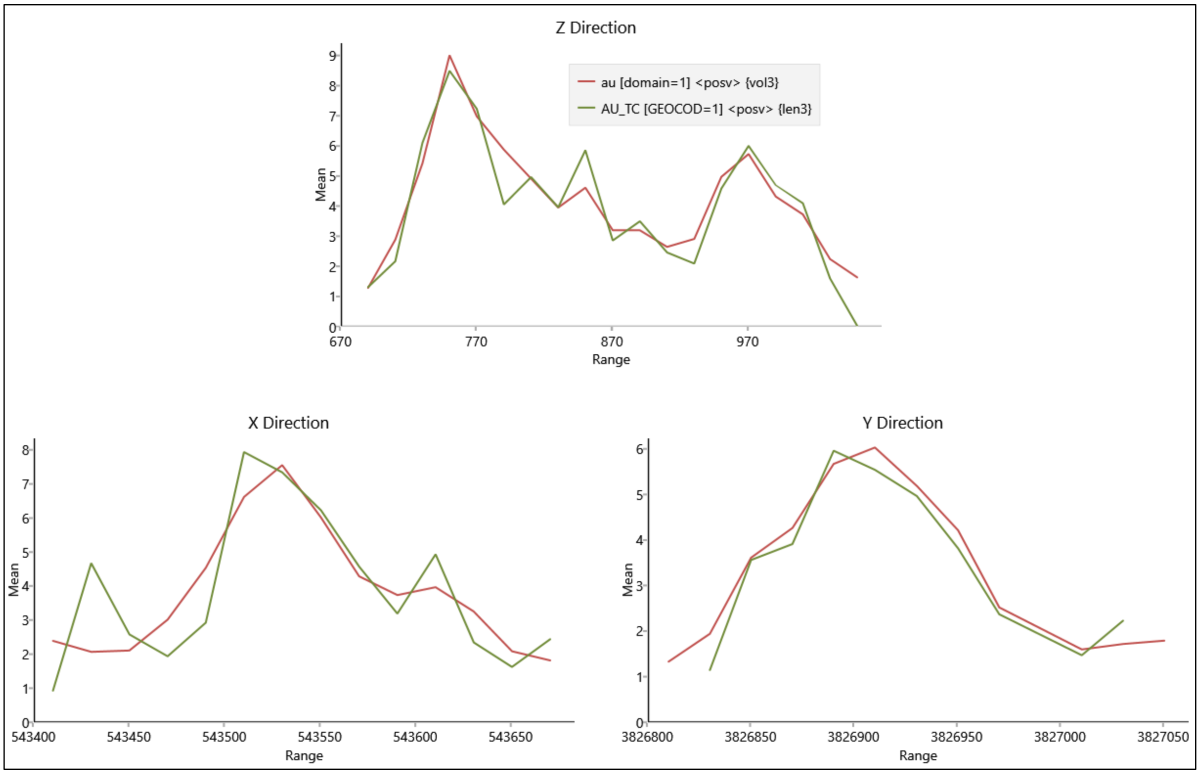
Domain	FUNCTIONS	Composite		BM		Ratio	
		Au TC	Ag TC	Au BM	Ag BM	Au g/t	Ag g/t
1	Number Of Samples	2,404	603	6,408	6,408		
	Minimum	0.004	0.005	0.430	0.385		
	Maximum	100	26.0	36.339	5.144		
	Range	99.996	25.995	35.909	4.759		
	Average	**4.51**	**2.22**	**4.52**	**1.935**	**0.3%**	**-12.9%**
	Standard Deviation	8.97	3.22	4.11	0.680		
	Variance	80.44	10.35	16.90	0.462		
	Skewness	1.99	1.45	0.00	0.001		
	Coef. Of Variance	6.04	4.55	0.91	0.351		
	Median	1.77	1.50	3.14	1.862		
3	Number Of Samples	3,167	192	11,621	11,621		
	Minimum	0.002	0.005	0.004	0.126		
	Maximum	0.4	2.2	0.391	0.751		
	Range	0.438	2.195	0.387	0.625		
	Average	**0.10**	**0.42**	**0.10**	**0.36**	**0.3%**	**-12.5%**
	Standard Deviation	0.14	0.42	0.07	0.10		
	Variance	0.02	0.18	0.00	0.01		
	Skewness	1.34	1.02	0.00	0.00		
	Coef. Of Variance	1.54	2.18	0.67	0.29		
	Median	0.04	0.25	0.09	0.34		

Source: OceanaGold, 2024



Source: OceanaGold, 2024
Cross section looking NW showing good correlation between underlying composite data vs. estimated blocks for Au g/t. The section showing estimated blocks prior to zeroing (Au and Ag) for Dike

Figure 14-24: Cross Section looking NW Showing Composite vs. Block Model



Source: OceanaGold, 2024
Swath plot for Au g/t between block model (red) vs. composite (green) weighted by length, X direction (10 m slices), Y direction (10 m slices), and Z direction (10 m slices)

Figure 14-25: Swath Plot for Au g/t between Block Model (Red) vs. Composite (Green)

In addition to various model validation, 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 mineralization. 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 mineralization. More sensitivities recommended.
- Continuously monitor the spatial and statisticall distribution of very-high grade samples. Investigate potential geological controls for extreme grades to ensure they are appropriately modeled and estimated.

<u>Horseshoe Resource Classification</u>

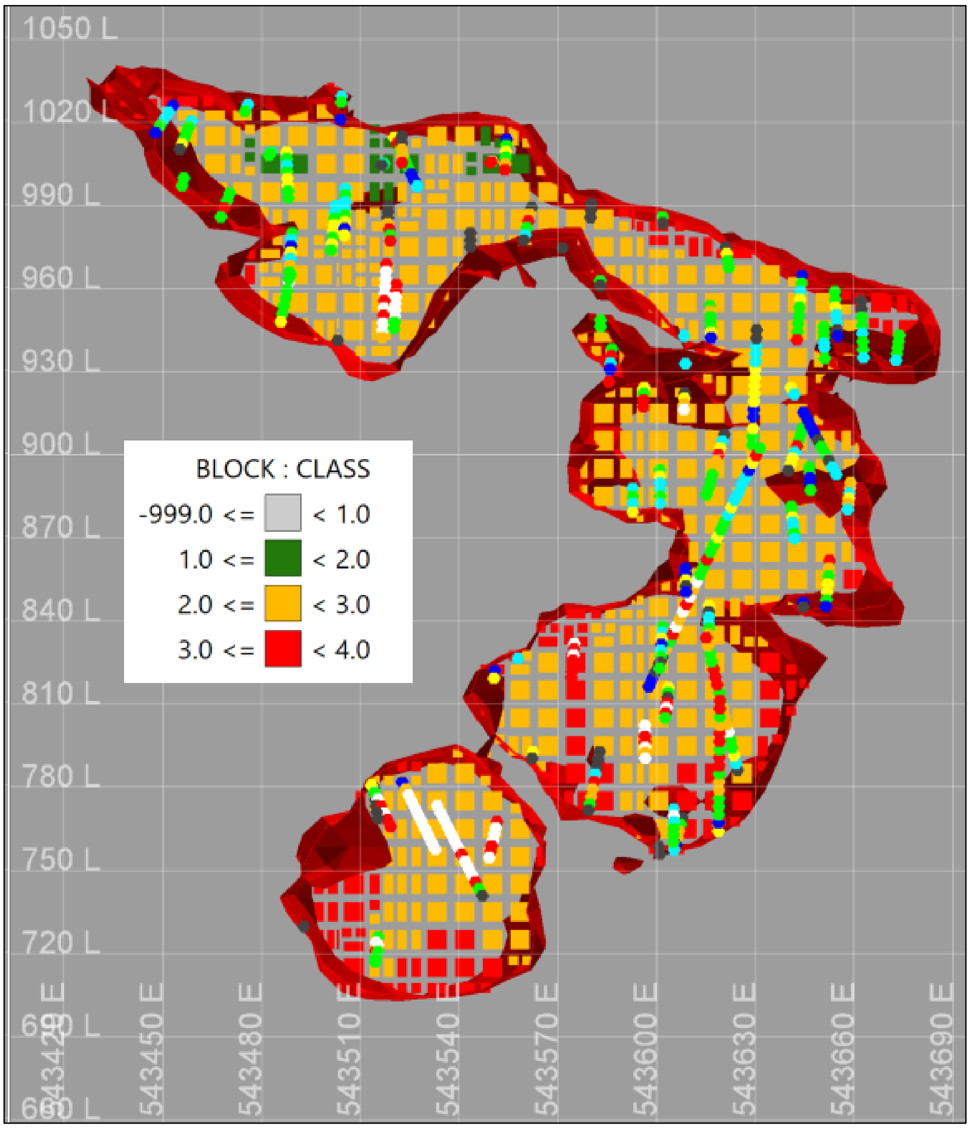
Mineral Resources are classified as Measured, Indicated and Inferred in accordance with CIM guidelines. Classification of the Mineral Resources reflects the relative confidence of the grade estimates, kriging properties and the continuity of the mineralization. This classification is based primarily on the sample spacing and geological complexity.

The resource classification consists of two stages in 2023

First, a provisional classification using wireframes / solids:

- Measured: The area is defined within approx. 15 m x 15 m UG GC drill spacing using numerical distance function in leapfrog with ~8.5 m radius.
- Indicated: The area is guided by approx. 30 m x 30 m drill spacing using numerical distance function in leapfrog with ~15 m radius.
- Inferred: The remaining blocks within Domain 1 mineralization.

Second stage is to assign a post estimation script in Vulcan™ based on lithology type as shown in (Figure 14-26).

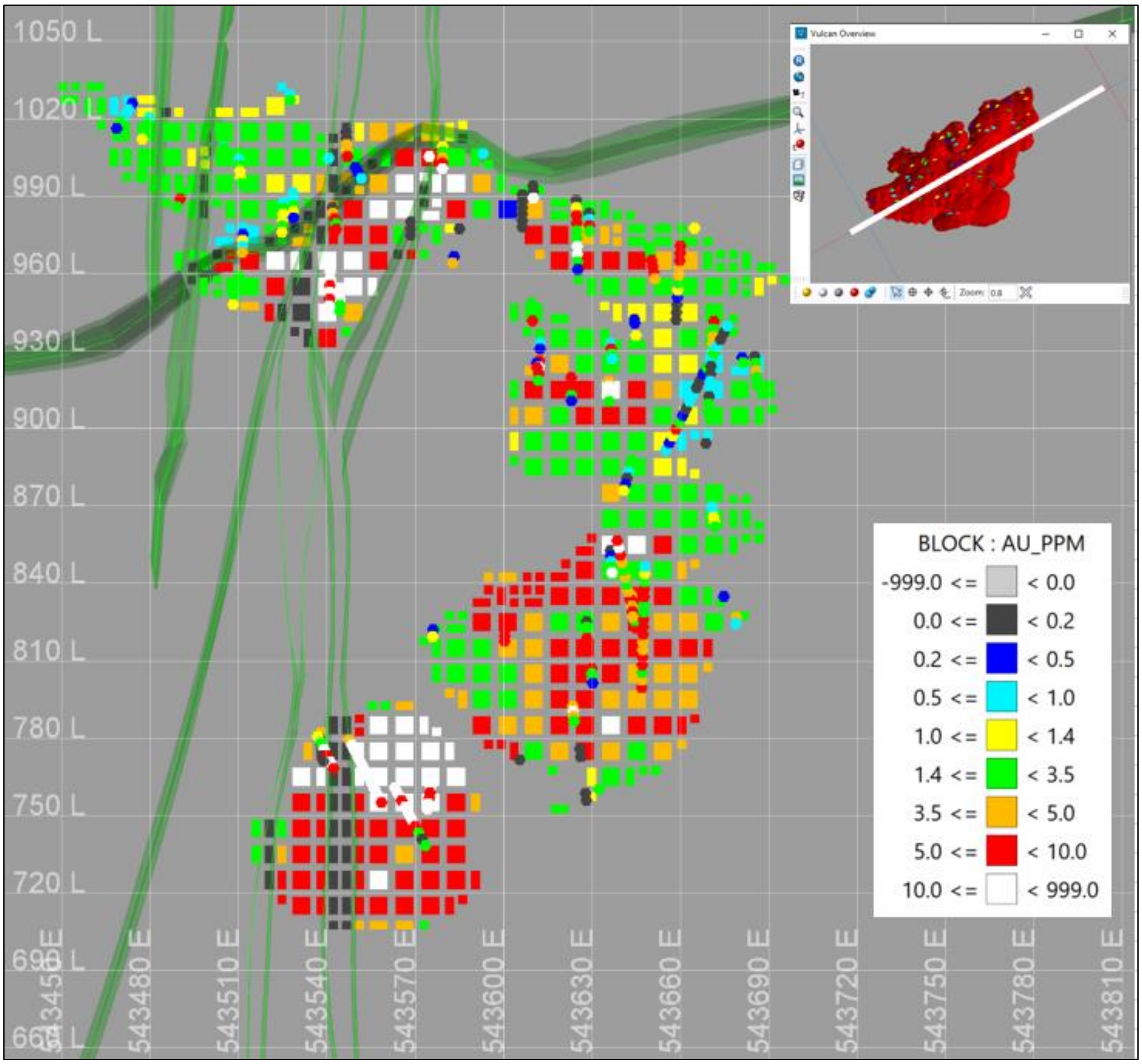


Source: OceanaGold, 2024

Figure 14-26: Cross Section Looking NW 2023 Resource Classifications

Assigned Zero Grade for Dike

Due to uncertainty on the dike interpretation, a precautionary approach was taken; all grades for model blocks with centroids within the dike wireframes were set to zero after estimation (Figure 14-27). The weakly to unmineralized dike grades however were included for grade estimation for estimation, prior to this step. The approach is acknowledged to be slightly conservative but deemed appropriate, given the geological uncertainty. Once the confidence in the interpretation improves, via underground drilling and mapping; hard boundary estimates may be implemented.



Source: OceanaGold, 2024
Cross section (in white) looking NW showing dike interpretation (green). The blocks values for Au g/t within the dike wireframe have been zeroed

Figure 14-27: Cross Section Looking NW showing Dike Interpretation

Horseshoe Mineral Resource Statement

The Horseshoe Mineral Resource statement is based on the OK model as presented in Table 14-13. A CoG of 1.55 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.No mining dilution has been applied.

Table 14-13: Horseshoe Underground Mineral Resource Statement as of December 31, 2023

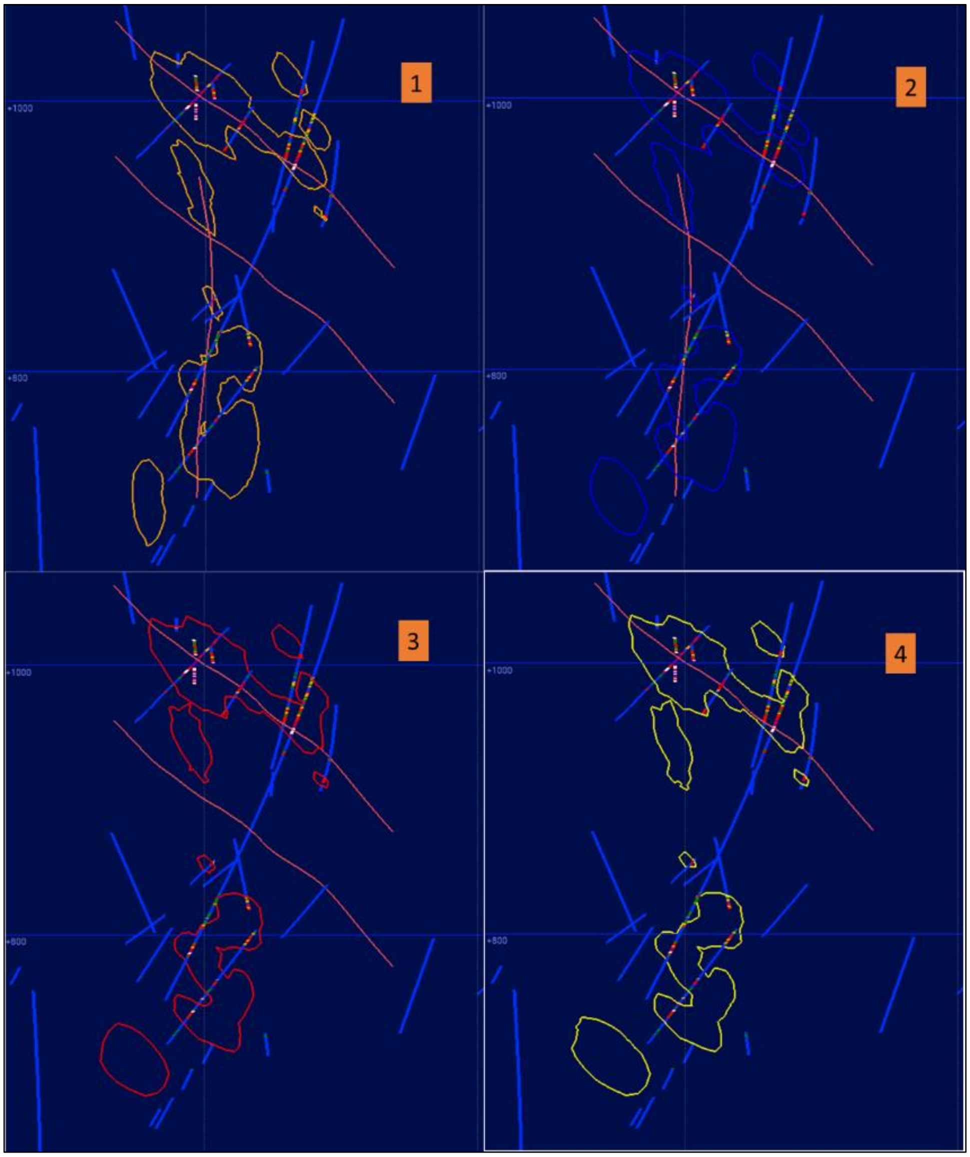
Class	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (Moz)	Contained Ag (Moz)
Measured	0.1	5.04	2.0	0.02	0.01
Indicated	3.7	5.49	2.3	0.65	0.3
Measured & Indicated	3.8	5.48	2.3	0.67	0.3
Inferred	1.8	4.10	2.2	0.20	0.1

Source: OceanaGold, 2024

- Cut-off grade 1.55 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

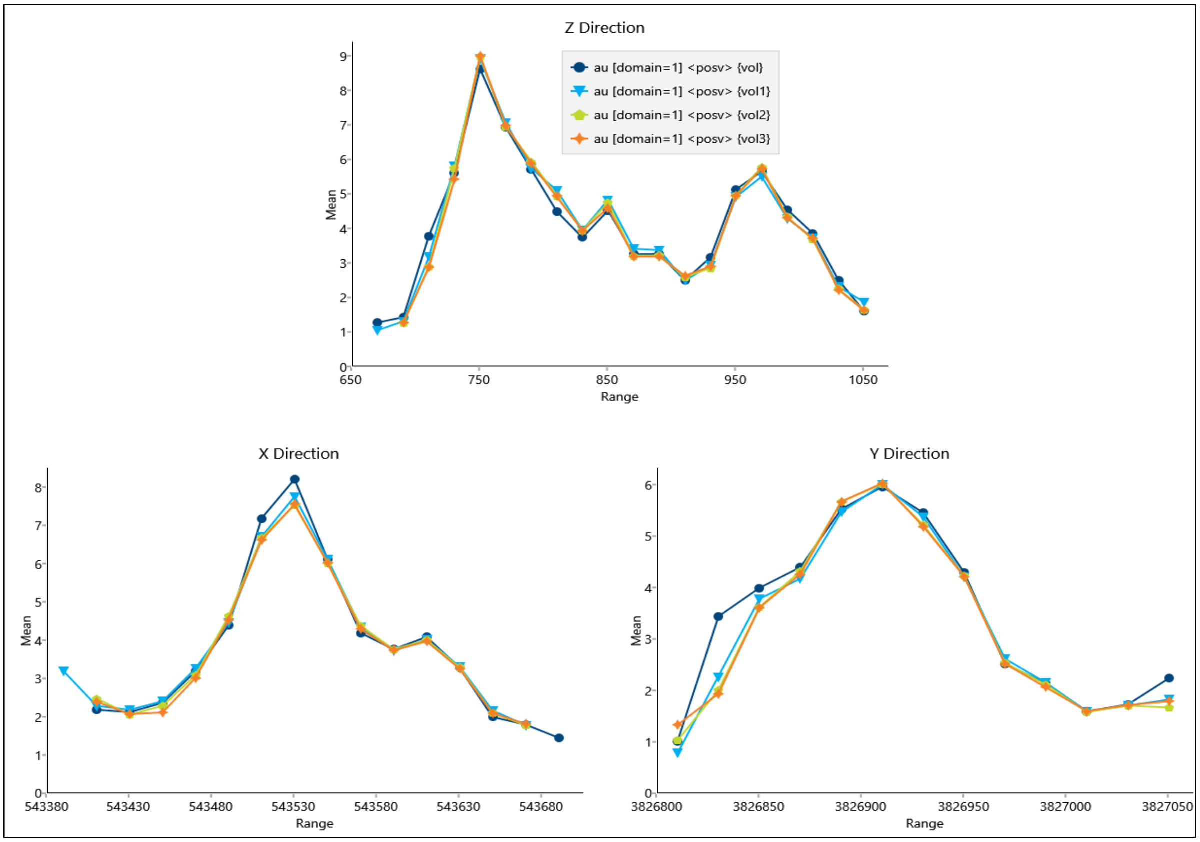
Following up on Jun Cowan structure review in September 2023, a range of estimation domaining strategies were evaluated by varying the structure trend of mineralization controls for lower part of the Horseshoe (Figure 14-28).



Source: OceanaGold, 2024
Cross section looking South-West showing several implicit models in Leapfrog using various structure trends (Maroon). 1 and 2 used similar structure trends however no.2 has softened strength for vertical trend which control the lower part of Horseshoe

Figure 14-28: Cross Section Looking South-West showing Several Implicit Models

The sensitivity estimates were validated and compared statistically and visually. The estimates showed consistency with less than 4% variation, which is not material (Figure 14-29).



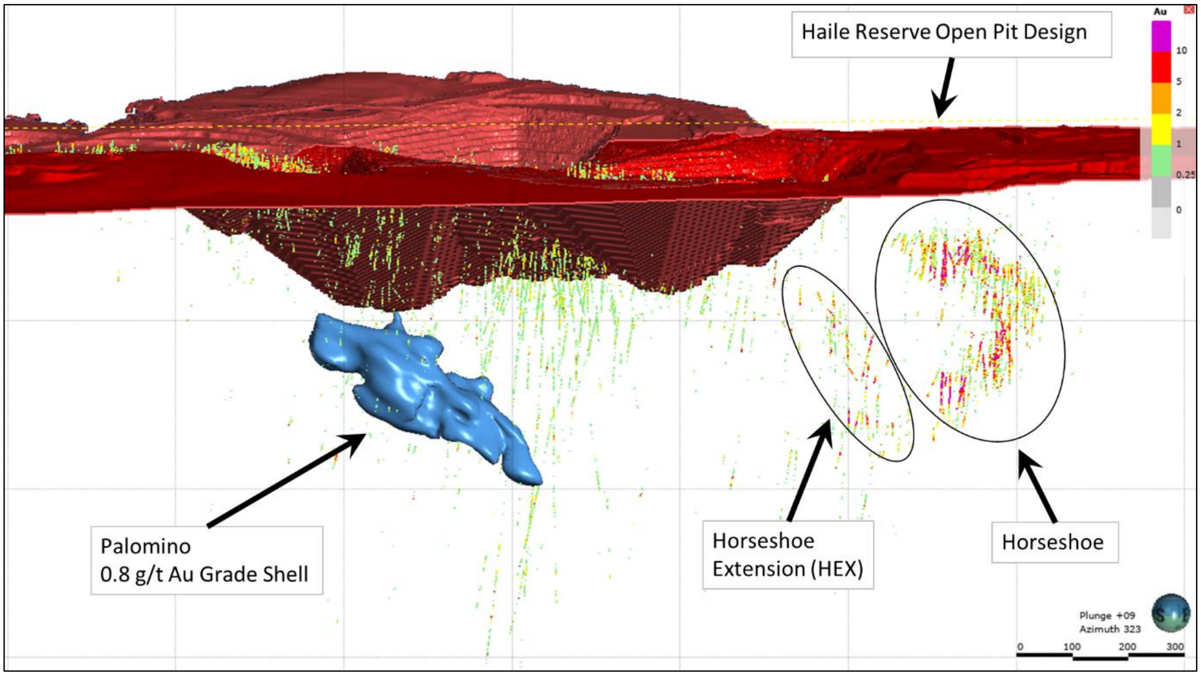
Source: OceanaGold, 2024
Sensitivity test swath plots for Au g/t showing Minimum Variation between Estimates

Figure 14-29: Sensitivity Test Swath Plots for Au g/t

14.2.2 Palomino Mineral Resource Estimate

The Palomino deposit is a medium grade deposit, located approximately 650 m southwest of Horseshoe, and 300 m below surface (Figure 14-30).

The Palomino resource estimate is based on the current drillhole database, interpreted lithologies, geologic controls and current topographic data. The cut-off date for drilling was for samples with assays received on October 11, 2023. No mining has occurred at Palomino.



Source: OceanaGold, 2023

Figure 14-30: Long-Section Looking NNW, showing Palomino Mineralization, Horseshoe, HEX and, Entire Haile Drilling Intercept Dataset Shown (colored by Au g/t)

Geologic Model and Controls on Gold Mineralization

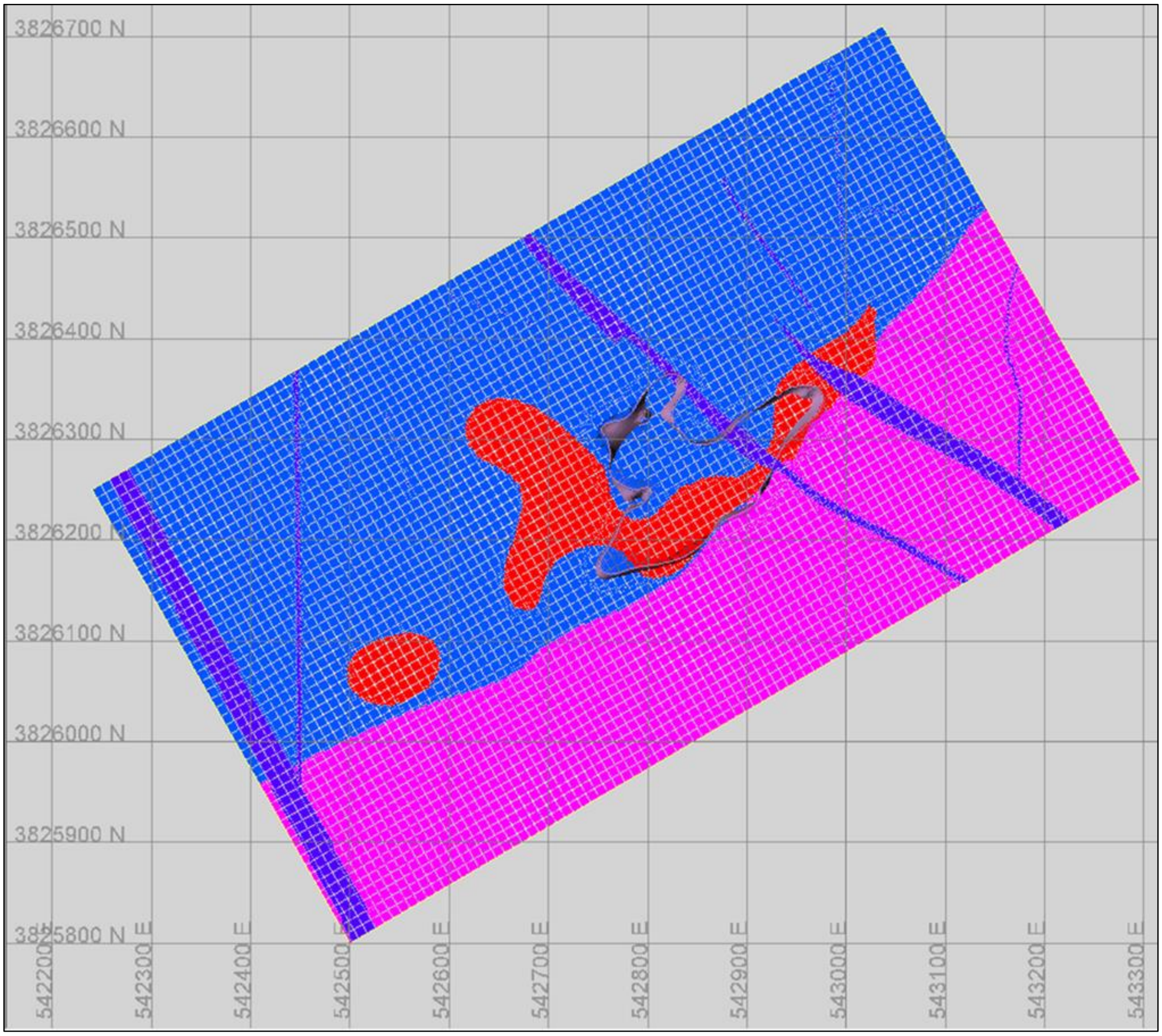
The deposit dimensions are approximately 400 m long x 70 m high x 90 m wide. Lozenge-shaped mineralized zones strike ENE, dip northwest, and plunge gently northeast. Diamond drillhole spacing ranges from 20 to 70 m.

Fine-grained gold is hosted in pyritic and silicified siltstone and intrusives along a steeply SE-dipping, ENE-striking contact with overlying dacite flows. Mineralization is locally 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, rhyodacite and diabase and lamprophyre dikes. These four rock types constitute the lithologies coded in the block model (Figure 14-31).

Gold and silver estimates were constrained within an implicitly modeled 0.8 g/t Au indicator [4]at a 25% probability of being above threshold (0.8 g/t threshold shell) created using Leapfrog® software (Version 2023.1), with appropriate trend surfaces to represent the controls on mineralization at Palomino.

[4] This approximates a 0.4 g/t Au Indicator at a 50% probability of being above threshold.

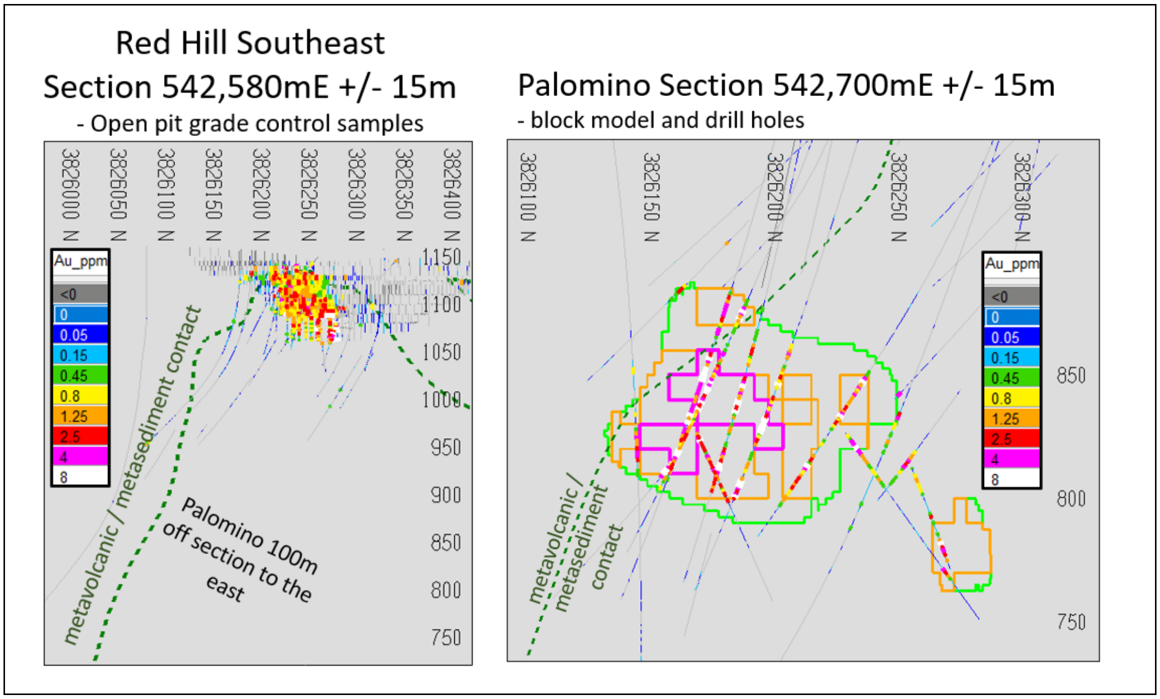


Source: OceanaGold, 2023
Blue – metasediment / pink – metavolcanic / red – rhyodacite / dk purple – diabase dikes / purple wireframe – 0.8 g/t Au threshold shell.

Figure 14-31: Plan View (725 mRL) of Lithological Relationship

In 2018, prior to being mined in the open pit, Red Hill SE was identified as a good analogue for Palomino. Red Hill is near Palomino, and like Palomino, is located on a steep SE 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. Red Hill Grade Control (left) Palomino Resource Drilling (right), relative to the Metavolcanic / Metasediment contact. Figure 14-32 shows a cross-section through both Red Hill and Palomino, which shows a shallow north-west dipping structural component to the control on mineralization.



Source: OceanaGold, 2022
Red Hill Grade Control (left) Palomino Resource Drilling (right), relative to the Metavolcanic / Metasediment contact.

Figure 14-32 Cross-Section of Red Hill Grade Control Drilling and Palomino Resource Drilling

Bulk Density

In situ density determinations have been carried out at regular intervals on drill core samples using the immersion method, which involves weighing the sample both in air and in water. The measurements are then averaged for each lithology.

Model in situ dry bulk densities (BD) are based on domain averages, as shown in Table 14-14. The BD was assigned for each lithology type (all lithologies within model area are fresh).

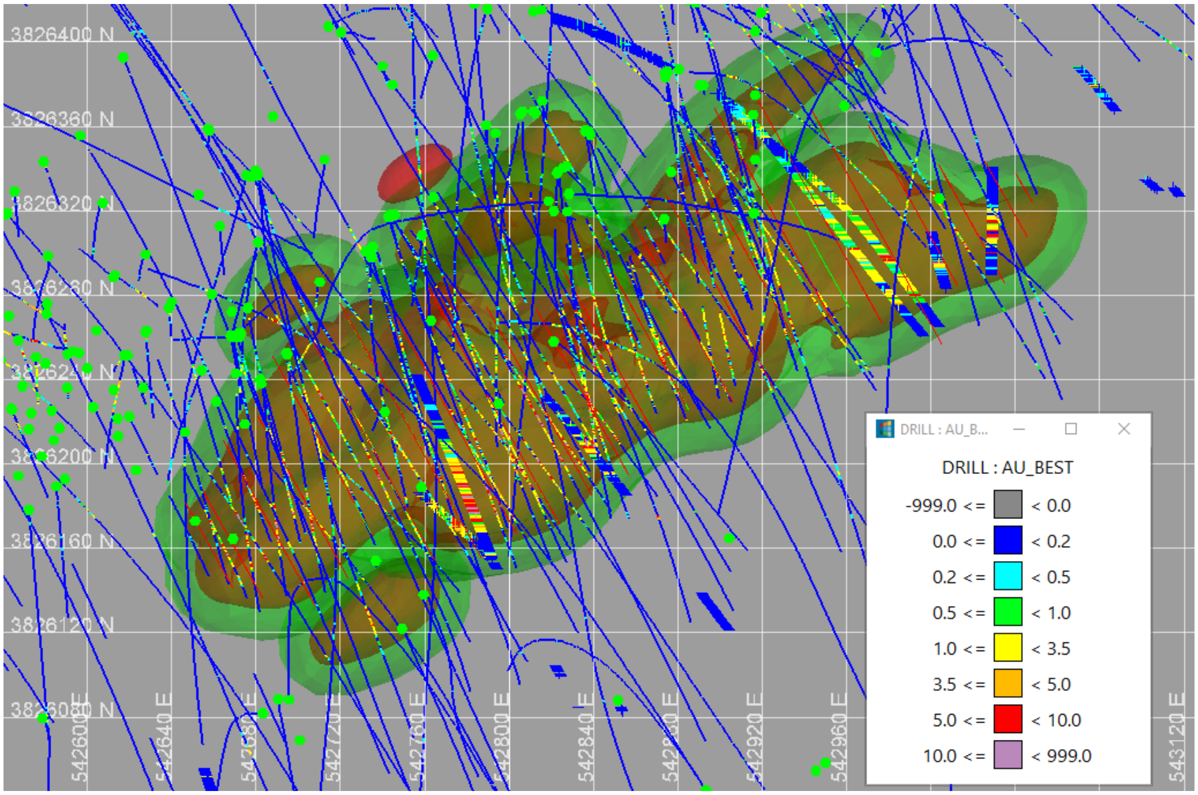
Table 14-14: Densities Assigned in the Block Model via Lithology

Bulk Density Assignment Criteria			
Diabase Dike	**Meta Volcanics**	**Rhyodacite**	**Meta Sediments**
2.88	2.7	2.77	2.78

Source: OceanaGold, 2023

Grade Boundary Domaining

Gold and silver estimation was constrained within an implicitly modeled wireframes using Leapfrog® software, using a 0.8 g/t Au threshold at a 25% probability, with appropriate trend surfaces to represent the controls on mineralization at Palomino. A dilution shell was also added to the model, to account potential mining dilution. A hard domain was generated by expanding the 0.8 g/t Au threshold wireframe by 15 m. The two domains used in the estimate are shown in Figure 14-33.



Source: OceanaGold, 2023

Figure 14-33: Plan View of the 0.8 g/t Au Domain (red) and the Dilution Wireframe (green)

Compositing and Top Capping

Compositing was completed in Vulcan™ software to 3 m downhole lengths with no breaks at domain contacts. The 3 m length was chosen to reflect the low degree of mining selectivity and the parent block size used. The merge function was used, where intervals less than or equal to 1.5 m are merged with the adjacent sample, resulting in lengths ranging from 1.5 to 4.5 m with a mean of 3 m.

Statistical analysis of the composite data for Au and Ag has resulted in a capping value of 28 g/t Au and 15 g/t Ag for 0.8 g/t Au threshold shell. Capping analysis for the dilution domain resulted 4.5 g/t Au and 2.5 g/t Ag. Table 14-15 summarizes the length weighted statistics of 3 m composites within the 0.8 g/t Au threshold shell (pug_0p8=1) and dilution domain (pug 0p8=3).

Table 14-15 Basic Statistics for 3 m Composites by Domain

Variable	Domain	Count	Minimum	Maximum	Mean	Variance	CV
AU_PPM	pug_0p8=1	2475	0.003	39.8	2.5	12.4	1.4
	pug_0p8=3	1943	0.003	37.2	0.2	1.0	4.2
AU_CUT	pug_0p8=1	2475	0.003	28	2.5	11.7	1.4
	pug_0p8=3	1943	0.003	4.5	0.2	0.2	1.9
AG_PPM	pug_0p8=1	599	0.029	22.2	2.5	8.8	1.2
	pug_0p8=3	174	0.007	10.7	0.6	1.3	2.0
AG_CUT	pug_0p8=1	599	0.029	15	2.4	7.7	1.1
	pug_0p8=3	174	0.007	2.5	0.5	0.3	1.2

Source: OceanaGold, 2023

Block Model Generation

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 14-16.

Table 14-16: Palomino Block Model Dimensions and Origin

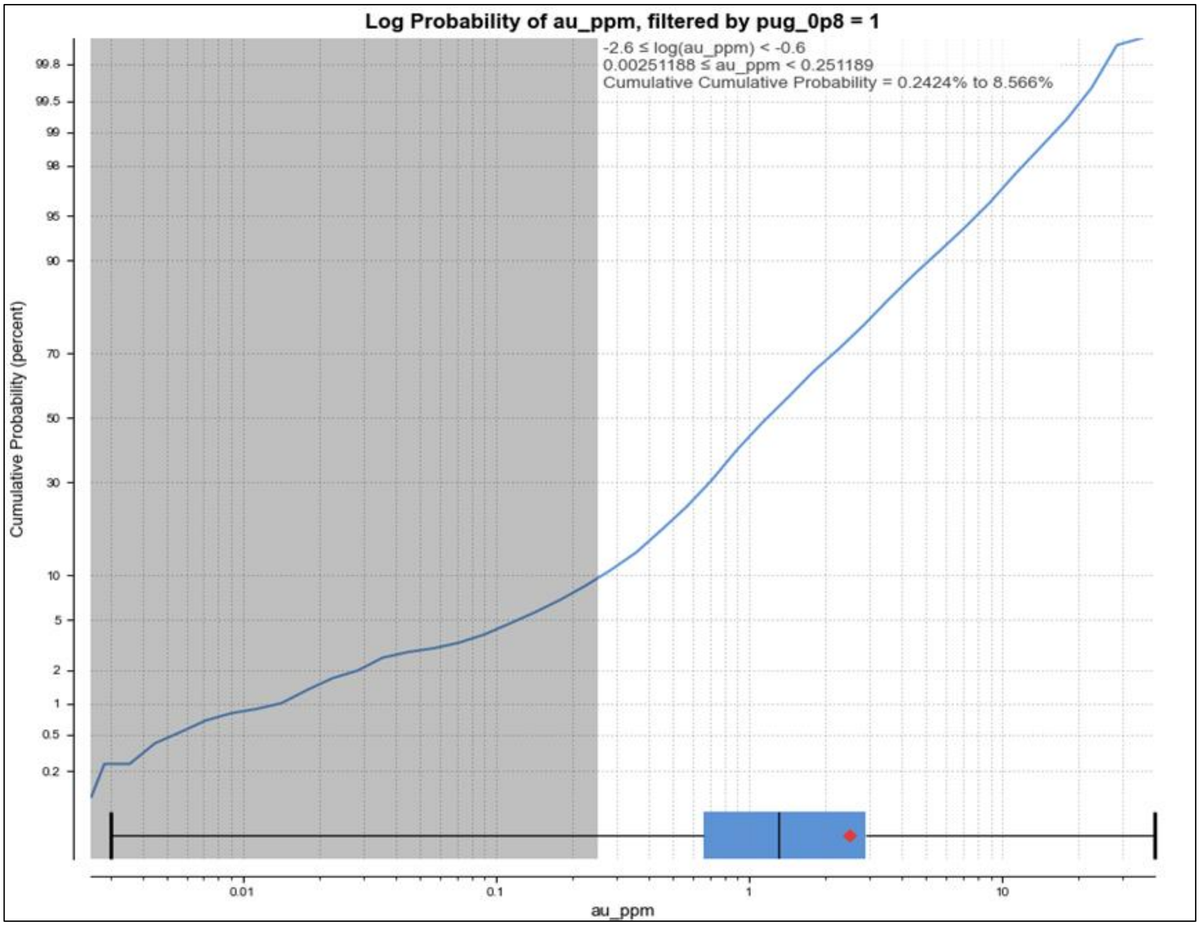
Variable	X	Y	Z
Rotation	060º		
Origin	542,500	3,825,800	250
Length	920	520	750
Block Size (Parent)	10	10	10
Sub-block size	2.5	2.5	2.5
No. of Blocks (Parent)	92	52	75

Source: OceanaGold, 2023

Palomino Estimation Methodology

Gold (Au) Estimation

Examination of the data suggested a broad bimodal distribution of grade within the 0.8 g/t Au threshold shell, (Figure 14-34), the log-probability plot shows the two grade populations, above and below a 0.25 g/t Au threshold (grey area shows population below 0.25 g/t Au).



Source: OceanaGold, 2023

Figure 14-34: Log-Probability Plot of 3 m Composite Data within 0.8 g/t Au Threshold Shell

A probability kriging methodology was used to separate the higher grade (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 0.25 g/t Au indicator, is shown in Table 14-17.

Table 14-17: Statistics of Top-Cut 3 m Composite Data within 0.8 g/t Au Threshold – High / Low Indicator Domains

Variable	Domain	Probability (% data)	Count	Minimum	Maximum	Mean	Variance	CV
AU_CUT	LG_ind_0.25=0	8.5%	211	0.003	0.2	0.11	0.01	0.69
AU_CUT	HG_ind_0.25=1	91.5%	2,264	0.25	28.00	2.70	12.21	1.29

Source: OceanaGold, 2023

Gold grades were estimated into 10 m E x 10 m N x 10 m RL parent blocks with Vulcan™ modeling software using OK 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 resolution, estimation was into the parent block. Search parameters were optimized via Kriging Neighborhood Analysis (KNA). LG Probability domain (within 0.8 g/t Au threshold shell) KNA optimization example is shown in Figure 14-35.

Example variograms of LG probability and LG Au Indicator - Au grade, are presented in Figure 14-36 and Figure 14-37 respectively, The OK parameters used and estimation search strategy used are summarized in Table 14-18 and Table 14-19 respectively.

Metasediment, metavolcanic and rhyodacite 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 0.25 g/t Au Indicator for data within the 0.8 g/t Au threshold shell
- Search orientation essentially parallel to the plane of gold continuity and matches the variogram orientation
- Estimate LG indicator probability (LG ind prob). Calculate HG indicator probability (HG ind prob), where HG ind prob = 1 – LG ind prob
- Build a variogram for the Au grade in the LG and HG domains
- Estimate Au grade for HG and LG Domain
- Limit data to four 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))

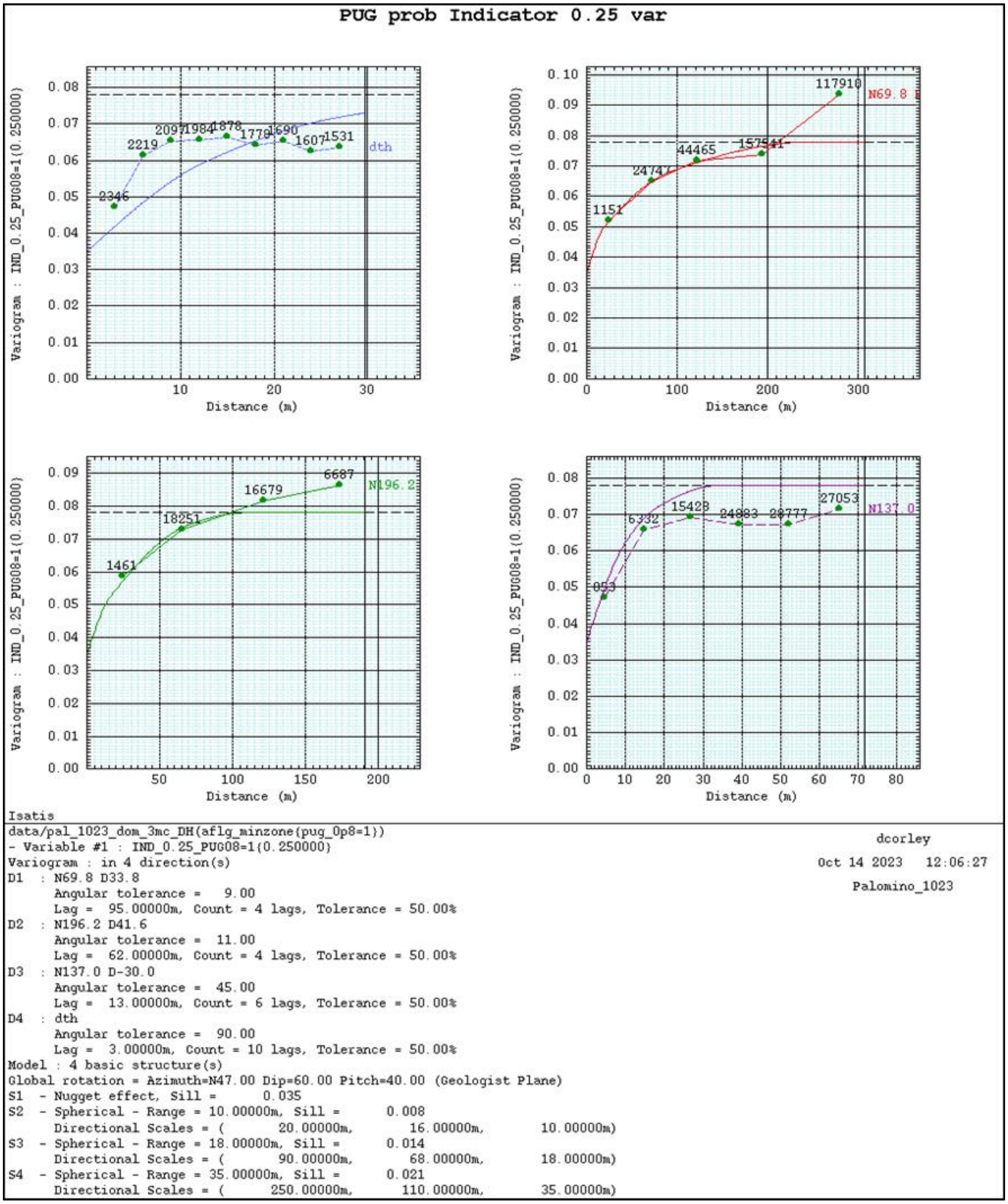
Kriging Neighbourhood Analysis -1 Sector 4 samp per hole) AU PAL PR IND dom1 10x10x10 - no LVA applied

# Angular Sectors	Min # Samp	Opt. # Sam Per Sector	# Samples per Hole	Search U	Search V	Search W	# Cells	OK Grade Mean	OK.Std Mean	# Samples	Sample Mean Dist	Sum P. Wts Max	Lagrange	Var Z(x)* Mean	Covar Z(x)|Z(x)* Mean	Corr Z(x)|Z(x)* Mean	Slope Z(x)|Z(x)* Mean	KV	Neg weight
1	5	7	4	250	100	40	163320	0.139	0.21	6.7	72.2	1.000	-0.02300	0.03000	0.00700	0.23	0.23	0.0449	0.00%
1	5	9	4	250	100	40	163320	0.143	0.20	8.3	76.1	1.000	-0.01900	0.02600	0.00700	0.24	0.26	0.0416	0.00%
1	5	10	4	250	100	40	163320	0.144	0.20	9.0	78.0	1.000	-0.01900	0.02500	0.00700	0.24	0.27	0.0408	0.00%
1	5	11	4	250	100	40	163320	0.144	0.20	9.7	79.6	1.000	-0.01800	0.02500	0.00600	0.24	0.27	0.0404	0.00%
1	5	12	4	250	100	40	163320	0.144	0.20	10.4	80.9	1.000	-0.01800	0.02400	0.00600	0.24	0.28	0.0400	0.00%
1	5	13	4	250	100	40	163320	0.146	0.20	11.0	82.4	1.000	-0.01700	0.02300	0.00600	0.24	0.29	0.0392	0.00%
1	5	14	4	250	100	40	163320	0.146	0.20	11.6	83.7	1.000	-0.01700	0.02300	0.00600	0.25	0.30	0.0388	0.00%
1	5	15	4	250	100	40	163320	0.146	0.20	12.2	84.9	1.000	-0.01600	0.02300	0.00600	0.25	0.30	0.0388	0.00%
1	5	16	4	250	100	40	163320	0.147	0.20	12.8	85.9	1.000	-0.01600	0.02200	0.00600	0.25	0.30	0.0384	0.00%
1	5	18	4	250	100	40	163320	0.148	0.20	13.87	87.93	1.003	-0.01500	0.02200	0.00600	0.249	0.314	0.0380	-0.30%
1	5	20	4	250	100	40	163320	0.148	0.19	14.9	89.6	1.007	-0.01500	0.02100	0.00600	0.25	0.32	0.0376	-0.70%
1	5	22	4	250	100	40	163320	0.149	0.19	15.8	91.1	1.017	-0.01500	0.02100	0.00600	0.25	0.33	0.0372	-1.70%
1	5	24	4	250	100	40	163320	0.150	0.19	16.7	92.4	1.020	-0.01500	0.02100	0.00600	0.25	0.33	0.0372	-2.00%
1	5	26	4	250	100	40	163320	0.151	0.19	17.6	93.6	1.021	-0.01500	0.02100	0.00600	0.25	0.34	0.0369	-2.10%
1	5	28	4	250	100	40	163320	0.151	0.19	18.4	94.7	1.027	-0.01400	0.02000	0.00600	0.25	0.34	0.0369	-2.70%
1	5	30	4	250	100	40	163320	0.152	0.19	19.2	95.7	1.034	-0.01400	0.02000	0.00600	0.26	0.34	0.0369	-3.40%
1	5	32	4	250	100	40	163320	0.152	0.19	19.9	96.5	1.039	-0.01400	0.02000	0.00600	0.26	0.35	0.0369	-3.90%
1	5	34	4	250	100	40	163320	0.152	0.19	20.6	97.3	1.041	-0.01400	0.02000	0.00600	0.26	0.35	0.0365	-4.10%

Rotation 50|68|38 Geo | Discretisations 5:5:5

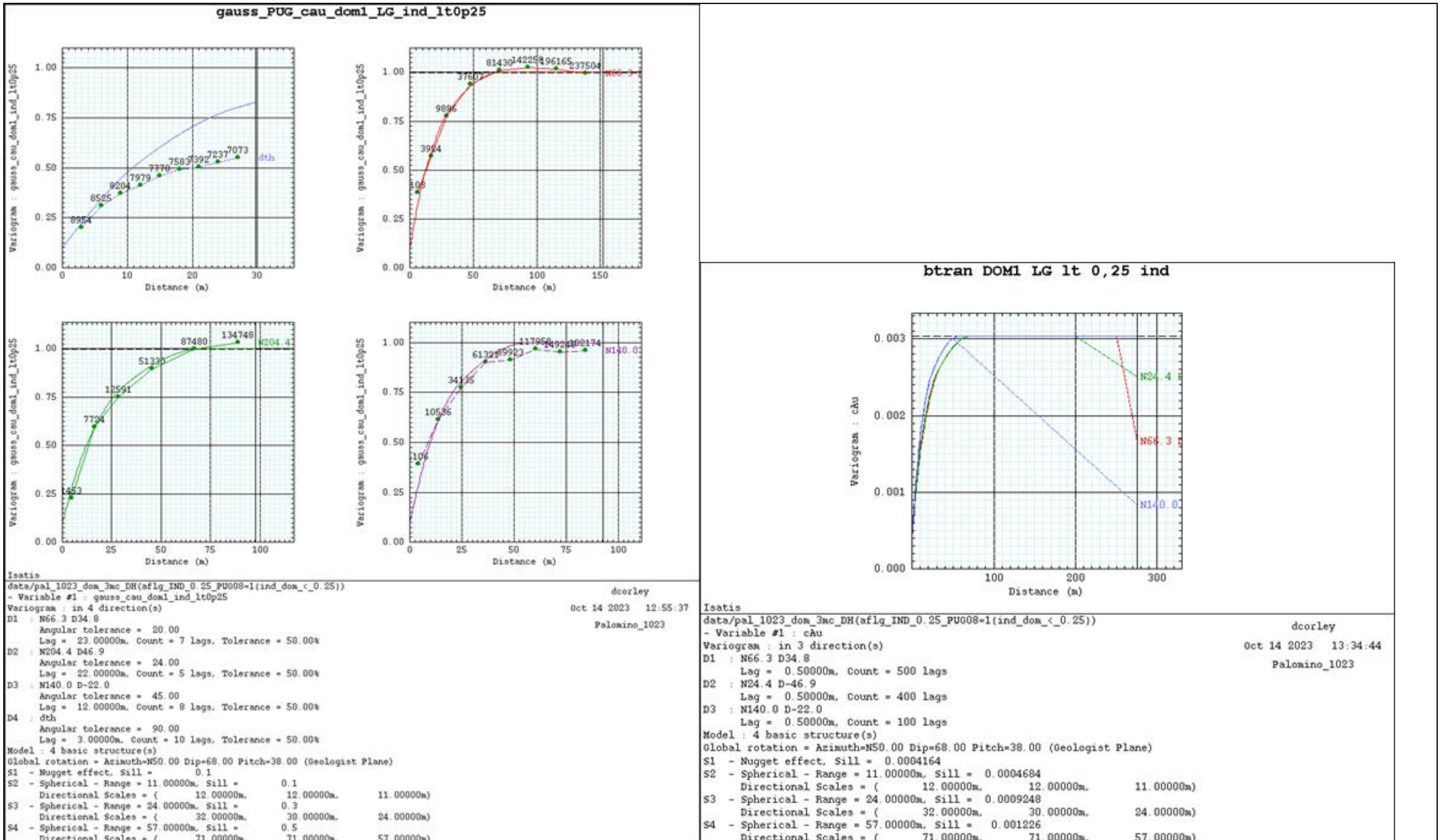
Source: OceanaGold, 2023

Figure 14-35: KNA Optimization – LG Probability Domain (blue line shows optimum samples)



Source: OceanaGold, 2023
(top left) – downhole / (top right) – Major / (bottom left) – Semi-Major / (bottom right) – Minor

Figure 14-36: Variogram_0.25 g/t Au Probability Indicator within 0.8 g/t Au Threshold Shell



Gaussian Variogram (LEFT) - (top left) – downhole / (top right) – Major / (bottom left) – Semi-Major / (bottom right) – Minor
Back-transformed Variogram (RIGHT)

Source: OceanaGold, 2023

Figure 14-37: Gaussian/Back Transformed Variogram – LG Au Indicator, Au Grade

Table 14-18: Palomino OK Parameters

Estimation Domain	Variogram Structure	Nugget	Sill Differential	Rotations (Vulcan™, Bearing, Plunge, Dip)	Ranges (m) (M, SM, Min)
LG ind Prob	1st Spherical	0.449	0.359	66.3 °, -34.8°, -62.9°	20,16,10
	2nd Spherical		0.180	66.3 °, -34.8°, -62.9°	90,68,18
	3rd Spherical		0.195	66.3 °, -34.8°, -62.9°	250,110,35
HG Au Grade	1st Spherical	0.300	0.315	66.3 °, -34.8°, -62.9°	12,10,9
	2nd Spherical		0.212	66.3 °, -34.8°, -62.9°	57,38,35
	3rd Spherical		0.173	66.3 °, -34.8°, -62.9°	220,105,53
LG Au Grade	1st Spherical	0.137	0.154	66.3 °, -34.8°, -62.9°	12,12,11
	2nd Spherical		0.305	66.3 °, -34.8°, -62.9°	32,30,24
	3rd Spherical		0.404	66.3 °, -34.8°, -62.9°	71,71,57
Au Grade (Domain 1) OK	1st Spherical	0.324	0.391	66.3 °, -34.8°, -62.9°	13,10,7
	2nd Spherical		0.146	66.3 °, -34.8°, -62.9°	33,20,16
	3rd Spherical		0.139	66.3 °, -34.8°, -62.9°	145,38,32
Dilution Shell (Domain 3)	1st Spherical	0.277	0.478	66.3 °, -34.8°, -62.9°	12,10,10
	2nd Spherical		0.139	66.3 °, -34.8°, -62.9°	30,24,23
	3rd Spherical		0.106	66.3 °, -34.8°, -62.9°	125,45,50

Source: OceanaGold, 2023

Table 14-19: LG Domain Probability and Au Grade Estimation Search Distances

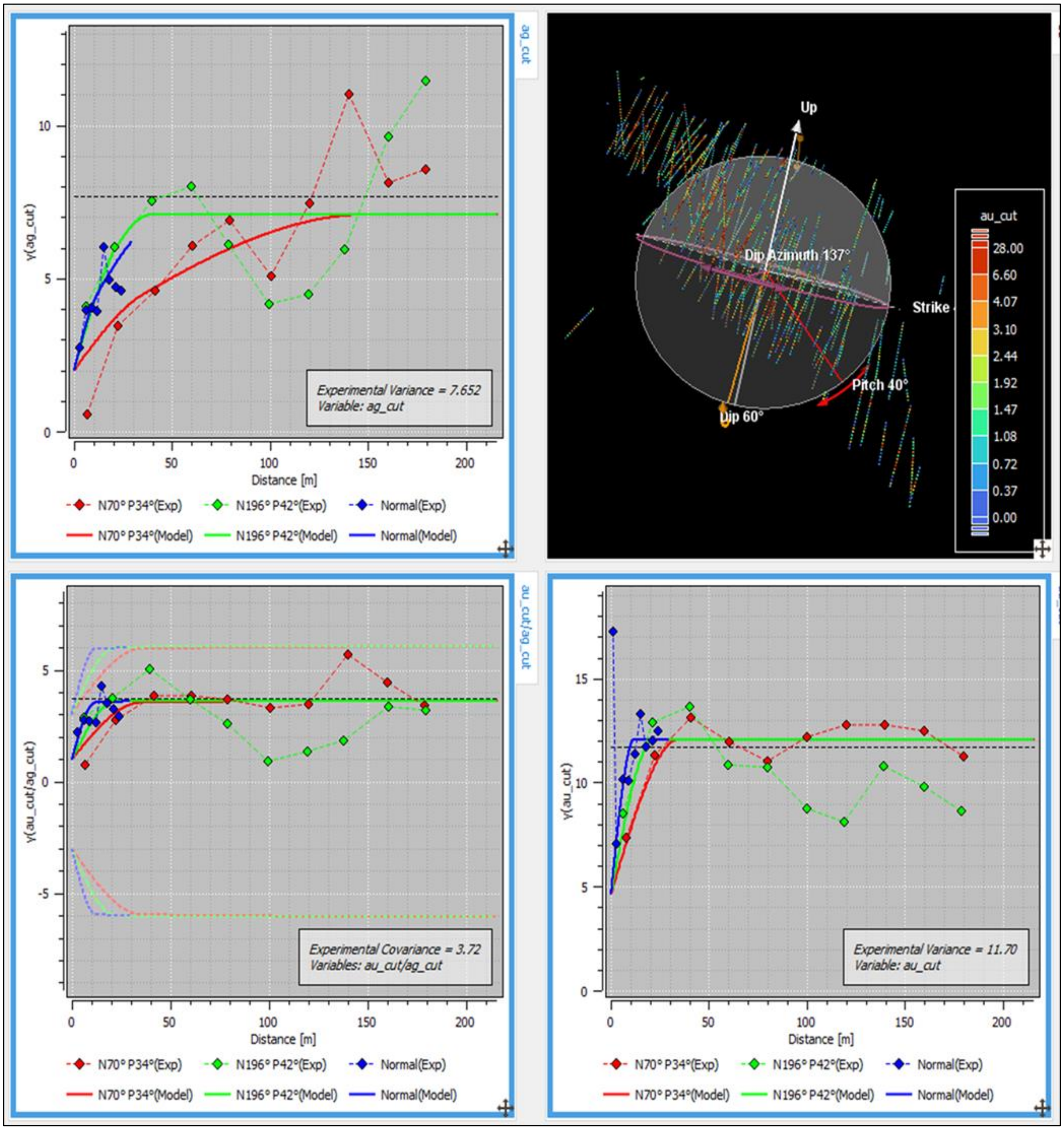
Estimation Domain	Estimation Pass	Min Sample	Max Sample	Max. Sample / DH	Search Range (m) (M, SM, Min)
LG Ind Prob	1	7	20	4	200,100,30
	2	4	20	N/A	400,200,150
HG Au Grade	1	9	18	4	200,100,50
	2	4	18	N/A	400,200,150
LG Au Grade	1	5	15	4	80,80,50
	2	3	15	N/A	300,140,110
Au Grade (Domain 1) OK	1	7	18	4	150,50,30
	2	4	18	N/A	300,100,100
Dilution Shell (Domain 3)	1	7	20	4	120,50,40
	2	4	20	N/A	250,150,100

Source: OceanaGold, 2023

Silver (Ag) Estimation

Silver grades for 0.8 g.t Au threshold shell (Domain 1) were estimated into parent blocks built within Isatis-Neo modeling software using Ordinary Co-Kriging (COK) on 3 m composites. Only 773 silver composites were available compared to 4,418 for gold. The COK estimates leverage the spatial correlation between Au vs. Ag using cross-variogram (Figure 14-38) to estimate at locations where gold assays were present, but no silver assays were available.

Estimated Parent block Ag grades from the Isatis-Neo block model were copied into the final Vulcan™ block model. A small skin of sub-blocks at the edge of Domain 1 were un-estimated, and OK was used to populate this region. OK was also used for Domain 3 (dilution) Ag estimation.



Source: OceanaGold, 2023
(top left) – Experimental Au Variogram (Major red / SM Green/ Min Blue) / (top right) – Rotation used for variogram / (bottom left) – Experimental CoK Variogram Au|Ag / (bottom right) – Experimental Ag Variogram

Figure 14-38: Co-Kriging Variogram Au | Ag – Domain = 1 – used for Ag Estimate

Palomino Mineral Resource Classification

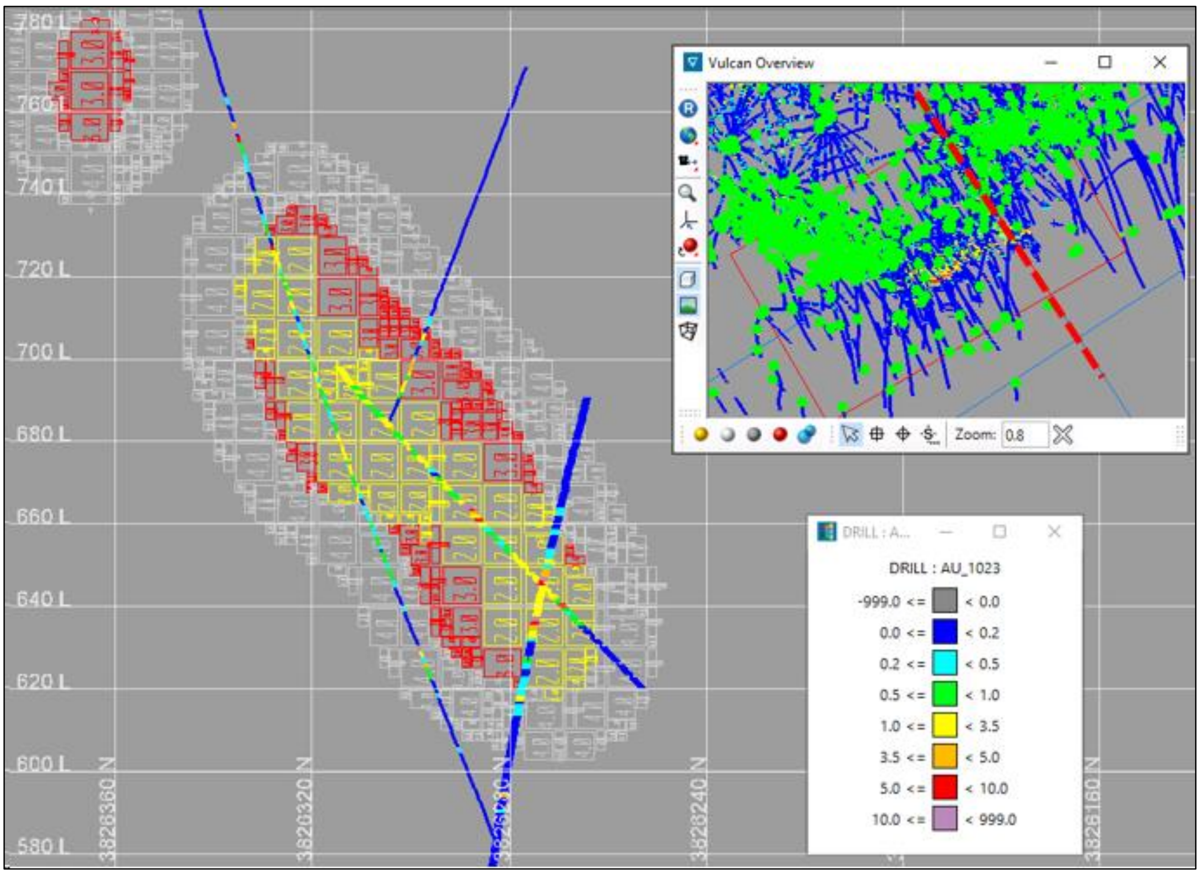
Mineral Resources are classified as Indicated and Inferred in accordance with CIM guidelines. There are no Measured Resources. Classification of the Mineral Resources reflects the relative confidence of the grade estimates and the continuity of the mineralization. This classification is based primarily 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 most 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 25m or less. This 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.

There were three isolated areas outside the main mineralized domain, which were defined by limited data, but were within the 0.8 g/t Au threshold shell used. These areas were set as unclassified.

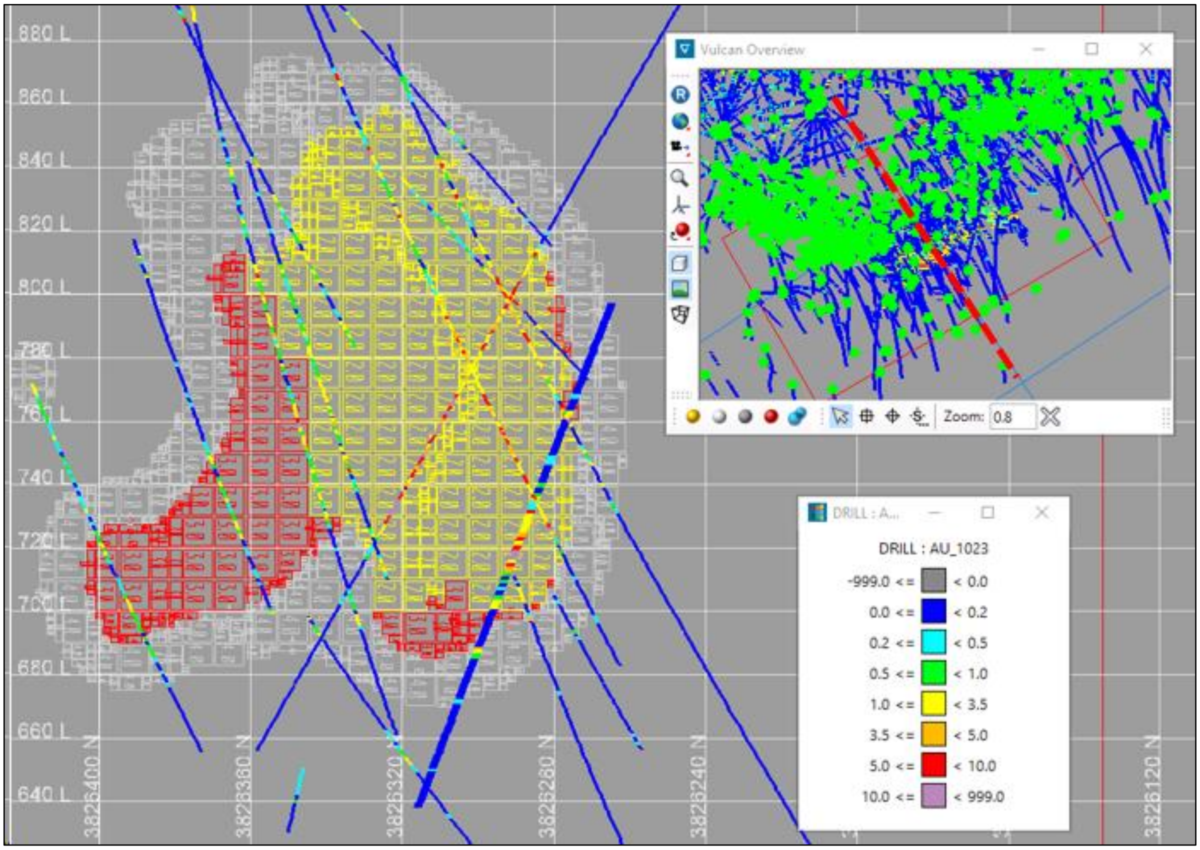
Also, any Indicated blocks within the Metavolcanics were downgraded to Inferred (in keeping with the classification process used at the UG Horseshoe deposit), this step resulted in approximately 5 koz of gold being reclassified from Indicated to Inferred.

The dilution shell (15m expansion of the mineralized domain) was also estimated but was left as unclassified (Figure 14-39 and Figure 14-40).



Source: OceanaGold, 2023
Yellow – Indicated / Red - Inferred / Grey – unclassified. Section location shown in plan-view window – red dashed line.

Figure 14-39: Oblique Section (+/- 15 m window) showing Blocks Coded by Classification



Source: OceanaGold, 2023
Yellow – Indicated / Red - Inferred / Grey – unclassified. Section location shown in plan-view window – red dashed line.

Figure 14-40: Oblique Section (+/- 10 m) showing Blocks Coded by Classification

<u>Palomino Model Validation</u>

<u>Au Estimate</u>

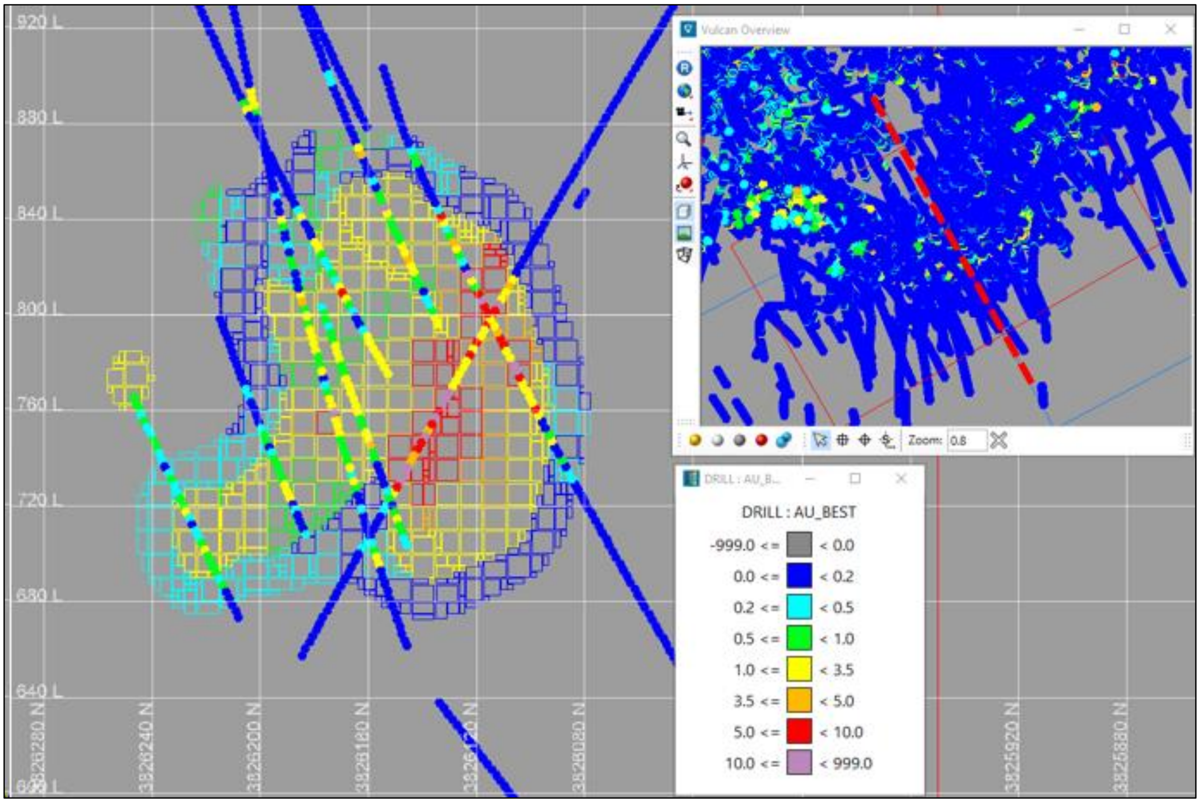
Several techniques were used to validate the block model; QA/QC analysis was performed on drilling data as per database procedures, and visual validation of the supporting drill data Au grades and estimated final block Au grades was performed, Figure 14-41).

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

The model was peer reviewed as well as independently externally reviewed by ERM Consultants, based in Brisbane (Adams & Konopa, 2024):

- A review of the model was to support an ongoing evaluation of Palomino as part of a Pre-Feasibility Study.
- Focus of the review was the technical aspects of the estimation and the classification process.
- No material issues were identified. The geological models honor the input data and the current understanding of the deposit geological framework and structural controls. The grade

estimates have undergone adequate validation and statistically compare well to the input data when assessed on a global basis.



Source: OceanaGold, 2023

Figure 14-41 Section of Final Block Au Grade, with Informing 3 m Topcut Composite Drillhole Data (window +/- 10 m)

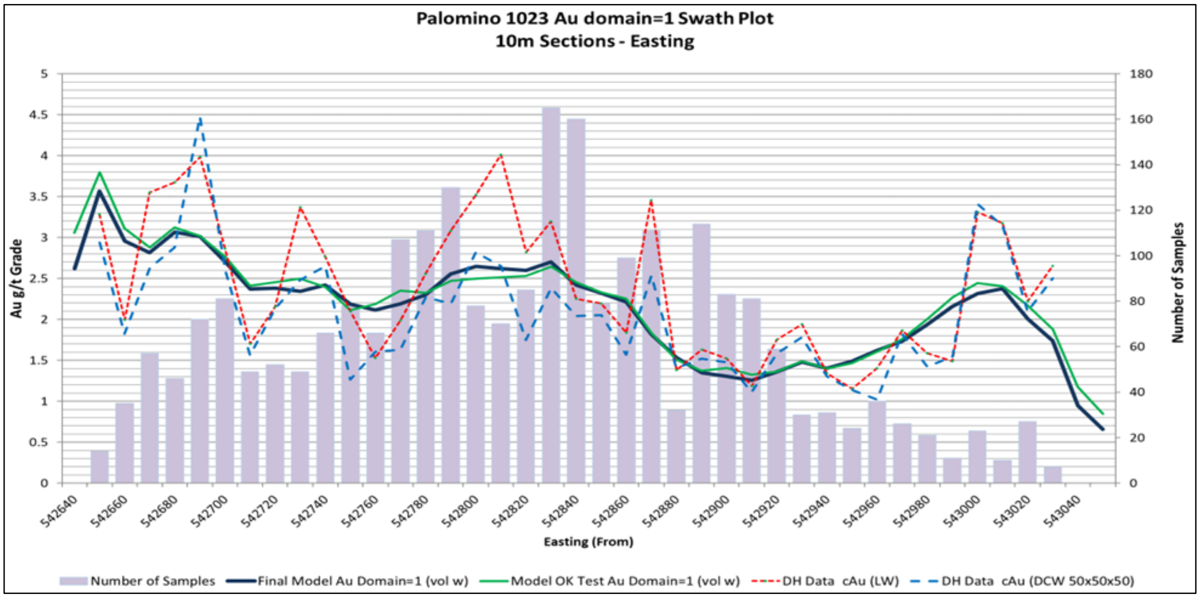
Global comparison of the 3m top capped drill data (with an appropriate declustering weighting applied of 50mE x 50mN x 50mRL), was compared to the final calculated block grade (block volume weighted) within the mineralized 0.8 g/t grade shell. This shows good correlation as shown in Table 14-20.

Table 14-20 Comparison Global Statistics – Block Model (vol weighted) vs. 3 m Top Capped DH (50 mE x 50 mN x 50 mRL declustered weighting) within 0.8 g/t Au Threshold Shell

Variable	Count	Minimum	Maximum	Mean	Variance	CV
Final BM Grade (vol Weighted)	20,305	0	12.1	2.2	2.3	0.7
DH 3m Top Capped (declust. Weighted)	2,475	0.003	28	2.0	9.4	1.5

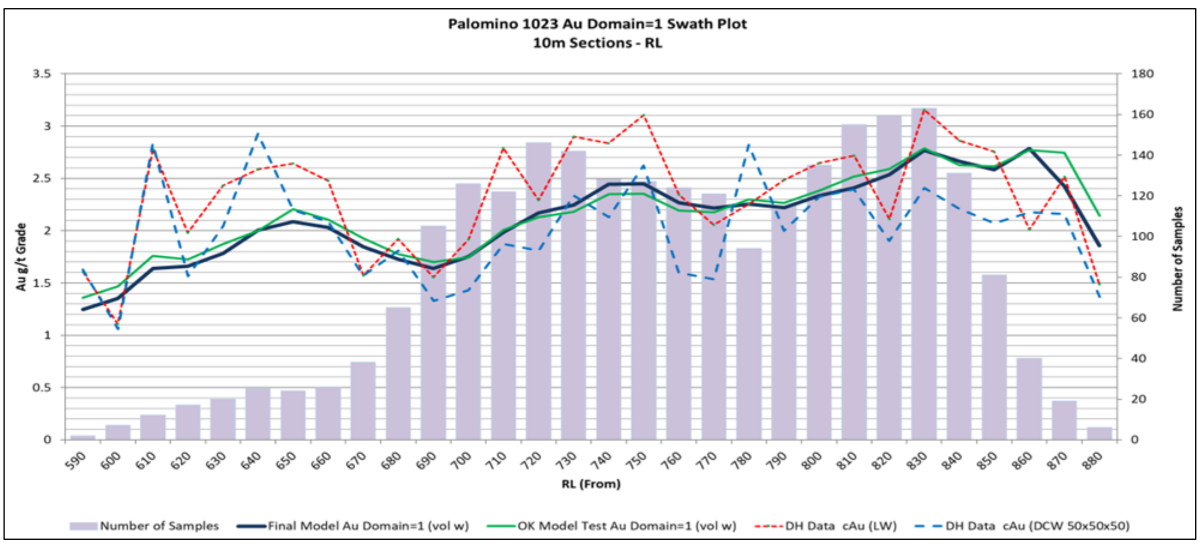
Source: OceanaGold, 2023

Swath plots were used to compare the estimation with underlying topcut composite grades for each domain. Figure 14-42 and Figure 14-43 show an acceptable local correlation between the composites and the block estimation grade for the mineralized domain. An OK check estimate model was also run and is shown in the swath plots, and locally where there are limited, high grade samples, the OK estimated grades are elevated (smoothed).



Source: OceanaGold, 2023

Figure 14-42: 0.8 g/t Threshold - Easting Swath Plot - Final Block Grade vs. DH 3 m Top Capped (length and declustered weighted) vs. OK Check Estimate



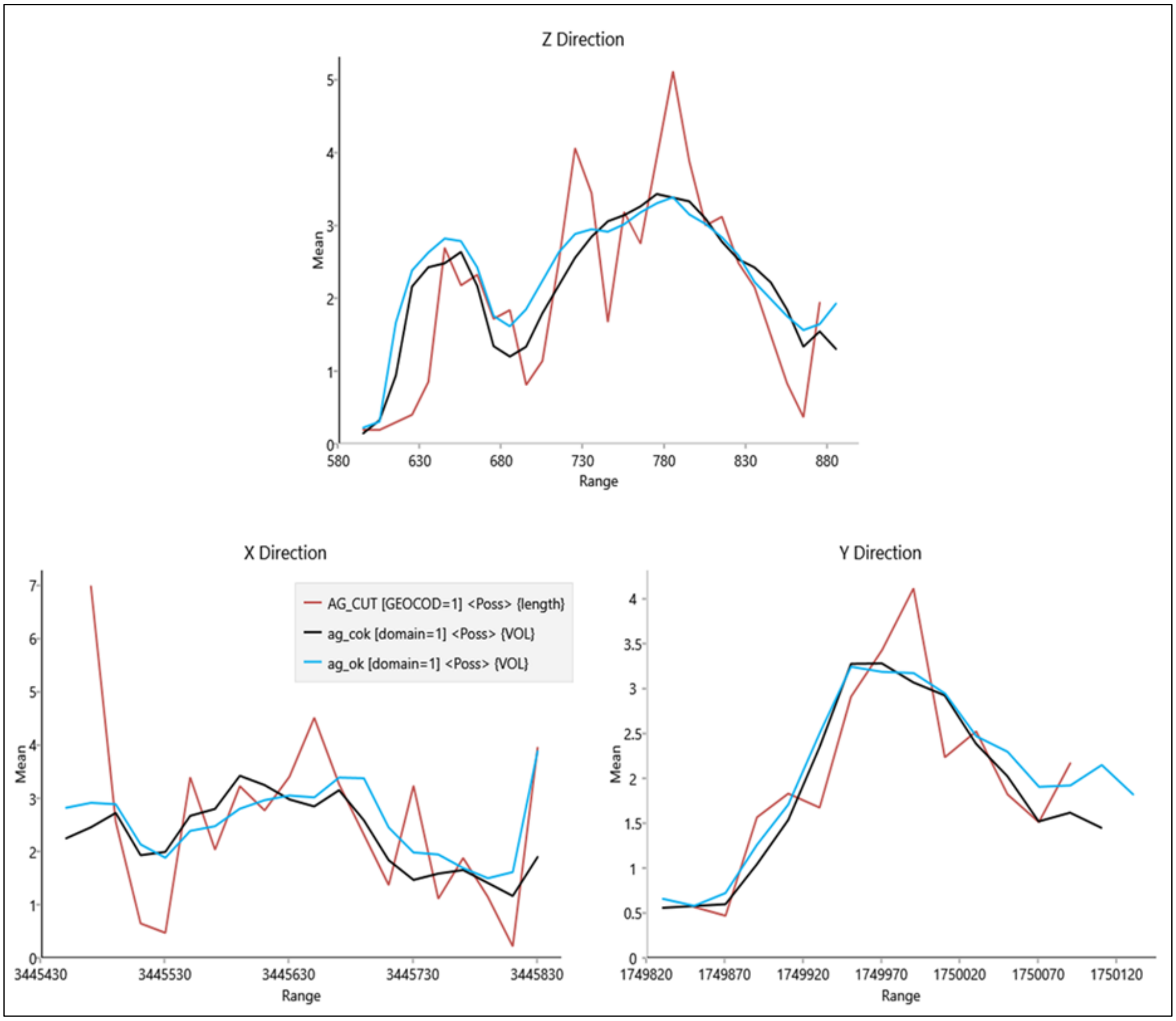
Source: OceanaGold, 2023

Figure 14-43: 0.8 g/t Threshold - RL Swath Plot - Final Block Grade vs. DH 3 m Top Capped (length and declustered weighted) vs. OK Check Estimate

Ag Estimate

Ag was estimated into Domain 1 and 3 using OK estimation technique. However, due to the relatively small amount of Ag samples available in Domain 1, it appeared that locally, the Ag was overestimating the input composted DDH data. A COK methodology was used taking advantage of the spatial correlation between Au vs. Ag using cross-variogram.

Swath plots in Easting, Northing and RL directions comparing the OK, COK and input 3 m composited top capped data (see Figure 14-44), shows the COK locally slightly lowers the estimated Ag value, better matching the input data.



Source: OceanaGold, 2023
Input 3 m composite top-capped DDH data (length weighted) and OK and COK Ag estimates (vol. weighted)

Figure 14-44: Swath-Plot Ag (0.8 g/t Au Threshold) – by RL (Z), Easting (X) and Northing (Y)

Palomino Mineral Resource Statement

The Palomino Mineral Resource estimate is presented in Table 14-21. The reported resource is constrained within a conceptual stope design based on a gold price of US$1,700/oz, approximating a 1.55 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 Au threshold shell remain unclassified.

The Palomino Mineral Resource statement is based on the OK model. It is constrained within a conceptual stope design using a CoG of 1.55 g/t Au, which assumes underground mining methods and is based on a mining cost of US$49.50/t, processing cost of US$14.84/t, administration cost of US$4.93/t, tailing cost of US$1.84/t and on a gold price of US$1,700/oz.

Table 14-21: Palomino Underground Mineral Resource Statement as of December 31, 2023

Class	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Contained Au (Moz)	Contained Ag (Moz)
Measured	-	-	-	-	-
Indicated	4.5	3.10	2.8	0.45	0.4
Measured & Indicated	4.54	3.10	2.8	0.45	0.4
Inferred	0.8	2.20	2.1	0.05	0.05

Source: OceanaGold, 2024

- Cut-off grade 1.55 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 diffuse 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 14-22 presents the combined open pit, stockpile, and underground resource statement for Haile.

Table 14-22: Haile Combined Open Pit, Stockpile and Underground Resource Statement December 31, 2023

Type	Class	Tonnes (Mt)	Au Grade (g/t)	Contained Au (Moz)	Ag Grade (g/t)	Contained Ag (Moz)
Open Pit	Measured	1.67	1.19	0.0.6	2.5	0.14
	Indicated	34.4	1.58	1.75	2.5	2.8
	Measured & Indicated	**36.1**	**1.56**	**1.81**	**2.5**	**2.9**
	Inferred	2.8	0.89	0.08		
Stockpiles	Measured	2.1	0.89	0.06		
	Indicated	0.0	0.00	0.0		
	Measured & Indicated	**2.1**	**0.89**	**0.08**		
	Inferred	0.0	0.0	0.0		
Underground	Measured	0.12	5.04	0.02	2.1	0.01
	Indicated	8.3	4.18	1.11	2.6	0.68
	Measured & Indicated	**8.4**	**4.19**	**1.13**	**2.6**	**0.69**
	Inferred	2.5	3.7	0.29	2.1	0.2
Combined	Measured	3.88	1.15	0.14	1.2	0.14
	Indicated	42.6	2.08	2.85	2.5	3.5
	Measured & Indicated	**46.5**	**2.00**	**3.00**	**2.4**	**3.6**
	Inferred	5.4	2.2	0.4		0.2

Source: OceanaGold, 2023

- Cut-off grades for the open pit, Horseshoe underground, and Palomino underground are 0.45 g/t / 0.55 (primary / oxide), 1.55 g/t Au, based on a gold price of US$1,700/oz.
- 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 threshold shell.
- Mineral Resources include Mineral Reserves and are reported on an in situ basis.
- It is reasonably expected that most of the Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.
- 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 Mineral Resources were estimated under the supervision of Jonathan Moore, MAusIMM CP(Geo), a Qualified Person.

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 operating mine within the United States, or that would materially affect the Mineral 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 were 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 and silver price of US$18/oz Ag. Reserves stated in this report are dated effective as of December 31, 2023.

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. Mineralized material is truck hauled from the pits to an existing crusher/processing facility. Overburden is categorized, and truck hauled to the appropriate Overburden Storage Area (OSA).

15.1.1 Introduction

Dilution and ore recovery have been applied to the resource block model to account for a portion of mineralized material that is expected to be mined by face shovel excavators. The resource block model was then used for open-pit optimization without further 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. This has limited impact on the Mineral Reserve, with an effective global dilution of 2.8% and mining recovery of 99.8%. This is discussed further in Section 16.1.6.

The open-pit Mineral 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 and silver price of US$18/oz Ag. 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 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 30° to 45°. This includes a 5° allowance for ramps and geotechnical catch benches.

Table 15-1: Pit Optimization Parameters

Parameter	Unit	Value
Base Mining Cost	US$/t mined	2.53
Incremental Mining Cost	US$/t mined / 5 m bench	0.015
Pit Exit	m RL	1140
Sustaining CapEx	US$/t mined	0.40
PAG Rehabilitation Cost	US$/t mined PAG waste	0.65
Ore Mining Premium	US$/t ore	0.37
Processing Cost	US$/t ore	14.84
G&A Cost	US$/t ore	4.93
Ore Rehandle Cost	US$/t ore	0.44
TSF Expansion	US$/t ore	1.84
Gold Recovery - Primary (1)	%	(1-(0.2152*Au grade^-0.3696)) +0.025
Gold Recovery – Oxide	%	67%
Silver Recovery	%	70%
Mill Throughput	Mtpa	3.8 (2)
Gold Price	US$/oz Au	1,500
Silver Price	US$/oz Ag	18
Gold Refining & Selling Cost	US$/oz (Au+Ag)	3.00
Calculated Au Cut-off Grade	US$/t	0.6
Royalties	%	0.0
Discount Rate	%	5.0

Source: OceanaGold, 2023
(1) Recovery equation has further 2.5% uplift added to recovery of material > 1.7 g/t Au
(2) Actual throughput is prorated at the average of 3.8 Mtpa for open-pit ore and 3.2 Mtpa for underground ore

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			Non-PAG/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		

Unit	Data Evaluated	Data Not Evaluated	Not Applicable	Notes
General Costs and Revenue Elements or Factors				
General and Administrative Costs	X			Operating Mine
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			Mine life, exploration potential.
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 2024

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 36.4 Mt with an average gold grade of 1.56 g/t and silver grade of 2.4 g/t. The Mineral Reserve statement, as of December 31, 2023 for the Haile Open Pit is presented in Table 15-3.

Table 15-3: Haile Open Pit Mineral Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven (1)	3.6	1.03	1.6	0.12	0.2
Probable	32.8	1.62	2.4	1.71	2.5
Proven + Probable	36.4	1.56	2.3	1.82	2.7

Source: OceanaGold, 2023
(1) Includes 2.0 Mt of stockpile material grading 0.9 g/t Au and 0.9 g/t Ag

- Reserves are based on a US$1,500/oz Au gold price and US$18/oz Ag silver 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 for gold 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%.
- Metallurgical recovery for silver is applied at 70%.
- Reserves are converted from resources through the process of pit optimization, pit design, production schedule and supported by a positive cash flow model.
- 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 under the supervision of David Londono of OceanaGold, a Qualified Person.

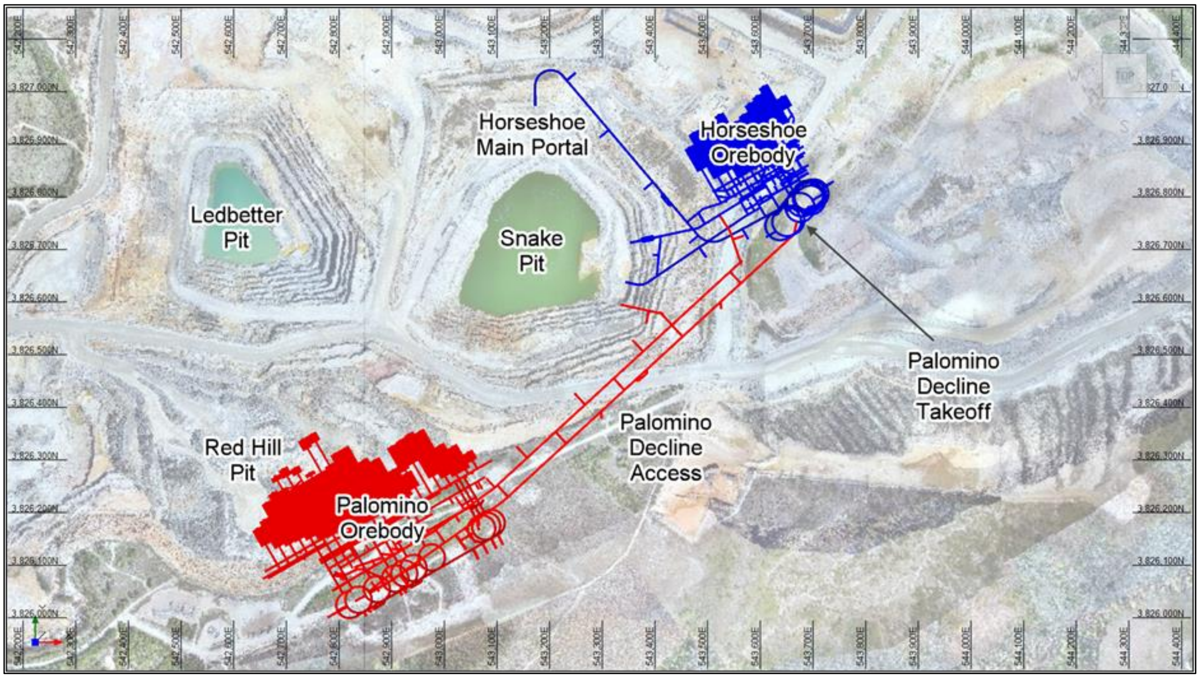
Relevant Factors

OceanaGold knows of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors that might materially affect the open pit Mineral Reserve estimate.

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 areas evaluated for underground mining are referred to as “Horseshoe” and “Palomino”. Horseshoe is located to the northeast of the Snake Pit and Palomino is to the southwest of Snake Pit, as shown in Figure 15-1.



Source: OceanaGold, 2023

Figure 15-1: General Site Layout and Location of the UG Reserve Area (in blue and 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 at HUG and 15 m wide at PUG 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.

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.

Dilution

The mining dilution estimate is based on the ELOS (equivalent linear overbreak/slough) methodology (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 (rock)	0.50
Sidewalls (backfill)	0.25
Endwalls (rock and backfill)	0.15
Bottom (backfill)	0.05

Source: OceanaGold, 2024

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, 2024

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 0.25 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. 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 shapes created during the mine design process.

Horseshoe underground mining reserves of 3.83 Mt (diluted) with an average grade of 4.23 g/t Au are presented in Table 15-6.

Table 15-6: Horseshoe Underground Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	0.1	4.53	2.0	0.01	0.01
Probable	3.7	4.23	1.7	0.50	0.2
Proven + Probable	3.8	4.23	1.7	0.52	0.2

Source: OceanaGold, 2023

Palomino underground mining reserves of 4.03 Mt (diluted) with an average grade of 2.91 g/t Au are presented in Table 15-7.

Table 15-7: Palomino Underground Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	-	-	-	-	-
Probable	4.0	2.91	2.7	0.38	0.3
Proven + Probable	4.0	2.91	2.7	0.38	0.3

Source: OceanaGold, 2023

The Mineral Reserve statement, as of December 31, 2023, for the Haile Underground (Horseshoe and Palomino) is presented in Table 15-8.

Table 15-8: Haile Underground Reserves Estimate as of December 31, 2023

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Au Contained (Moz)	Ag Contained (Moz)
Proven	0.1	4.53	2.0	0.01	0.01
Probable	7.7	3.54	2.0	0.88	0.5
Proven + Probable	7.8	3.56	2.0	0.89	0.5

Source: OceanaGold

- 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 85%.
- The reserve estimate is based on a mine design using an elevated cut-off grade of 1.87 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 1.74 g/t Au and an incremental development cut-off grade of 0.59 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.
- 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 Brianna Drury of OceanaGold, a Qualified Person.

<u>Relevant Factors</u>

The authors know\ of no existing environmental, permitting, legal, socio-economic, marketing, political, or other factors which could materially affect the underground Mineral Reserve estimate.

15.3 Open Pit and Underground Combined Reserves Statement

Table 15-9 presents the combined open pit and underground Mineral Reserves statement for Haile.

Table 15-9: Combined Reserve Statement for OceanaGold's Haile Gold Mine as of December 31, 2023

Type	Category	Tonnes (Mt)	Au Grade (g/t)	Ag Grade (g/t)	Au Contained (Moz)	Ag Contained (Moz)
OP	Proven (1)	3.6	1.03	1.6	0.12	0.18
	Probable	32.8	1.62	2.4	1.71	2.54
	Proven + Probable	36.4	1.56	2.4	1.82	2.72
UG	Proven	0.10	4.53	1.95	0.01	0.01
	Probable	7.76	3.54	2.18	0.89	0.54
	Proven + Probable	7.86	3.56	2.18	0.90	0.55
OP + UG	Proven	3.7	1.13	1.62	0.13	0.19
	Probable	40.6	1.98	2.36	2.59	3.08
	Proven + Probable	44.3	1.91	2.30	2.72	3.27

Source: OceanaGold

(1) Includes 2.0 Mt of stockpile material grading 0.9 g/t Au and 0.9 g/t Ag

- Mineral Reserves are based on a gold price of US$ 1,500/oz Au and silver price of US$18/oz Ag.
- 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 85% for the underground material.
- Metallurgical recovery for silver is applied at 70%.
- 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.
- The reserve estimate is based on a mine design using an elevated cut-off grade of 1.87 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 1.74 g/t Au and an incremental development cut-off grade of 0.59 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.
- 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 supported by a positive cash-flow model.
- The open pit Mineral Reserves were estimated under the supervision of David Londono of OceanaGold, a Qualified Person. The underground Mineral Reserves were estimated by Brianna Drury of OceanaGold, a Qualified Person.

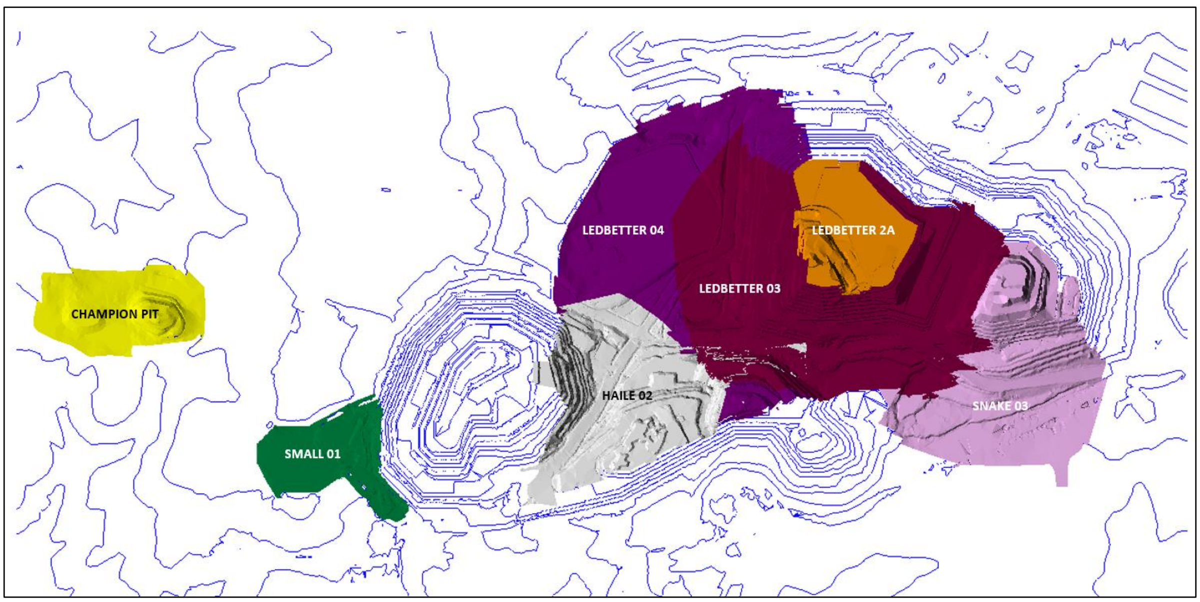
16 Mining Methods

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, 2023 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 both the Horseshoe and Palomino underground mines 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, 2023

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-shovel 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.

Overburden typically refers to material overlying a mineral deposit that must be removed prior to mining, and waste is rock that must be broken and disposed of to gain access to and excavate the ore. Operationally at Haile, the term overburden is typically used for all waste from the open-pits.

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 overburden classification is not necessary. Saprolite is mined without blasting where possible. Saprolite is sampled for overburden 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 0.8 to 1.3 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 Non-PAG/PAG definition. Blastholes in ore are typically sampled three times at 3.3 m sample intervals.

Flitch height is variable. Overburden 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 excavators, while the majority of waste is mined with hydraulic face shovels. Front-end loaders may be used in either application in back-up capacity. The haul truck fleet is primarily 175 tonne payload units with smaller 140 tonne payload units used for ancillary duties and backup production.

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. (Design Angle Recommendations for the Haile Gold Mine, 2022) 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
- Slope Angle Evaluation for the Haile Gold Mine (CNI 2018)
- 2021 Mill Zone Design Review
- 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 within the metasediments and metavolcanics 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 (cross joints) 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 to establish boundary conditions for the FEFLOW analyses. 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.

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			

Source: Call & Nicholas Inc., 2021

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			

Source: Call & Nicholas Inc., 2021

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

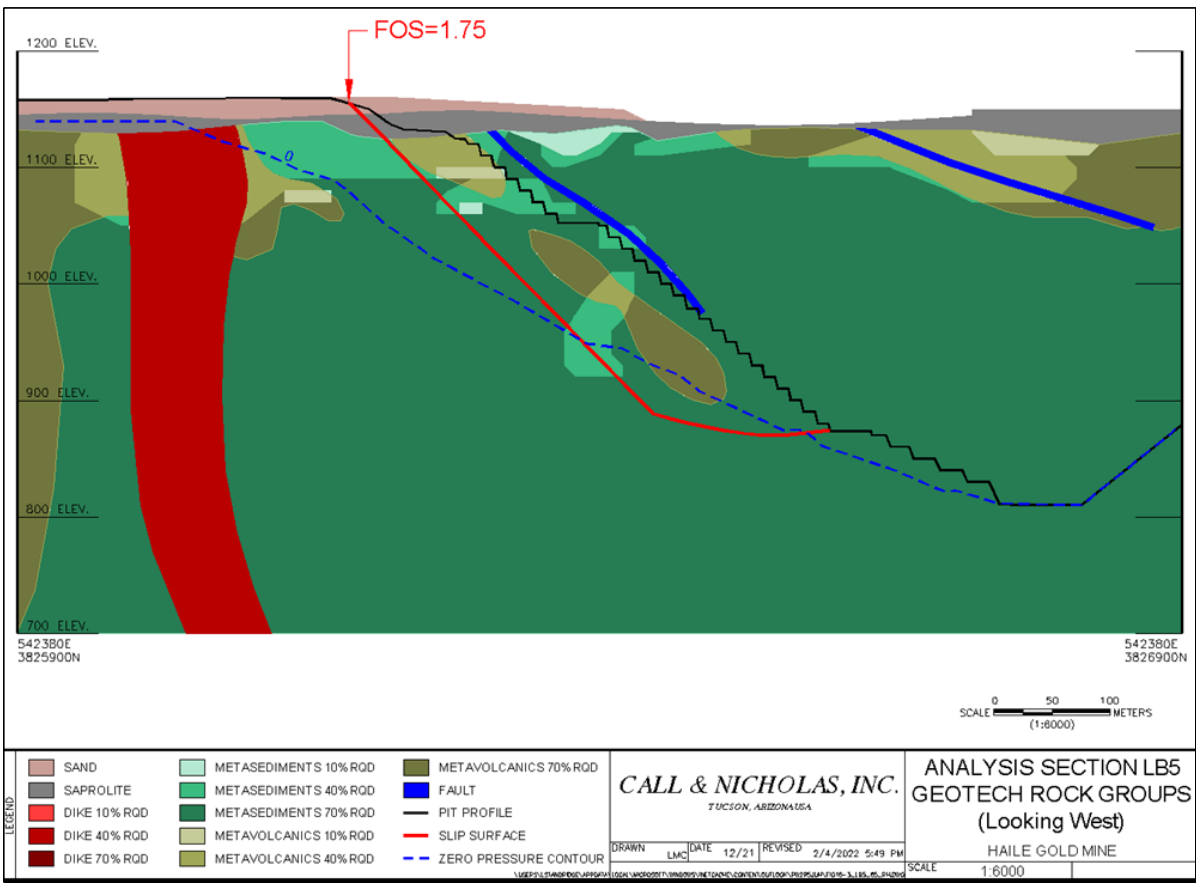
Range of Wall DDR (°)	Metavolcanics - Bench Design							
	ISA (°)	Height (m)	BFA (°)	CBW (m)	ISA (°)	Height (m)	BFA (°)	CBW (m)
345 - 015	42	10	78	9.0	20 m Bench Heights not Recommended			
015 - 045	48	10	78	6.9	50	20	78	12.5
045 - 075	49	10	78	6.6	50	20	78	12.5
075 - 105	47	10	78	7.2	50	20	78	12.5
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

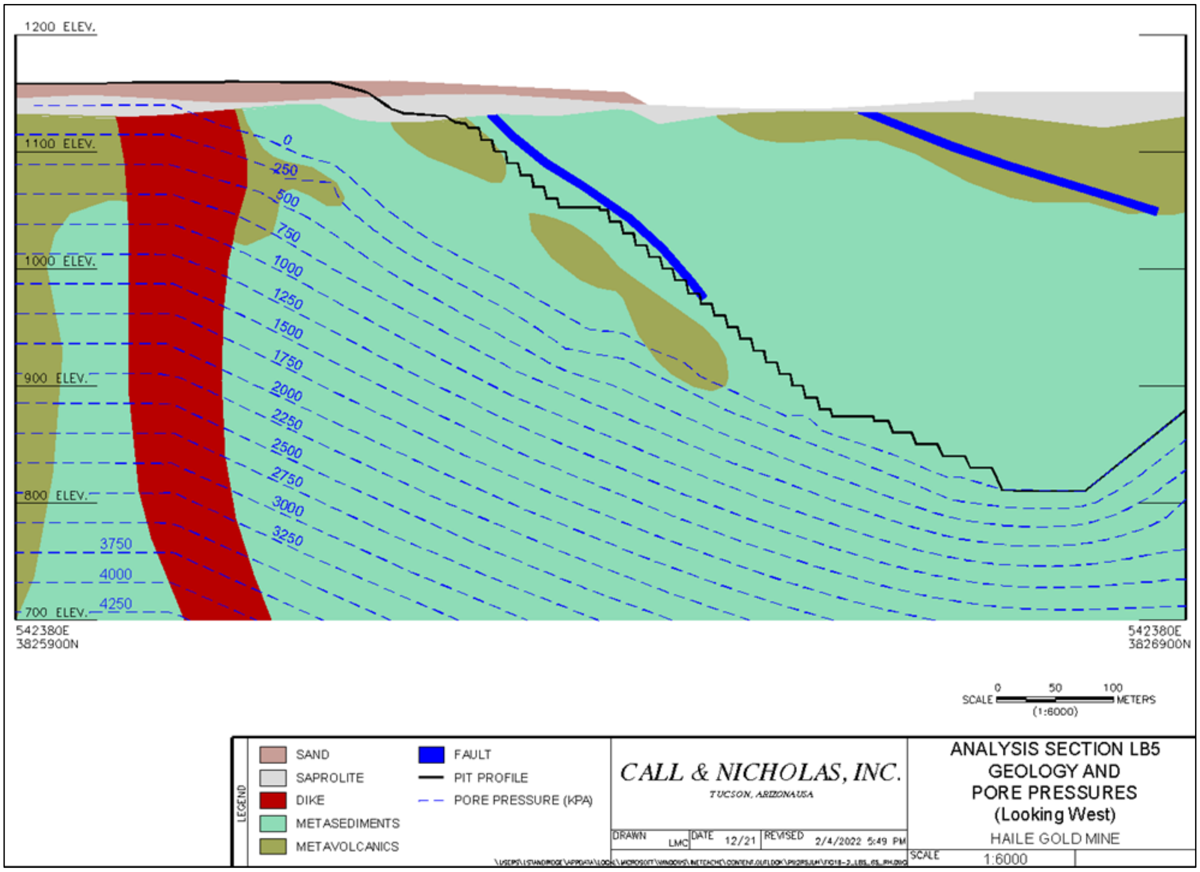
Rock Type	Strength Type	Density (kN/m^3)	Cohesion (kPa)	Phi (°)
Meta Sediments	10% RQD	28.0	397.6	25.2
	40% RQD		859.7	27.5
	70% RQD		1867.9	30.9
	Foliation Anisotropy		251.2	21.5
	Cross Joint Anisotropy		489.5	22.0
Meta Volcanics	10% RQD	25.8	465.4	28.6
	40% RQD		1008.3	30.7
	70% RQD		2192.7	33.7
	Foliation 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

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 backhoe loading units and mining practices currently used for the majority of ore mining at Haile. As part of the mining sequence, some ore is expected to be mined using face shovel loading units near the top of the orebody, during the transition from bulk waste mining to selective mining. Dilution and recovery factors have been applied to the upper zones of the mineralization in the resource model prior to optimization in recognition of the different mining method. The factors and application method for dilution and recovery are detailed in Section 16.1.6.

The optimization included Measured and Indicated Mineral Resource categories with a gold price of US$1,500/oz Au and silver price of US$18/oz Ag. 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 2023 produced in December 2023. There are no material differences between the forecast and actual 31 December 2023 surfaces.

Whittle optimization parameters are summarized in Table 16-7.

Table 16-7: Pit Optimization Parameters

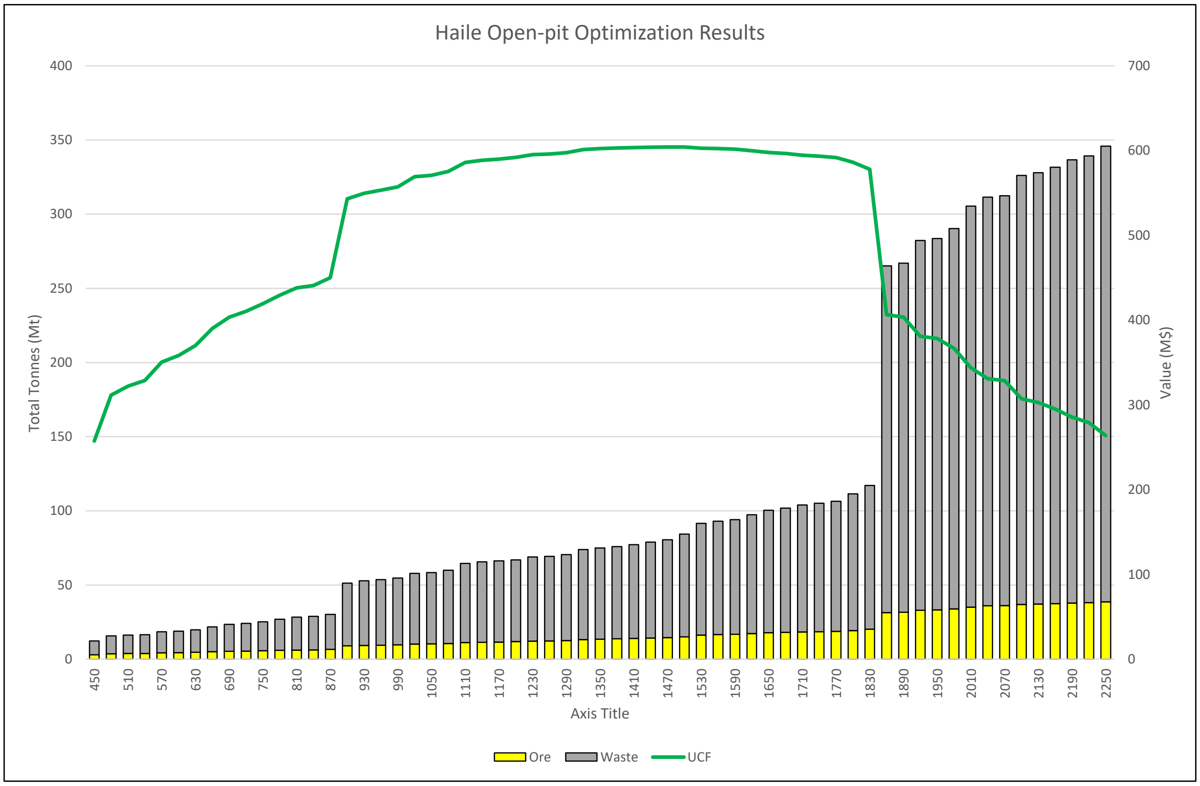
Parameter	Unit	Value
Base Mining Cost	US$/t mined	2.53
Incremental Mining Cost	US$/t mined / 5 m bench	0.015
Pit Exit	m RL	1140
Sustaining CapEx	US$/t mined	0.40
PAG Rehabilitation Cost	US$/t mined PAG waste	0.65
Ore Mining Premium	US$/t ore	0.37
Processing Cost	US$/t ore	14.84
G&A Cost	US$/t ore	4.93
Ore Rehandle Cost	US$/t ore	0.44
TSF Expansion	US$/t ore	1.84
Gold Recovery – Primary > 1.7 g/t Au	%	(1-(0.2152*Au grade^-0.3696)) +0.025
Gold Recovery – Primary < 1.7 g/t Au	%	(1-(0.2152*Au grade^-0.3696))
Gold Recovery – Oxide	%	67%
Silver Recovery	%	70%
Mill Throughput	Mtpa	3.8[1]
Gold Price	US$/oz Au	1,500
Silver Price	US$/oz Ag	18
Gold Refining & Selling Cost	US$/oz (Au+Ag)	3.00
Calculated Au Cut-off Grade	US$/t	0.6
Royalties	%	0.0
Discount Rate	%	5.0

Source: OceanaGold, 2023
[1] Actual throughput is prorated at the average of 3.8 Mtpa for open-pit ore and 3.2 Mtpa for underground ore

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.

Rehabilitation costs have been added to 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 paid, is made by Whittle during optimization.

The open pit Mineral Reserves are reported within a pit design based on open pit optimization results are shown in Figure 16-4 and Table 16-8.



Source: OceanaGold, 2024

Figure 16-4: Pit Optimization Tonnes by Revenue Factor

The ultimate pit design used for reporting Mineral Reserves is the existing Life-of-Mine ultimate pit from the previous Technical Report (SRK, 2022) with minor modifications. As noted in Table 16-8, due to cost increases the equivalent shell to the designed ultimate pit is based on a higher gold price than the Mineral Reserve reporting price of US$1,500/oz. The difference between the Revenue Factor 1.0 shell and the ultimate pit shell is roughly equivalent to the Ledbetter 4 pit phase. The decision to maintain a larger open-pit than is indicated by the optimization process, notably retaining Ledbetter Phase 4, has been made based on a number of factors, including:

- Positive total project cashflow
- Maintaining life-of-mine to 2035
- Potential cost reductions through planned continuous improvement processes
- Support for extended underground operations

In addition to the above, an underground vs. open-pit trade-off study has been commenced specifically targeting the mineralization within and in close proximity to Ledbetter Phase 4. This study is expected to be completed in March 2025. If the underground option is proven to be economically preferable, this will improve the overall project value, however with a potential reduction in overall Mineral Reserves.

Table 16-8: Optimization Results for Selected Shell on US$30 Gold Price Increments

Gold Price	Rev Factor	Total	Waste	Strip Ratio	Ore			Contained Metal		Recovered Metal		Recovery		UCF	Avg Cash Cost	Mining Cost	Process Cost	Selling Cost
US$/oz Au	#	Mt	Mt	Wt:Ot	Mt	Au (g/t)	Ag (g/t)	Au Moz	Ag Moz	Au Moz	Ag Moz	Au %	Ag %	US$ million	US$/oz Au [(1)]	US$/t	US$/t	US$/oz Au
750	0.50	25.2	19.5	3.4	5.7	2.67	2.6	0.49	0.5	0.43	0.3	87.0%	70.0%	419	518	3.66	22.42	5.39
780	0.52	27.0	21.0	3.6	5.9	2.66	2.7	0.51	0.5	0.44	0.4	87.0%	70.0%	429	527	3.66	22.42	5.41
810	0.54	28.4	22.3	3.6	6.1	2.64	2.6	0.52	0.5	0.45	0.4	86.9%	70.0%	438	535	3.65	22.42	5.42
840	0.56	28.9	22.7	3.6	6.2	2.63	2.6	0.53	0.5	0.46	0.4	86.9%	70.0%	441	538	3.65	22.42	5.42
870	0.58	30.2	23.7	3.6	6.6	2.58	2.6	0.54	0.6	0.47	0.4	86.8%	70.0%	450	549	3.66	22.42	5.47
900	0.60	51.3	42.2	4.7	9.0	2.48	2.5	0.72	0.7	0.63	0.5	86.7%	70.0%	543	634	3.70	22.42	5.43
930	0.62	52.9	43.6	4.7	9.3	2.47	2.5	0.74	0.7	0.64	0.5	86.6%	70.0%	550	639	3.70	22.42	5.44
960	0.64	53.6	44.2	4.7	9.4	2.45	2.5	0.74	0.8	0.64	0.5	86.6%	70.0%	553	643	3.70	22.42	5.46
990	0.66	54.6	45.0	4.7	9.6	2.44	2.5	0.75	0.8	0.65	0.5	86.5%	70.0%	557	647	3.70	22.42	5.47
1020	0.68	57.9	47.7	4.7	10.2	2.38	2.5	0.78	0.8	0.68	0.6	86.4%	70.0%	569	661	3.69	22.42	5.54
1050	0.70	58.3	48.0	4.6	10.3	2.37	2.5	0.79	0.8	0.68	0.6	86.4%	70.0%	571	663	3.69	22.42	5.55
1080	0.72	59.9	49.3	4.7	10.6	2.35	2.5	0.80	0.8	0.69	0.6	86.3%	70.0%	575	670	3.69	22.42	5.56
1110	0.74	64.6	53.4	4.8	11.2	2.32	2.5	0.83	0.9	0.72	0.6	86.2%	70.0%	586	687	3.68	22.42	5.62
1140	0.76	65.6	54.2	4.8	11.4	2.30	2.5	0.84	0.9	0.73	0.6	86.2%	70.0%	589	692	3.68	22.42	5.65
1170	0.78	66.2	54.7	4.7	11.5	2.29	2.5	0.85	0.9	0.73	0.7	86.1%	70.0%	590	695	3.68	22.42	5.67
1200	0.80	66.9	55.1	4.7	11.8	2.26	2.5	0.86	1.0	0.74	0.7	86.0%	70.0%	592	700	3.68	22.42	5.71
1230	0.82	68.9	56.8	4.7	12.1	2.23	2.5	0.87	1.0	0.75	0.7	86.0%	70.0%	595	708	3.68	22.42	5.74
1260	0.84	69.3	57.0	4.7	12.2	2.22	2.5	0.88	1.0	0.75	0.7	85.9%	70.0%	596	710	3.68	22.42	5.76
1290	0.86	70.4	57.9	4.6	12.5	2.20	2.5	0.89	1.0	0.76	0.7	85.9%	70.0%	598	717	3.67	22.42	5.78
1320	0.88	73.9	60.7	4.6	13.2	2.15	2.5	0.91	1.1	0.78	0.7	85.7%	70.0%	601	733	3.67	22.42	5.83
1350	0.90	75.0	61.5	4.6	13.5	2.13	2.5	0.92	1.1	0.79	0.8	85.6%	70.0%	602	739	3.66	22.42	5.85
1380	0.92	75.9	62.2	4.5	13.7	2.11	2.5	0.93	1.1	0.80	0.8	85.6%	70.0%	603	744	3.66	22.42	5.87
1410	0.94	77.2	63.3	4.5	14.0	2.09	2.5	0.94	1.1	0.80	0.8	85.5%	70.0%	604	750	3.66	22.42	5.90
1440	0.96	78.9	64.7	4.6	14.2	2.07	2.5	0.95	1.1	0.81	0.8	85.5%	70.0%	604	757	3.65	22.42	5.91
1470	0.98	80.5	66.0	4.6	14.5	2.06	2.5	0.96	1.1	0.82	0.8	85.4%	70.0%	604	764	3.65	22.42	5.94
1500	1.00	84.3	69.2	4.6	15.1	2.02	2.5	0.98	1.2	0.84	0.8	85.3%	70.0%	604	780	3.64	22.42	6.01
1530	1.02	91.6	75.4	4.6	16.2	1.96	2.4	1.02	1.3	0.87	0.9	85.1%	70.0%	603	811	3.62	22.42	6.06
1560	1.04	93.0	76.5	4.6	16.6	1.94	2.4	1.03	1.3	0.88	0.9	85.1%	70.0%	602	818	3.62	22.42	6.07
1590	1.06	94.1	77.4	4.6	16.8	1.93	2.4	1.04	1.3	0.88	0.9	85.0%	70.0%	602	822	3.62	22.42	6.09
1620	1.08	97.4	80.1	4.6	17.3	1.90	2.4	1.06	1.3	0.90	0.9	84.9%	70.0%	600	835	3.61	22.42	6.10
1650	1.10	100.4	82.6	4.6	17.8	1.88	2.4	1.07	1.4	0.91	0.9	84.8%	70.0%	598	848	3.61	22.42	6.11
1680	1.12	101.8	83.8	4.6	18.0	1.87	2.4	1.08	1.4	0.92	1.0	84.8%	70.0%	597	854	3.60	22.42	6.12
1710	1.14	103.9	85.6	4.7	18.4	1.85	2.4	1.09	1.4	0.93	1.0	84.8%	70.0%	595	862	3.60	22.42	6.15
1740	1.16	105.1	86.6	4.7	18.5	1.85	2.4	1.10	1.4	0.93	1.0	84.7%	70.0%	593	866	3.60	22.42	6.16
1770	1.18	106.5	87.8	4.7	18.7	1.84	2.3	1.11	1.4	0.94	1.0	84.7%	70.0%	592	872	3.60	22.42	6.17
1800	1.20	111.4	92.2	4.8	19.3	1.82	2.3	1.13	1.5	0.95	1.0	84.6%	70.0%	586	889	3.60	22.42	6.20
1830	1.22	117.1	96.9	4.8	20.2	1.78	2.3	1.15	1.5	0.98	1.0	84.5%	70.0%	578	911	3.59	22.42	6.20
1860	1.24	265.2	233.9	7.5	31.4	1.68	2.4	1.69	2.4	1.42	1.7	84.2%	70.0%	406	1,221	3.64	22.42	6.50
1890	1.26	267.0	235.5	7.5	31.6	1.67	2.4	1.70	2.4	1.43	1.7	84.1%	70.0%	404	1,224	3.64	22.42	6.51
1920	1.28	282.3	249.3	7.6	33.0	1.66	2.3	1.76	2.5	1.48	1.7	84.1%	70.0%	381	1,249	3.65	22.42	6.53
1950	1.30	283.5	250.3	7.5	33.2	1.65	2.3	1.76	2.5	1.48	1.7	84.1%	70.0%	378	1,252	3.65	22.42	6.54

Source: OceanaGold, 2024
Blue – Revenue Factor 1
[(1)] Silver as by-product credit

16.1.4 Mine Design

Reserve 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
- Non-PAG/PAG determination
- Mining dilution and recovery

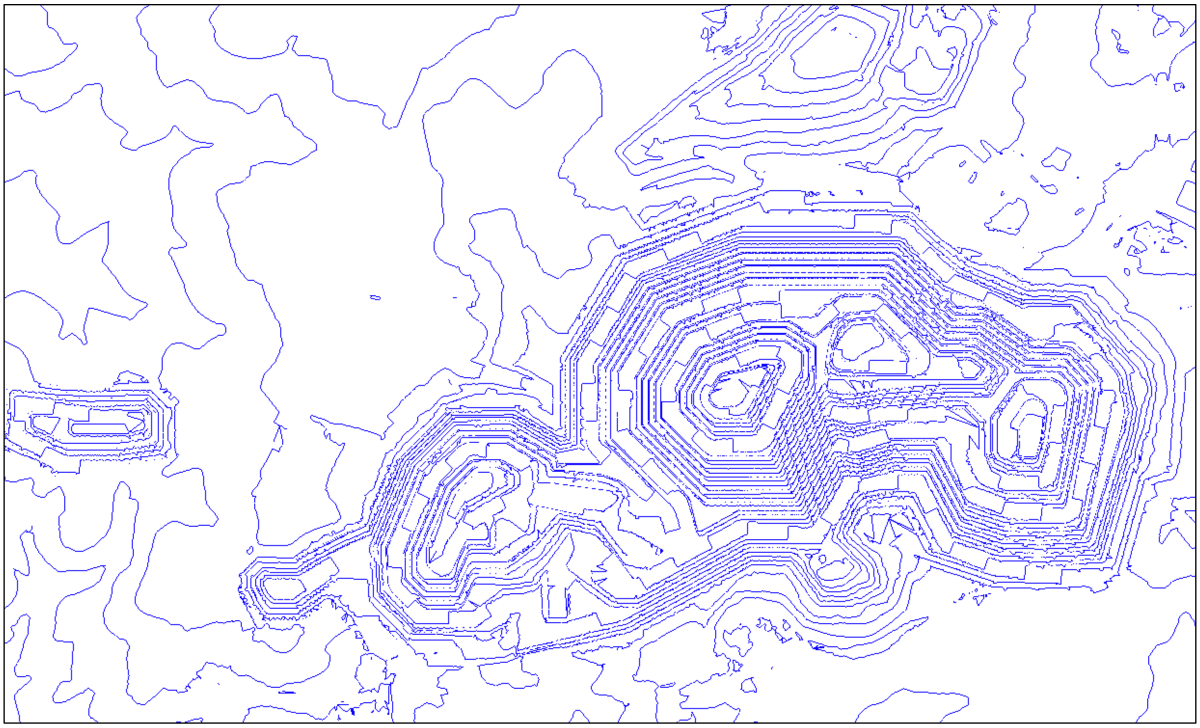
The Non-PAG/PAG determinations govern the routing of overburden material to either lined PAG OSAs, in-pit backfill (yellow only) or unlined Non-PAG OSAs.

Pit Design

OceanaGold used the optimization shells as a guide for practical phase and ultimate pit 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 m to 300 m
- Flat switchbacks
- Bench heights, berm widths and bench face angles in accordance with current site-specific design criteria

Figure 16-5 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 pit phase has at least one ramp for scheduling purposes.



Source: OceanaGold, 2023

Figure 16-5: LoM Pit Design

16.1.5 Overburden/Geochemical

Overburden mined at Haile consists of cover soil, Coastal Plain Sands (CPS), and bedrock with variable degrees of weathering and oxidation. Mined materials are grouped as soft (i.e., cover soil, CPS, and saprolite) and hard (i.e., bedrock comprised of metavolcanics and metasediments with variable oxidation) rock units. Cover soil is salvaged from the disturbance footprints prior to open pit mining or placement of mine waste, either as overburden or tailings, and stockpiled to be used to facilitate reclamation of the mine waste storage facilities. CPS is mined without drilling and blasting and classified as Green (Non-PAG) material without additional testing. Saprolite is mined without blasting where possible and is sampled and tested for overburden classification and management. Bedrock is drilled and blasted, and the cuttings from each blast hole in waste zones are sampled and tested for overburden classification. The current open pit mine plan is summarized in Table 16-13 and includes 282,624 kt of overburden. Annual production by material type, based on the reserve block model, is illustrated in Figure 16-9.

Oxidation in bedrock generally extends 20 to 60 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, comprising 0.1% to 10% by volume. 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 rock (i.e., overburden) to identify, manage and mitigate geochemical risks at Haile as part of the open pit plan (Schafer, 2019). 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 Overburden Management Plan (OMP) (Schafer, 2015) has three geochemical categories of overburden based on total sulfur abundance (S_T) and net neutralization potential (NNP). The NNP is a measure of overall acid generation potential (AP) calculated as the difference between the neutralization potential (NP) and AP. The geochemical categories of overburden are:

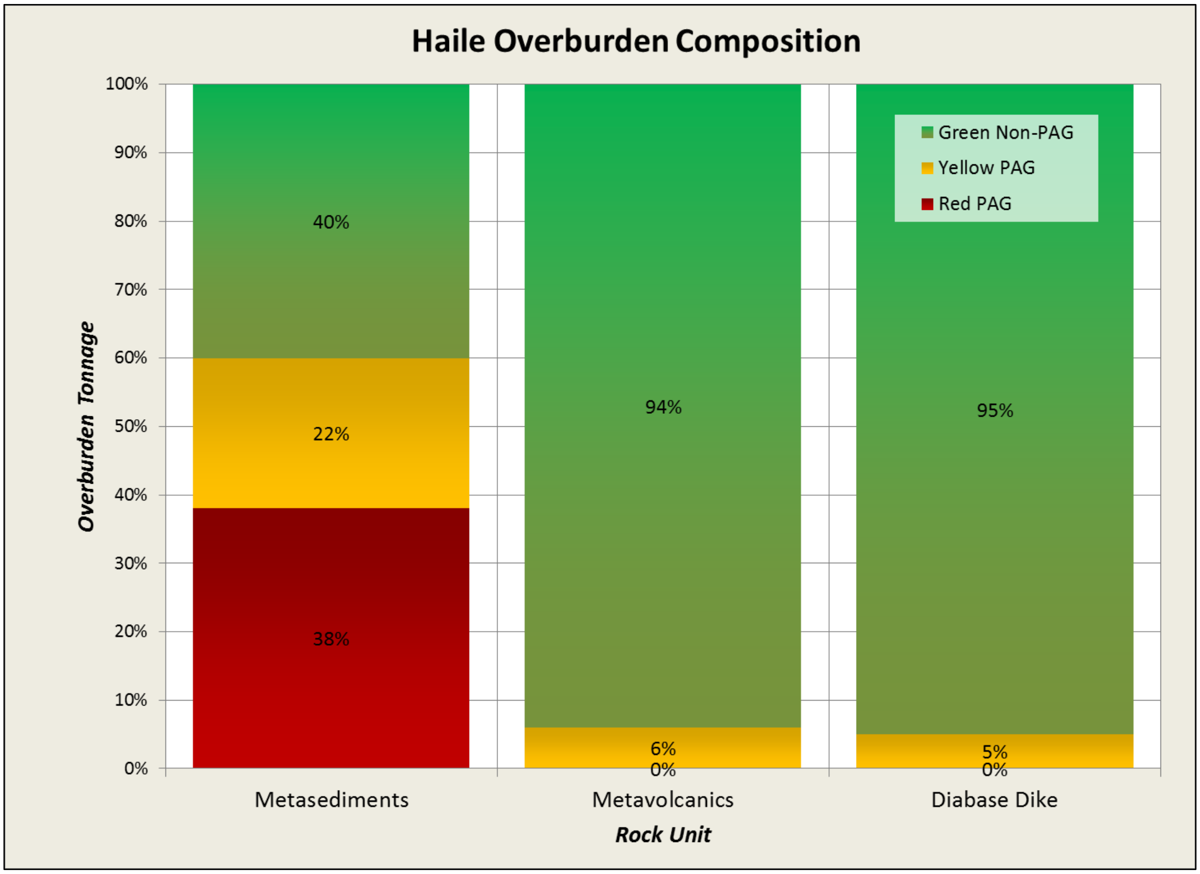
- Red PAG – Strongly acid generating:
  - NNP < -31.25 kg $CaCO_3$/t
- Yellow PAG – Moderately acid generating:
  - S_T > 0.2 % or NNP between -31.25 and 0 kg $CaCO_3$/t
- Green Non-PAG – Non acid generating, oxidized:
  - S_T < 0.2 % and NNP > 0 kg $CaCO_3$/t

The geochemical categories of overburden and their associated management approaches are summarized in Table 16-9. A summary of the distribution of the three categories throughout the bedrock lithologies is presented in Figure 16-6. The quantities of Red PAG, Yellow PAG, and Green Non-PAG overburden presented in Figure 16-6 and Table 16-9 are based on the exploration drilling dataset and are not limited to the mine design area. The quantities are intended to provide a general overview of each material type's characteristics.

Kinetic testing of "Red" and "Yellow" material confirmed the applicability of the overburden 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 release was much lower than for Red PAG samples due to the lower initial abundance of pyritic sulfur.
- Green Non-PAG samples maintained near neutral pH and maintained low to non-detectable sulfate levels.

Supplemental testwork completed between 2020 and 2023 confirms that the current PAG criteria of NNP < 0 and > -31.25 and S_T > 0.2 % are conservative, i.e., some, but not all, Yellow PAG material may become acidic in time (Schafer, 2024).



Source: Schafer, 2015

Figure 16-6: Material Classifications by Rock Type

Sulfur and carbon assays from blastholes are used to populate block models and assign overburden class for each blast pattern. The overburden materials are placed as follows:

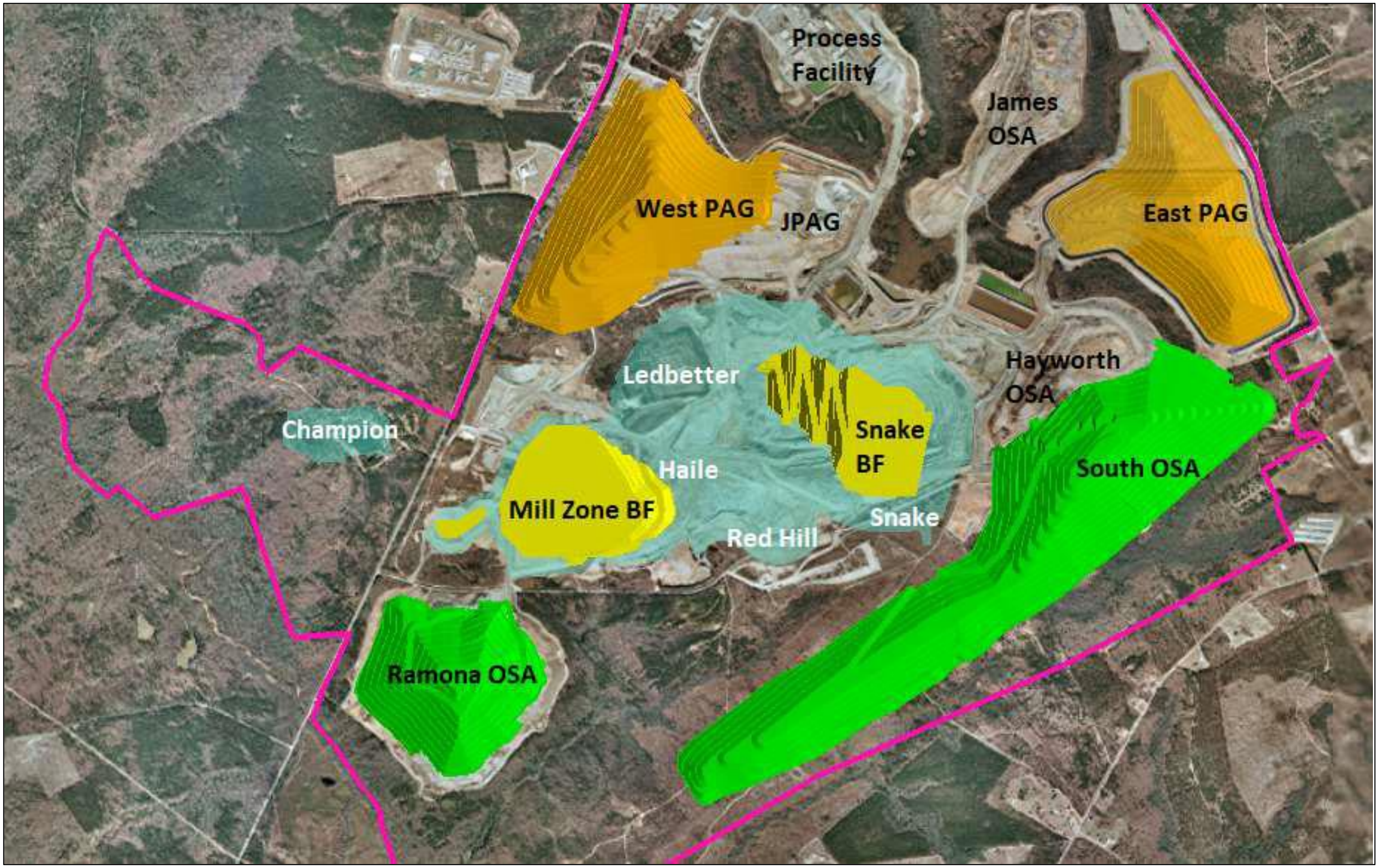
- All Red PAG is sent to the East PAG or West PAG OSAs. These are lined facilities, and both are currently in use. Material will be placed in lifts and compacted by haulage fleet traffic. For closure, the top of these facilities will be covered with saprolite, followed by an HDPE geomembrane liner and then a layer of growth media. The growth media will be seeded to establish vegetation.
- Yellow PAG can be stored in the East PAG or West PAG OSAs, or below a prescribed water table within pits. Yellow PAG material will be mixed with lime at a rate of 0.9 kg (2 lb.) lime per 0.9 t (2,000 lb.) 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 Non-PAG material. Yellow PAG can be utilized in-pit for road construction.
- Green Non-PAG 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.

Table 16-9: Overburden Classification at Haile

Operational Testing Criterion	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 acidity, metals, and sulfate (>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 between -31.25 and 0 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, may also be placed in lifts as subaqueous pit backfill (2 lbs/t lime added and 5-ft saprolite cover)
Green Non-PAG - Non Acid Generating Overburden			
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 Sands is Green Non-PAG.	40% Metasediment Unit, 94% Metavolcanics, 95% Saprolite and all Coastal Plain Sands	Contact water may have moderately acidic to alkaline pH (4.0 to 8.0), low sulfate (<1,000 mg/L) and metals non-detectable.	Placed in unlined overburden piles. Runoff will not require treatment assuming it meets stormwater 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 16-7.



Yellow inside pits are backfill material.
Source: OceanaGold, 2024

Figure 16-7: Final Pit Design and Ultimate Overburden Storage Site Plan

16.1.6 Mine Production Schedule

Cut-Off Grade

OceanaGold have used a stockpiling strategy with multiple CoGs to determine direct processing ore, stockpiled ore, and waste in the mine production schedule. The base assumptions for the calculation of break-even CoG during operations and at the end of mine life are detailed in Table 16-10. Primary and Oxide ores have different processing recovery responses, and separate CoG values have been applied. The breakeven CoG during normal operation is 0.6 g/t for Primary material and 0.7 g/t for Oxide material. However, while the mine is operating a cut-over grade of approximately 0.7 g/t has been used to delineate direct feed ore, while the end of mine life marginal CoG is used for material sent to the stockpile, being 0.5 g/t Au for Primary and to 0.6 g/t Au for Oxide. The end of mine life marginal CoG has been used for reporting of Mineral Reserves.

Table 16-10: Cut-off Grade Calculation

Description	Units	Operating BE CoG	Operating BE CoG	End of Mine Life CoG	End of Mine Life CoG
Material Type	Type	Primary	Oxide	Primary	Oxide
Gold Price	US$/oz	1,500	1,500	1,500	1,500
Smelting & Refining	US$/oz	3.00	3.00	3.00	3.00
Au Recovery [(1)]	Source	Opti Study	Opti Study	Opti Study	Opti Study
Au Recovery (at CoG)	%	74.2%	68.0%	72.4%	68.0%
Operating Costs	**Units**	**Values**	**Values**	**Values**	**Values**
Ore Premium (D&B)	US$/t ore	0.37	0.37	0.37	0.37
G&A	US$/t ore	4.93	4.93	1.80	1.80
Tailings	US$/t ore	1.84	1.84	1.84	1.84
Processing	US$/t ore	14.84	14.84	14.00	14.00
PAG rehab	US$/pag t mined	0.65	0.65	0.65	0.65
Rehandle Cost	US$/t ore	0.44	0.44	0.44	0.44
Total PCAF (Whittle)	**US$/t ore**	**22.42**	**22.42**	**18.45**	**18.45**
Total Cost (with PAG)	**US$/t ore**	**21.77**	**21.77**	**17.80**	**17.80**
Breakeven Cut-off Grade	g/t	0.61	0.67	0.51	0.54
Applied Cut-off Grade	g/t	0.60	0.70	0.50	0.60

Source: OceanaGold, 2024
[(1)] Recovery at Primary CoG based on recovery formula: (1-(0.2152*Au grade^-0.3696))

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.

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
Snake: Phase 3	1.00	1.00	1.00	1.10	0.95	1.05	1.18	0.87	1.02
Ledbetter: Phase 2A	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
Red Hill Phase 1	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

Source: OceanaGold, 2022
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.

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 remaining inventory in the ultimate pit design is broken into 7 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 reserve block model. Scheduling is completed using 10 m bench heights until 30% of the contained ounces for all large phases have been mined, then 5 m bench heights used for mining the majority of the phase to match the reserve block model. Small Phase 1 and Champion Phase 1 are scheduled on 5 m bench heights for the entire phase.

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

Table 16-12: Phase Inventory (1/1/2024 to End of Mine Life)

Phases	Ore (Measured + Indicated)					Waste	Total	
	Tonnes (kt)	Au g/t	Ag g/t	Contained Au (koz)	Contained Ag (koz)	Tonnes (kt)	SR	Tonnes (kt)
Snake: Phase 3	4,560	1.44	1.9	211	275	40,276	8.8	44,836
Ledbetter: Phase 2A	2,135	2.51	2.7	172	185	8,111	3.8	10,246
Ledbetter: Phase 3	8,835	2.13	2.5	604	705	73,040	8.3	81,875
Ledbetter: Phase 4	14,509	1.34	2.5	626	1,151	136,298	9.4	150,807
Small: Phase 1	863	0.71	2.5	20	69	2,321	2.7	3,184
Haile: Phase 2	3,768	1.05	2.4	127	287	17,784	4.7	21,552
Champion: Phase 1	1,031	0.94	3.4	31	111	4,794	4.6	5,825
Total	**35,700**	**1.56**	**2.4**	**1,792**	**2,785**	**282,624**	**7.9**	**318,325**

Source: OceanaGold, 2024

Production Schedule Targets

The production schedule has a start date of 1 January 2024 and is based on the end of year survey dated 31 December 2023. Scheduling is completed using RPM Global's OPMS scheduling software.

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)
2024	2,359	2.03	2.52	33,992	36,351
2025	2,239	1.79	2.24	34,367	36,606
2026	5,281	2.14	2.41	37,372	42,653
2027	5,047	1.53	2.46	41,170	46,217
2028	5,876	1.31	2.08	36,634	42,511
2029	3,397	0.97	2.30	38,892	42,289
2030	1,983	0.90	2.86	19,550	21,533
2031	1,092	1.10	2.37	16,551	17,643
2032	2,226	1.32	2.59	14,860	17,086
2033	4,130	1.50	2.53	8,163	12,293
2034	2,071	2.34	2.90	1,073	3,144
Total	**35,700**	**1.56**	**2.43**	**282,624**	**318,325**

Source: OceanaGold, 2024

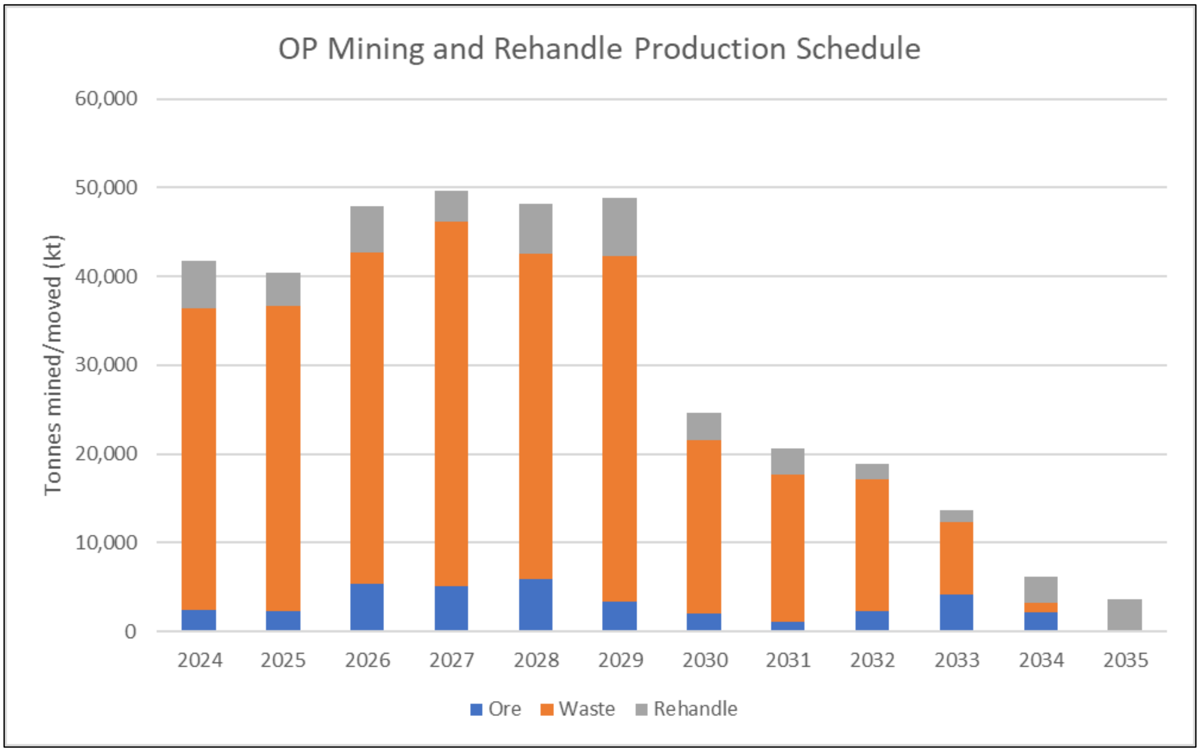
Points of note on the schedule:

- Low ore production in 2024 and 2025 relates to completion of Ledbetter 2A in 1Q 2025 and Ledbetter Phase 3 reaching consistent ore in 3Q 2025
- Excess Yellow PAG overburden stored in in-pit dumps from operations to date is scheduled to be rehandled on an as-required basis across 2024 to 2029

- Total mining movement rates reduce from 2030 on to control low-grade stockpile size given the availability of underground ore for processing feed
- Low ore production and grade in 2030 and 2031 relates waste stripping in Ledbetter Phase 4

At the end of the mine life, a low-grade stockpile inventory of 1.3 Mt at 0.6 g/t Au and 2.9 g/t Ag remains unprocessed due to limitations on current TSF capacity. This material has been excluded from the Mineral Reserve reported in Section 15. This material will be stored within the PAG cell footprint and is currently assumed to be rehabilitated with that facility. Further investigation of TSF capacity requirements and constraints will be completed in conjunction with the Trade-off Study noted in Section 16.1.3.

Figure 16-8 illustrates the annual production schedule for ore and waste tonnes, including rehandle completed using the OP mining fleet.

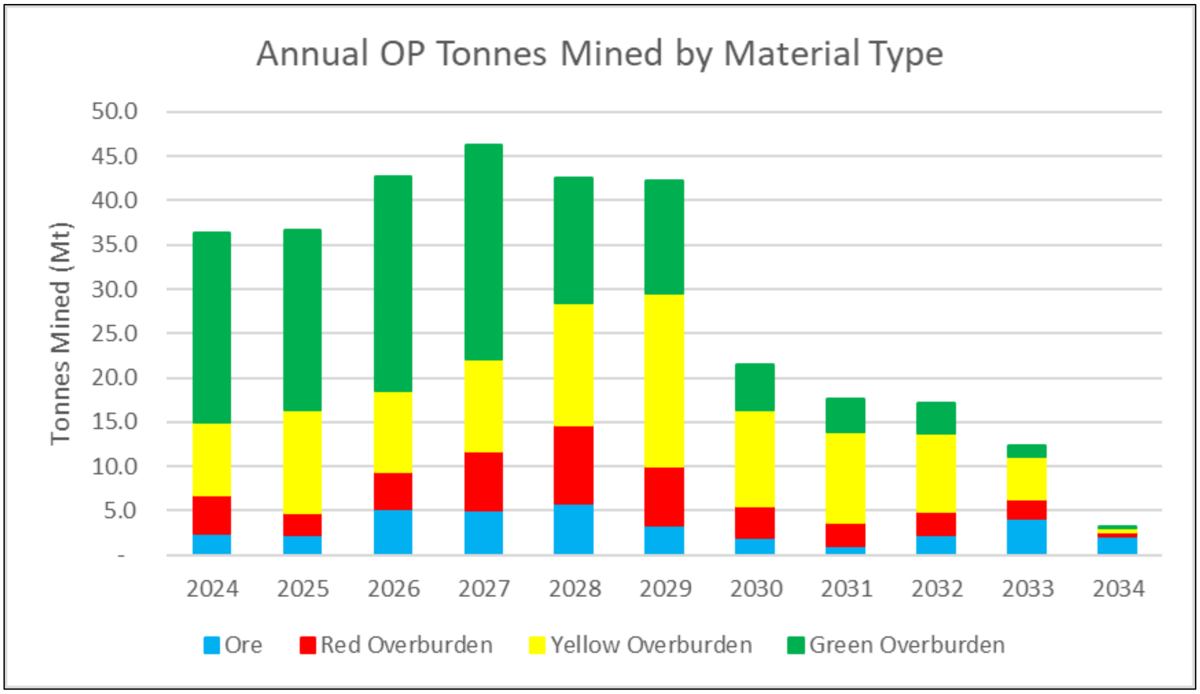


Source: OceanaGold, 2024

- Low ore production in 2024 and 2025 relates to completion of Ledbetter 2A in 1Q 2025 and Ledbetter Phase 3 reaching consistent ore in 3Q 2025
- Total mining movement rates reduce from 2030 on to control low-grade stockpile size given the availability of underground ore for processing feed.
- Low ore production and grade in 2030 and 2031 relates waste stripping in Ledbetter Phase 4

Figure 16-8: Annual Production Schedule

Figure 16-9 breaks down the PAG (Red and Yellow) overburden, Non-PAG (Green) overburden, and Ore tonnes for the LoM. The proportion of Red/Yellow to Green overburden generally increases with depth, highlighted by the decreasing quantities of Green overburden produced toward the end of the OP mine life.



Source: OceanaGold, 2024

Figure 16-9: Material Type Annual Schedule

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

Description	Description	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	LoM Total
UG Ore	RoM (kt)	562	624	778	761	1,079	969	867	747	749	643	80	-	7,859
	RoM Au (g/t)	4.0	4.0	3.4	4.6	3.4	3.6	3.7	2.9	3.0	3.0	4.0	0.0	3.6
	RoM Ag (g/t)	1.8	1.7	1.7	1.8	2.3	2.5	2.1	2.0	3.0	2.6	3.5	0.0	2.2
Total OP Ore	RoM (kt)	2,359	2,239	5,281	5,047	5,876	3,397	1,983	1,092	2,226	4,130	2,071	0	35,700
	RoM Au (g/t)	2.0	1.8	2.1	1.5	1.3	1.0	0.9	1.1	1.3	1.5	2.3	0.0	1.6
	RoM Ag (g/t)	2.5	2.2	2.4	2.5	2.1	2.3	2.9	2.4	2.6	2.5	2.9	0.0	2.4
	EOP Stock (kt)	1,236	410	2,772	4,879	8,221	8,925	8,094	6,267	5,527	6,576	4,934	1,296	
	RoM Au (g/t)	0.8	0.7	1.1	0.9	0.8	0.7	0.6	0.6	0.6	0.7	0.6	0.6	
	RoM Ag (g/t)	2.1	3.6	2.5	2.3	2.2	2.2	2.2	2.2	2.3	2.3	2.4	2.9	
UG Mill Feed	Ore Feed (kt)	562	624	778	761	1,079	969	867	747	749	643	80	-	7,859
	Ore Feed Au (g/t)	4.0	4.0	3.4	4.6	3.4	3.6	3.7	2.9	3.0	3.0	4.0	0.0	3.6
	Ore Feed Ag (g/t)	1.8	1.7	1.7	1.8	2.3	2.5	2.1	2.0	3.0	2.6	3.5	0.0	2.2
	Ore Feed Contained Au (koz)	72	81	85	114	118	114	103	70	72	61	10	-	900
	Ore Feed Contained Ag (koz)	32	34	42	43	81	77	58	48	73	53	9	-	551
OP Mill Feed	Ore Feed (kt)	3,116	3,065	2,919	2,939	2,535	2,693	2,814	2,920	2,966	3,081	3,713	3,638	36,398
	Ore Feed Au (g/t)	1.8	1.5	2.9	2.2	2.2	1.3	1.0	0.9	1.1	1.7	1.7	0.6	1.6
	Ore Feed Ag (g/t)	1.6	2.0	2.5	2.7	2.2	2.4	2.5	2.2	2.4	2.5	2.6	2.2	2.3
	Ore Feed Contained Au (koz)	180	152	275	209	179	111	90	81	109	167	199	71	1,824
	Ore Feed Contained Ag (koz)	165	198	235	260	183	204	229	202	231	252	307	257	2,721
Total Mill Feed	**Total Feed (kt)**	**3,678**	**3,690**	**3,698**	**3,701**	**3,614**	**3,661**	**3,681**	**3,666**	**3,714**	**3,723**	**3,793**	**3,638**	**44,257**
	Total Feed Au (g/t)	**2.1**	**2.0**	**3.0**	**2.7**	**2.6**	**1.9**	**1.6**	**1.3**	**1.5**	**1.9**	**1.7**	**0.6**	**1.9**
	Total Feed Ag (g/t)	**1.7**	**2.0**	**2.3**	**2.5**	**2.3**	**2.4**	**2.4**	**2.1**	**2.5**	**2.5**	**2.6**	**2.2**	**2.3**
	Total Feed Contained Au (koz)	**252**	**233**	**360**	**323**	**297**	**225**	**193**	**151**	**182**	**228**	**209**	**71**	**2,724**
	Total Feed Contained Ag (koz)	**197**	**232**	**277**	**303**	**264**	**281**	**287**	**250**	**304**	**305**	**316**	**257**	**3,272**

Source OceanaGold, 2024

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	Number of 10 Meter Benches by Year										
	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Ledbetter Phase 2A	7.0	1.5	-	-	-	-	-	-	-	-	-
Ledbetter Phase 3	6.0	8.5	6.0	3.5	1.5	-	-	-	-	-	-
Small Phase 1	1.5	2.5	-	-	-	-	-	-	-	-	-
Haile Phase 2	-	-	6.5	2.5	4.0	-	-	-	-	-	-
Snake Phase 3	-	5.0	7.0	5.5	5.0	-	-	-	-	-	-
Ledbetter Phase 4	-	-	-	4.5	4.5	5.0	2.0	3.5	4.0	6.0	4.5
Champion	-	-	-	-	-	1.5	4.0	-	-	-	-

Source OceanaGold, 2024

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 (Caterpillar 785 and Komatsu 730E). Blasthole drilling and wall control drilling is performed with a fleet of Sandvik DR410i, Sandvik Leopard DI650i, 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 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	2
Operating Days/year	363
Scheduled hours/shift	12
Shifts/day	2
Scheduled hours/year	8,712

Source: OceanaGold, 2024

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
Availability	84%	86%	84%	82%
Use of Availability	80%	85%	84%	85%
Operating Hours per Year	5,887	6,404	6,181	6,106

Source: OceanaGold, 2024

Overburden is drilled with either 171 mm or 200 mm diameter and ore is drilled with either 115 mm or 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 1,500 tonnes per hour (t/h). For PC4000 excavators, the estimated productivity is 2,600 t/h.

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

- Green overburden (to Green OSA and TSF)
- Red PAG (to dedicated PAG cells)
- Yellow PAG (to dedicated PAG cells or in-pit dumps)
- Inferred (to relevant OSA or PAG cells)
- Ore (to ROM or to stockpile)
- Yellow PAG 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 PAG cells)

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 scheduled 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 eight tracked dozers. In addition, three motor graders cover the pit, dump, and surface roads. A front-end loader (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	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
PC4000 Excavator	2	2	2	2	2	2	2	2	2	1	1
PC3000 Excavator	1	1	1	1	1	1	1	1	1	1	1
CAT 6020B Excavator	1	0	1	1	1	1	1	1	1	1	0
Cat 785 Trucks	1	1	1	1	1	1	1	1	2	2	1
Komatsu 730E Trucks	16	18	19	19	17	17	9	8	8	8	8
Sandvik DR410i Drill	4	4	4	4	4	4	4	4	4	2	2
Sandvik Leopard DI650i & Epiroc D65 Drill	3	4	4	4	4	4	2	2	2	3	2

Source: OceanaGold, 2024

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 2024.

Table 16-19: Labor Levels in 2024

Labor Category	Year 2024
Management and Technical Services	
Mine Operations Management	3
Mine Engineering/Geology/Survey	44
Total	**47**
Operations Labor	
Mine Foreman/Supervisors	22
Operators	169
Blast Crew/ Explosives Ops	0
Other	9
Total	200
Maintenance Labor	
Maintenance Super/Shift Foreman	15
Fitter/Welder/Electrician/Serviceman	82
Maintenance Planner	5
Maintenance Clerk	1
Total	**103**
Grand Total	**350**

Source: OceanaGold, 2024

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

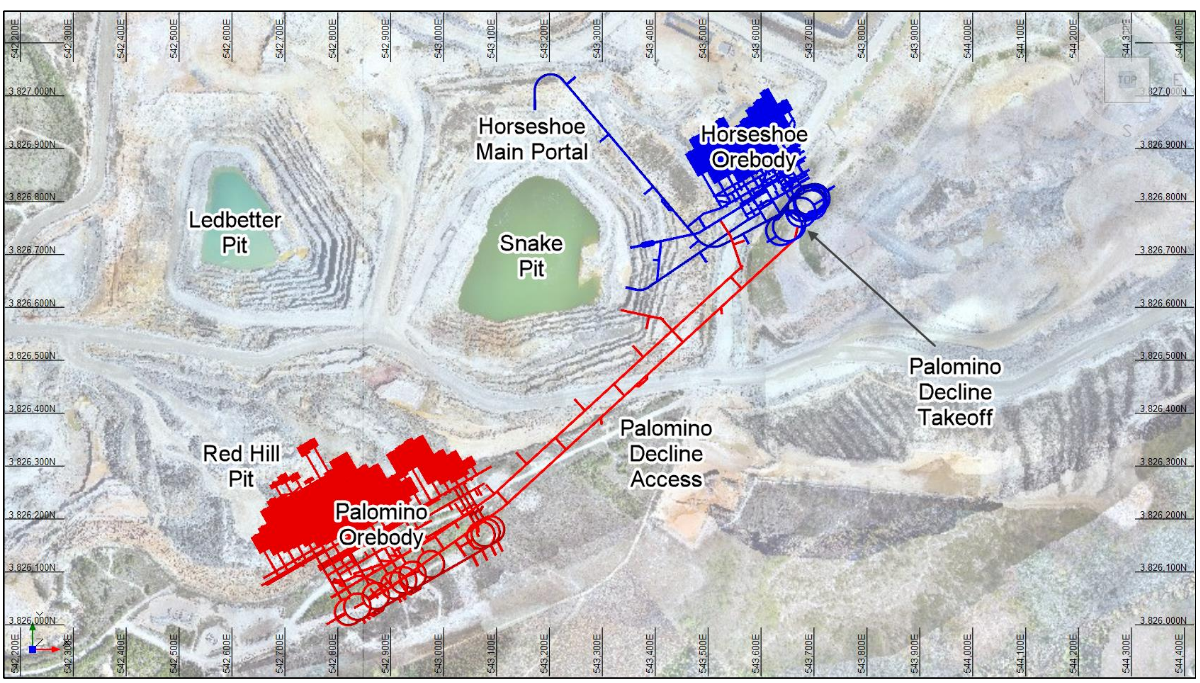
Site stratigraphic data, meteorological data and groundwater pressure/level data from numerous monitoring wells and vibrating wire piezometers installed in each of the hydrostratigraphic units, and groundwater production data, were used to develop the conceptual groundwater model and calibrate the numerical 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 70% 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 Horseshoe underground deposit was discovered in 2010. Economic mineralization at Horseshoe 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 400NW 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, 40NW 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. The Horseshoe deposit was the subject of numerous studies during the 2010’s and deemed suitable for extraction via underground methods. In late 2022, Horseshoe underground development commended with access and infrastructure via multiple portals within Snake Pit. In October 2023, first production was achieved from Horseshoe.

The Palomino deposit was first discovered in 2011. Palomino is approximately 650 m southwest of the Horseshoe deposit at 300 m to 500 m below the surface. Deposit dimensions are approximately 300 m long x 50 m to 100 m thick x 100 m to 150 m wide. Lozenge-shaped mineralized zones strike ENE, dip northwest, and plunge gently northeast. Fine-grained gold is hosted in pyritic and silicified siltstone and instrusives, along a steeply SE-dipping, ENE-striking contact with dacite flows. Mineralization is truncated by several NNW-striking, sub-vertical 1 m to 25 m thick diabase dikes. In 2017, a drill program was undertaken at Palomino showing good grade continuity. At the end of 2019, an initial Inferred resource as declared with subsequent drill programs focusing on infill drilling and conversion of Inferred material to Indicated. Figure 16-10 shows the location of Palomino relative to Horseshoe.



Source: OceanaGold 2023

Figure 16-10: Horseshoe and Palomino Plan View

Based on the orientation, depth, and geotechnical characteristics of the Horseshoe and Palomino mineralization, a transverse sublevel open stoping method (longhole) has been selected. The stopes will be 15 m to 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

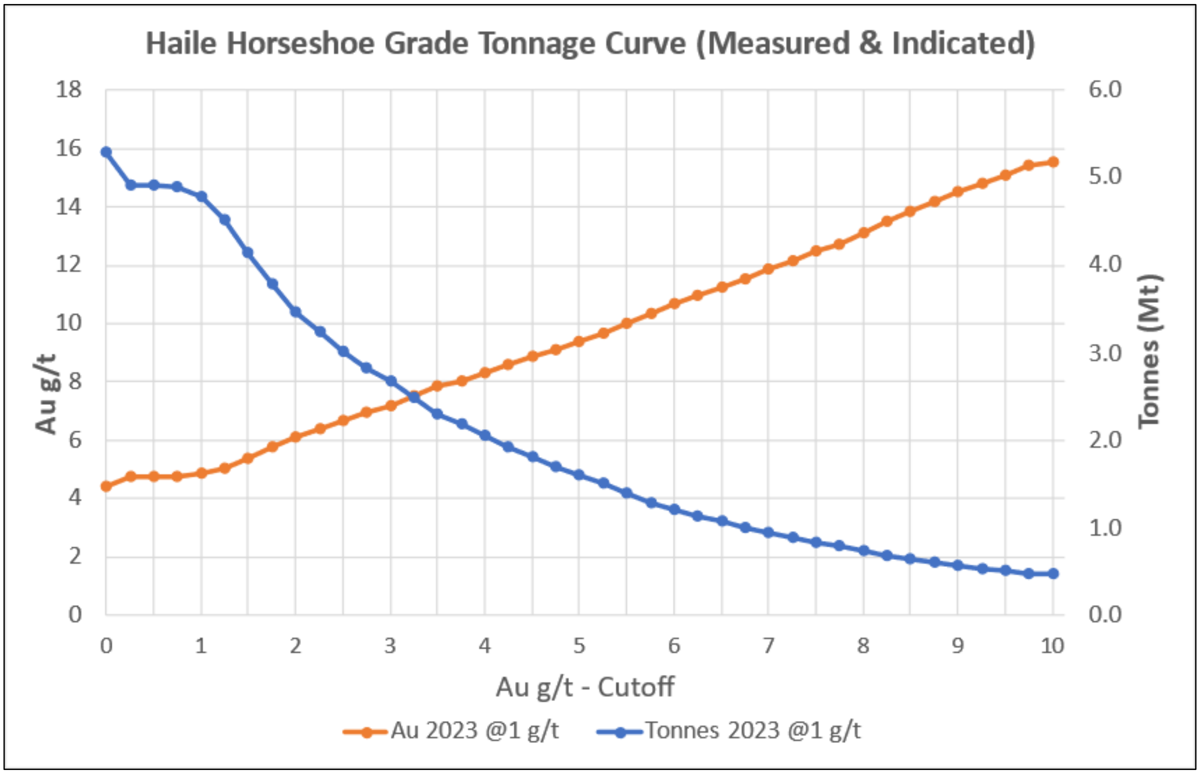
Current estimated project costs and the calculated Au CoG are shown in Table 16-20. For mine design, an elevated cut-off grade of 1.87 g/t Au was used, with adjacent lower grade stopes included in the design. Incremental material is included in the reserves based on an incremental stope cut-off grade of 1.74 g/t Au and an incremental development cut-off grade of 0.59 g/t Au.

Table 16-20: Underground Cut-off Grade Calculation

Parameter	Operating CoG	Incremental Stoping CoG	Marginal Development CoG	Unit
Mining cost [1]	55.22	49.50	-	US$/t
Process cost	14.84	14.84	14.84	US$/t
Tailings	1.84	1.84	1.84	
G&A	4.93	4.93	4.93	US$/t
Total Cost	**$76.83**	**$71.11**	**$21.61**	**US$/t**
Gold price	1,500.00	1,500.00	1,500.00	US$/oz
Average Au mill Recovery [2]	85%	85%	76%	
Smelting & Refining	3.00	3.00	3.00	US$/oz
CoG	**1.87**	**1.74**	**0.59**	**g/t**

Source: OceanaGold, 2023
[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

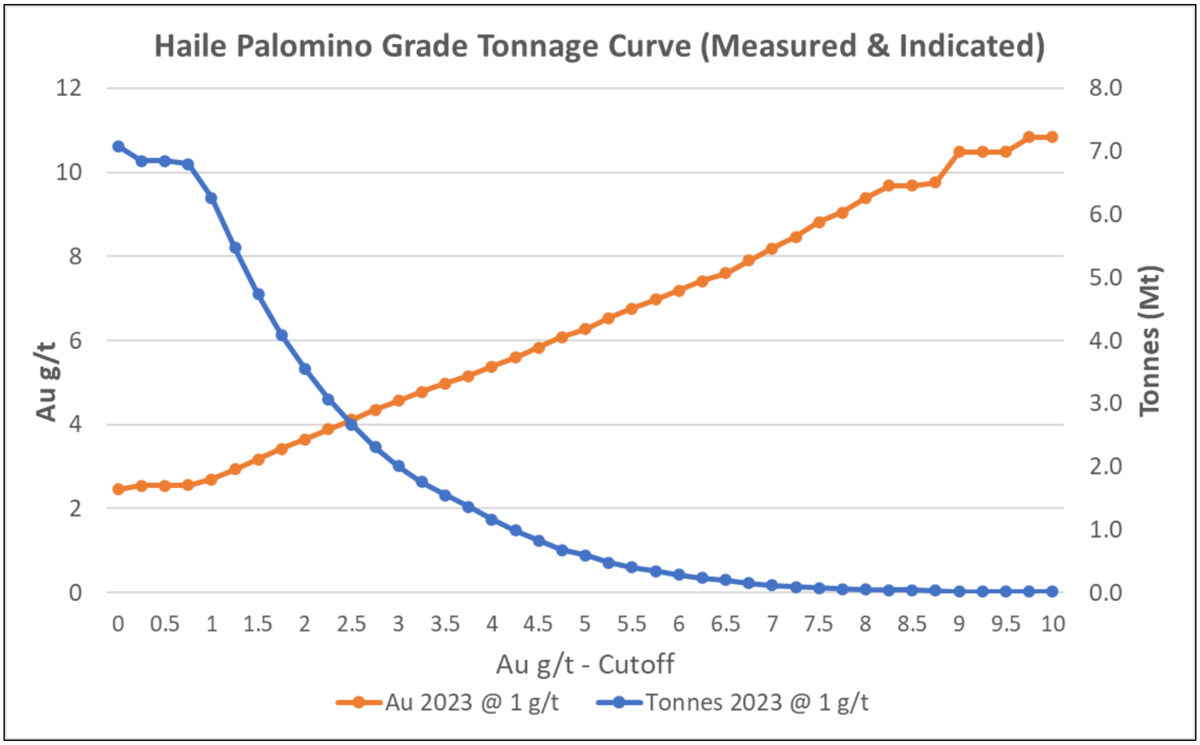
Figure 16-11 shows a grade-tonnage curve for the Horseshoe deposit.



Source: OceanaGold, 2024

Figure 16-11: Horseshoe Underground Model Grade/Tonne Curve Based on Au Cut-off

Figure 16-12 shows a grade-tonnage curve for the Palomino deposit.



Source: OceanaGold, 2024

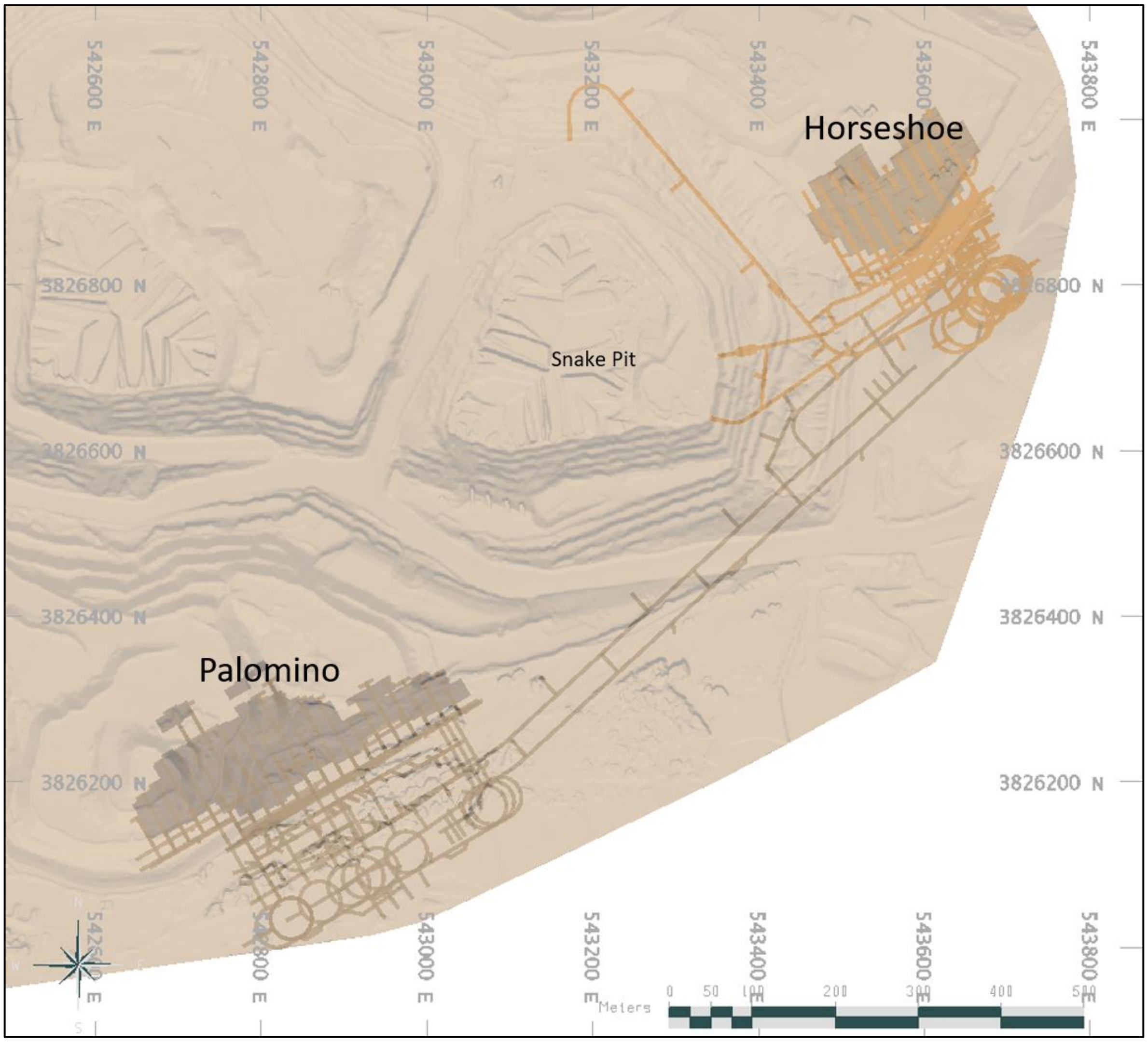
Figure 16-12: Palomino Underground Model Grade/Tonne Curve Based on Au Cut-Off

16.2.2 Underground Geotechnical

Mr. John Tinnuci, of SRK Consulting, was the QP for underground geotechnical studies in previous NI43-101 updates which included the Horseshoe Underground Mine. Mr. Robert Cook, of Call & Nicholas, Inc. (CNI) has assumed the role of underground QP as of the 2024 update which includes the addition of Palomino Underground. As part of this transition, the QP has reviewed previous work at Horseshoe and updated the document based on current mining practices and recent geotechnical studies conducted by CNI, including the geotechnical prefeasibility level study for Palomino.

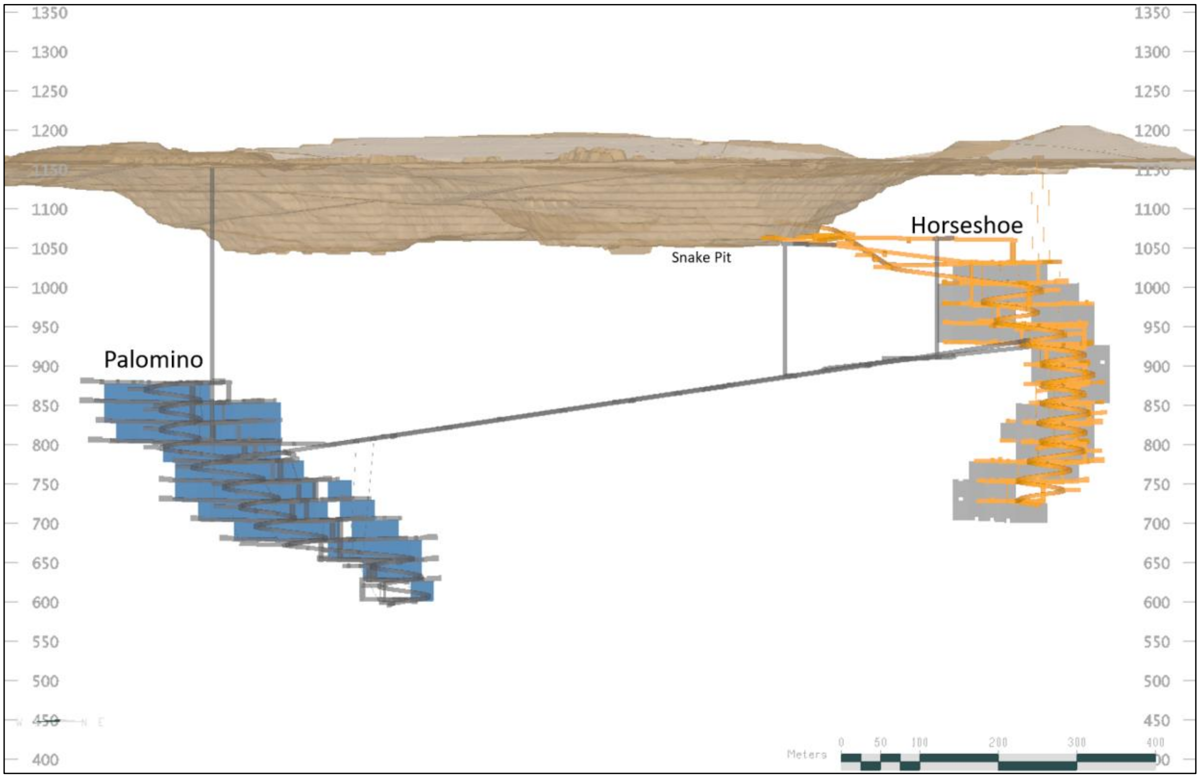
Underground Mining Deposits

Underground mining targets at Haile include the Horseshoe and Palomino Mines. While the Horseshoe Mine is in active production, Palomino is at a prefeasibility level of study. Given that the two deposits are at varying levels of study, there exist different levels of data and analyses (commensurate to their level of study) between the areas, as well. A plan view of the two deposits is presented in Figure 16-13, and section view is presented in Figure 16-14.



Source: CNI, 2024

Figure 16-13: Horseshoe and Palomino Underground Locations (Plan View)



Source: CNI, 2024

Figure 16-14: Horseshoe and Palomino Underground Locations (Looking 330 Deg. Azimuth)

Rock Quality Data

While HGM regularly collects rock quality designation (RQD) on all drilled holes, detailed geotechnical data for rock-mass characterization and measurement of geologic fabric were collected in drilling campaigns in the following years:

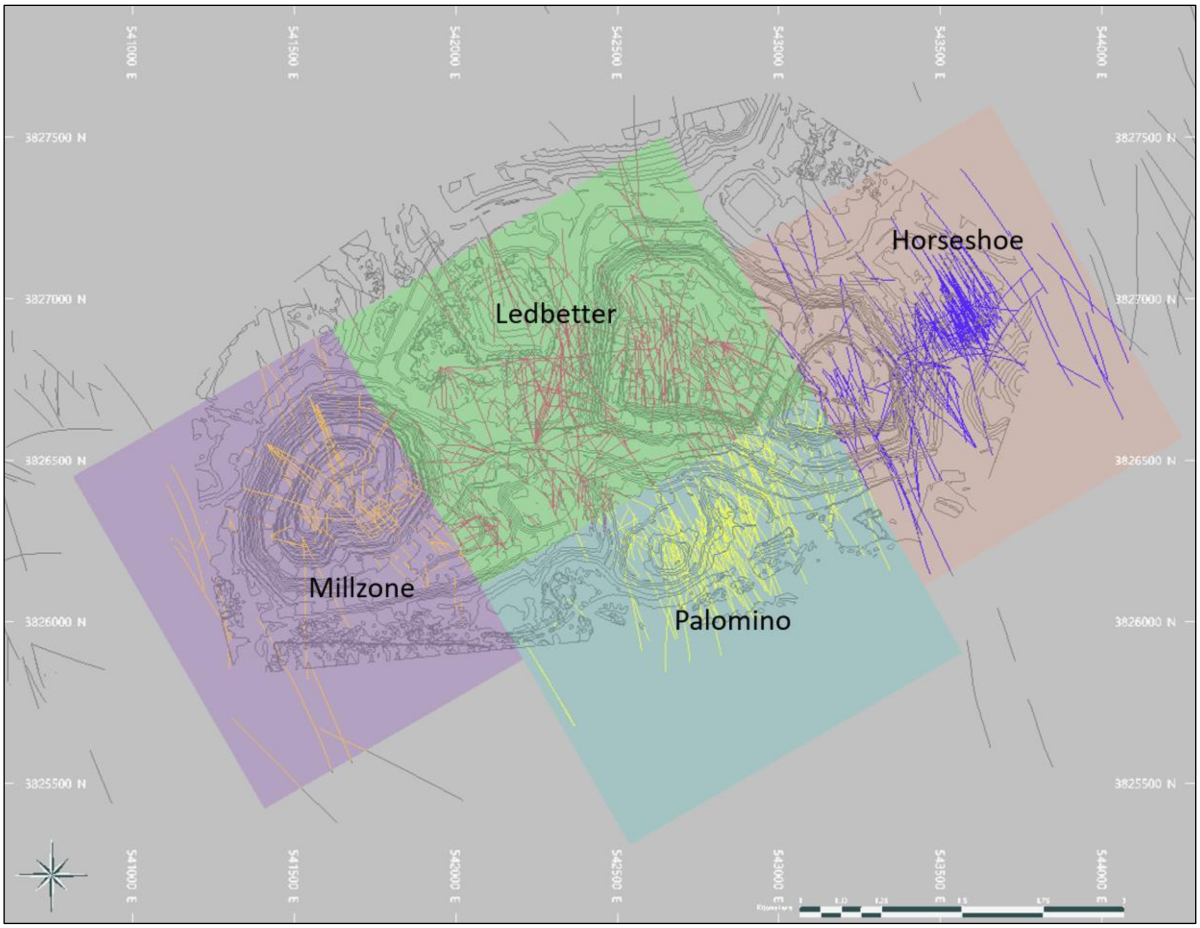
- Golder, 2010
- SRK, 2016 and 2017 (2 campaigns)
- CNI, 2023

The purpose of the geotechnical drilling campaigns was to provide data to be used in feasibility studies. All data from these campaigns were combined into a single database from which a geotechnical block model was developed by CNI (most recently updated in 2023). The geotechnical block model extends across most of the HGM property and includes all underground mining targets. A total of 1,245 core holes were used in the model which had both geological and geotechnical information. All drillholes were logged for rock quality designation (RQD) data, while a smaller subset of the core holes was logged for detailed geotechnical data, including parameters for the calculation of the Modified Rock Tunneling Quality Index Q' (Barton, Lien, and Lunde, 1974). Table 16-21 presents a summary of drillhole data within the targeted mining areas as presented in Figure 16-15. This 43-101 filing includes the underground mining areas of Horseshoe and Palomino.

Table 16-21: Drillhole Data Used in Underground Geotechnical Block Models

Location	RQD		Q'	
	DH #	Meters	DH #	Meters
Horseshoe	230	86122.62	13	3487.82
Ledbetter	439	107624.4	15	2654.95
Palomino	238	83458.04	15	6292.7
Mill Zone	176	31783.62	5	776.52

Source: CNI, 2024



Source: CNI, 2024

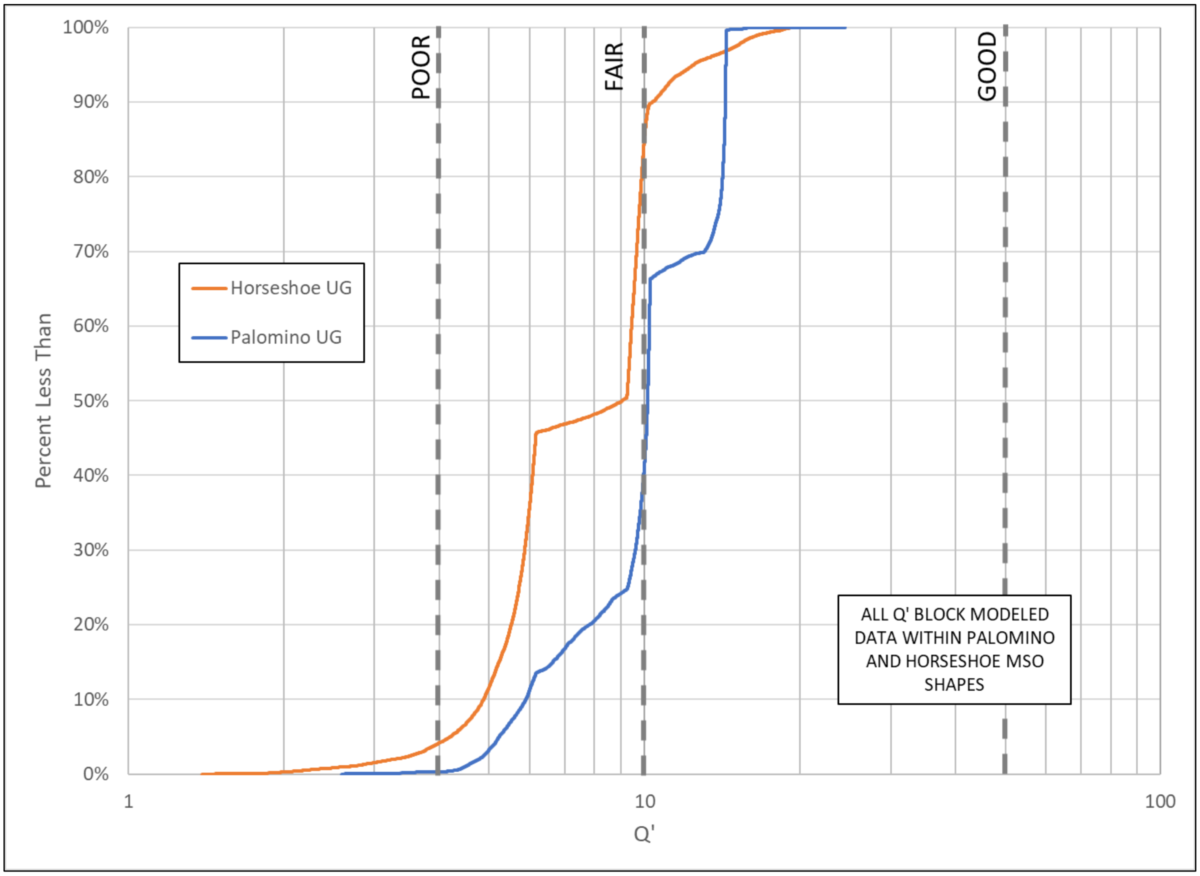
Figure 16-15: Geotechnical Block Model Deposit Locations

The geotechnical block model has been the basis for the review of the Horseshoe UG mine design and the geotechnical evaluation for the Palomino UG mine prefeasibility study. Distributions of the rock quality (Q') from the geotechnical block model are presented in Figure 16-16. Summaries of rock quality by stope muck levels at Palomino and Horseshoe are presented in Table 16-22. The rock qualities by muck level were utilized in estimating stope dimensions.

Table 16-22: Rock Qualities by Muck Level at Palomino and Horseshoe

Palomino			Horseshoe		
Muck Level	**50% Q'**	**75% Q'**	**Muck Level**	**50% Q'**	**75% Q'**
850	14.3	10.2	1025	5.5	4.9
825	10.1	8.4	1000	5.7	5.1
800	10.2	9.9	975	6.1	5.5
775	10.2	10.0	950	6.1	5.6
750	10.2	9.9	925	6.1	5.6
725	10.1	9.8	900	9.3	6.0
700	10.2	9.4	875	9.5	6.4
675	9.9	6.1	850	9.7	7.9
650	5.6	5.1	825	9.8	9.4
625	5.8	5.3	800	9.7	9.3
600	5.9	5.7	775	9.5	6.1
			750	9.4	6.0
			725	5.6	5.0
			700	5.7	4.9

Source: CNI, 2024



Source: CNI, 2024

Figure 16-16: Rock Quality Distributions within the Horseshoe and Palomino UG Areas

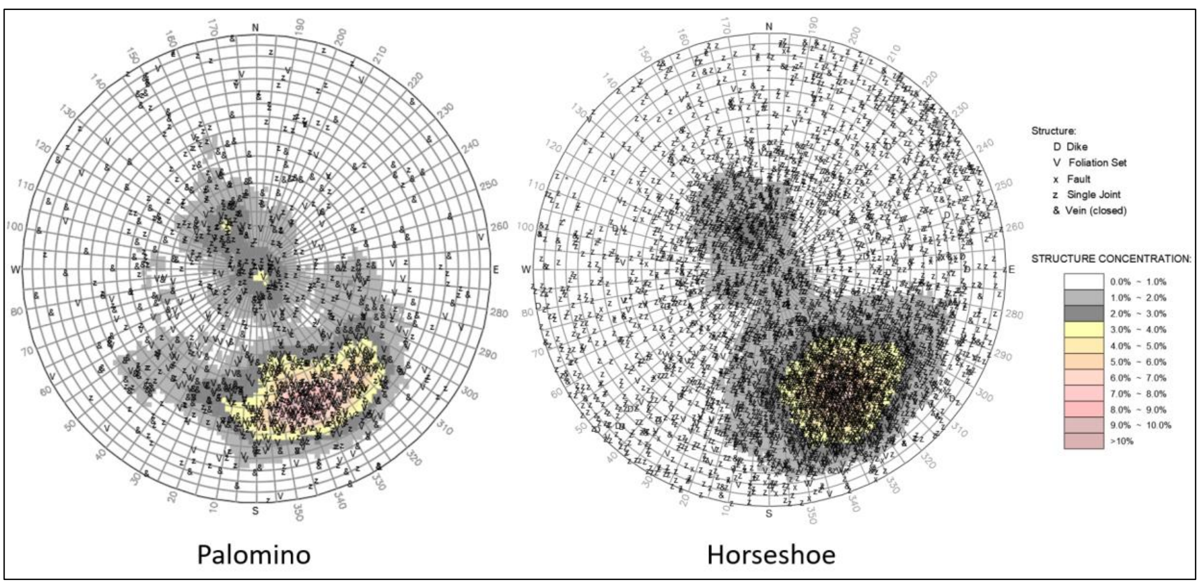
Oriented Core Data

Discontinuity orientation data from drillholes was collected using acoustic televiewer survey and the RELFEX ACT core orienting system. A summary of the drillholes with oriented core data relative to the underground projects is presented in Table 16-23. A total of 42 drillholes were logged for discontinuity orientations at Horseshoe UG, whereas a total of 52 drillholes were logged for discontinuity orientations at Palomino UG. Lower hemisphere, equal areas Schmidt nets are presented for the two underground mines in Figure 16-17 In general, Palomino is less intensely foliated, and has a shallower orientation to the foliation compared to Horseshoe. Furthermore, there is less steeply dipping structure at Palomino than at Horseshoe.

Table 16-23: Drillholes with Measured Discontinuity Orientations

Horseshoe UG				Palomino UG				
DDH0307	DDH0605	DDH0800	DDH1061	DDH0343	DDH0858	DDH1103	DDH1125	DDH1164
DDH0342	(1) DDH0606	DDH0837	DDH1069	DDH0581	DDH0861	DDH1106	DDH1126	DDH1167
DDH0429	(1) DDH0607	DDH0852	DDH1081	DDH0588	DDH0863	DDH1109	DDH1131	DDH1169
DDH0519	(1) DDH0608	DDH0942	DDH1091	DDH0594	DDH0873	DDH1110	DDH1133	DDH1171
DDH0522	(1) DDH0609	DDH0945	DDH1093	DDH0600	DDH0876	DDH1112	DDH1139	DDH1175
DDH0568	(1) DDH0610	DDH0946	DDH1096	DDH0750	DDH0959	DDH1113	DDH1145	RCT0172
(1) DDH0596	(1) DDH0611	DDH0949	DDH1097	DDH0768	DDH0967	DDH1115	DDH1148	(1) DDH1196
(1) DDH0601	DDH0774	DDH0951	DDH1101	DDH0775	DDH1064	DDH1116	DDH1152	(1) DDH1194
(1) DDH0602	DDH0785	DDH0955		DDH0776	DDH1068	DDH1120	DDH1154	
(1) DDH0603	DDH0788	DDH0970		DDH0820	DDH1074	DDH1121	DDH1156	
(1) DDH0604	DDH0798	DDH0977		DDH0854	DDH1099	DDH1123	DDH1159	

Source: CNI, 2024
(1) Orientation Data collected using Acoustic Televiewer Survey



Source: CNI, 2024

Figure 16-17: Palomino and Horseshoe UG Stereonets

In Situ Stress Measurement

The 2017 field program led by SRK included in situ stress measurements which were conducted by Agapito Associates, Inc. (AAI) using a downhole overcoring technique developed by Sigra, Pty. (Sigra) of Brisbane, Australia. A summary of the in-situ stress measurements is presented in Table 16-24. The major horizontal stress (Sigma 1) is more than twice the vertical stress (Sigma 3).

Table 16-24: In Situ Stress Measurement Summary

	Sigma 1	Sigma 2	Sigma 3
Bearing/Plunge (Deg.)	088/00	178/00	178/90
Stress Gradient (MPa/m)	0.0561	0.0376	0.0259

Source: OceanaGold, 2020

Rock Strength Laboratory Testing

A summary of the laboratory testing program for rock strength is provided in Table 16-25.

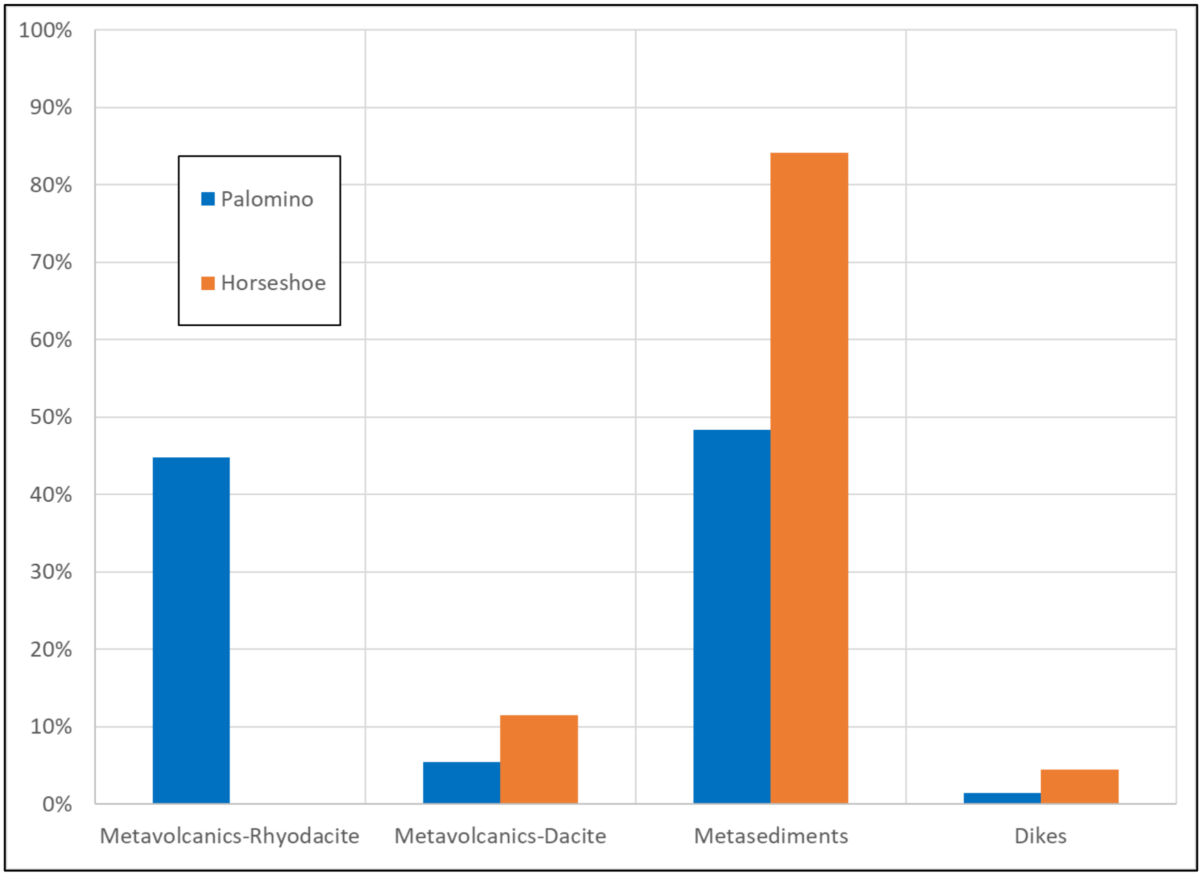
Table 16-25: Summary of the Laboratory Testing

Year	Laboratory	Rock Type	Uniaxial Compression	Uniaxial Compression (with E&V)	Triaxial Compression	Disk Tension	Small Scale Direct Shear	Total
(1) 2009 (Golder OP)	ATT	Metasediments	1	1	4		2	**13**
		Dike	1	1		-	-	
		Metavolcanics	2	1		-	-	
(1) 2011 (Golder UG)	ATT	Metasediments	12	-	-	-	-	**12**
		Dike						
		Metavolcanics						
(1) 2016 (SRK UG)	Agapito	Metasediments	7	8	5	1	3	**37**
		Dike	1	1	-	-	1	
		Metavolcanics	2	2	3	1	2	
(1) 2017 (SRK UG)	Agapito	Metasediments	11	9	10	10	6	**125**
		Dike	3	4	-	2	1	
		Metavolcanics	21	16	15	9	8	
2017 (CNI)	CNI	Metasediments	-	-	-	-	7	**18**
		Dike	-	-	-	-	5	
		Metavolcanics	-	-	-	-	6	
2020 (CNI)	CNI	Metasediments	-	-	13	23	-	**113**
		Dike	-	-	15	30	-	
		Metavolcanics	-	-	16	16	-	
2023 (CNI)	CNI	Metasediments	3	2	5	6	5	**59**
		Dike	-	-	-	-	-	
		Metavolcanics	6	-	12	17	3	
Total			**70**	**45**	**98**	**115**	**49**	**377**

(1) Source: SRK, 2017 & CNI, 2024

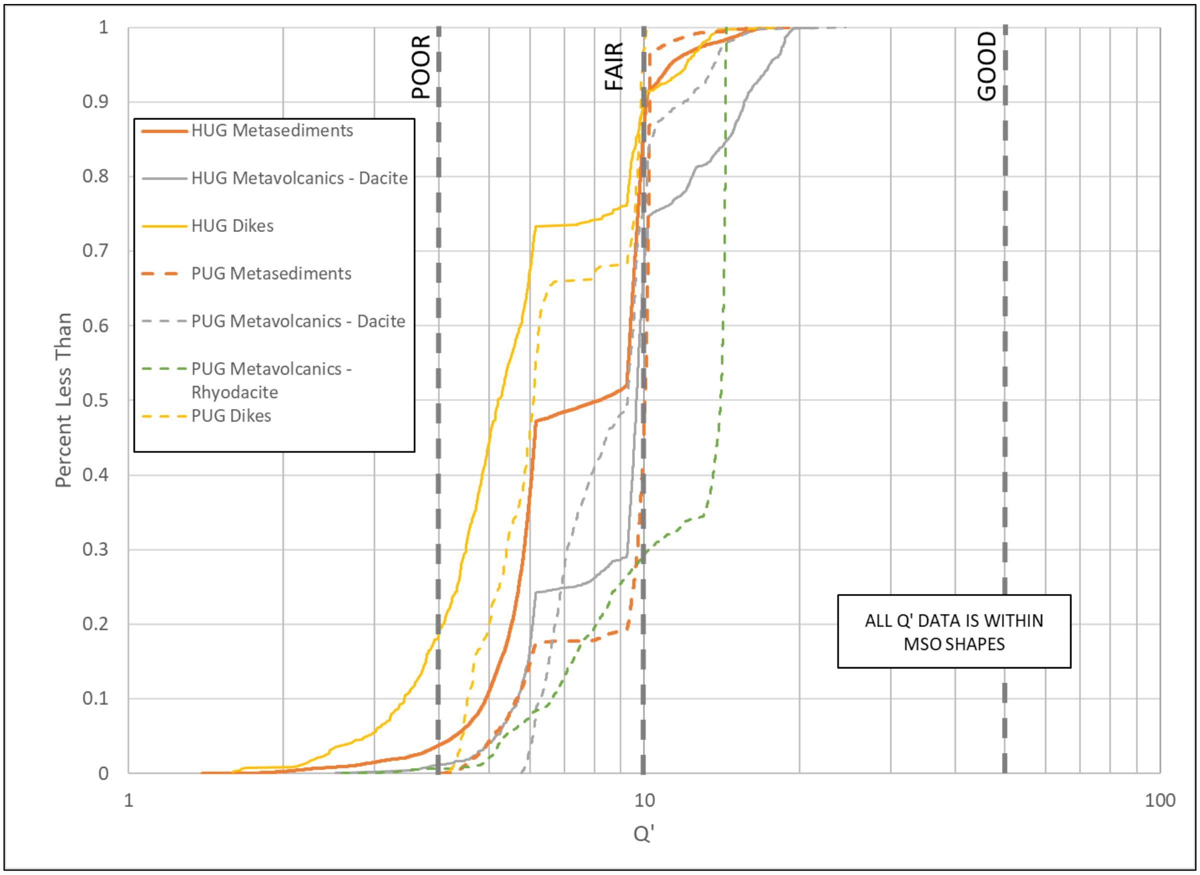
Rock-Mass Characterization

Both properties have similar geology consisting of metasediments and metavolcanics (dacite and rhyodacite) overlain by some volcanic tuff, coastal sands, and saprolite. Ore is predominantly hosted within the metasediments at Horseshoe, whereas the ore at Palomino is hosted within both the metasediments and metavolcanics (rhyodacite and dacite). A distribution of Palomino and Horseshoe stope targets by rock type is presented in Figure 16-18. As presented in Figure 16-16, most of the rock quality within the mining areas at Horseshoe and Palomino is of fair to good rock quality using Barton's rock quality classification. Palomino has superior rock quality compared to Horseshoe. In general, the metavolcanics are better rock quality compared to the metasediments at both areas, due to less intense foliation within the metavolcanics compared to the metasediments. Figure 16-19 presents distributions of rock quality by rock type at Horseshoe and Palomino.



Source: CNI, 2024

Figure 16-18: Palomino and Horseshoe Stopes by Rock Type Percentage

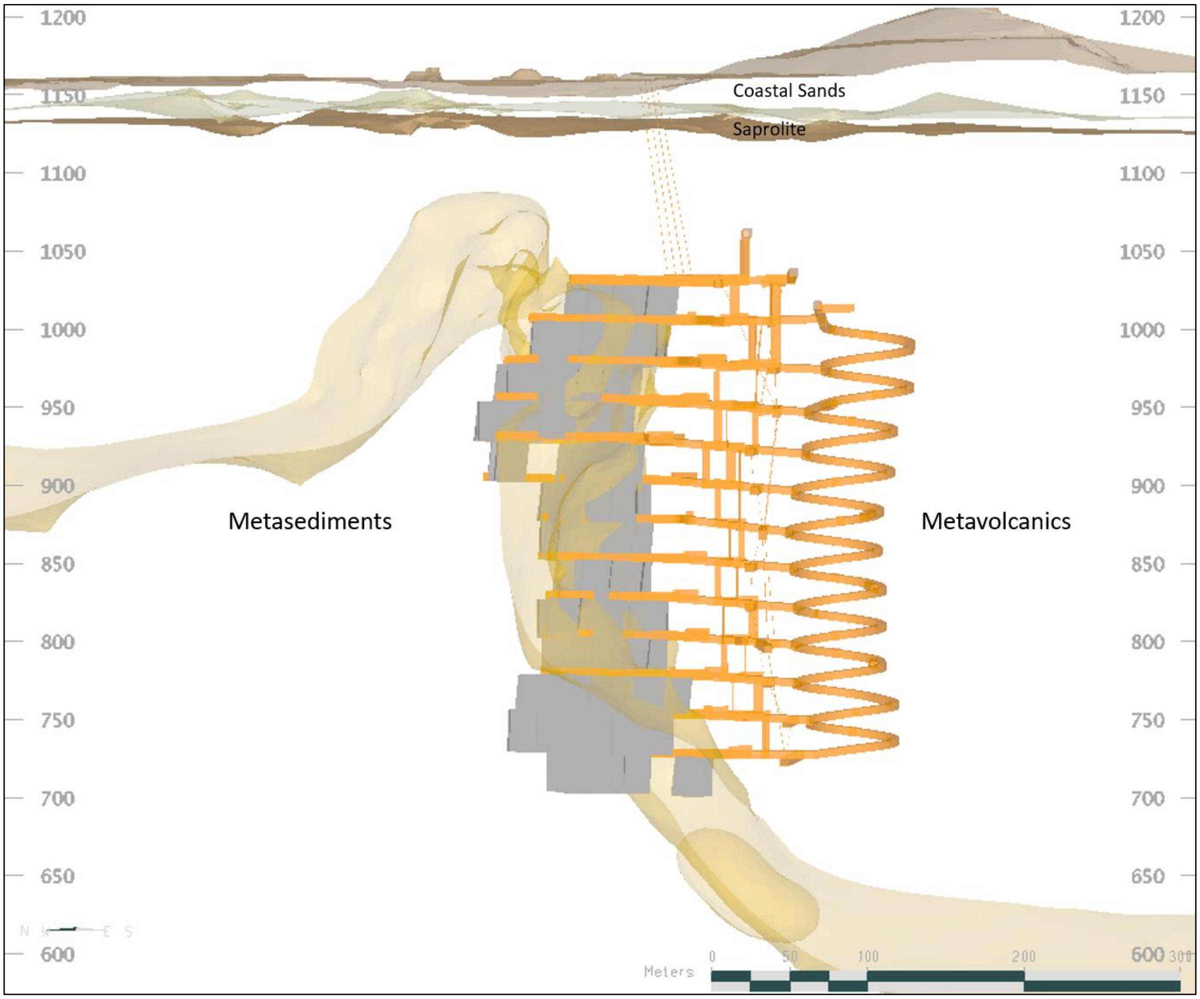


Source: CNI, 2024

Figure 16-19: Palomino and Horseshoe Rock Qualities by Rock Type

Horseshoe Underground Rock Strengths

A northwest-southeast section view through the Horseshoe Mine is illustrated in Figure 16-20. The upper elevations contain thin layers of Coastal Sands and Saprolite followed by the basement rocks of metasediments (MS) and metavolcanics (MV). The upper elevations (nominal 100 m thick) of the metavolcanics beneath the saprolite are more fractured and weathered than the deeper metavolcanics. The mineralized zone is almost entirely within the metasediments. Most of the capital infrastructure is within the metavolcanics.



Source: CNI, 2024

Figure 16-20: Generalized Geotechnical Cross Section – Horseshoe (Looking NE)

Laboratory uniaxial compression strength test (UCS) results show that the strength of the weathered metavolcanic rocks is 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 16-26 presents a summary of the laboratory testing results at Horseshoe.

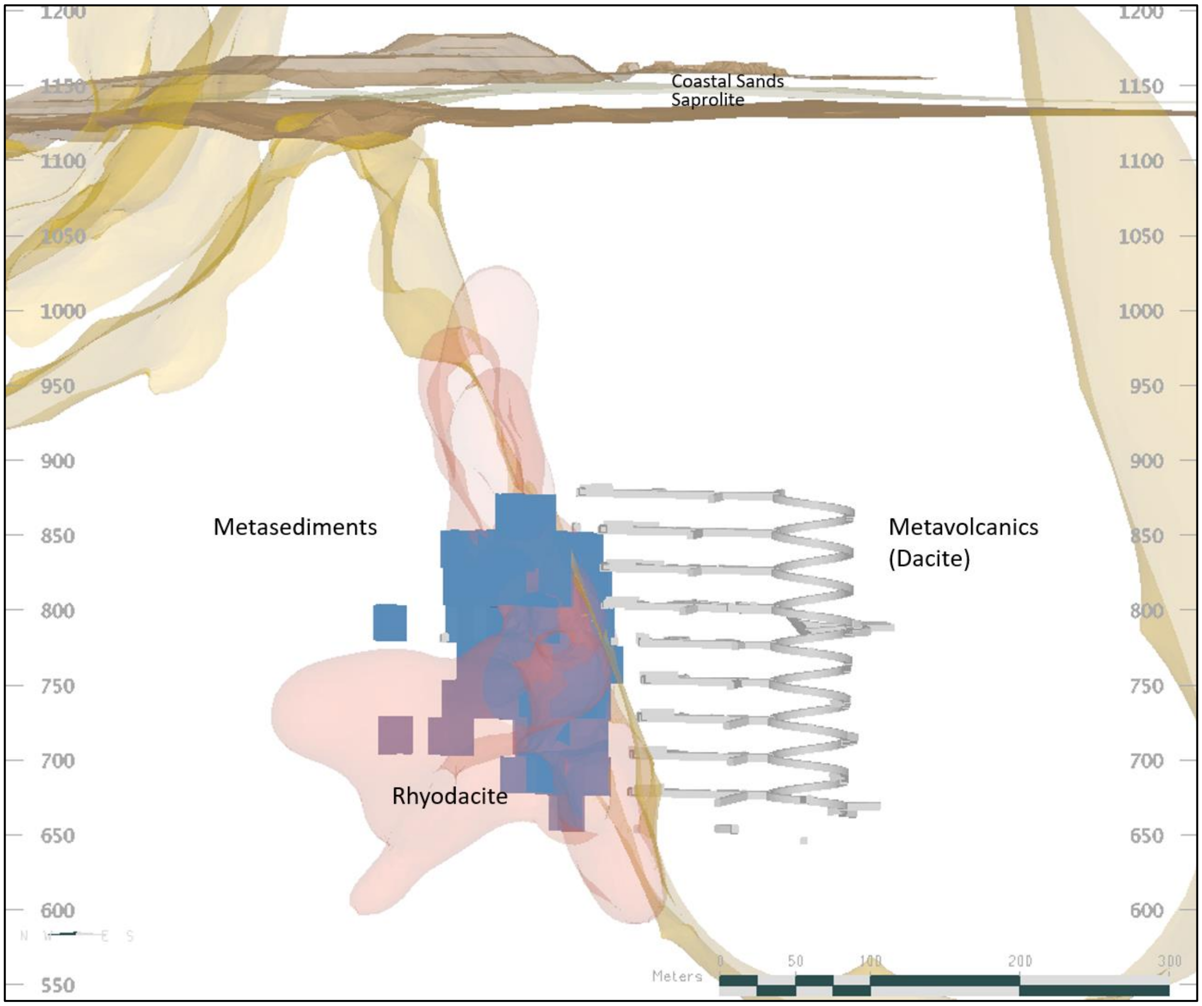
Table 16-26: Summary of Strength Properties (mi and σ_{ci}) - Horseshoe

Geotechnical Unit		Density (t/m^3)	Laboratory Test		Intact Rock Properties from TCS	
Lithology	**Weathered**		**σ_t (MPa)**	**UCS (MPa)**	**σ_{ci} (MPa)**	**mi**
Metavolcanics	Weathered	2.60	4.4	60	61	14
	Fresh	2.70	12.7	153	158	13
Metasediments Foliated	Fresh	2.76	6.2	50.5	59	10
Metasediments Not Foliated		2.75	11.8	127.8	138	12
Dike	Fresh	2.92	12.94	146	-	-

Source: SRK, 2017

Palomino Underground Rock Strengths

A northwest-southeast section view through the Palomino Mine is illustrated in Figure 16-21. The upper elevations contain thin layers of Coastal Sands and Saprolite followed by the basement rocks of metasediments (MS) and metavolcanics (MV) where the metavolcanics consist of Rhyodacite and Dacite. The upper elevations (nominal 100 m thick) of the metavolcanics beneath the saprolite are tuffaceous and more fractured and weathered than the deeper metavolcanics. The mineralized zone is predominantly in the Rhyodacite. Most of the capital infrastructure is within the metavolcanics (Dacite). Table 16-27 presents a summary of the laboratory testing results at Palomino.



Source: CNI, 2024

Figure 16-21: Generalized Geotechnical Cross Section – Palomino (Looking NE)

Table 16-27: Summary of Strength Properties (mi and σ_{ci}) - Palomino

Geotechnical Unit	Density (kg/m^3)	Laboratory Test		Intact Strength Properties	
		σ_t (MPa)	UCS (MPa)	σ_{ci} (MPa)	m_i
Metasediments	2767	8.6	45.2	45.2	7.1
Metavolcanics (Dacite + Rhyodacite)	2732	7.6	109.0	109.0	10.2
Dike	2920	12.9	146.0	146.0	--

Source: CNI, 2024

Stope Design Parameters

Transverse longhole stope (LHS) mining has been selected as the mining method for both orebodies. The LHS method requires a top cut, which is used as a drilling platform, and a bottom cut, which is used as a mucking level. The pillar between the top cut and the bottom cut is excavated by initiating a small vertical opening (slot raise), and then by line blasts that progressively open up a large excavation with four walls (two side walls and two end walls) and a back (roof). All ore is drawn from the bottom cut sublevel. Backfill is placed to fill the void space. Backfilled pillars can then be used as the sidewalls for subsequent secondary stopes. Stopes that have total strike lengths in excess of their stable length can be paneled such that consolidated backfill is placed once the stope is at its maximum stable length. Subsequent panels can be re-slotted against the poured backfill and stoping can re-commence until the entire strike length of the stope has been mined. Risks associated with subsidence are generally eliminated due to the placement of backfill in the completed stopes. The total open stope height is 25 m, with the overdrive drift above the open stope, resulting in a full exposure height of 30 m (sill-to-back). Stope panel dimensions currently vary between the two projects:

- Horseshoe: 20 m wide x 25 m high (30 m high sill to back) x 25 m long; to improve stability, CNI has recommended that the stopes be narrowed to 15 m wide for all mining below the first sill pillar. HGM are currently incorporating this design change.
- Palomino: 15 m wide x 25 m high (30 m high sill to back), lengths vary by rock type and mining level as summarized in Table 16-28.

Table 16-28: Summary of Stope Dimensions by Mining Level at Palomino

Palomino UG												
Muck Level	DESIGN DIMENSIONS (m)											
	MS/MV/RD Design Dimensions (m)			MS Design Dimensions (m)			MV Design Dimensions (m)			RD Design Dimensions (m)		
	Height	Width	Length	Height	Width	Length	Height	Width	Length	Height	Width	Length
850	30.0	15.0	40.2	30.0	15.0	40.1	30.0	15.0	46.9	30.0	15.0	68.1
825	30.0	15.0	38.5	30.0	15.0	39.1	30.0	15.0	33.3	30.0	15.0	40.1
800	30.0	15.0	38.9	30.0	15.0	38.6	30.0	15.0	34.9	30.0	15.0	47.9
775	30.0	15.0	35.6	30.0	15.0	35.6	30.0	15.0	39.8	30.0	15.0	35.6
750	30.0	15.0	34.5	30.0	15.0	34.5	30.0	15.0	33.2	30.0	15.0	35.0
725	30.0	15.0	32.6	30.0	15.0	32.5	30.0	15.0	31.9	30.0	15.0	53.0
700	30.0	15.0	30.3	30.0	15.0	30.4	30.0	15.0	37.2	30.0	15.0	29.7
675	30.0	15.0	24.7	30.0	15.0	20.5	30.0	15.0	35.6	30.0	15.0	45.1
650	30.0	15.0	20.3	30.0	15.0	20.3	30.0	15.0	25.0	30.0	15.0	42.9
625	30.0	15.0	19.1	30.0	15.0	18.8	30.0	15.0	23.3			
600	30.0	15.0	18.8	30.0	15.0	18.7						

Source: CNI, 2024

Stability Graph Method for Stope Dimensions

The Mathews Stability Graph Method (1980) was used to evaluate stope dimensions at both sites. This method is an empirical design tool based on case histories from hard rock mines which typically have good to very good quality rock.

The Stability Graph method accounts for key factors influencing open stope design, including rock-mass strength and structure, stresses surrounding the opening, and the shape and orientation of the stope. The method is based on two calculated factors: N’ (modified stability number) and S (hydraulic radius). The stability number (N’) is comprised of the following components:

$$N' = Q' * A * B * C$$

where: Q’ = Modified Q Tunneling Quality Index
A = Rock Stress Factor
B = Joint Orientation Factor
C = Gravity Adjustment Factor

The hydraulic radius (S) is calculated as follows:

$$S = (\text{area of stope face - m}^2) / (\text{perimeter of stope face - m})$$

N' and S values are used to classify the excavations as one of the following:

- Stable zone
- Stable without support
- Stable with support
- Supported transition zone
- Caving zone

The analysis assumes the following:

- Stress field orientations and magnitudes as summarized in Table 16-24
- Q' based on the 75% reliability values from modeled blocks within each 30 m mining level by rock type (Table 16-22)
- Stopes oriented at 330° azimuth alignment
- Flat stope backs and vertical stope walls
- Stope walls oriented approximately parallel to the primary joint orientation
- A nominal unconfined compressive strength (UCS) of 75 MPa
- Stopes of 20 m width at Horseshoe and 15 m width at Palomino. The 20 m width at Horseshoe is based on the previous analyses conducted by SRK.

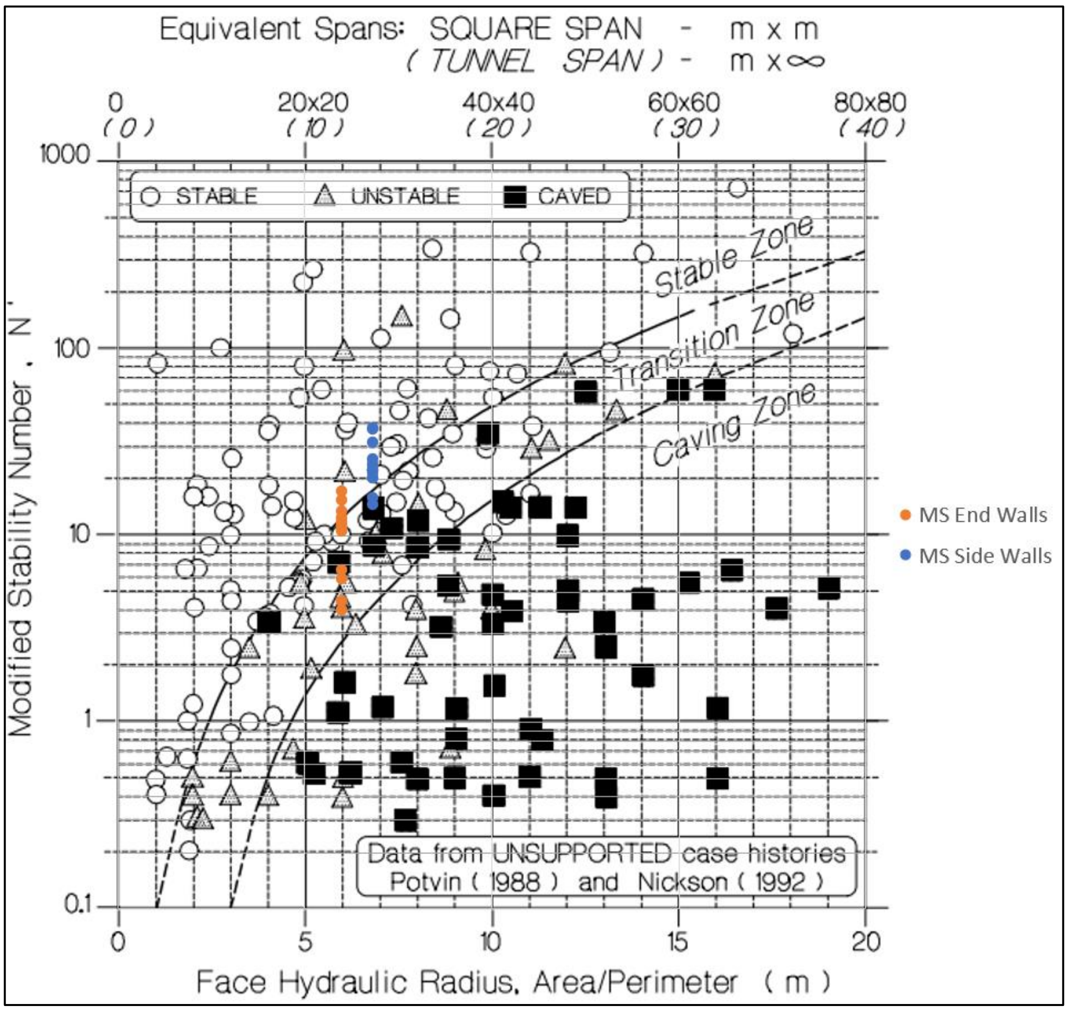
Mathews Stability Graph Results

Each mining sublevel was analyzed to predict maximum stable stope configurations by rock type. The stability charts updated by Hutchinson and Diederichs (1996) were utilized for all stope dimension evaluations. For non-supported surfaces such as end walls and side walls, it is recommended that the upper boundary of the transition zone between stable and caving cases be used for design (Hutchinson & Diederichs, 1996). For stope backs, the stability number was plotted to the stable with support line and assumes effective cable bolt support across the stope spans. Stopes were optimized for length for each 30 m stope sublevel while maintaining constant stope widths. With effective support installed within stope backs, stability is controlled by sidewall dimensions. Results of the Mathews stability graph analyses for side and end walls for Horseshoe and Palomino are presented in Table 16-29 through Table 16-30 and Figure 16-22 through Figure 16-23.

Table 16-29: Stability Graph Results - Horseshoe

Muck Level	Horseshoe					
	MS Stability Number N'			MS Hydraulic Radius (m)		
	Backs	Side Walls	End Walls	Backs	Side Walls	End Walls
1000	0.98	20.45	17.20	6.55	7.20	6.76
975	0.86	21.83	15.46	6.42	7.38	6.50
950	0.74	22.38	13.52	6.28	7.45	6.19
925	0.63	22.44	11.80	6.12	7.45	5.89
900	0.59	24.14	11.17	6.06	7.65	5.77
875	0.54	25.74	10.55	5.98	7.84	5.65
850	0.57	31.45	11.53	6.04	8.43	5.84
825	0.59	37.66	12.40	6.07	9.01	6.00
800	0.52	37.11	11.03	5.95	8.96	5.75
775	0.29	22.44	6.57	5.52	7.45	4.75
750	0.25	20.47	5.84	5.43	7.21	4.55
725	0.20	15.84	4.45	5.29	6.56	4.12
700	0.20	14.75	4.02	5.28	6.39	3.97

Source: CNI, 2024



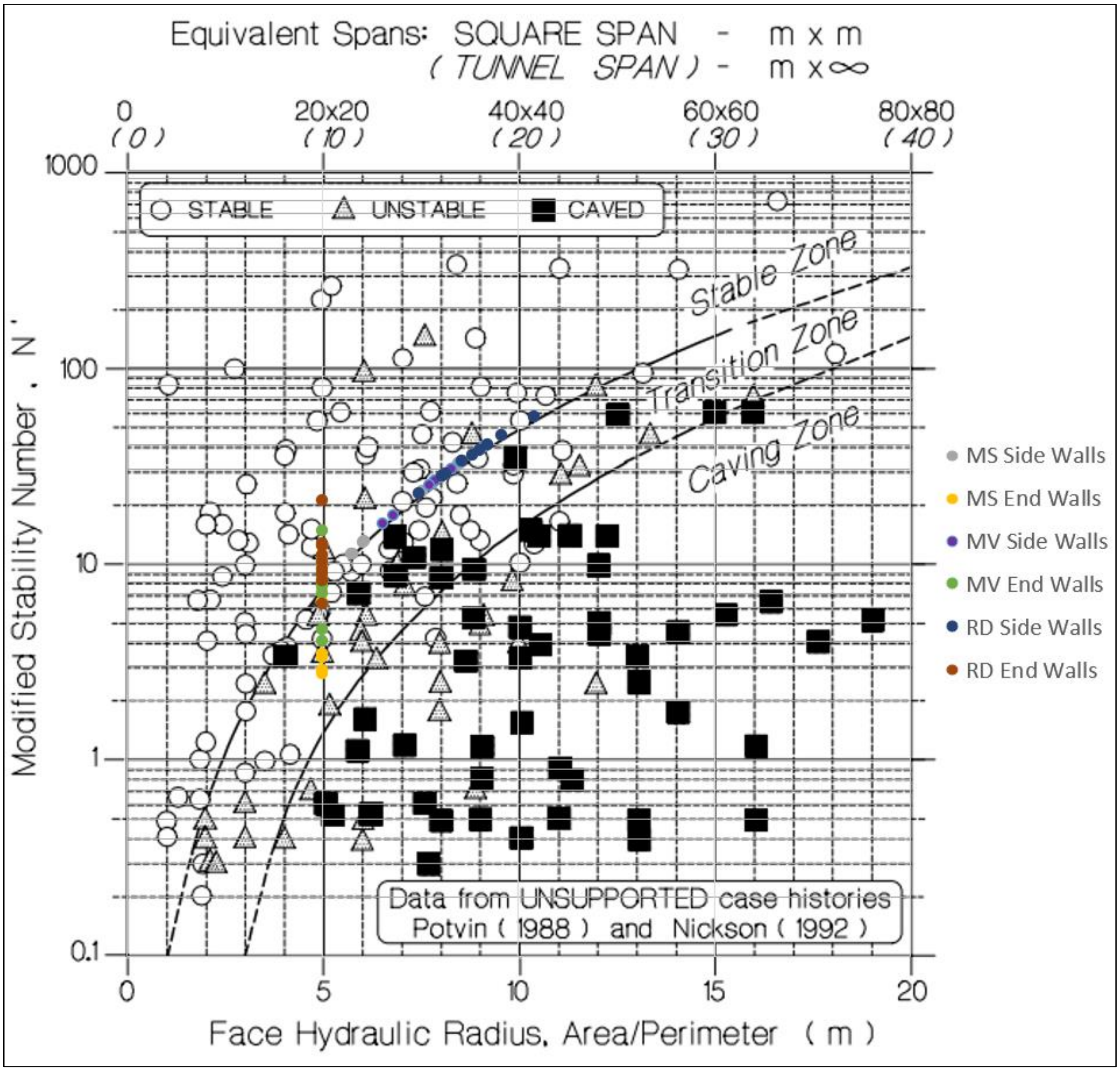
Source: CNI, 2024

Figure 16-22: Stability Graph Results – Horseshoe

Table 16-30: Stability Graph Results - Palomino

Muck Level	Palomino								
	MS Stability Number N'			MV Stability Number N'			RD Stability Number N'		
	Backs	Side Walls	End Walls	Backs	Side Walls	End Walls	Backs	Side Walls	End Walls
850	0.60	32.95	12.30	0.72	39.23	14.65	1.02	55.89	20.87
825	0.51	32.04	10.75	0.41	26.23	8.80	0.52	32.97	11.06
800	0.45	31.56	9.67	0.40	27.89	8.55	0.57	40.20	12.32
775	0.40	28.58	8.91	0.45	32.64	10.17	0.39	28.52	8.89
750	0.35	27.49	8.08	0.33	26.17	7.70	0.35	27.90	8.20
725	0.32	25.37	7.15	0.31	24.83	6.99	0.56	44.50	12.54
700	0.31	23.20	6.39	0.40	30.16	8.31	0.30	22.48	6.19
675	0.18	12.91	3.46	0.40	28.57	7.66	0.53	37.67	10.10
650	0.19	12.72	3.34	0.26	17.55	4.61	0.54	35.65	9.36
625	0.18	11.22	2.85	0.25	15.84	4.02			
600	0.19	11.09	2.76						
Muck Level	MS Hydraulic Radius (m)			MV Hydraulic Radius (m)			RD Hydraulic Radius (m)		
	Backs	Side Walls	End Walls	Backs	Side Walls	End Walls	Backs	Side Walls	End Walls
850	6.08	8.58	5.98	6.24	9.15	6.37	6.59	10.41	7.26
825	5.93	8.49	5.69	5.77	7.89	5.29	5.95	8.58	5.75
800	5.83	8.44	5.48	5.73	8.07	5.23	6.03	9.23	5.98
775	5.73	8.14	5.31	5.83	8.55	5.58	5.73	8.14	5.31
750	5.64	8.03	5.13	5.60	7.88	5.04	5.65	8.07	5.15
725	5.57	7.80	4.90	5.55	7.73	4.86	6.01	9.58	6.02
700	5.55	7.54	4.70	5.74	8.30	5.18	5.53	7.46	4.65
675	5.24	6.09	3.76	5.75	8.14	5.03	5.97	9.01	5.56
650	5.27	6.05	3.71	5.45	6.81	4.17	5.98	8.83	5.41
625	5.23	5.78	3.50	5.42	6.56	3.97			
600	5.26	5.76	3.46						

Source: CNI, 2024



Source: CNI, 2024

Figure 16-23: Stability Graph Results – Palomino

Dilution Estimates Using the ELOS Method

The equivalent length of overbreak was estimated using the ELOS (Equivalent Length of Slough) chart (Clark & Pakalnis, 1997). The ELOS chart is an extension of the Mathews stability graph, using empirical evidence to estimate the amount of overbreak for different ground conditions at varying hydraulic radii. Intentionally mining stopes of poorer rock quality at widths beyond their stable configuration will lead to additional sloughing. The ELOS method is widely used to predict dilution in LHS mining.

Table 16-31 presents the ELOS design zones. Dilution estimates are presented in Table 16-32.

Table 16-31: ELOS Design Zones

ELOS Range	ELOS Design Zones
ELOS <0.5 m	Blast damage only; surface is self-supporting.
ELOS = 0.5 m - 1.0 m	Minor sloughing; some failure from unsupported stope wall should be anticipated before a stable shape configuration is achieved.
ELOS = 1.0 m - 2.0 m	Moderate sloughing; significant failure from unsupported stope wall is anticipated before reaching stable shape configuration.
ELOS > 2.0 m	Severe sloughing; large failures from unsupported stope wall should be anticipated. Wall collapse is possible.

Source: CNI, 2024

Table 16-32: Summary of Dilution ELOS Estimates

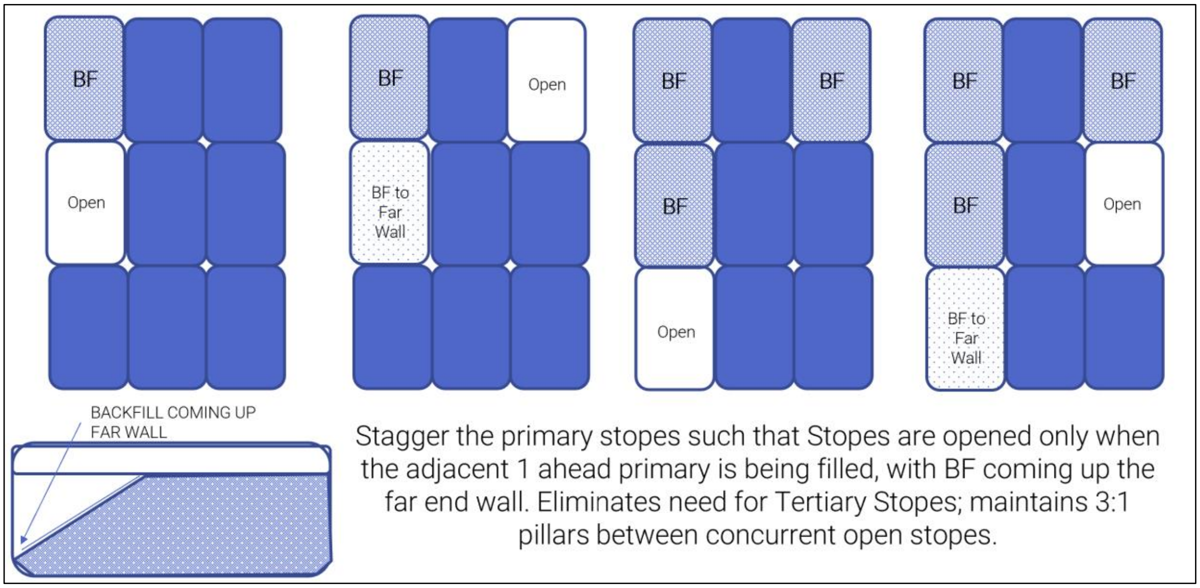
Type	ELOS Value (m)
Sidewalls (rock)	0.50
Sidewalls (backfill)	0.25
Endwalls (rock and backfill)	0.15
Bottom (backfill)	0.05

Source: CNI, 2024

Longhole Stope Mining Sequencing

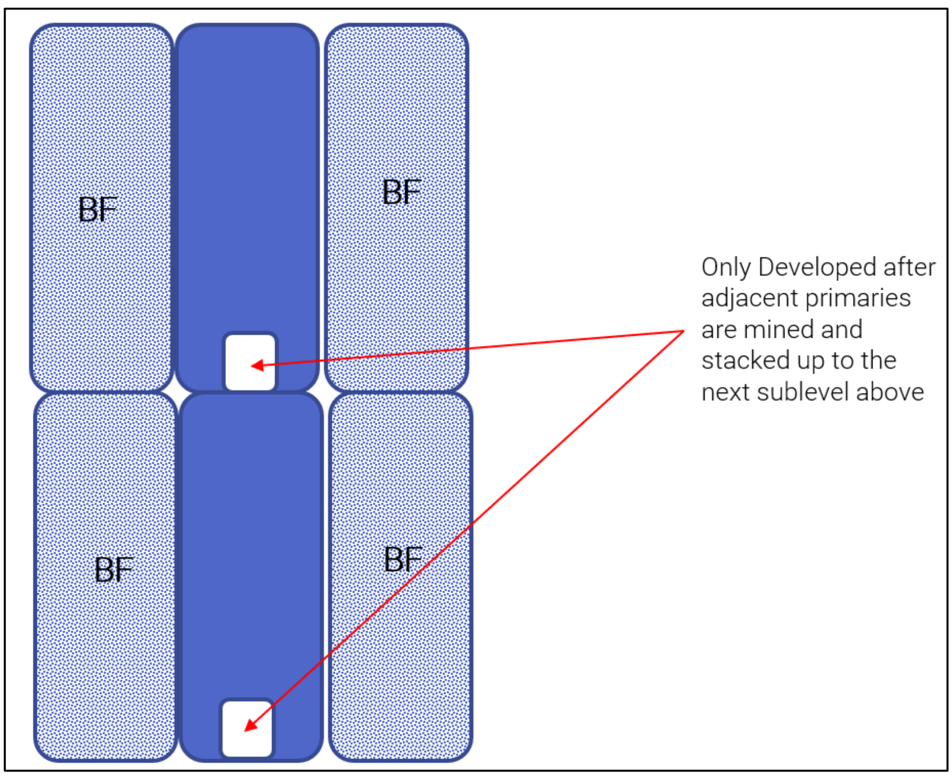
A staggered 1-3-5 sequence will be used, as presented in Figure 16-24. The sequence offers the advantage of allowing several primary stopes to be mined simultaneously, which increases productivity. To maintain pillar stability, both sides of a pillar cannot be mined simultaneously. Stopes should be staggered such that panels are backfilled before opening the nearest stope in section. By utilizing this sequence with a staggered leading panel, a 3x pillar width (of rock or backfill) is maintained between concurrently open stope panels. Furthermore, one full stope sublevel must be mined above a secondary pillar before recovering it. Stope top cut and bottom cut development cannot commence until the adjacent stopes are filled to the entire vertical extent (Figure 16-25).

As the stope sequence progresses, mining induced stress redistributions will occur which may be detrimental to later stage stope recovery and accesses. The impacts of these detrimental stress distributions should be considered as the Project progresses to the next phase of study.



Source: CNI, 2024

Figure 16-24: Staggered 1-3-5 Sequence Plan View



Source: CNI, 2024

Figure 16-25: 1-3-5 Sublevel Vertical Sequence Section View

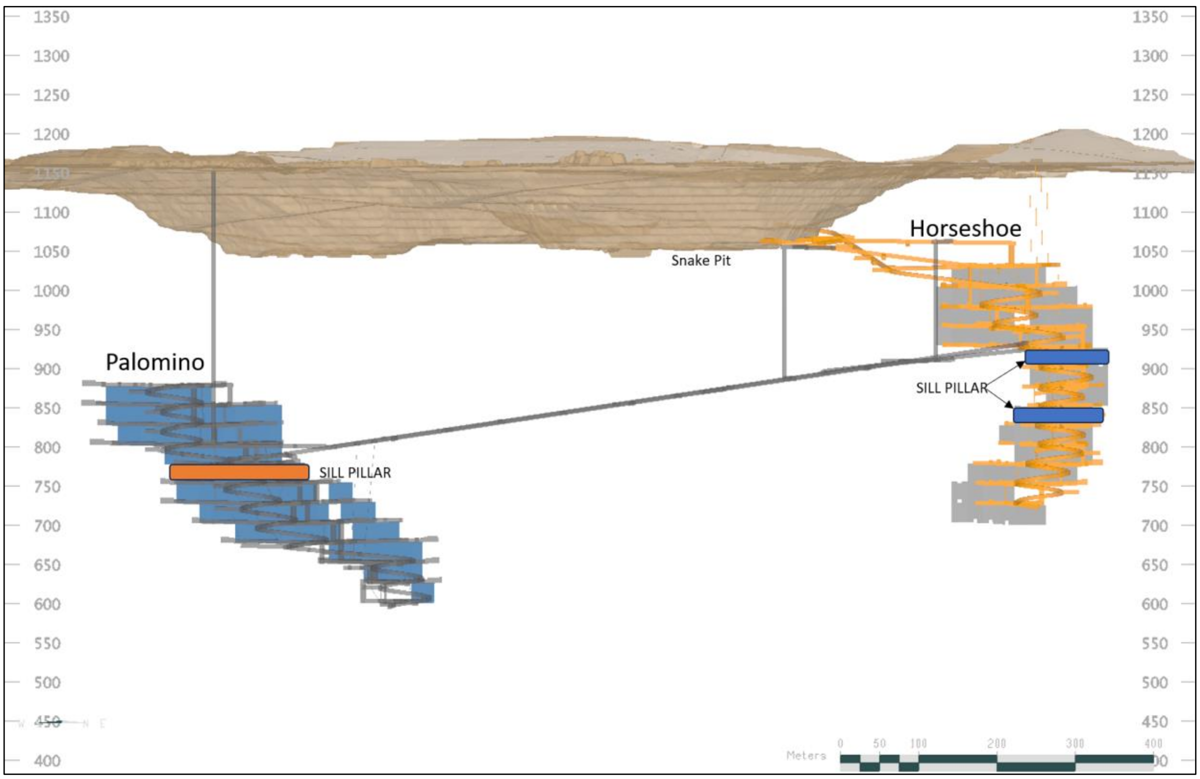
Access Pillars and Sill Pillars

To minimize mining-induced damage to long-term access drifts (footwall drives), the setback distances used in the design for both properties include:

- Haulage setback of 18 m to 22 m from stopes
- Main ramp setback of 64 m to 70 m from stopes

Sill pillars are planned at both underground mines to divide stoping blocks so that the uppermost stopes may be brought into production prior to completing development to the lower levels (Figure 16-26):

- Horseshoe: Sill levels between the 825 and 850 Levels and between the 900 and 925 Levels
- Palomino: Sill levels between the 750 and 775 Levels



Source: CNI, 2024

Figure 16-26: Sill Pillar Locations

While no mining strategy has been selected for these pillars, HGM have estimated a nominal 80% recovery. CNI believes this is a reasonable estimate. As more mining experience is gained at Horseshoe, potential late-stage recovery methods such as room and pillar, cut and fill, or underhand stoping can be further evaluated.

Backfill

To achieve full recovery of the orebodies, stopes must be backfilled with consolidated fill material. Stope panels will be backfilled with cemented rock backfill (CRF) which will be trucked into nearby

CRF stockpiles and then remoted into open stopes by loaders to fill them to their original top cut sill elevation and then top cuts can be compact filled to achieve tight filling to the back.

The purpose of the CRF is to support and confine the sidewalls of primary stopes and allow subsequent stope panels to be re-slotted against CRF endwalls. Consequently, the CRF must remain stable at a full vertical stope height when adjacent secondary stopes are opened, or when re-slotting a stope panel. CRF strength estimates were calculated using the frictionless wedge model proposed by Mitchell et al. (1982) For a 30 m total exposed wall height, 600 Kpa is required to achieve a 1.5 factor of safety. The current design target strength at Horseshoe is 700 kPa. A 3.8% cement content has been planned for which should reliably achieve the strength target. HGM are currently conducting laboratory strength testing on CRF cylinders to confirm that the recipe achieves the design strength.

Secondary stopes will be mined between backfilled pillars. In these cases, uncemented fill may be used to fill the stopes. However, when secondary stopes are paneled, a portion of the secondary stope will require CRF fill in order for the subsequent slot to be blasted against. In these cases, the secondary stope panel should be filled to angle of repose up to the original sill height of the top cut.

Ground Support and Drift Stability

Ground support varies based on the permanence and exposure of each individual mine opening. In general, discrete support requirements have been established for permanent development and production mining headings.

Horseshoe Mine Permanent Support (Decline and Footwall Drives)

Permanent development at Horseshoe includes 2 bolting systems:

- The decline is supported using galvanized Split Set (SS-46) friction stabilizers and #8 rebar as presented in Figure 16-27.
- Footwall drives (level accesses) which must remain open through the entirety of mine life, will be supported exclusively with galvanized Split Set (SS-46) bolts.

Because these friction type bolts are susceptible to corrosion, there is some geotechnical risk to this support strategy. However, given the short mine life (less than 10 years) associated with Horseshoe, these bolt types may still have a suitable service life. To manage this risk, HGM will perform regular inspections and conduct scheduled pull testing to identify areas which have deteriorated due to bolt corrosion and rehab as necessary.

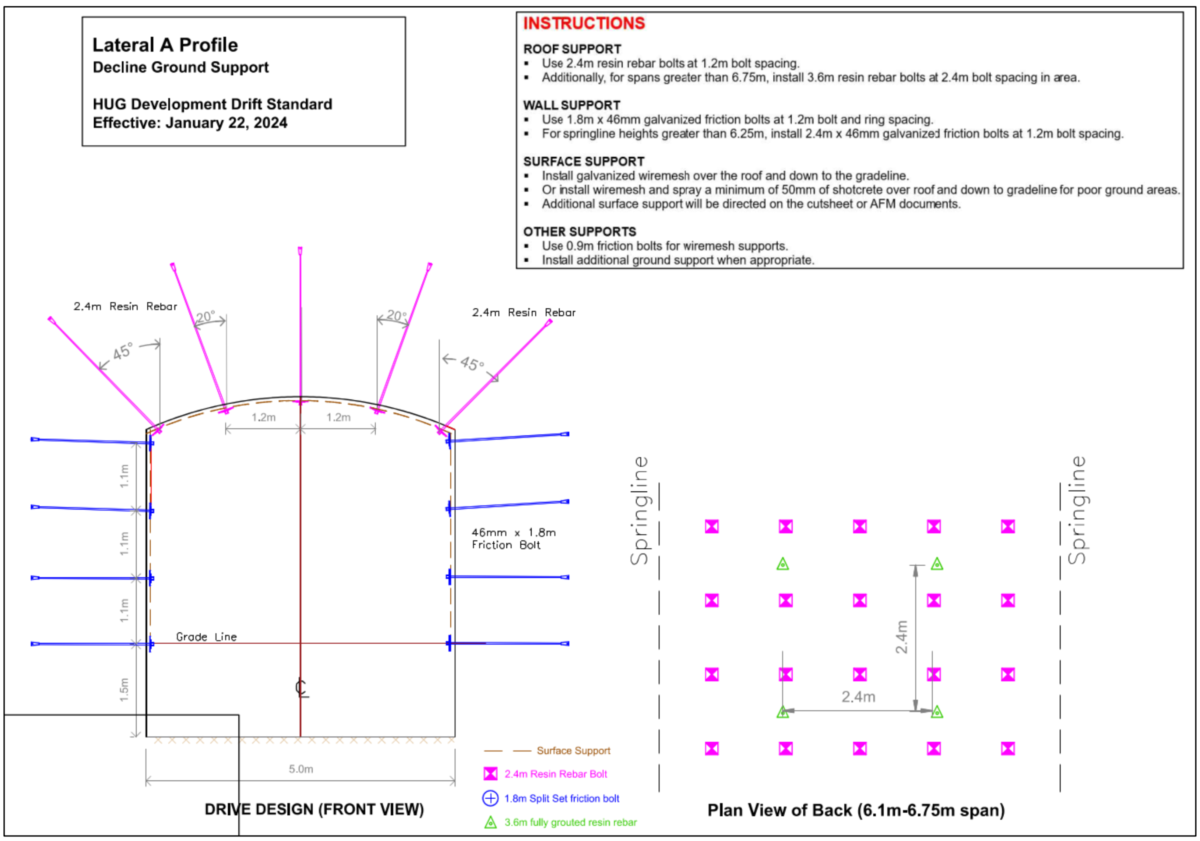
All bolting strategies achieve a safety factor exceeding 2.0 based on kinematic analyses. Table 16-33 presents surface support requirements, which is based on the ground support chart by Grimstad et al. (2002). Fiber reinforced shotcrete (fibercrete) is applied to areas with rock quality less than Q equal to 1.0, or areas where additional stability is warranted (Type 2 and Type 3 support). All advancing faces are supported with short (minimum 0.9 m) friction bolts and mesh.

Figure 16-28 presents the rock quality distributions along the permanent development. A majority of all development has a rock quality, Q rating, greater than 4. Consequently, most development at Horseshoe will not require shotcrete or fibercrete.

Table 16-33: Summary of Surface Support Categories for Horseshoe

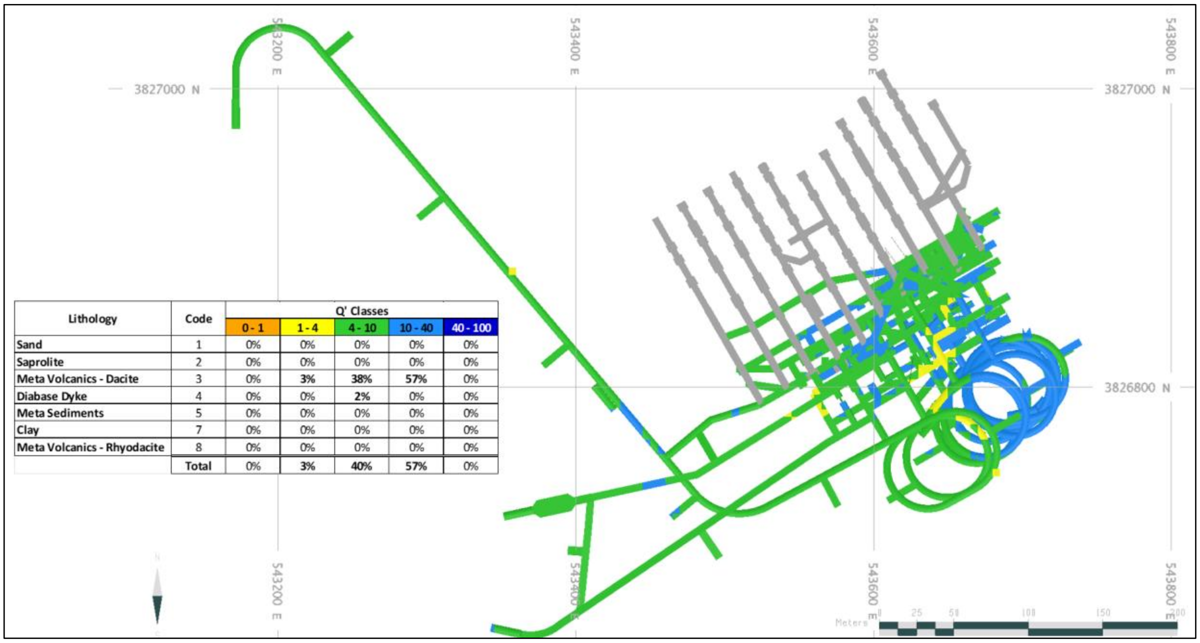
Support Category	Q value	Surface Support	Remarks
Type 1	> 1.0	Welded wire mesh	Majority of all ground conditions in the decline and access drives and ore drives (beyond 20 m), Decline passing bay
Type 2	> 1.0	50 mm fibercrete	Ore drives, escapeway drive, footwall drive (first 20 m), loading bay drives, level sump drives, ventilation drives, CRF stockpile drives, Jumbo starter drives, etc.
Type 3	0.1 - 1.0	75 - 100 mm fibercrete	First 20 m of all HUG portals, substation drives, Fan Chamber, Pump station drives

Source: OceanaGold, 2024



Source: OceanaGold, 2024

Figure 16-27: Decline Ground Support Layout



Lithology	Code	Q' Classes				
		0 - 1	1 - 4	4 - 10	10 - 40	40 - 100
Sand	1	0%	0%	0%	0%	0%
Saprolite	2	0%	0%	0%	0%	0%
Meta Volcanics - Dacite	3	0%	3%	38%	57%	0%
Diabase Dyke	4	0%	0%	2%	0%	0%
Meta Sediments	5	0%	0%	0%	0%	0%
Clay	7	0%	0%	0%	0%	0%
Meta Volcanics - Rhyodacite	8	0%	0%	0%	0%	0%
	Total	0%	3%	40%	57%	0%

Source: CNI, 2024

Figure 16-28: Rock Quality On Permanent Development - Horseshoe

Horseshoe Mine Temporary Support (Production Headings and Ore Drives)

The backs and sidewalls of ore drives, and slot drives will be supported with 1.8 m long x 46 mm (SS-46) diameter black friction bolts. These drives are not expected to have a service life greater than one year, and as a result do not require corrosion protection. Surface support includes fibercrete when rock quality with Q less than 1.0 is encountered and is prescribed by the geotechnical engineer.

Cable bolts are installed 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). The ground support design includes bulbed single strand cables (8.3 m length) installed in stope crowns at a minimum density of 0.33 cables per meters squared. Plating is conducted as needed. Cable bolt requirements are based on empirical charts developed by Hutchinson and Diederichs (1996).

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).

Palomino Mine Permanent Support (Decline and Footwall Drives)

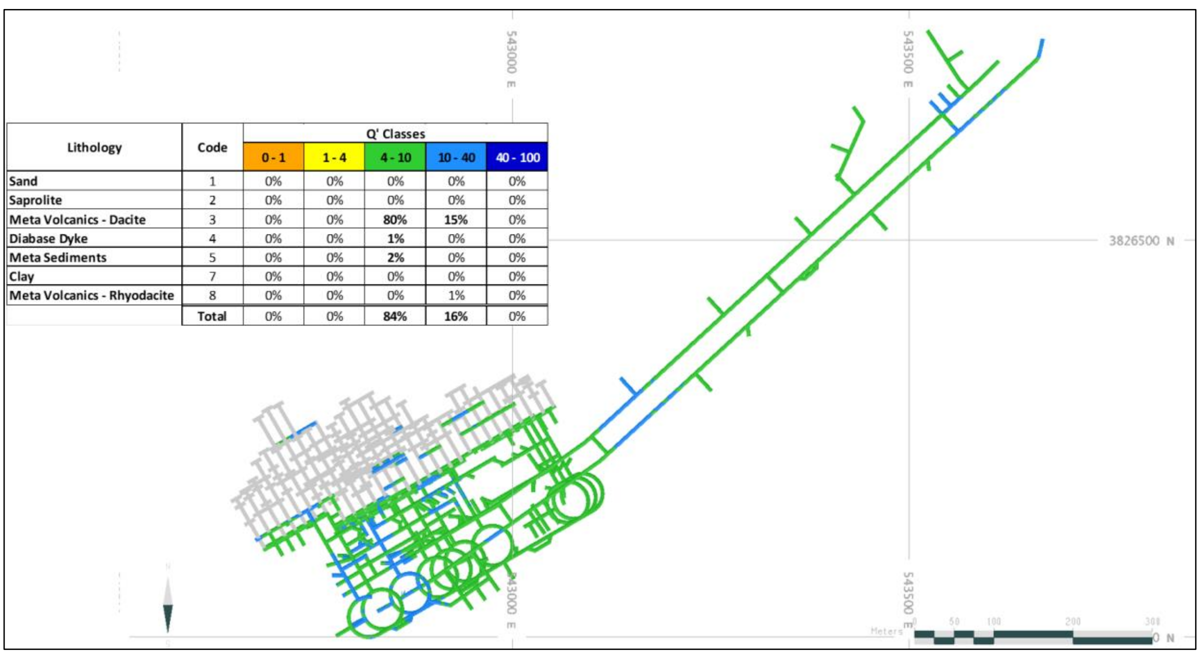
Planned decline support at Palomino includes fully encapsulated rebar bolts only. However, if the performance of galvanized Split Set bolts at Horseshoe demonstrates good longevity in their service life, this support type may also be applied to Palomino. The proposed ground support for the Palomino PFS is presented in Table 16-34. Support requirements are based on both kinematic and empirical methods (Grimstad et al. support chart). All bolting layouts achieve a safety factor exceeding 2.0 based on kinematic analyses. All advancing faces are supported with short (minimum 0.9 m) friction bolts and mesh.

Figure 16-29 presents the rock quality distributions along the permanent development. Nearly all development has a rock quality, Q rating, greater than 4. Consequently, most development at Palomino will not require shotcrete or fibercrete.

Table 16-34: Summary of Permanent Support Categories for Palomino PFS

Support Category	Q value	Estimated RMR76\GSI	Estimated % of Development	Advance Length (m)	Support Type
Category 1	> 2.0	> 50	91%	4.0	2.4 m #7 rebar on 1.2 m x 1.2 m spacing with welded mesh (10 cm / 6 Ga.) to within 1.5 m of sill
Category 2	0.7 - 2.0	41 - 50	3%	3.5	2.4 m #7 rebar on 1.2 m x 1.2 m spacing with welded mesh (10 cm / 6 Ga.) and 5 cm of shotcrete to within 1.5 m of sill
Category 3	0.07 - 0.7	20 - 40	3%	2.5	10 cm of fiber reinforced shotcrete (FRS) and 2.4 m #7 rebar on 1.2 m x 1.2 m spacing with welded mesh (10 cm / 6 Ga.) down to sill
Category 4	< 0.07	< 20	3%	1.2	15 cm of FRS and 2.4 m #7 rebar on 1.2 m x 1.2 m spacing with welded mesh (10 cm / 6 Ga.) down to sill with 6 count #7 rebar arch spaced each 2.4 m and fully encased in shotcrete; forepoling (spiling)

Source: CNI, 2024



Source: CNI, 2024

Figure 16-29: Rock Quality On Permanent Development - Palomino

Palomino Mine Temporary Support (Production Headings and Ore Drives)

Temporary ground support for stope crowns and production ore drives at Palomino is summarized in Table 16-35. These drives are not expected to have a service life greater than one year, and as a result do not require corrosion protection and friction bolts are an acceptable alternative to rebar bolts. Surface support includes fibercrete when rock quality with Q less than 1.0 is encountered (Category 2).

Cable bolts are installed 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). The ground support design includes bulbed single strand cables (8.3 m length) installed in stope crowns at a minimum density of 0.25 cables per meters squared. Plating will be conducted as needed. Cable bolt requirements are based on empirical charts developed by Hutchinson and Diederichs (1996).

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).

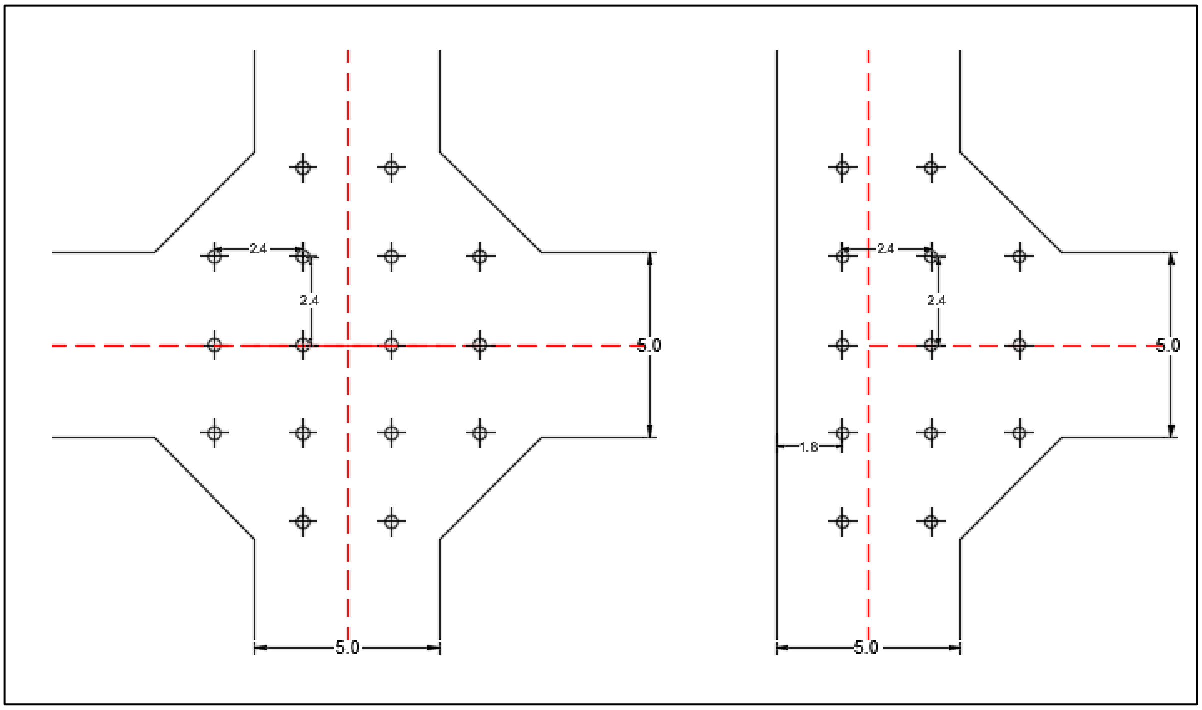
Table 16-35: Summary of Production Support Categories for Palomino PFS

Primary / Secondary	Support Category	Q Value	Estimated RMR76\GSI	% of MSO Shapes	Advance Length (m)	Support Type
Primary Support	Category 1	> 1.0	> 44	95	4.0	2.4 m #7 Rebar [1] on 1.2 m x 1.2 m spacing with welded mesh (10cm / 6Ga.) to within 1.5 m of sill
	Category 2	0.7 - 1.0	< 44	5	2.5	2.4 m #7 Rebar [1] on 1.2 m x 1.2 m spacing with welded mesh (10cm / 6Ga.) and 5cm of Shotcrete to within 1.5 m of sill
Secondary Support	All Stope Top Cuts					8.3 m Cable Bolts (Single Strand) on 2.0 m x 2.0 m spacing in the backs (minimum 3 each per row); installed prior to stoping

Source: CNI, 2024
[1] 12 Ton Capacity Friction Bolts are Acceptable Alternative to Rebar in Headings with less than 1 year service life

Intersection Support (Horseshoe and Palomino)

Intersections will include cable bolt support (6.3 m Single Strand Cables) in addition to the primary support at both Palomino and Horseshoe. Figure 16-30 presents the cable bolt layout for intersections. Cable bolt specifications are based on empirical charts developed by Hutchinson and Diederichs (1996).



Source: CNI, 2024

Figure 16-30: Cable Bolt Layout for Intersections

Mine Access and Shaft Design

Horseshoe Mine Portal Stability

CNI conducted stability analyses to specify ground support requirements for batter faces at the Horseshoe Mine portal locations. All three access portals (production decline, intake decline, exhaust decline) were reviewed for kinematic stability and discrete support systems for the batter faces were developed. The stability of these portal faces is being monitored for stability using multiple point borehole extensometers (MPBXs), with no measurable deformation currently recorded.

Palomino Ventilation Shaft

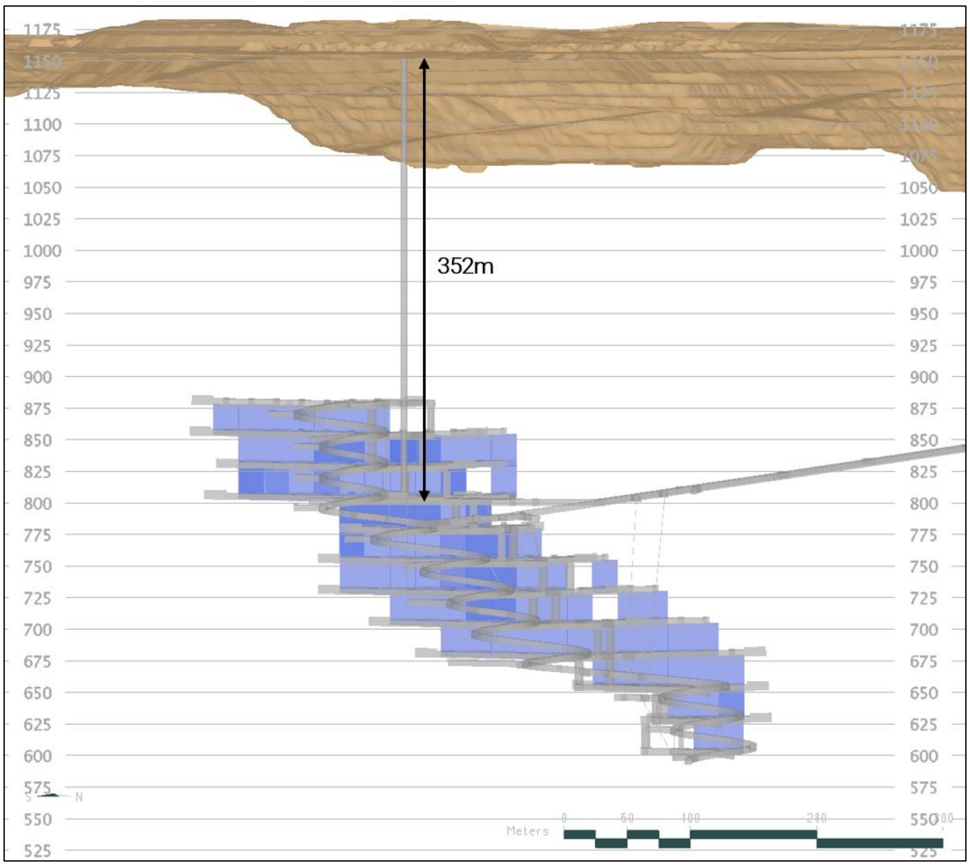
Ventilation at Palomino will be provided via a 352-m shaft located south of the deposit (Figure 16-31). Preliminary shaft support and geotechnical recommendations are based on logging and testing from a pilot hole (DDH-1196) drilled in support of the PFS study. The upper 21 m from existing topography consist of coastal sands and saprolite which are not amenable to shaft collar placement. Consequently, a collar pre-sink of the uppermost 21 m is recommended so that the shaft collar is situated on rock. More competent rock (metavolcanics) can be reached if the pre-sink extends to 70 m depth from surface.

The thickness of the shaft liner will be 0.5 m. The thicker shaft liner is due to the high horizontal stresses present at HGM. To account for the shaft liner and still maintain the specified 5 m internal diameter required for ventilation, the excavated, outer diameter of the shaft will be 6.0 m. Liner thickness proximal to the shaft station (nominal 22.5 m length) will increase to 0.9 m as presented in Figure 16-32.

Support of the shaft will be in two stages. Initial support will be upon excavation and is considered temporary; final support is installed once the shaft has been sunk 1 or 2 times the shaft diameters (6.0 to 12 m). CNI suggests a safe unsupported height of 9 m; the maximum lag distance between the temporary support and final support should not exceed 12 m. The support requirements are as follows:

- Temporary support consists of 1.8 m friction bolts (Split Set or inflatable friction type) and welded-wire mesh.
- Final support consists of steel fiber reinforced concrete with a minimum compressive strength of 30 MPa.

Table 16-36 presents a summary of shaft recommendations.



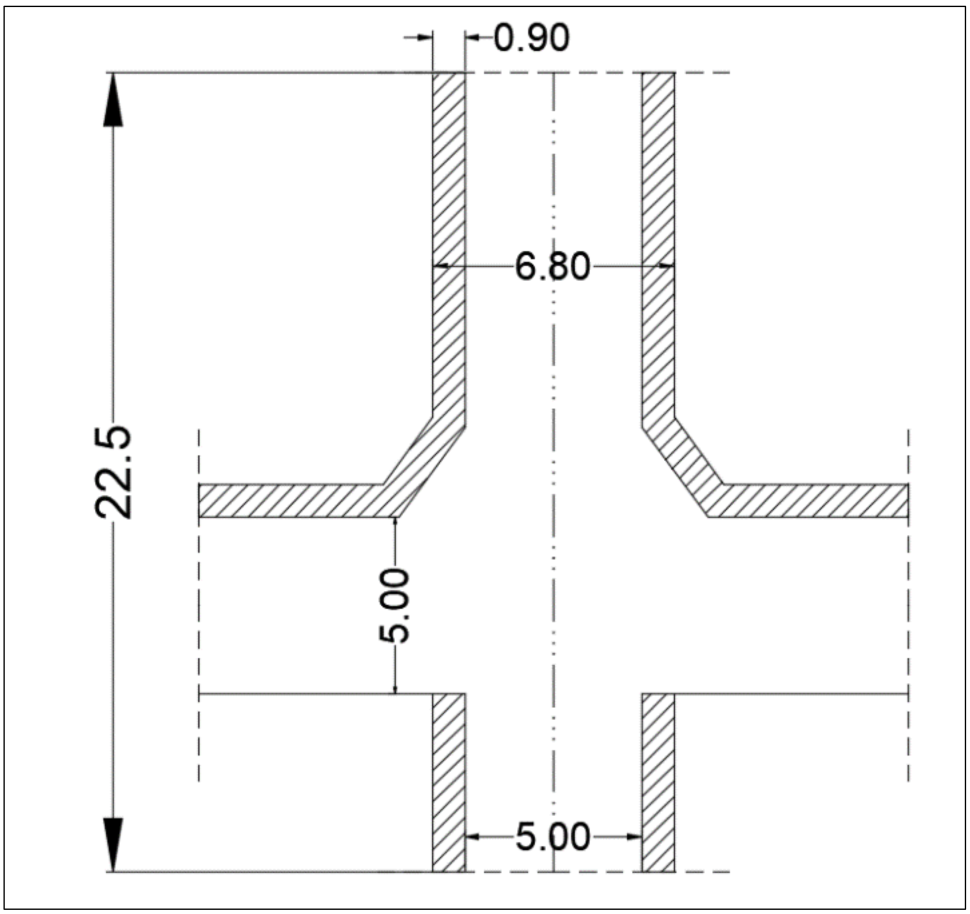
Source: CNI, 2024

Figure 16-31: Palomino Ventilation Shaft

Table 16-36: Summary of Preliminary Shaft Recommendations

Collar Pre-Sink Minimum Depth (m)	21.0
Excavated Outer Diameter (m)	6.0
Liner Thickness (m)	0.5
Inner Diameter (m)	5.0
Temporary Support	1.8 m friction bolts; welded wire mesh
Final Support	Steel reinforced concrete liner (0.73 m) of minimum 30 MPa concrete
Recommended Lag Distance between Temporary and Final Support (m)	9
Liner Thickness Proximal (22.5 m) to Shaft Station (m)	0.9

Source: CNI, 2024



Source: CNI, 2024

Figure 16-32: Shaft Station Liner Zone

16.2.3 Hydrogeology and Mine Dewatering

The hydrogeology of the Site is summarized in Section 16.1.9. The Horseshoe underground mine workings will extend to a depth of approximately 1,470 ft (448 m) below ground surface and will be accessed by a decline from Snake Pit, which is currently under construction. The planned Palomino underground mine workings will extend to a depth of approximately 1,560 ft (475 m) below ground surface and will be accessed drift development from the Horseshoe main decline. The planned underground mine developments 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 combined Horseshoe and Palomino underground workings have been estimated using the groundwater numerical model (Section 16.1.9) and range from approximately 390 to 650 gpm (25 to 41 L/sec), with an average rate of 510 gpm (32 L/sec). Estimated dewatering rates from the Horseshoe underground workings range from approximately 240 to 410 gpm (15 to 26 L/sec), with an average rate of 300 gpm (19 L/sec). Estimated dewatering rates from the Palomino underground workings range from approximately 150 to 240 gpm (9 to 15 L/sec), with an average rate of 210 gpm (13 L/sec). The timing and volume of extracted groundwater from the underground workings is expected to be manageable.

16.2.4 Geochemical

The current underground mine design includes 1,972 kt of development rock and 7,859 kt of ore (Table 16-40). Development rock is approximately 20% of the total material to be mined from the underground development. All development rock from underground mining is handled as Red PAG, due to the small quantity and the logistics of testing required to classify and route it according to the geochemical classification. During mining, the development rock from the underground mine is hauled to surface and stockpiled on a clay liner before open pit mining operations moves it to an OSA with a geomembrane liner, either the East PAG OSA or West PAG OSA.

Most of the development rock generated from underground mining will be for mine access and ventilation development. Access and ventilation workings in the current mine design will intersect the metavolcanics unit, which was extensively characterized by geochemical testing and is not believed to present a significantly greater risk for ARDML than the same material in other areas of the Project. However, characterization is recommended to obtain confirmatory data on the geochemical properties of the new mining areas, to apply to geochemical modeling for water quality predictions.

CRF is assumed as the base case backfill material for the current underground mine design. Depending on the mining sequence, uncemented rock fill could be used instead of, or in addition to, CRF. CRF is produced at the batch plant on site with aggregate comprised of bedrock lithologies classified as Green (Non-PAG) overburden from open pit mining. SRK completed geochemical characterization of CRF samples (SRK, 2017), but upon further review, found that the overburden materials collected for CRF testing were more representative of Red PAG rock than Green Non-PAG rock. Because the aggregate sourced for CRF is limited to Green Non-PAG rock, the characterization results are not representative of the CRF, and are not presented in the geochemical characterization report (Schafer, 2019). Cemented paste backfill is being evaluated as an alternative to CRF. Geochemical characterization results of paste backfill samples have not been published to date.

16.2.5 Stope Optimization

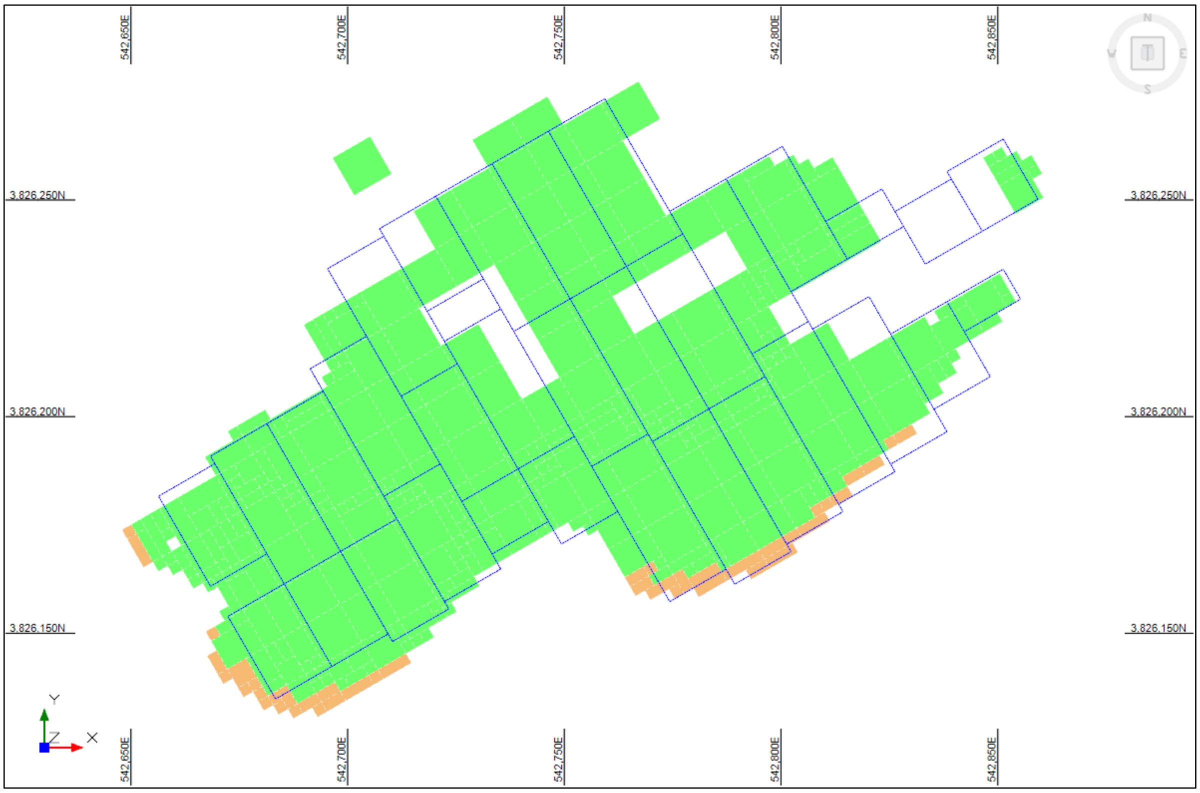
Stope optimization was completed on prior versions of the model. Results from those prior runs were used for comparison during the mine design process.

16.2.6 Mine Design

Stope Design

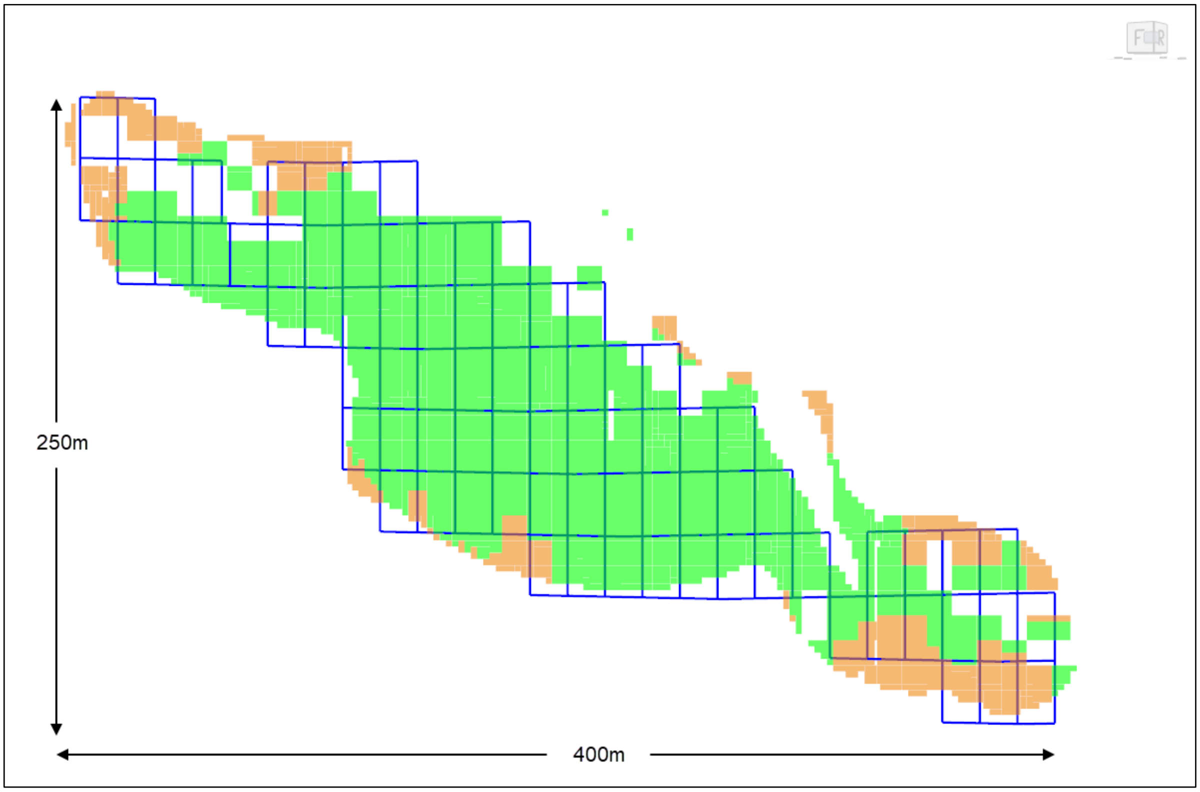
For stope designs, a combination of manual and automated processes has been used to create stope shapes. Stope shapes created manually involved vertical slices through the orebody along 2 m strike length intervals. The block model was then interrogated, filtered on cut-off, and the slices were combined to create minable stopes. Stope Optimizer (SO), which is part of the Deswik suite, was also used to create stope shapes and is a more automated process.

Plan and section views of Palomino stopes (blue) overlayed against the block model (green = indicated, orange = inferred) are illustrated in Figure 16-33 and Figure 16-34.



Source: OceanaGold, 2023

Figure 16-33: Palomino Plan View Clipped at 800 m RL

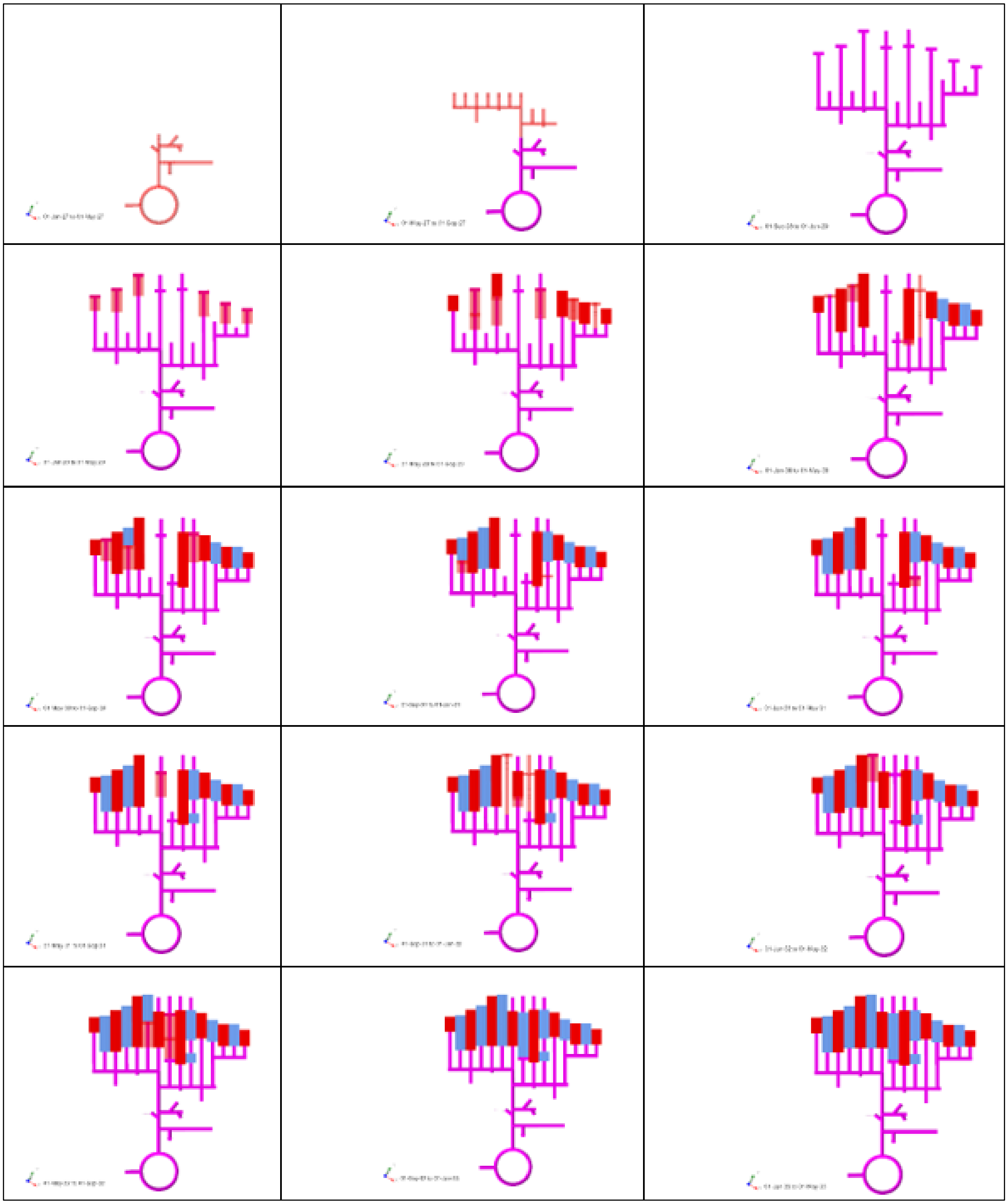


Source: OceanaGold, 2023

Figure 16-34: Palomino Stopes Section Looking Northwest

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.

Stopes are 15 to 20 m wide while length will vary based on the extent of mineralization and geotechnical considerations and recommendations. A sublevel spacing of 25 m (floor to floor) has been used. 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. A development and production sequence for a typical underground level at Horseshoe and Palomino is illustrated in Figure 16-35. Capital development is complete before primary ore drives and stopes (red) are extracted. Secondary ore drives and stopes (blue) are not mined until the adjacent primary stopes are mined and backfilled.



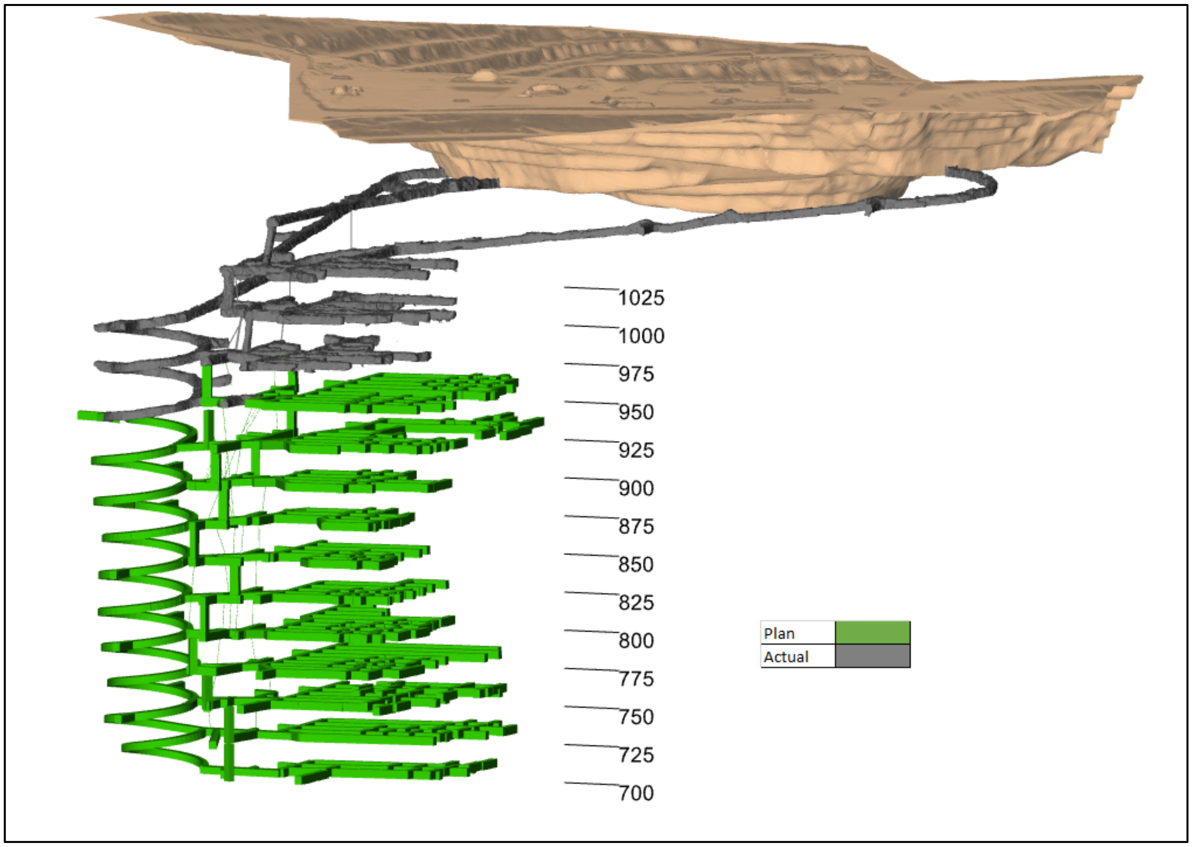
Source: OceanaGold, 2023

Figure 16-35: Typical Underground Level Sequence Over Time

Slot drives are incorporated into designs to provide sufficient void and contingency for initial stope firings. Slot drive development cannot commence until the adjacent stope has had been backfilled and sufficient curing time.

Development Design

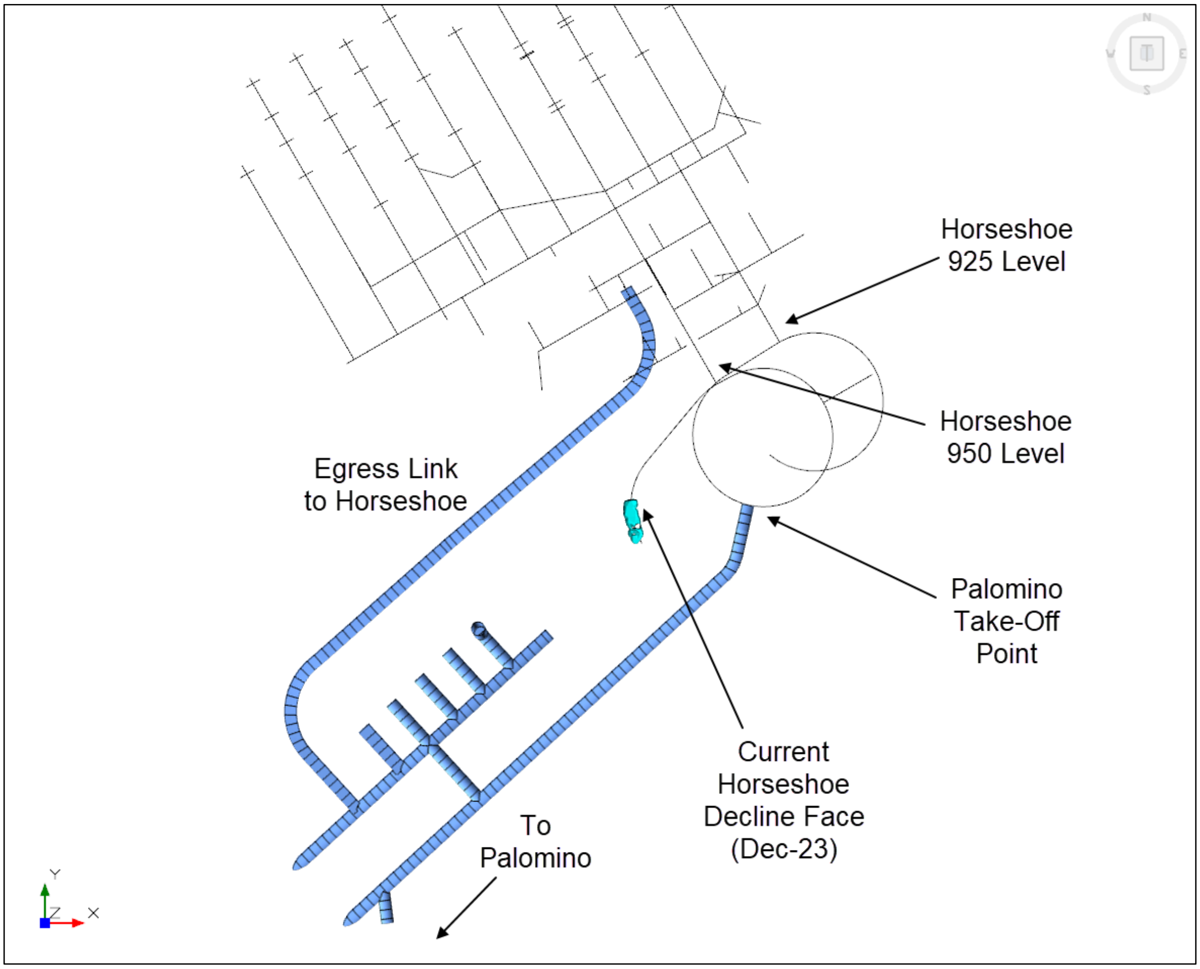
The Horseshoe underground mine is accessed via a decline from the surface. The decline portal is located on an open pit bench approximately 80 m below the natural surface. Two ventilation drift portals are also located on an open pit bench. Current Horseshoe development is illustrated in Figure 16-36, with the decline face at just past the 950 as at February 2024.



Source: OceanaGold, 2024

Figure 16-36: Horseshoe Development Asbuilt and Plan

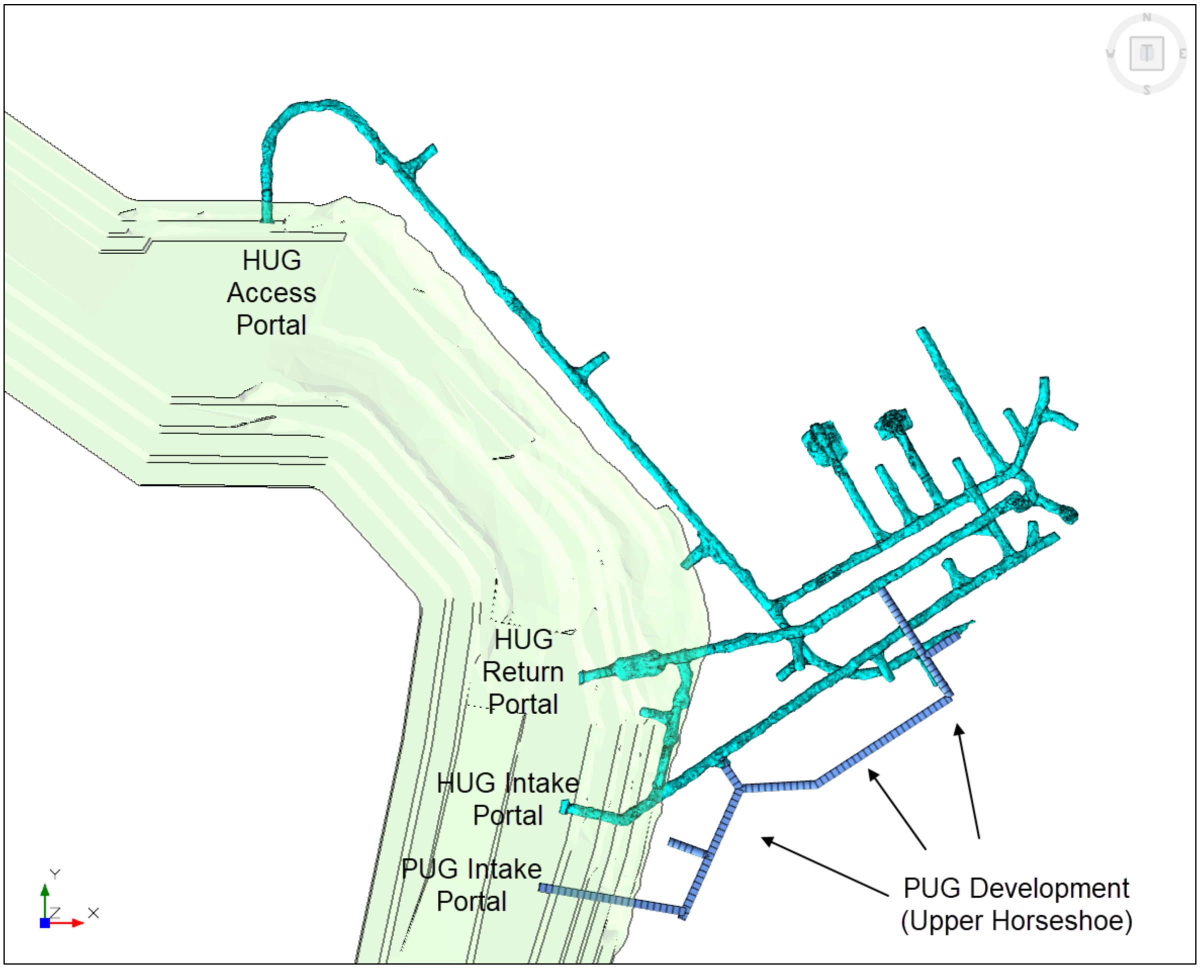
Planned Palomino access is via the main Horseshoe decline. The take-off point for the Palomino decline is between the Horseshoe 950 and 925 Levels and is illustrated in Figure 16-37. This location has been chosen based on an optimal landing point RL at Palomino (based on a 1:7 decline), and the current Horseshoe schedule which shows capital development advanced far enough at depth to allow for a 2026 start date for the Palomino decline that will minimize interaction with planned Horseshoe activities.



Source: Source: OceanaGold, 2024

Figure 16-37: Palomino Decline Take-Off Point

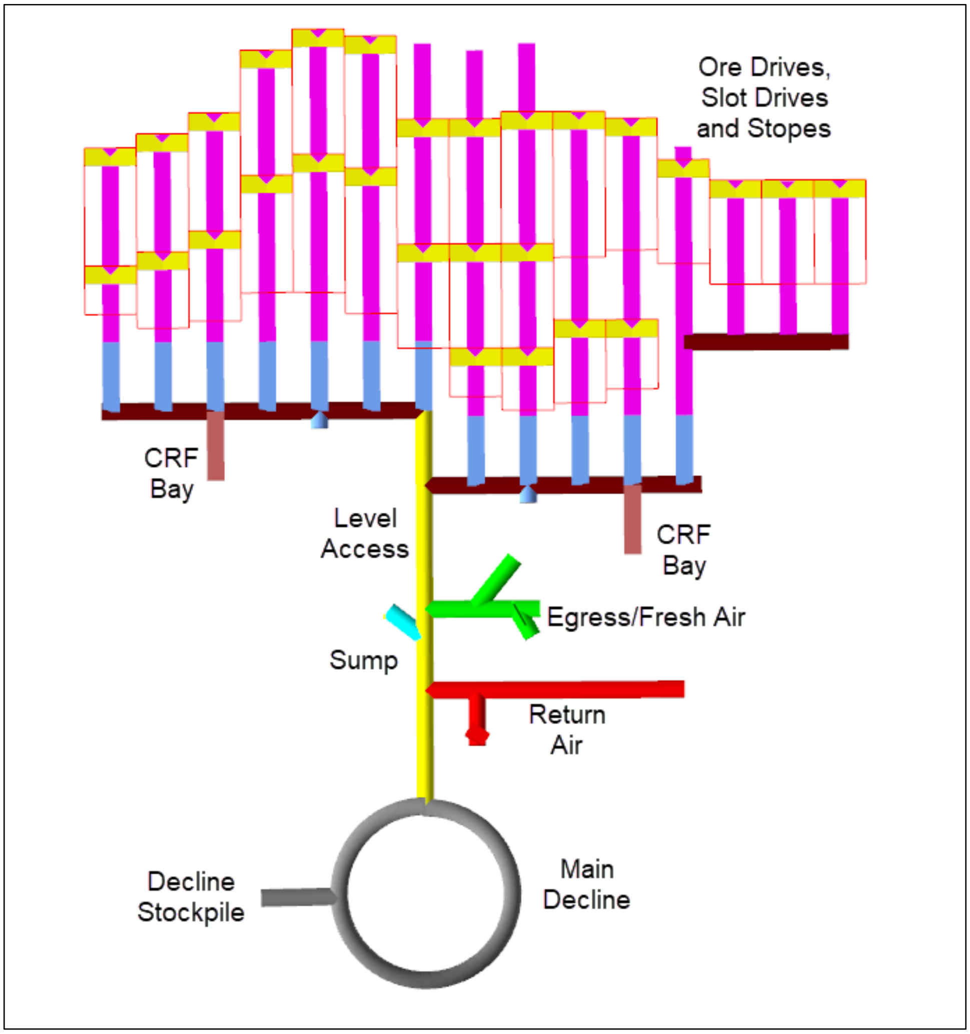
Additional development is required in the upper Horseshoe precinct for Palomino and includes development connections for initial and steady state ventilation purposes and is shown in Figure 16-38.



Source: OceanaGold, 2023

Figure 16-38: Upper Horseshoe Vent Connections required for Palomino Ventilation

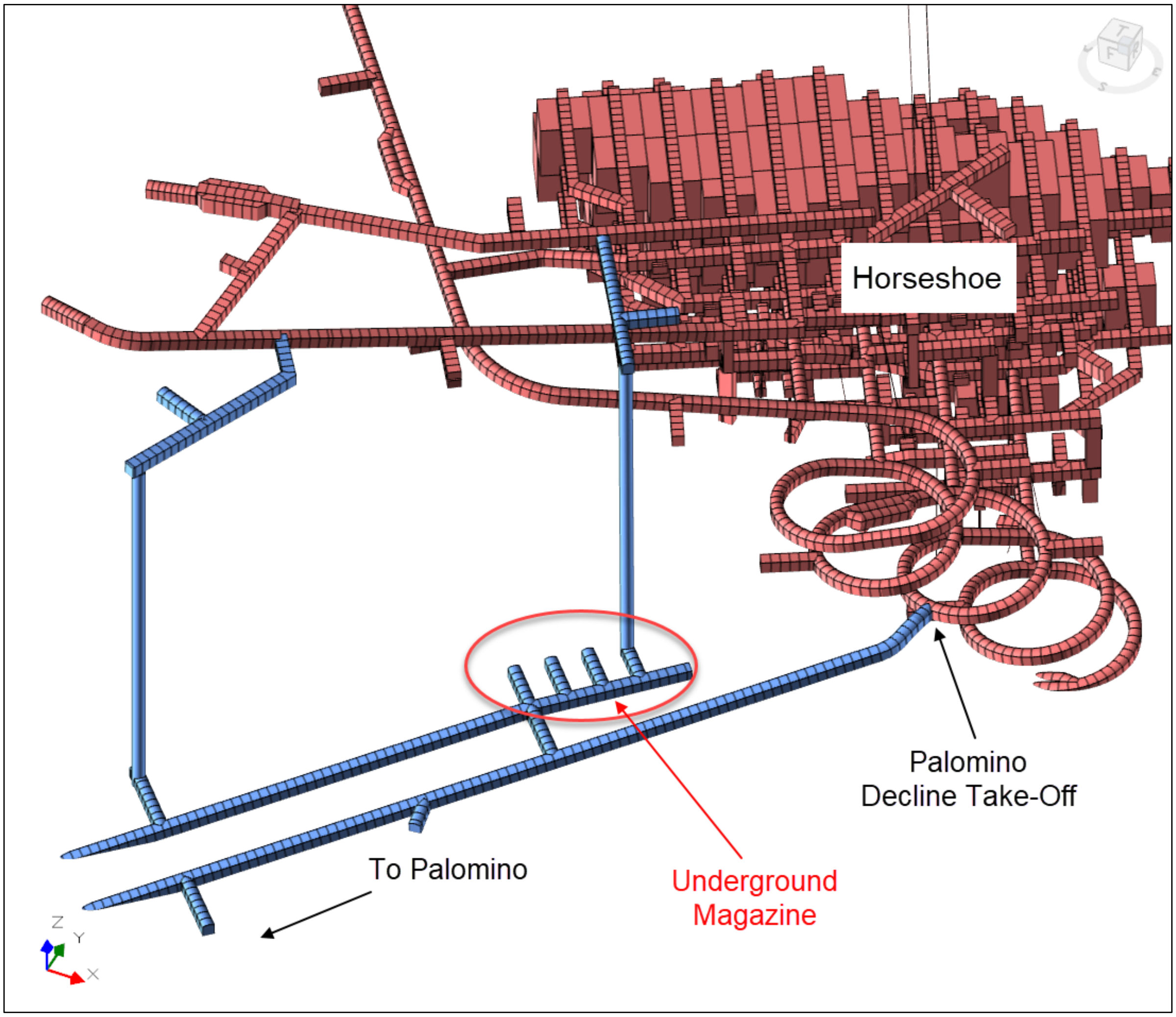
Horseshoe and Palomino stope accesses are connected to a level access located in the footwall in waste material. The level accesses are offset a minimum of 25 m from the stopes. The level accesses connect to the interlevel ramp system which is located in the footwall and is offset approximately 75 m from the footwall accesses. Each level is connected to the ventilation system and emergency egress system. Figure 16-39 shows a typical level section.



Source: OceanaGold, 2023

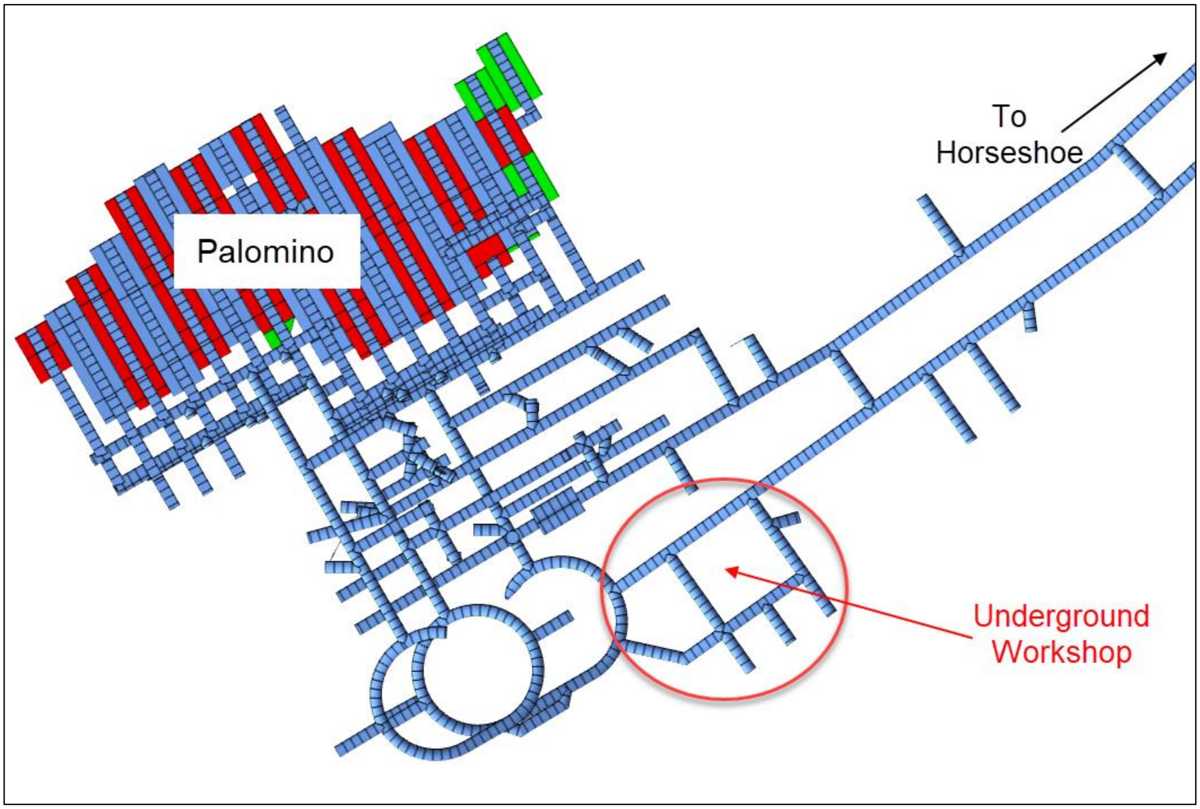
Figure 16-39: Typical Level Section

Currently for Horseshoe, explosives are stored in a surface magazine. An underground magazine is planned to service both Horseshoe and Palomino. Preferred location is illustrated in Figure 16-40 and is in close proximity to return air and situated strategically between both orebodies. Currently all maintenance is carried out on the surface. With the addition of Palomino into underground mining schedules, an underground maintenance facility has been included in designs and costs to account for increased fleet size and increased distances required to tram equipment to the surface for servicing. The preferred location of the underground maintenance facility is towards the base of the main Palomino access and is illustrated in Figure 16-41. The workshop will be utilized by all underground equipment, however major services and/or breakdowns could still be conducted from the surface. The CRF facilities currently located on the surface are operational and no additional infrastructure is required underground.



Source: OceanaGold, 2023

Figure 16-40: Underground Magazine Location



Source: OceanaGold, 2023

Figure 16-41: Underground Workshop Location Plan View

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.

All development rock from underground mine development is handled as Red PAG and is discussed in 16.2.4.

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 16-37.

Table 16-37: Productivity Rates

Activity Type	Rate (1)
Main Decline and Decline Stockpiles	5.0 m/day
Level Access, Ventilation Access, Sumps, Escapeway Access	5.0 m/day
Footwall Drive, Ore Drives	5.0 m/day
Stope Slot Raiseboring and Vent Raise Slot Raiseboring	10 m/day
Stope Drilling and Vent Raise Drilling	250 m/day
Stope Bogging	1,500 t/day
Stope Backfilling (CRF)	1,000 m^3/day
Ventilation Raises (Firing and Bogging)	25 m/day
Escapeway Raises	10 m/day
Drain Holes and Service Holes	25 m/day
Surface Shaft – Blind Sink	0.6 m/day
Surface Shaft – Slash and Line	1.7 m/day
Internal Raisebore Shafts (3 m and 5 m diameter)	6 m/day

Source: OceanaGold, 2024
(1) All rates are per face. Multiple areas/faces are mined together to generate the mining schedule.

General Parameters

General schedule parameters applicable to all underground mining activities are presented in Table 16-38.

Table 16-38: 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, 2023

Key assumptions regarding ore and waste material characteristics are detailed in Table 16-39.

Table 16-39: 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, 2023

16.2.8 Mine Production Schedule

The Horseshoe and Palomino underground mine production schedules are based on the productivity assumptions shown in Table 16-37. 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 from each orebody 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.

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. Horseshoe and Palomino production and development summaries are illustrated in Table 16-40.

Table 16-40: Underground (Horseshoe and Palomino) Mining Summary

General Summary	Horseshoe	Palomino	Total UG
Ore Tonnes (kt)	3,828	4,031	7,859
Ore Au (g/t)	4.24	2.91	3.56
Waste Tonnes (kt)	833	1,139	1,972
Total Tonnes Moved (kt)	**4,662**	**5,170**	**9,831**
Ore Summary			
Development Ore Tonnes (kt)	309	500	808
Stope Production Tonnes (kt)	3,519	3,531	7,051
Horizontal Development Summary			
ACC - Level Access	679	875	1,554
CRF - CRF Stockpile	242	306	549
DEC – Decline	1,550	3,514	5,064
DSM - Decline Sump		73	73
DSP - Decline Stockpile	138	368	506
EWD - Escapeway Drive	205	691	896
FAD - Fresh Air Drive	195	675	870
FAR - Fresh Air Rise	1,087	1,598	2,685
LSM - Level Sump	48	194	241
MAG - Underground Magazine		90	90
OED - Ore Exploration Drive	1,231	1,645	2,876
OPD - Ore Production Drive	5,018	6,690	11,709
PBY - Passing Bay	99	86	185
PSN - Pump Station Drive	94	186	280
RAD - Return Air Drive	552	937	1,490
SLD - Slot Drive	2,333	1,463	3,796
SUB - Substation	60	198	258
WKS - Underground Workshop		229	229
XCT – Crosscut	78		78
Total Horizontal Development Length (m)	**13,610**	**19,818**	**33,427**
Vertical Development Summary			
DH_ Drainhole	149	215	364
EWR - Escapeway Rise	199	167	366
FAR - Fresh Air Rise	65	204	269
RAR - Return Air Rise	205	666	871
RMH_ Rising Main	154	134	288
SH_ Service Hole	289	79	368

Source: OceanaGold, 2023

Annual mining schedules for Horseshoe, Palomino, and combined underground at Haile (Horseshoe and Palomino) are illustrated in Table 16-41, Table 16-42, and Table 16-43.

Table 16-41: Horseshoe Annual Mining Schedule

Year	Ore Tonnes (kt)	Au (g/t)	Ounces (koz)	Waste Tonnes (kt)	Backfill Volume (m^3)
2024	561.7	3.99	72.1	292.9	197,853
2025	624.5	4.05	81.2	276.3	221,018
2026	778.4	3.40	85.1	187.9	286,414
2027	761.2	4.64	113.6	41.2	279,642
2028	639.7	3.70	76.1	15.5	278,594
2029	331.7	5.49	58.6	19.0	141,736
2030	131.0	8.42	35.5	0.6	53,705
2031	-	-	-	-	-
2032	-	-	-	-	-
2033	-	-	-	-	-
2034	-	-	-	-	-
Total	**3,828.2**	**4.24**	**522.2**	**833.4**	**1,458,961**

Source: OceanaGold, 2023

Table 16-42: Palomino Annual Mining Schedule

Year	Ore Tonnes (kt)	Au (g/t)	Ounces (koz)	Waste Tonnes (kt)	Backfill Volume (m^3)
2024	-	-	-	-	-
2025	-	-	-	-	-
2026	-	-	-	160.3	-
2027	0.2	1.61	0.0	262.0	-
2028	439.2	2.95	41.7	182.9	99,041
2029	636.8	2.69	55.0	333.6	241,306
2030	735.9	2.84	67.7	186.2	225,299
2031	746.7	2.91	69.9	1.3	310,851
2032	748.8	3.01	72.5	12.8	286,982
2033	642.6	2.95	61.0	-	307,671
2034	80.4	4.05	10.5	-	72,355
Total	4,030.7	2.91	377.7	1,139.0	1,543,506

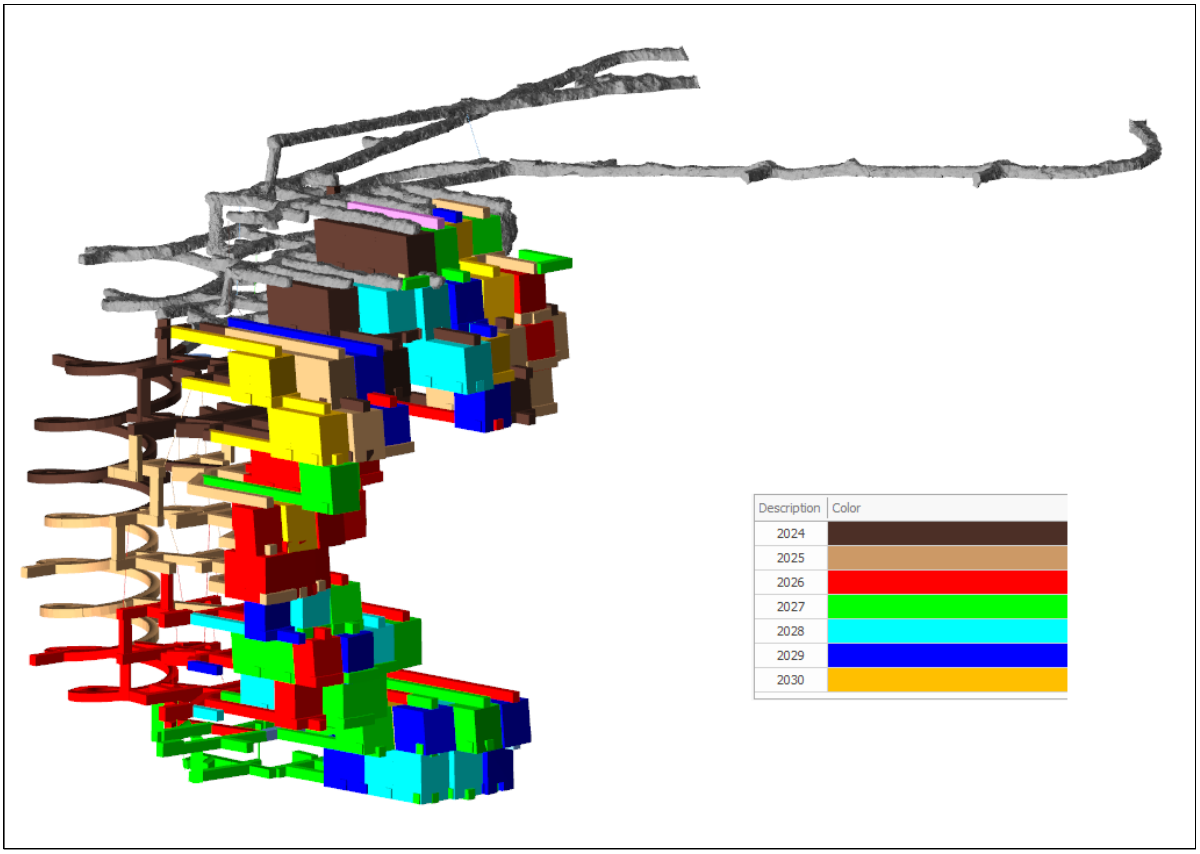
Source: OceanaGold, 2023

Table 16-43: Underground (Horseshoe and Palomino) Annual Mining Schedule

Year	Ore Tonnes (kt)	Au (g/t)	Ounces (koz)	Waste Tonnes (kt)	Backfill Volume (m^3)
2024	561.7	3.99	72.1	292.9	197,853
2025	624.5	4.05	81.2	276.3	221,018
2026	778.4	3.40	85.1	348.2	286,414
2027	761.4	4.64	113.6	303.2	279,642
2028	1,078.9	3.40	117.8	198.4	377,636
2029	968.5	3.65	113.6	352.6	383,041
2030	866.9	3.68	102.6	186.2	279,005
2031	746.7	2.91	69.9	1.3	310,851
2032	748.8	3.01	72.5	12.8	286,982
2033	642.6	2.95	61.0	-	307,671
2034	80.4	4.05	10.5	-	72,355
Total	7,858.9	3.56	899.9	1,972.3	3,002,467

Source: OceanaGold, 2023

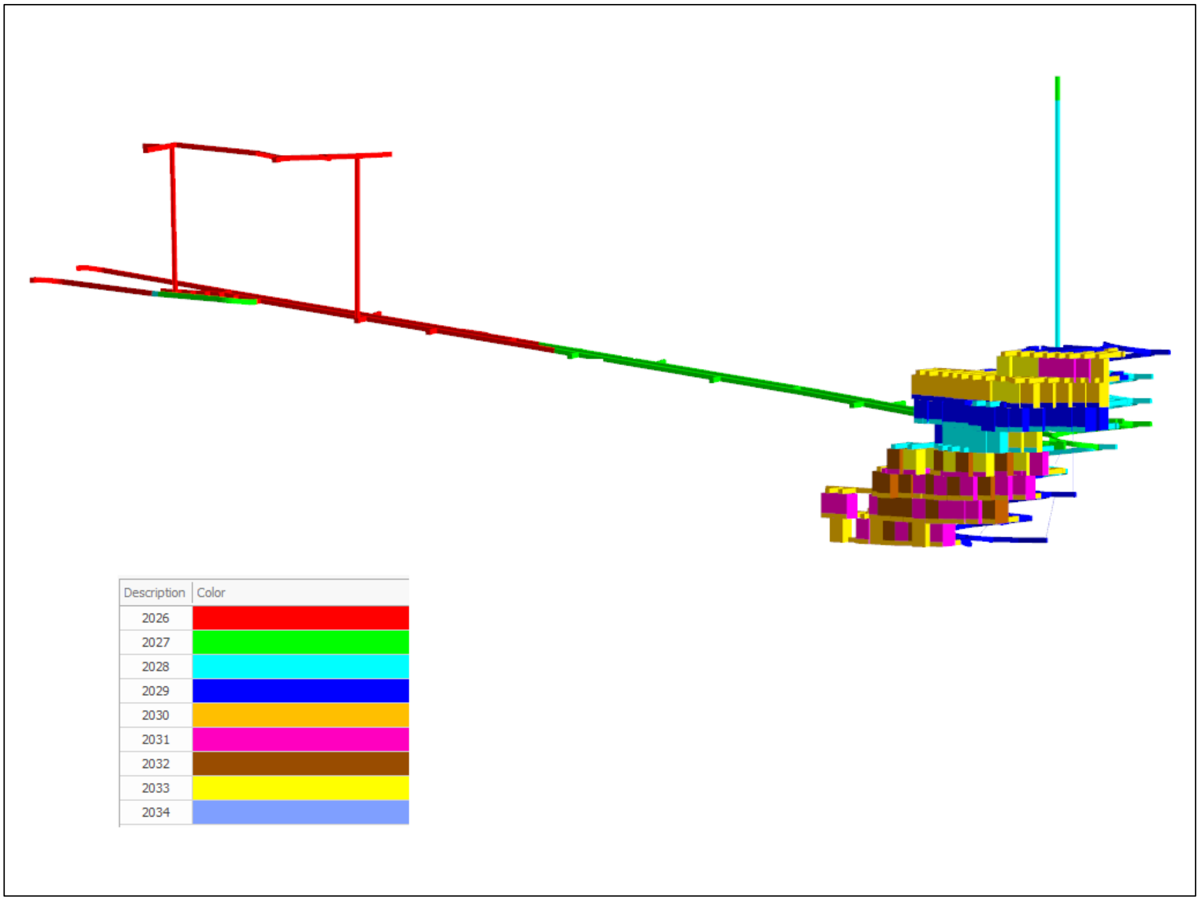
Horseshoe mining schedules colored by year is illustrated in Figure 16-42.



Source: OceanaGold, 2023

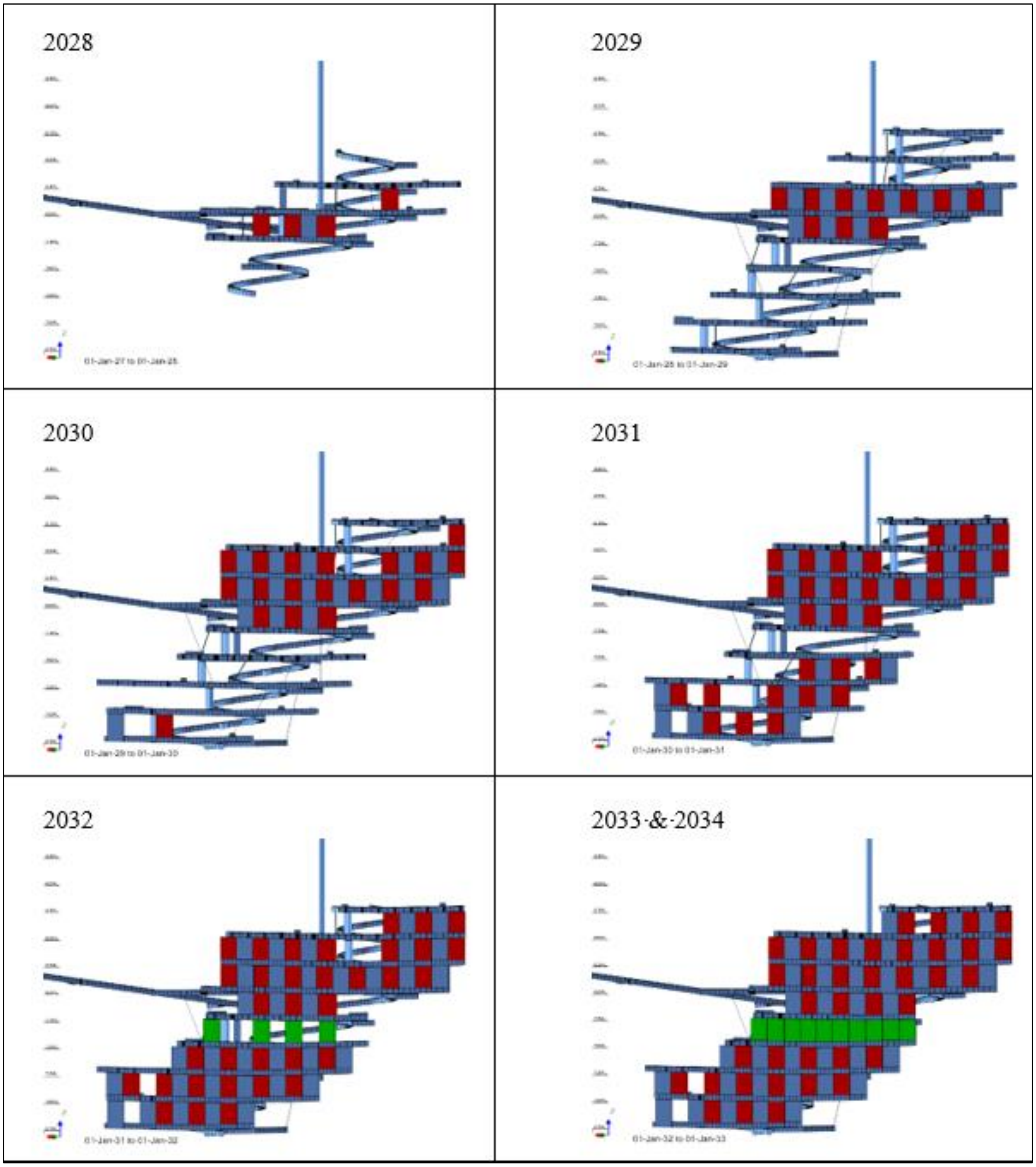
Figure 16-42: Horseshoe Mine Production Schedule Colored by Year

Palomino mining schedules colored by year is illustrated in Figure 16-43 whilst the stoping sequence by year is illustrated in Figure 16-44.



Source: OceanaGold, 2023

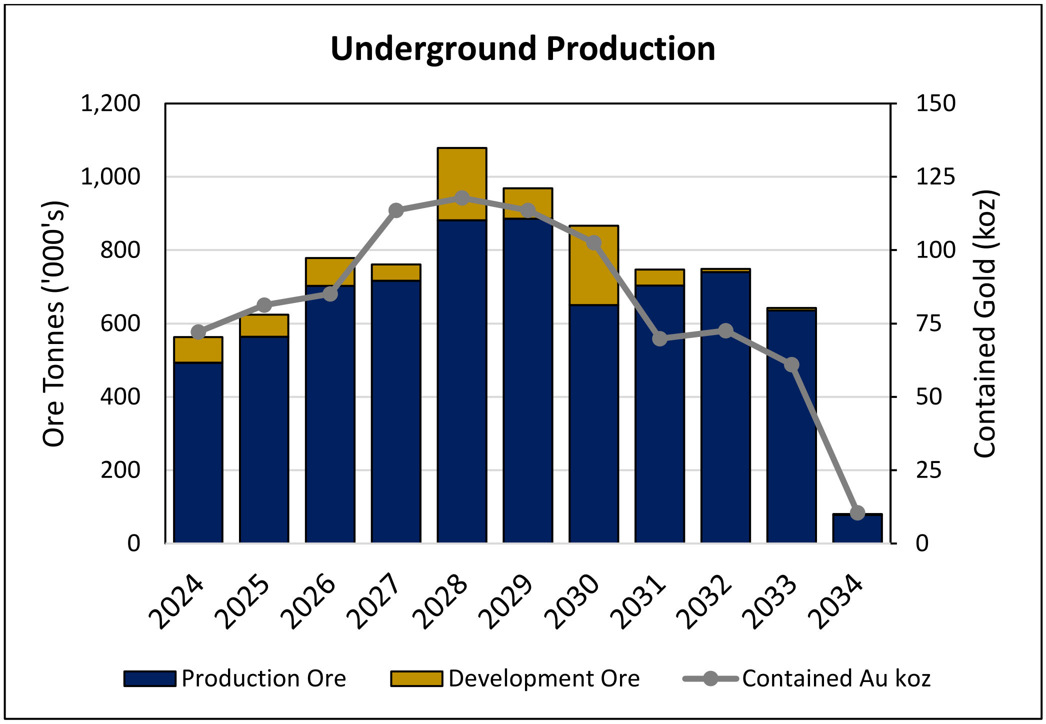
Figure 16-43: Palomino Mine Production Schedule Colored by Year



Source: OceanaGold, 2023

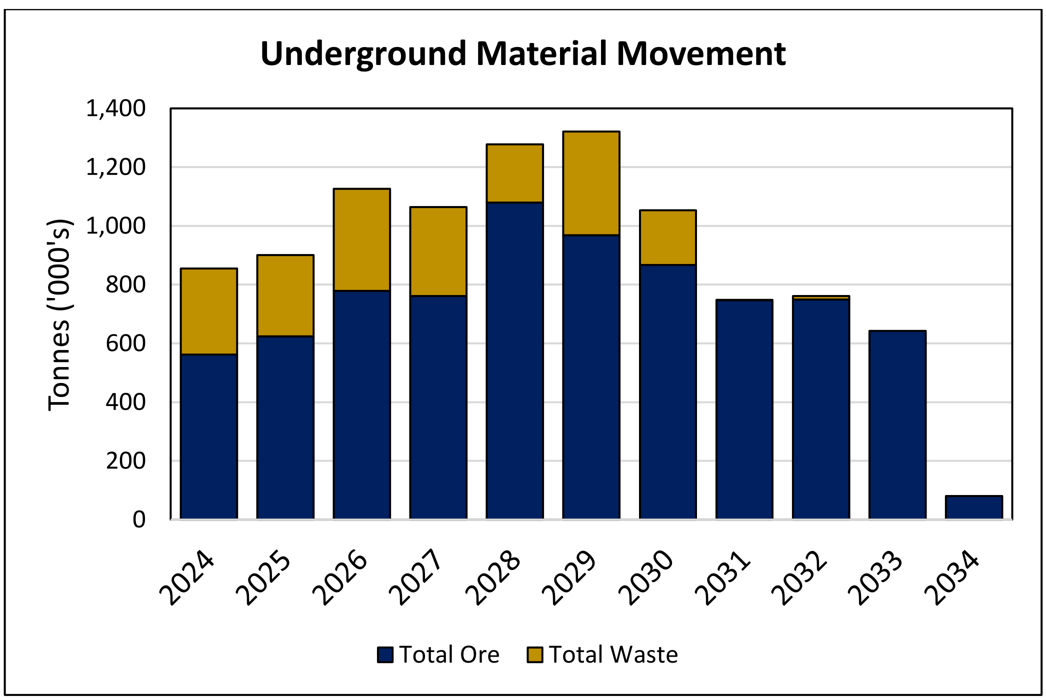
Figure 16-44: Palomino Stoping Sequence By Year

Annual underground ore production (Horseshoe and Palomino) is shown in Figure 16-45 and total underground material movement is shown in Figure 16-46.



Source: OceanaGold, 2023

Figure 16-45: Annual Underground Ore Production



Source: OceanaGold, 2023

Figure 16-46: Annual Underground Material Movement

Detailed annual underground production schedules are shown in Table 16-44.

Table 16-44: Detailed Annual Underground Schedule

	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
General Summary												
Total Ore (kt)	**7,859**	**562**	**624**	**778**	**761**	**1,079**	**969**	**867**	**747**	**749**	**643**	**80**
Total Waste (kt)	**1,972**	**293**	**276**	**348**	**303**	**198**	**353**	**187**	**1**	**13**	**0**	**0**
Total Tonnes (kt)	**9,831**	**855**	**901**	**1,127**	**1,065**	**1,277**	**1,321**	**1,054**	**748**	**762**	**643**	**80**
Total Lateral Development (m)	**33,427**	**4,301**	**4,053**	**4,914**	**4,170**	**5,086**	**5,201**	**4,879**	**584**	**109**	**99**	**30**
Total Metal - Au (koz)	**900**	**72**	**81**	**85**	**114**	**118**	**114**	**103**	**70**	**73**	**61**	**10**
Total Grade - Au (g/t)	**3.56**	**3.99**	**4.05**	**3.40**	**4.64**	**3.40**	**3.65**	**3.68**	**2.91**	**3.01**	**2.95**	**4.05**
Ore Breakout												
Stoping Ore (kt)	7,051	492	564	703	717	882	886	650	703	740	635	78
Stoping Ore Grade – Au (g/t)	3.56	3.88	3.95	3.29	4.60	3.49	3.63	4.04	2.91	3.01	2.94	4.06
Development Ore (kt)	808	70	61	76	45	197	82	217	43	8	8	2
Development Ore Grade -Au (g/t)	3.54	4.80	4.96	4.44	5.33	2.97	3.80	2.61	2.85	3.04	3.89	3.60
Development Summary												
Total Lateral Development (m)	**33,427**	**4,301**	**4,053**	**4,914**	**4,170**	**5,086**	**5,201**	**4,879**	**584**	**109**	**99**	**30**
Total Vertical Development (m)	**2,525**	**508**	**427**	**408**	**390**	**117**	**675**	**0**	**0**	**0**	**0**	**0**
Lateral Development Ore (m)	10,212	827	721	898	532	2,551	1,046	2,831	568	109	99	30
Lateral Development Waste (m)	23,215	3,473	3,332	4,017	3,638	2,535	4,155	2,048	16	0	0	0
Lateral Development Breakout												
ACC - Level Access	1,554	297	263	119	180	199	496	0	0	0	0	0
CRF - CRF Stockpile	549	88	67	74	31	90	128	71	0	0	0	0
DEC - Decline	5,064	669	717	1,012	1,416	631	620	0	0	0	0	0
DSM - Decline Sump	73	0	0	13	39	0	21	0	0	0	0	0
DSP - Decline Stockpile	506	57	60	127	182	59	20	0	0	0	0	0
EWD - Escapeway Drive	896	67	83	213	139	47	346	0	0	0	0	0
FAD - Fresh Air Drive	870	195	0	245	279	69	83	0	0	0	0	0
FWD - Footwall Drive	2,685	473	318	271	204	341	704	375	0	0	0	0
LSM - Level Sump	241	16	31	0	27	10	157	0	0	0	0	0
MAG – Underground Magazine	90	0	0	90	0	0	0	0	0	0	0	0
OED - Ore Exploration Drive	2,876	322	420	325	270	423	825	291	0	0	0	0
OPD - Ore Production Drive	11,709	1,398	1,267	1,413	518	2,350	803	3,611	350	0	0	0
PBY - Passing Bay	185	46	53	29	28	0	29	0	0	0	0	0
PSN - Pump Station Drive	280	45	26	24	68	20	98	0	0	0	0	0
RAD - Return Air Drive	1,490	232	243	389	177	86	363	0	0	0	0	0
SLD - Slot Drive	3,796	357	406	573	387	631	444	526	234	109	99	30
SUB - Substation	258	40	20	0	119	8	66	5	0	0	0	0
WKS - Underground Workshop	229	0	0	0	106	123	0	0	0	0	0	0
XCT - Crosscut	78	0	78	0	0	0	0	0	0	0	0	0

	Totals	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Vertical Development Breakout												
DH_ Drainhole	364	47	102	0	0	26	190	0	0	0	0	0
EWR - Escapeway Rise	366	65	102	33	19	19	128	0	0	0	0	0
FAR - Fresh Air Rise	269	65	0	163	20	0	21	0	0	0	0	0
RAR - Return Air Rise	871	83	77	194	321	72	124	0	0	0	0	0
RMH_ Rising Main	288	81	73	0	0	0	134	0	0	0	0	0
SH_ Service Hole	368	167	74	19	30	0	79	0	0	0	0	0

Source: OceanaGold, 2023

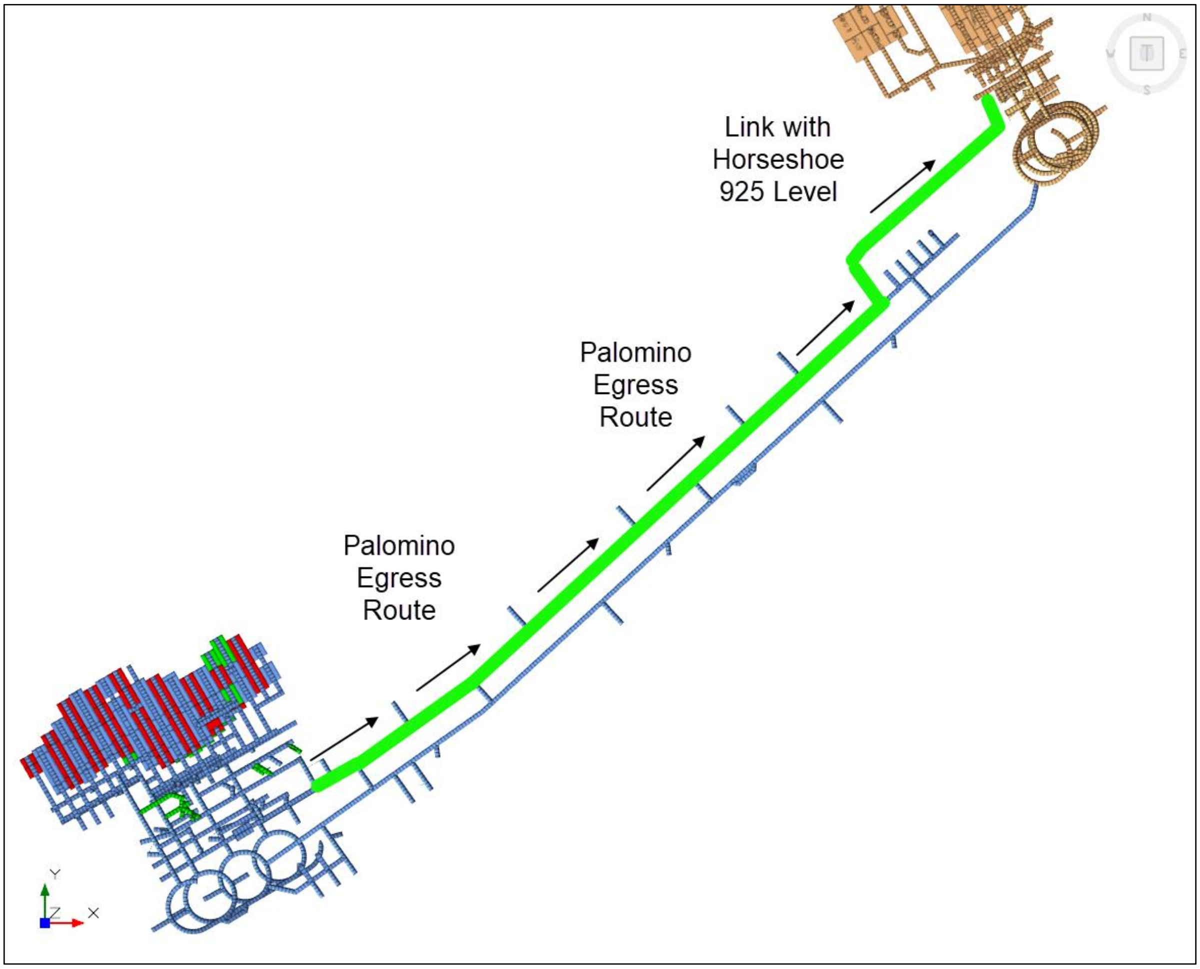
16.2.9 Mining Operations

Mine Access

The mine is accessed via a portal within Snake Pit. The decline is designed at a maximum gradient of 14%. A minimum turning radius of 27.5 m is used, which is suitable for the underground haul trucks for the operation.

Ventilation, power, water discharge, supply water, and communications are installed at the portal and carried down the decline to support the initial development operation. Secondary egress in Horseshoe and Palomino will be via 1.1 m diameter raisebored escapeway raises fitted with ladderways. The escapeway path for Horseshoe begins at the fresh air portal within Snake Pit, then across a link drive to the exhaust drive, then follows the fresh-air system to the 900 m RL, then finally follows the return air system to the bottom of the mine.

For Palomino, during tunnel construction, refuge chambers will be required in the active areas and will be located in strategic locations and be of sufficient size to accommodate the number of workers within the area. At the orebody, secondary egress will follow the same designs as Horseshoe, with secondary egress via 1.1 m diameter raisebored escapeway raises fitted with ladderways within the fresh air system. The escapeway system at the Palomino orebody links up with the dual Palomino fresh air tunnel at the 775 m RL level. Egress to the surface from here is via the second fresh air tunnel which then links into the proposed escapeway system at Horseshoe at the 925 m RL level as shown in Figure 16-47.



Source: OceanaGold,2023

Figure 16-47: Palomino Egress via Horseshoe - Plan View

Egress for Palomino via the surface exhaust shaft (within return air) and an additional intake shaft near Snake Pit have been deemed unsuitable.

Stoping

Stopes will be mined using the sublevel open stoping method. Individual stope blocks are designed to be 15 to 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 within a slot drive that is mined on the northern wall of the stope adjacent to the recently backfilled stope. A bottom up, primary/secondary extraction sequence will be followed utilizing sill pillars to ensure early access to production ore. Primary and secondary stopes will be backfilled with CRF, whilst stopes situated above sill pillars will require higher strength CRF (increased cement content) to ensure stability during extraction.

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. All blasting with be with bulk emulsion.

Ore will be mucked from the bottom stope access using a 14.9-t LHD. Cable bolts will be installed in stope crowns and brows to ensure stability. 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

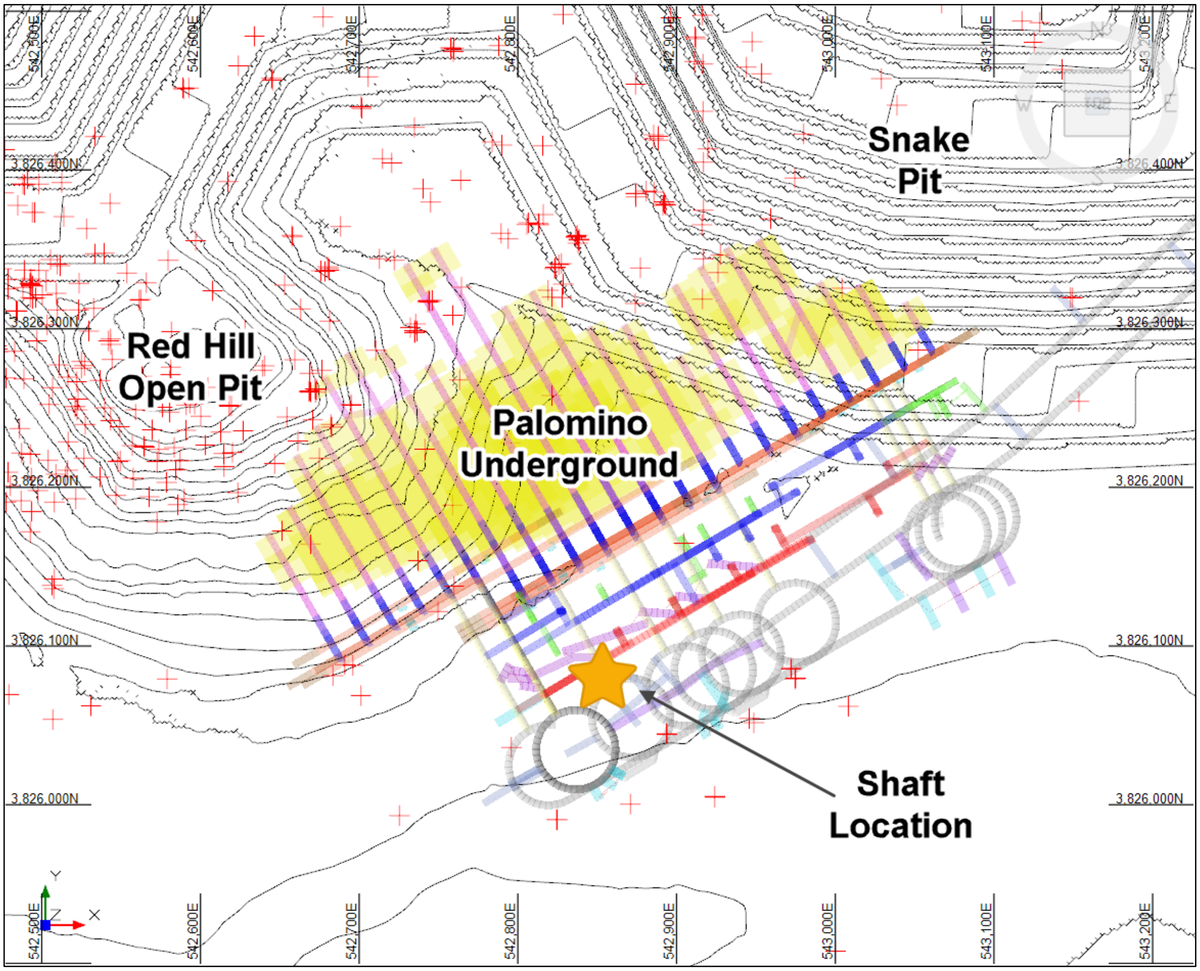
Lateral development includes interlevel ramps, level accesses, stope accesses, and short connecting drifts for ventilation. The interlevel ramp system will be a continuation of the main access decline and will have the same maximum gradient (14%). Level accesses will be 5 m wide x 5.5 m high with a flat 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

Inter-level ventilation rises at Horseshoe and Palomino 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. Palomino will require additional vertical development to deliver adequate quantities of air to active headings during initial development, and during steady-state production, to minimize the impact on the Horseshoe ventilation network. Two internal raisebored shafts are proposed at the top of the Palomino access decline. The first shaft (3 m diameter) will connect into the top of the Horseshoe return air circuit and provide return air during Palomino decline development and return air for the magazine during steady-state production. The second shaft (5 m diameter) will connect into an additional fresh air portal in Snake Pit and provide fresh air when Palomino commences production. A ~360 m long, 5 m diameter surface shaft at the Palomino orebody, illustrated in Figure 16-48, is planned and will act as an exhaust shaft once complete.

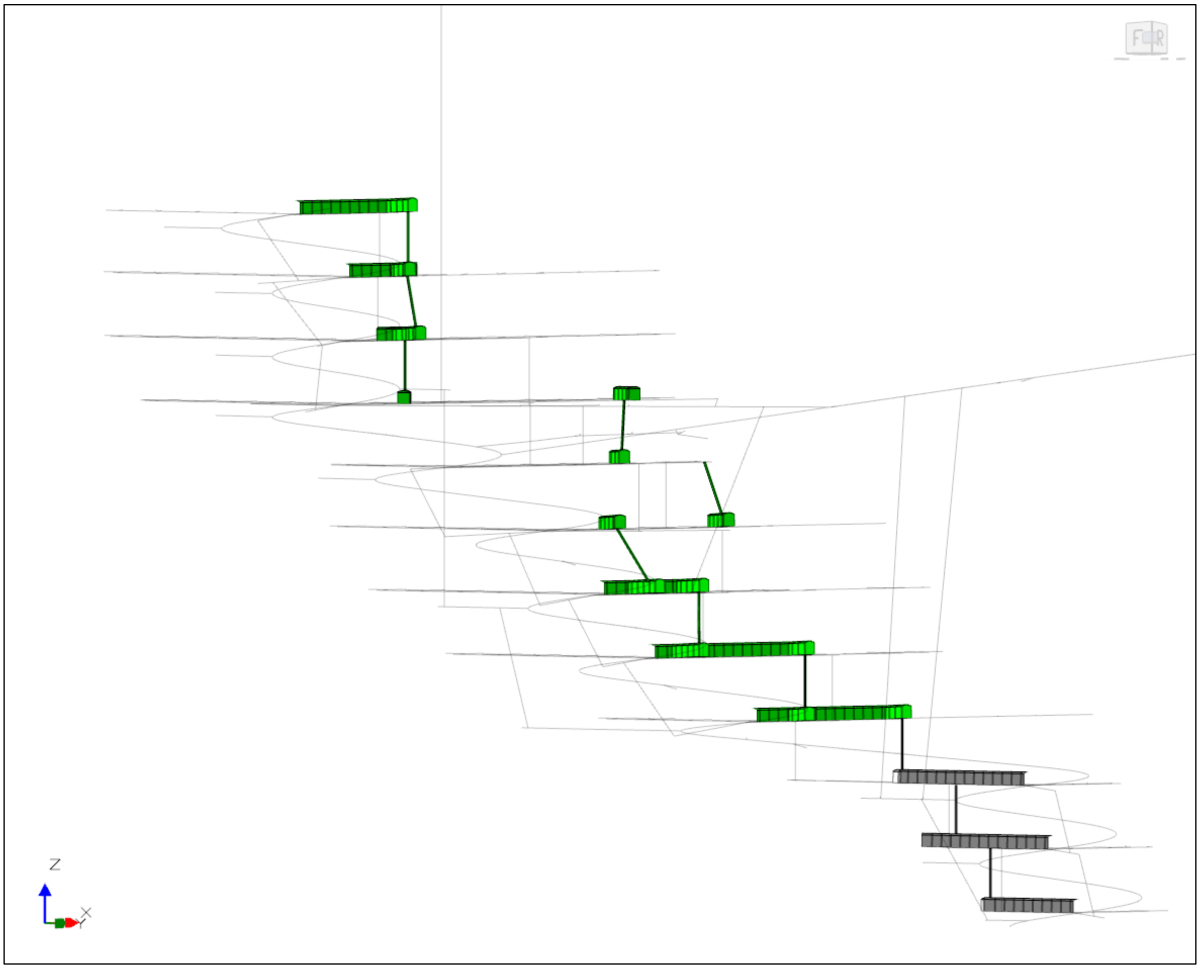


Source: OceanaGold, 2023

Figure 16-48: Palomino Surface Shaft Collar Location

Geotechnical analysis has shown zones of poor ground (coastal sands and saprolite) for the top section of the shaft (0 to 70 m) and more competent material (dacite metavolcanics) for the remainder of the shaft. Various methods of shaft construction have been assessed. A combination of blind sink through the upper portion of the shaft, then a slash and line methodology into a 2.4 m raisebore for the remainder of the shaft, is the preferred method of shaft construction as it mitigates against geotechnical risks and potential delays for shaft construction, which will impact upon Palomino ramp-up and production rates. Further analysis of surface shaft construction will be undertaken during subsequent study phases.

Escapeways between levels will be sub-vertical and raisebored at a diameter of 1.1 m and is illustrated in Figure 16-49. 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.



Source: OceanaGold, 2023

Figure 16-49: Interlevel Escapeway Connections

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 2 to 3.1 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 when Horseshoe and Palomino are both in production, thirteen 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. Green Non-PAG overburden will be supplied by surface operations to the UG ROM:

- The Crush Screen Plant (CSP) average output will be 900 m^3 per day producing a crusher run aggregate containing both course and fine aggregates.

- Average yearly backfill requirement of 260,000 m^3 will require approximately 470,000 tonnes of aggregate.

The CRF batch plant is located in the UG Infrastructure area:

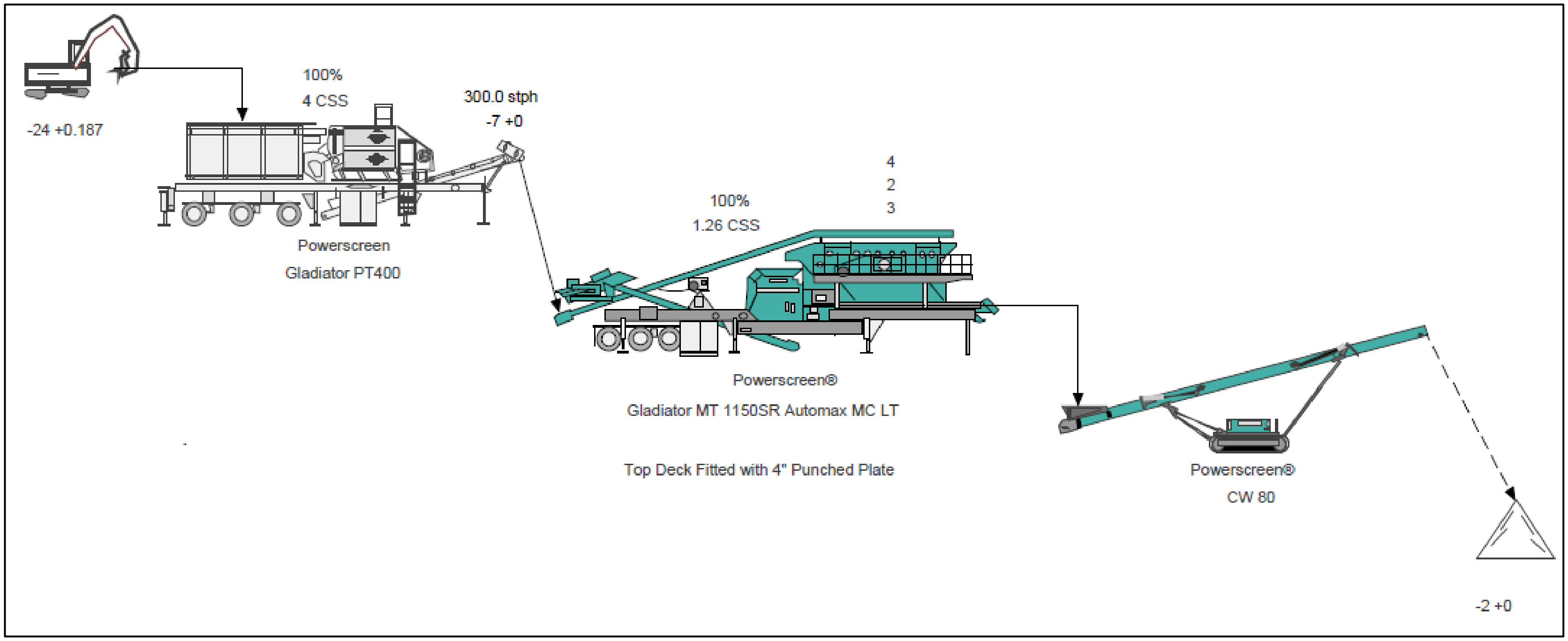
- The automated batch plant mixer makes 5 m^3 x 3 m^3 CRF batches per haul truck.
- The batching time is about 15 minutes to make the five batches.
- The batch plant will typically produce 45 m^3/hr x 20 hours per day, i.e., 900 m^3/day.
- Typical annual requirement of 264,000 m^3 would require 293 CRF days per year.

The underground CRF production sequence will work as follows:

- The haul trucks will tip their loads into a CRF stockpile bay located on each footwall drive.
- An LHD will move the CRF from the stockpile to the stope.

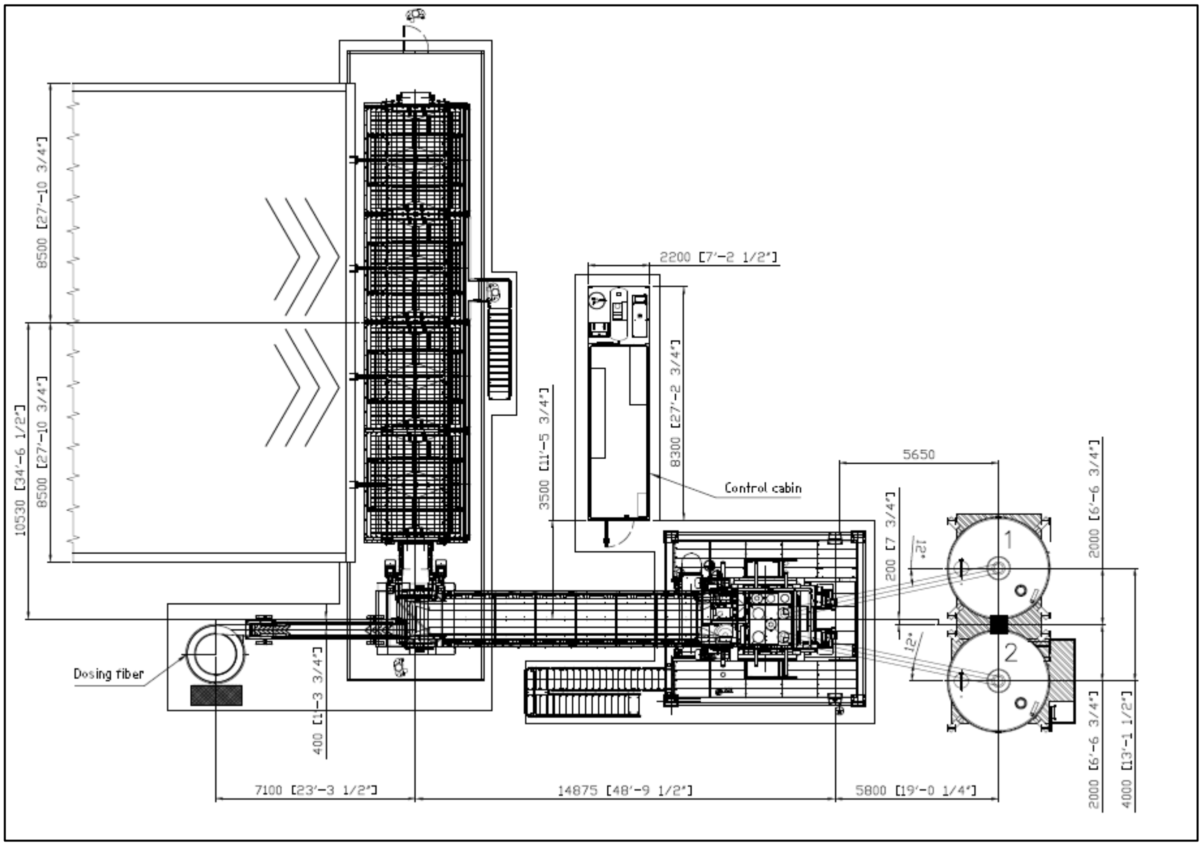
Two different CRF mix designs will be required: (1) low strength, and (2) high strength. The low strength mix will require about 4.2% cement by dry mass. The high strength mix will require about 5.2% by dry mass.

Figure 16-50 shows the crushing and screening plant and Figure 16-51 shows the CRF plant.



Source: OceanaGold, 2024

Figure 16-50: Crushing and Screening Plant for Backfill Aggregate



Source: OceanaGold, 2024

Figure 16-51: CRF Backfill Plant

Low strength CRF (4.2% cement) is required for backfill on all primary and secondary stopes except for stopes above planned sill pillars. High strength (5.2% cement) and high quality CRF is required in stopes directly above planned sill pillars to ensure crown stability during extraction. Table 16-45 shows the total LoM volume breakdown of the rock fill (low strength) and CRF (high strength).

Table 16-45: LoM Backfill Quantities

Backfill Summary	(m^3)
Total Backfill Volume	**1,458,961**
CRF High Strength (4.2% cement)	58,358
CRF Low Strength (5.2% cement)	1,400,603

Source: OceanaGold, 2024

Waste Rock

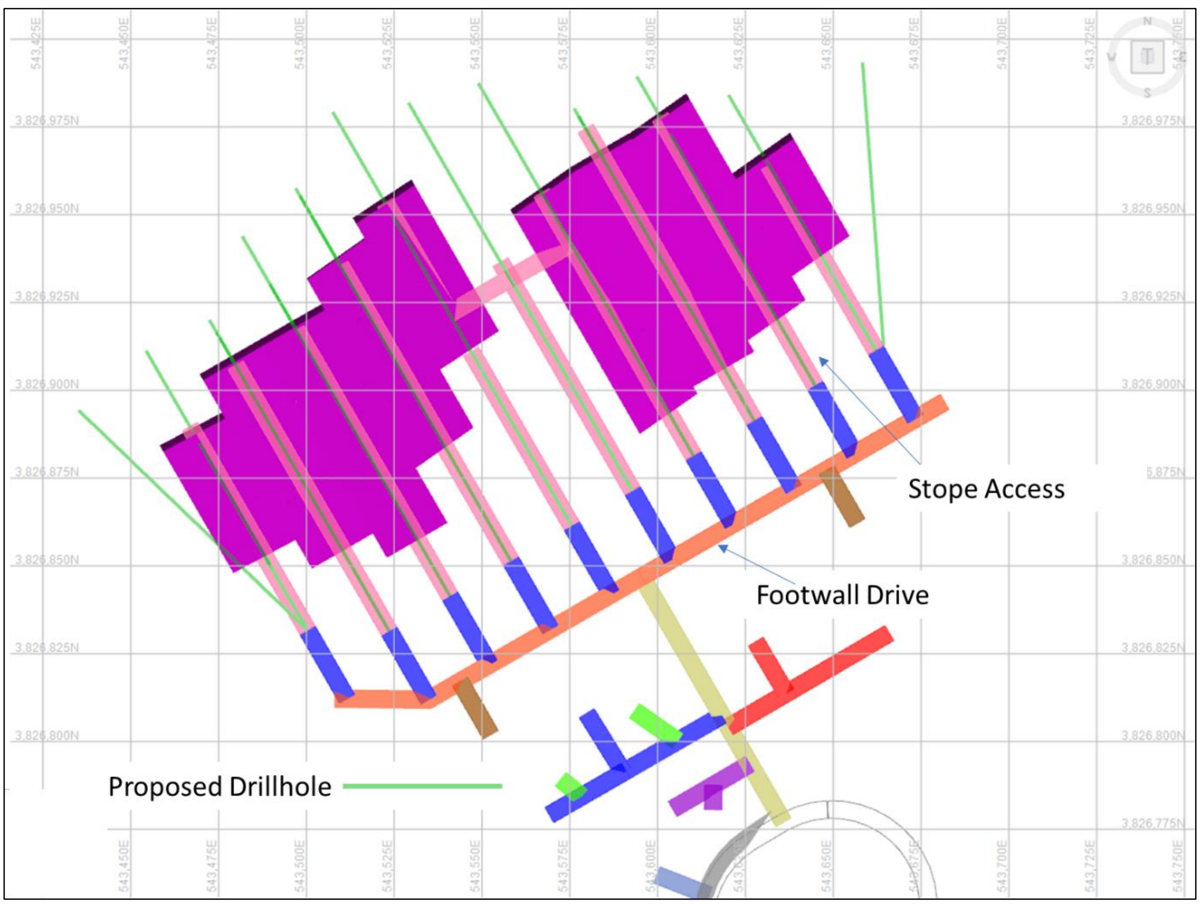
Waste rock from the underground will be hauled to the surface to be rehandled by surface operations to a lined PAG facility. Although there is scope to haul some waste to secondary stopes, it is not included in this study.

Grade Control

The characterization of ore vs. waste will be completed through diamond core drilling of the stope accesses prior to mining as shown in Figure 16-51. 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 form the start of each ore drive on every production level. Holes will target the ore production drives and into the stope(s). Stopes will have a drillhole pierce on both the hanging wall and footwall of mineralization. The drilling in 2023 totals approximately 5,000 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. Geologic and production block models will be updated with this information and ore/waste grade boundaries will be determined prior to mining the stope accesses. All waste will be classified as Red and will be stockpiled according to the Overburden Management Plan. Geochemical sampling techniques for the underground mine will follow existing underground core sampling procedures. 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 24 to 48 hours.



Source: OceanaGold,2023

Figure 16-51: Level Section showing Planned Diamond Drilling from Stope Accesses

Additionally, as the stope accesses are being mined, a mine geologist will observe and map the exposed rock on each rib. If necessary, channel samples will be collected from the ribs or face of ore development drives. Channel samples will be orientated perpendicular to the dominate fabric if collected from the ribs. Sample breaks will be determined by the geologist and will occur at lithologic or alteration boundaries. Generally, samples will be kept below 1 m in total length. Channels can be sawn or chipped depending on the nature of the rock being collected.

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 LoM development model. A 3D ventilation model is maintained 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).

No harmful strata gases are encountered at this site. No crushers or fixed ore/waste conveyances (continuous acute dust sources) are currently designed underground. 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 between levels are 4m x 6m or 5m x 5m. Auxiliary ventilation supplied on each level by 110kw fan with 1.4m vent duct in the main decline and level access. Ore drives are supplied by 1.2m vent duct. Model friction factors, resistances, shock losses, etc. were used based on available data and standard best practice as shown in Table 16-46.

Table 16-46: General Ventilation Modeling Parameters

Design Point	Friction Factor (kg/m^3)	Resistance (Ns^2/m^8)	Shock X	Notes
Standard Development Drive	0.011			Standard base value for general horizontal development
Ramp	0.011		Auto Low	Slightly elevated shock loss to account the additional losses associated with the spiral development
Ventilation Development (Fresh Air Drive and Return Air Drive)	0.011		Auto Mid	Elevated shock loss associated with sharp direction changes between ventilation raises and drifts.
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
Equipment Door with Man Door		20		The return drift is accessed by a crosscut drift with airlock equipment doors and mandoor. The doors are reinforced and shotcreted to reduce leakage.

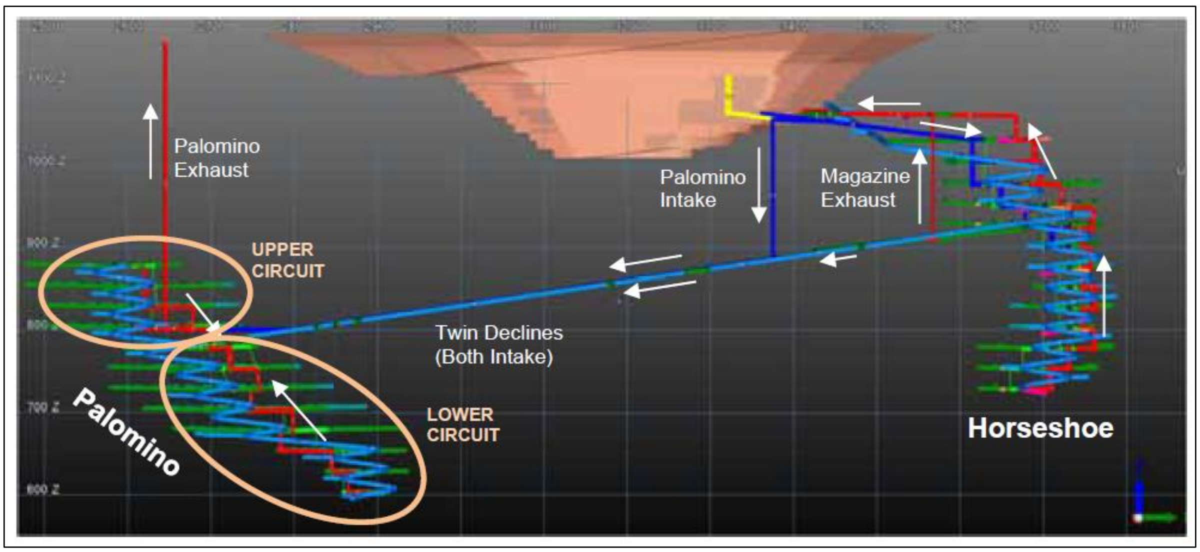
Source: OceanaGold, 2024

Ventilation studies for Palomino have focused on requirements for an interlinked ventilation network for both orebodies (Horseshoe and Palomino) and ensuring minimal interaction and changes to the current planned Horseshoe network. Ventilation for the initial Palomino access decline can be supplied via the Horseshoe primary ventilation network through the addition of secondary (110 kW) fans underground. This setup will allow for initial development of the access declines and some development at the orebody. However, once Palomino is in production, additional primary ventilation infrastructure is required as the Horseshoe ventilation network will not be able to provide enough air to ventilate both orebodies. Major ventilation infrastructure requirements for the inclusion of Palomino in the mine plan include:

- Initial ventilation for Palomino development will be provided via secondary fans and a return air connection (via raisebore) to the top of the current Horseshoe return air circuit. Once Palomino is in steady state production, this shaft will serve as an exhaust for the underground magazine.
- An additional intake opening in Snake Pit is strongly recommended. An option to utilize the current intake portal for both orebodies was assessed and is technically feasible. However, this introduces hazards around elevated velocity (>13 m/s) entering through this portal due to the large increase of volume required for both orebodies. This introduces hazards for

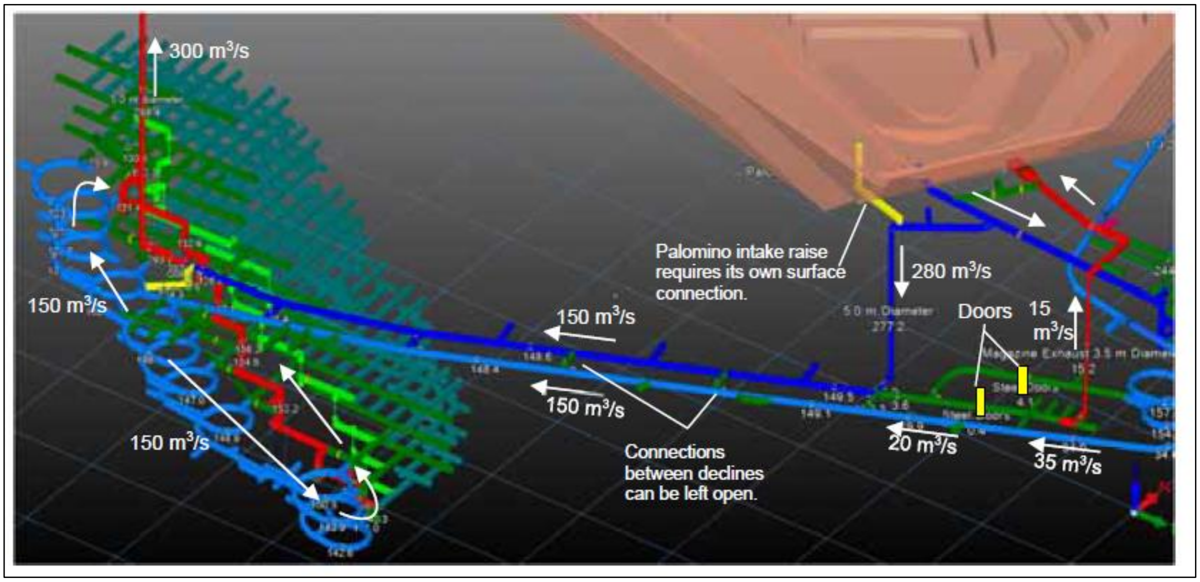
personnel who need to access through this portal to inspect and maintain the Horseshoe exhaust fans. Therefore, an additional opening into Snake Pit (portal) approximately 70 m south of the current intake portal has been incorporated into Palomino designs.

- A ~330 m long, 5 m diameter surface shaft is planned at the Palomino orebody which will act as a return air rise during steady-state operations, with intake provided by the main Palomino access tunnels and the aforementioned additional intake portal.
- Primary fan configuration is two single stage axial fans mounted in parallel in an underground chamber (very similar to current Horseshoe setup). Recommended airflow is 290 m3/s with a total motor size of 740kW (370kW per fan). It is proposed to operate an upper and lower circuit for Palomino, similar to the planned Horseshoe arrangement, and illustrated in Figure 16-52 and Figure 16-53.
- Secondary ventilation and interlevel return-air connections for Palomino are in-line with current Horseshoe design parameters.



Source: OceanaGold, 2023

Figure 16-52: Long-section Palomino Steady-State Airflow Distribution



Source: OceanaGold, 2023

Figure 16-53: Oblique view Palomino Steady-State Airflow Distribution

Airflow Requirements

The 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 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 conservativeness in the evaluation. A general leakage value of 20% was assumed to provide the minimum airflow requirement. Designed airflow for the mine exceeds the ventilation required for the equipment based on the MSHA/NIOSH nameplate dilution values, meeting the requirements from CFR57.5067. Overall, the minimum airflow requirement for the Horseshoe mine is approximately 250 m^3/s for the support of CRF method as shown in Table 16-47.

The diesel equipment fleet operating in the Palomino ventilation circuit would be similar to the fleet numbers assumed for the Horseshoe ventilation design (i.e. three LHDs, four trucks and associated ancillary equipment). The nominal design primary airflow rate for Palomino should therefore also be similar to the design flow originally used for Horseshoe (i.e., 270 m^3/s). However, for this report the total design airflow rate for Palomino was increased by 10% to 300 m^3/s to cater for the small proportion (20 to 30 m^3/s) of polluted airflow that will be drawn into the Palomino circuit from the Horseshoe decline.

Table 16-47: Cemented Rock Fill Equipment Load and Minimum Airflow Calculation

Equipment (Main Equipment Only)	Rated Power (kW)	2024	2025	2026	2027	2028	2029	2030
LHD (LH517i) - Tier 4	315	4	4	4	4	4	4	4
Haul Truck (TH 551i) - Tier 4i	515	6	6	6	6	6	6	6
Spray Mech - Tier 3	90	1	1	1	1	1	1	1
Agi Truck - Tier 3	120	1	1	1	1	1	1	1
Large IT - Tier 4	225	1	1	1	1	1	1	1
Small IT - Tier 4	130	1	1	1	1	1	1	1
12k Grader - Tier 4	225	1	1	1	1	1	1	1
Charge-Up - Tier 3	120	1	1	1	1	1	1	1
Scissor Lift - Tier 3	90	1	1	1	1	1	1	1
Multimec- Tier 3	90	2	2	2	2	2	2	2
Diesel Dilution at 0.04 $m^3/s/kW$ (m^3/s)		221.2	221.2	221.2	221.2	221.2	221.2	221.2
General Leakage Allowance		10%	10%	20%	20%	20%	20%	20%
Total Minimum Airflow Requirement (m^3/s)		**243**	**243**	**265**	**265**	**265**	**265**	**265**

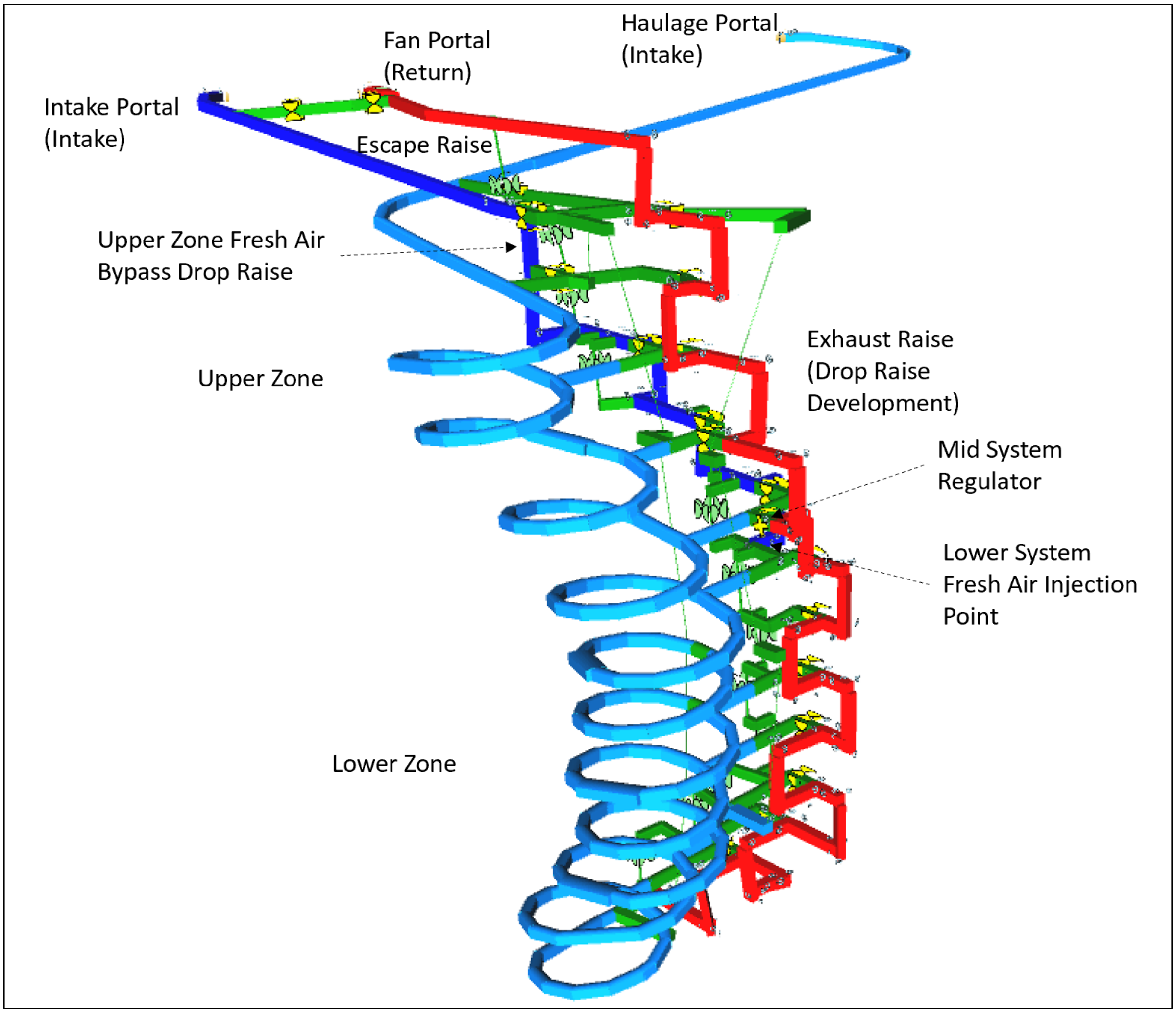
Source: OceanaGold, 2024

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 operates at approximately 40 m3/s at 1.5 kPa, 110kW (allowing for approximately 15% leakage).

Ventilation Model

The LoM ventilation model is maintained using the measured/indicated/inferred reserve mine design to analyze the long range potential needs of the mine. 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 16-54.



Source: OceanaGold, 2023

Figure 16-54: Ventilation System Layout

Horseshoe Underground Mine is ventilated using a parallel exhaust fan system (2 fans). This allows HUG to achieve the pressure and volume needed to properly ventilate the mine while also giving redundancy to the main ventilation system in the event of a fan outage. The fans are controlled by variable frequency drives, giving the mine increased control of airflow in the system.

The fresh air raise system will also contain the secondary raises from level to level. This provides fresh air to the secondary escapeway raise system, which gives miners the ability to escape through fresh air out the secondary intake portal out to the surface.

Table 16-48: Main Fan Requirements

Scenario	Total Fan Requirement			
	Airflow (System) (m^3/s)	Total Pressure (kPa)	Efficiency (%)	Power (kW)
Stage 2 (full system – steady state)	270	1.702	82%	660

Source: OceanaGold, 2024
System efficiency presented in this table is based on aerodynamic bench test, efficiency at full mine development may be different.

The parallel Zitron ZVN 1-24-330/6 330 kW fans installed at Horseshoe underground for the steady state ventilationare designed for the long-term ventilation of the mine. However, it is best practice to limit leakage and reduce airway resistance.

16.2.11 Mine Services

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 hour shift is supported by a four-crew rotation.

Table 16-49 shows the required workforce by year.

Table 16-49: Mine Labor by Year

Category	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Supervision	7	7	7	7	7	7	7	7	4	4	4
Tech Services	21	21	21	26	26	26	26	26	21	21	13
UG Mining	117	125	125	129	149	150	147	139	112	78	53
Maintenance	68	64	64	64	64	64	64	64	64	42	42
Total	**213**	**217**	**217**	**226**	**246**	**247**	**244**	**236**	**201**	**145**	**112**

Source: OceanaGold, 2023

The distribution of owner and contractor labor is shown in Table 16-.

Table 16-: Mine Labor by Year: Owner versus Contractor

Category	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
OceanaGold	173	180	184	193	246	247	244	236	201	145	112
Contractors	40	37	33	33	0	0	0	0	0	0	0
Total	**213**	**217**	**217**	**226**	**246**	**247**	**244**	**236**	**201**	**145**	**112**

Source: OceanaGold, 2023

Mine Mobile Equipment

The mine mobile equipment requirements as summarized in Table 16-50 are based on the production schedule.

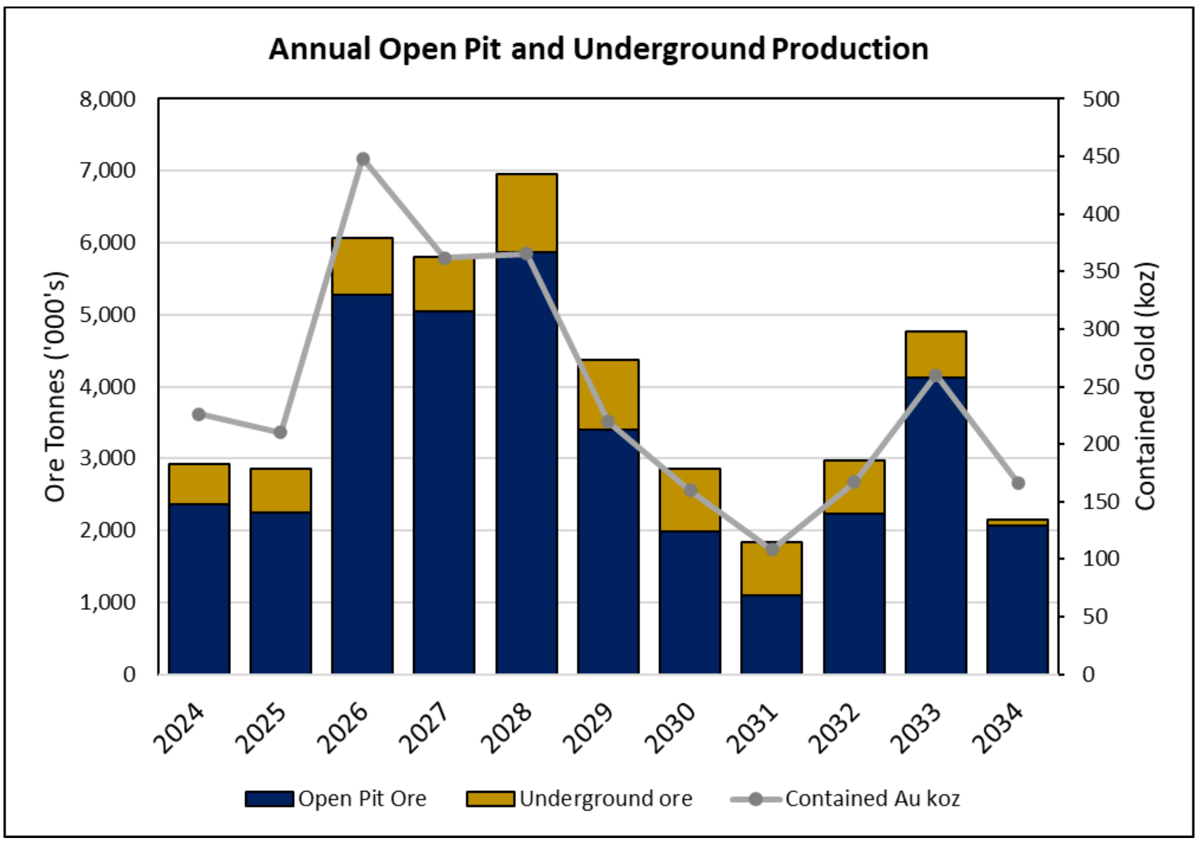
Table 16-50: Mobile Equipment Fleet

Category	Make	Model	Max Units
Development Drill	Sandvik	DD421-60C	4
Bolter	Sandvik	DS421C Bolter	2
Longhole Drill	Sandvik	DL421-7C	3
LHD	Sandvik	LH517	8
Truck	Sandvik	TH551	13
Shotcreter	Spraymec	6050WP	1
Transmixer	UTIMEC	MF 500	1
Wheel Loader	Caterpillar	962H	2
Wheel Loader	Caterpillar	938K	3
Grader	12K	12K	1
Transporter	UTIMEC	Multimec	3
Charge-up	Charmec	MC 605	3
Scissor Lift	UTIMEC	Scissor	3

Source: OceanaGold, 2023

16.3 Combined Open Pit and Underground Production Schedule

Figure 16-55 and Table 16-51 show the combined open pit and underground production schedule annually.



Source: OceanaGold, 2023

Figure 16-55: Annual Open Pit and Underground Production

A schedule optimization process is planned to be completed in 2024 that will aim to identify the ideal start date for the Palomino Underground, open-pit production rate, and maximum stockpile size to maximize cashflow for the overall project.

Table 16-51: Combined Open Pit and Underground Production Schedule [(1)]

Year		2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	LoM Total
Total (OP+UG) Production													
Ore	kt	2,921	2,863	6,059	5,808	6,955	4,365	2,850	1,839	2,974	4,772	2,151	43,559
Ore	t/d	8,003	7,844	16,600	15,912	19,055	11,962	7,808	5,038	8,151	13,077	5,893	10,849
Au Grade	g/t	2.41	2.29	2.30	1.94	1.63	1.56	1.75	1.84	1.75	1.69	2.40	1.92
Contained Au	koz	226	210	448	362	365	219	160	109	167	260	166	2,692
Ag Grade	g/t	2.37	2.13	2.32	2.37	2.12	2.34	2.63	2.22	2.70	2.53	2.93	2.38
Contained Ag	koz	223	196	451	442	474	328	241	131	259	389	202	3,335
Underground													
Ore	kt	562	624	778	761	1,079	969	867	747	749	643	80	7,859
Ore	t/d	1,539	1,711	2,133	2,086	2,956	2,654	2,375	2,046	2,052	1,761	220	1,957
Au Grade	g/t	3.99	4.05	3.40	4.64	3.40	3.65	3.68	2.91	3.01	2.95	4.05	3.56
Contained Au	koz	72	81	85	114	118	114	103	70	72	61	10	900
Ag Grade	g/t	1.75	1.70	1.68	1.78	2.33	2.47	2.10	2.01	3.03	2.57	3.49	2.18
Contained Ag	koz	32	34	42	43	81	77	58	48	73	53	9	551
Waste	kt	293	276	348	303	198	353	187	1	13	-	-	1,972
Open Pit													
Ore	kt	2,359	2,239	5,281	5,047	5,876	3,397	1,983	1,092	2,226	4,130	2,071	35,700
Ore	t/d	6,446	6,134	14,468	13,827	16,056	9,307	5,433	2,993	6,081	11,314	5,673	8,885
Au Grade	g/t	2.03	1.79	2.14	1.53	1.31	0.97	0.90	1.10	1.32	1.50	2.34	1.56
Contained Au	koz	154	129	363	248	247	105	57	39	95	199	156	1,792
Ag Grade	g/t	2.52	2.24	2.41	2.46	2.08	2.30	2.86	2.37	2.59	2.53	2.90	2.43
Contained Ag	koz	191	162	409	398	393	251	182	83	186	336	193	2,785
Strip Ratio	t/t	14.4	15.4	7.1	8.2	6.2	11.4	9.9	15.2	6.7	2.0	0.5	7.9
Waste	kt	33,992	34,367	37,372	41,170	36,634	38,892	19,550	16,551	14,860	8,163	1,073	282,624

[(1)] Does not include stockpile material
Source: OceanaGold, 2024

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 flotation 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 led 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 (1.2 MW) and 10 inch cyclone pack
- Installation of an M10000 Isamill (3 MW) 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 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.

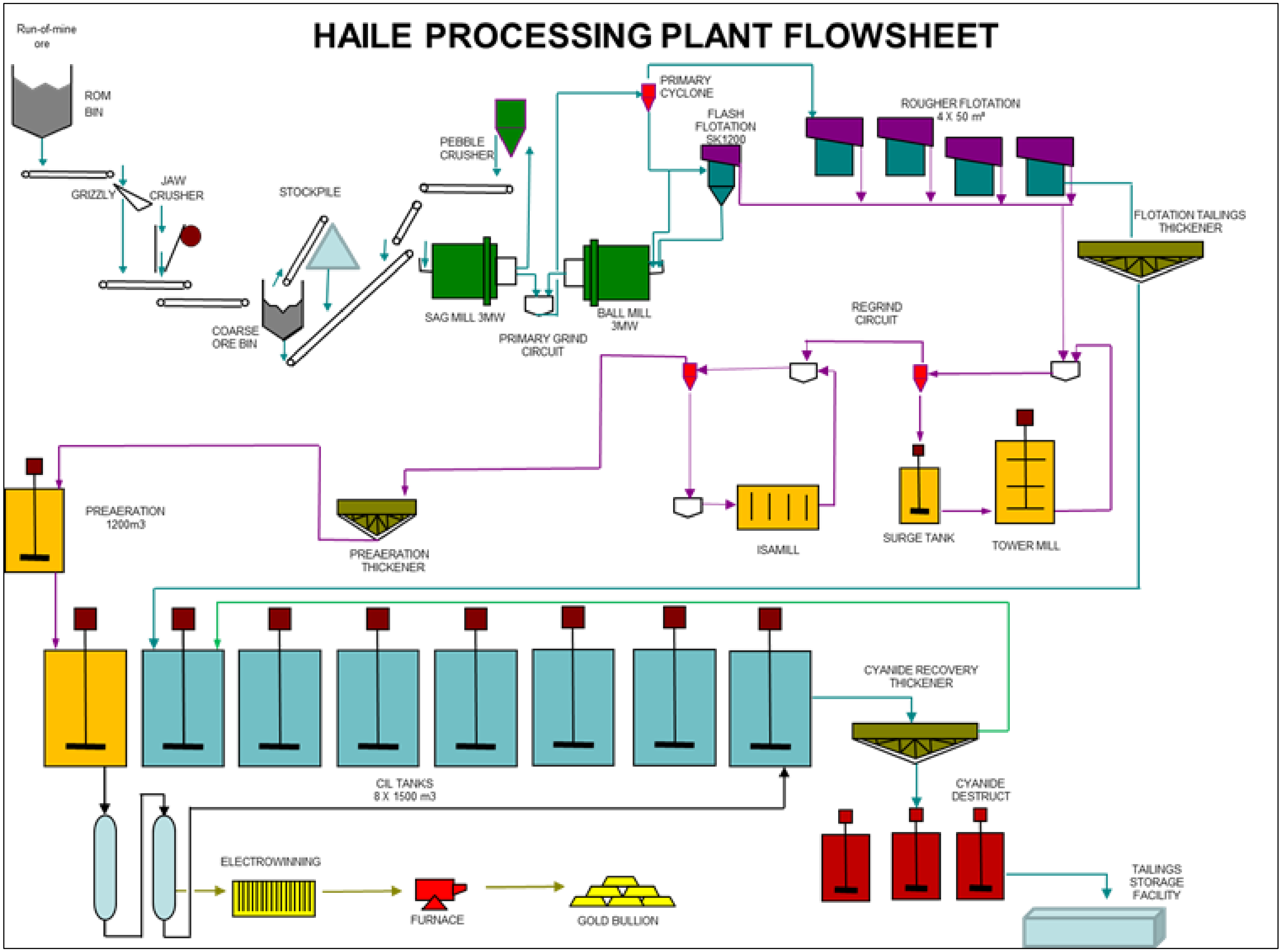
The tailing slurry exiting the CIL is dewatered in a similar thickening stage to the flotation tailings for recovery of the cyanide solution and reduction of the volume of slurry needing to be treated by oxidation of the residual cyanide. The detoxified tailing slurry is pumped for long-term storage 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 plant is shown in Figure 17-1.

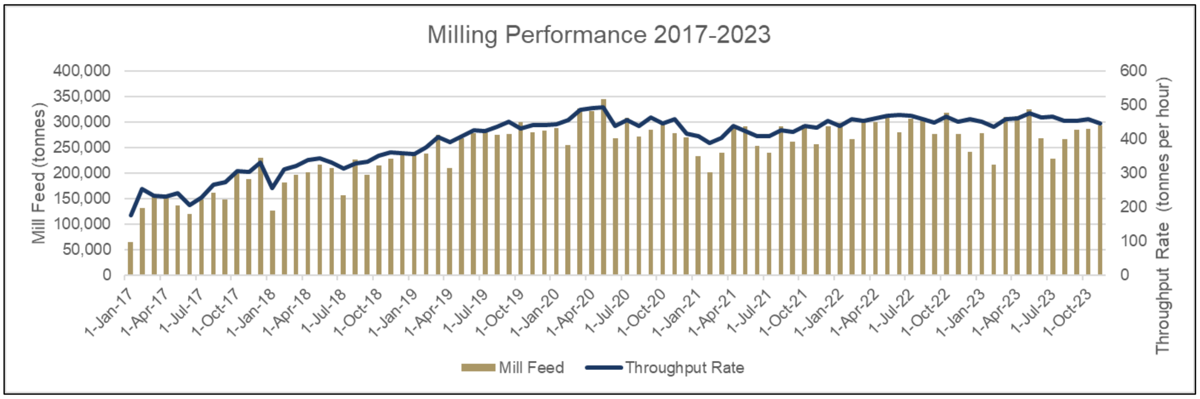


Source: OceanaGold, 2022

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 2022-23 period have varied as a function of ore competency and oxidized clay content in feed between 426 t/h and 479 t/h.



Source: OceanaGold, 2024

Figure 17-2: Mill Throughput Performance Since Startup

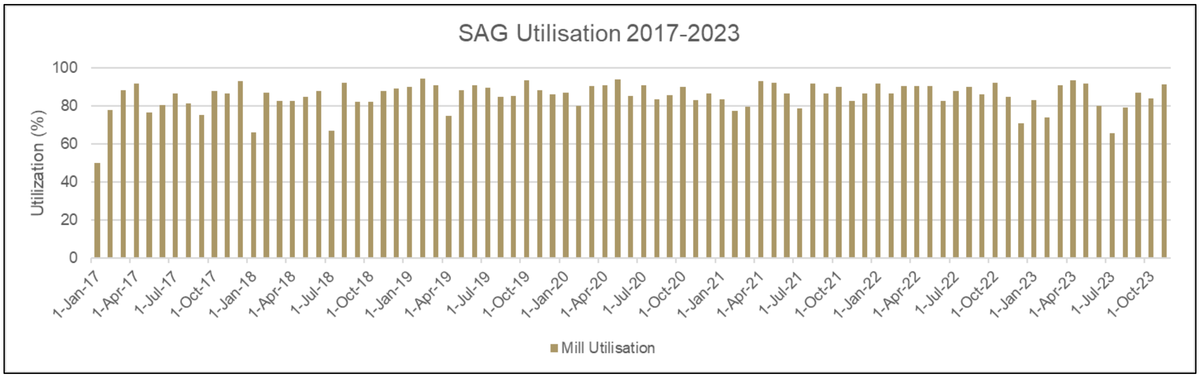
Mill throughput restrictions are tracked by operations and are characterized by cause. The main drivers of reduced throughput are upstream constraints in the crushing circuit during high rainfall events increasing ore moisture, ore competency affecting SAG mill feed rate, pumping capacity constraints in the thickening/final tailings systems or equipment failure within the plant that leads to reduced throughput but not plant stoppage. No single area is the primary constraint on throughput, downtime and restriction data is used to prioritize capital investment such as the upgrade to the final tailings pumps and upgrade of the CIL interstage screens in 2021and thickener underflow pumps in planned in 2024.

SAG mill utilization has progressively improved since startup from 2017-2021 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 84% to 90% range over 2021-23 and was affected significantly in periods of high rainfall and subsequent high moisture levels in the ore supply affecting the primary crusher utilization.

In 2023 a number of unplanned outages occurred that are not regarded as reoccurring events including:

- failure of the SAG mill gearbox input shaft bearing at 50,000 hours service ~162 hours lost
- Failures in the lime slaking system 94 hours
- Issues with the flocculant mixing system leading to thickener outages 51 hours

Engineering rectifications and condition monitoring programs are in place to better forecast failures to minimize future downtime. Long term budget assumptions for utilization target a 92% overall utilization. Historical utilization data since start up is shown in Figure 17-3.



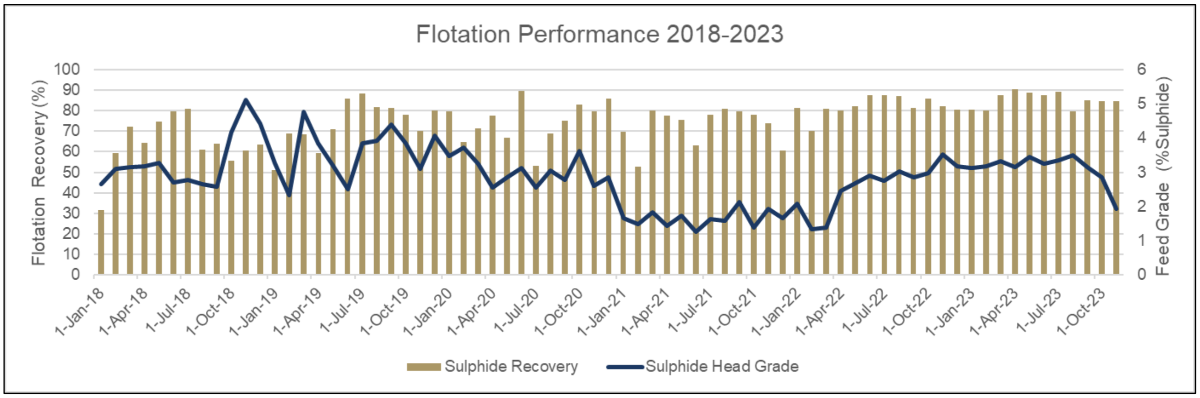
Source: OceanaGold, 2024

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.

Flotation circuit performance is shown below in Figure 17-4 over the last 6 years with the feed grade tracking as expected within the plant design between 2% to 3.5% sulfide sulfur and recovery of sulfide sulfur to the flotation concentrate in the 80-90% range.



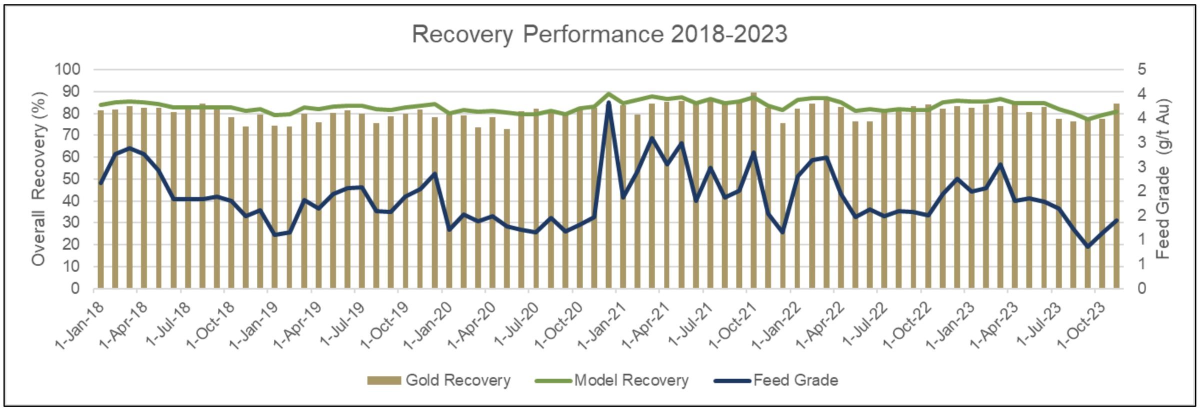
Source: OceanaGold, 2024

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 downcomers 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 was addressed during 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 capable of achieving regrind product sizing in the 14-16 micron range.
- 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 recovery elevated by 2% from 2021 and plant head grade.



Source: OceanaGold, 2024

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
- Upgrades to pumps in the pre-aeration thickener circuit to allow running the Isamill cyclone circuit at lower density to achieve a slightly finer concentrate sizing prior to CIL
- Modification of the CIL interstage screens with alternate screen panel baskets from Derrick to increase the flux rate and overall volumetric capacity of the CIL circuit
- Implementation of an expert control module to run the cyanide destruct circuit to reduce reagent consumption and better control the residual CN_{WAD} levels in the final tail

17.4 Process Unit Costs

Process unit cost history is shown in Figure 17-6ince operations commenced. Unit cost compliance to budget hs from a combination of increased throughput on the fixed cost base and improvements in maintenance planning processes and increasing knowledge of component service life from 2019. In 2023 a number of unbudgeted costs were incurred in the process plant that are not expected to be recurring events including:

- Costs associated with a rental water treatment plant to assist in reducing contact water stored on site in open puts to supplement the existing WTP during and post upgrade, adding US$1.40/t, plant now demobilized from site
- External consultant costs associated with the TSF embankment repair design during 2023 adding a one off US$0.30/t
- Impacts from lower plant utilization and reduced milled tonnes reduced efficiency on the fixed cost component of the plant equivalent to US$1.46/t

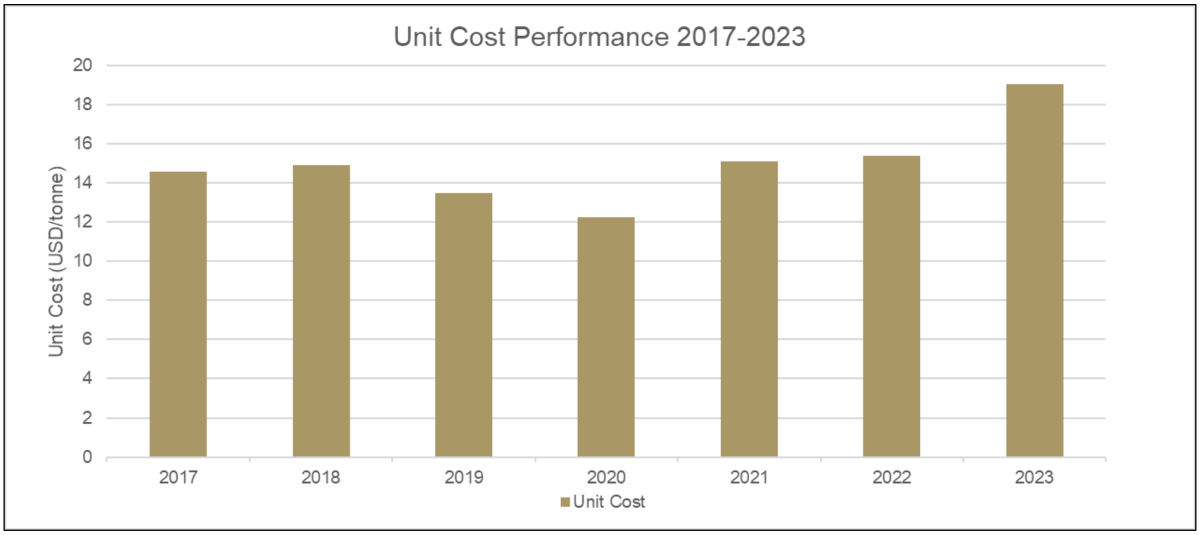
- Electric power consumption for the HUG area was allocated to the process plant until Sep 2023 when systems were established to allocate appropriately

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 cost

Table 17-1: Key Consumable Consumption Rates

Consumable		**Consumption Rate**
2” Ball Mill Balls	kg/tonne ore	0.410
5 SAG Balls	kg/tonne ore	0.360
.75” Tower Mill Balls	kg/tonne ore	0.134
2.5 mm ISA Mill Media	kg/tonne ore	0.067
Carbon	kg/tonne ore	0.050
Promoter	kg/tonne ore	0.010
Frother	kg/tonne ore	0.004
SIBX	kg/tonne ore	0.030
CuSO4	kg/tonne ore	0.120
Flocculant	kg/tonne ore	0.030
NaCN	kg/tonne ore	0.572
Lime	kg/tonne ore	1.880
Ammonium Bisulfate	kg/tonne ore	0.950
HCl	kg/oz Au Produced	1.985
Caustic	Kg/oz Au Produced	1.926
Natural Gas	M^3/oz Au Produced	8
Water Treatment Plant		
Sulfuric Acid	Kg/m^3 discharged	0.011
Flocculant	Kg/m^3 discharged	0.005
Citric Acid	Kg/m^3 discharged	0.033
Caustic Soda Solution	Kg/m^3 discharged	0.051
Hypochlorite	Kg/m^3 discharged	0.051
Ferric Chloride	Kg/m^3 discharged	0.029
TMT-15	Kg/m^3 discharged	0.001
AF304	Kg/m^3 discharged	0.003
$NaMnO_4$	Kg/m^3 discharged	0.039
Sodium Bisulfate	Kg/m^3 discharged	0.005

Source: OceanaGold, 2024



Source: OceanaGold, 2024

Figure 17-6: Process Unit Cost History Since Startup

At the current forecast mill throughput rates of 3.7 Mtpa, process costs are budgeted at an average of US$15.10/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 3.6-3.8 depending on the proportion of underground ore in mill feed 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 labor rates for HGM personnel over the last 24 months
- Increased costs for key reagents from inflationary pressures on suppliers
- inflationary increases to the unit electricity cost

Unit costs have allowed for a progressive increase in water treatment plant capacity from the current 1,000 gpm to 2,640 gpm achieved during 2023 and the associated increased chemical consumption. In 2023 the increased water treatment plant rates contributed $1.93/t to the overall process unit cost with the plant ramped up to maximum capacity for the year.

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 conventional single stage pH adjustment water treatment process using lime and coagulants followed by microfiltration to remove dissolved metals and suspended solids.

In 2022 the original two stage pH adjustment water treatment plant was expanded with two additional trains consisting of a first stage pH adjustment and multiflo clarifier followed by three parallel trains of microfiltration and then pH adjustment to increase capacity from a nominal 1,100 gpm to 2,640 gpm. Commissioning of the upgraded plant occurred from May to August 2023 with the plant now consistently operating up to design capacity to treat accumulated contact water and release it from site.

18 Project Infrastructure

18.1 Tailing Storage Facility

Currently, the Duckwood TSF has been permitted to be constructed in six stages to a crest elevation of 204 m around the maximum storm water pond and the embankment crest sloping in the north corner of the facility. Currently the ultimate capacity of the facility is approximately 63 Mt. The first three stages of the embankment construction have been completed to a crest elevation of 183 m and the Stage IV embankment raise is currently being constructed. Once completed in the 3rd Quarter of 2024, the crest elevation will be at 192 m and the facility will have an approximate capacity of 43 Mt. The ultimate Duckwood TSF layout is presented in Figure 18-1 and Figure 18-2 shows the typical embankment cross-section.

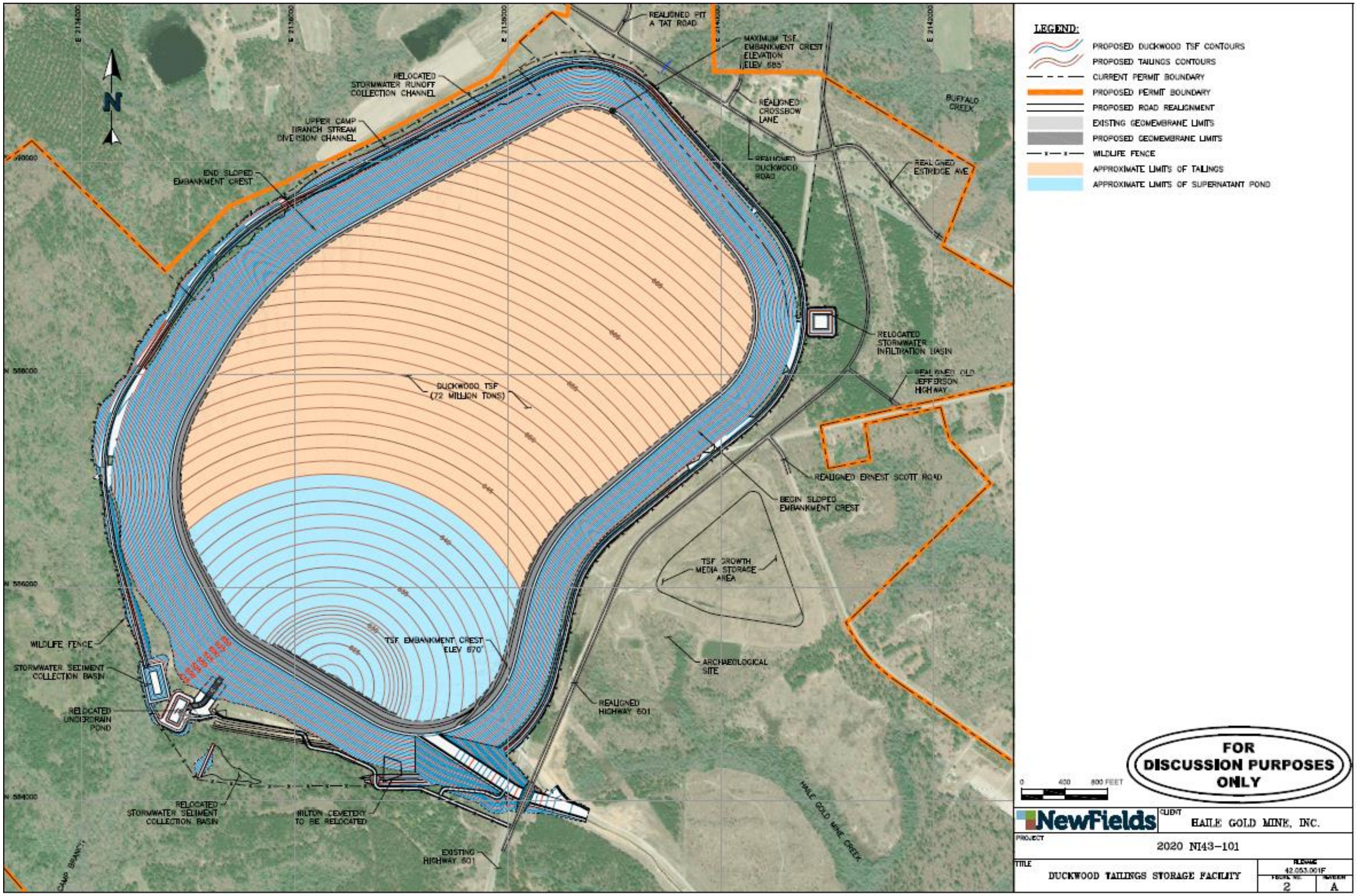
Stage V construction involves raising the embankment, and the following improvements:

- 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 relocating the existing 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 banks of spigots installed on tailings distribution line along the northwest – southeast crestline and outside the supernatant pond limits. Tailings deposition will strategically progress around the facility to maintain slightly sloped surface to drain process water and direct precipitation towards the decant pond that is in the southwest 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.

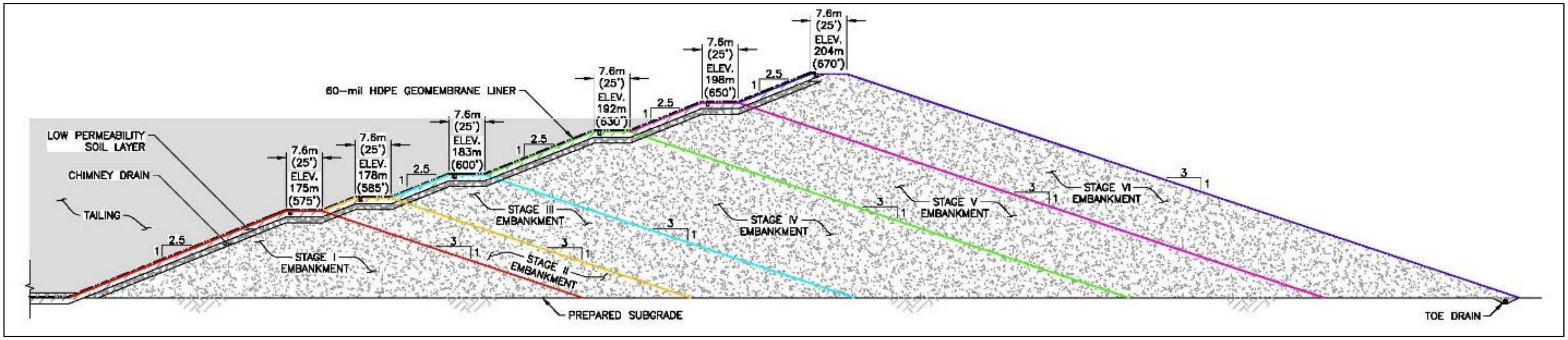
The TSF is designed as a zero-discharge facility. This facility been sized to accommodate the anticipated tailing storage and operating pool requirements, the Probable Maximum Precipitation (PMP) storm event, and an additional 4 ft of freeboard at all times. Note, the existing large supernatant pond has been accounted for in the TSF design and the goal is to reduce the size of the pond via an enhanced evaporation system and increase the volume of water recycled to the mill as much as possible before construction of Stage 5.

A routine inspection of the TSF identified an area of maintenance required in the TSF, noting that it was confirmed to be structurally sound and non-compromising to operations. Pumping of some of the TSF solution from the TSF to the completed Mill Zone pit is underway in order for the maintenance work area to be accessed and attended to during 2024. This plan has been approved by relevant government agencies and the associated costs are included in this report.



Source: OceanaGold, 2022

Figure 18-1: Tailing Storage Facility Layout



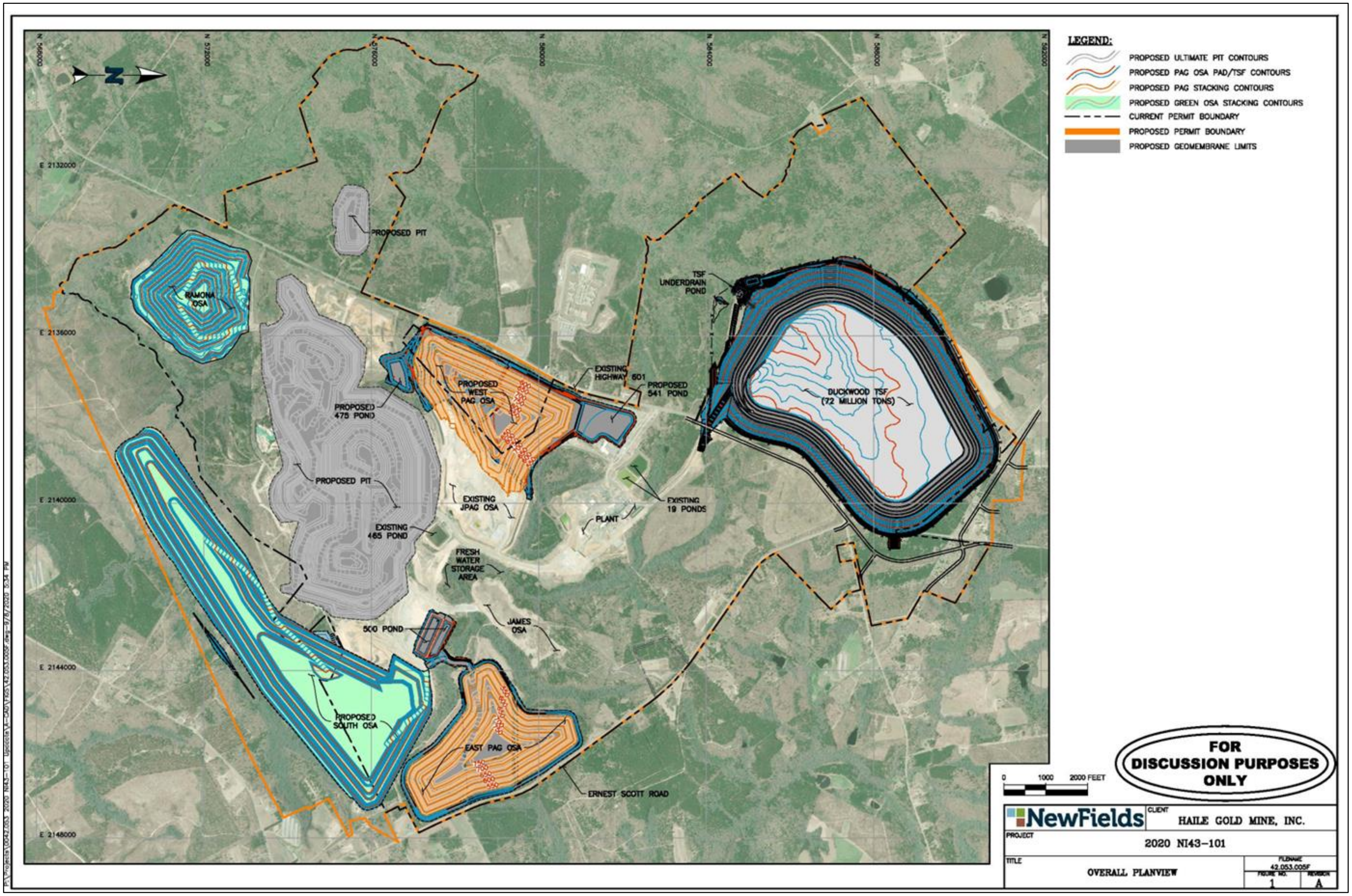
Source: OceanaGold, 2022

Figure 18-2: Tailing Storage Facility Typical Section

18.2 Overburden Storage

During the mine life from the present, six different Overburden Storage Areas (OSAs) and pit backfill will be utilized for the storage of approximately 303 Mt of additional material generated from the open pit and underground development. The material generated from the development of the pits will be classified as either potentially acid generating (PAG) or Non-PAG overburden material. 173 Mt 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 130 Mt 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-PAG 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

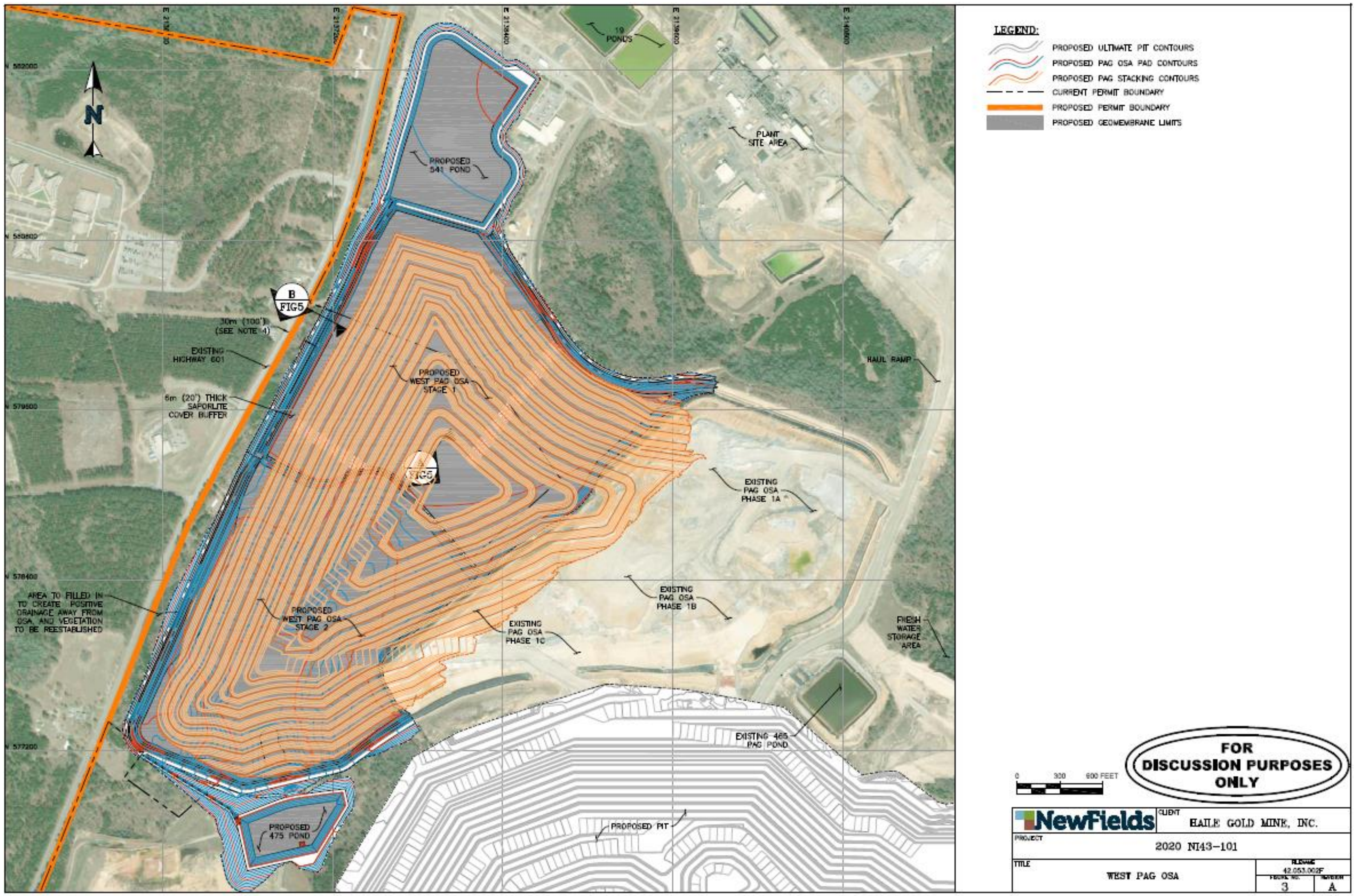
With the current mining permit, Johnny’s PAG (JPAG) OSA, East PAG, West PAG, and pit backfill are designated as the dedicated facilities for storing the PAG material. It is estimated that the current mine plan will generate an additional 173 Mt PAG material of which 73 Mt will need to be stored above ground in a lined OSA facility.

The West PAG OSA, presented in Figure 18-4, has a total capacity of approximately 72 Mt and 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, 541 and 475 Ponds. The recently constructed 541 Pond included in the north expansion has a capacity of 135 million litres (ML), including an extra 37 ML for additional contact water storage for staging to the water treatment plant (WTP) and eliminates the need for the 29 Pond proposed in the SEIS. The remaining 95 ML is sized to contain the predicted runoff from the 100 year/24-hour storm event on the first two phase s (note the first phase was recently constructed and the second phase construction is underway). The south expansion will drain to the 475 Pond that will replace the EIS 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 third phase. When complete, the perimeter runoff collection channels for the full West PAG build out will drain to either the 465 (existing), 541 (recently constructed) or 475 Pond future). 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.

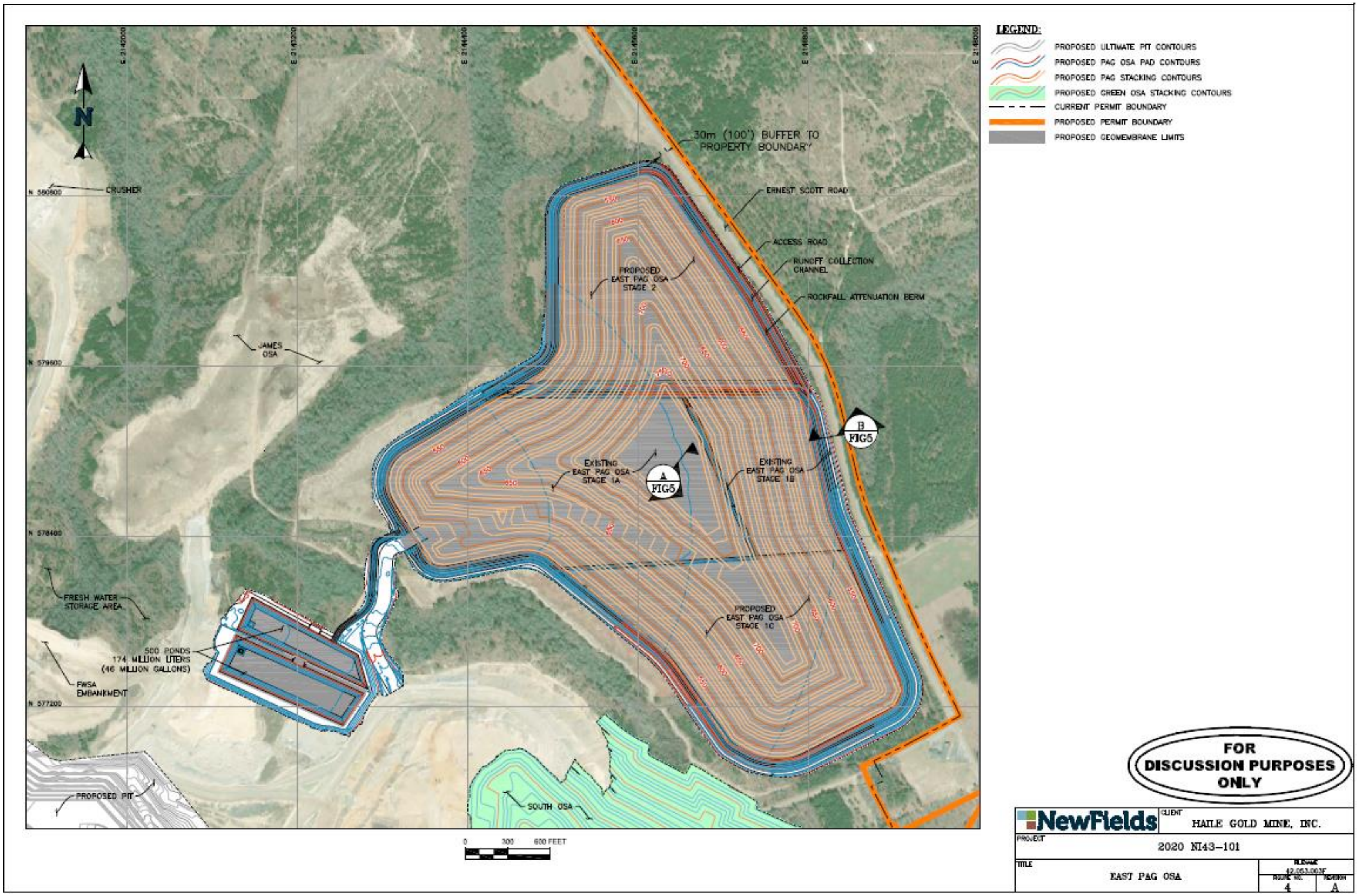
The ultimate footprint of West PAG OSA will have an overall footprint of approximately 126 ha and the PAG material will be loaded with an overall slope of 3(H):1(V).

Figure 18-5 shows the East PAG Overburden Storage Area. The remaining capacity in East PAG (1 Mt), plus the West PAG (72 Mt, and pit backfill ((100 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

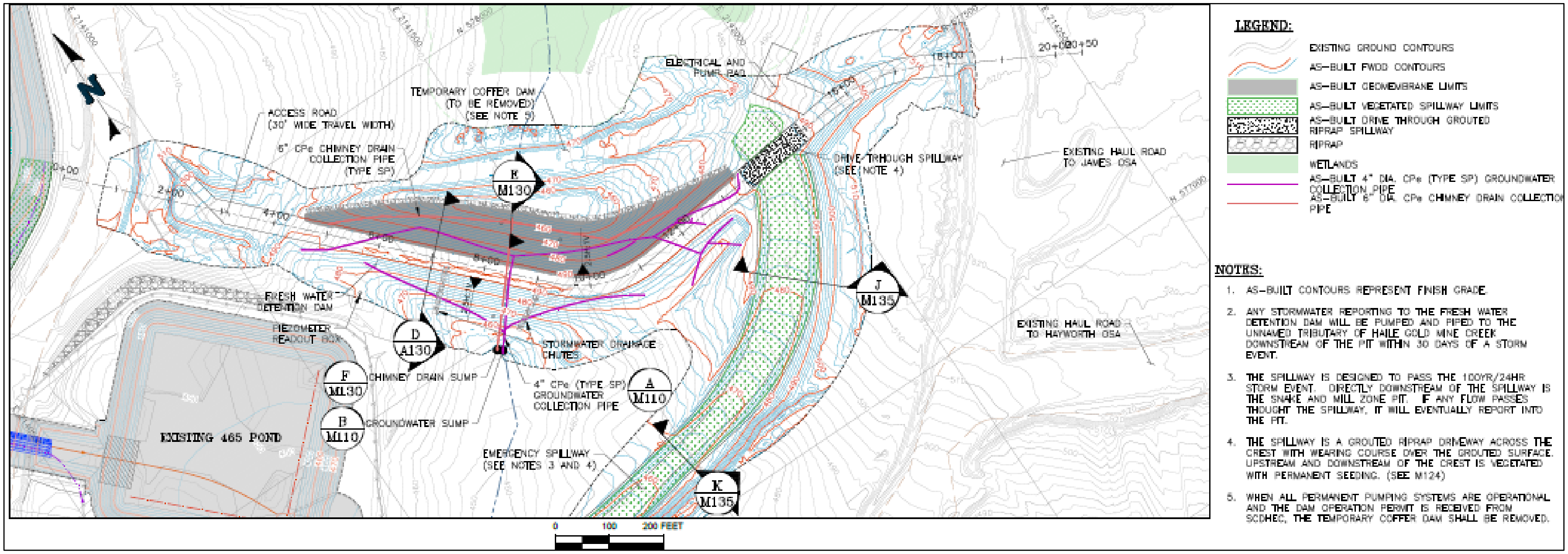


Source: OceanaGold, 2022

Figure 18-5: East PAG Overburden Storage Area Expansion

18.4 Site Wide Water Management

The existing site wide water management plan will remain unchanged. Incoming water from Upper Haile Gold Mine Creek will be detained by the Fresh Water Storage Dam (FWSD) at the upper reaches of the watershed. 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 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), freshwater 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 precipitation 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.

18.6 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. Mill process water will be sourced from TSF reclaim pond and make-up water will be sourced from pit run-in and surface catchment at the FWSD. PAG OSAs can also be used as mill make up water if necessary. The underground operations water supply will be sourced from WTP treated water via an HDPE line to the underground facilities. 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.

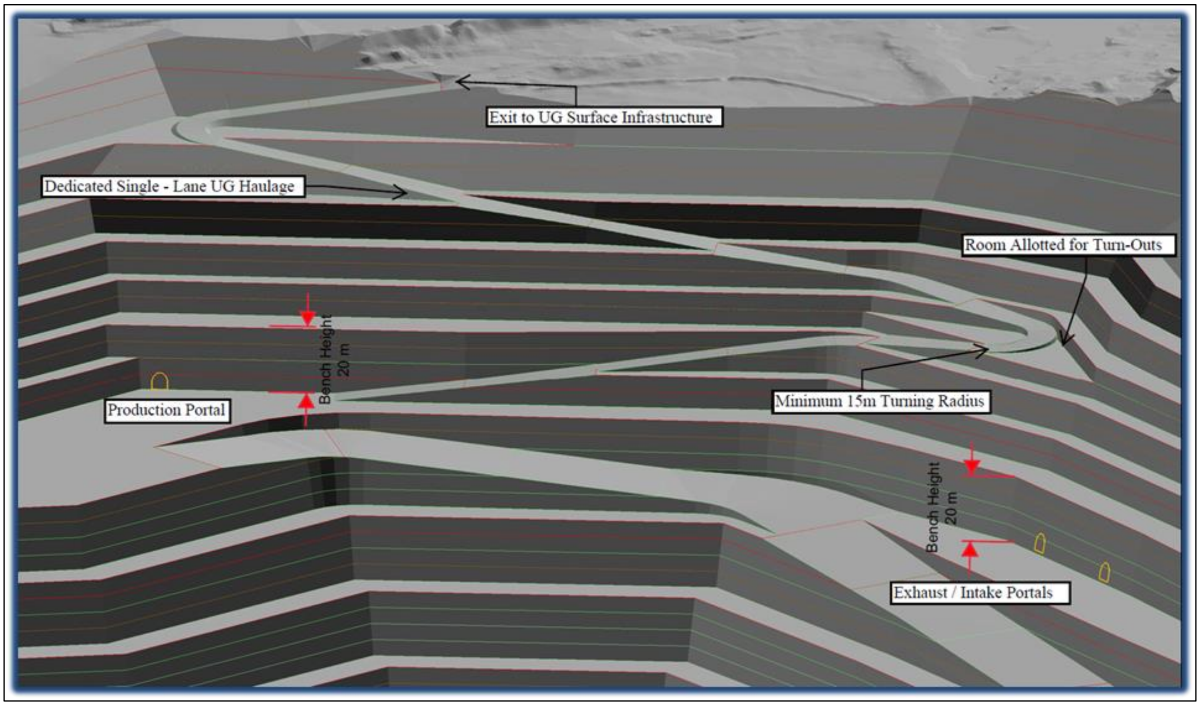
18.7 Surface Roads and Bridges

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.

18.8 Underground Access

The underground is 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. Figure 18-7 shows the underground access route, with the locations of the production portal and intake/exhaust portals annotated. The main fans for Horseshoe are located underground inside the exhaust portal.

Palomino is accessed off of the Horseshoe main decline between the 950 and 925 RL. An additional intake portal or shaft is required to support Palomino mining with location to be confirmed during Feasibility Study work. The location of the exhaust ventilation shaft has been identified to the south of Snake Pit.



Source: OceanaGold, 2023

Figure 18-7: Portal Access (Looking North)

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 offices, warehouse, contact sediment pond, , electrical transformers, and an underground maintenance shop as shown in Figure 18-8. The design has been developed to minimize interaction between the underground mobile equipment and the surface haul road traffic.

Source: OceanaGold, 2023

Figure 18-8: Underground Surface Infrastructure Detail

The ROM Pad has been designated for stockpiling all underground ore and waste rock material. Ore and PAG material will be rehandled into surface trucks and transported to either the processing plant or surface PAG dumps. The facility also has an area allocated for extraction of waste metal from the ore prior to transportation to the processing plant that is both clay and HDPE lined with a 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 near 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 underground operations) 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 has been extended to the underground yard and a 24.9 kV to 13.8 KV transformer installed to serve the CRF plant. Two additional transformers (24.9kV to 480V) provide power to the aboveground facilities on the surface. A separate extension feeds a 6 MVA (24.9 to 13.8 kV) transformer to provide power to the underground operation located on the surface and feeds the underground distribution system.

As part of the Palomino Prefeasibility study further extension of the overhead power line to supply ventilation fans and an underground requirement via borehole cable was examined and capital costs included within the overall estimate for the Palomino study. No further significant upgrades to the site supply were identified beyond the current work in place for the Horseshoe operation.

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 with an Effective Date of January 31, 2020, has an amendment extending the contract to January 30, 2025, 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: Pageland, 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
  - Purchased through a retailer, McCarthy Tires,
- Magotteaux:
  - Grinding Media (all but 5inch which is purchased through Moly Cop on a contract)
  - Registered Address: Franklin, Tennessee
  - End Date of Contract: April 24, 2026
- Roberts Shell:
  - Fuel (off road diesel):
  - Registered Address: Kershaw, South Carolina
  - End Date of Contract: March 17, 2026

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). Due to the 2022 Supplemental Environmental Impact Statement NEPA process, Haile received the permits required for current operations.

On 24 May 2018, Haile applied to the US Army Corp of Engineers (USACE) to initiate the National Environmental Policy Act (NEPA) process and launch a Supplemental Environment Impact Statement (SEIS). USACE has jurisdictional responsibility for all Waters of the United States and works cooperatively with US Environmental Protection Agency (US EPA), and SC DHEC for modifications such as this that have impacts to wetlands, groundwater and surface water conditions and air emissions. Haile 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.

To adjust mine plan expansions, a modified application of the 404 Permit under the Clean Water Act of 1972 (CWA) was submitted in Quarter 4 2020. The final SEIS was published on 19 August 2022. The record of decision and modified 404 permit was received on 12 December 2022. Various permitting approvals/certifications were also required from SC DHEC, including modification of Haile's Mine Operating Permit which was received on 14 December 2022, and 401 Water Quality certification which was received 8 November 2022. Other federal and state agencies included in the review process during the SEIS included: US EPA, United States Fish and Wildlife Service (US FWS), SC DNR, South Carolina State Historic Preservation Office (SC SHPO), South Carolina Department of Transportation (SC DOT) and Catawba Indian Nation. NEPA process also allows non-governmental organizations (NGOs) and other interested parties an opportunity for review and comment on the anticipated impacts.

Since December 14, 2022, SC DHEC has approved two additional modifications to Haile's Mine Operating Permit. An expansion of the Horseshoe underground operation was approved on February 21, 2024, and the Palomino underground operation was approved on March 15, 2024.

Haile is unique in that mining occurs wholly on private land owned by Haile Gold Mine and does not impact federal/public (BLM or USFS) lands that would be subject to projected modifications from these surface management agencies.

This provides financial assurance to the State of South Carolina that funds will be available (in the event of default by Haile Gold Mine) 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.

Table 20-1 is a summary of the current HGM permits as issued under the 2014 EIS, 2022 SEIS, and subsequent state permitting processes.

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.
US Army Corps of Engineers	404 Permit – SAC-1992-24122-4IA	December 19, 2022	Modified 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	November 8, 2022	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	December 14, 2022	Mine Operating Permit – Regulation of closure and reclamation.
SCDHEC, Bureau of Solid and Hazardous Waste Management	Permit No. SCD987596806	4 Apr, 2022	Conditionally exempt very small quantity generator
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 SWPPP General permit	General Permit for Stormwater Discharges for Small and Large mining (Activities Permit) SCR100000	1 July, 2022	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	Wastewater discharges associated with mining activity Permit No. SC0040479	1 June, 2022	Discharge of wastewater in connection with mining activities

Agency	Permit/Authorization Number	Date Received	Description
SCDHEC, Office of Environmental Quality, Bureau of Air Quality	Bureau of Air Quality, StateTitle V No. 1460-0070	1 July, 2021	Authorizes the operation of this facility and the equipment specified herein in accordance with valid construction permits, and the plans, specifications, and other information.
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

All required mining permits have been received.

20.2 Environmental Studies

There are no current environmental studies required at this time.

20.3 Environmental Issues

As required by Haile's Mine Operating Permit, a progressive US$103 million (M) Bond and a US$20 million Reclamation Trust Agreement is in place between HGM and SCDHEC. Currently, US$92.7 million has been paid under the agreed upon schedule. The bond has increased 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.

21 Capital and Operating Costs

21.1 Capital Expenditure Estimates

A summary of the total capital expenditure is provided in Table 21-1; the basis of the capital estimate is discussed below.

Table 21-1: Total Capital Expenditure Summary (US$000's)

Description	Sustaining Capital	Non-Sustaining Capital	Total
OP Mining	542,606	-	542,606
Pre-Strip	408,813	-	408,813
Operational	133,793	-	133,793
UG Mining	30,361	129,212	159,573
Horseshoe	16,659	29,527	46,186
Palomino	13,702	99,685	113,387
Processing	25,828	16,536	42,364
Infrastructure	133,532	-	133,532
Exploration	372	15,523	15,895
Rehabilitation (Non-Sustaining) (1)	-	115,599	115,599
Other	-	4,966	4,966
Total LoM Net Capex	**$732,699**	**$281,836**	**$1,014,535**

Source: OceanaGold, 2024
(1) Captured as Capex in Cashflow

21.1.1 Basis for Capital Expenditure Estimates

The capital estimates throughout this section have a base or effective date of December 31, 2023. 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

Some of the major mining equipment are 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	US$000's	3,079
Principal Payment - Capital Leases: Sustaining	US$000's	32,452
Interest Expense - Capital Leases	US$000's	1,460
Total – Capital Leases	**US$000's**	**$36,991**

Source: OceanaGold, 2024

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.

Table 21-3: Open Pit Mining Capital

Item	Units	LoM Total
OP Capitalized Pre-Strip	US$000's	408,813
OP Mining PP&E	US$000's	125,276
OP Tech Services PP&E	US$000's	892
Pit Dewatering	US$000's	4,900
Site Works	US$000's	2,725
Total	**US$000's**	**$542,606**

Source: OceanaGold, 2023

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	24,467
Interest Expense – Capital Leases	US$000's	1,922
Total – Capital Leases	**US$000's**	**$26,389**

Source: OceanaGold, 2023

Projected capital costs for underground mining are summarized in Table 21-5.

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	Total Cost (US$000's)
Growth Capital Development	52,405
Sustaining Capital Development	15,245
UG Capital Purchases	15,116
Palomino Project	74,737
Palomino Studies	2,070
Total	**$159,573**

Source: OceanaGold, 2023

Process Plant

The Haile process plant has been progressively upgraded from 2017 to 2023 to increase milling rates up to a 3.8 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 LoM totaling US$42.4 million is principally related to:

- Progressive infrastructure upgrades to support tailings dam lifts and the associated impact on tailings pumping.
- 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.
- Other sustaining capital spend to mitigate wear and tear

A progressive tailing pumping study was completed in 2019 to confirm pumping requirements for each stage of the dam raising at the higher 3.8 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$133.5 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 2024 Haile LoM plan. The major infrastructure items, such as the original JPAG and EPAG, tailings storage facility Stage 1, 2, and 3, 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 LoM plan is as shown in Table 21-6 and covers upgrade of Lynches River substation to cover additional sitewide requirements, permitting and exploration drilling.

Table 21-6: Other Capital

Item	Units	LoM Total
Substation Upgrade	US$000’s	3,950
Land Acquisitions	US$000’s	0
Permitting	US$000’s	1,016
On-Site Exploration Drilling	US$000’s	15,895
Total	**$000’s**	**$20,861**

Source: OceanaGold, 2023

21.2 Operating Cost Estimates

The total RoM operating cost unit rate of US$45.50/t processed is summarized in Table 21-7.

Table 21-7: RoM Operating Cost Summary

Description	US$000’s	US$/t mined
OP Mining (US$/t rock mined)	1,070,195	3.36
Capitalized Pre-Strip	(408,813)	
OP Mining – Operational Material	**661,382**	
UG Mining (US$/t rock mined)	533,669	54.28
Capitalized Development	(67,650)	
UG Mining – Operational Material	**466,019**	
Description	**US$000’s**	**US$/t Ore Processed**
Subtotal Mining (Operational Material Only)	**1,127,402**	**25.47**
Processing	668,051	15.09
G&A Cost	242,173	5.47
Refining/Freight Costs	7,385	0.17
Total Operating Costs	**$2,045,011**	**$46.21**

Source: OceanaGold, 2024

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. There is a US$12.2 million of bond commitment remaining on the US$20 million obligation, which will be returned at the end of LOM, hence the reversal.

Table 21-8: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond (Expenditure Only)	12,200	0.28
Environmental Bond Release	(20,000)	(0.45)
Interest Expense - Capital Leases	3,382	0.08
Principal Payment - Capital Leases: Sustaining	32,452	0.73
Principal Payment - Capital Leases: Non-Sustaining	27,546	0.62
Total Non-Operating Costs	**$55,579**	**$1.26**

Source: OceanaGold, 2024

21.2.1 Basis for Operating Cost Estimates

The operating cost estimates throughout this section has a base or effective date of December 31, 2023. 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.

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.

The average mining cost over the life of the mine is US$3.36 per tonne of material mined. The cost by activity is presented in Table 21-9.

Table 21-9: Open Pit Mining Cost Summary

Description	US$/t mined (1)	Total US$000's
Drill and Blast	0.75	238,917
Load and Haul	1.11	353,381
Ancillary	0.36	114,322
Maintenance Management	0.52	164,763
Operations Management	0.12	37,110
Dewatering	0.12	37,425
Technical Services	0.38	124,277
Total	**$3.36**	**$1,070,195**

Source: OceanaGold, 2024
(1) Ore + Waste

Underground

Projected operating costs for underground mining have been developed based on the LoM production schedule. The average cost of underground mining is US$54.28/t and the cost by activity is presented in Table 21-10.

Table 21-10: Underground Mining Cost Summary

Description	US$/t mined	Total US$000's
Diamond Drilling - Res Def	0.51	5,040
Labor	25.26	248,360
Backfill	0.29	2,821
Fuel	1.95	19,158
Power	2.15	21,155
Equipment Operating	3.13	30,790
Equipment Maintenance	8.39	82,441
Explosives	1.79	17,623
Ground Support	2.68	26,394
Grout/Shotcrete/Cement	4.25	41,817
Services	2.75	27,079
Contracts	1.12	10,991
Total	**$54.28**	**$533,669**

Unit costs are based on ore and waste tonnes.
Source: OceanaGold, 2023

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 LoM 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	US$/t milled	Total US$000's
Power	1.81	80,086
Labor	3.32	147,056
Grinding Media	1.51	66,858
Reagents	4.15	183,499
Mill Liners	0.60	26,567
Maintenance Materials	1.66	73,613
Miscellaneous	1.71	75,581
Assay	0.33	14,791
Total	**$15.09**	**$668,051**

Source: OceanaGold, 2024

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$7.4 million over the LoM or US$0.17/t processed.

Gold

- 99.996% payable Au
- US$0.60/troy oz Au treatment charge
- US$0.43/troy oz Au shipment cost

Silver

- 99.0% payable Ag
- US$0.60/troy oz Ag treatment charge
- US$0.43/troy oz Ag shipment costs

General and Administration

General and administration (G&A) costs are using Haile 2023 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.

The total G&A will increase slightly to reflect support for UG expansion, then 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 LoM total is US$242 million, or US$5.47/t processed.

Table 21-12: General and Administration Cost Summary (US$000's)

Description	2023	2024 to 2030 p.a. OP + UG
General Administration	10,127	9,467
Commercial	6,544	4,538
Human Resources	1,487	1,145
Information Technology	750	919
Health, Safety and Environmental	7,094	7,391
Supply Chain	2,358	2,955
Total	**28,359**	**26,415**

Source: OceanaGold, 2024

Indirect Costs

Indirect costs are estimated to be US$76 million for the LoM or US$1.71/t processed. These costs have been sourced from the 2023 actual expenditures. Indirect costs include environmental bond 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	12,200
Environmental Bond Release	(20,000)
Principal/Interest Payment – Capital Leases	63,380
Total	**$55,579**

Source: OceanaGold, 2024

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 2023 and 2024. 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 2023 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, 2024
OP Operations Start	January 1, 2024
UG Operations Start	January1, 2024
OP Mine Life	11 years
UG Mine Life	11 years
Mill Operations	12 years
Discount Rate	5%

Source: SRK, 2024

All costs incurred prior to January 2024 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

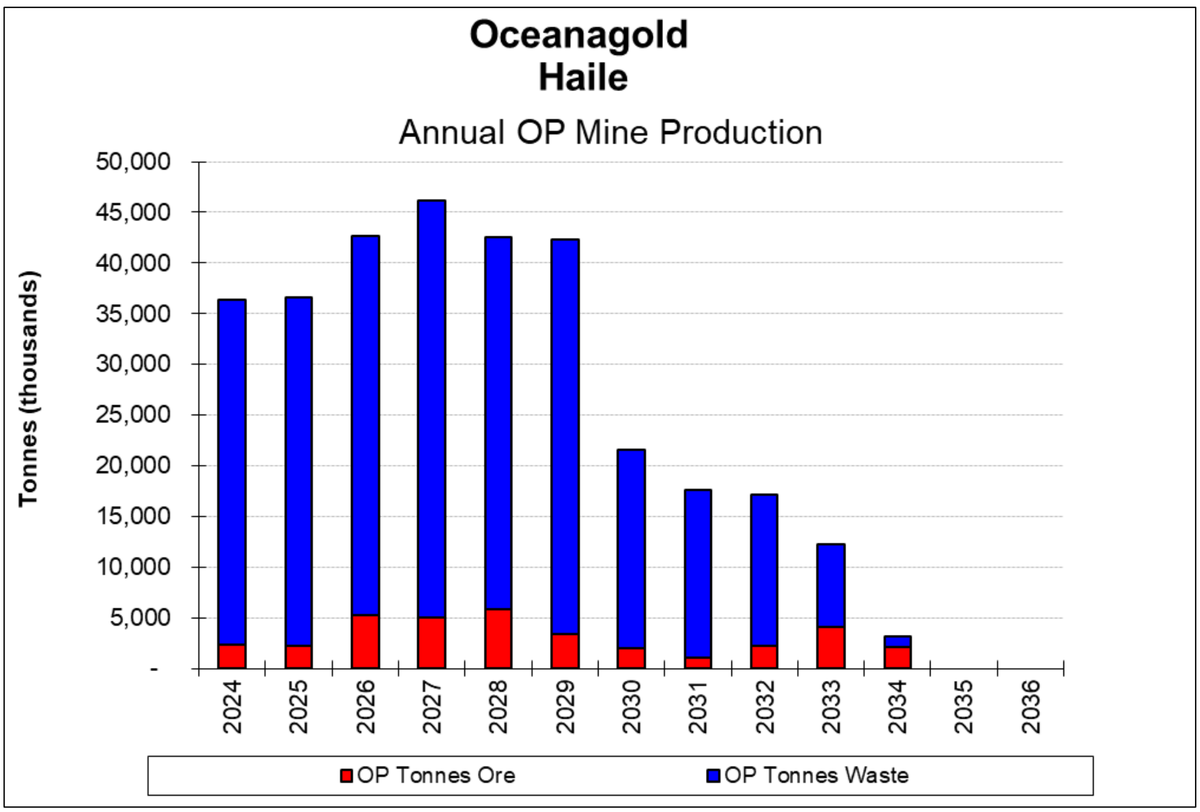
Table 22-2 is a summary of the estimated production over a 12-year mine life for the combined open pit/underground operations. Ore mined refers to Proven and Probable Mineral Reserves. Figure 22-1 and Figure 22-2 show LoM production by year for OP and UG operations.

Lower forecast ore production rates in 2024 and 2025 results from the completion of Ledbetter 2A in 1Q 2025 and ramp up of Ledbetter Phase 3

Table 22-2: Life-of-Mine Production Summary

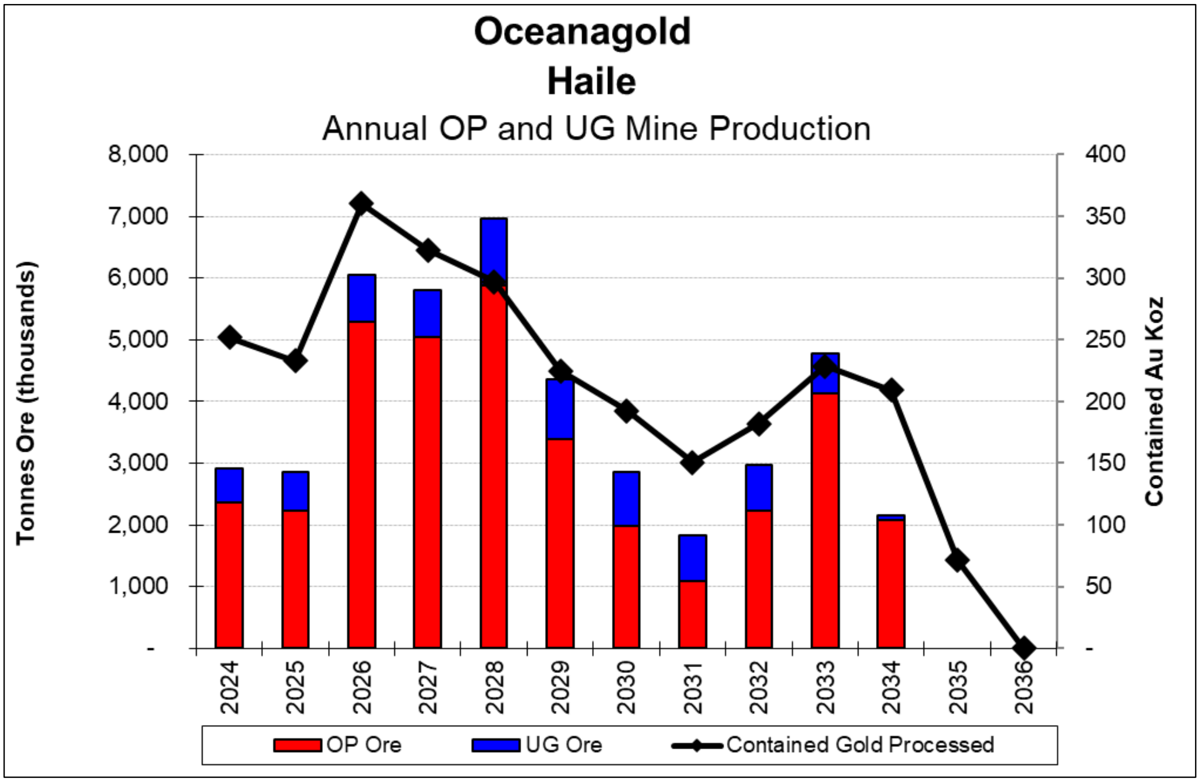
Description	Units	Value
OP Ore Mined	kt	35,700
OP Waste Mined	kt	282,625
OP Total Material Mined	**kt**	**318,325**
OP Mined Gold Grade	g/t	1.56
OP Mined Silver Grade	g/t	2.43
OP Contained Gold	koz	1,792
OP Contained Silver	koz	2,785
UG Ore Mined	kt	7,859
UG Waste Mined	kt	1,972
UG Mined Gold Grade	g/t	3.56
UG Mined Silver Grade	g/t	2.18
UG Contained Gold	koz	900
UG Contained Silver	koz	551
Total Ore Mined	**kt**	**43,559**
Waste Mined	kt	284,597
Total Material Mined	**kt**	**328,156**
Mined Gold Grade	g/t	1.92
Mined Silver Grade	g/t	2.38
Total Contained Gold	**koz**	**2,692**
Total Contained Silver	**koz**	**3,335**

Source: SRK, 2024



Source: SRK, 2024

Figure 22-1: Annual Open Pit Mine Production



Source: SRK, 2024

Figure 22-2: Annual Open Pit and Underground Ore Production

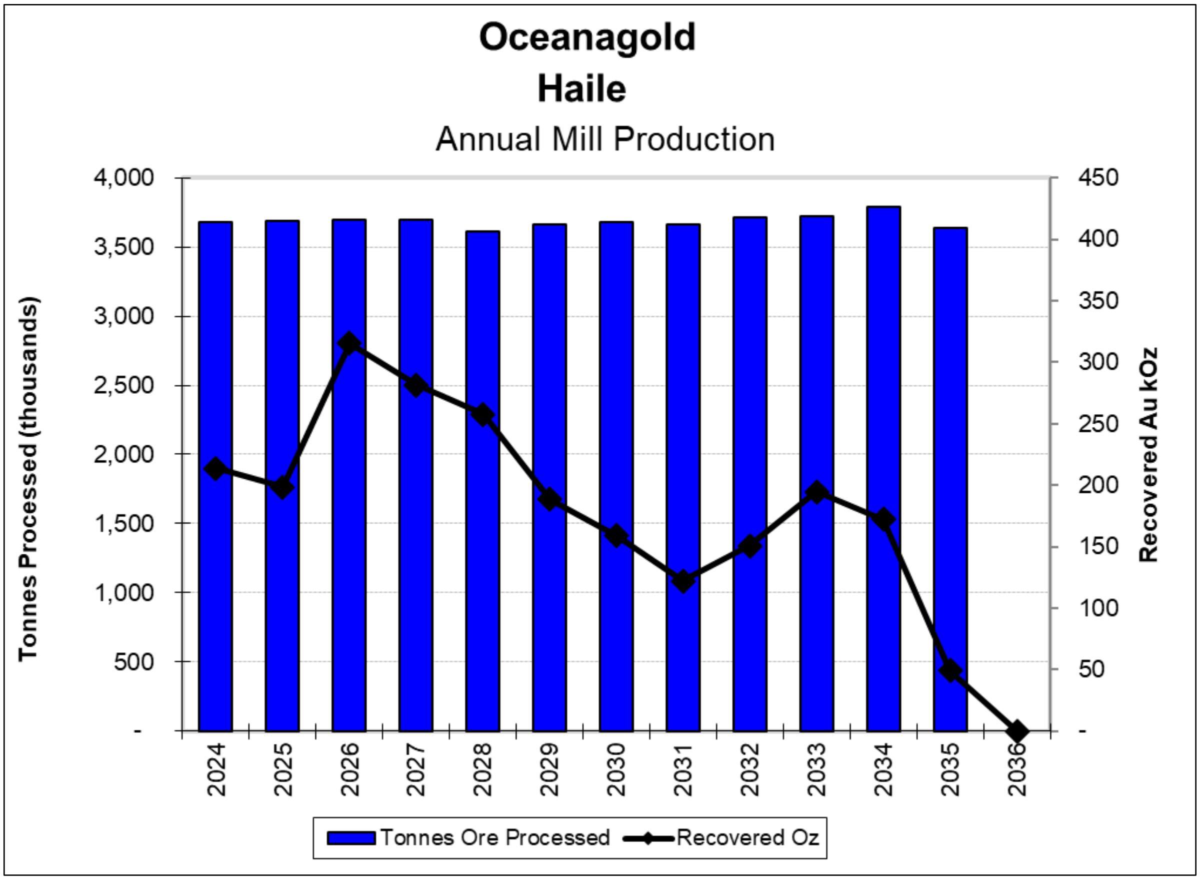
22.2.2 Mill Production

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. Note that differences in mined ore and ounces compared to processed ore and ounces are due to stockpile strategies and management.

Table 22-3: Life-of-Mine Process Production Summary

Description	Units	Value
Total Ore Processed	**kt**	**44,257**
Daily Process Capacity	t/d	10,528
Processed Grade	g/t	1.91
Contained Gold	koz	2,724
Contained Silver	koz	3,272
Gold Recovery	%	84.5
Silver Recovery	%	70.0
RoM Recovered Gold	koz	2,303
RoM Recovered Silver	koz	2,290

Source: SRK, 2024



Source: SRK, 2024

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$0.60/troy oz Au treatment charge
- US$0.43/troy oz transportation cost.

Silver is also included in the current 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$0.60/troy oz Ag treatment charge
- US$0.43/troy oz transportation cost

The silver by-product credit of US$41 million over LoM represents 1.2% 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$46.21/t processed is summarized in Table 22-4.

Table 22-4: RoM Operating Cost Summary

Description	US$000's	US$/t Mined
OP Mining ($/t rock mined (ore and waste)) - All Material	1,070,19	3.36
OP Mining ($/t rock mined (ore and waste)) - (excl. capitalized cost)	661,382	2.08
UG Mining ($/t rock mined (ore and waste)) – All Material	533,669	54.28
UG Mining ($/t rock mined (ore and waste)) - (excl. capitalized cost)	466,019	47.40
	US$000's	**US$/t Ore Processed**
Subtotal Mining (Operational Material Only)	1,127,402	25.47
Processing	668,051	15.09
G&A Cost	242,173	5.47
Refining/Freight Costs	7,385	0.17
Total Operating Costs	**$2,045,011**	**$46.21**

Source: SRK, 2024

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. Note that the negative cost presented in the environmental bond item is the result of a return of cash at the end of mine life.

Table 22-5: RoM Indirect Costs Summary

Description	US$000's	US$/t Ore Processed
Environmental Bond	(7,800)	(0.18)
Principal and Interest - Capital Leases: Sustaining	34,281	0.77
Principal and interest - Capital Leases: Non-Sustaining	29,098	0.66
Total Non-Operating Costs	**55,579**	**1.26**

Source: SRK, 2024

Total LoM capital costs totaling US$970 million including rehabilitation 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$733 million over LoM.

Table 22-6: Life-of-Mine Capital Costs (US$000's)

Description	Sustaining Capital	Non-Sustaining Capital	Total
Operations Information Technology	1,370	-	1,370
General Operations Expenditure	160,837		160,837
Brownfields Exploration	372	-	372
Operations Based Mining Projects	146,063	-	146,063
Open Pit Mine	408,813	-	408,813
Growth Information Technology	15,245	-	15,245
General Growth Expenditure	-	14,602	14,602
Greenfields Exploration	-	15,523	15,523
Stand-alone Mining Projects	-	83,708	83,708
Rehabilitation (Non-Sustaining)[1]	-	115,599	115,599
Underground Mine	-	52,405	52,405
Total LoM Net Capex	**$732,699**	**$281,836**	**$1,014,535**

Source: SRK, 2024
(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 utilizing mid-period discounting, the Project returns an after-tax NPV of US$256 million.

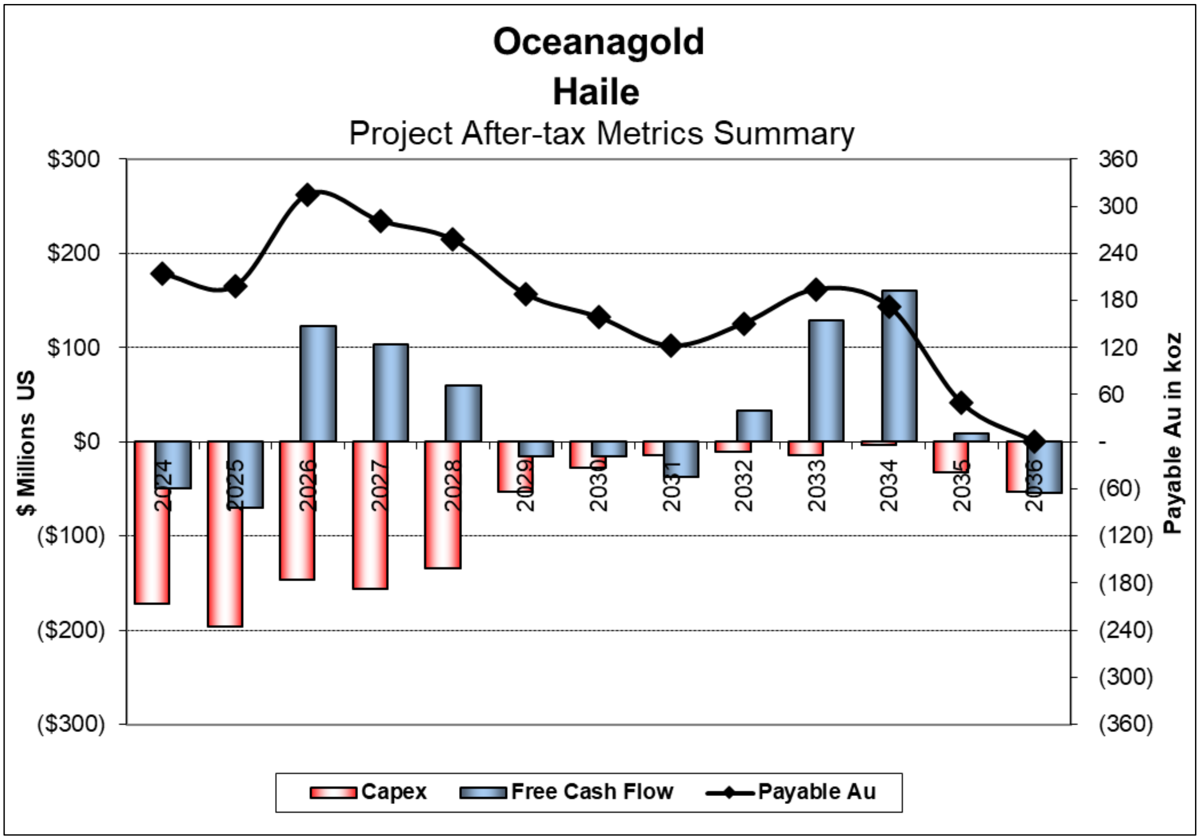
Note that because the Project is operating and is valued on a total project basis with prior capital treated as sunk, and not by an incremental analysis of the UG and mill expansions, an IRR value is not relevant in this analysis.

Figure 22-4 presents annual cash flow metrics vs. recovered gold production and shows that the Project does not generate positive free cash flow in 2024 and 2025 due to the level of capital expenditure and lower grade material at that time and a period from 2029 to 2031 due lower grades through the mill and lower forecast recoveries. Cash flow is forecast to be negative at the end of the operational life due to closure costs.

Table 22-7: Indicative Economic Results

Description	US$000's
Market Prices	
Gold (US$/oz)	1,500
Payable Gold (koz)	2,303
Revenue	
Gross Gold Revenue	3,454,270
Silver By-Product Credit (at US$18 / oz Ag)	40,810
Total Gross Revenue	**$3,495,080**
Operating Costs	
OP Mining	(661,382)
UG Mining	(466,019)
Processing	(668,051)
Site G&A	(242,173)
Selling/Refining	(7,385)
Non-Operating Costs	(55,579)
Total Operating Costs	**($2,100,591)**
Operating Margin (EBITDA)	1,394,490
Taxes	
Income Tax	(4,699)
Total Taxes	**($4,699)**
Working Capital	-
Operating Cash Flow	1,389,790
Capital	
Sustaining Capital	(732,699)
Non-Sustaining Capital	(281,836)
Total Capital	**($1,014,535)**
Metrics	
Pre-Tax Free Cash Flow	379,954
After-Tax Free Cash Flow	375,255
Pre-Tax NPV at 5%	260,540
After-Tax NPV at 5%	256,340

Source: SRK, 2024



Source: SRK, 2024

Figure 22-4: Project After-Tax Metrics Summary

Early in the operational life, negative cashflow years are incurred due to high capital spend. In 2029 to 2031, the mill processes lower grade material with a lower recovery resulting in negative cashflow at the US$1,500/oz Au reserve price. Overall, the operation generates positive LoM cashflow.

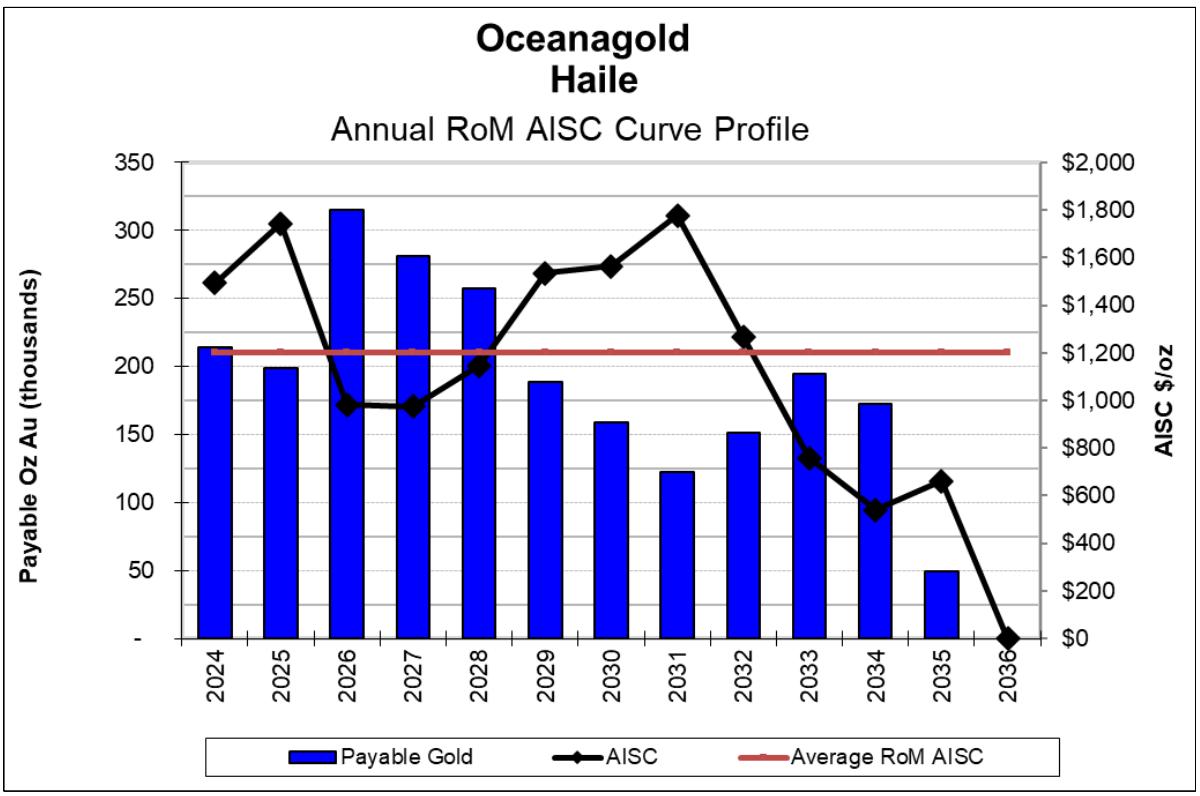
Table 22-8 shows the build-up of a RoM AISC of US$1,200/oz Au, net of a US$18/oz silver by-product credit, over the 12-year life of the Project.

Table 22-8: RoM AISC Contribution

Total RoM Payable Gold Sales in koz		2,303
Description	**US$000's**	**US$/oz**
OP Mining	661,382	287
UG Mining	466,019	202
Processing	668,051	290
Site G&A	242,173	105
Selling/Refining/Freight	7,385	3
Direct Cash Costs Before By-Product Credit	$2,045,011	$888
Silver By-Product Credit	(40,810)	(18)
Direct Cash Costs After By-Product Credit	$2,004,201	$870
Capital Leases: Sustaining	34,281	15
Environmental Bond	(7,800)	(3)
Sustaining Capex	732,699	318
Total RoM AISC	**$2,763,381**	**$1,200**

Source: SRK, 2024

Figure 22-5 shows the annual RoM AISC trend during the mine operations against an overall average RoM AISC of US$1,2009/payable oz over the 12-year LoM at an annual average production rate of 192 koz Au per year during operation. The AISC variations are mainly driven by annual gold production levels and fluctuating capital spend and can range from US$541 to US$1,777 per oz in a given year.



Source: SRK, 2024

Figure 22-5: Annual AISC Curve Profile

22.3 Taxes, Royalties and Other Interests

As the Project is currently in operation, a comprehensive depreciation analysis for the remainder of the mine life was provided by OceanaGold for incorporation into the TEM. The main taxation assumptions utilized within the model are as follows:

- Corporate Income Tax (CIT) rates are 21% for Federal and 5% for South Carolina
- Existing Net Operating Loss (NOL) pools are not considered
- Federal Depletion allowance is estimated for each period by applying a depletion rate of US$196/oz provided by Oceana Gold
- A Tax Depreciation allowance schedule was provided by OceanaGold. The tax depreciation schedule provided yields US$1,644 million of depreciation from the whole year 2024 through LoM:
    - The total tax deductions calculated for the operation from 2024 through the LoM are presented in Table 22-9

Table 22-9: Haile Tax Deductions (US$000's)

Tax Deductions	LoM US$(000's)
Depreciation	1,644,459
Depletion	451,050
Total Tax Deductions	**$2,095,510**

Source: OceanaGold, 2024

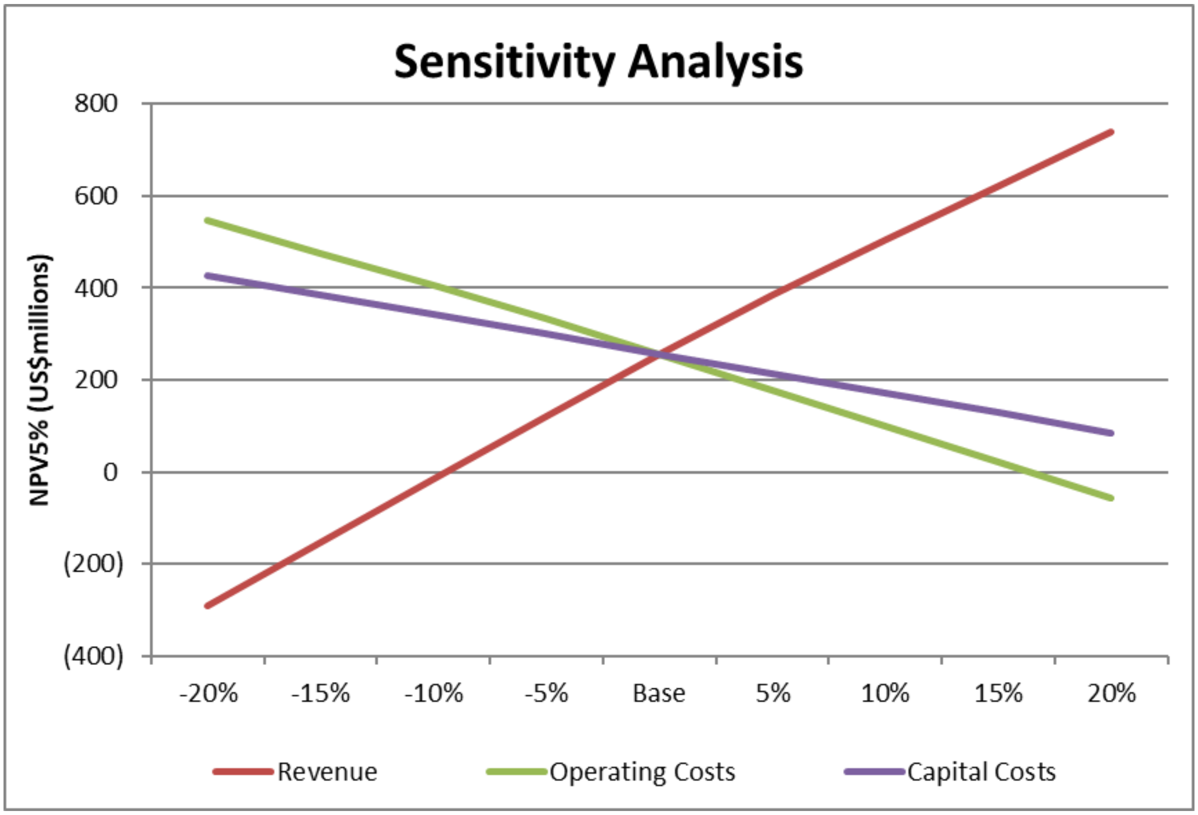
As a result of these deductions, the operation, as modeled is expected to pay minimal income tax over the life of the operation. As this project is being modeled in isolation at fixed prices and costs, actual results may differ.

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 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.



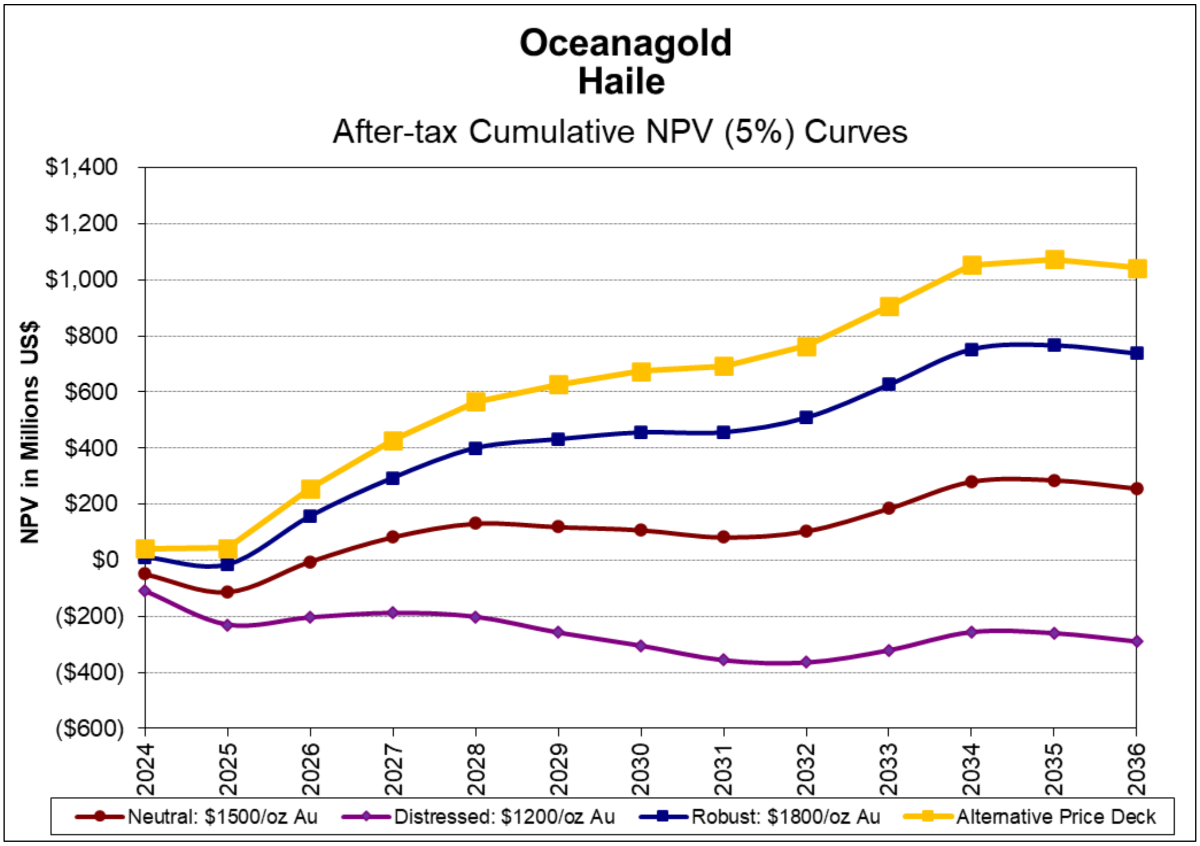
Source: SRK, 2024

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$1,800/oz and constant “Distressed” price of US$1,200/oz, and a price deck provided by Oceana Gold that is more in line with the current price environment (Alternative Price Profile), which is a flat price deck at US$2,000/oz Au and US$24/oz Ag.

Figure 22-7 shows the gold price sensitivity analysis.

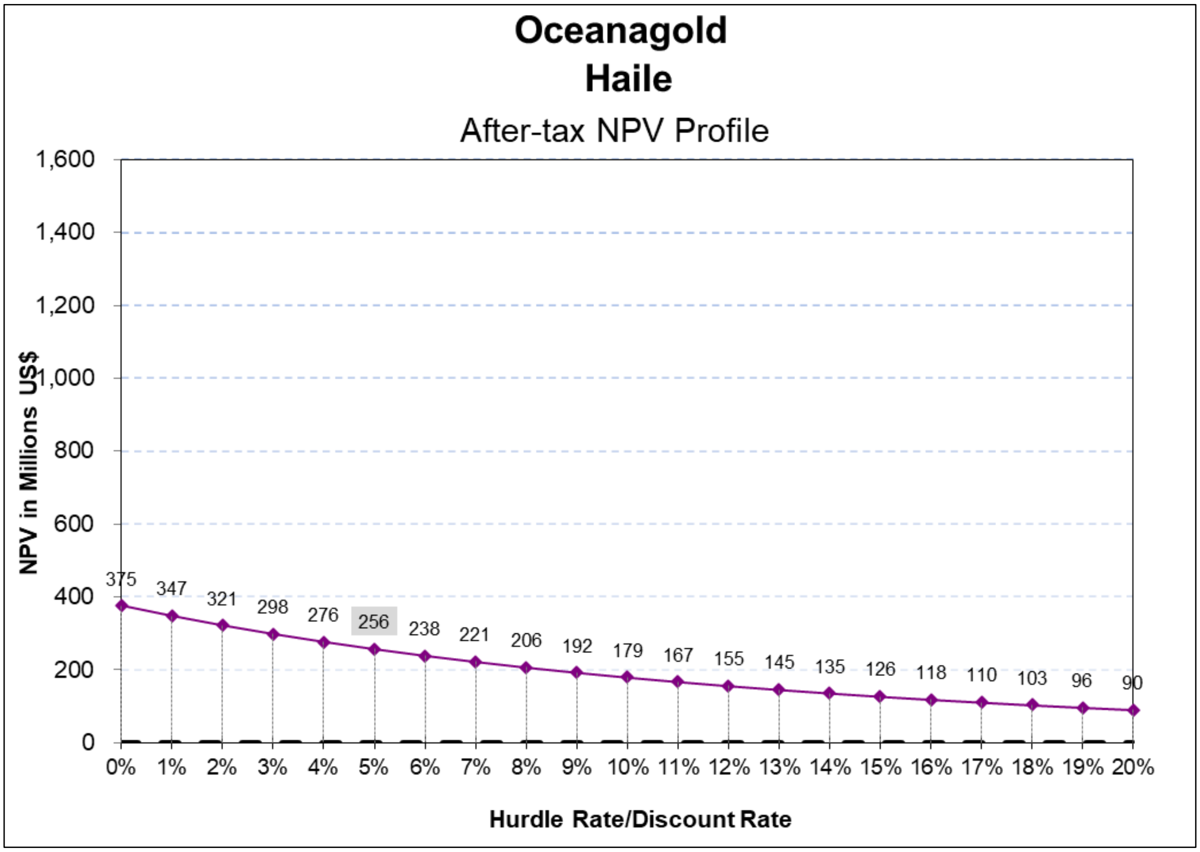


Source: SRK, 2024

Figure 22-7: Gold Price Sensitivity Analysis

22.4.3 Discount Rate Sensitivity

A sensitivity analysis of discount rates presented in Figure 22-8 shows that the Project as currently modeled would be NPV positive through a 20% discount rate.



Source: SRK, 2024

Figure 22-8: Discount Rate Sensitivity Analysis

22.5 OceanaGold 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 22-10 at the Alternative price profile. This price profile remains 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 the reserve case price and higher than the Alternative price profile.

Table 22-10: Indicative Economic Results at Ocean Gold Price Profile

Description	US$000's
Market Prices	
Gold (US$/oz)	$2,000
Payable Gold (koz)	2,303
Revenue	
Gross Gold Revenue	4,605,693
Silver By-Product Credit (at ~ US$23 / oz Ag)	54,414
Total Gross Revenue	**$4,660,107**
Operating Costs	
OP Mining	(661,382)
UG Mining	(466,019)
Processing	(668,051)
Site G&A	(242,173)
Selling/Refining	(7,385)
Non-Operating Costs	(55,579)
Total Operating Costs	**($2,100,591)**
Operating Margin (EBITDA)	2,559,516
Taxes	
Income Tax	(153,526)
Total Taxes	**($153,526)**
Working Capital	-
Operating Cash Flow	**$2,405,990**
Capital	
Sustaining Capital	(732,699)
Non-Sustaining Capital	(281,836)
Total Capital	**($1,014,535)**
Metrics	
Pre-Tax Free Cash Flow	1,544,981
After-Tax Free Cash Flow	1,391,455
Pre-Tax NPV at 5%	1,177,167
After-Tax NPV at 5%	1,043,576

Source: SRK, 2024

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.

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 3,666 drillholes including 1,473 core holes for 466,907 m (60% m) and 2,193 reverse-circulation (RC) holes for 307,372 m (40% 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

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. The deposits occur within a variably deformed ENE-trending structural zone at or near the contact between metamorphosed Neoproterozoic volcanic and sedimentary rocks. The deposits are hosted in laminated siltstones and volcanic rocks of the Upper Persimmon Fork Formation and are 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. The age of gold mineralization is assumed at ~549 Ma, based on closely associated molybdenite dated using Re-Os (Mobley et al., 2014), which postdates peak volcanism. Pressure shadows around pyrite grains, stretched pyrite and pyrrhotite grains, and flattened hydrothermal breccia clasts indicate that some deformation has occurred subsequent to sulfide mineralization. The bulk geometry and orientation of the deposits implies emplacement subsequent to folding. The Re-Os date 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 structurally controlled, low-sulfidation, disseminated gold deposit.

25.2 Resource Estimation

OceanaGold has in place a number of resource model governance processes, including model validation, peer review, production reconciliation as well as coaching and team-based training. Where appropriate, independent model estimates are completed and compared. Where appropriate, external model reviews are undertaken.

OceanaGold continue to develop and improve these processes.

25.2.1 Open Pit

The drillhole database and resource estimation methodology are appropriate for the purposes of estimating the open pit gold resources. In addition to QAQC processes, 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, suggest that the annual reconciliation variance previously experienced will remain a feature of the resource estimates.

The commencement of underground mining at Horseshoe means that there will be three confluent mill feed sources (open pit, underground and stockpiles - putting aside individual pit stages or underground development areas). Short-term (1 to 2 months) multi-source mine to mill reconciliation may be a challenge. However, long term performance is expected to be acceptable.

OceanaGold will continue to optimize gold estimation processes, 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 will continue to guide exploration over the next five years, particularly in the search 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 continued growth of the Horseshoe underground reserve (0.52 Moz) in 2023 and announcement of an initial reserve at Palomino in 2024 (0.38 Moz).

In-house core drilling continues by OceanaGold focused on high-grade underground targets proximal to the sedimentary-volcanic contact. Underground development at the Horseshoe deposit in 2023 has provided access for underground drill stations, and as development continues, will extend access 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 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

Geotechnical field characterization programs have been undertaken to assess the expected rock quality at the underground targets. These programs included geotechnical core logging, laboratory strength testing, in situ stress measurements and oriented core logging of discontinuities. The results of these programs have provided adequate quantity and quality of data for feasibility-level design of Horseshoe and prefeasibility-level design of Palomino.

Geotechnical assessments of the Horseshoe and Palomino orebody shapes and ground conditions has determined that longhole open stoping mining is an appropriate mining method. The design has been laid out using empirical design methods based on similar case histories. 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.

Mining

Longhole stoping is seen as the appropriate mining method for the Horseshoe and Palomino deposit geometries. The large stope sizes minimize cost and grades are not overly diluted. Mine planning work considered revenue for Au and an elevated CoG of 1.74 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 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 7 years and has been progressively upgraded from its original nameplate capacity of 2.3 Mtpa and planned to treat up to3.8 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.

Ongoing future ore test programs are conducted on material in the LoM plan and on any new proposed resources. The laboratory flowsheet has been effective at predicting the recovery performance of the sulfide ore sources tested and treated in the plant.

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 dry/meeting building, underground yard, an upgrade and addition to the power distribution system, water supply system, maintenance and warehouse facilities, and installation of a portable crusher/screen plant and CRF plant at the underground yard.

Palomino development will require additions to the power distribution, air and water supplies. Establishment of an exhaust shaft is required to provide necessary airflow requirements for the Palomino deposit.

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.

In 1Q 2024, South Carolina Department of Health and Environmental Control approved two modifications to Haile's Mine Operating Permit. An expansion of the Horseshoe underground operation was approved on February 21, 2024, and the Palomino underground operation was approved on March 15, 2024.

Permits currently held by the HGM may be kept, modified, terminated, or replaced during the mining process.

25.9 Economic Analysis

The Project consists of an operating surface and underground mine with a mill. The milling facility is mainly fed by the OP mine. The mill feed is supplemented with ore from a UG 1.1 Mtpa max annual capacity operation.

The Project is expected to produce 2.3 Moz of payable gold over a 12-year mine life at an average rate of 192 koz Au per year during full production years with a LoM AISC of US$1,1200/oz Au.

The Project is expected to incur sustaining capital in the amount of US$732.7 million over the modeled life and a non-sustaining capital spend, including rehabilitation costs, of US$281.8 million for total capital expenditure of US$1,014.5 million.

Project metrics using a constant US$1,500/oz gold price include pre-tax and after-tax NPV 5% values of US$256 million. 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, 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 followed 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 will continue to guide exploration over the next five years, particularly in the search for underground deposits.

Haile 3D geologic models continue to be integrated with metallurgical data to facilitate geometallurgical modeling. Continue using portable XRF testing or other technologies to further refine the geology interpretation (in tandem with in-pit studies).

26.1.2 Resource Estimation

As underground mining ramps up at Horseshoe, there will 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. OceanaGold will monitor and improve systems to track feed source performance (for example, regular stope cavity lidar surveys, open pit and underground track factor determination).

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 resource block model. This will improve the integration of 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 have provided some insight into this. The sensitivity modeling study needs to be leveraged to quantify and manage 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

A rolling front of pre-production infill drilling at approximately 15 m x 15 m spacing will be maintained 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 (penetration rate may be a challenge)
- Close spaced, short, ore-drive parallel diamond drilling has been chosen as the preferred option for grade control drilling.

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 vs. Underground Trade-off Study – Ledbetter Phase 4

An opportunity has been identified to potentially increase project cashflow by mining the mineralization within the current Ledbetter 4 open-pit phase by underground methods instead. A pre-feasibility study is required on the underground option to confirm the most appropriate option with a view to adjusting the Mineral Reserve estimate. This is underway and is expected to be completed by March 2025. This will include all elements required for an underground PFS similar to that reported for the Palomino Underground in this report, such as:

- Geotechnical data program and analysis
- Mineral Resource estimate update
- Underground design and scheduling
- Infrastructure design
- Operating and Capital cost estimate updates
- Cashflow analysis

Site-wide Schedule Optimization

The timing for the commencement of the Palomino Underground is influenced by the timing of availability of higher grade ore from the open-pit, given that underground ore offsets open-pit production.

There is an opportunity to improve project NPV by running a schedule optimization process that will identify the ideal timing for the commencement of Palomino, and also identify the open-pit production rate that balances maximum stockpile balance against presentation of grade to the processing plant.

The schedule optimization project will be completed in parallel and as an input to the Ledbetter Open-pit vs. Underground Trade-off Study and the Palomino Underground Feasibility Study.

Open Pit

An internal continuous improvement program and resources are in place. Progress existing initiatives and identify additional initiatives focusing on drill / blast practices, equipment performance, technician and operator training, and other cost reduction opportunities.

Continuous improvement initiatives currently being actioned are:

- Increase Mining Capacity in Load and Haul
- Uplift Fleet Availability
- Dispatch and Radio Network Improvements
- Build and deploy a maintenance parts tracking process
- Work order process optimization
- Improve drill plan management
- Implement Geometallurgical model

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:

- Grade control drilling to delineate ore/waste and guide material destination
- Investigate UG RC drilling
- Completion of Palomino through Feasibility Study for further de-risking and evaluation of mine design, assumptions, and cost estimations

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 for the Palomino deposit.

A structured geomet program should continue to focus on understanding expected ore competency to allow improved scheduling and blending to maximize 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. The remaining requirements are overburden storage facilities and the costs for these are included as sustaining capital.

Underground Infrastructure

Although the base case design considers cemented 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 that supported the SEIS 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.
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

Study	Scope of Work
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
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 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 not included in the cost schedules presented in Section 21.

Table 26-2: Summary of Costs for Recommended Work

Discipline	Program Description	Cost (US$)	No Further Work is Recommended Reason:
Geology and Mineralization	External reviews	30,000	
Status of Exploration, Development and Operations	OceanaGold exploration programs and development drilling. External assay laboratories used.	1,500,000	
Mineral Processing and Metallurgical Testing	External laboratory testing – future ore sources	400,000	
pXRF analyses to augment geological interpretation	Collect more data and analyze		Salaries only
Mineral Resource Estimate – Open Pit	Develop Geometallurgical model		Salaries only
Mining	Ledbetter Open-pit vs. Underground Trade-off Study (Pre-feasibility Study)	1,000,000	
Mining	Palomino Underground Feasibility Study	2,000,000	
Mining	Site-wide mine schedule optimization	100,000	
Mining	General improvement initiatives		Salaries only
Environmental Studies	Additional studies and legal support	2,100,000	
Total US$		**$7,130,000**	

Source: OceanaGold 2024

27 References

Adams, C. Konopa, S., 2024, November 2023 Palomino Resource Model Review – Memo, Reference R108:2024: ERM Consultants, Brisbane. 23 January 2024.

Ayuso, R. A., Wooden, J. L., Foley, N. K., Seal, R. R., and Sinha, A. K., 2005, U-Pb Zircon Ages and Pb Isotope Geochemistry of Gold Deposits in the Carolina Slate Belt of South Carolina: Economic Geology, v. 100, p. 225-252.

Barker, C. A., Secor, D. T., Pray, J. R., and Wright, J. E., 1998, Age and Deformation of the Longtown Metagranite, South Carolina Piedmont: A Possible Constraint on the Origin of the Carolina Terrane: Journal of Geology, v. 106, p. 713-725.

Barton, N.R., Lien, R., and Lunde, J. 1974. Engineering classification of rock masses for the design of tunnel support. Rock Mech. 6(4), 189-239.

BGC Engineering Inc., 2017. “Haile Gold Mine Optimization Study - Open Pit Slope Designs”, Final report issued to OceanaGold Corp. July 21, 2017

Butler, J. R., and Secor, D. T., 1991, The Central Piedmont, *in* Horton, W., and Zullo, V., eds., The Geology of the Carolinas: Knoxville, TN, University of Tennessee Press, p. 59-78.

CIM (2014). Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014.

Clark, L., and R. Pakalnis (1997). An Empirical Design Approach for Estimating Unplanned Dilution from Open Stope Hangingwalls and Footwalls. Presentation at 99th Canadian Institute of Mining annual conference, Vancouver, B.C.

Clark, S., 1999, Geologic Maps and Block Diagrams of the Barite Hill Gold-Silver Deposit and Vicinity, South Carolina and Georgia: U.S. Geological Survey Open-File Report 99-148A

Daniels, D., 2005 South Carolina Aeromagnetic and Gravity Maps and Data: USGS Open-File Report 2005-1022

Feiss, P. G., Vance, R. K., and Weslowki, D. J., 1993, Volcanic rock hosted gold and base metal mineralization associated with Neoproterozoic-Early Paleozoic back-arc extension in the Carolina Terrane, southern Appalachian Piedmont: Geology, v. 21, p. 439-442.

Foley, N. K., Ayuso, R. A., and Seal, R. R., 2001, Remnant colloform pyrite at the Haile gold deposit, South Carolina: A textural key to genesis: Economic Geology, v. 96, p. 891-902.

Foley, N. K., and Ayuso, R. A., 2012, Gold deposits of the Carolina slate belt, southeastern United States: Age and origin of the major gold producers: USGS open file report 2012-1179

Fullagar, P. D., and Butler, J. R., 1979, 325 to 265 Ma-old granitic plutons in the Piedmont of the southeastern Appalachians: American Journal of Science, v. 279, p. 161-185

Gillon, K.A., Spence, W.H., Duckett, R.P., and Benson, C.J., 1995, Geology of the Ridgeway gold deposits, Ridgeway, South Carolina: Society of Economic Geologists Guidebook Series, v. 24, p. 53–94.

Gillon, K.A., Mitchell, T.L., Dinkowitz, S.R., Barnett, R.L., 1998, The Ridgeway gold deposits -A window to the evolution of a Neoproterozoic intra-arc basin in the Carolina Terrane, South Carolina: South Carolina Geology, 40, p. 29–70.

Golder (2010). Haile Gold Mine, Lancaster County, South Carolina, Report on Feasibility Level Pit Slope Evaluation, Project No. 093-91767.300, October 2010.

Grimstad, E., Kankes, K., Bhasin, R., Magnussen, A., and Kaynia, A. 2002. Rock mass quality Q used in designing reinforced ribs of sprayed concrete and energy absorption. In proceedings: International Symposium on Sprayed Concrete. Davos 2002, pp. 134-142.

Hardy, L. S., 1989, Hydrothermal potassium feldspar at the Haile gold mine, South Carolina: Economic Geology, v. 84, p. 2307-2310.

Hayward, N., 1992, Controls on syntectonic replacement mineralization in parasitic antiforms, Haile gold mine, Carolina Slate Belt, U. S. A.: Economic Geology, v. 87, p. 91-112.

Hibbard, J. P., Shell, G., Bradley, P., Samson, S., and Wortman, G., 1998, The Hyco shear zone in North Carolina and southern Virginia: implications for the Piedmont zone-Carolina zone boundary in the southern Appalachians: American Journal of Science, v. 298, p. 85 - 107.

Hibbard, J.P., van Staal, C.R., and Rankin, D.W., 2010, Comparative analysis of the geological evolution of the northern and southern Appalachian orogen: Late Ordovician-Permian: in Tollo, R.P., Bartholomew, M.J., Hibbard, J.P., and Karabinos, P.M., eds., From Rodinia to Pangea: The Lithotectonic Record of the Appalachian Region: Geological Society of America Memoir 206, p. 51-69.

Hibbard, J.P., Stoddard, E. F., Secor, D. T., and Dennis, A. J., 2002, The Carolina Zone: overview of Neoproterozoic to early Proterozoic peri-Gondwanan Terranes along the eastern flank of the southern Appalachians; Earth Science reviews vol. 57, pp. 299-339.

Hutchinson, Douglas J, and Mark S. Diederichs. Cablebolting in Underground Mines. Richmond, B.C: BiTech Publishers, 1996. Print.

Jory, J., 2019, Haile legacy 1975-1994 drillhole data validation 2019, OceanaGold internal report, 11 p.

Larson, A., and Worthington, J.,1989, Geophysical mapping at the Haile mine: report for Piedmont Mining Company, January 1989, 22p.

Maddry, J.W., and Kilbey, T.R., 1995, Geology of the Haile gold mine, selected mineral deposits of the Gulf Coast and southeastern United States: Part II. Gold deposits of the Carolina slate belt: Society of Economic Geologists Guidebook Series, v. 24, p. 147−172.

Mathews, K. E., Hoek, D.C., Wyllie, D.C., and Stewart, S.B.V. 1980. Prediction of stable excavation spans for mining at depths below 1000 metres in hard rock. Report to Canada Centre for Mining and Energy Technology (CANMET), Department of Energy and Resources; DSS File No. 17SQ.23440-0-90210. Ottawa, 1980.

McCauley, C. K., and Butler, J. B.,1966, Gold resources of South Carolina, Bulletin 32, State Development Board, South Carolina, 76 p.

Mitchell, RJ, Olsen, RS & Smith, JD 1982, ‘Model studies on cemented tailings used in mine backfill’, Canadian Geotechnical Journal, vol. 19, no. 1, pp. 14–28.

Mobley, R.M., Yogodzinski, G.M., Creaser, R.A., and Berry, J.M., 2014, Geologic History and Timing of Mineralization at the Haile Gold Mine, South Carolina: Economic Geology, v. 109, p. 1863-1881.

Newton, Edmund; Gregg, D.B.; Mosier, McHenry. (1940). “Operations at the Haile gold mine, Kershaw, SC” Information Circular: Volume 7111, U.S Department of the interior: Bureau of Mines.

Nystrom, P. G., Jr., Willoughby, R. H., and Price, L. K., 1991, Cretaceous and Tertiary stratigraphy of the upper Coastal Plain, South Carolina: *in* Horton, J. W., Jr., and Zullo, V. A. eds., The Geology of the Carolinas: Knoxville, Tennessee, The University of Tennessee Press, p. 221-240

Pardee, J.T.; Park Jr., C.T. (1948) Gold deposits of the southern Piedmont: USGS Professional Paper 213.

Schafer Limited LLC (2015). Revised Haile Gold Mine Overburden Management Plan. September 2015. 25 pp.

Schafer Limited LLC, 2019, Mine Expansion Plan Geochemistry Update – Haile Gold Mine. April 22, 2019.

Schafer Limited LLC, 2024, Personal communication regarding additional testwork and evaluation of site-specific PAG criteria for the Haile Mine. January 5, 2024.

Scheetz, J.W., Stonehouse, J. M., and Zwaschka, M. R., 1991, Geology of the Brewer Gold Mine in South Carolina: Mining Engineering, pp. 38-42

Secor, Donald T. Jr., and Snoke, Arthur W.,1986: Character of the Alleghanian orogeny in the Southern Appalachians: Part II. Geochronological constraints on the tectonothermal evolution of the eastern Piedmont in South Carolina: GSA Bulletin, Volume 97, Number 11.

Secor, D. T., Jr., and Wagener, H. D., 1968, Stratigraphy, structure and petrology of the Piedmont in Central South Carolina: Columbia, South Carolina: South Carolina Geological Survey, Geologic Notes, v. 12, p. 67-84.

Speer, W. E., and Maddry, J. W., 1993, Geology and recent discoveries at the Haile gold mine, Lancaster County, South Carolina: South Carolina Geology, v. 35, p. 9-26.

Spence, W. H., Maddry, J. W., Worthington, J. E., Jones, E. M., and Kiff, I. T., 1980, Origin of the gold mineralization at the Haile gold mine, Lancaster County, South Carolina: Mining Engineering, v. 32, p. 70-73.

Stein, H. J., Markley, R. J., Morgan, J. W., Hannah, J. L., Zak, K., and Sundblad, K., 1997, Re-Os dating of shear-hosted Au deposits using molybdenite: *in* Papunen, H., ed., Mineral Deposits, Research and Explorations: Where Do They Meet? Proceedings of the Fourth Biennial SGA Meeting, Turku, Finland, 11-13 August 1997, p. 313-317.

Tomkinson, M. J., 1988, Gold mineralization in phyllonites at the Haile Mine, South Carolina: Economic Geology, v. 83, p. 1392-1400.

Tosdal, R.M., 2020, Haile structural geology evaluation, OceanaGold internal report.

Weis, T., 2016, Extended Haile geophysical interpretation: report for OceanaGold, December 2016, 143 p.

Whitney, J.A., Paris, T.A., Carpenter, R.H., and Hartley, M.F., 1978: Volcanic evolution of the southern slate belt of Georgia and South Carolina - a primitive oceanic arc, Journal of Geology, vol. 86, p137-192.

Worthington, J. E., 1993, The Carolina slate belt and its gold deposits: Reflections after a quarter century, South Carolina: Geology, v. 35, p. 1-8.

Worthington, J. E., and Kiff, I.T., 1970, A suggested volcanogenic origin for certain gold deposits in the slate belt of the Carolina piedmont: Economic Geology, v. 65, p. 529-537.

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.
Hanging wall	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
Overburden	Material that overlies a mineral deposit and must be removed prior to mining. At Haile, overburden is waste material mined from the open pits.
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).
Waste	Rock that must be broken and disposed of to gain access to and excavate the ore. Costs of mining and processing exceed the value of recoverable metals. At Haile, waste refers to material produced from underground mine development.

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

Abbreviation	Unit or Term
CIT	Corporate Income Tax
cm^3	cubic centimeter
CoG	cut-off grade
CPLEX	IBM ILOG CPLEX Optimizer
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	Fracture frequency 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
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	Haile Mine Venture, joint venture between Amax and Piedmont
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
kN	kilonewton
kN/m^3	kilonewton per cubic meter

Abbreviation	Unit or Term
Koz	thousand troy ounce
KS	Kolmogorov-Smirnoff
Kt	thousand tonnes
kV	kilovolt
kW	kilowatt
L	liter
Lb	pound
Leapfrog	Seequent 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
Mt	million tonnes
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

Abbreviation	Unit or Term
QP	Qualified Persons
QSP	quartz-sericite-pyrite
RC	reverse circulation
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
µm	microns

Appendices

Appendix A: Certificates of Qualified Persons

OCEANAGOLD

CERTIFICATE OF QUALIFIED PERSON

I, David Read Carr, MAusIMM CP (Met) do hereby certify that:

1. I am Group Manager Metallurgy of OceanaGold Corporation, Suite 1020, 400 Burrard Street, Vancouver, British Columbia V6C 3A6.
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, 2023 (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 31 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, 2020, 2022, August 28, 2023 for 27 days and November 6, 2023 for 39 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 (New Zealand) 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, the "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated September 30, 2020, the "NI 43-101 Technical Report Haile Gold Mine Lancaster County, South Carolina" dated March 30, 2022 and in my role as Group Manager Metallurgy 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 28th Day of March, 2024.

"Signed" *"Stamped"*

David Read Carr, MAusIMM CP (Met)

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 Executive Vice President, Chief Operating Officer Americas of OceanaGold, 6911 Snowy Owl Rd, 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, 2023 (the “Technical Report”).
3. I graduated with a degree in Mine and Metallurgical Engineering from Universidad Nacional de Colombia in 1984. In addition, I have obtained a Master or Sciences in Earth and Systems Engineering from Colorado School of Mines in 2000 and a master’s in business administration from Regis University in 2005. I am a Registered member of the Society of Mine Engineers, registration number 04038617. I have worked as a Mine Engineer for a total of 40 years since my graduation from university. My relevant experience includes 9 years as mine planning engineer and analyst at Cerrejon Coal Mine in Colombia, 9 years as Senior Planning Engineer with World Minerals in California, and as Senior Operations Manager with several companies around the world, including Barrick, AngloGold Ashanti, Kirkland Lake and Detour Lake Mine and 2.5 years as Executive Vice President and COO at OceanaGold Corporation.
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 work at Haile since July 15th, 2021.
6. I am responsible for environmental and open pit Mineral Reserves, the open pit portions of Section 15 and 16.3, Section 16 opening statements, Sections 16.1, 16.1.1, 16.1.3, 16.1.4, 16.1.6, 16.1.7, 16.1.8, 18.7, 19, 20, the open pit capital and operating costs portion of section 21, the other/G&A portions of the operating costs in section 21, the tailings/overburden capital and 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 currently working as Executive Vice President and COO of Americas.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is being the Executive Vice President and COO of the Americas, based at Haile.
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 28th Day of March 2024.

“Signed” *“Stamped”*

David Londono, BSC, MEng, MSc Earth and Systems Engineering, MBA, SME-RM

OCEANAGOLD

CERTIFICATE OF QUALIFIED PERSON

I, Jonathan Moore, BSc Geology (Hons), MAusIMM(CP) do hereby certify that:

1. I am Group Manager, Resource Development, of OceanaGold (New Zealand) Limited, 22 Maclaggan Street, Dunedin, New Zealand.
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, 2023 (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 and grade control 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 visited the Haile property annually since 2015. Most recently, I visited the Haile property on November 20, 2023 for 19 days.
6. I am 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.
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. I have been employed by OceanaGold or its subsidiaries since May 1996.
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 28th Day of March, 2024.

"Signed" *"Stamped"*

Jonathan Moore, BSc Geology (Hons), MAusIMM(CP)

CERTIFICATE OF QUALIFIED PERSON

I, Brianna Drury, BEng Mining, RM-SME do hereby certify that:

1. I am Underground Engineering Superintendent 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, 2023 (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 14 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 3.5 years with OceanaGold as the underground project manager and more recently the underground engineering superintendent.
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 u underground Mineral Reserves, the underground portions of Section 15 and 16.3, Sections 16.2, 16.2.1, 16.2.5 - 16.2.11, 18.8, 18.9, the underground mining capital and operating costs portion of Section 21, 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-l0lFl 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 28th Day of March, 2024.

"Signed" *"Stamped"*

Brianna Drury, BEng Mining, RM-SME

CERTIFICATE OF QUALIFIED PERSON

I, Larry Standridge, PE, MSE Geotechnical do hereby certify that:

1. I am Principal Engineer, Geotechnical 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, 2023 (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 23 years since my graduation from university. My relevant experience includes 19 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, registration as a Professional Engineer in the state of Arizona (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, 2017 for 2 days, October 10, 2018 for 2 days, January 14, 2020 for 3 days, May 18, 2021 for 3 days, and January 24, 2024 for 1 day.
6. I am 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.
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 28th Day of March, 2024.

"Signed" *"Stamped"*

____

Larry Standridge, PE, MSE Geotechnical

CERTIFICATE OF QUALIFIED PERSON

I, Robert Cook, PE, RM-SME do hereby certify that:

1. I am Principal I Geological Engineer of Call & Nicholas, 2475 N Coyote Dr. Tucson, AZ 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, 2023 (the "Technical Report").
3. I graduated with a degree in Geological Engineering from University of Arizona in 2008. I am a Registered Member of the Society for Mining, Metallurgy, & Exploration (SME). I have worked as a Geological Engineer for a total of 15 years since my graduation from university. My relevant experience includes 5 years as an underground geological engineer for Newmont Mining Corporation in operations using the Longhole open stoping (LHOS) method as well as consulting on LHOS projects for more than 9 years with Call and Nicholas, Inc..
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, 2017 for 2 days, July 27, 2021 for 2 days, January 16, 2023 for 4 days, and January 22, 2024 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 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 28th Day of March, 2024.

"Signed" *"Stamped"*

Robert Cook, PE, RM-SME

CERTIFICATE OF QUALIFIED PERSON

I, Jay Newton Janney-Moore, PE, RM-SME 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, 2023 (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 on September 24, 2019 for 3 days, August 31, 2021 for 3 days, October 18, 2022 for 1 day, and September 26, 2023 for 2 days.
6. I am responsible for tailing and overburden storage, Sections 18.1, 18.2, 18.3, 18.4, 18.5, the tailings/overburden capital and operating cost portions of section 21, 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, June 30, 2020, and March 30, 2022.
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 28th Day of March, 2024.

"Signed" *"Stamped"*

Jay Newton Janney-Moore

CERTIFICATE OF QUALIFIED PERSON

I, William Lucas Kingston, MSc, PG, RM-SME 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, 2023 (the "Technical Report").
3. I graduated with a degree in Geology rom 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 30 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, 18.6 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.
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 28th Day of March, 2024.

"Signed" *"Stamped"*

William Lucas Kingston, MSc, PG.

SRK Consulting (U.S.), Inc.
Suite 400
999 Seventeenth Street
Denver, CO 80202

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Matthew Sullivan, BS Economics, BS Metallurgy do hereby certify that:

1. I am Principal Consultant (Mining Economics) of SRK Consulting (U.S.), Inc., 999 Seventeenth Street, Suite 400, 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, 2023 (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 11 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 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 Haile Gold Mine Lancaster County, South Carolina" dated March 30, 2022.
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 28th Day of March, 2024.

"Signed" *"Stamped"*

Matthew Sullivan, BS Economics, BS Metallurgy

SRK Consulting (U.S.), Inc.
5250 Neil Road, Suite 300
Reno, Nevada 89502

T: (775) 828-6800
F: (775) 828-6820

reno@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Brooke Miller Clarkson, C.P.G. do hereby certify that:

1. I am a Principal Consultant of SRK Consulting (U.S.), Inc., 5250 Neil Road, Reno, Nevada 89502.
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, 2023 (the “Technical Report”).
3. I graduated with a degree in Bachelor of Arts degree in Geology from Lawrence University in 2002. In addition, I have obtained a Master of Science degree in Geological Sciences from The University of Oregon in 2004. I am a Certified Professional Geologist of the American Association of Professional Geologists. I have worked as a Geologist for a total of 18 years since my graduation from university. My relevant experience includes mining and exploration geology, geochemical characterization, and geologic modeling.
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 14, 2023 for one day.
6. I am responsible for geochemistry, Sections 2, 3, 16.1.5, 16.2.4, 24, 27, 28 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 28th Day of March, 2024.

“Signed” *“Stamped”*

Brooke Miller Clarkson, CPG

Appendix B: LoM Annual Cash Flow Forecast

Technical-Economic Model Summary

srk consulting

Company: Oceanagold
Haile

YEAR				2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036
Time Before Mine Closure			LoM	16	15	14	13	12	11	10	9	8	7	6	5	4
Production Timeline			Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Discount Factors	5%	US$ & Metric Units	or Avg.	0.9759	0.9294	0.8852	0.8430	0.8029	0.7646	0.7282	0.6936	0.6605	0.6291	0.5991	0.5706	0.5434
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	$1,500
Silver		US$/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
Physicals																
Total OP Ore Mined		kt	35,700	2,359	2,239	5,281	5,047	5,876	3,397	1,983	1,092	2,226	4,130	2,071	-	-
Total OP Waste Mined		kt	282,624	33,992	34,367	37,372	41,170	36,634	38,892	19,550	16,551	14,860	8,163	1,073	-	-
Total OP Material Mined		kt	318,325	36,351	36,606	42,653	46,217	42,511	42,289	21,533	17,643	17,086	12,293	3,144	-	-
Strip Ratio		w:o	7.9	14.4	15.4	7.1	8.2	6.2	11.4	9.9	15.2	6.7	2.0	0.5	-	-
Total UG Ore Mined		kt	7,859	562	624	778	761	1,079	969	867	747	749	643	80	-	-
Total Ore Tonnes, Mined		kt	43,559	2,921	2,863	6,059	5,808	6,955	4,365	2,850	1,839	2,974	4,772	2,151	-	-
Total Ore Tonnes, Processed		kt	44,257	3,678	3,690	3,698	3,701	3,614	3,661	3,681	3,666	3,714	3,723	3,793	3,638	-
Gold Grade, Processed		g/t	1.91	2.13	1.96	3.03	2.71	2.56	1.91	1.63	1.28	1.52	1.91	1.72	0.61	-
Silver Grade, Processed		g/t	2.30	1.66	1.96	2.33	2.55	2.27	2.38	2.43	2.12	2.54	2.55	2.59	2.20	-
Contained Gold, Processed		koz	2,724	252	233	360	323	297	225	193	151	182	228	209	71	-
Contained Silver, Processed		koz	3,272	197	232	277	303	264	281	287	250	304	305	316	257	-
Average Recovery, Gold		%	84.5%	84.9%	85.2%	87.6%	87.1%	86.7%	83.8%	82.5%	81.0%	83.0%	85.1%	82.4%	69.2%	-
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%	-
Recovered Gold, Dore		koz	2,303	214	198	315	281	257	188	159	122	151	194	172	49	-
Recovered Silver, Dore		koz	2,290	138	162	194	212	185	196	201	175	213	214	221	180	-
Payable Gold, Dore		koz	2,303	214	198	315	281	257	188	159	122	151	194	172	49	-
Payable Silver, Dore		koz	2,267	136	161	192	210	183	194	199	173	210	211	219	178	-
Cash Flow																
Gross Revenue		**$000s**	**3,454,270**	**320,853**	**297,717**	**473,003**	**422,056**	**386,187**	**282,390**	**238,300**	**183,188**	**226,350**	**291,676**	**258,653**	**73,898**	**-**
OP Mining Cost		$000s	(661,382)	(48,078)	(32,622)	(59,819)	(35,325)	(51,224)	(117,170)	(85,676)	(80,341)	(67,935)	(51,676)	(24,633)	(6,883)	-
UG Mining Cost		$000s	(466,019)	(36,088)	(34,899)	(43,855)	(39,753)	(61,325)	(52,663)	(61,248)	(51,285)	(42,520)	(33,659)	(8,724)	-	-
Process Cost		$000s	(668,051)	(61,942)	(58,089)	(56,325)	(56,528)	(56,151)	(56,397)	(55,853)	(55,618)	(55,952)	(54,985)	(54,606)	(45,604)	-
Site G&A Cost		$000s	(242,173)	(28,928)	(26,998)	(26,908)	(26,985)	(26,508)	(24,885)	(23,689)	(19,732)	(18,216)	(8,420)	(7,818)	(3,084)	-
Dore Refining/Freight Costs		$000s	(7,385)	(587)	(595)	(954)	(830)	(769)	(618)	(548)	(408)	(548)	(672)	(625)	(232)	-
By-Product Credit		$000s	40,810	2,453	2,893	3,460	3,781	3,288	3,500	3,581	3,119	3,788	3,806	3,938	3,203	-
Direct Cash Costs		**$000s**	**(2,004,201)**	**(173,170)**	**(150,310)**	**(184,401)**	**(155,640)**	**(192,689)**	**(248,233)**	**(223,434)**	**(204,264)**	**(181,385)**	**(145,607)**	**(92,468)**	**(52,601)**	**-**
Royalties		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Environmental Bond		$000s	7,800	(1,800)	(1,800)	(1,800)	(1,800)	(1,800)	(1,200)	(1,000)	(1,000)	-	-	-	20,000	-
BLANK		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital Leases: Sustaining		$000s	(34,281)	(12,064)	(18,548)	(3,669)	-	-	-	-	-	-	-	-	-	-
Capital Leases: Non-Sustaining		$000s	(29,098)	(14,389)	(4,039)	(5,541)	(2,691)	(2,438)	-	-	-	-	-	-	-	-
Non-Operating Cash Costs		**$000s**	**(55,579)**	**(28,254)**	**(24,387)**	**(11,010)**	**(4,491)**	**(4,238)**	**(1,200)**	**(1,000)**	**(1,000)**	**-**	**-**	**-**	**20,000**	**-**
Total Project Costs		**$000s**	**(2,059,780)**	**(201,424)**	**(174,697)**	**(195,410)**	**(160,131)**	**(196,927)**	**(249,433)**	**(224,434)**	**(205,264)**	**(181,385)**	**(145,607)**	**(92,468)**	**(32,601)**	**-**
Operating Margin		**$000s**	**1,394,490**	**119,429**	**123,021**	**277,592**	**261,924**	**189,260**	**32,957**	**13,866**	**(22,076)**	**44,965**	**146,069**	**166,185**	**41,298**	**-**
Earnings Before Interest, Taxes & Depreciation		**$000s**	**1,394,490**	**119,429**	**123,021**	**277,592**	**261,924**	**189,260**	**32,957**	**13,866**	**(22,076)**	**44,965**	**146,069**	**166,185**	**41,298**	**-**
Depreciation Allowance		$000s	(1,644,459)	(157,780)	(97,050)	(224,871)	(184,657)	(201,121)	(128,281)	(97,197)	(82,868)	(116,211)	(172,010)	(149,193)	(33,221)	-
Depletion Allowance		$000s	(451,050)	(41,896)	(38,875)	(61,764)	(55,111)	(50,427)	(36,874)	(31,117)	(23,920)	(29,556)	(38,086)	(33,774)	(9,649)	-
Earnings Before Taxes		**$000s**	**(701,020)**	**(80,247)**	**(12,905)**	**(9,042)**	**22,156**	**(62,288)**	**(132,198)**	**(114,448)**	**(128,864)**	**(100,802)**	**(64,027)**	**(16,783)**	**(1,573)**	**-**
Income tax		$000s	(4,699)	(230)	(1,824)	(1,185)	(1,460)	-	-	-	-	-	-	-	-	-
Net Income		**$000s**	**(705,719)**	**(80,477)**	**(14,729)**	**(10,227)**	**20,696**	**(62,288)**	**(132,198)**	**(114,448)**	**(128,864)**	**(100,802)**	**(64,027)**	**(16,783)**	**(1,573)**	**-**
Non-Cash Add Back - Depreciation		$000s	1,644,459	157,780	97,050	224,871	184,657	201,121	128,281	97,197	82,868	116,211	172,010	149,193	33,221	-
Non-Cash Add Back - Depletion		$000s	451,050	41,896	38,875	61,764	55,111	50,427	36,874	31,117	23,920	29,556	38,086	33,774	9,649	-
Working Capital		$000s	0	3,590	5,827	(7,669)	(670)	4,884	4,634	(816)	(396)	(1,848)	(2,956)	(2,596)	201	(2,185)
Operating Cash Flow		**$000s**	**1,389,790**	**122,789**	**127,024**	**268,738**	**259,794**	**194,144**	**37,592**	**13,050**	**(22,472)**	**43,117**	**143,113**	**163,589**	**41,499**	**(2,185)**
Operations Information Technology		$000s	(1,370)	(500)	-	(250)	-	-	(120)	(500)	-	-	-	-	-	-
General Operations Expenditure		$000s	(160,837)	(28,553)	(30,120)	(18,054)	(11,070)	(15,599)	(19,060)	(15,715)	(10,130)	(10,013)	(1,774)	(750)	-	-
Operations Land and Building (Acquisitions)		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Brownfields Exploration		$000s	(372)	-	(275)	(97)	-	-	-	-	-	-	-	-	-	-
Operations Based Mining Projects		$000s	(146,063)	(24,762)	(50,639)	(28,371)	(13,452)	(9,850)	(11,063)	(6,275)	(1,650)	-	-	-	-	-
Open Pit Mine		$000s	(408,813)	(74,738)	(91,891)	(70,619)	(92,000)	(73,436)	(6,129)	-	-	-	-	-	-	-
Rehabilitation (Sustaining)		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Underground Mine		$000s	(15,245)	(3,982)	(2,367)	(2,259)	(241)	(2,102)	(2,906)	(1,389)	-	-	-	-	-	-
Total Sustaining Capital Spend		**$000s**	**(732,699)**	**(132,536)**	**(175,292)**	**(119,649)**	**(116,763)**	**(100,986)**	**(39,278)**	**(23,879)**	**(11,780)**	**(10,013)**	**(1,774)**	**(750)**	**-**	**-**
Growth Information Technology		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Growth Expenditure		$000s	(14,602)	(3,000)	(6,650)	(3,452)	(500)	(500)	(500)	-	-	-	-	-	-	-
Land and Building (Growth)		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Greenfields Exploration		$000s	(15,523)	(7,368)	(4,008)	(1,702)	(755)	(1,689)	-	-	-	-	-	-	-	-
Stand-alone Mining Projects		$000s	(83,708)	(7,975)	(995)	(16,350)	(36,031)	(22,356)	-	-	-	-	-	-	-	-
Open Pit Mine		$000s	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Rehabilitation (Non-Sustaining)		$000s	(115,599)	(7,212)	-	-	(1,624)	-	-	(3,537)	(2,819)	(46)	(12,878)	(2,716)	(32,226)	(52,541)
Underground Mine		$000s	(52,405)	(14,062)	(9,641)	(5,329)	(496)	(8,329)	(13,750)	(799)	-	-	-	-	-	-
Total Non-Sustaining Capital		**$000s**	**(281,836)**	**(39,617)**	**(21,294)**	**(26,833)**	**(39,407)**	**(32,874)**	**(14,250)**	**(4,336)**	**(2,819)**	**(46)**	**(12,878)**	**(2,716)**	**(32,226)**	**(52,541)**
Total Capital		**$000s**	**(1,014,535)**	**(172,153)**	**(196,586)**	**(146,482)**	**(156,170)**	**(133,860)**	**(53,528)**	**(28,215)**	**(14,599)**	**(10,059)**	**(14,652)**	**(3,466)**	**(32,226)**	**(52,541)**
Metrics																
Economic Metrics																
a) Pre-Tax																
Free Cash Flow		$000s	**379,954**	(49,133)	(67,738)	123,441	105,084	60,284	(15,936)	(15,165)	(37,071)	33,058	128,462	160,123	9,272	(54,727)
Cumulative Free Cash Flow		$000s		(49,133)	(116,871)	6,570	111,654	171,938	156,002	140,836	103,765	136,824	265,285	425,409	434,681	379,954
NPV @	5.00%	$000s	**260,540**	(47,949)	(62,958)	109,266	88,588	48,400	(12,186)	(11,044)	(25,711)	21,836	80,812	95,933	5,291	(29,739)
Cumulative NPV		$000s		(47,949)	(110,906)	(1,640)	86,948	135,348	123,162	112,119	86,408	108,244	189,056	284,988	290,279	260,540
IRR		%	**53.4%**													
b) After-Tax																
Free Cash Flow		$000s	**375,255**	(49,363)	(69,562)	122,256	103,624	60,284	(15,936)	(15,165)	(37,071)	33,058	128,462	160,123	9,272	(54,727)
Cumulative Free Cash Flow		$000s		(49,363)	(118,925)	3,331	106,955	167,239	151,302	136,137	99,066	132,124	260,586	420,709	429,982	375,255
NPV @	5.00%	$000s	**256,340**	(48,174)	(64,653)	108,218	87,357	48,400	(12,186)	(11,044)	(25,711)	21,836	80,812	95,933	5,291	(29,739)
Cumulative NPV		$000s		(48,174)	(112,827)	(4,609)	82,748	131,148	118,963	107,919	82,208	104,044	184,856	280,789	286,079	256,340
IRR		%	**51.8%**													
Sales Metrics																
LoM Dore Sales		koz Au	2,303	214	198	315	281	257	188	159	122	151	194	172	49	-
LoM All-In Sustaining Costs (AISC)		$000s	2,763,381	319,570	345,950	309,518	274,204	295,475	288,711	248,313	217,044	191,397	147,381	93,218	32,601	-
LoM AISC Metric		$ / oz Au	$1,200	$1,494	$1,743	$982	$975	$1,148	$1,534	$1,563	$1,777	$1,268	$758	$541	$662	$0