Anaconda Mining

From the ground up

# NI 43-101 TECHNICAL REPORT AND FEASIBILITY STUDY FOR THE GOLDBORO GOLD PROJECT, EASTERN GOLDFIELDS DISTRICT, NOVA SCOTIA

PREPARED FOR: ANACONDA MINING INC.

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Mineral Resource Estimate: November 15, 2021

Mineral Reserves Estimate: December 15, 2021

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# NI 43-101 TECHNICAL REPORT AND FEASIBILITY STUDY

# FOR THE

# GOLDBORO GOLD PROJECT
# EASTERN GOLDFIELDS DISTRICT, NOVA SCOTIA

**Prepared for:**

Anaconda Mining Inc.

Project # 20048-02

Technical Report Effective Date: December 16, 2021

Mineral Resource Effective Date: November 15, 2021

Mineral Reserves Effective Date: December 15, 2021

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Technical Report Effective Date: December 16, 2021

Mineral Resource Effective Date: November 15, 2021

Mineral Reserves Effective Date: December 15, 2021

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The undersigned prepared this Technical Report, titled NI 43-101 Technical Report and Feasibility Study for the Goldboro Gold Project, Eastern Goldfields District, Nova Scotia, with an effective date of December 16, 2021 in support of the public disclosure for public listing. The format and content of this report conforms to National Instrument 43-101 of the Canadian Securities Administrators.

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| REV. NO | ISSUE DATE | PREPARED BY | REVIEWED BY | APPROVED BY | DESCRIPTION OF REVISION |
|---|---|---|---|---|---|
| 2.4 | Dec. 23, 2021 | Nordmin | Client | | Draft Review |
| 3.3 | Jan. 8, 2022 | Nordmin | QPs | | Draft Review |
| 3.6 | Jan. 11, 2022 | Nordmin | Client, QP’s | | Final Review |
| | Jan 11, 2022 | Nordmin | QP’s | QP’s | Final |

# FORWARD-LOOKING INFORMATION

This document includes certain “forward-looking” information and "forward-looking statements" within the meaning of applicable securities legislation, together, forward-looking statements. All statements, other than statements of historical facts constitute forward-looking statements. Forward-looking statements estimates and statements that describe the Company's future plans, objectives or goals, including words to the effect that the Company or management expects a stated condition or result to occur. Forward-looking statements may be identified by such terms as "believes", "anticipates", "expects", "estimates", "may", "will", "could", "would", "if", "yet", "potential", "undetermined", "objective", "plan" or similar expressions. Since forward-looking statements are based on assumptions and estimates and address future events and conditions, by their very nature, they involve inherent risk, and uncertainties. Although these statements are based on information currently available to the authors of this Technical Report and the Company, the authors provides no assurance that actual results will meet the expectations set forth herein. Risks, uncertainties, and other factors, known and unknown, involved with forward-looking statements could cause actual events, results, performance, prospects, and opportunities to differ materially from those expressed or implied by such forward-looking statements. Forward-looking statements in this Technical Report include, but are not limited to, the Company's objectives, goals, future plans, statements, exploration results, potential mineralization, estimation of Mineral Resources and Reserves, exploration, and mine development plans, the timing of the commencement of operations and estimates of market conditions. Furthermore, forward-looking statements include disclosure regarding the economics and project parameters presented in the feasibility study, including, without limitation, IRR, all-in sustaining costs, NPV and other costs and economic information, possible events, conditions or financial performance that is based on assumptions about future economic conditions and courses of action; the timing and costs of future development and exploration activities on the Company's projects; success of development and exploration activities; permitting time lines and requirements; time lines for further studies; planned exploration and development of properties and the results thereof; and planned expenditures and budgets and the execution thereof. Factors that could cause actual results to differ materially from such forward-looking statements include, but are not limited to the failure to identify Mineral Resources, failure to convert estimated Mineral Resources to Mineral Reserves, the failure to make a development or production decision, the nature of metallurgical test results, geotechnical challenges, delays in obtaining or failures to obtain required governmental, environmental, or other project approvals, political risks, inability to fulfill the duty to accommodate First Nations and other Indigenous Peoples, uncertainties relating to epidemics, pandemics and other public health crises, including COVID-19 or similar such viruses, uncertainties relating to the availability and costs of financing needed in the future, changes in equity markets, inflation, changes in foreign currency exchange rates, fluctuations in commodity prices, delays in the development of projects, capital and operating costs varying significantly from estimates and the other risks involved in the mineral exploration and development industry, and those risks set out in the Company's public documents filed on the System for Electronic Document Analysis and Retrieval ("SEDAR") at www.sedar.com. Although the authors believes that the assumptions and factors used in preparing the forward-looking statements in this Technical Report are reasonable, undue reliance should not be placed on such forward-looking statements, which only apply as of the date of this Technical Report, and no assurance can be given that such events will occur in the disclosed time frames or at all. The authors disclaim any intention or obligation to update or revise any forward-looking statements, whether as a result of new information, future events, or otherwise, other than as required by applicable law.

# CAUTIONARY NOTE TO U.S. READERS REGARDING ESTIMATES OF MEASURED, INDICATED AND INFERRED RESOURCES

This Technical Report uses the terms "Measured" and "Indicated" Mineral Resources and "Inferred" Mineral Resources. The Company advises U.S. investors that, while these terms are recognized by the U.S. Securities and Exchange Commission (SEC) under Regulation S-K subpart 1300, there are differences between the definitions ascribed to such terms under Regulation S-K subpart 1300 and the Canadian Institute of Mining (CIM) Standards.

The estimation of "Measured" and "Indicated" Mineral Resources involves greater uncertainty as to their existence and economic feasibility than the estimation of proven and probable reserves. The estimation of "Inferred" resources involves far greater uncertainty as to their existence and economic viability than the estimation of other categories of resources. It cannot be assumed that all or any part of a "Measured," "Indicated," or "Inferred" Mineral Resource will ever be upgraded to a higher category.

Under Canadian rules, estimates of "Inferred Mineral Resources" may not form the basis of feasibility studies, prefeasibility studies, or other economic studies, except in prescribed cases, such as in a preliminary economic assessment under certain circumstances. Under SEC rules, estimates of "Inferred Mineral Resources" may not be considered when assessing the economic viability of a mining project, and may not be converted to a mineral reserve. Under U.S. standards, mineralization may not be classified as a "mineral reserve" unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the reserve determination is made. U.S. investors are cautioned not to assume that any part or all of a "Measured," "Indicated," or "Inferred" Mineral Resource exists or is economically or legally mineable. Information concerning descriptions of mineralization and resources contained herein may not be comparable to information made public by U.S. companies subject to the reporting and disclosure requirements under the SEC's Regulation S-K subpart 1300.

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# 1. SUMMARY

## 1.1 Current Technical Report

Nordmin Engineering Ltd. (Nordmin) was retained by Anaconda Mining Inc. (Anaconda or the Company) to prepare a Canadian National Instrument 43-101 (NI 43-101) Technical Report (Technical Report) and Feasibility Study for the Goldboro Gold Project (Goldboro or the Project), situated approximately 185 kilometres (km) northeast of Halifax, Nova Scotia within Nova Scotia's Eastern Goldfields District, Canada.

This Technical Report supports the disclosures in the Company's news release dated December 16, 2021, entitled "Anaconda Mining Reports Positive Phase 1 Open Pit Feasibility Study for the Goldboro Gold Project." All measurement units used in this Technical Report are metric unless otherwise noted. Currency is expressed in Canadian dollars (C$) The Technical Report uses Canadian English.

Mineral Resources and Reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (May 2014; the 2014 CIM Definition Standards) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; 2019 CIM Best Practice Guidelines).

The effective date of this Technical Report is December 16, 2021. The effective date of the Mineral Resource Estimate is November 15, 2021 and the effective date of the Mineral Reserve Estimate is December 15, 2021.

## 1.2 Project Description, Location, and Access

The Project is comprised of three domains known as the West Goldbrook (WG), Boston Richardson (BR), and East Goldbrook (EG) Gold Systems. The WG Gold System is separated from the BR Gold System by a north trending, near vertical fault with tens of metres of apparent offset. The EG Gold System is separated from the BR Gold System by a thick greywacke sequence or marker unit.

The Goldboro Property (the Property) is situated on the eastern shore of Nova Scotia, Canada. The Property's central point is approximately located at 45° 12' 2.6" N latitude and 61° 39' 2.0" W longitude. The Property consists of 37 contiguous claims, registered through the Company's wholly-owned subsidiary Orex Exploration Inc., covering a total area of approximately 592 hectares held under Exploration Licence No. 05888. This title is in its 43rd year of issue and is renewed every two years, with the next renewal date on November 29, 2022.

The Property is located approximately 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro, on the eastern shore of Isaac's Harbour, in Guysborough County, Nova Scotia, Canada. The elevation is nominally 70 m above sea level.

All-weather highway, Route 316 links the village of Goldboro to the town of Antigonish. A secondary gravel road named Goldbrook Road, accessible from Route 316, crosses the Property, and passes near the historical BR shaft and exploration decline. Smaller logging roads and trails provide good access to most areas of the Property.

## 1.3 History

Gold mineralization on the Property was first discovered in 1862 by Howard Richardson of the Geological Survey of Canada in quartz veins within the Isaac's Harbour anticline. The gold bearing BR Belt (slate and quartz) was subsequently discovered by Howard Richardson in 1892. The Richardson Gold Mining Company (Richardson Gold Mining) began production from the belt in 1893 at an average reported grade of 13.03 grams per tonne (g/t) gold milled. Milling recoveries were reported to be in the 50% to 60% range.

From 1901 to 1905, three gold bearing belts were intersected in the Dolliver Mountain mine, located 2 km west of the Boston Richardson Mine. In 1904, 7,195 tonnes were milled at a grade of 0.87 g/t to produce 205 ounces (oz) of gold. In 1905, several bodies of quartz and slate were intersected by a 152 metres (m) deep drill hole at the bottom of the main shaft along the anticlinal axis, but results were unsatisfactory, and mining at Dolliver Mountain mine ceased.

From 1909 to 1910, the WG exploration shaft intersected five gold bearing belts. Three of these were mill tested, but the milling results were considered unsatisfactory, and the mine was abandoned.

The total gold recovery from 1893 to 1910 for the Property has been estimated to be 376,303 tonnes at an average recovered gold grade of 4.11 g/t to produce 54,871 oz. However, mill recovery is reported to be approximately 67%. Operations at the mine continued on a small scale in 1911 and 1912.

In 1981, Patino Mines (Québec) Ltd. completed a geophysical program covering the Upper Seal Harbour district. In 1984, Onitap Resources Inc. (Onitap) acquired 37 claims overlying the Property. Between 1984 and 1988, Onitap conducted diamond drilling programs, airborne Very Low Frequency Electromagnetic (VLF-EM) surveys, and surface Induced Polarization (IP) surveys. During this period, several new mineralized belts were discovered.

Orex Exploration Inc. (also known as Exploration Orex Inc.) (Orex) acquired the Property from Onitap in 1988. Excepting a period of inactivity from 1996 to 2004, Orex pursued both surface and underground exploration programs, including large amounts of core drilling, metallurgical testing programs, resource estimation programs, and economic assessments of the Property.

Osisko Mining Corporation (Osisko), under the terms of an agreement with Orex, carried out an extensive core drilling assessment of the Property during the 2010 to 2012 period.

In March of 2017, the Company acquired control of the Property under the terms of a court approved Plan of Arrangement whereby Orex became a wholly-owned subsidiary of the Company. Work programs carried out in all years between 2017 to 2021 by the Company primarily focused on expansion and infill drilling of the Goldboro Gold Deposit (the Deposit) as well as conducting an underground bulk sample (the Bulk Sample) in 2018.

## 1.4 Geological Setting, Mineralization, and Deposit Types

### 1.4.1 Geological Setting and Mineralization

The Project is located within the Appalachian Orogen and is underlain by the rocks of the Cambrian to Ordovician aged Meguma Supergroup. These include sedimentary rocks of the Goldenville Formation and overlying, younger Halifax Formation. A minimum 1.5 km thick stratigraphic section of the Goldenville Formation is centred on the Deposit and in the form of a regional upright anticline, with Halifax Formation rocks located 1.6 km to the south.

At the Deposit, the Goldenville Formation consists of alternating greywacke and argillite beds with an approximate true thickness of 950 m. The base of the stratigraphic sequence intersected in the BR Gold System consists of roughly 325 m of alternating greywacke and argillite, with varying proportions of both rock types, ranging in thickness from less than 1 m up to 10 m. This is overlain by the Marker Horizon, which consists of a 40 m to 50 m greywacke bed that is commonly intersected during drilling and in underground workings. The Marker Horizon appears to thin or is offset by the New Belt Fault on the northern limb of the anticline toward the west. Above the Marker Horizon is the EG Gold System, approximately 560 m thick, consisting of alternating greywackes, and argillites. Within the EG Gold System there is a second, thick, greywacke sequence varying in thickness from 20 m to 60 m. This may represent a new marker unit within the stratigraphy.

The structure of the Project Area is dominated by the Upper Seal Harbour Anticline. The anticline folds all levels of stratigraphy observed in core and underground to form an upright, tight anticline that plunges 20° eastward. The enveloping surface to bedding also dips moderately eastward at 20°. Younging is upward, orthogonal to the hinge, and limbs of the fold. An axial planar cleavage is well developed at all levels of stratigraphy but pervasive within the hinge of the fold. The intersection of the axial planar cleavage forms an intersection lineation commonly observed on bedding surfaces that plunge parallel to the fold axis. All bedding and first-generation structures are refolded by open reclined folds that modify the axial plane and limbs of the Upper Seal Harbour Anticline. The axial plane of second generation folds dips shallowly, and an axial planar cleavage is observed in both the drill core and within underground workings.

All earlier structures are deformed by late brittle faults. One generation of these faults, which includes the New Belt Fault, are steeply dipping, and occur both parallel, and cross-cutting regionally folded stratigraphy. These faults also disrupt the stratigraphy on the northern limb of the fold structure in the WG and BR Gold Systems, although kinematics, and displacement are not known. A second generation of faults strike northerly and are steeply dipping, these offset the axial trace of the anticline. The WG Fault forms the boundary between the WG and BR Gold Systems. Displacement along the WG Fault indicates roughly 50 m of normal, west side down movement, and approximately 30 m of right lateral movement.

Turbiditic rocks in the hinge zone of the Upper Seal Harbour Anticline have been variably altered with carbonate, sericite, sulphide, tourmaline, and chlorite assemblages that post-date growth of regional metamorphic mineral assemblages. The nature of alteration varies as a function of lithology and proximity to mineralization. Greywacke/sandstone units have varying amounts of biotite and muscovite that have likely detrital, metamorphic, and alteration origins. The greywacke and quartz-rich units generally exhibit weaker alteration than the finer argillite /mudstone units but when altered the greywacke/quartz-rich units exhibit bleaching that consists of both albite and sericite alteration. These units also exhibit irregular quartz alteration proximal to cleavage fractures in the rock; these zones also arsenopyrite in some instances.

In contrast, the siltstone/mudstone/argillite units exhibit the greatest changes in alteration mineralogy proximal to veins. Background siltstones are generally layered and laminated and are brown-green with minor biotite and chlorite, whereas proximal to well mineralized veins they exhibit black to black-green colouration and are pervasively altered to chlorite with biotite, sericite, albite, quartz, carbonate, and sulphide. Often these zones have chlorite-biotite, as well as carbonate spots, and they are cut by quartz veins. Further, they ubiquitously have arsenopyrite proximal to veins that host mineralization and in the various belts; arsenopyrite ranges from mm-scale up to several centimetres and locally contains pressure shadows with quartz ±carbonate. The alteration extent within these argillites, however, is limited spatially (m-scale) due to individual beds having limited spatial extent. Despite their limited distribution the argillite beds are disproportionally veined compared to other rock types. The whole rock geochemistry of the argillites demonstrates gains in potassium oxide ($K_2O$), iron oxide ($Fe_2O_3$), sodium oxide ($Na_2O$), and aluminum oxide ($Al_2O_3$) proximal to mineralization and this decreases at distance from mineralization. Multi-element assays illustrate that locally these argillites are enriched in Au, arsenic (As), sulphur (S), lead (Pb), cadmium (Cd), iron (Fe), barium (Ba), potassium (K), sodium (Na), manganese (Mn), calcium (Ca), strontium (Sr), and phosphorus (P), particularly with increasing abundances of mineralization.

Gold and sulphide mineralization is associated with both wall rock and veins. Argillites contain diagenetic pyrite (locally framboidal), pyrrhotite, and arsenopyrite. There are several generations of veins with the majority of gold associated with vein generations later generations where gold occurs both in veins and wall rock, with the majority of coarse gold in veins associated with second generation arsenopyrite, galena, and to a lesser extent chalcopyrite and sphalerite. Microscopically, gold occurs as inclusions in arsenopyrite, often spatially proximal to galena inclusions. Gold also occurs as coarser grains or wires along grain edges and cracks in the arsenopyrite, indicative of potential coalescence and remobilization from grain interiors to grain margins.

Pyrrhotite is a commonly occurring sulphide phase in wall rock and typically is present as disseminated blebs, sometimes flattened in bands along foliation planes, or as irregular blebs at quartz vein contacts. Pyrrhotite also occurs in both wall rock and veins as a fracture coating phase and as very fine stringers. Chalcopyrite is almost exclusively confined to quartz veins and is present as fine-grained blebs concentrated along microfractures. Galena in small amounts is present in association with quartz vein hosted visible gold and within the wall rock. Sphalerite is rarely observed, but where present occurs as mm-scale fractures within quartz veins.

The gold mineralization observed in both core and microscopically is reflected in the multi-element geochemistry in the Deposit. Preliminary evaluations of the assay database illustrate that there are strong Au-As-S-Pb-Cd associations reflective of the mineralogy. There are also local enrichments in zinc (Zn), copper (Cu), Fe, nickel (Ni), and cobalt (Co) reflective of the presence of sphalerite, chalcopyrite, pyrite and pyrrhotite.

### 1.4.2 Deposit Type

The turbidite-hosted gold deposits of Nova Scotia have been compared to similar-age turbidite-hosted quartz vein deposits elsewhere in the world, particularly those in the Bendigo and Ballarat areas of the Lower Paleozoic Lachlan Fold Belt in the state of Victoria, Australia, and have historically been similarly classified. Robert et al. (1997) recognized this deposit class and proposed that the gold deposits of Nova Scotia be identified as a member of the ‘Turbidite-hosted, quartz-carbonate vein deposit (Bendigo Type)’ category. Ryan and Ramsay (1996) also addressed the similarity of Nova Scotia turbidite-hosted gold deposits with those in Victoria. As noted by Gervais et al. (2009), categorization within the USGS classification system of mineral deposits presented by Berger (1986) places the Deposit in the broad 36a category of ‘Low Sulphide Gold-Quartz Vein Deposits.’

The Deposit is a turbidite-hosted orogenic gold deposit hosted within a sequence of alternating argillites and greywacke metamorphosed to greenschist facies. These deposit types are typically characterized by the formation of gold bearing quartz veins within the argillite units, commonly referred to as mineralized belts (Belts), that are interbedded with greywacke units. There are currently 68 stacked mineralized belts ranging in thickness from 1 m to 20 m in the Deposit. The metasedimentary units on the Property are folded into the tight, gently east-plunging Upper Seal Harbour Anticline and gold mineralization has typically been deposited at various positions and times during the fold formation process. Veins, which form during deformation, occur in three major geometries commonly referred to as reefs: saddle reefs, leg reefs, and spur reefs. Saddle reefs occur about the apex of the fold and are the dominant vein types within some deposits. Leg reefs extend down the limbs of the fold, beyond the saddle reef, and are generally parallel with the metasedimentary layers. These are also commonly termed BP veins in the Nova Scotia goldfields. Spur reefs are veins that cross between layers and may be in the apex of the fold or on its limbs. This style of vein is in part captured under the term “angular” in the Nova Scotia goldfields.

The Deposit contains all three types of reefs outlined above but is also characterized by mineralization within the argillite forming the Belts. Because the Deposit contains saddle, leg, and spur reefs, and often has gold mineralization within the argillite hosting the veins, it has the potential to contain significantly more gold resources than deposits of a similar style that contain gold only within the quartz veins (reefs) themselves. The trace of the Upper Seal Harbour Anticline transects the Property and is found near the Dolliver Mountain prospect 2 km to the west of the Deposit, demonstrating that the structure which hosts gold continues for several kilometres.

## 1.5 Exploration

The Company acquired its interest in the Property early in 2017 under terms of a court approved Plan of Arrangement whereby Orex became a subsidiary of the Company. On this basis, work completed by Orex and others prior to the acquisition is considered historical in terms of current NI 43-101 technical reporting.

A summary of historical exploration was presented in Section 1.3. Work completed by the Company on the Property since its acquisition in 2017 includes the completion of 46,149.1 m of diamond drilling and three Mineral Resource Estimates. Additionally, the Company conducted an underground Bulk Sample from which a total of 13,028 tonnes of mineralized material was mined and stockpiled on surface with 10,023 wet metric tonnes (wmt) (9,785 dry metric tonnes [dmt]) shipped to the mill at Point Rousse near Baie Verte, NL, for processing into gold doré bars which were subsequently refined into bullion. The Company has also completed two phases of detailed metallurgical studies on both high-grade and low-grade mineralization from the Deposit and found recoveries averaging 96.4% for open pit and underground mineralized material.

In 2020 the Company retained Nordmin to conduct an assessment of the Project. Through an interactive process with the Company, Nordmin undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project.

The re-examination resulted in an additional 27,596.5 m of diamond drilling in 2020 and 2021, for incorporation into the geological model and represents the geological characteristics of the Deposit as observed in drill core and the Bulk Sample. Furthermore, this program determined the importance of low-grade mineralization associated with and adjacent to high-grade mineralization.

The results of this analysis and modelling are the subject of this Technical Report, which includes a Mineral Resource Estimate with an effective date of November 15, 2021.

## 1.6 Drilling

Work completed by the Company on the Property since its acquisition in March of 2017 includes several years of drilling programs, the completion of several Mineral Resource updates and associated technical reports.

A total of 65,968 m of surface and underground diamond drilling was completed between 1984 and 2011. Orex was corporately involved in all programs from 1988 through 2011, and earlier programs were carried out by Onitap, Petromet Resources Ltd., and Greenstrike Gold Corp. In 2010, reverse circulation (RC) drilling equipment was used by Osisko to explore near surface gold mineralized structures on the Property by recovering basal till and bedrock samples for gold assaying and whole rock analysis. The program consisted of 64 RC drill holes completed in the EG, BR Ramp, and WG Areas. Assay results from the RC drill program were not used for the Mineral Resource Estimate.

The Company has completed a total of 55,803.0 m of diamond drilling on 299 drill holes since acquiring the Project in 2017. Drilling since 2017 has largely been focused on infill and expansion drilling designed to update and upgrade the Mineral Resource at the Project as well as collect samples for metallurgical testing.

In addition to the drilling and associated metallurgical programs, the Company retained Nordmin in 2020 to conduct extensive remodelling of the Deposit geology and to also model low-grade mineralization found within the altered wall rock adjacent to high-grade veins.

All drilling completed for the Company from 2017 to 2020 was provided by Logan Drilling, recovering NQ, or HQ size core using conventional wireline drilling equipment. Core logging, geological interpretations and mineralogical/geochemical studies, core sampling, downhole surveying, and collar location surveying was completed in the same manner for each program under the project supervision of Mr. Paul McNeill, P.Geo., Mr. Steve Barrett, P.Geo., Ms. Tanya Tettelaar, P.Geo., Ms. Alana Haysom, P.Geo. and Mr. David A. Copeland, P.Geo., all employees of the Company, and geological consultant Dr. Stephen Piercey, P.Geo. Downhole orientation surveys were conducted under supervision of site technical staff using a Reflex downhole instrument at nominal 30 m intervals. Drill collars were surveyed using a differential GPS by Company employees or contractors.

In 2017, the Company completed diamond drilling in 13 drill holes (BR-17-01 to BR-17-13) totalling 4,196.3 m. The first five drill holes of the program were designed to acquire samples for metallurgical testing, verify historical drilling, and test the potential extents of the Deposit at depth.

During 2018 the Company completed 61 drill holes (BR-18-17 to BR-18-71) totalling 18,277.3 m focused on infilling areas of Inferred resources as outlined in the 2018 Preliminary Economic Assessment (2018 PEA) filed on March 2, 2018 and expanding the Deposit along strike and down plunge, and at depth along the host fold structure. Drilling focused on testing the down plunge, down dip, and along strike extension of the BR Gold System, EG Gold System, and WG Gold System. In addition, several holes tested the depth extent of the BR Gold System to depths of 525 m.

During 2019 the Company completed 33 drill holes (BR-19-72 to BR-19-104) totalling 5,733.8 m with the purpose of both infilling certain portions of the Deposit while expanding the Deposit eastward.

Infill drilling at the BR Gold System consisted of drilling select areas in order to upgrade from Inferred Mineral Resources to Measured and Indicated Mineral Resources. Infill and expansion drilling of the near surface mineralization potential of the EG Gold System in proximity to the optimized open pit shell as well as deeper exploration holes successfully intersected gold mineralization in all drill holes.

From June 2020 to September 15, 2021, the Company completed 192 drill holes (BR-20-105 to BR-20-295) on the Property totalling 27,595.7 m of drilling. The 2020 and 2021 programs focused on targeting under-drilled areas of the Deposit to upgrade Mineral Resources from the Inferred to Indicated and Measured Resource categories within the WG, BR and EG Gold Systems with a focus on testing near surface mineralization within conceptual open pits as part of the Feasibility Study (FS). Drilling also focused on testing areas with the conceptual open pit that had seen little historical drilling.

Representative assays (core length) from the diamond drilling program from 2017 to 2021 include:

- 34.70 g/t gold over 3.5 m (82.0 m to 85.5 m) in hole BR-17-09.
- 24.34 g/t gold over 3.8 m (389.9 m to 393.7 m) in hole BR-17-06.
- 252.76 g/t gold over 0.4 m (76.6 m to 77.0 m) in hole BR-18-15.
- 31.04 g/t gold over 1.0 m (6.2 m to 7.2 m) in hole BR-18-17.

- 11.27 g/t gold over 13.5 m (201.0 m to 214.5 m), including 15.63 g/t gold over 1.4 m and 44.33 g/t gold over 2.5 m in hole BR-18-22.
- 4.13 g/t gold over 20.5 m (324.5 m to 345.0 m), including 9.93 g/t gold over 7.5 m and 79.34 g/t gold over 0.5 m in hole BR-18-23.
- 10.55 g/t gold over 6.1 m (223.0 m to 229.1 m), including 18.78 g/t gold over 3.1 m in hole BR-18-22.
- 5.10 g/t gold over 9.6 m (116.0 m to 125.6 m), including 25.82 g/t gold over 1.5 m in hole BR-18-22.
- 7.22 g/t gold over 6.5 m (310.5 m to 317.0 m), including 16.00 g/t gold over 2.0 m in hole BR-18-23.
- 752.54 g/t gold over 0.5 m (145.0 m to 145.5 m) in hole BR-18-25.
- 56.67 g/t gold over 1.0 m (132.5 m to 133.5 m) in hole BR-18-25.
- 23.24 g/t gold over 2.5 m from (21.5 m to 24.0 m) in hole BR-18-28.
- 7.12 g/t gold over 4.5 m from (193.5 m to 194.0 m) in hole BR-18-29.
- 2.21 g/t gold over 25.5 m (506.1 m to 531.6 m), including 12.39 g/t gold over 3.2 m in hole BR-18-30.
- 77.69 g/t gold over 0.5 m (64.5 m to 65.0 m) in hole BR-18-42.
- 32.42 g/t gold over 2.6 m (300.3 m to 302.9 m), including 201.68 g/t gold over 0.4 m in hole BR-18-59.
- 8.79 g/t gold over 8.0 m (483.0 m to 491.0 m), including 64.40 g/t gold over 0.8 m in hole BR-18-44.
- 5.36 g/t gold over 2.2 m (321.1 m to 323.3 m) in hole BR-19-75.
- 102.43 g/t gold over 0.7 m (142.0 m to 142.7 m) in hole BR-19-86.
- 72.40 g/t gold over 0.6 m (21.0 m to 21.6 m) in hole BR-19-87.
- 28.52 g/t gold over 2.0 m (125.6 m to 127.6 m), including 112.87 g/t gold over 0.5 m in hole BR-20-146.
- 1.85 g/t gold over 15.0 m (184.0 m to 199.0 m), including 29.13 g/t gold over 0.5 m and 9.12 g/t gold over 0.5 m in hole BR-20-147.
- 6.63 g/t gold over 5.3 m from (35.7 m to 41.0 m), including 47.67 g/t gold over 0.5 m in hole BR 20-134.
- 6.05 g/t gold over 11.7 m (189.9 m to 201.6 m), including 12.55 g/t gold over 2.5 m and 5.52 g/t gold over 7.2 m (210.8 m to 218.0 m) in hole BR-20-142.
- 7.76 g/t gold over 4.4 m (39.8 m to 44.2 m), including 22.07 g/t gold over 1.0 m and 9.42 g/t gold over 1.0 m in hole BR-20-135.
- 1.34 g/t gold over 18.7 m (97.3 m to 116.0 m), including 6.41 g/t gold over 1.0 m in hole BR 20-135.
- 1.86 g/t gold over 14.5 m (55.5 m to 70.0 m) and 2.78 g/t gold over 4.5 m (44.0 m to 48.5 m), and 1.31 g/t gold over 5.5 m (19.0 m to 24.5 m) in hole BR-20-221.
- 1.37 g/t gold over 21.5 m (72.0 m to 93.5 m), including 26.80 g/t gold over 0.5 m in hole BR-20-219.

- 2.08 g/t gold over 9.0 m (130.0 m to 139.0 m), including 12.20 g/t gold over 1.0 m in hole BR-20-216.
- 1.61 g/t gold over 14.7 m (64.9 m to 79.6 m) including 5.49 g/t gold over 1.5 m in diamond drill hole BR-21-285.
- 1.71 g/t gold over 9.5 m (91.2 m to 100.7 m) including 9.78 g/t gold over 1.0 m in diamond drill hole BR-21-285.
- 16.09 g/t gold over 1.5 m (87.3 m to 88.8 m) in diamond drill hole BR-21-271.
- 1.33 g/t gold over 4.5 m (144.5 m to 149.0 m) in diamond drill hole BR-21-259.
- 1.20 g/t gold over 5.0 m (156.0 m to 161.0 metres) in diamond drill hole BR-21-25;
- 7.88 g/t gold over 7.9 m (364.3 to 372.2 m), including 21.38 g/t gold over 1.5 m and 17.32 g/t gold over 1.5 m in diamond drill hole BR-21-291.
- 6.19 g/t gold over 2.6 m (94.6 m to 97.2 m), including 24.80 g/t gold over 0.6 m in diamond drill hole BR-21-299.
- 3.67 g/t gold over 4.2 m and 14.10 g/t gold over 0.5 m within a broader zone consisting of 1.91 g/t gold over 12.6 m (279.4 m to 292.0 m), in diamond drill hole BR-21-295.

## 1.7 Sampling, Analysis, and Data Verification

### 1.7.1 Sampling and Analysis

Drill holes from programs completed between 1984 and 2011 are included in the current Mineral Resource Estimate database. The sampling approaches in programs carried out prior to 2005 generally reflect sampling of visibly determined Belts, respective of major geological units, plus varying amounts of adjacent material. Exceptions to this, which include continuous core sampling and/or total core rather than half core sampling, pertain to certain historical metallurgical programs. Continuous mineralized zone sampling, respective of major lithologic units, pertains to 2005, and later programs.

Drill core samples from surface drilling programs carried out in 2005 (HQ core) and 2008 (NQ core) were generated by Orex during this period. Samples were sent to ALS facilities in either Val-d'Or, Québec (2005) or Timmins, Ontario (2008) (ALS is independent of the Company). Standard rock sample crushing and grinding procedures at ALS were followed by initial FA fusion-FA finish analysis of 50 g pulp splits.

If the initial result met or exceeded a 2.5 g/t gold threshold, analysis of a second coarse reject split was carried out using a gravimetric finish. Composite metallurgical samples were created from coarse reject materials selected by Orex consultants. These were submitted to SGS Lakefield (SGS is independent of the Company) for whole sample metallurgical testing. A quality assurance (QA) and quality control (QC) program that included analysis of Certified Reference Material (CRM), field duplicates, coarse reject duplicates, pulp split duplicates, and blank samples was carried out with respect to both the 2005 and 2008 programs, and results of these programs are presented in the report.

The 2010 to 2011 Osisko program was conducted and included drilling of NQ sized core that was logged, photographed, sampled, bagged, tagged, and sealed at the Goldboro site by qualified persons. Logging utilized Gemcom Gems™ Logger software, and project protocols included progressive, systematic, and secure off site backup of digital drilling, logging, and sampling data. At ALS, each sample was crushed to 70% < 2 mm, split to 250 g using a riffle splitter, pulverized to 85%

at < 0.075 mm, and made into a 50 g sample of the pulp. The 50 g pulp was fire assayed with atomic absorption spectrometry (AAS) finish (ALS codes Au-AA24 and Au-AA26). Samples exceeding the AAS threshold were re-assayed using a gravimetric finish (ALS code Au-GRA22). All samples containing visible gold were directly assigned for processing using the total metallic screen method with FA-AA or gravimetric finish.

A review of assessment reporting related to the various drilling programs carried out during the 1984 to 2005 period showed that, with the exception of a metallurgical and check sampling program carried out in 1995, no structured programs designed to systematically monitor QA/QC issues for drill core were in place. Orex drilling programs in 2005 and 2008 and Orex-Osisko programs in 2010 and 2011 were subject to rigorous QA/QC programs, with some procedural changes incorporated during the period.

During 2017 to the effective date of the current Mineral Resource Estimate, drill core samples were collected systematically down the hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 m to 1.0 m depending on the nature and width of veining and mineralization samples, while trying to best honour geological contacts. Samples were collected of quarter-sawn drill core and shipped to Eastern Analytical (who is independent of the Company) for analysis via standard 30 g FA with AA finish. Samples were also analyzed at Eastern Analytical via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold, and select samples were submitted for 34-element ICP analysis.

Sample bags are sealed with zip ties to ensure sample integrity and securely shipped to Eastern Analytical for analysis. Drill core is stored in racks at the Company core storage facility at the Project site. Security of site operations, core, samples, and core storage are addressed on an ongoing basis by site staff.

### 1.7.2 Data Verification

Core sample records, lithologic logs, laboratory reports and associated drill hole information for all drill programs completed in the 1984 to 2011 period were digitally compiled for use in Gemcom-Surpac Version 6.2.1® (SurpacTM) deposit modelling software. Historical and current drilling program information was reviewed, and digital records of historical drilling were checked for both consistency and accuracy against original source documents available through Nova Scotia Department of Natural Resources (NSDNR) or received from Orex. All 2010 and 2011 drill hole coordination and orientation data inputs were checked, and validation of approximately 20% of the assay dataset for sample interval and assay value information against corresponding source documents was carried out.

From 2011 until current, all drill hole data was compiled into a validated Microsoft Access® database that Nordmin reviewed digitally using a combination of Datamine and Target software programs.

The QP completed a spot check verification on the Project of:

- Drill holes—62 (12%) of the lithologies, 1,042 (10%) of the geotechnical measurements, 3,843 (8%) of the assays.
- Chip samples—84 (6%) of the lithologies, 168 (12%) of the assays.

The geology was validated for lithological units from the Company's Geovia GEMS logger. The geological contacts and lithology are aligned with the core contacts and lithology and are acceptable for use.

## 1.8 Mineral Resource and Mineral Reserve Estimates

### 1.8.1 Mineral Resource Estimate

Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project. The Deposit consists of three domains referred to as the BR, EG, and WG Gold Systems. The WG Gold System is separated from the BR Gold System by a north trending, near vertical fault with tens of metres of apparent offset. The EG Gold System is separated from the BR Gold System by a thick greywacke sequence or marker unit. Stratigraphic younging is from west to east with the anticlinal fold plunging shallowly to the east.

From a modelling perspective, each individual Gold System in the Deposit was separated into its own domain. Each domain was further sub-domained into Higher-Grade Belts and Lower-Grade Domains.

Detailed wireframing was completed based on plan-oriented sections to mirror likely mining patterns based on the geometry of the Deposit. Special attention was given to consistent smoothing of the wireframe linework to mimic the underlying geological controls on mineralization, including geological bedding, regularly dipping north, and south limbs of the large-scale anticlinal fold geometry and down the plunge of the anticline. Historical workings of three underground mines, which traced the outline of the fold geometry down the fold plunge and along anticlinal limbs coincident with gold mineralization were also used to orient wireframes. All wireframes are independent of each other without overlap across wireframes or across domains.

Explicit modelling was used to create the Mineral Resource, which allows for mineralization to better reflect the Deposit geology and associated geochemistry.

Multiple test scenarios were evaluated to determine the optimum processes and parameters to use to achieve the stated criteria. Each scenario was based on nearest neighbour (NN), inverse distance squared (ID2), inverse distance cubed (ID3), and ordinary kriging (OK) interpolation methods.

All test scenarios were evaluated based on global statistical comparisons, visual comparisons of composite samples versus block grades, and the assessment of overall smoothing. Based on results of the testing, it was determined that all scenarios including the draft and final resource estimation methodology would constrain the mineralization by using hard wireframe boundaries to control the spread of high-grade and low-grade mineralization. OK was selected as the most representative interpolation method as the most representative of all domains in the Project.

Block models were defined with parent blocks at 2.0 m x 2.0 m x 2.0 m (Northing x Easting x Elevation). All wireframe volumes were filled with blocks from the prototype. Block volumes were compared to the wireframe volumes to confirm there were no significant differences. Block volumes for all wireframes were found to be within reasonable tolerance limits. Sub-blocking was allowed to maintain the geological interpretation and to accommodate the Higher-Grade Belts and Lower-Grade Domains (wireframes), the specific gravity (SG), and the category application. Sub-blocking has been allowed to the following minimums:

- 2.0 m x 2.0 m x 2.0 m blocks are sub-blocked two-fold to 0.5 m x 0.5 m in the N-S and E-W directions with a variable elevation calculated based on the other sizes.

Block models were not rotated nor clipped to topography. Because dynamic anisotropy requires the full, folded wireframes for calculation, blocks were permitted to estimate above surface but had an "air" code applied and were removed from reporting. The Mineral Resource Estimate was conducted using Datamine Studio RM™ version 1.8.32.0 within the North American Datum 1983 (NAD83) Modified Transverse Mercator (MTM) Zone 4 datum.

Four block models were independently estimated, WG, EG, the Marker Horizon unit, and BR. These then had extraneous fields removed and were combined into one overall resource block model.

The Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines and have an effective date of November 15, 2021. The Project hosts:

- Total Open Pit (at a 0.45 g/t cut-off) and Underground (at a 2.40 g/t cut-off) Mineral Resources including 9,255,000 tonnes and 1,057,963 oz of Measured Resources grading 3.56 g/t gold, 12,338,000 tonnes and 1,523,014 oz of Indicated Resources grading 3.84 g/t gold, and 3,181,000 tonnes and 484,250 oz of Inferred Resources grading 4.73 g/t gold.
- Open Pit Mineral Resources (at a 0.45 g/t cut-off) including 7,680,000 tonnes and 680,518 oz of Measured Resources grading 2.76 g/t gold, 7,988,000 tonnes and 741,220 oz of Indicated Resources grading 2.89 g/t gold, and 975,000 tonnes and 66,237 oz of Inferred Resources grading 2.11 g/t gold.
- Underground Mineral Resources including 1,576,000 tonnes and 377,445 oz of Measured Resources grading 7.45 g/t gold, 4,350,000 tonnes and 781,794 oz of Indicated Resources grading 5.59 g/t gold, and 2,206,000 tonnes and 418,013 oz of Inferred Resources grading 5.89 g/t gold.

The Mineral Resource Estimate presented in Table 1-1 is based on validated results of 681 surface and underground drill holes, for a total of 120,550 m of diamond drilling completed between 1984 and the effective date of November 15, 2021, as well as 1,230 chip samples comprised of 822.7 m from the Bulk Sample (2018 to 2019). The Mineral Resource Estimate includes 7,488.3 m of diamond drilling in 62 drill holes since the Previous Mineral Resource Estimate effective February 7, 2021. Nine drill holes totalling 1,001.9 m were removed from the database due to inconsistent sample lengths.

**Table 1-1: Mineral Resource Estimate, Open Pit (0.45 g/t Cut-off) and Underground (2.40 g/t Cut-off)**

| Resource Type | Gold Cut-off (g/t) | Category | Tonnes ('000) | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|---|
| Open Pit | 0.45 | Measured | 7,680,000 | 2.756 | 680,518 |
| | | Indicated | 7,988,000 | 2.886 | 741,220 |
| | | Measured + Indicated | 15,668,000 | 2.822 | 1,421,738 |
| | | Inferred | 975,000 | 2.113 | 66,237 |
| Underground | 2.40 | Measured | 1,576,000 | 7.450 | 377,445 |
| | | Indicated | 4,350,000 | 5.590 | 781,794 |
| | | Measured + Indicated | 5,925,000 | 6.085 | 1,159,239 |
| | | Inferred | 2,206,000 | 5.893 | 418,013 |
| **Combined Open Pit and Underground*** | **0.45 and 2.40** | **Measured** | **9,255,000** | **3.555** | **1,057,963** |
| | | **Indicated** | **12,338,000** | **3.839** | **1,523,014** |
| | | **Measured + Indicated** | **21,593,000** | **3.718** | **2,580,977** |
| | | **Inferred** | **3,181,000** | **4.734** | **484,250** |

** Combined Open Pit and Underground Mineral Resources; The Open Pit Mineral Resource is based on a 0.45 g/t gold CoG, and the Underground Mineral Resource is based on 2.40 g/t gold CoG.*

**Mineral Resource Estimate Notes**

1. Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
2. Mineral Resources are inclusive of Mineral Reserves.
3. Open pit Mineral Resources are reported at a cut-off grade (CoG) of 0.45 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,600/oz) and metallurgical recovery factor of 89% around cut-off as calculated from ((GRADE-(0.0262*LN(GRADE)+0.0712))/GRADE*100)-0.083
4. Underground Mineral Resource is reported at a CoG of 2.40 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,600/oz) and a gold processing recovery factor of 97%.
5. Assays were variably capped on a wireframe-by-wireframe basis (Table 14-11, Table 14-12 and Table 14-13).
6. SG was applied using weighted averages to each individual wireframe.

7. Mineral Resource effective date November 15, 2021.
8. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
9. Excludes unclassified mineralization located within mined out areas.
10. Reported from within a mineralization envelope accounting for mineral continuity.

#### 1.8.1.1 Input Parameters for Mineral Resource Calculation

Open Pit

For the open pit Mineral Resource (Table 1-1) the economic limits for the two open pits were determined using Geovia's Whittle™ 4.7 software which uses the Lerchs-Grossmann (LG) algorithm. The LG algorithm progressively identifies economic blocks, taking into account waste stripping, resulting in the highest possible total value mined within the open pit shell subject to the specified pit slope constraints. The pit limit analysis was performed on the resource block model. The parameters used to generate the pit shell are shown in Table 1-2.

**Table 1-2: Open Pit Limit Analysis Parameters**

| Parameter | Value |
|---|---|
| Currency Used for Evaluation | C$ |
| Block Size | In-Situ model regularized to 2.0 m x 2.0 m x 4.0 m |
| Overall Stope Angle | Rock: Varied by Sector, Range 42°- 46°<br>Overburden: 25° |
| Open Pit Mining Cost | 0.8 Mining Cost Adjustment Factor (MCAF) for Overburden<br>$5.10/t_{mined}$ Rock<br>+$0.016/t per 8 m |
| Process Cost<br>*Includes assumptions for Milling, G&A, tailings, and rehabilitation* | $25.75/t_{processed}$ |
| Selling Cost<br>*Includes doré transportation, refining, and royalty* | $5/oz |
| Percent Payable | 99.95% |
| Metal Price | US$1,600 per ounce of gold<br>US$1:C$1.25<br>C$2,000 per ounce of gold |
| Process Recovery | Based on Grade – Recovery Curve:<br>$\frac{Block\ Grade - (0.0262 \times ln(Block\ Grade) + 0.0712)}{(Block\ Grade \times 100) - 0.083}$ |
| Mining Loss & Dilution | Included within Reblocked/ Regularized Block Model<br>Plus 5% factor for mining loss within optimization program<br>Overall effect of ~26% additional tonnes and ~8% reduction in metal |
| Resources Used to Generate Pit Shell | Measured + Indicated (no Inferred Resources were used to create the open pit physical limits) |
| Pit Shell Selection | Revenue Factor (RF) 0.80 for Mine Planning |
| Production Rate Assumption | 4,000 tonnes per day (t/d) |

Three boundary constraints were used in the pit limit analysis for the Deposit:

- A 40 m (X-Y) offset from the Natural Gas pipeline easement, on the west side of the property;
- A 50 m (X-Y) offset from the edges of the Gold Brook Lake; and
- A 20 m (X-Y) offset from the centerline of Gold Brook.

The block models were created in Datamine using 2 m x 2 m x 2 m parent cell and variable sub-celling to 1 m. For the open pit evaluation, the resource block model in Datamine format was reblocked to a regularized block model in Datamine format using Deswik.CAD. Default waste blocks and overburden blocks were added to the model. The envisioned selective mining excavator, at the onset of the analysis, will likely have a bucket width of approximately 2 m. Mining is planned at an 8 m operating bench height.

To classify the material contained within the open pit limits as material for processing or material for waste, the milling cut-off grade is used. This break-even cut-off grade is calculated to cover the costs of processing, general and administrative costs, and selling costs using the economic and technical parameters listed in Table 1-2. Mineral Resource material contained within the pit and above the cut-off grade is classified as potential mill feed (PMF), while resource material below the cut-off grade is classified as waste. The cut-off grade has been estimated to be 0.45 g/t gold for the open pit.

Underground

For the underground Mineral Resource analysis, parameters used to calculate the CoG are shown in Table 1-3. The underground Mineral Resource CoG is estimated to be 2.40 g/t gold. The Mineral Resource Estimate excludes unclassified mineralization located within mined out areas.

**Table 1-3: Underground Limit Analysis Parameters**

| Parameter | Value |
|---|---|
| Currency Used for Evaluation | C$ |
| Block Size | In-Situ sub-blocked model with parent blocks at 2.0 m x 2.0 m x 2.0 m |
| Underground Mining Cost<br>*Includes assumptions for operating waste development, surface rehandle* | $96.25/t$_{processed}$ |
| Process Cost<br>*Includes assumptions for Milling, G&A, tailings, indirect costs* | $44.30/t$_{processed}$ |
| Underground Support Cost<br>*Includes assumptions for sustaining underground capital, infill diamond drilling* | $22.50/t$_{processed}$ |
| Selling Cost<br>*Includes doré transportation, refining, and royalty* | $24.84/troy ounce |
| Percent Payable | 99.95% |
| Metal Price | $1.550 US$ per troy ounce<br>Exchange Rate: 1 US$=1.3 C$<br>$2.000 C$/troy ounce (rounded) |
| Process Recovery | 97% |
| Production Rate Assumption | 1,200 t/d |

### 1.8.2 Mining and Mineral Reserve Estimate

#### 1.8.2.1 Mining Methods

Conventional open pit mining methods will be used to extract a portion of the Deposit. This method was selected considering the deposit’s size, shape, orientation, and proximity to the surface. Drilling, blasting, loading, and hauling will be used to mine the open pit material within the designed pit to meet the mine production schedule.

Open pit mining will include conventional drilling and blasting with a combination of a backhoe type excavator, hydraulic excavator, and front-end loader type excavator loading broken rock into haul trucks, which will haul the material from the bench to the crusher, run of mine (ROM) stockpile or waste stockpiling areas depending on the material type. Ancillary equipment includes dozers, graders, and various maintenance, support, service and utility vehicles. This Technical Report considers a mining contractor operator scenario.

The FS is based on a conventional truck-shovel open pit mining operation within two pits. The open pit production period is approximately 10.9 years with 1 year of pre-production (prior to process plant start-up). It is envisaged that the PMF will be loaded directly into the processing plant crusher hopper but there will be a need for a ROM stockpile to allow for stoppages, for stockpiling in the pre-production period, and possibly some blending. The operation scenario for the FS involves:

- Open pit mining at an average mining rate of 12.8 Mt per year.
- Gold process facility with a 1.46 Mtpa (4,000 t/d) capacity.

- Approximate 6 month ramp up period in Year 1 (YR1) for process facility.
- 1 year pre-production mining period to coincide with the tailings management facility (TMF) Initial Stage development.

#### 1.8.2.2 Mineral Reserve

The Mineral Reserve Estimate for the Project is reported using the May 10, 2014, Standards for Mineral Resources and Mineral Reserves and the 2019 CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019).

Mineral Reserves are based on the engineering and economic analysis described in Sections 16 to Section 22 of this Technical Report. Changes in the following factors and assumptions may affect the Mineral Reserve Estimate:

Factors that may affect the Mineral Reserve:

- Metal prices
- Interpretations of mineralization geometry and continuity of mineralization zones
- Kriging assumptions
- Geomechanical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate
- Operating cost assumptions
- Process plant recoveries
- Mining loss and dilution
- Ability to meet and maintain permitting and environmental licence conditions
- Historical mining depletion

Nordmin prepared a Mineral Reserve Estimate for the Project using a combination of Geovia's Whittle 4.7.4 and Geovia's Surpac 2021 software packages for estimating the economic pit limit for the open pit and block model interrogation.

The Mineral Reserve Estimate for the Deposit is based on the resource block model estimated by Nordmin and described in Section 14. The block model contained Measured, Indicated and Inferred Mineral Resources, however only Measured and Indicated Mineral Resources were used. Inferred Mineral Resources in the block model were not included in the Probable Mineral Reserve and remain classified as waste; Inferred Mineral Resources do not meet the standards required for inclusion in Mineral Reserves.

Mineral Reserves for the Deposit incorporate mining dilution and mining loss assumptions for the open pit mining method.

The reference point at which Mineral Reserves are defined, is the point where the ore is delivered to the processing facility, which includes the ROM stockpile.

The following subsections outline the procedures used to estimate the Mineral Reserves. A mineral Reserve Estimate of 15.8 Mt of Probable Reserve grading 2.26 g/t gold resulted from the detailed design of the final pit and a LOM scheduling with the cut-off grade (Table 1-4).

**Table 1-4: Mineral Reserve Estimate**

| Category | Area | Au Cut-off Grade | Tonnage (t) | Diluted Au Grade (g/t) | Contained Au Metal (oz) |
|---|---|---|---|---|---|
| Probable Mineral Reserve | East Pit | 0.45 g/t | 5,468,300 | 2.54 | 446,000 |
| Probable Mineral Reserve | West Pit | 0.45 g/t | 10,330,600 | 2.12 | 704,200 |
| Probable Mineral Reserve | Overall Total | 0.45 g/t | 15,798,900 | 2.26 | 1,150,200 |

Notes:

- The independent and Qualified Person for the Mineral Reserve Estimate, as defined by NI 43-101, is Joanne Robinson, P.Eng. of Nordmin Engineering Ltd.
- The effective date of the Mineral Reserves estimate is December 15, 2021.
- The Mineral Reserve Estimate is based metallurgical recovery algorithms, that result in an overall average recovery of 95.8%.
- Metal prices are set at US$1,600/oz Au with an exchange rate assumption of 1US$:1.25C$ resulting in C$2,000/oz.
- The Mineral Reserve was derived from a pit limit analysis and detailed pit design. A cut-off grade of 0.45 g/t was based on parameters described in Table 15-2.
- The Mineral Reserve Estimate incorporates mining dilution and mining loss assumptions through regularization of block size to 2 m x 2 m x 4 m. An additional 5% mining loss assumption was incorporated. The overall impact is approximately 26% additional tonnes and approximately 8% reduction in Au Metal.

## 1.9 Metallurgical Testwork

The testwork relied upon in this Project includes a recent metallurgical program completed in 2021 as well as historical data previously published. The recent 2021 programs included additional comminution testing for Bond Ball Mill Work Index, as well as cyanide destruction, arsenic precipitation, tailings thickening and rheology.

In the 2021 metallurgical program, twenty four samples, representing mineralization from the two open pits were selected from available NQ drill core to provide spatial representation to assess variability of the hardness as measured in the Bond ball mill work index. The Bond ball mill work index results are characterized as hard, with 75$^{th}$ percentile value of 15.5 kWh/t. The average of 15.0 kWh/t is slightly lower than the average from the 2020 program which was 15.2 kW/t. The 75$^{th}$ percentile value from the 2020 program was 15.7 kWh/t, also slightly higher

A Master Composite Sample, assembled from drill core from both open pits, was submitted for gravity concentration followed by cyanidation of gravity tailings testing in a stirred reactor. Sampleswere also used to measure leach kinetics at specified increments of 2, 6, 8, 24 and 36 hours, at which point the leach was terminated.

Gravity concentration recovered 64.3% of the contained Au with a mass recovery of 0.057%. Cyanidation of the gravity tailings recovered an additional 33.5% of the contained Au for an overall recovery of 97.8% Au. The calculated head grade from the sample was 5.05 g/t Au, which was higher than the planned grade of 1.65 g/t Au. The test was run with low cyanide concentrations which returned a sodium cyanide consumption of 0.19 kg/t. The test also validated the design leach retention time of 36 hours.

Cyanide destruction testing using the $SO_2$/air method was completed to determine process design requirements to achieve a discharge weak acid cyanide concentration ($CN_{WAD}$) of less than 0.5 mg/L and total cyanide ($CN_{TOT}$) of 0.5 mg/L. The required conditions to achieve this target include 120 minutes of retention time using an addition ratio of 10 g $SO_2$/g $CN_{WAD}$. Arsenic precipitation of the cyanide destruction product with ferric sulphate reduced arsenic in solution to below 0.5 mg/L and is in line with industrial practice.

Solid/liquid separation testwork was performed on Master Composite detoxified final leach tailings to determine tailings thickener requirements. The program included both static and dynamic tests. The required underflow density for tailings deposition is 60% solids (w/w). The conditions that meet this requirement and provide acceptable overflow clarity provide a settling rate of 0.7 t/m²/h with 50 g/t flocculant addition at pH 10.

## 1.10 Recovery Methods

The process plant was designed using conventional processing unit operations. It has been designed to treat up to 4,000 t/d based on an availability of 92% or 8,059 hours per year. The crushing plant section design is set at 64% availability and the gold room availability is set at 52 weeks per year. The plant will operate two shifts per day, 365 days per year, and will produce doré bars.

The process plant includes the following:

- Three stages crushing of ROM material
- A covered, crushed material stockpile to provide buffer capacity ahead of the grinding circuit
- Ball mill with trommel screen followed by cyclone classification
- Gravity recovery of ball mill discharge followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution
- Trash screening
- Leach + carbon adsorption (L/carbon-in-pulp (CIP))
- Acid washing of loaded carbon and Pressure Zadra type elution followed by electrowinning and smelting to produce doré
- Carbon regeneration by rotary kiln
- Cyanide destruction of tailings using the $SO_2$/air process followed by arsenic precipitation
- Carbon safety screening
- Tailings thickening
- Reagent storage and distribution
- Water and air services
- Potable water treatment and distribution

Key process design criteria listed in Table 1-5 were derived from testwork presented in Section 13 and previously summarized in Section 1.9.

**Table 1-5: Key Process Design Criteria**

| Design Parameter | Units | Value |
|---|---|---|
| Plant Throughput | t/d | 4,000 |
| Gold Grade – Design Mill Head | g/t | 2.58 |
| Crushing Plant Availability | % | 64 |
| Mill Availability | % | 92 |
| Bond Crusher Work Index (CWi), 75th percentile | kWh/t | 23 |
| Bond Rod Mill Work Index (BWi), 75th percentile | kWh/t | 17.6 |
| Bond Ball Mill Work Index (BWi), 75th percentile | kWh/t | 15.7 |
| SMC Axb, 25th percentile | - | 30.4 |
| Bond Abrasion Index (Ai) | g | 0.228 |
| Material Specific Gravity | $t/m^3$ | 2.75 |
| Primary Grind size (P80) | µm | 100 |
| Primary Crusher | - | Jaw, 1 m x 1.3 m |
| Secondary Crusher | - | Standard Cone, 1.32 m diam. |
| Tertiary Crusher | - | Shorthead Cone, 1.32 m diam. |
| Ball Mill Dimensions | - | 5.2 m diam. x 7.9 m EGL |
| Ball Mill Installed Power | MW | 3.5 |
| Leach Residence Time | h | 30 |
| CIP Residence Time | h | 6 |
| Gravity Gold Recovery (design) | % Au | 40 |
| Total Gold Recovery (life of mine) | % Au | 96 |
| Leach pH target range | - | 10.5-11 |
| Leach-CIP Operating Density | % w/w solids | 44 |
| Leach Sodium Cyanide Addition | kg/t | 0.5 |
| Leach Hydrated Lime Addition | kg/t | 1.0 |
| Leach & CIP Tanks | # | 3 + 6 |
| Tonnes of Carbon per Elution Column | t | 3 |
| Detoxification Residence Time | min | 120 |
| Detoxification Tanks | # | 2 (Parallel) |
| Detoxification $SO_2$ Addition (as SMBS) | $SO_2$:$CN_{WAD}$ ratio(w/w) | 10 |
| Detoxification Lime Addition | kg/t | 0.80 |
| Detoxification Discharge $CN_{WAD}$, Design | mg/L | <0.5 |
| Detoxification Discharge $CN_{TOT}$, Design | mg/L | 0.5 |
| Arsenic Precipitation Residence Time, Design | min | 10 |
| Ferric Sulphate Addition Ratio | Fe:As Ratio (w/w) | 10 |
| Thickener Underflow Density | % w/w solids | 60 |

Source: Ausenco, 2021

## 1.11 Infrastructure

The main Project infrastructure components include the mine and process plant supporting infrastructure, site accommodation facilities, TMF, external and internal access roads, power supply and distribution, freshwater supply and distribution, and the water treatment plant.

The Property is situated on the eastern shore of Nova Scotia, Canada, with the central point of the Property being approximately located at 45° 12’ 2.6” N latitude and 61° 39’ 2.0” W longitude.

The Property will have access to the substantial infrastructure, services, and skilled labour in the area. There will be reduced infrastructure cost requirements due to its location near Route 316 compared to a remote mine site location. The Property is approximately 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro, on the eastern shore of Isaac’s Harbour, in Guysborough County, Nova Scotia, Canada. A secondary gravel road (Goldbrook Road), accessed from Route 316, crosses the Property, and passes near the historic Boston Richardson shaft and exploration decline. Smaller logging roads and trails provide good access to most areas of the Property. The elevation is nominally 70 m above sea level. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

The majority of the earthworks will be realized in the preparation of the mine infrastructure, process plant and TMF infrastructures. Haulage roads on site will be built to withstand frequent heavy traffic between the proposed open pit, ROM stockpile and TMF. They will be wide enough to accommodate two trucks passing between the pits and ROM stockpile at 16.5 m with a grade no greater than 10%. The road to and from the tailing’s management facility will be 11 m wide for one-way traffic by haul trucks.

In total, approximately 5,300 $m^2$ of ancillary buildings (not including the employee accommodations and process plant buildings) have been provided.

These ancillary infrastructure buildings will be pre-engineered steel structures founded on conventional spread footing foundations. Space has been provided for future buildings provided by the mining contractor or in the case of expansion during operations.

Power for the site is anticipated to be provided from a nearby Nova Scotia Power 25 kV distribution line installed along Route 316. A 1.6 km tap line would be installed along a new right of way to the mine site main substation. Nova Scotia Power would upgrade their existing distribution system as necessary to be able to provide the additional power required. Peak power demand for the site is estimated to be 10 MW, with the average demand estimated to be 7.5 MW. A network of 13.8 kV overhead distribution lines would be installed at site to provide power sourced from the main substation for the mine and surface infrastructure

Water supply infrastructure includes one intake structure, two booster stations, one transmission watermain from Gold Brook Lake to the mill freshwater tank and to the potable water treatment unit; and distribution piping to supply potable water throughout the Project site (mill, emergency response transport (ERT) facility, plant office, general office, mine dry, core storage, truck shop and employee accommodation). A transmission watermain from Gold Brook Lake to the processing plant buildings is to provide a raw water source to support mill process operations and site wide potable water, hence the watermain flowrate was estimated based on the potable and process water demands (22 $m^3$/h).

Gold Brook Lake was considered as the source water, the treatment requirements were established based on the Canadian Drinking Water Guideline, and potable water treatment was sized assuming an equal flowrate for both potable water and wastewater (16 $m^3$/h).

Two separate wastewater treatment units were developed to service employee accommodation (with 350 people) and other buildings/facilities including mill, ERT facility, plant office, general office, mine dry, core storage, truck shop (with 84 people). Sewage flow rates as well as treatment requirements were adopted from the Atlantic Canada Wastewater Guidelines Manual for Collection, Treatment and Disposal, 2006.

The mine water management plan (MWMP) and associated design measures have been developed based on the proposed feasibility level mine site arrangement with inputs from the Company and the Consultants. The MWMP will be implemented during the construction phase and will be adjusted as necessary throughout the mine operations and closure phase.

Site contact water will be managed to meet the following regulatory discharge requirements prior to discharge to the natural environment:

- Metal and Diamond Mining Environmental Regulations (MDMER) Objectives
- Canadian Council of Ministers of the Environment (CCME) Guidelines for the Protection of Aquatic Life
- Tier 1 Nova Scotia Environment Quality Standards (EQS) for Surface Water
- Site specific criteria (based on background)

Based on predictive water quality modelling, it is understood that the water quality at some locations may be acceptable for discharge to the environment with total suspended solids (TSS) removal as the only form of treatment (where MDMER objectives are met). Where this is not the case, contingency measures (shutoff valves and pumps etc.) will be put in place to redirect water towards the nearest water treatment system (WTS) in case of exceedances. The primary objectives of the MWMP are as follows:

- Provide mechanism to dewater and treat ponded water within the Project Area to allow for development and excavation of mine infrastructure (e.g., pit, waste piles, haul road etc.).
- Capture, treat and provide controlled discharge for all site contact water during construction and operations.
- Divert all off site clean water away from the mine site infrastructure to reduce the total volume of water entering the settling ponds for treatment.

### 1.11.1 Tailings Management Facility

Knight Piésold Ltd. (Knight Piésold) completed a FS level design for the TMF at the Project. The TMF will provide secure storage for tailings, PAG1 waste rock (potentially acid generating (PAG) waste rock designed to be deposited in the TMF), and process water. Co-disposal will include management of both tailings and PAG1 waste rock in the TMF. The embankments include for adequate freeboard to provide ongoing tailings storage, PAG1 waste rock, water cover, operational water management, temporary storage of runoff resulting from the Environmental Design Flood (EDF) and safe conveyance of runoff up to and including the Inflow Design Flood (IDF) through a spillway. The TMF will be constructed as a paddock style, single cell facility located on a side hill northeast of Gold Brook Lake. A geomembrane lining system will be installed along the TMF basin floor and on the upstream face of the perimeter embankments to minimize seepage exiting the facility. The embankments will be raised in stages using downstream construction methods throughout the mine life.

Tailings will be pumped from the process plant to the TMF as a conventional thickened slurry via pipeline(s) and deposited into the TMF. The PAG1 waste rock will be segregated during mining operations and hauled directly to the TMF. The PAG1 waste rock pile in the TMF basin will be constructed similar to conventional waste rock piles (i.e., spread by a dozer in controlled lifts and compacted by the mine haul fleet) towards the northeast portion of the facility. The working surface of the PAG1 waste rock pile will be maintained above the elevation the tailings and supernatant throughout the mine life.

Meteoric and supernatant inflows to the TMF basin will be temporarily stored prior to reclaim by a floating pump barge in the basin to the process plant. Water reclaim, and treatment and release will be conducted such as to always maintain a 2 m minimum water cover over the deposited tailings surface. Excess water beyond the storage of the required water cover level and allowable operating range will be transferred to the TMF water treatment plant as required for treatment prior to release to the environment. The TMF will be equipped with an emergency overflow spillway in each embankment stage to accommodate flows above the EDF and up to the IDF.

### 1.11.2 Polishing Pond

A polishing pond will be constructed as an external pond to store water for the TMF water treatment plant (WTP) operations. The polishing pond will be constructed southwest of the TMF. The polishing pond has been designed to store approximately four days of TMF WTP discharge capacity plus some extra capacity contingency. The polishing pond embankment will be constructed in one stage as zoned rockfill dam. The polishing pond basin and upstream embankment face will be lined with a smooth 80 mil High Density Poly Ethylene (HDPE) geomembrane overlying a 12 oz./sq. yd. non-woven geotextile.

## 1.12 Mi'kmaq Engagement & Public Consultation

The Company recognizes the asserted Aboriginal and Treaty Rights and Title of Nova Scotia Mi'kmaq and maintains ongoing engagement with Kwilmu'kw Maw-klusuaqn Negotiation Office (KMKNO) and representatives of Paqtnkek Mi'kmaw Nation. On June 2, 2019, the Company and the Assembly of Nova Scotia Mi'kmaw Chiefs signed a Memorandum of Understanding (MOU) that outlines a process that the parties may use to develop a Mutual Benefits Agreement (MBA) that reflects a desire to build a mutually beneficial relationship with respect to the Project. This process is ongoing. The Company maintains its commitment to work collaboratively with Nova Scotia Mi'kmaq regarding environmental and cultural priorities, as well as social and economic opportunities throughout the life of the Project. Information shared through ongoing Mi'kmaq engagement as well as completion of a Mi'kmaq Ecological Knowledge Study (MEKS), has been reflected in the development of the Project to date and will continue throughout the life of the Project.

Public engagement has been ongoing with the Municipality of the District of Guysborough (MODG), as well as residents and property owners in the region since 2017. This includes regular meetings of Company senior executives and project consultants with the MODG Council. A Community Liaison Committee (CLC) was established to foster environmental stewardship, and act as a conduit for transparent and ongoing communications between community, stakeholders, and the Company on all matters pertaining to potential development. The Company has held three open house meetings in Goldboro and will seek additional opportunities for community engagement throughout the life of the Project.

## 1.13 Permitting and Compliance Activities

To date, the Company has arranged access to the Property for the purpose of exploration through agreements with both private and Crown entities. Much of the Property, including all the BR, EG historical workings, is underlain by Crown Land. Similarly, access to private lands, and securing agreements with landowners has been generally manageable. At the effective date of this Technical Report, the Company held access agreements that specifically apply to surface core drilling. The Company has the necessary Crown Land permits for additional drilling and trenching or expects to receive them through normal exploration permitting process.

The Project will require the acquisition of some privately owned property. The Company has engaged a third-party to complete the relevant property assessments, negotiate property acquisitions, and manage and document the process. Should some individuals refuse to sell through this process, the Company would then pursue the expropriation process to acquire the property which could cause delays, however, would unlikely cause any delays in the overall permitting process.

The Company continues to successfully manage the Industrial Approval (IA) related to the underground Bulk Sample collected in 2018.

The presence of past mining operation infrastructure, including several historical tailings sites associated with the past operation of the historical Boston Richardson Mine within the Gold Brook Lake-Seal Harbour Lake watershed, are recognized as important environmental site factors. Provincial regulators indemnified Orex in 1995 from any environmental liabilities resulting from historical mining activities, assuming that old tailings storage areas are not impacted during exploration or mining activities. A historic tailings management plan will be developed in consultation with Nova Scotia Environment and Climate Change (NSECC) to manage the areas that will be directly disturbed by the Project.

The Company plans to submit an Environmental Assessment Registration Document (EARD) to Nova Scotia Environment (NSE) for a Class 1 Environmental Assessment in Q2 2022. As such, baseline studies and related modelling efforts are ongoing. Critical provincial authorizations are required to proceed with mine development, operation, and reclamation, including an IA. Applications to federal authorities are also required, including a Fisheries Act Authorization through Fisheries and Oceans Canada (DFO) for alteration and destruction of fish habitat and a Schedule 2 addition for tailings placement. Applications for these approvals or permits have not been made at the effective date of this Technical Report, but will be submitted at various points throughout 2022/23

## 1.14 Environmental Studies

The Project is subject to regulation under the NS Environment Act, Part IV. An EARD for the proposed project will be submitted for Class 1 Environmental Assessment in Q2 2022. The EARD will be authored by the Company and GHD, utilizing extensive baseline data collected at the Project site by the Company and its consultants since the Company acquired the Project in 2017. Baseline studies, combined with predictive modelling, will inform project planning and provide the required information for various authorizations and permits. Mitigation measures to avoid, reduce or offset for potential effects will be developed and supported by EARD.

## 1.15 Market Studies and Contracts

The Company has not completed any formal marketing studies with respect to gold production that will result from the mining and processing from the Project, which is assumed to be in the form of gold doré bars for the purposes of this study. However, the Company was able to make reference to its existing refining contract at its Point Rousse operation with the Royal Canadian Mint to refine its gold doré bars into bullion, and a precious metals sales agreement with Auramet International LLC for the purposes of selling bullion. The Company has relied on its contracts for transportation, security, and insurance with respects to the refining of its gold doré bars

Gold produced will likely be sold on the spot market by precious metals marketing professionals retained on behalf of the Company, at terms and conditions typical of similar contracts for the sale of refined London Bullion Market Association (LBMA)[1] Good Delivery gold bullion. There are active and liquid gold markets throughout the world where gold can be bought and sold, and market pricing can be ascertained.

## 1.16 Capital and Operating Costs

The capital cost estimate was prepared by Nordmin and the Consultants with an expected accuracy range of ±15% weighted average accuracy of actual costs. Base pricing is in Q3 of 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25. The estimate includes direct and indirect costs, (such as engineering, procurement, construction and start up of facilities) as well as owners costs and contingency associated with mine and process facilities and on site/off site infrastructure. Total LOM capital costs, including initial, sustaining and reclamation costs, are $384.5 million. The initial capital estimate is $271.1 million and includes amounts indirect and contingency assumptions. A contingency of $31.7 million has been included in the estimate of initial capital costs, which amounts to 16% of direct initial capital costs or 11% of the total.

| Item / Description | Units | Pre-Production Phase | % of Total | Production Phase | Closure Phase | Total |
|---|---|---|---|---|---|---|
| **Capital Cost** | | | | | | |
| **Capital Cost Estimate** | | **Initial** | | **Sustaining** | | **TOTAL** |
| Open Pit Mining | M$ | 25.5 | 9% | 1.6 | | 27.1 |
| Process Plant | M$ | 70.5 | 25% | | | 70.5 |
| Tailings Management | M$ | 20.6 | 7% | 42.4 | | 63.1 |
| Infrastructure and Site Development | M$ | 49.8 | 18% | 7.4 | | 57.2 |
| Water Management & Treatment | M$ | 14.4 | 5% | 11.7 | | 26.1 |
| General Site Equipment | M$ | 1.1 | 0% | | | 1.1 |
| Employee Accommodations | M$ | 12.1 | 4% | | | 12.1 |
| **Subtotal Capital Costs** | **M$** | **193.9** | **70%** | **63.1** | **0.0** | **257.0** |
| Indirect CapEx | M$ | 45.4 | 16% | | | 45.4 |
| *Mill Labour during pre-production* | *M$* | *0.79* | *0%* | | | *0.8* |
| *G&A Labour during pre-production* | *M$* | *2.18* | *1%* | | | *2.2* |
| *Other Indirects* | *M$* | *14.6* | *5%* | | | *14.6* |
| *EPCM* | *M$* | *27.8* | *10%* | | | *27.8* |

[1]. https://www.lbma.org.uk/good-delivery/good-delivery-rules-and-governance. London Bullion Market Association (LBMA) is the international trade association representing the global Over The Counter (OTC) bullion market and defines itself as "the global authority on precious metals". LBMA Good Delivery is the de facto world standard, which accredits refiners who produce bars which satisfy high standards in terms of purity, quality and physical appearance.

| | | | | | | |
|---|---|---|---|---|---|---|
| Contingency | M$ | 31.7 | 11% | | | 31.7 |
| **Subtotal Capital Costs** | **M$** | **271.1** | **97%** | **63.1** | **0.0** | **334.2** |
| Rehabilitation & Closure, Bond Cost | M$ | 0.7 | 0% | 10.1 | 30.3 | 41.0 |
| Other CapEx – Habitat Compensation | M$ | 0.0 | 0% | 9.3 | | 9.3 |
| Working Capital | M$ | 6.7 | 2% | -6.7 | | 0.0 |
| **Total Capital Costs** | **M$** | **278.5** | **100%** | **75.7** | **30.3** | **384.5** |

The operating cost estimate was prepared by Nordmin and the Consultants with an expected accuracy range ± 15% weighted average accuracy of actual costs based on the third quarter of 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25, unless otherwise stated. The LOM operating costs, including selling costs, are estimated to be $1,064.0 million.

| **Operating Costs** | | **C$ million** |
|---|---|---|
| Open Pit Mining | | 691.0 |
| Processing | | 212.5 |
| General and Administration | | 137.5 |
| Water Management and Treatment | | 18.4 |
| Refining Charges | | 3.6 |
| Transportation Charges | | 1.0 |
| **Total** | | **1,064.0** |

## 1.17 Economic Analysis

An engineering economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this Technical Report are based upon the services performed by:

- Nordmin for open pit mining and surface infrastructure and their associate consultants at Optimize Group Inc. (Optimize) for the pit slope stability.
- Knight Piésold for TMF.
- GHD for site water management.
- Ausenco Engineering Canada Inc. (Ausenco) for processing.
- Lorax Environmental (Lorax) for geochemistry.
- McCallum Environmental (MEL) for consultation and permitting.

The Company provided the inputs with respect to the tax impact of the economic model, including calculation of federal and provincial income taxes, provincial mining taxes, and available tax attributes that are applicable to the Project.

The Project includes an open pit and associated infrastructure, surface infrastructure to support the mine operations (i.e., maintenance and office facilities), water management features, ROM stockpiling area, processing facility, TMF, and employee accommodation facility.

The economic model for the Project indicates a pre-tax free cash flow of $755.1 million over approximately and 11-year mine life, a pre-tax Net Present Value (NPV) 5% of $483.8 million and a pre-tax IRR of 31.2%. On an after-tax basis, the Project could generate free cash flow of $529.0 million, and after-tax NPV (5%) of $328.2 million and an after-tax IRR of 25.5%. The Project is most sensitive to commodity prices.

## 1.18 Risks and Opportunities

There are inherent risks for any mining project Like Goldboro, including:

- Changes to long term metal price assumptions.
- Changes to the input values for mining, processing, and general and administrative costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.
- EA Timing, requirements and supporting documentation.
- The assumption that the electric power line will be available on time for the construction of the project.
- Discussions with various First Nation and Indigenous communities.
- Logistics of securing and moving adequate services, labour and supplies could be affected by epidemics, pandemics and other public health crises, including COVID-19 or similar such viruses.

### 1.18.1 Risks

The risk analysis defined 96 risks and their associated potential mitigation strategies (Appendix E).

Ten risks were considered a pre-response consequence rating of ‘moderate or major’, or ‘severe’, and a likelihood rating of ‘likely’ or ‘almost certain’. If the related action plans are initiated, the post response consequence rating for these high-risk items reduces to three risks outlined below.

The key group of risks with an action plan assigned are the following:

#### 1.18.1.1 Geology, Mining and Milling

Operations – Grade control sampling and sample turnaround will be key to delivering the expected grades to the mill. Any reduction in grade will affect the overall production, milling and ounce production. To mitigate this risk, the Company should complete infill drilling/sampling within the first two to three years of mining, create an appropriate reconciliation process from resource through to reserve, updating the short/long term mine plan, mineral reserve, mine design, and mill efficiency/production. As such, extensive sampling protocols, locations and sizes will be required to improve mineralization patterns, reduce ore/waste assignments and improve the non-potentially acid generating (NPAG) and PAG waste rock association.

The cyanide detoxification process primarily breaks down $CN_{WAD}$ and total cyanide ($CN_{TOT}$) which may or may not be directly reduced. The $CN_{TOT}$ present in the mill discharge is a function of the variable iron species present in the cyanide detox feed stream. Additional testing should be conducted to confirm the variability of $CN_{TOT}$ in the process tailings discharge water to the storage facility.

Additionally, no geotechnical information is available for the process plant area with the exception of the crushing and stockpile areas. This testing should be also completed as part of future work prior to the next phase of detailed engineering

#### 1.18.1.2 Safety, Health, Environment and Community

While the Company is mitigating environmental impacts, having meaningful engagement with rightsholders and stakeholders, and has the support of the municipality, there are groups and individuals that are against all forms of gold mining in Nova Scotia. There is accordingly the potential for organized protests, which could cause delays. To mitigate this risk the Company will maintain ongoing opportunities for all rightsholders and stakeholders to have an occasion to raise questions and concerns about the Project. Additionally, the Company will ensure that accurate information about the Project is publicly available and will identify and correct inaccurate information in the public domain.

The Project requires the acquisition of some privately owned property. There may be individuals who refuse to sell regardless of the offer. If the property is essential to the Project the Company may have to pursue expropriation as a last resort, which could cause delays. To mitigate this risk the Company has engaged a third-party Turner Drake & Partners Ltd. (Turner Drake) to complete property assessments, negotiate property acquisitions, and manage and document the process. The Company is not seeking to acquire any dwellings and will continue to work collaboratively will property owners, through its third-party service provider, for mutually agreeable outcomes. If the parties were unable to find a mutually agreeable outcome, the Company would then pursue the expropriation process to acquire the property which could cause delays, however, would unlikely cause any delays in the overall permitting process. The expropriation process is an established Provincial process used where required for land acquisition to enable development and infrastructure construction in Nova Scotia and has also been used to enable the construction and development of a now operating gold mine.

### 1.18.2 Opportunities

The Deposit has been drill tested to only 500 m deep and remains open at depth; analogous geological systems, such as in the Victorian Goldfields, are known to continue at depth for multiple kilometres. The most recent Mineral Resource Estimate dated November 15, 2021, demonstrated a substantial amount of underground Mineral Resources, including Measured and Indicated Resources of 1,159,000 ounces of gold (5,925,000 tonnes @ 6.09 g/t gold) and Inferred Mineral Resources of 418,000 ounces of gold (2,206,000 tonnes @ 5.89 g/t gold). A previous study on the Project has indicated that there could be potential to further explore underground mining with an open pit mining scenario.

A future study will consider upgrading and expanding potentially mineable underground Mineral Resources as part of the longer-term mine development plan. Once surface mining has commenced and a significant portion of the ore body has been exposed, the Company will likely undertake infill and expansion drilling, possibly from drifts off benches within the open pits, allowing for more effective and less expensive diamond drilling. Successful conversion of a major portion of the Inferred Resources to the Measured and Indicated category would allow for additional technical studies to be carried out with the intent of adding a ramp-access underground mining component to the Project, allowing for a combined open pit/underground scenario and significantly increasing the potential mine life of the Project.

## 1.19 Conclusions

The results of the FS for the Company indicate that the Project has technical and financial merit using the inputs from various advanced studies. The Company anticipates filing an EARD for the Project in Q2 2022, which will focus on surface mine plan outlined in the FS over a mine life of approximately 11 years.

If a production decision is made with respects to the surface mining operation outlined in the FS, the Company will then consider further opportunity to incorporate underground mining, including the initiation of infill and expansion drilling from drifts off benches in the open pit, allowing for more effective and less expensive diamond drilling. Pending those results, the Company would then consider a supplementary study that will focus on adding an underground mining phase to the Project.

## 1.20 Recommendations

The recommended program is focused on advancing technical and related studies toward an EARD being submitted for Class 1 Environmental Assessment in Q2 2022. Throughout 2022, the Company will continue with ongoing work to support further technical studies including geotechnical drilling, expanded surface water monitoring, metallurgical test programs and infill drilling. Additionally, the Company will continue exploration activities designed to target potential opportunities to expand the Project further, with emphasis on infill drilling to convert in-pit inferred mineral resources to Indicated mineral resources as well as demonstrating the continuation of the deposits along strike to the west of the existing Mineral Resource.

Table 1-6 tabulates the budget recommendations and associated costs with advancing activities such as process, metallurgical, environmental and tailings facility studies, permitting, delineation drilling and detailed engineering.

**Table 1-6: Recommended Budget**

| Activity | Value (C$) |
|---|---:|
| Processing and Metallurgy Studies | 160,000 |
| Permitting and Environmental Studies | 2,269,000 |
| Detailed Engineering | 5,645,000 |
| Tailings Management Facility | 530,000 |
| Delineation Drilling | 2,000,000 |
| **Subtotal** | **10,604,000** |
| Contingency (10%) | 1,591,000 |
| **Total** | **12,195,000** |

The budget recommendations presented in Table 1-7 are focused on infill and expansion drilling, resource modelling and updated mine design.

**Table 1-7: Recommended Budget**

| Activity | Value (C$) |
|---|---:|
| Infill and Expansion Drilling - Existing Mine Plan | 1,000,000 |
| Growth Exploration | 1,500,000 |
| Resource Modelling and Updated Mine Design | 700,000 |
| **Total** | **3,200,000** |

# 2. INTRODUCTION

## 2.1 Terms of Reference

This Technical Report, including the FS, for the Project was prepared as a NI 43-101 Technical Report and Feasibility Study for the Company by Nordmin

The Mineral Resources Estimate is considered effective as of November 15, 2021. The effective date of the Mineral Reserves Estimate is December 15, 2021.

This Technical Report supersedes all prior technical reports, Mineral Resource Estimates and Mineral Reserve Estimates prepared for the Project. As of the date of this Technical Report, the Company anticipates using these Mineral Resources for future drill targeting and Mineral Resource upgrades.

The Company is a TSX and OTCQX-listed gold mining, development, and exploration company, with their head office located at:

20 Adelaide St. East
Suite 915
Toronto, Ontario, Canada M5C 2T6

The quality of information, conclusions, and estimate contained herein are consistent with the level of effort involved in Nordmin'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 outlined in this Technical Report.

This Technical Report is intended to be used by the Company; this permits the Company to file this report on SEDAR as a NI 43-101 Technical Report with the Canadian Securities Administrators. Nordmin understands that the Company may use this Technical Report for a variety of corporate purposes. The responsibility for this disclosure remains with the Company. The user of this document should ensure that this is the most recent Technical Report for the Project, as it is not valid if a new Technical Report has been issued.

This Technical Report provides a Mineral Resource/Mineral Reserve, and a classification of the Mineral Resource/Mineral Reserve prepared in accordance with the CIM, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, May 10, 2014 (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (November 2019; 2019 CIM Best Practice Guidelines).

## 2.2 Qualified Persons

The consultants preparing this Technical Report are specialists in the fields of geology, exploration, open pit mining, water management, environmental, infrastructure development, mineral processing, metallurgical testing, and Mineral Resource, and Mineral Reserve estimation, and classification.

The consultants nor any associates employed in the preparation of this Technical Report are insiders, associates, affiliates, or has any beneficial interest in the Company. 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 the Company and the Consultants. The consultants are being paid a fee for the work in accordance with reasonable professional consulting practices.

This Technical Report was prepared by the QPs listed in Table 2-1, and their responsibilities for each section are indicated. These individuals, by virtue of their education, experience, and professional association, are considered a QP as defined in the NI 43-101 standard, for this Technical Report, and are a member in good standing of a relevant professional institution. QP Certificates of the Authors are provided in Appendix A of this Technical Report.

**Table 2-1: QP – Section Responsibility**

| Section and Title | Qualified Person | Company |
|---|---|---|
| 1: Summary | All QPs | Various |
| 2: Introduction | Glen Kuntz, P.Geo | Nordmin |
| 3: Reliance on Other Experts | Glen Kuntz, P.Geo | Nordmin |
| 4: Property Description and Location | Glen Kuntz, P.Geo | Nordmin |
| 5: Accessibility, Climate, Local Resources, Infrastructure, and Physiography | Glen Kuntz, P.Geo | Nordmin |
| 6: History | Glen Kuntz, P.Geo | Nordmin |
| 7: Geological Setting and Mineralization | Glen Kuntz, P.Geo | Nordmin |
| 8: Deposit Types | Glen Kuntz, P.Geo | Nordmin |
| 9: Exploration | Glen Kuntz, P.Geo | Nordmin |
| 10: Drilling | Glen Kuntz, P.Geo | Nordmin |
| 11: Sample Preparation, Analyses, and Security | Glen Kuntz, P.Geo | Nordmin |
| 12: Data Verification | Glen Kuntz, P.Geo | Nordmin |
| 13: Mineral Processing and Metallurgical Testing | Tommaso Roberto Raponi, P.Eng. | Ausenco |
| 14: Mineral Resource Estimate | Glen Kuntz, P.Geo | Nordmin |
| 15: Mineral Reserve Estimate | Joanne Robinson, P.Eng. | Nordmin |
| 16: Mining Methods | Joanne Robinson, P.Eng., João Paulo dos Santos, MAusIMM (CP) | Nordmin/ Optimize |
| 17: Recovery Methods | Tommaso Roberto Raponi, P.Eng. | Ausenco |
| 18: Project Infrastructure | Steve Pumphrey, P.Eng., Harold Harkonen, P.Eng., Reagan McIsaac, P.Eng., Tommaso Roberto Raponi, P.Eng., Andrew Betts, P.Eng. | Nordmin/Knight Piésold/ Ausenco/ GHD |
| 19: Market Studies and Contracts | Glen Kuntz, P.Geo | Nordmin |
| 20: Environmental Studies, Permitting, and Social, or Community Impact | Timo Kirchner, P.Geo., Jeff Parks, P.Geo., Andrew Betts, P.Eng., Reagan McIsaac, P.Eng. | Lorax/GHD/ Knight Piésold |
| 21: Capital and Operating Costs | Joanne Robinson, P.Eng., Steve Pumphrey, P.Eng., Reagan McIsaac, P.Eng. Tommaso Roberto Raponi, P.Eng., Andrew Betts, P.Eng. | Nordmin/Knight Piésold/ Ausenco/GHD |
| 22: Economic Analysis | Joanne Robinson, P.Eng | Nordmin |
| 23: Adjacent Properties | Glen Kuntz, P.Geo | Nordmin |
| 24: Other Relevant Data and Information | Glen Kuntz, P.Geo | Nordmin |
| 25: Interpretation and Conclusions | All QPs | Various |
| 26: Recommendations | All QPs | Various |
| 27: References | Glen Kuntz, P.Geo | Nordmin |
| 28: Glossary | Glen Kuntz, P.Geo | Nordmin |

The following summarizes the dates of the QP site visit to the Project:

- Glen Kuntz, P.Geo., completed a site visit from January 18 to January 21, 2021 and August 16 and August 17, 2021.
- Joanne Robinson, P.Eng., completed a site visit from August 16 to August 17, 2021.
- Steve Pumphrey, P.Eng., completed a site visit from August 16 to August 17, 2021.
- Andrew Betts, P.Eng., completed a site visit on July 29, 2021.
- Reagan McIsaac, P.Eng., completed a site visit on November 29, 2021.

## 2.3 Effective Dates

- The effective date of the Mineral Resource Estimate is November 15, 2021.
- The effective date of the Mineral Reserves Estimate is December 15, 2021.
- The effective date of the Technical Report is December 16, 2021.

## 2.4 Information Sources and References

This Technical Report has been prepared by independent consultants who are QP's under NI 43-101 and prepared in accordance with NI 43-101, Form 43-101F1, and Companion Policy 43-101CP. Subject to the conditions and limitations set forth herein, the independent consultants believe that the qualifications, assumptions, and information used by them are reliable, and efforts have been made to confirm this to the extent practicable. However, none of the consultants involved in this study can guarantee the accuracy of all information in this Technical Report.

This Technical Report is based, in part, on internal Company technical reports and maps, published government reports, Company letters and memoranda, and public information as listed in Section 27. Several sections from reports authored by other consultants have been directly quoted or summarized in this Technical Report and are so indicated where appropriate.

A draft copy of this Technical Report has been reviewed for factual errors by the Company regarding the Company, the history of the Project, and the Mineral Resource/Mineral Reserves Estimates prepared by Nordmin.

Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Technical Report.

The authors of this report have taken all steps in their professional judgment to verify and confirm the accuracy of the information contained in this report and other than with respect to this matters set forth in Section 3 hereof, do not disclaim any responsibility for this Technical Report.

## 2.5 Previous Reporting

### 2.5.1 Previous Mineral Resource Estimate

The Mineral Resource Estimate (effective date of November 15, 2021) discussed herein (Section 14.9) supersedes historical and past Mineral Resource Estimates presented in this section.

The following historical information is relevant to provide context but is not current and should not be relied upon. The QPs responsible for the preparation of this Technical Report have not done sufficient work to classify the historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating any historical estimates as Mineral Resource Estimates.

- Bourgoin, M.; MRB & Associates. 2004. *Technical Report, Goldboro Property, Guysborough County, Nova Scotia, NTS 11F/4D.*
- Puritch, E., Armstrong, T. and Horvath, A.S. 2006. *Technical Report and Resource Estimate on the Goldboro Property, Guysborough County, Nova Scotia*.
- Gervais, D.; Carrier, A.; Brousseau, K.; InnovExplo Inc. 2009. *Technical Report on the 2009 Mineral Resource Estimate for the Goldboro Property.*
- Cullen, M.; Yule, S.; Mercator Geological Services Ltd. 2013. *Mineral Resource Estimate Technical Report for the Goldboro Property, Guysborough County, Nova Scotia, Canada. Effective Date: February 11, 2013.* 2013.
- Cullen, M.; Yule, S.; Mercator Geological Services Ltd. 2017. Updated Mineral Resource Estimate Technical Report for the Goldboro Property, Guysborough County, Nova Scotia, Canada. Effective Date: February 28, 2017.
- Robinson, J.; Cullen, M.; Liukko, G.; Bertelegni, S.; Thibault, J.D.; Slepcev, G.; Mercator Geological Services Ltd. 2018**.** *Goldboro Project Preliminary Economic Assessment.* 2018. NI 43-101 Technical Report.
- McCracken, T.; Ghouralal, S.; Bertelegni, S.; Thibault, J.D.; WSP. 2018. *Goldboro Project, Mineral Resource Update and Preliminary Economic Assessment, Guysborough County, Nova Scotia.* 2018.
- McCraken, T.; WSP Canada Inc. 2019. *Goldboro Gold Project Resource Update Phase 2 Guysborough County, Nova Scotia.*
- Kuntz, G.; Nordmin Engineering Ltd., 2021. *NI 43-101 Technical Report and Mineral Resource Estimate, Goldboro Gold Project, Eastern Goldfields District, Nova Scotia.*
- Kuntz, G.; Nordmin Engineering Ltd., 2021. *NI 43-101 Technical Report and Preliminary Economic Assessment, Goldboro Gold Project, Eastern Goldfields District, Nova Scotia.*

### 2.5.2 Previous Mineral Reserve Estimates

There are no previous Mineral Reserve estimates calculated for the Project.

## 2.6 Acknowledgements

Nordmin would like to thank and acknowledge the following people who have contributed to the preparation of this report and the underlying studies under the supervision of the QPs:

Nordmin Personnel

Christian Ballard, P.Geo, Senior Geologist, Brett Stewart, Technical Design Specialist, Annika Van Kessel, G.I.T., Sirena Jacobsen, Geological Technician, Joanne Robinson, P.Eng., Senior Mining Engineer, Open Pit Mining, and Steve Pumphrey, P.Eng., Consulting Engineer – Civil/Structural, and João Paulo dos Santos, MAusIMM, Mine Geotechnical Engineer, Optimize Group.

Anaconda Employees

Kevin Bullock, P.Eng., President & CEO, Robert Dufour, CPA, Chief Financial Officer, Paul McNeill, P.Geo., Vice President, Exploration, David Copeland, P.Geo, Chief Geologist, Alana Haysom, P.Geo., Exploration & Resource Geologist, Tanya Tettelaar, P.Geo., Exploration Manager, Michelle English, Exploration Geologist – Goldboro Lead.

Anaconda Consultants

Glen Kuntz, P.Geo., Consulting Specialist – Geology/Mining with Nordmin Engineering Ltd., Timo Kirchner, P.Geo., Environmental Geoscientist of Lorax Environmental, Tommaso Roberto Raponi, P.Eng., Principal Metallurgist of Ausenco Engineering Canada Inc., Reagan McIsaac, Ph.D., P.Eng., Senior Engineer of Knight Piésold Ltd., Andrew Betts, P.Eng., Water Resource Engineer of GHD and Meghan Milloy, Vice President of McCallum Environmental.

## 2.7 Units of Measure

Unless otherwise noted, the following measurement units, formats, and systems are used throughout this Technical Report.

- Measurement Units: all references to measurement units use the International System of Units (SI, or metric) for measurement. The primary linear distance unit, unless otherwise noted, are metres (m).
- General Orientation: unless otherwise stated, all references to orientation and coordinates in this Technical Report are presented as MTM Projection, NAD83, Zone 4, false easting 304800 min metres. Elevations on the Project are set using a historic mine grid of elevation above sea level + 4,940 m.
- Currencies outlined in the Technical Report are stated in Canadian dollars (C$" unless otherwise noted.
- Gold values for work performed by the Company and previous operators are reported as grams per tonne or parts per billion. A conversion factor of 31.1035 is used to convert grams to troy ounces.

The symbols and abbreviations used in this Technical Report are outlined in Section 28.4.

# 3. RELIANCE ON OTHER EXPERTS

Nordmin and the Consultants have assumed and relied on the fact that all the information and existing technical documents listed in the References, Section 27 of this Technical Report, are accurate and complete in all material aspects. The authors of this report have taken all steps in their professional judgment to verify and confirm the accuracy of the information contained these documents and do not disclaim any responsibility for this Technical Report. However, the authors cannot guarantee its accuracy and completeness. We reserve the right, but will not be obligated, to revise the Technical Report and conclusions if additional information becomes known after the date of this Technical Report.

## 3.1 Mineral Tenure, Surface Rights, Property Agreements, and Royalties

Copies of the tenure documents, operating licences, permits, and work contracts were reviewed by Nordmin; independent verification of land title and tenure reported in Section 4 was not performed. Nordmin did not independently verify the legality of any underlying agreement(s) that may exist concerning the licenses or other agreement(s) between third parties but has instead relied on the Company to have conducted the proper legal due diligence.

## 3.2 Environmental, Permitting, and Liability Issues

The QP has fully relied upon the Company concerning the Project environmental, legal, and permitting matters relevant to the Technical Report.

## 3.3 Taxes

The QP has fully relied upon the Company for the tax impact of the economic model, including calculation of federal and provincial income taxes, provincial mining taxes, and available tax attributes that are applicable to the Project.

# 4. PROPERTY DESCRIPTION AND LOCATION

The Property is situated on the eastern shore of Nova Scotia, Canada. The Property's central point is approximately located at 45° 12’ 2.6” N latitude and 61° 39’ 2.0” W longitude (Figure 4-1).

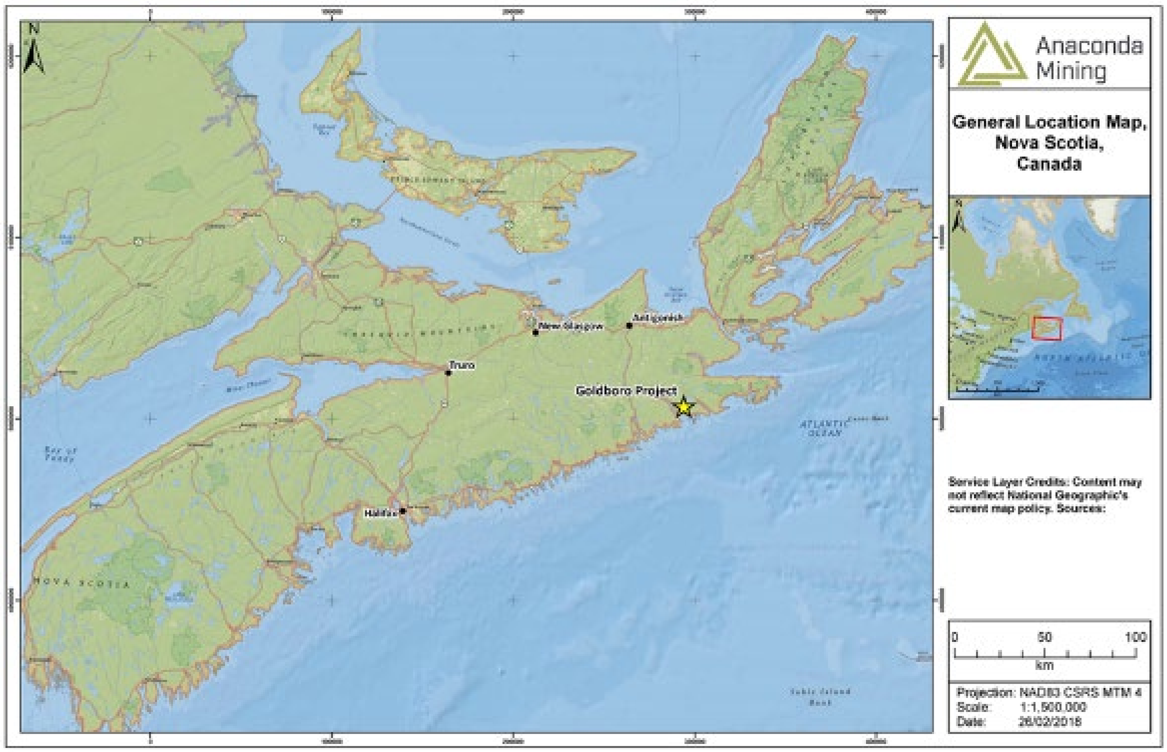



*Figure 4-1: General location map of the Project on the eastern shore of Nova Scotia*

## 4.1 Property Land Tenure

The Property consists of 37 contiguous claims covering a total area of approximately 592 hectares held under Exploration Licence No. 05888 (Table 4-1 and Figure 4-2). This title is in its 42nd year of issue and is renewed every two years, with the next renewal date on November 29, 2023.

**Table 4-1: Claims List**

| Exploration Licence | NTS Sheet | Number of Claims |
|---|---|---|
| 05888 | 11-F-A | 37 |

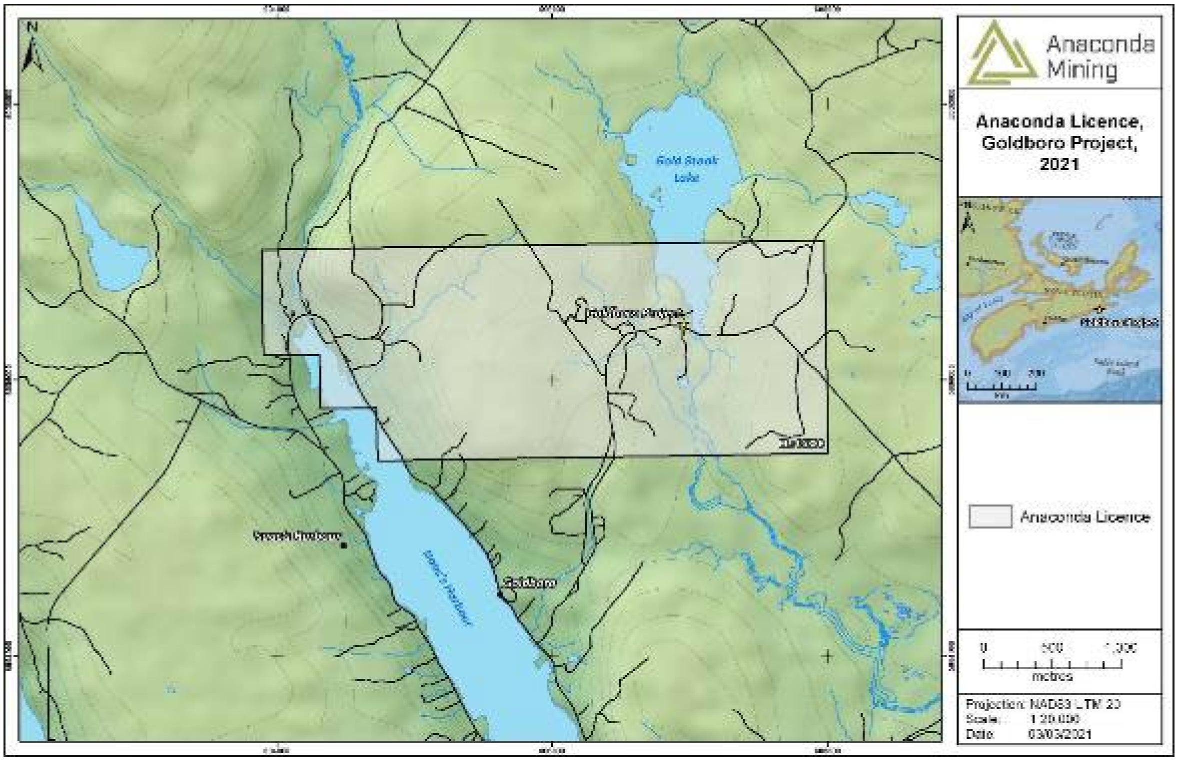



*Figure 4-2: Property location map, Exploration Licence No. 05888*

The surface rights are held by various private landowners and by the Nova Scotia government. The Company maintains a 100% interest in the exploration rights through its wholly-owned subsidiary Orex (with Orex having acquired the exploration rights from Onitap Resources Inc. (Onitap) in 1988), and at the effective date of this Technical Report, no lien, mortgage, royalty, or other rights in favour of third parties was registered with NSDNR. NSDNR records also show that sufficient assessment work credits were available at the effective date to maintain the Property in good standing for several years. Exploration expenditures of $800 per claim per year (or the equivalent in banked expenditure credits) and renewal application fees of $320 per claim per year are required to keep Licence 05888 in good standing. Nordmin has not been advised of any liens, encumbrances, or royalties associated with this licence. At the report effective date, Licence 05888 was still registered to Orex and had not been transferred to the Company.

Nordmin consulted NSDNR records for the purposes of this report and documented the information set out above. However, reliance has been placed on the Company with respect to confirmation of title validity and for comment with respect to encumbrances, liens, or royalties, if any, that apply to the title. A legal search of the title was not carried out by Nordmin for current report purposes. However, Nordmin has no reason to question the validity of the information and opinions presented above. There is no requirement to legally survey exploration licences in Nova Scotia.

## 4.2 Underlying Agreements

There are no underlying agreements relating to the Property.

### 4.2.1 Permits and Authorization

To date, the Company has arranged access to the Property for the purpose of exploration through agreements with both private and Crown entities. Much of the Property, including all the BR, and EG historical workings, is underlain by Crown Land. Similarly, access to private lands, and securing agreements with landowners has generally been manageable. The Company has advised Nordmin that it knows of no reason that future access to the areas required for exploration would be withheld.

At the effective date of this Technical Report, the Company held access agreements that specifically apply to surface core drilling. The Company has the necessary Crown Land permits for additional drilling and trenching or expects to receive them through normal exploration permitting process.

Nordmin is of the opinion that due to the location and size of the Property, its brownfield to undeveloped nature, and proximity to year-round habitations, it is reasonable to conclude that the site can support future mining operations and infrastructure.

### 4.2.2 Environmental Site Conditions

The presence of past mining operation infrastructure, including several historical tailings sites associated with the past operation of the historical Boston Richardson Mine and location within the Gold Brook Lake-Seal Harbour Lake watershed are recognized as important environmental site factors. Provincial regulators indemnified Orex in 1995 from any environmental liabilities resulting from historical mining activities, assuming that old tailings storage areas are not impacted during exploration or mining activities.

The Company continues to successfully manage the Industrial Approval related to the 13,028 tonne Bulk Sample collected in 2018.

No obvious environmental issues were identified during the 2021 Nordmin site visits, and Nordmin is not aware of any subsequently developed site conditions that would materially alter the nature of this assertion.

### 4.2.3 Environmental Approvals Required for Future Mining

Critical permits required to proceed with mine development, operation and reclamation include the Environmental Assessment Registration and Industrial Approval authorization pursuant to the NSE Act. Baseline studies are in progress, and the Company plans to submit an EARD to NSE for a Class 1 Environmental Assessment in Q2 2022. The Company will apply for the required Industrial Approval in Q4 2022.

It will also be necessary for the Company to make an application for and receive various permits associated with Mining and Crown Land access, mining, and milling permits, water use, wetland alteration, and sewage treatment to support authorization for future mining at this site. Applications to federal authorities are also required, including but not limited to a Fisheries Act Authorization through Fisheries and Oceans Canada (DFO) for alteration and destruction of fish habitat, as well as a Schedule 2 addition for tailings placement. These applications will be made in 2022, as regulations and associated timelines dictate.

Applications for these approvals or permits have not been made at the effective date of this Technical Report.

### 4.2.4 Mining Rights in Nova Scotia

The Project is located in the province of Nova Scotia, a jurisdiction that has a well-established permitting process, with mineral and mining rights reserved largely to the Crown. Rights and access to lands for mineral exploration and mining are governed under the Mineral Resources Act. Mineral rights in Nova Scotia are acquired and located via map staking, whereby mineral licences can be acquired in order to explore for minerals. Permitting of routine exploration activities (prospecting, geochemistry, geophysics, diamond drilling, etc.) is regulated and approved by the Province under the Mineral Resources Act and other Provincial Acts covering, but not limited to, water resources and the environment.

Permitting and approval of mining projects is coordinated between the municipal, provincial, and federal regulatory agencies, along with systematic community and indigenous consultation. As is the case for similar mine developments in Canada, the Project is subject to the federal and provincial Environmental Assessment process. Due to the complexity and size of such projects, various federal and provincial agencies have jurisdiction to provide authorizations or permits that enable Project construction to proceed.

Federal agencies that have significant regulatory involvement include the Canadian Environmental Assessment Agency, Environment and Climate Change Canada, Natural Resources Canada, and DFO.

### 4.2.5 Stakeholder Consultation

Productive and open relationships with all stakeholders and rightsholders are a key component of the Project. The Company has an active strategy for stakeholder consultation and Mi'kmaq engagement. These include, but are not limited to, ongoing communications with the Assembly of Nova Scotia Mi'kmaw Chiefs, KMKNO, Paqtnkek Mi'kmaw Nation, the MODG, the Goldboro Gold Project CLC, and the general public.

The Company maintains an active information sharing relationship with officials of KMKNO and representatives of Patqnkek Mi'kmaw Nation, including more than a dozen meetings since June 2017. On June 2, 2019, the Company and the Assembly of Mi'kmaw Chiefs signed an MOU that governs the process by which the parties shall negotiate a MBA, which reflects a desire to build a mutually beneficial relationship with respect to the Project. Baseline information for Indigenous Peoples was gathered through the ongoing engagement with the Mi'kmaq of Nova Scotia and completion of a Mi'kmaq Ecological Knowledge Study (MEKS), with the purpose of identifying and documenting land and resource use (within 5 km of the Project), which is recognized as holding great importance to the Mi'kmaq people.

Consultations have been ongoing with the MODG and people within the community. This includes annual meetings with MODG, quarterly newsletters sent to every household in the municipality as well as the creation of the CLC. The CLC was set up to foster environmental stewardship and act as a vehicle for transparent and ongoing communications between the community, stakeholders, and the Company on matters pertaining to potential development at the Project and to ensure regular communication between the community and the Company. In addition, the Company has held public open houses within the Community. The Company and MODG signed a Community Benefits Agreement in January 2022.

# 5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

## 5.1 Accessibility

The Property is located 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro on the eastern shore of Isaac's Harbour, in Guysborough County, Nova Scotia, Canada. The elevation is approximately 70 m above sea level. All-weather highway, Route 316 links the village of Goldboro to the town of Antigonish. A secondary gravel road named Gold Brook Road, accessible from Route 316, crosses the Property, and passes near the historical BR shaft and portal. Smaller logging roads and trails provide good access to most areas of the Property.

## 5.2 Local Resources and Infrastructure

The villages of Goldboro and Isaac's Harbour offer minimal essential services. However, basic services are readily available in the village of Sherbrooke, located approximately 45 highway km west of the Property, and full services, including a hospital, mechanical, and retail facilities, are present in the town of Antigonish, located approximately 70 km to the northwest (population approximately 4,500). The nearest commercial airport is Halifax Stanfield International Airport, located 3.5 hours from the Property.

The Strait of Canso Superport is located in Port Hawkesbury-Mulgrave area, approximately 60 km northeast of the Property and provides access to ocean-going shipping. The Cape Breton and Central Nova Scotia Railway mainline between Sydney, Nova Scotia, and Truro, Nova Scotia, passes within 50 km of the Project and ocean access for smaller vessels is possible from wharves in the local area along Isaac's Harbour and in nearby Country Harbour.

The population is sparse in this area, and the current local workforce does not reflect an extensive history of mining industry exposure. A substantial component of the area's economy is related to forestry sector activities, and some transfer of skills to potential future mining projects could be expected.

Central to the Property is the historical Goldboro mine site which includes the past producing Boston Richardson, East and West Goldbrook Mines. The Boston Richardson Mine includes a vertical, three-compartment shaft that was rehabilitated to a depth of 122 m in the late 1980s. At that time, a hoist capable of operating to depths of 600 m below surface was installed along with a new timber headframe. The headframe has since been dismantled. A service building that served as a warehouse, geology office, shaft house, and hoist room is also present on the Property and was in good repair at the time of the Nordmin 2021 site visit. This building is currently being used as a core logging and covered core storage facility. External core racks are also present at the site (Figure 5-1, Figure 5-2 and Figure 5-3).

*Figure 5-1: Core logging facility on the Property*


*Figure 5-2: Internal (covered) core storage racks on the Property*

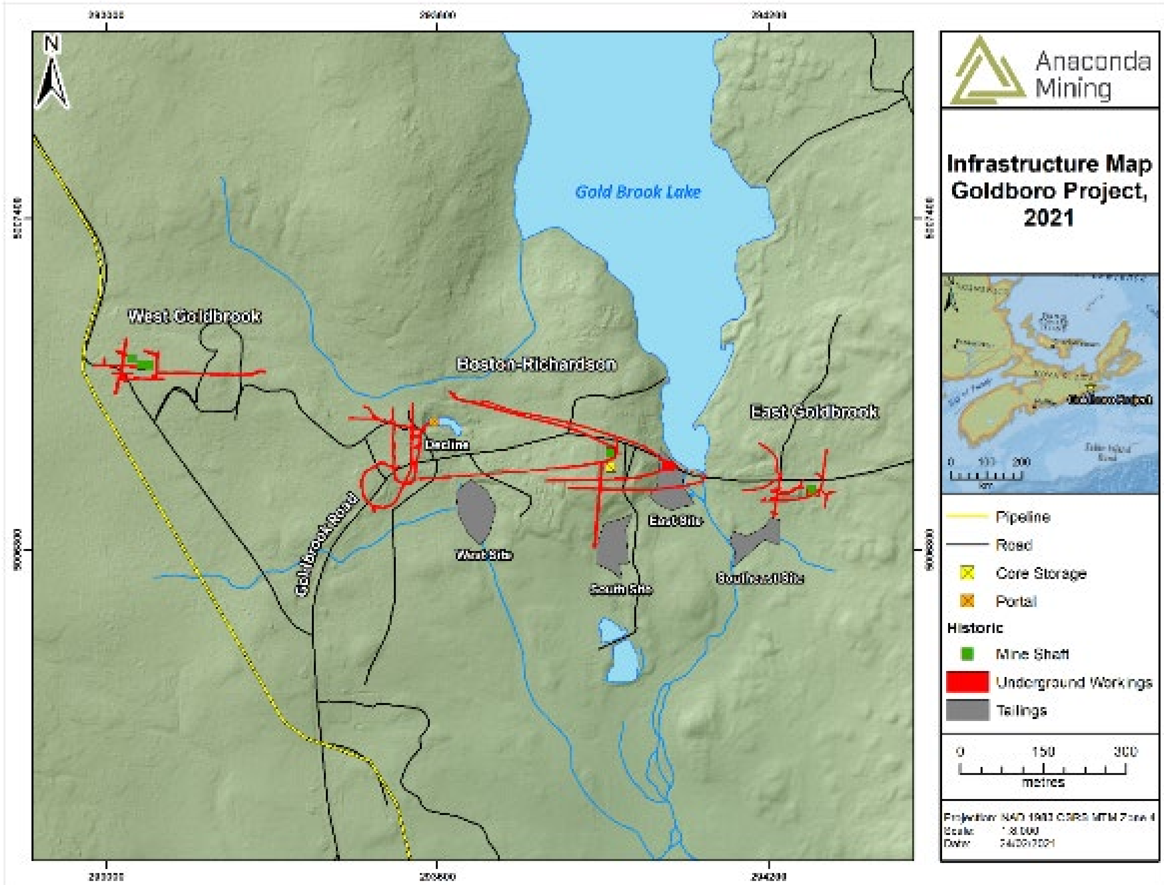



*Figure 5-3: Infrastructure surface map for the Property*

A portal accesses a decline at the top of the BR Gold System located approximately 350 m west of the Boston Richardson shaft. The decline measures approximately 5 x 3 m in cross section and was developed during the 1988 exploration program. The ramp was driven with a grade of 15% and has a total inclined length of 416 m from surface to the 76 m Level. Underground development in 1988 provided access to two 4 m x 3 m cross cuts, one at the 38 m Level and another at the 76 m Level.

In 2018 site infrastructure was temporarily improved with the completion of the Bulk Sample. Three-phase electricity has been restored to the site, the underground workings were temporarily dewatered, re-screened and re-bolted for stability, and underground ventilation was installed. The historical polishing ponds were used to treat water pumped from the re-established workings during the 2018/2019 Bulk Sample program. Following the completion of the Bulk Sample the underground workings flooded. A small storage building remains near the portal.

There are four historical tailings sites on the Property (Figure 5-3). These are referred to as West, South, Southeast, and East sites, with the material in the last comprising two separate areas along the stream flowing south of Gold Brook Lake. The Company has been indemnified by the Government of Nova Scotia with respect to historical tailings unless they are disturbed.

The Maritimes Northeast Limited natural gas pipeline crosses the western portion of the Property, and a former producing gas fractionation plant is located 3.5 km to the south. This plant processed gas from ExxonMobil Canada Ltd.'s Sable Project offshore production facilities near Sable Island, approximately 160 km to the east. Production ceased in the summer of 2019. A second gas pipeline also makes landfall in this area and served Encana Ltd.'s formerly producing Panuke gas project, with production ceasing in May 2018 (Figure 5-3 and Figure 5-4). Extensive work has also been completed by Pieridae Energy in this area with respect to the potential establishment of a liquefied natural gas (LNG) import facility, a petrochemical facility, and an export LNG facility.

The Gold Brook Lake and Gold Brook drainage system crosses the Property; other streams of lesser size are also present that could provide sufficient future plant and potable water requirements.

Sufficient undeveloped lands exist adjacent to the site to support potential tailings storage areas, potential waste disposal areas, heap leach pads, and potential processing plant sites.

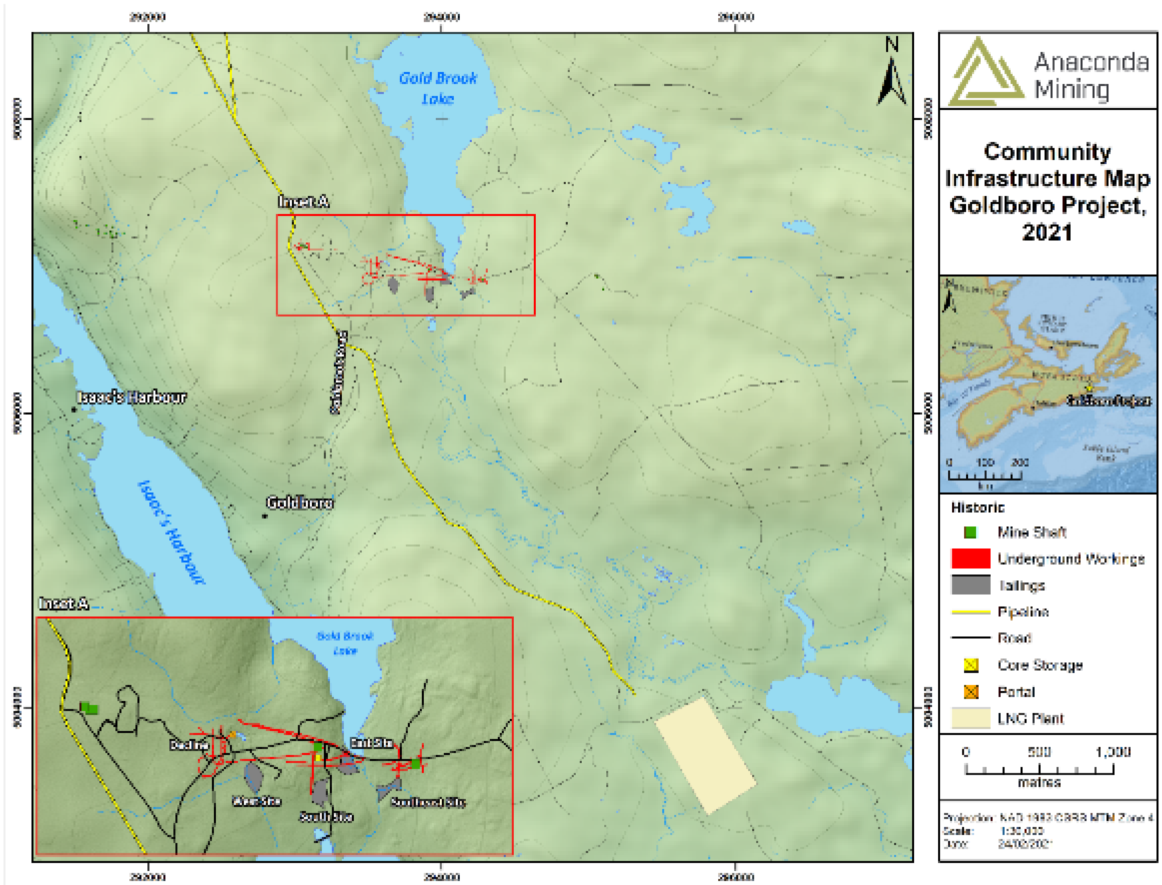



*Figure 5-4: Regional infrastructure*

## 5.3 Climate

The climate is moderated by the Atlantic Ocean and is typical of the eastern Canadian coast. Records for the Stillwater-Sherbrooke weather station, located 25 km west of the Property, recorded an average winter temperature of -3.8°C and an average summer temperature of 16.4°C for the 50-year period ending in 2012. The lowest recorded temperature was -39°C in February of 1985 and the highest recorded temperature was 35°C in June of 1976. The average winter monthly snowfall is 29 centimetres, and the average depth is 11 centimetres (Environment Canada). Field programs can be carried out throughout the year, with schedule allowances made for winter weather and wet site conditions during the spring thaw.

## 5.4 Physiography

The topography of the Property is characterized by gently rolling hills, with elevations ranging from 65 m to 80 m above sea level. A portion of the Property in the northeast is covered by Gold Brook Lake, which drains southward. The Property is moderate to heavily forested and covered with boulder-filled gravels, sandy clay, till, and muskeg. Swamp covers approximately one-fifth of the Property. The vegetation of the area includes secondary growth of tag alders, maple, birch, spruce, balsam, and tamarack. Outcrops of bedrock are rare on the Property.

# 6. HISTORY

Gold mineralization on the Property was first discovered in 1862 by Howard Richardson of the Geological Survey of Canada in quartz veins within the Isaac's Harbour anticline. The gold bearing BR Belt (slate and quartz) was subsequently discovered by Howard Richardson in 1892. Richardson Gold Mining began production from the belt in 1893.

From 1901 to 1905, three gold bearing belts were intersected in the Dolliver Mountain mine, located 2 km west of the Boston Richardson Mine.

From 1909 to 1910, the WG exploration shaft intersected five gold bearing belts. Three of these were mill tested, but the milling results were considered unsatisfactory, and the mine was abandoned.

In 1981, Patino Mines (Québec) Ltd. completed a geophysical program covering the Upper Seal Harbour district. In 1984, Onitap Resources Inc. (Onitap) acquired 37 claims overlying the Property.

Orex acquired the Property from Onitap in 1988. Excepting a period of inactivity from 1996 to 2004, Orex pursued both surface and underground exploration programs, including large amounts of core drilling, metallurgical testing programs, resource estimation programs, and economic assessments of the Property.

Osisko, under the terms of an agreement with Orex, carried out an extensive core drilling assessment of the Property during the 2010 to 2012 period.

In March of 2017, the Company acquired control of the Property under the terms of a court approved Plan of Arrangement whereby Orex became a wholly-owned subsidiary of the Company.

The summarized description of mining and exploration history prior to the acquisition of the Property by Orex in 1988 is presented herein. It reflects a compilation and modification of previous summaries by Faribault (1886), Malcolm (1976), and Gervais et al. (2009). A summary within Gervais et al. (2009) directly reflects earlier reporting by Rousseau (1990) and Roy (1998). Description of exploration work conducted prior to 1988 are outlined in Table 6-1, Table 6-2 and Table 6-3. Descriptions of exploration work carried out by Orex during the 1988 through 2017 period, up to the time of the merger with the Company, are presented, and have been taken directly from Cullen and Yule (2017), with minor modification as required to meet current report context regarding the Company (Table 6-4, Table 6-5 and Table 6-6). Exploration work conducted by the Company from 2017 to 2019 up to and including work disclosed in the NI 43-101 Technical Report, effective date August 21, 2019, (McCraken, T.; WSP Canada Inc., 2019) is outlined in Table 6-7.

## 6.1 Summarized Mining and Exploration History

**Table 6-1: Historic Regional Exploration 1858 to 1892**

| Activity | Operator |
|---|---|
| **1858** | |
| The first confirmed bedrock discovery of gold in Nova Scotia | Lieutenant C. L'Estrange |
| **1863** | |
| Gold was discovered in quartz veins by Howard Richardson on the Isaac's Harbour anticline. | Geological Survey of Canada |
| **1892** | |
| Discovery of the gold bearing BR Belt. | Howard Richardson |

**Table 6-2: Previous Project Exploration and Mining 1893 to 1910**

| Activity | Date | Performed By |
|---|---|---|
| **1893 to 1900** | | |
| Gold production commenced at an average reported grade of 13.03 g/t gold milled, with milling recoveries reported in the 50% to 60% range. | 1893 | The Richardson Gold Mining Company (Richardson Gold Mining) |
| Full capacity production with a 40-stamp mill. | 1896 | Richardson Gold Mining |
| Three mine shafts in operation from one shaft house. | 1897 | Richardson Gold Mining |
| 21,882 tonne milled with a total recovery of 4,479 oz. | 1898 | Richardson Gold Mining |
| 136 tonne of concentrates were recovered from mill tailings and treated by a Wilfley concentrator. | 1899 | Richardson Gold Mining |
| Substantial efforts were made to re-timber old workings. | 1900 | Richardson Gold Mining |
| **1901 to 1910** | | |
| Mill capacity was increased to 60 stamps, with the addition of two more Wilfley concentrators and an extensive cyanide plant from the Caribou Gold District. Work commenced at the Dolliver Mountain mine, located 2 km west of the Boston Richardson Mine. | 1901 | Richardson Gold Mining |
| 26,308 tonne of ore at an average gold grade of 4.08 g/t produced 3,459 oz. of gold. | 1902 | Richardson Gold Mining |
| Mining was suspended after an extensive collapse, attributed to insufficient support of the hanging wall, which destroyed the main shaft. | 1903 | Richardson Gold Mining |
| 8,059 tonnes an average gold grade of 0.87 g/t of gold 8,059, producing 205 oz. at the Dolliver Mountain mine. | 1904 | The Boston Richardson Mining Company (Boston Richardson Mining) |
| Several bodies of quartz and slate were intersected by a 152 m deep drill hole at the bottom of the main shaft along the anticlinal axis, but results were unsatisfactory, and mining at Dolliver Mountain mine ceased. | 1905 | Boston Richardson Mining |

| Activity | Date | Performed By |
|---|---|---|
| Recoveries approached 80% of the gold in concentrates, and half of the material mined was held as a reserve in the mine. | 1906 | Boston Richardson Mining |
| The EG property that adjoins the BR property to the east was acquired by F.S. Andrews and others. | 1907 | Boston Richardson Mining, F.S. Andrews and others |
| A shaft in EG was sunk 175 feet (53 m), and three promising gold bearing belts were explored in 1908. Operations were suspended on August 15, 1908, due to financial difficulties but were later resumed. | 1908 | Boston Richardson Mining |
| The mine was taken over by the New England Mining Company and processed an additional 37,572 tonne at an average grade of 4.14 g/t produced 5,024 oz. of gold. | 1909 | New England Mining Company (New England Mining) |
| 24,440 tonne milled an average gold grade of 5.17 g/t, producing 4,063 oz. | 1910 | New England Mining |
| Unsatisfactory mill test results from five gold bearing belts within the WG exploration shaft contributed to mine abandonment. | 1910 | New England Mining |

**Table 6-3: Previous Project Exploration and Mining 1926 to 1987**

| Activity | Date | Performed By |
|---|---|---|
| **1926 to 1987** | | |
| The Metal Mining and Smelting Corporation of Canada Ltd. took over the property and treated tailings to recover auriferous arsenopyrite through 1927. | 1926 to 1927 | Metal Mining and Smelting Corporation of Canada Ltd. |
| Locarno Copper Mines Ltd. sank a shaft on the WG property, west of the earlier shaft. In 1931, a metallurgical test recovered 1.61 oz. of gold from 1 tonne of feed, producing an average gold grade of 50.1 g/t. | 1929 to 1931 | Locarno Copper Mines Ltd. |
| Old workings at EG were dewatered and resampled with grades between 1.61 and 4.26 g/t. | 1931 to 1934 | Renada Mines Ltd. |

| Activity | Date | Performed By |
|---|---|---|
| The shaft at WG was dewatered, and cross-cutting was carried out. This work ceased due to financial difficulties. | 1956 | Canso Mining Corporation |
| Detailed geophysical program confirmed geological continuity of anticlinal structure from Dolliver Mountain to BR. | 1981 | Patino Mines (Québec) Ltd. |
| 37 claims were acquired, covering the BR, EG, WG, and Dolliver Mountain prospects. Surface drilling campaign of one core hole (BR-84-01) totalling 529 m. | 1984 | Onitap Resources Inc. |
| Surface drilling campaign of five drill holes (BR-85-01 to BR-85-04, and BR-85-01 A) totalling 390 m. These holes intersected several belts. | 1985 | Onitap Resources Inc. |
| Two surface drilling campaigns of five drill holes (BR-87-01 to BR-87-05) and 33 drill holes (BR-87-06 to BR-87-38), totalling 1,924 m and 11,621 m respectively; four new mineralized belts were intersected stratigraphically beneath the Historically producing BR Belt. | 1987 | Petromet Resources Ltd. and Greenstrike Gold Corp., Onitap Resources Inc. |
| Ground IP and helicopter-borne geophysical (VLF-EM and magnetics) surveys. | 1987 | Onitap Resources Inc. |

**Table 6-4: Previous Project Exploration 1988 to 1995**

| Activity | Date | Performed By |
|---|---|---|
| **1988 to 1996** | | |
| Surface drilling campaign completed 41 drill holes (BR88-39 to -79) totalling 10,822 m 14 new belts discovered. | 1988 | Orex, Onitap Resources Inc. |
| Access decline and two levels (38 m and 76 m) established; underground drilling campaign completed; three drill holes (BR-88-80 to BR-88-82) totalling 459 m; rehabilitation of Boston Richardson shaft to 122 m, hoist, and headframe installed. | 1988 | Orex, Onitap Resources Inc., Narex Ore Search Consultants Inc. |
| Underground drilling campaign completed with 112 drill holes (88 U-01 to 88 U-04, 89 U-05 to 89 U-26, and 90 U-27 to 90 U-112) totalling 4,979 m). | 1988 to 1990 | Orex |

| Activity | Date | Performed By |
|---|---|---|
| Surface drilling campaign of 26 drill holes completed (BR-89-83 to BR-89-108) totalling 2,811 m. | 1989 | Orex |
| Mill testing was completed on core material from six drill holes (BR-88-48, BR-88-60, BR-88-61, BR-88-62, BR-88-85, and BR87-35 A) that intersected the first five belts of the BR Gold System near the anticlinal apex. On the basis of unreproducible core sample results, it was concluded that bulk sampling was the best way to determine gold grade without substantive discrepancy attributable to nugget effect being encountered( (Parent, 1989), from (Dionne & Vachon, 1989)). | 1989 | Orex |
| Two underground bulk samples (Sample A and Sample B (~7 tonnes each) were mined out from South limb of the N1 belt at 38 m level and 76 m level, respectively; Samples A and B graded 2.54 g/t and 4.07 g/t gold, respectively. | 1989 to 1990 | Orex, Lakefield Research, Falconbridge Limited. |
| The Property was optioned to Minnova Exploration Inc. (Minnova) with the right to acquire 60% interest in the Property by investing a total sum of $5 million in exploration work during the next three years (Labelle, J.P.; Orex Exploration Inc., 1991). | 1991 | Orex, Minnova |
| Surface drilling campaign of four twinned drill holes (BR-91-109 to BR-91-113) targeting BR88-48, BR88-62, BR88-60, and BR87-35 A totalling 722 m. | 1991 | Minnova |
| Results from crushing, grinding, and continuous-grind-leach (CGL) processing methods of the 1991 drilling campaign showed that gold was not being fully recovered by such methods due to various factors, the most prominent being incomplete leaching by the cyanide solution; Minnova subsequently withdrew from the option agreement. | 1991 | Minnova |

| Activity | Date | Performed By |
|---|---|---|
| An evaluation of vat leaching (VL) as a method for testing and processing the ore from the Property was initiated. After four hours of leaching, recovery of 84% of the contained gold was achieved at grades in the 4 g/t to 6 g/t range. Novagold subsequently dropped the option due to financial difficulties (Bourgoin, M.; MRB & Associates, 2004). | 1992 | Novagold Resources Inc. |
| Surface drilling campaign of four twinned drill holes (BR-91-114 to BR-91-117) targeting BR-91-109, BR-88-85, BR-91-110, and BR-91-113, respectively, and three follow up twinned holes (BR-93-114B to BR-93-116B) targeting BR-93-114 and BR-93-116 totalled 593 m and provided material for further metallurgical testing. | 1993 | Orex |
| The Centre de Recherches Minérales achieved recoveries of less than 85% gold. | 1993 | Orex, The Centre de Recherches Minérales |
| Lithology-specific sampling program focused on a 4,000 tonne surface stockpile generated during the underground program, which provided material for the VL tests carried out by Novagold. | 1995 | Orex, Placer Dome Inc. (Placer) |
| Placer entered an option to invest $30 million to acquire 65% in the Property after receiving assays from the VL test. | 1995 | Orex, Placer |
| Surface drilling campaign of seven holes near ramp portal (BR95-119 to -125) totalling 1263 m to investigate open pit potential. | 1995 | Orex, Placer |
| Based on the results of its Property assessment, Placer opted to cease involvement in the Property after completion of the 1995 drilling program. | 1995 to 1996 | Orex |

**Table 6-5: Comparison of Placer Dome Stockpile Processing Results**

| Lithology | Calculated Gold Head Grade (g/t) | Assayed Gold Head Grade (g/t) |
|---|---|---|
| Greywacke | 1.00 | 1.00 |
| Slate | 2.64 | 2.82 |
| Quartz veins (bull quartz) | 16.18 | 6.16 |
| Mineralized quartz veins | 33.62 | 36.37 |

Source: Gagnon et al, 1996

**Table 6-6: Previous Project Exploration 2004 to 2017**

| Activity | Date | Performed By |
|---|---|---|
| **2004 to 2017** | | |
| Review of geology and evaluation of characteristics of Meguma-style deposit as compared to Bendigo and Ballarat deposits (Australia); review of historical sampling and analytical grade determination procedures; preparation of a Mineral Resource Estimate in accordance with NI 43-101 and the CIM Standards at its August 31, 2004, effective date. | 2004 | Orex, MRB & Associates Ltd. (MRB), InnovExplo inc. (InnoveExplo), Horvath Consulting, Tech2Mine Inc. |
| Conclusions drawn in the 2004 Resource Estimate recognize that historical sampling, processing, and nugget effect required a larger, more representative sample size to reflect the grade adequately. | 2004 | Orex., MRB, InnovExplo, Horvath Consulting, Tech2Mine Inc. |
| Surface drilling campaign of 23 drill holes completed (BR-05-001 to BR-05-023) totalling 2,422 m); twinned four historical drill holes for comparative analysis; metallurgical testing. | 2005 | Orex |
| Updated Mineral Resource Estimate for the 'Ramp Area' in 2006 by P&E Mining Consultants Inc. (P&E Mining) using results from the 2005 drill campaign in combination with validated historical drilling results. | Aug-06 | Orex, P&E Mining |
| Phase 2A and 2B programs consisting of 45 infill drill holes completed (BR-08-01 to BR-08-44, and BR-08-20 A) totalling 12,065 m; the focus was placed on a 975 m long area between the WG and EG Systems to infill on previous holes and to test for mineralization extensions east and west of the BR System. | 2008 | Orex |

| Activity | Date | Performed By |
|---|---|---|
| Updated Mineral Resource Estimate for the Property. | Jul-09 | Orex, InnovExplo |
| Orex signed an option to joint venture agreement with Osisko to acquire a 50% undivided interest in the Property on or before September 29, 2013. | Nov-09 | Orex, Osisko |
| Phase 2D, 2E, 2F drilling campaigns of 59 drill holes completed (OSK10-01 to OSK10-59) totalling 12,998 m; unsampled intervals from 97 historical drill holes were assayed. | 2010 | Orex, Osisko |
| Completion of 64 RC holes (OSK10RC-130 to 194) totalling 505 m in the EG, BR Ramp, and WG Areas; positive results from EG included 10.85 g/t gold (Au) and 25.65 g/t gold in bedrock chip samples. | 2010 | Orex, Osisko, Minerals Engineering Centre (MEC) |
| Drilling campaigns of six holes (OSK11-01 to OSK11-06) and four holes (OSK11-07 to OSK11-10) completed, totalling 2,375 m and 765.4 m, at EG and between WG, and Dolliver Mountain, respectively. | 2011 | Orex, Osisko |
| Osisko terminated its option to acquire an interest in the Property. | Sep-11 | Orex, Osisko |
| An updated geological model for the Deposit was created, with a specific focus on the definition of discrete, higher-grade mineralized belts with underground mining potential, contrasting earlier corporate focus on large open pit potential. | 2012 to 2013 | Orex, Mercator Geological Services Limited (Mercator) |
| The revised geological model developed, in conjunction with results from 69 drill holes completed in 2010 and 2011, provided the basis for a new Resource Estimate by Mercator Geological Services Limited. | 2012 to 2013 | Orex, Mercator |
| MineTech International Limited (MineTech) completed a 2014 PEA of the Project based on the 2013 Mineral Resource Estimate prepared by Mercator. This analysis was focused on the assessment of underground mining project viability and was presented in the NI 43-101 Technical Report prepared by MineTech for Orex (Roy, D.; Hannon, P.; Dickie, L.; MineTech International Limited, 2014). | 2014 | Orex, MineTech |

| Activity | Date | Performed By |
|---|---|---|
| A revision to the 2013 Mineral Resource Estimate was provided to address potential discrepancies in the resource model, and management pursued opportunities for further property assessment and potential development. | Feb-17 | Orex, Mercator |
| Plan of Arrangement between Orex and the Company whereby Orex became a wholly-owned subsidiary of the Company. | May-17 | Anaconda, Orex |

**Table 6-7: Project Exploration 2017 to 2019**

| Activity | Date | Performed By |
|---|---|---|
| Thirteen drill holes were drilled targeting the BR and EG Gold Systems, totalling 4,196.3 m (BR-17-01 to BR-17-13). This includes drill holes BR-17-01 to BR-17-05 that were drilled at HQ diameter and twinned historical drill holes BR-87-34, BR-88-22, BR-88-48, BR-05-03, and BR-08-17. The core from these five drill holes was also used for metallurgical analysis. | 2017 | Anaconda, Orex |
| The EG, BR, and WG Gold Systems were targeted by 61 drill holes totalling 18,277.3 m (BR-18-17 to BR-18-71, BR-18-23 A, BR-18-39 A, and BR-18-44 A). Four drill holes (BR-18-68 to BR-18-71) were drilled at HQ diameter to collect core samples for metallurgical analysis. | 2018 | Anaconda, Orex |
| Conducted an underground Bulk Sample. 13,028 tonnes mined with 10,023 wmt (9,785 dmt) shipped to the mill at Point Rousse near Baie Verte, NL. Average head grade of 3.81 g/t, gravity recoveries consistent with metallurgical test work and strong reconciliation. | 2018 | Anaconda, Orex |
| A drill program consisting of 33 drill holes (BR-19-72 to BR-19-104) totalling 5,733.8 m that targeted the EG, BR, and WG Gold Systems was completed. | 2019 | Anaconda, Orex |

### 6.1.1 Bulk Sample

In the fall of 2018 and early winter of 2019, the Company completed a 13,028 tonne Bulk Sample within the BR Gold System. The objectives of the Bulk Sample were to confirm the geological interpretation of the Deposit, test for spatial and grade continuity of the mineralized structures, validate results and key assumptions of the December 2019 Mineral Resource model, and test certain types of mining methods.

Of the total Bulk Sample, 10,023 wmt (9,785 dmt) were shipped to the Company's Pine Cove Mill in Baie Verte, NL for processing into gold doré bars. The remaining portion of the total Bulk Sample is currently stockpiled on the Property.

Key Takeaways of the Bulk Sample

- Successfully tested an area within the 2019 Mineral Resource Estimate with respect to continuity of gold grade and geological interpretation, confirming the position, and continuity of mineralized zones.
- The average diluted mine grade based on grade control samples of 3.61 g/t gold reconciles well with the average undiluted grade of the December 2019 Mineral Resource block model of 3.81 g/t gold in the area of the Bulk Sample.
- The average head grade of 3.81 g/t gold from the Pine Cove Mill shows a positive reconciliation of 5.5% to the mine grade of 3.61 g/t gold, demonstrating an upside bias within an acceptable range.
- High gravity recovery of 51% for higher grade areas of the Deposit, confirming metallurgical test work.
- Demonstration of excellent ground conditions.

Geology and Test Mining

The Bulk Sample was extracted from the BR Gold System within the Deposit, leveraging existing decline, and cross cuts developed in the late 1980s to minimize the development needed to access mineralized zones. The 2019 Mineral Resource for the specific mine areas targeted for the Bulk Sample predicted 11,274 tonnes of Mineral Resource at an average undiluted grade of 3.81 g/t gold, capped at 80 g/t. A total of 13,028 tonnes of mineralized material was mined and stockpiled on surface, at an average diluted grade of 3.15 g/t gold based on grade control samples capped at 80 g/t gold.

All underground chip, muck, and drill sludge samples were shipped daily to the contract assay lab at the Dufferin Mine, near Sheet Harbour, Nova Scotia, where they were analyzed for gold by FA. A total of 769 samples (coarse reject material) assaying greater than 1.50 g/t gold were shipped to Eastern Analytical for check analysis via metallic screen FA. QC samples were systematically inserted into each sample batch and consisted of two prepared powdered gold standards (CRM) and a natural blank. Results were assessed for accuracy, precision, and contamination on an ongoing basis.

Six areas of development were completed on three levels at mine elevations of 4965 (Level 1), 4930 (Level 2 West, Level 2 East, Level 2 Ramp), and 4920 (Level 3 West and Level 3 East) (Figure 6-1). New development, in excellent ground conditions, included three drifts off the existing cross cuts on Level 1 and Level 2 as well as the creation of a newly developed ramp to Level 3 with two additional drifts resulting in a total of 213 m of new development. A total of 86 drift rounds were blasted, which allowed for inspection, mapping, and sampling from the faces of the development headings. Information was also collected from the faces of 13 slot raise rounds during the development of the stopes. Drifting was advanced using a single boom jumbo, typically with one round blasted per 12-hour shift. Stoping was completed using a longhole method using both up-holes and down-holes.

Blasted rock was mucked using 1.5 and 2.0 cubic yard scoop trams, and then loaded into a 7.0 cubic yard truck to haul to surface.

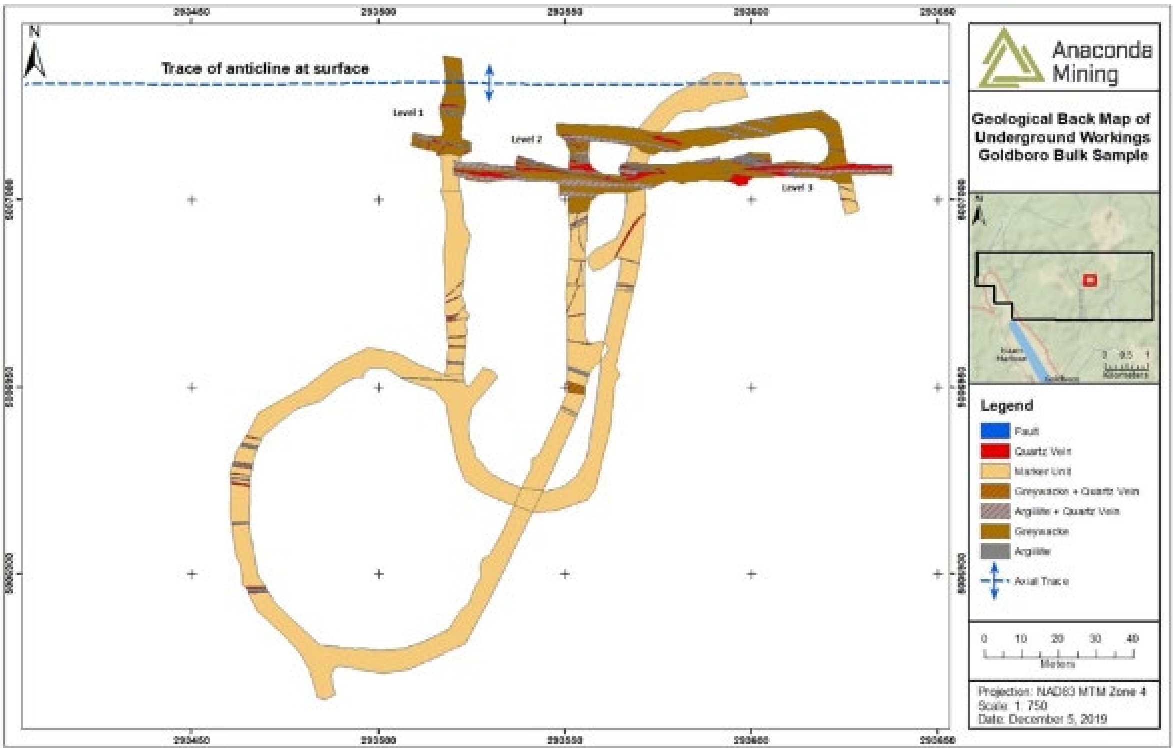



*Figure 6-1: Plan view of Bulk Sample workings showing the geology as mapped during the Bulk Sample*

Development faces were geologically mapped, photographed, and sampled (Figure 6-2, Figure 6-3, Figure 6-4 and Figure 6-5). Chip and muck sampling was completed systematically to determine the average grade for each round. A total of 2,148 muck and 1,380 chip samples were collected, as well as drilling and sampling of 518 test holes and 132 long holes, in addition to QC samples.

*Figure 6-2: Level 1 West Round 1 - thin argillite-quartz veins adjacent to extensional quartz veins (looking west)*

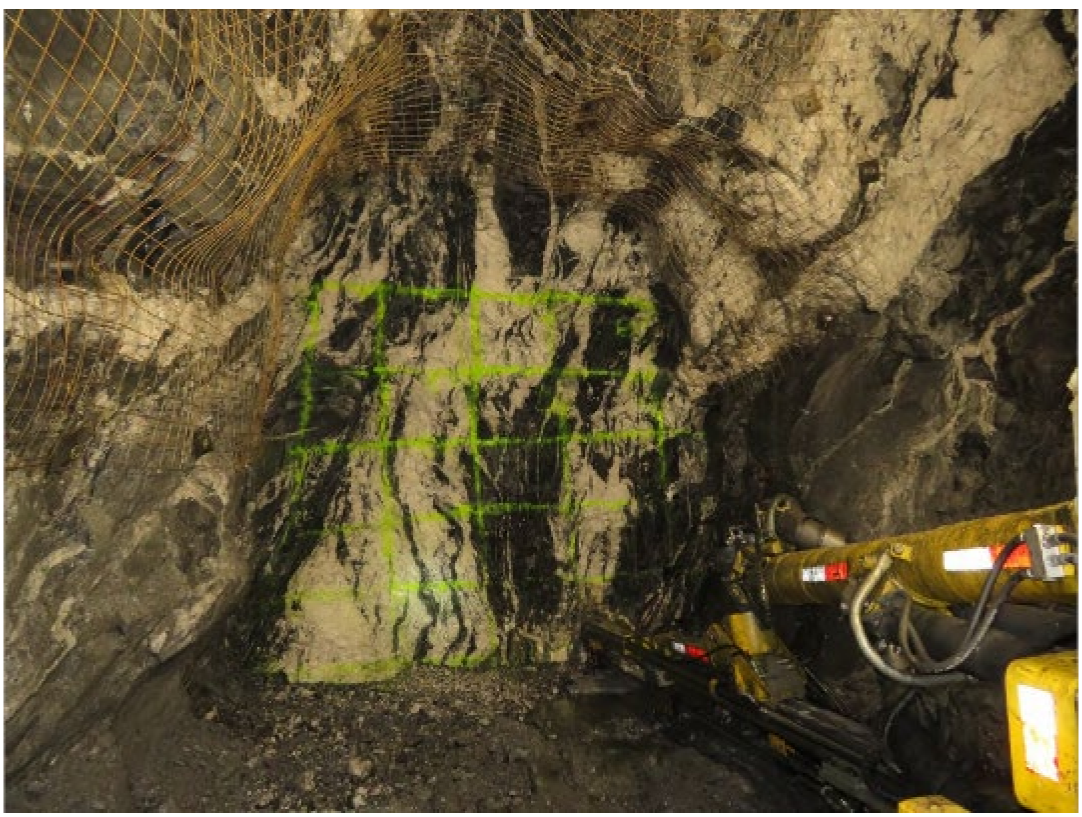


*Figure 6-3: Level 2 West Round 6 - North and South Zone of Belt 1 of the BR Gold System (looking west)*

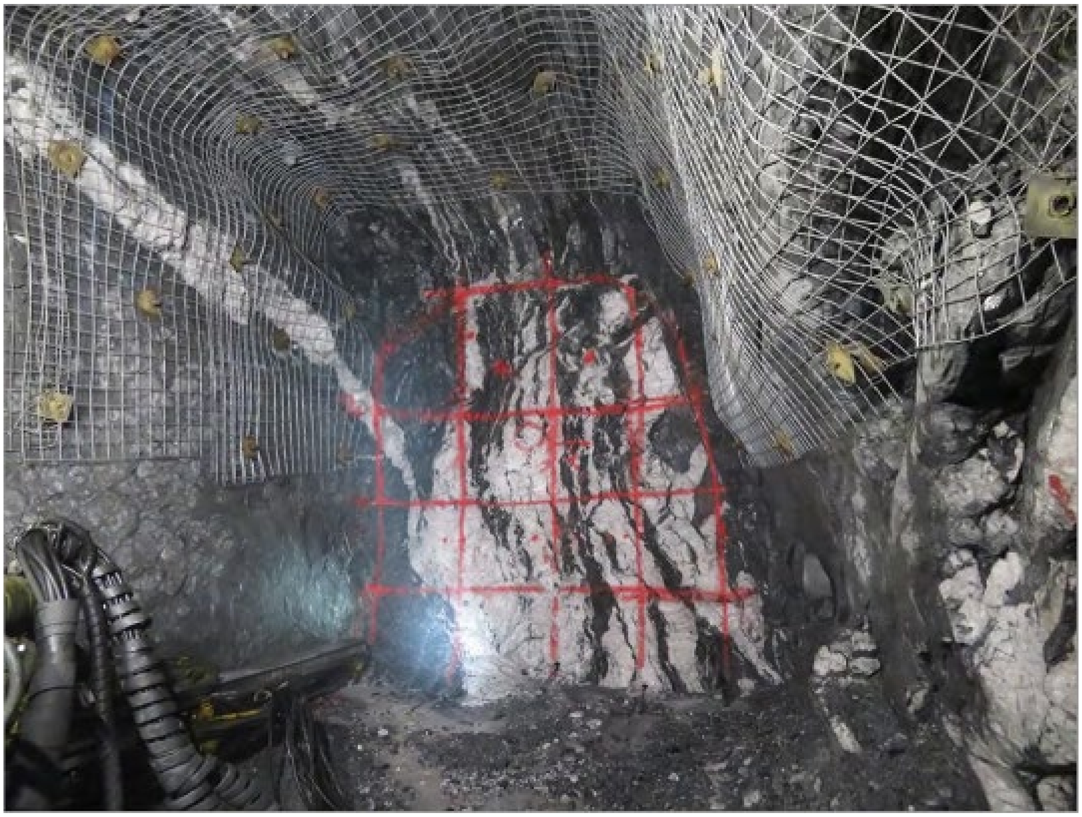


*Figure 6-4: Level 3 East Round 2 (looking east)*

*Figure 6-5: Visible gold along margin of quartz vein-argillite contact in Level 3 East Round 3*

Surface Stockpiling and Shipping

Rock from each round/blast was trucked and stockpiled separately on surface. Upon receipt of chip and muck sample assays, many of the rounds were consolidated into larger stockpiles based on average grade; high-grade (>3.0 g/t gold), low-grade (1.0 g/t to 3.0 g/t gold), and rock that was considered waste (<1.0 g/t gold) for the Bulk Sample exercise. 982 tonnes of waste material were used to line the ore pad and build safety berms around the site.

Of the total 13,028 tonnes of material brought to surface a subsample based on grade was selected from the existing high and low-grade stockpiles for shipping to Point Rousse (Table 6-7). A total of 6,735 tonnes at an average grade of 4.45 g/t gold of high-grade (>3.0 g/t gold) and 3,288 tonnes at an average grade of 1.87 g/t gold of low-grade (1.0 g/t to 3.0 g/t gold) stockpiled material was shipped. Of the total tonnes extracted, 3,005 tonnes remain stockpiled at the site.

**Table 6-7: Total Bulk Sample Material Shipped and Processed at the Pine Cove Mill in Baie Verte, NL**

| Bulk Sample Shipped (Diluted) | | | Head Grade Through Mill | | | Variance (%) |
|---|---|---|---|---|---|---|
| Tonnes (wet) | g/t Gold[2] | Ounces | Tonnes (dry) | g/t Gold[3] | Contained Ounces | |
| 10,023 | 3.61 | 1,152 | 9,785 | 3.81 | 1,197 | 5.5% |

Processing, Grade Reconciliation, and Gold Recovery

The 9,785 dmt were milled via a combination of gravity recovery followed by sulphide flotation and cyanide leaching. The mill recovered 610 oz of gold from the gravity concentration circuit, with an additional 360 oz being recovered through the flotation and leaching circuits, for a total of 970 oz of gold recovered and an overall recovery of 81%. Concentrate from the gravity circuit was further cleaned using a shaker table and additional gravity concentration processes to ultimately produce gold doré bars.

The overall recovery from the Pine Cove Mill was less than those predicted by the metallurgical test work, but this is because the Pine Cove Mill flow sheet does not match those of the metallurgical test work. The Pine Cove Mill uses a flotation circuit rather than whole ore leach.

Composite samples were taken from key sampling points in the process, including gold contained in tailings, were used in conjunction with the actual gold recovered and milled tonnage to calculate an average head grade of 3.81 g/t gold. Mill feed tonnages used in the sample processing reconciliation were provided daily. The Company collected 6-hour composite samples during the processing of the Bulk Sample, which was assayed by Eastern Analytical. Daily composite samples were taken on feed and tail streams for reconciliation purposes. Mill throughput for the Bulk Sample was estimated based on the feed belt scale, which was calibrated monthly. All samples were sent out to an external independent lab for preparation and assaying for both gold and arsenic. Gold assays were obtained by screened metallics FA. For all the other samples, the gold assay was obtained by FA-AA finish or gravimetric finish.

[2] Mine grade was based on muck samples taken from every second truck during the mining process, estimated using metallic screening with an 80 g/t gold top cut.

[3] Contained ounces and head grade calculated based on gold produced from the Pine Cove Mill plus gold contained in tailings.

The reconciled results from the processing of the Bulk Sample material are presented in Table 6-8.

**Table 6-8: Reconciled Recoveries from the Processing of the Bulk Sample at the Pine Cove Mill**

| Tonnes Processed (Dry) | Head Grade | Contained Ounces | Gravity Concentrate | | Flotation Concentrate Leach | | Overall Recovery | Recovered Ounces |
|---|---|---|---|---|---|---|---|---|
| | Au (g/t) | Au | Au Rec (%) | Au Oz | Au Rec (%) | Au Oz | (%) | Au |
| 9,785 | 3.81 | 1,197 | 51% | 610 | 30% | 360 | 81% | 970 |

The calculated head grade from processing was 3.81 g/t gold, a positive reconciliation of 5.5% compared to the average mined grade of 3.61 g/t gold. The Bulk Sample results indicate that good reconciliation of the diluted material mined compared to the block model was achieved, and the reconciliation of the calculated head grade to the average mine grade indicates that the capping of 80 g/t gold is appropriate, with an upside bias.

The gravity recovery compared well to bench-scale testing as part of metallurgical testing completed by Base Metallurgical Laboratories Ltd., under the supervision of Ausenco Engineering Canada Inc. (Ausenco) (Anaconda Mining Inc., 2019); (Anaconda Mining Inc., 2021).

## 6.2 Previous Mineral Resource Estimates

### 6.2.1 2004 Technical Report Dated August 31, 2004

The Mineral Resource Estimate developed by the MRB & Associates study is presented in Table 6-9 and was considered to be in accordance with NI 43-101 and the CIM Standards at its August 31, 2004, Technical Report effective date. These resources are now historical in nature and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

**Table 6-9: Historic MRB Resource Estimate–August 31, 2004**

| Zone | Cut-off (Au g/t) | Resource Category | Metric Tonnes | Au g/t | Ounces Au |
|---|---|---|---|---|---|
| **Boston Richardson Area** | 0.00 | Measured Resource | 755,000 | 1.21 | 29,300 |
| | 0.00 | Indicated Resource | 12,500,000 | 0.75 | 301,000 |
| | 0.00 | **Subtotal** | **13,255,000** | **0.78** | **330,300** |
| | 0.00 | Inferred Resource | 7,000,000 | 0.78 | 175,500 |
| **West Goldbrook** | 0.00 | Inferred Resource | 8,600,000 | 0.50 | 138,200 |

Foremost among conclusions and recommendations of the 2004 Technical Report by MRB was the recognition that historical sampling, processing, and analytical gold determination protocols have led to consistent under-estimation of gold grades above the 1 g/t level due to extreme nugget effect; and that the most reliable method of obtaining accurate and precise grade determinations for mineralized samples is by metallurgical testing/processing of adequately sized representative samples.

### 6.2.2 2006 Technical Report Revised Date September 18, 2006

Orex carried out an exploration drilling program in 2005 that concentrated on a 1,500 m long by 225 m wide area centred on section 8675E of the BR Gold System. A total of 23 drill holes (2,355 m) were completed in this program, and samples were prepared, and analyzed using revised protocols. The results were used in combination with validated historical drilling results to support a NI 43-101 Resource Estimate for the 'Ramp Area' in 2006 by P&E Mining Consultants Inc. (P&E Mining) (Puritch, Armstrong, & Horvath, 2006). This estimate was limited to the central deposit area where ramp development, bulk sampling, and metallurgical processing of composited drill core samples had been completed, and estimation results are presented in Table 6-10. These resources are now historical in nature and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

**Table 6-10: Historic P&E Mining Resource Estimate–Effective August 21, 2006**

| Zone | Cut-off (Au g/t) | Resource Category | Tonnes | Grade (Au g/t) | Contained Ounces (Au) |
|---|---|---|---|---|---|
| **Ramp Area** | 0.5 | Measured Resource | 481,000 | 3.40 | 52,600 |
| | 0.5 | Indicated Resource | 2,624,000 | 2.17 | 183,400 |
| | 0.5 | Measured Plus Indicated | 3,105,000 | 2.36 | 236,000 |

A Conceptual Target as defined under NI 43-101 was identified by P&E Mining at the time of resource estimation. The Conceptual Target consisted of the potential gold inventory increase that would result if the Ramp Area Indicated Resources were increased in gold grade by a 1.23 g/t. This resulted in the definition of potential within the existing Indicated category tonnage for a gold inventory increase of 103,500 oz. Confirmation of such an increase would require metallurgical processing of all available core materials from each contributing drill hole in the Indicated resource volume (Puritch, Armstrong, & Horvath, 2006). The P&E Mining Resource Estimate is now historical in nature and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

### 6.2.3 2009 Technical Report Dated September 15, 2009

Another drilling program was carried out in 2008 by Orex, and this consisted of 45 drill holes (12,201.5 m). The focus was placed on a 975 m long area between the WG and EG Gold Systems to infill on previous holes and to test for mineralization extensions east and west of the BR System. InnovExplo subsequently provided a revised Mineral Resource Estimate for the Property in a Technical Report dated September 15, 2009 (Table 6-11).

**Table 6-11: Historic InnovExplo Resource Estimate–September 15, 2009**

| Cut-off (Au g/t) | Resource Category | Tonnes | Grade (Au g/t) | Contained Ounces (Au) |
|---|---|---|---|---|
| **1.5** | Measured Resource | 270,000 | 4.99 | 43,300 |
| **1.5** | Indicated Resource | 2,441,000 | 4.51 | 353,900 |
| **1.5** | Measured Plus Indicated | 2,711,000 | 4.56 | 397,200 |
| **1.5** | Inferred | 3,438,000 | 3.67 | 405,900 |

The InnovExplo Resource Estimate is now historical in nature and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

### 6.2.4 2013 Technical Report Dated April 15, 2013

On March 30, 2012, Orex retained Mercator Geological Services Limited (Mercator) to manage work programs and complete an updated Mineral Resource Estimate (Cullen, M.; Yule, S.; Mercator Geological Services Ltd., 2013). An updated geological model for the Deposit was created, with a specific focus on the definition of discrete, higher-grade mineralized belts with underground mining potential. The updated Mineral Resource Estimate with an effective date of February 11, 2013, included results from 69 drill holes completed in 2010 and 2011 (Table 6-12). This Mineral Resource Estimate has been superseded and is now considered historical in nature and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify this historical estimate as current Mineral Resources or Mineral Reserves, and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

**Table 6-12: Historic Mercator Resource Estimate–Effective February 11, 2013**

| Gold Cut-off g/t | Resource Category | Boston Richardson Zone | | West Goldbrook Zone | | East Goldbrook Zone | | Total Goldboro Deposit | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Tonnes* | Au g/t | Tonnes * | Au g/t | Tonnes* | Au g/t | Tonnes* | Au g/t | Ounces* |
| **2.00** | Measured | 171,000 | 5.39 | | | | | 171,000 | 5.39 | 29,600 |
| | Indicated | 1,472,000 | 5.44 | 473,000 | 5.34 | 473,000 | 6.37 | 2,418,000 | 5.60 | 435,300 |
| | **Subtotal** | **1,643,000** | **5.43** | **473,000** | **5.34** | **473,000** | **6.37** | **2,589,000** | **5.59** | **465,000** |
| | Inferred | 953,000 | 5.04 | 345,000 | 4.40 | 1,245,000 | 5.45 | 2,543,000 | 5.15 | 421,100 |

*Notes:

- Tonnages have been rounded to the nearest 1,000 tonnes, and ounces have been rounded to the nearest 100; average grades and contained ounces may not sum due to rounding.
- The resource statement gold CoG is 2.0 g/t.
- The 2.0 g/t gold resource statement CoG reflects a reasonable expectation of economic development by underground mining methods based on a three-year trailing average gold price of US $1,492.
- Contributing 1.0 m assay composite populations were capped at gold grades of 80 g/t or 120 g/t and separately interpolated.
- A bulk density factor of 2.7 kg/m$^3$ was applied to all blocks.

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

In 2014, MineTech International Limited (MineTech) completed the 2014 PEA of the Project based on the 2013 Mineral Resource Estimate prepared by Mercator. This analysis was focused on the assessment of underground mining project viability and was presented in the NI 43-101 Technical Report prepared by MineTech for Orex (Roy, D.; Hannon, P.; Dickie, L.; MineTech International Limited, 2014).

### 6.2.5 2017 Technical Report Dated April 1, 2017

In February of 2017, Mercator provided a revision to the 2013 Mineral Resource Estimate, which superseded the 2014 PEA Technical Report by MineTech. The revised Mineral Resource Estimate, with an effective date of February 28, 2017, by Mercator was prepared to correct a non-material assay compositing error that affected certain incompletely sampled drill core intervals from historical drilling programs used in the 2013 estimate (Table 6-13). The 2013 and 2017 Mineral Resource Estimates, as well as the 2014 PEA, have been superseded by subsequent work, are historical in nature, and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves, and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

**Table 6-13: Historic Mercator Mineral Resource Estimate–Effective February 28, 2017**

| Gold Cut-off g/t | Resource Category | Boston Richardson Zone | | West Goldbrook Zone | | East Goldbrook Zone | | Total Goldboro Deposit | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Tonnes* | Au g/t | Tonnes* | Au g/t | Tonnes* | Au g/t | Tonnes* | Au g/t | Ounces* |
| **2.00** | Measured | 171,000 | 5.39 | | | | | 171,000 | 5.39 | 29,600 |
| | Indicated | 1,507,000 | 5.27 | 464,000 | 5.39 | 414,000 | 6.91 | 2,385,000 | 5.58 | 427,800 |
| | **Subtotal** | **1,678,000** | **5.28** | **464,000** | **5.39** | **414,000** | **6.91** | **2,556,000** | **5.57** | **457,400** |
| | Inferred | 10,083,000 | 4.56 | 459,000 | 4.42 | 1,127,000 | 4.11 | 2,669,000 | 4.35 | 372,900 |

* Notes:

- This revised Resource Estimate was prepared in accordance with National Instrument 43-101 and the CIM Standards.
- Tonnages have been rounded to the nearest 1,000, and ounces have been rounded to the nearest 100; average grades and contained ounces may not sum due to rounding.
- The 2.0 g/t gold resource statement threshold grade reflects a reasonable expectation of economic extraction by underground mining methods.
- Contributing 1.0 m assay composite populations were capped at gold grades of 80 g/t or 120 g/t and separately interpolated.
- A bulk density factor of 2.7 kg/m$^3$ was applied to all blocks.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

### 6.2.6 2018 Technical Report Dated March 1, 2018

In 2017 an updated Mineral Resource Estimate was completed by Mercator Geological Services Ltd. (Michael Cullen, P.Geo., Independent QP) with an effective date of December 31, 2017, (Robinson, J.; Cullen, M.; Liukko, G.; Bertelegni, S.; Thibault, J.D.; Slepcev, G.; Mercator Geological Services Ltd., 2018). The Mineral Resource Estimate comprises an open pit resource including 1,059,00 tonnes of Measured and Indicated resource at a grade of 3.01 g/t gold (102,500 ounces) and 45,000 tonnes of Inferred Mineral Resource at a grade of 2.54 g/t gold (3,700 ounces) at a CoG of 0.5 g/t gold; and an underground resource including 2,586,000 tonnes of Measured and Indicated Mineral Resources at a grade of 5.09 g/t gold (422,900 ounces) and 2,497,000 tonnes of Inferred Resource at a grade of 4.28 g/t gold (343,600 ounces) at a CoG of 2.0 g/t gold (Table 6-14). The 2017 updated Mineral Resource Estimate has been superseded by subsequent work, is historical in nature, and should not be relied upon. A QP, as defined in NI 43-101, has not done sufficient work to classify these historical estimates as current Mineral Resources or Mineral Reserves, and the Company is not treating or considering these as being current Mineral Resources or Mineral Reserves.

**Table 6-14: Historic Mercator Mineral Resource Estimate–Effective December 31, 2017**

| Classification | Resource Type | Cut-off (Au g/t) | Mercator 2017 | | |
|---|---|---|---|---|---|
| | | | Tonnes (Rounded) | Grade (Au g/t) | Ounces (Rounded) |
| Measured | Open Pit | 0.5 | 397,000 | 2.88 | 36,800 |
| | Underground | 2.0 | 22,000 | 4.70 | 3,300 |
| Indicated | Open Pit | 0.5 | 662,000 | 3.09 | 65,800 |
| | Underground | 2.0 | 2,564,000 | 5.09 | 419,600 |
| Measured and Indicated | Open Pit | **0.5** | 1,059,000 | 3.01 | 102,500 |
| | Underground | **2.0** | 2,586,000 | 5.09 | 422,900 |
| Inferred | Open Pit | 0.5 | 45,000 | 2.54 | 3,700 |
| | Underground | 2.0 | 2,497,000 | 4.28 | 343,600 |

Notes:

- Mineral Resources were prepared in accordance with NI 43-101 and the CIM Standards (2014).
- Open pit Mineral Resources are reported at a CoG of 0.5 g/t gold within the WSP base case pit shell and are based on a gold price of C$1,550/oz and a gold processing recovery factor of 95%. These include PEA base case open pit resources that have an estimated life of mine strip ratio of 7.3:1 (waste tonnes: PEA tonne).
- Appropriate mining costs, processing costs, metal recoveries, and inter-ramp pit slope angles were used by WSP to generate the base case pit design.
- Rounding may result in apparent summation differences between tonnes, grade, and contained metal content. Tonnages have been rounded to the nearest 1,000 tonnes; ounces have been rounded to the nearest 100 ounces.
- Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.
- Contributing assay composites were capped at 80 g/t gold.
- A density factor of 2.70 g/cm$^3$ was applied to all blocks.

- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

### 6.2.7 2018 Technical Report Dated December 10, 2018

In 2018 the Company contracted WSP to provide an update to the Mineral Resource Estimate to include the 2017 to 2018 drilling. The update was based on validated results of 316 surface drill holes and 119 underground drill holes, for a total of 79,104 m of diamond drilling that was completed between 1984 and June 2018. The updated Mineral Resource Estimate, effective date July 19, 2018, prepared by McCracken et al. (2018) in a NI 43-101 Technical Report supersedes the 2017 Mineral Resource Estimate presented in the 2018 PEA. Table 6-15 summarizes the 2018 Mineral Resource Estimate.

**Table 6-15: Historic WSP Mineral Resource Estimate–Effective July 19, 2018**

| Resource Type | Au Cut-off (g/t) | Category | Tonnes (Rounded) | Au | Troy Ounces (Rounded) |
|---|---|---|---|---|---|
| **Open Pit** | 0.5 | Measured | 608,700 | 2.80 | 54,900 |
| | | Indicated | 247,600 | 3.72 | 29,600 |
| | | Measured & Indicated | 856,300 | 3.07 | 84,500 |
| | | Inferred | 58,500 | 4.10 | 7,700 |
| **Underground** | 2.00 | Measured | 1,003,100 | 5.10 | 164,400 |
| | | Indicated | 1,918,600 | 5.74 | 353,800 |
| | | Measured & Indicated | 2,921,700 | 5.52 | 518,200 |
| | | Inferred | 2,067,900 | 6.70 | 445,500 |
| **Combined** | 0.50/2.00 | Measured | 1,611,800 | 4.23 | 219,300 |
| | | Indicated | 2,166,200 | 5.50 | 383,400 |
| | | Measured & Indicated | 3,778,000 | 4.96 | 602,700 |
| | | Inferred | 2,126,400 | 6.63 | 453,200 |

Notes:

- Mineral Resources were prepared in accordance with NI 43-101 and the CIM Standards (2014).
- Open pit Mineral Resources are reported at a CoG of 0.5 g/t gold within the WSP base case pit shell and are based on a gold price of C$1,550/oz and a gold processing recovery factor of 95%.
- Appropriate mining costs, processing costs, metal recoveries, and inter-ramp pit slope angles were used by WSP to generate the base case pit design.
- Rounding may result in apparent summation differences between tonnes, grade, and contained metal content. Tonnages have been rounded to the nearest 1,000 tonnes; ounces have been rounded to the nearest 100 ounces.
- Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.
- Contributing assay composites were capped at 80 g/t gold.
- A density factor of 2.70 g/cm$^3$ was applied to all blocks.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

### 6.2.8 2019 Technical Report Dated December 18, 2019

The Company retained WSP in February of 2019 to provide an update to the Mineral Resource Estimate to reinterpret the geology and the mineralized belts based on updated diamond drilling and revised interpretation. The Mineral Resource was based on validated results of 485 surface and underground drill holes for a total of 93,916 m of diamond drilling that was completed between 1984 and August 21st, 2019. The Mineral Resource included 27,467 m of drilling conducted by the Company, including 15,112 m of diamond drilling in 57 drill holes since the Previous Mineral Resource Estimate of July 19, 2018. The effective date of this Mineral Resource is August 21, 2019. Table 6-16 summarizes the 2019 Mineral Resource Estimate.

**Table 6-16: Historic WSP Mineral Resource Estimate–Effective August 21, 2019**

| Resource Type | Au Cut-off (g/t) | Category | Tonnes('000) | Au (g/t) | Troy Ounces |
|---|---|---|---|---|---|
| **Open Pit** | 0.5 | Measured | 844.00 | 2.40 | 65,200 |
| | | Indicated | 111.00 | 2.63 | 9,400 |
| | | Measured +Indicated | 955.00 | 2.43 | 74,600 |
| | | Inferred | 22.00 | 2.79 | 2,000 |
| **Underground** | 2 | Measured | 967.00 | 6.08 | 189,200 |
| | | Indicated | 2,174.00 | 6.22 | 434,800 |
| | | Measured +Indicated | 3,141.00 | 6.18 | 624,000 |
| | | Inferred | 2,985.00 | 7.12 | 683,200 |
| **Combined** | 0.50/2.00 | Measured | 1,811.00 | 4.37 | 254,400 |
| | | Indicated | 2,285.00 | 6.05 | 444,200 |
| | | Measured +Indicated | 4,096.00 | 5.30 | 698,600 |
| | | **Inferred** | **3,007.00** | **7.09** | **685,100** |

Notes:

- Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards (2014). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
- Open pit Mineral Resources are reported at a CoG of 0.5 g/t gold that is based on a gold price of C$1,753/oz (approximately US$1,350/oz) and a gold processing recovery factor of 95%.
- Underground Mineral Resource is reported at a CoG of 2.0 g/t gold that is based on a gold price of C$1,753/oz (approximately US$1,350/oz) and a gold processing recovery factor of 95%.
- Appropriate mining costs, processing costs, metal recoveries, and inter-ramp pit slope angles were used by WSP to generate the pit shell.
- Appropriate mining costs, processing costs, metal recoveries, and stope dimensions were used by WSP to generate the potential underground resource.
- Rounding may result in apparent summation differences between tonnes, grade, and contained metal content.

- Tonnage and grade measurements are in metric units. Contained gold ounces are in troy ounces.
- Contributing assay composites were capped at 80 g/t gold.
- A bulk density factor was calculated for each block based on a regression formula.

### 6.2.9 2021 Technical Report Dated March 30, 2021

The Company retained Nordmin in April 2020 to prepare an NI 43-101 Technical Report and Mineral Resource Estimate for the Project. The Mineral Resource was based on validated results of 635 surface and underground drill holes for a total of 113,132.9 m of diamond drilling (completed between 1984 and February 7, 2021) and 1,230 chip samples totalling 822.7 m from the 2018 to 2019 Bulk Sample (Table 6-17).

**Table 6-17: Historic Nordmin Mineral Resource Estimate – Effective February 7, 2021**

| Resource Type | Gold Cut-off (g/t) | Category | Tonnes ('000) | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|---|
| **Open Pit** | 0.44 | Measured | 6,137 | 2.73 | 538,500 |
| | | Indicated | 5,743 | 2.99 | 551,300 |
| | | Measured + Indicated | 11,880 | 2.860 | 1,089,900 |
| | | Inferred | 1,580 | 1.75 | 89,000 |
| **Underground** | 2.60 | Measured | 1,384 | 7.36 | 327,700 |
| | | Indicated | 2,772 | 5.93 | 528,600 |
| | | Measured + Indicated | 4,156 | 6.41 | 856,200 |
| | | Inferred | 3,726 | 5.92 | 709,100 |
| **Combined Open Pit and Underground*** | 0.44 and 2.60 | Measured | 7,521 | 3.58 | 866,200 |
| | | Indicated | 8,515 | 3.95 | 1,079,900 |
| | | Measured + Indicated | 16,036 | 3.78 | 1,946,100 |
| | | Inferred | 5,306 | 4.68 | 798,100 |

* Combined Open Pit and Underground Mineral Resources; The open pit Mineral Resource is based on a 0.44 g/t gold CoG, and the underground Mineral Resource is based on 2.60 g/t gold CoG.

Notes

1. Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
2. Open pit Mineral Resources are reported at a CoG of 0.44 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,550/oz) and a gold processing recovery factor of 96%.
3. Underground Mineral Resource is reported at a CoG of 2.60 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,550/oz) and a gold processing recovery factor of 97%.
4. Assays were variably capped on a wireframe-by-wireframe basis.

5. SG was applied using weighted averages to each individual wireframe.
6. Mineral Resource effective date February 7, 2021.
7. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
8. Excludes unclassified mineralization located within mined out areas.
9. Reported from within a mineralization envelope accounting for mineral continuity.

### 6.2.10 2021 Technical Report Dated August 5, 2021

The Company retained Nordmin in February 2021 to prepare an NI 43-101 Technical Report and Preliminary Economic Assessment for the Project. The Mineral Resource remained unchanged from the technical report dated March 30, 2021. The Mineral Resources was based on validated results of 635 surface and underground drill holes for a total of 113,132.9 m of diamond drilling (completed between 1984 and February 7, 2021) and 1,230 chip samples totalling 822.7 m from the 2018 to 2019 Bulk Sample (Table 6-18).

**Table 6-18: Historic Nordmin Mineral Resource Estimate – Effective February 7, 2021**

| Resource Type | Gold Cut-off (g/t) | Category | Tonnes ('000) | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|---|
| **Open Pit** | 0.44 | Measured | 6,137 | 2.73 | 538,500 |
| | | Indicated | 5,743 | 2.99 | 551,300 |
| | | Measured + Indicated | 11,880 | 2.860 | 1,089,900 |
| | | Inferred | 1,580 | 1.75 | 89,000 |
| **Underground** | 2.60 | Measured | 1,384 | 7.36 | 327,700 |
| | | Indicated | 2,772 | 5.93 | 528,600 |
| | | Measured + Indicated | 4,156 | 6.41 | 856,200 |
| | | Inferred | 3,726 | 5.92 | 709,100 |
| **Combined Open Pit and Underground*** | 0.44 and 2.60 | Measured | 7,521 | 3.58 | 866,200 |
| | | Indicated | 8,515 | 3.95 | 1,079,900 |
| | | Measured + Indicated | 16,036 | 3.78 | 1,946,100 |
| | | Inferred | 5,306 | 4.68 | 798,100 |

* Combined Open Pit and Underground Mineral Resources; The open pit Mineral Resource is based on a 0.44 g/t gold CoG. The underground Mineral Resource is based on 2.60 g/t gold CoG.

Notes

1. Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
2. Open pit Mineral Resources are reported at a CoG of 0.44 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,550/oz) and a gold processing recovery factor of 96%.

3. Underground Mineral Resource is reported at a CoG of 2.60 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,550/oz) and a gold processing recovery factor of 97%.
4. Assays were variably capped on a wireframe-by-wireframe basis.
5. SG was applied using weighted averages to each individual wireframe.
6. Mineral Resource effective date February 7, 2021.
7. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
8. Excludes unclassified mineralization located within mined out areas.
9. Reported from within a mineralization envelope accounting for mineral continuity.

## 6.3 Historical Mineral Reserve Estimate

There are no historical Mineral Reserve estimates calculated for the Project.

## 6.4 Past Production

Gold in quartz veins was first discovered in 1862 in the Upper Seal Harbour Gold District by Howard Richardson of the Geological Survey of Canada on the Isaac’s Harbour anticline. The gold bearing BR Belt (slate and quartz) was subsequently discovered by Howard Richardson in 1892. Richardson Gold Mining began production from the mineralized vein system in 1893 at an average reported grade of 13.03 g/t gold. Milling recoveries were reported to be in the 50% to 60% range.

By 1896 the Boston Richardson Mine was working at full capacity with a 40-stamp mill, and in 1897, three shafts were being worked from one shaft house. The main shaft was situated on the east side on the apex of the anticline and had an inclination of–21 ° to the south. In 1898, 21,882 tonnes were milled from this mine, producing 2,479 oz. of gold recovered.

In 1899, 136 tonnes of concentrates were recovered from mill tailings and treated by a Wilfley concentrator. By 1900, substantial efforts were made to re-timber old workings, and large pillars were installed locally to stabilize the ground. In 1901, the mill was increased in size to 60 stamps, two more Wilfley concentrators were in operation, and an extensive cyanide plant was brought to the Property from the Caribou Gold District. In 1902, 26,308 tonnes of ore were milled at a grade of 4.08 g/t to produce 3,459 oz. In 1903, mining was suspended after an extensive collapse, attributed to insufficient support of the hanging wall, destroyed the main shaft. Boston Richardson Mining took over ownership of the Property in 1903.

From 1901 to 1905, three gold bearing belts were intersected in the Dolliver Mountain mine, located 2 km west of the Boston Richardson Mine. In 1904, 7,195 tonnes were milled at a grade of 0.87 g/t to produce 205 oz. of gold. In 1905, several bodies of quartz and slate were intersected by a 152 m deep drill hole at the bottom of the main shaft along the anticlinal axis, but results were unsatisfactory, and mining at Dolliver Mountain mine ceased.

In 1906, half the broken material mined was hoisted, and the remainder was held as a reserve in the mine. The bromo-cyanide plant brought from the Caribou Gold District was in operation and recovered up to 80 % of the gold in concentrates in 1906 and 1907. Re-concentrated tailings from the cyanide plant grading approximately 40% arsenic were shipped to Germany for further processing and gold recovery.

In 1907, the EG property that adjoined the BR property to the east was acquired by F.S. Andrews and others. A shaft was sunk 53 m (175 feet), and three promising gold bearing belts were explored in

1908. One of these was reported as being well mineralized, but no other work was carried out on the Property at that time. Operations were suspended on August 15, 1908, due to financial difficulties but were later resumed.

In 1909, New England Mining took over the Boston Richardson Mine and processed an additional 37,572 tonne at an average grade of 4.14 g/t to produce 5,024 oz. of gold. Of this total, 82.6% was recovered by crushing and amalgamation, and 17.4% was recovered by bromo-cyanide processing. Of the 533 tonnes of arsenical concentrate produced, 405 tonnes were shipped to Swansea, Wales, for final gold recovery. In 1910, 24,440 tonnes at an average gold grade of 5.17 g/t were milled to produce 4,063 oz. Of this total, 715 oz were recovered by the bromo-cyanide treatment, and a further 480 tonnes were shipped to Wales during the year for final gold recovery.

From 1909 to 1910, the WG exploration shaft intersected five gold bearing belts. Three of these were mill tested, but the milling results were considered unsatisfactory, and the mine was abandoned.

The total gold recovery from 1893 to 1910 for the Property has been estimated to be 376,303 tonnes at an average recovered gold grade of 4.11 g/t to produce 54,871 oz. However, mill recovery is reported to be approximately 67% (Roy, M.; Groupe Conseil Gesplaur, 1998). Operations at the mine continued on a small scale in 1911 and 1912.

In 1926, the Metal Mining and Smelting Corporation of Canada Ltd. took over the Property and treated tailings to recover auriferous arsenopyrite through 1927.

From 1929 to 1931, Locarno Copper Mines Ltd. sank a shaft on the WG property west of the earlier shaft. In 1931, 1 tonne of ore was metallurgically tested for an average grade of 50.1 g/t recovered and 1.61 oz. of gold.

From 1931 to 1934, Renada Mines Ltd. dewatered and sampled the old workings at EG and obtained gold assay results ranging between 1.61 g/t and 4.26 g/t.

In 1956, Canso Mining Corporation dewatered the shaft at WG and carried out cross-cutting. This work ceased due to financial difficulties.

# 7. GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 Regional Geology

There are five tectonostratigraphic zones within the Appalachian Belt in eastern Canada (Figure 7-1); these include the Humber, Dunnage, Gander, Avalon, and Meguma zones.

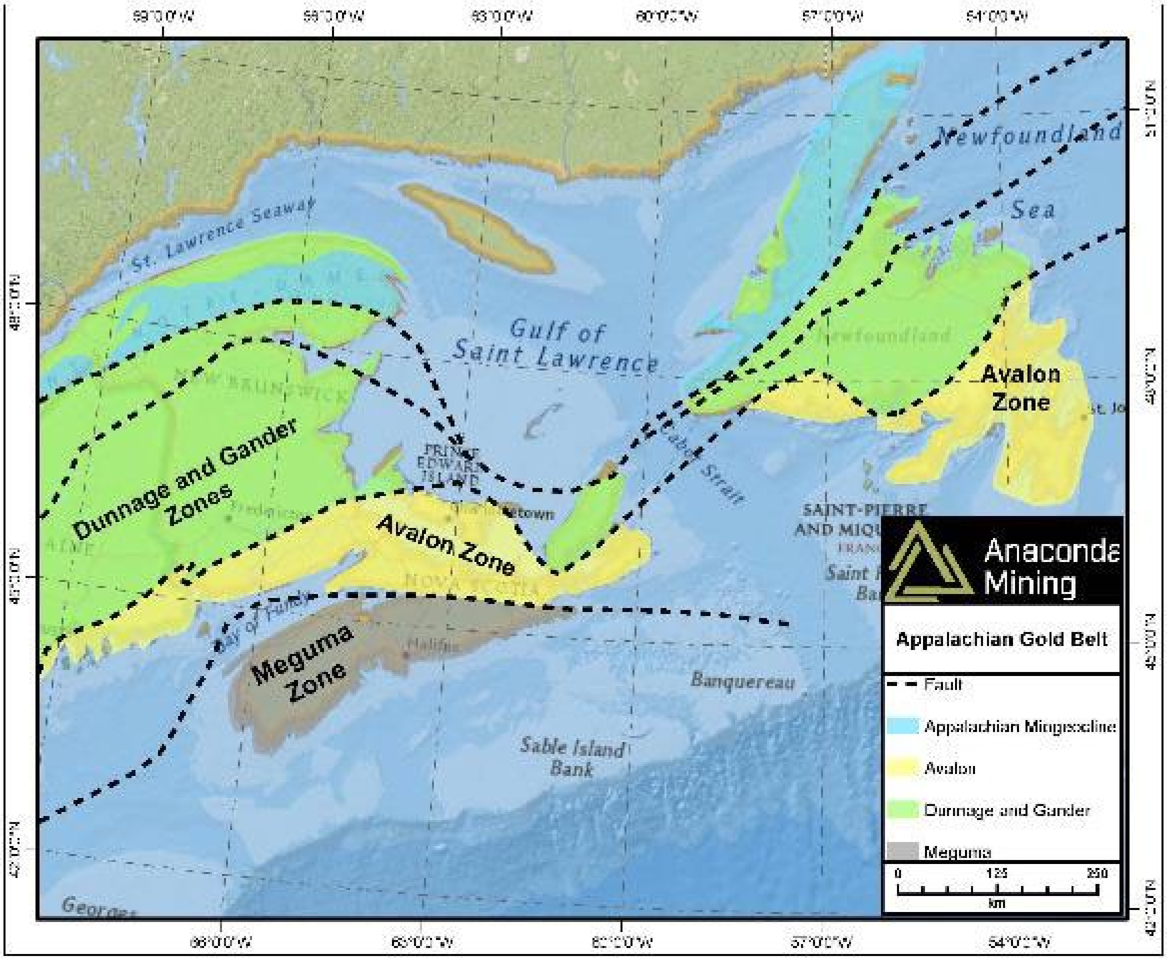



*Figure 7-1: Tectonostratigraphic Zones of the Northern Appalachian Orogen*

The Meguma zone underlies most of the southern mainland of Nova Scotia and is structurally juxtaposed against the Avalon zone to the north along the Cobequid-Chedabucto Fault System during the (Neo-Acadian) Orogeny (Smith & Kontak, 1996). The Meguma zone is predominately made up of the Meguma Group consisting of Cambro-Ordovician sedimentary rocks deposited along the continental margin of the Gondwana paleo-continent during closure of the Iapetus and Rheic oceans (Smith & Kontak, 1996). The Meguma Group includes the Goldenville Formation, a basal sandy flysch sequence estimated to be 6.7 km thick, and the overlying Halifax Formation, a shaley flysch sequence approximately 11.8 km thick (Sangster & Smith, 2007).

The massive, thick-bedded greywacke sequence of the Goldenville Formation is dark grey (carbonaceous) to light grey in colour and contains thin argillite horizons that commonly separate the thick, coarser beds. The Goldenville Formation grades upwards through manganese-rich strata into a basal Halifax Formation unit that consists of sulphidic black slate. The manganese-rich section, along with Tremadocian fossils, marks the transition between the two formations. Black carbonaceous sulphidic slate and thinly bedded to cross-laminated siltstone comprises much of the Halifax Formation, but lithologies in the uppermost stratigraphy consist mostly of grey-green slate and siltstone (Sangster & Smith, 2007).

The Meguma Group is pervasively folded and characterized by kilometre-scale wavelengths and E-W to NE-SW axial trace directions. Folds are upright to slightly inclined, with plunges to both east and west. Doubly-plunging fold trends produce domal structural culminations that in many instances correspond with historic gold producing districts. Cleavages are also a predominant structural feature and include regional slaty cleavage, AC cleavage, and pressure solution cleavage. The bedding-cleavage intersection lineation reflects local plunge variations and indicates a general non-cylindrical character (Horne, 1996).

The Meguma Group in the eastern part of Nova Scotia (Figure 7-2) was metamorphosed to greenschist-amphibolite facies grade during the mid-Devonian Acadian Orogeny (ca. 400 Ma) and was subsequently intruded by peraluminous granite, granodiorite, and minor mafic intrusions of mid-Devonian to Carboniferous age (ca. 375 Ma) (Sangster & Smith, 2007).

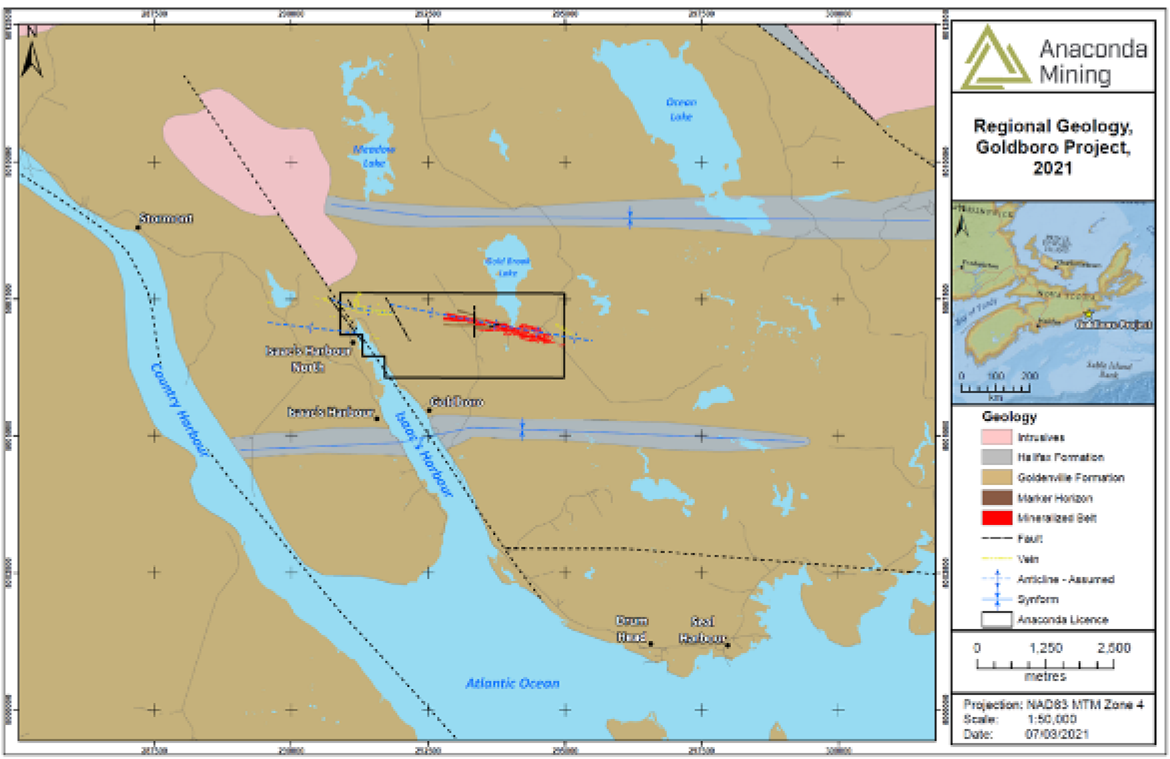



*Figure 7-2: Regional geology of Eastern Nova Scotia*

## 7.2 Local and Property Geology

The Project is underlain by rocks within the Goldenville Formation dominated by greywacke and argillite. Rocks of the overlying Halifax Formation are located approximately 1.6 km south of the Project (Figure 7-3). The stratigraphy dips steeply to the south exposing a stratigraphic section of the Goldenville Formation from the Goldboro region to the Halifax Formation that spans approximately 1.5 km of true thickness.

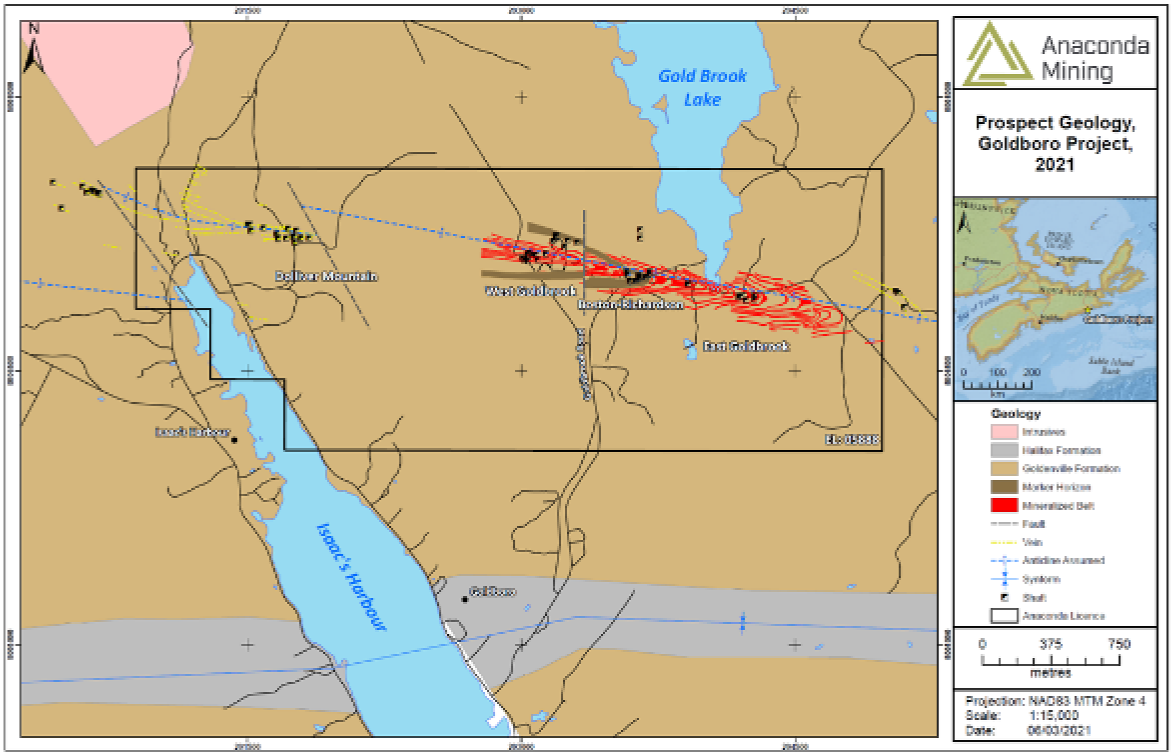



*Figure 7-3: Location of mineralized belts at the Project showing the outline of eastward plunging anticlinal fold*

At the Deposit, the Goldenville Formation consists of alternating greywacke and argillite beds with an approximate true thickness of 950 m. The base of the stratigraphic sequence intersected in the BR Gold System consists of roughly 325 m of alternating greywacke and argillite, with varying proportions of both rock types, ranging in thickness from less than 1 m up to 10 m. This is overlain by the Marker Horizon, which consists of a 40 m to 50 m greywacke bed that is commonly intersected during drilling and in underground workings (Figure 7-4). The Marker Horizon appears to thin or is offset by the New Belt Fault on the northern limb of the anticline toward the west. Above the Marker Horizon is the EG Gold System, approximately 560 m thick, consisting of alternating greywackes, and argillites. Within the EG Gold System there is a second, thick, greywacke sequence varying in thickness from 20 m to 60 m. This may represent a new marker unit within the stratigraphy.

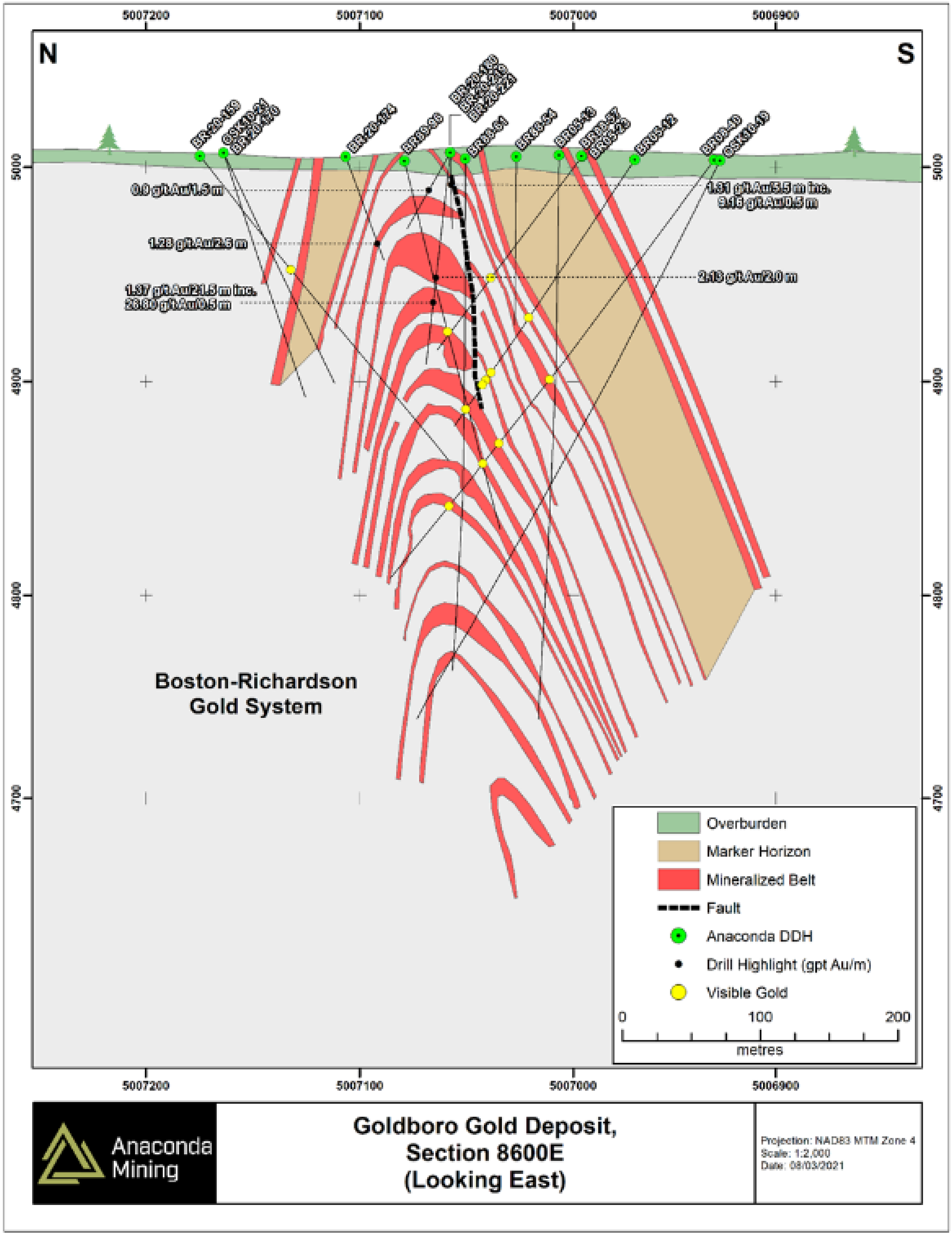



*Figure 7-4: Typical cross section at 9300E through the Deposit area*

The structure of the Project Area is dominated by the Upper Seal Harbour Anticline Figure 7-4. The anticline folds all levels of stratigraphy observed in core and underground to form an upright, tight anticline that plunges 20° eastward. The enveloping surface to bedding also dips moderately eastward at 20°. Younging is upward, orthogonal to the hinge, and limbs of the fold. An axial planar cleavage is well developed at all levels of stratigraphy but pervasive within the hinge of the fold. The intersection of the axial planar cleavage forms an intersection lineation commonly observed on bedding surfaces that plunge parallel to the fold axis. All bedding and first-generation structures are refolded by open reclined folds that modify the axial plane and limbs of the Upper Seal Harbour Anticline. The axial plane of second generation folds dips shallowly and an axial planar cleavage is observed in the core and within underground workings.

All earlier structures are deformed by late brittle faults. One generation of these faults, which includes the New Belt Fault, are steeply dipping, and occur both parallel, and cross-cutting regionally folded stratigraphy. These faults also disrupt the stratigraphy on the northern limb of the fold structure in the WG and BR Gold Systems, although kinematics, and displacement are not known. A second generation of faults strike northerly and are steeply dipping, these offset the axial trace of the anticline. The WG Fault forms the boundary between the WG and BR Gold Systems. Displacement along the WG Fault indicates roughly 50 m of normal, west side down movement, and approximately 30 m of right lateral movement.

Gold mineralization at the Deposit occurs in both quartz veins and within the argillite that hosts the veins, as well as within the rocks adjacent to the modelled argillites and quartz veins, including both lesser argillite with greywacke. Disseminated, euhedral arsenopyrite is pervasively associated with gold mineralization, and is commonly observed within the host rock and is usually present in mineralized quartz veins. Other sulphides associated with mineralized quartz veins are pyrrhotite, chalcopyrite, pyrite, and minor amounts of sphalerite and galena. Wall rock generally contains more pyrrhotite and arsenopyrite than directly associated quartz veins. Gold bearing quartz veins are stratabound with lesser discordant quartz veins and vein arrays (Figure 7-5). Native gold is nuggety in nature, and grains range from microscopic up to several centimetres in size (Figure 7-6) and is found in all rock types with visible gold generally associated with quartz veins. Within quartz veins, gold is present as free gold in quartz, and within arsenopyrite grains, along grain boundaries and internal fractures, and is non-refractory in nature. Native gold also occurs as a disseminated phase in altered argillite and argillite/greywacke intervals adjacent to and separate from quartz veins, demonstrating its association with both quartz veins and the altered wall rock.

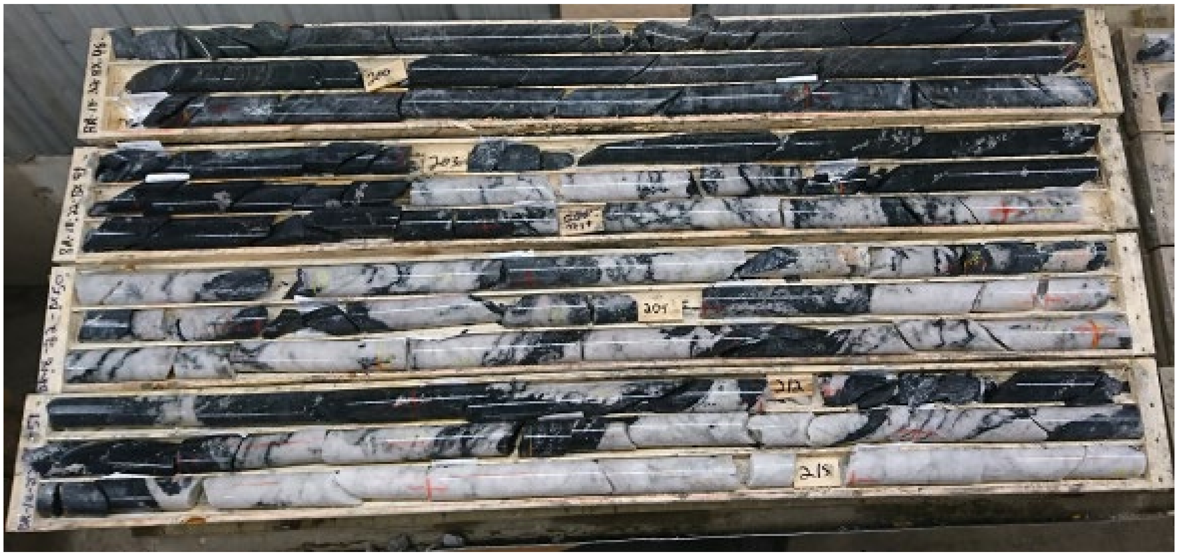


*Figure 7-5: Mineralized quartz veins in graphitic argillite: Hole BR-18-22*


*Figure 7-6: Visible gold in graphitic argillite: Hole BR-17-03.*

***A 3 mm grain of visible gold in a crack seal, bedding parallel (BP) white quartz vein.***

## 7.3 Structure

### 7.3.1 Structural Geology and Deformation Phases

Five deformation phases (D1 through D5) are recognized in the Meguma Group, and the descriptions presented below are modified slightly after those of Smith et al. (1985), Corey (1992), and Smith and Kontak (1988), unless otherwise indicated, as well as observations by Company geologists.

The first deformation phase (D1) is marked by an early BP foliation (S1) and is not easily recognized on the Property. The second deformation phase (D2) resulted in the development of northeast to east trending, upright regional folds with well developed, steeply dipping axial planar pressure solution and slate cleavages. Pressure solution cleavage is well developed in thicker (1 m to 50 m) greywacke beds on the Property. Slate cleavage is ubiquitous in the finer clastic units and bedding transpositions along this fabric are commonly seen. The D3 deformation phase consists of flat to shallowly-lying pressure solution cleavage developed locally and defined by the growth of oriented andalusite and biotite porphyroblasts in argillites. The D2 and D3 fabric elements have been recognized in outcrop on the Property in the Dolliver Mountain area, approximately 2 km west of the Boston Richardson Mine, observed in the underground workings at BR, and in drill core throughout the Deposit. The D4 deformation occurs as discrete brittle faults that both parallel and crosscut regionally folded stratigraphy and developed following peak metamorphic conditions and gold mineralization. The New Belt Fault is an example of D4 deformation and strikes approximately 085° and ranges in dip between 90° and 85°. Displacement and kinematics along this structure has not been defined, although a general dextral sense of displacement has been modelled by correlating mineralized belts within the BR Gold System. The D5 deformation phase was recognized in the form of late NW-trending brittle faults plus locally developed crenulation cleavage and kink bands that are seen in argillites.

The N-S trending, WG Fault, which separates the WG Gold System from the BR Gold System, is interpreted as a D4 structure. Recent drilling by the Company in the vicinity has confirmed historical interpretations that the WG Gold System is the offset western extension of the BR Gold System. The displacement of the Marker Horizon across the WG Fault demonstrates it is an oblique normal fault with roughly 50 m of normal, west side down movement, and approximately 30 m of right lateral movement.

### 7.3.2 Structural Setting, Vein Styles, and Genetic Model

The following discussion of structural setting, vein styles, and genetic model was modified after descriptions presented in Gervais et al. (2009) and observations by Company employees and consultants.

Gold mineralization at the Project occurs in quartz veins and wall rocks adjacent to the veins. At the deposit scale, the veins form a swarm and are clearly located in the flexure zone (hinge and adjacent limbs) of the Upper Seal Harbour Anticline. The gold bearing veins are found in a 175 m to 275 m wide envelope centred on the axial surface. The veins occur mostly on the limbs of the fold, but also in the hinge, and all are hosted by turbiditic metasedimentary rocks consisting of greywacke and argillite.

Gold bearing quartz veins occur as both stratabound and cross-cutting entities and are most prevalent within BP stratigraphy, occurring approximately 80 m north or south of the Upper Seal Harbour Anticline's axial surface. Deformation by compression and shearing associated with regional folding is greatest near this axial zone. A flexural slip model of fold evolution is considered to be most applicable in this area due to drilling defined thickening of individual slate units within the fold hinge zone relative to fold limb zone positions, and pervasive development of axial planar cleavage along which bedding transposition occurs in symmetry with that expected for a flexural slip process.

Gervais et al. (2009) and recent observations also indicate that progressive heterogeneous strain is localized within less competent argillites relative to greywacke. This is interpreted to be due to competency contrast between the argillites and greywackes. Contemporaneous fracturing and ductile deformation provide a mechanism for the circulation of hydrothermal fluids from which gold bearing quartz veins were developed (Gervais, D.; Carrier, A.; Brousseau, K.; InnovExplo Inc., 2009).

## 7.4 Alteration and Mineralization

High-grade gold mineralization at the Deposit occurs in both quartz veins and host argillite (Figure 7-5). These high-grade zones are BP and generally continuous around the fold hinge and down the north and south limbs. These mineralized zones are referred to as Belts. Sixty-eight Belts have been modelled within the Deposit and are referred to as Higher-Grade Belt within the model. A more disseminated, generally lower-grade, style of mineralization occurs in the wall rock adjacent to the quartz veins and can extend several metres outward from and between Higher-Grade Belts. These are referred to as Lower-Grade Domains and can include disseminated gold mineralization within altered, sulphide (arsenopyrite) bearing wall rock including greywacke and argillite, gold bearing quartz sulphide veins of variable orientation that do not correlate geometrically to adjacent Higher-Grade Belts.

Gold is associated with sulphide bearing quartz veins and altered, sulphidic wall rock. Arsenopyrite is the most common sulphide species present, although pyrite, pyrrhotite, chalcopyrite, galena, and sphalerite are also associated. Gold commonly occurs as a free-milling phase within quartz veins but is also present in direct association with vein hosted arsenopyrite. In such cases, it commonly occurs as inclusions within arsenopyrite, as free particles associated with microfractures cutting arsenopyrite crystals, and as free particles attached to arsenopyrite crystal surfaces (Ryan & Smith, 1998). Pyrite also coats fracture and cleavage planes closest to vein contacts and occurs as fine-grained, disseminated subhedral crystals. Pyrite locally exhibits wispy and blebby textures and frequently shows association with late faults (Gervais, D.; Carrier, A.; Brousseau, K.; InnovExplo Inc., 2009).

Pyrrhotite is a commonly occurring sulphide phase in wall rock and typically is present as disseminated blebs, sometimes flattened in bands along foliation planes, or as irregular blebs at quartz vein contacts. Pyrrhotite also occurs in both wall rock and veins as a fracture coating phase and as very fine stringers. Chalcopyrite is almost exclusively confined to quartz veins and is present as fine-grained blebs concentrated along microfractures. Galena in small amounts is present in association with quartz vein hosted visible gold and within the wall rock. Sphalerite is rarely observed, but where present occurs as mm-scale blebs within or along fractures within quartz veins (Gervais, D.; Carrier, A.; Brousseau, K.; InnovExplo Inc., 2009).

Native gold is nuggety in nature, and grains range from microscopic up to several centimetres in size (Figure 7-6) and is found in all rock types, with visible gold generally associated with quartz veins. Within quartz veins, gold is present as free gold in quartz, and within arsenopyrite grains, along grain boundaries and internal fractures, and is non-refractory in nature. Native gold also occurs as a disseminated phase in altered argillite and argillite/greywacke intervals adjacent to and separate from quartz veins, demonstrating its association with both quartz veins and the altered wall rock.

Turbiditic rocks in the hinge zone of the Upper Seal Harbour Anticline have been variably altered with carbonate, sericite, sulphide, tourmaline, and chlorite assemblages that post-date growth of regional metamorphic mineral assemblages (Roy & Labelle, 1990). The nature of alteration varies as a function of lithology and proximity to mineralization. Further, the alteration is in some cases cryptic and not pervasive, particularly distal from mineralization. Greywacke/sandstone units have varying amounts of biotite and muscovite that have likely detrital, metamorphic, and alteration origins. The greywacke and quartz-rich units generally exhibit weaker alteration than the finer argillite /mudstone units, but when altered, the greywacke/quartz-rich units exhibit bleaching that consists of both albite and sericite alteration. These units also exhibit irregular quartz alteration proximal to cleavage fractures in the rock; these zones also arsenopyrite in some instances.

In contrast, the siltstone/mudstone/argillite units exhibit the greatest changes in alteration mineralogy proximal to veins. Background siltstones are generally layered and laminated and are brown-green with minor biotite and chlorite, whereas proximal to well mineralized veins they exhibit black to black-green colouration and are pervasively altered to chlorite with biotite, sericite, albite, quartz, carbonate, and sulphide. Often these zones have chlorite-biotite, as well as carbonate spots, and they are cut by quartz veins. Further, they ubiquitously have arsenopyrite proximal to veins that host mineralization and in the various “belts”; arsenopyrite ranges from mm-scale up to several centimetres and locally contains pressure shadows with quartz ±carbonate. The alteration extent within these argillites, however, is limited spatially (metre-scale) due to individual beds having limited spatial extent. Despite their limited distribution, the argillite beds are disproportionally veined compared to other rock types. The whole rock geochemistry of the argillites demonstrates gains in $K_2O$, $Fe_2O_3$, $Na_2O$, and $Al_2O_3$ proximal to mineralization, and this decreases at distance from mineralization (Szmihelsky, Piercey, Copeland, & Tettelaar, 2021). Internal Company evaluation of the multi-element assay database also illustrates that locally these argillites are enriched in Au, As, S, Pb, Cd, Fe, Ba, K, Na, Mn, Ca, Sr, and P, particularly with increasing abundances of mineralization in the argillites.

Gold and sulphide mineralization is associated with both wall rock and veins. Argillite units contain diagenetic pyrite (locally framboidal), pyrrhotite, and arsenopyrite (generation 1 arsenopyrite). Szmihelsky et al. (2021) noted that there were four generations of veins in the Deposit (paragenetically V1 to V4) with the majority of gold associated with vein generations V3 and V4 where gold occurs both in veins and wall rock, with the majority of coarse gold in veins associated with arsenopyrite (generation 2), galena, and to a lesser extent chalcopyrite and sphalerite (Figure 7-7, Figure 7-8, Figure 7-9 and Figure 7-10). Microscopically, gold occurs as inclusions in arsenopyrite, often spatially proximal to galena inclusions (Figure 7-7 and Figure 7-8). Gold also occurs as coarser grains or wires along grain edges and cracks in the arsenopyrite, indicative of potential coalescence and remobilization from grain interiors to grain margins. This process is likely due to the prolonged thermal history in this area, was likely important in determining grade in the Deposit, and is common in many orogenic gold environments (Voisey, et al., 2020); (Hastie, Schindeler, Kontak, & Lafrance, 2021).

The gold mineralization observed in both core and microscopically is reflected in the multi-element geochemistry in the Deposit. Preliminary evaluations of the assay database illustrate that there are strong Au-As-S-Pb-Cd associations within the database, reflective of the mineralogy outlined above. There are also local enrichments in Zn, Cu, Fe, Ni, and Co reflective of the presence of sphalerite, chalcopyrite, pyrite and pyrrhotite.

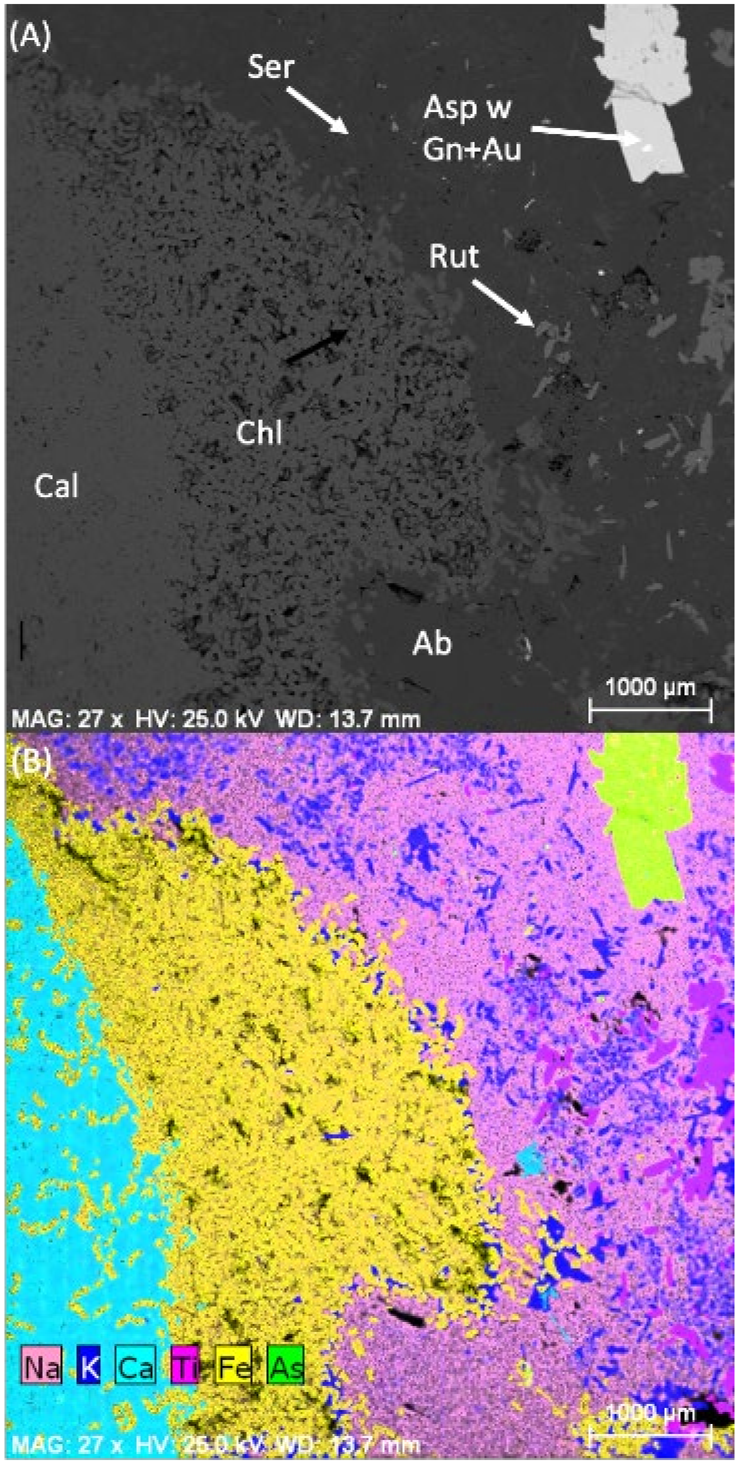



*Figure 7-7: Scanning electron microscope images of hydrothermally altered and mineralized argillite*

*Scanning electron microscope images of hydrothermally altered and mineralized argillite. A) Backscatter electron (BSE) image of an alteration assemblage containing calcite, chlorite, albite, sericite, and rutile associated with arsenopyrite grains that contain gold and galena inclusions. B) Energy dispersive spectroscopy map of semi-quantitative sodium, potassium, calcium, titanium (Ti), iron, and arsenic distribution illustrating spatial and intergrowth patterns of hydrothermal alteration phases in the mineralized argillites /mudstones.*

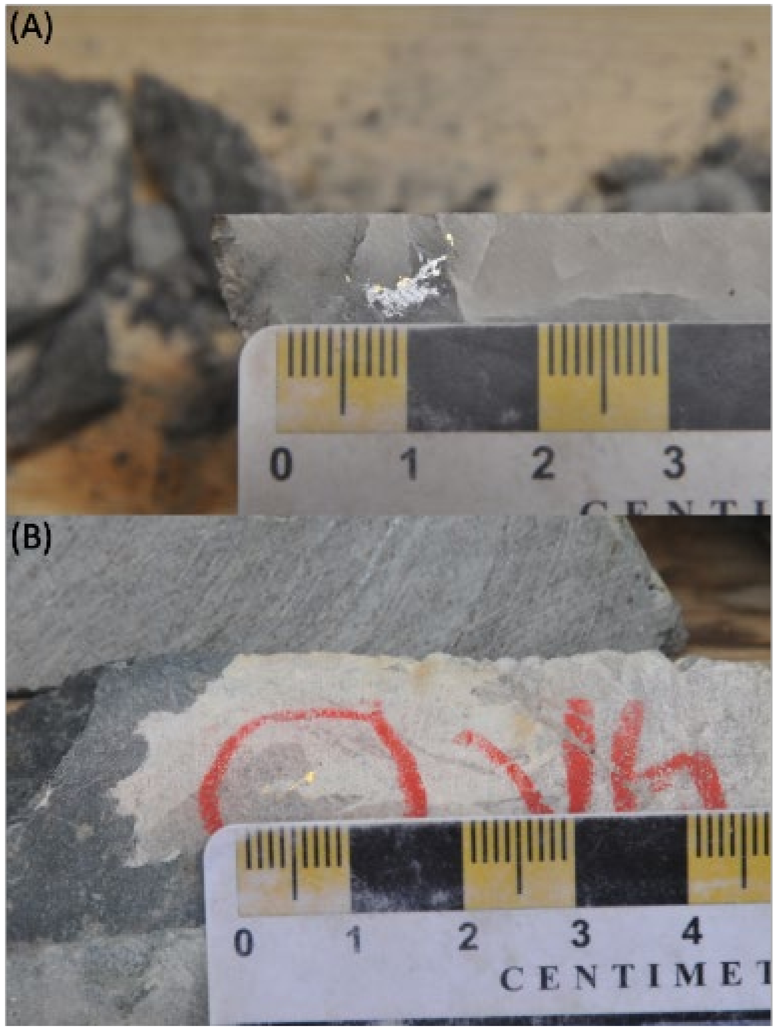



*Figure 7-8: Photos of gold in quartz veins from the Deposit*

*A) Quartz vein with visible gold spatially associated with galena. B) Free gold in quartz veins hosted by a black argillite.*

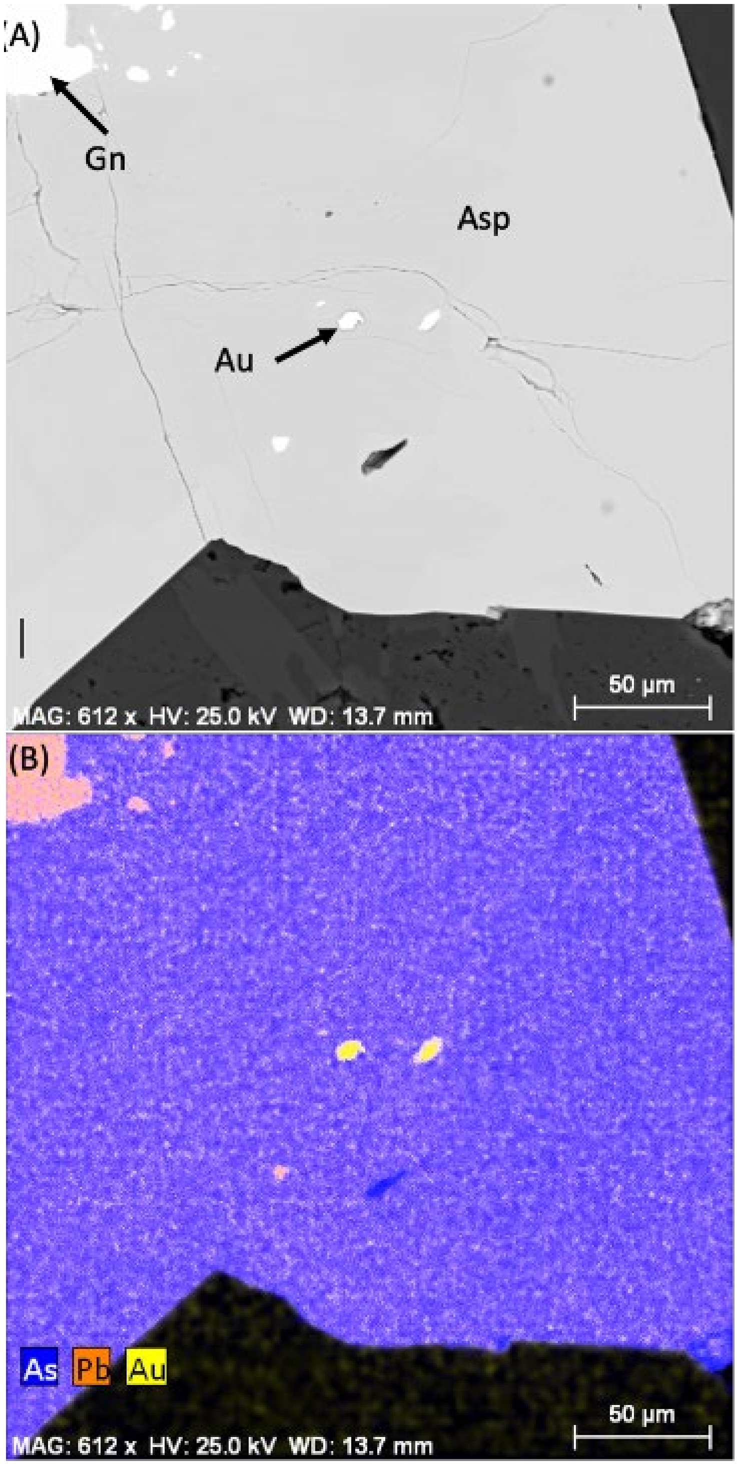



*Figure 7-9: Scanning electron microscope images of mineralization from the Deposit*

*A) BSE image of an arsenopyrite grain with inclusions of microscopic gold occurring as blebs within arsenopyrite spatially proximal to galena. B) Energy-dispersive X-ray spectroscopy map of semi-quantitative arsenic, lead, and gold distribution in arsenopyrite illustrating the location of gold and galena within arsenopyrite.*

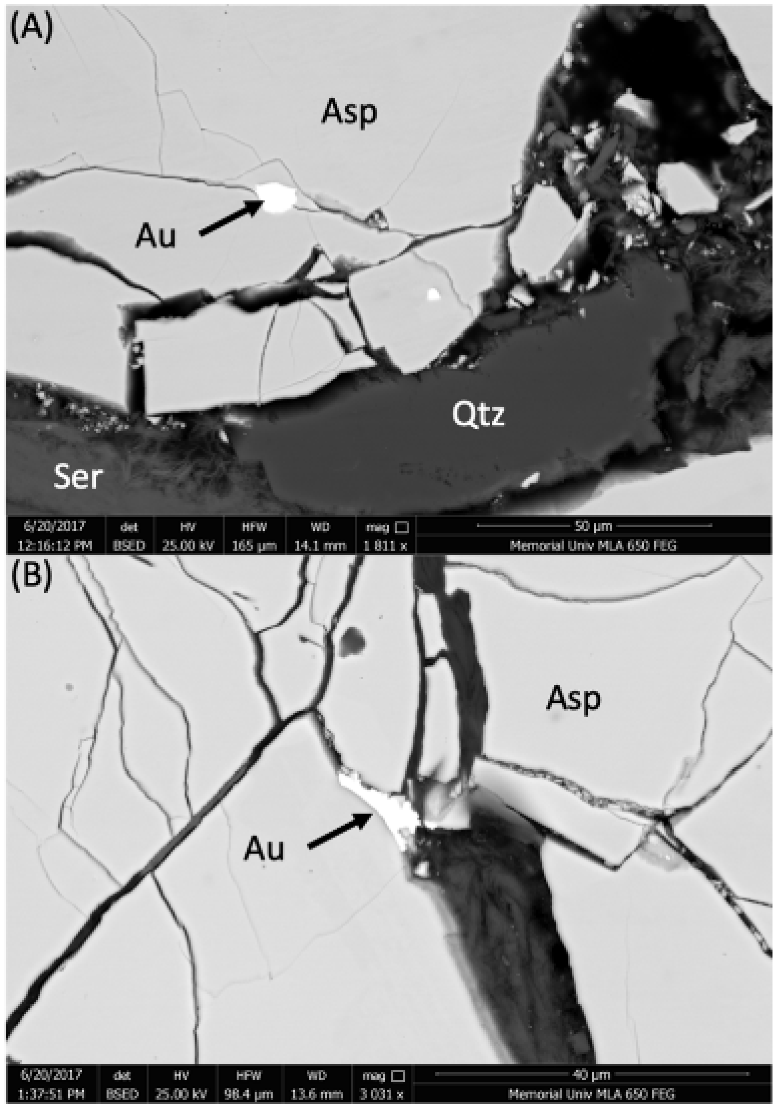



*Figure 7-10: Scanning electron microscope images of mineralization from the Deposit*

*A) BSE image with blebs of gold as an inclusion within pyrite as well as a coarser bleb located in a crack within the arsenopyrite grain. B) Coarse aggregate of gold located along a crack in arsenopyrite. These aggregated grains of gold are interpreted to be due to remobilization and potential aggregation/upgrading in fractures.*

# 8. DEPOSIT TYPES

The turbidite-hosted gold deposits of Nova Scotia have been compared to similar-age turbidite-hosted quartz vein deposits elsewhere in the world, particularly those in the Bendigo and Ballarat areas of the Lower Paleozoic Lachlan Fold Belt in the state of Victoria, Australia, and have historically been similarly classified. Robert et al. (1997) recognized this deposit class and proposed that gold deposits of Nova Scotia be identified as a member of the 'Turbidite-hosted, quartz-carbonate vein deposit (Bendigo Type)' category. Ryan and Ramsay (1996) also addressed the similarity of Nova Scotia turbidite-hosted gold deposits with those in Victoria. As noted by Gervais et al. (2009), categorization within the USGS classification system of mineral deposits presented by Berger (1986) places the Deposit in the broad 36a category of 'Low-Sulphide Gold-Quartz Vein Deposits.'

The Deposit is a turbidite-hosted orogenic gold deposit hosted within a sequence of alternating argillites and greywacke metamorphosed to greenschist facies (Figure 8-1). These deposit types are typically characterized by the formation of gold bearing quartz veins within the argillite units, commonly referred to as mineralized Belts, that are interbedded with greywacke units. There are currently 68 stacked mineralized Belts ranging in thickness from 1 m to 20 m in the Deposit. The metasedimentary units on the Property are folded into the tight, gently east-plunging Upper Seal Harbour Anticline and gold mineralization has typically been deposited at various positions and times during the fold formation process. Veins, which form during deformation, occur in three major geometries commonly referred to as reefs: saddle reefs, leg reefs, and spur reefs. Saddle reefs occur about the apex of the fold and are the dominant vein types within some deposits. Leg reefs extend down the limbs of the fold, beyond the saddle reef, and are generally parallel with the metasedimentary layers. These are also commonly termed BP veins in the Nova Scotia goldfields. Spur reefs are veins that cross between layers and may be in the apex of the fold or on its limbs. This style of vein is in part captured under the term "angular" in the Nova Scotia goldfields.

The Deposit contains all three types of reefs outlined above but is also characterized by mineralization within the argillite forming the Belts. Because the Deposit contains saddle, leg, and spur reefs, and often has gold mineralization within the argillite hosting the veins, it has the potential to contain significantly more gold resources than deposits of a similar style that contain gold only within the quartz veins (reefs) themselves. The trace of the Upper Seal Harbour Anticline transects the Property and is found near the Dolliver Mountain prospect 2 km to the west of the Deposit, demonstrating that the structure which hosts gold continues for several kilometres.

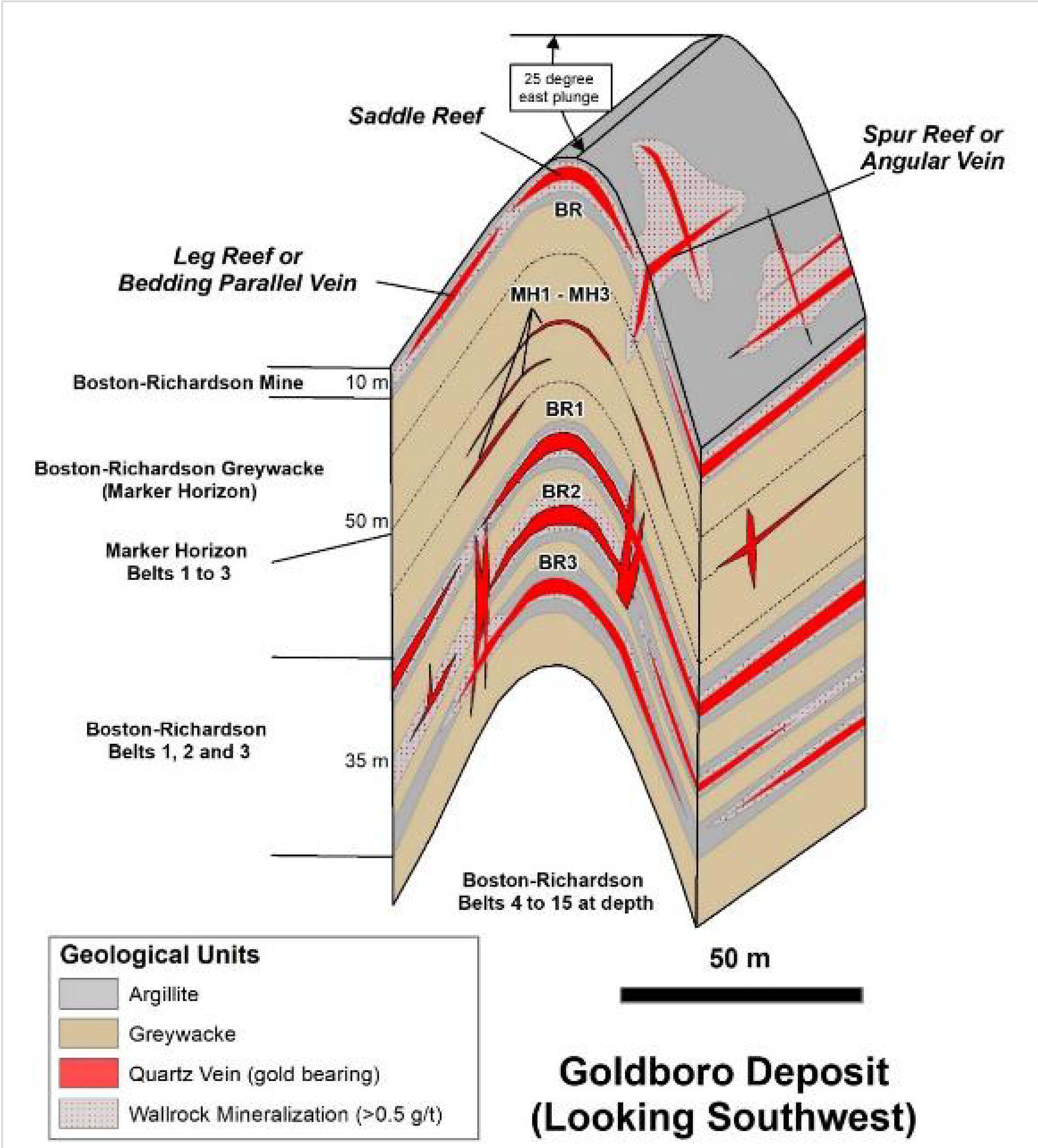



*Figure 8-1: Generalized model of mineralization within the Deposit*

# 9. EXPLORATION

The Company acquired its interest in the Property early in 2017 under terms of a court approved Plan of Arrangement whereby Orex became a wholly owned subsidiary of the Company. On this basis, work completed by Orex and others prior to the acquisition is considered historical in terms of current NI 43-101 technical reporting.

A summary of historical exploration was presented in Section 6. Work completed by the Company on the Property since its acquisition in 2017 includes the completion of 46,149.1 m of diamond drilling and three Mineral Resource Estimates. Additionally, the Company conducted an underground Bulk Sample from which a total of 13,028 tonnes of mineralized material was mined and stockpiled on surface with 10,023 wmt (9,785 dmt) shipped to the mill at Point Rousse near Baie Verte, NL for processing into gold doré bars, as outlined in Section 6.1.1. The Company has also completed two phases of detailed metallurgical studies on both high-grade and low-grade mineralization from the Deposit and found recoveries averaging 96%, as outlined in Section 13.

In 2020 the Company retained Nordmin to conduct an assessment of the Project. Through an interactive process with the Company, Nordmin undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project. This resulted in more detailed geological modelling to better represent geological characteristics of the Deposit as observed in drill core and during the Bulk Sample and the recognition of the importance of low-grade mineralization associated with and adjacent to high-grade mineralization. The results of this analysis and modelling, and the incorporation of an additional 17,941.7 m of diamond drilling, are the subject of the previous Technical Report with a Mineral Resource Effective date of February 7, 2021. Since then, an additional 73 diamond drill holes totalling 10,145 m has been used to further delineate the mineralization and geology within the Boston Richardson domain. The results of this analysis and modelling are the subject of this Technical Report with a Mineral Resource Effective date of November 15, 2021.

# 10. DRILLING

## 10.1 Introduction

The current Mineral Resource Estimate includes drilling results obtained by several operators from 1984 to 2021. Table 10-1 provides a chronological summary of associated drill holes. The Company has completed a total of 46,149.1 metres of diamond drilling on 228 drill holes since acquiring the project in 2017. Drilling since 2017 has largely been focused on infill and expansion drilling designed to update and upgrade the Mineral Resource at the Project as well as collect samples for metallurgical testing.

Representative geological and assay cross sections based on the combined drilling database used in the current Mineral Resource Estimate are included in Section 7 and Section 14. Figure 10-1 provides a summary view of drill collar distribution. A tabulation of drill collar coordinates and hole orientation data is provided in Appendix B.

**Table 10-1: Diamond Drilling Program Summary for the 1984 to 2019 Period**

<table>
<tr><th>Company</th><th>Year</th><th>Area</th><th>Metres</th><th>No. Holes</th><th>Series</th></tr>
<tr><td>Onitap</td><td>1984</td><td>BR</td><td>529</td><td>1</td><td>BR-84-01</td></tr>
<tr><td>Onitap</td><td>1985</td><td>West Goldbrook Mine</td><td>390</td><td>5</td><td>BR-85-01 to BR-87-04, incl. BR87-01 A</td></tr>
<tr><td>Petromet Resources Ltd. & Greenstrike Gold Corp.</td><td rowspan="2">1987</td><td>Eastern part of the Property</td><td>1,924</td><td>7</td><td>BR-87-01 to<br>BR-87-05 A</td></tr>
<tr><td>Onitap</td><td>BR Belt and EG property</td><td>11,621</td><td>33</td><td>BR-87-06 to BR87-38</td></tr>
<tr><td rowspan="4">Orex</td><td rowspan="2">1988</td><td>Upper Seal Harbour fold (8325E to 9100E) & West Goldbrook Mine</td><td>10,822</td><td>41</td><td>BR-88-39 to BR-88-79</td></tr>
<tr><td>Near decline</td><td>459</td><td>3</td><td>BR-88-80 to BR-88-82</td></tr>
<tr><td rowspan="2">1988 to 1990</td><td>Underground 76 m Level (8637.5E to 8762.5E)</td><td>4,979</td><td>112</td><td>88 U-01 to 88 U-04<br>89 U-05 to 88 U-26<br>90 U-27 to 90 U-112</td></tr>
<tr><td>WG (8150E to 8600E)</td><td>2,811</td><td>26</td><td>BR-89-83 to BR-89-108</td></tr>
</table>

| Company | Year | Area | Metres | No. Holes | Series |
|---|---|---|---|---|---|
| Minnova | 1991 | Twinned BR-88-48, BR-88-62, BR-88-60, and BR-87-35 A | 722 | 5 | BR-91-109 to BR91-113 |
| Orex | 1993 | Twinned BR-88-48/BR-91-109, BR-88-85, BR-88-62/BR-91-110 and BR-91-113 | 593 | 7 | BR-93-114 to BR-93-117, BR-93-114B, and BR-93-116B |
| Placer | 1995 | Near ramp portal | 1,263 | 7 | BR-95-119 to BR-95-125 |
| Orex | 2005 | BR Belt at 8675E | 2,422 | 23 | BR-05-001 to BR-05-023 |
| | 2008 | WG and EG | 12,065 | 45 | BR-08-01 to BR-08-44 and BR-08-20 A |
| Osisko | 2010 | Underground Ramp, WG, and Dolliver Mountain | 12,998 | 59 | OSK10-01 to OSK10-59 |
| Osisko | 2011 | EG and WG-Dolliver Mountain | 2,375 | 10 | OSK11-1 to OSK11-10 |
| Anaconda | 2017 | BR and EG | 4,196.3 | 13 | BR-17-01 to BR-17-13 |
| Anaconda | 2018 | BR, EG, and WG | 18,277.3 | 61 | BR-18-14 to BR-18-71 |
| Anaconda | 2019 | EG, BR, and WG | 5,733.8 | 33 | BR-19-72 to BR-19-104 |
| Anaconda | 2020 | EG, BR, and WG | 17,941.7 | 121 | BR-20-105 to BR-20-224 |
| Anaconda | 2021 (To September 15) | EG, BR, and WG | 9,654 | 71 | BR-21-225 to BR-20-295 |

Gervais et al. (2009), Puritch et al. (2006) and Bourgoin et al. (2004) previously described details of drilling programs carried out in support of their respective Mineral Resource Estimates and documented specific details of such programs. A review of these descriptions as well as underlying support documents by Nordmin showed that all programs were carried out to industry standards of their respective periods by competent technical and professional staff. All programs included detailed and systematic geological logging, sampling, and reporting procedures as well as systematic recording of downhole survey data, which, with the exception of a few early holes, was captured using modern borehole survey instrumentation.

Nordmin considers all drilling programs carried out between 1984 and 2011 (Figure 10-1) to meet requirements for the use of associated data for Mineral Resource estimation purposes. The relationship between sample length and true thickness for the drilling programs described in this section varies due to the intersection angle of the drill holes and mineralization. In most cases, true thickness ranges from 60% to 85% of the reported sample length. Drill program reporting does not highlight core loss as being a consistent and significant factor with respect to the assessment of the Property, but triple-tube procedures were applied during the 1995 program to negate potential losses in check sampling carried out by Placer.

In addition to the programs noted above, the Company completed a 228 hole (46,149.1 m) core drilling program consisting of 224 completed drill holes and four abandoned drill holes from 2017 to 2021 to expand and infill the Deposit as well as to obtain sample material for metallurgical testing.

All drilling completed for the Company from 2017 to 2021 was provided by Logan Drilling, recovering NQ, or HQ size core using conventional wireline drilling equipment. Core logging and sampling, downhole surveying, and collar location programs were completed in the same manner for each program under the project supervision of Mr. Paul McNeill, P.Geo., Mr. Steve Barrett, P.Geo., Ms. Tanya Tettelaar, P.Geo., Ms. Alana Haysom, P.Geo., Mr. David A. Copeland, P.Geo., and Michelle English, all employees of the Company.

Drill core samples were collected systematically down each hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 m to 1.0 m depending on the nature and width of veining and mineralization while trying to best honour geological contacts. Samples were collected of half-sawn drill core and shipped to Eastern Analytical for analysis via standard 30 g FA with AA finish. Samples were also analyzed at Eastern Analytical via screen metallics using the entire sample for samples assaying greater than 0.5 g/t gold and select samples for 34 element ICP analysis. Check assays on select fine fraction pulps from the metallic screen FA sample were analyzed for gold at ALS in North Vancouver, BC.

Downhole orientation surveys were conducted under the supervision of site technical staff using a Reflex downhole instrument at nominal 30 m intervals. Drill collars were surveyed using a differential GPS by Company employees or contractors.

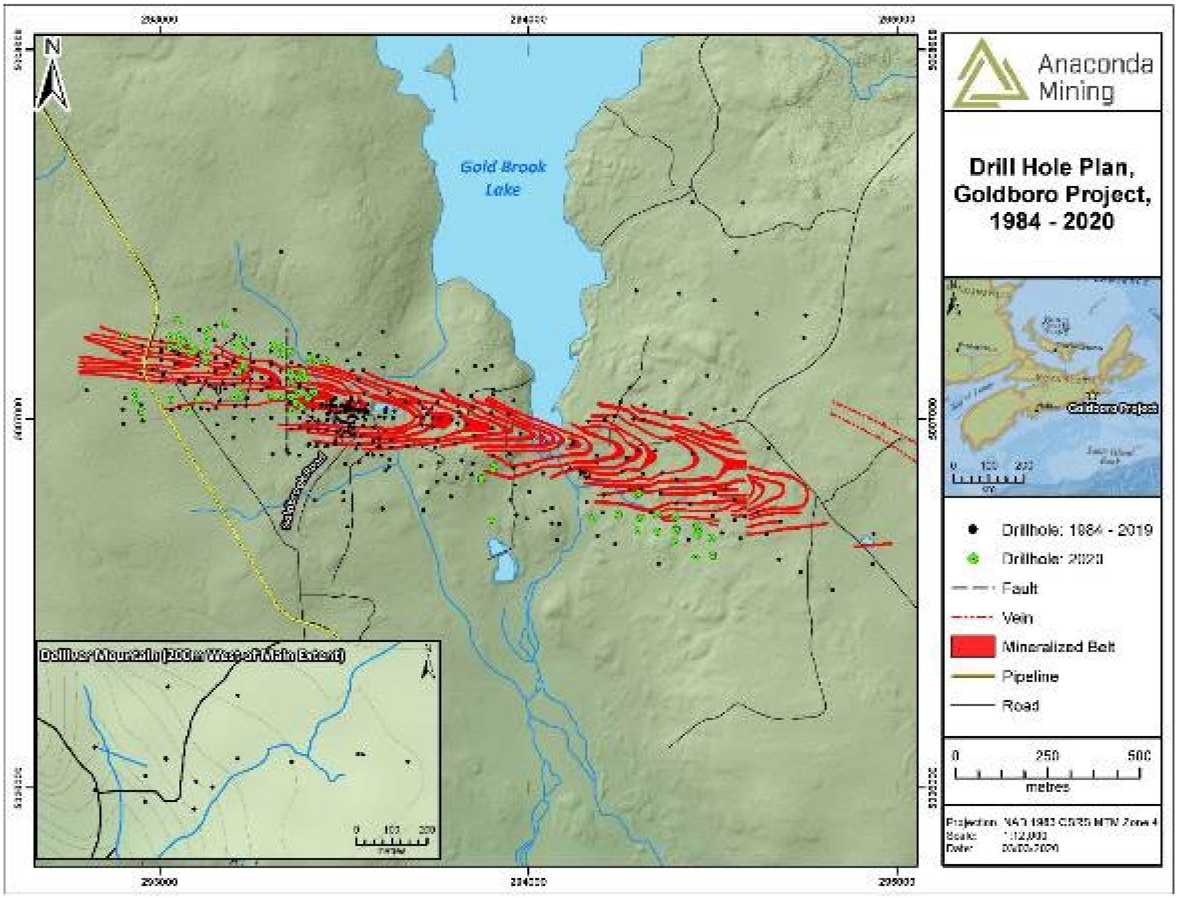



*Figure 10-1: Drill hole location plan view, 1984 to 2020*

## 10.2 1988 and 1989 Programs

After acquiring the Property in 1988, Orex conducted a surface, and underground exploration program that focused on the central portion of the Property. The surface drilling component of the program was comprised of 41 drill holes (BR88-39 to BR88-79, totalling 10,709 m) along the projected anticlinal axis of the Upper Seal Harbour Anticline, between local grid sections 8325E and 9100E. Ten of these holes were drilled in the former West Goldbrook Mine area. At the end of the program, an additional three holes were drilled in the vicinity of the underground decline to investigate near surface continuity of mineralized belts. NQ size core was recovered during the programs and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided by Narex Ore Search Consultants Inc. in 1988 and by Orex staff in 1989. Downhole orientation surveys were conducted under the supervision of site technical staff.

For the 1989 exploration program, Orex was interested in developing a better understanding of the displacement and consequence of the east trending fault that affects the axial zone of the anticline. One hundred and eight (108) underground drill holes (BR-89-U-05 to BR-89-U-112) for a total of 4,740 m were also drilled from the 76 m Level on 11 local grid sections (8637.5E to 8762.5E) spaced 12.5 m apart. Each section drilled was comprised of a fan of ten holes (when possible) spreading above the apex drift toward the 38 m Level, with inclinations of 0°, +20°, +40°, +60°, and +75° to the north and south. This set up allowed the holes to intersect the BR Gold System at right angles. In late 1989, another surface drilling campaign was completed that included 26 drill holes in the WG area, between sections 8150E and 8600E, totalling 2,822 m of drilling (BR89-83 to BR89-108).

## 10.3 1991 and 1993 Programs

In 1991, Orex optioned the Property to Minnova, whose initial objective was to twin drill holes BR88-48, BR88-62, BR88-60, and BR87-35 A as BR91-109, BR91-110, BR91-111, and BR91-112, respectively. This program was designed to validate gold values returned for the earlier holes. CGL was used for initial analysis. Analytical results for the program were much lower than expected and this was attributed to poor core recovery of heavily broken mineralized zones resulting in the loss of free gold during the drilling process. A fifth hole (BR-91-113) was drilled using a 'triple-tube' system to increase core recovery and was collared on section 8712.5E. Holes BR-88-62 and BR-91-110 were also drilled on this section. Core recovery was excellent for BR-91-113, and analytical results were within the range of gold grades encountered during the underground drilling campaign of 1989 to 1990 (Labelle, J.P.; Orex Exploration Inc., 1991).

In an attempt to resolve the question of whether high-grades were present over large intervals in the central portion of the anticline, Orex drilled four holes in the fall of 1993. Holes BR-93-114, BR-93-115, BR-93-116, and BR-93-117 were collared at the locations of Minnova test holes BR-88-48/BR-91-109, BR-88-85, BR 88-62/BR-91-110 and BR-91-113, respectively, and two more holes (BR-93-114 A and BR-93-116 A) were drilled as twins of BR-93-114 and BR-93-116 to ensure that high core recovery was attained over the total stratigraphic package. A total length of 593 m of drilling was carried out during this program. NQ sized core was collected with a triple-tube core barrel, high performance diamond bits were used, and controlled drilling conditions were established. Sludge samples (water return and drill cuttings) were also collected and assayed by the metallic screen method. Logan Drilling was contracted to provide drilling services, and core logging, and sampling services were provided by Orex staff. Downhole orientation surveys were conducted under the supervision of site technical staff.

The results from the sludge assays were much higher than the core assays and were considered unrepresentative. Six drill holes intersected similar rock formations to those observed in previous programs. The sample processing and gold determination methods applied were broadly similar to the CGL method used by Minnova in 1991. As described by Roy (1995) this began with crushing the 1 m samples to 100 % < 10 mesh, followed by splitting into four equal subsamples using a carousel. One subsample was ground in a small ball mill for 24 hours in the presence of cyanide solution and gold levels in both liquid and solid materials were determined to provide a metallurgical balance and calculated head grade. The head grades were lower than expected but similar to those obtained by Minnova. These results indicated that core recovery was not a significant contributing factor to gold grade discrepancies; however, data showed that gold dissolution was less than 90% in more than three-quarters of the samples. In one instance where a large gold nugget was present, gold dissolution was less than 5%. The Centre de Recherches Minérales could not establish a methodology to ensure that greater than 85% gold was being taken into the cyanide solution, and the program was therefore curtailed. Core from holes BR93-116, BR93-116B, and BR93-117 were not assayed completely due to cessation of the program (Roy, 1995).

## 10.4 1995 Programs

In 1995, after receiving the assay results from their sampling of the 4,000 tonne surface stockpile, Placer entered into an option to joint venture agreement with Orex under which a 65% interest in the Property could be earned. Placer became the operator of the Project and subsequently completed a seven-hole core drilling program (holes BR-95-119 to BR-95-125) that totalled 1,263 m of drilling. Results of the program were reported by Gagnon et al. (1996). Triple-tube equipment recovering HQ size core was used and 1 m whole rock samples were split for comparison of gold grade determination processing techniques. Logan Drilling was contracted to provide drilling services, and core logging, and sampling services were provided by Orex staff. Downhole orientation surveys were conducted under the supervision of site technical staff.

Standard FA-metallics screen (CMS), GCNL, and CGL methods were used to determine gold grades for multiple sample splits generated from the 1995 core samples. Placer's analysis of results showed that the GCNL processing approach provided the best representation of total contained gold levels. More specifically, screen metallics assaying of 45 samples from the BR Gold System returned gold grades between 0.02 g/t and 2.48 g/t, while GCNL results for corresponding splits returned gold grades between 0.07 g/t and 3.65 g/t. CGL results for additional splits of the same intervals returned gold grades between 0.03 g/t and 2.52 g/t, with best grades occurring in the south limb of the anticline (Gagnon, D.; Griffin, K.; Ings, D.; Sketchley, D.; Placer Dome Canada Limited, 1996).

## 10.5 2005 Programs

The drilling program carried out by Orex in 2005 consisted of 23 drill holes, designated BR05-001 to BR-05-023, from which HQ size core was recovered. A total of 2,355 m of drilling was completed in the program and this was concentrated in a 225 m wide zone centred on section 8675E of the BR Gold System. The drill pattern was designed to allow two separate shallow areas of mineralization to be similarly tested. Four holes were twinned from previous holes for comparative analysis and mineralized zones were assayed using FA and screen metallics methods with AA or gravimetric finish. At the end of the program, sample composites made by combining reject samples from multiple drill holes previously analyzed by screen metallics methods were sent for metallurgical testing.

HQ size core was recovered during the program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. William Shaw, P. Geo., of W. G. Shaw and Associates Ltd. of Antigonish, Nova Scotia (W.G. Shaw). Downhole orientation surveys were conducted under the supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan and Associates Ltd. (C. J. MacLellan) using differential GPS methods after completion and were spotted using handheld GPS units. Cement plugs were placed in holes immediately below the bedrock surface and casing was pulled. Holes were marked with a numbered wooden stake.

A Mineral Resource Estimate partly supported by metallurgical sample results from the Ramp Area of the Deposit was carried out by P&E, after completion of the 2005 drilling program, results of which were presented earlier in report Section 6. P&E recommended completion of a two-phase diamond drill program for the purpose of:

1. Upgrading the Indicated resource within the Ramp Area to the Measured category.
2. Upgrading a Conceptual Target of gold grade enhancement over a 1 km total strike length to the Indicated category.
3. Testing for extensions to the mineralized zones over a 2.5 km strike length.
4. Defining additional Inferred resources within the 2.5 km drilled deposit strike length.

## 10.6 2008 Program

Orex carried out an exploration drilling program in 2008 that consisted of 45 drill holes (BR-08-01 to BR-08-44, and BR-08-20 A) comprising a total of 12,201.5 m and focused on the area between WG and EG. The purpose of this drill program was to infill as follow up to the previous drill programs and to test potential extensions of gold mineralization to the east and west of the BR Gold System.

NQ size core was recovered during the 2008 program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under the project supervision of Mr. William Shaw, P. Geo. Downhole orientation surveys were conducted under the supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan using differential GPS methods after completion and were spotted using handheld GPS units. Cement plugs were placed in holes immediately below the bedrock surface, and the casing was pulled. Holes were marked with a numbered wooden stake.

A Mineral Resource Estimate, prepared in accordance with NI 43-101, was subsequently completed in 2009 by InnovExplo and results were reported by Gervais et al. (2009) with an effective date of September 15, 2009. Gervais et al. (2009) recommended a two-phase exploration program, the first phase of which consisted of a 35-drill hole program for a total of approximately 8,750 m to add additional Inferred resources. These drill holes were proposed for the area from local grid section 8100E for a distance of 1,500 m west along the strike of the anticlinal fold structure, including the WG, and Dolliver Mountain areas. The second phase of recommended exploration consisted of pilot milling and metallurgical testing of at least a 10,000 tonne Bulk Sample recovered from the resource model area. The design and execution of the second phase program were considered conditional on the success of the first phase.

## 10.7 2010 Programs

### 10.7.1 Diamond Core Drilling

During the first half of 2010, Osisko completed 59 drill holes (OSK10-01 to OSK10-59) for a total of 12,989 m in the Ramp, WG, and Dolliver Mountain areas. NQ size core was recovered, and drilling was carried out by Logan Drilling. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. Jean Lafleur, P. Geo., and geological staff from W.G. Shaw carried out field operations, including core logging, and sampling activities. Mercator also assisted with core logging through the provision of one geologist for part of the program. Downhole orientation surveys were conducted under the supervision of site technical staff using Reflex downhole instrumentation at nominal 50 m intervals. Drill collars were surveyed by C.J. MacLellan using differential GPS methods after completion and were spotted using handheld GPS units. Cement plugs were placed in holes immediately below the bedrock surface, and the casing was pulled. Holes were marked with a numbered wooden stake. Drilling was carried out under separate programs identified as 2D, 2E, and 2F.

Angled holes were drilled in 2010 from both north and south sides of the approximately 100 m wide BR segment of the Upper Seal Harbour Anticline and terminated in the hinge area to prevent downdip drilling on the opposing limb. As such, each hole penetrated about 50% of the total width of the mineralized target corridor that occurs in the core of the anticline.

In January 2010, Osisko began the 2F program with the objective of replacing incomplete and/or non-compliant historical drill results with compliant data, primarily in the deeper zones of the Ramp Area and extending westward toward WG. Some of the historic drill holes in this area were not sampled in their entirety, and others had been sampled on intervals considered too long for current interpretive purposes. The 2F program also included infill drilling, which provides validation of the theory that a larger structural domain of gold mineralization is present on the Property, centred on the hinge zone of the Upper Seal Harbour Anticline. An important component of this assessment was to determine whether consistent and significant lower-grade gold mineralization was present in unsampled intervals previously assumed to be barren of gold mineralization. Eighteen drill holes were completed for a total of 4,730 m of drilling in the 2F program. This was concentrated in a target area measuring approximately 250 m in east-west strike length along the anticlinal structure at elevations between 75 m and 250 m below surface.

The 2D drilling program was comprised of 25 drill holes for a total of 4,894 m and targeted a 350 m strike length along the anticline at depths between 25 m and 200 m below surface. These holes were drilled to replace historic drill holes having incomplete or otherwise deficient sampling and to assess possible downdip extensions to known mineralization. Resource extension drilling was also completed in the area between the west limit of past drilling and the Dolliver Mountain area.

The 2E program consisted of 16 drill holes for a total of 3,371.5 m that tested a 200 m strike length between WG and the Dolliver Mountain area, as well as a 500 m strike length that included the Dolliver Mountain mine and potential westward extensions of associated geology. A 600 m gap in drilling between the two areas remained untested.

### 10.7.2 Reverse Circulation Drilling

During the middle of 2010, RC drilling equipment was used to explore near surface gold mineralized structures on the Property by recovering basal till and bedrock samples for gold assaying and whole rock analysis. RC drilling was conducted by Archibald Drilling and Blasting of Truro, Nova Scotia, at hole spacings of 25 m to 50 m along north-south fences oriented across the interpreted trend of bedrock gold mineralization. Mercator provided site supervision and reporting services for the program.

The program consisted of 64 RC drill holes (OSK10RC-130 to OSK10RC-194) for a total of 505 m. These were completed in the EG, BR Ramp, and WG Areas. The program was successful in the identification of future diamond drilling target areas. Encouraging results from the under-explored EG area included 10.85 g/t gold and 25.65 g/t gold in two bedrock samples, each of which measured 1 m in length. No estimates of sample true width were assigned to these samples. Selected highlights from the RC drilling campaign were disclosed in a November 18, 2010 news release by Orex and significant results are tabulated in Table 10-2.

**Table 10-2: Significant 2010 Orex RC Drilling Results**

| Hole No. | Location | From (m) | To (m) | Width (m) | Type | Assay (Au ppm) |
|---|---|---|---|---|---|---|
| OSK10RC-135 | EG | 7 | 8 | 1 | Bedrock | 1.38 |
| OSK10RC-137 | EG | 4 | 5 | 1 | Bedrock | 10.85 |
| OSK10RC-147 | EG | 5 | 6 | 1 | Bedrock | 25.65 |
| OSK10RC-186 | Ramp Area | 3 | 4 | 1 | Overburden | 1.47 |
| OSK10RC-194 | WG | 4 | 5 | 1 | Overburden | 1.17 |
| OSK10RC-194 | WG | 6 | 7 | 1 | Bedrock | 4.23 |
| OSK10RC-194 | WG | 8 | 9 | 1 | Bedrock | 2.80 |

Overburden samples were collected at 1 m downhole intervals above the bedrock-overburden interface, and bedrock chip samples were collected at 1 m downhole intervals, beginning at the interface, and generally extending to a depth of 1 m to 3 m into rock. Bedrock drilling was advanced to a depth of 12 m in selected holes. All samples collected during the program were delivered by

Mercator or Armour Transport to the MEC at Dalhousie University in Halifax, Nova Scotia, where processing and analysis were carried out.

## 10.8 2011 Program

The 2011 core drilling campaign by Osisko focused on two areas within the Property, the first being a 750 m interval from EG to the east property boundary, with drill holes on three local grid sections (9650E to 10150E) spaced 250 m apart; and the second being a 600 m interval between WG and Dolliver Mountain, where drill holes on three sections were spaced 200 m apart in an area of no historical drilling. NQ size core was recovered during the program and Logan Drilling was contracted to provide drilling services. Core logging and sampling services were provided on a consulting basis under project supervision of Mr. Bruce Mitchell, P.Geo., and geological staff from W.G. Shaw carried out field operations, including core logging, and sampling activities.

A total of six drill holes (OSK11-01 to OSK11-06) were completed at EG for a total of 2,375.4 m. Results for these define narrow, high-grade quartz veins, often with visible gold, that are associated with wider zones of lower-grade gold values in argillite. These occur along an interval of approximately 500 m in strike length that extends to the eastern property boundary. The mineralization appears to be plunging shallowly to the east, and additional drilling in the EG area is required to delineate the mineralized zones further.

A total of four drill holes (OSK11-07 to OSK11-10) were completed in the gap area between WG and Dolliver Mountain for a total of 765.4 m. Three out of the four holes intersected occasional, narrow, high-grade quartz veins, but wall rock was not well mineralized. One drill hole (OSK11-07) was abandoned before reaching its target depth due to drilling problems.

## 10.9 2017 Program

After acquiring the Project in mid-2017, the Company completed diamond drilling in 13 drill holes (BR-17-01 to BR-17-13) totalling 4,196.3 m. The first five drill holes of the program were designed to acquire samples for metallurgical testing, verify historical drilling, and test the potential extents of the Deposit at depth.

### 10.9.1 2017 Metallurgical Drilling

During summer 2017, the Company completed a five-hole (BR-17-01 to 05), 643 m, diamond drill program that tested the BR and EG Gold Systems (Figure 10-2). Drilling of the five holes, using HQ-sized core, was completed in order to collect samples for metallurgical test work on the mineralization, with each of the completed holes twinning a historic drill hole.

Drill core samples were collected systematically down each hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 m to 1.0 m depending on the nature and width of veining and mineralization while trying to best honour geological contacts. Samples were collected of quarter-sawn drill core and shipped to Eastern for analysis via standard 30 g FA with AA finish. Samples were also analyzed at Eastern via total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold and all samples for 34 element ICP analysis. Half of the sampled drill core assaying greater than 0.5 g/t gold and totalling approximately 25 kg in mass was shipped to Thibault in Fredericton, New Brunswick, for metallurgical test work, while the remaining quarter of the core was retained as an archive split.

Results from the twinned metallurgical drilling confirmed the accuracy of historical drilling. Multiple occurrences of visible gold and assays with high gold grades are present in all five initial 2017 drill holes. Preliminary test work produced a total gold recovery of 91.9%.

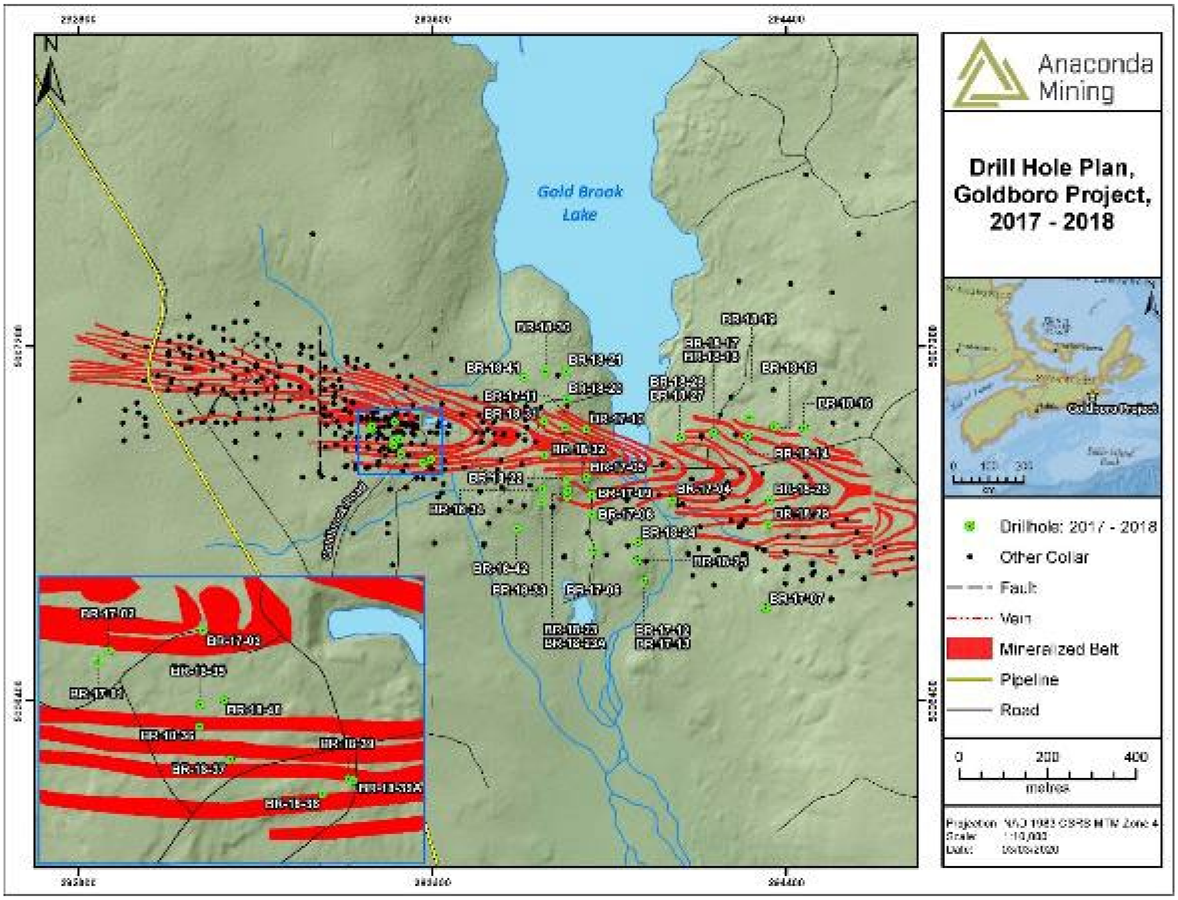



*Figure 10-2: Drill hole location plan view, 2017 to 2018 (up to BR-18-42)*

### 10.9.2 2017 Expansion and Infill Drilling

In 2017, the Company completed eight additional drill holes (BR-17-06 to BR-17-13) totalling 3,553.3 m on the Property and designed to locally infill and test mineralization at depth in the BR Gold System. Hole locations for this program are displayed in Figure 10-2.

Representative assays from the 2017 program include:

- 34.70 g/t gold over 3.5 m (82.0 m to 85.5 m) in hole BR-17-09.
- 24.34 g/t gold over 3.8 m (389.9 m to 393.7 m) in hole BR-17-06.
- 9.12 g/t gold over 3.2 m (293.8 m to 2.97 m) in hole BR-17-08.
- 31.56 g/t gold over 1.0 m (259.0 m to 260.0 m) in hole BR-17-08.
- 59.97 g/t gold over 0.5 m (272.7 m to 273.2 m) in hole BR-17-06.
- 17.68 g/t gold over 0.5 m (69.6 m to 70.1 m) in hole BR-17-10.

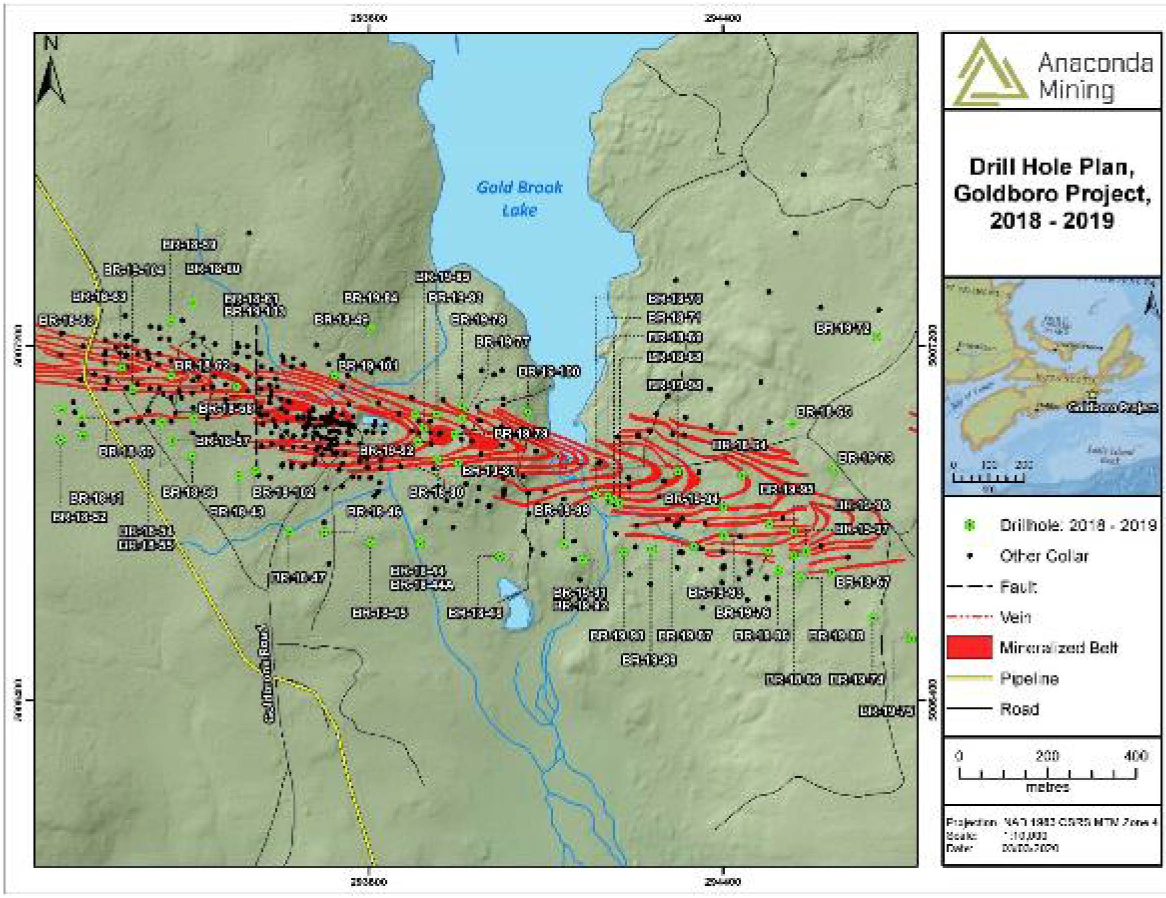



*Figure 10-3: Drill hole location plan view, 2018 (BR-18-43) to 2019 (BR-19-104)*

## 10.10 2018 Program

During 2018 the Company completed 61 drill holes (BR-18-17 to BR-18-71) totalling 18,277.3 m with the purpose of both infilling certain portions of the Deposit while expanding the Deposit eastward.

The 2018 drill program focused on infilling areas of Inferred Mineral Resources and expanding the Deposit. Drilling focused on testing the down plunge, down dip, and along strike extension of the BR Gold System, EG Gold System, and WG Gold System. In addition, several holes tested the depth extent of the BR Gold System to depths of 525 m.

### 10.10.1 Boston Richardson and East Goldbrook Infill and Expansion

Drilling also focused on infilling under-drilled areas of the Deposit in order to upgrade Mineral Resources from the Inferred to Indicated category with a focus on sections 9050E, 9100E, 9150E, 9250E, 9350E, 9450E, 9500E, 9550E and 9650E (Figure 10-3).

BR-18-35 to BR-18-40 on local grid sections 8700E to 8800E were designed to test the BR Gold System in an area of proposed underground bulk sampling. This drilling successfully intersected mineralization modelled from historical drilling and was used to refine selected locations for a bulk sampling program.

Infill drilling of the BR Gold System from 9000E to 9250E intersected high-grade intervals in predicted previously modelled mineralized Belts as well as intersecting new mineralized Belts below historical drilling, extending mineralization at depth by up to 150 m over a strike length of 250 m. This drilling was used to refine the geological interpretation suggesting that the anticlinal fold axis was slightly inclined to the north.

Infill drilling of the EG Gold System from 9000E to 9500E typically intersected zones of gold mineralization, both low-, and high-grade, but showing overall continuity between mineralized Belts. Additionally, drilling from 9500E to 9650E intersected three new near surface mineralized Belts in the EG Gold System.

Representative assays from the BR and EG portion of the drill program include:

- 252.76 g/t gold over 0.4 m (76.6 m to 77.0 m) in hole BR-18-15.
- 31.04 g/t gold over 1.0 m (6.2 m to 7.2 m) in hole BR-18-17.
- 12.87 g/t gold over 2.0 m (130.6 m to 132.6 m) in hole BR-18-18.
- 25.31 g/t gold over 1.0 m (62.0 m to 63.0 m) in hole BR-18-18.
- 9.29 g/t gold over 2.1 m (420.6 m to 422.7 m) in hole BR-18-21.
- 11.27 g/t gold over 13.5 m (201.0 m to 214.5 m), including 15.63 g/t gold over 1.4 m and 44.33 g/t gold over 2.5 m in hole BR-18-22.
- 4.13 g/t gold over 20.5 m (324.5 m to 345.0 m), including 9.93 g/t gold over 7.5 m and 79.34 g/t gold over 0.5 m in hole BR-18-23.
- 10.55 g/t gold over 6.1 m (223.0 m to 229.1 m), including 18.78 g/t gold over 3.1 m in hole BR-18-22.
- 5.10 g/t gold over 9.6 m (116.0 m to 125.6 m), including 25.82 g/t gold over 1.5 m in hole BR-18-22.
- 7.22 g/t gold over 6.5 m (310.5 m to 317.0 m), including 16.00 g/t gold over 2.0 m in hole BR-18-23.
- 752.54 g/t gold over 0.5 m (145.0 m to 145.5 m) in hole BR-18-25.
- 56.67 g/t gold over 1.0 m (132.5 m to 133.5 m) in hole BR-18-25.
- 17.00 g/t gold over 1.0 m (39.0 m to 40.0 m) in hole BR-18-25.
- 6.55 g/t gold over 2.5 m (84.5 m to 87.0 m) in hole BR-18-24.
- 62.01 g/t gold over 1.5 m (108.5 m to 110.0 m) in hole BR-18-26.
- 12.66 g/t gold over 1.7 m (27.8 m to 29.5 m) in hole BR-18-26.
- 23.24 g/t gold over 2.5 m from (21.5 m to 24.0 m) in hole BR-18-28.
- 7.12 g/t gold over 4.5 m from (193.5 m to 194.0 m) in hole BR-18-29.
- 2.21 g/t gold over 25.5 m (506.1 m to 531.6 m), including 12.39 g/t gold over 3.2 m in hole BR-18-30.
- 24.49 g/t gold over 1.0 m (177.0 m to 178.0 m) in hole BR-18-34.
- 4.82 g/t gold over 3.6 m (384.7 m to 388.3 m), including 9.90 g/t gold over 1.1 m in hole BR-18-33.
- 3.00 g/t gold over 7.5 m (270.5 m to 278.0 m) in hole BR-18-34.
- 63.88 g/t gold over 1.0 m (378.0 m to 379.0 m) in hole BR-18-41.
- 77.69 g/t gold over 0.5 m (64.5 m to 65.0 m) in hole BR-18-42.
- 6.05 g/t gold over 3.7 m (472.0 m to 475.7 m), including 28.12 g/t gold over 0.7 m in hole BR-18-42.
- 5.87 g/t gold over 1.5 m (451.6 m to 453.1 m) in hole BR-18-42.

### 10.10.2 West Goldbrook Infill and Expansion

Fifteen drill holes, BR-18-43 and BR-18-50 to BR-18-63 were designed to infill portions of the WG Gold System over 400 m of existing strike length. Vertically drilled holes BR-18-61 to BR-18-63 were also intended to test the system along the fold hinge below previously modelled mineralization. The infill drilling intersected mineralized Belts in areas of known Inferred Mineral Resources and demonstrated continuity of mineralization, providing the requisite geological data to potentially convert those Inferred Resources to the Indicated category. Expansion drilling intersected the host fold structure, alteration, and mineralization characteristic of the Deposit to a depth of 450 m, demonstrating that the Deposit continues up to 175 m below the previously modelled WG Gold System. Additionally, drill holes BR-18-43, BR-18-51, BR-18-52, BR-18-56, and BR-18-57 intersected 45 to 50 m of massive greywacke. The atypical bedding thickness and lithological and alteration characteristics are comparable to the Marker Horizon, which stratigraphically overlies the BR Gold System. This evidence confirmed the historical hypothesis that the WG Gold System is the fault-offset western continuation of the BR Gold System. Representative assays from the WG portion of the drill program include:

- 78.07 g/t gold over 1.1 m (196.7 m to 197.8 m) in hole BR-18-63.
- 32.42 g/t gold over 2.6 m (300.3 m to 302.9 m), including 201.68 g/t gold over 0.4 m in hole BR-18-59.
- 24.06 g/t gold over 2.0 m (138.0 m to 140.0 m), including 55.58 g/t gold over 0.5 m in hole BR-18-61.
- 20.02 g/t gold over 2.0 m (226.5 m to 228.5 m), including 78.29 g/t gold over 0.5 m in hole BR-18-56.
- 25.45 g/t gold over 1.5 m (199.3 m to 200.8 m), including 46.54 g/t gold over 0.8 m in hole BR-18-59.
- 11.15 g/t gold over 1.0 m (179.0 m to 180.0 m) in hole BR-18-51.

### 10.10.3 Boston Richardson Expansion at Depth

Six drill holes, BR-18-44 to BR-18-49 between local grid sections 8600E to 9100E, targeted a previously untested deeper area of the BR Gold System over 350 m of strike and to depths of 525 m. Drilling expanded two mineralized Belts an additional 200 m along strike and expanded five other Belts over 350 m along strike. Nineteen occurrences of visible gold were observed in the six drill holes and the character of the mineralization in those holes is consistent with results seen throughout the BR Gold System to date. The BR Gold System remains open for further expansion at depth and down plunge. Representative assays from this drilling include:

- 8.79 g/t gold over 8.0 m (483.0 m to 491.0 m), including 64.40 g/t gold over 0.8 m in hole BR-18-44.
- 51.89 g/t gold over 1.0 m (224.5 m to 225.5 m) in hole BR-18-46.
- 5.15 g/t gold over 4.0 m (390.9 m to 394.9 m), including 10.08 g/t gold over 1.5 m in hole BR-18-47.
- 21.06 g/t gold over 1.0 m (200.1 m to 201.1 m) in hole BR-18-48.
- 6.39 g/t gold over 2.0 m (457.2 m to 459.2 m) and 3.35 g/t gold over 4.5 m (539.0 to 543.5 m), including 25.68 g/t gold over 0.4 m in hole BR-18-49.

## 10.11 2019 Program

During 2019 the Company completed 33 drill holes (BR-19-72 to BR-19-104) totalling 5,733.8 m with the purpose of both infilling certain portions of the Deposit while expanding the Deposit eastward.

### 10.11.1 Boston Richardson Infill

Infill drilling at the BR Gold System consisted of drilling select areas in order to upgrade from Inferred Mineral Resources to Measured and Indicated Mineral Resources. Representative assays include:

- 1.87 g/t gold over 2.1 m (28.9 m to 31.0 m) in hole BR-19-79.
- 2.43 g/t gold over 2.0 m (46.0 m to 48.0 m) in hole BR-19-80.
- 1.89 g/t gold over 3.0 m (20.9 m to 23.9 m) in hole BR-19-83.
- 1.20 g/t gold over 2.2 m (13.4 m to 15.6 m) in hole BR-19-84.

### 10.11.2 East Goldbrook Infill and Expansion

Infill and expansion drilling of the near surface mineralization potential of the EG Gold System in proximity to the optimized open pit shell as well as deeper exploration holes successfully intersected gold mineralization in all drill holes. Geological interpretation of these zones demonstrates the continuity of several mineralized Belts. Representative assays from this drilling include:

- 17.79 g/t gold over 0.5 m (192.8 m to 193.3 m) in hole BR-19-74.
- 5.36 g/t gold over 2.2 m (321.1 m to 323.3 m) in hole BR-19-75.
- 102.43 g/t gold over 0.7 m (142.0 m to 142.7 m) in hole BR-19-86.
- 72.40 g/t gold over 0.6 m (21.0 m to 21.6 m) in hole BR-19-87.
- 32.62 g/t gold over 0.9 m (290.7 m to 291.6 m) in hole BR-19-87.
- 16.65 g/t gold over 2.0 m (167.5 m to 169.5 m), including 65.49 g/t gold over 0.5 m in hole BR-19-88.
- 50.60 g/t g/t gold over 1.0 m (246.0 m to 247.0 m) in hole BR-19-89.
- 12.23 g/t gold over 2.0 m (214.3 m to 216.3 m) in hole BR-19-89.
- 6.03 g/t gold over 2.9 m (200.7 m to 203.6 m) in hole BR-19-90.
- 27.12 g/t gold over 2.5 m (51.3 m to 53.8 m), including 133.11 g/t gold over 0.5 m in hole BR-19-97.

## 10.12 2020 Drilling

From June to December 2020, the Company completed 121 drill holes (BR-20-105 to BR-20-224) on the Property totalling 17,941.7 m of drilling (Figure 10-4). Core size was typically NQ diameter except BR-20-133, BR-20-157, BR-20-163, BR-20-167, BR-20-169, and BR-20-171, which were drilled as HQ size core to provide material for metallurgical studies.

The 2020 program focused on targeting under-drilled areas of the Deposit to upgrade Mineral Resources from the Inferred to Indicated and Measured Resource categories within the WG and EG Gold Systems. Drilling also focused on testing areas with the conceptual open pit that had seen little historical drilling.

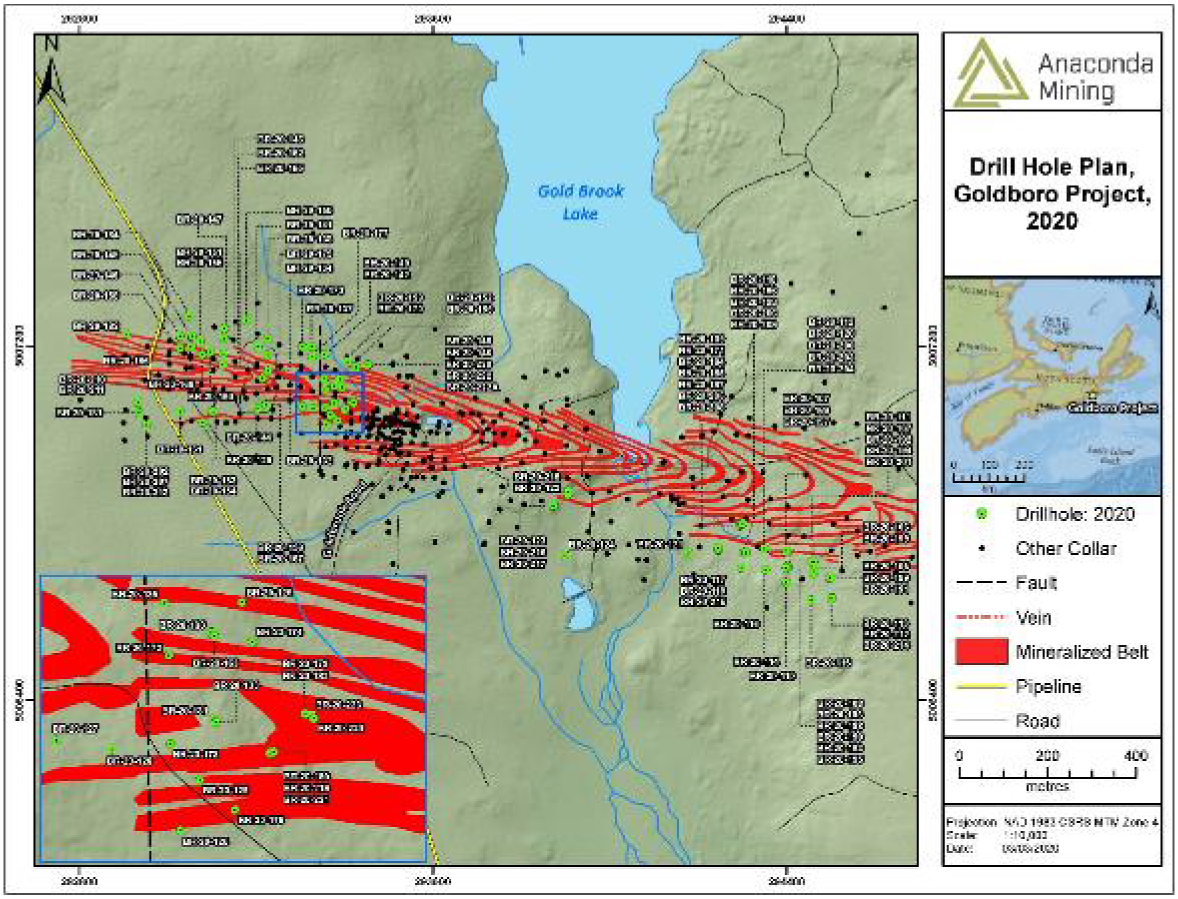



*Figure 10-4: Drill hole location plan view, 2020*

### 10.12.1 2020 West Goldbook Infill

In WG, 38 drill holes totalling 6,058 m were designed to infill portions of the WG Gold System over a 400 m of existing strike length. The majority of drill holes were intended to test areas of both the north and south limb within 150 m of surface. Holes BR-20-175, BR-20-207, and BR-213, however, targeted deeper mineralization on the south limb at the western end of the WG system. Drill hole BR-20-157 was drilled using HQ diameter core and was used for both FA and metallurgical analysis.

The infill drilling intersected mineralized Belts, including high-grade intercepts, within the target areas. These drill results confirmed and further refine the geometry and location of the modelled mineralized resource and allowed the resource to be upgraded to the Measured and Indicated category throughout much of the WG system. The drill results also indicate the resource remains open to the west, with the deeper drilling confirming the presence of mineralized Belts to a vertical depth of 280 m. Representative assays from the WG portion of the drill program include:

- 28.52 g/t gold over 2.0 m (125.6 m to 127.6 m), including 112.87 g/t gold over 0.5 m in hole BR-20-146.
- 1.85 g/t gold over 15.0 m (184.0 m to 199.0 m), including 29.13 g/t gold over 0.5 m and 9.12 g/t gold over 0.5 m in hole BR-20-147.
- 8.68 gold over 1.9 m (127.6 m to 129.5 m), including 17.60 g/t gold over 0.9 m in hole BR-20-160.

- 6.63 g/t gold over 5.3 m from (35.7 m to 41.0 m), including 47.67 g/t gold over 0.5 m in hole BR 20-134.
- 3.99 g/t gold over 4.5 m (186.0 m to 190.5 m), including 26.80 g/t gold over 0.5 m in hole BR-20-177.
- 4.29 g/t gold over 3.6 m (27.0 m to 30.6 m), including 26.70 g/t gold over 0.5 m in hole BR-20-158.
- 3.72 g/t gold over 3.5 m (76.0 m to 79.5 m), including 12.70 g/t gold over 0.5 m in hole BR-20-182.

### 10.12.2 2020 Boston Richardson Infill

Thirty drill holes, totalling 3,250.2 m, were drilled over a 100 m strike length at the western end of the BR Gold System, adjacent to the north-south trending fault that separates the WG Gold System from the Boston Richardson. As with the WG infill drilling, the purpose of the drilling was to target shallow (<150 m vertical depth) areas of Inferred Mineral Resource with the goal of upgrading resources to Indicated or Measured categories. Drilling also targeted areas within the conceptual pit with a low drill density. Drill holes BR-20-133 and BR-20-163 were drilled using HQ diameter core. In addition to FA, BR-20-163 was also sent for metallurgical analysis.

Drilling successfully intersected mineralized Belts, including high-grade intercepts, on both the north, and south limb with the target areas. These mineralized intersections helped refine the Mineral Resource model and aided in upgrading the categorization of resources to Indicated and Measured. Representative assays from the BR portion of the drill program include:

- 6.65 g/t gold over 2.0 m (23.0 m to 25.0 m), including 17.40 g/t gold over 0.5 m in hole BR-20-176.
- 6.05 g/t gold over 11.7 m (189.9 m to 201.6 m), including 12.55 g/t gold over 2.5 m and 5.52 g/t gold over 7.2 m (210.8 m to 218.0 m) in hole BR-20-142.
- 7.76 g/t gold over 4.4 m (39.8 m to 44.2 m), including 22.07 g/t gold over 1.0 m and 9.42 g/t gold over 1.0 m in hole BR-20-135.
- 1.34 g/t gold over 18.7 m (97.3 m to 116.0 m), including 6.41 g/t gold over 1.0 m in hole BR 20-135.
- 1.86 g/t gold over 14.5 m (55.5 m to 70.0 m) and 2.78 g/t gold over 4.5 m (44.0 m to 48.5 m), and 1.31 g/t gold over 5.5 m (19.0 m to 24.5 m) in hole BR-20-221.
- 1.37 g/t gold over 21.5 m (72.0 m to 93.5 m), including 26.80 g/t gold over 0.5 m in hole BR-20-219.
- 2.08 g/t gold over 9.0 m (130.0 m to 139.0 m), including 12.20 g/t gold over 1.0 m in hole BR-20-216.
- 8.56 g/t gold over 2.0 m (39.0 m to 41.0 m), including 16.70 g/t gold over 1.0 m in hole BR-20-224.
- 2.66 g/t gold over 5.6 m (121.2 m to 126.8 m), including 9.72 g/t gold over 0.8 m in hole BR-20-210.

### 10.12.3 2020 East Golbrook Infill

Fifty-two drill holes totalling 8,533.5 m were drilled targeting the EG Gold System. Five drill holes were drilled in the centre of the EG System between section lines 9070E and 9100E. The remaining 47 drill holes were drilled at the eastern end of the EG System between 9375E and 9700E Figure 10-5, and Figure 10-6. The purpose of the drilling was to target areas of near surface Inferred Mineral Resource proximal to planned development with the goal of upgrading them to Indicated or Measured Mineral Resource. All the drill holes were drilled on the south limb of the anticline. Drill holes BR-20-167, BR-20-169, and BR-20-171 were drilled at HQ diameter and for FA and metallurgical analysis.

Drilling intersected 13 mineralized zones, including several very high-grade zones (>50 g/t) and 29 visible gold occurrences. These mineralized zones were consistently intersected near the areas projected by the geological model, which has led to increased confidence in the model. Representative assays from the EG portion of the drill program include:

- 871.23 g/t gold over 0.5 m (52.9 m to 53.4 m) and 47.87 g/t gold over 1.5 m (63.1 m to 64.6 m) in hole BR-20-193.
- 20.08 g/t gold over 2.4 m (50.6 m to 53.0 m), including 74.40 g/t gold over 0.6 m in hole BR-20-195.
- 16.79 g/t gold over 3.0 m (113.0 m to 116.0 m) in hole BR-20-111.
- 9.78 g/t gold over 5.0 m (353.0 m to 358.0 m), including 46.86 g/t over 1.0 m in hole BR-20-114.
- 27.46 g/t gold over 1.0 m (230.0 m to 231.0 m) within a zone grading 2.54 g/t gold over 14.0 m in hole BR-20-113.
- 3.55 g/t gold over 10.9 m (234.0 m to 244.9 m), including 26.43 g/t gold over 1.0 m in hole BR-20-120.
- 59.21 g/t gold over 1.0 m (113.0 m to 114.0 m) in hole BR-20-117.
- 2.28 g/t gold over 13.0 m (289.0 m to 302.0 m), including 22.28 g/t gold over 1.0 m in hole BR-20-116.
- 1.45 g/t gold over 18.0 m (269.0 m to 287.0 m), including 9.39 g/t gold over 1.0 m in hole BR-20-119.
- 0.55 g/t gold over 17.9 m (23.1 m to 41.0 m) in hole BR-20-205.

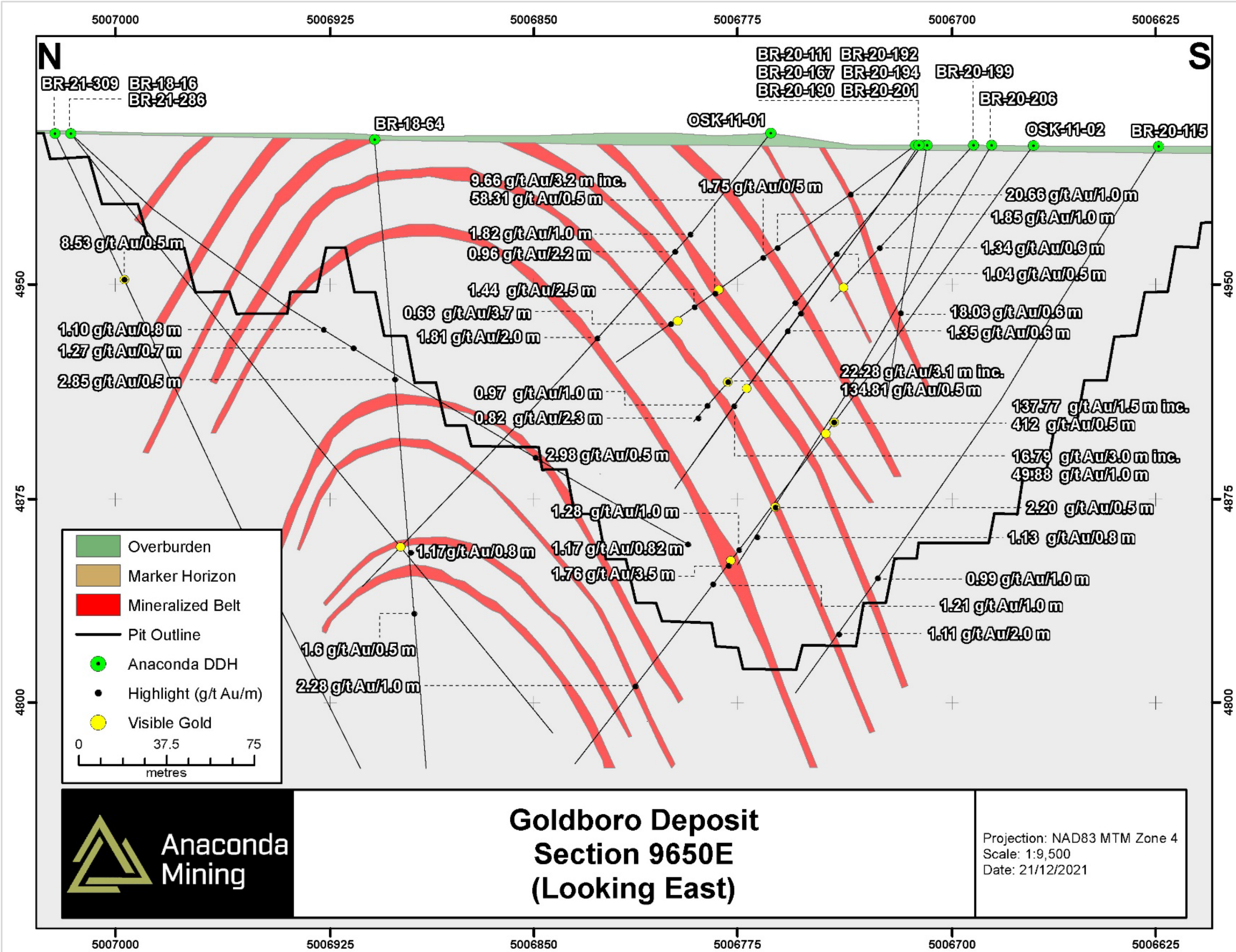



*Figure 10-5: Section 9550E (looking east)*

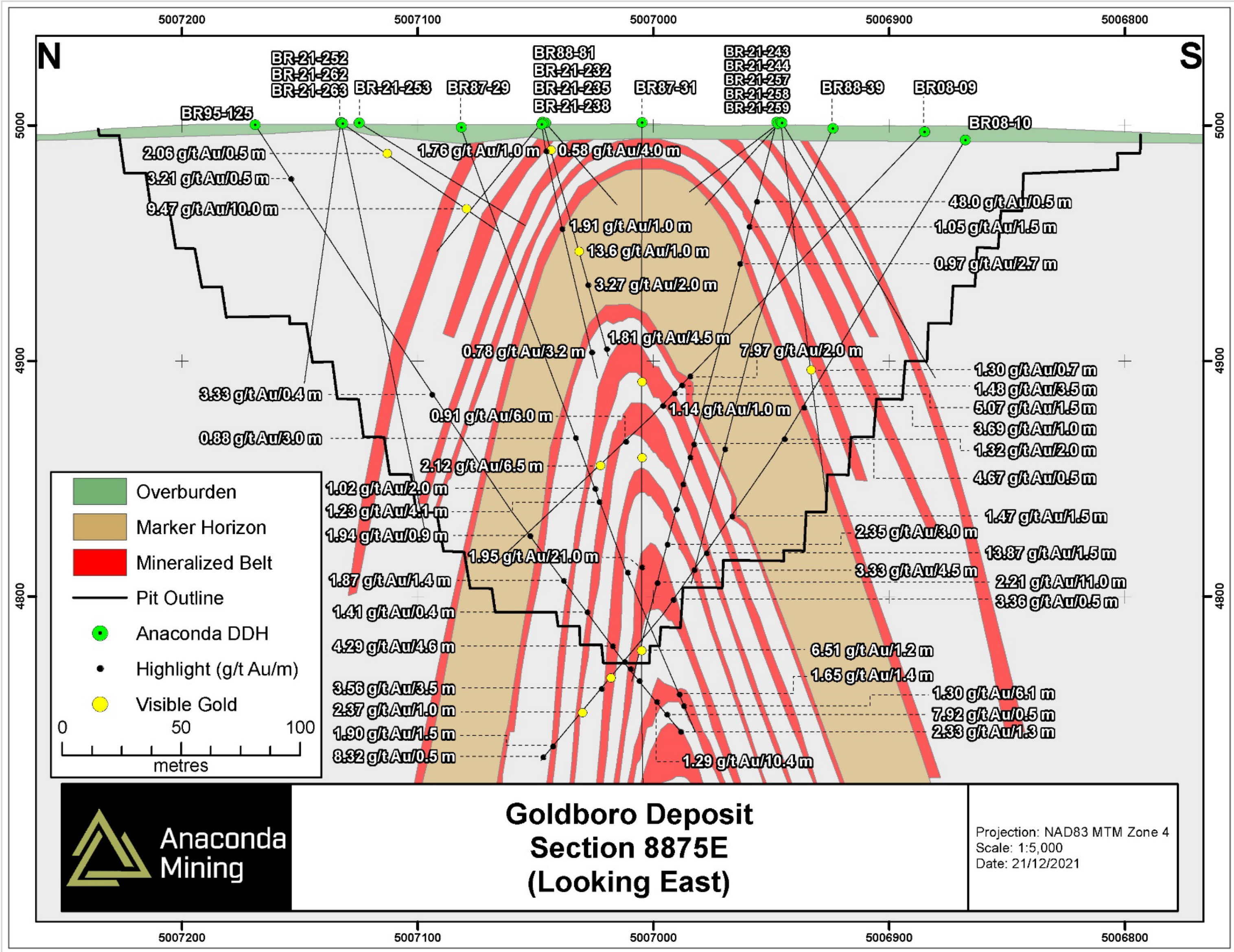



*Figure 10-6: Section 8600E (looking east)*

## 10.13 2021 Drilling

During 2021, 30 drill holes, totalling 3,250.2 m, were drilled across the area of the open pits contemplated in the Preliminary Economic Assessment (dated August 5, 2021) to confirm mineralization and upgrade resource categories. Of these drill holes, 25 were HQ diameter drill holes for the purpose of geotechnical and metallurgical testing. These mineralized zones were consistently intersected near the areas projected by the geological model, which has led to increased confidence in the model. Representative intercepts of the program include:

- 2.21 g/t gold over 11.0 m (196.5 to m 207.5 m) in diamond drill hole BR-21-259.
- 1.61 g/t gold over 14.7 m (64.9 m to 79.6 m), including 5.49 g/t gold over 1.5 m in diamond drill hole BR-21-285.
- 1.71 g/t gold over 9.5 m (91.2 m to 100.7 m), including 9.78 g/t gold over 1.0 m in diamond drill hole BR-21-285.
- 16.09 g/t gold over 1.5 m (87.3 m to 88.8 m) in diamond drill hole BR-21-271.
- 1.33 g/t gold over 4.5 m (144.5 m to 149.0 m) in diamond drill hole BR-21-259.
- 1.20 g/t gold over 5.0 m (156.0 m to 161.0 m) in diamond drill hole BR-21-25.
- 7.88 g/t gold over 7.9 m (364.3 m to 372.2 m), including 21.38 g/t gold over 1.5 m and 17.32 g/t gold over 1.5 m in diamond drill hole BR-21-291.
- 6.19 g/t gold over 2.6 m (94.6 m to 97.2 m), including 24.80 g/t gold over 0.6 m in diamond drill hole BR-21-299.
- 3.67 g/t gold over 4.2 m and 14.10 g/t gold over 0.5 m within a broader zone consisting of 1.91 g/t gold over 12.6 m (279.4 m to 292.0 m), in diamond drill hole BR-21-295.

## 10.14 Core Recovery

There is minimal core loss at the Project. The rocks are highly competent, except for local intersections of minor fault zones where a non-material amount of core may be lost. Core loss is not a consistent or significant factor.

## 10.15 Comments on Section 10

In the opinion of the QP, the quantity and quality of the lithological, collar, downhole survey, and SG data collected in the exploration programs are sufficient to support the Mineral Resource Estimate.

# 11. SAMPLE PREPARATION, ANALYSES, AND SECURITY

Drill holes from programs completed between 1984 and 2011 are included in the database used for the current Resource Estimate. The sampling approaches in programs carried out prior to 2005 generally reflect sampling of visibly determined Belts, respective of major geological units, plus varying amounts of adjacent material. Exceptions to this, which include continuous core sampling and/or total core rather than half core sampling, pertain to certain historic metallurgical programs. Continuous mineralized zone sampling, respective of major lithologic units, pertains to 2005 and later programs.

Drill core samples from surface drilling programs carried out in 2005 (HQ core) and 2008 (NQ core) were generated by Orex during this period. Samples were sent to ALS Chemex (ALS) facilities in either Val-d'Or, Québec (2005) or Timmins, Ontario (2008). ALS was independent of Orex and is independent of the Company. Standard rock sample crushing and grinding procedures at ALS was followed by initial FA fusion-FA finish analysis of 50 g pulp splits.

If the initial result met or exceeded a 2.5 g/t gold threshold, analysis of a second coarse reject split was carried out using a gravimetric finish. Composite metallurgical samples were created from coarse reject materials selected by Orex consultants, and these were submitted to SGS Lakefield for whole sample metallurgical testing. SGS was independent of Orex and is independent of the Company. A QA/QC program that included analysis of CRM, field duplicates, coarse reject duplicates, pulp split duplicates, and blank samples was carried out with respect to both the 2005 and 2008 programs, and results of these programs are presented in the report.

The 2010 to 2011 Osisko program was carried out under project supervision of Mr. J. Lafleur, P. Geo. and site supervision by consultant Mr. Bruce Mitchell, P. Geo. W.G. Shaw and Associates Ltd. provided most core logging, sample cutting, and field support staff for both programs and Mercator supplied one P. Geo. staff geologist to assist with the 2011 core logging. All of the NQ sized core was logged, photographed, sampled, bagged, tagged, and sealed at the Goldboro site by qualified personnel. Logging utilized Gemcom Gems™ Logger software, and project protocols included progressive, systematic, and secure off site backup of digital drilling, logging, and sampling data. At ALS, each sample was crushed to 70% < 2 mm, split to 250 g using a riffle splitter, pulverized to 85% at < 0.075 mm, and made into a 50 g sample of the pulp. The 50 g pulp was fire assayed with AAS finish (ALS codes Au-AA24 and Au-AA26). Samples exceeding the AAS threshold were re-assayed using a gravimetric finish (ALS code Au-GRA22). All samples containing visible gold were directly assigned for processing using the total metallic screen method with FA-AA or gravimetric finish.

A review of assessment reporting related to the various drilling programs carried out during the 1984 to 2005 period showed that, with the exception of the metallurgical and check sampling program carried out by Placer in 1995, no structured programs designed to systematically monitor QA/QC issues for drill core were in place. Orex drilling programs in 2005 and 2008 and Orex-Osisko programs in 2010 and 2011 were subject to rigorous QA/QC programs, with some procedural changes incorporated during the period.

During 2017 to the effective date of the current Mineral Resource Estimate, drill core samples were collected systematically down the hole based on the occurrence of visual alteration, mineralization, and quartz veining. Samples ranged in length from 0.3 m to 1.0 m depending on the nature and width of veining and mineralization samples while trying to best honour geological contacts. Samples were collected of quarter-sawn drill core and shipped to Eastern Analytical for analysis via standard 30 g FA with AA finish. Samples were also analyzed at Eastern Analytical via the total pulp metallics method (screen metallic) using the entire sample for samples assaying greater than 0.5 g/t gold, and all samples were submitted for 34-element ICP analysis.

## 11.1 Assay Sample Preparation and Analysis

The drill core is split with a mechanical drill saw, with one-half of the core placed in a pre-marked plastic sample bag and the other half returned to the core box. Sample bags are sealed with zip ties to ensure sample integrity (Figure 11-1) and shipped to Eastern Analytical for analysis through 30 g FA with an AA finish. Samples with assays higher than 0.5 g/t were analyzed by screen metallics, and all samples were submitted for 34-element ICP analysis.

*Figure 11-1: Mechanical drill saw with pre-marked sample bags*

## 11.2 Specific Gravity Sampling

A total of 3,360 SG measurements were provided from the Company using the weight in air versus the weight in water method (Archimedes) by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

Resulting in an average SG of 2.72 kg/m$^3$.

## 11.3 Quality Assurance/Quality Control Programs

QC measures were set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling, assaying, data management, and database integrity. Appropriate documentation of QC measures and regular analysis of QC data is essential as a safeguard for Project data and form the basis for the QA program implemented during exploration.

Analytical QC measures typically involve internal and external laboratory procedures implemented to monitor the precision and accuracy of the sample preparation and assay data. These measures are also important to identify potential sample sequencing errors and to monitor for contamination of samples.

Sampling and analytical QA/QC protocols typically involve taking duplicate samples and inserting QC samples (CRMs and blanks) to monitor the assay results' reliability throughout the drill program.

Umpire check assays are typically performed to evaluate the primary lab for bias and involve re-assaying a set proportion of sample rejects and pulps at a secondary umpire laboratory.

### 11.3.1 Historical Programs

#### 11.3.1.1 1984 to 2005

There was no QA/QC in place for the drill core.

#### 11.3.1.2 2005 to 2011

Field, coarse rejects, and pulp duplicate samples were routinely prepared and analyzed. Additionally, a blank, and CRM standard was routinely inserted in the sample stream. The duplicates were inserted every 50 samples while the blanks and CRM's were inserted every 25 samples. The Company does not have the QA/QC database for the 2005 to 2011 period.

#### 11.3.1.3 2017 to 2021

Diamond Drill Hole (DDH) Standards

The Company submitted five different CRMs as part of its QA/QC process for a total of 1,362 CRMs from 2017 to 2021 (Table 11-1). CDN-GS-1Z fell within the range of mean ± two standard deviations for gold (Figure 11-2). All other CRM's, including CDN-GS-1 M to CDN-GS-10E, show slight variability and had outliers/failures for the mean ± two standard deviations for gold (Figure 11-3 through Figure 11-6). No significant carryover is evident, and this does not impact the current assessment.

**Table 11-1: Deposit CRM Result Summary**

| Standard | Amount |
|---|---:|
| CDN-GS- 1M | 75 |
| CDN-GS- 1U | 258 |
| CDN-GS- 1Z | 344 |
| CDN-GS- 9D | 343 |
| CDN-GS- 10E | 342 |

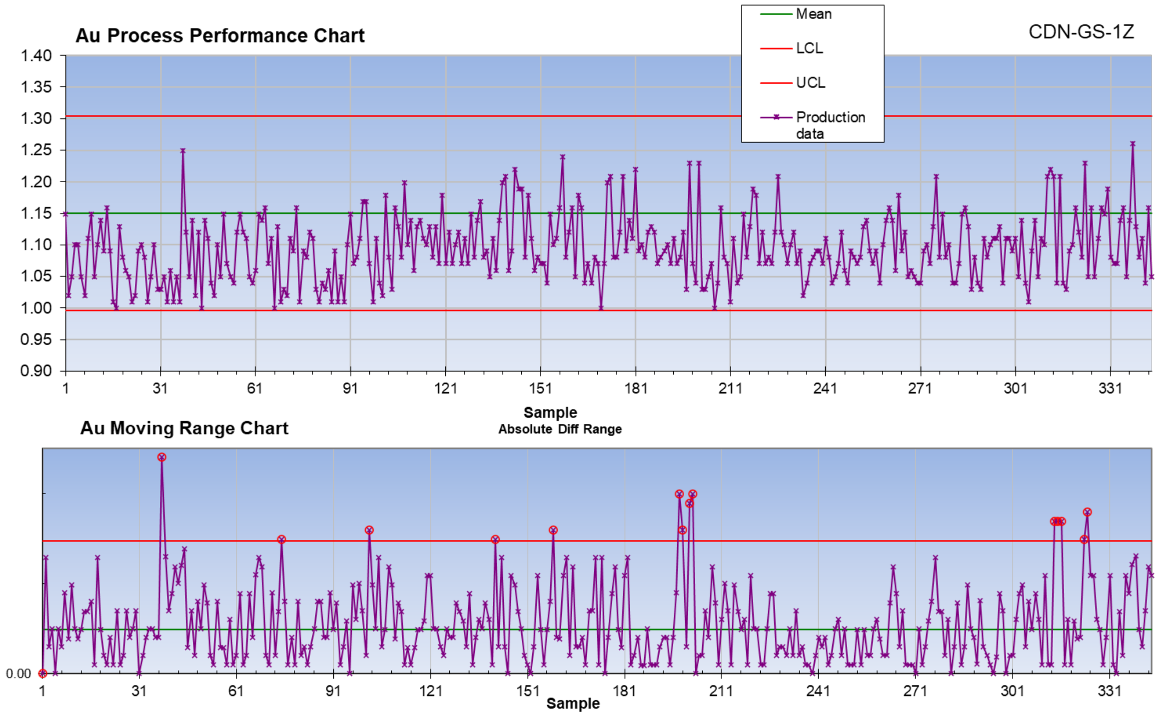



Figure 11-2: Standard CDN-GS-1Z gold (g/t)

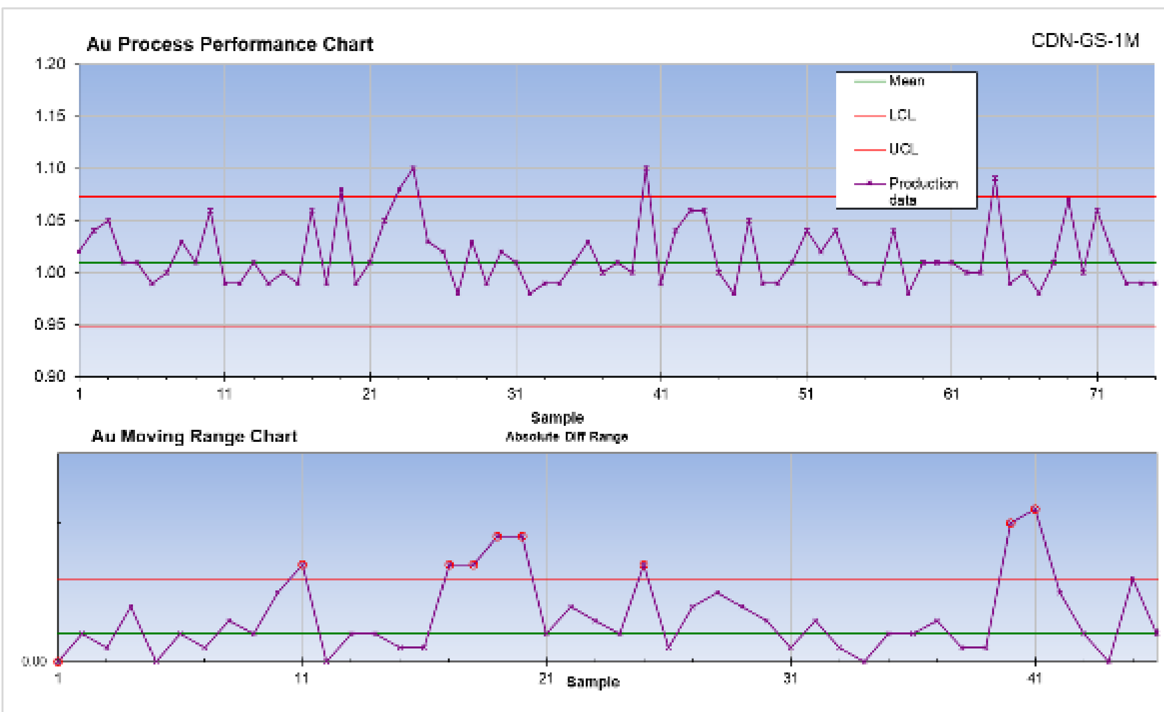



*Figure 11-3: Standard CDN-GS-1M gold (g/t)*

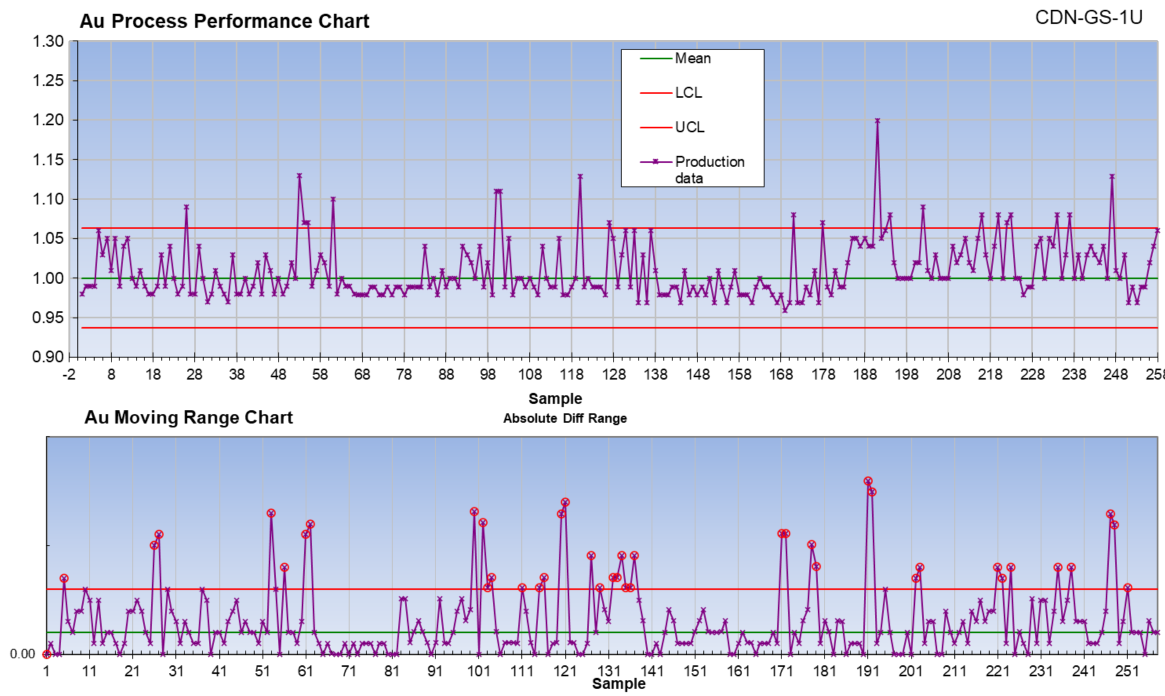



*Figure 11-4: Standard CDN-GS-1U gold (g/t)*

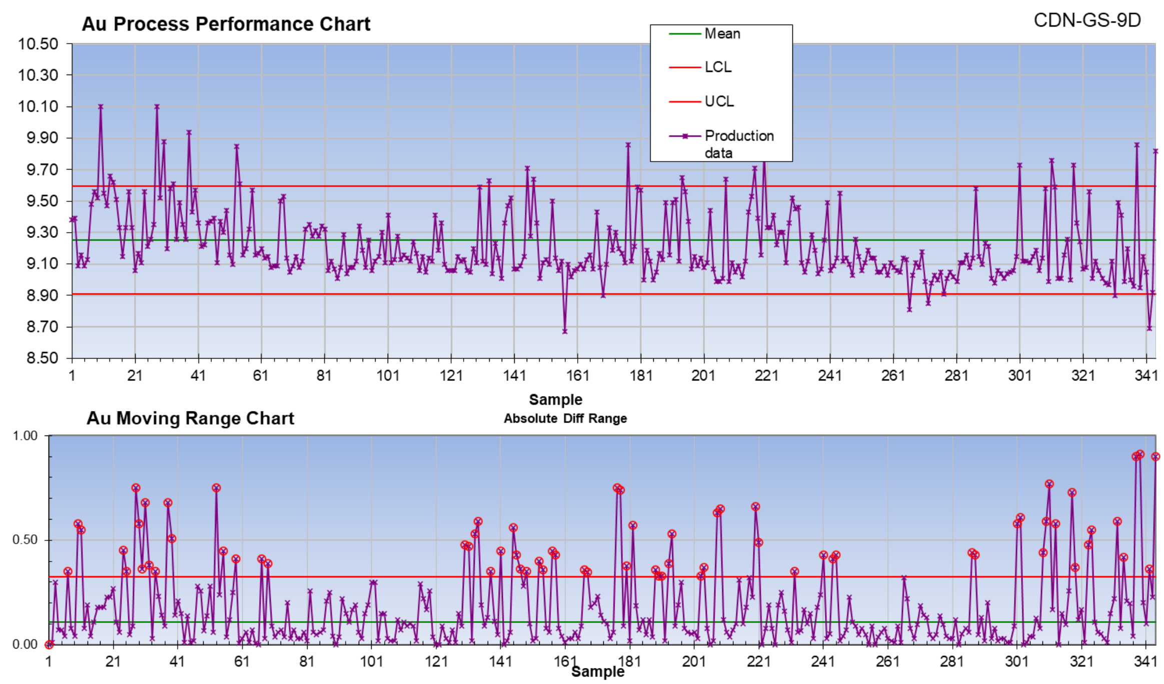



*Figure 11-5: Standard CDN-GS-9D gold (g/t)*

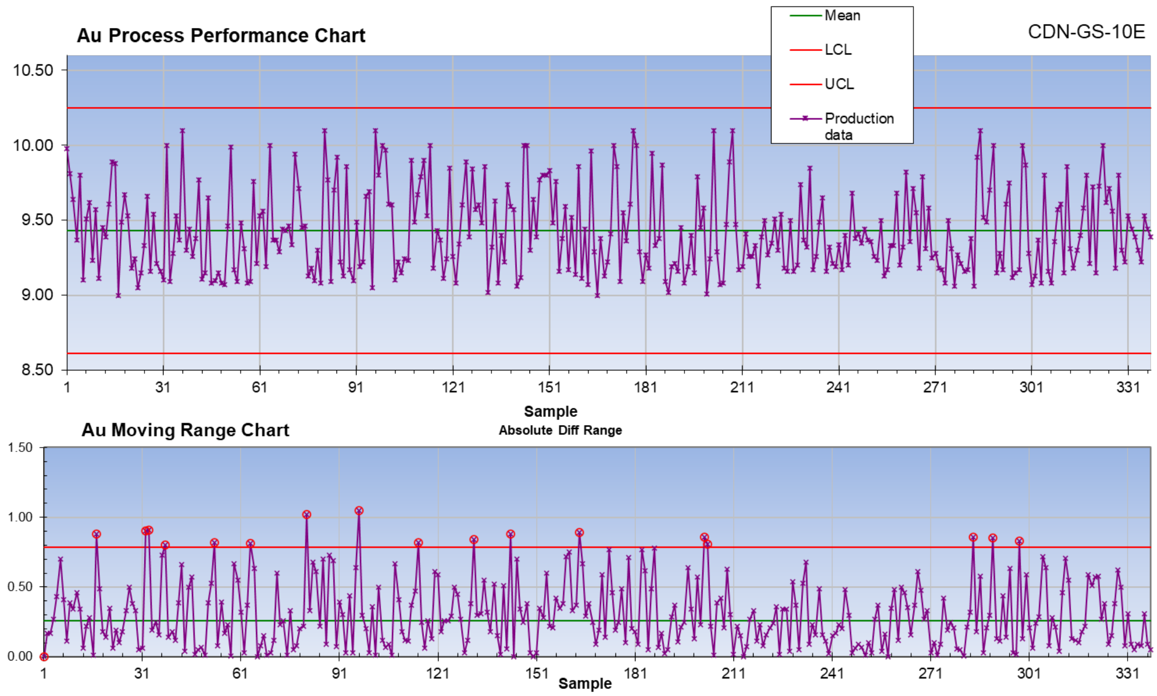



*Figure 11-6: Standard CDN-GS-10E gold (g/t)*

<u>DDH Blanks</u>

The Company submitted 1,379 blanks as part of its QA/QC process during the years 2017 to 2021 (Figure 11-7). No significant carryover is evident, and this does not impact the current assessment.

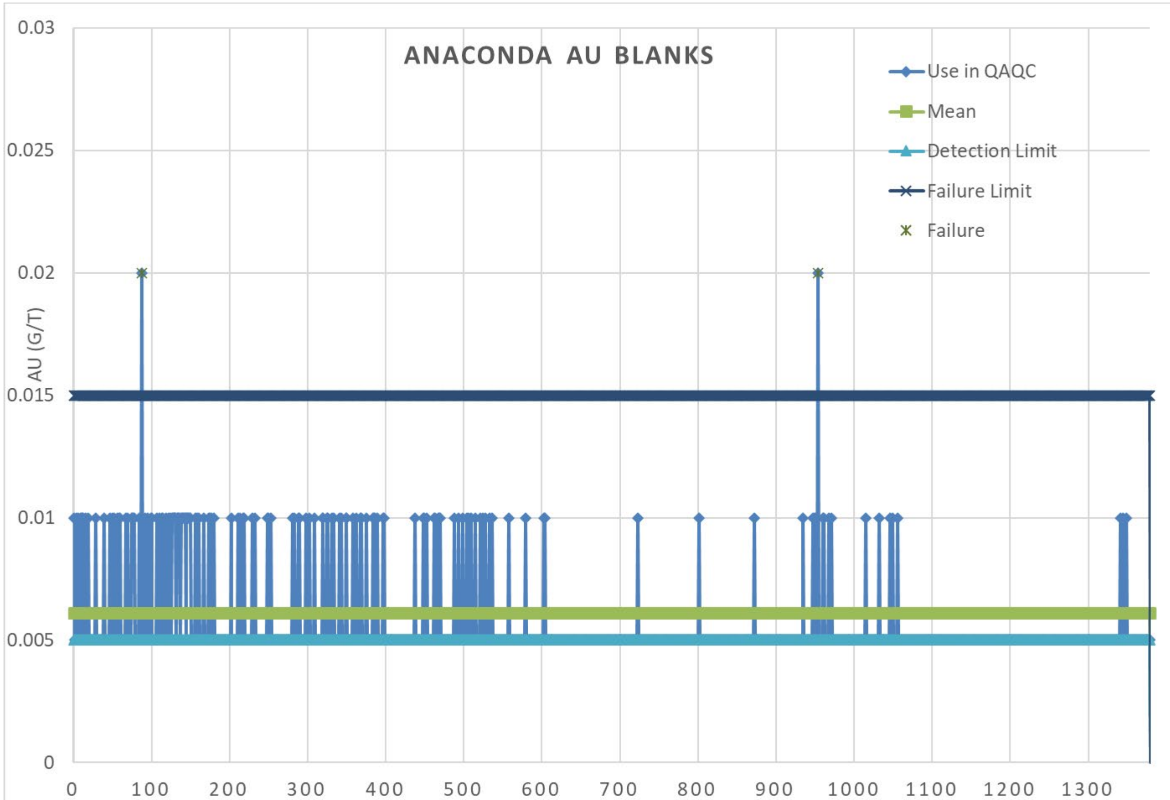



*Figure 11-7: Blank coarse blank gold (g/t)*

<u>DDH Laboratory Duplicates</u>

The lab submitted 1,299 duplicates as part of its QA/QC process between 2017 and 2020. The lab duplicates for gold show good agreement with few outliers (Figure 11-8).

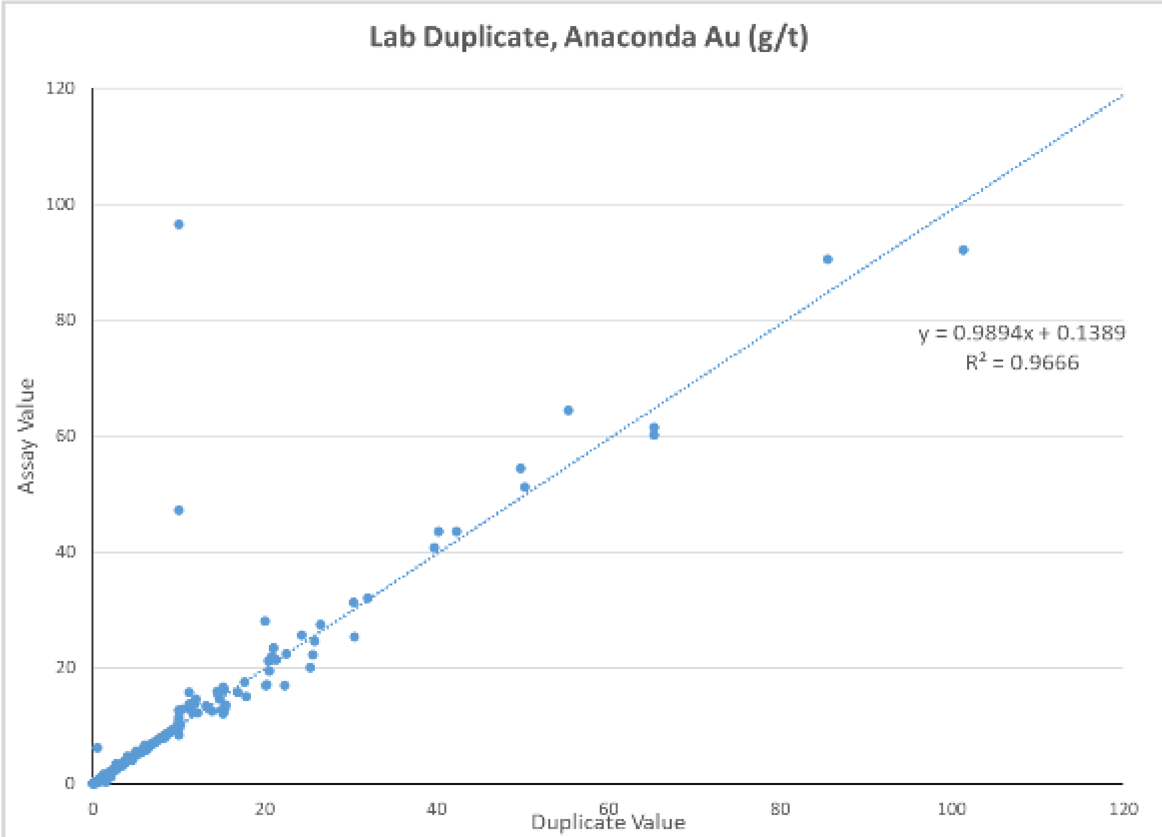



*Figure 11-8: Lab duplicates for gold (g/t)*

<u>Chip Standards</u>

The Company submitted two different CRM as part of its QA/QC process with a total of 83 CRMs from 2018 to 2019. CDN-GS-10E fell within the range of mean ± two standard deviations for gold (Figure 11-9). CDN-GS-1 U shows slight variability and has outliers/failures for the mean ± two standard deviations for gold (Figure 11-10). No significant carryover is evident, and this does not impact the current assessment.

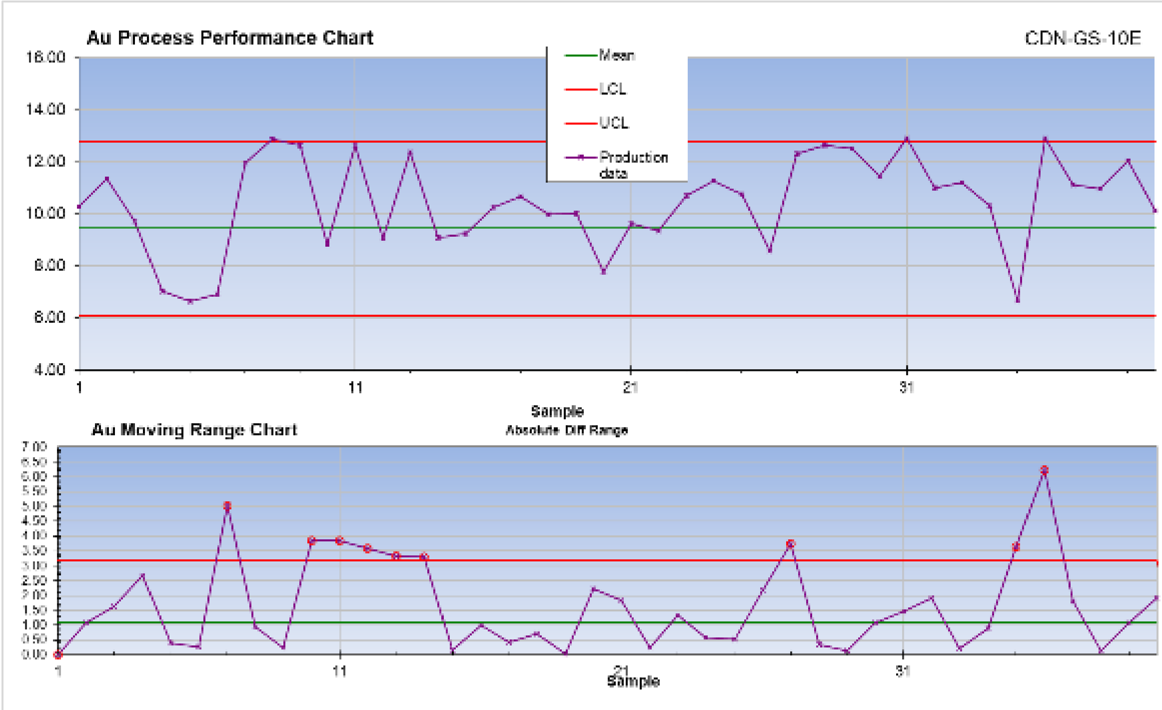



*Figure 11-9: Standard CDN-GS-10E gold (g/t)*

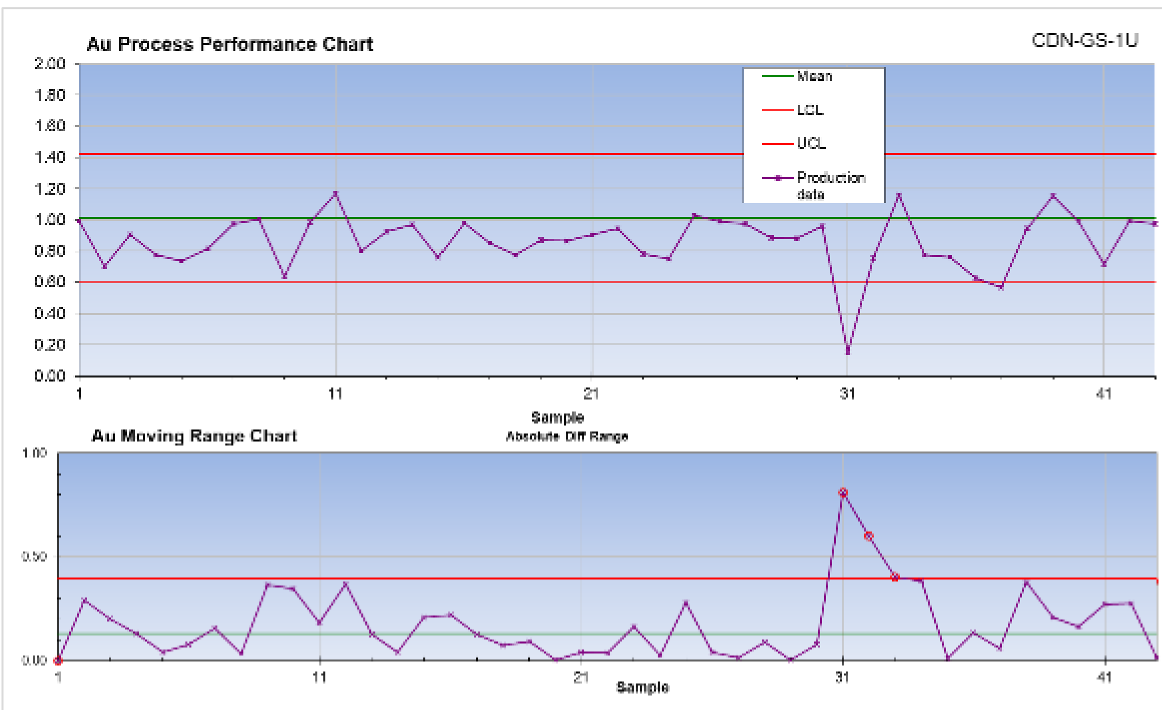



*Figure 11-10: Standard CDN-GS-1U gold (g/t)*

## Chip Blanks

The Company submitted 40 blanks as part of its QA/QC process (Figure 11-11). No significant carryover is evident, and this does not impact the current assessment.

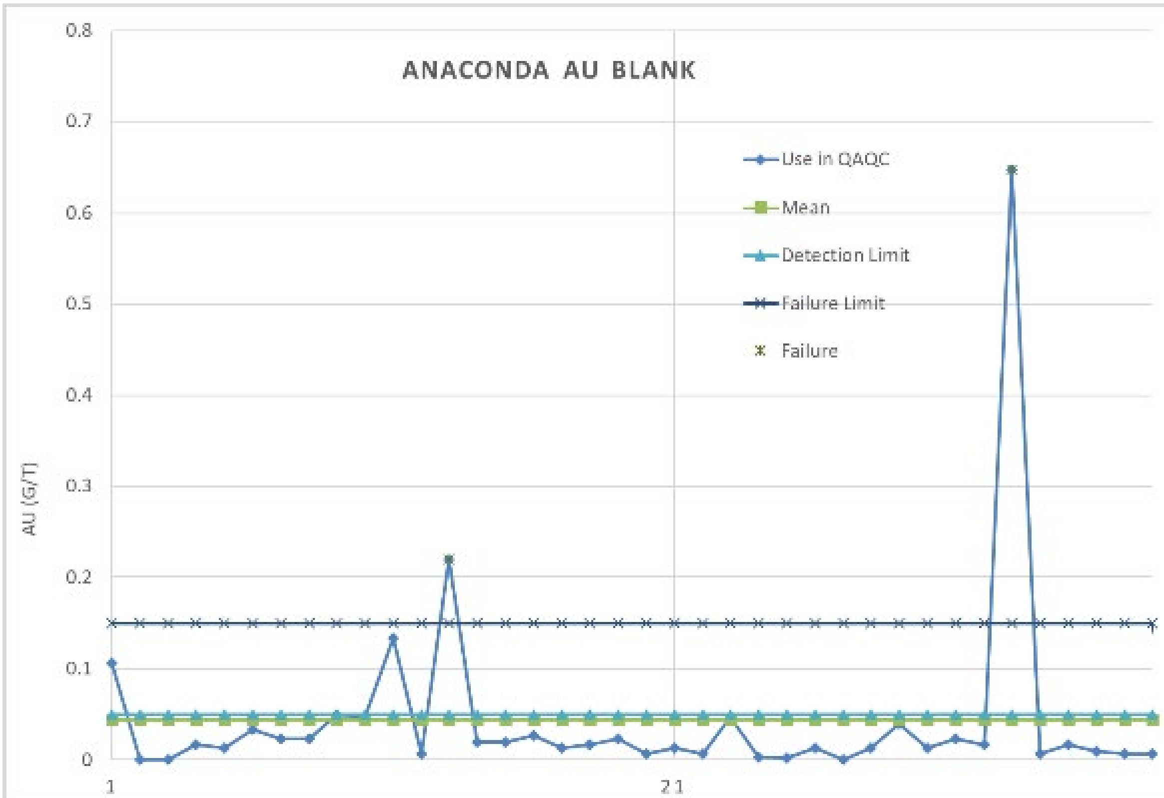



*Figure 11-11: Blank coarse blank gold (g/t)*

## Chip Laboratory Duplicates

The lab submitted 28 chip duplicates as part of its QA/QC process between 2018 to 2019. The lab duplicates for gold show good agreement with few outliers (Figure 11-12).

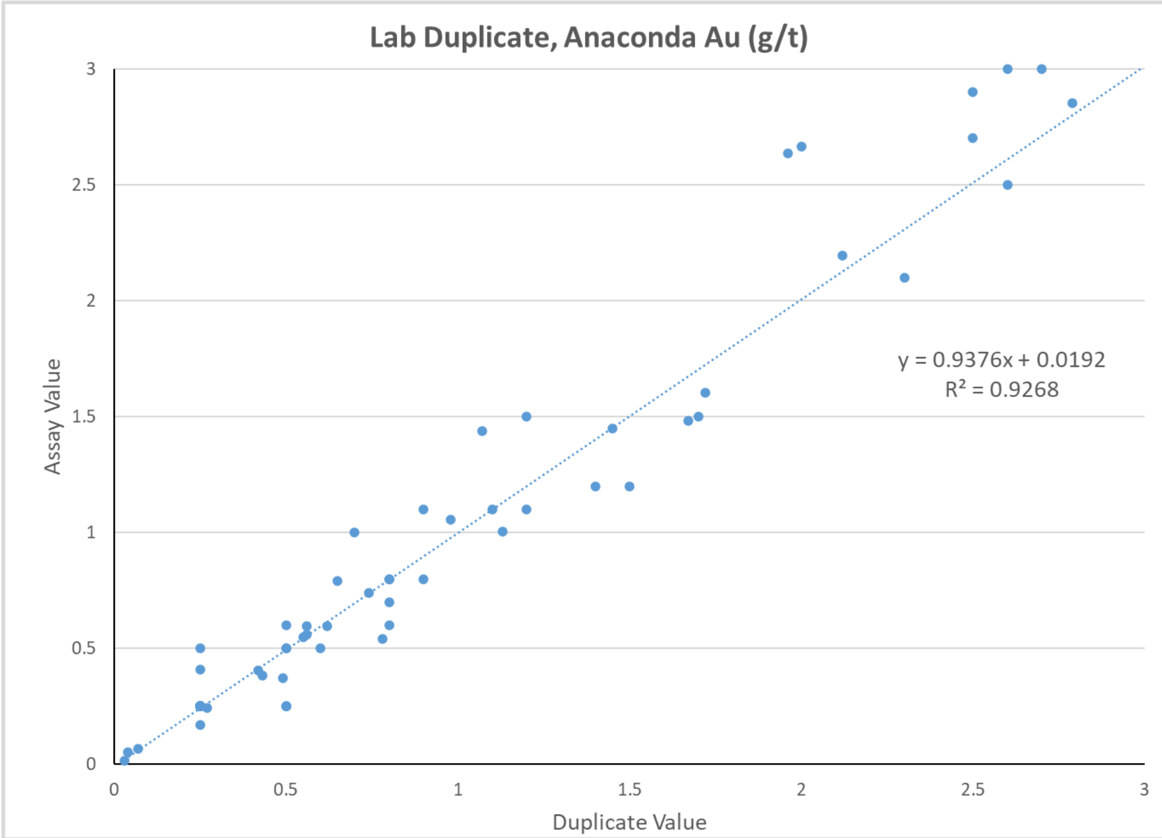



*Figure 11-12: Lab duplicates for gold (g/t)*

## 11.4 Security and Storage

Sample bags are sealed with zip ties to ensure sample integrity and securely shipped to Eastern Analytical for analysis. Drill core is stored in racks at the Company core storage facility at the Project site, with restricted access (Figure 11-13). Security of site operations, core, samples, and core storage are addressed on an ongoing basis by site staff.

*Figure 11-13: Company core storage facility*

## 11.5 Qualified Person's Opinion on the Adequacy of Sample Preparation, Security, and Analytical Procedures.

Nordmin has been supplied with all raw QA/QC data and has reviewed and completed an independent check of the results for all of the Project sampling programs. Between 1984 and 2005 there was no QA/QC in place; however, since that time extensive infill drilling and underground sampling by means of a Bulk Sample was completed and appropriate QA/QC applied. No apparent bias was determined between the historical programs in comparison with the programs since 2005. As such, the QP has determined that the use of the data that was collected between 1984 and 2005 is reliable and can be used within the Mineral Resource calculation.

It is Nordmin's opinion that the sample preparation, security, and analytical procedures used by all parties are consistent with standard industry practices and that the data is suitable for the 2021 Mineral Resource Estimate. Nordmin identified further recommendations to the Company to ensure the continuation of a robust QA/QC program but has noted that there are no material concerns with the geological or analytical procedures used or the quality of the resulting data.

# 12. DATA VERIFICATION

Nordmin completed several data validation checks throughout 2021. The verification process included a site visit to the Project January 18 to January 21, 2021, and on August 16 to August 17, 2021, by the QP to review surface geology, drill core geology, geological procedures, chain of custody of drill core, sample pulps, and for the collection of independent samples for metal verification. The data verification included:

- A survey spot check of drill collars.
- Historical mine workings.
- A spot check comparison of assays from the drill hole database against original assay records (lab certificates).
- A spot check of drill core lithologies recorded in the database versus the core located in the core storage shed.
- A review of the QA/QC performance of the drill programs.

Nordmin has also completed additional data analysis and validation, as outlined in Section 11.

## 12.1 Nordmin Site Visit 2021

A site visit to the Project was carried out between January 18 and January 21, 2021, and August 16 to August 17, 2021, by Glen Kuntz, P.Geo., QP for Mineral Resources. Mr. Kuntz was accompanied by the Company President and CEO, VP Exploration, and project geologist, who collectively have been involved with the Project for several years. Activities during the site visit included the:

- Review of the geological and geographical setting of the Deposit (BR, WG, and EG Domains).
- Review and inspection of the site geology, low, med, and high-grade mineralization, and structural controls with respect to gold distribution.
- Review of the drilling, logging, sampling, analytical and QA/QC procedures.
- Review of the chain of custody of samples from the field to the assay lab.
- Review of the drill logs, drill core, storage facilities, and independent assay verification on selected core samples.
- Confirmation of a variety of drill hole collar locations.
- Review of the structural measurements recorded within various drill logs and how they are utilized within the Company's geological/structural model.
- Validation of a portion of the drill hole database.
- Review of current "mineralized" waste stockpile.
- Location of proposed infrastructure (mill, waste management, waste dumps, and previously used underground infrastructure – portal).

The Company geologists completed the geological mapping, core logging, and sampling associated with the drill programs. Therefore, Nordmin used the Company's database to review the core logging procedures, the collection of samples, and the chain of custody associated with the drilling and sampling programs. The Company provided Nordmin with excerpts from the drill database (Geovia GEMS Logger) for the Project and electronic copies of the original logging and assay reports (Figure 12-1).

*Figure 12-1: Reviewing drill core using the Company's drill logging program (Geovia GEMS Logger)*

No significant issues were identified during the site visit. Two suggestions that should be incorporated into the Company's workflow include:

- Regular detailed drill audit.
- Insertion of a blank sample after all samples with noted visible gold or expected high-grade gold.
- Sampling of all material between the high-grade Belts within the proposed open pit areas.

The Company employs a rigorous QA/QC protocol, including the routine insertion of field duplicates, laboratory pulp duplicates, blanks, and certified reference standards. Nordmin was provided with an excerpt from the database for review.

The collection and use of the structural information was reliable and representative of the drilled structure features.

The geological data collection procedures and the chain of custody were found to be consistent with industry standards and following the Company's internal procedural documentation, and Nordmin was able to verify the quality of geological and sampling information and develop an interpretation of gold grade distributions appropriate for the Mineral Resource Estimate.

### 12.1.1 Field Collar Validation

The QP confirmed the various historic, 2020 and 2021 drill collar locations used within the Mineral Resource Estimate. Each drill collar that was drilled by the Company had a picket with a metal tag outlining the drill hole name, azimuth, and dip (Figure 12-2).

*Figure 12-2: Drill collars pickets outlining the drill hole name, azimuth, and dip*

Some of the drill holes had casing within the hole. Nordmin reviewed the hole collars within the database compared to a handheld GPS and determined that the collar locations are within acceptable error limits (Table 12-1, Table 12-2, Figure 12-3, and Figure 12-4).

**Table 12-1: Nordmin January Site Visit Check Collar MTM Coordinates Versus Drill Hole Database Coordinates**

| BHID | Original | | Check | |
|---|---|---|---|---|
| | Easting (NAD 83 MTM 4) | Northing (NAD 83 MTM 4) | Easting (NAD 83 MTM 4) | Northing (NAD 83 MTM 4) |
| BR-19-83 | 293750 | 5007047 | 293753.7 | 5007045 |
| BR-19-84 | 293704 | 5007047 | 293704.7 | 5007044 |
| BR-18-23 | 293901 | 5006870 | 293904 | 5006861 |
| BR-18-48 | 293899 | 5006731 | 293896.5 | 5006725 |
| OSK10-50 | 293030 | 5007223 | 293004.1 | 5007211 |
| OSK10-45 | 293058 | 5007223 | 293050.7 | 5007218 |
| OSK10-49 | 293028 | 5007193 | 293002.8 | 5007179 |
| OSK10-09 | 292906 | 5007226 | 292907.6 | 5007228 |
| BR-19-87 | 294348 | 5006742 | 294334.6 | 5006746 |

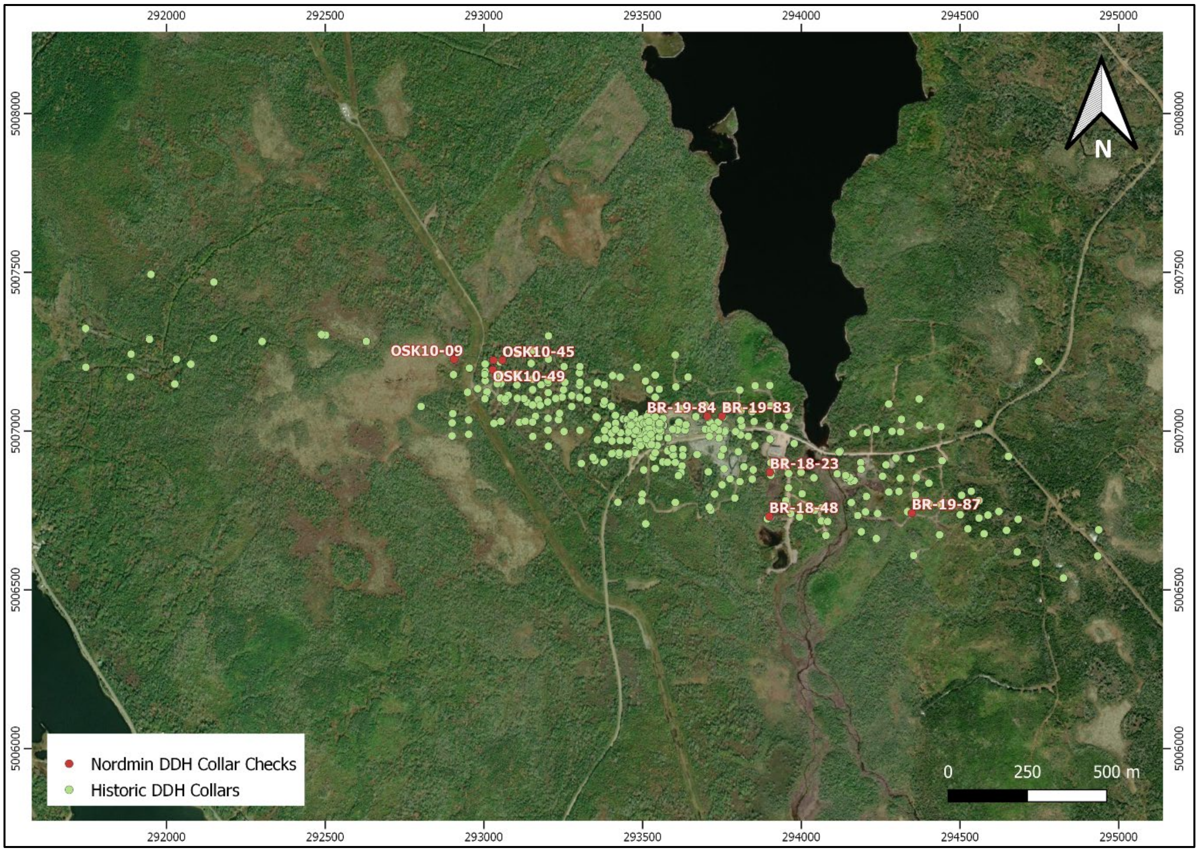



*Figure 12-3: Historic drill collars and Nordmin January 2021 site visit check collar locations*

**Table 12-2: Nordmin August Site Visit Check Collar MTM Coordinates Versus Database Collar Coordinates**

| BHID | Original | | Check | |
|---|---|---|---|---|
| | Easting (NAD 83 MTM 4) | Northing (NAD 83 MTM 4) | Easting (NAD 83 MTM 4) | Northing (NAD 83 MTM 4) |
| BR-21-256 | 293452.77 | 5006908.04 | 293449.32 | 5006905.77 |
| BR-21-262,263 | 293651 | 5007133.2 | 293650.36 | 5007130.33 |
| BR-21-257,258,259 | 293676 | 5006946.3 | 293674.63 | 5006944.71 |
| BR-21-287,288,289 | 294730.38 | 5006869.42 | 294729.58 | 5006867.94 |

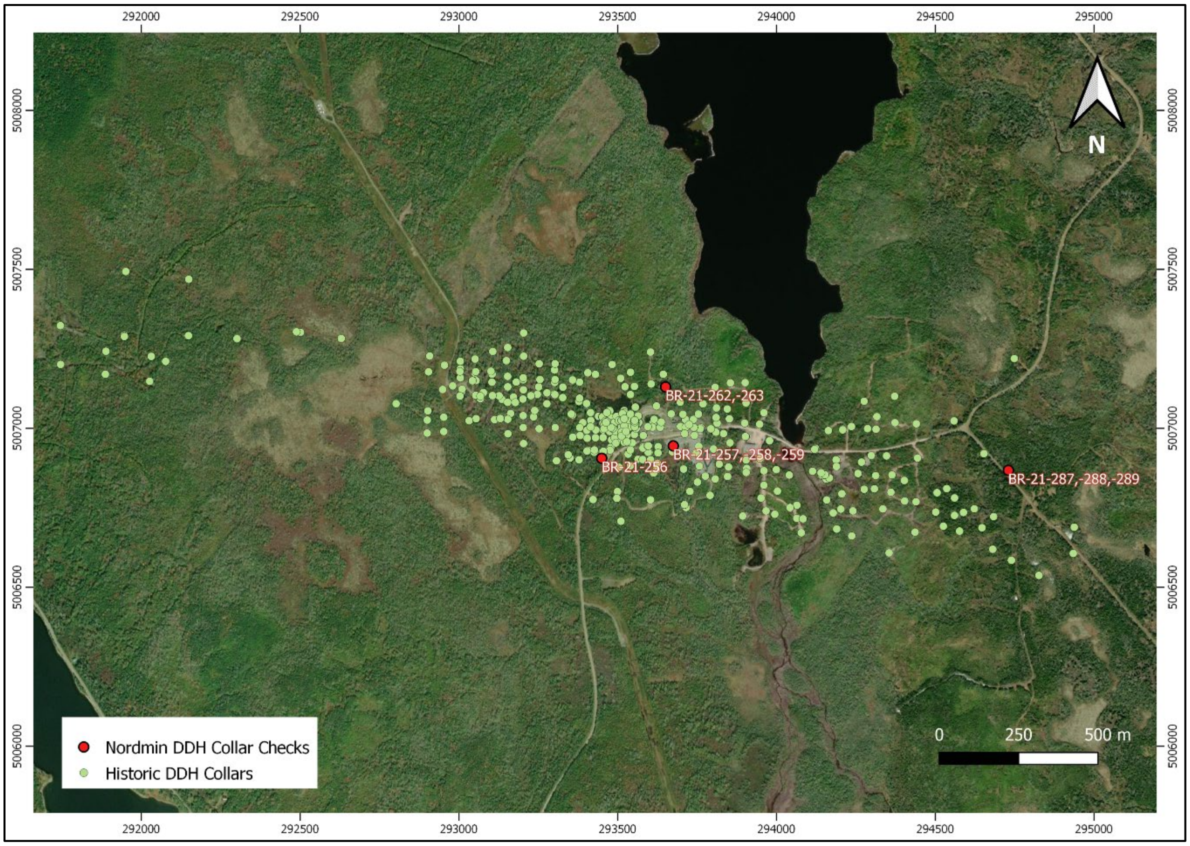



*Figure 12-4: Historic drill collars and Nordmin 2021 site visit check collar locations*

### 12.1.2 Core Logging, Sampling, and Storage Facilities

The Company drill holes were logged, photographed, and sampled on site at the core logging facility. All of the core is stored on site in one main location at the core storage facility (Figure 11-13 and Figure 12-5 through Figure 12-7). Coarse rejects that have not been consumed for geochemical analysis, as well as all pulps, are archived in the Company's secure storage facility in Springdale, NL.

During the visit to the core shack, Nordmin observed the cutting and sampling of the core that was selected by Nordmin to be quartered (Figure 11-1). The cutting and sampling was consistent with the Company's surface sampling procedures.

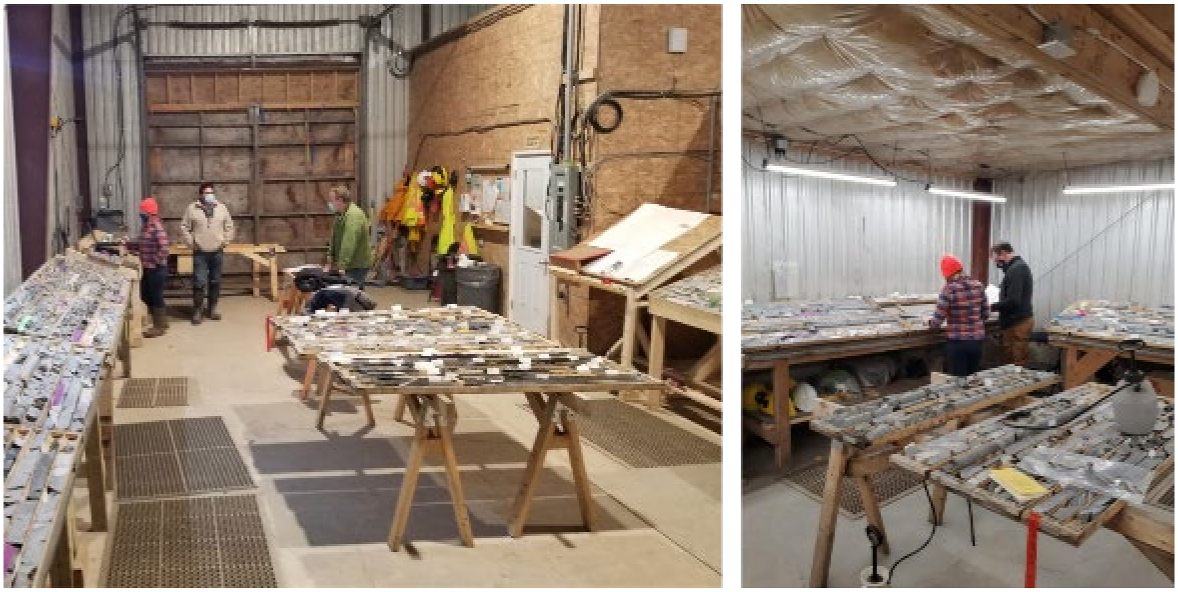


*Figure 12-5: Company logging and sampling facility*

*Figure 12-6: Company core cutting facility*

*Figure 12-7: Company core logging and storage facility*

The Company did inherit core from the drill programs of previous operators. The Company is actively preserving the historical core by re-boxing and moving the core into its main on site core storage facility (Figure 12-8).

*Figure 12-8: Historical Company core being reboxed and moved into the current storage facility*

### 12.1.3 Independent Sampling

The QP selected intervals from multiple Company drill holes for a total of 185 verification samples (Table 12-3). The samples were marked with flagging tape, and the core was quarter cut to represent the same sample length (Figure 12-9) and compared to pulps from previous assays.

**Table 12-3: Drill Program Holes Selected for Verification Sampling During the January 2021 Site Visit**

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-105 | 133.6 | 134.1 | 384865 | 384865 |
| BR-20-107 | 95.0 | 96.0 | 385497 | 385497 |
| BR-20-108 | 97.0 | 98.0 | 385536 | 385536 |
| BR-20-108 | 155.3 | 155.9 | 385587 | 385587 |
| BR-20-110 | 190.0 | 191.0 | 385697 | 385697 |
| BR-20-112 | 267.0 | 268.0 | 385819 | 385819 |
| BR-20-112 | 278.2 | 279.0 | 385832 | 385832 |
| BR-20-114 | 79.0 | 80.0 | 385843 | 385843 |
| BR-20-114 | 297.3 | 298.2 | 385941 | 385941 |
| BR-20-114 | 356.0 | 357.0 | 385993 | 385993 |
| BR-20-120 | 67.0 | 68.0 | 386238 | 386238 |
| BR-20-120 | 234.0 | 235.0 | 386361 | 386361 |
| BR-20-120 | 241.0 | 242.0 | 386368 | 386368 |
| BR-20-120 | 265.2 | 266.2 | 386381 | 386381 |
| BR-20-120 | 270.2 | 271.2 | 386386 | 386386 |
| BR-20-120 | 277.0 | 278.0 | 386393 | 386393 |
| BR-20-124 | 229.0 | 230.0 | 386569 | 386569 |
| BR-20-124 | 291.8 | 292.3 | 386613 | 386613 |
| BR-20-132 | 17.0 | 17.5 | 386637 | 386637 |
| BR-20-132 | 22.7 | 23.2 | 386644 | 386644 |
| BR-20-132 | 23.2 | 24.2 | 386645 | 386645 |
| BR-20-132 | 24.2 | 24.7 | 386646 | 386646 |
| BR-20-132 | 24.7 | 25.5 | 386647 | 386647 |
| BR-20-132 | 72.0 | 72.5 | 386707 | 386707 |
| BR-20-134 | 11.0 | 12.0 | 386767 | 386767 |
| BR-20-134 | 12.0 | 13.0 | 386768 | 386768 |
| BR-20-134 | 37.3 | 37.8 | 386801 | 386801 |
| BR-20-134 | 53.7 | 54.6 | 386823 | 386823 |
| BR-20-134 | 59.0 | 60.0 | 386831 | 386831 |
| BR-20-134 | 69.5 | 70.0 | 386833 | 386833 |

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-134 | 121.0 | 122.0 | 386894 | 386894 |
| BR-20-109 | 27.4 | 28.4 | 387503 | 387503 |
| BR-20-109 | 56.0 | 57.0 | 387511 | 387511 |
| BR-20-109 | 116.0 | 117.0 | 387526 | 387526 |
| BR-20-109 | 198.0 | 199.0 | 387580 | 387580 |
| BR-20-109 | 213.0 | 214.0 | 387593 | 387593 |
| BR-20-109 | 217.0 | 218.0 | 387597 | 387597 |
| BR-20-113 | 196.0 | 197.0 | 387719 | 387719 |
| BR-20-113 | 205.0 | 206.0 | 387730 | 387730 |
| BR-20-115 | 183.0 | 184.0 | 387817 | 387817 |
| BR-20-115 | 209.0 | 210.0 | 387837 | 387837 |
| BR-20-117 | 154.0 | 155.0 | 387932 | 387932 |
| BR-20-117 | 172.0 | 173.0 | 387941 | 387941 |
| BR-20-119 | 197.0 | 198.0 | 388056 | 388056 |
| BR-20-119 | 229.0 | 230.0 | 388068 | 388068 |
| BR-20-119 | 270.0 | 271.0 | 388110 | 388110 |
| BR-20-119 | 273.0 | 274.0 | 388113 | 388113 |
| BR-20-119 | 286.0 | 287.0 | 388128 | 388128 |
| BR-20-119 | 295.0 | 296.0 | 388137 | 388137 |
| BR-20-119 | 305.0 | 306.0 | 388147 | 388147 |
| BR-20-122 | 183.0 | 184.0 | 388402 | 388402 |
| BR-20-123 | 28.0 | 29.0 | 388459 | 388459 |
| BR-20-125 | 17.0 | 18.0 | 388611 | 388611 |
| BR-20-127 | 39.0 | 40.0 | 388680 | 388680 |
| BR-20-127 | 67.4 | 67.9 | 388708 | 388708 |
| BR-20-130 | 97.5 | 98.5 | 388943 | 388943 |
| BR-20-130 | 100.5 | 101.2 | 388946 | 388946 |
| BR-20-131 | 14.0 | 15.0 | 388964 | 388964 |
| BR-20-131 | 31.0 | 31.5 | 388979 | 388979 |
| BR-20-131 | 43.0 | 43.7 | 388998 | 388998 |
| BR-20-131 | 71.0 | 72.0 | 389039 | 389039 |
| BR-20-131 | 72.0 | 72.7 | 389040 | 389040 |
| BR-20-131 | 93.2 | 94.0 | 389077 | 389077 |
| BR-20-131 | 111.5 | 112.0 | 389096 | 389096 |

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-131 | 131.5 | 132.0 | 389123 | 389123 |
| BR-20-131 | 142.0 | 143.0 | 389137 | 389137 |
| BR-20-131 | 143.5 | 144.0 | 389139 | 389139 |
| BR-20-131 | 144.5 | 145.0 | 389141 | 389141 |
| BR-20-135 | 18.3 | 18.8 | 389153 | 389153 |
| BR-20-135 | 26.5 | 27.5 | 389160 | 389160 |
| BR-20-135 | 29.5 | 30.5 | 389163 | 389163 |
| BR-20-135 | 43.0 | 43.5 | 389185 | 389185 |
| BR-20-135 | 43.5 | 44.2 | 389186 | 389186 |
| BR-20-135 | 97.3 | 97.9 | 389240 | 389240 |
| BR-20-135 | 111.0 | 111.6 | 389257 | 389257 |
| BR-20-135 | 111.6 | 112.6 | 389258 | 389258 |
| BR-20-135 | 115.0 | 116.0 | 389262 | 389262 |
| BR-20-140 | 128.6 | 129.3 | 389457 | 389457 |
| BR-20-136 | 90.0 | 90.5 | 495083 | 495083 |
| BR-20-139 | 122.7 | 123.4 | 495185 | 495185 |
| BR-20-139 | 127.0 | 127.7 | 495190 | 495190 |
| BR-20-146 | 45.9 | 46.4 | 495287 | 495287 |
| BR-20-146 | 118.0 | 118.5 | 495320 | 495320 |
| BR-20-151 | 38.0 | 39.0 | 495391 | 495391 |
| BR-20-151 | 114.4 | 115.3 | 495437 | 495437 |
| BR-20-155 | 77.2 | 78.2 | 495531 | 495531 |
| BR-20-158 | 91.4 | 91.9 | 495597 | 495597 |
| BR-20-158 | 103.1 | 103.6 | 495620 | 495620 |
| BR-20-160 | 47.3 | 48.0 | 495672 | 495672 |
| BR-20-160 | 118.2 | 118.7 | 495731 | 495731 |
| BR-20-162 | 26.1 | 26.9 | 495805 | 495805 |
| BR-20-162 | 79.5 | 80.0 | 495860 | 495860 |
| BR-20-162 | 115.0 | 115.5 | 495909 | 495909 |
| BR-20-165 | 28.0 | 29.0 | 495990 | 495990 |
| BR-20-165 | 34.5 | 35.2 | 495997 | 495997 |
| BR-20-165 | 51.3 | 51.8 | 496022 | 496022 |
| BR-20-165 | 51.8 | 52.4 | 496023 | 496023 |
| BR-20-168 | 28.5 | 29.0 | 496067 | 496067 |

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-170 | 72.5 | 73.0 | 496161 | 496161 |
| BR-20-174 | 110.6 | 111.3 | 496331 | 496331 |
| BR-20-176 | 38.0 | 39.0 | 496370 | 496370 |
| BR-20-176 | 45.2 | 46.0 | 496379 | 496379 |
| BR-20-178 | 54.5 | 55.0 | 496413 | 496413 |
| BR-20-178 | 60.7 | 61.7 | 496419 | 496419 |
| BR-20-183 | 13.0 | 14.0 | 496454 | 496454 |
| BR-20-184 | 29.7 | 30.7 | 496495 | 496495 |
| BR-20-185 | 45.0 | 45.5 | 496557 | 496557 |
| BR-20-185 | 54.0 | 54.5 | 496568 | 496568 |
| BR-20-185 | 83.8 | 84.7 | 496602 | 496602 |
| BR-20-187 | 45.0 | 46.0 | 496638 | 496638 |
| BR-20-187 | 49.0 | 50.0 | 496643 | 496643 |
| BR-20-179 | 8.0 | 8.5 | 496656 | 496656 |
| BR-20-179 | 23.8 | 24.3 | 496677 | 496677 |
| BR-20-179 | 30.0 | 30.5 | 496687 | 496687 |
| BR-20-191 | 83.7 | 84.2 | 496838 | 496838 |
| BR-20-191 | 106.0 | 107.0 | 496860 | 496860 |
| BR-20-191 | 111.6 | 112.1 | 496866 | 496866 |
| BR-20-191 | 134.6 | 135.1 | 496883 | 496883 |
| BR-20-196 | 69.5 | 70.2 | 496952 | 496952 |
| BR-20-196 | 87.2 | 88.1 | 496973 | 496973 |
| BR-20-196 | 95.6 | 96.6 | 496984 | 496984 |
| BR-20-207 | 202.2 | 203.0 | 497220 | 497220 |
| BR-20-207 | 230.0 | 230.5 | 497261 | 497261 |
| BR-20-207 | 269.0 | 269.5 | 497316 | 497316 |
| BR-20-207 | 294.0 | 294.5 | 497339 | 497339 |
| BR-20-207 | 301.0 | 301.5 | 497354 | 497354 |
| BR-20-213 | 183.0 | 183.5 | 497465 | 497465 |
| BR-20-138 | 115.9 | 116.4 | 498002 | 498002 |
| BR-20-138 | 124.1 | 125.1 | 498009 | 498009 |
| BR-20-141 | 91.7 | 92.2 | 498068 | 498068 |
| BR-20-143 | 70.1 | 70.6 | 498142 | 498142 |
| BR-20-143 | 155.6 | 156.1 | 498203 | 498203 |

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-147 | 106.5 | 107.0 | 498286 | 498286 |
| BR-20-147 | 191.0 | 192.0 | 498320 | 498320 |
| BR-20-147 | 198.0 | 199.0 | 498329 | 498329 |
| BR-20-150 | 32.8 | 33.5 | 498503 | 498503 |
| BR-20-161 | 95.5 | 96.0 | 498617 | 498617 |
| BR-20-161 | 143.5 | 144.0 | 498667 | 498667 |
| BR-20-164 | 114.0 | 115.0 | 498726 | 498726 |
| BR-20-166 | 24.6 | 25.6 | 498741 | 498741 |
| BR-20-173 | 87.2 | 87.7 | 498806 | 498806 |
| BR-20-173 | 87.7 | 88.2 | 498807 | 498807 |
| BR-20-177 | 91.8 | 92.3 | 498931 | 498931 |
| BR-20-177 | 119.3 | 119.8 | 498946 | 498946 |
| BR-20-177 | 188.0 | 188.5 | 498993 | 498993 |
| BR-20-197 | 136.9 | 137.4 | 499613 | 499613 |
| BR-20-200 | 81.0 | 82.0 | 499662 | 499662 |
| BR-20-200 | 100.0 | 101.0 | 499681 | 499681 |
| BR-20-204 | 49.0 | 50.0 | 499813 | 499813 |
| BR-20-210 | 107.0 | 108.0 | 499988 | 499988 |
| BR-20-142 | 56.3 | 57.3 | 501006 | 501006 |
| BR-20-142 | 107.5 | 108.0 | 501035 | 501035 |
| BR-20-142 | 189.9 | 190.9 | 501104 | 501104 |
| BR-20-142 | 191.5 | 192.1 | 501106 | 501106 |
| BR-20-142 | 193.7 | 194.2 | 501110 | 501110 |
| BR-20-142 | 195.7 | 196.2 | 501114 | 501114 |
| BR-20-142 | 210.8 | 211.8 | 501134 | 501134 |
| BR-20-142 | 221.5 | 222.0 | 501154 | 501154 |
| BR-20-148 | 172.3 | 172.8 | 501435 | 501435 |
| BR-20-148 | 190.5 | 191.0 | 501456 | 501456 |
| BR-20-156 | 42.0 | 42.5 | 501561 | 501561 |
| BR-20-156 | 64.3 | 65.0 | 501579 | 501579 |
| BR-20-156 | 65.0 | 65.5 | 501580 | 501580 |
| BR-20-159 | 202.2 | 202.7 | 501732 | 501732 |
| BR-20-175 | 63.3 | 63.8 | 501789 | 501789 |
| BR-20-175 | 92.0 | 92.5 | 501813 | 501813 |

| Hole ID | From (m) | To (m) | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-175 | 162.5 | 163.0 | 501873 | 501873 |
| BR-20-180 | 18.0 | 18.5 | 501965 | 501965 |
| BR-20-180 | 22.5 | 23.0 | 501976 | 501976 |
| BR-20-180 | 29.0 | 29.5 | 501987 | 501987 |
| BR-20-182 | 70.1 | 70.6 | 502042 | 502042 |
| BR-20-182 | 76.0 | 76.5 | 502047 | 502047 |
| BR-20-182 | 77.5 | 78.0 | 502052 | 502052 |
| BR-20-194 | 113.0 | 114.0 | 502286 | 502286 |
| BR-20-199 | 51.5 | 52.1 | 502306 | 502306 |
| BR-20-201 | 68.5 | 69.0 | 502361 | 502361 |
| BR-20-201 | 99.5 | 100.5 | 502390 | 502390 |
| BR-20-201 | 110.5 | 111.1 | 502398 | 502398 |
| BR-20-205 | 26.8 | 27.3 | 502432 | 502432 |
| BR-20-205 | 32.0 | 33.0 | 502438 | 502438 |
| BR-20-205 | 37.0 | 37.5 | 502443 | 502443 |
| BR-20-205 | 37.5 | 38.0 | 502444 | 502444 |
| BR-20-205 | 63.9 | 64.4 | 502465 | 502465 |
| BR-20-208 | 39.0 | 39.5 | 502515 | 502515 |
| BR-20-209 | 30.4 | 31.0 | 502553 | 502553 |

**Table 12-4 Drill Program Holes Selected for Verification Sampling During the August 2021 Site Visit**

| Hole ID | From | To | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-142 | 210.8 | 211.8 | 501134 | 501134 |
| BR-20-142 | 211.8 | 212.3 | 501135 | 501135 |
| BR-20-142 | 212.3 | 212.8 | 501136 | 501136 |
| BR-20-142 | 212.8 | 213.3 | 501137 | 501137 |
| BR-20-142 | 213.3 | 214.2 | 501138 | 501138 |
| BR-20-142 | 214.2 | 214.7 | 501139 | 501139 |
| BR-20-142 | 214.7 | 215.7 | 501140 | 501140 |
| BR-20-142 | 215.7 | 216.3 | 501141 | 501141 |
| BR-20-142 | 216.3 | 216.8 | 501142 | 501142 |
| BR-20-142 | 216.8 | 217.3 | 501143 | 501143 |
| BR-20-142 | 217.3 | 218 | 501144 | 501144 |
| BR-20-142 | 218 | 218.5 | 501145 | 501145 |
| BR-20-142 | 218.5 | 219 | 501146 | 501146 |

| Hole ID | From | To | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-20-142 | 219 | 219.5 | 501147 | 501147 |
| BR-20-142 | 219.5 | 220 | 501148 | 501148 |
| BR-20-142 | 220 | 220.5 | 501151 | 501151 |
| BR-20-142 | 220.5 | 221 | 501152 | 501152 |
| BR-20-142 | 221 | 221.5 | 501153 | 501153 |
| BR-20-142 | 221.5 | 222 | 501154 | 501154 |
| BR-20-142 | 222 | 222.6 | 501155 | 501155 |
| BR-20-142 | 222.6 | 223.6 | 501156 | 501156 |
| BR-20-114 | 349 | 350 | 385986 | 385986 |
| BR-20-114 | 350 | 351 | 385987 | 385987 |
| BR-20-114 | 351 | 352 | 385988 | 385988 |
| BR-20-114 | 352 | 353 | 385989 | 385989 |
| BR-20-114 | 353 | 354 | 385990 | 385990 |
| BR-20-114 | 354 | 355 | 385991 | 385991 |
| BR-20-114 | 355 | 356 | 385992 | 385992 |
| BR-20-114 | 356 | 357 | 385993 | 385993 |
| BR-20-114 | 357 | 358 | 385994 | 385994 |
| BR-20-114 | 358 | 359 | 385995 | 385995 |
| BR-21-246 | 48.5 | 49 | 546557 | 546557 |
| BR-21-246 | 49 | 49.5 | 546558 | 546558 |
| BR-21-246 | 49.5 | 50 | 546559 | 546559 |
| BR-21-246 | 50 | 50.5 | 546560 | 546560 |
| BR-21-246 | 50.5 | 51 | 546561 | 546561 |
| BR-21-246 | 51 | 52 | 546562 | 546562 |
| BR-21-246 | 52 | 53 | 546563 | 546563 |
| BR-21-246 | 53 | 53.5 | 546564 | 546564 |
| BR-21-246 | 53.5 | 54 | 546565 | 546565 |
| BR-21-246 | 54 | 54.5 | 546566 | 546566 |
| BR-21-246 | 54.5 | 55 | 546567 | 546567 |
| BR-21-246 | 55 | 55.5 | 546568 | 546568 |
| BR-21-246 | 55.5 | 56 | 546569 | 546569 |
| BR-21-246 | 56 | 56.5 | 546570 | 546570 |
| BR-21-246 | 56.5 | 57 | 546571 | 546571 |
| BR-21-246 | 57 | 57.5 | 546572 | 546572 |
| BR-21-246 | 57.5 | 58 | 546573 | 546573 |
| BR-21-246 | 58 | 58.5 | 546576 | 546576 |

| Hole ID | From | To | Previous Sample ID | Nordmin Check Sample ID |
|---|---|---|---|---|
| BR-21-246 | 58.5 | 59 | 546577 | 546577 |
| BR-21-246 | 59 | 59.5 | 546578 | 546578 |
| BR-21-246 | 59.5 | 60 | 546579 | 546579 |
| BR-20-221 | 43 | 44 | 502859 | 502859 |
| BR-20-221 | 44 | 44.5 | 502860 | 502860 |
| BR-20-221 | 44.5 | 45 | 502861 | 502861 |
| BR-20-221 | 45 | 45.5 | 502862 | 502862 |
| BR-20-221 | 45.5 | 46 | 502863 | 502863 |
| BR-20-221 | 46 | 46.5 | 502864 | 502864 |
| BR-20-221 | 46.5 | 47 | 502865 | 502865 |
| BR-20-221 | 47 | 47.5 | 502866 | 502866 |
| BR-20-221 | 47.5 | 48 | 502867 | 502867 |
| BR-20-221 | 48 | 48.5 | 502868 | 502868 |
| BR-20-221 | 48.5 | 49 | 502869 | 502869 |

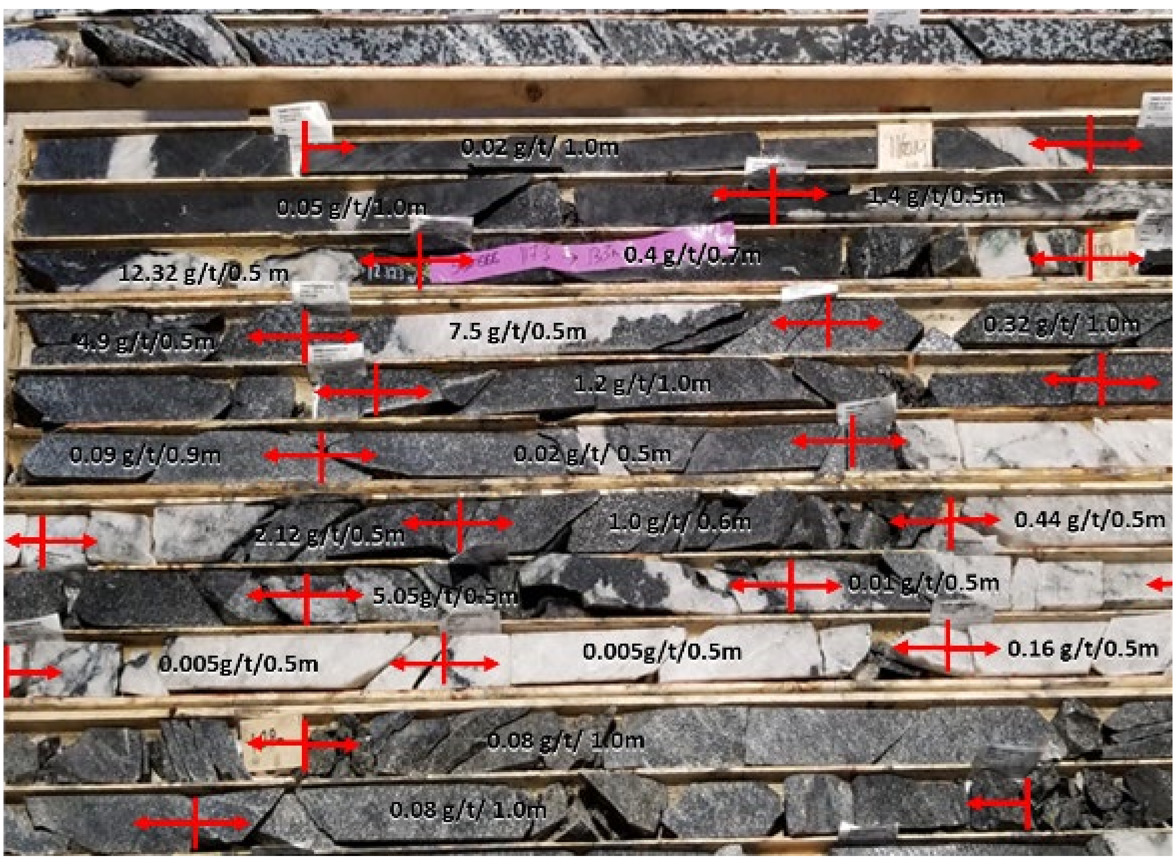



*Figure 12-9: DDH BR-20-134 between 115.3 m and 130.0 m outlining gold grades over an interval of approximately 14.7 m*

Multiple intervals of samples were chosen to review assays results over larger areas and to test local variability. Since the intent of the proposed mining method is open pit, many intervals were checked with a goal to emulate multiple benches that combine both the High-Grade Mineralized "Belts" and Low-Grade Mineralized Zones that surrounds the Belt. The samples that were chosen were over a range of low, medium, and higher-grade material (Figure 12-10).


*Figure 12-10: QP selection of core samples for review of assays results over larger areas and to test local variability*

The QP assay results were compared to the Company database and were summarized in scatter plots for gold (Table 12-5, Table 12-6, Figure 12-11 and Figure 12-12). Despite some significant sample variance in a few samples, most assays compared within reasonable tolerances for the mineralization types, and no material bias was evident.

**Table 12-5: Quarter Core Sampling Conducted by Nordmin, January 2021**

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-105 | 133.6 | 134.1 | 0.5 | 0.5 | 0.6 |
| BR-20-107 | 95.0 | 96.0 | 1.0 | 0.4 | 0.3 |
| BR-20-108 | 97.0 | 98.0 | 1.0 | 1.6 | 2.3 |
| BR-20-108 | 155.3 | 155.9 | 0.6 | 0.5 | 0.6 |
| BR-20-110 | 190.0 | 191.0 | 1.0 | 0.5 | 0.6 |
| BR-20-112 | 267.0 | 268.0 | 1.0 | 1.0 | 0.9 |
| BR-20-112 | 278.2 | 279.0 | 0.8 | 0.3 | 0.5 |
| BR-20-114 | 79.0 | 80.0 | 1.0 | 5.0 | 2.8 |
| BR-20-114 | 297.3 | 298.2 | 0.9 | 0.5 | 0.6 |
| BR-20-114 | 356.0 | 357.0 | 1.0 | 46.9 | 18.4 |
| BR-20-120 | 67.0 | 68.0 | 1.0 | 0.4 | 0.2 |
| BR-20-120 | 234.0 | 235.0 | 1.0 | 1.1 | 1.2 |

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-120 | 241.0 | 242.0 | 1.0 | 26.4 | 8.0 |
| BR-20-120 | 265.2 | 266.2 | 1.0 | 0.6 | 0.3 |
| BR-20-120 | 270.2 | 271.2 | 1.0 | 1.7 | 1.8 |
| BR-20-120 | 277.0 | 278.0 | 1.0 | 0.8 | 0.9 |
| BR-20-124 | 229.0 | 230.0 | 1.0 | 0.7 | 0.2 |
| BR-20-124 | 291.8 | 292.3 | 0.5 | 2.1 | 2.2 |
| BR-20-132 | 17.0 | 17.5 | 0.5 | 0.7 | 1.0 |
| BR-20-132 | 22.7 | 23.2 | 0.5 | 1.2 | 1.8 |
| BR-20-132 | 23.2 | 24.2 | 1.0 | 3.3 | 2.3 |
| BR-20-132 | 24.2 | 24.7 | 0.5 | 1.9 | 2.0 |
| BR-20-132 | 24.7 | 25.5 | 0.8 | 1.5 | 1.9 |
| BR-20-132 | 72.0 | 72.5 | 0.5 | 0.7 | 0.4 |
| BR-20-134 | 11.0 | 12.0 | 1.0 | 1.4 | 1.6 |
| BR-20-134 | 12.0 | 13.0 | 1.0 | 1.3 | 6.1 |
| BR-20-134 | 37.3 | 37.8 | 0.5 | 6.2 | 5.0 |
| BR-20-134 | 53.7 | 54.6 | 0.9 | 1.7 | 0.5 |
| BR-20-134 | 59.0 | 60.0 | 1.0 | 0.6 | 0.3 |
| BR-20-134 | 69.5 | 70.0 | 0.5 | 0.8 | 0.3 |
| BR-20-134 | 121.0 | 122.0 | 1.0 | 1.2 | 1.4 |
| BR-20-109 | 27.4 | 28.4 | 1.0 | 4.6 | 4.2 |
| BR-20-109 | 56.0 | 57.0 | 1.0 | 3.5 | 0.7 |
| BR-20-109 | 116.0 | 117.0 | 1.0 | 0.9 | 1.5 |
| BR-20-109 | 198.0 | 199.0 | 1.0 | 0.9 | 1.1 |
| BR-20-109 | 213.0 | 214.0 | 1.0 | 0.4 | 0.3 |
| BR-20-109 | 217.0 | 218.0 | 1.0 | 1.5 | 1.9 |
| BR-20-113 | 196.0 | 197.0 | 1.0 | 0.4 | 0.0 |
| BR-20-113 | 205.0 | 206.0 | 1.0 | 0.7 | 1.0 |
| BR-20-115 | 183.0 | 184.0 | 1.0 | 1.3 | 1.0 |
| BR-20-115 | 209.0 | 210.0 | 1.0 | 0.3 | 1.4 |
| BR-20-117 | 154.0 | 155.0 | 1.0 | 1.2 | 0.3 |
| BR-20-117 | 172.0 | 173.0 | 1.0 | 1.8 | 0.2 |
| BR-20-119 | 197.0 | 198.0 | 1.0 | 4.0 | 2.3 |
| BR-20-119 | 229.0 | 230.0 | 1.0 | 1.1 | 1.0 |
| BR-20-119 | 270.0 | 271.0 | 1.0 | 2.3 | 1.6 |
| BR-20-119 | 273.0 | 274.0 | 1.0 | 0.6 | 0.7 |
| BR-20-119 | 286.0 | 287.0 | 1.0 | 0.7 | 1.1 |
| BR-20-119 | 295.0 | 296.0 | 1.0 | 0.3 | 0.4 |
| BR-20-119 | 305.0 | 306.0 | 1.0 | 6.3 | 5.0 |

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-122 | 183.0 | 184.0 | 1.0 | 0.3 | 3.7 |
| BR-20-123 | 28.0 | 29.0 | 1.0 | 0.9 | 1.5 |
| BR-20-125 | 17.0 | 18.0 | 1.0 | 1.0 | 0.8 |
| BR-20-127 | 39.0 | 40.0 | 1.0 | 0.3 | 0.3 |
| BR-20-127 | 67.4 | 67.9 | 0.5 | 2.0 | 1.7 |
| BR-20-130 | 97.5 | 98.5 | 1.0 | 0.5 | 0.4 |
| BR-20-130 | 100.5 | 101.2 | 0.7 | 0.5 | 0.7 |
| BR-20-131 | 14.0 | 15.0 | 1.0 | 0.3 | 0.2 |
| BR-20-131 | 31.0 | 31.5 | 0.5 | 6.3 | 6.0 |
| BR-20-131 | 43.0 | 43.7 | 0.7 | 0.9 | 1.4 |
| BR-20-131 | 71.0 | 72.0 | 1.0 | 0.9 | 0.9 |
| BR-20-131 | 72.0 | 72.7 | 0.7 | 0.7 | 0.3 |
| BR-20-131 | 93.2 | 94.0 | 0.8 | 0.7 | 0.5 |
| BR-20-131 | 111.5 | 112.0 | 0.5 | 0.4 | 0.7 |
| BR-20-131 | 131.5 | 132.0 | 0.5 | 20.1 | 9.6 |
| BR-20-131 | 142.0 | 143.0 | 1.0 | 0.7 | 0.7 |
| BR-20-131 | 143.5 | 144.0 | 0.5 | 0.9 | 2.0 |
| BR-20-131 | 144.5 | 145.0 | 0.5 | 1.9 | 2.0 |
| BR-20-135 | 18.3 | 18.8 | 0.5 | 1.1 | 1.0 |
| BR-20-135 | 26.5 | 27.5 | 1.0 | 0.5 | 0.4 |
| BR-20-135 | 29.5 | 30.5 | 1.0 | 0.8 | 1.1 |
| BR-20-135 | 43.0 | 43.5 | 0.5 | 6.0 | 8.2 |
| BR-20-135 | 43.5 | 44.2 | 0.7 | 4.5 | 5.4 |
| BR-20-135 | 97.3 | 97.9 | 0.6 | 1.1 | 0.8 |
| BR-20-135 | 111.0 | 111.6 | 0.6 | 0.8 | 1.0 |
| BR-20-135 | 111.6 | 112.6 | 1.0 | 3.6 | 0.8 |
| BR-20-135 | 115.0 | 116.0 | 1.0 | 2.0 | 2.0 |
| BR-20-140 | 128.6 | 129.3 | 0.7 | 0.5 | 0.1 |
| BR-20-136 | 90.0 | 90.5 | 0.5 | 0.6 | 0.8 |
| BR-20-139 | 122.7 | 123.4 | 0.7 | 1.4 | 0.0 |
| BR-20-139 | 127.0 | 127.7 | 0.7 | 0.0 | 2.2 |
| BR-20-146 | 45.9 | 46.4 | 0.5 | 1.0 | 0.3 |
| BR-20-146 | 118.0 | 118.5 | 0.5 | 0.5 | 0.6 |
| BR-20-151 | 38.0 | 39.0 | 1.0 | 0.3 | 0.2 |
| BR-20-151 | 114.4 | 115.3 | 0.9 | 2.1 | 1.7 |
| BR-20-155 | 77.2 | 78.2 | 1.0 | 0.7 | 0.7 |
| BR-20-158 | 91.4 | 91.9 | 0.5 | 0.5 | 0.5 |
| BR-20-158 | 103.1 | 103.6 | 0.5 | 0.8 | 0.6 |

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-160 | 47.3 | 48.0 | 0.7 | 1.1 | 1.2 |
| BR-20-160 | 118.2 | 118.7 | 0.5 | 0.3 | 0.3 |
| BR-20-162 | 26.1 | 26.9 | 0.8 | 1.9 | 2.6 |
| BR-20-162 | 79.5 | 80.0 | 0.5 | 0.4 | 0.6 |
| BR-20-162 | 115.0 | 115.5 | 0.5 | 9.1 | 9.5 |
| BR-20-165 | 28.0 | 29.0 | 1.0 | 1.0 | 0.9 |
| BR-20-165 | 34.5 | 35.2 | 0.7 | 0.5 | 1.1 |
| BR-20-165 | 51.3 | 51.8 | 0.5 | 4.3 | 4.4 |
| BR-20-165 | 51.8 | 52.4 | 0.6 | 0.6 | 0.8 |
| BR-20-168 | 28.5 | 29.0 | 0.5 | 2.1 | 3.0 |
| BR-20-170 | 72.5 | 73.0 | 0.5 | 0.6 | 0.5 |
| BR-20-174 | 110.6 | 111.3 | 0.7 | 0.3 | 0.4 |
| BR-20-176 | 38.0 | 39.0 | 1.0 | 1.4 | 0.6 |
| BR-20-176 | 45.2 | 46.0 | 0.8 | 2.2 | 2.5 |
| BR-20-178 | 54.5 | 55.0 | 0.5 | 1.3 | 0.4 |
| BR-20-178 | 60.7 | 61.7 | 1.0 | 0.8 | 1.0 |
| BR-20-183 | 13.0 | 14.0 | 1.0 | 3.5 | 0.6 |
| BR-20-184 | 29.7 | 30.7 | 1.0 | 1.2 | 1.4 |
| BR-20-185 | 45.0 | 45.5 | 0.5 | 0.8 | 0.8 |
| BR-20-185 | 54.0 | 54.5 | 0.5 | 0.6 | 0.8 |
| BR-20-185 | 83.8 | 84.7 | 0.9 | 4.5 | 2.0 |
| BR-20-187 | 45.0 | 46.0 | 1.0 | 0.3 | 0.5 |
| BR-20-187 | 49.0 | 50.0 | 1.0 | 0.7 | 0.7 |
| BR-20-179 | 8.0 | 8.5 | 0.5 | 10.1 | 2.2 |
| BR-20-179 | 23.8 | 24.3 | 0.5 | 1.7 | 1.5 |
| BR-20-179 | 30.0 | 30.5 | 0.5 | 0.6 | 0.8 |
| BR-20-191 | 83.7 | 84.2 | 0.5 | 1.2 | 1.1 |
| BR-20-191 | 106.0 | 107.0 | 1.0 | 0.7 | 1.2 |
| BR-20-191 | 111.6 | 112.1 | 0.5 | 10.2 | 9.9 |
| BR-20-191 | 134.6 | 135.1 | 0.5 | 1.0 | 1.0 |
| BR-20-196 | 69.5 | 70.2 | 0.7 | 0.4 | 0.2 |
| BR-20-196 | 87.2 | 88.1 | 0.9 | 2.4 | 0.2 |
| BR-20-196 | 95.6 | 96.6 | 1.0 | 0.9 | 1.0 |
| BR-20-207 | 202.2 | 203.0 | 0.8 | 1.1 | 1.2 |
| BR-20-207 | 230.0 | 230.5 | 0.5 | 2.0 | 0.7 |
| BR-20-207 | 269.0 | 269.5 | 0.5 | 0.5 | 0.5 |
| BR-20-207 | 294.0 | 294.5 | 0.5 | 4.0 | 4.0 |
| BR-20-207 | 301.0 | 301.5 | 0.5 | 0.6 | 1.0 |

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-213 | 183.0 | 183.5 | 0.5 | 0.6 | 0.2 |
| BR-20-138 | 115.9 | 116.4 | 0.5 | 0.5 | 1.4 |
| BR-20-138 | 124.1 | 125.1 | 1.0 | 1.0 | 0.8 |
| BR-20-141 | 91.7 | 92.2 | 0.5 | 1.3 | 1.2 |
| BR-20-143 | 70.1 | 70.6 | 0.5 | 0.3 | 0.4 |
| BR-20-143 | 155.6 | 156.1 | 0.5 | 0.7 | 0.8 |
| BR-20-147 | 106.5 | 107.0 | 0.5 | 1.1 | 2.1 |
| BR-20-147 | 191.0 | 192.0 | 1.0 | 1.1 | 1.8 |
| BR-20-147 | 198.0 | 199.0 | 1.0 | 1.2 | 0.9 |
| BR-20-150 | 32.8 | 33.5 | 0.7 | 1.2 | 1.2 |
| BR-20-161 | 95.5 | 96.0 | 0.5 | 0.6 | 0.4 |
| BR-20-161 | 143.5 | 144.0 | 0.5 | 1.0 | 0.8 |
| BR-20-164 | 114.0 | 115.0 | 1.0 | 1.1 | 0.4 |
| BR-20-166 | 24.6 | 25.6 | 1.0 | 0.4 | 2.8 |
| BR-20-173 | 87.2 | 87.7 | 0.5 | 0.8 | 0.8 |
| BR-20-173 | 87.7 | 88.2 | 0.5 | 0.8 | 0.9 |
| BR-20-177 | 91.8 | 92.3 | 0.5 | 0.4 | 0.4 |
| BR-20-177 | 119.3 | 119.8 | 0.5 | 0.7 | 0.6 |
| BR-20-177 | 188.0 | 188.5 | 0.5 | 1.8 | 1.9 |
| BR-20-197 | 136.9 | 137.4 | 0.5 | 0.5 | 0.4 |
| BR-20-200 | 81.0 | 82.0 | 1.0 | 1.5 | 1.8 |
| BR-20-200 | 100.0 | 101.0 | 1.0 | 20.1 | 7.0 |
| BR-20-204 | 49.0 | 50.0 | 1.0 | 0.3 | 0.4 |
| BR-20-210 | 107.0 | 108.0 | 1.0 | 1.1 | 0.2 |
| BR-20-142 | 56.3 | 57.3 | 1.0 | 11.1 | 3.0 |
| BR-20-142 | 107.5 | 108.0 | 0.5 | 2.3 | 1.5 |
| BR-20-142 | 189.9 | 190.9 | 1.0 | 8.2 | 8.0 |
| BR-20-142 | 191.5 | 192.1 | 0.6 | 2.5 | 0.9 |
| BR-20-142 | 193.7 | 194.2 | 0.5 | 5.6 | 5.7 |
| BR-20-142 | 195.7 | 196.2 | 0.5 | 1.9 | 1.8 |
| BR-20-142 | 210.8 | 211.8 | 1.0 | 11.7 | 22.5 |
| BR-20-142 | 221.5 | 222.0 | 0.5 | 0.6 | 0.4 |
| BR-20-148 | 172.3 | 172.8 | 0.5 | 0.8 | 0.9 |
| BR-20-148 | 190.5 | 191.0 | 0.5 | 0.9 | 1.1 |
| BR-20-156 | 42.0 | 42.5 | 0.5 | 4.2 | 3.2 |
| BR-20-156 | 64.3 | 65.0 | 0.7 | 0.3 | 0.3 |
| BR-20-156 | 65.0 | 65.5 | 0.5 | 2.3 | 1.1 |
| BR-20-159 | 202.2 | 202.7 | 0.5 | 3.3 | 3.0 |

| Drill Hole ID | From (m) | To (m) | Length | Half Core Au (ppm) | Quarter Core Au (ppm) |
|---|---|---|---|---|---|
| BR-20-175 | 63.3 | 63.8 | 0.5 | 1.1 | 1.1 |
| BR-20-175 | 92.0 | 92.5 | 0.5 | 0.9 | 1.2 |
| BR-20-175 | 162.5 | 163.0 | 0.5 | 1.5 | 1.7 |
| BR-20-180 | 18.0 | 18.5 | 0.5 | 1.3 | 0.5 |
| BR-20-180 | 22.5 | 23.0 | 0.5 | 0.4 | 1.0 |
| BR-20-180 | 29.0 | 29.5 | 0.5 | 0.8 | 1.5 |
| BR-20-182 | 70.1 | 70.6 | 0.5 | 1.9 | 2.8 |
| BR-20-182 | 76.0 | 76.5 | 0.5 | 0.9 | 1.2 |
| BR-20-182 | 77.5 | 78.0 | 0.5 | 0.4 | 0.5 |
| BR-20-194 | 113.0 | 114.0 | 1.0 | 1.1 | 1.2 |
| BR-20-199 | 51.5 | 52.1 | 0.6 | 0.9 | 1.4 |
| BR-20-201 | 68.5 | 69.0 | 0.5 | 1.1 | 1.3 |
| BR-20-201 | 99.5 | 100.5 | 1.0 | 0.6 | 0.8 |
| BR-20-201 | 110.5 | 111.1 | 0.6 | 0.7 | 0.8 |
| BR-20-205 | 26.8 | 27.3 | 0.5 | 0.5 | 0.9 |
| BR-20-205 | 32.0 | 33.0 | 1.0 | 0.6 | 0.8 |
| BR-20-205 | 37.0 | 37.5 | 0.5 | 1.6 | 2.0 |
| BR-20-205 | 37.5 | 38.0 | 0.5 | 0.9 | 0.9 |
| BR-20-205 | 63.9 | 64.4 | 0.5 | 4.1 | 12.7 |
| BR-20-208 | 39.0 | 39.5 | 0.5 | 1.7 | 1.6 |
| BR-20-209 | 30.4 | 31.0 | 0.6 | 0.4 | 0.0 |

**Table 12-6: Quarter Core Sampling Conducted by Nordmin, August 2021**

| Hole ID | From | To | Length | 1/4 Core (Au ppm) | 1/2 Core (Au ppm) |
|---|---|---|---|---|---|
| BR-20-142 | 210.8 | 211.8 | 1 | 0.51 | 48.502 |
| BR-20-142 | 211.8 | 212.3 | 0.5 | 0.39 | 1.977 |
| BR-20-142 | 212.3 | 212.8 | 0.5 | 0.17 | 0.702 |
| BR-20-142 | 212.8 | 213.3 | 0.5 | 0.15 | 0.16 |
| BR-20-142 | 213.3 | 214.2 | 0.9 | 1.37 | 0.38 |
| BR-20-142 | 214.2 | 214.7 | 0.5 | 2.65 | 440.333 |
| BR-20-142 | 214.7 | 215.7 | 1 | 20.4 | 6.613 |
| BR-20-142 | 215.7 | 216.3 | 0.6 | 0.22 | 0.1 |
| BR-20-142 | 216.3 | 216.8 | 0.5 | 2.78 | 0.469 |
| BR-20-142 | 216.8 | 217.3 | 0.5 | 5.22 | 28.553 |
| BR-20-142 | 217.3 | 218 | 0.7 | 1.63 | 23.088 |
| BR-20-142 | 218 | 218.5 | 0.5 | 15.9 | 0.089 |
| BR-20-142 | 218.5 | 219 | 0.5 | 0.05 | 0.02 |
| BR-20-142 | 219 | 219.5 | 0.5 | 0.02 | 0.029 |

| Hole ID | From | To | Length | 1/4 Core (Au ppm) | 1/2 Core (Au ppm) |
|---|---|---|---|---|---|
| BR-20-142 | 219.5 | 220 | 0.5 | 0.01 | 0.029 |
| BR-20-142 | 220 | 220.5 | 0.5 | 0.06 | 0.22 |
| BR-20-142 | 220.5 | 221 | 0.5 | 1.94 | 2.88 |
| BR-20-142 | 221 | 221.5 | 0.5 | 0.38 | 0.11 |
| BR-20-142 | 221.5 | 222 | 0.5 | 0.11 | 0.571 |
| BR-20-142 | 222 | 222.6 | 0.6 | 0.34 | 0.13 |
| BR-20-142 | 222.6 | 223.6 | 1 | 0.14 | 0.16 |
| BR-20-114 | 349 | 350 | 1 | 0.21 | 0.349 |
| BR-20-114 | 350 | 351 | 1 | 0.34 | 0.22 |
| BR-20-114 | 351 | 352 | 1 | 0.47 | 3.279 |
| BR-20-114 | 352 | 353 | 1 | 0.08 | 0.119 |
| BR-20-114 | 353 | 354 | 1 | 0.65 | 0.614 |
| BR-20-114 | 354 | 355 | 1 | 0.34 | 0.26 |
| BR-20-114 | 355 | 356 | 1 | 0.29 | 0.25 |
| BR-20-114 | 356 | 357 | 1 | 0.27 | 46.862 |
| BR-20-114 | 357 | 358 | 1 | 0.14 | 0.554 |
| BR-20-114 | 358 | 359 | 1 | 0.45 | 0.4 |
| BR-21-246 | 48.5 | 49 | 0.5 | 1.77 | 1.79 |
| BR-21-246 | 49 | 49.5 | 0.5 | 1.59 | 1.235 |
| BR-21-246 | 49.5 | 50 | 0.5 | 1.7 | 0.861 |
| BR-21-246 | 50 | 50.5 | 0.5 | 0.43 | 0.694 |
| BR-21-246 | 50.5 | 51 | 0.5 | 0.32 | 49.395 |
| BR-21-246 | 51 | 52 | 1 | 0.09 | 0.289 |
| BR-21-246 | 52 | 53 | 1 | 0.06 | 0.029 |
| BR-21-246 | 53 | 53.5 | 0.5 | 10.7 | 0.95 |
| BR-21-246 | 53.5 | 54 | 0.5 | 9.16 | 3.188 |
| BR-21-246 | 54 | 54.5 | 0.5 | 5.01 | 7.769 |
| BR-21-246 | 54.5 | 55 | 0.5 | 0.4 | 0.01 |
| BR-21-246 | 55 | 55.5 | 0.5 | 0.02 | 0.01 |
| BR-21-246 | 55.5 | 56 | 0.5 | 1.8 | 0.409 |
| BR-21-246 | 56 | 56.5 | 0.5 | 4.44 | 2.616 |
| BR-21-246 | 56.5 | 57 | 0.5 | 12.1 | 1.578 |
| BR-21-246 | 57 | 57.5 | 0.5 | 1.01 | 0.936 |
| BR-21-246 | 57.5 | 58 | 0.5 | 1.37 | 5.03 |
| BR-21-246 | 58 | 58.5 | 0.5 | 2.3 | 4.416 |
| BR-21-246 | 58.5 | 59 | 0.5 | 0.04 | 0.02 |
| BR-21-246 | 59 | 59.5 | 0.5 | 76 | 93.322 |
| BR-21-246 | 59.5 | 60 | 0.5 | 2.85 | 3.495 |
| BR-20-221 | 43 | 44 | 1 | 1.12 | 0.05 |

| Hole ID | From | To | Length | 1/4 Core (Au ppm) | 1/2 Core (Au ppm) |
|---|---|---|---|---|---|
| BR-20-221 | 44 | 44.5 | 0.5 | 0.46 | 0.05 |
| BR-20-221 | 44.5 | 45 | 0.5 | 0.06 | 0.05 |
| BR-20-221 | 45 | 45.5 | 0.5 | 0.04 | 0.05 |
| BR-20-221 | 45.5 | 46 | 0.5 | 4.63 | 0.05 |
| BR-20-221 | 46 | 46.5 | 0.5 | 0.06 | 0.05 |
| BR-20-221 | 46.5 | 47 | 0.5 | 0.71 | 0.05 |
| BR-20-221 | 47 | 47.5 | 0.5 | 4.37 | 0.05 |
| BR-20-221 | 47.5 | 48 | 0.5 | 2.9 | 0.05 |
| BR-20-221 | 48 | 48.5 | 0.5 | 1.6 | 0.05 |
| BR-20-221 | 48.5 | 49 | 0.5 | 0.01 | 0.05 |

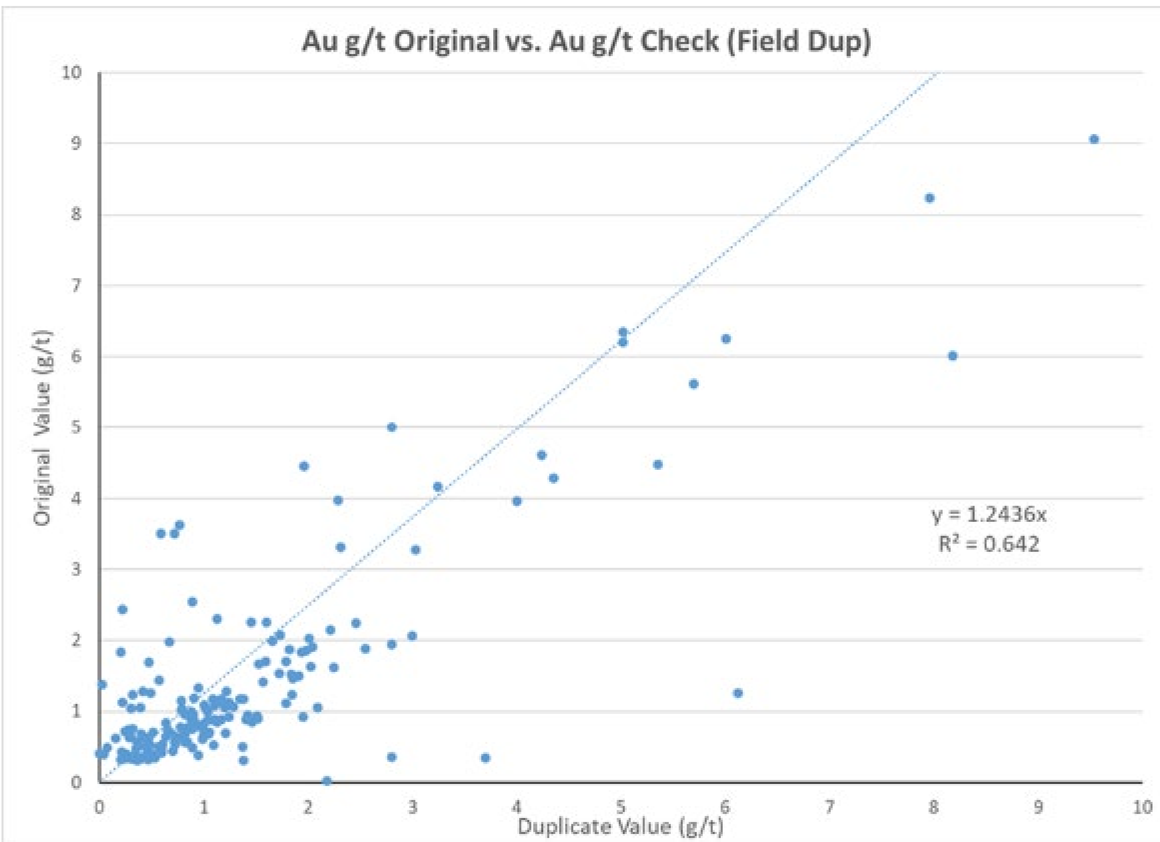



*Figure 12-11: Scatter plot comparison of gold (g/t) verification drill core samples, January 2021*

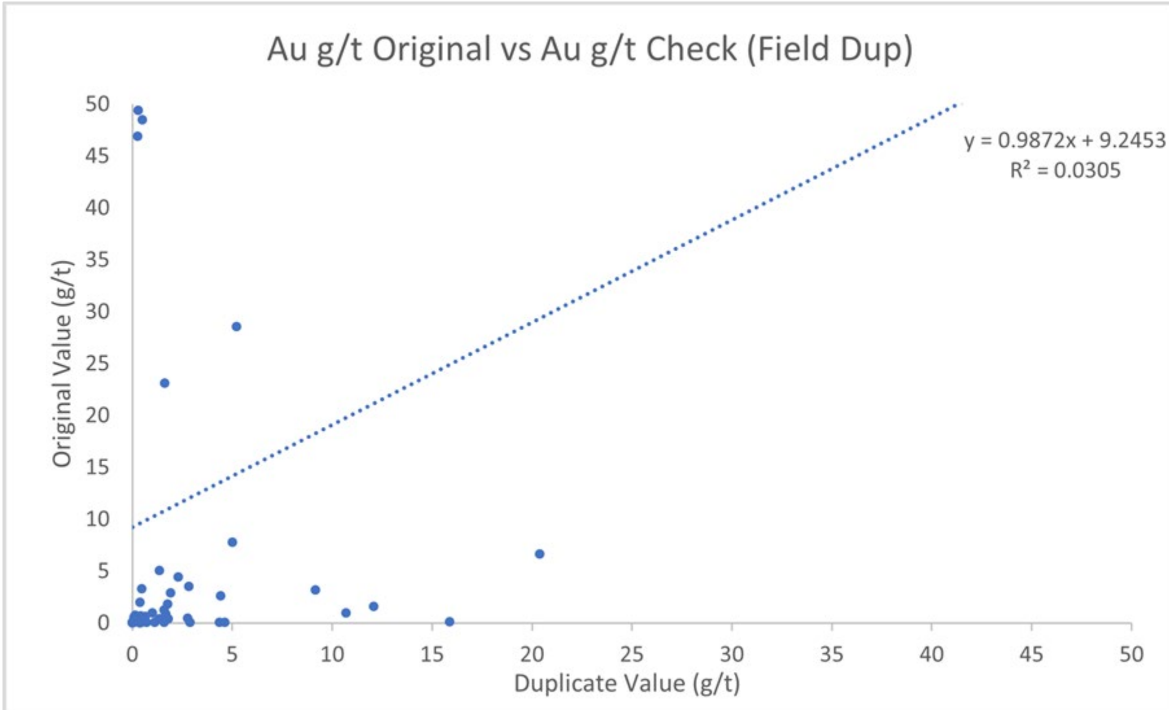



*Figure 12-12: Scatter plot comparison of gold (g/t) verification drill core samples, August 2021*

The drill core samples selected by the QP for verification analysis were individually placed into plastic sample bags, which were then packaged together and shipped to Eastern Analytical (Figure 12-13), for analysis using the Company's analytical procedures.


*Figure 12-13: Verification samples sent to Eastern Analytical*

The Company uses local Devonian granite boulder material from a rock quarry in Stormont, Nova Scotia, as its sample for blank material. The blank material is well tested and is consistent in trace elements of gold and silver (Figure 12-14 and Figure 12-15).

*Figure 12-14: Blank material from granite boulder rock quarry in Stormont, Nova Scotia*

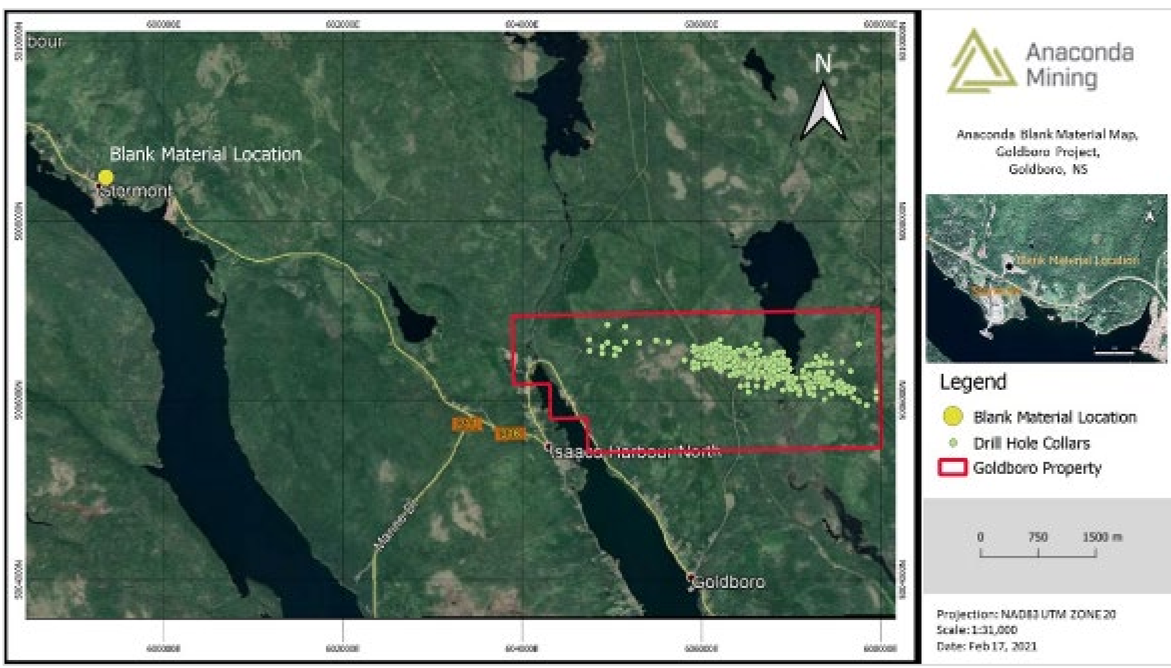



*Figure 12-15: Location of blank material in relation to the Project site*

### 12.1.4 Geological Interpretation, Surface Drilling, and Mineralized Surface Stockpiles Validation

The QP examined approximately 41 drill holes and approximately 350 samples located across the three domains as part of the due diligence, including reviewing the various geological section interpretations with the Company's geological team (Figure 12-16).

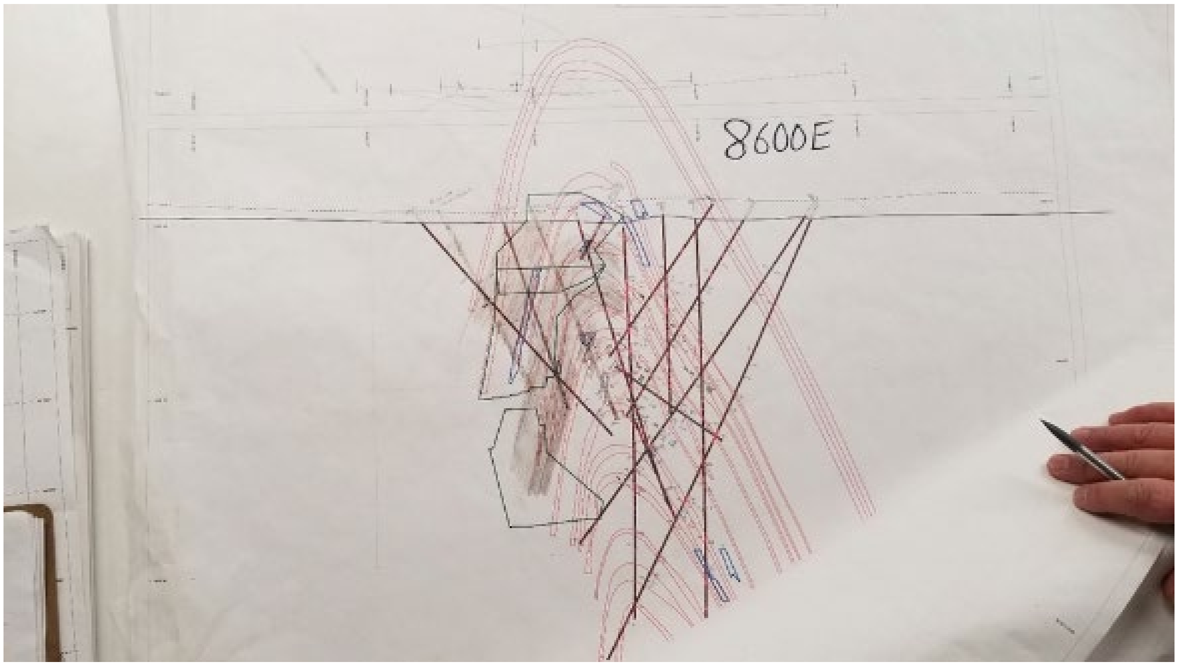



*Figure 12-16: Project geological section 8600E*

Nordmin examined the current stockpile on site with underground mineralization from the Bulk Sample previously completed using the portal (Figure 12-17). The material was reviewed from a geological, mineralization, and grade perspective. Various samples were examined to determine if the material and geological interpretation were indicative of what Nordmin used as the Mineral Resource Estimate basis. Nordmin reviewed various hand specimens and larger pieces demonstrating the same geological fabric, alteration, and style of mineralization observed within the drill core (Figure 12-18).

*Figure 12-17: Onsite mineralized stockpile*


*Figure 12-18: Large sample and hand specimen and from the mineralized stockpile*

Nordmin examined the underground portal area, local geology, and surface material that is located within the proposed open pit area. The geological logs confirm the expected depths of the local overburden and corresponding rock types (Figure 12-19). Figure 12-20 illustrates the overburden is approximately 10 m to 15 m in thickness, and the top of bedrock is bolted and screened.

*Figure 12-19: Hole BR-17-05 in the vicinity of the underground portal displaying similar overburden material to what is located within the underground portal area.*

*Figure 12-20: The flooded underground portal demonstrates the depth of overburden to bedrock*

Various drill hole collar sites were reviewed, including both historical and from the most recent 2020 drill program. Figure 12-21 illustrates the previous historical drill collars with the more recent infill 2020 drill program collar locations. The drill spacing from the field is in agreement with what is located within the drill hole database.

*Figure 12-21: Drill collar locations of BR-20-108, BR-20-114, BR-20-193 (2020)*

Various 2020 drill intervals were compared by the QP to the remaining core to determine if the logging and sampling intervals matched the information within the logs. The logging and sampling observed by the QP were consistent and representative. The drill core outlined in Figure 12-22 and Figure 12-23 were to emulate the multiple open pit benches that combine both the High-Grade Mineralized "Belts" and Low-Grade Mineralized Zones that surround the Belt.

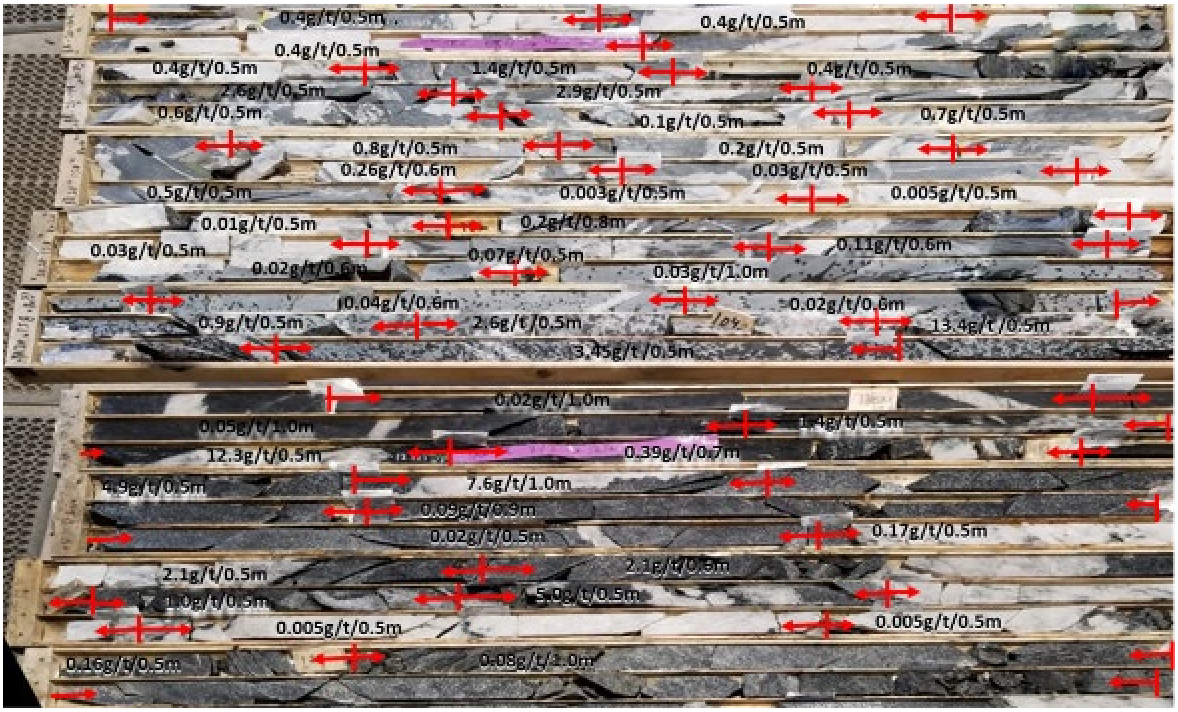



*Figure 12-22: Hole BR-20-158 (top four boxes) from 80.8 m to 105.1 m and hole BR-20-134 (bottom four boxes) from 115.3 m to 130 m outlining both broad lower-grade mineralization and the higher-grade zones within.*

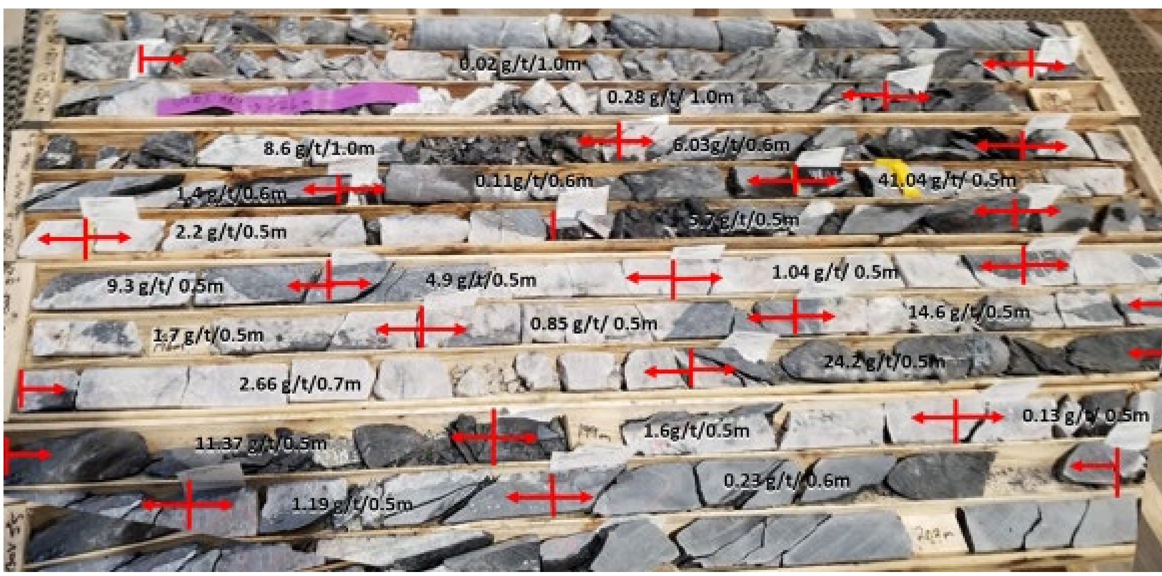



*Figure 12-23: Hole BR-20-142 from 186.5 m to 202.5 m outlining both broad lower-grade mineralization and high-grade zones within.*

## 12.2 Database Validation

Core sample records, lithologic logs, laboratory reports and associated drill hole information for all drill programs completed in the 1984 to 2011 period were digitally compiled for use in Gemcom-Surpac Version 6.2.1® (Surpac™) deposit modelling software. Historical and current drilling program information was reviewed, and digital records of historic drilling were checked for both consistency and accuracy against original source documents available through NSDNR or received from Orex. All

2010 and 2011 drill hole coordination and orientation data inputs were checked, and validation of approximately 20% of the assay dataset for sample interval and assay value information against corresponding source documents was carried out.

From 2011 until current, all drill hole data was compiled into a validated Microsoft Access® database that Nordmin reviewed digitally using a combination of Datamine and Target software programs.

The QP completed a spot check verification on the Project of:

- Drill holes—62 (12%) of the lithologies, 1,042 (10%) of the geotechnical measurements, 3,843 (8%) of the assays.
- Chip samples—84 (6%) of the lithologies, 168 (12%) of the assays.

The geology was validated for lithological units from the Company's Geovia GEMS logger. The geological contacts and lithology are aligned with the core contacts and lithology and are acceptable for use.

## 12.3 Review of the Company's QA/QC

The Company has a robust QA/QC process in place, as previously described in Section 11. The Company geologists actively monitor the assay results throughout the drill programs and summarize the QA/QC results, reporting weekly and monthly. Most of the CRMs performed as expected within tolerances of two to three standard deviations of the mean grade. Nordmin is satisfied that the QA/QC process performs as designed to ensure the assay data quality.

## 12.4 QP's Opinion

Upon completion of the data verification process, no apparent bias was determined between the historical programs in comparison with the programs since 2005. It is the QP's opinion that the geological data collection and QA/QC procedures used by the Company are consistent with standard industry practices and that the geological database is of suitable quality to support the Mineral Resource Estimate.

# 13. MINERAL PROCESSING AND METALLURGICAL TESTING

## 13.1 Introduction and Summary

The objective of the metallurgical study was to quantify the metallurgical response of mineralized material from the Deposit. The program was designed with the intent to confirm the parameters for process design criteria for grinding, gravity concentration, leaching, cyanide destruction and tailings thickening. The metallurgical program was conducted at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as project BL883 in October and November 2021, and was performed on samples from West and East Pits. BaseMet Labs is independent of the Company.

## 13.2 Historical Testwork

Summaries of the historic testwork are found in previous technical reports on the Project can be found in Section 2.5. The specific testwork programs previously completed for these technical reports on the Goldboro are shown in Table 13-1, including a brief summary of the testwork performed.

**Table 13-1: Reference Documents**

| Report | Testing Performed |
|---|---|
| Thibault and Associates Inc. Report, (Project Number 6429 Phase I) March 8, 2018 | Preliminary testing on a single sample. Testing included grinding, flotation, gravity concentration leaching, and cyanide destruction. |
| Base Metallurgical Laboratories Inc., BL0395 Report #1 July12, 2019 | Flowsheet development program on underground samples. Testing included comminution tests, gravity concentration, leaching, cyanide destruction and arsenic precipitation. |
| JKTech, SMC Test Report for JKTech Job No. 19008/P3" dated February 2019 | SMC tests on selected samples from BL0395 program. |
| Base Metallurgical Laboratories Inc., BL685 Metallurgical Study of the Goldboro Project | Flowsheet development program on open pit samples. Testing including comminution tests, gravity concentration, leaching, cyanide destruction and arsenic precipitation. |

## 13.3 Feasibility Study Testwork

The 2021 metallurgical test program was designed to support the FS and is based on results from the 2020 program, primarily to determine hardness variability, optimize cyanide detox for lower cyanide discharge requirements and generate thickener design data. The program was conducted at BaseMet Labs in Kamloops, British Columbia. The program scope included:

- Additional comminution tests (bond ball mill work index) on 24 samples to characterize hardness variability.
- Bulk gravity and leach test to provide slurry for cyanide destruction optimization and tailings thickener sedimentation tests.

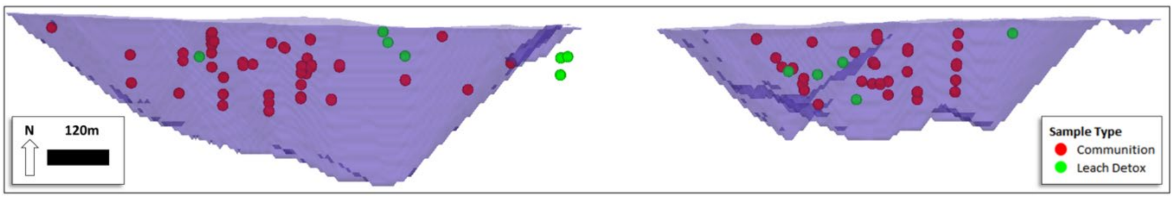
### 13.3.1 Samples

#### 13.3.1.1 Sample Description

Twenty four samples (Comp 1 to Comp 24), representing mineralization from the two open pits were selected from available NQ drill core to provide spatial representation to assess variability of the hardness as measured in the bond ball mill work index. These samples were also assayed.

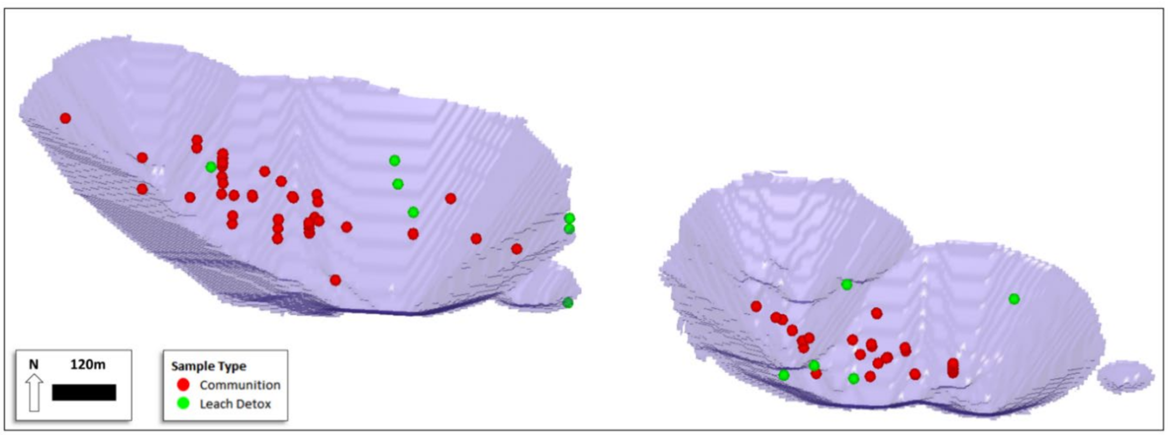
A large Bulk Sample was assembled from available drill core to provide a sample represent initial production (years 1 to 5). The sample was subjected to head assays, bulk gravity and leaching, with leached slurry then used for cyanide destruction optimization and thickener sedimentation testing.

Figure 13-1 and Figure 13-2 provide the location of the drill core intervals selected for all samples.




Source: Nordmin, 2021

*Figure 13-1: Location of metallurgical samples open pits shown on longitudinal section looking north*




Source: Nordmin, 2021

*Figure 13-2: Location of metallurgical samples open pits shown in plan view*

#### 13.3.1.2 Sample Analysis

Screened metallics gold assays were conducted on the 25 composites due to the occurrence of coarse free gold, as evident in previous programs. Samples of 0.5 kg from each composite were pulverized and then screened at 106 µm with the oversize and undersize fractions assayed separately. The head grade was calculated from the weighted assays from the two fractions. The results are shown in Table 13-2. Several samples contain coarse gold within the coarse fraction that is significantly above the overall mass. The lower grade samples also display similar effects with coarse gold. Collectively, these align with the amenability of the samples tested for GRG previously demonstrated in the reports listed in Section 13.2. The predicted head grade for the Master Composite was 1.65 g/t Au from the drill core assays. The calculated head grade from the screen metallics analysis is significantly higher at 4.31 g/t Au. This value was validated in the bulk leach test.

**Table 13-2: Screen Metallics Sample Assays**

| Sample | +106 µm Fraction | | -106 µm Fraction | Calc. Grade (g/t Au) |
|---|---|---|---|---|
| | Au (g/t) | Au Dist. (%) | Au (g/t) | |
| Comp 1 | 4.19 | 25.5 | 0.72 | 0.91 |
| Comp 2 | 0.85 | 6.4 | 0.62 | 0.63 |
| Comp 3 | 0.41 | 2.3 | 0.75 | 0.73 |
| Comp 4 | 1.58 | 8.8 | 0.36 | 0.39 |
| Comp 5 | 16.5 | 46.4 | 1.03 | 1.82 |
| Comp 6 | 13.7 | 38.7 | 0.93 | 1.45 |
| Comp 7 | 4.04 | 26.6 | 0.63 | 0.81 |
| Comp 8 | 0.86 | 8.6 | 0.51 | 0.52 |
| Comp 9 | 5.12 | 29.2 | 0.70 | 0.93 |
| Comp 10 | 0.52 | 4.1 | 0.51 | 0.51 |
| Comp 11 | 0.64 | 4.9 | 0.57 | 0.57 |
| Comp 12 | 1.38 | 11.3 | 0.68 | 0.72 |
| Comp 13 | 5.24 | 49.4 | 0.33 | 0.61 |
| Comp 14 | 9.11 | 34.5 | 1.06 | 1.52 |
| Comp 15 | 0.68 | 7.8 | 0.48 | 0.49 |
| Comp 16 | 1.05 | 10.1 | 0.57 | 0.60 |
| Comp 17 | 1.16 | 11.8 | 0.44 | 0.47 |
| Comp 18 | 0.48 | 4.0 | 0.57 | 0.56 |
| Comp 19 | 1.09 | 6.4 | 0.68 | 0.70 |
| Comp 20 | 1.26 | 9.2 | 0.75 | 0.77 |
| Comp 21 | 1.26 | 2.7 | 0.75 | 0.60 |
| Comp 22 | 0.68 | 4.4 | 0.83 | 0.82 |
| Comp 23 | 0.68 | 27.0 | 0.83 | 0.54 |
| Comp 24 | 0.20 | 2.9 | 0.31 | 0.30 |
| Master | 50.9 | 67.4 | 1.49 | 4.31 |

Source: BaseMet, 2021

The head analysis of the samples is shown in Table 13-3. Sulphur occurs primarily as sulphide sulphur and is associated predominantly with arsenopyrite. There is some arsenic solubility, as noted in the dissolved arsenic concentrations in the cyanide detox tests. Soluble arsenic was removed to required concentrations through ferric sulphate precipitation. Copper concentrations are low and below where they would be considered cyanide consuming.

**Table 13-3: Open Pit Test Program Head Analysis**

| Composite | Assay | | | | |
|---|---|---|---|---|---|
| | Cu (g/t) | Ag (g/t) | Fe (%) | S (%) | As (g/t) |
| Comp 1 | 49.6 | 0.1 | 4.00 | 0.28 | 4,542 |
| Comp 2 | 30.8 | 0.1 | 3.32 | 0.44 | 10,458 |
| Comp 3 | 33.2 | 0.4 | 3.88 | 0.40 | 8,404 |
| Comp 4 | 21.6 | 0.1 | 3.64 | 0.40 | 9,286 |
| Comp 5 | 54.8 | 0.2 | 4.40 | 0.62 | 153,300 |

| Composite | Assay | | | | |
|---|---|---|---|---|---|
| | Cu (g/t) | Ag (g/t) | Fe (%) | S (%) | As (g/t) |
| Comp 6 | 30.0 | 0.2 | 3.40 | 0.66 | 164,100 |
| Comp 7 | 26.0 | 0.2 | 3.40 | 0.36 | 9,681 |
| Comp 8 | 71.2 | 0.3 | 4.04 | 0.44 | 5,611 |
| Comp 9 | 40.0 | <0.1 | 3.04 | 0.22 | 3,976 |
| Comp 10 | 25.6 | 0.2 | 4.40 | 0.59 | 143,800 |
| Comp 11 | 31.2 | 0.2 | 4.36 | 0.71 | 183,500 |
| Comp 12 | 48.0 | 0.3 | 4.48 | 0.62 | 11,072 |
| Comp 13 | 18.8 | 0.7 | 2.84 | 0.25 | 4,321 |
| Comp 14 | 35.2 | 0.2 | 4.04 | 0.24 | 2,775 |
| Comp 15 | 23.6 | 0.1 | 2.80 | 0.16 | 6,938 |
| Comp 16 | 22.4 | 0.1 | 3.24 | 0.07 | 1,677 |
| Comp 17 | 25.6 | 0.1 | 3.28 | 0.19 | 4,418 |
| Comp 18 | 36.4 | 0.2 | 3.44 | 0.27 | 6,228 |
| Comp 19 | 34.8 | 0.1 | 4.24 | 0.27 | 5,983 |
| Comp 20 | 20.4 | 0.1 | 3.20 | 0.24 | 5,934 |
| Comp 21 | 20.4 | 0.1 | 3.24 | 0.18 | 4,470 |
| Comp 22 | 32.0 | 0.1 | 3.80 | 0.31 | 3,128 |
| Comp 23 | 35.6 | 0.1 | 4.12 | 0.14 | 2,333 |
| Comp 24 | 20.0 | 0.3 | 2.84 | 0.23 | 5,562 |
| Master | 34.0 | 0.2 | 3.52 | 0.30 | 6,492 |

Source: BaseMet, 2021

### 13.3.2 Comminution Testing

The comminution testing included only bond ball work index testing of the 24 samples, based on an analysis of the comminution testing completed in earlier metallurgical testing programs. The samples selected provide full spatial coverage of both pits. Results are summarised in Table 13-4.

**Table 13-4: Bond Ball Mill Work Index Testing Results**

| Test | Units | Average | $75^{th}$ percentile | Minimum | Maximum |
|---|---|---|---|---|---|
| Bond Ball Mill Work Index | kWh/t | 15.0 | 15.5 | 14.0 | 16.2 |

Source: BaseMet, 2021

The bond ball mill work index results are characterized as hard ($75^{th}$ percentile). The average is slightly lower than the average from the 2020 program which was 15.2 kW/t. The $75^{th}$ percentile value from the 2020 program was 15.7 kWh/t, also slightly higher.

### 13.3.3 Gravity Separation

The entire Master Composite sample was processed through gravity concentration prior to leach testing with a 75 mm diameter Knelson concentrator followed by a Mozley Super Panner with a target mass recovery of 0.05%. The gravity concentrate graded 5,676 g/t Au with a mass recovery of 0.057%. Overall gravity gold recovery was 64.3%.

### 13.3.4 Leach Testing

#### 13.3.4.1 Bulk Leach Testing

The Master Composite Sample was submitted for cyanidation testing using the entire gravity tailing in a stirred reactor and sampled to measure leach kinetics at specified increments of 2, 6, 8, 24 and 36 hours, at which point the leach was terminated.

The leaching test conditions are summarized below:

- Pulp Density = 45% Solids (w/w)
- Pulp pH: 11.0 (maintained)
- Dissolved oxygen: approximately 7 – 8 mg/L from air addition
- Sodium cyanide (NaCN) concentration = 0.25 g/L (maintained to 8 hours)
- Activated carbon added for the last 12 hours to simulate tailings with respect to gold concentration and available cyanide.
- Target grind of 80% passing 100 µm

The leach results are presented in Table 13-5

**Table 13-5: Master Composite Leach Test Results**

| Reagent Consumption (kg/t) | | Calc. Head Grade (g/t Au) | Gravity Recovery (% Au) | Cumulative Recovery (% Au) | | | | |
|---|---|---|---|---|---|---|---|---|
| NaCN | CaO | | | 2 h | 6 h | 8 h | 24 h | 36 h |
| 0.19 | 1.44 | 5.05 | 64.3 | 70.8 | 76.1 | 78.8 | 85.9 | 97.8 |

Source: BaseMet, 2021

The results show high recovery that exceed the modeled recovery at this head grade using the recovery model. Cyanide consumption is very low with the tailings containing 0.14 g/L free NaCN. The results also validate the 36 hour retention time selected for the 2021 PEA process design as leach extraction continues and plateaus between 24 and 36 hours.

### 13.3.5 Cyanide Destruction

#### 13.3.5.1 The $SO_2$/Air Process

The chemical reaction for the oxidation of weak-acid dissociable cyanide ($CN_{WAD}$) using sodium metabisulphite ($Na_2S_2O_5$ as a source of $SO_2$) is widely used throughout the industry. The technology is proven and capable of achieving low $CN_{WAD}$ concentrations.

Process development testing for the $SO_2$/air process is completed in two stages. The first stage is batch testing, followed by second stage continuous testing. The batch reactor is first filled with feed slurry and the required copper sulphate is added. The reactor content is then treated in batch mode with sodium metabisulphite ($Na_2S_2O_5$ or SMBS) as the $SO_2$ source and air to reduce the concentration of $CN_{WAD}$ in solution to target less than 0.5 mg/L. The oxidation reduction potential (ORP) of the pulp is monitored with a Pt/Ag/AgCl combination electrode, while the residual $CN_{WAD}$ concentration in the solution phase is analyzed during the test determined using the Modified Potentiometric Titration method. Iniital target batch retention times are between 45 and 60 minutes. The batch test serves to produce treated material with low residual $CN_{WAD}$, the product is used as starting feed material for the initial continuous test. Final solutions are submitted for analysis at the completion of each test or run.

A 2-L reactor was used for both batch and continuous tests. For the continuous tests, an overflow nozzle on the reactor transferred treated slurry to a storage tank.

#### 13.3.5.2 Master Composite Cyanide Destruction Testing

The low free cyanide concentration at the end of the leach test resulted in an initial concentration of only 25.0 mg/L $CN_{WAD}$. This is significantly lower than earlier tests which had initial concentrations of 125 – 150 mg/L $CN_{WAD}$. The results are presented in Table 13-6.

Previous testing was directed to achieve a discharge concentration of less than 3 mg/L $CN_{WAD.}$ The current program was designed to achieve a discharge concentration of less than 0.5 mg/L $CN_{WAD}$ in support of achieving a desired discharge effluent concentration of 0.5 mg/L $CN_{TOT.}$ The program included five testing iterations achieved the lower target. All tests were conducted at a pulp density of 40% solids (w/w). Air and or oxygen was added to maintain a minimum dissolved concentration of 6.0 mg/L.

**Table 13-6: Master Composite Cyanide Destruction Testing Results**

| Test | Objective | Ret. Time (min) | Reactor Chemistry (Solution) | | | | | Reagent Add. (g/g $CN_{WAD}$) | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | pH | $CN_t$ mg/L | $CN_{WAD}$ mg/L | Cu mg/L | Fe mg/L | $SO_2$ Equiv. | Lime | Cu mg/L sol. |
| ***Master Comp: CN40*** | | | 10.3 | 44.1 | 25.0 | 6.40 | 6.8 | | | |
| CND-C1 | $SO_2$: $CN_{WAD}$ 5:1 & 125 mg/L Cu | 60 | 8.4 | 1.5 | 1.5 | 0.22 | <1 | 5.0 | - | 125 |
| CND-C2 | $SO_2$: $CN_{WAD}$ 5:1 & 150 mg/L Cu | 120 | 8.6 | 0.9 | 0.9 | - | - | 5.0 | - | 150 |
| CND-C3 | $SO_2$: $CN_{WAD}$ 10:1 & 150 mg/L Cu | 120 | 8.7 | 0.7 | 0.7 | - | - | 10.0 | - | 150 |
| CND-C4 | $SO_2$: $CN_{WAD}$ 10:1 & 150 mg/L Cu | 120 | 8.5 | 0.5 | 0.5 | - | - | 10.0 | - | 150 |
| CND-C5 | $SO_2$: $CN_{WAD}$ 10:1 & 150 mg/L Cu | 120 | 8.5 | 0.7 | 0.5 | 0.07 | 0.04 | 10.0 | - | 150 |

Source: BaseMet, 2021

The retention time was increased to 120 minutes from the initial 60 minutes as a first step. The initial ratio of $SO_2$:$CN_{WAD}$ ratio of 5.0:1 results was increased to 10.0:1 to achieve the required discharge concentration. The discharge target also required a copper concentration of 150 mg/L $Cu^{2+}$.

Due to the low initial $CN_{WAD}$ concentration, minimal lime additions were required during the test runs. The additions were low enough to make measurements difficult. A nominal value of 0.48 kg/t of hydrated lime will be included in the process operating costs.

The Master Composite cyanide destruction testing included the following conditions to achieve the $CN_{WAD}$ target of 0.5 mg/L.

- Pulp density = 40% solids (w/w);
- Pulp pH = 8.4 (maintained);
- Retention time = 120 minutes;
- Dissolved oxygen concentration = 6.0 mg/L minimum;
- $SO_2$:$CN_{WAD}$ addition rate = 10.0:1 equivalent;
- Copper addition = 150 mg/L $Cu^{2+}$

### 13.3.6 Arsenic Precipitation

The elevated arsenic concentrations in the samples tested from arsenopyrite (and in the PEA testing) result in soluble arsenic concentrations in the tailings solution that require treatment. Iron compounds, such as ferric sulphate, are commonly used for the removal of soluble metals such as arsenic. The precipitation of arsenic with iron results in stable ferric compounds (arsenates and hydroxides) that are suitable for long term disposal when the Fe:As molar ratio is at least 4:1 (3:1 weight basis) with a pH > 5. The treatment is commonly done on wastewater streams but can also be employed on tailings slurry streams. The strong oxidizing conditions in the $SO_2$/air process and pH 8.5 provide good conditions for the precipitation of arsenic with iron.

#### 13.3.6.1 Master Composite Arsenic Precipitation

This program included testing to achieve a soluble arsenic concentration of 0.5 mg/L. This was completed using a single stage treatment with a retention of 30 minutes with a feed concentration of 26 mg/L As. Ferric sulphide was dosed based on an 8:1 Fe to As ratio to produce this result which is consistent with typical industry practices.

### 13.3.7 Solid Liquid Separation

Solid/liquid separation testwork was performed on Master Composite detoxified final leach tailings. For the detox tailings sample, both static and dynamic tests were performed.

Dynamic settling tests were conducted to determine thickener sizing parameters for the project. Feed characterization is presented in Table 13-7.

**Table 13-7: Master Composite Thickener Feed Sample Characterization**

| Parameter | Units | Value |
|---|---|---|
| Solids SG | - | 2.82 |
| 80% Passing Size | µm | 100 |

Source: BaseMet, 2021

Magnafloc 10 (MF10) flocculant was selected for dynamic settling tests. These tests were all performed initially targeting a natural pH and using 15% solids (w/w) concentration for the feed slurry. The pH was increased to 10 to improve overflow clarity (reduced turbidity). Table 13-8 presents the results obtained. The highest underflow density achieved was 62.8% solids (w/w), with a settling rate of 0.5 t/m²/h at natural pH and 40 g/t flocculant addition, however the resulting overflow clarity was too high. The required underflow density for tailings deposition is 60% solids (w/w). The conditions that meet this requirement and provide acceptable overflow clarity provide a settling rate of 0.7 t/m²/h with 50 g/t flocculant addition and at pH 10 requiring 470 g/t lime addition.

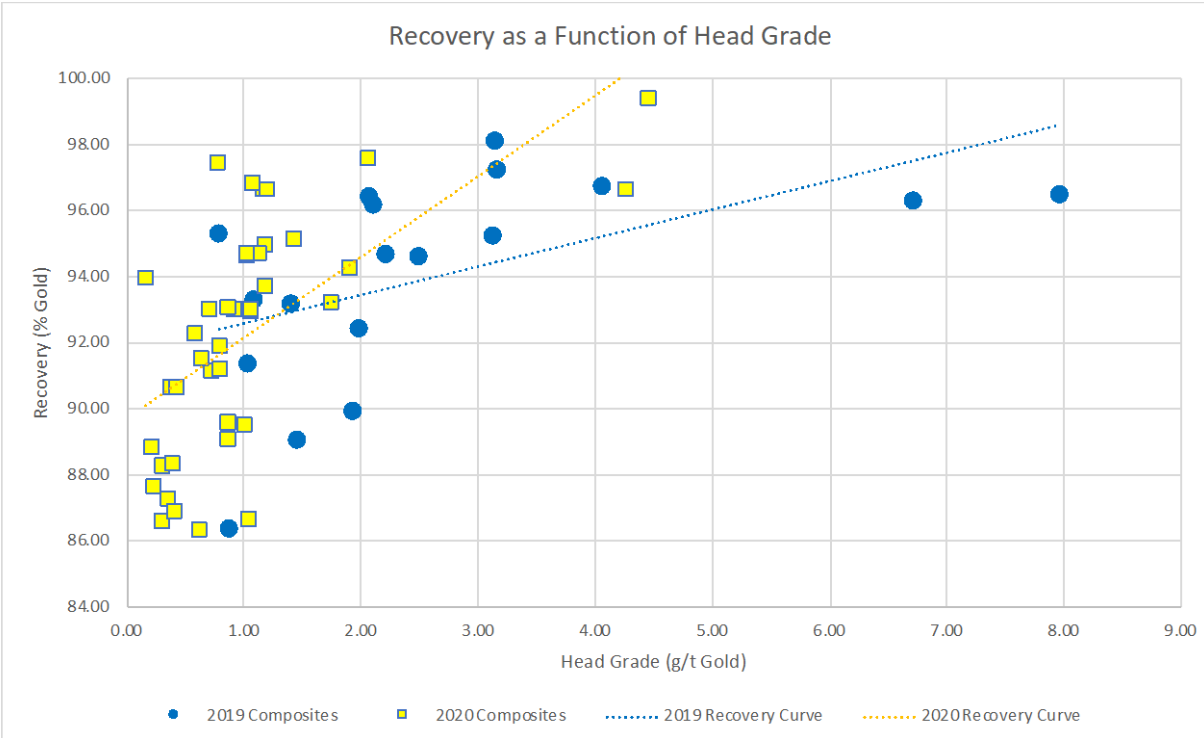
**Table 13-8: Master Composite Dynamic Settling Test Results**

| Parameter | Test No. | | | | | |
|---|---|---|---|---|---|---|
| | D1-A | D1-B | D1-C | D1-D | D1-E | D1-F |
| Unit Area (t/m²/h) | 0.5 | 0.7 | 1.0 | 0.7 | 0.7 | 0.5 |
| Rise Rate (m/h) | 3.1 | 4.4 | 6.2 | 4.4 | 4.4 | 3.1 |
| Flocculant Dosage (g/t) | 40 | 40 | 40 | 50 | 50 | 50 |
| pH | 8.1 | 8.1 | 8.1 | 8.1 | 10 | 10 |
| Underflow Density (% solids w/w) | 62.8 | 60.5 | 56.8 | 61.7 | 60.6 | 60.7 |
| Overflow Clarity (mg/L) | 756 | 849 | 570 | 405 | 231 | 95 |

Source: BaseMet, 2021

## 13.4 Conditions from Previous Testwork

The 2019 and 2020 leach and gravity test recoveries are shown in Figure 13-3. The results show the results generally follow the same relationship. The 2020 samples provide more definition at lower grades expected for the open pit resource. The 2020 results were used to model recovery.




Source: Ausenco, 2021

*Figure 13-3: Recovery head grade model*

The open pit leach test results were analyzed to provide a recovery model for use with the mine production schedule to provide gold recovery and production data. Test results for tests at the target grind of 80% passing 100 µm were included (as the previous underground study) in the modeling exercise for all composite samples. Leach residue assays at 36 hours as a function of calculated head grades formed the basis for the model. Kinetic samples with calculated extraction over 100% were not used.

In addition to the predicted extraction, plant losses including:

- Soluble losses of 0.010 g/t Au for head grades > 2.0 g/t Au.
- Carbon losses of 40 g/t.
- Fine carbon assays of 80 g/t Au for carbon losses.
- Other plant losses of 0.2% Au

Using the model shown based on the recovery as a function of head grade including plant losses of 0.83% Au the following equation is used to predict plant gold recovery:

$$(GRADE - (0.0262 * LN(GRADE) + 0.0712))/GRADE * 100 - 0.083$$

# 14. MINERAL RESOURCE ESTIMATE

## 14.1 Introduction

In 2020, Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project.

The current Mineral Resource Estimate is the result of refinements to the geological interpretations of the Deposit and the resultant mineralized wireframes, changes to the search orientation strategy, recognition of the importance of low-grade mineralization not previously captured in earlier models and additional drilling completed in 2021.

Detailed wireframing was completed based on plan and section-oriented sections to mirror likely mining patterns based on the Deposit geometry. Special attention was given to consistent smoothing of the wireframe linework to mimic the underlying geological controls on mineralization including geological bedding which regularly dips north, and the south limbs of the large-scale anticlinal fold geometry and down the plunge of the anticline.

The search orientation strategy determined to be most representative of the mineralization at the Deposit was to use a combination of an overall search ellipsoid for each domain and to allow dynamic anisotropy in the estimation process. Dynamic anisotropy is a search adjustment applied to estimation, which considers local variation of the wireframe orientation. The dynamic anisotropy approach was applied to the three Gold Systems (WG, BR, and EG Gold Systems) and adjusts the search ellipsoid on a block-to-block basis controlled by the orientation for all mineralized wireframes. Nordmin's opinion is that dynamic anisotropy allows for a much more accurate estimation of grade and mineralization due to the tightly folded nature of Belts and lower-grade mineralized zones.

In 2021, the Company completed 74 infill drill holes (10,145.4 m) across these three domains. Nordmin used these infill drill holes and reviewed the lithological, structural, spatial mineralization controls and grade variances within the Project. The review concluded that each Gold System required updated modelling of many of the tightly folded Higher-Grade Belts and the surrounding Lower-Grade Domains. In previous modelling, the significance of the Lower-Grade Domains was not thoroughly understood nor included within previous resource estimates. The combination of Higher-Grade Belts and Lower-Grade Domains and the resultant Mineral Resource Estimate supports various surface and underground mining methods.

## 14.2 Drill Hole Database

The Mineral Resource Estimate was estimated from the main database comprised of 681 DDHs consisting of 121,550 m completed between 1984 and the effective date of November 15, 2021, including 55,803 m completed by Anaconda, as well as 1,230 chip samples comprised of 822.7 m from the 2018 to 2019 Bulk Sample. The drill holes used in the Mineral Resource Estimate are displayed in Figure 14-1 and Figure 14-2 and demonstrated in Table 14-1.

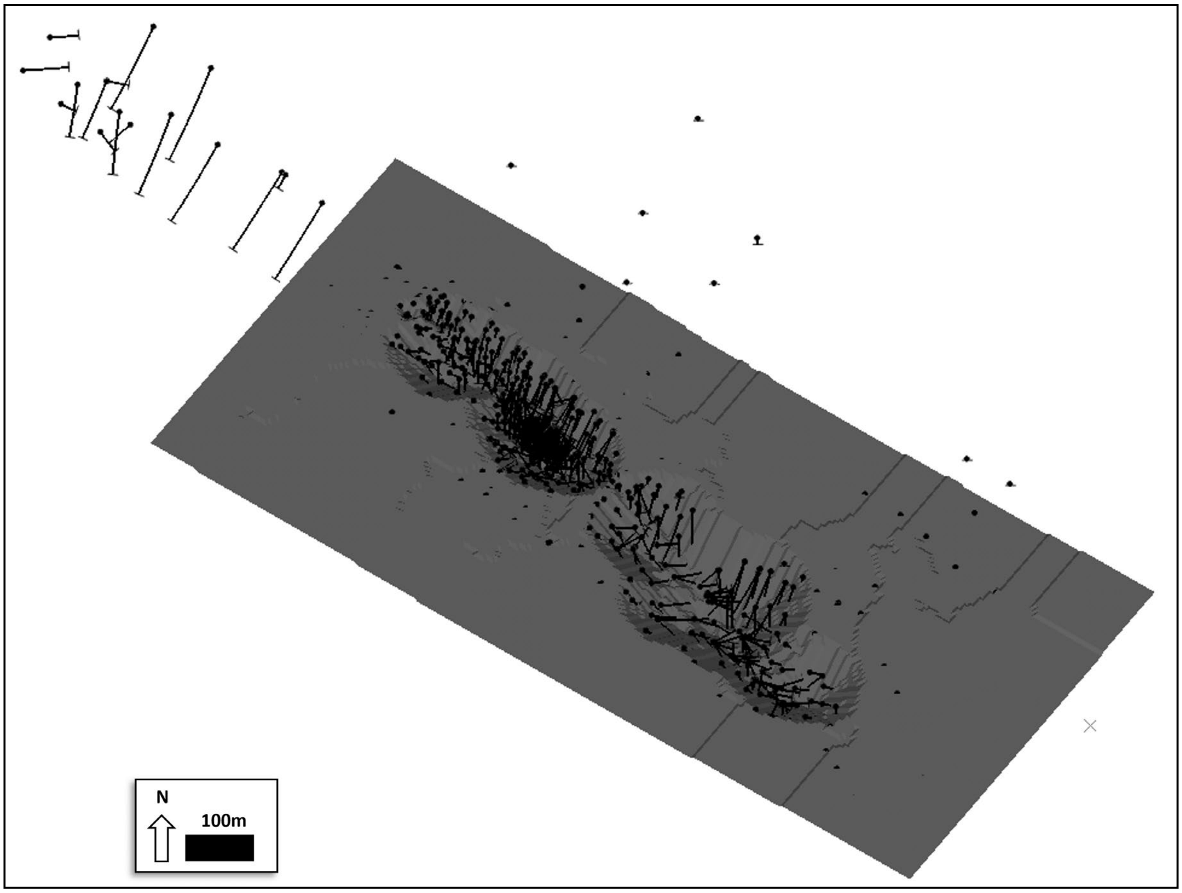



*Figure 14-1: Project drilling overview, plan section (Source: Nordmin 2021)*

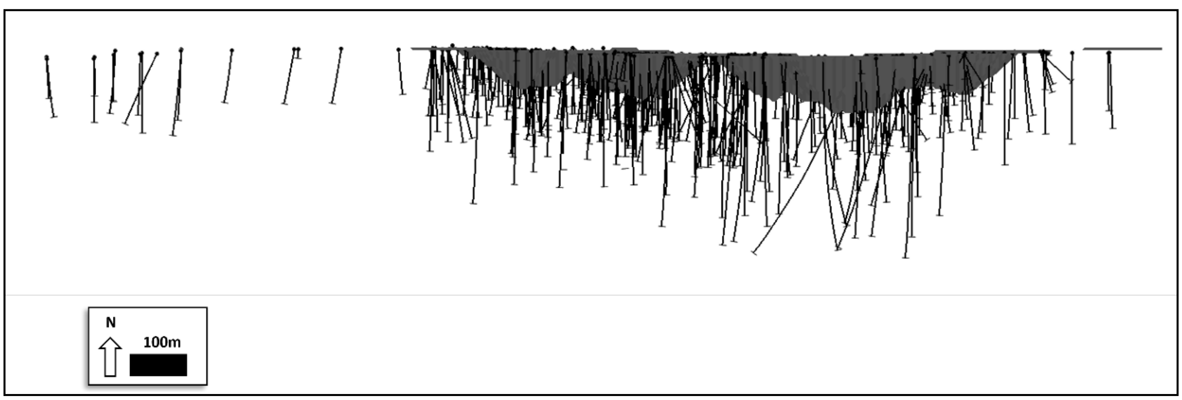



*Figure 14-2: Project drilling overview, long section, looking north (Source: Nordmin 2021)*

**Table 14-1: Diamond Drilling and Chip Sampling**

| | Diamond Drilling | | Chip Sampling | |
|---|---|---|---|---|
| **Year** | **Count** | **Length (m)** | **Count** | **Length (m)** |
| Unknown | 4 | 596.6 | - | - |
| 1984 | 1 | 529.6 | - | - |
| 1985 | 5 | 389.9 | - | - |
| 1987 | 40 | 13,545.7 | - | - |
| 1988 | 50 | 11,990.4 | - | - |
| 1989 | 39 | 2,571.8 | - | - |
| 1990 | 86 | 3,981.3 | - | - |
| 1991 | 5 | 722.4 | - | - |
| 1993 | 6 | 593.1 | - | - |
| 1995 | 7 | 1,262.9 | - | - |
| 2005 | 23 | 2,422.3 | - | - |
| 2008 | 44 | 12,064.5 | - | - |
| 2010 | 59 | 12,997.6 | - | - |
| 2011 | 10 | 2,375.6 | - | - |
| 2017 | 13 | 4,196.3 | - | - |
| 2018 | 61 | 18,277.3 | 1,230 | 822.7 |
| 2019 | 33 | 5,733.8 | | |
| 2020 | 121 | 17,941.7 | - | - |
| 2021 | 71 | 9,653.9 | - | - |
| **TOTAL** | **681** | **121,549.6** | **1,230** | **822.7** |

Gold assays exist for 103,880 samples from 682 DDHs, and a full ICP suite exists for 18,247 samples from 153 drill holes. A total of 1,230 gold assays from chip samples are available. All historical assays included in the Mineral Resource Estimate have been reviewed and validated based upon the available information. Table 14-2 summarizes the drill holes and samples used in the Mineral Resource model.

**Table 14-2: DDH Database Summary**

| | **Overall** |
|---|---|
| **Number of Drill Holes** | 681 |
| **Number of Survey Records** | 3,563 |
| **Number of Gold Assay Records** | 71,209 |
| **Number of ICP Assay Records** | 32,311 |
| **Number of Lithology Records** | 31,338 |
| **Number of Chip Samples** | 1,230 |

## 14.3 Geological Domaining

In 2020, Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project. Gold mineralization at the Project occurs in both quartz veins and within argillite, which hosts the veins, and within the rocks adjacent to the modelled argillites and quartz veins, including argillite with greywacke. Disseminated, euhedral arsenopyrite is pervasively associated with gold mineralization and is commonly observed within the host rock and is usually present in mineralized quartz veins. Wall rock generally contains more pyrrhotite and arsenopyrite than directly associated quartz veins.

The Deposit consists of three domains referred to as the BR, EG, and WG Gold Systems. The WG Gold System is separated from the BR Gold System by a north trending, near vertical fault with tens of metres of apparent offset. The EG Gold System is separated from the BR Gold System by a thick greywacke sequence or marker unit. Stratigraphic younging is from west to east with the anticlinal fold plunging shallowly to the east.

From a modelling perspective, each of the Deposit Gold Systems was separated into its own domain. Each domain was further sub-domained into Higher-Grade Belts and Lower-Grade Domains.

The review concluded that each Gold System required extensive remodelling of the tightly folded Higher-Grade Belts and the creation of surrounding Lower-Grade Domains. Previously, the significance of the Lower-Grade Domains was not thoroughly understood nor included within previous resource estimates. The combination of Higher-Grade Belts and Lower-Grade Domains and the resultant Mineral Resource Estimate supports various surface and underground mining methods.

Detailed wireframing was completed based on plan-oriented sections to mirror likely mining patterns based on the geometry of the Deposit. Special attention was given to consistent smoothing of the wireframe linework to mimic the underlying geological controls on mineralization, including geological bedding, regularly dipping north, and south limbs of the large-scale anticlinal fold geometry and down the plunge of the anticline. Historical workings of three underground mines, which traced the outline of the fold geometry down the fold plunge and along anticlinal limbs coincident with gold mineralization were also used to orient wireframes. Wireframes were created between 10 m to 25 m cross sections and joined section to section resulting in irregular geometries and plunge lines not representative of the underlying geology. All wireframes are independent of each other without overlap across wireframes or across domains.

Explicit modelling was used to create the Mineral Resource, which allows for mineralization to better reflect the Deposit geology and associated geochemistry. Nordmin's opinion is that the explicit modelling approach minimizes risks compared to the use of implicit modelling for the Project.

In 2021, the Company completed 74 infill drill holes (10,145.4 m) across these three domains. Nordmin utilized these infill drill holes and reviewed the lithological, structural, spatial mineralization controls and grade variances within the Project, updating and editing the model as appropriate.

The modelled domain wireframes can be seen in Figure 14-3 and Figure 14-4, and the wireframes are summarized in Table 14-3.

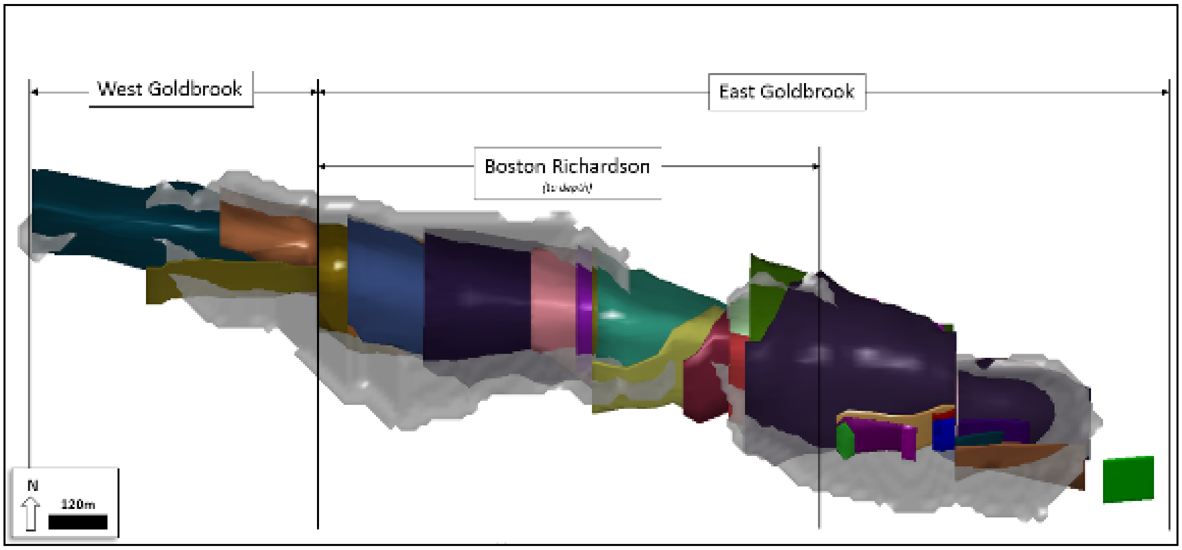



*Figure 14-3: Project domaining, plan view (Source: Nordmin 2021)*

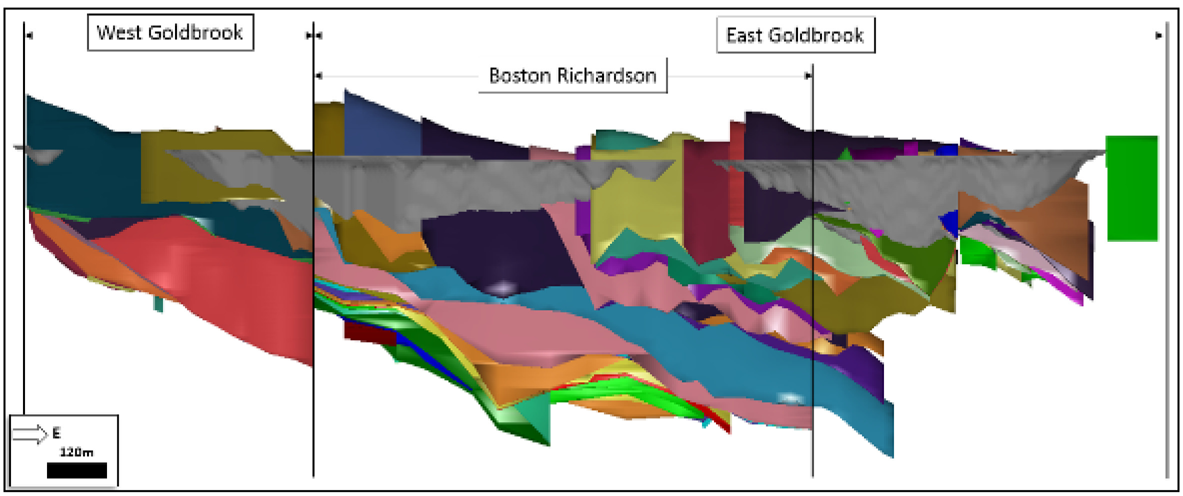



*Figure 14-4: Project domaining, long section (Source: Nordmin 2021)*

**Table 14-3: Domaining**

| Domain | Belt Count | Higher-Grade Belts | Lower-Grade Domains |
|---|---|---|---|
| WG | 17 | 1,2,3,3a,3b,4,5,5a,5b,6,6 a,6b,7,8,9,10,11 | Lower-Grade North and South |
| BR | 16 | 1,2,3,3 a,4,4a,5,5a,6,7,8,9,10,11,12,13 | Lower-Grade North and South |
| EG | 34 | 1,2,3,4,4 a,4b,5,5a,6,7,8,9,10,10 a,11,11 a, 12,12 a,13,14,14a,15,15a,15b,16,17,18, 19,20,21,22,23,24,25 | Lower-Grade North and South |
| Marker Horizon | 1 | 1 | |

The Higher-Grade Belt wireframes were modelled using the following criteria:

- An approximate CoG of 1.5 g/t gold.
- Structural model: structural trends were observed while developing the model.
- Geology model and lithological boundaries: wireframes were permitted to follow lithological boundaries and trends where appropriate.

The Lower-Grade Domain wireframes were modelled using the following criteria:

- An approximate CoG of approximately 0.1 g/t gold.
- Structural model: Structural trends were observed while developing the model.
- Geology model and lithological boundaries: wireframes were permitted to follow lithological boundaries and trends where appropriate.
- Each domain has two lower-grade zones that are split along the axis of the antiform.

Wireframes were primarily created on 5 m to 25 m sections depending on drill density and the geographical location of existing hand-drawn sections and were adjusted on plan views to edit and smooth each wireframe where required. When not cut-off by drilling, the wireframes terminate at plunge and depth. No wireframe overlapping exists, including within or across domains, and all domains are independent of each other (Figure 14-5).

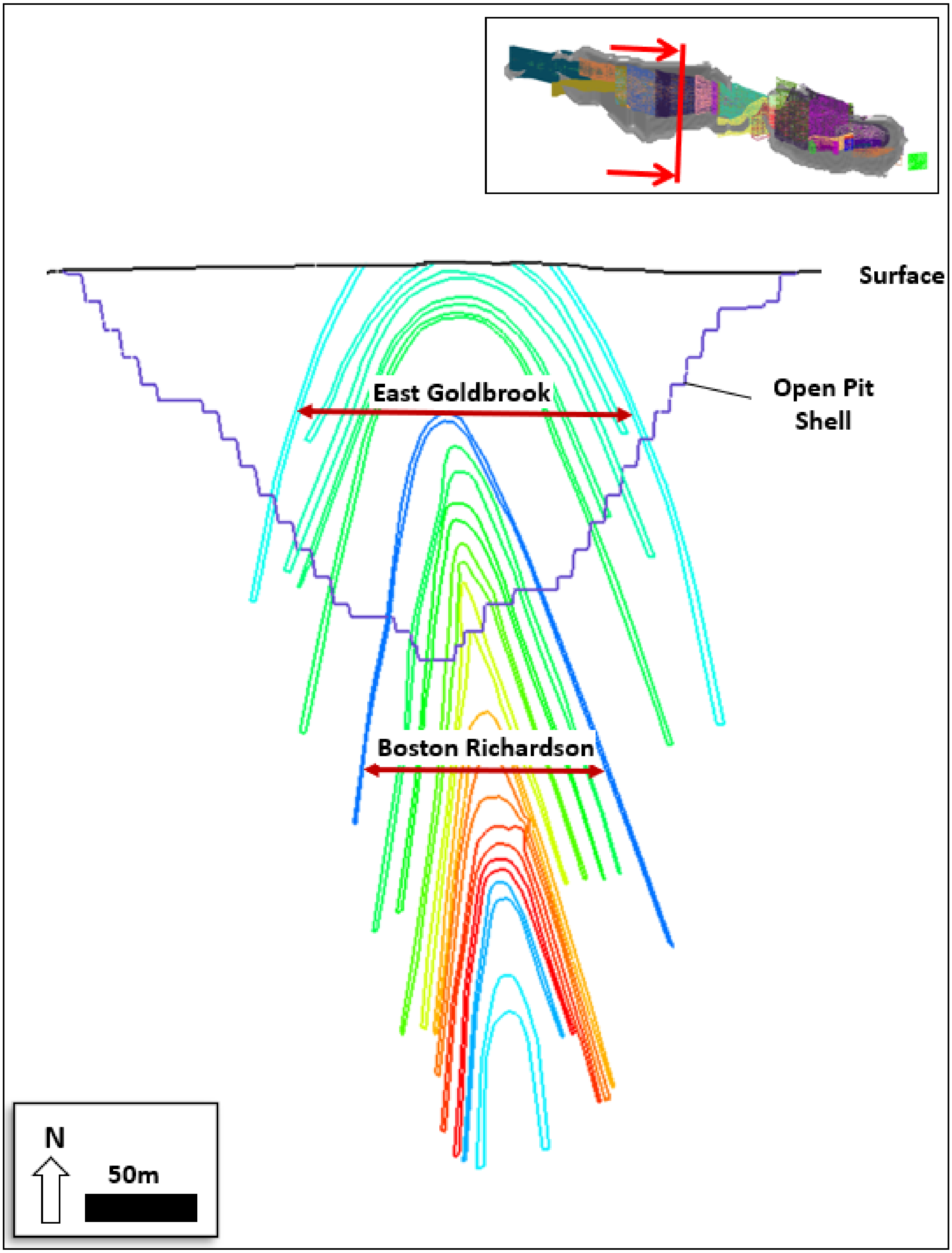



*Figure 14-5: Project domaining and belt wireframing, cross section (Source: Nordmin 2021)*

Regional and local structures were considered in addition to grade during the creation of the wireframes. Fault wireframes were built/extrapolated from surface and underground mapping, and belt wireframes were allowed to follow these structures where appropriate.

Three main fault structures exist within the Deposit. The WG Fault defines the separation between the WG Gold System to the west and the BR and EG Gold Systems to the east. The New Belt Fault affects Belts within the BR and EG Gold Systems, and the EG Fault affects the EG Domain (Figure 14-6 and Figure 14-7).

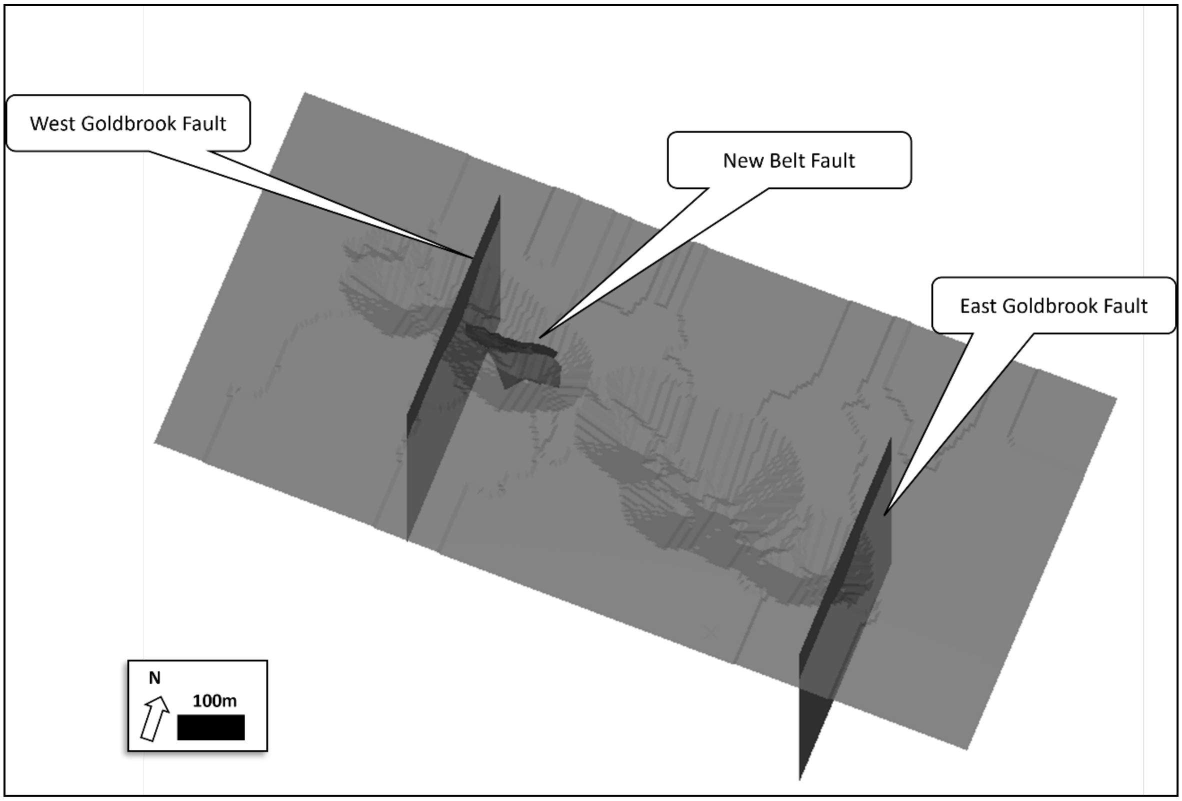



*Figure 14-6: Structural components, plan view (Source: Nordmin 2021)*

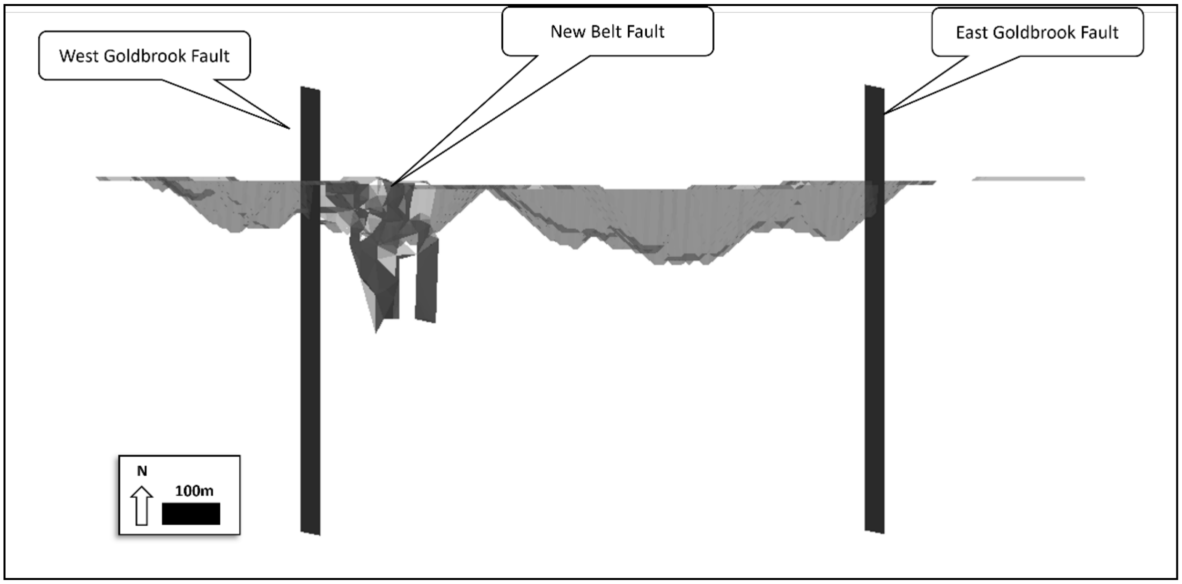



*Figure 14-7: Structural components, long section (Sourece: Nordmin 2021)*

The New Belt Fault is located within the centre of the proposed west open pit. It primarily affects the BR Belts along with a few of the select EG Belts. The offset of the fault was incorporated into the belt wireframes, and was identified via section maps, fault modelling, and assays (Figure 14-8).

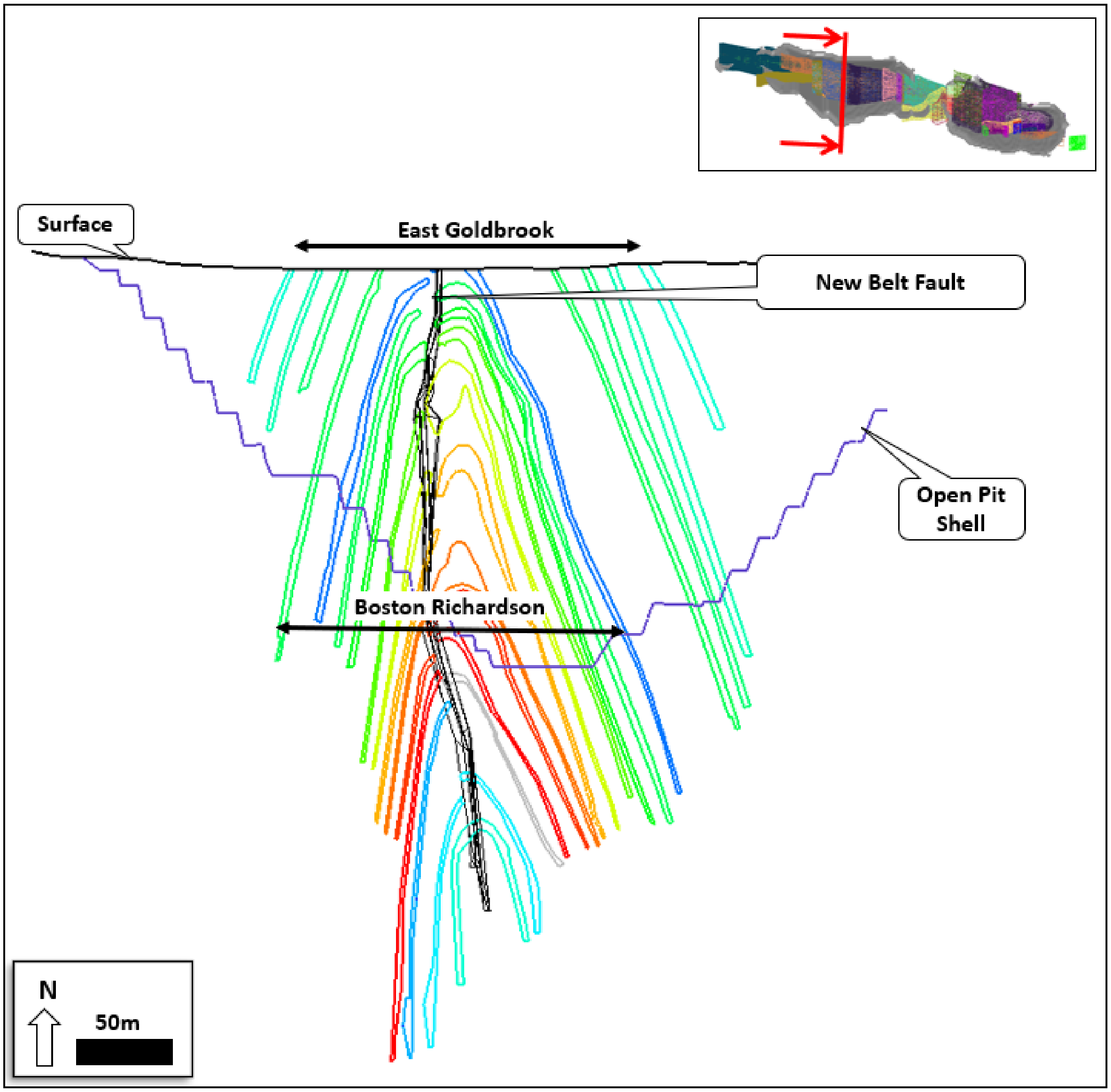



*Figure 14-8: New Belt Fault (Source: Nordmin 2021)*

## 14.4 Exploratory Data Analysis

The exploratory data analysis was conducted on raw drill hole data to determine the nature of the gold distribution, correlation of grades within individual rock units, and the identification of high-grade outlier samples. Nordmin used a geostatistical package (X10 Geo) to complete various descriptive statistics, histograms, probability plots, and XY scatter plots to analyze the grade population data. The findings of the exploratory data analysis were used to help define modelling procedures and parameters used in the Mineral Resource Estimate.

Descriptive statistics were used to analyze the grade distribution of each sample population, determine the presence of outliers, and identify correlations between grade and rock types for each mineral belt and low-grade zone.

Several drill holes were removed from the flagging process due to inaccurate and unrecoverable interval lengths with elevated grades (Table 14-4).

**Table 14-4: Removed Drill Holes**

| Drill Holes Removed |
|---|
| BR89-83 |
| BR89-85 |
| BR89-91 |
| BR89-92 |
| BR89-93 |
| BR89-94 |
| BR89-97 |
| BR89-104 |
| BR89-105 |

An excerpt from drill holes BR89-85 and BR89-95 is shown in Table 14-5.

**Table 14-5: Excerpt from Drill Holes with Inappropriate Sample Lengths**

| Hole | From (m) | To (m) | Length (m) | Au (g/t) |
|---|---|---|---|---|
| BR89-85 | 38.9 | 144.86 | 105.96 | 10.63 |
| BR-89-95 | 5.3 | 13.49 | 8.19 | 0.04 |
| BR-89-95 | 13.49 | 15.56 | 2.07 | 0.16 |
| BR-89-95 | 15.56 | 19.9 | 4.34 | 0.06 |
| BR-89-95 | 19.9 | 22.58 | 2.68 | 1.13 |
| BR-89-95 | 22.58 | 34.81 | 12.23 | 0.08 |
| BR-89-95 | 34.81 | 35.66 | 0.85 | 10.59 |
| BR-89-95 | 35.66 | 46.72 | 11.06 | 0.1 |
| BR-89-95 | 46.72 | 48.68 | 1.96 | 0.14 |
| BR-89-95 | 48.68 | 64.33 | 15.65 | 0.13 |
| BR-89-95 | 64.63 | 68.14 | 3.51 | 0.51 |
| BR-89-95 | 68.14 | 97.29 | 29.15 | 0.11 |
| BR-89-95 | 97.29 | 104.67 | 7.38 | 8.17 |

Individual drill hole tables (collar, survey, assay, etc.) were merged to create one single master drill hole file. The process splits assay intervals to allow for all records in all tables to be included. Values in Table 14-6, Table 14-7, Table 14-8 and Table 14-9 are based on analysis of this master file; counts will differ when compared with the original data.

**Table 14-6: WG Domain, Assays by Domain and Belt, Drill Holes, and Chips**

| | | Drill Holes | | | Underground Chips | | |
|---|---|---|---|---|---|---|---|
| Domain | Belt | Sample Count | Au Sample Count | ICP Sample Count | Sample Count | Au Sample Count | ICP Sample Count |
| West Goldbrook | 1 | 75 | 75 | 30 | - | - | - |
| | 2 | 245 | 245 | 130 | - | - | - |
| | 3a | 162 | 162 | 106 | - | - | - |
| | 3b | 25 | 25 | 323 | - | - | - |
| | 4 | 181 | 181 | 86 | - | - | - |
| | 5 | 25 | 25 | 3 | - | - | - |
| | 5a | 300 | 300 | 184 | - | - | - |
| | 5b | 10 | 10 | - | - | - | - |
| | 6 | 15 | 15 | 9 | - | - | - |
| | 6a | 412 | 412 | - | - | - | - |
| | 6b | 60 | 60 | 31 | - | - | - |
| | 7 | 170 | 170 | 61 | - | - | - |
| | 8 | 145 | 145 | 61 | - | - | - |
| | 9 | 66 | 66 | 37 | - | - | - |
| | 10 | 2 | 2 | - | - | - | - |
| | 11 | 15 | 15 | 6 | - | - | - |
| | Low-Grade | 165 | 12,302 | 3,556 | - | - | - |

**Table 14-7: BR Domain, Assays by Domain and Belt, Drill Holes, and Chips**

| | | Drill Holes | | | Underground Chips | | |
|---|---|---|---|---|---|---|---|
| Domain | Belt | Sample Count | Au Sample Count | ICP Sample Count | Sample Count | Au Sample Count | ICP Sample Count |
| Boston Richardson | 1 | 1015 | 1018 | 196 | 90 | 90 | - |
| | 2 | 1498 | 1498 | 294 | 303 | 303 | - |
| | 3 | 1920 | 1920 | 334 | 31 | 31 | - |
| | 3a | 111 | 111 | 21 | - | - | - |
| | 4 | 1807 | 1807 | 383 | - | - | - |
| | 4a | 228 | 228 | 41 | - | - | - |
| | 5 | 1173 | 1173 | 223 | - | - | - |
| | 5a | 62 | 62 | 10 | - | - | - |
| | 7 | 669 | 669 | 147 | - | - | - |
| | 8 | 350 | 350 | 120 | - | - | - |
| | 9 | 227 | 227 | 103 | - | - | - |
| | 10 | 102 | 102 | 62 | - | - | - |
| | 11 | 95 | 95 | 69 | - | - | - |
| | 12 | 46 | 46 | 32 | - | - | - |
| | 13 | 19 | 19 | 19 | - | - | - |
| | Low-Grade | 17,167 | 17,167 | 3,346 | 165 | 165 | - |

**Table 14-8: EG Domain, Assays by Domain and Belt Drill Holes, and Chips**

| Domain | Belt | Drill Holes: Sample Count | Drill Holes: Au Sample Count | Drill Holes: ICP Sample Count | Underground Chips: Sample Count | Underground Chips: Au Sample Count | Underground Chips: ICP Sample Count |
|---|---|---|---|---|---|---|---|
| East Goldbrook | 1 | 6 | 6 | 3 | - | - | - |
| | 2 | 11 | 11 | 8 | - | - | - |
| | 3 | 10 | 10 | 4 | - | - | - |
| | 4 | 179 | 179 | 136 | - | - | - |
| | 4a | 28 | 28 | 24 | - | - | - |
| | 4b | 58 | 58 | 43 | - | - | - |
| | 5 | 230 | 230 | 190 | - | - | - |
| | 5a | 180 | 180 | 128 | - | - | - |
| | 6 | 168 | 168 | 110 | - | - | - |
| | 7 | 177 | 177 | 103 | - | - | - |
| | 8 | 195 | 195 | 100 | - | - | - |
| | 9 | 423 | 423 | 169 | - | - | - |
| | 10 | 333 | 333 | 131 | - | - | - |
| | 10a | 91 | 91 | 31 | - | - | - |
| | 11 | 353 | 353 | 145 | - | - | - |
| | 11a | 238 | 238 | 92 | - | - | - |
| | 12 | 393 | 393 | 167 | - | - | - |
| | 12a | 46 | 46 | 5 | - | - | - |
| | 13 | 539 | 539 | 198 | - | - | - |
| | 14 | 614 | 614 | 180 | 1 | 1 | - |
| | 14a | 473 | 473 | 222 | | | |
| | 15 | 763 | 763 | 339 | - | - | - |
| | 15a | 309 | 309 | 111 | - | - | - |
| | 15b | 318 | 318 | 104 | 6 | 6 | - |
| | 16 | 405 | 405 | 162 | 5 | 5 | - |
| | 17 | 8 | 8 | 8 | - | - | - |
| | 18 | 7 | 7 | 7 | - | - | - |
| | 19 | 25 | 25 | 25 | - | - | - |
| | 20 | 21 | 21 | 21 | - | - | - |
| | 21 | 6 | 6 | 4 | - | - | - |
| | 22 | 9 | 9 | 9 | - | - | - |
| | 23 | 1 | 1 | - | - | - | - |
| | 24 | 2 | 2 | 2 | - | - | - |
| | 25 | 4 | 4 | 4 | - | - | - |
| | Low-Grade | 29,829 | 29,829 | 4,404 | 52 | 52 | 29,829 |
| **Marker Horizon** | | 5,323 | 5,323 | 1,153 | 95 | 95 | - |

**Table 14-9: Drill Hole and Chip Assay Count Summary**

| Sample Type | Count Type | Total |
|---|---|---|
| Drill Hole Assays | Drill Hole Sample Count | 71,208 |
| | Au Sample Count | 71,208 |
| | ICP Sample Count | 32,311 |
| Chip Samples | Chip Sample Count | 1230 |
| | Au Sample Count | 1230 |
| | ICP Sample Count | 0 |

Figure 14-9 through Figure 14-11 outlines the histogram, probability plot, and box plot for each domain and the corresponding subdomains. The complete data analysis is available in Appendix C.

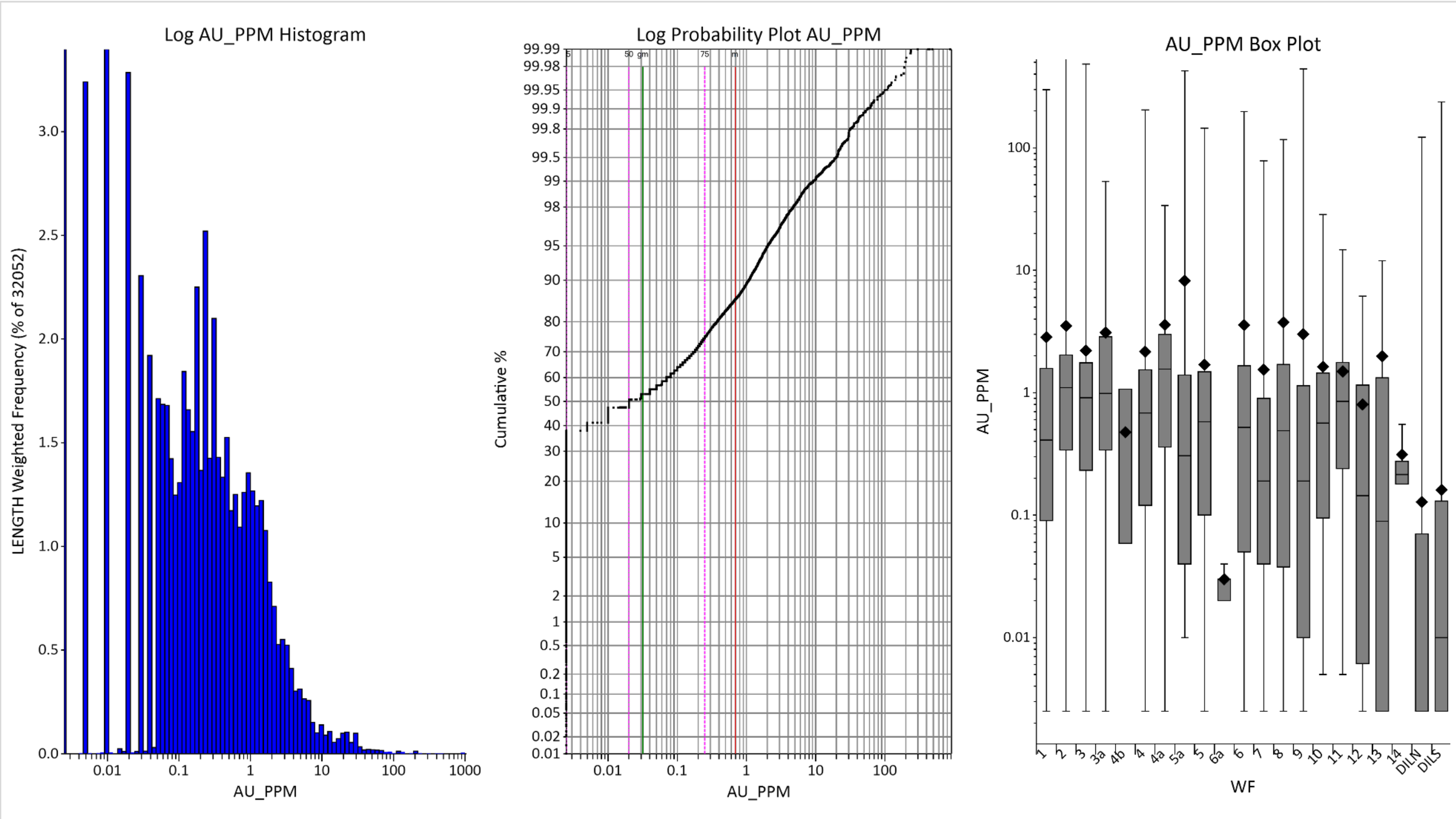



*Figure 14-9: Histogram, probability plot and box plot for the BR Domain and corresponding subdomains*

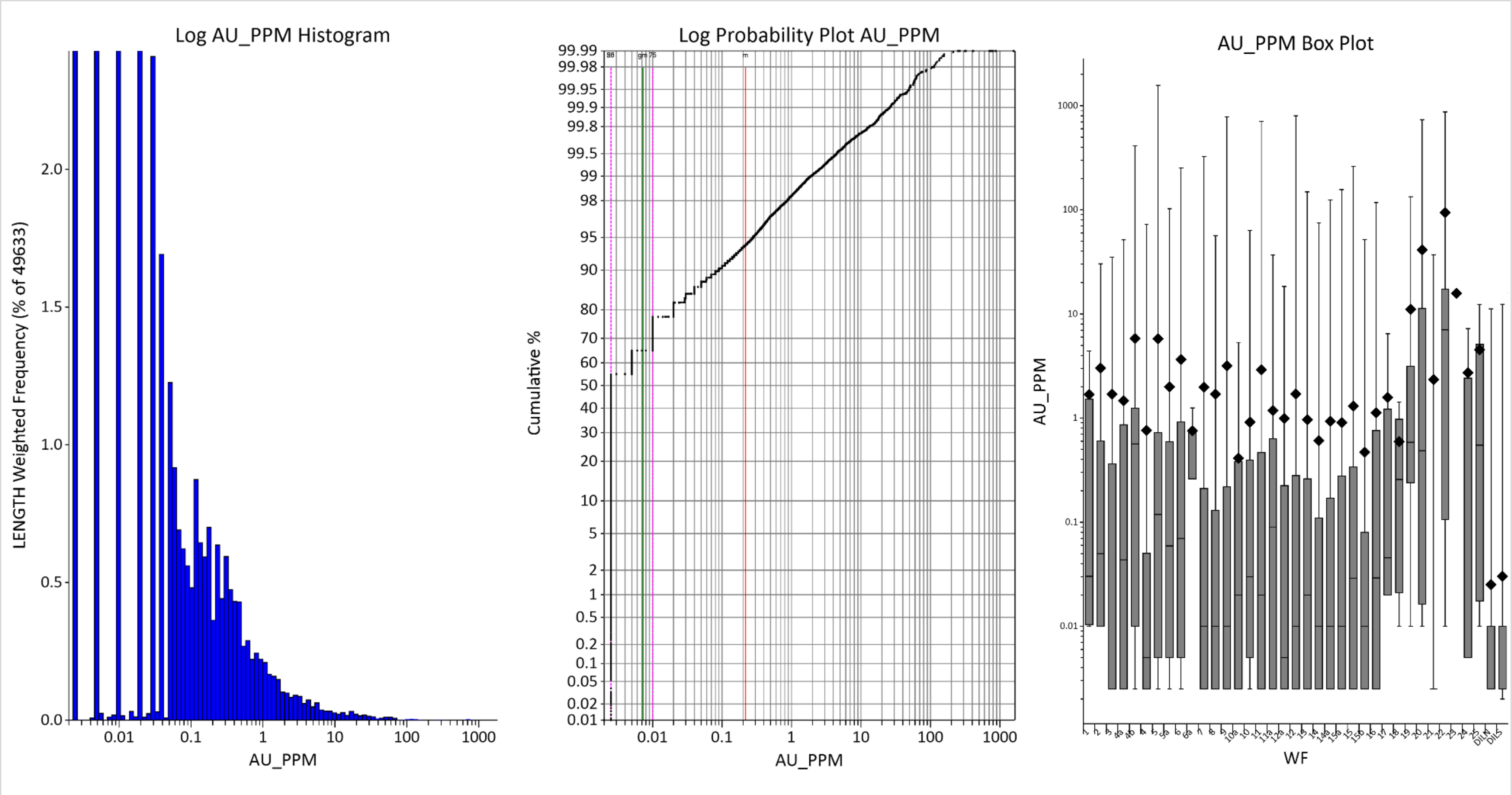



*Figure 14-10: Histogram, probability plot and box plot for the EG Domain and corresponding subdomains*

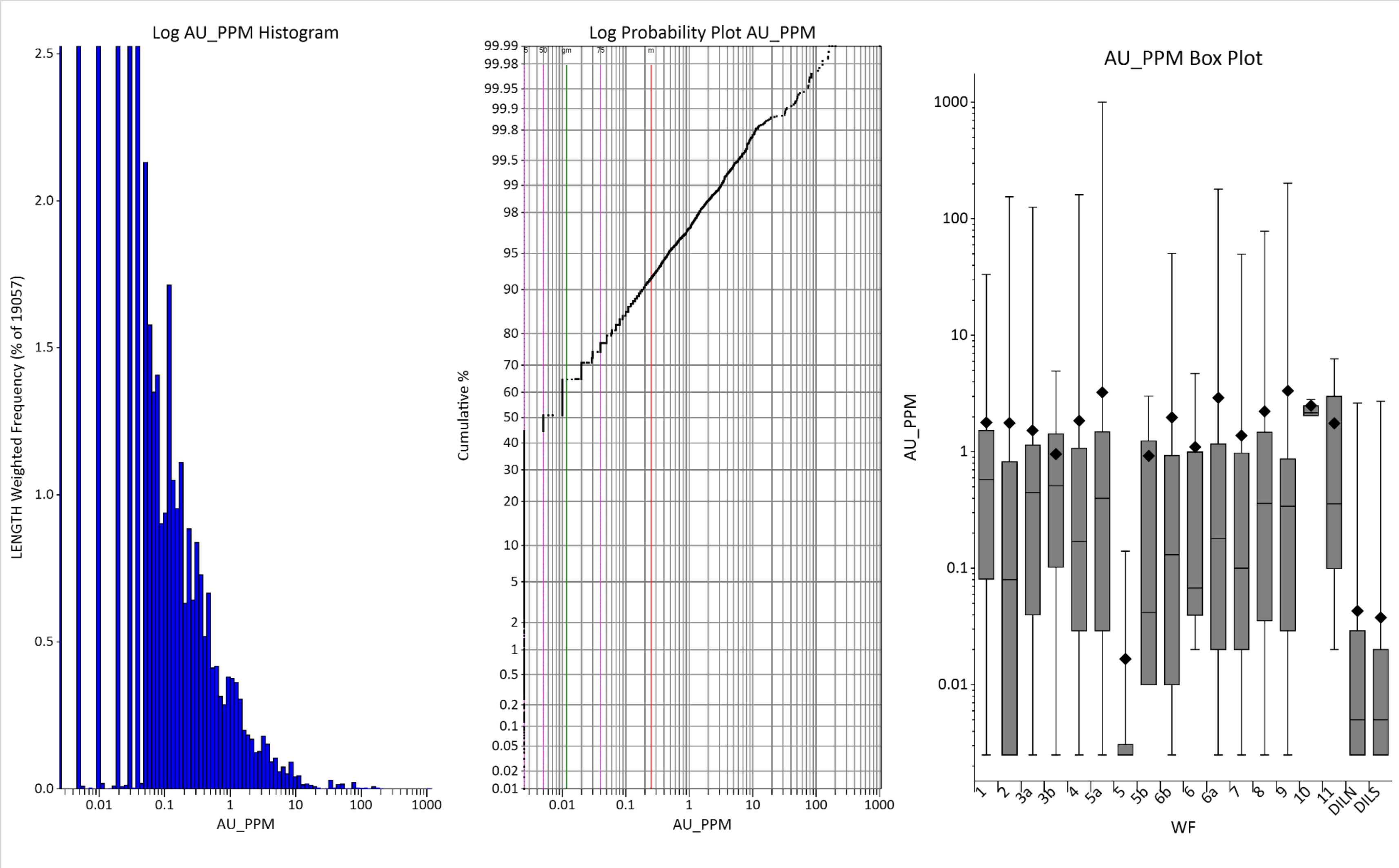



*Figure 14-11: Histogram, probability plot and box plot for the WG Domain and corresponding subdomains*

## 14.5 Data Preparation

Prior to grade estimation, the data was prepared in the following matter for each of the three domains:

- All unsampled intervals were assigned a half-minimum detection limit value (0.0025 g/t gold).
- The raw assay data was manually "flagged" to intersecting Belts and low-grade zones by assigning codes representative to the domain and wireframe.
- Each belt and low-grade zone's flagged assays were statistically analyzed to define appropriate capping, modelling procedures, and parameters.
- High-grade outlier samples in each zone were top-cut to a variable maximum value (capped).

### 14.5.1 Non-Sampled Intervals and Minimum Detection Limits

Table 14-10 summarizes the drill hole assays at minimum detection used in the resource model. The assay table received by Nordmin contained half-minimum detection gold values substituted for assays below minimum detection. When non-assayed gold intervals exist for payable and non-payable fields, half-minimum detection values were substituted to remove bias from the block model. Values in Table 14-10 are based on the master drill hole file defined in Section 14.4.

**Table 14-10: Assays at Minimum Detection**

| Field | Count | Minimum Detection Limit | Count at Minimum Detection | % at Minimum Detection |
|---|---|---|---|---|
| Au (g/t) | 105,109 | 0.005 | 36,006 | 34.25 |
| Ag (g/t) | 41,694 | 0.1 | 26,539 | 63.6 |
| As (g/t) | 41,694 | 5 | 6,129 | 14.6 |
| Cd (g/t) | 41,694 | 0.5 | 28,684 | 68.7 |
| Cu (g/t) | 41,694 | 5 | 894 | 0.0 |
| Pb (g/t) | 41,694 | 1 | 164 | 0.4 |
| S (%) | 41,694 | 0.01 | 3,855 | 9.24 |

### 14.5.2 Outlier Analysis and Capping

Grade outliers are high-grade assay values that are much higher than the general population of samples and have the potential to bias (inflate) the quantity of metal estimated in a block model. Geostatistical analysis using XY scatter plots, cumulative probability plots, and decile analysis was used by Nordmin to analyze the raw drill hole assay data for each domain to determine appropriate grade capping. Statistical analysis was performed by the X10 Geo software package. Table 14-11, Table 14-12 and Table 14-13 are summaries of the results from the capping analysis per domain.

The raw assay data was manually "flagged" to intersecting Belts and low-grade zones. Each belt and low-grade zone's flagged assays were statistically analyzed to define appropriate capping, modelling procedures, and parameters. Nordmin reviewed the previous historical estimate capping method and determined the global 80 g/t gold cap was not representative of the gold distribution for the entire Deposit. Therefore, the assays were variably capped to prevent excessive high-grade from skewing the estimation in each wireframe. The overall difference between an 80 g/t gold cap and a variable cap is less than 2% over the entire Deposit.

**Table 14-11: WG Domain, Outlier Analysis, and Capping**

| | | | | | Capped | | | | | | | Uncapped | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Belt | Metal | Cap (g/t) | # of Samples | Min | Max | Mean | # Capped | % Capped | % Metal Lost | CV | Min | Max | Mean | CV |
| West Goldbrook | 1 | Au | No cap | 77 | 0.003 | 33.38 | 1.78 | 0 | 0.00% | | 2.35 | 0.0025 | 33.38 | 1.78 | 2.35 |
| | 2 | Au | 100.00 | 321 | 0.003 | 154.5 | 1.67 | 3 | 0.9 | 5.2 | 5.05 | 0.003 | 154.50 | 1.76 | 5.51 |
| | 3 a | Au | 80.00 | 221 | 0.003 | 80 | 1.39 | 1 | 0.5% | 8.3 | 3.67 | 0.003 | 125.7 | 1.52 | 4.75 |
| | 3b | Au | No cap | 34 | 0.003 | 4.95 | 0.95 | 0 | 0 | 0 | 1.37 | 0.003 | 4.95 | 0.95 | 1.37 |
| | 4 | Au | 80.00 | 189 | 0.003 | 80 | 1.59 | 1 | 0.5 | 14 | 3.69 | 0.003 | 161.3 | 1.85 | 5.31 |
| | 5 | Au | No cap | 5 | 0.003 | 0.14 | 0.016 | 0 | 0 | 0 | 2.79 | 0.003 | 0.14 | 0.016 | 2.79 |
| | 5 a | Au | 100.00 | 387 | 0.003 | 100 | 1.76 | 3 | 0.8 | 46 | 3.43 | 0.003 | 1005 | 3.24 | 12.6 |
| | 5b | Au | No cap | 7 | 0.01 | 3.00 | 0.923 | 0 | 0 | 0 | 1.57 | 0.01 | 3.00 | 0.92 | 1.57 |
| | 6 | Au | No cap | 9 | 0.02 | 4.7 | 1.09 | 0 | 0 | 0 | 1.52 | 0.02 | 4.7 | 1.09 | 1.52 |
| | 6 a | Au | 120.00 | 516 | 0.003 | 120 | 2.71 | 3 | 0.6 | 7 | 4.53 | 0.003 | 180 | 2.91 | 4.97 |
| | 6b | Au | No cap | 67 | 0.003 | 50.48 | 1.97 | 0 | 0 | 0 | 3.13 | 0.003 | 50.48 | 1.97 | 3.13 |
| | 7 | Au | No cap | 171 | 0.003 | 49.8 | 1.37 | 0 | 0 | 0 | 3.81 | 0.003 | 49.8 | 1.37 | 3.81 |
| | 8 | Au | No Cap | 153 | 0.003 | 78 | 2.22 | 0 | 0 | 0 | 3.7 | 0.003 | 78 | 2.22 | 3.7 |
| | 9 | Au | 80.00 | 78 | 0.003 | 80 | 2.26 | 1 | 1.3 | 32 | 3.65 | 0.003 | 201.7 | 3.34 | 5.76 |
| | 10 | Au | No Cap | 2 | 2.047 | 2.806 | 2.49 | 0 | 0 | 0 | 0.22 | 2.047 | 2.806 | 2.49 | 0.22 |
| | 11 | Au | No Cap | 15 | 0.02 | 6.3 | 1.75 | 0 | 0 | 0 | 1.30 | 0.02 | 6.3 | 1.75 | 1.30 |
| | Lower-Grade Domain | Au | No cap | 14,222 | 0.003 | 2.717 | 0.04 | 0 | 0 | 0 | 2.76 | 0.003 | 2.71 | 0.04 | 2.76 |

**Table 14-12: BR Domain, Outlier Analysis, and Capping**

| | | | | | | Capped | | | | | | | Uncapped | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Belt | Type | Metal | Cap (g/t) | # of Samples | Min | Max | Mean | Number Capped | % Capped | % Metal Lost | CV | Min | Max | Mean | CV |
| Boston Richardson | 1 | Chip | Au | 75.00 | 86 | 0.07 | 75.00 | 9.89 | 2 | 2.3% | 45.0 | 2.14 | 0.07 | 526.67 | 10.3 | 5.05 |
| | | DDH | Au | 160.00 | 963 | 0.003 | 160 | 2.67 | 4 | 0.4 | 5.7 | 3.94 | 0.003 | 298.1 | 2.84 | 4.72 |
| | 2 | Chip | Au | 70.00 | 298 | 0.0025 | 70.00 | 4.18 | 5 | 1.65% | 41 | 2.55 | 0.0025 | 673.33 | 6.62 | 5.94 |
| | | DDH | Au | 100.00 | 1,408 | 0.003 | 100 | 2.44 | 2 | 0.1 | 30 | 2.61 | 0.003 | 876.00 | 3.5 | 9.24 |
| | 3 | Chip | Au | 95.00 | 31 | 0.15 | 28.06 | 2.81 | 0 | 0.00% | 0 | 2.11 | 0.153 | 28.07 | 2.81 | 2.11 |
| | | DDH | Au | 120.00 | 1,804 | 0.003 | 120.00 | 2.12 | 3 | 0.20 | 3.6 | 3.07 | 0.003 | 483 | 2.2 | 4.29 |
| | 3a | DDH | Au | 21.00 | 111 | 0.003 | 21.00 | 2.77 | 3 | 2.7 | 9.8 | 1.65 | 0.003 | 53.02 | 3.10 | 2.04 |
| | 4 | DDH | Au | No Cap | 1,657 | 0.003 | 204.3 | 2.16 | 0 | 0 | 0 | 3.73 | 0.003 | 204.3 | 2.16 | 3.73 |
| | 4a | DDH | Au | No cap | 183 | 0.003 | 33.7 | 3.58 | 0 | 0 | 0 | 1.72 | 0.003 | 33.7 | 3.58 | 1.72 |
| | 5 | DDH | Au | No Cap | 1,067 | 0.003 | 144.2 | 1.69 | 0 | 0 | 0 | 0 | 0.003 | 144.2 | 1.69 | 3.95 |
| | 5a | DDH | Au | 120.00 | 68 | 0.01 | 120 | 4.69 | 1 | 1.5 | 43 | 3.48 | 0.01 | 425.00 | 8.20 | 5.68 |
| | 6 | DDH | Au | No Cap | 559 | 0.003 | 198.2 | 3.56 | 0 | 0 | 0 | 5.14 | 0.003 | 198.2 | 3.56 | 5.14 |
| | 7 | DDH | Au | No Cap | 523 | 0.003 | 78.17 | 1.54 | 0 | 0 | 0 | 4.26 | 0.003 | 78.17 | 1.54 | 4.26 |
| | 8 | DDH | Au | 85.00 | 280 | 0.003 | 85.00 | 3.75 | 2 | 0.7 | 1 | 2.48 | 0.003 | 116.5 | 3.75 | 2.56 |
| | 9 | DDH | Au | 90.00 | 204 | 0.003 | 90.00 | 1.80 | 1 | 0.5 | 40 | 3.92 | 0.003 | 440.3 | 3.01 | 8.82 |
| | 10 | DDH | Au | No cap | 82 | 0.005 | 28.55 | 1.63 | 0 | 0.00% | 0 | 2.59 | 0.005 | 28.55 | 1.63 | 2.59 |
| | 11 | DDH | Au | No cap | 86 | 0.005 | 14.65 | 1.5 | 0 | 0.00% | 0 | 1.42 | 0.005 | 14.65 | 1.5 | 1.42 |
| | 12 | DDH | Au | No cap | 45 | 0.003 | 6.155 | 0.8 | 0 | 0.00% | 0 | 1.78 | 0.003 | 6.15 | 0.8 | 1.78 |
| | 13 | DDH | Au | No cap | 24 | 0.003 | 11.96 | 1.98 | 0 | 0.00% | 0 | 1.93 | 0.003 | 11.96 | 1.98 | 1.93 |
| | Lower-Grade Domain | Au | Au | 5.0 | 21,933 | 0.003 | 5.0 | 0.119 | 86 | 0.4 | 17 | 2.89 | 0.003 | 237 | 0.14 | 11.19 |

**Table 14-13: EG Domain, Outlier Analysis, and Capping**

| | | | | | | Capped | | | | | | | Uncapped | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Belt | Type | Metal | Cap (g/t Gold) | # of Samples | Min | Max | Mean | # Capped | % Capped | % Metal Lost | CV | Min | Max | Mean | CV |
| East Goldbrook | 1 | DDH | Au | No cap | 4 | 0.010 | 4.43 | 1.68 | 0 | 0.00% | | 1.55 | 0.010 | 4.43 | 1.68 | 1.55 |
| | 2 | DDH | Au | No cap | 10 | 0.010 | 30.3 | 3.03 | 0 | 0.00% | | 2.70 | 0.010 | 30.3 | 3.03 | 2.70 |
| | 3 | DDH | Au | No cap | 10 | 0.003 | 35.1 | 1.69 | 0 | 0.00% | | 4.80 | 0.003 | 35.1 | 1.69 | 4.80 |
| | 4 | DDH | Au | 50.0 | 183 | 0.003 | 50.00 | 0.76 | 1 | 0.5 | 10 | 5.30 | 0.003 | 72.40 | 0.76 | 6.29 |
| | 4a | DDH | Au | No cap | 28 | 0.003 | 51.37 | 1.46 | 0 | 0.00% | | 4.89 | 0.003 | 51.37 | 1.46 | 4.89 |
| | 4b | DDH | Au | 120.00 | 63 | 0.003 | 412 | 5.83 | 3 | 6.3 | 59 | 4.46 | 0.003 | 412 | 5.83 | 7.07 |
| | 5 | DDH | Au | 120.00 | 237 | 0.003 | 120.00 | 1.55 | 3 | 1.3 | 73 | 6.04 | 0.003 | 1570.00 | 5.76 | 14.67 |
| | 5a | DDH | Au | 100.00 | 196 | 0.003 | 100.00 | 1.98 | 1 | 0.5 | 0.6 | 4.62 | 0.003 | 102.4 | 1.99 | 4.66 |
| | 6 | DDH | Au | 120.00 | 171 | 0.003 | 120.00 | 3.24 | 1 | 0.6 | 11 | 3.78 | 0.003 | 252.76 | 3.65 | 4.77 |
| | 6a | DDH | Au | No Cap | 2 | 0.26 | 1.25 | 0.75 | 0 | 0 | 0 | 0.93 | 0.26 | 1.25 | 0.75 | 0.93 |
| | 7 | DDH | Au | 110.00 | 176 | 0.003 | 110.00 | 1.32 | 1 | 0.6 | 33 | 5.91 | 0.003 | 323.4 | 1.98 | 9.44 |
| | 8 | DDH | Au | No Cap | 195 | 0.003 | 56.67 | 1.70 | 0 | 0 | 0 | 3.97 | 0.003 | 56.67 | 1.7 | 3.97 |
| | 9 | DDH | Au | 150.00 | 423 | 0.003 | 150.00 | 1.30 | 3 | 0.7 | 59 | 7.65 | 0.003 | 780.00 | 3.179 | 13.46 |
| | 10 | DDH | Au | No cap | 333 | 0.003 | 63.30 | 0.916 | 0 | 0.00% | | 4.57 | 0.003 | 63.30 | 0.92 | 4.57 |
| | 10a | DDH | Au | No cap | 91 | 0.003 | 5.33 | 0.41 | 0 | 0.00% | | 2.23 | 0.003 | 5.33 | 0.41 | 2.23 |
| | 11 | DDH | Au | 120.00 | 353 | 0.003 | 120.00 | 1.7 | 3 | 0.8 | 41 | 5.85 | 0.003 | 709.00 | 2.91 | 9.97 |
| | 11a | DDH | Au | No cap | 238 | 0.003 | 36.7 | 1.18 | 0 | 0.00% | | 3.26 | 0.003 | 36.70 | 1.18 | 3.26 |
| | 12 | DDH | Au | 140.00 | 395 | 0.003 | 120.00 | 1.12 | 2 | 0.5 | 30 | 6.12 | 0.003 | 800.00 | 1.7 | 13.60 |
| | 12a | DDH | Au | No cap | 46 | 0.003 | 18.32 | 0.99 | 0 | 0.00% | | 3.47 | 0.003 | 18.32 | 0.99 | 3.47 |
| | 13 | DDH | Au | 110.00 | 539 | 0.003 | 110.00 | 0.91 | 1 | 0.2 | 5.1 | 6.22 | 0.003 | 148.9 | 0.96 | 6.97 |
| | 14 | Chip | Au | No cap | 1 | 0.273 | 0.27 | 0.27 | 0 | 0.00% | | 0.0 | 0.273 | 0.27 | 0.27 | 0.00 |
| | | DDH | Au | No Cap | 611 | 0.003 | 74.9 | 0.61 | 0 | 0 | 0 | 5.73 | 0.003 | 74.9 | 0.61 | 5.73 |
| | 14a | DDH | Au | No cap | 468 | 0.003 | 124.5 | 0.93 | 0 | | | 6.41 | 0.003 | 124.5 | 0.93 | 6.41 |
| | 15 | DDH | Au | 100.00 | 757 | 0.003 | 100.00 | 1.21 | 7 | 0.9 | 7.1 | 4.67 | 0.003 | 262.00 | 1.30 | 6.08 |
| | 15a | DDH | Au | 1200.00 | 309 | 0.003 | 120.00 | 0.87 | 4.0 | 1.3 | 4.1 | 5.85 | 0.003 | 156.50 | 0.90 | 6.64 |
| | 15b | Chip | Au | No cap | 6 | 0.010 | 2.29 | 0.52 | 0 | 0.00% | | 1.81 | 0.010 | 2.29 | 0.52 | 1.81 |
| | | DDH | Au | No Cap | 313 | 0.003 | 51.89 | 0.47 | 0 | 0 | 0 | 6.94 | 0.003 | 51.89 | 0.47 | 6.94 |

| | | | | | | Capped | | | | | | | Uncapped | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Domain Name | Belt | Type | Metal | Cap (g/t Gold) | # of Samples | Min | Max | Mean | # Capped | % Capped | % Metal Lost | CV | Min | Max | Mean | CV |
| | 16 | Chip | Au | No cap | 5 | 1.197 | 84.33 | 20.34 | 0 | 0.00% | | 1.54 | 1.197 | 84.33 | 20.34 | 1.54 |
| | | DDH | Au | No cap | 392 | 0.003 | 117.50 | 1.13 | 0 | 0.00% | | 4.47 | 0.003 | 117.50 | 1.13 | 4.47 |
| | 17 | DDH | Au | No cap | 8 | 0.02 | 6.5 | 1.58 | 0 | 0.00% | | 1.64 | 0.02 | 3.5 | 1.58 | 1.64 |
| | 18 | DDH | Au | No cap | 7 | 0.01 | 1.43 | 0.60 | 0 | 0.00% | | 1.01 | 0.01 | 1.42 | 0.60 | 1.01 |
| | 19 | DDH | Au | No Cap | 25 | 0.01 | 133.1 | 11.11 | 0 | 0 | 0 | 2.74 | 0.01 | 133.11 | 11.11 | 2.74 |
| | 20 | DDH | Au | 80.00 | 21 | 0.01 | 80.00 | 9.4 | 1 | 4.8 | 17 | 1.99 | 0.01 | 735.4 | 41.14 | 3.93 |
| | 21 | DDH | Au | No cap | 6 | 0.003 | 37.1 | 2.3 | 0 | 0.00% | | 5.33 | 0.003 | 37.1 | 2.33 | 5.33 |
| | 22 | DDH | Au | 120.00 | 9 | 0.01 | 120.00 | 20.2 | 1 | 11.1 | 9.1 | 1.82 | 0.01 | 871.23 | 93.85 | 2.96 |
| | 23 | DDH | Au | No cap | 1 | 0.0025 | 15.80 | 15.80 | 0 | 0.00% | | 0.00 | 0.0025 | 15.80 | 15.80 | 0.00 |
| | 24 | DDH | Au | No cap | 2 | 0.005 | 7.26 | 2.72 | 0 | 0.00% | | 1.88 | 0.0025 | 7.26 | 2.72 | 1.88 |
| | 25 | DDH | Au | No cap | 4 | 0.01 | 12.33 | 4.53 | 0 | 0.00% | | 1.32 | 0.01 | 12.33 | 4.53 | 1.32 |
| | Marker Horizon | DDH | Au | 80.00 | 4,149 | 0.0025 | 80.00 | 0.05 | 3 | 0.07% | 18.0 | 20.78 | 0.0025 | 122.50 | 0.06 | 27.02 |
| | Lower-Grade Domain | DDH | Au | 5.0 | 39,656 | 0.002 | 5.0 | 0.03 | 4 | 0.01 | 1.0 | 4.06 | 0.002 | 12.42 | 0.03 | 4.66 |

### 14.5.3 Compositing

Compositing of samples is a technique used to give each sample a relatively equal length to reduce the potential for bias due to uneven sample lengths; it prevents the potential loss of sample data and reduces the potential for grade bias due to the possible creation of short and potentially high-grade composites that are generally formed along the zone contacts when using a fixed length.

The raw sample data was found to have a moderately consistent range of sample lengths. Samples captured within all zones were composited to 1.0 m regular intervals based on the observed modal distribution of sample lengths, which supports a 2.0 m x 2.0 m x 2.0 m block model (Northing x Easting x Elevation) with two sub-blocking levels (a minimum size of Northing = 0.5 m x Easting = 0.5 m x Variable Elevation). An option to use a slightly variable composite length was chosen to allow for backstitching shorter composites that are located along the edges of the composited interval. All composite samples were generated within each Higher-Grade Belt and Lower-Grade Domains. There are no overlaps along boundaries. The composite samples were statistically validated to ensure no material loss of data or change to each sample population's mean grade.

Table 14-14 provides the composite counts by belt/zone for each domain.

**Table 14-14: Composite Counts by Belt/Zone for Each Domain**

| West Goldbrook | | Boston Richardson | | Marker Horizon | | East Goldbrook | |
|---|---|---|---|---|---|---|---|
| Belt | Au Composite Count | Belt | Au Composite Count | Belt | Au Composite Count | Belt | Au Composite Count |
| 1 | 56 | 1 | 898 | Marker Horizon | 9,076 | 1 | 3 |
| 2 | 230 | 2 | 1,411 | | | 2 | 6 |
| 3 a | 158 | 3 | 1,400 | | | 3 | 15 |
| 3b | 32 | 3 a | 85 | | | 4 | 137 |
| 4 | 179 | 4 | 1,317 | | | 4 a | 31 |
| 5 | 41 | 4 a | 176 | | | 4b | 58 |
| 5 a | 241 | 5 | 985 | | | 5 | 173 |
| 5b | 5 | 5 a | 41 | | | 5 a | 151 |
| 6 | 10 | 6 | 449 | | | 6 | 128 |
| 6 a | 350 | 7 | 361 | | | 7 | 156 |
| 6b | 47 | 8 | 167 | | | 8 | 149 |
| 7 | 135 | 9 | 120 | | | 9 | 323 |
| 8 | 117 | 10 | 43 | | | 10 | 224 |
| 9 | 46 | 11 | 54 | | | 10 a | 78 |
| 10 | 2 | 12 | 27 | | | 11 | 241 |
| 11 | 17 | 13 | 22 | | | 11 a | 147 |
| | | | | | | 12 | 331 |
| | | | | | | 12 a | 31 |
| | | | | | | 13 | 396 |
| | | | | | | 14 | 469 |
| | | | | | | 14 a | 356 |
| | | | | | | 15 | 518 |
| | | | | | | 15 a | 250 |
| | | | | | | 15b | 282 |
| | | | | | | 16 | 324 |
| | | | | | | 17 | 6 |
| | | | | | | 18 | 5 |
| | | | | | | 19 | 15 |
| | | | | | | 20 | 8 |
| | | | | | | 21 | 6 |
| | | | | | | 22 | 3 |
| | | | | | | 23 | 2 |
| | | | | | | 24 | 2 |
| | | | | | | 25 | 2 |
| Lower-Grade Domain | 14,929 | Lower-Grade Domain | 21,386 | | | Lower-Grade Domain | 38,297 |

### 14.5.4 Specific Gravity

A total of 3,596 SG measurements exist for all three domains within the Project, provided from laboratory measurements. This included 885 SG measurements for BR Domain, 2,115 from EG Domain, and 596 from WG Domain. Measurements were taken from DDH samples using the weight in air versus the weight in water method (Archimedes) by applying the following formula:

$$Specific\ Gravity = \frac{Weight\ in\ Air}{(Weight\ in\ Air - Weight\ in\ Water)}$$

Nordmin determined that the required amount and distribution of SG measurements did not exist for direct estimation of the entire block model. Instead, SG weighted averages were calculated from the available SG assays, and each belt was assigned an SG value as described in Table 14-15.

To summarize, SG values were assigned as follows:

- BR Domain:
  - Each belt was assigned an independent weighted average SG.
- Low-grade was assigned a weighted average SG of 2.73.
- EG Domain:
  - Belts 1 through 7: A weighted average SG of 2.73 was assigned.
  - Belts 8 through 16: Each belt was assigned an independent weighted average SG.
  - Belts 17 through 25: A weighted average of 2.76 was applied.
- Low-Grade: A weighted average of 2.73 was applied.
- WG Domain:
  - A weighted average SG of 2.753 was applied to all Belts.
  - A weighted average SG of 2.715 was assigned to low-grade.

**Table 14-15: Specific Gravity**

| West Goldbrook | | Boston Richardson | | East Goldbrook | |
|---|---|---|---|---|---|
| **Belt** | **SG Assigned** | **Belt** | **SG Assigned** | **Belt** | **SG Assigned** |
| 1 | 2.753 | 1 | 2.75 | 1 | 2.73 |
| 2 | 2.753 | 2 | 2.72 | 2 | 2.73 |
| 3a | 2.753 | 3 | 2.77 | 3 | 2.73 |
| 3b | 2.753 | 3a | 2.84 | 4 | 2.73 |
| 4 | 2.753 | 4 | 2.75 | 4a | 2.73 |
| 5 | 2.753 | 4a | 2.75 | 4b | 2.73 |
| 5a | 2.753 | 5 | 2.80 | 5 | 2.73 |
| 5b | 2.753 | 5a | 2.80 | 5a | 2.73 |
| 6 | 2.753 | 6 | 2.79 | 6 | 2.73 |
| 6a | 2.753 | 7 | 2.77 | 7 | 2.73 |
| 6b | 2.753 | 8 | 2.74 | 8 | 2.75 |
| 7 | 2.753 | 9 | 2.76 | 9 | 2.76 |
| 8 | 2.753 | 10 | 2.77 | 10 | 2.70 |
| 9 | 2.753 | 11 | 2.76 | 10a | 2.71 |
| 10 | 2.753 | 12 | 2.78 | 11 | 2.73 |
| 11 | 2.753 | 13 | 2.74 | 11a | 2.71 |
| | | | | 12 | 2.76 |
| | | | | 13 | 2.75 |
| | | | | 14 | 2.76 |
| | | | | 14a | 2.75 |
| | | | | 15 | 2.79 |
| | | | | 15a | 2.73 |
| | | | | 15b | 2.77 |
| | | | | 16 | 2.76 |
| | | | | 17 | 2.76 |
| | | | | 18 | 2.76 |
| | | | | 19 | 2.76 |
| | | | | 20 | 2.76 |
| | | | | 21 | 2.76 |
| | | | | 22 | 2.76 |
| | | | | 23 | 2.76 |
| | | | | 24 | 2.76 |
| | | | | 25 | 2.76 |
| Lower-Grade Domain | 2.715 | Lower-Grade Domain | 2.73 | Lower-Grade Domain | 2.73 |
| | | | | **Marker Horizon** | 2.73 |

## 14.6 Block Model Mineral Resource Estimation

### 14.6.1 Block Model Strategy and Analysis

A series of upfront test modelling was completed to define an estimation methodology to meet the following criteria:

- Representative of the Deposit geology, structural models, and geological controls on mineralization.
- Accounts for the variability of grade, orientation, and continuity of mineralization.
- Controls the smoothing (grade spreading) of grades and the influence of outliers.
- Accounts for most of the mineralization.
- Is robust and repeatable within domains.
- Supports multiple high-grade and low-grade zones.

Multiple test scenarios were evaluated to determine the optimum processes and parameters to use to achieve the stated criteria. Each scenario was based on NN, ID2, ID3, and OK interpolation methods.

All test scenarios were evaluated based on global statistical comparisons, visual comparisons of composite samples versus block grades, and the assessment of overall smoothing. Based on the results of the testing, it was determined that all scenarios, including the draft and final resource estimation methodology, would constrain the mineralization by using hard wireframe boundaries to control the spread of high-grade and low-grade mineralization. OK was selected as the most representative interpolation method.

### 14.6.2 Block Model Definition

Block model shape and size is typically a function of the geometry of the deposit, the density of sample data, drill hole spacing, and the selected mining unit. Block models were defined with parent blocks at 2.0 m x 2.0 m x 2.0 m (Northing x Easting x Elevation). Block model parameters are defined in Table 14-16.

All wireframe volumes were filled with blocks from the prototype (which used the parameters in Table 14-16). Block volumes were compared to the wireframe volumes to confirm there were no significant differences. Block volumes for all wireframes were found to be within reasonable tolerance limits. Sub-blocking was allowed to maintain the geological interpretation and to accommodate the Higher-Grade Belts and Lower-Grade Domains (wireframes), the SG, and the category application. Sub-blocking has been allowed to the following minimums:

- 2.0 m x 2.0 m x 2.0 m blocks are sub-blocked three-fold to 0.25 m x 0.25 m in the N-S and E-W directions with a variable elevation calculated based on the other sizes.

**Table 14-16: Block Model Definition**

| | NAD83 MTM Zone 4 Coordinates | | | | | |
|---|---|---|---|---|---|---|
| **Item** | **Block Origin** | **Block Maximum** | **Block Extent (m)** | **Block Dimension (m)** | **Number of Blocks** | **Minimum Subblock (m)** |
| Easting | 292,200 | 295,400 | 3,200 | 2 | 1,600 | 0.25 |
| Northing | 5,006,200 | 5,007,800 | 1,600 | 2 | 800 | 0.25 |
| Elevation | 4,300 | 5,302 | 1,002 | 2 | 501 | Variable |

Block models were not rotated nor clipped to topography. Because dynamic anisotropy requires the full, folded wireframes for calculation, blocks were permitted to estimate above surface but had an "air" code applied and were removed from reporting. This is also the case for blocks within overburden and organics. The Mineral Resource Estimate was conducted using Datamine Studio RM™ version 1.8.32.0 within the NAD83 MTM Zone 4 datum.

Four block models were independently estimated, WG, EG, the Marker Horizon unit, and BR. The Marker Horizon unit block model was then merged overtop of the EG block model. The three block models then had extraneous fields removed and were combined into one overall resource block model.

### 14.6.3 Interpolation Method

The Project block models were estimated using NN, ID2, ID3, and OK interpolation methods for global comparisons and validation purposes. The OK method was selected for the Mineral Resource Estimate; it was selected over NN, ID2, and ID3 as the method best controlling estimation and smoothing of grades and was the most representative of all domains in the Project.

### 14.6.4 Search Strategy

Zonal controls were used to constrain the grade estimates to each wireframe. These controls prevented the samples from individual wireframes from influencing the block grades of others, acting as a "hard boundary" between the wireframes.

The search orientation strategy determined to be most representative of the mineralization at the Deposit was to use a combination of an overall search ellipsoid for each domain and to allow dynamic anisotropy in the estimation process. Dynamic anisotropy is a search adjustment applied to estimation which considers local variation of the wireframe orientation. It is Nordmin's opinion that dynamic anisotropy allows for a much more accurate estimation of grade and mineralization due to the tightly folded nature of mineralized Belts and lower-grade mineralized zones.

Estimation passes were defined with carefully-selected search distances. The first pass is correlated to a Measured categorization, the second pass is correlated to an Indicated categorization, and the third pass is correlated to an Inferred categorization. These three passes of increasing distance were as follows (major axis x semi-major axis x minor axis). Overall search parameters can be found in Table 14-17.

- **High-Grade Wireframes (Belts):**
  - First Pass: 25 m x 15 m x 10 m
  - Second Pass: 31.3 m x 18.8 m x 12.5 m
  - Third Pass: 150 m x 90 m x 60 m
- **Low-Grade Wireframes:**
  - First Pass: 15 m x 10 m x 5 m
  - Second Pass: 18.8 m x 12.5 m x 6.3 m
  - Third Pass: 60 m x 40 m x 20 m

**Table 14-17: Search Parameters**

| | | Ellipsoid Rotation Angles | | | Ranges, Search Pass 1 (m) | | | Ranges, Search Pass 2 (m) | | | Ranges, Search Pass 3 (m) | | | Composites, Pass 1 | | Composites, Pass 2 | | Composites, Pass 3 | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Domain** | **Type** | **1 (X)** | **2 (Z)** | **3 (Y)** | **1** | **2** | **3** | **1** | **2** | **3** | **1** | **2** | **3** | **Min** | **Max** | **Min** | **Max** | **Min** | **Max** |
| All | Higher-Grade Belts | 10 | -84 | 23 | 25 | 15 | 10 | 31.3 | 18.8 | 12.5 | 150 | 90 | 60 | 3 | 8 | 3 | 8 | 2 | 8 |
| All | Lower-Grade Domains | 10 | -84 | 23 | 25 | 15 | 10 | 18.8 | 12.5 | 6.3 | 60 | 40 | 20 | 3 | 8 | 3 | 8 | 2 | 8 |

### 14.6.5 Assessment of Spatial Grade Continuity

Datamine, X10 Geo, and Sage 2001 was used to determine the geostatistical relationships of the Deposit. Independent variography was performed on composite data for each wireframe in each domain. Experimental grade variograms were calculated from the capped/composited sample gold data to determine the approximate search ellipse dimensions and orientations.

The analyses considered the following:

- Downhole variograms were created and modelled to define the nugget effect.
- Experimental pairwise relative correlogram variograms were calculated to determine directional variograms for the strike and down dip orientations.
- Variograms were modelled using an exponential with practical range.
- Directional variograms were modelled using the nugget defined in the downhole variography, and the ranges for strike, perpendicular to strike, and down dip directions.
- Variograms outputs were re-oriented to reflect the orientation of the mineralization.

Individual variograms were created for each Higher-Grade Belts and Lower-Grade Domains.

- The analysis of these individual variograms demonstrated that gold continuity could be appropriately defined by one main variogram across all domains.

Variography parameters used are provided in Table 14-18.

**Table 14-18: Variography Parameters**

| | Rotation Angles | | | | Structure 1 | | | Structure 2 | | | Nugget |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Domain | 1 | 2 | 3 | Axes | Range 1 | Range 2 | Range 3 | Range 1 | Range 2 | Range 3 | |
| All | 21 | 14 | 50 | Z-Y-Z | 7.2 | 31.7 | 6.5 | 1307 | 385 | 7.8 | 0.046 |

## 14.7 Grade Distribution Between High-Grade Belts and Lower-Grade Zones

Open pit constrained resources have been outlined by domain in Table 14-19, Table 14-20, and Table 14-21 for each high-grade belt and lower-grade zone. High-grade belt material accounts for approximately 84.0% of pit tonnage, and lower-grade zone material accounts for approximately 16.0% of pit tonnage.

**Table 14-19: WG Open Pit Constrained Totals, Cut-off=0.45 g/t Gold**

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| West Goldbrook Measured | | | | |
| 1 | 0.45 | 95,319 | 1.52 | 4,660 |
| 2 | 0.45 | 225,646 | 2.63 | 19,096 |
| 3a | 0.45 | 137,568 | 1.62 | 7,155 |
| 3b | 0.45 | 10,917 | 1.07 | 376 |
| 4 | 0.45 | 69,545 | 1.82 | 4,071 |
| 5a | 0.45 | 203,279 | 2.98 | 19,447 |
| 6a | 0.45 | 60,338 | 3.37 | 6,540 |
| 6b | 0.45 | 9,165 | 5.73 | 1,688 |
| 11 | 0.45 | 2,468 | 2.07 | 164 |
| Lower-Grade | 0.45 | 133,812 | 0.62 | 2,655 |
| **TOTAL** | **0.45** | **948,059** | **2.16** | **65,852** |

| Belt/ Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| West Goldbrook Indicated | | | | |
| 1 | 0.45 | 200,619 | 2.12 | 13,648 |
| 2 | 0.45 | 221,541 | 4.50 | 32,068 |
| 3a | 0.45 | 180,552 | 2.16 | 12,546 |
| 3b | 0.45 | 13,535 | 1.03 | 449 |
| 4 | 0.45 | 34,609 | 1.61 | 1,796 |
| 5a | 0.45 | 108,726 | 2.41 | 8,439 |
| 6a | 0.45 | 67,871 | 2.63 | 5,731 |
| 6b | 0.45 | 35,528 | 4.39 | 5,017 |
| 11 | 0.45 | 41,834 | 1.72 | 2,308 |
| Lower-Grade | 0.45 | 83,630 | 0.59 | 1,581 |
| **TOTAL** | **0.45** | **988,446** | **2.63** | **83,584** |

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| West Goldbrook Inferred | | | | |
| Lower-Grade | 0.45 | 13,097 | 0.58 | 243 |
| **TOTAL** | **0.45** | **13,097** | **0.58** | **243** |

*Note: Totals may not add due to rounding*

**Table 14-20: BR Open Pit Constrained Totals, Cut-off=0.45 g/t Gold**

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **Boston Richardson Measured** | | | | |
| 1 | 0.45 | 414,661 | 3.10 | 41,294 |
| 2 | 0.45 | 640,069 | 3.12 | 64,176 |
| 3 | 0.45 | 673,250 | 2.71 | 58,689 |
| 3a | 0.45 | 5,616 | 3.40 | 614 |
| 4 | 0.45 | 441,740 | 2.40 | 34,084 |
| 4a | 0.45 | 112,111 | 3.77 | 13,578 |
| 5 | 0.45 | 396,414 | 2.85 | 36,319 |
| 5a | 0.45 | 26,601 | 4.51 | 3,856 |
| 6 | 0.45 | 337,473 | 4.66 | 50,542 |
| 7 | 0.45 | 305,206 | 2.95 | 28,925 |
| 8 | 0.45 | 228,501 | 4.10 | 30,131 |
| 9 | 0.45 | 116,661 | 5.01 | 18,781 |
| 10 | 0.45 | 42,060 | 2.29 | 3,102 |
| 11 | 0.45 | 12,093 | 1.18 | 459 |
| Lower-Grade | 0.45 | 598,847 | 0.74 | 14,243 |
| **TOTAL** | **0.45** | **4,351,304** | **2.85** | **398,795** |

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **Boston Richardson Indicated** | | | | |
| 1 | 0.45 | 255,422 | 1.73 | 14,190 |
| 2 | 0.45 | 351,907 | 2.15 | 24,297 |
| 3 | 0.45 | 271,851 | 3.20 | 27,974 |
| 3a | 0.45 | 3,755 | 4.89 | 590 |
| 4 | 0.45 | 216,796 | 1.89 | 13,147 |
| 4a | 0.45 | 18,930 | 1.97 | 1,200 |
| 5 | 0.45 | 165,271 | 3.89 | 20,693 |
| 5a | 0.45 | 3,268 | 5.20 | 546 |
| 6 | 0.45 | 142,297 | 4.11 | 18,820 |
| 7 | 0.45 | 124,452 | 2.46 | 9,833 |
| 8 | 0.45 | 95,879 | 4.87 | 15,017 |
| 9 | 0.45 | 68,587 | 8.67 | 19,110 |
| 10 | 0.45 | 40,531 | 1.66 | 2,160 |
| 11 | 0.45 | 11,914 | 1.06 | 404 |
| Lower-Grade | 0.45 | 116,737 | 0.75 | 2,824 |
| **TOTAL** | **0.45** | **1,887,597** | **2.81** | **170,807** |

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **Boston Richardson Inferred** | | | | |
| 1 | 0.45 | 331 | 1.27 | 14 |
| 2 | 0.45 | 4,159 | 1.78 | 238 |
| Lower-Grade | 0.45 | 2,852 | 0.94 | 86 |
| **TOTAL** | **0.45** | **7,342** | **1.43** | **337** |

Note: Totals may not add due to rounding

### Table 14-21: EG Open Pit Constrained Totals, Cut-off=0.45 g/t Gold

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **East Goldbrook Measured** | | | | |
| | | | | |
| | | | | |
| 3 | 0.45 | 874 | 9.18 | 258 |
| 4 | 0.45 | 54,333 | 2.79 | 4,869 |
| 4a | 0.45 | 7,889 | 8.39 | 2,128 |
| 4b | 0.45 | 62,536 | 6.32 | 12,709 |
| 5 | 0.45 | 149,640 | 4.60 | 22,139 |
| 5a | 0.45 | 123,903 | 4.41 | 17,574 |
| 6 | 0.45 | 109,867 | 4.88 | 17,227 |
| 7 | 0.45 | 72,592 | 6.97 | 16,264 |
| 8 | 0.45 | 60,555 | 4.98 | 9,700 |
| 9 | 0.45 | 166,111 | 1.70 | 9,083 |
| 10 | 0.45 | 70,600 | 0.98 | 2,223 |
| 10a | 0.45 | 67 | 1.26 | 3 |
| 11 | 0.45 | 19,410 | 4.49 | 2,799 |
| 11a | 0.45 | 40,276 | 2.60 | 3,369 |
| 12 | 0.45 | 39,545 | 1.88 | 2,385 |
| 13 | 0.45 | 11,729 | 0.63 | 239 |
| 14 | 0.45 | 49,014 | 1.00 | 1,570 |
| 14a | 0.45 | 186,108 | 2.93 | 17,538 |
| 15 | 0.45 | 136,159 | 2.88 | 12,595 |
| 15a | 0.45 | 25,296 | 1.64 | 1,330 |
| 15b | 0.45 | 96,225 | 1.21 | 3,730 |
| 16 | 0.45 | 216,784 | 3.20 | 22,299 |
| 17 | 0.45 | 2,291 | 1.69 | 124 |
| 18 | 0.45 | 1,645 | 0.66 | 35 |
| 19 | 0.45 | 16,037 | 9.70 | 5,001 |
| 20 | 0.45 | 21,206 | 7.65 | 5,215 |
| 21 | 0.45 | 2,826 | 9.70 | 881 |
| 22 | 0.45 | 4,208 | 27.36 | 3,701 |
| | | | | |
| | | | | |
| Lower-Grade | 0.45 | 386,254 | 0.69 | 8,612 |
| **TOTAL** | **0.45** | **2,380,280** | **2.93** | **224,419** |

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **East Goldbrook Indicated** | | | | |
| 1 | 0.45 | 82,266 | 1.44 | 3,820 |
| 2 | 0.45 | 18,163 | 9.71 | 5,670 |
| 3 | 0.45 | 83,980 | 3.00 | 8,110 |
| 4 | 0.45 | 358,447 | 1.60 | 18,449 |
| 4a | 0.45 | 220,675 | 2.31 | 16,406 |
| 4b | 0.45 | 298,594 | 3.76 | 36,058 |
| 5 | 0.45 | 416,642 | 3.55 | 47,498 |
| 5a | 0.45 | 325,795 | 3.66 | 38,291 |
| 6 | 0.45 | 404,201 | 5.74 | 74,553 |
| 7 | 0.45 | 276,375 | 4.19 | 37,245 |
| 8 | 0.45 | 153,939 | 3.45 | 17,061 |
| 9 | 0.45 | 271,368 | 1.95 | 17,001 |
| 10 | 0.45 | 136,426 | 1.19 | 5,240 |
| 10a | 0.45 | 31,116 | 0.74 | 742 |
| 11 | 0.45 | 137,711 | 2.25 | 9,946 |
| 11a | 0.45 | 48,575 | 1.87 | 2,915 |
| 12 | 0.45 | 64,395 | 1.05 | 2,180 |
| 13 | 0.45 | 39,027 | 0.71 | 889 |
| 14 | 0.45 | 76,454 | 0.97 | 2,384 |
| 14a | 0.45 | 163,899 | 1.87 | 9,868 |
| 15 | 0.45 | 290,664 | 2.04 | 19,045 |
| 15a | 0.45 | 20,850 | 1.48 | 993 |
| 15b | 0.45 | 135,082 | 3.74 | 16,232 |
| 16 | 0.45 | 366,108 | 1.64 | 19,360 |
| 17 | 0.45 | 35,213 | 2.14 | 2,420 |
| 18 | 0.45 | 50,752 | 0.61 | 988 |
| 19 | 0.45 | 104,189 | 11.10 | 37,177 |
| 20 | 0.45 | 52,797 | 9.80 | 16,636 |
| 21 | 0.45 | 4,371 | 6.85 | 963 |
| 22 | 0.45 | 10,972 | 21.58 | 7,613 |
| 23 | 0.45 | 7,941 | 15.80 | 4,034 |
| 25 | 0.45 | 16,570 | 4.66 | 2,484 |
| Lower-Grade | 0.45 | 392,008 | 0.64 | 8,103 |
| **TOTAL** | **0.45** | **5,136,975** | **2.95** | **487,318** |

Note: Table continues

| Belt/Zone | Gold Cut-off (g/t) | Tonnes | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|
| **East Goldbrook Inferred** | | | | |
| 1 | 0.45 | 54,788 | 1.50 | 2,637 |
| 2 | 0.45 | 18,655 | 9.70 | 5,815 |
| 3 | 0.45 | 40,007 | 2.60 | 3,347 |
| 4 | 0.45 | 177,128 | 1.60 | 9,132 |
| 4a | 0.45 | 54,275 | 0.97 | 1,689 |
| 4b | 0.45 | 59,730 | 2.31 | 4,445 |
| 5 | 0.45 | 195,091 | 0.98 | 6,137 |
| 5a | 0.45 | 114,932 | 1.06 | 3,910 |
| 6 | 0.45 | 100,233 | 6.57 | 21,168 |
| 9 | 0.45 | 1,152 | 3.34 | 124 |
| 10 | 0.45 | 2,149 | 1.25 | 86 |
| 13 | 0.45 | 574 | 0.58 | 11 |
| 14 | 0.45 | 1,464 | 1.10 | 52 |
| 14a | 0.45 | 1,091 | 1.52 | 53 |
| 15 | 0.45 | 1,819 | 2.36 | 138 |
| 16 | 0.45 | 3,301 | 0.67 | 71 |
| 17 | 0.45 | 1,221 | 2.28 | 89 |
| 18 | 0.45 | 194 | 0.59 | 4 |
| 19 | 0.45 | 158 | 10.29 | 52 |
| 23 | 0.45 | 7,941 | 15.80 | 4,034 |
| Lower-Grade | 0.45 | 118,848 | 0.70 | 2,663 |
| **TOTAL** | **0.45** | **954,749** | **2.14** | **65,657** |

Note: Totals may not add due to rounding

## 14.8 Estimation of Non-Payables

Non-payable elements were estimated using the NN and ID2 interpolation methods and included silver, arsenic, cadmium, copper, lead, and sulphur.

## 14.9 Block Model Validation

The block model validation process included visual comparisons between block estimates and composite grades in plan and section, local versus global estimates for NN, ID2, ID3, and OK, as well as swath plots. Block estimates were visually compared to the drill hole composite data in all domains and corresponding zones to ensure agreement. No material grade bias issues were identified, and the block model grades compared well to the composite data.

### 14.9.1 Visual Block Model Validation

The validation of the interpolated block model was assessed by using visual assessments and validation plots of block grades versus capped assay grades and composites. The review demonstrated a good comparison between local block estimates and nearby assays and composites without excessive smoothing in the block model. Figure 14-12 through Figure 14-15 provide the visual comparisons, displaying gold composite grades versus block model grades. Visual comparisons for all elements are available in Appendix D.

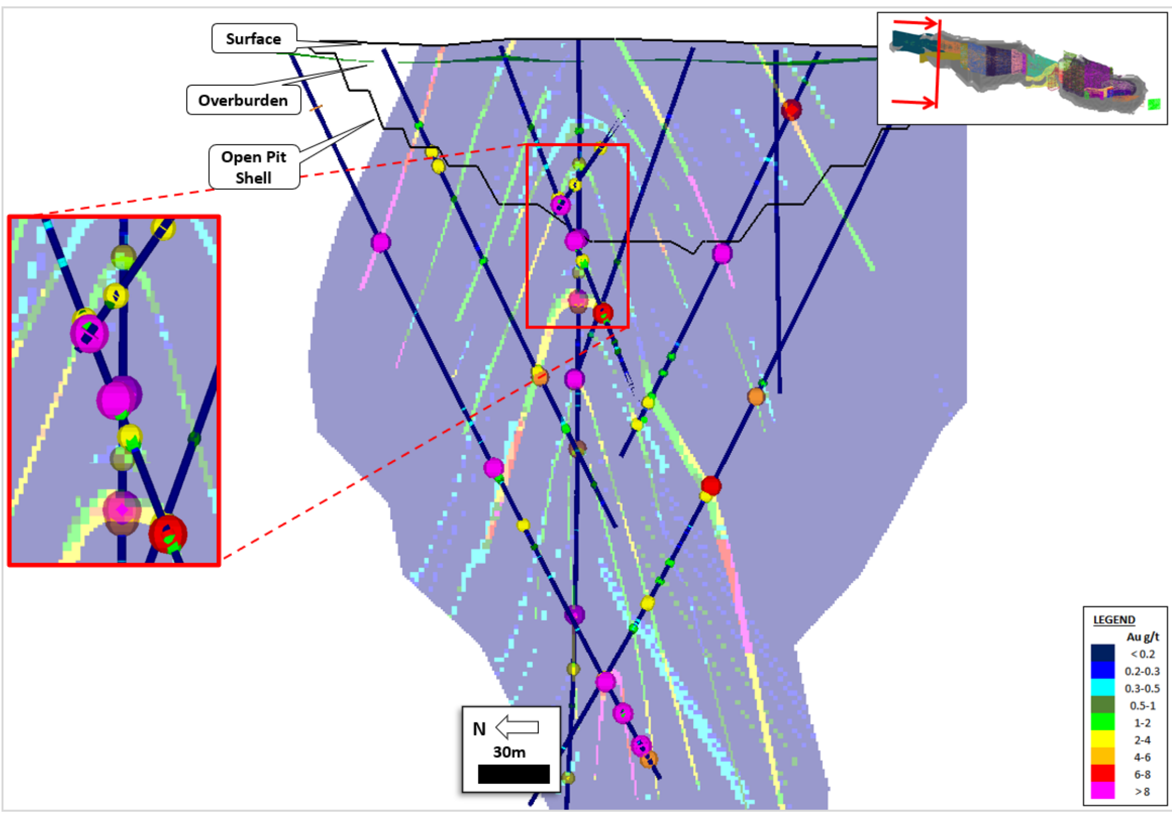



*Figure 14-12: Block model validation, cross section located within the WG Domain*

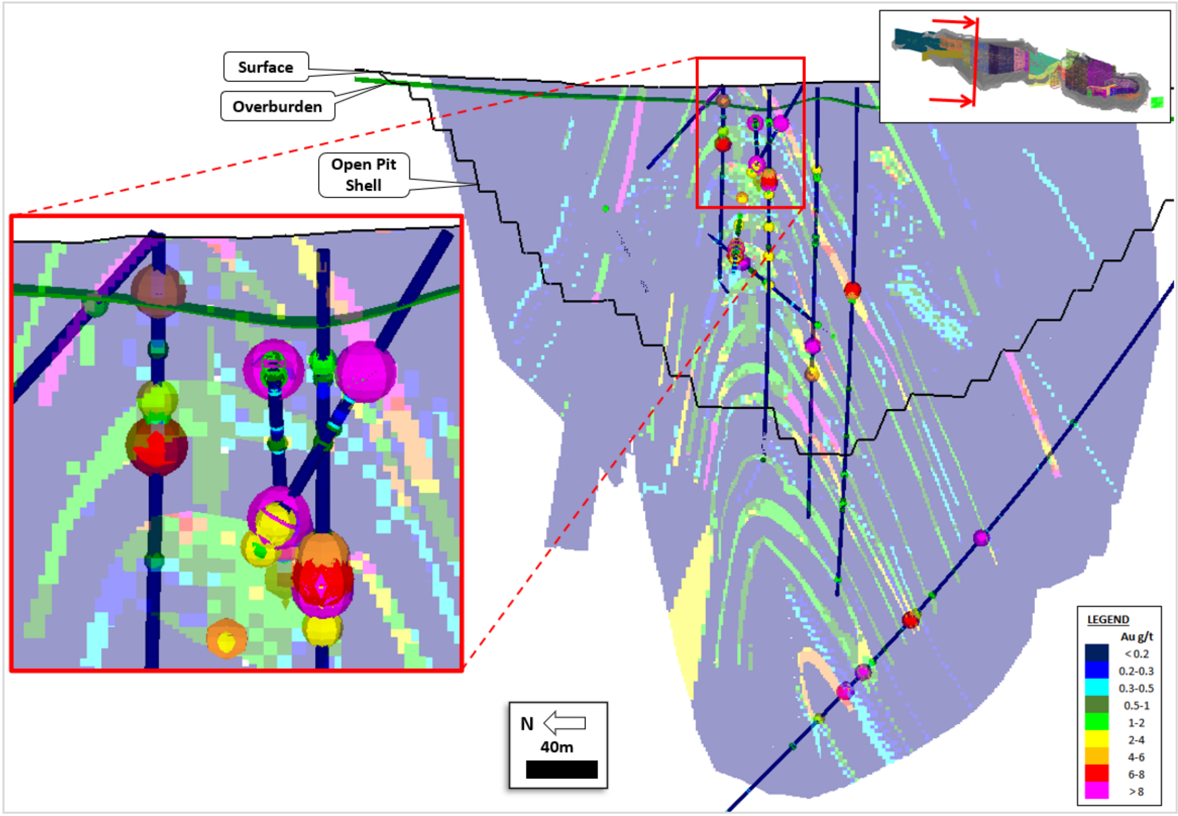



*Figure 14-13: Block model validation, cross section located within the WG and BR Domains*

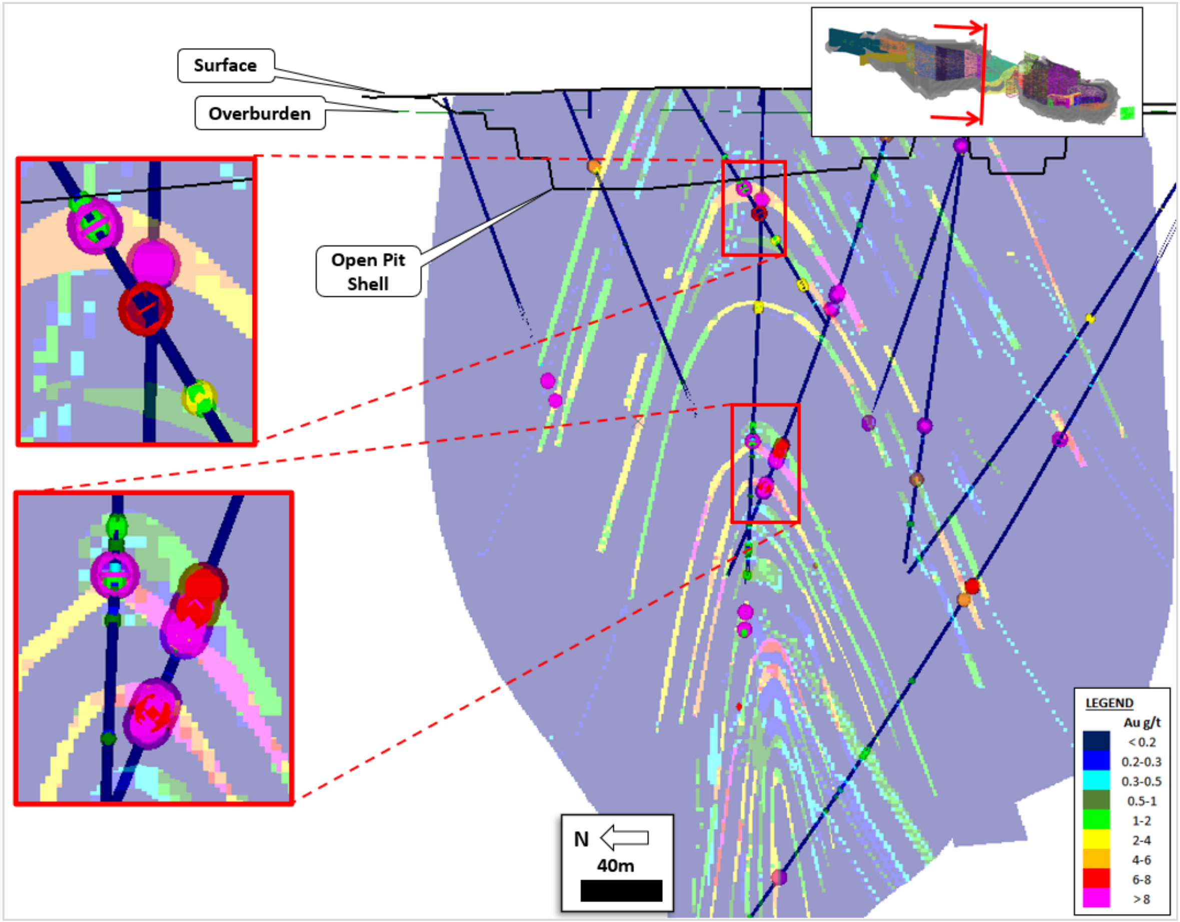



*Figure 14-14: Block model validation, cross section located at the west end of the BR and EG Domains*

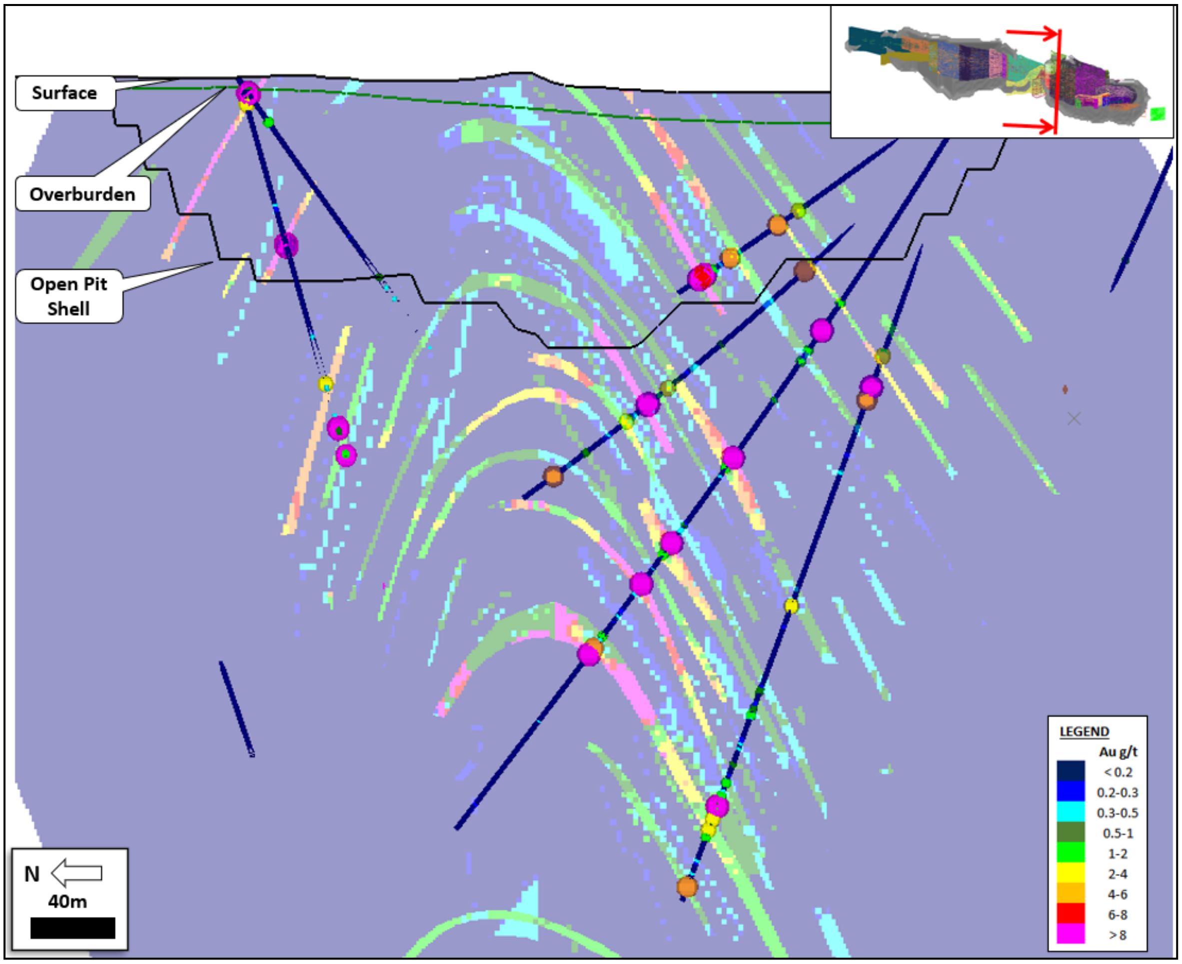



*Figure 14-15: Block model validation, cross section located in EG Domain*

### 14.9.2 Swath Plots

A swath plot is a graphical representation of grade distribution derived by a series of sectional "swaths" throughout the Deposit. Swath plots were generated for gold from slices throughout each domain. They compare the block model grades for NN, ID2, ID3, and OK to the drill hole composite grades to evaluate any potential local grade bias. Review of the swath plots did not identify bias in the model that is material to the 2021 Mineral Resource Estimate, as there was a strong overall correlation between the block model OK grade and the capped composites used in the 2021 Mineral Resource Estimate, as demonstrated in Table 14-16, Table 14-17, and Table 14-18. For these figures, the composite grade (S_AUCAP) is compared across swaths with the four gold estimation grades from the block model.

Fields include (all are in g/t):

- M_TONNES: Block model tonnage
- NRECORDS: Number of records
- S_AUCAP: Composite capped gold grade
- M_AUOK: Block model estimated gold grade, OK
- M_AUID2: Block model estimated gold grade, ID2
- M_AUID3: Block model estimated gold grade, ID3
- M_AUNN: Block model estimated gold grade, NN

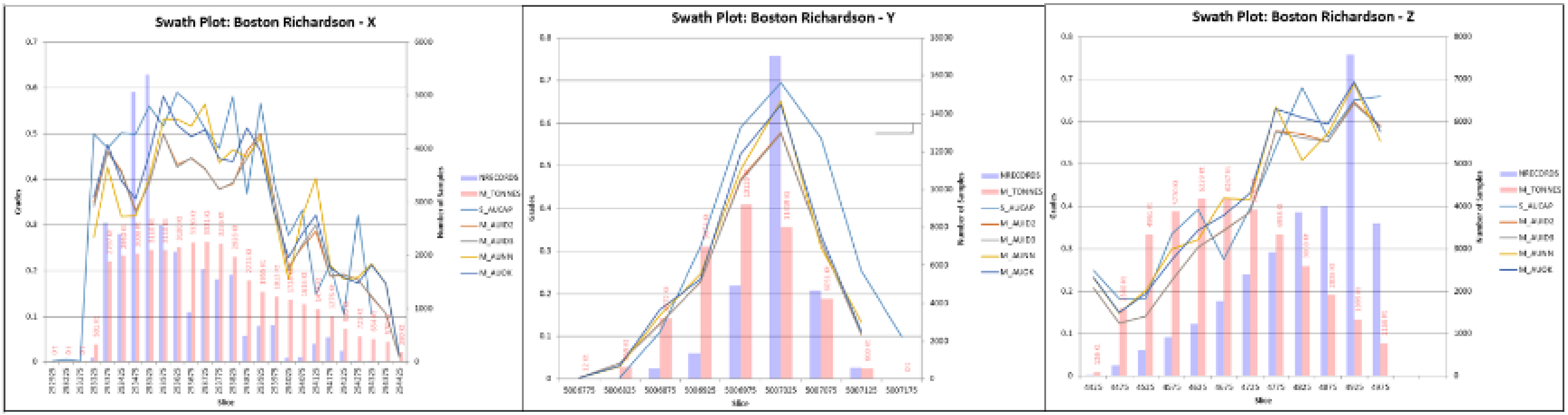



*Figure 14-16: Swath plots, BR, X, Y, and Z directions*

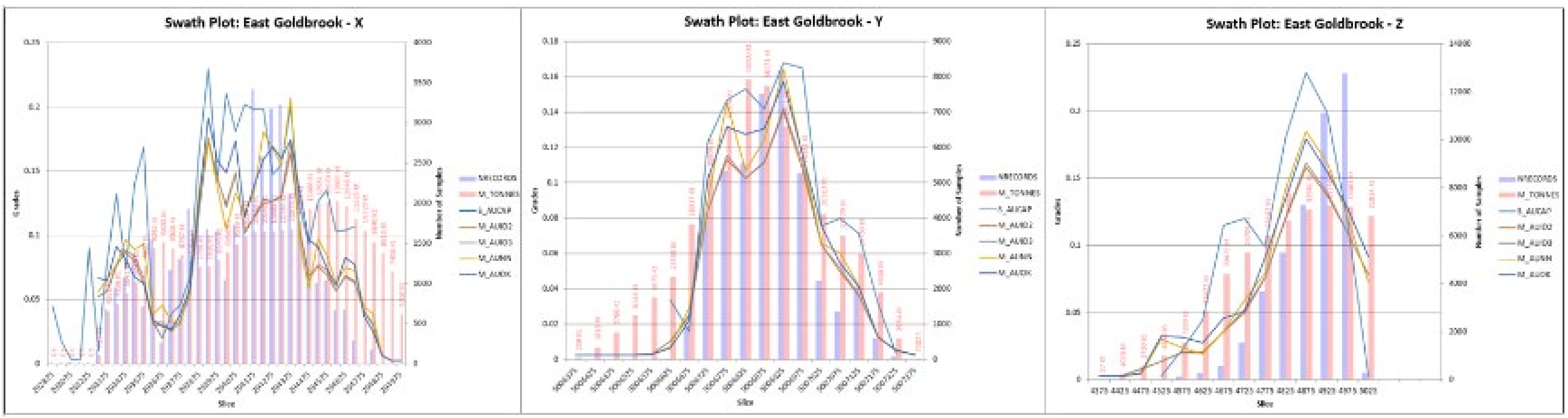



*Figure 14-17: Swath plots, EG, X, Y, and Z directions*

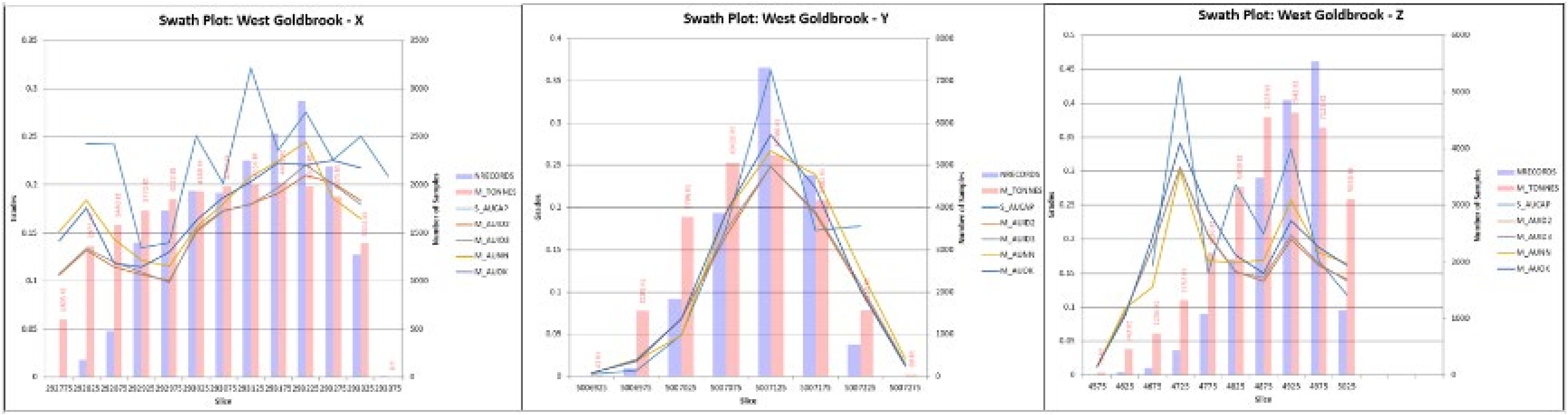



*Figure 14-18: Swath plots, WG, X, Y, and Z directions*

## 14.10 Interpolation Comparison

Estimation was completed using NN, ID2, ID3, and OK interpolation methods. The results are presented in Table 14-17. This includes all material that has been classified as Measured, Indicated, and Inferred.

Estimation differences between NN, ID2, and ID3 vs. OK interpolation methods are between 1.9% and 10.3%.

## 14.11 Mineral Resource Classification

The Mineral Resource Estimate was classified in accordance with the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Mineral Resource classifications were assigned to regions of the block model based on the QPs confidence and judgment related to geological understanding, continuity of mineralization in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

The classification was initially applied from the estimation pass. Blocks populated in pass 1 were classified as Measured, blocks populated in pass 2 were classified as Indicated, and blocks populated in pass 3 were classified as Inferred. Subsequently, block models were analyzed, and it was determined that several classification adjustments were required. These adjustments were as follows:

- Independent wireframes were built within specific areas that have relatively low drill density and/or high kriging variance. All material within these wireframes was classified as Inferred.
- Specific areas that contain lower-grade mineralized zones with a Measured or Indicated classification were downgraded to a lower classification in areas with poor drill density toward the bottom of the modelled Deposit.

Classification can be seen in Figure 14-19 and Figure 14-20.

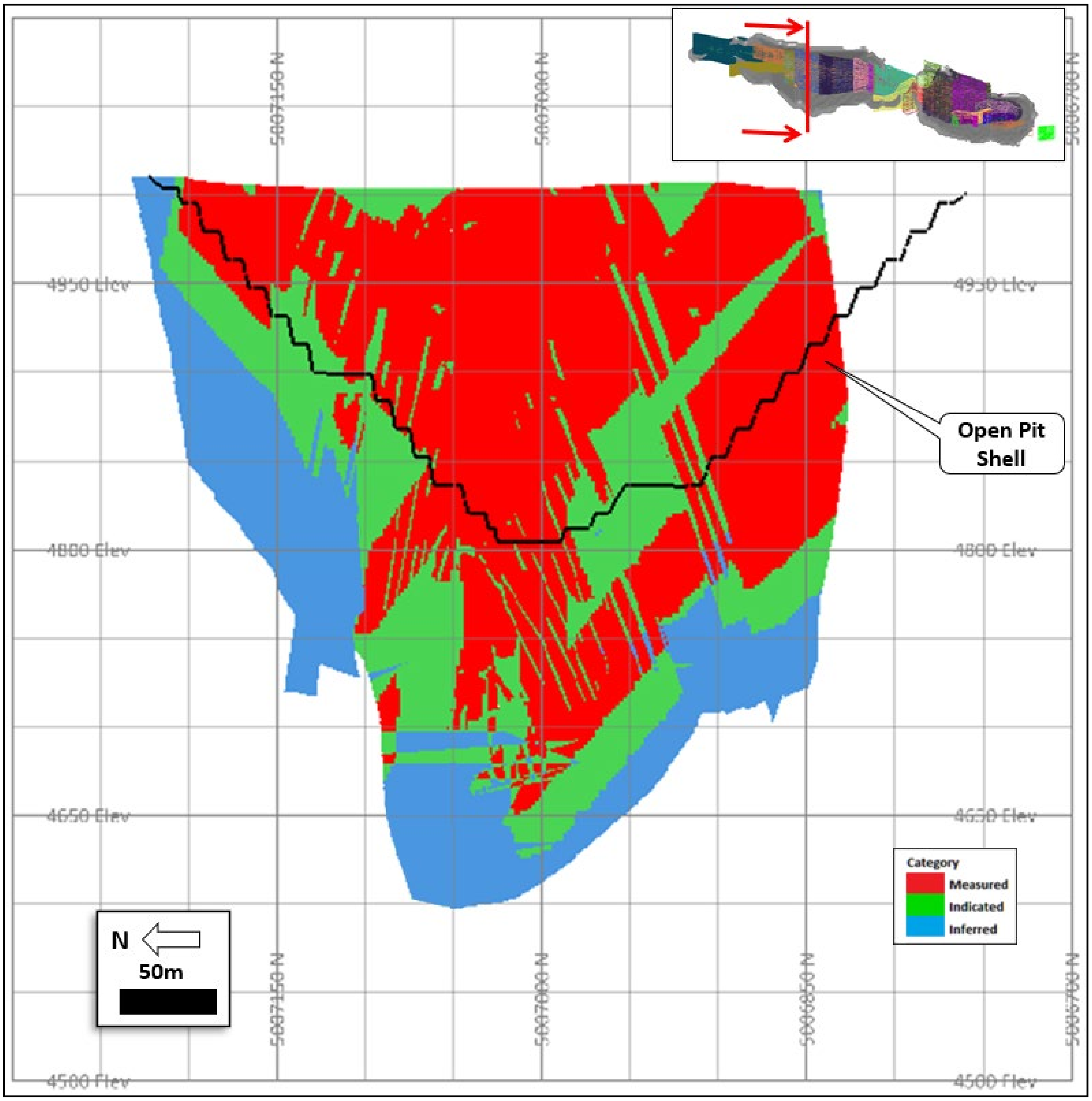



*Figure 14-19: Classification, Project long-section*

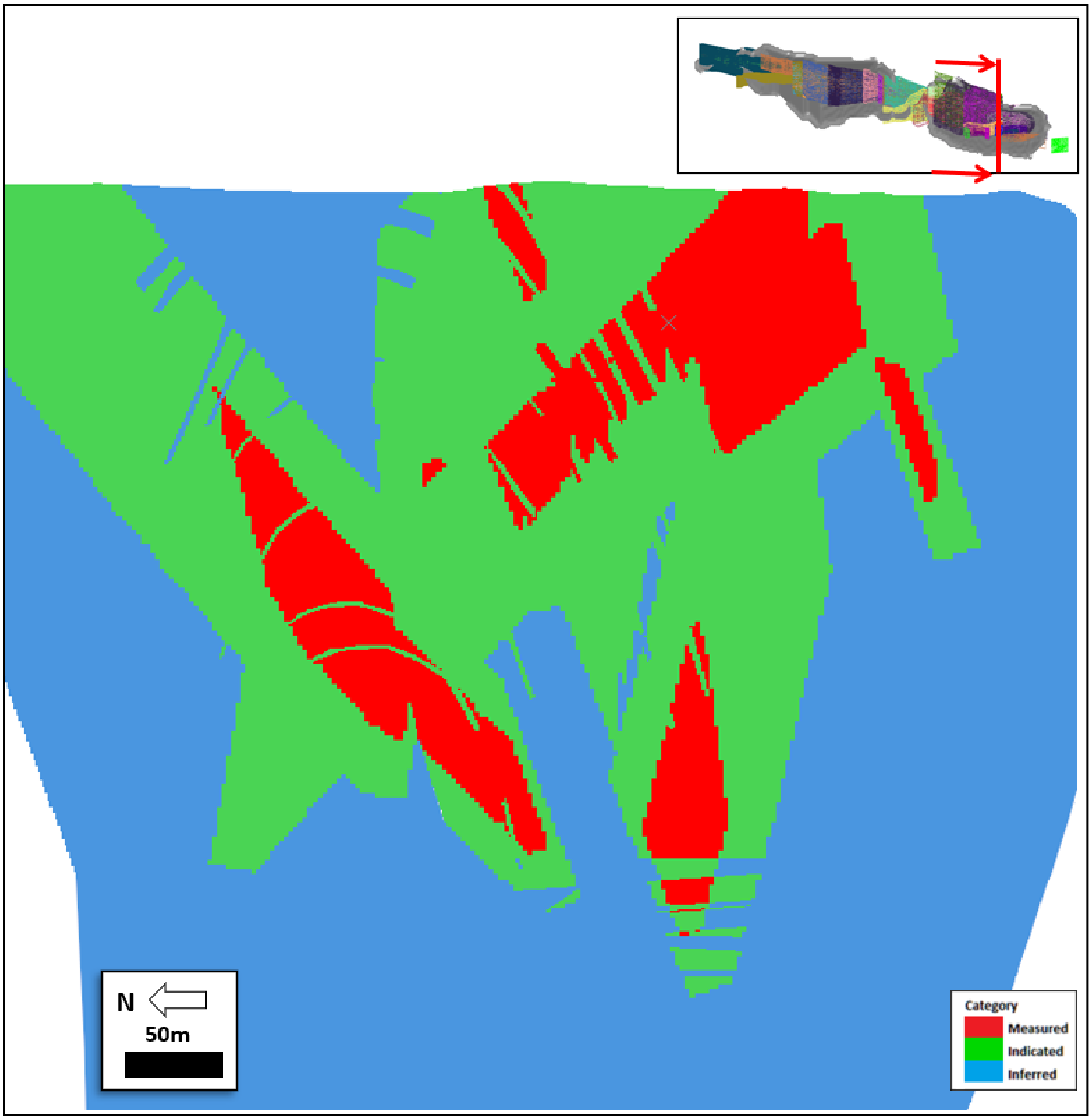



*Figure 14-20: Classification, cross section displaying WG Domain*

## 14.12 Reasonable Prospects of Eventual Economic Extraction

### 14.12.1 Open Pit

For the open pit Mineral Resource (Table 14-24), a pit limit analysis was undertaken using the LG algorithm in Geovia's Whittle 4.7 software to determine physical limits for a pit shell constrained Mineral Resource. The parameters used to generate the pit shell are shown in Table 14-22.

**Table 14-22: Open Pit Limit Analysis Parameters**

| Parameter | Value |
|---|---|
| Currency Used for Evaluation | C$ |
| Block Size | In-Situ model regularized to 2.0 m x 2.0 m x 4.0 m |
| Overall Stope Angle | Rock: Varied by Sector, Range 42°-46°<br>Overburden: 25° |
| Open Pit Mining Cost | 0.8 MCAF for Overburden<br>$5.10/$t_{mined}$ Rock<br>$0.016/t per 8 m |
| Process Cost<br>*Includes assumptions for Milling, G&A, tailings, and rehabilitation* | $25.75/$t_{processed}$ |
| Selling Cost<br>*Includes doré transportation, refining, and royalty* | $5.00/ ounce Au |
| Percent Payable | 99.95% |
| Metal Price | US$1,600 per ounce<br>Exchange Rate: 1 US$=1.25 C$<br>C$2,000 per troy ounce (rounded) |
| Process Recovery | Based on Grade – Recovery Curve:<br>$\frac{Block\ Grade - (0.0262 \times ln(Block\ Grade) + 0.0712)}{(Block\ Grade \times 100) - 0.083}$ |
| Mining Loss & Dilution | Included within Reblocked/ Regularized Block Model<br>Plus 5% factor for mining loss within optimization program<br>Overall effect of ~26% additional tonnes and ~8% reduction in metal |
| Resources Used to Generate Pit Shell | Measured + Indicated (no Inferred Resources were used to create the open pit physical limits) |
| Pit Shell Selection | RF 0.80 for Mine Planning |
| Production Rate Assumption | 4,000 t/d |

Three boundary constraints were used in the pit limit analysis for the Deposit:

- A 40 m (X-Y) offset from the natural gas pipeline easement, on the west side of the property;
- A 50 m (X-Y) offset from the edges of the Gold Brook Lake; and
- A 20 m (X-Y) offset from the centerline of Gold Brook.

The block models were created in Datamine using 2 m x 2 m x 2 m parent cell and variable sub-celling to 1 metre. For the open pit evaluation, the resource block model in Datamine format was reblocked to a regularized block model in Datamine format using Deswik.CAD. Default waste blocks and overburden blocks were added to the model. The envisioned selective mining excavator, at the onset of the analysis, will likely have a bucket width of approximately 2 m. Mining is planned at an 8 m operating bench height.

To classify the material contained within the open pit limits as material for processing or material for waste, the milling cut-off grade is used. This break-even cut-off grade is calculated to cover the costs of processing, general and administrative costs, and selling costs using the economic and technical parameters listed in Table 14-22. Mineral Resource material contained within the pit and above the cut-off grade is classified as PMF, while resource material below the cut-off grade is classified as waste. The cut-off grade has been estimated to be 0.45 g/t gold for the open pit.

### 14.12.2 Underground

For the underground Mineral Resource (Table 14-24), the parameters used to calculate the CoG are shown in Table 14-23.

**Table 14-23: Underground Limit Analysis Parameters**

| Parameter | Value |
|---|---|
| Currency Used for Evaluation | C$ |
| Block Size | In-Situ sub-blocked model with parent blocks at 2.0 m x 2.0 m x 2.0 m |
| Underground Mining Cost<br>*Includes assumptions for operating waste development, surface rehandle* | $96.25/t$_{processed}$ |
| Process Cost<br>*Includes assumptions for Milling, G&A, tailings, indirect costs* | $44.30/t$_{processed}$ |
| Underground Support Cost<br>*Includes assumptions for sustaining underground capital, infill diamond drilling* | $22.50/t$_{processed}$ |
| Selling Cost<br>*Includes doré transportation, refining, and royalty* | $24.84/troy ounce |
| Percent Payable | 99.95% |
| Metal Price | US$1,600 per troy ounce<br>Exchange Rate: 1 US$=1.3 C$<br>$2,000 per troy ounce (rounded) |
| Process Recovery | 97% |
| Production Rate Assumption | 1,200 t/d |

The underground Mineral Resource CoG is estimated to be 2.40 g/t gold. For resource cut-off calculation purposes, a mining recovery of 100% and a mining dilution of 0% were applied. The Mineral Resource Estimate excludes unclassified mineralization located within mined out areas.

## 14.13 Mineral Resource Estimate

The Mineral Resources were classified using the 2014 CIM Definition Standards and the 2019 CIM Best Practice Guidelines and have an effective date of November 15, 2021. The Project hosts:

- Total Open Pit (at a 0.45 g/t cut-off) and Underground (at a 2.40 g/t cut-off) Mineral Resources including 9,255 thousand tonnes and 1,057,963 oz of Measured Resources grading 3.56 g/t gold, 12,338 thousand tonnes and 1,523,014 oz of Indicated Resources grading 3.84 g/t gold, and 3,181 thousand tonnes and 484,250 oz of Inferred Resources grading 4.73 g/t gold.

- Open Pit Mineral Resources (at a 0.45 g/t cut-off) Mineral Resources including 7,680 thousand tonnes and 680,518 oz of Measured Resources grading 2.76 g/t gold, 7,988 thousand tonnes and 741,220 oz of Indicated Resources grading 2.89 g/t gold, and 975 thousand tonnes and 66,237 oz of Inferred Resources grading 2.11 g/t gold.
- Underground Mineral Resources including 1,576 thousand tonnes and 377,445 oz of Measured Resources grading 7.45 g/t gold, 4,3500 thousand tonnes and 781,794 oz of Indicated Resources grading 5.59 g/t gold, and 2,206 thousand tonnes and 418,013 oz of Inferred Resources grading 5.89 g/t gold.

The Mineral Resource Estimate presented in Table 14-24 are based on validated results of 681 surface and underground drill holes, for a total of 120,550 m of diamond drilling completed between 1984 and the effective date of November 15, 2021, as well as 1,230 chip samples comprised of 822.7 m from the 2018 to 2019 Bulk Sample. The Mineral Resource Estimate includes 7,488.3 m of diamond drilling in 62 drill holes since the Previous Mineral Resource Estimate effective February 7, 2021. Nine drill holes totalling 1,001.9 m were removed from the database due to inconsistent sample lengths (Table 14-4 and Table 14-5).

**Table 14-24: Mineral Resource Estimate, Open Pit (0.45 g/t Cut-off) and Underground (2.40 g/t Cut-off)**

| Resource Type | Gold Cut-off (g/t) | Category | Tonnes ('000) | Gold Grade (g/t) | Gold Troy Ounces |
|---|---|---|---|---|---|
| Open Pit | 0.45 | Measured | 7,680,000 | 2.756 | 680,518 |
| | | Indicated | 7,988,000 | 2.886 | 741,220 |
| | | Measured + Indicated | 15,668,000 | 2.822 | 1,421,738 |
| | | Inferred | 975,000 | 2.113 | 66,237 |
| Underground | 2.40 | Measured | 1,576,000 | 7.450 | 377,445 |
| | | Indicated | 4,350,000 | 5.590 | 781,794 |
| | | Measured + Indicated | 5,925,000 | 6.085 | 1,159,239 |
| | | Inferred | 2,206,000 | 5.893 | 418,013 |
| **Combined Open Pit and Underground*** | **0.45 and 2.40** | **Measured** | **9,255,000** | **3.555** | **1,057,963** |
| | | **Indicated** | **12,338,000** | **3.839** | **1,523,014** |
| | | **Measured + Indicated** | **21,593,000** | **3.718** | **2,580,977** |
| | | **Inferred** | **3,181,000** | **4.734** | **484,250** |

** Combined Open Pit and Underground Mineral Resources; The Open Pit Mineral Resource is based on a 0.45 g/t gold CoG, and the Underground Mineral Resource is based on 2.40 g/t gold CoG.*

**Mineral Resource Estimate Notes**

1. Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.
2. Mineral Resources are inclusive of Mineral Reserves.
3. Open pit Mineral Resources are reported at a CoG of 0.45 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,600/oz) and metallurgical recovery factor of 89% around cut-off as calculated from ((GRADE-(0.0262*LN(GRADE)+0.0712))/GRADE*100)-0.083
4. Underground Mineral Resource is reported at a CoG of 2.40 g/t gold that is based on a gold price of C$2,000/oz (approximately US$1,600/oz) and a gold processing recovery factor of 97%.
5. Assays were variably capped on a wireframe-by-wireframe basis (Table 14-11, Table 14-12 and Table 14-13).
6. SG was applied using weighted averages to each individual wireframe.
7. Mineral Resource effective date November 15, 2021.
8. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.
9. Excludes unclassified mineralization located within mined out areas.
10. Reported from within a mineralization envelope accounting for mineral continuity.

### 14.13.1 Cautionary Statement Regarding Mineral Resource Estimates

Until mineral deposits are actually mined and processed, Mineral Resources must be considered as estimates only. Mineral Resource Estimates that are not Mineral Reserves do not have demonstrated economic viability. The estimation of Mineral Resources is inherently uncertain, involves subjective judgment about many relevant factors and may be materially affected by, among other things, environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant risks, uncertainties, contingencies, and other factors described in the foregoing Cautionary Statements. The quantity and grade of reported "Inferred" Mineral Resource Estimates are uncertain in nature and there has been insufficient exploration to define "Inferred" Mineral Resource Estimates as an "Indicated" or "Measured" Mineral Resource and it is uncertain if further exploration will result in upgrading "Inferred" Mineral Resource Estimates to an "Indicated" or "Measured" Mineral Resource category. The accuracy of any Mineral Reserve and Mineral Resource Estimates is a function of the quantity and quality of available data, and of the assumptions made and judgments used in engineering and geological interpretation, which may prove to be unreliable and depend, to a certain extent, upon the analysis of drilling results and statistical inferences that may ultimately prove to be inaccurate. Mineral Reserve and Mineral Resource Estimates may have to be re-estimated based on, among other things: (i) fluctuations in mineral prices; (ii) results of drilling, and development; (iii) results of test stoping and other testing; (iv) metallurgical testing and other studies; (v) results of geological and structural modelling including stope design; (vi) proposed mining operations, including dilution; (vii) the evaluation of mine plans subsequent to the date of any estimates; and (viii) the possible failure to receive required permits, licences, and other approvals. It cannot be assumed that all or any part of an “Inferred,” “Indicated” or “Measured” Mineral Resource Estimate will ever be upgraded to a higher category.

## 14.14 Mineral Resource Sensitivity to Reporting Cut-off

The sensitivity of the Mineral Resource Estimate to a range of CoG’s for each category in the open pit and underground by domain are contained in Table 14-25, Table 14-26 and Table 14-27.

**Table 14-25: Mineral Resource Sensitivity to Reporting Cut-off, WG Domain**

| Resource Type | Category | CoG (Au g/t) | Tonnes | Gold Grade (g/t) | Gold (oz) |
|---|---|---|---|---|---|
| Open Pit | Measured | 0.40 | 1,019,338 | 2.04 | 66,824 |
| | | 0.42 | 986,388 | 2.09 | 66,391 |
| | | **0.45** | **948,059** | **2.16** | **65,852** |
| | | 0.47 | 906,337 | 2.24 | 65,233 |
| | | 0.49 | 870,943 | 2.31 | 64,688 |
| | Indicated | 0.40 | 1,033,771 | 2.53 | 84,193 |
| | | 0.42 | 1,005,721 | 2.59 | 83,824 |
| | | **0.45** | **988,446** | **2.63** | **83,584** |
| | | 0.47 | 960,848 | 2.69 | 83,174 |
| | | 0.49 | 950,751 | 2.72 | 83,018 |
| | Inferred | 0.40 | 18,886 | 0.53 | 321 |
| | | 0.42 | 16,205 | 0.55 | 285 |
| | | **0.45** | **13,097** | **0.58** | **243** |
| | | 0.47 | 9,573 | 0.62 | 190 |
| | | 0.49 | 9,510 | 0.62 | 189 |
| Underground | Measured | 2.20 | 317,949 | 6.75 | 68,998 |
| | | 2.30 | 300,363 | 7.01 | 67,726 |
| | | **2.40** | **288,969** | **7.20** | **66,866** |
| | | 2.50 | 278,075 | 7.38 | 66,007 |
| | | 2.60 | 270,075 | 7.53 | 65,352 |
| | Indicated | 2.40 | 629,132 | 6.57 | 132,859 |
| | | 2.50 | 588,943 | 6.86 | 129,950 |
| | | **2.40** | **560,082** | **7.10** | **127,772** |
| | | 2.50 | 535,049 | 7.31 | 125,799 |
| | | 2.60 | 510,646 | 7.54 | 123,801 |
| | Inferred | 2.20 | 457,304 | 5.49 | 80,744 |
| | | 2.30 | 433,119 | 5.67 | 78,992 |
| | | **2.40** | **411,225** | **5.85** | **77,343** |
| | | 2.50 | 384,085 | 6.09 | 75,192 |
| | | 2.60 | 359,880 | 6.33 | 73,206 |

**Table 14-26: Mineral Resource Sensitivity to Reporting Cut-off, BR Domain**

| Resource Type | Category | CoG (Au g/t) | Tonnes | Gold Grade (g/t) | Gold (oz) |
|---|---|---|---|---|---|
| Open Pit | Measured | 0.40 | 4,594,342 | 2.72 | 402,115 |
| | | 0.42 | 4,497,979 | 2.77 | 400,849 |
| | | **0.45** | **4,351,304** | **2.85** | **398,795** |
| | | 0.47 | 4,285,815 | 2.89 | 397,827 |
| | | 0.49 | 4,208,357 | 2.93 | 396,631 |
| | Indicated | 0.40 | 1,963,695 | 2.72 | 171,855 |
| | | 0.42 | 1,940,628 | 2.75 | 171,553 |
| | | **0.45** | **1,887,597** | **2.81** | **170,807** |
| | | 0.47 | 1,864,898 | 2.84 | 170,472 |
| | | 0.49 | 1,840,113 | 2.88 | 170,088 |
| | Inferred | 0.40 | 7,430 | 1.42 | 338 |
| | | 0.42 | 7,423 | 1.42 | 338 |
| | | **0.45** | **7,342** | **1.43** | **337** |
| | | 0.47 | 7,285 | 1.43 | 336 |
| | | 0.49 | 7,285 | 1.43 | 336 |
| Underground | Measured | 2.20 | 494,420 | 6.40 | 101,781 |
| | | 2.30 | 472,141 | 6.60 | 100,169 |
| | | **2.40** | **451,775** | **6.79** | **98,635** |
| | | 2.50 | 437,340 | 6.93 | 97,497 |
| | | 2.60 | 420,429 | 7.11 | 96,110 |
| | Indicated | 2.20 | 1,665,779 | 4.83 | 258,807 |
| | | 2.30 | 1,606,687 | 4.93 | 254,533 |
| | | **2.40** | **1,531,842** | **5.05** | **248,889** |
| | | 2.50 | 1,469,935 | 5.16 | 244,008 |
| | | 2.60 | 1,411,599 | 5.27 | 239,232 |
| | Inferred | 2.20 | 1,318,215 | 4.52 | 258,807 |
| | | 2.30 | 1,223,105 | 4.70 | 254,533 |
| | | **2.40** | **1,145,374** | **4.86** | **248,889** |
| | | 2.50 | 1,077,571 | 5.01 | 244,008 |
| | | 2.60 | 1,019,890 | 5.15 | 239,232 |

**Table 14-27: Mineral Resource Sensitivity to Reporting Cut-off, EG Domain**

| Resource Type | Category | CoG (Au g/t) | Tonnes | Gold Grade (g/t) | Gold (oz) |
|---|---|---|---|---|---|
| Open Pit | Measured | 0.40 | 2,463,181 | 2.70 | 214,150 |
| | | 0.42 | 2,378,900 | 2.79 | 213,041 |
| | | **0.45** | **2,245,870** | **2.92** | **211,191** |
| | | 0.47 | 2,161,160 | 3.02 | 209,943 |
| | | 0.49 | 2,096,932 | 3.10 | 208,950 |
| | Indicated | 0.40 | 5,409,644 | 2.82 | 490,860 |
| | | 0.42 | 5,298,957 | 2.87 | 489,402 |
| | | **0.45** | **5,107,949** | **2.96** | **486,744** |
| | | 0.47 | 4,945,541 | 3.05 | 484,352 |
| | | 0.49 | 4,853,821 | 3.09 | 482,937 |
| | Inferred | 0.40 | 1,051,268 | 1.98 | 66,972 |
| | | 0.42 | 1,014,245 | 2.04 | 66,485 |
| | | **0.45** | **954,749** | **2.14** | **65,657** |
| | | 0.47 | 931,587 | 2.18 | 65,316 |
| | | 0.49 | 909,189 | 2.22 | 64,970 |
| Underground | Measured | 2.20 | 863,677 | 7.62 | 211,553 |
| | | 2.30 | 837,044 | 7.79 | 209,631 |
| | | **2.40** | **819,849** | **7.90** | **208,333** |
| | | 2.50 | 802,165 | 8.02 | 206,939 |
| | | 2.60 | 783,543 | 8.15 | 205,413 |
| | Indicated | 2.20 | 2,504,945 | 5.25 | 423,109 |
| | | 2.30 | 2,378,556 | 5.41 | 413,969 |
| | | **2.40** | **2,255,880** | **5.58** | **404,697** |
| | | 2.50 | 2,145,492 | 5.74 | 396,005 |
| | | 2.60 | 2,043,682 | 5.90 | 387,663 |
| | Inferred | 2.20 | 719,826 | 7.21 | 166,952 |
| | | 2.30 | 677,946 | 7.52 | 163,931 |
| | | **2.40** | **649,687** | **7.75** | **161,795** |
| | | 2.50 | 624,063 | 7.96 | 159,780 |
| | | 2.60 | 598,259 | 8.20 | 157,486 |

## 14.15 Comparison with the Previous Resource Estimate

Changes from the Mineral Resource Estimate effective February 7, 2021 are summarized in Table 14-28.

**Table 14-28: Mineral Resource Estimate Statement for the Project with Comparison to Previous Mineral Resource**

| | | November 15, 2021 | | | February 7, 2021 | | | Percentage Change from February 2021 | |
|---|---|---|---|---|---|---|---|---|---|
| **Resource Type** | **Category** | **Tonnes ('000)*** | **Au (g/t)*** | **Troy Ounces*** | **Tonnes ('000)**** | **Au (g/t)**** | **Troy Ounces**** | **Tonnes ('000)** | **Troy Ounces** |
| Open Pit | Measured | 7,680 | 2.76 | 680,518 | 6,137 | 2.73 | 538,536 | 25.1% | 26.4% |
| | Indicated | 7,988 | 2.89 | 741,220 | 5,743 | 2.99 | 551,287 | 39.1% | 34.5% |
| | Measured + Indicated | 15,668 | 2.82 | 1,421,738 | 11,880 | 2.86 | 1,089,823 | 31.9% | 30.5% |
| | Inferred | 975 | 2.11 | 66,237 | 1,580 | 1.75 | 88,953 | -38.3% | -25.5% |
| Underground | Measured | 1,576 | 7.45 | 377,445 | 1,384 | 7.36 | 327,667 | 13.9% | 15.2% |
| | Indicated | 4,350 | 5.59 | 781,794 | 2,772 | 5.93 | 528,567 | 56.9% | 47.9% |
| | Measured + Indicated | 5,925 | 6.09 | 1,159,239 | 4,156 | 6.41 | 856,234 | 42.6% | 35.4% |
| | Inferred | 2,206 | 5.89 | 418,013 | 3,726 | 5.92 | 709,114 | -40.8% | -41.1% |
| **Combined Open Pit and Underground** | **Measured** | **9,255** | **3.56** | **1,057,963** | **7,521** | **3.58** | **866,203** | **23.1%** | **22.1%** |
| | **Indicated** | **12,338** | **3.84** | **1,523,014** | **8,515** | **3.95** | **1,079,854** | **44.9%** | **41.0%** |
| | **Measured + Indicated** | **21,593** | **3.72** | **2,580,977** | **16,036** | **3.78** | **1,946,057** | **34.7%** | **32.6%** |
| | **Inferred** | **3,181** | **4.73** | **484,250** | **5,306** | **4.68** | **798,067** | **-40.0%** | **-39.3%** |

** Combined Open Pit and Underground Mineral Resources. Open Pit Mineral Resource is based on a 0.45 g/t gold CoG; Underground Mineral Resource is based on a 2.40 g/t CoG. The Mineral Resource Estimate has an effective date of November 15, 2021.*

*** Refer to the Company's technical report entitled "Anaconda Mining Inc., NI 43-101 Technical Report and Mineral Resource Estimate for the Goldboro Gold Project, Eastern Goldfields District, Nova Scotia" with an effective date of February 7, 2021.*

## 14.16 Factors That May Affect the Mineral Resources

Areas of uncertainty that may materially impact the Mineral Resource Estimate include:

- Changes to long-term metal price assumptions.
- Changes to the input values for mining, processing, and G&A costs to constrain the estimate.
- Changes to local interpretations of mineralization geometry and continuity of mineralized zones.
- Changes to the density values applied to the mineralized zones.
- Changes to metallurgical recovery assumptions.
- Changes in assumptions of marketability of the final product.
- Variations in geotechnical, hydrogeological, and mining assumptions.
- Changes to assumptions with an existing agreement or new agreements.
- Changes to environmental, permitting, and social licence assumptions.
- Logistics of securing and moving adequate services, labour, and supplies could be affected by epidemics, pandemics and other public health crises, including COVID-19, or similar such viruses.

## 14.17 Comments on Section 14

The QP is not aware of any environmental, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Technical Report.

The QP is of the opinion that Mineral Resources were estimated using industry-accepted practices and conform to the 2014 CIM Definition Standards and 2019 CIM Best Practice Guidelines. Technical and economic parameters and assumptions applied to the Mineral Resource Estimate are based on Nordmin's internal calculations and feedback from the Company to determine if they were appropriate.

# 15. MINERAL RESERVE ESTIMATE

## 15.1 Introduction

NI 43-101 defines the terms "mineral reserve", "probable mineral reserve" and "proven mineral reserve" have the meanings ascribed to those terms by the Canadian Institute of Mining, Metallurgy and Petroleum, as the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019).

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.

**Modifying Factors** are considerations used to convert Mineral Resources to Mineral Reserves. These include, but are not restricted to, mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social, and governmental factors.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, 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. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report.

The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

## 15.2 Mineral Reserve Estimate

Changes in the following factors and assumptions may affect the Mineral Reserve Estimate:

- Metal prices
- Interpretations of mineralization geometry and continuity of mineralization zones
- Kriging assumptions
- Geomechanical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate
- Operating cost assumptions
- Process plant recoveries
- Mining loss and dilution
- Ability to meet and maintain permitting and environmental license conditions
- Historical mining depletion

Nordmin prepared a Mineral Reserve Estimate for the Project using a combination of Geovia's Whittle 4.7.4 and Geovia's Surpac 2021 software packages for estimating the economic pit limit for the open pit and block model interrogation.

The Mineral Reserve Estimate for the Deposit is based on the resource block model estimated by Nordmin and described in Section 14. The block model contained Measured, Indicated and Inferred Mineral Resources, however only Measured and Indicated Mineral Resources were used. Inferred

Mineral Resources in the block model were not included in the Probable Mineral Reserve and remain classified as waste; Inferred Mineral Resources do not meet the standards required for inclusion in Mineral Reserves.

Mineral Reserves for the Deposit incorporate mining dilution and mining loss assumptions for the open pit mining method.

The reference point at which Mineral Reserves are defined, is the point where the ore is delivered to the processing facility, which includes the ROM stockpile.

The following subsections outline the procedures used to estimate the Mineral Reserves. Following the detailed design of the final pit and a LOM scheduling with the cut-off grade, a total of 15.8 Mt of diluted ore exists inside the mine design. The detailed pit design is discussed in the following sections, while the production plan is discussed in Section 16.

Table 15-1 presents the reserves inside the designed pit.

**Table 15-1: Mineral Reserve Estimate**

| Category | Area | Au Cut-off Grade | Tonnage (t) | Diluted Au Grade (g/t) | Contained Au Metal (oz) |
|---|---|---|---|---|---|
| Probable Mineral Reserve | East Pit | 0.45 g/t | 5,468,300 | 2.54 | 446,000 |
| Probable Mineral Reserve | West Pit | 0.45 g/t | 10,330,600 | 2.12 | 704,200 |
| Probable Mineral Reserve | Overall Total | 0.45 g/t | 15,798,900 | 2.26 | 1,150,200 |

Source: Nordmin, 2021

Notes:

- The independent and Qualified Person for the Mineral Reserve Estimate, as defined by NI 43-101, is Joanne Robinson, P.Eng. of Nordmin Engineering Ltd.
- The effective date of the Mineral Reserves estimate is December 15, 2021.
- The Mineral Reserve Estimate is based metallurgical recovery algorithms, that result in an overall average recovery of 95.8%.
- Metal prices are set at US$1,600/oz gold with an exchange rate assumption of US$1:C$1.25 resulting in C$2,000/oz gold.
- The Mineral Reserve was derived from a pit limit analysis and detailed pit design. A cut-off grade of 0.45 g/t was based on parameters described in Table 15-2.
- The Mineral Reserve Estimate incorporates mining dilution and mining loss assumptions through regularization of block size to 2 m x 2 m x 4 m. An additional 5% mining loss assumption was incorporated. The overall impact is approximately 26% additional tonnes and approximately 8% reduction in Au Metal.

## 15.3 Open Pit Mine Design

Conventional open pit mining methods will be used to extract a portion of the Deposit. This method was selected considering the Deposit's size, shape, orientation, and proximity to the surface. Drilling, blasting, loading, and hauling will be used to mine the open pit material within the designed pit to meet the mine production schedule.

The following sub-section details the aspects of the pit design.

### 15.3.1 Pit Limit Analysis

Economic pit limits were determined using Geovia's Whittle™ 4.7 software which uses the LG algorithm. The LG algorithm progressively identifies economic blocks, taking into account waste stripping that results in a highest possible total value mined within the open pit shell, subject to the specified pit slope constraints.

The pit limit analysis was evaluated on the Deposit, using the Measured and Indicated Mineral Resources.

#### 15.3.1.1 Input Parameters

A 3D geological block model and other economical and operational variables are used as inputs in the software program. These variables include overall pit slope angle, mining costs, processing costs, selling costs, metal prices, metal recoveries, and other variables listed in Table 15-2.

Although these parameters are not necessarily final, a reasonable degree of accuracy is required since the analysis is an iterative process. The economic parameters used at the time of the pit limit analysis may not necessarily conform to those stated in the economic model.

**Table 15-2: Pit Limit Analysis Parameters**

| Currency Used for Evaluation | C$ |
|---|---|
| Block Size | 2.0 m x 2.0 m x 4.0 m |
| Overall Slope Angle | Rock: Varied by Sector – Range 42° – 46°<br>Overburden: 25° |
| Mining Cost<br>includes assumptions for contract mining, Owners Mine Technical team, pit dewatering, reclamation, water treatment | 0.8 MCAF Overburden material<br>$5.10/t_{mined}$ Rock<br>+ $0.016/t per 8 m |
| Process Cost<br>includes assumptions for Milling, G&A, tailings, reclamation, water treatment | $25.75/t_{processed}$ |
| Selling Cost<br>includes transportation, refining, and % payable | $5,00 /oz gold<br>99.95% payable |
| Metal Price | US$1,600/oz<br>US$1.00:C$1.25 Exchange Rate Factor<br>C$2,000/oz |
| Process Recovery | Based on Recovery Curve Algorithms |

| Currency Used for Evaluation | C$ |
|---|---|
| | (GRADE-(0.0262*LN(GRADE)+0.0712))/GRADE*100-0.083 |
| Mining Loss & Dilution | Included within Reblocked / Regularized Block Model<br>Plus 5% factor for Mining Loss within optimization program<br>Overall effect of ~26% additional tonnes and ~8% reduction in Metal |
| Resources Used for Pit Shell Generation | Measured, Indicated |
| Pit Shell Selection | Revenue Factor RF 0.80 for Mine Planning |

###### 15.3.1.1.1 Resource Block Model

The resource model was created using subblocks, smaller than the parent blocks, as a means of improving the resolution of the model at geological boundaries. This technique is designed to maximise the resolution of the in-situ boundaries of the mineralization in the Mineral Resource model. The subcelled resource block model, as described in Section 14, was used as the basis for the pit limit analysis, with the following edits:

- The four block models for the Boston-Richardson, West Goldbrook, East Goldbrook, and Marker Horizon were combined into one model for the open pit analysis
- The regularized model block size is 2 m wide x 2m long x 4m high, created from the sub-celled resource model with parent block size of 2 m wide x 2m long x 2m high and smaller for sub-cell blocks. The resource block model in Datamine format was reblocked to a regularized block model in Datamine format using Deswik.CAD. The codes in the sub-celled resource model for resource categories: Measured, Indicated, and Inferred were applied to blocks in the regularized model based on which code had the greatest volume
- The type of material: soil, till, tailings, and fresh rock were also applied to blocks in the regularized model based on which code had the greatest volume.
- Grades from the sub-celled resource model were applied to the regularized model on a mass weighted basis.
- Non mineralized material was added to the regularized model with the following specific gravities:
    - Soil, 1.60
    - Till, 2.10
    - Historic Tailings, 2.10
    - Fresh Rock, 2.715
    - Previously Mined, 1.00

The block model uses the MTM Zone 4 NAD83 (NAD_1983_CSRS_MTM_4 ) coordinates in the X-Y. The Z coordinates would be considered a local mine grid coordinate system (5000 elevation values). The pit optimization and design were completed using the same coordinate system.

###### 15.3.1.1.2 Mine Dilution and Mine Loss

The block model imported for use in the pit limit analysis is considered a diluted block model.

For mine planning, it was decided to reblock the subblock models to blocks of regular size which matched half the mining bench height. This reblocking process is known as regularization. Ideally, the regularization would reblock the model to a block size that represents the mining selectivity.

The regularization process creates blocks that cut across the mineralized-waste boundaries, thus adding dilution to the PMF material. This also drives some of the regularized blocks below the cut-off grade and these become mining loss.

The model was regularized to 2.0 m x 2.0 m x 4.0 m block size. The envisioned Ore Mining Excavator, at the onset of the analysis, was considered to have a bucket width of approximately 2 m, the rock is planned to be mined on 8 m operating bench height with ore mining being excavated on half bench, if necessary for ore control.

Table 15-3 tabulates a preliminary comparison using a preliminary pit design version.

As a result of the regularization, approximately 24% additional tonnage and 3% metal loss were estimated to be incorporated into the model. An additional 5% loss factor assumption was applied within the optimization software.

**Table 15-3: Comparison Regularized Block Model with Subcelled Resource Block Model**

| Au COG g/t | | Resource Category | SUBCELLED MODEL | | | | PARENT MODEL – 2x2x2 | | | | REBLOCK MODEL – 2x2x4 | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | Tonnage Kt | Au g/t | Contained Au Metal kg | | Tonnage Kt | Au g/t | Contained Au Metal kg | | Tonnage Kt | Au g/t | Contained Au Metal kg |
| 0.45 | Above COG | Measured | 6,508 | 2.890 | 18,808 | | 8,251 | 2.425 | 20,008 | | 8,620 | 2.301 | 19,839 |
| | | Indicated | 5,247 | 3.035 | 15,927 | | 5,627 | 2.507 | 14,106 | | 6,007 | 2.317 | 13,917 |
| | | **Subtotal** | **11,755** | **2.955** | **34,735** | | **13,878** | **2.458** | **34,114** | | **14,627** | **2.308** | **33,756** |
| | | Inferred | 669 | 1.780 | 1,191 | | 737 | 1.506 | 1,110 | | 759 | 1.370 | 1,040 |
| | Below COG | Measured | 53,778 | 0.044 | 2,380 | | 51,867 | 0.052 | 2,722 | | 51,348 | 0.057 | 2,916 |
| | | Indicated | 28,052 | 0.039 | 1,081 | | 27,647 | 0.047 | 1,290 | | 28,288 | 0.053 | 1,511 |
| | | Inferred | 8,956 | 0.018 | 164 | | 8,839 | 0.024 | 210 | | 8,921 | 0.029 | 261 |
| | | **Subtotal** | **91,456** | **0.053** | **4,815** | | **89,090** | **0.060** | **5,333** | | **89,316** | **0.064** | **5,728** |
| | | Difference | | | | | 2,123 | | -621 | | 2,871 | | -979 |
| | | Variance | | | | | 18% | | -2% | | 24% | | -3% |

#### 15.3.1.1.3 Overall Slope Angle

Preliminary overall pit slope angle assumptions were based on information from the previous Preliminary Economic Assessment study for the Project, prepared by Nordmin, July 2021.

The preliminary analysis included 10 geotechnical sectors with the slope angle assumptions shown in Table 15-4.

**Table 15-4: Preliminary Basis for Overall Slope Angle Assumptions**

| Parameter | | Sector | | | | |
|---|---|---|---|---|---|---|
| | | 1 and 6 | 2 and 7 | 3 and 8 | 4 | 5 and 9 |
| Slope Orientation | | 185 º | 275 º | 355 º | 355 º | 85 º |
| Geomechanical Classes | | II | II | II | III | II / III |
| Bench Face Angle | | 80º | 85º | 85º | 70º | 70º |
| Bench Height | Operational | 5 m | 5 m | 5 m | 5 m | 5 m |
| | Final Wall | 15 m | 15 m | 15 m | 15 m | 15 m |
| Berm width | | 7.5 m | 7.5 m | 7.5 m | 7.5 m | 7.5 m |
| Inter-ramp Angle (IRA) | | 56º | 59º | 59º | 49º | 49º |

The inter-ramp angles were further reduced to account for a 25 m haul road. The resulting overall slope angles used in the pit limit analysis were in the range of 42$^{o}$ and 46$^{o}$ for the various sectors.

#### 15.3.1.1.4 Operating Costs

The operating costs are preliminary and are used for pit limit analysis, mine planning, and Mineral Reserve Estimate purposes (Table 15-5). Detailed operating costs are developed based on a detailed mine design and plan and discussed in Section 21.

The basis for the operating costs for the pit limit analysis are updated mine contractor budget quotes and where appropriate the results of the previous 2021 PEA study (Nordmin, July 2021).

**Table 15-5: Pit Limit Analysis Cost Estimate Details**

| ITEM | VALUE | UNIT |
|---|---|---|
| **Mining Cost** | | |
| Mining Contractor | 4.75 | $/t mined |
| Owner Mining Technical Team | 0.11 | $/t mined |
| Reclamation | 0.08 | $/t mined |
| Pit Dewatering Allowance | 0.06 | $/t mined |
| Water Treatment | 0.10 | $/t mined |
| **Total** | **5.10** | **$/t mined** |
| Allowance for Vertical Depth | 0.016 | $/t/8 m |
| **Processing Cost** | | |
| Process Operations | 12.91 | $/t processed |
| Incremental Ore Mining | -0.10 | $/t processed |
| G&A | 7.83 | $/t processed |
| Water Treatment | -0.31 | $/t processed |
| Tailings Operations, Progression | 2.32 | $/t processed |
| Reclamation | 2.55 | $/t processed |
| Habitat Compensation | 0.56 | $/t processed |

| ITEM | VALUE | UNIT |
|---|---|---|
| **Total** | **25.75** | **$/t processed** |
| **Selling Cost** | | |
| Refining | 3.90 | $/oz |
| Transportation | 1.10 | $/oz |
| **Total (rounded)** | **5.00** | **$/oz** |

###### 15.3.1.1.5 Metallurgical Recovery

The assumptions for the metallurgical recoveries are based on the preliminary Grade-Recovery curve developed by AUSENCO. The equation used:

- [(G – (0.0262 X LN(G) + 0.0712))/G X 100 – 0.083]/100

Where G= grade, 0.083 represents plant loss, equation as written returns a value of 0.95 for example.

The equation is applied to the Au grade of each block in the block model.

###### 15.3.1.1.6 Metal Price

A selling price of C$2,000/oz gold was used in the pit limit analysis. This is based on US$1,600/oz gold with an exchange factor of US$1.00:C$1.25.

###### 15.3.1.1.7 Selling Costs

Selling costs were based on the PEA results and rounded up. The assumptions were based on information provided by the Company and their experience with their existing operation.

###### 15.3.1.1.8 Boundary Constraints

Three boundary constraints were used in the pit limit analysis for the Deposit:

- A 40 m (X-Y) offset from the Natural Gas pipeline easement, on the west side of the property
- A 50 m (X-Y) offset from the edges of the Gold Brook Lake
- A 20 m (X-Y) offset from the centerline of the stream flowing south from Gold Brook Lake

###### 15.3.1.1.9 Cut-off Grade

To classify the material contained within the open pit limits as material for processing or material for waste, the milling cut-off grade is used. This break-even cut-off grade is calculated to cover the costs of processing, general and administrative costs, and selling costs using the economic and technical parameters listed in Table 15-2. Mineral Resource material contained within the pit shell and above the cut-off grade is classified as PMF, while resource material below the cut-off grade is classified as waste.

The cut-off grade has been estimated to be 0.45 g/t gold for the open pit.

##### 15.3.1.2 Pit Limit Analysis Results

The pit limit analysis process results in a series of nested pit shells, each corresponding to a RF. The revenue factor scales the metal price only, and no costs are factored by the RF. The RF 1 corresponds to a gold price 2,000 C$/oz. Table 15-6 summarizes the LG nested pit shell results for the Deposit at a selection of revenue factors.

**Table 15-6: LG Nested Pit Shell Results**

| RF | Pit Shell Label | Total Rock (Mt) | Waste (Mt) | PMF (Mt) | Strip Ratio | Gold Grade (g/t) |
|---|---|---|---|---|---|---|
| 0.3 | 11 | 12.3 | 11.1 | 1.1 | 10.0 | 5.87 |
| 0.4 | 16 | 22.1 | 19.8 | 2.3 | 8.8 | 4.54 |
| 0.5 | 21 | 52.1 | 47.2 | 5.0 | 9.5 | 3.68 |
| 0.6 | 26 | 66.5 | 59.8 | 6.7 | 8.9 | 3.22 |
| 0.7 | 31 | 114.0 | 103.4 | 10.6 | 9.7 | 2.84 |
| 0.8 | 36 | 150.2 | 136.1 | 14.1 | 9.6 | 2.55 |
| 0.9 | 41 | 172.0 | 155.4 | 16.6 | 9.4 | 2.36 |
| 1 | 46 | 182.5 | 164.1 | 18.4 | 8.9 | 2.22 |
| 1.1 | 51 | 196.0 | 175.6 | 20.4 | 8.6 | 2.09 |

#### 15.3.1.2.1 Pit Optimization Methodology

The nested pit shell generation step does not take into consideration the time value of money. This factor is considered during the schedule optimization step of the Analysis.

A basic schedule is applied to the nested pit shells to produce a ‘pit-by-pit’ graph. An objective of the pit-by-pit graph is to illustrate the impact of scheduling on the pit shells and to provide guidance on selection of an optimum pit shell to use as a guide in the detailed pit design. The tonnages reported within the pit shells is based on the base case metal prices, which is 2,000 C$/oz. The optimum pit limit is chosen by estimating the pit size where an incremental increase in pit size does not significantly increase the pit resource and where the economic return starts to decline.

Figure 15-1 illustrates the pit-by-pit graph generated for the Deposit. The three schedules represented are:

- The **Best Case** schedule consists of mining out nested Pit Shell 1, the smallest pit, and then mining out each subsequent pit shell from the top down, before starting the next pit shell. This schedule is seldom feasible because the pushbacks are usually too narrow. Its usefulness lies in setting an upper limit to the achievable Present Value (PV).
- The **Worst Case** schedule consists of mining each bench completely before starting on the next bench. This schedule’s usefulness lies in setting a lower limit to the Present Value. If, as is sometimes the case, worst case and best case schedules differ by only a few percent then, for that pit, mining sequence is relatively unimportant from an economic point of view.
- If, as is usually the case, the difference between worst and best case is significant, a more realistic mining schedule can be approximated, between the two extremes, by specifying a sequence of pit outlines to push back to, this is the **Specified Case**. Chosen pushbacks should satisfy mining constraints and produce a PV curve that is as close as possible to the best case PV curve.

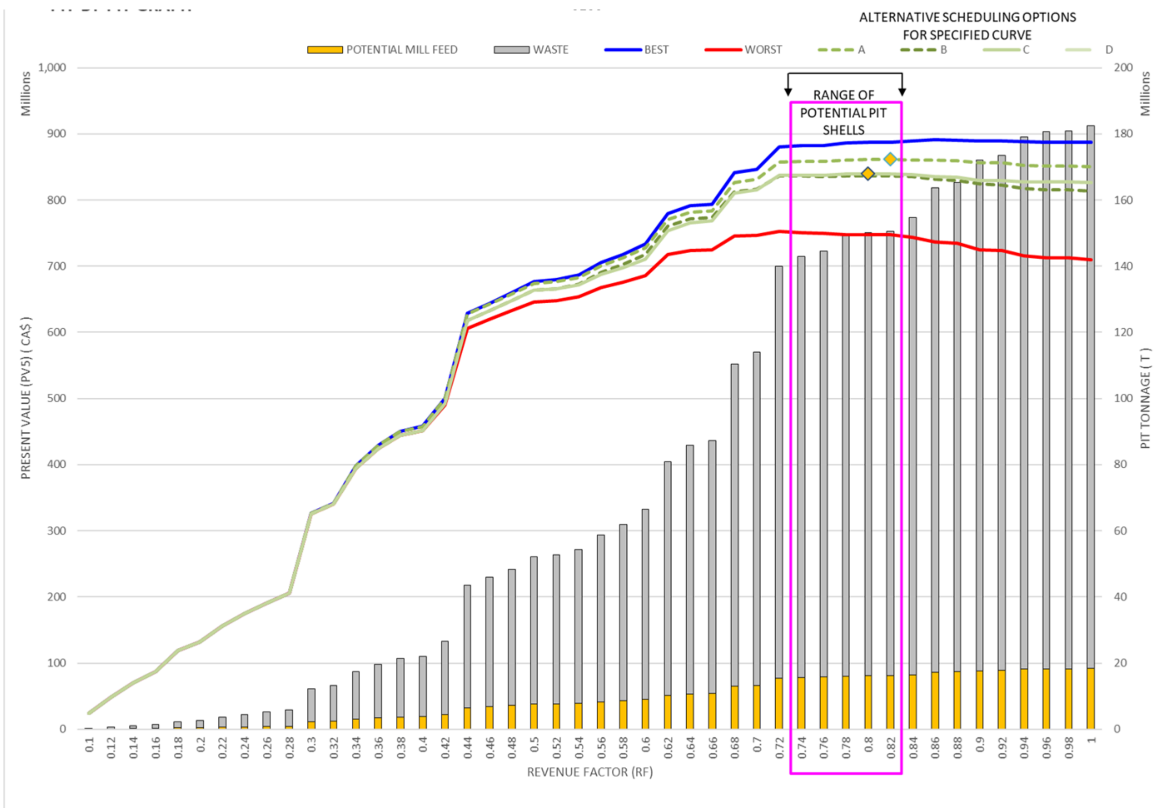



*Figure 15-1: Pit optimization results, pit-by-pit graph*

**Note:** The Present Value (PV5) shown on Figure 15-1 is used only as a guide in pit shell selection.

In choosing an optimum pit, it is important to understand the level of risk the project is willing to accept. By accepting more risk, the pit size can increase, thereby increasing mining tonnages at the possibility that the NPV of the pit could suffer.

The pit shells selected to use as a guide for the detailed pit design is pit shell with RF 0.80, which was the pit shell with the maximum PV on the specified curve in Figure 15-1.

### 15.3.1.3 Open Pit Design

The objective of the detailed pit design is to follow the outline of the selected pit shell while incorporating bench designs, minimum mining widths, and haulage ramps. Figure 15-2 and Table 15-7 summarizes the slope design sectors, the slope design recommendations, and summarizes the slope design assumptions applied, while Table 15-8 summarizes the haul ramp design assumptions. Optimization of pit design, ramp locations, pit exits, and interaction of pit walls with historical mine openings should be considered in the next level of study.

#### 15.3.1.3.1 Pit Slope Stability Considerations

Nordmin commissioned Optimize to develop a FS level geotechnical study for the open pit mining for the Project.

Optimize undertook kinematic and limit equilibrium stability analysis to evaluate bench scale, inter-ramp, and overall slope stability of proposed pit slopes in order to generate design recommendations. Details from the analysis are presented in Section 16.

A summary of the details with respect to pit limit analysis and pit design are presented in this section. Figure 15-2 illustrates the geotechnical design sectors and Table 15-7 illustrates the slope design parameters used in the pit design.

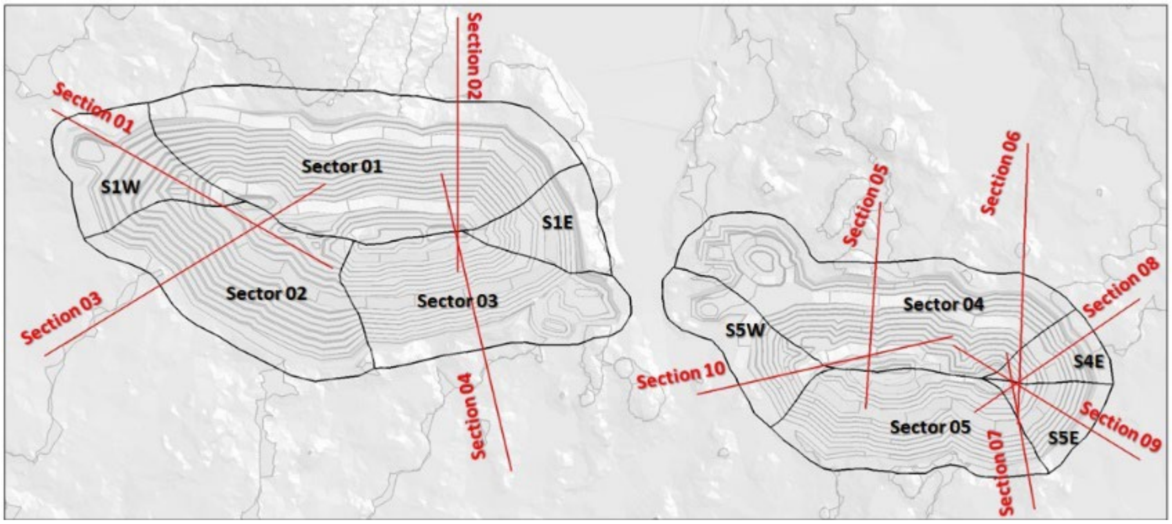



*Figure 15-2: Geotechnical design sectors*

**Table 15-7: Slope Design Assumptions**

| Zone Label | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Previous Sector Label | | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 7 | 8 | 9 | OVB |
| FS Sector Label | | S01 | S1E | S03 | S02 | S1W | S04 | S4E | S5E | S05 | S5W | OVB |
| Previous Recommendations | | | | | | | | | | | | |
| Bench Face Angle | o | 80 | 85 | 85 | 70 | 70 | 80 | 85 | 85 | 85 | 70 | |
| Berm Width | m | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | |
| Bench Height | m | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | |
| Interramp Angle | o | 56 | 59 | 59 | 49 | 49 | 56 | 59 | 59 | 59 | 49 | |
| | *Calculated* | *55.9* | *59.6* | *59.6* | *49.2* | *49.2* | *55.9* | *59.6* | *59.6* | *59.6* | *49.2* | |
| Interim Memo | | | | | | | | | | | | |
| Bench Face Angle | o | 80 | 85 | 85 | 70 | 70 | 80 | 85 | 85 | 85 | 85 | |
| Berm Width | m | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | 7.5 | |
| Bench Height | m | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | |
| Interramp Angle – Measured from Previous design | o | 56 | 59 | 59 | 48 | 43 | 56 | 59 | 57 | 55 | 55 | |
| **Used in FS Pit Design** | | | | | | | | | | | | |
| FS Bench Face Angle | o | 80 | 85 | 85 | 70 | 70 | 80 | 85 | 80 | 80 | 70 | 40 |
| FS Berm Width | m | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 | 8 |
| FS Bench Height | m | 16 | 16 | 16 | 16 | 16 | 16 | 16 | 16 | 16 | 16 | 8 |
| FS Interramp Angle | o | 55.9 | 59.6 | 59.6 | 49.2 | 49.2 | 55.9 | 59.6 | 55.9 | 55.9 | 49.2 | 25 |

According to the analysis, and summarized in Section 16, the change in bench height and berm width did not significantly impact the factor of safety calculated since most of the rock varies between Geomechanical classes II and III, which are notoriously good strength classes. However, as mentioned before, the height increase could lead to more instabilities in the bench scale. However, the increase in the berm width is considered a conservative approach with plus 0.3 m wider than the formula used to design this parameter. This conservative approach could absorb potential falling blocks. The geometrical changes to the bench height and berm width are not considered an issue if the inter-ramp and overall slope are maintained for each of the sectors.

##### 15.3.1.3.2 Haul Ramp Design

The haul ramp design is based on the largest truck planned for the pit. As the project is planned to be operated by mining contractor, the ultimate haul truck selection is not known at this time. However, based on budgetary quotes received for the FS level study, the largest haul truck noted from the quotations was a 90- tonne rigid frame truck. Table 15-8 summarizes the calculations for the haul ramp width. Figure 15-3 illustrates a typical haul ramp profile.

**Table 15-8: Ultimate Pit Design Assumptions – Haul Ramp Design**

| Item | Units | Value |
|---|---|---|
| **Haul Truck Parameters** | | |
| Example Model | | CAT 777 |
| Payload (T, Heaped 2:1) | Tonne | 90 |
| Operating Width, W | m | 6.1 |
| Width Factor (Of Truck Width) | | |
| Double Lane | | 3x |
| Single Lane | | 2x |
| Running Surface Width | | |
| Double Lane | m | 18.3 |
| Single Lane | m | 12.2 |
| **Safety Berm Parameters** | | |
| Tire Type | | 27.00R49 |
| Tire Overall Diameter | m | 2.6 |
| Factor (Of Tire Size) | | 0.5 |
| Berm Height (Calculated) | m | 1.3 |
| Berm Height (Minimum) | m | 1.0 |
| Slope | degrees | 37 |
| Road Berm Allowance | m | 3.5 |
| **Road Drainage Ditch Parameters** | | |
| Road Drainage Allowance | m | 2.5 |
| **Total Ramp Width** | | |
| Double Lane | m | 25 |
| Single Lane | m | 18 |
| **Other Ramp Design Parameters** | | |
| Ramp Gradient | % | 10 |

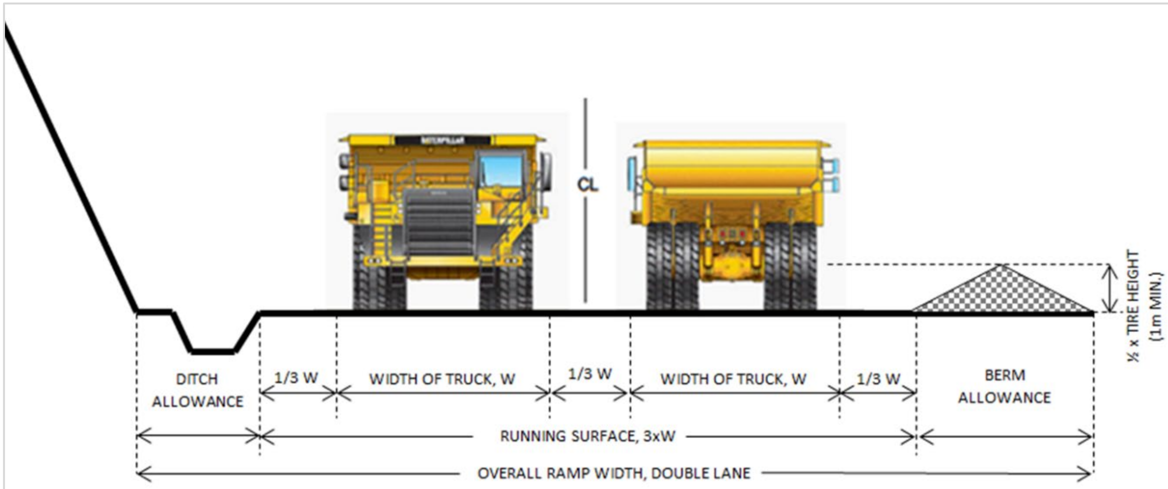
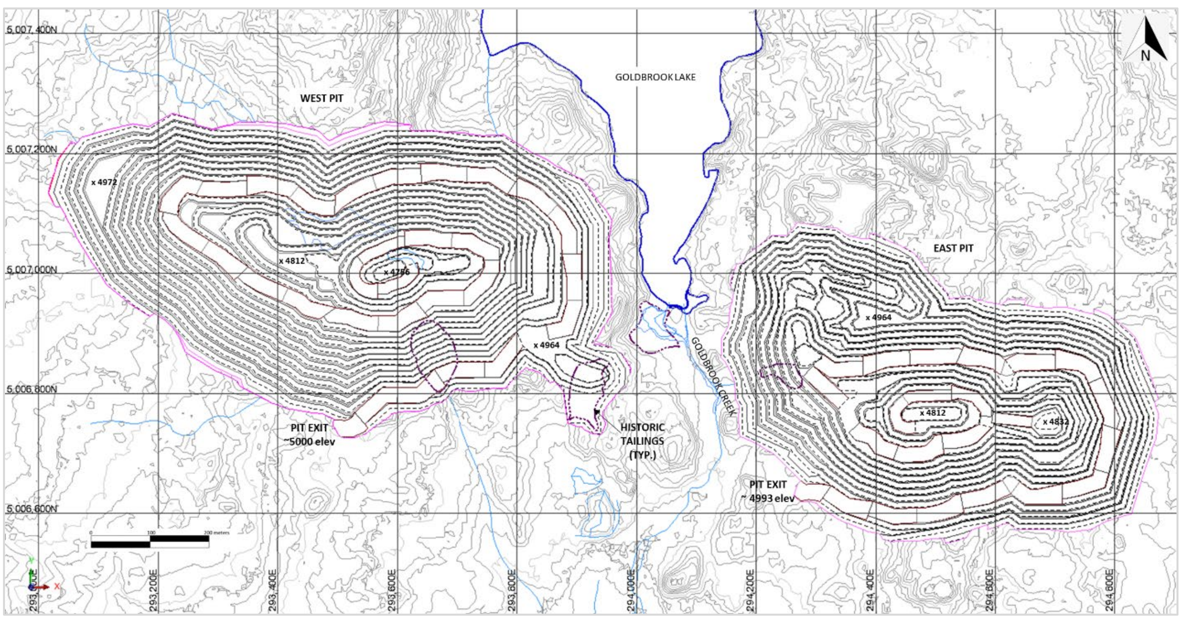



*Figure 15-3: Schematic of haul ramp geometry*

Access to the final three bench levels (24 m vertical) have been designed with single lane access.

#### 15.3.1.4 Pit Design Results

A pit design was created using the pit limit analysis pit shell as a guide and the slope design parameters described in previous section. Figure 15-4 illustrates the pit design.




Source: Nordmin 2021

*Figure 15-4: Ultimate pit design*

The pit entrances are on the south side at approximately 5000 Level. The pit designs were adjusted to incorporate all of the historic tailings area if they were within the pit limit. Table 15-9 summarizes the planned material quantities. Table 15-10 summarizes the approximate dimensions of the ultimate pit design.

**Table 15-9: Ultimate Pit Design Results, Pit Contents**

| Rock Category | Tonnes (T) | Au (g/t) |
|---|---|---|
| **MILL FEED MINED** | | |
| East Pit | 5,468,300 | 2.54 |
| West Pit | 10,330,600 | 2.12 |
| **Total** | **15,798,900** | **2.26** |
| **WASTE MINED** | | |
| NPAG Rock | 107,539,400 | |
| PAG1 Rock | 10,721,000 | |
| PAG2 Rock | 234,100 | |
| Till Overburden | 7,522,800 | |
| Soil Overburden | 563,400 | |
| Historic Tailings Overburden | 170,200 | |
| **Total** | **126,750,800** | |
| **Total Material Mined** | **142,549,700** | |
| LOM Strip Ratio | 8.0 | |

Source: Nordmin 2021

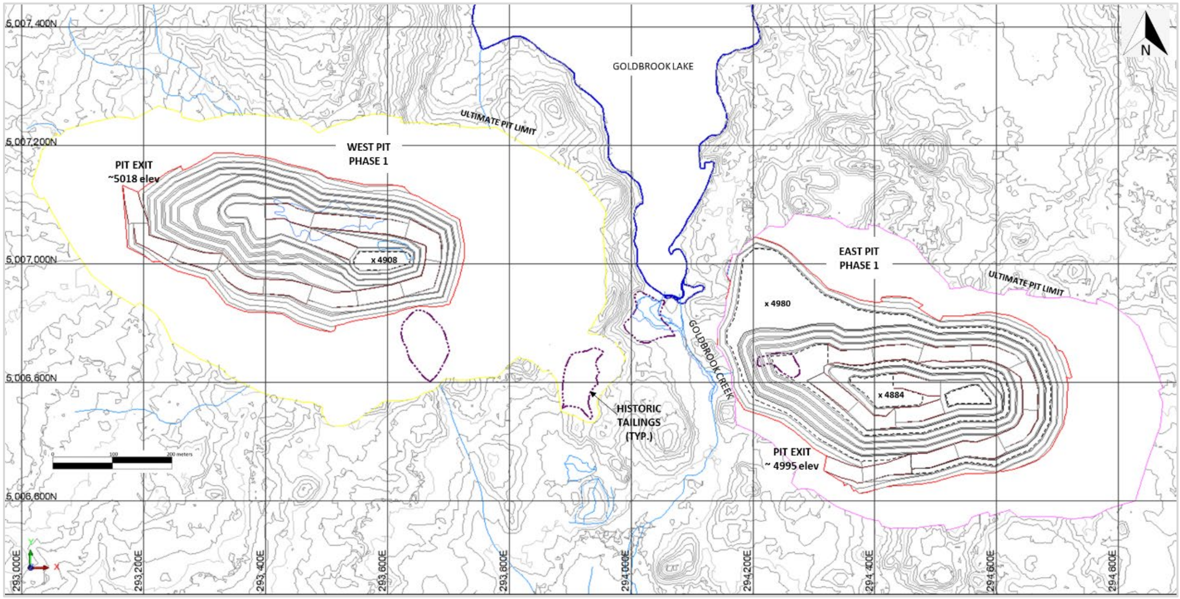
**Table 15-10: Ultimate Pit Design Results, Approximate Pit Dimensions**

| Item | Unit | Goldboro Pit |
|---|---|---|
| Length | m | West Pit – 1025 m<br>East Pit – 775 m |
| Width | m | West Pit – 520 m<br>East Pit – 410 m |
| Depth | m | West Pit – 250 m<br>East Pit – 190 m |

Source: Nordmin 2021

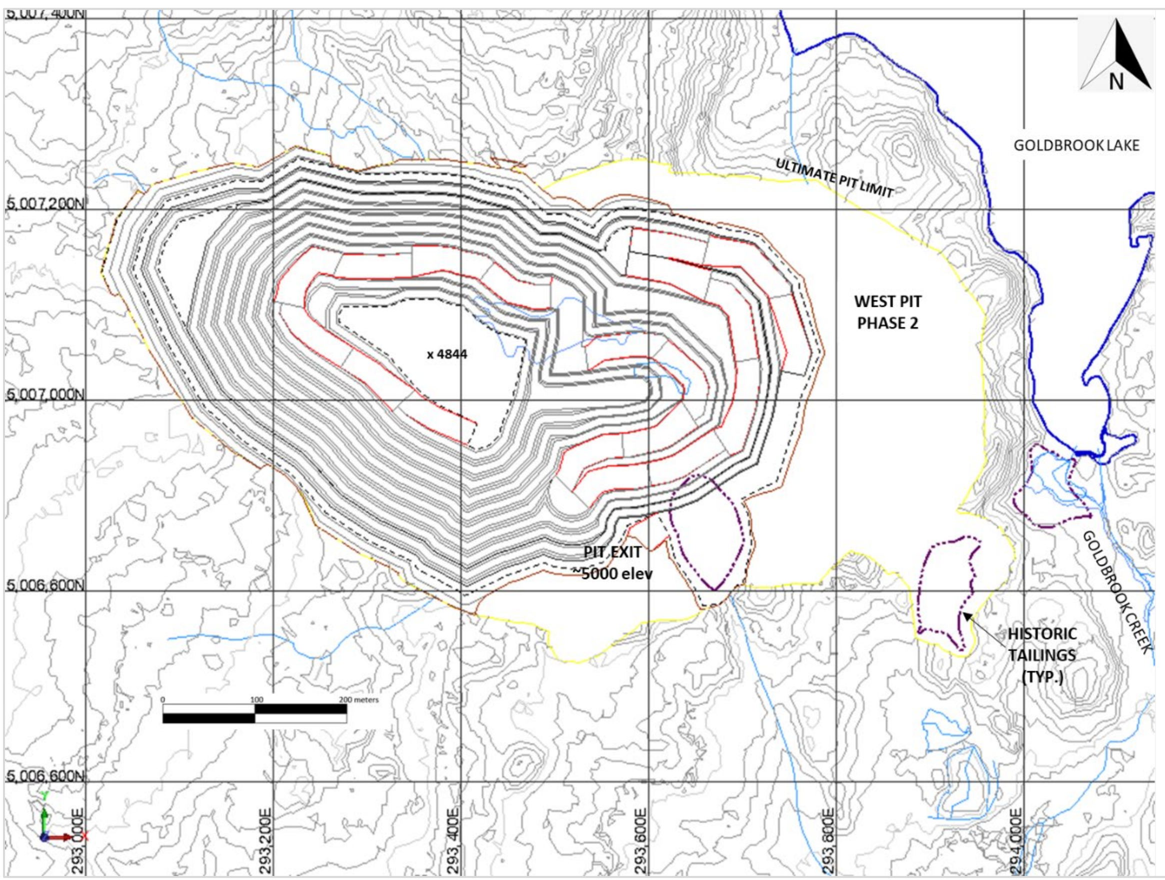
As part of the planning and scheduling process, the intermediate pits leading to ultimate pit limits are determined to see how the pit will evolve over time. In the pit limit analysis using Whittle software, a number of nested pit shells are generated which use varying metal prices, from a low to a high value. This generated a number of pit shells of increasing size and decreasing average unit value per tonne of ore contained within each pit shell. Smaller RF nested pit shells, pit shell 17 (RF0.42) and pit shell 26 (RF0.60), were used as an approximate reference to represent the initial phasing.

The open pit mine plan for the FS assumed a total of five phases – the initial pit in the East Pit, the initial pit in the West Pit, an intermediate phase in the West Pit, expansion to the Ultimate Pit in the East Pit and expansion to the Ultimate Pit in the West Pit. Figure 15-5 and Figure 15-5 illustrate the phase designs for the pit.




Source: Nordmin 2021

*Figure 15-5: Phase design, Initial Phase*

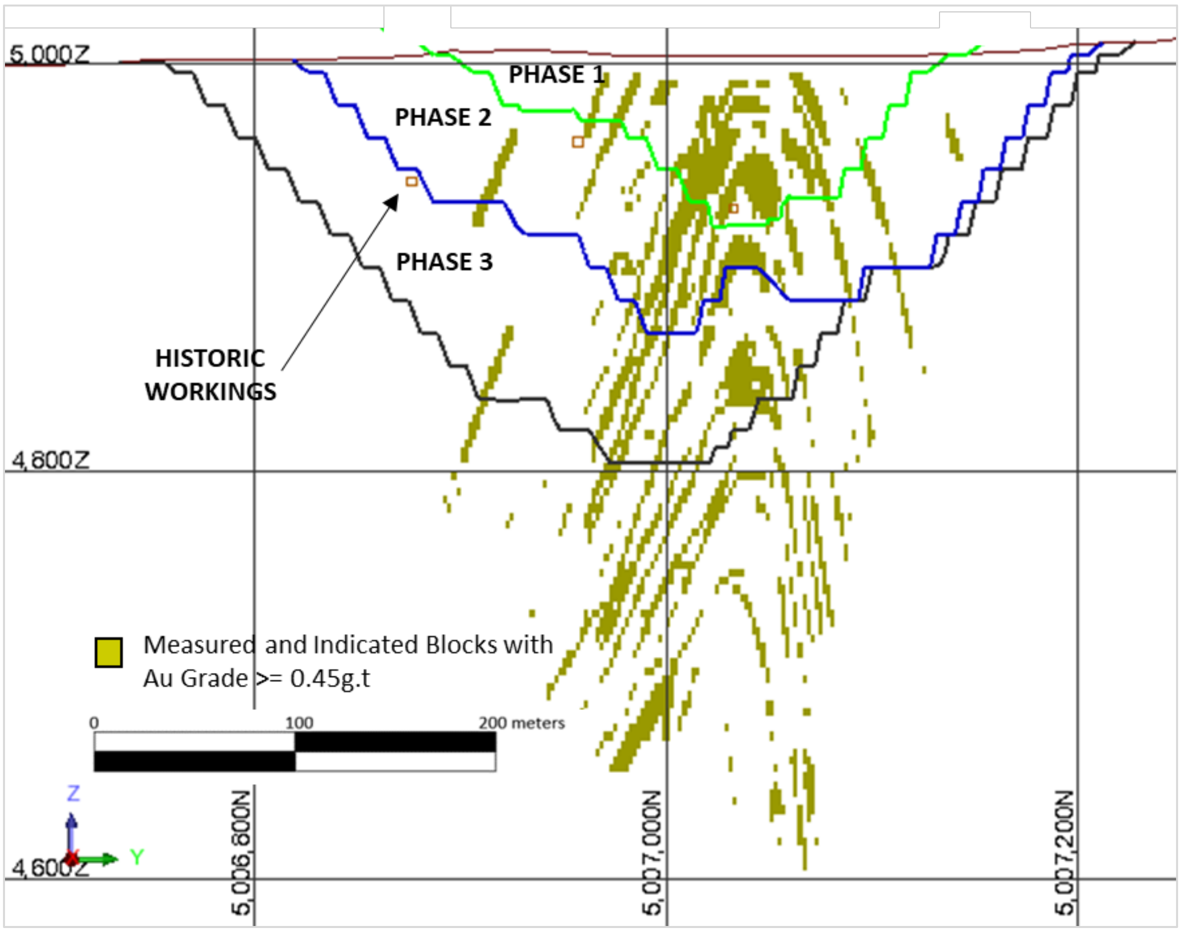



Source: Nordmin 2021

*Figure 15-6: Phase design, West Pit Phase 2*

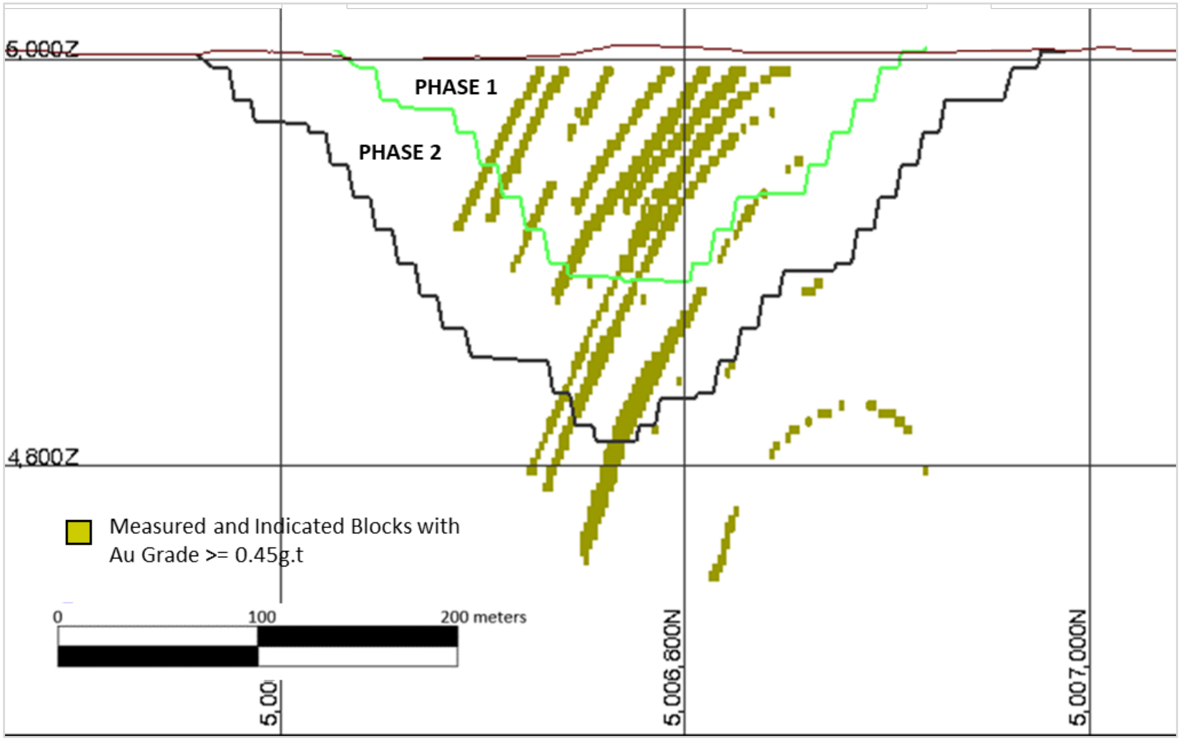
Figure 15-7 through Figure 15-9 illustrate various sections through the ultimate pit showing the FS level pit design, phases, and the Measured and Indicated resource blocks above cut-off grade.

Figure 15-10 to Figure 15-12 illustrate plan view sections showing the FS level pit design, phases, and the Measured and Indicated resource blocks above cut-off grade.



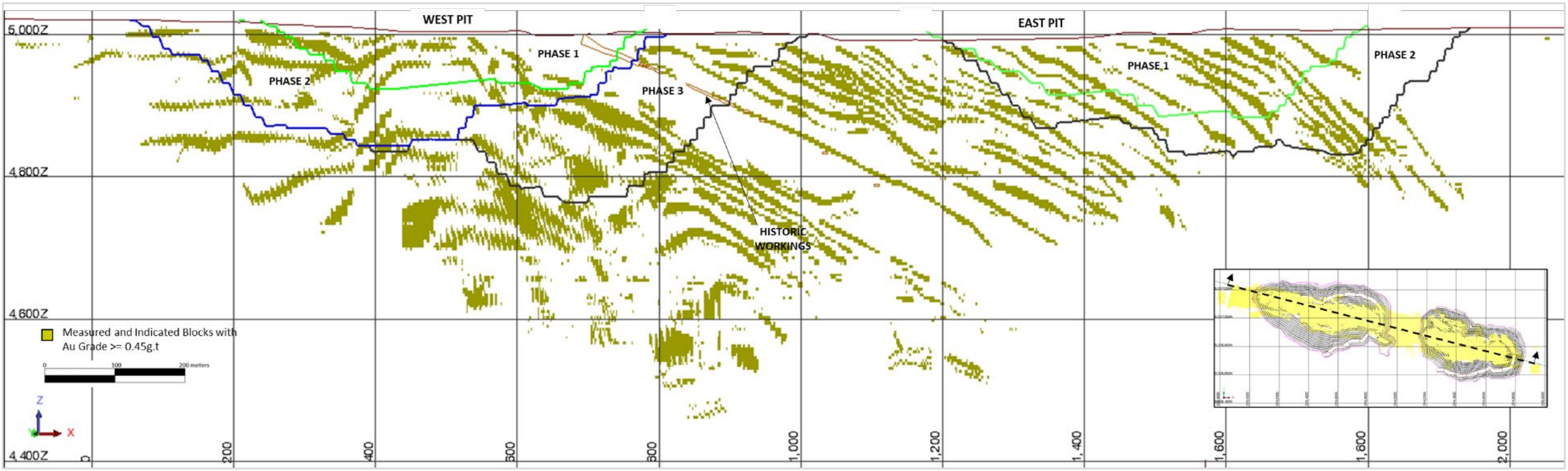

Source: Nordmin 2021

*Figure 15-7: Cross section 293500E (looking West), West Pit*



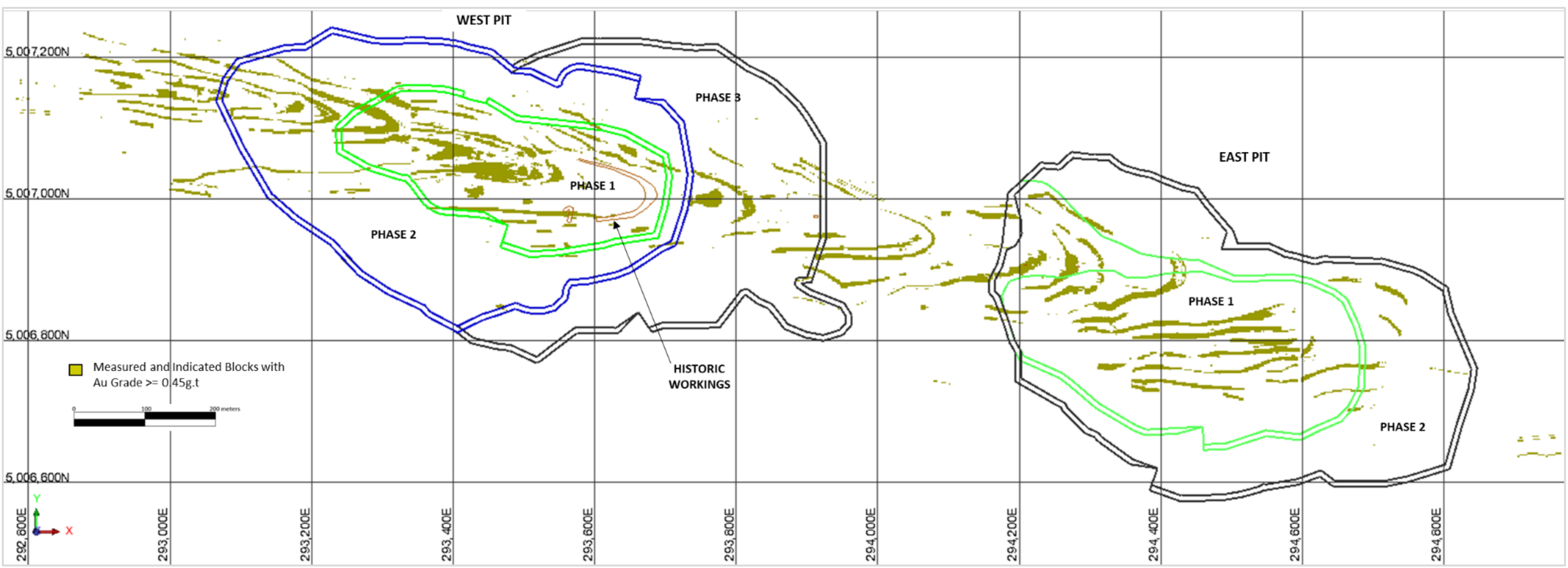

Source: Nordmin 2021

*Figure 15-8: Cross section 294500E (looking West), East Pit*



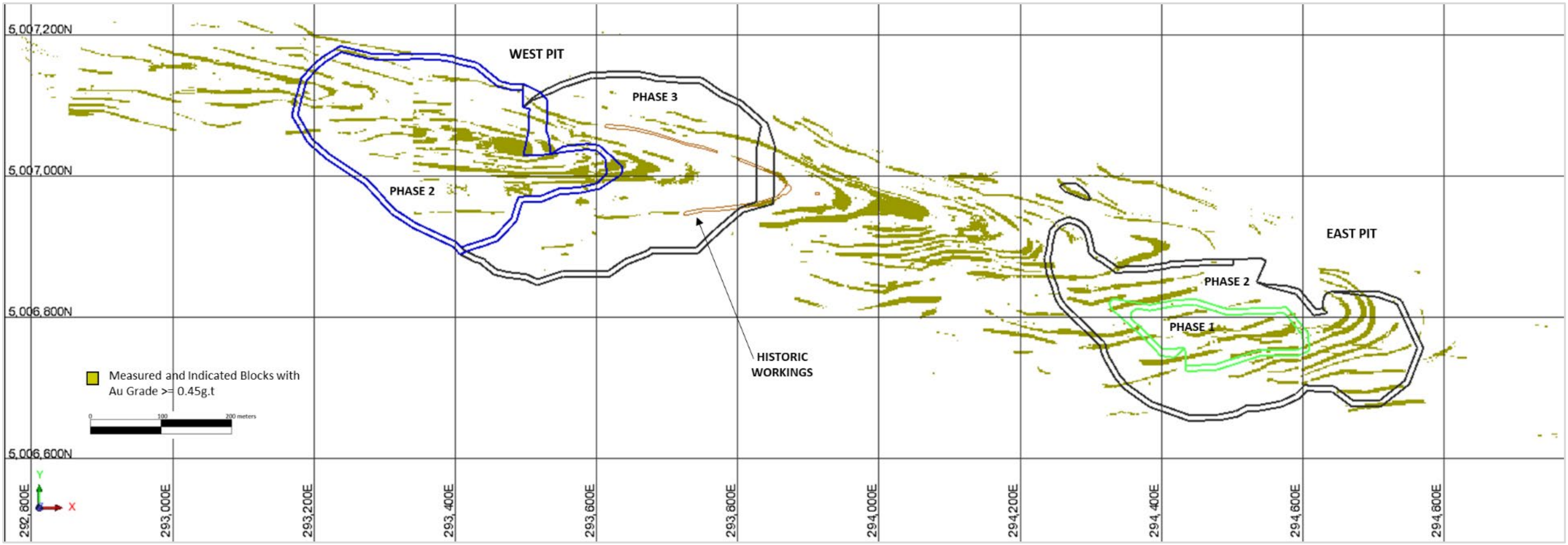

Source: Nordmin 2021

*Figure 15-9: Long Section (looking Northeast)*



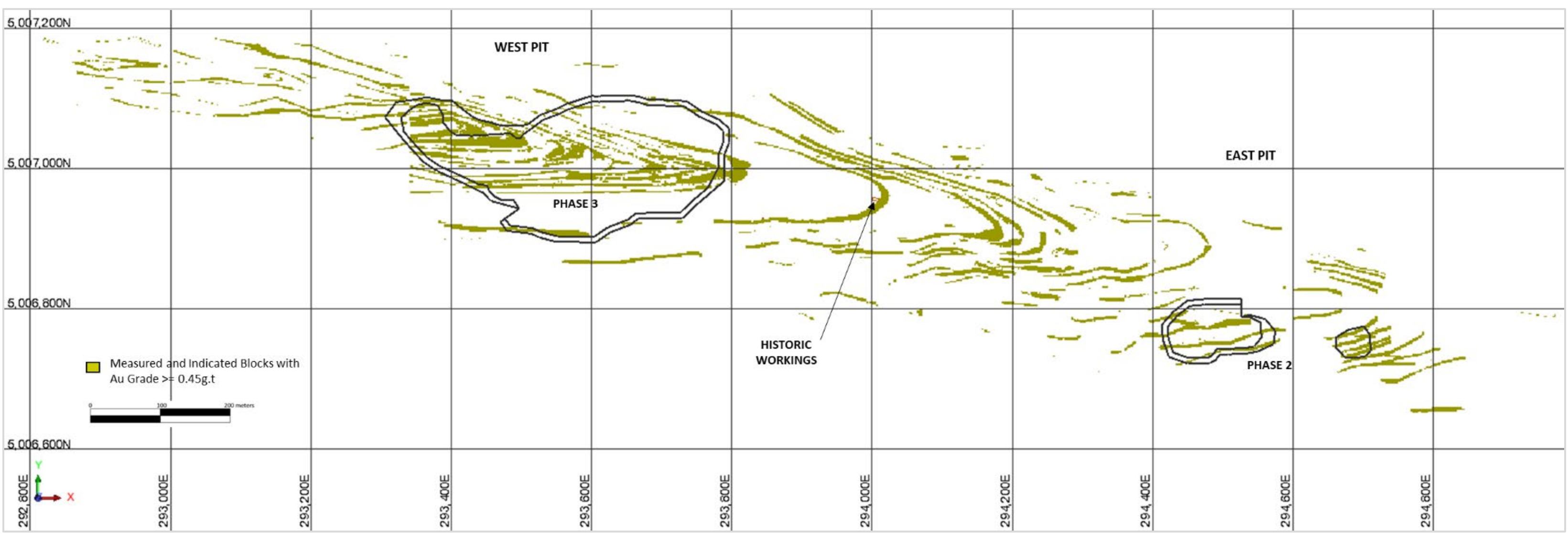

Source: Nordmin 2021

*Figure 15-10: Plan view 4980 elevation*




Source: Nordmin 2021

*Figure 15-11: Plan view 4900 elevation*




Source: Nordmin 2021

*Figure 15-12: Plan view 4836 elevation*

# 16. MINING METHODS

## 16.1 Introduction

This section outlines the parameters and procedures used by Nordmin to perform the FS level mine planning work for the Project at a proposed mill feed production rate of 1.46 Mtpa.

This FS utilizes the Mineral Reserve described in Section 15 with an effective date of December 15, 2021.

Open pit mining was considered a viable option for the study given that the mineralization is on or near surface.

Open pit mining will include conventional drilling and blasting with a combination of a backhoe type excavator, hydraulic excavator, and front-end loader type excavator loading broken rock into haul trucks, which will haul the material from the bench to the crusher, ROM stockpile or waste stockpiling areas depending on the material type. Ancillary equipment includes dozers, graders, and various maintenance, support, service and utility vehicles.

This Technical Report considers a mining contractor operator scenario.

## 16.2 Summary

The operation scenario studied for the FS involves:

- Open Pit mining at an average mining rate of 12.8 Mt per year.
- Gold process facility with a 1.46 Mtpa (4,000 t/d) capacity.
- Approximate 6-month ramp up period in Year 1 (YR1) for process facility.
- 1-year pre-production mining period to coincide with Tailings Management Facility Initial Stage development.

The FS is based on a conventional truck-shovel open pit mining operation within two pits. The open pit production period is approximately 10.9 years with 1 year of pre-production (prior to process plant start-up).

Details of the open pit LOM plans are summarized in Table 16-1.

**Table 16-1: Summary of Open Pit LOM Mine Plan**

| | OVERBURDEN | | | WASTE ROCK | | | TOTAL WASTE | MILL FEED MINED | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | TILL | SOIL | HISTORIC TAILS | NPAG | PAG1 | PAG2 | | MILL FEED | | CONTAINED OUNCES |
| | Kt | Kt | Kt | Kt | Kt | Kt | Kt | kt | g/t Au | Oz Au |
| **PRE-PROD Y-1** | 1,398 | 178 | 29 | 2,161 | 67 | 1 | 3,835 | 221 | 2.32 | 16,494 |
| **YR1 – H1** | 1,534 | 88 | 16 | 2,140 | 166 | 3 | 3,948 | 318 | 2.14 | 21,853 |
| **YR1 – H2** | 1,079 | 78 | 0 | 4,745 | 607 | 10 | 6,519 | 730 | 2.34 | 54,936 |
| **YR1 TOTAL** | 2,614 | 166 | 16 | 6,885 | 773 | 13 | 10,467 | 1,048 | 2.28 | 76,789 |
| **YR2** | 1,861 | 97 | 13 | 10,308 | 794 | 37 | 13,110 | 1,460 | 2.33 | 109,403 |
| **YR3** | 190 | | 10 | 9,273 | 795 | 25 | 10,294 | 1,460 | 2.54 | 119,140 |
| **YR4** | 4 | | | 13,010 | 1,181 | 26 | 14,221 | 1,460 | 1.82 | 85,494 |
| **YR5** | | | | 15,953 | 1,478 | 33 | 17,464 | 1,460 | 1.87 | 87,543 |
| **YR6** | | | | 8,583 | 539 | 12 | 9,134 | 1,460 | 2.57 | 120,616 |
| **YR7** | 1,456 | 122 | 101 | 8,738 | 730 | 21 | 11,169 | 1,460 | 2.58 | 120,933 |
| **YR8** | | | | 15,399 | 1,152 | 36 | 16,588 | 1,460 | 1.95 | 91,300 |
| **YR9** | | | | 10,550 | 1,657 | 22 | 12,229 | 1,600 | 2.21 | 113,677 |
| **YR10** | | | | 5,557 | 1,316 | 7 | 6,880 | 1,960 | 2.16 | 135,962 |
| **YR11** | | | | 1,123 | 238 | 0 | 1,361 | 750 | 3.02 | 72,854 |
| **LOM** | **7,523** | **563** | **170** | **107,539** | **10,721** | **234** | **126,751** | **15,799** | **2.26** | **1,150,205** |

## 16.3 Open Pit Evaluation

Conventional open pit mining methods will be used to extract a portion of the Deposit. This method was selected considering the Deposit's size, shape, orientation, and proximity to the surface. Drilling, blasting, loading, and hauling will be used to mine the open pit material within the designed pit to meet the mine production schedule.

**Table 16-2: Planned Open Pit Mining Inventory, Tonnage and Grade by Phase**

| Area Description | | Activity Description | | Units | Totals |
|---|---|---|---|---|---|
| EAST PIT | PHASE 1 | | | | |
| | | MILL FEED | TO ROM | kt | 2,570.3 |
| | | | GRADE | g/t | 2.59 |
| | | WASTE ROCK | | kt | 15,623.6 |
| | | | NPAG | kt | 13,651.3 |
| | | | PAG1 | kt | 274.8 |
| | | | PAG2 | kt | 3.4 |
| | | | TILL | kt | 1,470.6 |
| | | | SOIL | kt | 177.8 |
| | | | HISTORIC TAILINGS | kt | 45.5 |
| | | **TOTAL MINED** | **PHASE 1** | **kt** | **18,193.9** |
| | PHASE 2 | | | | |
| | | MILL FEED | TO ROM | kt | 2,897.9 |
| | | | GRADE | g/t | 2.49 |
| | | WASTE ROCK | | kt | 32,250.3 |
| | | | NPAG | kt | 30,372.0 |
| | | | PAG1 | kt | 497.7 |
| | | | PAG2 | kt | 13.8 |
| | | | TILL | kt | 1,285.6 |
| | | | SOIL | kt | 81.2 |
| | | | HISTORIC TAILINGS | kt | 0.0 |
| | | **TOTAL MINED** | **PHASE 2** | **kt** | **35,148.3** |
| WEST PIT | PHASE 1 | | | | |
| | | MILL FEED | TO ROM | kt | 2,119.5 |
| | | | GRADE | g/t | 2.16 |
| | | WASTE ROCK | | kt | 8,682.3 |
| | | | NPAG | kt | 4,315.6 |
| | | | PAG1 | kt | 2,685.2 |
| | | | PAG2 | kt | 48.6 |
| | | | TILL | kt | 1,544.8 |
| | | | SOIL | kt | 88.1 |
| | | | HISTORIC TAILINGS | kt | 0.0 |
| | | **TOTAL MINED** | **PHASE 1** | **kt** | **10,801.8** |
| | PHASE 2 | | | | |
| | | MILL FEED | TO ROM | kt | 3,802.6 |

| Area Description | | Activity Description | | Units | Totals |
|---|---|---|---|---|---|
| | | | GRADE | g/t | 2.16 |
| | | WASTE ROCK | | kt | 33,504.3 |
| | | | NPAG | kt | 27,444.8 |
| | | | PAG1 | kt | 4,022.9 |
| | | | PAG2 | kt | 153.2 |
| | | | TILL | kt | 1,765.8 |
| | | | SOIL | kt | 94.0 |
| | | | HISTORIC TAILINGS | kt | 23.5 |
| | | **TOTAL MINED** | **PHASE 2** | **kt** | **37,306.9** |
| | PHASE 3 | | | | |
| | | MILL FEED | TO ROM | kt | 4,408.5 |
| | | | GRADE | g/t | 2.07 |
| | | WASTE ROCK | | kt | 36,690.3 |
| | | | NPAG | kt | 31,755.6 |
| | | | PAG1 | kt | 3,240.3 |
| | | | PAG2 | kt | 15.0 |
| | | | TILL | kt | 1,455.9 |
| | | | SOIL | kt | 122.3 |
| | | | HISTORIC TAILINGS | kt | 101.2 |
| | | **TOTAL MINED** | **PHASE 3** | **kt** | **41,098.9** |

### 16.3.1 Waste Rock Disposal Design

Waste rock generated from the open pit will require the development of waste rock storage areas. The waste generated from the open pit includes waste rock, till, historical tailings, and topsoil and organics.

The proposed mine plan will generate approximately 126.8 million tonnes of waste material, which includes overburden. Assuming a swell factor of 30%, a volume of 61.7 million $m^3$ of waste storage is required.

The stockpile size requirements are provided in Table 16-10. The storage capacity has been designed to accommodate waste rock generated from the open pit operations.

**Table 16-3: Waste Rock Storage Area Design Details**

| Area ID | Comment | Volume (M m$^3$) | Top Elev. (m) |
|---|---|---|---|
| NE | Waste material from East Pit, Years 3 to 8 | 10.7 | 130 |
| SE | Waste material from East Pit and West Pit, Year -1 to 4 | 11.1 | 140 |
| NW | Waste material from West Pit, Year 4 to Year 8 | 14.1 | 145 |
| SW | Segregated Till Stockpile, East & West Pits, Year 1 to Year 7 | 3.4 | 100 |
| TMF | NPAG Waste Rock for Embankment | 9.0 | |
| TMF | Historic Tailings & PAG1 Waste Rock | 5.2 | |
| East Pit Backfill | Waste material from West Pit, Year 8 to Year 11 (following completion of East Pit) | 7.5 | |
| Pads, Roads | During construction | 0.2 | |
| Organics | Organics Material to Organic Stockpiles | 0.5 | |

The waste rock storage area has been located in proximity to the mining areas to minimize waste haulage distances. To prepare the area, topsoil is removed and stockpiled in an organics stockpile area for long term storage and later use during reclamation. The foundation is prepared to address any geotechnical concerns.

Waste rock is then end dumped from the haul trucks forming lifts. Trucks dump near, but at a safe distance from, the edge of the lift. Lifts will be constructed such that the final waste rock storage areas have an overall slope angle that does not require rework at closure, thus reducing reclamation costs. A 27° overall reclaimed slope was assumed.

PAG waste rock labelled PAG1 is designed to be deposited in the TMF, co-placed with tailings. Historic Tailings mined within the footprint of the pits is designed to be deposited in the TMF.

NPAG waste rock, PAG waste rock labelled PAG2, and till overburden are designed to be deposited in the waste rock storage areas (NE, SE, NW). The SW storage area is currently designed to be a segregated till stockpile.

Where possible, waste rock material would be used in road construction, pad construction, and tailings embankment, thus reducing the footprint required for the waste rock storage areas.

Design footprints for the waste rock storage areas should be optimized in the next iteration of study to balance among other things, open pit haulage times and environmental objectives.

### 16.3.2 Open Pit Mine Schedule

An open pit mine production schedule was developed with the main objective of delivering 1.46 Mtpa of ore material to the processing facility.

Table 16-11 presents a summary of material movement from the open pit.

**Table 16-4: Open Pit LOM Schedule**

| | | | | | TOTALS | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
| EAST PIT | PHASE 1 | | | | | | | | | | | | | | | | |
| | | MILL FEED | TO ROM | kt | **2,570.3** | 220.8 | 694.3 | 872.4 | 782.8 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | | | GRADE | g/t | **2.59** | 2.32 | 2.40 | 2.48 | 2.95 | | | | | | | | |
| | | WASTE ROCK | | kt | **15,623.6** | 3,834.8 | 4,804.6 | 3,999.4 | 2,984.8 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | | | NPAG | kt | **13,651.3** | 2,161.0 | 4,619.2 | 3,935.3 | 2,935.9 | | | | | | | | |
| | | | PAG1 | kt | **274.8** | 67.0 | 96.3 | 63.3 | 48.2 | | | | | | | | |
| | | | PAG2 | kt | **3.4** | 1.3 | 0.6 | 0.7 | 0.7 | | | | | | | | |
| | | | TILL | kt | **1,470.6** | 1,398.3 | 72.4 | | | | | | | | | | |
| | | | SOIL | kt | **177.8** | 177.8 | | | | | | | | | | | |
| | | | HISTORIC TAILINGS | kt | **45.5** | 29.4 | 16.1 | | | | | | | | | | |
| | | TOTAL MINED | PHASE 1 | kt | **18,193.9** | **4,055.6** | **5,498.9** | **4,871.8** | **3,767.6** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | PHASE 2 | | | | | | | | | | | | | | | | |
| | | MILL FEED | TO ROM | kt | **2,897.9** | 0.0 | 10.9 | 37.6 | 80.2 | 144.2 | 531.6 | 1,174.1 | 875.1 | 44.2 | 0.0 | 0.0 | 0.0 |
| | | | GRADE | g/t | **2.49** | | 5.1 | 3.0 | 1.9 | 1.5 | 1.7 | 2.5 | 3.0 | 5.5 | | | |
| | | WASTE ROCK | | kt | **32,250.3** | 0.0 | 1,989.1 | 3,462.4 | 3,419.8 | 4,355.8 | 7,968.4 | 7,325.9 | 3,624.9 | 104.1 | 0.0 | 0.0 | 0.0 |
| | | | NPAG | kt | **30,372.0** | | 913.8 | 3,196.4 | 3,329.5 | 4,280.6 | 7,820.4 | 7,186.3 | 3,542.2 | 102.9 | | | |
| | | | PAG1 | kt | **497.7** | | 0.4 | 9.0 | 52.9 | 71.3 | 143.6 | 137.0 | 82.3 | 1.2 | | | |
| | | | PAG2 | kt | **13.8** | | 0.4 | 0.5 | 1.7 | 3.8 | 4.4 | 2.6 | 0.4 | | | | |
| | | | TILL | kt | **1,285.6** | | 996.5 | 253.4 | 35.8 | | | | | | | | |
| | | | SOIL | kt | **81.2** | | 78.1 | 3.1 | | | | | | | | | |
| | | | HISTORIC TAILINGS | kt | **0.0** | | | | | | | | | | | | |
| | | TOTAL MINED | PHASE 2 | kt | **35,148.3** | 0.0 | **2,000.0** | **3,500.0** | **3,500.0** | **4,500.0** | **8,500.0** | **8,500.0** | **4,500.0** | **148.3** | 0.0 | 0.0 | 0.0 |
| WEST PIT | PHASE 1 | | | | | | | | | | | | | | | | |
| | | MILL FEED | TO ROM | kt | **2,119.5** | 0.0 | 343.0 | 420.9 | 509.9 | 845.7 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | | | GRADE | g/t | **2.16** | | 1.94 | 2.28 | 2.23 | 2.14 | | | | | | | |
| | | WASTE ROCK | | kt | **8,682.3** | 0.0 | 3,673.4 | 1,776.9 | 1,726.1 | 1,505.9 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | | | NPAG | kt | **4,315.6** | | 1,352.4 | 1,119.5 | 1,073.2 | 770.5 | | | | | | | |
| | | | PAG1 | kt | **2,685.2** | | 675.9 | 643.4 | 637.9 | 728.1 | | | | | | | |
| | | | PAG2 | kt | **48.6** | | 12.2 | 14.0 | 15.1 | 7.3 | | | | | | | |
| | | | TILL | kt | **1,544.8** | | 1,544.8 | | | | | | | | | | |
| | | | SOIL | kt | **88.1** | | 88.1 | | | | | | | | | | |
| | | | HISTORIC TAILINGS | kt | **0.0** | | | | | | | | | | | | |
| | | TOTAL MINED | PHASE 1 | kt | **10,801.8** | 0.0 | **4,016.4** | **2,197.8** | **2,236.1** | **2,351.6** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 |
| | PHASE 2 | | | | | | | | | | | | | | | | |
| | | MILL FEED | TO ROM | kt | **3,802.6** | 0.0 | 0.0 | 129.1 | 87.1 | 470.1 | 928.4 | 285.9 | 512.6 | 747.9 | 641.6 | 0.0 | 0.0 |
| | | | GRADE | g/t | **2.16** | | | 1.3 | 1.2 | 1.3 | 2.0 | 2.8 | 2.1 | 2.2 | 3.0 | | |
| | | WASTE ROCK | | kt | **33,504.3** | 0.0 | 0.0 | 3,870.9 | 2,162.9 | 8,359.5 | 9,495.6 | 1,807.9 | 2,727.5 | 3,350.2 | 1,729.7 | 0.0 | 0.0 |
| | | | NPAG | kt | **27,444.8** | | | 2,056.9 | 1,934.0 | 7,958.7 | 8,132.2 | 1,396.6 | 2,102.9 | 2,587.0 | 1,276.4 | | |
| | | | PAG1 | kt | **4,022.9** | | | 77.8 | 56.4 | 381.8 | 1,334.4 | 402.1 | 604.0 | 726.8 | 439.6 | | |
| | | | PAG2 | kt | **153.2** | | | 21.7 | 7.8 | 14.9 | 29.0 | 9.3 | 20.6 | 36.4 | 13.7 | | |
| | | | TILL | kt | **1,765.8** | | | 1,607.5 | 154.5 | 3.9 | | | | | | | |
| | | | SOIL | kt | **94.0** | | | 94.0 | | | | | | | | | |
| | | | HISTORIC TAILINGS | kt | **23.5** | | | 12.9 | 10.3 | 0.3 | | | | | | | |

| | | | | TOTALS | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
| | **TOTAL MINED** | **PHASE 2** | **kt** | **37,306.9** | 0.0 | 0.0 | **4,000.0** | **2,250.0** | **8,829.6** | **10,424.0** | **2,093.8** | **3,240.1** | **4,098.0** | **2,371.3** | 0.0 | 0.0 |
| **PHASE 3** | | | | | | | | | | | | | | | | |
| | **MILL FEED** | **TO ROM** | **kt** | **4,408.5** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 72.3 | 667.9 | 958.4 | 1,959.6 | 750.4 |
| | | **GRADE** | **g/t** | **2.07** | | | | | | | | 1.0 | 1.4 | 1.6 | 2.2 | 3.0 |
| | **WASTE ROCK** | | **kt** | **36,690.3** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 4,816.2 | 13,133.6 | 10,499.5 | 6,879.9 | 1,361.1 |
| | | **NPAG** | **kt** | **31,755.6** | | | | | | | | 3,092.9 | 12,709.2 | 9,273.6 | 5,557.1 | 1,123.0 |
| | | **PAG1** | **kt** | **3,240.3** | | | | | | | | 44.0 | 424.4 | 1,217.5 | 1,316.3 | 238.0 |
| | | **PAG2** | **kt** | **15.0** | | | | | | | | | 0.0 | 8.3 | 6.6 | 0.1 |
| | | **TILL** | **kt** | **1,455.9** | | | | | | | | 1,455.9 | | | | |
| | | **SOIL** | **kt** | **122.3** | | | | | | | | 122.3 | | | | |
| | | **HISTORIC TAILINGS** | **kt** | **101.2** | | | | | | | | 101.2 | | | | |
| | **TOTAL MINED** | **PHASE 3** | **kt** | **41,098.9** | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | **4,888.5** | **13,801.6** | **11,457.8** | **8,839.5** | **2,111.5** |
| **MINED MATERIAL** | | | | **142,549.7** | **4,055.6** | **11,515.3** | **14,569.6** | **11,753.6** | **15,681.2** | **18,924.0** | **10,593.8** | **12,628.6** | **18,047.9** | **13,829.1** | **8,839.5** | **2,111.5** |
| **OTHER MATERIAL MOVEMENTS** | | | | | | | | | | | | | | | | |
| | Stockpile Rehandle | | kt | **1,000.3** | | 220.8 | | | | | | | | | 140.0 | 639.5 |
| | | | kt | **0.0** | | | | | | | | | | | | |
| | | TOTAL | kt | **1,000.3** | **0.0** | **220.8** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **0.0** | **140.0** | **639.5** |
| STRIP RATIO | | | | 8.0 | 17.4 | 10.0 | 9.0 | 7.1 | 9.7 | 12.0 | 6.3 | 7.6 | 11.4 | 7.6 | 3.5 | 1.8 |
| MINED TONNES PER DAY | | | | | 11,111 | 31,549 | 39,917 | 32,202 | 42,962 | 51,847 | 29,024 | 34,599 | 49,446 | 37,888 | 24,218 | 11,256 |
| **MILLED MATERIAL** | | | | | | | | | | | | | | | | |
| **MILL FEED** | | | | | | | | | | | | | | | | |
| Milled Tonnage | **TOTAL** | THROUGH ROM | **kt** | **15,798.9** | 0.0 | **1,269.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,460.0** | **1,389.9** |
| | | GRADE | **g/t** | **2.26** | | **2.29** | **2.33** | **2.54** | **1.82** | **1.87** | **2.57** | **2.58** | **1.95** | **2.26** | **2.14** | **2.59** |
| | | FROM PIT | **kt** | **14,798.6** | | 1,048.2 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,320.0 | 750.4 |
| | | FROM STPL | **kt** | **1,000.3** | | 220.8 | | | | | | | | | 140.0 | 639.5 |
| | | Contained Oz | **oz** | **1,150,205** | 16,494 | 76,789 | 109,403 | 119,140 | 85,494 | 87,543 | 120,616 | 120,933 | 91,300 | 113,677 | 135,962 | 72,854 |
| | | Recovered Oz | oz | **1,102,313** | | 89,417 | 104,929 | 114,553 | 81,343 | 83,362 | 116,013 | 116,327 | 87,064 | 101,822 | 96,174 | 111,309 |

Open Pit progression plan figures for a select years are illustrated in Figures 16-1 through 16-8. The figures illustrate the conceptual progression of the East and West pits and the waste rock storage areas.

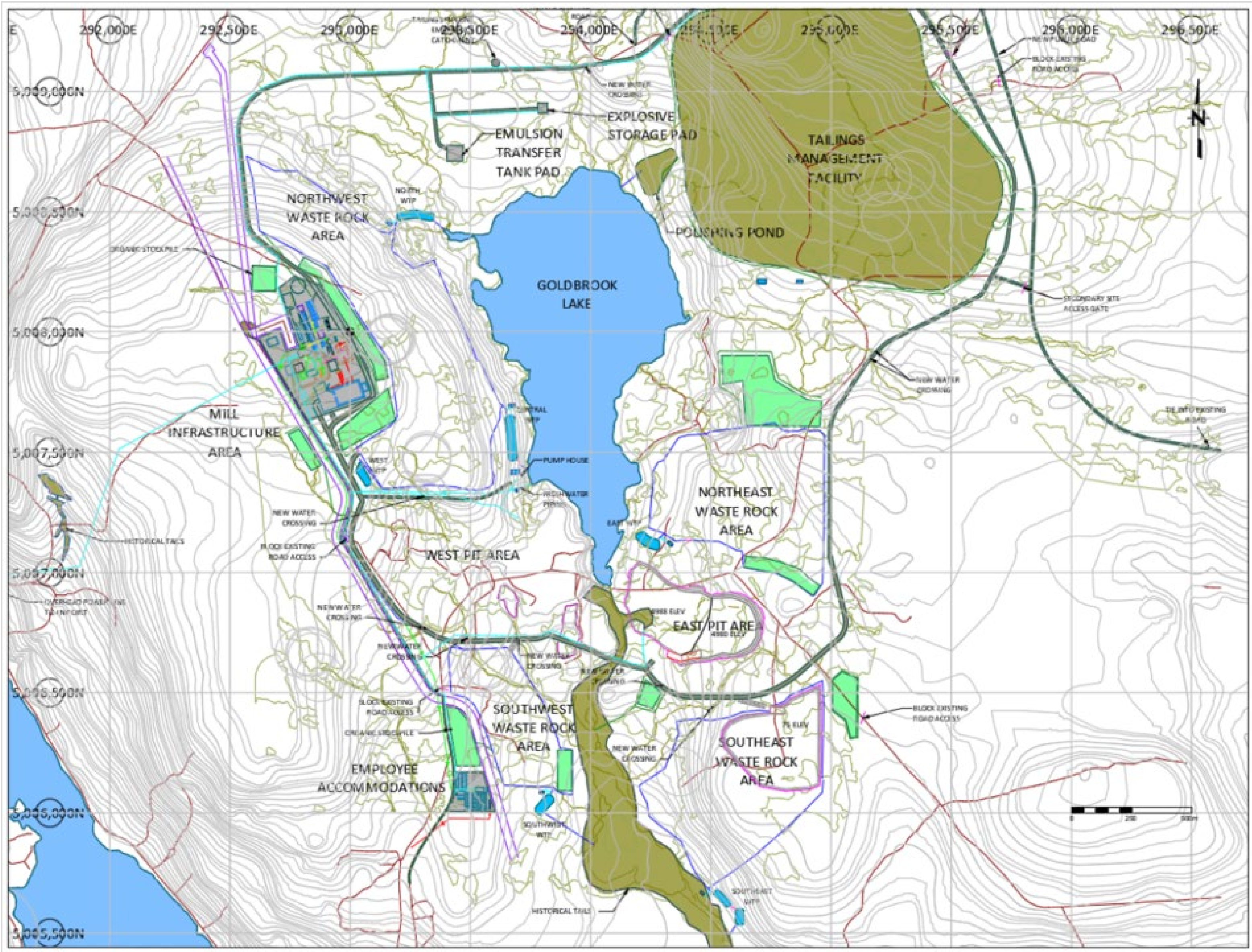



*Figure 16-1: Pit Progression Plan, Year -1*

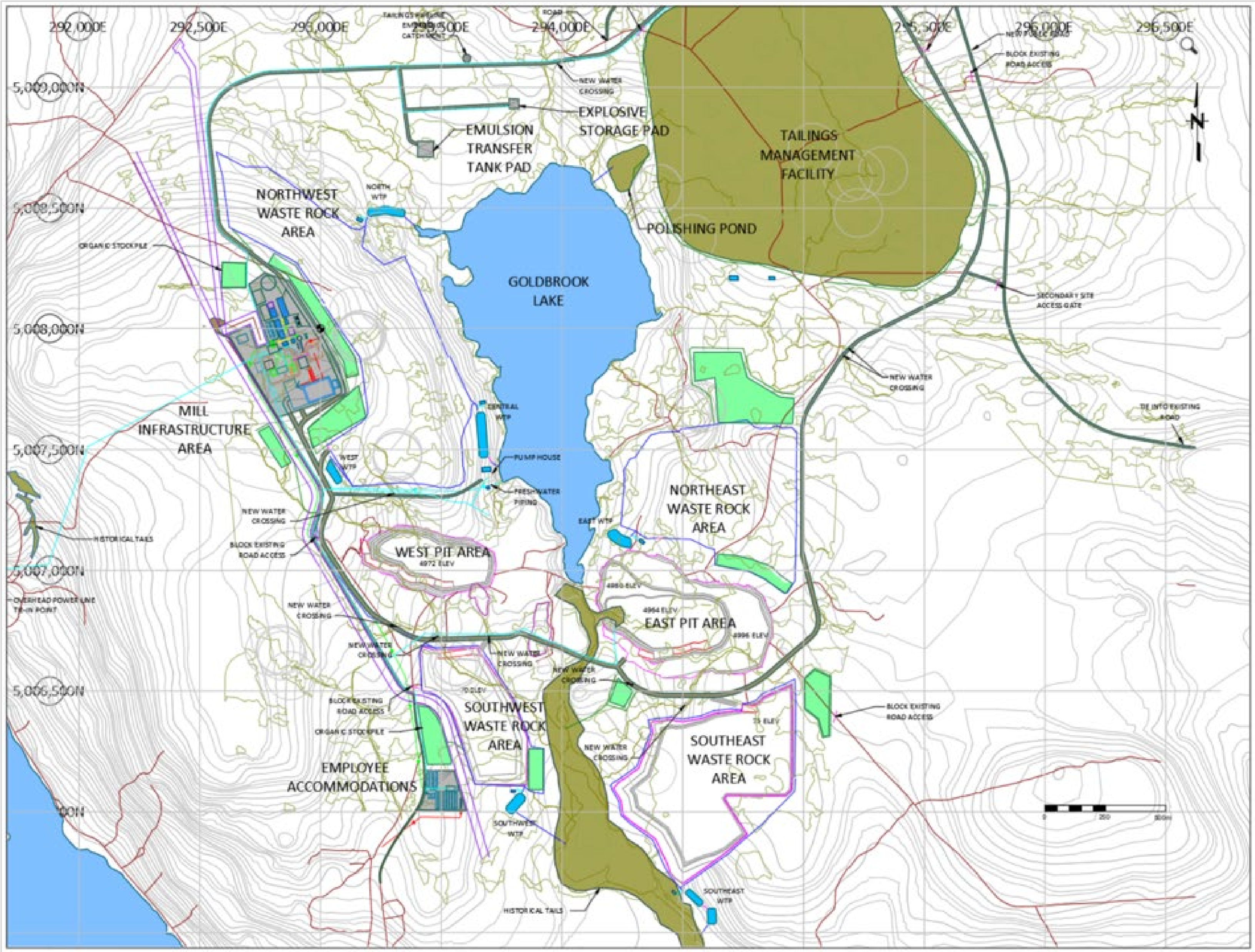



*Figure 16-2: Pit Progression Plan, Year 1*

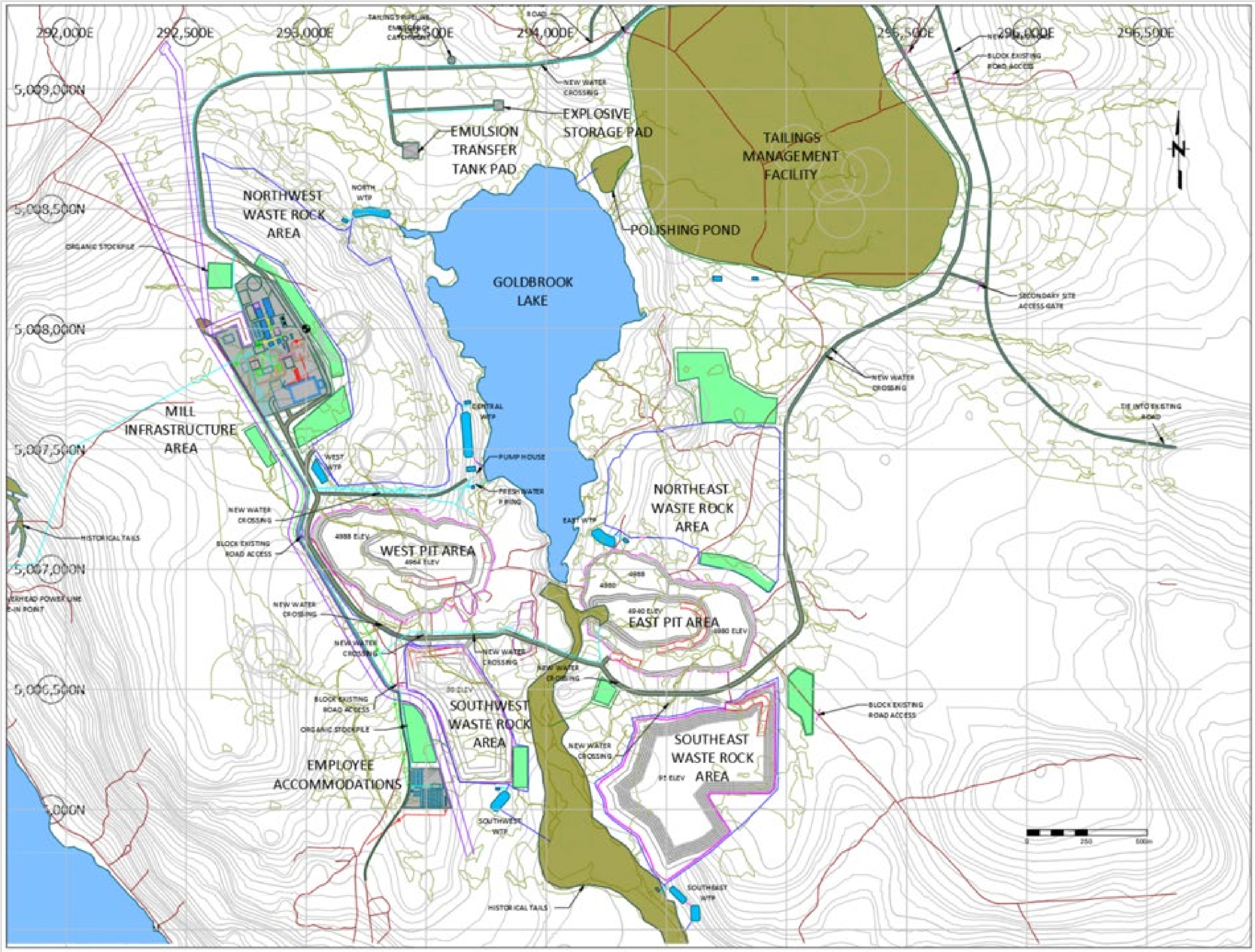



*Figure 16-3: Pit Progression Plan, Year 2*

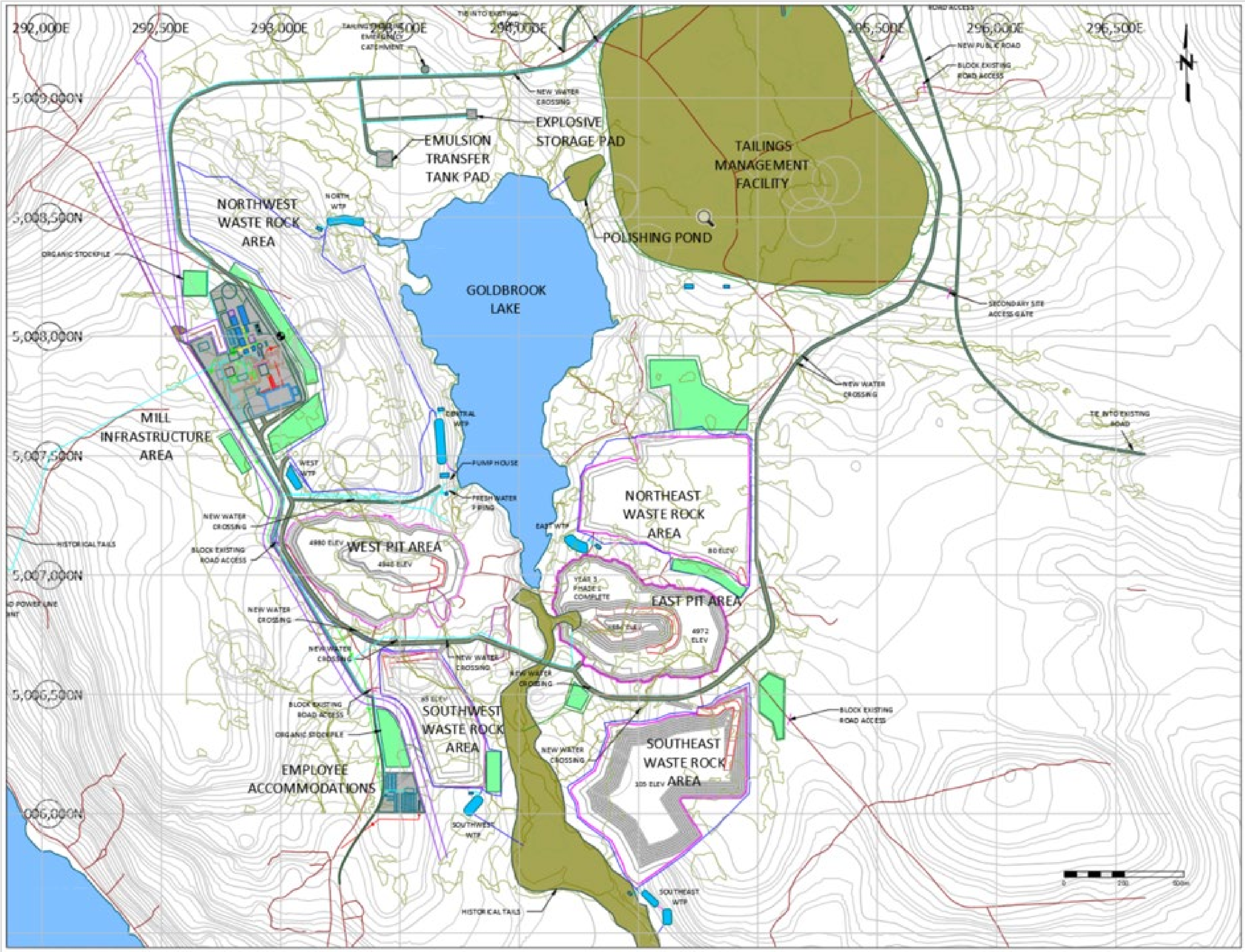



*Figure 16-4: Pit Progression Plan, Year 3*

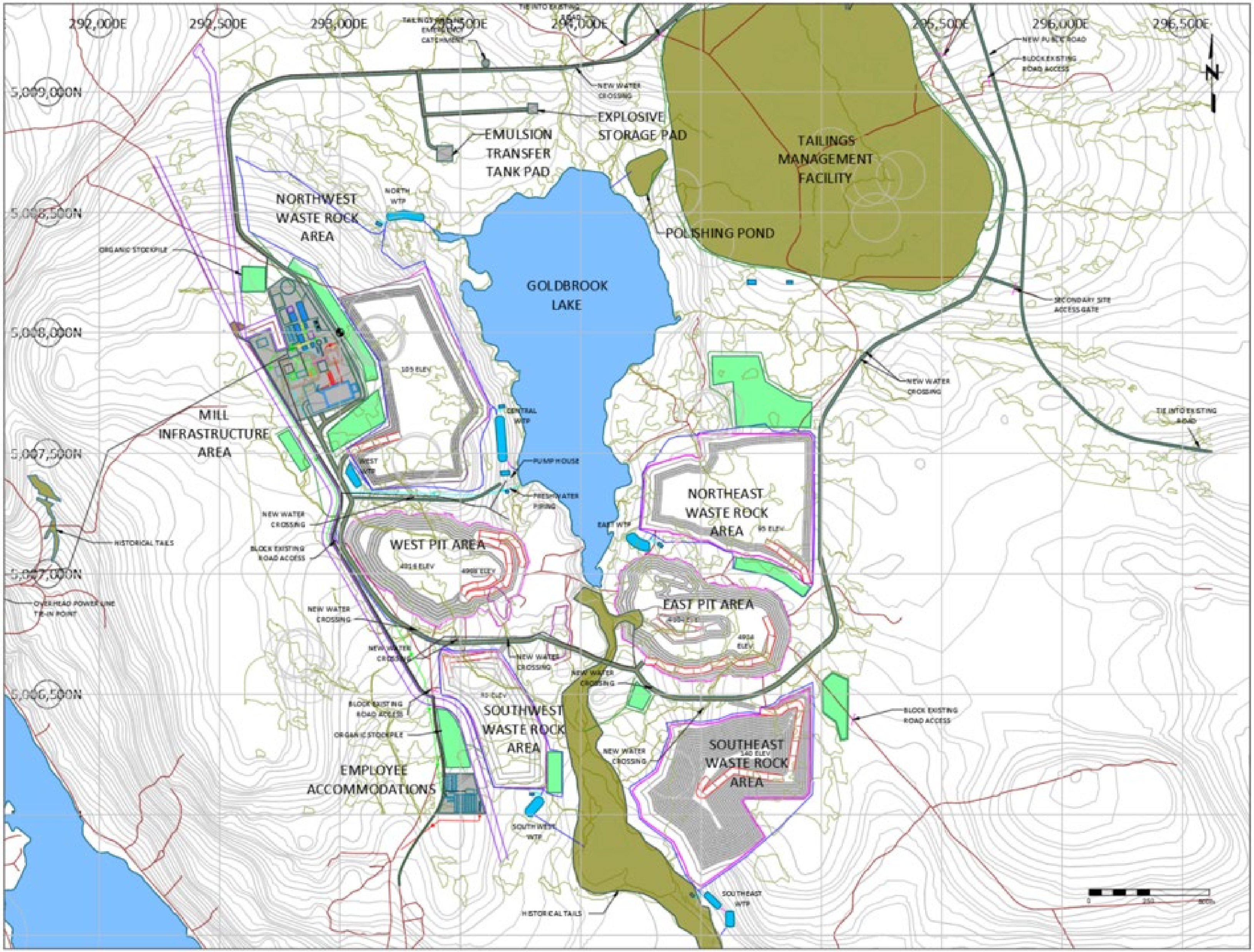



*Figure 16-5: Pit Progression Plan, Year 5*

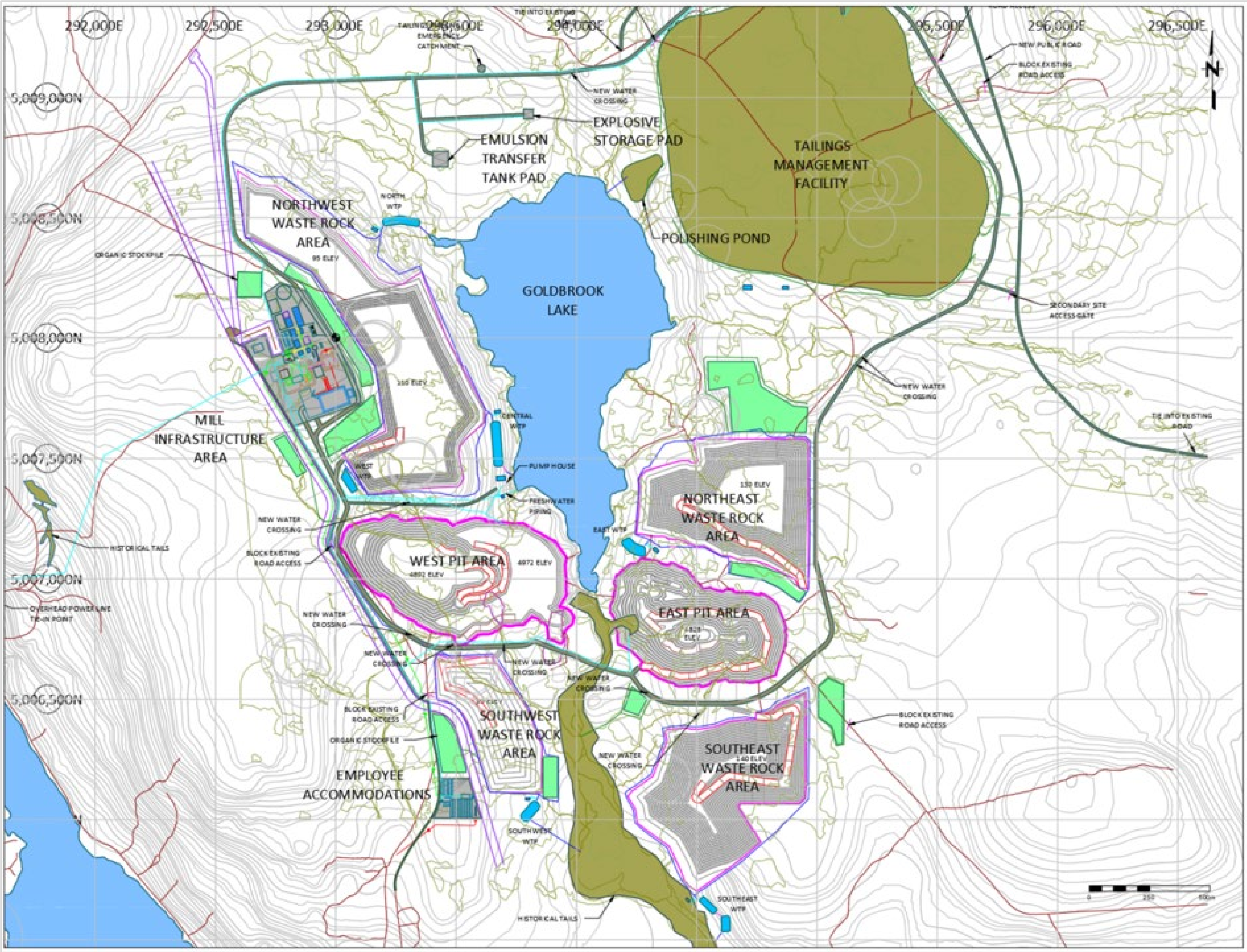



*Figure 16-6: Pit Progression Plan, Year 7*

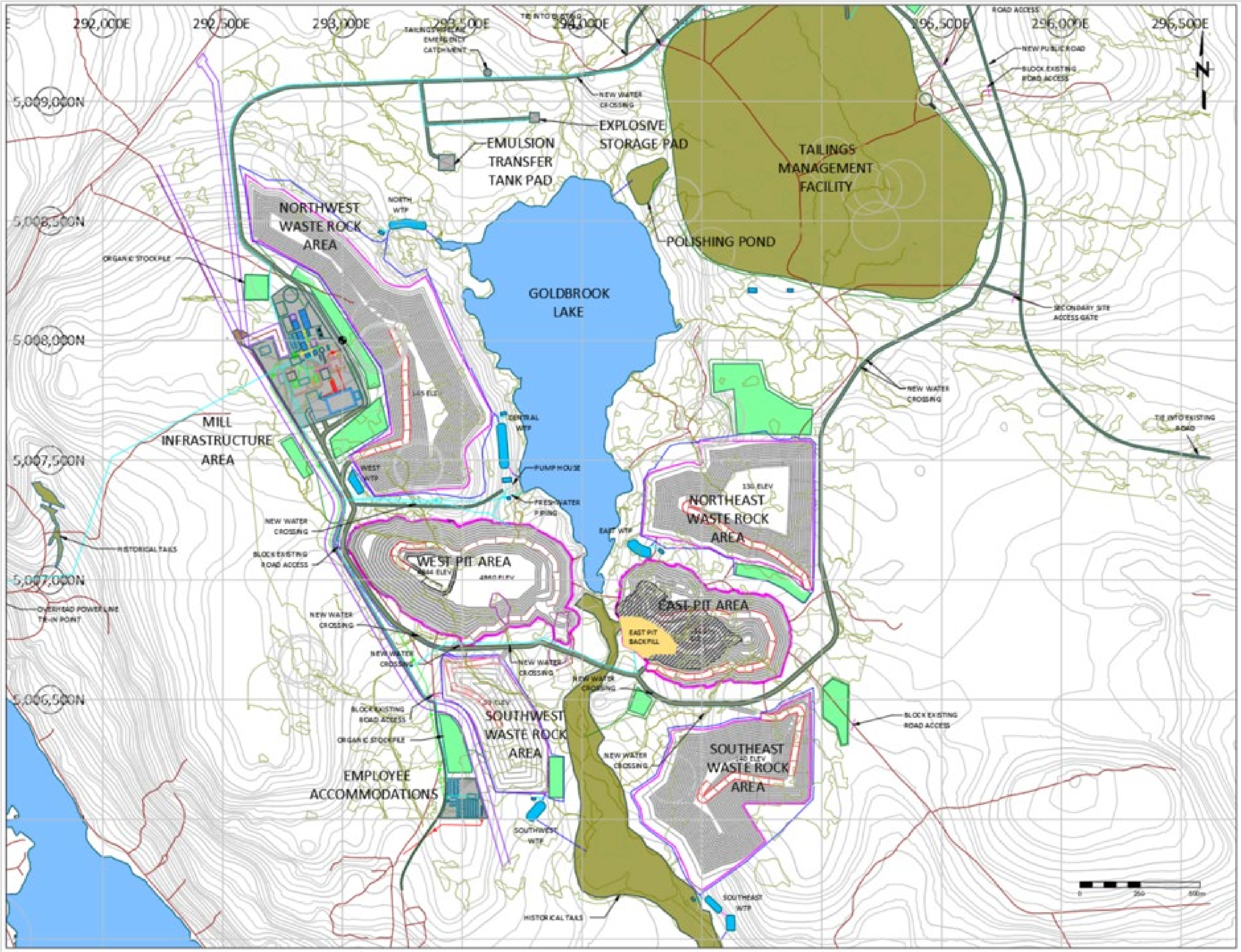



*Figure 16-7: Pit Progression Plan, Year 9*

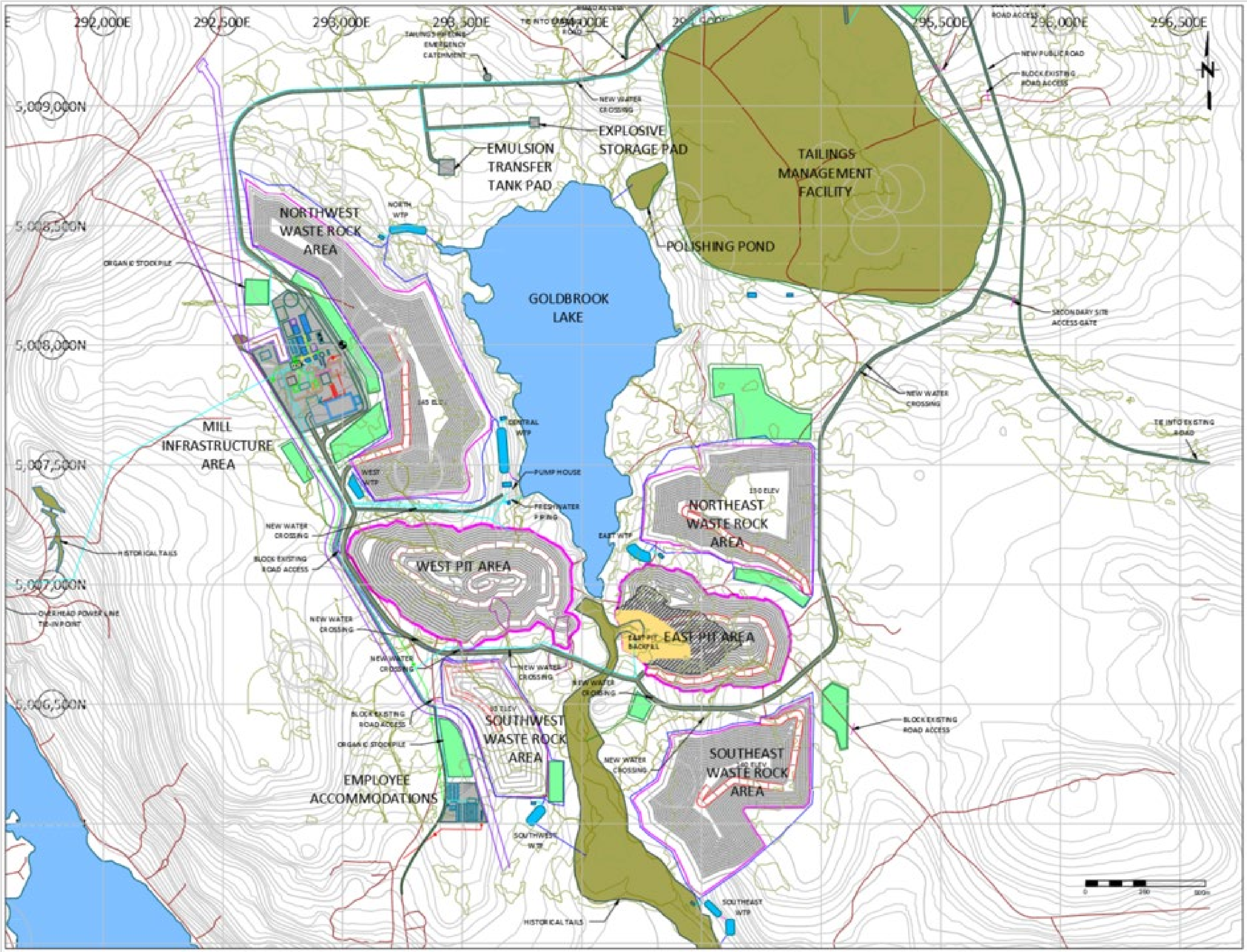



*Figure 16-8: Pit Progression Plan, Year 11*

### 16.3.3 Open Pit Mining Operation

This section includes indicative parameters for drilling, blasting, loading, and hauling. A mining contractor would be responsible, with the guidance of the Company technical staff, to provide suitable equipment and operational procedures to meet the target production.

The objective of equipment selection for this level of study is to produce an estimate of costs suitable for a FS level study and not necessarily to design an optimized equipment fleet. The mining equipment will depend on the available equipment of the selected mining contractor. The equipment listed in Table 16-5 was selected to match the mine production schedule.

**Table 16-5: Major Mining Equipment, Open Pit, Peak Requirements**

| Fleet | Size / Capacity | Peak # Units |
|---|---|---|
| Haul Truck | 90t | 14 |
| Haul Truck | 40t | 5 |
| Excavator | 4.5 $m^3$ | 1 |
| Excavator | 8 $m^3$ | 3 |
| Primary Drill | 6.5 inch / 165 mm | 4 |
| Track Dozer | | 4 |
| Wheel Dozer | | 1 |
| Grader | | 2 |

An annual estimate of the major mine equipment fleet typical for this size of operation is listed in Table 16-6.

**Table 16-6: Major Mobile Mine Equipment, Number of Estimated Units by Year**

| ACTIVITY | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Drill | 1 | 2 | 2 | 2 | 3 | 4 | 3 | 3 | 4 | 3 | 2 | 2 |
| Waste Excavator, 8 $m^3$ | 1 | 3 | 3 | 2 | 3 | 3 | 2 | 2 | 3 | 2 | 1 | 1 |
| Mineral Excavator, 4.5 $m^3$ | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Hauling Units | 3 | 10 | 10 | 10 | 13 | 16 | 11 | 14 | 15 | 13 | 11 | 11 |
| Tracked Dozer | 3 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 2 | 2 | 2 |
| Wheel Dozer | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 | 1 |
| Grader | 1 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |

#### 16.3.3.1 Drilling and Blasting

Drilling is expected to be performed on eight-metre benches. For ore-control may be excavated on 4 m intervals.

Table 16-7 presents the assumptions used in estimating the annual equipment production hours.

**Table 16-7: Estimation of Annual Production Hours**

| | Units | Drill |
|---|---|---|
| Calendar Days | | 365 |
| Calendar Hours | | 24 |
| Total Time (TH) | | 8,760 |
| Mechanical Availability | m | 85% |
| Available Time | Hours | 7,446 |
| Operator Standby (i.e., weather delays) | Hours | 360 |
| Utilized Time | Hours | 7,086 |
| Operating Delay Hours | hours | 1,521 |
| Production Hours | Hours | 5,565 |
| Efficiency Factor | % | 82 |
| Valuable Production Time | Hours | 4,563 |

Table 16-8 shows the blasthole productivity for an 8 m drill bench.

**Table 16-8: Blasthole Drill Productivity**

| Blasthole Drill Productivity | Units | Waste | Mineral |
|---|---|---|---|
| Hole Diameter | mm | 165 | 165 |
| Bench Height | m | 8 | 8 |
| Subgrade | m | 1.53 | 1.47 |
| Bank Density | $t/m^3$* | 2.72 | 2.73 |
| Powder Factor | kg/t | 0.22 | 0.24 |
| Rock Mass per Hole | t | 656 | 601 |
| Burden | m | 5.1 | 4.9 |
| Spacing | m | 5.9 | 5.6 |
| Drilling Rate (overall) | m/h** | 27 | 25 |

* $t/m^3$ = tonnes per cubic metre **m/h = metres per hour

### 16.3.3.2 Blasting

It is recommended that the blasting services be provided by the mining contractor, or a specialist blasting services provider. Typical blasting parameters are set out in Table 16-9.

**Table 16-9: Blasting Parameters for Production Blast Holes**

| Description | Unit | Waste | Mineral |
|---|---|---|---|
| Explosive density | $g/cm^3$ | 1.20 | 1.20 |
| Powder factor | kg /t | 0.22 | 0.24 |
| Explosive length | m | 5.7 | 5.6 |
| Explosive per hole | kg | 147 | 142.9 |
| Stemming height | m | 3.8 | 3.9 |

For the purposes of estimating the requirements it has been assumed that overburden material will be free digging and that the explosive product is emulsion.

During operations, safe working procedures for drilling and blasting when approaching historical workings will require development.

#### 16.3.3.3 Loading and Hauling

Based on mining contractor budget quotes, a front end loader type of excavator (~7 $m^3$ bucket capacity) and a hydraulic excavator (~8 $m^3$ bucket capacity) were proposed for waste handling. A backhoe type excavator (~4-4.5 $m^3$ bucket capacity) was proposed for ore handling. The haulage units proposed ranged from a single fleet of ~63t haul truck to a combination fleet of ~90t rigid frame truck with ~40t articulated haul truck.

An 8.0 $m^3$ bucket capacity and 4.5 $m^3$ bucket capacity were assumed when estimating the loading fleet requirements. The 90t haul truck with 40t articulated haul truck were assumed when estimating the hauling fleet.

Table 16-10 presents the assumptions used in estimating the annual equipment production hours.

**Table 16-10: Estimation of Annual Production Hours**

| | Units | LOAD | HAUL |
|---|---|---|---|
| Calendar Days | Days | 365 | 365 |
| Calendar Hours | Hours | 24 | 24 |
| Non-scheduled Days per year | # | 10 | 10 |
| Scheduled Shifts per day | # | 2 | 2 |
| Scheduled Hours per shift | Hours | 12 | 12 |
| Scheduled Hours per year | Hours | 8,520 | 8,520 |
| Total Delays per shift | Hours | 2 | 2 |
| Operating Hours per shift | Hours | 10 | 10 |
| Shift Utilization | % | 83.3 | 83.3 |
| Effective Utilization | % | 75.0 | 75.0 |
| Unit Availability | % | 80-85 | 85 |
| Operating Hours per year | Hours | 5,680 – 6035 | 6033 |

Loading fleet numbers have been estimated on first principles based on the operating hours required to achieve the production schedule, calculated by cycle times, and estimates of the equipment's rated capacities and productivities. The loading unit productivity assumptions are listed in Table 16-17.

**Table 16-11: Loading Unit Productivity Assumptions**

| Description | Unit | Waste | Mineral |
|---|---|---|---|
| Bucket capacity | $m^3$ | 8.0 | 4.5 |
| Bucket fill factor | % | 90 | 90 |
| Dry density | t/bcm | 2.72 | 2.73 |
| Moisture | % | 5 | 5 |
| Swell factor | % | 40 | 40 |
| Tonnes/Pass | | 14.0 | 7.9 |
| Haul Truck | | 90t | 40t |
| Passes | | 6.0 | 5.0 |
| Total Load Time | min | 4.3 | 3.6 |
| Productivity | Tph (operating hour) | 939 | 523 |
| | tph (shift hour) | 782 | 436 |

Haul truck fleet numbers have been estimated on first principles based on the operating hours required to achieve the production schedule, calculated cycle times, and estimates of the equipment's rated capacities and productivities. A limited number of typical haul profiles specific to the detailed pit design were estimated.

Table 16-18 shows the average annual haul cycle travel times estimated for the Project.

**Table 16-12: Average Annual Haul Cycle Travel Times**

| AREA | UNITS | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| EAST PIT TO | | | | | | | | | | | | | |
| ROM | min | 12.6 | 13.5 | 14.4 | 15.9 | 19.3 | 15.5 | 17.4 | 20.5 | 25.4 | 28.0 | | |
| SE Waste rock storage areas (WRSA) | min | 6.9 | 7.7 | 8.4 | 10.4 | 13.3 | | | | | | | |
| NE WRSA | min | | | | | 14.1 | 12.5 | 14.7 | 19.0 | 26.0 | 28.6 | | |
| SW WRSA | min | | 6.9 | 6.3 | 8.1 | 9.1 | | | | | | | |
| TMF | min | 22.0 | 22.9 | 23.5 | 24.4 | 26.4 | 24.9 | 26.8 | 29.9 | 34.8 | 37.4 | | |
| OTHER | min | 7.09 | | | | | | | | | | | |
| | | | | | | | | | | | | | |
| WEST PIT TO | | | | | | | | | | | | | |
| ROM | min | | 6.7 | 8.4 | 8.5 | 10.3 | 15.2 | 16.6 | 18.8 | 17.7 | 17.1 | 20.1 | 22.7 |
| SE WRSA | min | | 13.3 | 15.0 | 14.9 | 18.0 | 21.7 | | | | | | |
| NW WRSA | min | | | | | | 12.4 | 18.3 | 20.6 | 18.1 | 16.3 | | |
| SW WRSA | min | | 8.0 | 9.7 | 8.2 | 11.2 | 13.5 | | | 9.1 | | | |
| TMF | min | | 25.8 | 27.5 | 28.0 | 29.4 | 31.5 | 32.9 | 35.2 | 28.0 | 33.5 | 35.9 | 39.4 |
| EAST PIT BACKFILL | min | | | | | | | | | | 18.18 | 15.58 | 19.80 |

Additional time for loading (varied by excavator and haul truck combination) and dumping (1.25 min) was considered.

### 16.3.3.4 Ancillary Service and Support Equipment

The primary pit operations will be supported by additional equipment including track dozers with ripper attachments, road graders, water truck, utility loaders and excavators, and maintenance service vehicles, dependent on the selected mining contractor. A typical list of expected ancillary equipment is provided in (**Table 16-13**).

It is envisaged that the rock will be loaded directly into the processing plant crusher hopper but there will be a need for a ROM stockpile to allow for stoppages, for stockpiling in the pre-production period, and possibly some blending.

**Table 16-13: Estimated List of Ancillary Mine Equipment**

| ANCILIARY EQUIPMENT | NO. OF UNITS |
|---|---|
| Blast Crew Truck | 1 |
| Blaster's Truck | 1 |
| Skid Steer | 1 |
| Air Trac – Secondary Drill | 1 |
| Loader – Medium | 1 |
| Water Truck | 1 |
| Snow Plow | 1 |
| Utility Excavator | 1 |

| ANCILIARY EQUIPMENT | NO. OF UNITS |
|---|---|
| Utility Loader | 1 |
| Maintenance Field and Service Trucks | 4 |
| Fuel/Lube Truck | 1 |
| Float Truck | 1 |
| Crane | 1 |
| Tire Handler | 1 |
| Forklift | 2 |
| Light Vehicles | 7 |
| Crew Buses | 2 |
| Ambulance, Fire Truck | 2 |
| Light Vehicles – Owner Team | 10 |
| Portable Light Towers | 5 - 10 |
| Dewatering Pumps | 3 |

#### 16.3.3.5 Pit Dewatering

The progressive development of the open pit will result in increasing water infiltration from precipitation and groundwater inflows. As the pit deepens and increases in footprint, it will be necessary to control water inflow through the construction if in-pit dewatering systems such as dewatering wells, drainage ditches, sumps, pipelines, and pumps.

An allowance has been included in the open pit capital and operating costs for in-pit dewatering through in-pit sumps.

The range of monthly average flow rate was estimated at 23.9-81.6 L/s for the West pit and 18.1-64.4 L/s for the East Pit.

Booster pumps ae expected to be required when the pits pass the 160 m.

#### 16.3.3.6 Labour Requirements

The open pit personnel estimates will be dependent on the selected mining contractor and the type of equipment fleet they have available. Table 16-14 lists the potential labour requirements for the open pit operations.

The estimate is based on the estimated equipment fleet required to achieve the production schedule.

A combination of rotation schedules is envisioned. The majority of operations and maintenance crews were assumed to be 12 hours per day, 7 days per week.

**Table 16-14: Open Pit Mining Personnel Estimate**

| ACTIVITY | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Drill and Blast Crew | | 12 | 12 | 12 | 16 | 20 | 16 | 16 | 20 | 16 | 12 | 12 |
| Loading and Hauling Operators | 16 | 36 | 48 | 48 | 63 | 79 | 52 | 67 | 65 | 63 | 52 | 52 |
| Support Operators | 23 | 29 | 29 | 29 | 29 | 32 | 29 | 29 | 29 | 23 | 23 | 23 |
| Operations Supervision | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 | 4 |

| ACTIVITY | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Mine Maintenance | 11 | 17 | 19 | 19 | 23 | 27 | 21 | 23 | 25 | 21 | 19 | 19 |
| Mine Management | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Mine Technical Team | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 | 2 |
| Owner's Technical Team | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 | 9 |
| Health and Safety | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| **Total Estimated Open Pit Mining** | **78** | **114** | **128** | **128** | **151** | **178** | **138** | **155** | **169** | **143** | **126** | **126** |

## 16.4 Geotechnical Evaluation

Optimize was initially commissioned to develop a geotechnical gap analysis for the Project to verify that the existing geotechnical information used to support the PEA Technical Report (August 5, 2021) could be used as a base for the FS. However, the gap analysis confirmed that further geotechnical drilling would be required to complete an FS level of study and the Company completed 18 additional geotechnical boreholes to support the FS. The geotechnical team evaluated the following information:

- Drill holes
- Geotechnical logging
- Geotechnical parameters and laboratory tests
- 3D geotechnical model
- Pit sectorization
- Kinematic analysis
- Stability analysis

The following sections are excerpts from the summary report (Optimize, 2021).

### 16.4.1 3D Geotechnical Model

The geomechanical model named "Classe_Model_Nord" was divided into four rock mass classes according to RMR89 (varying from Class II to Class V). The model included the three geotechnical drill holes drilled in 2019 along with the 18 new geological-geotechnical boreholes drilled in 2021. The 21 holes were combined to verify the coherence between geomechanical classes and differentiate the geological rock types (argillite and greywacke) for mine waste and mineralized zones.

Collectively, the geotechnical information within these 21 holes was used to support the orientation, characteristics and geometry of the various geological structures within the Deposit.

The geotechnical holes used to support the FS are in Table 16-15 and Figure 16-9.

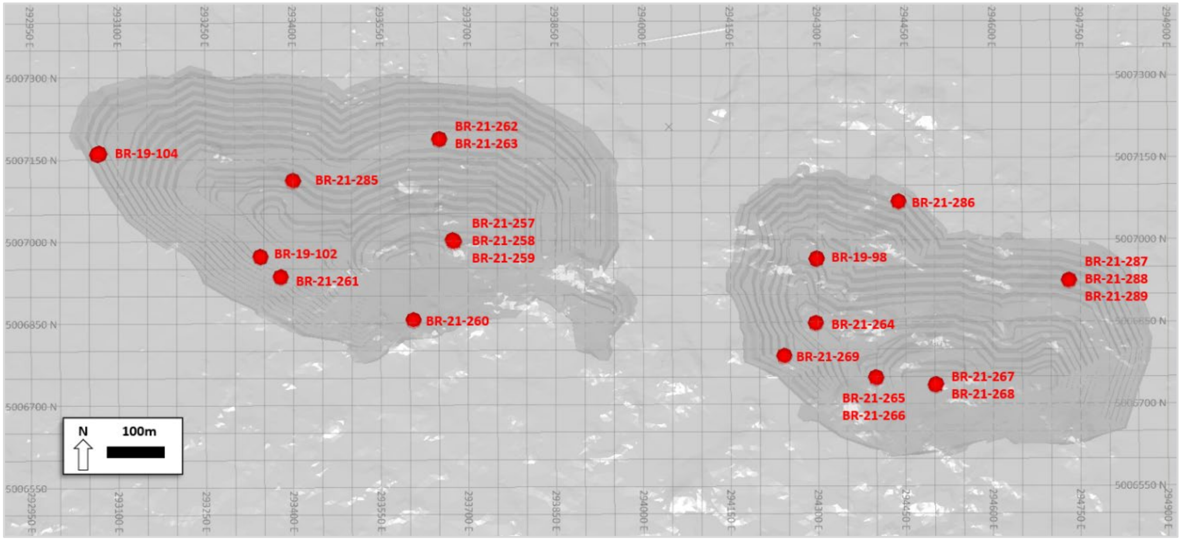



*Figure 16-9: Location of boreholes used to update the geomechanical model.*

**Table 16-15: Information from all Boreholes Used to Update the Model and Presented in 3D Model**

| Hole ID | East (E) | North (N) | Elevation (Z) | Depth | Televiewer | Geotech | Geology |
|---|---|---|---|---|---|---|---|
| BR-21-257 | 293676.0 | 5006946.3 | 5001.0 | 131.0 | - | ✓ | ✓ |
| BR-21-258 | 293675.7 | 5006946.7 | 5000.7 | 158.0 | - | ✓ | ✓ |
| BR-21-259 | 293673.7 | 5006949.2 | 5000.3 | 240.5 | 2021 | ✓ | ✓ |
| BR-21-260 | 293606.0 | 5006804.1 | 4998.2 | 401.0 | 2021 | ✓ | ✓ |
| BR-21-261 | 293378.3 | 5006880.8 | 5005.1 | 86.0 | - | ✓ | ✓ |
| BR-21-262 | 293651.0 | 5007133.2 | 5001.8 | 185.0 | 2021 | ✓ | ✓ |
| BR-21-263 | 293651.2 | 5007133.4 | 5001.8 | 122.0 | 2021 | ✓ | ✓ |
| BR-21-264 | 294296.2 | 5006799.0 | 4992.0 | 194.0 | - | ✓ | ✓ |
| BR-21-265 | 294399.0 | 5006697.2 | 4996.8 | 110.0 | - | ✓ | ✓ |
| BR-21-266 | 294399.3 | 5006696.5 | 4996.8 | 146.0 | 2021 | ✓ | ✓ |
| BR-21-267 | 294502.0 | 5006683.8 | 4999.5 | 293.0 | 2021 | ✓ | ✓ |
| BR-21-268 | 294501.5 | 5006682.2 | 4999.3 | 113.0 | 2021 | ✓ | ✓ |
| BR-21-269 | 294241.8 | 5006739.1 | 4991.1 | 302.0 | 2021 | ✓ | ✓ |
| BR-21-285 | 293410.0 | 5007054.0 | 5006.0 | 218.0 | - | ✓ | ✓ |
| BR-21-286 | 294438.0 | 5007021.0 | 5004.0 | 269.0 | - | ✓ | ✓ |
| BR-21-287 | 294730.0 | 5006869.0 | 5011.0 | 203.0 | - | ✓ | ✓ |
| BR-21-288 | 294730.0 | 5006869.0 | 5011.0 | 164.0 | - | ✓ | ✓ |
| BR-21-289 | 294730.0 | 5006869.0 | 5011.0 | 110.0 | - | ✓ | ✓ |
| BR-19-98 | 294298.0 | 5006914.5 | 4997.5 | 101.0 | 2019 | - | - |
| BR-19-102 | 293343.7 | 5006917.0 | 5007.7 | 143.0 | 2019 | - | - |
| BR-19-104 | 293065.0 | 5007100.0 | 5022.6 | 85.0 | 2019 | - | - |

An updated geotechnical model was constructed using the previous and new geomechanical and geological information. Additionally, the various edits to the mineralized zones since the PEA Technical Report (August 5, 2021) were incorporated. The geotechnical model was subdivided, and various geotechnical slope stability analyses were performed (Figure 16-10).

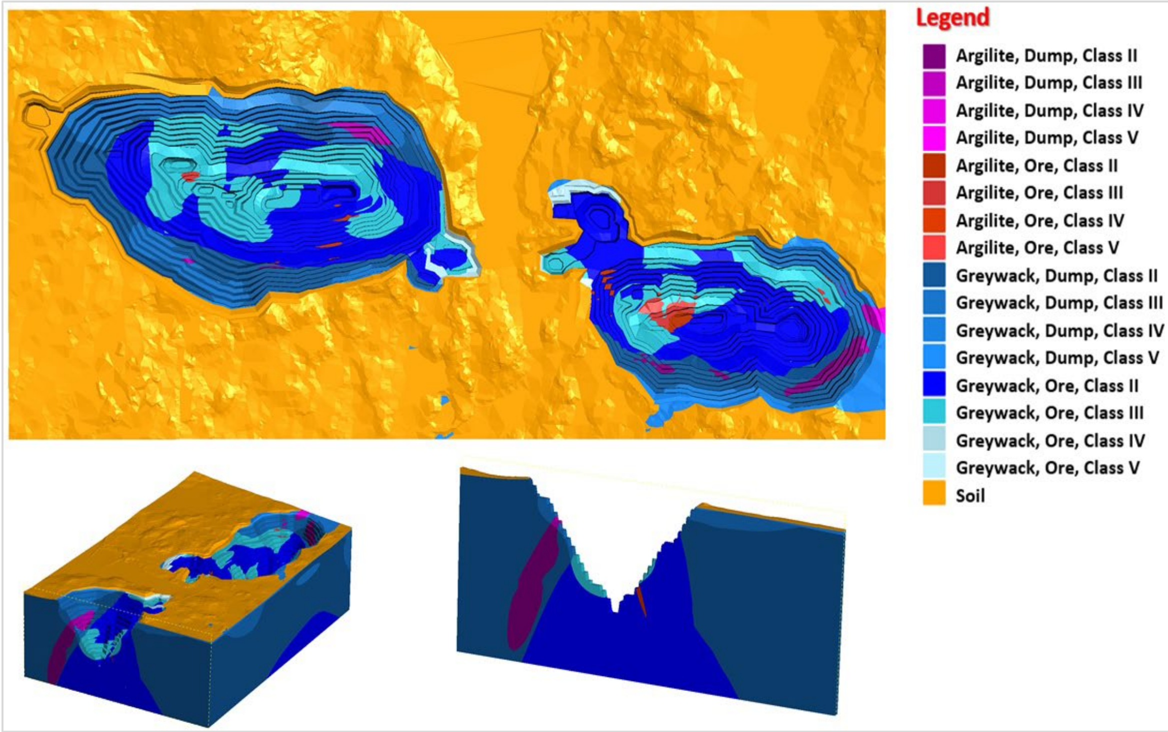



*Figure 16-10: 3D Geotechnical model and cross section from the 3D geotechnical model.*

### 16.4.2 Laboratory Tests

The current geotechnical database consists of 44 laboratory tests. Twenty-three of these tests were for uniaxial compressive strength (UCS), completed in 2017 and 2019. In 2021, a total of 21 UCS tests (performed per ASTM D4345-19) were conducted to determine the strength and elastic properties (Young's modulus and Poisson's ratio – ASTM D2845-08 and ASTM D7012-14e1). The statistical results analyses were defined for each type of rock and are presented in Table 16-16 and Table 16-17.

**Table 16-16: Statistics Results for Argilite's Geotechnical Parameters**

| | Argilite Direction parallel | | | | Argilite Direction oblique | | | |
|---|---|---|---|---|---|---|---|---|
| | Density ($kN/m^3$) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio | Density ($kN/m^3$) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio |
| Average | 2.76 | 64.14 | 67.20 | 0.29 | 2.73 | 76.67 | - | - |
| Minimum | 2.63 | 32.00 | 51.36 | 0.20 | 2.67 | 30.00 | - | - |
| Maximum | 2.83 | 119.07 | 76.70 | 0.36 | 2.78 | 142.00 | - | - |
| Standard Deviation | 0.04 | 16.32 | 7.10 | 0.05 | 0.04 | 27.22 | - | - |
| Coefficient Variation | 2% | 25% | 11% | 16% | 1% | 36% | - | - |

| | Argilite Direction parallel | | | | Argilite Direction oblique | | | |
|---|---|---|---|---|---|---|---|---|
| | Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio | Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio |
| # Validation Samples | 17 | 17 | 7 | 7 | 6 | 6 | - | - |

**Table 16-17: Statistics Results for Graywacke's Geotechnical Parameters**

| | Graywacke Direction parallel | | | | Graywacke Direction oblique | | | |
|---|---|---|---|---|---|---|---|---|
| | Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio | Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio |
| Average | 2.70 | 72.99 | 73.03 | 0.20 | 2.71 | 103.32 | 80.48 | 0.20 |
| Minimum | 2.58 | 35.00 | 60.98 | 0.28 | 2.68 | 49.00 | 74.67 | 0.17 |
| Maximum | 2.77 | 113.00 | 73.95 | 0.30 | 2.73 | 137.84 | 83.99 | 0.25 |
| Standard deviation | 0.04 | 17.87 | 6.49 | 0.01 | 0.01 | 29.81 | 3.87 | 0.03 |
| Coefficient Variation | 1% | 24% | 9% | 4% | 0.4% | 28.9% | 4.8% | 14.7% |
| # Validation Samples | 22 | 22 | 7 | 6 | 7 | 7 | 3 | 3 |

Based on the laboratory tests results, the argillite and graywacke intact rock have a low anisotropic strength behaviour. Both rock masses are highly fractured and have a significant dispersion within the orientations of the various structures. The current laboratory tests do not allow for a reliable definition of the anisotropy degree; as such, the more conservative items such as the parallel strength and deformability parameters were also utilized for the FS. Table 16-18 presents the geotechnical parameters obtained from the laboratory tests interpretation used in the analysis.

**Table 16-18: Geotechnical Parameters Used in the Analysis**

| Argilite | | | | Graywacke | | | |
|---|---|---|---|---|---|---|---|
| Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio | Density (kN/m³) | UCS (MPa) | Young's Modulus (GPa) | Poisson's Ratio |
| 2.7 | 60 | 67.20 | 0.29 | 2.7 | 70 | 73.0 | 0.20 |

### 16.4.3 Open Pit Slope Design

#### 16.4.3.1 Bench Height

The bench height is a direct function of the excavation equipment type and size planned for the open pit mine. The open pit design considered a 5m operational bench height in initial studies, with a triple bench (15 m) for the final pit wall and long period pushbacks. Therefore, the geotechnical study considered this geometrical constraint as the bench height.

#### 16.4.3.2 Berm Width

The berm width must be wide enough to contain falling blocks, narrow enough to not significantly affect the overall slope angle and not increase the strip ratio substantially or disrupt operations. Therefore, the dimension of this parameter followed the criterion that relates the bench height with the berm width, as shown in Equation 1.

*Equation 1. Berm width ratio for falling blocks. Source: Read and Stacey (2009)*

$$bench\ width\ (m) = 0.2 * bench\ height\ (m) + 4.5m = 0.2 * 15 + 4.5 = 7.5m$$

The berm width suggested for this study is 7.5 m wide, considering the final bench height of 15 m.

#### 16.4.3.3 Pit Sectorization

The proposed open pit design was divided into five geometrical sectors (Figure 16-11). As the main slope direction within some sectors has shown considerable variations, they were divided into subsectors. Table 16-19 summarizes the average direction of each slope of each sector for bench, inter-ramp and global scale.

Ten sections were drawn up in the mine geotechnical domains to verify the factor of safety (FoS) of the operational pit on an overall scale, demonstrated in Figure 16-11.

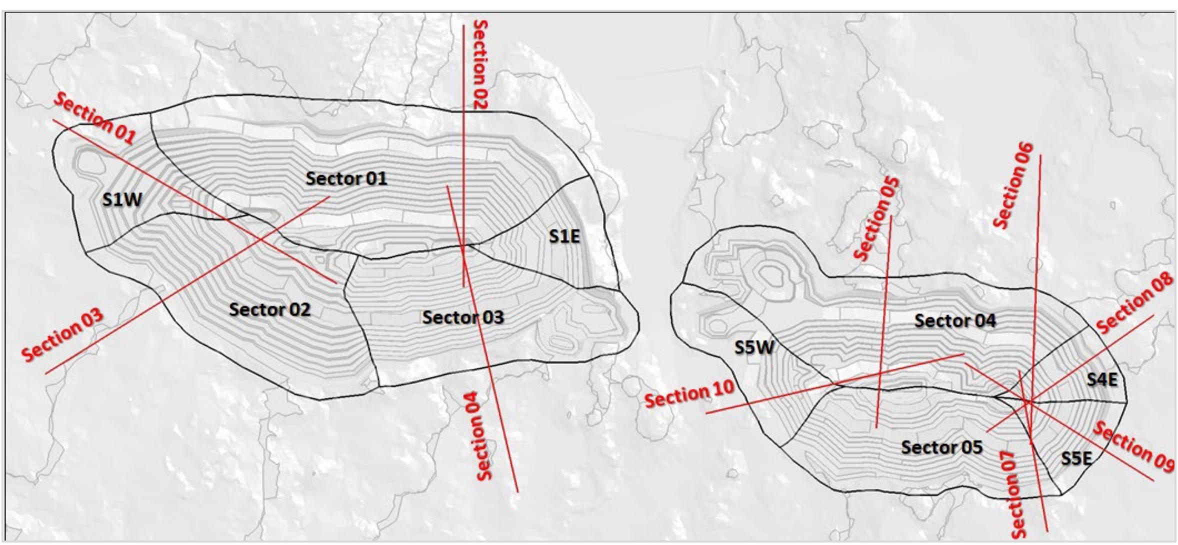



*Figure 16-11: Geomechanical sectors division for proposed open pits and vertical cross sections position for each sector, considered in the PEA Technical Report (August 5, 2021).*

**Table 16-19: Average Dip Directions and Bench, Inter-Ramp and Overall Angle Used in the PEA Technical Report (August 5, 2021)**

| Design Sector | Sub Sector | Slope Dip Direction | Bench Face Angle | Inter-Ramp Angle | Slope Height |
|---|---|---|---|---|---|
| **Sector 01** | **S1** | 170 | 80 | 56 | 15 |
| | **S1W** | 275 | 70 | 43 | 15 |
| | **S1E** | 120 | 85 | 59 | 15 |
| **Sector 02** | **S2** | 58 | 70 | 48 | 15 |
| **Sector 03** | **S3** | 347 | 85 | 59 | 15 |
| **Sector 04** | **S4** | 184 | 80 | 56 | 15 |
| | **S4E** | 235 | 85 | 59 | 15 |
| **Sector 05** | **S5** | 350 | 85 | 55 | 15 |
| | **S5W** | 300 | 85 | 55 | 15 |
| | **S5E** | 77 | 85 | 57 | 15 |

### 16.4.3.4 Kinematic Analysis

Kinematics analyses were performed using the televiewer structural data and the projection of the mineralized zones concerning the sectorization of the proposed pit design (Figure 16-11). The analysis identified the main failure mechanisms for each sector and subsector of the open pit.

In total, 80 kinematics analyses were completed. Table 16-20 summarizes each sector's relative structurally controlled failure potential, considering the bench scale. Table 16-21 presents the results for the inter-ramp scale.

**Table 16-20: Results of Kinematics Analyses for Bench Slope Scale**

| Design Sector | | | | Probability of Failure (Bench Scale) | | | |
|---|---|---|---|---|---|---|---|
| Sector | Sub Sector | Slope Orientation | Bench Face Angle | Planar | Wedge | Flexural Toppling | Direct Toppling (Base Plane) |
| **Sector 01** | S1 | 170 | 80 | Low | Moderate | High | Low |
| | S1W | 275 | 70 | - | Low | - | - |
| | S1E | 120 | 85 | - | Moderate | - | - |
| **Sector 02** | S2 | 58 | 70 | - | Low | - | Low |
| **Sector 03** | S3 | 347 | 85 | - | - | High | Low |

| Design Sector | | | | Probability of Failure (Bench Scale) | | | |
|---|---|---|---|---|---|---|---|
| Sector | Sub Sector | Slope Orientation | Bench Face Angle | Planar | Wedge | Flexural Toppling | Direct Toppling (Base Plane) |
| Sector 04 | S4 | 184 | 80 | High | High | Moderate | High |
| | S4E | 235 | 85 | - | Low | Low | - |
| Sector 05 | S5 | 350 | 85 | Low | Low | High | Low |
| | S5W | 300 | 85 | Low | Low | - | Low |
| | S5E | 77 | 85 | - | Moderate | - | Low |

**Table 16-21: Results of Kinematics Analyses for Inter-Ramp Slope Scale**

| Design Sector | | | | Probability of Failure (Inter Ramp Scale) | | | |
|---|---|---|---|---|---|---|---|
| Sector | Sub Sector | Slope Orientation | Inter-Ramp Angle | Planar | Wedge | Flexural Toppling | Direct Toppling (Base Plane) |
| Sector 01 | S1 | 170 | 56 | - | - | High | Low |
| | S1W | 275 | 43 | - | - | - | - |
| | S1E | 120 | 59 | - | Moderate | - | - |
| Sector 02 | S2 | 58 | 48 | - | - | - | - |
| Sector 03 | S3 | 347 | 59 | - | - | High | - |
| Sector 04 | S4 | 184 | 56 | - | - | - | - |
| | S4E | 235 | 59 | - | Low | - | - |
| Sector 05 | S5 | 350 | 55 | - | - | High | Low |
| | S5W | 300 | 55 | - | Low | - | Low |
| | S5E | 77 | 57 | - | Moderate | - | - |

### 16.4.4 Stability Analysis

#### 16.4.4.1 Geotechnical Parameters

For the development of the studies, the following criteria and assumptions for the slope stability analyses were assumed:

- According to Wesseloo & Read (2009), the results of the stability analysis developed for mining slopes in terms of deterministic (FoS) and probabilistic (probability of failure [PoF]) approaches must meet the typical values of the acceptance criteria of projects presented in Table 16-22.

**Table 16-22: Typical FoS and PoF Acceptance Criteria Values (Wesseloo & Read, 2009)**

| | | Acceptance criteria[a] | | |
|---|---|---|---|---|
| Slope scale | Consequences of failure[b] | FoS (min) (static) | FoS (min) (dynamic) | PoF (max) P[FoS ≤ 1] |
| Bench | Low–high | 1.1 | NA | 25–50% |
| Inter-ramp | Low | 1.15–1.2 | 1.0 | 25% |
| | Medium | 1.2 | 1.0 | 20% |
| | High | 1.2–1.3 | 1.1 | 10% |
| Overall | Low | 1.2–1.3 | 1.0 | 15–20% |
| | Medium | 1.3 | 1.05 | 5–10% |
| | High | 1.3–1.5 | 1.1 | ≤5% |

a: Needs to meet all acceptance criteria
b: Semi-quantitatively evaluated (see Figure 13.9)

- The bench scale analyses of the slopes were considered drained.
- The overall and inter-ramp analyses were considered for two hydrogeological scenarios due to the lack of specific studies:
  - Dry condition: Slope without the presence of water level or at a drained condition due to the effectiveness of the depressurization system. In this last scenario, the groundwater level is distant from the potential failure surfaces; and
  - Saturated condition: Slope with a groundwater level approximately set back at least 5 m from the face of the slope.
- The pseudo-static condition analyses, Kh.g equal to 0.063 g for a return time of 2,475 years was considered, obtained through the Seismic Risk Hazard Site from the Government of Canada.
- Due to the lack of data, the strength properties of the overburden material were estimated, with a value cohesion equal to 0 kPa and a friction angle equal to 29°.
- The disturbance factor was estimated as D=0.7, with an average thickness of 20 m.
- The relationship between in situ vertical and horizontal stresses was estimated to be k=1.
- Slope stability analyses were run on Slide2 software, from Rocscience®. The FoS was calculated considering Mohr-Coulomb (isotropic and anisotropic), Barton-Bandis and generalized Hoek-Brown strength criteria. The water level was simulated throughout a hydrogeological model. The FoS was calculated using the GLE/Morgenstern-Price method. The Cuckoo Search research option was adopted to be efficient and fast for global optimization for critical non-circular failure surfaces.

Strength and deformability parameters used in the slope stability analyses are presented in Table 16-23. As there were no values for mi parameter, they were based on the data available within Hoek's tables (2001).

**Table 16-23: Geotechnical Parameters (Strength and Deformability) used in Stability Analysis**

| Geomechanics Class | Argilite | | | | | | Graywacke | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | GSI* | UCS* (MPa) | mi *– intact rock | D (disturbance factor) | Young's Modulus – Intact rock Ei (GPa) | Poisson's Ratio | GSI* | UCS* (MPa) | mi *– intact rock | D (disturbance factor) | Young's Modulus – Intact rock Ei (GPa) | Poisson's Ratio |
| II | 64±5 | 60±16 | 7±4 | 0 | 67.20 | 0.29 | 64±5 | 70±18 | 18±4 | 0 | 73.0 | 0.20 |
| III | 48±5 | 60±16 | 7±4 | 0 | 67.20 | 0.29 | 48±5 | 70±18 | 18±4 | 0 | 73.0 | 0.20 |
| IV | 24±5 | 60±16 | 7±4 | 0 | 67.20 | 0.29 | 24±5 | 70±18 | 18±4 | 0 | 73.0 | 0.20 |
| II – D 0.7 | 64±5 | 60±16 | 7±4 | 0.7 | 67.20 | 0.29 | 64±5 | 70±18 | 18±4 | 0.7 | 73.0 | 0.20 |
| II – D 0.7 | 48±5 | 60±16 | 7±4 | 0.7 | 67.20 | 0.29 | 48±5 | 70±18 | 18±4 | 0.7 | 73.0 | 0.20 |
| IV – D 0.7 | 24±5 | 60±16 | 7±4 | 0.7 | 67.20 | 0.29 | 24±5 | 70±18 | 18±4 | 0.7 | 73.0 | 0.20 |

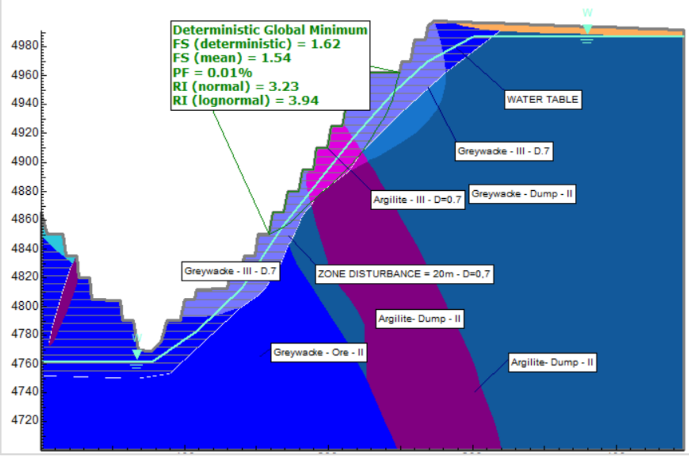
* - variables and ranges utilized in probabilistic analyses.

### 16.4.4.2 Geotechnical Analysis

Ten sections were sketched for the defined mine geotechnical domains to verify the FoS of the operational pit on an overall scale. Figure 16-11 shows the location of the analysis sections.

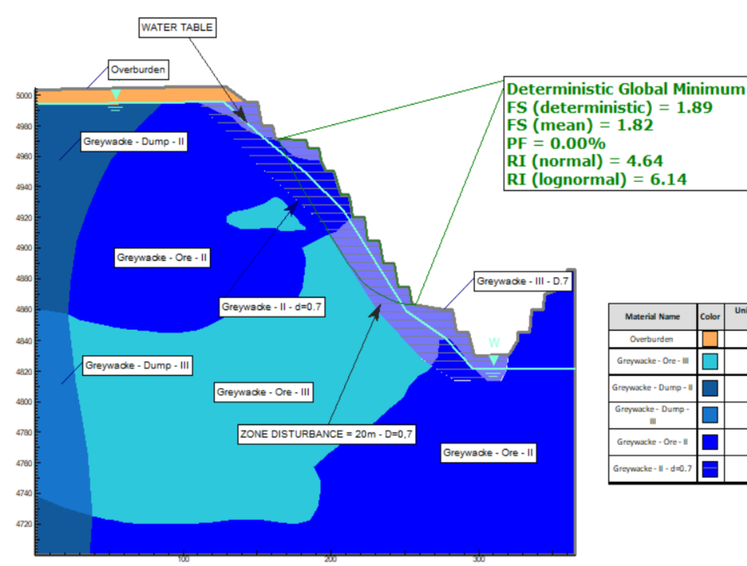
The stability analyses demonstrate a very similar failure pattern governed by the excavation disturbance zone and the groundwater level, the exception being in section S04, in which the influence of the lowest geomechanical class was verified.

Figure 16-12 and Figure 16-13 display the stability analyses of sections S02 and S05 which are the most representative sections for the Project. Table 16-24 summarizes the overall and inter-ramp stability analysis results.




| Material Name | Color | Unit Weight (kN/m3) | Strength Type | Cohesion (kPa) | Phi (deg) | UCS (kPa) | GSI | mi | D |
|---|---|---|---|---|---|---|---|---|---|
| Overburden | | 19 | Mohr Coulomb | 0 | 28 | | | | |
| Argilite Dump II | | 27 | Generalized Hoek Brown | | | 60000 | 64 | 7 | 0 |
| Argilite Dump III | | 27 | Generalized Hoek Brown | | | 60000 | 48 | 7 | 0 |
| Greywacke Ore III | | 27 | Generalized Hoek Brown | | | 70000 | 64 | 18 | 0 |
| Greywacke Dump II | | 27 | Generalized Hoek Brown | | | 70000 | 64 | 18 | 0 |
| Greywacke Dump III | | 27 | Generalized Hoek Brown | | | 70000 | 48 | 18 | 0 |
| Greywacke Ore II | | 27 | Generalized Hoek Brown | | | 70000 | 64 | 18 | 0 |
| Argilite III D=0.7 | | 27 | Generalized Hoek Brown | | | 60000 | 48 | 7 | 0.7 |
| Greywacke II d=0.7 | | 27 | Generalized Hoek Brown | | | 70000 | 64 | 18 | 0.7 |
| Greywacke III D.7 | | 27 | Generalized Hoek Brown | | | 70000 | 48 | 18 | 0.7 |

*Figure 16-12: S02 Geotechnical model cross section*




| Material Name | Color | Unit Weight (kN/ m3 ) | Strength Type | Cohesion (kPa) | Phi (deg) | UCS (kPa) | GSI | mi | D |
|---|---|---|---|---|---|---|---|---|---|
| Overburden | | 19 | Mohr-Coulomb | 0 | 28 | | | | |
| Greywacke - Ore - III | | 27 | Generalized Hoek-Brown | | | 70000 | 64 | 18 | 0 |
| Greywacke - Dump - II | | 27 | Generalized Hoek-Brown | | | 70000 | 64 | 18 | 0 |
| Greywacke - Dump - III | | 27 | Generalized Hoek-Brown | | | 70000 | 48 | 18 | 0 |
| Greywacke - Ore - II | | 27 | Generalized Hoek-Brown | | | 70000 | 64 | 18 | 0 |
| Greywacke - II - d=0.7 | | 27 | Generalized Hoek-Brown | | | 70000 | 64 | 18 | 0.7 |

*Figure 16-13: S05 Geotechnical model cross section*

**Table 16-24: FoS for the Analyzed Sections**

| Section | Stability Analysis Using Conditions Dewatered | Stability Analysis Using Conditions Saturated | | Stability Analysis Using Pseudo Static Condition | Stability Analysis Using the Stress-Strain Method and Conditions Dewatered | Stability Analysis Using the Stress-Strain Method and Conditions Saturated |
|---|---|---|---|---|---|---|
| | $F_oS$ | $F_oS$ | PoF | $F_oS$ | SRF | SRF |
| S01 | 2.81 | 2.25 | >1x10$^{-4}$ | 2.01 | 2.56 | 2.01 |
| S02 | 1.93 | 1.62 | 1x10$^{-4}$ | 1.44 | 1.59 | 1.30 |
| S03 | 3.79 | 2.46 | >1x10$^{-4}$ | 2.22 | 3.01 | 1.94 |
| S04 | 3.27 | 2.12 | >1x10$^{-4}$ | 1.89 | 2.29 | 2.04 |
| S05 | 2.60 | 1.89 | >1x10$^{-4}$ | 1.74 | 2.19 | 1.59 |
| S06 | 2.35 | 1.87 | 1x10$^{-4}$ | 1.67 | 2.01 | 1.52 |
| S07 | 3.35 | 2.55 | >1x10$^{-4}$ | 2.34 | 3.20 | 2.59 |
| S08 | 2.22 | 1.81 | >1x10$^{-4}$ | 1.65 | 2.19 | 1.84 |
| S09 | 3.39 | 2.63 | >1x10$^{-4}$ | 2.38 | 3.30 | 2.53 |
| S10 | 2.94 | 2.28 | >1x10$^{-4}$ | 2.07 | 2.24 | 1.80 |

#### 16.4.4.3 Geometrical Slope Configuration

The pit sector configuration is shown in Figure 16-11 and the pit design parameters in Table 16-25.

**Table 16-25 - Slope Configurations**

| Sector | Sub Sector | Slope Orientation | Berm Width | Bench Height | | Bench Face Angle | Inter-Ramp Angle |
|---|---|---|---|---|---|---|---|
| | | | | Operational | Final Wall | | |
| **Sector 01** | S1 | 170 | 7.5 m | 5 m | Triple Bench (15 m) | 80 | 56 |
| | S1W | 275 | 7.5 m | 5 m | Triple Bench (15 m) | 70 | 43 |
| | S1E | 120 | 7.5 m | 5 m | Triple Bench (15 m) | 85 | 59 |
| **Sector 02** | S2 | 58 | 7.5 m | 5 m | Triple Bench (15 m) | 70 | 48 |

<table>
<tr><th rowspan="2">Sector</th><th rowspan="2">Sub Sector</th><th rowspan="2">Slope Orientation</th><th rowspan="2">Berm Width</th><th colspan="2">Bench Height</th><th rowspan="2">Bench Face Angle</th><th rowspan="2">Inter-Ramp Angle</th></tr>
<tr><th>Operational</th><th>Final Wall</th></tr>
<tr><td>Sector 03</td><td>S3</td><td>347</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>85</td><td>59</td></tr>
<tr><td rowspan="2">Sector 04</td><td>S4</td><td>184</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>80</td><td>56</td></tr>
<tr><td>S4E</td><td>235</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>85</td><td>59</td></tr>
<tr><td rowspan="3">Sector 05</td><td>S5</td><td>350</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>85</td><td>55</td></tr>
<tr><td>S5W</td><td>300</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>85</td><td>55</td></tr>
<tr><td>S5E</td><td>77</td><td>7.5 m</td><td>5 m</td><td>Triple Bench (15 m)</td><td>85</td><td>57</td></tr>
</table>

### 16.4.5 FS Design Changes

Following the FS initial design work by Optimize, Nordmin requested Optimize to evaluate a geometrical change in two geometrical constraints:

1. bench height, and
2. berm width.

This assessment was requested to calculate a possible reduction of dilution and increase in productivity.

#### 16.4.5.1 Bench Height

The geometrical change to the bench height of a one-meter increase resulted in considering an 8 m operational bench height, with a double bench (16 m) for the final pit wall and long period pushbacks.

This one-meter increase was considered a minor change in the geometrical aspect considering the 2021 PEA. The height increase directly impacted the structural PoF presented in Table 16-22 and the FoS. More geological structures would be exposed within the same bench face.

Optimize reviewed the one metre change and determined that it would not significantly affect the geotechnical conditions of the proposed open pit.

#### 16.4.5.2 Berm Width

Considering the new bench height of 16 m, Optimize achieved a berm width of 7.7 m following the criterion that relates the bench height with the berm width, as shown in Equation 2.

*Equation 2. Berm width ratio for falling blocks. Source: Read and Stacey (2009)*

$$bench\,width\,(m) = 0.2 * bench\,height\,(m) + 4.5m = 0.2 * 16 + 4.5 = 7.7m$$

Optimize and Nordmin concluded that the proposed berm width of 8.0 m wide would improve safety.

#### 16.4.5.3 Qualitative Consideration and Conclusions

The change in bench height and berm width do not significantly impact the FoS calculated since most rocks vary between Geomechanical classes II and III, which are relatively medium-strength classes. However, the height increase could lead to more instabilities in the bench scale, and further test work is required.

The increase in the berm width is considered a conservative approach. It utilizes a berm that is 0.3 m wider than the formula indicated. This more conservative approach could absorb potential falling blocks.

It can be concluded that the final slopes of the final operational open pit have an adequate security level for an FS. The slope stability analyses and geomechanical geometries proposed have a satisfactory FoS.

However, some additional recommendations are made for the next stages:

- Perform additional drill holes to increase the knowledge of Sectors 01, 1E, S4E, S5E and 5 and to improve the geotechnical model.
- Incorporate additional borehole samples description, based on RMR parameters, to increase the geotechnical information.
- Use the cores to get new laboratory parameters, especially those from Hoek's triaxial cell and Brazilian tests;
- Perform 3D hydrogeologic model studies to evaluate the drawdown, as the stability of the final open slopes are intimately related to the position of the water level.

It is important that the benches and bench faces are cleaned so that they remain functional during mining. Detailed geotechnical mapping should be conducted after cleanup to verify the competency of the rock mass, and the orientation, population, and location of critical joint sets.

Data from pit development will be used for ongoing slope stability analyses and design optimization. A pit slope monitoring program will be required. It will include frequent inspections of benches and crests for tension cracking and other signs of instability. It will also need to include survey scanning, movement detection systems, and groundwater monitoring. The monitoring system should be set up with priority given to the higher risk areas and configured for the anticipated failure mode(s).

# 17. RECOVERY METHODS

## 17.1 Overall Process Design

The provided testwork was analyzed, and several options of process routes were addressed in the initial stages of the FS. Based on the analysis, a conventional leach and carbon-in-pulp process route was chosen as the most suitable for the Deposit and Project economics. The unit operations selected are all typical for gold recovery, and the proposed flowsheet uses standard processes and technologies.

Key operating criteria for the process plant are:

- Nominal throughput of 4,000 t/d or 1.46 Mtpa
- Crushing plant availability of 64%
- Plant availability of 92% for grinding, leach plant, and gold recovery operations

## 17.2 Mill Process Plant Description

The process design is comprised of the following circuits:

- Three stage crushing of ROM material
- A covered, crushed ore stockpile to provide buffer capacity ahead of the grinding circuit
- A ball mill with cyclone classification
- Gravity concentration and intensive leaching of the concentrate
- Trash screening
- Leach + carbon adsorption (L/CIP)
- Acid washing of loaded carbon and Pressure Zadra type elution followed by electrowinning and smelting to produce doré
- Carbon regeneration by rotary kiln
- Cyanide destruction of tailings using $SO_2$/air process
- Arsenic precipitation
- Tailings thickening

### 17.2.1 Plant Design Criteria

Key process design criteria for the mill are listed in Table 17-1.

**Table 17-1: Process Design Criteria**

| Design Parameter | Units | Value |
|---|---|---|
| Plant Throughput | t/d | 4,000 |
| Gold Grade – Design Mill Head | g/t | 2.58 |
| Crushing Plant Availability | % | 64 |
| Mill Availability | % | 92 |
| Bond Crusher Work Index (CWi), 75th percentile | kWh/t | 23 |
| Bond Rod Mill Work Index (BWi), 75th percentile | kWh/t | 17.6 |
| Bond Ball Mill Work Index (BWi), 75th percentile | kWh/t | 15.7 |
| SMC Axb, 25th percentile | - | 30.4 |

| Design Parameter | Units | Value |
|---|---|---|
| Bond Abrasion Index (Ai) | g | 0.228 |
| Material Specific Gravity | $t/m^3$ | 2.75 |
| Primary Grind size (P80) | µm | 100 |
| Primary Crusher | - | Jaw, 1 m x 1.3 m |
| Secondary Crusher | - | Standard Cone, 1.32 m diam. |
| Tertiary Crusher | - | Shorthead Cone, 1.32 m diam. |
| Ball Mill Dimensions | - | 5.2 m diam. x 7.9 m EGL |
| Ball Mill Installed Power | MW | 3.5 |
| Leach Residence Time | h | 30 |
| CIP Residence Time | h | 6 |
| Gravity Gold Recovery (design) | % Au | 40 |
| Total Gold Recovery (life of mine) | % Au | 96 |
| Leach pH target range | - | 10.5-11 |
| Leach-CIP Operating Density | % solids (w/w) | 44 |
| Leach Sodium Cyanide Addition | kg/t | 0.5 |
| Leach Hydrated Lime Addition | kg/t | 1.0 |
| Leach & CIP Tanks | # | 3 + 6 |
| Tonnes of Carbon per Elution Column | t | 3 |
| Detoxification Residence Time | min | 120 |
| Detoxification Tanks | # | 2 (Parallel) |
| Detoxification $SO_2$ Addition (as SMBS) | $SO_2$:$CN_{WAD}$ ratio(w/w) | 10 |
| Detoxification Lime Addition | kg/t | 0.80 |
| Detoxification Discharge $CN_{WAD}$, Design | mg/L | <0.5 |
| Detoxification Discharge $CN_{TOT}$, Design | mg/L | 0.5 |
| Arsenic Precipitation Residence Time, Design | min | 10 |
| Ferric Sulphate Addition Ratio | Fe:As Ratio (w/w) | 8 |
| Thickener Underflow Density | % w/w solids | 60 |

### 17.2.2 Primary Crushing and Stockpiling

The crushing circuit is designed for an annual operating time of 5,631 hours or 64% availability at a capacity of 4,000 tpd.

ROM ore is transported from the mine to the process plant by haul trucks. The ROM bin is fitted with a static grizzly to keep large oversize rocks from blocking the primary crusher. A vibrating grizzly feeder ahead of the of the primary jaw crusher is used to screen out fine material and feed the jaw crusher. The primary crusher product combines with the grizzly feeder undersize and is conveyed to a secondary screen. Screen oversize is fed to a secondary cone crusher and the secondary crushed product is fed to the tertiary screen. The tertiary crusher operates in closed circuit with a tertiary screen. The combined undersize from both the secondary and tertiary screens is conveyed to the mill feed stockpile with 80% of the particle size distribution passing an aperture of 10 mm. The mill feed stockpile is equipped with two belt feeders to regulate feed at 181 t/h into the ball mill via a feed conveyor. Each feeder is capable of feeding the plant at design capacity independently.

The material handling and crushing circuit includes the following key equipment:

- ROM Bin
- Vibrating grizzly
- Fixed rock breaker
- Primary jaw crusher, 1 m x 1.3 m
- Secondary screen
- Secondary cone crusher, 1.32 m diam. head
- Tertiary Screen
- Tertiary shorthead cone crusher, 1.32 m diam. head
- Crushed ore reclaim belt feeders (equipped with VSDs)
- Conveyors
- Metal detection and rejection

Crushing circuit product is conveyed to a covered mill feed stockpile. The stockpile is designed to have a live capacity of 12 hours or 2,000 tonnes. The stockpile ensures that the processing plant operates independently of the mining and crushing activities and provides a more stable feed to the grinding circuit.

### 17.2.3 Grinding Circuit

The grinding circuit consists of a ball mill in closed circuit with hydrocyclones. The circuit is sized based on ball mill feed size of 80% passing 10 mm and product of 80% passing 100 µm.

Mill feed from the stockpile is reclaimed by two belt feeders. The feeders transfer the mill feed onto the ball mill feed conveyor which discharges the ore directly into the ball mill feed chute. Process water is added to the ball mill feed chute to create a slurry in the ball mill. The mill feed chute also receives oversized material from the gravity scalping screen and the underflow from the hydrocyclone cluster. The mill is operated in closed circuit where the product is discharged into a common pumpbox with independent pumps for both the hydrocyclone cluster and the gravity circuit. The mill discharge includes a trommel for scat removal. Ground material that is too coarse will be classified by the hydrocylone and will report back to the ball mill for further size reduction.

The hydrocyclone classification circuit will operate at a nominal circulating load of 400% which is a typical for the design cyclone overflow density and target grind size.

The cyclone cluster is configured to achieve a target design cyclone overflow product sizing of 80% passing of 100 µm. Leach testing showed that this product size is sufficient to achieve design recoveries of gold. The cyclone cluster will have pneumatically actuated valves that allow automated feed pressure control as well as manually actuated isolation valves for maintenance.

Process water is also added to the cyclone feed pumpbox to obtain the appropriate density prior to pumping to the cyclones. Cyclone overflow is sent to a trash screen prior to the leaching circuit to remove any large detritus that may accumulate in the leach tanks. The ball mill cyclone pumpbox is also equipped with a second pump which discharges slurry to the gravity concentration circuit where gravity gold is concentrated and treated.

The grinding circuit includes the following key equipment:

- 3,500 kW, 5.2 m diam. X 7.9 m EGL ball mill with a single pinion drive
- Cyclone feed pumpbox

- Classification cyclones
- Cyclone feed pump, gravity feed pump
- Trash screen

### 17.2.4 Gravity Circuit

The gravity circuit is fed from a dedicated pump on the cyclone feed pumpbox, with all tails and oversize stream recollected and pumped back to the ball mill feed. A side stream of ball mill discharge is pumped to the gravity sizing screen and the screen undersize is fed to a centrifugal gravity concentrator. A batch of concentrate is produced every 45 minutes and fed to the intensive cyanidation unit (ICU) holding tank. The gravity tails are fed back to the ball mill by gravity.

Gravity concentrate will be fed to the ICU cone. Process water will be fed to the cone for approximately 30 minutes to deslime the contents. The ICU solution tank contains sodium hydroxide, sodium cyanide, and leach aid diluted in freshwater. The reagent mixture is pumped through the bottom of the ICU cone producing a fluidized bed. Overflow returns to the solution tank. The pregnant leach solution is pumped to gold room for electrowinning and refining. The barren solids are pumped back to the cyclone feed pumpbox.

### 17.2.5 Leach and Adsorption Circuit

The leach-adsorption circuit consists of three leach tanks and six CIP tanks. Trash screen undersize flows to a pumpbox and then pumped to the leach circuit. Barren solution from electrowinning cells is periodically transferred to the leach circuit. The leach and CIP tanks have a total circuit residence time of 36 hours at 44% solids (w/w) density.

Hydrated lime slurry is added to adjust the operating pH to the desired set point of 10.5 to 11 and cyanide solution is added to the first leach tank. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIP circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. Slurry from the last CIP tank flows to the cyanide detoxification tanks.

The intertank screen in each CIP tank retains the carbon while allowing the slurry to flow to the downstream tank. This counter-current process operates until the gold loaded onto carbon in the lead tank reaches its target concentration and is sent to acid wash. Recessed impeller pumps are used to transfer slurry between the CIP tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit.

The leach and carbon adsorption circuit includes the following key equipment:

- Leach feed pump
- Leach tanks and agitators, 15.3 m diam. X 16.7 m height, 100 kW agitator
- CIP tanks and agitators, 7.2 m diam. X 7.8 m height, 15 kW agitator
- Loaded carbon screen
- Intertank carbon screens
- Carbon sizing screen

### 17.2.6 Cyanide Destruction

CIP tailings at 44% solids (w/w) flow by gravity to the cyanide destruction tanks which operate in parallel. The water used for acid rinse and carbon transfer is also included in the feed to detoxification circuit. As a result, the percentage solids in the feed to the detoxification circuit is estimated to be closer to 43% solids (w/w).

The tanks provide a total residence time of approximately 120 mins to reduce weak acid dissociable cyanide ($CN_{WAD}$) concentration from 100 mg/L to less than 0.5 mg/L. Total cyanide ($CN_{TOT}$) is expected to be 0.5 mg/L due to limited iron species present in the leach feed.

Cyanide destruction is undertaken using the $SO_2$/air method. The reagents required are air, lime, copper sulphate, and sodium metabisulphite (SMBS). Each cyanide destruction tank is equipped with air addition points and an agitator to ensure that the reagents are thoroughly mixed with the tailings slurry.

The detoxification tanks feed the arsenic precipitation tank via gravity flow.

The main equipment in this area includes:

- Cyanide destruction tanks and agitators, 7.8 m diam. X 7.8 m height, 30 kW agitator

### 17.2.7 Arsenic Precipitation

The arsenic precipitation tank precipitates the dissolved arsenic by the addition of ferric sulphate at a design ratio by weight of 8:1 iron to arsenic in solution. The arsenic concentration in solution is approximately 6.3 mg/L based on metallurgical testing. The arsenic precipitation tank provides a residence time in excess of the design value of 10 minutes residence time for the reaction to occur. Tailings from the arsenic precipitation tank flows by gravity to the carbon safety screen to ensure no carbon is lost to tailings. The screen oversize (recovered carbon) is collected in a bin for return to the CIP circuit. The screen undersize is pumped to the tailings thickener.

The main equipment in this area includes:

- Arsenic precipitation tank and agitator, 6.5 m diam. X 7.2 m height, 30 kW agitator
- Carbon safety screen

### 17.2.8 Tailings Thickening

Detoxified tailings slurry flows from the arsenic precipitation tank to the carbon safety screen where any carbon in slurry is removed in the screen oversize. Screen undersize is pumped to the tailings thickener where it is dewatered to obtain an underflow density of 60% solids (w/w). The thickener overflow is recycled to the process water tank while the underflow is pumped to the tailings storage facility. Flocculant is added to the thickener feed to improve the solids settling rate. Excess water from the process water tank is sent to effluent treatment.

The main equipment in this area includes:

- High-rate thickener, 18 m diam.
- Overflow tank for process water storage
- Final tailings pumps

### 17.2.9 Carbon Acid Wash, Elution and Regeneration Circuit

#### 17.2.9.1 Carbon Acid Wash

Prior to gold elution, loaded carbon is treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide destruction tank.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

The main equipment in this area includes:

- Acid wash carbon column – 3 tonne capacity
- Hydrochloric acid feed pump
- Spent solution discharge sump pump

#### 17.2.9.2 Carbon Stripping (Elution) and Electrowinning

The gold stripping (elution) circuit uses the Pressure Zadra process.

A high-cyanide, caustic solution is recirculated through a pressure elution column at 140°C to strip the precious metals from the carbon. The precious metal-rich solution from the column exchanges heat with barren solution going to the column. Cooled rich solution then flows through electrowinning cells to deposit the gold and silver on the cathodes before the solution is recycled back to the elution column.

The stripping circuit includes the following key equipment:

- Carbon elution column – 3 tonne capacity
- Strip solution heater (propane-fired) with heat exchangers
- Strip eluate, and pregnant solution tanks

After completion of the elution process, stripped carbon is hydraulically transferred from the elution column to the eluted carbon dewatering screen. The screened carbon is fed into the kiln feed hopper then metered into the carbon regeneration kiln.

#### 17.2.9.3 Gold Room

Gold/silver sludge is recovered from the electrowinning cells and smelted to produce doré bars. Separate electrowinning cells are provided for the ICU pregnant solution and the carbon elution pregnant solution.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high pressure spray water and gravitates to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold doré. The electrowinning and smelting process takes place within a secure and supervised gold room equipped with access control, intruder detection, and closed circuit television equipment.

The electrowinning circuit and gold room include the following key equipment:

- Electrowinning cells with rectifiers
- Sludge pressure filter
- Drying oven
- Flux mixer
- Smelting furnace with cascade doré moulds and slag handling system
- Doré vault and safe

- Dust and fume collection system
- Gold room security system

#### 17.2.9.4 Carbon Reactivation

Carbon is reactivated in a propane-fired rotary kiln. Dewatered barren carbon from the stripping circuit is held in a 3-tonne kiln feed hopper. A screw feeder metres the carbon into the reactivation kiln, where it is heated to 650° to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln is quenched in water and screened on a carbon sizing screen located on top of the CIP tanks to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, while carbon screen oversize is directed to the CIP circuit.

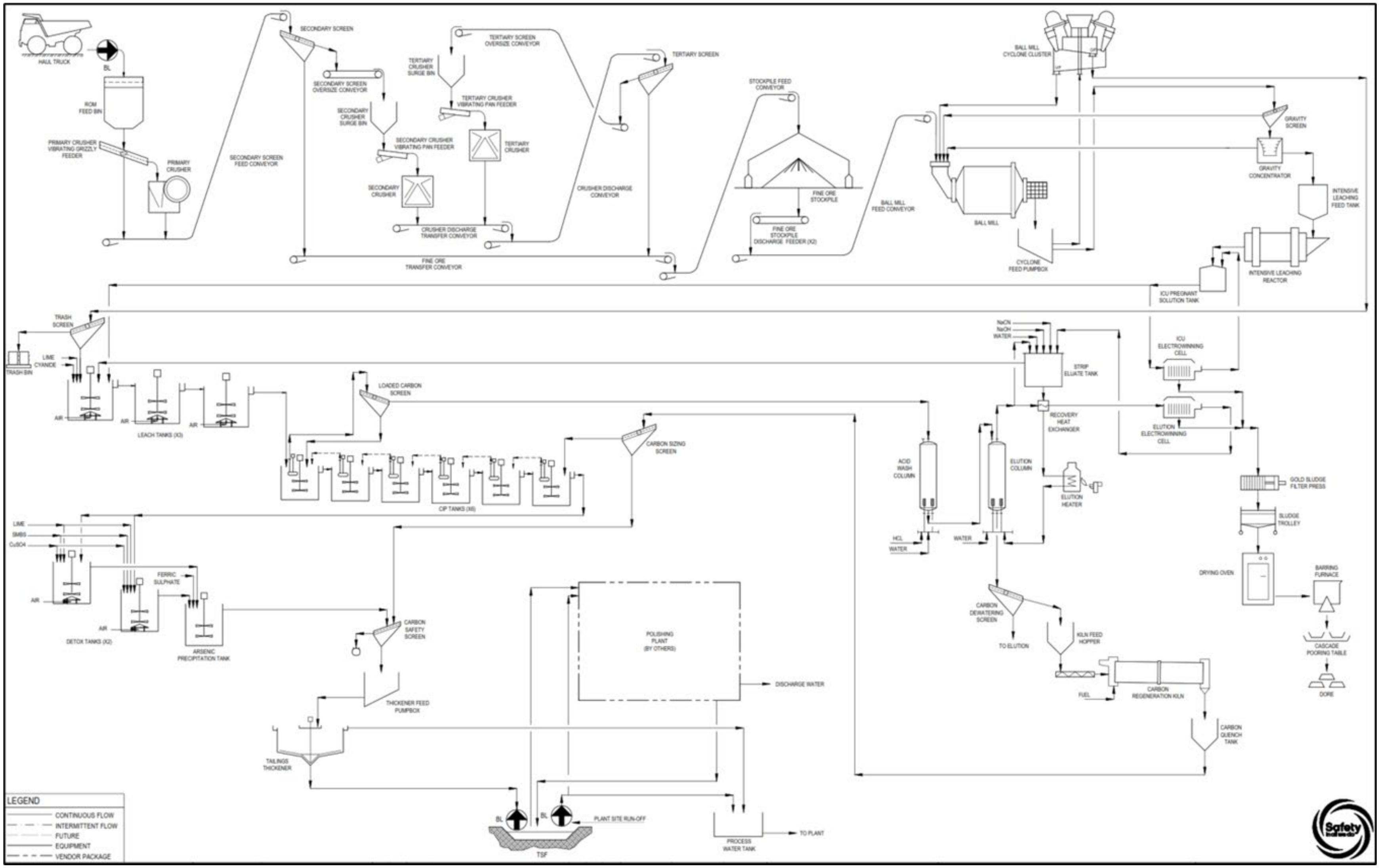
As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit includes the following key equipment:

- Carbon dewatering screen
- Regeneration kiln (propane) including feed hopper and screw feeder
- Carbon quench tank

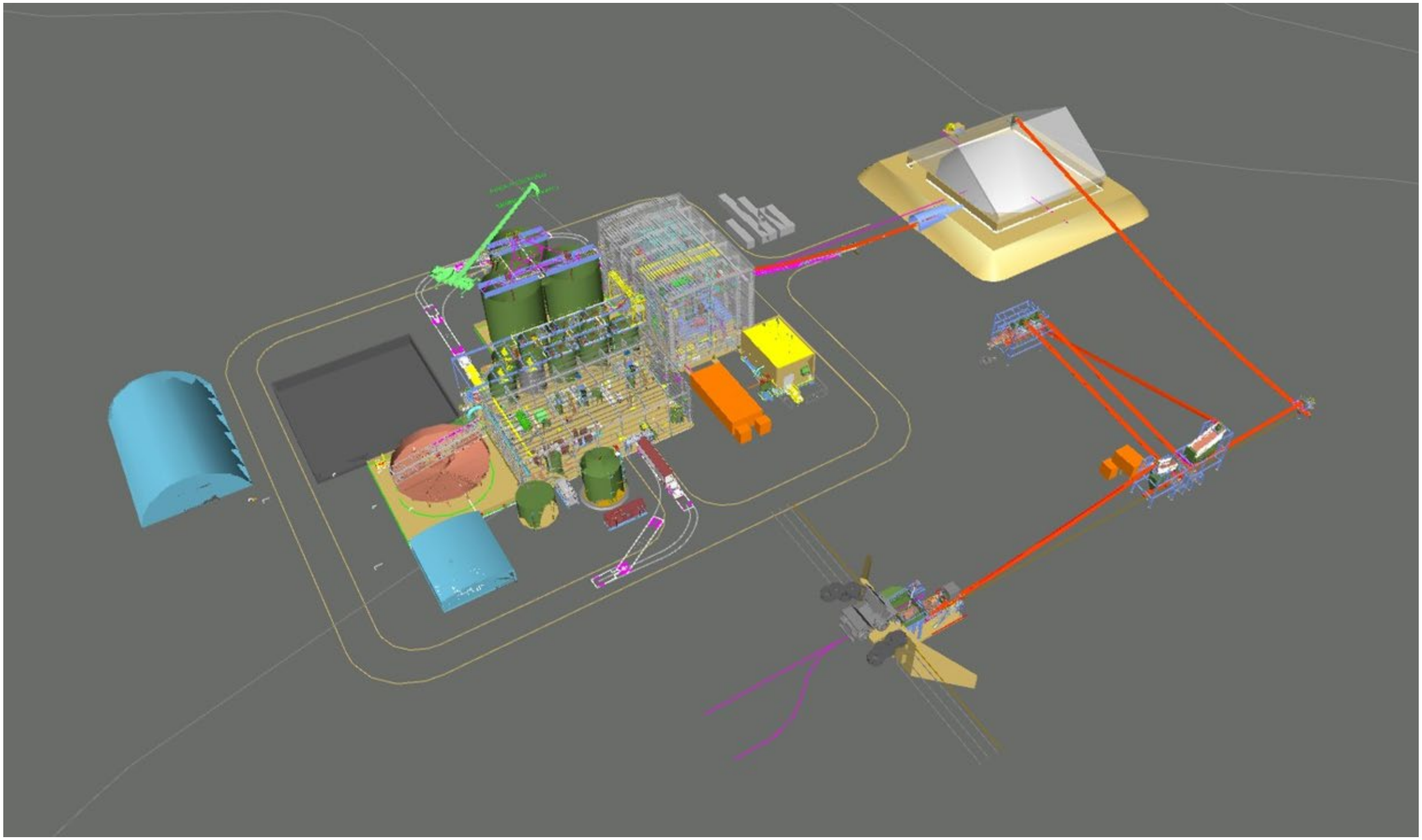
### 17.2.10 Flowsheet and Layout Drawings

An overall process flow diagram showing the unit operations in the selected process flowsheet is presented in Figure 17-1. Isometric views of the proposed plant extracted from the three dimensional model are provided in Figure 17-2 to Figure 17-7.



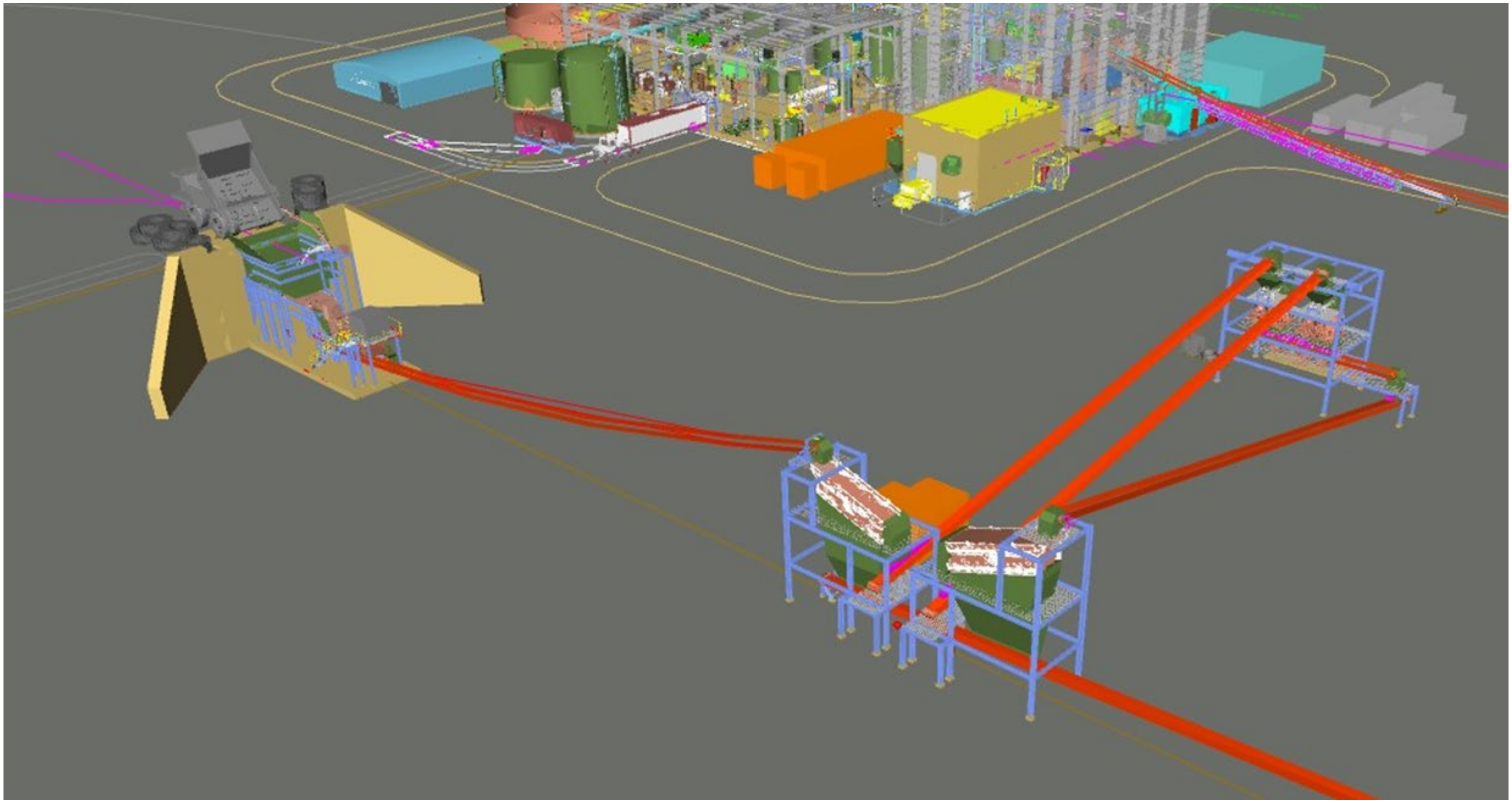

Source: Ausenco, 2021

*Figure 17-1: Overall process flow diagram*



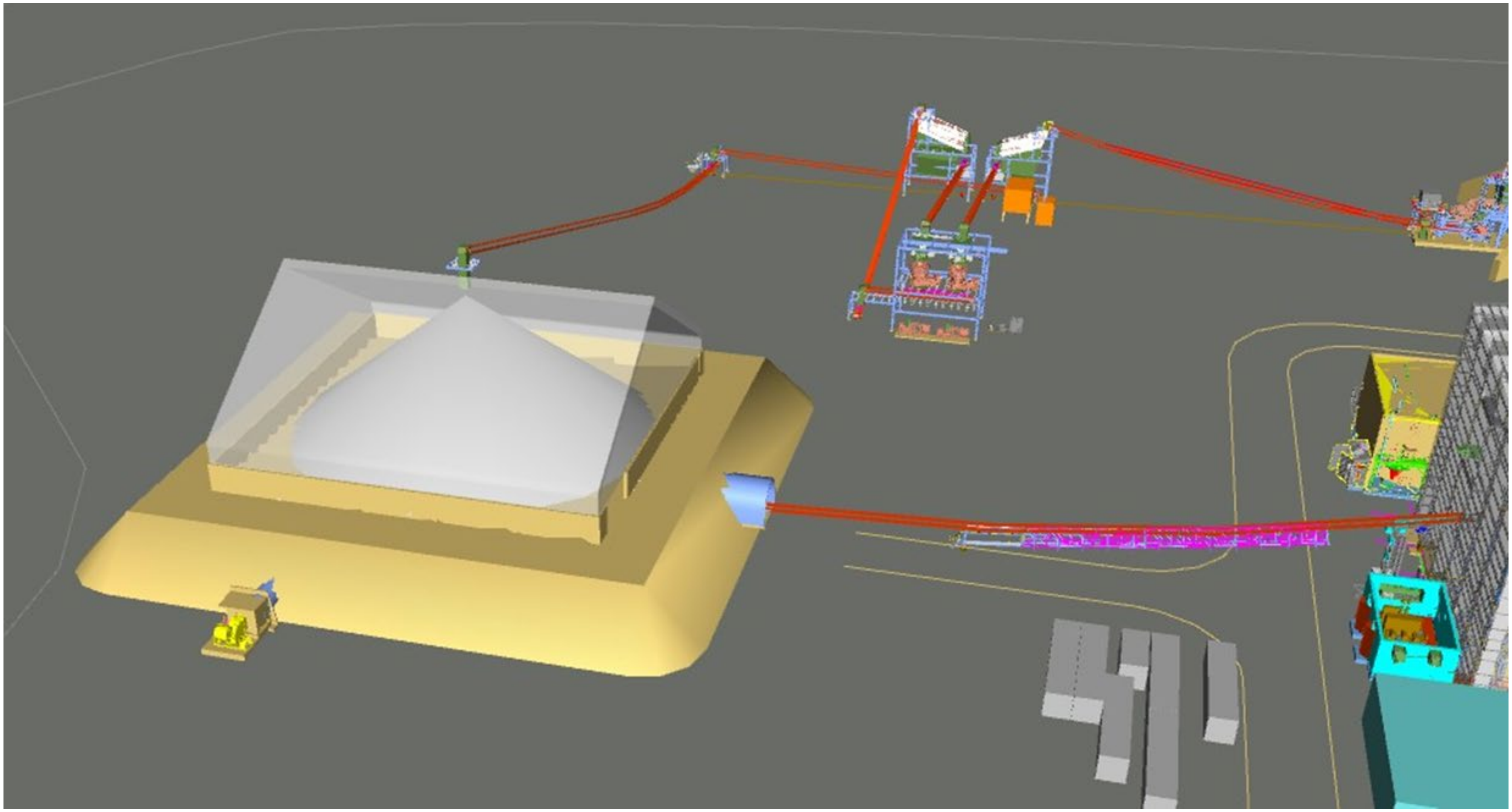
Source: Ausenco, 2021

*Figure 17-2: Overall plant layout*



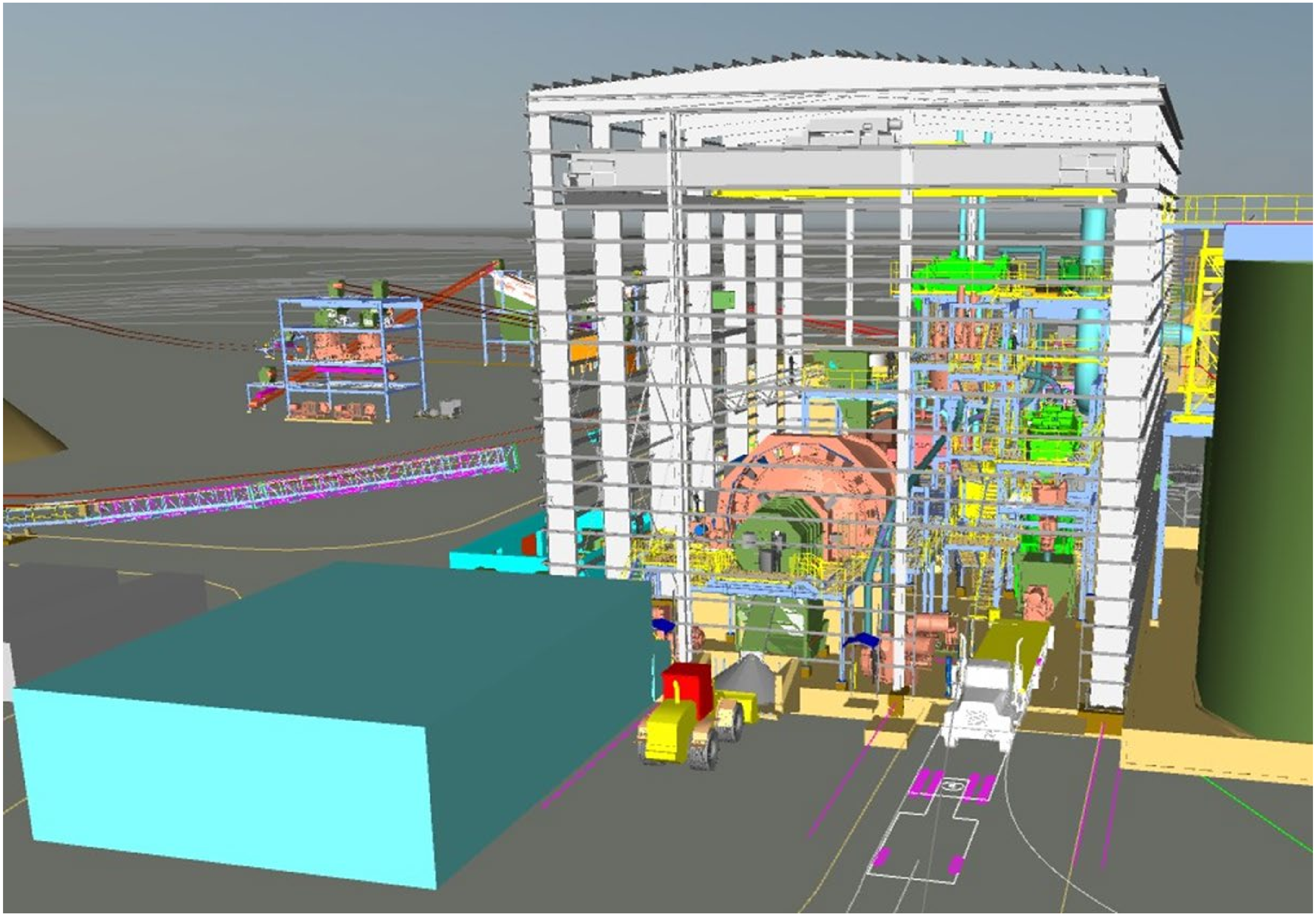
Source: Ausenco, 2021

*Figure 17-3: Crushing area*



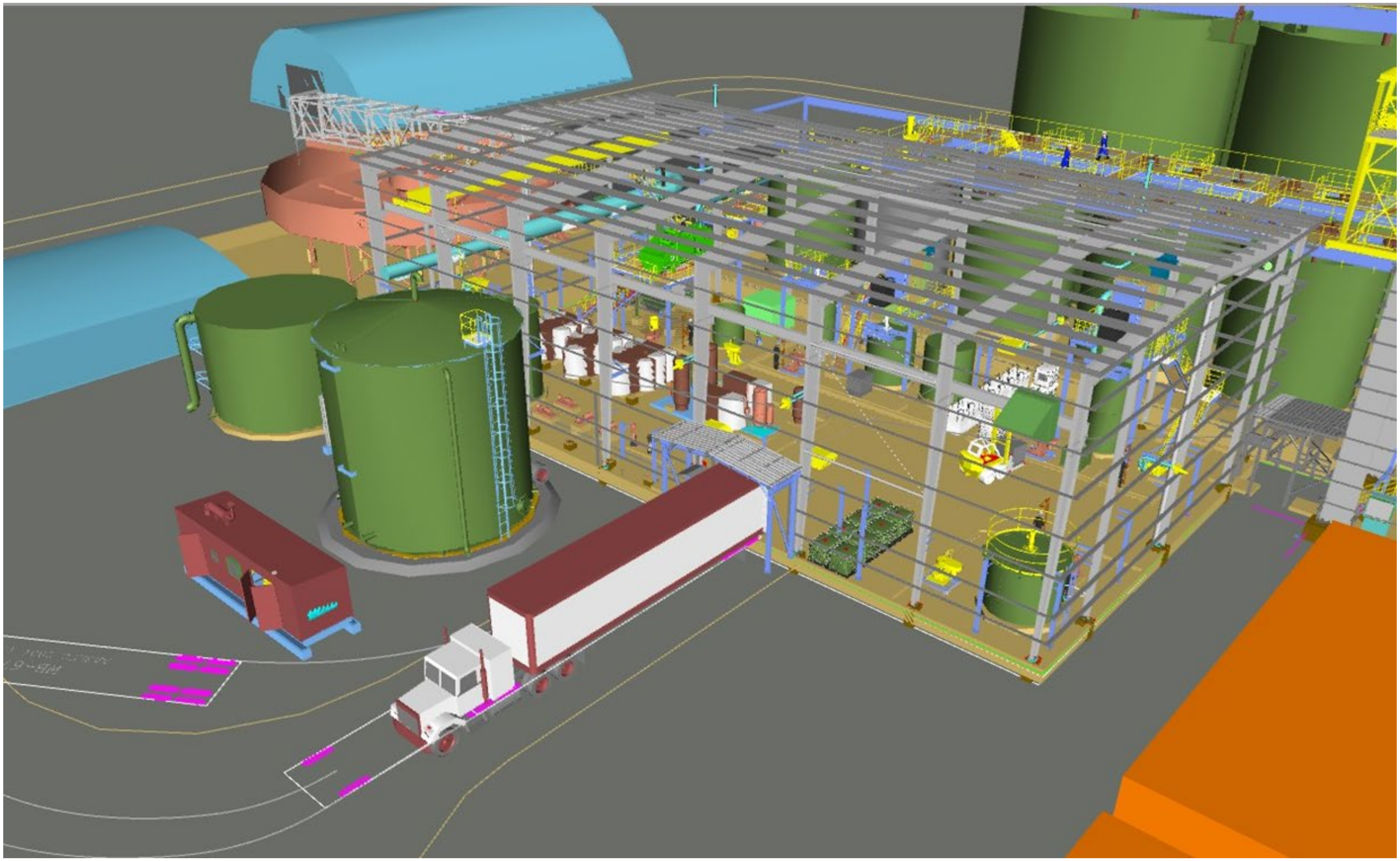
Source: Ausenco, 2021

*Figure 17-4: Stockpile area*



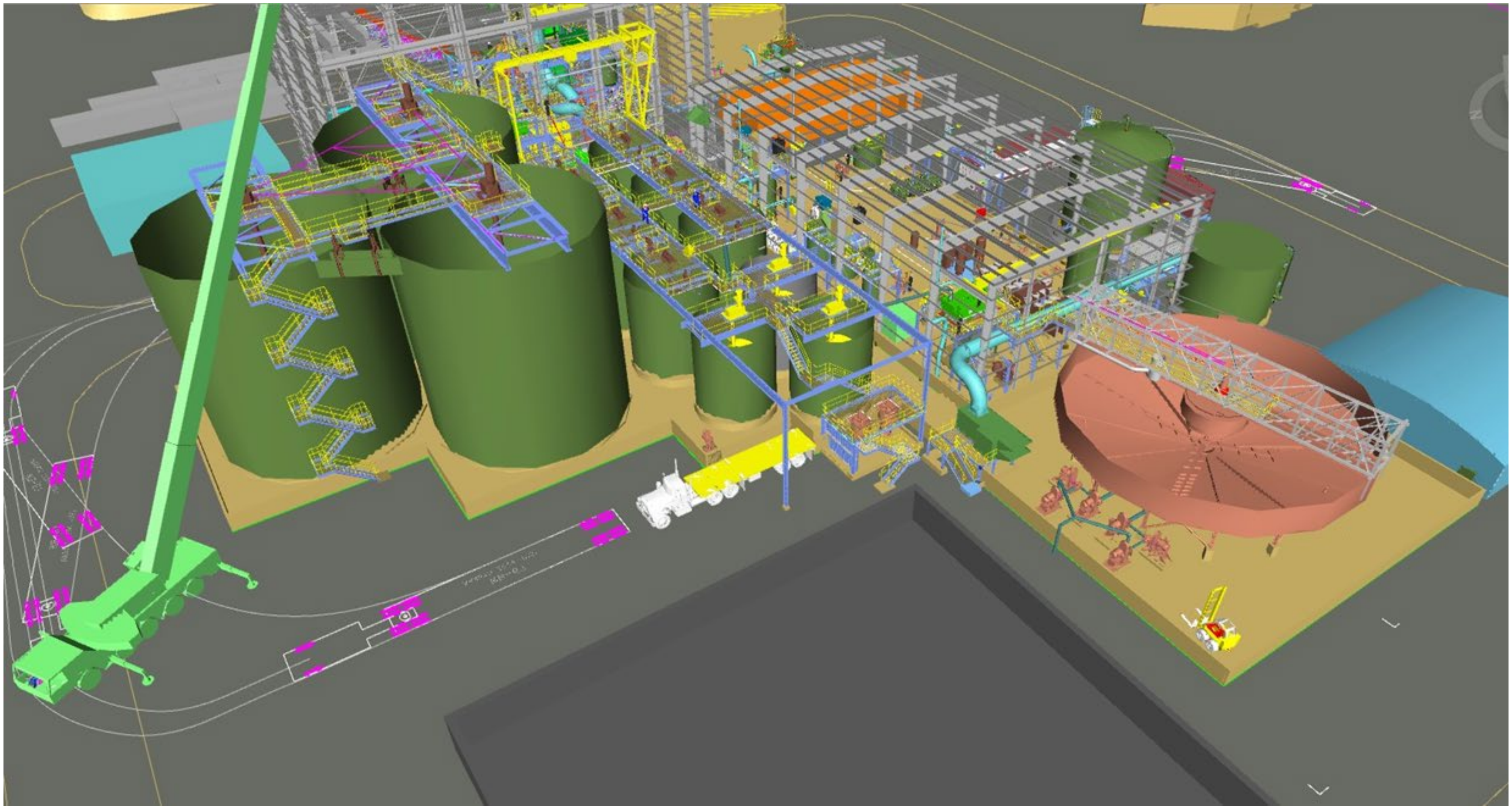
Source: Ausenco, 2021

*Figure 17-5: Grinding area*



Source: Ausenco, 2021

*Figure 17-6: Plant services and reagents area*



Source: Ausenco, 2021

*Figure 17-7: Leach, CIP, cyanide detoxification, arsenic precipitation and tailings thickening*

## 17.3 Reagent Handling and Storage

Each set of compatible reagent mixing, and storage systems are located within curbed containment areas to prevent incompatible reagents from mixing. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, and Material Safety Data Sheet (MSDS) stations are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

The following reagent systems are required for the process:

- Hydrated lime
- Sodium cyanide
- Hydrochloric acid
- Copper sulphate pentahydrate
- Sodium metabisulphite
- Sodium hydroxide
- Sulphamic acid
- Flocculant
- Activated carbon
- Smelting fluxes
- Ferric sulphate

### 17.3.1 Hydrated Lime

Hydrated lime is delivered in bags, which are lifted using a frame and hoist into the hydrated lime bag breaker on top of the mixing/storage tank. The solid reagent discharges into the tank and is slurried in process water to achieve the required dosing concentration of 25% solids (w/w). The slurried lime is pumped through a ring main with distribution points at leaching, and in cyanide destruction. An extraction fan is provided over the lime bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

### 17.3.2 Sodium Cyanide (NaCN)

Sodium cyanide is delivered to site in bags, which are lifted using a frame and hoist into the sodium cyanide bag breaker on top of the mixing/storage tank. The solid reagent discharges into the tank and is dissolved in freshwater to achieve the required dosing concentration of 20% sodium cyanide (w/v). After the mixing period is complete, cyanide solution gravitates to the cyanide storage tank. Sodium cyanide is delivered to the leach circuit and elution circuit with dedicated dosing pumps. An extraction fan is provided over the sodium cyanide bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

### 17.3.3 Copper Sulphate

Copper sulphate pentahydrate is delivered in solid crystal form in small bags and stored in the warehouse. Freshwater is added to the agitated copper sulphate mixing tank. A pallet of bags is lifted using a frame and hoist, and periodically a single bag is placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration of 20% copper sulphate (w/v).

Copper sulphate solution is transferred by gravity to the copper sulphate storage tank. Copper sulphate is delivered to the cyanide detoxification circuit using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

### 17.3.4 Sodium Metabisulphite (SMBS)

SMBS is delivered in the form of solid flakes in bulk bags and stored in the warehouse. Freshwater is added to the agitated SMBS mixing tank. Bags are lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required concentration of 20% SMBS (w/v). After the mixing period is complete, SMBS solution is transferred to the SMBS storage tank. SMBS is delivered to the cyanide detoxification circuit using the SMBS dosing pump. An extraction fan is provided over the SMBS mixing tank to remove $SO_2$ gas that may be generated during mixing. The SMBS mixing area is ventilated using the SMBS area roof fan.

### 17.3.5 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) is delivered in intermediate bulk containers (IBCs) as a 50% caustic solution (w/v) and stored adjacent to the elution circuit until required. During winter months, the reagent concentration will be adjusted prevent it from freezing in the IBCs. Dosing pumps automatically deliver the reagent to the required locations—elution circuit, electrowinning and cyanide mixing—to ensure the dosing requirements are met.

### 17.3.6 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in IBC as a solution and stored adjacent to the elution circuit until required. Hydrochloric acid is mixed with raw water (inline) to achieve the required 3% w/v concentration. Hydrochloric acid is delivered to the acid wash circuit using the hydrochloric acid dosing pump.

### 17.3.7 Ferric Sulphate ($Fe_2SO4_3$)

Ferric sulphate solution delivered in IBCs as a 60% solution (w/w). Ferric sulphate is delivered to the arsenic precipitation tank via dosing pump.

### 17.3.8 Flocculant

Powdered flocculant is delivered to site in bulk bags. A self-contained mixing and dosing system is installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powdered flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water.

Flocculant solution is diluted to a concentration of 0.50% flocculant (w/w) and agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant is transferred to the flocculant storage tank using the flocculant transfer pump. Flocculant is dosed to the tailings high-rate thickener using variable speed helical rotor style pump. Flocculant is further diluted just prior to the addition point via in-line mixer.

### 17.3.9 Activated Carbon

Activated carbon is delivered in solid granular form in bulk bags. When required, the fresh carbon is introduced to the carbon quench tank, or directly to the CIP tanks.

### 17.3.10 Antiscalant

Antiscalant is delivered as a solution in IBCs and stored until required. Antiscalant is dosed neat, without dilution. Positive displacement dosing pumps deliver the antiscalant to reduce the formation of scale in the elution and electrowinning circuits equipment as needed.

### 17.3.11 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and soda ash are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

### 17.3.12 Reagent Consumption and Storage

Reagent consumption rates are described in Table 17-2. Bulk storage requirements were calculated based on 6 weeks of supply for activated carbon, 2 weeks of supply for grinding media, and 10 days of supply for all other reagents.

**Table 17-2: Reagent Consumption and Storage**

| Reagent | Consumption | Consumption Rate | Packaging Unit | Stored Units |
|---|---|---|---|---|
| Sodium Cyanide | 1.1 | t/d | Bulk Bag | 12 |
| Hydrated Lime | 11 | t/d | Bulk Bag | 111 |
| Activated Carbon | 58 | t/y | Bulk Bag | 14 |
| Copper Sulphate | 0.26 | t/d | Bulk Bag | 3 |
| Flocculant | 0.2 | t/d | 25 kg bag | 74 |
| Hydrochloric Acid | 0.4 | $m^3$/day | Liquid, IBC | 4 |
| Sodium Hydroxide | 0.8 | $m^3$/day | Liquid, IBC | 6 |
| Sodium Metabisulphate | 6.6 | t/d | Bulk Bag | 67 |
| Ferric Sulphate | 1.9 | t/d | Liquid, IBC | 20 |
| Ball Mill Media | 3.0 | t/d | Bulk Bag | 18 |

## 17.4 Services and Utilities

The process plant will be serviced with compressed air which serves a number of functions throughout the plant. The major consumers of air are the leach and cyanide detoxification tanks as well as pneumatic instruments.

### 17.4.1 Plant / Instrument Air

High pressure air at 750 kPag is produced by compressors to meet plant requirements. The high pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

## 17.5 Water Supply

The water supply for the process plant is described below. A more detailed description of the site water balance is provided in Section 18.

### 17.5.1 Freshwater Supply System

Freshwater is supplied to a raw water storage tank. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- Gland water for pumps
- Reagent make-up
- Elution circuit make-up
- Freshwater is treated and stored in the potable water storage tank for use in safety showers and other similar applications
- Fire water for use in the sprinkler and hydrant system
- Cooling water for mill motors and mill lubrication systems (closed loop)

The process plant is expected to require 7 $m^3/h$ of freshwater with an additional 33 $m^3/h$ made up of either freshwater or from other influent water sources such as site runoff accumulated in the TSF.

### 17.5.2 Process Water Supply System

Overflow from the final tailings thickener are reused for process water requirements. Mine wastewater, mill contact water and freshwater provide any additional make-up water requirements.

### 17.5.3 Gland Water

One dedicated gland water pump is fed from the freshwater tank to supply gland water to all slurry pump applications in the main plant.

## 17.6 Power

Power consumption for the process plant was estimated from the electrical load list and derated by appropriate factors specific to each load to adjust the installed power to a nominal power demand. Additional details relating to power supply and site reticulation can be found in Section 18. A summary of the calculated power requirements is show below in Table 17-3.

**Table 17-3: Power Demand by WBS Area**

| WBS | Description | Installed Power (MW) | Maximum Demand (MW) | Nominal Demand (MW) |
|---|---|---|---|---|
| 3100 | Crushing, Stockpile, Reclaim | 1.6 | 1.0 | 0.7 |
| 3200 | Grinding | 4.5 | 3.6 | 3.2 |
| 3300 | Gravity and Intensive Leaching | 0.2 | 0.2 | 0.1 |
| 3400 | Leach – CIP | 0.8 | 0.6 | 0.5 |
| 3500 | Desorption, Regeneration, and Goldroom | 0.1 | 0.1 | 0.1 |
| 3600 | Cyanide Detoxification, Arsenic Precipitation, and Tailings Disposal | 0.4 | 0.4 | 0.3 |
| 3700 | Reagents | 0.1 | 0.1 | 0.0 |
| 3800 | Plant Services | 0.7 | 0.4 | 0.3 |
| 3900 | Process/Mill Buildings | 1.0 | 0.7 | 0.5 |
| **Total** | | **9.3** | **7.0** | **5.7** |

# 18. PROJECT INFRASTRUCTURE

## 18.1 Introduction

The main Project infrastructure components include mine and process plant supporting infrastructure, site accommodation facilities, TMF, external and internal access roads, power supply and distribution, freshwater supply and distribution, and water treatment plant. The infrastructure for the FS is situated within the locations shown in Figure 18-1. It should be noted that the figure shows the maximum extents of the pit and various stockpiles and not a specific phase of the project.

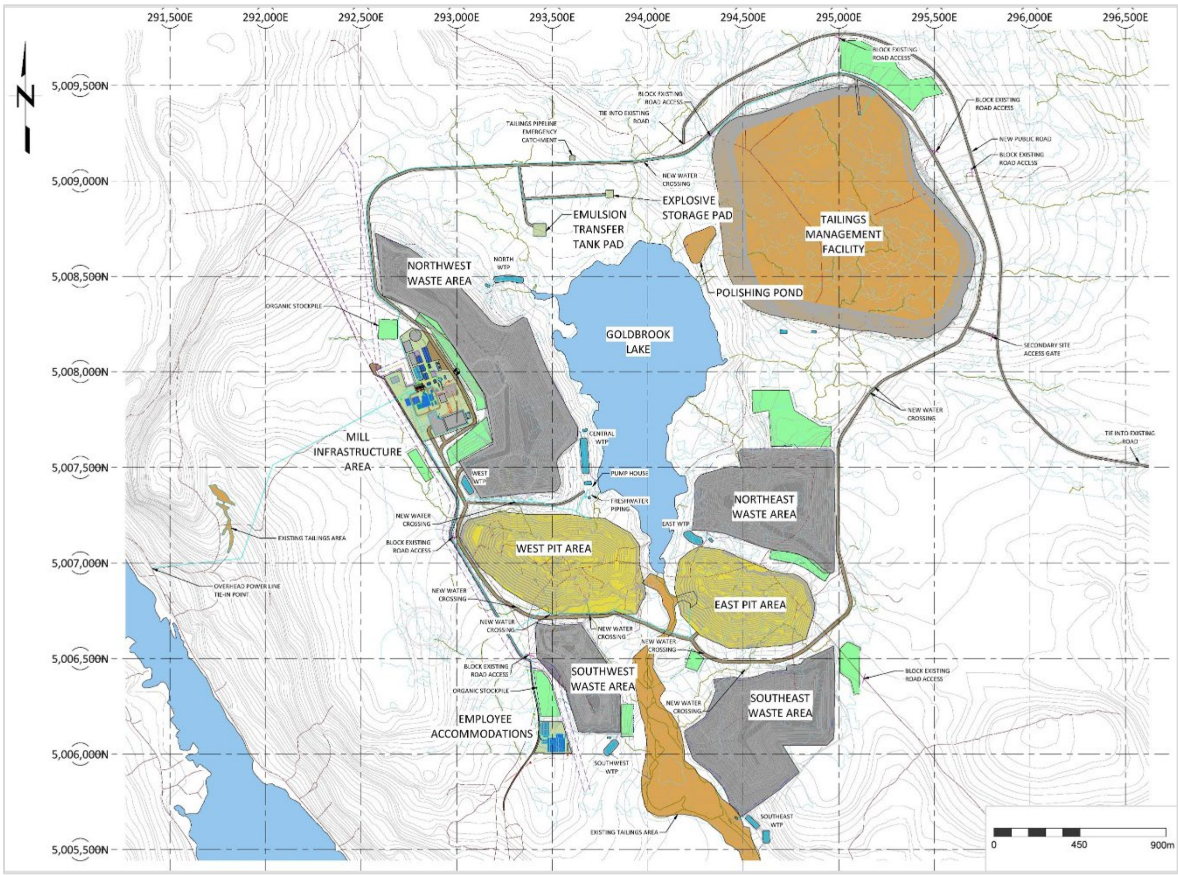



*Figure 18-1 Site conceptual general arrangement*

## 18.2 Project Logistics

The Property is situated on the eastern shore of Nova Scotia, Canada, with the central point of the Property being approximately located at 45° 12’ 2.6” N latitude and 61° 39’ 2.0” W longitude.

The Property will have access to the substantial infrastructure, services, and skilled labour in the area. There will be reduced infrastructure cost requirements due to its location near Route 316 compared to a remote mine site location. The Property is approximately 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro, on the eastern shore of Isaac’s Harbour, in Guysborough County, Nova Scotia, Canada. A secondary gravel road (Goldbrook Road), accessed from Route 316, crosses the Property, and passes near the historic Boston Richardson shaft and exploration decline. Smaller logging roads and trails provide good access to most areas of the Property. The elevation is nominally 70 m above sea level. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

## 18.3 On Site Infrastructure

The final locations of infrastructure at the mining site will be determined following further detailed geotechnical studies. At the current level of study, preliminary locations have been selected. An overall site plan is shown in Figure 18-2. These are the locations of buildings and other infrastructure based on information gathered from a site visit and geographic data including existing roads, water courses and wetlands.


*Figure 18-2: Infrastructure area*

When travelling to site via Goldbrook Road, personnel will report in at the security checkpoint located approximately 1.5 km from the Route 316 turn off. The security building will be a wood framed building and is located on the site employee accommodations pad. The employee accommodations will be modular building construction with a capacity of 350 and 175 beds during the construction and operations phases, respectively. The trailers will have a central kitchen/dining area and a portion will be converted to a recreation area after the construction phase is complete. The employee accommodations area will be equipped with several features to increase the safety for those staying on site. A key card system will be in place to access the kitchen and sleeping quarter facilities, guests will only be able to access the trailer for the room where they will be staying. CCTV will also be installed throughout the public areas such as kitchen/dining room and hallways. The employee accommodations layout is shown in Figure 18-3.

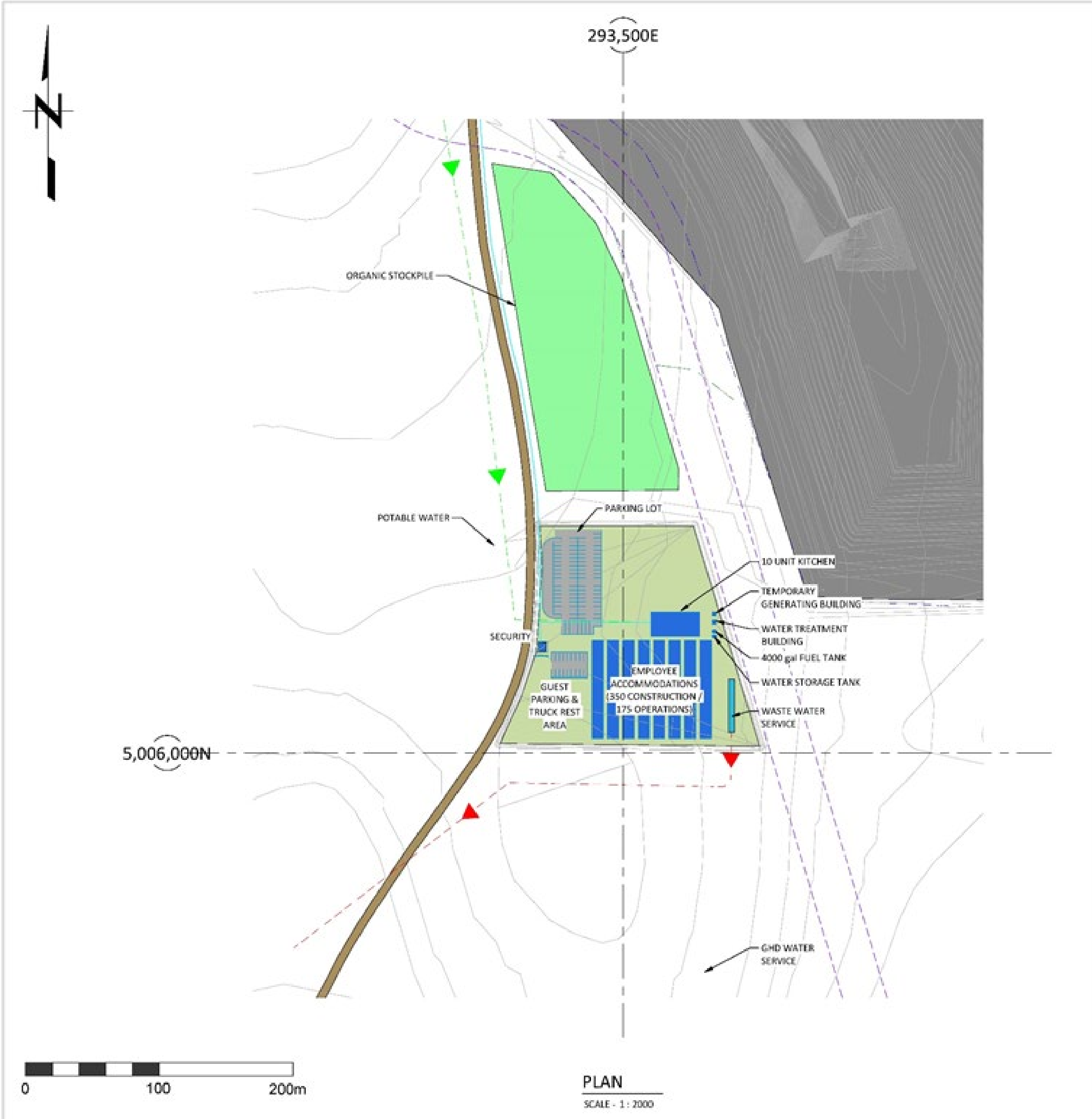



*Figure 18-3: Employee accommodations layout*

The primary infrastructure pad will house the process plant, run of mill stockpile (ROM), main switchyard and supporting ancillary buildings. The infrastructure pad will be outfitted with exterior lighting in high traffic areas such as the parking lots, exterior storage areas, laydown area, and employee accommodations to facilitate night work and safety. The general office building is a pre-engineered steel building and will provide office and cubicle space for 30 personnel including the mine managers, accounting, purchasing and HR personnel. The adjacent plant office will provide office space for ten of the mine technical personnel including, engineers, geologists, and surveyors.

The mine dry building is a pre-engineered steel building that contains a total of 175 lockers for the surface mine workers. The dry building includes gendered locker rooms, showers, and laundry rooms. A large meeting room has been provided for shift briefing prior to reporting to the various site workstations. The truck shop will be a pre-engineered steel building that houses three vehicle repair bays and is serviced by a 35-tonne bridge crane. Enough clearance has been provided to allow for the servicing of site haul trucks. Space for a welding fabrication area has also been provided for miscellaneous maintenance requirements. A bay for the truck wash was provided at the north end of the building with a containment wall for any runoff to be collected. Both truck shop and truck wash areas are equipped with a floor drain system complete with oil water separators. A second-floor mezzanine has been allowed for storage of materials or for expansion into future office space.

An ERT facility is located in the centre of infrastructure pad area. This building will house a two-vehicle garage for the site emergency response team and a medical examination room to treat any on site medical emergencies. A helipad on the north side of the site is provided in the case of a medical evacuation, or for helicopter arrivals to the site.

An 850 $m^2$ warehouse with storage racks has been provided for storage of spare parts, tool, equipment, and consumables. A total of 5,600 $m^2$ of laydown and exterior storage has been provided for rock cores and staging equipment and materials. An area to the north west of the site has been reserved for hazardous waste storage, this will be used to stage waste such as oil barrels, soil or materials contaminated with fuel and chemical containers before being removed from site.

## 18.4 Buildings and Facilities

The FS general mine and process surface facilities assumptions include the following:

- A truck shop / wash facility
- A process plant and laboratory
- Fuel storage facility
- Propane storage facility
- An explosive storage magazine
- Emulsion transfer tank
- Warehouse and laydown areas
- General office building
- Plant office building
- Mine dry building
- Core storage and a core yard
- ERT facility
- Helipad
- Hazardous waste storage
- Employee accommodations
- Water treatment facilities
- Main switchyard

In total, approximately 5,300 $m^2$ of ancillary buildings (not including the employee accommodations and process plant buildings) have been accounted for in the capital cost estimate.

The main operational and support buildings are located on a prepared granular pad to the northwest of the West Pit, outside of the 500 m buffer zone for blasting. The section of land gently slopes uphill to the west. The process plant, specifically the conveyor next to the ROM stockpile are closest to the open pit. These buildings are intended to be pre-engineered steel structures. Space has been provided for future buildings provided by the mining contractor or in the case of expansion during operations. Preliminary geotechnical recommendations indicate that the soil in this area is a compacted till with fines. At the time of this report geotechnical recommendations advise that the ancillary buildings may be founded on conventional spread footings placed at a depth great enough to avoid frost heave.

## 18.5 Existing Infrastructure

The only recoverable surface infrastructures on the site are Goldbrook Road (Figure 18-4), the access roads to the existing core shack and explosives storage (Figure 18-5), and the core shack itself. It was considered that all existing roads require partial clearing, minor granular refilling, culvert additions and/or repairs, and to be levelled with a grader. The existing core shack will be removed, as their current location is within the proposed open pit area.

Along Goldbrook Road is a natural gas pipeline, cathodic protection, and corresponding easement. The pipeline extends adjacent to the proposed infrastructure area with a pumping station at the end of the existing road. New infrastructure including earthworks must be constructed outside of this right of way.

*Figure 18-4: Goldbrook Road looking west showing end of powerline*

*Figure 18-5: Access road from Goldbrook Road to the core shack showing end of powerline*

## 18.6 Road Network and Access

The Property is accessible via Goldbrook Road, a gravel covered road approximately 2.5 km from Goldboro, Nova Scotia on Route 316, which is a provincial road along the southern shores of Nova Scotia (Figure 18-1).

Haulage roads on site will be built to withstand frequent heavy traffic between the proposed open pit, ROM stockpile and TMF. They will be wide enough to accommodate two trucks passing between the pits and ROM stockpile at 16.5 m with a grade no greater than 10%. The road to and from the tailing's management facility will be 11 m wide for one-way traffic by haul trucks.

Service roads other than those used by haulage trucks can be approximately 8 m wide and less resistant to heavy loads. The location of the new buildings and infrastructure areas were selected to maximize the use of the existing Goldbrook Road and other access roads on the mine site. The current layout of Goldbrook Road will intercept the proposed open pit and will therefore be realigned and offset at least 30 m from the open pit. Approximately 510 m of Goldbrook Road will require deforestation, grading, and granular refilling. An estimated 3,200 m of site roads will also be required in and around the mine site. In addition, an estimated of 5,500 m of public access roads will also be required as the existing access road is located in the proposed TMF location. Clearing work is also included for the existing road, just east of Gold Brook Lake, to locate the explosives storage buildings

## 18.7 Power Supply and Distribution

Power for the site is anticipated to be provided from a nearby Nova Scotia Power 25 kV distribution line installed along Route 316. A 1.6 km tap line would be installed along a new right of way to the mine site main substation. Nova Scotia Power would upgrade their existing distribution system as necessary to be able to provide the additional power required. Peak power demand for the site is estimated to be 10 MW, with the average demand estimated to be 7.5 MW. A network of 13.8 kV overhead distribution lines would be installed at site to provide power sourced from the main substation for the mine and surface infrastructure. Power supply and site distribution routes are shown in Figure 18-6.

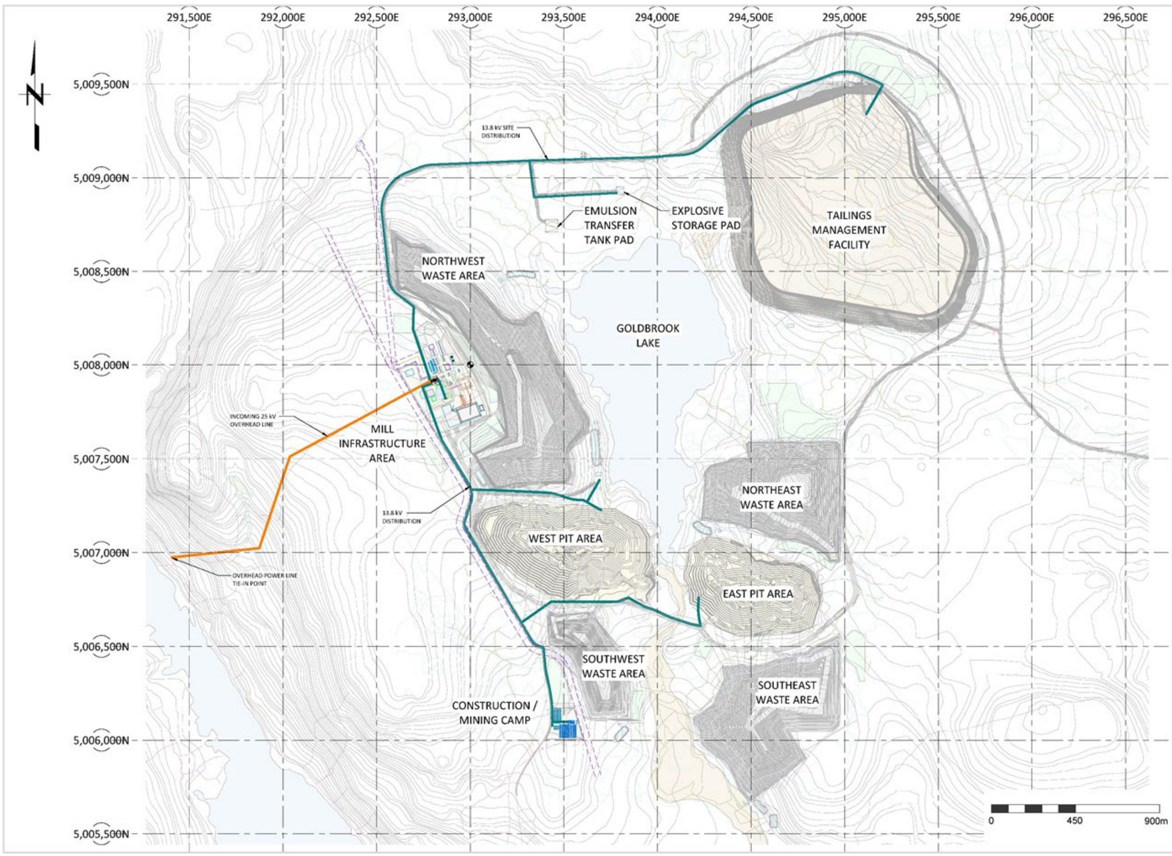



*Figure 18-6: Power site distribution*

## 18.8 Services and Utilities

### 18.8.1 Fuel

A cardlock diesel fuel station will be installed on site in order to refuel the haul trucks. The station will be located just off the main haul road, near the truck shop. The station will consist of two 100,000 L fuel storage tanks, each outfitted with high flow suction dispensers, fuel cardlock system, and environmental monitoring system. In addition to the leak detection instruments that make up the environmental monitoring system, the dispensing data can be used to reconcile fuel consumption with the delivery quantities and the fill level of the tank to ensure no fuel is unaccounted for. The station is planned to be located to the south east of the truck shop. The maximum safe capacity of the storage tanks will allow for 190,000 L of diesel fuel to be stored. This capacity represents the forecasted weekly consumption rate.

### 18.8.2 Propane Storage

A storage and distribution system for liquefied petroleum gas (LPG) will be required to provide heating fuel for the surface infrastructure and process buildings, as well as provide fuel to the process plant for the operation of the furnace and kiln in the gold room and carbon elution areas. An 18,000 USWG storage tank will be installed. The storage requirements were determined using the calculated heating loads of the surface infrastructure buildings, including the general office, plant office, mine dry, warehouse, truck shop, core storage, and emergency response transport facility. LPG will be distributed from the storage tank to the buildings via buried piping. Vaporizers will be located near the buildings to supply propane gas to the equipment.

### 18.8.3 Security

A security office and guardhouse are provided at the public entrance to the site near the employee accommodations area which will restrict the general public's access to the haul roads and pit areas. This structure will have offices for security guards and a security system for monitoring personnel coming to and leaving site. The security structure will be staffed continuously. Perimeter fencing will be built around the infrastructure area, explosive storage areas and other sensitive areas as required.

### 18.8.4 Communications

Cellular service is currently available at the site, as is Wi-Fi, but will need to be extended to the office and process plant area. UHF radio will be used in the pits and TMF, with a base station at the guardhouse.

## 18.9 Site Preparation and Earthworks

General site preparation includes deforestation, topsoil removal and storage, excavation, backfill material, grading, drainage ditches, and finishing surfaces to provide slopes and collect surface water. The general site work covers roads, ore and waste pads, truck shop, office buildings, warehouse, and other surface infrastructures, as well as the crushing area and process plant area. The site is forested, with small diameter trees and underbrush. Significant area of wetlands is also present throughout site. All organic material cleared will be stockpiled for mine close-out and reclamation efforts. Any pad material required can be quarried from the rock material in the open pits during the pre-stripping phase. The waste rock will be processed at a mobile screening station. An average depth of 0.6 m of topsoil has been estimated across the site by a geotechnical engineering professional.

The majority of the earthworks will be realized in the preparation of the mine infrastructure pad which will also contain the process plant, conveyors and ROM stockpile. Additional smaller pads will be built up for the employee accommodations and for the explosive storage magazine and emulsion tank.

## 18.10 Water Management

### 18.10.1 Water Conveyance and Distribution

Water supply infrastructure includes:

1. One intake structure;
2. Two booster stations;

3. One transmission watermain from Gold Brook Lake to the mill freshwater tank and to the potable water treatment unit; and,
4. Distribution piping to supply potable water throughout the Project site (mill, ERT facility, plant office, general office, mine dry, core storage, truck shop and employee accommodation).

A transmission watermain from Gold Brook Lake to the processing plant buildings is to provide a raw water source to support mill process operations and site wide potable water, hence the watermain flowrate was estimated based on the potable and process water demands (22 $m^3$/h).

### 18.10.2 Potable Water Treatment

The selection of the potable water treatment unit was completed in accordance with the Atlantic Canada Guidelines for the Supply, Treatment, Storage, Distribution and Operation of Drinking Water Supply Systems, 2004. In summary, Gold Brook Lake was considered as the source water, the treatment requirements were established based on the Canadian Drinking Water Guideline, and potable water treatment was sized assuming an equal flowrate for both potable water and wastewater (16 $m^3$/h).

### 18.10.3 Sanitary Waste Treatment/Disposal System

Two separate wastewater treatment units were developed to service employee accommodation (with 350 people) and other buildings/facilities including mill, ERT facility, plant office, general office, mine dry, core storage, truck shop (with 84 people). Sewage flow rates as well as treatment requirements were adopted from the Atlantic Canada Wastewater Guidelines Manual for Collection, Treatment and Disposal, 2006. Similar to the water treatment system, costing was provided for two containerized sewage wastewater treatment units (peak design flowrate of 14 $m^3$/h for employee accommodation and 2 $m^3$/h for other buildings/facilities).

### 18.10.4 Contact Water Treatment/Disposal System

The MWMP and associated design measures have been developed based on the proposed feasibility level mine site arrangement with inputs from the Company and the Consultants. The MWMP will be implemented during the construction phase and will be adjusted as necessary throughout the mine operations and closure phase.

Site contact water will be managed to meet the following regulatory discharge requirements prior to discharge to the natural environment:

- MDMER Objectives
- CCME Canadian Water Quality Guidelines for the Protection of Aquatic Life
- Tier 1 Nova Scotia EQS for Surface Water
- Site specific criteria (based on background data)

Based on predictive water quality modelling, it is understood that the water quality at some locations may be acceptable for discharge to the environment with TSS removal as the only form of treatment (where MDMER objectives are met). Where this is not the case, contingency measures (shutoff valves and pumps etc.) will be put in place to redirect water towards the nearest WTS in case of exceedances. The primary objectives of the MWMP are as follows:

- Provide mechanism to dewater and treat ponded water within the Project Area to allow for development and excavation of mine infrastructure (e.g., pit, waste piles, haul road etc.).

- Capture, treat and provide controlled discharge for all site contact water during construction and operations.
- Divert all off site clean water away from the mine site infrastructure to reduce the total volume of water entering the settling ponds for treatment.

#### 18.10.4.1 Design Basis Criteria

The criteria used for the design of the water management infrastructure are based on the feasibility level design site arrangement, regulatory discharge water quality requirements, operational requirements, and environmental site conditions. The design basis criteria for storm events utilized for the MWMP design are summarized in Table 18-1.

**Table 18-1: Water Management Design Basis Criteria Summary**

| Item | Design Basis |
|---|---|
| Contact Water Ditches and Culverts | • Designed to convey stormwater runoff resulting from the 1 in 100 year, 24-hour, climate change adjusted storm event (153 mm) |
| Non-contact Water Ditches and Culverts | • Culverts were designed to convey stormwater runoff resulting from the 1 in 5 year storm event with durations ranging from 20 minutes to 1.2 hours depending on time of concentration in contributing sub-watershed (15 mm to 27 mm)<br>• TMF embankment clean water ditches were designed to convey stormwater runoff resulting from the 1 in 100 year, 24-hour, climate change adjusted storm event (153 mm). |
| Settling Ponds | • Designed to detain runoff resulting from the 25 mm 4-hour storm event, 1 in 10-year 24-hour climate change adjusted storm event (112 mm) and 1 in 100-year 24-hour climate change adjusted storm event (153 mm) for a minimum of 24 hours to allow for TSS removal due to settling<br>• Designed to convey runoff resulting from the 1 in 100-year 24-hour climate change adjusted storm event (153 mm) to Gold Brook Lake (north) or engineered wetland (south) via a concrete outlet structure<br>• Designed to convey runoff from Hurricane Beth, modelled as a 48-hour storm event (296 mm), through an emergency spillway<br>• All discharge water from collection ponds must meet MDMER water quality requirements<br>• All discharge water must meet CCME, Tier 1 NSE WQS or site-specific requirements within the 100 m mixing zone of the natural watercourse receiver (Gold Brook Lake or Gold Brook) |

The Project was modelled using PCSWMM (Version 7.3.3095) which is a hydrologic and hydraulic modelling software that uses the EPA SWMM (Version 5.1.015) engine. PCSWMM was used to develop the design storm hydrographs, estimate peak flow rates and runoff volumes for the design storm events.

To add contingency to the design, the impacts due to climate change were considered when developing the design storm hydrographs. To account for the project duration (operation life of <20 years) and nearest climate region (Guysborough), a 5% increase in the total rainfall depth was applied to each design storm based on the requirements outlined in Climate Data Nova Scotia as the projected increase in annual precipitation for 2040[4].

The estimated runoff volumes resulting from the 24-hour, 1 in 100-year climate change adjusted storm event were used to determine the storage capacity of the settling ponds. The 25 mm 4-hour, 10-year and 100-year climate change adjusted storm events were used to design the settling pond outlet structures such that a minimum 24-hour detention time was achieved in each settling pond to allow for settling of TSS. Hurricane Beth (as recorded by Halifax International Airport Climate Station (Climate ID: 8202251) was used to size the emergency spillway of the settling ponds. The 100-year climate change adjusted storm event was used to size the ditches, culverts and other conveyance pathways leading to the settling ponds. Additional design details for the various water management design elements are provided in the following sections.

#### 18.10.4.2 Collection Ditches and Culverts

The MWMP consists of a series of surface water ditches and culverts collecting all Project stormwater runoff. The surface water ditches include contact water ditches, which collect runoff from all mine infrastructure, and clean water ditches. Clean water will be collected from areas that do not come in contact with mine waste such as runoff from the outside TMF embankment. The surface water clean water ditches collect storm water runoff from the TMF embankment, which is constructed from NPAG rockfill, and direct it away from the site. Culverts are dispersed throughout the site to convey stormwater below mine infrastructure (i.e., haul roads). The contact water ditches drain to one of five settling ponds located across the site.

Each ditch will be trapezoidal in section with 3H:1V side slopes and bottom widths and depths ranging from 0.3 m to 1.2 m. Ditch slopes range between 0.3% and 7.5% depending on the location across the site. Ditches will be excavated into the existing overburden and/or bedrock or formed by grading existing surface material to form the required channel cross-section. All excess material used to grade the channel to the required cross-section will be sloped to existing ground at a 3H:1V slope. For clean water ditches, the exposed slopes will be seeded upon reaching finished grade to prevent erosion.

Contact water ditches will be lined with a HDPE liner, underlain by geotextile, followed by a layer of sand and a layer of riprap to prevent infiltration of stormwater into the surficial groundwater and protect the ditch from erosion. The riprap layer in the liner system will be sized appropriately to prevent erosion during the 1 in 100-year 24-hour climate change adjusted storm event. Detailed riprap requirements will be determined during later design stages. Rock check dams will be put in place on ditches that have a slope of greater than 3% in addition to the riprap layer to prevent erosion. Rock check dams reduce the overall slope of the water surface, reducing the potential for erosion. Rock check dams also allow time for suspended sediment to settle out prior to reaching the settling pond. The ditches leaving the settling ponds will contain clean water following TSS and arsenic removal and any additional required water treatment via the WTS in the case of the northeast settling pond and northwest settling pond 2. The outlet of the effluent ditch into the receiving

[4] https://climatechange.novascotia.ca/climate-data?tid=8#climate-data-map

watercourse will be lined with an HDPE liner followed by a layer of sand and a layer of riprap to prevent erosion. Detailed outlet design will be determined during later design stages.

Culverts are to be circular corrugated steel pipe (CSP) culverts with diameters ranging from 450 mm to 1200 mm and lengths between 11.5 m and 25 m. Culvert slopes range between 0.6% and 2.9% across the site. Each culvert will include a riprap apron on the upstream and downstream sides of the culvert to prevent erosion around the inlet and outlet. The outlet riprap aprons are designed to include an energy dissipation basin. The energy dissipater reduces velocities in the downstream ditch, reducing the potential for erosion. The energy dissipation basin is to be lined with riprap specifically sized to withstand culvert exit velocities and reduce flow velocity downstream of the culvert.

#### 18.10.4.3 Settling Ponds

Settling ponds will be constructed to collect and treat contact water prior to discharging to Gold Brook Lake (north) and Gold Brook (south). Settling ponds are included for runoff from the northwest waste rock storage area, northeast waste rock storage area, southeast waste rock storage area, till, organics and stockpiles and process plant area. The ponds were designed to maintain a 0.3 m freeboard during the 1 in 100-year 24-hour climate change adjusted design storm event. All ponds were also designed with an emergency overflow spillway sized to convey Hurricane Beth sized storm event.

The ponds will be lined with an HDPE liner, underlain by geotextile. A 0.3 m layer of sand will be placed on top of the HDPE liner to protect against punctures and a 0.3 m layer of riprap on top of the sand layer. The riprap is to act as ballast to prevent the liner from being impacted by buoyancy forces of the nearby groundwater as well as provide erosion protection.

The ponds will be trapezoidal in cross-section with 3H:1V side slopes. The maximum depth in the settling ponds varies between 3.5 m to 4.25 m, including a permanent pool depth of 1 m. The surge pond has a maximum depth of 5 m but will be dry during normal operating conditions. The pond lengths vary from approximately 98 to 200 m, and widths vary from 40 to 51 m. The northwest surge pond and northwest settling pond 2 will be classified as dams due to the embankment berms exceeding the 2.5 m threshold.

The settling ponds (except for the northeast settling pond) each consist of a concrete outlet structure and emergency overflow spillway. The concrete outlet structure has been designed to control storm events up to and including the 1 in 100-year 24-hour climate change adjusted storm event through a series of orifices – to achieve a minimum detention time of 24-hours for TSS settling. The concrete outlet structure will be surrounded by a layer of riprap to reduce exit velocities and further assist with TSS settling. The emergency overflow channel will convey flows resulting from storm events greater than the 1 in 100 year, 24-hour climate change adjusted design storm event, up to and including Hurricane Beth. Northwest settling pond 2 and the northeast settling pond will direct the emergency overflow spillway toward the west and east open pits respectively to reduce the risk of uncontrolled discharge from the site. The surge pond, which remains dry during normal operating procedures, will have a 300 mm CSP culvert at the outlet to control flow and allow the pond to act as surge capacity during design storm events, reducing the peak inflow to northwest settling pond 2.

Effluent from the northeast pond will be pumped to the northwest settling pond 2. Effluent from northwest settling pond 2 will pass through the WTS prior to discharge into Gold Brook Lake. Effluent from northwest settling pond 1 will discharge into Gold Brook Lake. Effluent from the southeast settling pond will pass through an engineered wetland to reduce the nitrate and nitrite concentrations, prior to discharge into Gold Brook. Effluent from the southwest pond will discharge directly into Gold Brook. All settling ponds will discharge effluent at concentrations below the federal MDMER regulations as per the Fisheries Act.

#### 18.10.4.4 Water Balance

A mine water balance was conducted to simulate water transfers between site features and the receiving environment through operations, active closure and post-closure until the pit lakes are formed. The primary objective of the water balance was to estimate inflow rates from discrete sources to the settling ponds and natural assessment points to inform the predictive water quality analysis and determine the level of treatment required at the proposed site discharge locations. The mine water balance was also used to estimate time to fill the east and west pits to assess the transition time between active and post closure.

The mine water balance modelling was completed using the GoldSim (Version 12.1.5) software. Simulations were performed at a monthly time step over the duration of the mine life and baseline conditions. The model inputs include catchment areas, mine infrastructure footprints and pit volumes. Climatic and hydrologic inputs include yield (rainfall, snowmelt), evapotranspiration, lake evaporation, direct runoff coefficients and seepage coefficients to estimate runoff volumes to the various mine water management features and assessment points. Water withdrawals (water demand from the mill, employee accommodation, administrative areas) and inflow volumes calculated outside of the model (TMF discharge, groundwater inflows to the pit) were also included in the model. The mine water balance and groundwater models were coordinated to ensure the assumptions and outputs were consistent and compatible. The modelling considered average climate conditions over the mine life and baseline stages.

The mine water balance model was ultimately used to estimate the volume of contact water from stockpile seepage, pit wall runoff and TMF discharge, and volume of clean runoff to assess dilution potential at the natural assessment points in the predictive water quality modelling.

### 18.10.5 Erosion and Sediment Control Measures

Erosion control measures in the contact water ditches and settling ponds are to be maintained during operations including replacement of riprap, restoration of check dams if damaged and general visual inspection of the ditches and settling ponds. Experience at other mine sites in Nova Scotia indicates that significant sediment build up could occur in the collection ditches. The contact water ditches should be inspected regularly and cleaned out as needed to ensure sediment does not build up within the ditches or travel directly into the settling pond, reducing the available storage volume of the settling pond itself.

Riprap check dams are to be placed in ditches with slopes greater than 3%, and straw bale and filter sock check dams are to be placed in ditches with a slope of 0.5% to 3% as needed. Runoff collection ditches that are not lined with HDPE are to be hydroseeded, including ditches in the administrative area, surrounding the topsoil and organics stockpiles (where not containing contact water), and at the toe of the TMF berm.

### 18.10.6 Water Treatment

#### 18.10.6.1 Tailing Management Facility Water Treatment System

Arsenic, cobalt, iron, cyanide, ammonia, and nitrite are among exceeded elements that need treatment in TMF discharge water. The water treatment system for TMF will include a metal precipitation step using caustic followed by coagulation and flocculation. Increasing pH by adding caustic will precipitate metals in their hydroxide forms. The undissolved hydroxide particles then will be settled out by adding coagulant and polymer. A series of well-mixing tanks will mix injected chemicals and will lead to flocculant formation. The formed flocculants then will be settled out in a clarification unit. The supernatant will be passed to the next treatment step and the sludge from the bottom will be pumped back into the TMF. A cyanide destruction unit will provide enough retention time and mixing to let the injected chemicals react with cyanide species and allow for their removal. Furthermore, a moving bed bioreactor (MBBR) unit will remove residual cyanide, as well as ammonia and nitrite. The effluent of the MBBR will then be discharged into the final polishing pond before discharging into the environment.

#### 18.10.6.2 Waste-rock Storage Area Runoffs Water Treatment System

Modeling results show that metals such as arsenic, cobalt, copper, and iron will exceed discharge limits for the waste-rock stockpile runoffs. Nitrogen containing species such as nitrite, nitrate and ammonia also could exceed discharge limits for the waste-rock stockpile runoffs. These species are the result of blasting material used during mining activities.

The WTS for waste rock storage area runoffs includes a hydroxide precipitation using caustic or lime to increase the pH in order to precipitate exceeded metals in their insoluble hydroxide form. Coagulation is the second step, where a coagulant is added to accelerate precipitation of formed fine particles. Then the chemically dosed impacted water will pass through a mixing unit and will discharge into the settling pond. The suspended solids will then be settled within the settling pond and the effluent of the settling pond will pass through an engineered wetland for final polishing before discharging to the environment.

A flume will measure the flow of influent water ahead of the settling pond and will adjust the chemical dosing rates accordingly.

## 18.11 Plant Infrastructure

Process plant infrastructure is described in Table 18-2. The main buildings are detailed further in Section 18.11.1 and Section 18.11.2.

**Table 18-2: Process Plant Infrastructure**

| Description | Building Construction | L (m) | W (m) | H (m) | Area ($m^2$) |
|---|---|---|---|---|---|
| Stockpile Cover | Inflatable Tube Structure | 47 m Dia | | 24 | 2,290 |
| Mill Building | Pre-engineered | 38 | 26 | 26 | 975 |
| Main Reagents Building | Fabric | 40 | 24 | 13 | 957 |
| Reagents Storage Warehouse | Fabric | 18 | 18 | 5 | 334 |
| Goldroom | Pre-engineered | 16 | 11 | 10 | 184 |
| Mill Office | Modular | 18 | 12 | 5 | 216 |
| Mill Workshop | Fabric | 35 | 24 | 9 | 855 |
| Assay Laboratory | Containerized | (6) x 40-foot sea containers (2.4 m x 2.4 m x 12 m each) | | | |
| Control Room | Modular | 6 | 3 | 3.5 | 18 |

### 18.11.1 Process Plant Buildings

The process plant complex is comprised of the following separate buildings:

- Mill building (grinding and gravity).
- Main reagents building.
- Gold room.

The buildings will be supported on reinforced concrete footings and are complete with concrete slab on grade. To account for winter conditions, building roof and wall cladding will have fibreglass blanket insulation complete with vapour barrier. Overhead cranes will be available for equipment servicing in the process plant. Building heating for all buildings will be from propane fuel.

The mill building will be a pre-engineered building with a 40 tonne overheard crane includes a ground floor and one major mill operating floor and multiple equipment access platforms. The various equipment will be accessed by purpose-built mezzanine platforms for maintenance, service and sampling. The mill building will contain the ball mill, cyclone feed hopper/pumps, cyclone cluster and trash screen, as well as dedicated areas for the gravity circuit equipment, acid wash column, the elution column and regeneration equipment.

The reagent building will be a fabric building, with a 5 tonne overhead crane, and will contain the reagent mixing tanks, and dosing tanks (where applicable). The reagent profile consists of cyanide, lime, sodium hydroxide, hydrochloric acid, carbon, copper sulphate, sodium metabisulphite, ferric sulphate, flocculant, and antiscalant. Where possible totes of reagents will be used directly, to conserve space and tankage.

The gold room pre-engineered building will house the pregnant solution tank, electrowinning cells, sludge filters, furnace, drying oven and vault.

Exterior process facilities include:

- primary crusher modular structure.
- secondary and tertiary crushing modular structures.
- three leach tanks, each of which is 15.3 m in diameter.
- six carbon-in-pulp tanks, each of which is 7.2 m in diameter.
- two detoxification tanks that are 7.8 m in diameter.
- arsenic precipitation tank that is 6.5 m in diameter.

The tanks will be accessed by purpose-built mezzanine platforms and walkways to allow servicing, sampling and maintenance. An area crane will provide access to screens, tanks, pumps and agitators for the carbon-in-pulp, detoxification and arsenic precipitation tanks; removal/reinstallation of the leach tank agitators will be accomplished via a mobile crane. The tailings will report to an 18 m diameter tailings thickener located outdoors, before being pumped to the tailings storage facility.

### 18.11.2 Support Buildings

An inflatable tube structure will be used to cover the mill feed stockpile.

The mill office facility will be 18 m (long) by 12 m (wide) modular building and include a lunchroom, washrooms, men's and women's dry, lockers, first-aid and showers. The building will be equipped with heating, ventilation and air conditioning (HVAC), and will be located in close proximity to the mill building.

The mill workshop will be 24 m wide by 35 m long and will provide space for general maintenance and servicing of small equipment for the process plant, such as pumps, motors and mobile equipment. The building will be equipped with HVAC and will be located in close proximity to the process plant.

The laboratory will be containerized building on pre-cast concrete blocks, consisting of 6 x 40 foot (2.4 x 2.4 x 12 m long) containers. The laboratory will house equipment for typical mine and plant assays as well as office space. The laboratory will be located in close proximity to the process plant and administration buildings.

The reagents storage warehouse will be a fabric building, 18 m by 18 m, and will provide space for short-term storage of reagents for use in the process plant; it will be located in close proximity to the main reagents building.

### 18.11.3 Fire Systems

The process plant buildings and structures will be equipped with fire detection and protection systems generally consisting of portable fire extinguishers, smoke detectors, manual pull stations, alarm strobes/sounders and hose stations. Additionally, automated fire sprinkler systems will be provided around the ball mill hydraulic systems and stockpile reclaim conveyor.

The process plant area will have a fire water reticulation system (combined with the mine infrastructure area) consisting of fire water storage, a fire water pump station (including backup diesel-powered pump), and fire water piping (size DN250) around the perimeter of the mill/reagent/leach area, with fire water reticulation also extended to the primary, secondary and tertiary crushing and reclaim stockpile areas. Fire suppression, in the event of a fire, will be implemented via fire hydrants located around the above areas.

## 18.12 Tailings Management Facility

### 18.12.1 Overview

The TMF will be constructed as a paddock style, single cell facility located on a side hill northeast of Gold Brook Lake as shown in Figure 18-7. Co-disposal will include management of both 16.2 million tonnes of PAG tailings and 10.5 million tonnes PAG1 waste rock in the TMF. Tailings and PAG1 waste rock would be transported to the TMF independently and placed in separate locations within the fully lined TMF basin (i.e., co-placement). Following placement, the PAG1 waste rock will become inundated with ongoing thickened tailings slurry deposition and tailings supernatant water. This will maintain both the tailings and the PAG1 waste rock below a water cover and in a saturated state. Maintaining these PAG waste materials below the long-term phreatic surface within the TMF will prevent the onset of acid rock drainage (ARD) conditions and help reduce metal leaching (ML) from the PAG1 material.

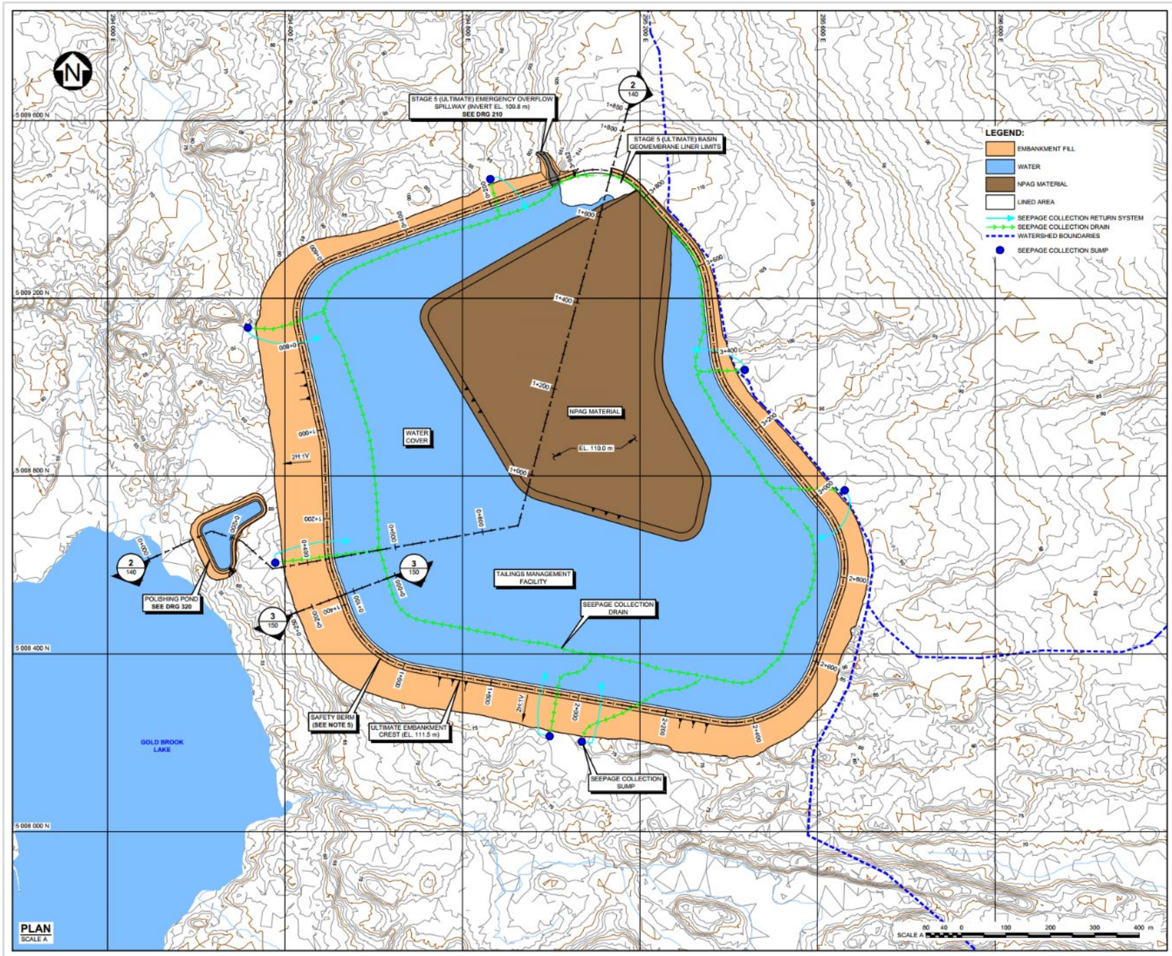



*Figure 18-7: TMF – general arrangement – ultimate facility*

## 18.12.2 Predictive Water Quality

A predictive water quality (PWQ) model for the TMF to support feasibility level design of the water management systems for the Project (by others) was completed. The objective was to establish a preliminary understanding of the estimated water quality of the monthly effluent discharged from the TMF to the TMF WTP over the life of the mine. Deterministic mass balance models were developed to predict the quality of the effluent during operations and closure (active and post-closure). Source terms were provided by Lorax for the component inputs for the model (i.e., PAG1 waste rock, NPAG waste rock, tailings process water, and topsoil stockpile material). The predicted water quality results were used to design the TMF WTP. The modelling suggests that natural cyanide degradation is not effective within the TMF. Cyanide destruction should be conducted within the plant to reduce the potential loadings that are discharged to the TMF WTP. The modelling indicates that the water quality inputs from the tailings slurry and the PAG1 waste rock are depleted during active closure and are fully diluted by Month 24. Post closure discharge loadings are solely influenced by the source terms from the topsoil cover and minimal infiltration through the NPAG waste rock beneath the cover.

### 18.12.3 TMF Embankments and Lining System

The TMF embankments will be constructed using NPAG waste rock from open pit mining operations. The embankment will be constructed of zoned earthfill and rockfill with a geosynthetic lining system installed along the TMF basin floor and on the upstream face of the perimeter embankments to minimize seepage exiting the facility. Transition/filter zones will be established between the liner and the embankment rockfill to ensure internal stability. The downstream slopes will be 2.5H:1V. The upstream slopes will be 2.5H:1V with 3 m of the previous stage's crest left as a bench when the next stage is constructed, to facilitate and tie-in the geomembrane liner, resulting in an overall slope for the ultimate embankment of 2.8H:1V. The crest width will be 15 m and the maximum embankment height above original ground is approximately 49 m. A typical cross-section for the TMF embankment is shown on Figure 18-8.

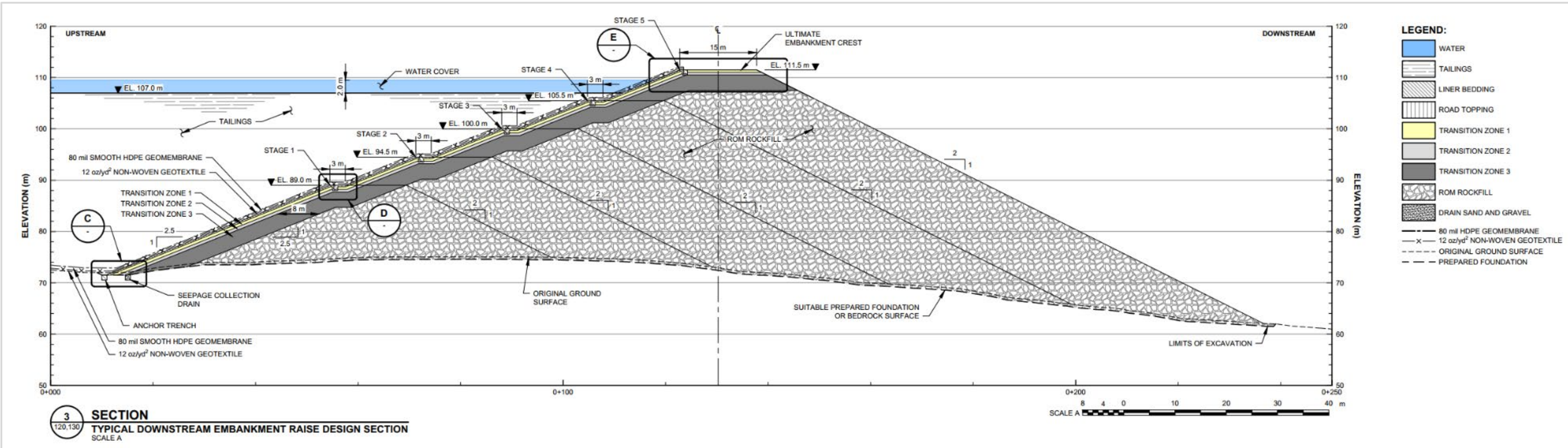


Figure 18-8: TMF – section

The lining system for the tailings storage portion consists of a smooth 80 mil HDPE geomembrane overlying a placed and compacted till liner bedding layer on the upstream face of the embankments; and would be placed overlying a 12 oz./sq. yd. non-woven geotextile within the basin. The lining system is to be placed on a prepared subgrade.

The lining system for the PAG1 waste rock portion of the facility consists of (from bottom to top) a 12 oz./sq. yd. non-woven geotextile, textured (both sides) 80 mil HDPE geomembrane and a geocomposite cushion layer. The initial lifts of PAG1 waste rock will be selectively processed and placed to further protect the underlying lining system under the PAG1 waste rock portion (i.e., consisting of 0.3 m thick layer of 50 mm minus material followed by control placement of 1 m thick layer of material).

The TMF embankments have been classified based on the FS arrangements and industry accepted guidelines published by the Canadian Dam Association (Canadian Dam Association (CDA), 2019; Canadian Dam Association (CDA), 2013). The TMF is classified as having a Dam Classification of Extreme. The Earthquake Design Ground Motion (EDGM) and IDF thresholds used for the design of the facility were selected based on these classifications.

The TMF and PAG1 waste rock is required to be stable under the design loading conditions. The stability of the embankment and PAG1 waste rock was evaluated considering loading conditions and minimum target FoS values recommended for mining dams (CDA, 2019). The FoS targets are met or exceeded for all sections and loading conditions evaluated.

The potential leakage through the geomembrane lining system was estimated with the operating pond and tailings at the maximum level for Stage 1 and the Ultimate stage. The seepage analyses considered leakage due to the presence of geomembrane defects. Seepage was estimated using the method of Bonaparte and Giroud (Bonaparte, & Giroud,, 1989) and Giroud (Giroud, 1997). The seepage rates are within a range that is manageable with the seepage collection system.

### 18.12.4 Staged Construction and Filling Schedule

The TMF design includes an initial starter embankment (Stage 1) followed by subsequent stages. Stages 2 through 5 of the TMF will be expanded using downstream construction methods throughout the life of the facility. Staged development of the TMF offers the following advantages:

- Reduction of initial capital expenditures
- Refining of design and construction methods as experience is gained with local conditions and/or as operating criteria change
- Adjustment of plans at a future date in order to remain current with "state-of-the-art" engineering and environmental practices, etc.

This staged approach will be used for the future design, construction, and operation of the facility as part of a continuous and integrated process to identify cost savings and enhance safety. The approach requires construction controls, monitoring, and review to improve the understanding of site specific conditions.

Embankment construction will be scheduled to provide sufficient storage capacity and freeboard in the TMF, including to temporarily store runoff resulting from the EDF and safely convey runoff resulting from the IDF. The design basis and operating criteria are based upon Canadian Dam Association Dam Safety Guidelines (Canadian Dam Association (CDA), 2013) (Canadian Dam Association (CDA), 2019) and site-specific design considerations.

The filling schedule for the TMF has been developed based on the basin characteristics and a projected settled dry density of 1.35 t/m$^3$. The filling and embankment staging plan for the TMF is provided on Figure 18-9.

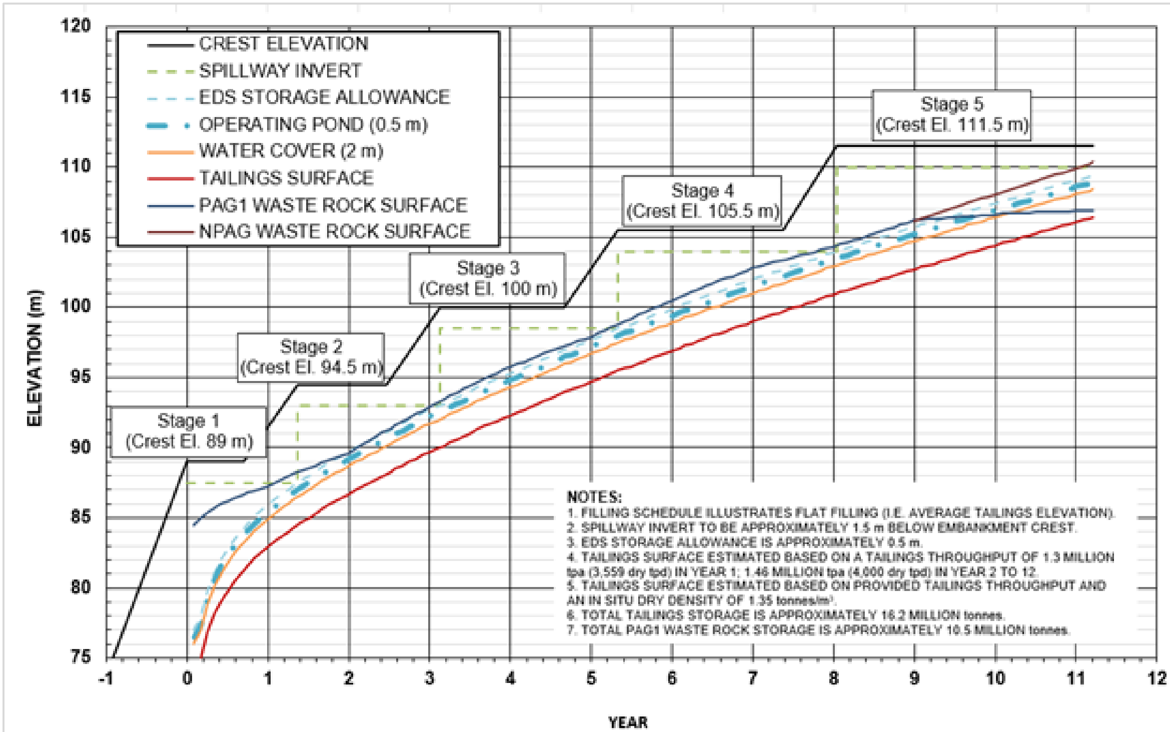



*Figure 18-9: TMF – filling schedule*

## 18.12.5 Tailings and PAG1 Waste Rock Management

Tailings will be pumped as a conventional thickened tailings slurry (typically 60% solids (w/w) content by weight) from the process plant to the TMF via pipelines. Tailings will be deposited sub-aqueously (below the water surface) from the upstream face of the TMF embankments at multiple locations around the perimeter of the TMF basin and from the co-placed PAG1 waste rock in the central portion of the TMF. This deposition strategy will develop a low permeability tailings deposit adjacent to these embankments. The tailings deposition strategy will allow for even filling of the basin to maintain a water cover over the tailings and maximize tailings storage within the impoundment.

The PAG1 waste rock will be segregated during mining operations and hauled directly to the TMF. The PAG1 waste rock pile in the TMF basin will be constructed similar to conventional waste rock piles (i.e., spread by a dozer in controlled lifts and compacted by the mine haul fleet). The PAG1 waste rock pile will initially start at a higher elevation than the initial deposition of tailings. Based on the tailings filling rate and the production schedule for the PAG1 waste rock, the working surface of the PAG1 waste rock pile will be maintained above the elevation the tailings and supernatant throughout the mine life.

## 18.12.6 Water Management and Freeboard

The primary water management objectives for the TMF are as follows:

- Maximize the recycle of process water and runoff water from the TMF to the plant site.

- Provide temporary containment of the EDF within the TMF basin and safe conveyance of the IDF from the TMF during operations.
- Maintain a minimum water cover (2 m) over the deposited tailings throughout operations.

Meteoric and supernatant inflows to the TMF basin will be temporarily stored prior to reclaim by a floating pump barge in the basin to the process plant. Water reclaim, and treatment and release will be conducted such as to always maintain a 2 m minimum water cover over the deposited tailings surface.

A water/solids balance was completed for the TMF. The water balance was completed for pre-production, operations (Years 1 through 12 [partial]), and closure, using average precipitation conditions. Analyses were also completed for abnormally wet conditions. Based on the water balance, under average conditions, the TMF will operate in a net water surplus and excess water beyond the storage of the required water cover level and allowable operating range will be transferred to the TMF water treatment plant as required for treatment prior to release to the environment. Under wet conditions the water treatment plant will need to operate at near its maximum capacity.

The estimated runoff volumes from the EDF (1 in 200 year, 72-hour storm event) and IDF (24-hour Spring PMP plus the corresponding melt of the 1 in 100 year snowpack) were used to determine the storage volume and corresponding wet freeboard depth required within the TMF to manage each event. The estimated peak flows for the IDF were used to design the emergency overflow spillways. Approximately 1.1 m of wet freeboard allowance is required during all stages of operation to manage the EDF and IDF. A dry freeboard allowance of 0.9 m is required to prevent overtopping from wave run-up for all stages of operations. Therefore, the total freeboard above the maximum operating water level is 2 m for all stages of the TMF.

### 18.12.7 Seepage Collection System

Seepage from the TMF will be collected in the seepage collection system which will consist of drains and sumps. The seepage collection drains will be installed in the foundation along the upstream toe of the TMF embankment to collect potential seepage below the embankment and safely route it to the nearest downstream seepage collection sump, located adjacent to the downstream toe of the embankment.

Water collected in the seepage collection sumps will be transferred back into the TMF using a pump-back system. If the collected water is suitable for release to the environment (i.e., meets the discharge criteria), then it may be discharged to the downstream receiving environment.

### 18.12.8 Polishing Pond

A polishing pond will be constructed as an external pond to store water for the TMF WTP operations (by GHD). The polishing pond will be constructed southwest of the TMF. The polishing pond has been designed to store approximately 20,000 m$^3$ of water, which is equivalent to roughly four days of TMF WTP discharge capacity plus some extra capacity contingency.

The polishing pond has been designed to meet the Canadian Dam Association Technical Guidelines for Mining Dams ( (Canadian Dam Association (CDA), 2013) (Canadian Dam Association (CDA), 2019)), and includes freeboard and design earthquake ground motion considerations to minimize operational risks. The polishing pond has been identified as having a Dam Hazard Classification of Significant based on the foreseeable consequences.

The polishing pond embankment will be constructed in one stage as zoned rockfill dam. The bulk fill within the embankment will consist of NPAG material, sourced from the open pits. The zoned embankment will be constructed with filter graded materials consisting of a liner bedding layer and one filter/transition zone (processed mine rock). The majority of the embankment fill will be constructed using unprocessed ROM rockfill. The downstream slopes will be 2.25H:1V. The upstream slopes will be 2.5H:1V. The crest width will be 6 m and the maximum embankment height above original ground is 6.5 m. A typical cross-section for the polishing pond embankment is shown on Figure 18-10.

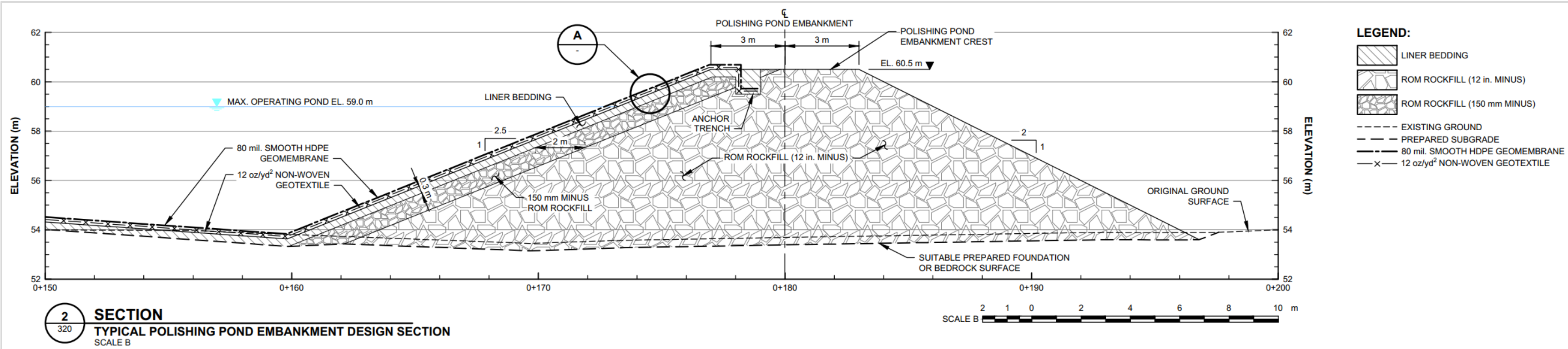



*Figure 18-10: Polishing pond – section*

The polishing pond basin and upstream embankment face will be lined with a smooth 80 mil HDPE geomembrane overlying a 12 oz./sq. yd. non-woven geotextile. The lining system in the basin is to be placed on a prepared subgrade.

The embankment has been designed to crest El. 60.5 m to provide the required water storage capacity in addition to freeboard contingencies for storm water runoff management (under normal operating conditions), excess water discharge, conveyance of the IDF through the emergency overflow spillway, and wave run-up. A wet freeboard allowance of 0.1 m has been included for conveyance of the IDF through the emergency overflow spillway, while a dry freeboard allowance of 0.4 m has been provided to prevent overtopping by wave run-up.

The polishing pond design includes a two-staged spillway on the south side of the pond. The lower stage is a small, riprap lined, trapezoidal operational spillway to convey flows equal to the TMF WTP discharge rate. The upper stage is the emergency overflow spillway sized to safely convey excess runoff, resulting from the IDF, to the environment.

### 18.12.9 Operations, Monitoring and Surveillance (OMS)

The facilities will be operated in compliance with applicable international and national guidelines and standards. An OMS Manual and Emergency Response and Preparedness Plan (ERP and Emergency Preparedness Plan (EPP)) for the TMF will be developed prior to operations. These documents will be used for operator training and support for the management of the TMF.

Monitoring of the TMF and associated infrastructure will be carried out at specified regular intervals to evaluate the performance of the TMF and to refine the operating practices. Regular inspections of the TMF and associated infrastructure will be completed as part of the TMF operations to confirm that the TMF is being operated in accordance with the design intent.

### 18.12.10 Reclamation and Closure

The conceptual TMF closure plan includes removing the tailings supernatant water and encapsulating the tailings and PAG1 waste rock with a closure cover during the final years of operation and active closure (approximately 2 years) to maintain the tailings and PAG1 material in a saturated state to prevent the onset of ARD conditions. The closure cover over the tailings and PAG 1 material will consist of a combination of a geosynthetic reinforcement layer (tailings area only), NPAG waste rock (nominal 2 to 3 m thick), till (0.45 m thick), and topsoil (0.15 m thick). Small riprap lined collection ditches will be constructed on the cover to route precipitation runoff from the cover and minimize erosion. The cover will be vegetated to improve site aesthetics and erosion protection. Ongoing monitoring will be performed for a period of time sufficient to confirm suitable water quality and ongoing stability for the facility. The reclamation and closure plan is a living document that will be updated throughout the project to reflect changing conditions and input from local regulators.

# 19. MARKET STUDIES AND CONTRACTS

## 19.1 Market Studies

The Company has not completed any formal marketing studies with respect to gold production that will result from the mining and processing from the Project, which is assumed to be in the form of gold doré bars for the purposes of this study. However, the Company was able to make reference to its existing refining contract at its Point Rousse operation with the Royal Canadian Mint to refine its gold doré bars into bullion, and a precious metals sales agreement with Auramet International LLC for the purposes of selling bullion. Gold produced will likely be sold on the spot market by precious metals marketing professionals retained on behalf of the Company, at terms and conditions typical of similar contracts for the sale of refined LBMA[5] Good Delivery gold bullion. There are active and liquid gold markets throughout the world where gold can be bought and sold, and market pricing can be ascertained.

Gold is typically quoted on a per ounce basis and denominated in US dollars. During 2020 the price of gold fluctuated between US$1,527 and US$2,067 per ounce. During 2021, the price of gold averaged US$1,799 during the year, including a high of US$1,943 and a low of US$1,684. For the purposes of this study, the Company has assumed a gold price of US$1,600 per ounce and a foreign exchange rate of US$1.00:C$1.25 based on forward consensus pricing, which is considered reasonable in the context of the current market. This results in a Canadian dollar price of gold of $2,000 per ounce for the purpose of this study.

## 19.2 Contracts

The Company was able to use its existing contract with the Royal Canadian Mint, whereby the Mint refines the Company's gold doré bars into LBMA Good Delivery gold bullion, which it believes is a reasonable basis for cost assumptions used in this study. The Company has also relied on its contracts for transportation, security, and insurance with respects to the refining of its gold doré bars.

The Company may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the gold mining industry in Canada. At this point there are no sales contracts in place and for the study it has been assumed that gold will be sold on the spot market by precious metals marketing professionals retained on behalf of the Company, similar to its existing process at its Point Rousse operation.

With respects to contracts in general, as part of the Company's socio-economic commitment to the region and other local stakeholders, where possible the Company will endeavour to work with local businesses as well as other businesses in the province of Nova Scotia. The Company's objective is to focus on opportunities for the residents and businesses of the region to participate in the Project, with the aim of being an active member of the community and participant in the sustainable development of the region. It is expected that the construction of the Project will be executed through an Engineering, Procurement and Construction Management (EPCM) contract. The Project will require an employee accommodation facility which is expected to be managed by a third party under contract. The FS has assumed that blasting, extraction, and haulage will be undertaken by a mine contractor.

[5] https://www.lbma.org.uk/good-delivery/good-delivery-rules-and-governance. London Bullion Market Association (LBMA) is the international trade association representing the global Over The Counter (OTC) bullion market and defines itself as "the global authority on precious metals". LBMA Good Delivery is the de facto world standard, which accredits refiners who produce bars which satisfy high standards in terms of purity, quality and physical appearance.

# 20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL, OR COMMUNITY IMPACT

## 20.1 Environmental Regulatory Setting and Approvals Process

The EARD will be submitted Q2 2022 and in accordance with the Environmental Assessment Regulations made under the Nova Scotia Environment Act. Consultation programs specific to this Project for the purposes of the Provincial Environmental Assessment (EA) process have been developed and will continue throughout the EA and permitting process. Information collected from these programs will be used in the preparation of the provincial Class 1 EARD for the Project and subsequent permitting packages. The Project does not exceed the federal Impact Assessment Act threshold therefore the Project is anticipated to be reviewed via the provincial process.

Following release from the provincial EA process numerous permits, leases, and approvals are required for initiation of the Project and will be sought throughout 2022 and 2023. The notable approvals and permits required to proceed to mine development, operation and reclamation are:

Federal

- Fisheries and Oceans Canada (DFO) *Fisheries Act* Authorization.
- Amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations (*Fisheries Act*) due to deposition of mine waste on waters frequented by fish.

Nova Scotia *Environment Act*

- EA Approval
- IA issued for all Project phases
- Water Withdrawal Permit
- Wetland Alteration Permit
- Watercourse Alteration Permit

Nova Scotia *Crown Lands Act*

- Crown Land Lease

Nova Scotia *Mineral Resources Act*

- Mineral Lease

Municipality of the District of Guysborough

- Municipal level approvals as required for minor items such as signage.

The Company is engaging with the appropriate regulators on the various requirements to facilitate submission.

## 20.2 Baseline Studies

Baseline conditions for each biophysical, ecological, and socio-economic valued component (VC) were studied to:

- Characterize the existing environment of the Project.
- Establish an understanding of the receiving environment.
- Provide sufficient context to enable an understanding of how the Project activities may affect the existing environment.

The Company initiated baseline environmental work at the site during the spring and summer of 2017. A careful review of the previous work was completed in 2020, and supplemental programs were developed and initiated in 2020. Additional detail on the baseline programs and studies are described below. Consultants involved in the baseline programs to date and referred to in this Section include the following:

- GEMTEC Consulting Engineers and Scientists Limited (GEMTEC)
- WSP
- MEL
- GHD
- Lorax
- Membertou Geomatics Solutions (MGS)
- Davis MacIntyre and Associates (Davis MacIntyre)

Consultation programs specific to the Project for the purposes of the Provincial EA process were developed and initiated with anticipated completion in 2022.

For the purposes of the EARD and this discussion, the Project Area is defined as the footprint of Project-related infrastructure plus a buffer of 100 m to 200 m.

### 20.2.1 Climate

The Project design and associated components studies includes several tasks that require climate data inputs, including the mine water management plan development, surface water baseline monitoring, groundwater quantity and quality monitoring, and TMF design.

The Project is located near the southeast coast of Nova Scotia within the Atlantic Coast Ecoregion, and it is expected that climate conditions experienced at the site would be best represented by a station within or near the same region/coast.

Precipitation data will be required for the time periods covering the previous and future monitoring to be conducted at the site. The Deming station east of the Project provides daily total precipitation, rainfall, snowfall, and mean, minimum, and maximum air temperature data. Lake evaporation data is derived from the monthly climate normals at Truro station, which is the nearest Environment and Climate Change Canada (ECCC) climate station that provides evaporation data to the Project. Current precipitation data, required to complete the surface baseflow monitoring program, will be taken from Malay Falls or Collegeville Auto stations as they are the nearest stations to the Project, along the Atlantic coast with sub-daily, current precipitation. A climate change factor of 5% will be applied to the IDF data calculated from the Deming station to allow for climate change contingencies following the NSECC projections for the 2011-2040 period.

### 20.2.2 Air Quality

GHD completed air emission estimates and dispersion modelling in 2019. GHD is currently updating the modelling based the Project design.

The emission rates from the Project-related sources will be calculated using United States Environmental Protection Agency (USEPA) AP-42 emission factors. Recent ambient air quality monitoring data was obtained from the National Pollutant Surveillance network. GHD will complete the air assessment using the 90$^{th}$ percentile measured concentration as “background”. Dispersion modelling will be performed using the USEPA multi-source dispersion model AERMOD.

The results of the 2019 model indicated that all particulate size fractions (Total Suspended Particulate [TSP], Particulate Matter less than 10 µm and 2.5 µm in size [$PM_{10}$), and $PM_{2.5}$]) were predicted to meet the assessment criteria for all averaging periods for the operations. The cumulative effects assessment for TSP showed that this species is predicted to meet the 24-hour and annual assessment criteria. Both 24-hour and annual $PM_{2.5}$ cumulative effects are also predicted to meet the assessment criteria.

A Best Management Practices Plan for dust control will be developed to minimize the re-suspension of road dust from on site vehicle movement during all phases of the Project. A monitoring program for airborne dust may also be implemented to validate that the Best Management Practices Plan is providing suitable and sufficient control of airborne particulates.

### 20.2.3 Noise

GHD completed a Noise Impact Study in 2019. The study included the evaluation of potential off-site environmental noise impacts up to 1500 m from the Project Area where any noise impacts beyond this distance is expected to be environmentally insignificant. The study evaluated the potential noise impacts generated during the construction phase and the operations phase on the nearest sensitive receptors. The Project is expected to increase ambient sound levels within the Project Area.

Additional modelling will be conducted by GHD in 2022 to determine the potential impact of the increased ambient sound levels (i.e., effects on wildlife) and to determine if additional mitigation measures are required to reduce the noise impacts at the nearby worst-case points-of-reception. GHD will use CadnaA (2021) software to create a 3D Acoustical Model to evaluate the potential impacts for both construction and operational project stages. Noise levels produced by equipment will be assessed at various worst-case points-of-reception and at the Project property line to determine the possible future impact on residents of the nearest communities. Predicted noise levels produced by operations are anticipated to be within the guideline limits specified by NSECC at all the identified worst-case points-of-reception. Based on these predictions, noise levels at all nearby residential receptors are expected to be within the NSECC noise level time-of-day limits.

### 20.2.4 Light

GHD completed a Light Impact Assessment for the Project in 2021. The Company provided GHD with a list of equipment that are expected to be sources of light during operations. Calculated light levels at the identified sensitive receptors were below the limits recommended by the Institution of Light Engineers (ILE) guidelines during both post- and pre-curfew conditions.

Mitigation measures will be put in place to minimize impacts of light on birds and terrestrial fauna. Some of the mitigation measures include using light spectrums that have less effect on wildlife and shielding lights where possible and practicable. As well, lighting at the site will be kept at a minimum during the nighttime while satisfying security and operational needs. Lights will be directed downwards and away from local residents and sensitive receptors. The surrounding topography and site infrastructure as well as distance, hills, and trees will substantially shield residents from the light sources from the site.

### 20.2.5 Geology, Soil and Sediment Quality

The Project is in the Eastern and Atlantic Coastal Ecoregions of the Acadian Ecozone. The Eastern Ecoregion is underlain by quartzite and slate of the Meguma Supergroup (Goldenville and Halifax Groups) with granitic intrusives throughout, blanketed by fine-textured till derived from these underlying and adjacent rocks. A variety of landforms are found in this ecoregion, including rolling till plains, drumlin fields, extensive rockland, and wetlands. The bedrock is highly visible in those areas where the glacial till is very thin, exposing the ridge topography.

The Project is entirely underlain by metasedimentary rocks of the Goldenville Group, and the local sequences consist of greywacke, arenite, and slate. The Goldenville grades upwards through manganese-rich strata into a basal Halifax Group unit that consists of sulphidic black slate with lithologies in the uppermost stratigraphy consisting mostly of grey-green slate and metasiltstone. The Halifax Group rocks are sulphide bearing and have the potential to become acidic when exposed to oxygen and water. Certain rocks of the Goldenville Group may also be a source of ARD, particularly in areas where highly mineralized zones are present.

Surficial materials in the area are described as ground moraine and streamlined drift. The on site surficial material consists of relatively uniform silty sand and gravel till with frequent cobbles and boulders. The silt and clay content of the till typically increases with depth, as the contact with the bedrock surface is approached. The depths of till to bedrock are in the range of 1 m to 7 m. The Project is underlain predominantly by the Danesville (50 cm + depth) and Halifax (60 cm + depth) Series soils and to a lesser extent Aspotogan Series and Peat deposits on the eastern and western extents. Halifax and Danesville are associated with coarse-loamy soils dominated with by sandy loam texture derived from granite, quartzite, or sandstone tills.

<u>Historic Tailings</u>

GHD conducted a Limited Phase I and Phase II Environmental Site Assessment (ESA) to characterize environmental contamination remaining from historical mining activity in the area. The purpose of this characterization is to ensure proper management of elevated metal concentrations and the potential surface water and groundwater impacts associated with disturbing historic tailings. Surface water and groundwater modelling will be used to help assess possible indirect impacts to historic tailings that are in proximity to, but not directly disturbed by, infrastructure associated with the Project.

The Phase I ESA consisted of a records review, site visit observations, an evaluation of information available from previous site work, and a screening of known areas of historic tailings within the area of Upper Seal Harbour. The Phase II ESA included an estimate of the spatial extent and volume of material that exists within the footprint that will require handling or treatment, as well as collection of surface soils samples from five previously identified historic tailings areas and review of analytical data. Concentrations of arsenic, selenium, and zinc in exceedance of the NS Tier 1 EQS were present in the historic tailing areas assessed.

All historic tailings disturbed by the development of the Project will be excavated and disposed of in the TMF. A Historic Tailings Management Plan will be developed to manage both direct and indirect impacts to areas of historic tailings.

### 20.2.6 Metal Leaching and Acid Rock Drainage

Geochemical characterization aims to understand and minimize the potential for and effects of metal leaching and acid rock drainage (ML/ARD) that result from the exposure of sulphide minerals contained in geologic materials (i.e., waste rock, ore, tailings, and overburden) during the construction and operation of the Project. Preliminary ML/ARD investigations were conducted in 2017 (GEMTEC, 2018) and 2019 (WSP, 2019) in which 86 samples were submitted for static test analyses. Building on this information, Lorax initiated a comprehensive geochemical baseline program in 2020 considering the current open pit dimensions and associated sampling gaps. During this sampling campaign, 229 samples comprising different material types were collected and analyzed for geochemical parameters (Table 20-1).

**Table 20-1: Summary of 2020-2021 Static Testing Sampling Summary**

| Analysis | Number of Waste Rock Samples | Number of Ore Samples | Number of Duplicates (Waste Rock) | Number of Tailings Samples | Number of Overburden Samples | Total |
|---|---|---|---|---|---|---|
| ABA | 174 | 14 | 7 | 7 | 27 | 229 |
| Solid Phase Metals | 174 | 14 | 7 | 7 | 27 | 229 |
| NAG pH | 11 | 0 | 0 | 6 | 0 | 17 |
| SFE | 36 | 5 | 0 | 3 | 0 | 44 |

Total sulphur contents contributing to the acid potential (AP) of the materials and neutralization potential (NP) were found to be relatively low across the deposit with median values for waste rock being 0.05% S and 9.1 kg $CaCO_3$/t, respectively. Operationally, the distinction and quantification of PAG and NPAG material is important for mine planning since the exposure of PAG mine rock or tailings is expected to have negative impacts on contact water quality. The ARD characteristics of Project geologic materials was defined through the Net Potential Ratio (NPR = NP/AP) as follows:

- PAG1 – NPR < 1 or 1 ≤ NPR ≤ 2 and total S ≥ 0.2 wt. %
- PAG2 – 1 ≤ NPR ≤ 2 and total S < 0.2 wt. %
- NPAG – NPR > 2

According to this classification, PAG (PAG1 and PAG2) proportions were calculated to amount to 38%, 93%, and 100% for waste rock, ore, and tailings, respectively. There is no significant geochemical trend with respect to PAG proportions across the different lithological units. Solid phase Ca/S ratios provide a good surrogate for NPR and, as a result, PAG tonnages were derived via geologic block modelling using the full exploration assay database. These tonnages were used for the development of contact water chemistry predictions (source terms) and ML/ARD management strategies.

A kinetic test program comprising humidity cell, saturated column, and field scale experiments using Project waste and ore materials was initiated in 2021 and is currently ongoing. These tests allow for the evaluation of pH, metal mobility and mineral reaction rates contributing to ML/ARD for a range of conditions.

To date, pH in kinetic test leachates from waste rock, ore, and tailings samples have remained circumneutral. Under these conditions, release rates of sulphate and most pH-sensitive metals are relatively low in waste rock and ore. Tailings are an exception where fine grain size and residual mill process reagents cause elevated concentrations of multiple species including sulphate, cyanide, iron, copper, and cobalt. Arsenic is enriched across the Deposit and is expected to be mobile under a range of geochemical conditions including neutral pH. In waste rock and ore, arsenic was found to be primarily hosted in arsenopyrite which is susceptible to dissolution under oxic conditions. Long-term pH and its effects on metal leaching rates is being investigated through ongoing testing.

Geochemical source terms were developed for contact water associated with the different mine facilities. Predictions are based on the geochemical characteristics of mine rock and tailings assessed through scaled static and kinetic experiments described above as well as site analogue data. Modelled site facilities include:

- WRSA, pit walls, and TMF embankments
- TMF contact water (process water and seepage)

- Overburden stockpiles (soil and till)

Geochemical source terms and the resulting water quality model results were used to inform material handling and water management/treatment strategies to minimize the water quality impacts on the receiving environment. Briefly, all PAG1 waste rock will be co-deposited and ultimately submerged in the TMF to inhibit sulphide oxidation and the onset of ARD from these materials. Similarly, a water cover will be maintained over the tailings throughout operations for the same purpose. At closure, the TMF water cover will be drained and replaced with a soil cover to shield tailings beaches from contact with the atmosphere. PAG2 waste rock produced during the life of mine was identified through geologic block modelling to be minor in tonnage (<1 Mt) and will be interlayered operationally with NPAG waste rock in the various surface WRSAs such that ARD from these facilities is prevented. The exposure time of PAG rock in the pit walls will be minimized via accelerated pit filling at closure.

### 20.2.7 Hydrogeology

To support the FS, a hydrogeologic model was created to develop a conceptual understanding of the groundwater flow regime at the site under pre-mine development (i.e., baseline) conditions. The model is based on the following: hydrogeologic data collected during a four-month pumping test of the existing mine workings between end of September 2018 and January 2019; inflow rates reported into the historical mine workings; a synoptic round of static groundwater elevation monitoring in 2019; topographic data; a review of major hydrologic and hydrogeologic features; hydraulic testing of boreholes and monitoring wells; and the analysis of borehole logs to identify major hydrostratigraphic units. The pumping test of the historic mine workings indicated limited drawdown in the monitoring wells, generally below 3 m, except one well in hydraulic connection with the workings and one monitoring well exhibiting a drawdown of 7 m.

Dewatering of the open pits are anticipated to have an influence on local groundwater regime. The influence of dewatering would be represented by a depressed water table and pressure head and redirected groundwater flow toward the mine. The hydrogeologic model was applied to predict the extend and magnitude of potential impacts to groundwater that could occur as a result of Project development. Specifically, the radius of influence (drawdown), changes to baseflow, and impacts to groundwater quality were evaluated. The drawdown radius of influence of the proposed pits is predicted not to affect existing groundwater users. Degraded groundwater quality at the domestic wells due to mining is not predicted. The nearest identified drilled/dug well is located approximately 1.5 km from the Project Area, outside the predicted maximum groundwater radius of influence of approximately 1 km. Predicted changes in baseflow are incorporated into the hydrologic and hydraulic modelling to evaluate the influence of dewatering on surface water resources (i.e., wetland, and fish). Predicted changes to groundwater quality, and groundwater discharge to surface water are incorporated into the surface water quality model to assess the combined groundwater and surface water impacts on the receiving environment.

For the EARD, GHD is updating hydrogeologic model and including the additional hydrogeologic data collected during the 2020 to 2021 drilling and monitoring well installation program. Groundwater quality and quantity monitoring is being conducted in 2021 from the updated monitoring well network to identify potential seasonal variations in observed conditions and to expand the hydrogeologic monitoring network to cover the proposed project footprint. Once updated, the hydrogeologic model will be reapplied to predict the extent and magnitude of potential impacts to groundwater based on the most up-to-date data.

### 20.2.8 Groundwater Quality

Groundwater sampling was conducted as part of the hydrogeologic assessment in December 2017 and as part of the Company's compliance program associated with the Bulk Sample in 2018-2019. 46 monitoring wells were installed on site in 2021 as part of the baseline data collection program for the EARD, with a further 73 wells planned to be installed in 2022. Baseline groundwater quality samples collected from on site monitoring wells contained concentrations of antimony, arsenic, lead, manganese, and uranium in exceedance of the Guidelines for Canadian Drinking Water Quality, and concentrations of aluminum, arsenic, copper, iron, and zinc in exceedance of the Nova Scotia Tier 1 EQS for groundwater discharging to surface water greater than 10 m away. Groundwater elevations recorded in 2021 ranged from 58.601 to 85.221 metres above sea level (masl).

Groundwater samples will be collected through all seasons and for select stations that provide Project specific data. Additional samples may also be collected from domestic wells in the local area. Monitoring wells installed in 2021-2022 will be equipped with data loggers to provide continuous water level readings.

### 20.2.9 Hydrology

A Hydrology Study was completed in 2018 and 2020 by WSP, with an aquatic assessment of a small pond called Beaver Pond and associated outlet to Gold Brook Lake completed by GEMTEC in 2018. The hydrological response and hydraulic capacity of existing culverts in the Project Area were assessed using precipitation data from neighbouring ECCC weather stations as well as site specific data (topography, soil, and precipitation). Existing flows within the area were quantified using a hydrologic model developed in PCSWMM (software for stormwater, wastewater, watershed, and water distribution systems) using the SWMM 5 engine.

WSP completed an initial baseline (existing conditions) model to reflect the land use and surface features observed from aerial photography.

The full mine development will affect many of the watercourses in the local area to varying degrees, both in terms of annual flow as well as seasonal low flows. Hydrology and hydrogeology studies indicate that during operation all watercourses except Gold Brook will be significantly impacted both in terms of surface flow and baseflow. Immediately following completed site remediation, the effected watercourses will continue to observe the same conditions as during mining operations. However, as the pits are allowed to fill, most of these watercourses will begin to recover and the water table will begin to rise, and the drawdown losses to the watercourses will be lessened.

GHD reviewed all previous modelling results and data collected, considering the advancement of the Project. Supplemental data collection began in 2020 and modelling efforts in 2021. Hydraulic data collected since 2018 were used along with outputs from the model completed by WSP. Hydrologic and hydraulic modelling was used to quantify the flows for the watercourses and to gauge the potential impacts mining operations may have on their respective watercourses. GHD model outcomes were similar to WSP indicating a loss of surface flow and baseflow in the smaller streams adjacent to site infrastructure. A water management plan will be developed for the Project to mitigate impacts associated Project.

### 20.2.10 Surface Water Quality

Water quality is well understood at the Project Area based on previous and ongoing baseline monitoring programs as well an ongoing compliance monitoring associated with the Bulk Sample (completed in 2019). The surveys completed include:

- Surface Monitoring by GEMTEC in 2017 as part of the Bulk Sampling program.
- Aquatic habitat surveys by GEMTEC in 2017.
- A stream survey of Gold Brook was conducted by GEMTEC in 2017.
- An aquatic assessment of Beaver Pond and associated outlet to Gold Brook Lake by GEMTEC in 2018.
- In spring 2019, MEL completed additional watercourse surveys.
- Surface water quality baseline monitoring took place in June 2019.
- Watercourse assessment in the northeast portion of the Project Area, several watercourses, and Gold Brook Lake in August 2019.
- Ongoing quality and quantity monitoring to support the FS and EARD initiated by GHD and the Company in early 2021.

The 2017 and 2018 GEMTEC assessments, as well as the subsequent MEL surveys in 2019, confirmed there are 13 unnamed watercourses, Gold Brook, and three waterbodies: Gold Brook Lake, Beaver Pond, and a historic settling pond. Watercourse surveys are currently ongoing in 2021 to support the enhanced Project.

Data from programs conducted from 2017 to 2019 were reviewed and a supplemental water quality program was developed for the Project which was initiated in 2021. Samples have been collected through all seasons and for stations that provide Project specific data and data for locations in the local area. Within the baseline data collected, elevated metals (aluminum, arsenic, and iron) are present.

As per the predictive geochemical source term data (Lorax 2021), there is potential for 19 parameters to exceed regulatory limits outlined in the MDMER regulatory objectives, CCME and NSE Tier 1 EQS regulatory guidelines or SSWQG established based on historical background data. These 19 parameters have been defined as constituents of concern for the Project. Predictive water quality modelling was be completed to determine the potential impact of Project contact water within the receiving watercourses and to determine the treatment requirements at each discharge location. Water treatment will be designed to meet applicable provincial or federal discharge requirements. Additional information on water treatment is further described in Section 18.12.1.

#### 20.2.11 Wetlands

GEMTEC and MEL conducted wetland assessments during a field reconnaissance program in the field season (June 1 to September 30) in 2017, 2019, 2020, and 2021. This wetland assessment included:

- The boundary delineation of any encountered wetlands.
- Identifying the wetland characteristics of each encountered wetland.
- An ecological functional assessment for each encountered wetland.

The study team identified 224 wetlands with a total area of 331 ha within the EARD Project Area. Impacts to wetland habitat will be required to facilitate pit development, TMF construction, and other infrastructure required for the Project. Review of the Project design and micro-siting of infrastructure will continue through the EARD to minimize impacts to wetlands.

To mitigate and reduce overall loss of wetland habitat and function, the following actions will be implemented by the Company within wetlands where direct impacts and potential indirect impacts to wetland habitat are expected:

- Ensure all wetlands are visually delineated (i.e., flagged).

- Complete detailed design and micro-siting of project infrastructure to look for opportunities to further avoid or minimize wetland impact.
- Leave buffers or re-vegetate disturbed slopes adjacent to wetlands to limit erosion and sediment release.
- Direct runoff through natural vegetation, wherever practicable.
- Develop and implement an Erosion and Sediment Control Plan.
- Acquire and adhere to wetland alteration permits.
- Work to identify wetland compensation projects (on site where possible and off site within the same watershed where possible) to offset Project wetland impacts.
- Development of a wetland monitoring plan for partially altered and potentially indirectly impacted wetlands.
- Ensure contractors and site personnel are aware of wetland locations and avoid all activities within wetland boundaries during site development and construction.
- Complete pre-construction site meetings for all relevant staff/contractors related to working around wetlands and watercourses to minimize unauthorized disturbance, such as the introduction of invasive species.

The effectiveness of mitigation measures will be confirmed through monitoring requirements, as described at the permitting stage through the IA. The Company will apply for approval to alter wetlands from NSECC and will abide by all site specific conditions of that approval, which will specify any timing windows in which alterations are permitted.

In addition, the Company is committed to engaging in wetland compensation activities for permanent wetland loss associated with the Project as required by the provincial wetland alteration process. A preliminary Wetland Compensation Plan will be developed and will be submitted to support the EARD with proposed wetland compensation options to offset wetland losses required to support the Project.

### 20.2.12 Fish Habitat Mapping and Characterization

An initial fish habitat assessment and fish population survey was carried out by GEMTEC in June 2018. Detailed fish habitat quantification, including Project specific mapping, and analysis, was completed by MEL in 2018, 2019, 2020, and 2021. Predictive mapping and species at risk screening was carried out using data acquired from the Atlantic Canada Conservation Data Centre (ACCDC) in 2018 and 2021. Surface water feature screening was completed with mapping and later confirmed during the field visits in 2021. Fish habitat survey tasks included:

- Documentation of fish habitat features.
- Collection of water quality data, collection of flow measurements.
- The execution of a live fish sampling program.
- Completion of eDNA surveys to support fish habitat quantification and fishing programs.
- Collection of benthic invertebrate community baseline conditions.

In 2018, GEMTEC identified 10 watercourses associated with outlet of Gold Brook Lake and Beaver Pond systems. To further assess the potential for fish and fish habitat, MEL completed additional watercourse identification and surveys including confirmation of conditions within the Gold Brook Lake outlet and Beaver Pond systems. Preliminary “spot-checks” were conducted in November 2018 to confirm the presence of watercourses and physical habitat conditions in the vicinity of Beaver

Pond system. Preliminary watercourse, wetland identification, and delineation were completed in the eastern portion of the Project Area in April 2019, follow up fish habitat surveys and fish collection occurred in June 2019. In addition, MEL conducted electrofishing surveys in selected watercourses within open reaches (i.e., without the use of barrier nets) to better understand the fish species present as well as the abundance and density.

The 2020 and 2021 MEL field programs involved four main tasks: a continuation of seasonal high flow trapping and barrier assessment; three rounds of fish sampling (i.e., electrofishing and trapping) within selected watercourses and waterbodies; detailed fish habitat characterization and quantification of watercourses predicted to be directly and indirectly affected by Project development; and, a continuation of baseline field delineation of watercourses and wetlands within the unmapped portions of the Project Area. Water quality measurements were recorded in-situ during fish and fish habitat surveys.

Watercourses within the Project Area have habitat ranging from flats to rapids. The dominant habitat type within the Project is riffle, runs, and flats with the presence of pools in a handful of watercourses. Substrate within watercourses consists of primarily muck/detritus or boulder, however some systems are characterized by rubble and cobble substrate. Gravel, sand, and silt substrate is generally lacking in aquatic habitats within the Project Area. The habitats present within Gold Brook Lake and associated watercourses within the Project Area support various life history stages of the fish species identified through extensive fish collection efforts completed through 2020-2021. Species observed within the Project Area include (in decreasing order of abundance): yellow perch, American eel, brook trout, golden shiner, banded killifish, and blacknose dace.

Priority species observed within the Project Area include the American eel and brook trout. American eel is listed by Committee on the Status of Endangered Wildlife in Canada (COSEWIC) as Threatened, and ranked by the ACCDC as S3, however this species is not currently listed under provincial or federal endangered species legislation. Brook trout is assessed by the ACCDC as S3. The Project Area is within the Southern Uplands designated unit for Atlantic salmon; however, it is not expected that Atlantic salmon are present within this watershed (Salmon Atlas, n.d.). Multi-season surveys through 2020 and 2021 did not result in detection of Atlantic salmon; furthermore, preferred habitat for Atlantic salmon is generally lacking throughout the Project Area. DFO states that freshwater habitat for Atlantic salmon should be composed of “clear, cold, fast-moving water with a gravel bottom for spawning and rocky areas for juvenile fish. Salmon are often found in pools that offer protection from predators and warm temperatures and where water flow conditions enable them to rest” (DFO, 2018). Although fast water (runs, rapids, and falls) are present in a few watercourses within the Project, the substrate is not conducive to Atlantic salmon. DFO also states that Atlantic salmon fry have a significant mortality rate when pH is below 5.4 and smolts have a significant mortality rate below 5.0. Approximately half of the streams within the Project have pH recorded below 5.4 (Farmer, 2000).

Detailed effects assessment is currently ongoing to support the pending submission of the EARD. Preliminary effects assessment demonstrates that there is approximately 9,000 m of linear watercourse length directly under infrastructure, most of which are small first order streams (less than 1 m wide). All major watercourse systems (linear watercourses and waterbodies) have been avoided through Project design and micro-sighting of infrastructure. There will be direct impact to a Beaver Pond to support the development of the west pit. This pond area is 4,220 $m^2$. Finally, the preliminary site wide water balance has predicted indirect flow reductions within watercourses not directly associated with infrastructure placement which may result in additional indirect impacts to fish resources (estimated at 4,160 $m^2$).

A conceptual fish habitat offset plan will be developed based on expected direct and indirect impacts of the Project to support the EARD submission. This fish habitat offset plan will describe proposed fish restoration projects to offset fisheries losses associated with the Project. Discussions are ongoing with DFO and the Mi'kmaq of Nova Scotia to support the impact assessment and the offsetting plan.

### 20.2.13 Terrestrial Resources

MEL completed biophysical field surveys and reporting in the fall of 2018, 2020, and 2021. The field components included:

- Habitat assessments.
- Lichen surveys, with a focus on species at risk and priority species.
- Vascular plant surveys, with a focus on species at risk and priority species.
- Spring and Fall avian migration surveys.
- Breeding avian surveys.
- Species specific surveys including common Nighthawk and owl.
- Mainland moose surveys.
- Evaluation of Abandoned Mine Openings (AMOs) for bat hibernacula, and review of Project Area for other possible bat hibernacula.

<u>Habitat Surveys</u>

In the fall of 2018 and the spring 2021 a desktop review was conducted using several geospatial data sets including NSECC Wetland inventory, NS forestry inventory, Ecological Land Classification, and aerial imagery. These data were used to plan the general survey route and highlight areas of interest prior to conducting the field assessments.

Field surveys took place in the fall 2018 and the spring – fall in 2021. Survey transects occurred concurrently with the wetland delineation program and all homogenous vegetation types were documented and photographed. Several classifications systems were used including Forest Ecosystem Classification System (FEC) (Neily et al, 2010), Natural Landscapes of Maine (NLM) (Gawler & Cutko, 2018) and Classification of Heathlands and Related Plant Communities on Barrens Ecosystems in Nova Scotia (Porter, Basquill & Lundholm, 2020). Surveyors opportunistically georeferenced and classified community types and their boundaries whenever they were encountered. The field data was then used to delineate the approximate boundaries using orthophotos at a 1:10,000 scale.

<u>Vascular and non-vascular plant surveys</u>

Prior to undertaking the field assessment, a detailed desktop review of known vascular and non-vascular plants (including lichens) observations and potential habitat for rare flora specific to the Project Area was conducted. The desktop review process involved several components: ACCDC database results; Nova Scotia Department of Natural Resources and Renewables (NSDNRR) predictive habitat mapping for Boreal Felt Lichen (*Erioderma pedicellatum*); Mersey Tobeatic Research Institute (MTRI) rare lichen database, results of habitat mapping; and mapped wetland habitat.

All suitable habitats were surveyed by MEL over the three years of surveys to support analysis of Project impact to the terrestrial environment. During the lichen surveys, mature trees that are appropriate for hosting priority lichen species were visually inspected by focusing on tree trunks, branches, and twigs. Seven priority lichen species were observed which include two species at risk:

blue felt lichen (*Pectenia plumbea*) and frosted glass whiskers (*Sclerophora peronella*) and five Species of Conservation Concern (SOCI): Slender Monk's Hood Lichen (*Hypogymnia vittata*), Corrugated Shingles Lichen (*Fuscopannaria cf. ahlneri*), A Shingle Lichen (*Fuscopannaria cf. sorediata*), Appressed Jellyskin Lichen (*Scytinium subtile*) and peppered moon lichen (*Sticta fuliginosa*).

Considering the size of the Project Area (~1,222 ha), vascular plant, and bryophyte diversity was low and approximately 139 vascular plant species and 40 bryophytes, with a total of approximately 179 species observed.[6] Five priority vascular plant species were observed including northern comandra (*Geocaulon lividum*) Wiegdand's sedge (*Carex wiegandii*), variegated scouring rush (*Equisetum variegatum*), Nova Scotia agalinis (*Agalinis neoscotica*) and southern twayblade (*Neottia bifolia*) were identified within the Project Area. The majority of these species, with the exception of variegated scouring rush, were found in the wetland habitat within the Project Area. Whereever possible, wetland habitat will be avoided, or impact minimized, in part to reduce impact to species at risk.

No rare or uncommon vegetation communities were identified within the Project Area and therefore, none are expected to be directly impacted. The landscape proposed to be impacted primarily consists of cutovers, regenerative, and mature softwood stands. Rare plants and lichens will be avoided where practicable with ongoing micro-sighting to support the EARD submission. Additionally, mitigations to reduce indirect effects to these species from the mining activity will be developed and deployed.

Avian Surveys

Breeding bird, spring and fall migration surveys, and species specific surveys (common nighthawk and owl) were completed by GEMTEC (2017) and MEL (2018, 2020, and 2021).

During the fall, spring, and breeding bird surveys, several species at risk (SAR) were observed which include Canada warbler (*Cardellina canadensis*; SAR: Threatened; Nova Scotia Endangered Species Act (NSESA): Endangered), olive-sided flycatcher (*Contopus cooperi*, SAR: Special Concern; NSESA: Threatened) and wood thrush (*Hylocichla mustelina*, SAR: Threatened) and several priority species were observed.

Boreal Owl (*Asio otus*; S2?B), Great-horned owl (Bubo virginiatus; S4) and Northern Saw Whet Owl (Aegolius acadius; S4B) were observed in 2017, however, only northern saw whet owl was observed in the 2021 surveys. No common nighthawks were observed in the 2017 – 2021 surveys.

Habitat loss to birds is expected to occur, however, mitigation will be implemented to reduce impacts. These mitigations include but not limited to clearing outside the breeding bird window (April 15 – August 31) when practicable. If clearing must occur within the breeding season, then nest sweeps by qualified avifauna biologists will occur. Mitigation measures will be implemented to reduce noise and light to limit sensory disturbance to wildlife.

**Mammals**

Moose surveys were completed by GEMTEC in 2018. MEL continued these surveys (tracks and Pellet Group Inventory [PGI]) in 2021, following the same transects that GEMTEC used as well as additional transects specific to cover the Project Area. Both in the 2018 and 2021 surveys, moose activity (browsing and tracks) was observed.

[6] Data is still being processed, however, the approximate number of species observed is 179.

### 20.2.14 Archaeology

All archaeological sites (including known and unknown sites) are protected through the *Special Places Protection Act* from disturbance, unless it is conducted under the supervision of a qualified archaeologist working under a Category C (Archaeological Resource Impact Assessment) Heritage Research Permit issued by the Nova Scotia Department of Communities, Culture, and Heritage (NSCCH).

Davis MacIntyre conducted an Archaeological Resource Impact Assessment of the proposed Project in the summer of 2017. The assessment included a historic background study as well as a field reconnaissance of all planned disturbance areas. In 2019, Davis MacIntyre conducted a follow up Archaeological Resource Impact Assessment, including additional field reconnaissance work. In 2021, Davis MacIntyre conducted an Archaeological Resource Impact Assessment of the Project in the area east of Gold Brook Lake. Previous assessments have been conducted for the Company to the west and southwest of this location. The assessment included a historic background study as well as a field reconnaissance of all areas to be impacted by mine development. Based on the results of the 2021 reconnaissance, most of the Project Area has been evaluated to be of low potential for archaeological resources related to occupation by the Mi'kmaq and their ancestors. The general mine area includes a registered archaeological site, related to historic mining activity which includes cellars and a mill site.

Some cultural activity (hunting blinds and quarrying) has been documented in the 2021, but all of the features noted have been evaluated to be of low significance, and therefore, there are no further recommendations for these features. It is also recommended that the areas of elevated archaeological potential (Low-Moderate 2 and Moderate Potential 6) should be subjected to shovel testing at 5-metre intervals (due to the small size of the area) to determine the presence or absence of archaeological resources, if this area is anticipated to be impacted by either borehole drilling, or any other future impact.

The recommendations of previous study reports, which have been approved by Communities, Culture, and Heritage, are still relevant, and the 2021 recommendations are pending regulatory approval.

### 20.2.15 Mi'kmaq Ecological Knowledge Study (MEKS)

The Company contracted MGS to complete a MEKS in 2017. The MEKS consisted of two major components:

- Mi'kmaq Traditional Land Use Activities, both past, and present.
- A Mi'kmaq Significance Species Analysis, considering the resources that are important to Mi'kmaq use.

The Mi'kmaq Traditional Land and Resource Use Activities component utilized interviews as the key source of information regarding Mi'kmaq use within the MEKS study area. The MEKS study area was defined as those areas within a 5 km radius of the Project Area boundaries and consisted of the following:

- Interviews were undertaken by the MEKS Team with Mi'kmaq knowledge holders from the communities of Paqtnkek, Pictou Landing, and Sheet Harbour during August and September 2017.
- A review of historic maps of Guysborough County show very little recorded evidence of Mi'kmaw settlements within proximity to the MEKS study area.

- A site visit including two Company employees and a Mi'kmaw knowledge holder took place over a three-day period in September 2017.

The MEKS recommended that the Company discuss the future steps of the Project with the Assembly of Nova Scotia Mi'kmaw Chiefs with regards to Mi'kmaw use in the area.

MGS was contracted again in 2021 to complete an updated MEKS , which will be used in the EARD preparation.

### 20.2.16 Socio-Economic Impacts

A high-level Economic Impact Study was completed by Group ATN Consulting in 2018 to estimate the impacts of the Project throughout operations and subsequent decommissioning.

The 2018 study indicated that the Project is a significant development and is taking place in a part of the province that has historically faced significant economic challenges. It was deemed that the Project will bring significant economic stimulus to the communities along Nova Scotia's Eastern Shore and within Northeastern Nova Scotia as a whole including creating hundreds of jobs through the various project phases within the community, increases in household income, significant capital spent on goods and services in Nova Scotia and tax revenues.

An updated and enhanced Socio-Economic Study is being conducted in 2021 based on the current Project and will be used in the EARD preparation. Section 22 provides information on the economic impact of the Project and should be referred to.

To further enhance local economic benefits, the Company prefers to award contracts to local businesses once operations commence. The demographic challenges in Nova Scotia are significant, and the development will help reverse some of the labour and economic disparity in the area.

## 20.3 Monitoring Programs

All emissions from the Project during construction and operation will be monitored, reported, and treated in accordance with provincial and federal regulations.

The Company understands that monitoring is a mechanism to gauge performance and measure against baseline conditions and effects as predicted in the EARD, as well as expectations of regulators, members of the public, the Mi'kmaq of Nova Scotia, and other stakeholders. Monitoring programs will continue throughout the life of the Project to determine the effects on the surrounding environment. However, potential adverse effects on valued environmental and socio-economic components are expected to be short-term and/or highly localized and can be effectively mitigated through the application of technically feasible mitigation and standard mining health, safety, and environment procedures.

Air Quality

National Pollutant Release Inventory reporting is a requirement of subsection 46(1) of the *Canadian Environmental Protection Act* (CEPA). The Company is aware of the legislation and will comply with reporting requirements, as applicable. Monitoring of particulate emissions will be conducted as required by NSECC. Dust Monitoring will take place at drier times to gauge the effectiveness of dust suppression mitigation.

### 20.3.1 Noise

Noise monitoring will take place at several locations within the area with a focus on residents and other sensitive receptors to establish an operational baseline as compared to pre-construction baseline noise. The locations chosen will be the smallest arc distance between receptors and the mill and the rims of the open pits. Ongoing noise monitoring will be conducted based on the final layout and mill design and in accordance with regulatory requirements outlined in the Environmental Assessment Approval and subsequent approvals.

### 20.3.2 Light

Lighting will be monitored to ensure light trespass is minimized and does not extend beyond the areas required for site/worker safety. If complaints are received at the mine concerning light trespass, a monitoring program will be developed, in consultation with NSECC that will aim to reduce light levels in non-active work areas, and redirect lighting in active work areas within the parameters of site/worker safety.

### 20.3.3 Geology, Soil, and Sediment Quality

The Company will regularly test rock to monitor the acid generating potential and provide the appropriate treatment or storage for the material. Areas of historic tailings directly disturbed by the Project will be remediated. A monitoring program for areas that will be indirectly impacted by the Project will be developed as part of the Historic Tailings Management Plan detailed in Section 20.4.1. Visual monitoring of erosion and sedimentation control measures will be required to measure the effectiveness of mitigation activities.

### 20.3.4 Groundwater

A groundwater monitoring plan has been developed to determine water quality and quantity in a series of wells located strategically around the Project Area. The monitoring network will be modified over time as the Project enters different stages of its life cycle. The well network currently consists of approximately 123 wells. It is estimated that the well network will consist of approximately 80 wells during the construction phase. Sampling will be undertaken on a quarterly basis. The samples will be monitored for general chemistry (RCAp-MS), hydrocarbons, total and dissolved cyanide, total and dissolved mercury, dissolved metals, total and dissolved phosphorus, chemical oxygen demand, dissolved organic carbon, TSS, and field parameters (including static water levels), and other compounds as required by legislation and approvals. Results will be compared to applicable provincial and federal guidelines.

Many IA's issued by NSECC prescribe a quarterly groundwater quality monitoring frequency, which is a conservative approach. As such, is it the Company plans to conduct quarterly groundwater monitoring over the duration of the operations phase to determine if there are cyclic changes in groundwater quality related to weather/climate and to monitor trends or mine infrastructure changes during operations.

During active closure (two years) and the first two years of post-closure, groundwater sampling will continue at a quarterly frequency. Following this period, the monitoring frequency will reduce to semi-annually for eight years, followed by annual monitoring for the remainder of the post-closure phase.

The data collected throughout the life of the mine will be used to validate prediction models and determine recovery to pre-construction groundwater levels once mining is completed.

### 20.3.5 Surface Water and Hydrology

Surface water in the vicinity of the Project will be monitored according to terms and conditions of any approvals.

Surface water quality assessment sites were selected to monitor baseline/construction, operation, and post-closure water quality and water quantity. These include both surveillance locations and compliance locations. Site selections were chosen to coincide with previous surface water monitoring completed by GHD from 2018 onward as part of baseline monitoring.

During the construction phase, nine water quality surveillance locations are proposed to be sampled weekly for the first three months of construction, and then monthly from thereon. A one-year duration for the construction phase is assumed for the purpose of costing the monitoring program. At each surveillance monitoring stations, water quality grab samples are collected and will be analyzed for general chemistry (RCAp), total and dissolved metals, total and dissolved mercury, methyl-mercury, dissolved organic carbon, Total Chemical Oxygen Demand, chlorophyll a, petroleum hydrocarbons, field parameters, and others as required by applicable regulations.

One compliance location will selected to meet the MDMER and CCME guidelines for cyanide (total, free), TSS, radium isotopes, and mercury (dissolved, total, methyl).

The collection of continuous water level and manual flow data at eight locations is proposed to continue on a monthly frequency to support and improve the existing stage-discharge relationship/rating curves established and further validate modelling completed for the Project.

During the operations phase (11 years), surface water monitoring locations will adapt to accommodate the phased approach of site operations. These preliminary monitoring sites have been identified to monitor settling pond locations, outlets, and receiving water bodies/waterways. Surface water monitoring locations will be finalized during the IA process. The locations of these stations may require some adjustments in the field post construction, where applicable.

The proposed frequency changes for all water quality sample locations from monthly to quarterly after the initial 8 months of monthly sampling. The frequency of water quantity monitoring should remain as monthly throughout the duration of operations to continue establishing stage-discharge relationship/rating curves and validating future modelling completed for the Project.

During the two years of active closure, nine surveillance locations are proposed and two to three compliance locations. Quarterly monitoring frequency will be undertaken for water quality. Nine water quantity monitoring locations are proposed and will continue to be monitored at a monthly frequency during active closure.

After the completion of active closure, the proposed surveillance locations, and frequency will be reduced. Three water quality surveillance locations are proposed for 14 years during the post-closure phase, with a quarterly monitoring frequency for the first two years of post-closure, followed by semi-annual monitoring for the remainder of post-closure. Water quality compliance monitoring will be undertaken quarterly at each pit lake and outlet for three years (minimum) once discharge to the natural waterway begins. If pit lake discharge is compliant over the three years of monitoring, it is anticipated that the compliance monitoring will not be required for the remainder of the post-closure phase. Monitoring will also be undertaken annually at each pit for the two years prior to the anticipated discharge.

Visual monitoring of erosion and sedimentation control measures to identify pathways to surface water bodies and wetlands will be conducted through all phases.

### 20.3.6 Wetlands

Monitoring will be used to assess the methods used to re-establish wetland vegetation and habitat and will provide early indication of problematic areas where these methods may not be achieving the desired results. Any additional wetland survey, monitoring, or follow up will be developed in accordance with all approvals issued and in consultation with NSECC.

### 20.3.7 Fish and Fish Habitat

Impacts to watercourses will require provincial Watercourse Alteration permitting and impacts to fish and fish habitat will require a Fisheries Act Authorization and an associated fish habitat offsetting plan. Detailed mitigation measures will be outlined and will include measures such as implementation of erosion and sediment control plans, construction, operation, and blasting protocols that prevent unnecessary impacts to fish and fish habitat, spill control, and contingency planning, water management, waste management, and monitoring of all environmental components to ensure minimal impacts and regulatory compliance. Final reclamation will see habitat reasonably restored to pre-existing conditions.

- Monitoring programs for fish and fish habitat will include:
    - Monitoring of predicted impacts in affected watercourses and waterbodies as required through the Fisheries Act Authorization.
    - An Environmental Effects Monitoring (EEM) Program under MDMER for mine discharge locations.
    - Required monitoring for offset project(s).

## 20.4 Waste Management Plan

A Waste Management Plan (WMP) will be developed to provide direction on waste handling, storage, transport, treatment, and disposal of the various wastes produced. The WMP provides a system to deal with waste streams and allow for the implementation of reduction and diversion opportunities. It also serves as an internal quality control document that provides clear and concise direction for Company staff and contractors regarding waste management policies and procedures that must be followed.

The foundation of the WMP is based on the regulatory framework for industrial waste management in Nova Scotia. The principal legislation guiding and governing waste management in Nova Scotia is the Environment Act, specifically the Solid Waste-Resource Management Regulations. These regulations cover the technical aspects of waste disposal, including handling, diverting, recovering, recycling, reducing, and reusing waste materials.

Routine monitoring of waste management activities will be conducted to ensure that the guidelines and procedures outlined in this plan are being followed. Routine monitoring may consist of informal or formal checks on personnel, equipment, and contractors and review of records related to waste management activities.

Monitoring may include:

- Location and condition of on site waste and recycling collection bins.
- Condition and organization of waste laydown and storage areas.
- Waste collection, transportation, and handling operations for Company employees and waste management contractors.
- Waste volumes from mine areas.
- Any other aspects or issues related to the waste management system.

Waste rock disposal and tailings are further described in Section 18.12.1.

### 20.4.1 Historic Tailings Management

Historic tailings have been identified at the Project that were generated by past mining operations in close proximity to the historic underground operations described previously in this FS. The historic tailings are well mapped and understood from a surface perspective through delineation programs completed by the Geological Survey of Canada, the Company, WSP, and GHD. Of the areas of historic tailings mapped at the Project, only four areas were identified within the limits of planned infrastructure and required further evaluation. The Company considered the presence of these historic tailings in the design of the Project to avoid areas where possible. The total surface area of the historic tailings that will be directly disturbed by the infrastructure is less than 1.8 hectares or less than 1% of the Project Area.

Historic Tailings located within the footprint of the Project will be excavated and placed in the TMF. Preferred management options will be detailed in the EARD and will include quantities, movement procedures, engineered containment, and associated monitoring programs for areas of historic tailings that will not be directly disturbed by the Project.

## 20.5 Mi’kmaq and Stakeholder Engagement Regarding Social Impact

The Company has actively pursued opportunities for dialogue and engagement with Nova Scotia Mi’kmaq (Assembly of Nova Scotia Mi’kmaw Chiefs and the KMKNO, MODG, the Goldboro CLC, and the public regarding the Project since May 2017.

The Company recognizes the asserted Aboriginal and Treaty Rights and Title of Nova Scotia Mi'kmaq. Ongoing dialogue and information sharing with the Assembly of Nova Scotia Mi’kmaw Chiefs is maintained by the Company and KMKNO staff. The Company has presented information about the Project to the Benefits Committee of the Assembly of Chiefs and has also met with the Band Council of Paqtnkek, the closest Mi’kmaw community to the Project. On June 2, 2019, the Company and the Assembly of Nova Scotia Mi'kmaw Chiefs signed a MOU for the process that the parties are using to develop an MBA with respect to the Project. This process is ongoing. The Company maintains its commitment to work collaboratively with Nova Scotia Mi'kmaq regarding environmental and cultural priorities, as well as social and economic opportunities throughout the life of the Project. Information shared through ongoing Mi’kmaq engagement as well as completion of a MEKS in 2017 has been reflected in the development of the Project. A new MEKS is in progress that will reflect any new information or considerations related to the current footprint. The Company welcomes an opportunity to engage with any Mi’kmaw Community’s Council or organization that has an interest in the Project.

The Company actively shares new information and engages regularly with MODG Council and staff, with the most recent update being delivered in June 2021. Public engagement activities will occur in 2021 to support the EARD for the Project, including additional open house engagement sessions.

In January 2022, the Company and MODG announced a Community Benefits Agreement that outlines direct financial benefits to support social and economic growth in the region during the permitting and construction process as well as a higher level of commitment when the Project reaches commercial production.

In terms of socio-economic considerations, Mi’kmaq, residents in the Goldboro region and the MODG have expressed interest in employment opportunities and economic development during public consultation and Mi’kmaq engagement.

The results of the public consultation and Mi’kmaq engagement will be considered in the environmental effects assessment of the EARD, including the Company’s commitments to include the Mi'kmaq in the development and implementation of mitigation and monitoring measures and proposed compliance and effects monitoring programs, as well as the Company's broader commitment to ongoing public consultation and Mi'kmaq engagement.

The Company has had numerous meetings with provincial and federal government departments and agencies since 2017; this is in addition to multiple "One Window" meetings with all related regulatory and permitting departments and agencies. Engagement with provincial and federal government departments and agencies will continue throughout the life of the Project.

## 20.6 Reclamation and Closure Plan

Reclamation Plan requirements are governed by the Mineral Resources Act. A preliminary Reclamation Plan is required for the EARD, followed by a detailed Reclamation Plan to be submitted as part of the IA and Mine Lease applications. A Final Reclamation Plan is required to be submitted six months prior to mine closure. The Reclamation Plan is a living document that will be updated throughout the Project to reflect changing conditions and input from local regulators.

The general concept for reclaiming the Project Area is to remove all infrastructure that can be dismantled and to return the site to a state that is concordant with the pre-existing conditions or future land use as identified in consultation with stakeholders and regulators. All static, physical aspects including remaining inert waste rock, and overburden piles will be used for reclamation purposes either as fill on slopes or as a base for topsoil, graded to appropriate slopes. Any remaining waste/overburden material will be graded appropriately and revegetated. The open pit shorelines will be developed to a 5:1 slope and the pits allowed to refill. Shorelines will vegetated to the predicted water edge to reduce erosion and the nearshore will be allowed to naturally revegetate. Soils, vegetation, and wildlife baseline data will be used as guidelines for the design, completion, and evaluation of surface reclamation. Final surface reclamation will blend affected areas with adjacent undisturbed lands to re-establish plant life and present a natural appearance. Surface reclamation efforts will strive to limit soil erosion by wind, water, sedimentation, and re-establish natural drainage patterns.

Some areas of the site will be subject to a progressive reclamation approach, thereby reducing the efforts needed at the time of closure and reducing the re-vegetation timeframes for some areas.

### 20.6.1 Process Plant and Site Works

The process plant closure plan is based on removal of all buildings, structures, and equipment from the Project site, including piping, and structural steel. Equipment and piping will be safely drained of all process materials prior to dismantling. Concrete foundations will be demolished and removed down to a depth of 0.5 m below grade; concrete below this elevation will be left in-situ and covered.

All other surface structures and equipment will be evaluated for appropriate post-closure re-use, sale, or disposal. Buildings and equipment will be decommissioned, decontaminated (as necessary), dismantled, and either salvaged or disposed of in an appropriate off site disposal facility. A New Glasgow facility has been identified as a suitable hazardous waste disposal site for any concrete or equipment that has been contaminated with fuel, oils, or other chemicals. All surface concrete will be demolished including slabs on grade, containment walls, foundation walls, and building piers to a depth of 0.5 m below grade. The graded surface areas will be covered with 0.5 m of soil from the organic stockpiles. The area will then be hydroseeded to promote plant growth and to stabilize the soil against erosion. The haul and access roads will remain in place for access to the open pits and TMF throughout closure monitoring.

### 20.6.2 Tailings Management Facility

Reclamation and closure of the TMF will be based on the following general goals and objectives:

- Reclamation goals and objectives will be considered during the design of the TMF.
- Reclamation goals and objectives will be periodically updated during construction and operations.
- Progressive reclamation will be implemented wherever possible.
- Upon cessation of operations, the TMF will be decommissioned and reclaimed to allow for future land use as guided by local regulators.
- Reclamation and closure construction will be designed to meet long-term physical and chemical stability objectives.

The conceptual TMF closure plan includes for encapsulating the tailings and PAG1 waste rock with a closure cover during the final years of operation and active closure (approximately two years) to maintain the tailings and PAG1 material in a saturated state to prevent the onset of ARD conditions. Small collection ditches will be constructed on the cover to route precipitation runoff from the cover and minimize erosion.

Generally, the closure work will consist of decommissioning and dismantling of all tailings delivery and distribution pipework, all water reclaim pipelines and the pump barge, all seepage recycle pipework and pumps, assuming that the seepage water meets water quality objectives and removal of the tailings supernatant water (including the water cover and operating cover).

Construction of a closure cover over the tailings and PAG1 material will be a combination of a geosynthetic reinforcement layer (tailings area only), NPAG waste rock (nominal 2 to 3 m thick), till (0.45 m thick), and topsoil (0.15 m thick). The NPAG waste rock cover will be placed over the PAG1 material during the final years of operations.

Small riprap lined collection ditches will be constructed to minimize erosion. Vegetation will be planted or allowed to naturally occur on the cover soil to improve site aesthetics and erosion protection.

Post-closure monitoring for the TMF will continue for a period of time sufficient to confirm suitable water quality and ongoing stability for the facility.

The open pits will be allowed to flood creating two open waterbodies with a shallow water wetland border and aquatic habitat. The shorelines will be graded to 5:1 to allow for egress. It is anticipated that the pits will take several years to refill and that final lake elevations will be similar to that of the elevation of nearby Gold Brook Lake.

The need for effluent treatment during post-closure will be fully assessed during the operation and early closure phases through sampling; however, current modelling predicts that treatment will not be required. Water levels will be monitored, and pit refilling rates will be compared to the conceptual model. The potential need for effluent treatment will be based on geochemical source terms that will be undergoing revision as additional information is collected and becomes available. In addition, mitigation methods at closure such as rapid filling of the open pits and redirecting site runoff to settling ponds could be employed to mitigate effluent quality concerns if they are identified. The modelling predicts that the East Pit will fill within 3 years after year 7 of mining and will discharge to Gold Brook in Year 14 and the West pit will fill within 14 years of end of mine life.

Surface and groundwater monitoring is planned to continue for 17 years post-closure or until released by the regulators.

All man-made slopes on the site will be reduced, if necessary, to maintain safe, and stable slopes that promote natural re-vegetation. WRSA will be built to engineered specifications for the material being stored. Any remaining exposed vertical rock faces will be redeveloped to 1:1 slope or the angle of repose as per geotechnical considerations. Removal of facilities and active rehabilitation of the site, including re-vegetation, will occur over an estimated two-year period.

Monitoring the reestablishment of site vegetation will be undertaken and be based on the final approvals with respect to reclamation. This will likely be one to three years following closure with maintenance and any remedial action (e.g., spot vegetation programs, erosion control) occurring on an as required basis to ensure that the goals of the approved final Reclamation Plan are achieved.

Both the Mining Lease and the IA set requirements for a Reclamation Plan and security bond to cover reclamation activities, including post operation closure activities and long-term closure, and reclamation monitoring. A reclamation security bond will be estimated and negotiated with the province during these applications. The security can be submitted to the province progressively as the disturbed footprint and infrastructure increases until the full extent of site development is realized. The cost of reclamation of the Project is summarized in Section 21.5.6. The bond is returned as reclamation activities are carried out.

## 20.7 Comments on Section 20

Mitigation measures to avoid, reduce or compensate for potential effects will need to be developed and supported by comprehensive environmental and social baseline investigations and engineering studies. The FS has made certain assumptions as to the timelines needed to complete prior consultation and collect the necessary seasonal baseline data to allow the EARD to be completed and submitted to the relevant regulatory authorities. There is a risk that these timeline assumptions are aggressive and may need to be refined during the Environmental Assessment application process.

# 21. CAPITAL AND OPERATING COSTS

## 21.1 Basis of Estimates

The capital cost estimate was prepared by Nordmin with an expected accuracy range of:

- ±15% weighted average accuracy of actual costs. Base pricing reflect the third quarter of 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25.

The estimate includes direct and indirect costs (such as engineering, procurement, construction, and start up of facilities) as well as owner costs and contingency associated with mine and process facilities and on site and off site infrastructure. The following areas are included in the estimate:

- Open pit mine development, equipment fleet, and support infrastructure and services to support mining operations (noting it is assumed the mining fleet will be supplied by a mine contractor).
- The TMF will provide secure storage for tailings and process water and protect groundwater and surface waters during operations and post-closure. The FS level design is based on a projected 10 year mine life at a nominal processing rate of approximately 4,000 t/d. The tailings throughput to the TMF is currently envisioned to be on average 3,200 t/d. The TMF has been sized to permanently store approximately 21.1 million tonnes of tailings, or 16.25 million $m^3$ at an average settled dry density of 1.3 tonnes/$m^3$.
- Direct costs include contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas. The process plant design point daily throughput is 4,000 t/d.
- The process plant is designed for ore treatment of 1,460,000 t/a or 4,000 t/d based on an availability of 8,059 h/y, or 92.0%.
- On site infrastructure (water treatment and distribution, electrical substation, and distribution, shops, and other general facilities).
- Off site infrastructure (water and power supply).

A small amount of engineering work, being in the range of 10% to 15% of total engineering for the Project was carried out to support the estimate. The estimate was based on the following Project specific information:

- FS level process flowsheet, design criteria and mass balance.
- Process and potable water demands are 6 $m^3$/h and 16 $m^3$/h respectively.
- Potable water treatment is sized assuming an equal flowrate for both potable water and wastewater (16 $m^3$/h).
- Sewage wastewater treatment units are designed for peak design flowrate of 14 $m^3$/h for employee accommodation and 2 $m^3$/h for the process plant area.
- Contact water ditches and culverts are designed to convey stormwater runoff resulting from the 1 in 100 year, 24-hour, climate change adjusted storm event.
- Non-contact water ditches and culverts are designed to convey stormwater runoff resulting from the 1 in 5 year storm event with durations ranging from 20 minutes to 1.2 hours depending on time of concentration in contributing sub-watershed. TMF embankment clean water ditches were designed to convey stormwater runoff resulting from the 1 in 100 year, 24-hour, climate change adjusted storm event.

- Settling ponds are designed to detain runoff resulting from the 25 mm 4-hour storm event, 1 in 10-year 24-hour climate change adjusted storm event and 1 in 100-year 24-hour climate change adjusted storm event for a minimum of 24 hours to allow for TSS removal due to settling. Designed to convey runoff resulting from the 1 in 100-year 24-hour climate change adjusted storm event to Gold Brook Lake or engineered wetland via a concrete outlet structure. All discharge water from collection ponds must meet MDMER water quality requirements. All discharge water must meet CCME, Tier 1 NSE WQS or site-specific requirements within the 100 mixing zone of the natural watercourse receiver (Gold Brook Lake or Gold Brook).
- The intake structure, pipe sizing, alignment, and booster station locations were detailed for pricing purposes. It should be noted that it is assumed that the potable water line can cross the existing gas line to convey potable water to employee accommodations site.
- The cost includes the supply and installation of a containerized potable water treatment unit (including instrumentation, programming cost, gravel pad, electrical and mechanical connections) and storage tank.
- The cost includes the supply and installation of a containerized sewage wastewater treatment unit (including lift station, instrumentation, programming cost, gravel pad, electrical and mechanical connections). Wastewater service lines from the employee accommodation and all buildings/facilities were priced including the collection laterals, intake lines and discharge lines from the wastewater treatment units.
- WTS costs were developed based on vendor supplied pricing, unit rates and industry practices for procurement, supply and installation of civil works.
- Geotechnical and survey data were not available at the time of developing the listed costs associated with Water management and Treatment.
- Costs include packaged water treatment system, piping, delivery, equipment installation, instrumentation and controls, electrical and buildings (chemical feed systems and chemical storage).
- Cost associated with heat transfer have not been allocated.
- Cost for ditching assume on site material can be utilized for rock lining.
- Feasibility level mine, process plant and TMF design criteria.
- Feasibility level general site layout.
- Feasibility level electrical supply.
- Massive earthworks quantities derived from preliminary 3D models.

Factored, end-product units and physical dimensions methods were used to estimate costs based on historical data from similar projects or facilities. Contractor quotes were provided for major cost items. The ratio or factored estimating method was used in estimating the cost of process plant components or areas where the cost of the specialized process equipment made up a significant portion of the total component or area cost. Nordmin and its Consultants used historical data available from similar projects; the end-product units estimating method was used to relate the end-product units (capacity units) of a plant component to construction costs. This allows an estimate to be prepared relatively quickly, knowing only the end-product unit capacity of the proposed component.

Data for the estimates have been obtained from numerous sources, including:

- Feasibility Level engineering design by Nordmin, Knight Piésold, GHD, Ausenco.
- Surface and Process Plant buildings general arrangement is a 3D model.

- Surface and Process plant structural steel and concrete quantities developed from the 3D model.
- A mechanical equipment list was created for the surface buildings and process plant.
- An electrical load list and equipment list was completed for the surface building and for the process plant.
- Contractor and vendor quotes from local and national contractors for project specific conditions.
- Historical pricing data from similar projects in the Atlantic Canada region.
- In-house benchmarking data from similar projects in the Atlantic Canada region.
- Topographical information.

The following assumptions were considered:

- All equipment and materials will be new.
- The main equipment will be purchased and manufactured in appropriate sizes to be transported by the existing main roads to the Project site.
- The execution work will be continuous without interruptions or stoppages.
- Contractors will be contracted under unit price contracts.
- The project will be executed through an EPCM contract.

The following are excluded from the capital cost estimate:

- Mining fleet, as mining activity is assumed to be undertaken by a mine contractor (and therefore costs related to mining equipment are captured in operating costs).
- Land acquisition costs for areas of planned surface and process infrastructure.
- Finance costs and interests during construction.
- Costs due to fluctuations in exchange rates.
- Changes in the design criteria.
- Changes in scope or accelerated schedule.
- Changes in Canadian legislation.
- Site mitigation (identification and removal of contaminated soils – oil, fuel spilled, heavy metals, pesticides, etc.).
- Other than specified obligations and taxes.
- Provisions for force majeure.
- Wrap-up insurance.
- Reschedule to recover delays due to:
    - Change in scope.
    - Force majeure.
    - Notice to proceed with construction.
    - Labour conflicts.
    - Non-availability of qualified and other labour.
    - Lack of geotechnical and environmental definitions.
    - Different soil conditions.
- The proposed Project includes approximately 2 years pre-production construction period, with a six-month ramp up production.

## 21.2 Labour Assumptions

The construction labour and equipment costs were included in the factors that were used in the estimation to account for installation costs or in the unit costs when applied.

## 21.3 Material Costs

All materials required for facilities construction are included in the capital cost estimate. Material costs include freight to the site. Material costs related to the processing plant such as concrete, structural steel, piping and fittings, and electrical cable were included within the installation factors applied to the mechanical equipment costs.

Material cost related to the processing plant platform, TMF and planned access roads were determined by material take off quantities from sketches/drawings, contractor supply and installation unit costs. All earthworks quantities were assumed to be neat in place, with no allowance for swell, waste, or compaction of materials. Industry standard allowances for swell and compaction were incorporated into the unit rate.

## 21.4 Contingency

The contingency was established deterministically applying the following percentage factors associated with a FS level estimate to capital costs:

- 15% on mining, processing plant, supporting on site infrastructure, utility systems, and general/indirect costs.
- 10% on mining pre-operating stripping.
- 15% on the TMF capital costs.

## 21.5 Capital Costs

Details of the initial and sustaining capital estimate are shown in Table 21-1.

The estimate of initial capital costs is $271.0 million including amounts for indirect and contingency assumptions, as outlined Table 21-1 (note that columns may not sum exactly due to rounding). A contingency of $31.7 million has been included in the estimate of initial capital costs, which amounts to 16% of direct initial capital costs or 11% of the total.

The sustaining capital, including rehabilitation and closure costs, fisheries and wetland compensation and the reversal of upfront working capital, is estimated at $113.4 million over the life of the Project.

**Table 21-1: Summary of Initial and Sustaining Capital**

| Item / Description | Units | Pre-Production Phase | % of Total | Production Phase | Closure Phase | Total |
|---|---|---|---|---|---|---|
| **Capital Cost** | | | | | | |
| **Capital Cost Estimate** | | **INITIAL CapEx** | | **SUSTAINING CapEx** | | **TOTAL** |
| Open Pit Mining | M$ | 25.5 | 9% | 1.6 | | 27.1 |
| Process Plant | M$ | 70.5 | 25% | | | 70.5 |
| Tailings Management | M$ | 20.6 | 7% | 42.4 | | 63.1 |
| Infrastructure and Site Development | M$ | 49.8 | 18% | 7.4 | | 57.2 |
| Water Management & Treatment | M$ | 14.4 | 5% | 11.7 | | 26.1 |
| General Site Equipment | M$ | 1.1 | 0% | | | 1.1 |
| Employee Accommodations | M$ | 12.1 | 4% | | | 12.1 |
| **Subtotal Capital Costs** | **M$** | **193.9** | **70%** | **63.1** | **0.0** | **257.0** |
| Indirect CapEx | M$ | 45.4 | 16% | | | 45.4 |
| *Mill Labour during pre-production* | *M$* | *0.79* | *0%* | | | *0.8* |
| *G&A Labour during pre-production* | *M$* | *2.18* | *1%* | | | *2.2* |
| *Other Indirects* | *M$* | *14.6* | *5%* | | | *14.6* |
| *EPCM* | *M$* | *27.8* | *10%* | | | *27.8* |
| Contingency | M$ | 31.7 | 11% | | | 31.7 |
| **Subtotal Capital Costs** | **M$** | **271.1** | **97%** | **63.1** | **0.0** | **334.2** |
| Rehabilitation & Closure, Bond Cost | M$ | 0.7 | 0% | 10.1 | 30.3 | 41.0 |
| Other CapEx – Habitat Compensation | M$ | 0.0 | 0% | 9.3 | | 9.3 |
| Working Capital | M$ | 6.7 | 2% | -6.7 | | 0.0 |
| **Total Capital Costs** | **M$** | **278.5** | **100%** | **75.7** | **30.3** | **384.5** |

### 21.5.1 Mining Capital Costs

#### 21.5.1.1 Open Pit Mining

The capital cost estimate for open pit mining is based on a mine contractor scenario. The total mining capital costs for the open pit portion of this Project are estimated to be $25.5 million and relates to the mining costs incurred during the one-year pre-production period. Table 21-2 summarizes the direct cost estimates.

**Table 21-2: Open Pit Mining Capital Cost Summary**

| Description | Pre-Production Period M $ |
|---|---|
| **Open Pit Mining** | |
| Pre-production mining | 14.8 |
| Contractor Costs | 8.6 |
| Owner's Costs | 2.1 |
| **Subtotal** | **25.5** |
| Contingency | 2.6 |
| **Total Capital Estimate** | **28.1** |

The open pit mining cost estimate is based on the average of budget quotes received from two regional mining contractors. The pre-production mining category includes drilling, blasting, loading, hauling and support equipment. The contractor cost category includes mobilization, temporary facilities, and contractor management. The owner's cost category includes the mine technical team, and allowances for items as blast monitoring, crush material road maintenance, pit dewatering.

### 21.5.2 Process Plant Capital Costs

The capital cost estimate was developed in Q4 2021 from a combination of vendor and contractor quotes and Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of 15% in accordance with the Association for the Advancement of Cost Engineering (AACE) International, Class 3. The estimate includes the process plant, and ancillary buildings.

The capital cost summary is presented in Table 21-2. The total initial capital cost for the processing plant is $70.5 M.

**Table 21-3: Process Plant Capital Cost Summary**

| WBS Level 2 | Description | Total Capital Costs (C$M) |
|---|---|---|
| **1000** | **General Mine** | **3.6** |
| 1300 | Tailings and Decant Return Piping | 3.6 |
| **3000** | **Process Plant** | **66.9** |
| 3100 | Crushing, Stockpile and Reclaim | 16.4 |
| 3200 | Grinding | 11.2 |
| 3300 | Gravity and Intensive Cyanidation | 1.3 |
| 3400 | Leach-CIP | 9.3 |
| 3500 | Desorption, Regeneration and Goldroom | 6.2 |
| 3600 | Cyanide Detoxification and Tailings Disposal | 3.4 |
| 3700 | Reagents | 2.6 |
| 3800 | Plant Services | 3.3 |
| 3900 | Process/Mill Buildings and Electrical Rooms | 13.3 |
| | **Subtotal Direct Costs – Process Plant** | **70.5** |
| | Indirects (contractor indirects, spares, first fills) | 7.6 |

| WBS Level 2 | Description | Total Capital Costs (C$M) |
|---|---|---|
| | Project Delivery (EPCM, Commissioning, Vendor Reps) | 16.0 |
| | Contingency | 14.1 |
| | **Total Initial Capital Costs – Process Plant** | **108.2** |

#### 21.5.2.1 Direct Costs

The definition of process equipment requirements was based on process flowsheets and process design criteria (refer to Section 17).

Once the mechanical equipment list was outlined, the mechanical scopes of work were derived and sent for budgetary pricing by Canadian and international equipment suppliers (see Table 21-3). Once the budgetary quotations were reviewed and integrated, in total 94% of the value of mechanical equipment was sourced from either new budgetary quotations or budgetary quotations from mid-2021 quoted for other Eastern Canadian projects, with the remainder of minor process equipment pricing sourced by benchmarking against other recent Canadian gold projects and studies.

**Table 21-4: Mechanical Packages**

| Pkg No. | Package Name | Source |
|---|---|---|
| 001 | Modular crushing, dry screening and conveyor package | New budgetary quote for the Project |
| 002 | Ball mill | New budgetary quote for the Project |
| 003 | Large leach /CIP / detox agitators | Recent pricing from Eastern Canadian projects |
| 004 | Carbon stripping, regeneration and electrowinning package | New budgetary quote for the Project |
| 005 | Water and slurry pumps | Recent pricing from Eastern Canadian projects |
| 006 | Thickener (tails) | New budgetary quote for the Project |
| 009 | Gravity Gold | New budgetary quote for the Project |
| 010 | Fire Systems | New budgetary quote for the Project |

Similar to the above, all major electrical equipment was sized based on the project equipment list. Once the electrical equipment list was outlined, scopes of work were derived and sent for budgetary pricing by Canadian and international equipment suppliers, as outlined in Table 21-4. All values for the electrical equipment were sourced from budgetary quotations.

**Table 21-5: Electrical Packages**

| Pkg No. | Package Name | Source |
|---|---|---|
| 001 | Integrated Electrical Rooms, including MCCs, VFDs, etc | New budgetary quote for the Project |
| 002 | Outdoor Transformers | New budgetary quote for the Project |

Platework supply costs were based on recent historical budget quotes from similar projects, adjusted to reflect the project sizing.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design was completed to a FS level of definition, allowing for the bulk material quantities (steel, concrete, earthworks, piping, cables, instruments, etc.) to be derived for the major commodities.

After the derivation of all the bulk material quantities, for the process plant and infrastructure areas, major construction contracts were formed, and tendered to experienced Eastern Canadian contractors for budgetary pricing bids (see Table 21-5). The scope of each contract is detailed in Section 21.5.2.2.

**Table 21-6: Construction Contracts**

| Pkg No. | Package Name |
|---|---|
| 502 | Structural, Mechanical, Piping |
| 503 | Pre-engineered buildings |
| 504 | Concrete Installation |
| 506 | Field-Erected Tank Supply + Install |

Costs for in plant-piping, electrical bulks and instrumentation were factored as a percentage of mechanical equipment cost, based on Eastern Canadian projects of similar size and complexity (Table 21-6).

Costs for process plant mobile equipment were based on Ausenco historical database pricing, and include only a down payment, with annual payments separately included in the operating costs.

**Table 21-7: Process Plant Total Initial Capital Direct Cost by Discipline**

| Code | Discipline | Initial Direct Capital Costs (C$M) |
|---|---|---|
| A | Architectural | 8.0 |
| C | Concrete | 5.2 |
| E | Electrical Equipment | 5.5 |
| F | Platework | 5.9 |
| I | Instrumentation | 0.9 |
| M | Mechanical Equipment | 25.2 |
| O | Mobile Equipment | 0.1 |
| P | Pipework | 6.2 |
| Q | Electrical Bulks | 1.7 |
| S | Structural Steel | 7.0 |
| V | 3rd Party Packages | 4.8 |
| | **TOTAL DIRECT COSTS** | **70.5** |

Building supply costs (inclusive of HVAC and lighting) were based on new quotes for the Project and recent budget quotes from similar projects and adjusted to reflect the project sizing. Building costs are presented in Table 21-7.

**Table 21-8: Process Plant Building Costs**

| Area Description | Building Description | Building Type | Initial Capital (C$M) |
|---|---|---|---|
| Crushing, Stockpile and Reclaim | Stockpile Cover | Inflatable Tube | 0.6 |
| Grinding | Mill Building | Pre-Engineered | 3.7 |
| Desorption, Regeneration and Gold Room | Gold Room | Pre-Engineered | 0.4 |
| Reagents | Reagents Building | Fabric | 1.2 |
| Infrastructure and Facilities | Mill Office | Modular | 0.4 |
| | Mill Workshop | Fabric | 0.9 |
| | Reagents Storage | Fabric | 0.4 |
| | Laboratory | Containerized | 2.0 |
| | Control Room | Modular | 0.03 |

### 21.5.2.2 Construction Contracts

#### 21.5.2.2.1 Mechanical Equipment, Structural Steel, Electrical and Off-Plot Piping

Scope

The estimate allows for the supply and installation of:

- All new mechanical equipment for the process plant.
- All new steel work in the process plant.
- Electrical equipment in the process plant.
- Tailings and decant return pipelines and decant pumps (not including earthworks for the pipelines)

Quantities

The mechanical equipment list has been developed and equipment sized by process and mechanical engineering. The mechanical equipment was specified utilizing project specific equipment datasheets highlighting agreed upon process performance criteria and were accompanied by typical engineering specifications.

All structural steel quantities were estimated from quantity take-offs from the 3D model by the civil/structural department. Structural steel take-offs include light, medium, heavy, and extra-heavy structural steel designations and miscellaneous steel including grating and handrail and stair treads.

An electrical equipment list was developed based on the mechanical equipment list, load list, single line diagrams and general arrangement drawings.

Quantities for the overland pipelines were estimated from quantity take-offs from the 2D site layout drawings provided by Nordmin, with the routing as agreed with Nordmin.

Supply

Structural steel supply and fabrication (including delivery to site) as well as installation was quoted by contractors as part of the SMP package by providing them with a bill of quantities for completion of unit rates.

Electrical equipment pricing is sourced from budget quotations for all major equipment (E-rooms, MCCs/switchgear/transformers).

Supply costs for the tailings/decant return pipelines have been sourced from budget quotes from the SMPE contractors.

Installation

Mechanical equipment, electrical equipment installation, structural steel installation and overland piping installation was quoted by contractors as part of the SMP package by providing them with a bill of quantities for completion of unit rates for each designated mechanical equipment item. The returned price schedules include for the direct and indirect costs to install the agreed upon scope for the mechanical equipment scope.

The returned rates were compared and evaluated, and the selected contractor rates have been carried in the estimate.

#### 21.5.2.2.2 Pre-Engineered Buildings

Scope

The estimate allows for the supply and construction of the pre-engineered buildings within the process plant facilities.

Building cranes, HVAC and building electrical services have been included in the contractor scope of supply and installation.

Building Design

Building datasheets were developed to describe the basic requirements, including sizing, load requirements and features. Datasheets were included with the contract packages as a basis for detailed design.

Pricing

Various suppliers were approached for the detailed design, supply and installation packages for the pre-engineered buildings. The returned costs were compared and evaluated, and the selected suppliers' estimates were included in the estimate.

#### 21.5.2.2.3 Field-Erected Tanks

Scope

The estimate allows for the supply and installation of large diameter bolted tanks associated with the process plant facility.

Pricing

Detailed design, supply, freight, and installation of bolted tanks was quoted by contractors as part of the field-erected tanks package by providing them with a bill of quantities for the completion of unit rates.

The returned prices were compared and evaluated, and the selected contractor prices have been used in the estimate.

#### 21.5.2.2.4 Concrete Installation

Scope

The scope of the civil concrete works allows for all new concrete work, and concrete installation.

Quantities

All concrete quantities were estimated from quantity take-offs from the model s by the civil/structural department. MTOs for major structures including foundations, footings, walls, pedestals, slab on grade and elevated concrete, detailed excavation, detailed backfill have been developed based on these calculations.

Pricing

Budget pricing was sourced from the market for supply and delivery of locally produced concrete.

The basis for the total cost of installed concrete is the cost of materials supply and installation costs:

- the cost of materials includes formwork, required embedments and reinforcement steel
- the cost for labour includes categorized installation hours multiplied by the direct labour rate and distributable rate as quoted by the contractor

Concrete install rates inclusive of formwork, reinforcement steel detailed excavation and backfill were quoted by contractors as part of the concrete installation package by means of a schedule of rates. The returned price schedules included the direct and indirect costs to supply and install the agreed concrete scope.

The returned rates were compared and evaluated, and the selected contractor rates have been used in the estimate.

#### 21.5.2.3 Project Indirects

Indirect costs are those that are required during the project delivery period to enable and support the construction activities. Indirect costs include:

- Temporary construction facilities and services.
- Commissioning representatives and assistance.
- On site materials transportation and storage.
- Spares (commissioning, and insurance).
- First fills

The Project indirects have been based on budget quotes received for the Project (for contractor indirects and main spares items), quantity take-offs (for first fills) and factored allowances (for spares costs not provided by vendors).

Indirect costs are included in the overall process plant cost summary in Table 21-2.

#### 21.5.2.4 Project Delivery

The Project delivery cost has been calculated at based on built up estimates of the EPCM and commissioning services scope for the Goldboro process plant scope, as required to suit the project execution schedule.

Project delivery costs are included in the overall process plant cost summary in Table 21-2.

#### 21.5.2.5 Estimate Sources

The estimate for the process plant has been derived from a combination of sources, as specifically detailed in the paragraphs above.

Table 21-8 lists the amount from each of these sources in the overall process plant estimate. A total of 86% of the process plant estimate was obtained from either new budgetary quotes obtained for the FS, or from recent 2021 budgetary quotes obtained for other similar eastern Canada gold projects.

**Table 21-9: Process Plant Estimate Sources**

| Description | Estimate Value (C$M) | % of Total Direct + Indirect |
|---|---|---|
| Budgetary Quotes + Mid-2021 Database Pricing | 80.8 | 86% |
| Historical Pricing | 4.2 | 4% |
| Factored Allowances | 9.1 | 10% |
| **Total Direct + Indirect Capital Costs** | **94.1** | **100** |

### 21.5.2.6 Growth Allowance

Estimate growth:

- Is intended to account for items that cannot be quantified based on current engineering status, but which are empirically known to appear;
- Accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at a FS level;
- Pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the Project.

Where an allowance has been used that is the result of factoring, no growth has been applied, as the factor has been surmised from a total cost.

For the process plant, growth has been calculated by commodity and by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The capital cost growth allowance is presented in Table 21-9.

**Table 21-10: Process Plant Growth Allowances**

| Commodity Code | Discipline | Growth Applied |
|---|---|---|
| A | Architectural | 4% |
| B | Earthworks | n/a |
| C | Concrete | 4% |
| E | Electrical | 4% |
| F | Platework | 4% |
| I | Instrumentation | n/a |
| M | Mechanical Equipment | 4% |
| O | Mobile Equipment | 8% |
| P | Pipework (off-plot piping only) | 4% |
| Q | Electrical Bulks | n/a |
| S | Structural Steel | 4% |
| U | Project Indirects | 4% |
| V | Third party packages / Other | 4% |
| W | EPCM | 0% |

#### 21.5.2.7 Contingency Provision

Contingency accounts for the difference in costs from the estimated and actual costs of materials and equipment. Typically, these costs become more identifiable as the engineering design of the project advances.

The contingency cost is derived from total installed costs based on the level of uncertainty for each area. The amount of risk is assessed with due consideration of the preliminary level of design work, and the manner in which pricing was derived. Ausenco recommends a contingency of 15% for initial capital.

The total estimated contingency is C$14.1 million for the initial capital cost estimate.

The estimate contingency will not allow for the following:

- Abnormal weather conditions.
- Changes to market conditions affecting the cost of labour or materials.
- Changes of scope within the general production and operating parameters.
- Effects of industrial disputations.
- Financial modelling.
- Technical engineering refinement.
- Estimate inaccuracy.

#### 21.5.2.8 Exclusions

The following costs and scope were excluded from the capital cost estimate:

- Taxes.
- Scope changes and project schedule changes and the associated costs.
- Any facilities/structures not mentioned in the Project summary description.
- Geotechnical unknowns/risks.
- Financing charges and interest during the construction period.

### 21.5.3 Tailings Management Facility

Cost estimates for the initial CapEx, sustaining CapEx, and the closure and reclamation activities for the TMF and polishing pond have been developed based on the FS level design, the current understanding of the site conditions and permitting obligations. The cost estimates are based on neat line quantities and material take-offs from the design drawings, neat line AutoCAD modelling, unit rate development, and contractor quotes. Lump sum placeholder estimates have been applied were necessary. A 15% contingency is included in the total capital cost estimates. The cost estimates are presented in 2021 Canadian dollars and do not include inflation for future work.

The estimated CapEx includes for the following main items:

- Earthworks costs associated with foundation preparation, material processing and embankment construction for the TMF.
- Earthworks costs for the polishing pond, and miscellaneous infrastructure required for the TMF operations.
- Installation of a seepage collection drains, seepage collection sumps, and pump back systems to collect potential embankment seepage.

- Supply and installation of geotechnical instrumentation to monitor embankment performance during operations.
- Indirect costs associated with TMF and polishing pond construction quality assurance and quality control.
- The cost estimate assumes that earthfill and rockfill for the construction of the embankments will be sourced from the open pit mining operations and the material:
  - is NPAG;
  - can be segregated during mining;
  - is suitable for construction in the embankments;
  - will be delivered to the TMF during construction as part of the mining operations (i.e., an overhaul); and
  - will be available as required during the construction of the TMF.

### 21.5.4 Infrastructure Capital Costs

The total estimated capital cost for site infrastructure is approximately $73.0 million including indirect costs and contingency. All infrastructure required for the initial capital period is listed in Table 21-11.

The cost associated with the site electrical substation and on site distribution was estimated based on conceptual system design budget vendor quotes and benchmarked costs for the major components. These costs were estimated based on preliminary routes and benchmark costs sourced from Nordmin's internal database.

The costs associated with the internal access roads were based on earthworks quantities estimated from 3D models of the site topography, the preliminary general site layout and typical sections of road cross sections. The costs were estimated from local earthmoving contractors' quotes including unit costs.

The general facilities cost accounts for the costs associated with items such as the general office building, plant office, core storage, mine dry, truck shop and warehouse. These costs were estimated based on mine requirements and budget cost estimations from pre-engineered building suppliers.

The water supply cost accounts for the costs associated with the freshwater catchment system, storage pond, pipeline, and freshwater storage tanks in the Project Area. These costs were estimated based on conceptual system design and a combination of unit and benchmark costs for the major components sourced from Nordmin, Knight Piésold and GHD internal databases.

**Table 21-11: Infrastructure Capital Costs**

| Description | Initial ($M) | Sustaining ($M) | Total ($M) |
|---|---|---|---|
| General Mine Site Development and Grading | 6.8 | 2.2 | 9.0 |
| Open Pit Site Preparation | 2.0 | | 2.0 |
| Waste Area Site Preparation | 2.2 | 4.3 | 6.5 |
| Internal Access and Haul Roads | 10.8 | | 10.8 |
| General Office Building, Dry, Plant Office, Warehouse and Security Facilities | 13.8 | | 13.8 |
| Main Switchyard and power distribution | 4.1 | | 4.1 |
| Hazardous Waste Storage | 0.2 | | 0.2 |
| Fuel Storage | 1.0 | | 1.0 |
| Explosive Storage | 0.3 | | 0.3 |
| Propane Storage (Heating Purposes) | 0.3 | | 0.3 |
| Employee Accommodations | 6.8 | | 6.8 |
| General Site Services | 0.8 | | 0.8 |
| Pit Dewatering | 0.7 | | 0.7 |
| **Infrastructure Subtotal** | **49.8** | **6.5** | **56.2** |
| Indirects | 6.6 | | 6.6 |
| Contingency 15% | 9.2 | 1.0 | 10.2 |
| **Total Surface Infrastructure Capital Costs** | **65.6** | **7.4** | **73.0** |

### 21.5.5 Water Management and Treatment

Table 21-12 summarizes the cost estimate for water management and treatment for the Project.

**Table 21-12: Water Management and Treatment Capital Costs**

| Description | Pre-Production Period ($M) | Production Period ($M) | Total ($M) |
|---|---|---|---|
| **Water Management & Treatment** | | | |
| 1.0 NW Area (Waste Stockpiles, Pump Stations, Administrative Area, Mill) | 4.2 | 5.3 | 9.5 |
| 2.0 SW Area (Till Stockpiles, Employee Accomodations) | 3.1 | | 3.1 |
| 3.0 SE Area (Waste Stockpiles, Pump Station) | 3.0 | | 3.0 |
| 4.0 NE Area (Waste Stockpiles, Pump Station) | | 0.6 | 0.6 |
| 5.0 TMF Area | 3.3 | 3.6 | 6.9 |
| 6.0 NW Settling (SWM) Pond (South) | | 0.3 | 0.3 |
| 7.0 NW Settling (SWM) Pond (North) | | 0.2 | 0.2 |
| 8.0 SW Settling (SWM) Pond | 0.2 | | 0.2 |
| 9.0 SE Settling (SWM) Pond | 0.2 | | 0.2 |
| 10.0 NE Settling (SWM) Pond | | 0.2 | 0.2 |
| 13.0 Post Closure Water Treatment & Conveyance | | | 0.0 |
| 14.0 Surface Water Monitoring Plan | 0.2 | | 0.2 |
| 15.0 Groundwater Monitoring Plan | 0.1 | | 0.1 |
| **Subtotal Costs** | **14.4** | **10.2** | **24.5** |
| Contingency 15% | 2.1 | 1.5 | 3.7 |
| Total Cost Estimate | **16.5** | **11.7** | **28.2** |

### 21.5.6 Reclamation and Closure

Table 21-13 summarizes the cost estimate for reclamation and closure for the Project.

**Table 21-13: Reclamation and Closure Capital Costs**

| Description | Production Period ($M) | Closure Period ($M) | Total ($M) |
|---|---|---|---|
| **Reclamation and Closure** | | | |
| TMF | | 13.2 | 13.2 |
| Restoration of Water Management Infrastructure | | 2.9 | 2.9 |
| Water Monitoring Plans | | 1.8 | 1.8 |
| Dismantling and Disposal of Site Buildings | | 7.0 | 7.0 |
| Restoration of Dismantled Infrastructure Footprint | | 1.4 | 1.4 |
| Salvage Value, Process Facility | | (1.7) | (1.7) |
| Salvage Value, Mobile Equipment & Other furnishings | | (0.8) | (0.8) |

| Description | Production Period ($M) | Closure Period ($M) | Total ($M) |
|---|---|---|---|
| Re-vegetate Waste Dumps | 2.8 | 0.9 | 3.7 |
| Site Security | | 0.2 | 0.2 |
| **Subtotal Costs** | **2.8** | **24.9** | **27.7** |
| Indirect Costs and Contingency | 1.0 | 4.8 | 5.8 |
| **Total Costs** | **3.7** | **29.7** | **33.4** |

### 21.5.7 Other Capital

Table 21-14 summarizes the cost estimate for other items identified for the Project.

**Table 21-14: Other Capital Costs**

| Description | Total (C$M) |
|---|---|
| Reclamation and Closure Bond Costs | 7.6 |
| Fisheries Offsetting | 1.9 |
| Wetland Compensation | 7.3 |
| **Total Costs** | **16.9** |

Additional factors for indirect CapEx and contingency were not added to the above items.

The fish habitat offsetting costs are based on the following assumptions:

- An estimated 9,000 m of linear watercourse length directly under the infrastructure with an average width of 1 m.
- There are four main stream systems that have been identified that may be indirectly affected by management of mine contact water and resulting flow reductions. The total lengths of potential impact are based on current infrastructure placement. Total length of indirect impacts 4,160 m or 8,320 $m^2$, with an assumed 50% loss and requirement for offsetting 4,160 $m^2$.
- 4,218 $m^2$ have been assumed area for waterbody direct impact.
- Total for offsetting area calculation totals 17,378 $m^2$ multiplied by 2.
- $500,000/ha as a reasonable construction cost for offsetting project(s).
- Monitoring will be required for offsetting projects for up to 7-12 years. For the purposes of this assessment, $200,000 has been estimated for monitoring.

Wetland compensation estimate is based on 225 hectares of impacted wetlands with a compensation cost assumption of $32,500/ha.

### 21.5.8 Indirect Capital and Contingency

Indirect capital and contingency costs are expected to include EPCM fees, indirect construction costs, owner's costs, and contingency. Table 21-15 summarizes the indirect capital and contingency estimates included in the economic analysis.

**Table 21-15: Summary of Indirect and Contingency included in Initial Capital Cost Estimate**

| Area | Indirect Capital Amount | Contingency Amount |
|---|---|---|
| | C$M | C$M |
| Owner's Costs | 3.8 | 0.5 |
| *Mill labour during pre-production* | *0.8* | |
| *G&A labour during pre-production* | *2.2* | |
| Infrastructure, Power, Site Development | 17.0 | 9.2 |
| Process Plant | 23.6 | 14.1 |
| Mining | 0.0 | 2.6 |
| TMF | 1.0 | 3.2 |
| Water Management | 0.0 | 2.1 |
| Other | | |
| **Total Costs** | **45.4** | **31.7** |

## 21.6 Operating Costs

The operating cost estimate was prepared by the Consultants with an expected accuracy range of ± 15% weighted average accuracy of actual costs based on the third quarter of 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25, unless otherwise stated. Table 21-16 summarizes the Operating Cost Estimate for the Project.

**Table 21-16: Summary of Operating Cost Estimate**

| Item / Description | Units | Pre-Production Phase | Production/Operating Phase | Closure Phase | Total | $/t mined | $/t milled | $/oz Au recovered | $/oz Au Payable |
|---|---|---|---|---|---|---|---|---|---|
| **Production Data** | | | | | | | | | |
| Total Mined Tonnage | kt | 4,056 | 138,494 | | 142,550 | | | | |
| Mill Feed Tonnage – Mined | kt | 221 | 15,578 | | 15,799 | | | | |
| Mill Feed Tonnage – Processed | kt | 0 | 15,799 | | 15,799 | | | | |
| Recovered Gold Ounces | oz | 0 | 1,102,334 | | 1,102,334 | | | | |
| **Operating Cost** | | | | | | | | | |
| Open Pit Mining | k$ | Costs occurred during this Phase are captured in the Initial Capital Section, Pre-production mining, and Owner's Cost Estimate of the Indirect Capital Estimate | 691,038 | | 691,038 | 4.99 | 43.74 | 627 | 627 |
| Processing | k$ | | 212,465 | | 212,465 | | 13.45 | 193 | 193 |
| General and Administration | k$ | | 137,467 | | 137,467 | | 8.70 | 125 | 125 |
| Water Management and Treatment | k$ | | 18,321 | 34 | 18,355 | | 1.16 | 17 | 17 |
| **Subtotal Costs** | **k$** | | **1,059,290** | **34** | **1,059,324** | | **67.05** | **961** | **961** |
| **Off site Costs** | | | | | | | | | |
| Refining Charges | k$ | 0 | 3,626 | | 3,626 | | 0.2 | 3 | 3 |
| Transportation Charges | k$ | 0 | 1,032 | | 1,032 | | 0.1 | 1 | 1 |
| **Subtotal Costs** | **k$** | **0** | **4,658** | **0** | **4,658** | | **0.29** | **4** | **4** |
| **Total Costs** (Operating Costs + Selling Costs) | **k$** | **0** | **1,063,948** | **34** | **1,063,982** | | **67.35** | **965** | **966** |

### 21.6.1 Process Plant Operating Costs

The estimate aligns with the principles of a Class 3 FS level estimate with a ±15% accuracy according to the AACE. The processing operating cost estimate includes costs relating to reagent and consumable consumption, plant maintenance, power use, the laboratory, labour, and processing mobile equipment.

The average yearly operating costs amount to C$19.7 million, or $13.50/t of ore milled. A breakdown of this value and its unit costs is presented in Table 21-12.

**Table 21-17: Average Annual Process Operating Costs**

| Cost Centre | M$ CAD/a | $ CAD / t |
|---|---|---|
| Process Reagents and Consumables | 8.3 | 5.71 |
| Process Plant Maintenance | 0.9 | 0.64 |
| Process Power | 4.5 | 3.06 |
| Laboratory | 0.6 | 0.41 |
| Process Labour (O&M) | 4.9 | 3.38 |
| Process Mobile Equipment | 0.4 | 0.30 |
| **Total** | **19.7** | **13.50** |

#### 21.6.1.1 Basis of Operating Costs

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q4 2021 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.25 Canadian dollar per US dollar was assumed.
- Fuel costs were assumed as C$1.00/L for petroleum diesel and C$1.05/L for gasoline. These values were assumed to be the same as in the previous study phase completed in 2021.
- The annual power costs were calculated using a unit price of C$0.102/kWh.
- Mill production is set at a rate of 4,000 t/d or 1.46 Mtpa
- Off site gold refining, insurance, and transportation costs are excluded (separately accounted for in the financial model).
- Grinding media consumption rates have been estimated based on the ore characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork results at a nominal basis.
- Mobile equipment costs provide for fuel and maintenance and annual payments, not the initial down payment.

#### 21.6.1.2 Labour

Staffing was estimated by benchmarking against similar projects, with projected salary costs for each position provided by the Company. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total operational labour averages 54 employees. The organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-13.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. Salaries were estimated by benchmarking against similar projects. The total cost per employee includes various health and retirement benefits, as well as the bonuses to be allocated. The total labour cost is C$3.38/t milled, or approximately 25% of the total processing cost.

**Table 21-18: Operations and Maintenance Staffing Plan**

| Labour / Contractor Summary | #/Shift | # Shifts | Quantity |
|---|---|---|---|
| Graduate Metallurgist (Plant Metallurgist) | 1 | 2 | 2 |
| Mill Maintenance Superintendent | 1 | 1 | 1 |
| Process Superintendent (Mill) | 1 | 1 | 1 |
| **Mill Staff Total** | | | **4** |
| Mill Foreman | 1 | 4 | 4 |
| Crusher Operator | 2 | 4 | 8 |
| Grinding Operator | 1 | 4 | 4 |
| Leach/Reagents Operator | 1 | 4 | 4 |
| Elution / Reagents Operator | 1 | 4 | 4 |
| Gravity/Goldroom Foreman/Operator | 1 | 4 | 4 |
| **Mill Operations Total** | | | **28** |
| Lab Manager (Chief Assayer) | 1 | 1 | 1 |
| Metallurgical Tech | 1 | 4 | 4 |
| Assay Lab Tech | 1 | 4 | 4 |
| **Laboratory Total** | | | **9** |
| Maintenance Planner | 1 | 1 | 1 |
| Millwright/Fitter (certified journey person) | 2 | 2 | 4 |
| Electrician (certified journey person) | 1 | 2 | 2 |
| Control Room Operator | 1 | 2 | 2 |
| Apprentice | 2 | 2 | 4 |
| **Mill Maintenance Total** | | | **13** |
| **Overall Total** | **20** | **46** | **54** |

#### 21.6.1.3 Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services. An estimated 43,846,771 kWh are required per year, resulting in an annual power cost of C$4.47 million, or C$3.06/t milled. This represents 22% of the total processing operating costs.

#### 21.6.1.4 Reagents and Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Major reagent costs were obtained from vendor quotations to the Project, including activated carbon, antiscalant, leach aid, ball mill media, ferric sulphate, copper sulphate, flocculant, hydrochloric acid, hydrated lime, sodium cyanide, sodium hydroxide, sodium metabisulphite (SMBS), and sulphamic acid. Other reagent cost was obtained through benchmarking for similar projects performed by Ausenco.

Other consumables (e.g., liners for the primary crusher, ball mill, and ball media for the mills) were estimated using:

- Metallurgical testing results (bond abrasion testing)
- Ausenco's in-house calculation methods, including simulations
- Forecast nominal power consumption

The breakdown of consumptions and costs is provided in Table 21-14.

**Table 21-19: Reagents/Consumables Annual Rates and Costs**

| Items | Unit | Unit Cost (CAD) | Cons. per Year | Cost (CAD/a) | Cost (CAD/t) |
|---|---|---|---|---|---|
| **3100 – Crushing, Stockpile and Reclaim** | | | | **801,904** | **0.55** |
| Cheek and swing jaw set | set | 8,712 | 3.0 | 26,135 | 0.02 |
| Secondary crusher mantle/bowl liner | set | 15,705 | 6.0 | 94,231 | 0.06 |
| Tertiary crusher mantle/bowl liner | set | 23,846 | 6.0 | 143,077 | 0.10 |
| Secondary crusher screen deck | set | 19,231 | 12.0 | 230,769 | 0.16 |
| Tertiary crusher screen deck | set | 25,641 | 12.0 | 307,692 | 0.21 |
| | | | | | |
| **3200 – Grinding** | | | | **2,063,705** | **1.41** |
| Ball mill media | t | 1,590 | 1236 | 1,964,833 | 1.35 |
| Ball Mill Liner | set | 149,806 | 0.66 | 98,872 | 0.07 |
| | | | | | |
| **3300 – Gravity and Intensive Cyanidation** | | | | **21,583** | **0.01** |
| NaOH | t | 680 | 0 | - | - |
| NaCN | t | 2,875 | 6 | 17,553 | 0.01 |
| Leach Aid | t | 12,000 | 0 | 4,030 | 0.00 |
| | | | | | |
| **3400 – Leach/CIP** | | | | **1,845,897** | **1.26** |
| Hydrated lime | t | 450 | 945 | 425,317 | 0.29 |
| NaCN | t | 2,875 | 409 | 1,175,300 | 0.81 |
| Activated carbon | t | 4,200 | 58 | 245,280 | 0.17 |
| | | | | | |
| **3510 – Desorption** | | | | **803,301** | **0.55** |
| HCl | t | 480 | 44 | 21,286 | 0.01 |
| NaCN | t | 2,875 | 3 | 7,469 | 0.01 |
| NaOH | t | 680 | 105 | 71,393 | 0.05 |
| Propane | kL | 450 | 813,158 | 572,951 | 0.39 |
| Sulphamic acid | t | 1,100 | 7 | 8,030 | 0.01 |
| Antiscalant | t | 4,500 | 15 | 65,700 | 0.05 |

| Items | Unit | Unit Cost (CAD) | Cons. per Year | Cost (CAD/a) | Cost (CAD/t) |
|---|---|---|---|---|---|
| **3540 – Goldroom** | | | | **35,835** | **0.02** |
| Borax | t | 1,530 | 3.9 | 5,914 | 0.00 |
| Silica | t | 732 | 1.9 | 1,414 | 0.00 |
| Nitre | t | 4,269 | 0.3 | 1,375 | 0.00 |
| Sodium carbonate | t | 960 | 0.3 | 309 | 0.00 |
| Crucibles | unit | 1,573 | 12.0 | 18,876 | 0.01 |
| Propane | L | 0.70 | 11,279 | 7,947 | 0.01 |
| | | | | | |
| **3600 – Detox / Tailings** | | | | **2,691,559** | **1.84** |
| Hydrated lime | t | 450 | 1171 | 527,144 | 0.36 |
| SMBS | t | 980 | 1421 | 1,392,987 | 0.95 |
| Copper sulphate | t | 3,200 | 95 | 304,751 | 0.21 |
| Ferric sulphate | t | 380 | 344 | 130,599 | 0.09 |
| Flocculant | t | 5,100 | 66 | 336,078 | 0.23 |
| | | | | | |
| **3800 – Plant Services** | | | | **135,722** | **0.09** |
| Antiscalant | t | 4,500 | 7 | 32,850 | 0.02 |
| Propane | L | 0.70 | 146,000 | 102,872 | 0.07 |
| | | | **Total** | **$8,343,035** | **$5.71** |
| | | | *Reagents - Subtotal* | $5,477,426 | **$3.75** |
| | | | *Consumables - Subtotal* | $900,776 | **$0.62** |
| | | | *Grinding Media - Subtotal* | $1,964,833 | $1.35 |

Reagents and consumables represent approximately 44% of the total process operating cost at C$5.71/t milled.

#### 21.6.1.5 Maintenance

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average factor from 3% to 4%. The factor was applied to mechanical equipment, platework, and piping. The total maintenance consumables operating cost is C$0.63/t milled, or approximately 5% of the total process operating cost.

#### 21.6.1.6 Laboratory Services

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year, estimated by Ausenco. Assay costs include mine samples, plant solid samples taken from various samplers throughout the plant, solution samples, tests on the loaded, barren, and regenerated carbon, bullion bar testing, cyanide detoxification sampling, and environmental sampling and assaying. The laboratory and assays comprise approximately 3% of the total process operating cost, and the forecasted annual requirement for internal assays will be 12,900 for the processing plant. An annual cost of $600,751 is estimated for the laboratory and assay activities.

#### 21.6.1.7 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares and tires, and annual registration and insurance fees. As mentioned, the prices of fuel used in these calculations are C$1.00/L for petroleum diesel and C$1.05/L for gasoline.

The cost of operating and maintaining the processing mobile vehicles is estimated as C$0.14/t milled, or 1% of the processing operating costs.

Financing costs for the first five years of operation, calculated based on a 5% annual interest rate, were estimated at C$0.16/t milled.

### 21.6.2 Open Pit Mining Operating Costs

The open pit mining operating cost estimate is summarized in Table 21-18. The costs and unit rates are derived from the average unit rates from two mining contractor budget quotes and applied to the FS Mine Plan. At the time of requesting quotes, a $1.00/L unit rate was used for diesel fuel pricing. The costs are presented without any contingency allowance.

**Table 21-20: Open Pit Mining Operating Costs Summary – by Activity**

| Description | Total LOM C$M | C$/t mined |
|---|---|---|
| Drilling | 97.0 | 0.68 |
| Blasting | 94.1 | 0.66 |
| Loading | 73.8 | 0.52 |
| Hauling | 303.9 | 2.13 |
| Support Equipment | 73.6 | 0.52 |
| Stockpile Rehandle | 1.5 | 0.01 |
| Subtotal | 643.9 | 4.52 |
| Contractor Management | 48.2 | 0.34 |
| Owner Technical Team and Costs | 26.0 | 0.18 |
| **Subtotal Costs** | **718.1** | **5.04** |
| Costs Occurring in Pre-Production Period | 25.5 | 6.30 |
| Sustaining Capital during Production | 1.6 | |
| **Total Operating Costs, Production Period** | **691.0** | **4.99** |

- Drilling costs include cost assumptions for wall control drilling.
- Overburden material was assumed not to require drilling and blasting.
- Blasting costs are based on the following unit rates.
  - Explosive products unit cost of 0.57$/t.
  - Blasting services unit cost of 0.13$/t.
- Loading and hauling costs are based on the following average contractor equipment hourly unit rates.
  - Hauling – 236 to 395$/hr, depending on contractor and hauling unit type.
  - Loading – 315 to 518$/hr, depending on contractor and loading unit type.
- Stockpile rehandling costs are based on 1.50$/t unit cost.
- Contractor costs were averaged between the two budget quotes and include such items as Management team, supervision, technical, administrative, health & safety.

- Owner's cost include Owner's technical team, allowances for pit dewatering via in-pit sump pumping, blast monitoring, horizontal drain holes, crush material for road maintenance. Ore control and blasthole sampling are deemed covered within the Process Operating Cost estimate.
- Sustaining capital includes purchase of pit dewatering pumps

**Table 21-21: Open Pit Mining, Owner's Technical Team Estimate**

| Description | Employees |
|---|---|
| Technical Services Superintendent | 1 |
| Senior Mining Engineer | 2 |
| Mine Planner | 2 |
| Senior Geologist | 1 |
| Production Geologist | 1 |
| Grade Control Technician | 2 |
| **G&A Total** | **9** |

### 21.6.3 Power Costs

The annual power cost is calculated from the overall plant power draw determined from the site equipment load list. An average power consumption for the site is based on an estimate that the average power demand is approximately 75% of the projected peak power demand. The cost for power is determined as C$0.102/kWh based on historical data.

### 21.6.4 Water Treatment

The water treatment operating cost estimate are summarized in Table 21-22

**Table 21-22: Water Treatment Operating Cost Estimate**

| Description | Total Operating Cost C$M |
|---|---|
| NW Area (Waste Stockpiles, Pump Stations, Administrative Area, Mill) | 7.0 |
| SW Area (Till Stockpiles, employee accommodation Area) | 0.1 |
| SE Area (Waste Stockpiles, Pump Station) | 1.0 |
| TMF Area | 7.9 |
| Surface Water Monitoring Plan | 1.3 |
| Groundwater Monitoring Plan | 1.2 |
| **Total Operating Costs** | **18.4** |
| **Cost Per Tonne Milled** | **1.16** |

The costs above include items for the following, occurring during the production/operations period:

- Operation cost of containerized potable water treatment system at 16 $m^3$/h.
- Operation costs of raw and potable pumps.
- Waste rock pile – water treatment operation costs.
- TMF treatment operations costs.

- Operation cost of containerized sewage wastewater treatment unit at peak design flowrate of 16 $m^3$/h.
- Assumes potable water treatment system and sewage wastewater treatment system are required for 11 years of operation and 2 years during closure.
- It is assumed that the generated sludge as the result of WTS will be pumped back to the settling ponds.

### 21.6.5 General and Administrative

G&A operating costs shown in Table 21-23 have been estimated based on assumed personnel requirements. The estimate accounted for 20 personnel. The general services include general management (not accounted for in mining and processing), accounting, human resources, purchasing, health and safety, environment, and employee accommodations and rotation costs. The costs are presented without any contingency allowance.

**Table 21-23: General and Administrative Operating Cost Estimate**

| Description | Total Operating Cost Estimate C$M | C$/t milled |
|---|---|---|
| Labour | 20.9 | |
| Services | 54.7 | |
| Employee Accommodation Costs | 52.7 | |
| Rotation Costs | 9.2 | |
| **Total Costs** | **137.5** | **8.70** |

The following assumptions have been made with respect to the G&A cost estimate (Table 21-23):

- Diesel fuel has been considered at a fixed rate of $0.97/L.
- Labour costs included a variable % burden assumption, based on position. The total cost per employee includes various health and retirement benefits, as well as bonus assumptions to be allocated.
- Employee accommodations and Rotation costs were estimated based on an average number of persons per year of the operation. Rotation costs for owner's team assumed travel costs from Halifax to site.

Table 21-24 provides the G&A labour and salary estimate.

**Table 21-24: General and Administrative Labour and Salary Estimate**

| Description | Employees |
|---|---|
| General Manager | 1 |
| Administrative Assistant | 1 |
| Human Resource Coordinator | 1 |
| Human Resource Staff | 1 |
| Controller / Accountant | 1 |
| Payroll Coordinator | 1 |
| Payroll Staff | 1 |
| Purchasing Agent | 1 |
| Warehouse Staff | 4 |

| Description | Employees |
|---|---|
| Health and Safety Coordinator | 1 |
| First Aid Attendants | 4 |
| IT Staff | 1 |
| Environmental Manager | 1 |
| Environmental Technician | 1 |
| **G&A Total** | **20** |

### 21.6.6 Selling Costs

The estimated selling costs are shown in Table 21-25.

**Table 21-25: Selling Cost Estimate**

| Description | Total LOM |
|---|---|
| Refining Charges | $3.6M |
| Transportation Charges | $1.0M |
| **Total Costs** | **$4.7M** |
| | $4.23 / ounce of gold |

The estimate is based on the following assumptions, which are based on existing contracts of the Company for the shipment and refining of gold doré:

- Weekly shipments.
- Flat rate transportation cost of $4,537/oz.
- Insurance charge of $0.31/$1,000.
- Freight charge of approximately $10.12/kg.
- Minimum refining charge of $1,500 per shipment.
- Deduction of $0.15/oz.

# 22. ECONOMIC ANALYSIS

## 22.1 Introduction

An economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the services performed by:

- Nordmin for open pit mining and surface infrastructure and their associate consultants at Optimize for the pit slope stability.
- Knight Piésold for TMF related matters.
- GHD for site water management.
- Ausenco for processing and metallurgy.
- Lorax for geochemistry.
- MEL for consultation, permitting and wetland compensation.

The Company provided the inputs with respect to the tax impact of the economic model, including the calculation of federal and provincial income taxes, provincial mining taxes, and available tax attributes that are applicable to the Project.

The Project includes two open pits and associated surface infrastructure to support the mine operations (i.e., maintenance and office facilities), water management features, a ROM stockpiling area, processing facility, a TMF, and an employee accommodation facility.

The economic model for the Project indicates a pre-tax free cash flow of $755.1 million over a mine life of approximately 11 years, a pre-tax NPV 5% of $483.8 million and a pre-tax IRR of 31.2%. On an after-tax basis, the Project could generate free cash flow of $529.0 million, and after-tax NPV (5%) of $328.2 million and an after-tax IRR of 25.5%. The Project is most sensitive to commodity prices. Table 22-1 summarizes the Project economics for the described base case.

**Table 22-1: Summary of Economic Analysis Results**

| ITEM | VALUE | UNITS |
|---|---|---|
| **Financial Analysis** | | |
| Gold Price Assumption | 1,600 | US$/oz |
| US$:C$ Exchange | 1:1.25 | |
| Gold Price Assumption – C$ | 2,000 | C$/oz |
| Pre-Tax NPV 5%[1] | 483.8 | C$M |
| Pre-Tax IRR[1] | 31.2 | % |
| Pre-Tax Payback[2] | 2.7 | years |
| After-Tax NPV 5%[1] | 328.2 | C$M |
| After-Tax IRR[1] | 25.5 | % |

| ITEM | VALUE | UNITS |
|---|---|---|
| After-Tax Payback[2] | 2.9 | years |
| Pre-Tax Unlevered Free Cash Flow | 755.1 | C$M |
| After-Tax Unlevered Free Cash Flow | 528.9 | C$M |
| LOM Income and Provincial Mining Taxes | 226.2 | C$M |
| Average Operating Cash Cost per Ounce Sold [3] | **C$966**<br>**US$773** | **$/oz** |
| Average All-In Sustaining Cost per Ounce Sold [3] | **C$1,062**<br>**US$849** | **$/oz** |
| **Production Data** | | |
| Life of Mine | Production Period<br>10.9<br>Pre-production Period<br>1 | Years |
| Processing Rate | 4,000 / 1.46 | tpd / Mtpa |
| Recovered Gold | 1.1 | Moz |
| Average Gold Recovery | 95.8 | % |
| Pre-production Mined Tonnage | 4.1 | Mt |
| Total Mined Tonnage (including pre-production) from Open Pit Mining | 142.6 | Mt |
| Total Milled Tonnage from Open Pit Mining | 15.8 | Mt |
| Overall Strip Ratio | 8.0 | waste:ore |
| Average Annual Gold Production | 100 | koz |
| Average Mill Feed Grade | 2.26 | g/t gold |

[1] Discounting period uses mid-period discounting interval, discounting occurs during construction / pre-production period

[2] Payback period is relative to the start of production, which is deemed to occur at the start of Year 1 of production

[3] Refers to "Non-IFRS Financial Measures". Please refer to the "Non-IFRS Financial Measures" section 22.7 below for more information.

## 22.2 Cautionary Statement

The results of the Economic Analysis are based on forward-looking information that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

Forward-looking statements in this section include, but are not limited to:

- Future prices of gold.
- Currency exchange rate fluctuations.
- Estimation of Mineral Reserves.
- Realization of Mineral Reserve Estimate.
- Estimated costs and timing of initial and sustaining capital expenditures and operating costs.

## 22.3 Principal Assumptions

The cash flow estimate includes only revenue, costs, taxes, and other factors applicable to the Project. Corporate obligations, financing costs (other than for the purposes of surety bonding), and taxes at the corporate level are excluded.

The economic model was prepared from estimated mining schedules developed on an annual basis with the technical assumptions outlined in the previous sections, together with the economic assumptions and estimated capital and operating costs described in Section 21. The cash flow model was based on the following:

- All costs are reported in Canadian dollars and referenced as '$', unless otherwise stated.
- As discussed in further detail in Section 19, a constant commodity pricing of C$2,000 per ounce of gold was assumed in the economic analysis, based on a long-term gold price assumption of US$1,600 per ounce and a constant exchange rate assumption of US$1: C$1.25.
- Annual gross revenue is determined by applying estimated metal prices with payable metal assumption to the annual recovered metal estimated for each operating year.
- No cost escalation beyond 2021.
- Constant 2021 dollar analysis, no provision for potential future inflation.
- Exploration costs for growth are deemed outside of the Project.
- Any additional Project study costs have not been included in the analysis.
- Excludes potential land acquisition costs for areas of planned mine and process infrastructure, if any.
- Reclamation costs and requirements for a reclamation performance bond have been included in the economic analysis, including progressive reclamation over the LOM.
- It has been assumed that the reclamation performance bond would be secured by a surety bond product or similar insurance instrument with an annual financing cost of approximately 2% based on the Company's existing surety bond products.
- Salvage value has been included in the economic analysis and applied to the Closure and Reclamation costs.
- Financing is assumed to be on a 100% equity basis; no debt or related financing costs have been included in the economic analysis.

## 22.4 Taxes and Royalties

There are no third-party royalties or gold streams on the Project.

The economic analysis of the Project has also been completed on an after-tax basis. It must be noted that there are many potential complex factors that affect the taxation of a mining project. The calculation of federal and provincial income and mining taxes, including the application of tax depreciation in the FS economic analysis, are preliminary like pre-tax FS economics and are intended only to give a general indication of the potential tax implications of the Project.

As of the time of this FS, the Project will be subject to the following taxes as they relate to the Project:

- A Canadian federal income tax rate of 15%.
- A Nova Scotia provincial income tax rate of 14%.
- A Nova Scotia provincial mining tax calculated as 1% on the Net Smelter Return (NSR) of gold sales.

The estimated federal and provincial income taxes were broadly calculated by deducting from gross revenues operating and reclamation costs and claiming certain tax deductions such as capital cost allowances (CCA), development expense deductions and exploration expense deductions. No corporate level costs were included in any tax estimates. The FS incorporated certain tax pools of the corporate entity that owns the Project (which is wholly-owned by Anaconda), including unused non-capital losses and tax pools relating to development and exploration expenses.

## 22.5 Economic Results

The results of the economic analysis are derived from the LOM schedule presented in Section 16, the processing and recovery methods discussed in Section 17, and capital and operating costs presented in Section 21.

Table 22-2 summarizes the production, capital and operating cost inputs for the economic analysis.

The estimate of initial capital costs is $271.1 million, including amounts for indirect costs and contingency, and as outlined in Table 22-2. A contingency of $31.7 million has been included in the estimate of initial capital costs, which amounts to 16% of the direct initial capital costs ($193.9 million), or 11% of the total initial capital estimate ($278.5 million).

The sustaining capital, including rehabilitation and closure costs, fisheries and wetland compensation and the reversal of upfront working capital, is estimated at $113.4 million over the life of the mine. Details of the estimate are shown in Table 22-2.

The operating costs, detailed in Table 22-2 , are the costs occurring during the production phase, are estimated at $67.05/t of material processed.

Figure 22-1 shows the cash flow model results. The cash flow is presented in Table 22-3.

Table 22-2: Summary of Input into the Economic Analysis (columns may not sum exactly due to rounding)

| Item / Description | Units | Pre-Production Phase | Production Phase | Closure Phase | Total | $/t mined | $/t milled | $/oz Au recovered | $/oz Au Payable |
|---|---|---|---|---|---|---|---|---|---|
| **1 – Production Data** | | | | | | | | | |
| Total Mined Tonnage | kt | 4,056 | 138,494 | | 142,550 | | | | |
| Mill Feed Tonnage – Mined | kt | 221 | 15,578 | | 15,799 | | | | |
| Mill Feed Tonnage – Processed | kt | 0 | 15,799 | | 15,799 | | | | |
| Recovered Gold Ounces | oz | 0 | 1,102,334 | | 1,102,334 | | | | |
| **2 – Operating Cost** | | | | | | | | | |
| Mining – OP | k$ | Costs occurred during this Phase are captured in the Initial Capital, Pre-production mining, and Owner's Cost Estimate of the Indirect Capital Estimate | 691,038 | | 691,038 | 4.99 | 43.74 | 627 | 627 |
| Processing | k$ | | 212,465 | | 212,465 | | 13.45 | 193 | 193 |
| General and Administration | k$ | | 137,467 | | 137,467 | | 8.70 | 125 | 125 |
| Water Management and Treatment | k$ | | 18,321 | 34 | 18,355 | | 1.16 | 17 | 17 |
| **1 – Subtotal Costs** | **k$** | | **1,059,290** | **34** | **1,059,324** | | **67.05** | **961** | **961** |
| **2 – Off site Costs** | | | | | | | | | |
| Refining Charges | k$ | 0 | 3,626 | | 3,626 | | 0.2 | 3 | 3 |
| Transportation Charges | k$ | 0 | 1,032 | | 1,032 | | 0.1 | 1 | 1 |
| **2 – Subtotal Costs** | **k$** | **0** | **4,658** | **0** | **4,658** | | **0.29** | **4** | **4** |
| **Subtotal Costs** (Operating Costs + Selling Costs) | **k$** | **0** | **1,063,948** | **34** | **1,063,982** | | **67.35** | **965** | **966** |
| **3 – Capital Cost** | | | | | | | | | |
| **Capital Cost Estimate** | | **Initial CapEx** | **Sustaining CapEx** | | **TOTAL** | | | | |
| Open Pit Mining | k$ | 25,503 | 1,554 | | 27,057 | | | | |
| Process Plant | k$ | 70,458 | 0 | | 70,458 | | | | |
| Tailings Management | k$ | 20,624 | 42,430 | | 63,054 | | | | |
| Infrastructure and Site Development | k$ | 49,769 | 7,423 | | 57,192 | | | | |
| Water Management & Treatment | k$ | 14,378 | 11,698 | | 26,075 | | | | |
| General Site Equipment | k$ | 1,126 | | | 1,126 | | | | |
| Employee Accommodations | k$ | 12,073 | | | 12,073 | | | | |
| **Subtotal Capital Costs** | **k$** | **193,930** | **63,105** | **0** | **257,035** | | | | |
| Indirect CapEx | k$ | 45,391 | included | | 45,391 | | | | |
| *Mill Labour during pre-production* | *k$* | *790* | | | *790* | | | | |
| *G&A during pre-production* | *k$* | *2,183* | | | *2,183* | | | | |
| *Other Indirects* | *k$* | *14,618* | | | *14,618* | | | | |
| *EPCM* | *k$* | *27,799* | | | *27,799* | | | | |
| Contingency | k$ | 31,729 | included | included | 31,729 | | | | |
| **Subtotal Capital Costs** | **k$** | **271,050** | **63,105** | **0** | **334,155** | | | | |
| Rehabilitation & Closure, Bond Cost | k$ | 700 | 10,055 | 30,294 | 41,049 | | | | |
| Other CapEx – Habitat Compensation | k$ | 0 | 9,253 | | 9,253 | | | | |
| Working Capital | k$ | 6,703 | -6,703 | | 0 | | | | |
| **Total Capital Costs with Contingency** | **k$** | **278,453** | **75,710** | **30,294** | **384,457** | | | | |
| **4 – All-in Sustaining Costs, Pre-Tax*** (Operating Costs + Off site Costs + Sustaining Capital + Closure Costs) | | | | | **1,169,986** | | | | **1061** US$ 849 |
| **5 – Tax Estimation** | | | | | | | | | |
| Income Taxes (Federal & Provincial) | k$ | 0 | 215,747 | -11,554 | 204,193 | | | | |
| Mining Taxes (Nova Scotia) | k$ | 0 | 21,989 | 0 | 21,989 | | | | |
| **Subtotal Costs** | **k$** | **0** | **237,736** | **-11,554** | **226,182** | | **14.32** | **205** | **205** |

*Refers to a non-IFRS financial measure. Refer to the "Non-IFRS Financial Measures" in section 22.7 for more information on such non-IFRS financial measure.

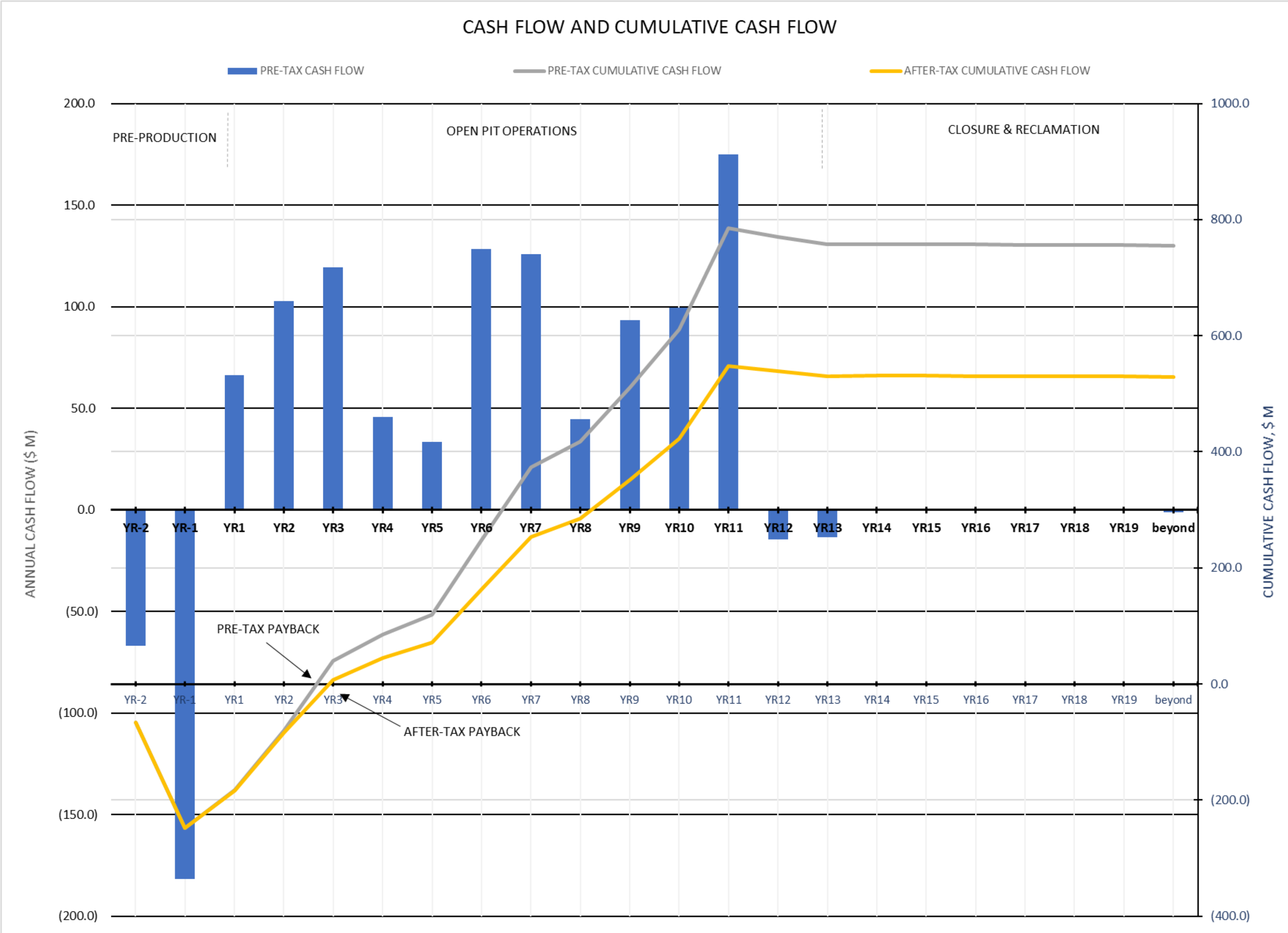



*Figure 22-1: Cash flow model results*

**Table 22-3: Cash Flow Model**

| | Total | Units | PRE-PRODUCTION | | RAMP-UP | PRODUCTION | | | | | | | | | | RECLAMATION & CLOSURE | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | YR-2 | YR-1 | YR1 | YR2 | YR3 | YR4 | YR5 | YR6 | YR7 | YR8 | YR9 | YR10 | YR11 | YR12 | YR13 | YR14 | YR15 | YR16 | YR17 | YR18 | YR19 | beyond |
| Mined Tonnage | 142,550 | kt | 0.0 | 4,055.6 | 11,515.3 | 14,569.6 | 11,753.6 | 15,681.2 | 18,924.0 | 10,593.8 | 12,628.6 | 18,047.9 | 13,829.1 | 8,839.5 | 2,111.5 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | |
| Mill feed production tonnage | 15,799 | kt | 0.0 | 0.0 | 1,269.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,460.0 | 1,389.9 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | 0.0 | |
| Mill feed head grades, Au | 2.26 | g/t | 0.00 | 0.00 | 2.29 | 2.33 | 2.54 | 1.82 | 1.87 | 2.57 | 2.58 | 1.95 | 2.26 | 2.14 | 2.59 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | |
| Contained Ounce Production | 1,150,205 | oz | 0.00 | 0 | 93,283 | 109,403 | 119,140 | 85,494 | 87,543 | 120,616 | 120,933 | 91,300 | 106,257 | 100,536 | 115,700 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Recovered Ounce Production | 1,102,334 | oz | 0 | 0 | 89,417 | 104,929 | 114,553 | 81,343 | 83,362 | 116,013 | 116,327 | 87,064 | 101,822 | 96,174 | 111,328 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Assumptions | | | | | | | | | | | | | | | | | | | | | | | | |
| Commodity price, Au | 1,600 | US$/oz | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | 1,600 | |
| | 2,000 | C$/oz | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 | 2,000 |
| Exchange Rate Factor | 1.25 | | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | 1.25 | |
| Gross Revenue | 2,203,565 | kC$ | 0 | 0 | 178,746 | 209,753 | 228,992 | 162,605 | 166,641 | 231,911 | 232,537 | 174,042 | 203,542 | 192,252 | 222,545 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Selling Costs | 4,658 | k$ | 0 | 0 | 411 | 428 | 440 | 402 | 404 | 443 | 444 | 408 | 424 | 418 | 435 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Operating Costs | 1,059,324 | k$ | 0 | 0 | 80,436 | 96,712 | 91,566 | 110,016 | 125,633 | 91,975 | 100,839 | 122,541 | 104,839 | 85,958 | 48,777 | 17 | 17 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Sustaining Capital Costs (incl. Contingency) | 63,105 | k$ | 0 | 0 | 4,534 | 7,225 | 15,544 | 4,243 | 4,243 | 9,080 | 4,875 | 4,875 | 4,243 | 4,243 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Capital Costs (incl. Contingency) | 271,050 | k$ | 66,681 | 174,694 | 29,676 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Closure Costs, incl bond costs | 41,049 | k$ | 207 | 493 | 493 | 557 | 557 | 626 | 1,598 | 608 | 608 | 1,784 | 585 | 2,082 | 557 | 14,714 | 13,424 | 143 | 141 | 139 | 137 | 135 | 133 | 1,327 |
| Other Costs | 9,253 | k$ | 0 | 0 | 1,293 | 2,109 | 1,463 | 1,463 | 1,463 | 1,463 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Working Capital | 0 | k$ | 0 | 6,703 | -4,469 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | -2,234 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Duties & Taxes | 226,182 | k$ | 0 | 0 | 1,783 | 2,093 | 29,111 | 8,345 | 6,323 | 36,749 | 35,469 | 12,144 | 26,945 | 29,047 | 49,727 | -5,722 | -4,981 | -851 | 0 | 0 | 0 | 0 | 0 | 0 |
| **Cash flow results** | | | | | | | | | | | | | | | | | | | | | | | | |
| Pre-tax cash flow | 755,126 | k$ | -66,887 | -181,890 | 66,371 | 102,722 | 119,422 | 45,855 | 33,300 | 128,342 | 125,773 | 44,435 | 93,450 | 99,551 | 175,012 | -14,731 | -13,441 | -143 | -141 | -139 | -137 | -135 | -133 | -1,327 |
| Cumulative Pre-Tax Cash Flow | | k$ | -66,887 | -248,777 | -182,407 | -79,685 | 39,737 | 85,592 | 118,892 | 247,234 | 373,007 | 417,442 | 510,892 | 610,443 | 785,454 | 770,724 | 757,283 | 757,139 | 756,998 | 756,859 | 756,722 | 756,586 | 756,453 | 755,126 |
| Discounted Pre-Tax Cash Flow | 483,790 | M$ | -66,887 | -177,506 | 61,687 | 90,926 | 100,675 | 36,816 | 25,463 | 93,463 | 87,230 | 29,350 | 58,787 | 59,643 | 99,860 | -8,005 | -6,956 | -71 | -66 | -62 | -58 | -55 | -51 | -390 |
| After-tax cash flow | 528,944 | k$ | -66,887 | -181,890 | 64,587 | 100,629 | 90,311 | 37,510 | 26,977 | 91,593 | 90,304 | 32,290 | 66,506 | 70,504 | 125,285 | -9,009 | -8,460 | 707 | -141 | -139 | -137 | -135 | -133 | -1,327 |
| Cumulative After-Tax Cash Flow | | k$ | -66,887 | -248,777 | -184,190 | -83,561 | 6,750 | 44,260 | 71,237 | 162,830 | 253,134 | 285,425 | 351,931 | 422,434 | 547,719 | 538,710 | 530,250 | 530,957 | 530,816 | 530,677 | 530,540 | 530,404 | 530,271 | 528,944 |
| Discounted After-Tax Cash Flow | 328,202 | M$ | -66,887 | -177,506 | 60,029 | 89,073 | 76,134 | 30,116 | 20,628 | 66,701 | 62,631 | 21,329 | 41,837 | 42,240 | 71,486 | -4,896 | -4,378 | 349 | -66 | -62 | -58 | -55 | -51 | -390 |

An NPV sensitivity calculation was performed at various discount rates, using mid-period discounting. The IRR was also calculated, which is defined as the rate at which the NPV of the cash flow equals zero. The payback period of the Project is defined as the point when the cumulative cash flow becomes positive.

Table 22-4 summarizes the economic indicators, both pre-tax and after-tax, for the estimated cash flow model.

**Table 22-4: Economic Indicators**

| Economic Indicators | Units | Pre-Tax | After-Tax |
|---|---|---|---|
| Payback Period *(from start production)* | Years | 2.7 | 2.9 |
| IRR | % | 31.2 | 25.5 |
| NPV @ 5% (base case) | C$M | 483.8 | 328.2 |
| NPV @ 8% | C$M | 369.7 | 243.4 |
| NPV @ 10% | C$M | 308.1 | 197.4 |

## 22.6 Sensitivity Analysis

The gold price sensitivity on an after-tax basis is presented in Table 22-5. Sensitivities to changes in other parameters are shown in Table 22-6, on an after-tax basis.

The Project's after-tax NPV was most sensitive to the factors impacting revenue such as gold commodity pricing and currency exchange rates, followed by the operating costs of the Project.

**Table 22-5: After-Tax Valuation Sensitivities to Gold Price**

| Description | | Unit | NPV (C$ M) | | | | |
|---|---|---|---|---|---|---|---|
| % Variation | | % | -20% | -10% | **0%** | +10% | +20% |
| Au Price[2] | | US$/oz | 1,280 | 1,440 | **1,600** | 2,200 | 2,400 |
| | | C$/oz | 1,600 | 1,800 | **2,000** | 2,200 | 2,400 |
| Discount Rate | 0% | C$M | 219.2 | 374.1 | 528.9 | 683.8 | 838.6 |
| | 3% | C$M | 140.0 | 269.5 | 398.0 | 526.3 | 654.5 |
| | 5% | C$M | 98.0 | 213.8 | **328.2** | 442.2 | 556.2 |
| | 8% | C$M | 47.2 | 146.2 | 243.4 | 340.1 | 436.6 |
| | 10% | C$M | 19.7 | 109.6 | 197.4 | 284.6 | 371.6 |
| IRR | | % | 11.7 | 18.9 | **25.5** | 31.7 | 37.5 |
| Payback Period[3] | | years | 6.1 | 4.7 | **2.9** | 2.5 | 2.2 |

**Table 22-6: After-Tax Valuation Sensitivity to Certain Parameters**

| Factor | | 20% | 10% | 0% | -10% | -20% |
|---|---|---|---|---|---|---|
| Operating Cost | C$M | 1,271.2 | 1,165.3 | 1,059.3 | 953.4 | 847.5 |
| | IRR | 19.0% | 22.3% | 25.5% | 28.7% | 31.7% |
| | NPV5% | 215.7 | 272.0 | 328.2 | 384.3 | 440.2 |
| | Payback (yrs)[3] | 5.0 | 3.4 | 2.9 | 2.7 | 2.5 |
| Initial Capital Cost[4] | C$M | 325.3 | 298.2 | 271.1 | 243.9 | 216.8 |
| | IRR | 20.5% | 22.8% | 25.5% | 28.7% | 32.6% |
| | NPV5% | 287.7 | 308.0 | 328.2 | 348.3 | 368.4 |
| | Payback (yrs)[3] | 3.9 | 3.4 | 2.9 | 2.7 | 2.4 |
| Sustaining Capital Cost | C$M | 75.7 | 69.4 | 63.1 | 56.8 | 50.5 |
| | IRR | 24.8% | 25.2% | 25.5% | 25.9% | 26.3% |
| | NPV5% | 314.6 | 321.4 | 328.2 | 335.0 | 341.8 |
| | Payback (yrs)[3] | 3.0 | 3.0 | 2.9 | 2.9 | 2.9 |

Notes for Table 22-5 and Table 22-6:

1. Non-IFRS Financial Measures.
2. Exchange rate of US$1.00:C$1.25 was used.
3. Payback is defined as achieving cumulative positive free cashflow after all cash costs and capital costs, including sustaining capital costs and is counted from the start of production.
4. Excludes initial working capital requirements.

## 22.7 Non-IFRS Financial Measures

This Technical Report includes certain non-IFRS performance measures as further described below. In the gold mining industry, these are common performance measures but may not be comparable to similar measures presented by other issuers. The Company believes that, in addition to conventional measures prepared in accordance with IFRS, certain investors use this information to evaluate the Company's performance and ability to generate cash flow. Accordingly, these numbers are intended to provide additional information and should not be considered in isolation or as a substitute for measures of performance prepared in accordance with IFRS.

*Average Operating Cash Costs per Ounce of Gold* - Anaconda calculates operating cash costs per ounce by dividing operating expenses, net of by-product revenue, by payable gold ounces. Operating expenses include mine site operating costs such as mining, processing and administration as well as selling costs and royalties, however, excludes depletion and depreciation and rehabilitation costs.

The equivalent historical non-IFRS financial measure is “Operating Cash Costs per Ounce of Gold” and the most comparable IFRS measure is cost of operations. For further details and a reconciliation of such non-IFRS measure to the consolidated statement of comprehensive income, please refer to the section “Non-IFRS Measures” in the Company’s Management’s Discussion and Analysis for the three and nine months periods ended September 30, 2021 and September 30, 2020, (the “Interim MD&A”), which is incorporated by reference herein and available on SEDAR at www.sedar.com.

*Average All-In Sustaining Costs per Ounce of Gold* - Anaconda has adopted an all-in sustaining cost performance measure that reflects all expenditures that are required to produce an ounce of gold from current operations. While there is no standardized meaning of the measure across the industry, the Company’s definition conforms to the all-in sustaining cost definition as set out by the World Gold Council in its guidance dated June 27, 2013. The World Gold Council is a nonregulatory, non-profit organization established in 1987 whose members include global senior mining companies. The Company believes that this measure will be useful to external users in assessing operating performance and the ability to generate free cash flow from current operations.

The Company defines all-in sustaining costs as the sum of operating cash costs (per above), sustaining capital (capital required to maintain current operations at existing levels), sustaining exploration, and rehabilitation and reclamation costs. All-in sustaining costs excludes initial capital expenditures, financing costs, corporate general and administrative costs and salvage value, and taxes. AISC per Ounce is calculated as AISC divided by payable gold ounces. The equivalent historical non-IFRS financial measure is “All-In Sustaining Costs per Ounce of Gold”. For further details and a reconciliation of such non-IFRS measure to the consolidated statement of comprehensive income, please refer to the section “Non-IFRS Measures” in the Company’s Interim MD&A.

# 23. ADJACENT PROPERTIES

On July 17, 2019, the Company announced the option to acquire a 100% interest in the Lower Seal Harbour Property from Crosby Gold Ltd. (at the time a 100%-owned subsidiary of Osprey Gold Development Ltd., which is now a subsidiary of Meguma Gold Corp) (Figure 23-1).

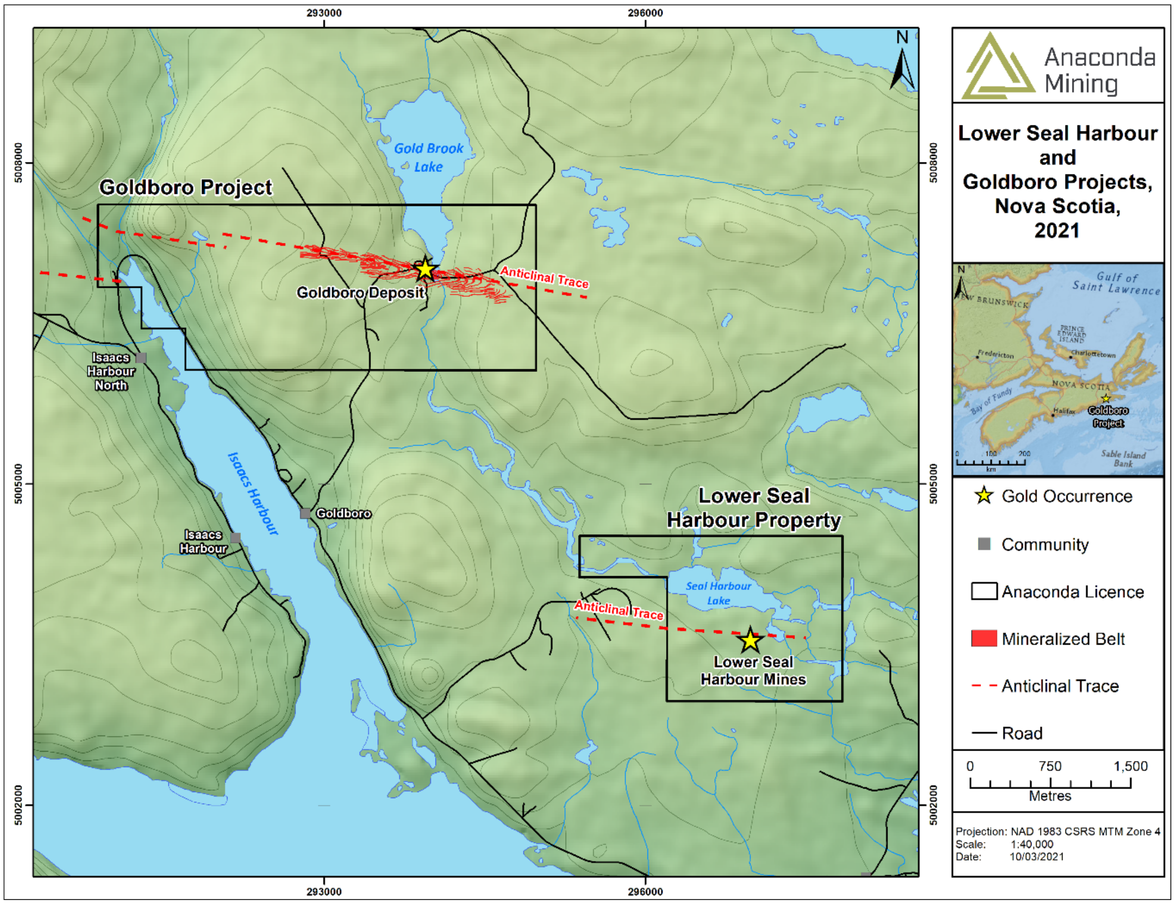



*Figure 23-1: Anaconda property locations on the eastern shore of Nova Scotia, including the Goldboro and Lower Seal Harbour Properties.*

The Lower Seal Harbour Property comprises two mineral licences totalling over 291 hectares and is located five kilometres south of the Project (Figure 23-2). The Company may exercise the option by making total cash payments of $85,000 and issuing of $85,000 worth of the Company's common shares over a 3-year period. The Company has assumed a 2.0% Gross Metal Royalty (GMR) granted to a private vendor under the terms of a preceding underlying agreement, with the right to buy-back 75% of the GMR for $375,000. Total work commitments on the Lower Seal Harbour Property under the agreement are $150,000 over four years.

The Lower Seal Harbour Mine produced 34,295 oz of gold after mining 394,905 tonnes between 1894 and 1949. Gold is hosted in both the quartz veining and anticlinal host greywacke and argillite of the Goldenville Formation, with accessory arsenopyrite, pyrite, and galena. Previously recognized mineralization is hosted solely on the limb of the fold with no past exploration undertaken in the hinge of the fold, one of the key mineralized environments at the Project (Parsons, et al., 2012).

Immediately east of the Project, St. Barbara Ltd. holds 34 claims and south east an additional 14 claims through its subsidiary Atlantic Mining NS Corp. (Nova Scotia Geoscience Atlas) (Figure 23-2). There is no public disclosure of activity on these claims at the effective date of this Technical Report.

Immediately to the west, south, and north of the Project, MegumaGold Corp. (through subsidiary 1156219 B.C. Ltd.) holds a total of 174 claims (Nova Scotia Geoscience Atlas) (Figure 23-2).

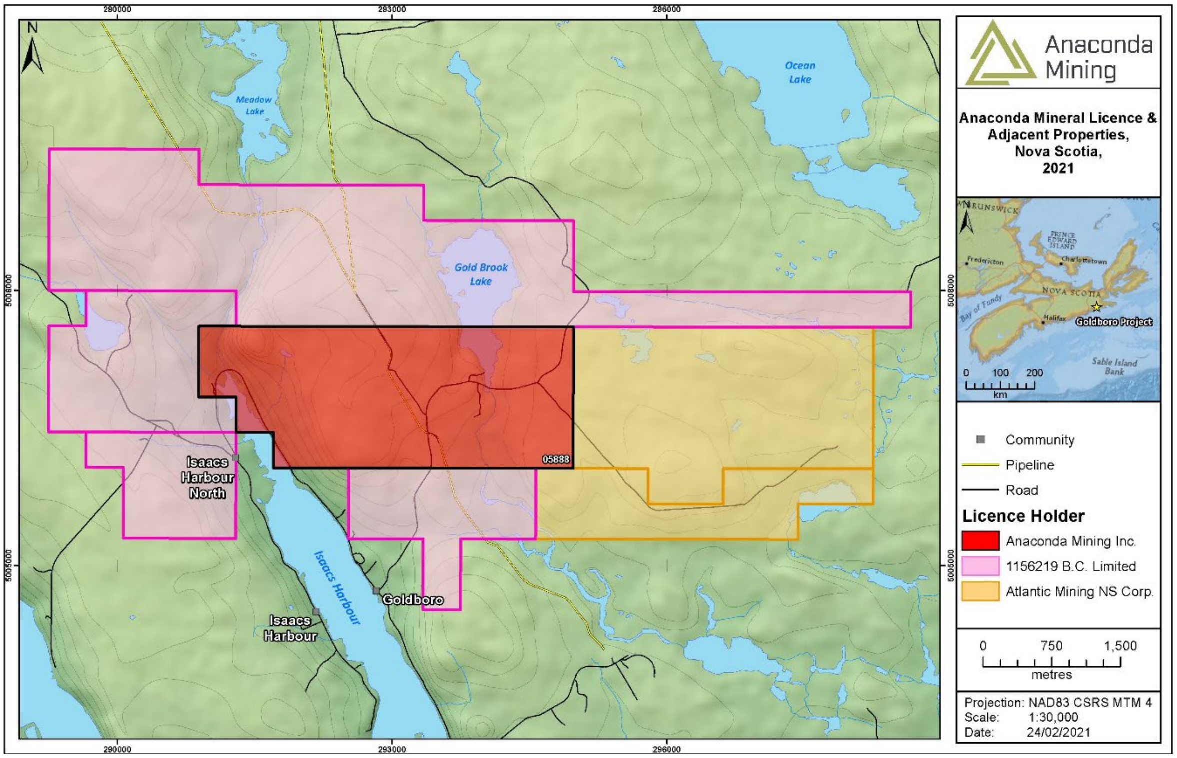



*Figure 23-2: Adjacent properties*

# 24. OTHER RELEVANT DATA AND INFORMATION

## 24.1 Constrution Schedule

### 24.1.1 Schedule Objectives

The scheduling objective is to deliver a fully constructed and commissioned mining facility as per the following timeline.

- The timeline is based on achieving ramp up to 100% of production capacity within first year of operation.
- The timeline is linked to both the mining related activities and the surface operations in both sequencing and duration. The construction of the main plant buildings and supporting infrastructure is the critical path.

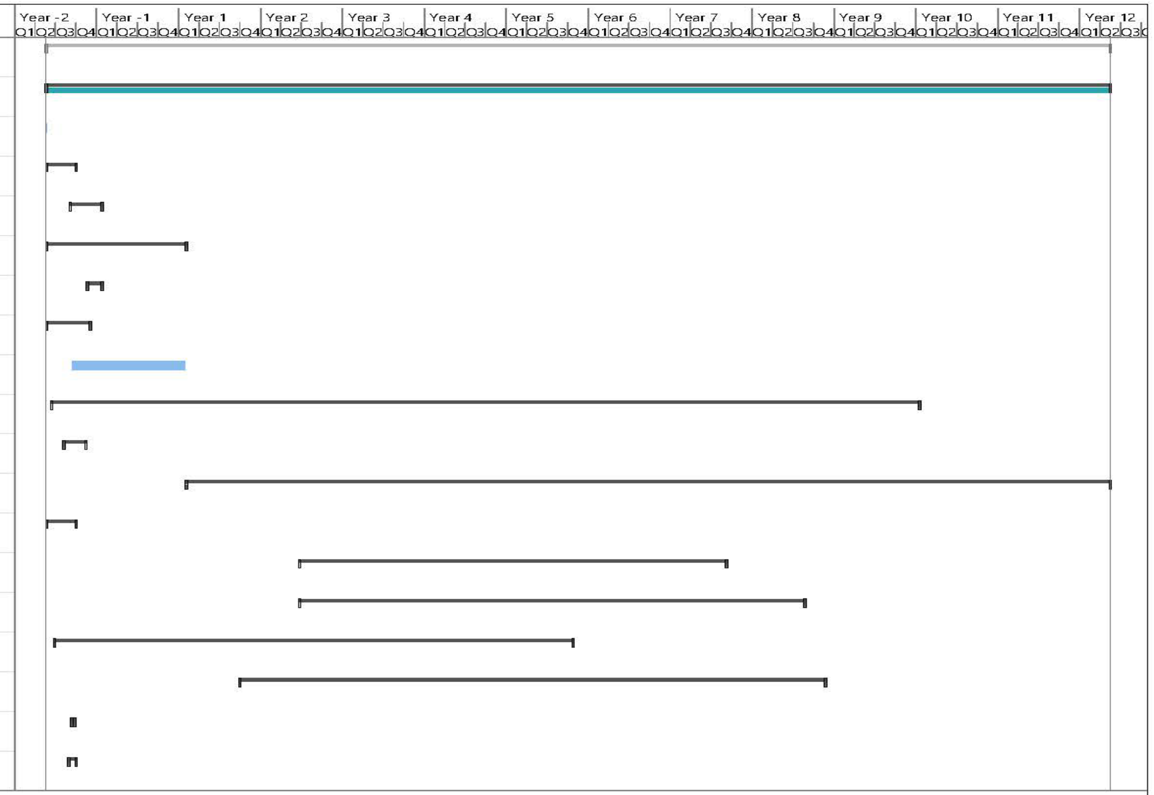
### 24.1.2 Construction Management and Schedule

Construction management functions will be performed by EPCM contractors, including planning, organizing, and resolving issues involving the site contractors. Ensuring that contractors' work is performed according to the Company's safety, quality, schedule, and cost requirements. Additionally, the EPCM contractors are required to provide the facilities and services, including security, to support the sub-contractors. This practice will ensure that quality standards are maintained and will improve the use of shared resources and equipment. The primary functions include planning and coordination, contractor management, quality assurance, resolving design engineering issues, quantity measurement, and materials management.

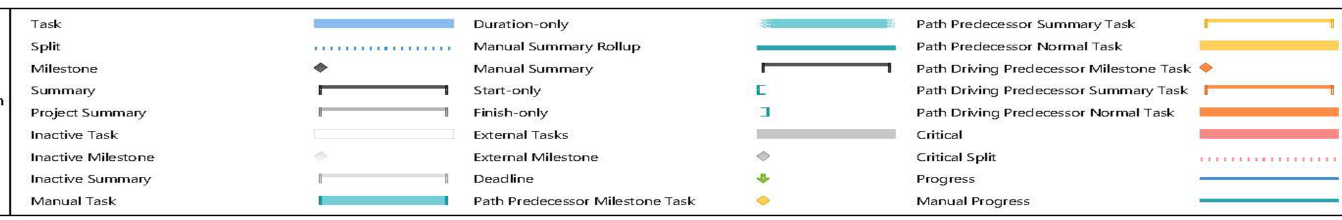
Table 24-1 provides the construction schedule prepared by Nordmin in conjunction with the Company.

**Table 24-1: Construction Schedule**

| ID | Task Name | Duration |
|---|---|---|
| 0 | Goldboro Construction Schedule - FINAL | 678 wks |
| 1 | Goldboro Construction | 678 wks |
| 2 | PROJECT START/KICK-OFF | 0.2 wks |
| 3 | Mine Site Civil Works and Site Preparation | 19 wks |
| 10 | Main Access & Mine Site Roads | 21 wks |
| 18 | Employee Accommodations | 89 wks |
| 32 | Explosive Storage | 10 wks |
| 38 | Haul Roads | 28 wks |
| 46 | Mill Build | 72 wks |
| 47 | East Pit Pre-mining | 553 wks |
| 52 | Screening plant for Ditches/Roads | 14 wks |
| 56 | West Pit Pre-Mining | 589 wks |
| 60 | Tailings Piping Civil Works | 19 wks |
| 65 | Northwest Waste Dump | 272 wks |
| 70 | Northeast Waste Dump | 322 wks |
| 75 | Southeast Waste Dump | 330.6 wks |
| 80 | Southwest Waste Dump | 373 wks |
| 85 | Construction Laydown Area | 2 wks |
| 87 | Exterior Storage | 5 wks |




Project: Goldboro Construction
Date: Wed 1/19/22

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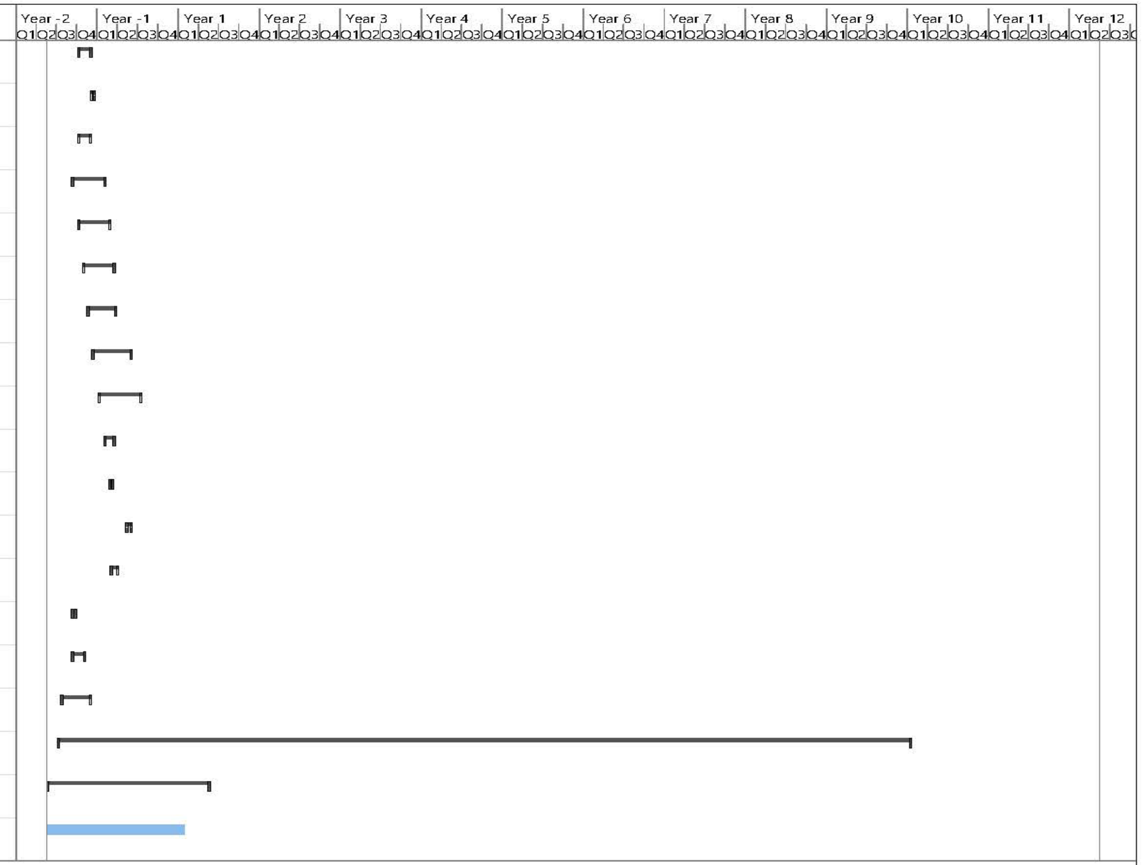
| ID | Task Name | Duration |
|---|---|---|
| 90 | **Core Storage** | **8 wks** |
| 95 | **Core Yard** | **2 wks** |
| 98 | **Security And Guest Parking** | **8 wks** |
| 105 | **Truck Shop / Truck Wash** | **21 wks** |
| 111 | **Plant Office** | **20 wks** |
| 116 | **ERT Facility** | **20 wks** |
| 121 | **General Office and Parking** | **18 wks** |
| 127 | **Surface (Open Pit) Mine Dry** | **25 wks** |
| 132 | **Warehouse** | **27 wks** |
| 137 | **Truck Scale** | **6 wks** |
| 142 | **Propane Storage** | **2 wks** |
| 145 | **Communication Process Plant (AUSENCO)** | **3 wks** |
| 147 | **Fuel Storage** | **4 wks** |
| 150 | **Heli Pad** | **2 wks** |
| 152 | **Main Switchyard** | **8 wks** |
| 158 | **Power Distribution** | **18.8 wks** |
| 166 | **Tailings Management Facility (KP)** | **548.8 wks** |
| 175 | **Water Treatment Plant** | **104 wks** |
| 181 | **Project Construction phase Start to Finish** | 88 wks |




Project: Goldboro Construction
Date: Wed 1/19/22

| | | |
|---|---|---|
| Task | Duration-only | Path Predecessor Summary Task |
| Split | Manual Summary Rollup | Path Predecessor Normal Task |
| Milestone | Manual Summary | Path Driving Predecessor Milestone Task |
| Summary | Start-only | Path Driving Predecessor Summary Task |
| Project Summary | Finish-only | Path Driving Predecessor Normal Task |
| Inactive Task | External Tasks | Critical |
| Inactive Milestone | External Milestone | Critical Split |
| Inactive Summary | Deadline | Progress |
| Manual Task | Path Predecessor Milestone Task | Manual Progress |

Page 2

The construction schedule has been developed based on the following assumptions.

**Assumption #1**

- During construction the weather will not interfere with the construction.
- The weather will not be so severe that work will have to stop for any long durations (10%).

**Assumption #2**

- The area will have the available trained and qualified workforce.
- The workforce will perform and meet the productivity targets required.
- On-the-job training will include apprenticeship and Indigenous training programs.
- Area production rates will be provided by local sources.

**Assumption #3**

- All equipment requirements to execute the construction work will be available and will fill the required equipment needs to work in multiple areas.
- D-11 Cats or equivalent will be used for excavation.

**Assumption #4**

- The equipment availability will be based at 85+%, and any parts are readily available for the repair and maintenance of the equipment for the duration of the construction.

**Assumption #5**

- All required permitting/nesting plans will have been completed prior to starting of the construction preventing any unnecessary delays to start up.

**Assumption #6**

- The concrete will be readily available from local resources.

**Assumption #7**

- All rebar steel, forming materials, tools, pumps, maintenance parts and supplies will be provided by the contractors and delivered to the site prior to starting the concrete construction phase of the work.

**Assumption #8**

- Gravel will be available off site for the required construction needs.
- The surplus cut materials will be used on haul roads and pads as required.
- The waste rock NPAG will be used on the tailing's impoundment.

**Assumption #9**

- All water works will be completed during the approved non-restricted period between June 1 through September 31.

**Assumption #10**

- All grubbing mitigation measures and plans will be approved by the Nova Scotia Department of Natural Resources and Renewables.
- Pre-clearing nest sweeping plans will be approved in keeping with the Nova Scotia Endangered Species Act (NS ESA) and the Species at Risk Act (SARA) management plans and recovery strategies.
- No interruptions on earthworks from any Government agency.

**Assumption #11**

- All predesigned fabricated buildings and all parts are delivered to the site prior to the starting of construction of the building as scheduled.
- All construction supplies and materials are readily available from local sources.

**Assumption #12**

- Delivery of all equipment and materials can be transported to the site on accessible roadways without any unnecessary delays.

**Assumption #13**

- There will be no local interference with the project start up that may create unnecessary delays.

**Assumption #14**

- A local crushing operation will provide required engineered fill materials for pre-construction foundation preparation, road finishing materials for the Project site and access roads as required.

**Assumption #15**

- Earthworks include clearing, grubbing, topsoil stripping, haulage and access roads, excavation, culverts, cut, haul and fill, drainage, water controls and sediment controls etc.
- Sediment controls will be started upon the completion of temporary access.

**Assumption #16**

- Productivity levels are at 100% performance.
- The percentage performance rate will be factored in upon agreement by all parties.

**Assumption #17**

- The use of a temporary generator system will be used and available during construction until permanent power is established.

**Assumption #18**

- Multiple crews will be working at the same time during construction works.

**Assumption #19**

- Long lead items will be ordered by procurement in advance of start up for construction.

**Assumption #20**

- Temporary accommodation will be provided for the earthworks crews during the time of construction of the employee accommodations.

**Assumption #21**

- Pit grubbing and clearing will begin upon the completion of the temporary access road.

**Assumption #22**

- The mill/concentrator construction is the critical task to the Project. Any delays in the commencement of construction or any materials or procurement delays during construction will directly affect the project end date.

**Assumption #23**

- The employee accommodations will be adequate for housing up to 352 people at any given time. This will include bathing, sleeping and food requirements.

**Assumption #24**

- A temporary security/EMT/helipad complex will be built upon the start of the project for personnel medical protection and equipment security until the permanent facilities have been built and established.

**Assumption #25**

- An estimated duration has been used in areas where the vendor bids have not been provided.
- Estimates are based on past knowledge and experiences.

# 25. INTERPRETATION AND CONCLUSIONS

## 25.1 Introduction

The QP's note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Technical Report.

## 25.2 Mineral Tenure, Surface Rights, Royalties, and Agreements

The Property is situated on the eastern shore of Nova Scotia, Canada. The Property's central point is approximately located at 45° 12' 2.6" N latitude and 61° 39' 2.0" W longitude.

The Property consists of 37 contiguous claims covering a total area of approximately 592 hectares held under Exploration Licence No. 05888. This title is in its 43rd year of issue and is renewed every two years, with the next renewal date on November 29, 2023.

To date, the Company has arranged access to the Property for the purpose of exploration through agreements with both private and Crown entities. Much of the Property, including all the BR, and EG historic workings, is underlain by Crown Land. Similarly, access to private lands, and securing agreements with landowners has not proven to be difficult. The Company has advised Nordmin that it knows of no reason that future access to the areas required for exploration would be withheld.

The presence of past mining operation infrastructure, including several historic tailings sites associated with the past operation of the historic Boston Richardson Mine and location within the Gold Brook Lake-Seal Harbour Lake watershed are recognized as important environmental site factors. Provincial regulators indemnified Orex in 1995 from any environmental liabilities resulting from historical mining activities, assuming that old tailings storage areas are not impacted during exploration or mining activities.

The Company continues to successfully manage the IA related to the Bulk Sample collected in 2018 to 2019.

No obvious environmental issues were identified during the 2021 Nordmin site visits, and Nordmin is not aware of any subsequently developed site conditions that would materially alter the nature of this assertion.

At the effective date of this Technical Report, the Company held access agreements that specifically apply to surface core drilling. The Company has the necessary Crown Land permits for additional drilling and trenching or expects to receive them through normal exploration permitting process.

## 25.3 Geology and Mineral Resource Modelling

The turbidite-hosted gold deposits of Nova Scotia have been compared to similar-age turbidite-hosted quartz vein deposits elsewhere in the world, particularly those in the Bendigo and Ballarat areas of the Lower Paleozoic Lachlan Fold Belt in the state of Victoria, Australia, and have historically been similarly classified. Robert et al. (1997) recognized this deposit class and proposed that it be identified as a member of the 'Turbidite-hosted, quartz-carbonate vein deposit (Bendigo Type)' category. Ryan and Ramsay (1996) also addressed the similarity of Nova Scotia turbidite-hosted gold deposits with those in Victoria. As noted by Gervais et al. (2009), categorization within the USGS classification system of mineral deposits presented by Berger (1986) places the Deposit in the broad 36a category of 'Low Sulphide Gold-Quartz Vein Deposits.'

The Deposit is a turbidite-hosted orogenic gold deposit hosted within a sequence of alternating argillites and greywacke metamorphosed to greenschist facies. These deposit types are typically characterized by the formation of gold bearing quartz veins within the argillite units, commonly referred to as mineralized Belts, that are interbedded with greywacke units. There are currently 68 stacked mineralized Belts ranging in thickness from 1 m to 20 m in the Deposit. The metasedimentary units on the Property are folded into the tight, gently east-plunging Upper Seal Harbour Anticline and gold mineralization has typically been deposited at various positions and times during the fold formation process. Veins, which form during deformation, occur in three major geometries commonly referred to as reefs: saddle reefs, leg reefs, and spur reefs. Saddle reefs occur about the apex of the fold and are the dominant vein types within some deposits. Leg reefs extend down the limbs of the fold, beyond the saddle reef, and are generally parallel with the metasedimentary layers. These are also commonly termed BP veins in the Nova Scotia goldfields. Spur reefs are veins that cross between layers and may be in the apex of the fold or on its limbs. This style of vein is in part captured under the term "angular" in the Nova Scotia goldfields.

Gold mineralization at the Deposit occurs in both quartz veins and within the argillite that hosts the veins, as well as within the rocks adjacent to the modelled argillites and quartz veins, including both lesser argillite with greywacke. Disseminated, euhedral arsenopyrite is pervasively associated with gold mineralization, and is commonly observed within the host rock and is usually present in mineralized quartz veins. Other sulphides associated with mineralized quartz veins are pyrrhotite, chalcopyrite, pyrite, and minor amounts of sphalerite and galena. Wall rock generally contains more pyrrhotite and arsenopyrite than directly associated quartz veins. Gold bearing quartz veins are stratabound with lesser discordant quartz veins and vein arrays. Native gold is nuggety in nature, and grains range from microscopic up to several centimetres in size and is found in all rock types with visible gold generally associated with quartz veins. Within quartz veins, gold is present as free gold in quartz, and within arsenopyrite grains, along grain boundaries and internal fractures, and is non-refractory in nature. Native gold also occurs as a disseminated phase in altered argillite and argillite/greywacke intervals adjacent to and separate from quartz veins, demonstrating its association with both quartz veins and the altered wall rock.

The Mineral Resource Estimate was estimated from the main database comprised of 681 DDHs consisting of 121,500 m completed between 1984 and the effective date of November 15, 2021, and 1,230 chip samples comprised of 822.7 m from the 2018 to 2019 Bulk Sample. Nordmin, through an interactive process with the Company, undertook a full re-examination of the mineralogical, lithological, structural, and geochemical correlations influencing the higher-grade and lower-grade gold areas within the Project. Gold mineralization at the Project occurs in both quartz veins and within argillite, which hosts the veins, and within the rocks adjacent to the modelled argillites and quartz veins, including argillite with greywacke. Disseminated, euhedral arsenopyrite is pervasively associated with gold mineralization and is commonly observed within the host rock and is usually present in mineralized quartz veins. Wall rock generally contains more pyrrhotite and arsenopyrite than directly associated quartz veins.

The Deposit consists of three domains referred to as the BR, EG, and WG Gold Systems. The WG Gold System is separated from the BR Gold System by a north trending, near vertical fault with tens of metres of apparent offset. The EG Gold System is separated from the BR Gold System by a thick greywacke sequence or marker unit. Stratigraphic younging is from west to east with the anticlinal fold plunging shallowly to the east.

From a modelling perspective, each of the Deposit Gold Systems was separated into its own domain. Each domain was further sub-domained into Higher-Grade Belts and Lower-Grade Domains.

In 2021 (prior to September 15), the Company completed 74 infill drill holes (10,145.4 m) across these three domains. Nordmin utilized these infill drill holes and reviewed the lithological, structural, spatial mineralization controls and grade variances within the Project, updating and editing the model as appropriate.

Explicit modelling was used to create the Mineral Resource, which allows for mineralization to better reflect the Deposit geology and associated geochemistry. Nordmin's opinion is that the explicit modelling approach minimizes risks compared to the use of implicit modelling for the Project.

## 25.4 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

The exploration programs completed by the Company and previous operators are appropriate for the deposit style. The programs have delineated the WG, BR, and EG Gold Systems as well as a number of exploration targets along the trend. Regional surface exploration indicates the potential to discover further targets that warrant further investigation.

The quantity and quality of the lithological, collar and downhole survey data collected in the various exploration programs by various operators are sufficient to support the Mineral Resource Estimate. The collected sampling is representative of the gold, sulphur, arsenic, and corresponding ICP data in the Deposit, reflecting areas of higher and lower grades. The analytical laboratories used for legacy and current assaying are well known in the industry, produce reliable data, are properly accredited, and are widely used within the industry.

Nordmin is not aware of any drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results. In Nordmin's opinion, the drilling, core handling, logging, and sampling procedures meet or exceed industry standards, and are adequate for the purpose of Mineral Resource estimation.

Nordmin considers the QA/QC protocols in place for the Project to be acceptable and in line with standard industry practice. Based on the data validation and the results of the standard, blank, and duplicate analyses, Nordmin is of the opinion that the assay and bulk density databases are of sufficient quality for Mineral Resource estimation for the Project.

## 25.5 Mineral Resource Estimate

The Mineral Resource Estimate for the Project conforms to industry best practices and is reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves and 2019 CIM Best Practice Guidelines. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues.

Mineral Resources were classified into Measured, Indicated, and Inferred Resource categories based on geological and grade continuity, in conjunction with data quality, spatial continuity based on variography, estimation pass, data density, and block model representativeness, specifically assay spacing and abundance, kriging variance, and search volume block estimation assignment.

The Mineral Resource Estimate has been defined based on an applied g/t gold CoG to reflect processing methodology and assumed revenue stream from gold.

The Mineral Resource Estimate is based on an open pit and underground mining methodology, process design for comminution, gravity concentration, leaching, carbon absorption, cyanide destruction, and carbon elution, and gold refining in the process plant.

The Mineral Resource Estimate is based on validated results of 635 surface and underground drill holes, for a total of 113,132.9 m of diamond drilling completed between 1984 and the effective date of February 7, 2021, as well as 1,230 chip samples comprised of 822.7 m from the 2018 to 2019 Bulk Sample. The Mineral Resource Estimate includes 45,408.7 m of drilling conducted by the Company. The Mineral Resource Estimate includes 7,488.3 m of diamond drilling in 62 drill holes since the Previous Mineral Resource Estimate effective February 7, 2021. Nine drill holes totalling 1,001.9 m were removed from the database due to inconsistent sample lengths.

The Project hosts:

- Total Open Pit (at a 0.44 g/t cut-off) and Underground (at a 2.40 g/t cut-off) Mineral Resources including 7,521,000 tonnes and 866,200 oz of Measured Resources grading 3.58 g/t gold, 8,515,000 tonnes and 1,079,900 oz of Indicated Resources grading 3.95 g/t gold, and 5,306,000 tonnes and 798,100 oz of Inferred Resources grading 4.68 g/t gold.
- Open pit Mineral Resources including 6,137,000 tonnes of Measured Resources and 538,500 oz grading 2.73 g/t gold, 5,743,000 tonnes and 551,300 oz of Indicated Resources grading 2.99 g/t gold, and 1,580,000 tonnes and 89,000 oz of Inferred Resources grading 1.75 g/t gold.
- Underground Mineral Resources including 1,384,000 tonnes and 327,700 oz of Measured Resources grading 7.36 g/t gold, 2,772,000 tonnes and (528,600 oz) of Indicated Resources grading 5.93 g/t gold, and 3,726,000 tonnes and 709,100 oz of Inferred Resources grading 5.92 g/t gold.

There is potential for an increase in the estimate if mineralization that is currently classified as Inferred can be upgraded to higher-confidence Mineral Resource categories and if any categorized mineralization within the Deposit can be expanded.

## 25.6 Mining and Mineral Reserves

#### 25.6.1.1 Mining Methods

Conventional OP mining methods will be used to extract a portion of the Deposit. This method was selected considering the deposit’s size, shape, orientation, and proximity to the surface. Drilling, blasting, loading, and hauling will be used to mine the OP material within the designed pit to meet the mine production schedule.

Open pit mining will include conventional drilling and blasting with a combination of a backhoe type excavator, hydraulic excavator, and front-end loader type excavator loading broken rock into haul trucks, which will haul the material from the bench to the crusher, ROM stockpile or waste stockpiling areas depending on the material type. Ancillary equipment includes dozers, graders, and various maintenance, support, service and utility vehicles.

The FS is based on a conventional truck-shovel open pit mining operation within two pits. The open pit production period is approximately 10.9 years with 1 year of pre-production (prior to process plant start-up). It is envisaged that the rock will be loaded directly into the processing plant crusher hopper but there will be a need for a ROM stockpile to allow for stoppages, for stockpiling in the pre-production period, and possibly some blending. The operation scenario for the FS involves:

- Open pit mining at an average mining rate of 12.8 Mt per year.
- Gold process facility with a 1.46 Mtpa (4,000 t/d) capacity.
- Approximate 6-month ramp up period in Year 1 (YR1) for the process facility.
- 1-year pre-production mining period to coincide with TMF initial stage development.

#### 25.6.1.2 Mineral Reserve

The Mineral Reserve Estimate for the Project conforms to industry best practices and is reported using the May 10, 2014, Standards for Mineral Resources and Mineral Reserves and the 2019 CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019).

Mineral Reserves are based on the engineering and economic analysis described in Sections 16 to Section 22 of this Technical Report. Changes in the following factors and assumptions may affect the Mineral Reserve Estimate:

Factors that may affect the Mineral Reserve

- Metal prices
- Interpretations of mineralization geometry and continuity of mineralization zones
- Kriging assumptions
- Geomechanical and hydrogeological assumptions
- Ability of the mining operation to meet the annual production rate
- Operating cost assumptions
- Process plant recoveries
- Mining loss and dilution
- Ability to meet and maintain permitting and environmental licence conditions
- Historical mining depletion

Nordmin prepared a Mineral Reserve Estimate for the Project using a combination of Geovia's Whittle 4.7.4 and Geovia's Surpac 2021 software packages for estimating the economic pit limit for the open pit and block model interrogation.

The Mineral Reserve Estimate for the Deposit is based on the resource block model estimated by Nordmin and described in Section 14. The block model contained Measured, Indicated and Inferred Mineral Resources, however only Measured and Indicated Mineral Resources were used. Inferred Mineral Resources in the block model were not included in the Probable Mineral Reserve and remain classified as waste; Inferred Mineral Resources do not meet the standards required for inclusion in Mineral Reserves.

Mineral Reserves for the Deposit incorporate mining dilution and mining loss assumptions for the open pit mining method.

The reference point at which Mineral Reserves are defined, is the point where the ore is delivered to the processing facility, which includes the ROM stockpile.

The detailed design of the final pit and the LOM scheduling with the cut-off grade resulted in a total Probable Mineral Reserve of 15,798,900 tonnes grading 2.26 g/t gold for total ounces of 1,150,200 (Table 15-1) is estimated.

## 25.7 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralisation type, appropriate to establish the optimal processing routes, and were performed using samples that are typical of the mineralisation styles found within the various mineralised zones in the current and past studies.

Samples selected for testing were representative of the various types and styles of mineralisation. Samples were selected from a range of depths within the deposits. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Recovery factors estimated are based on appropriate metallurgical testwork and are appropriate to the mineralisation types and the selected process route.

Based on the testwork results completed between 2018 and 2021 on composite samples representative of the Open Pit deposits, doré can be produced at a high recovery of gold. Cyanide concentrations in Mill tailings can be decreased to 0.5 mg/L $CN_{TOT}$ and arsenic concentrations reduced to 0.5 mg/L.

## 25.8 Recovery Methods

The process plant is designed to process ore at a rate of 4,000 tpd to produce gold doré bars.

The process plant flowsheet designs were based on testwork results and industry standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

## 25.9 Infrastructure

The main Project infrastructure components include mine and process plant supporting infrastructure, site accommodation facilities, TMF, external and internal access roads, power supply and distribution, freshwater supply and distribution, and a water treatment plant.

The Property is situated on the eastern shore of Nova Scotia, Canada, with the central point of the Property being approximately located at 45° 12’ 2.6” N latitude and 61° 39’ 2.0” W longitude.

The Property will have access to the substantial infrastructure, services, and skilled labour in the area. There will be reduced infrastructure cost requirements due to its location near Route 316 compared to a remote mine site location. The Property is approximately 175 km northeast of the city of Halifax, 60 km southeast of the town of Antigonish, and 1.6 km north of the village of Goldboro, on the eastern shore of Isaac’s Harbour, in Guysborough County, Nova Scotia, Canada. A secondary gravel road (Goldbrook Road), accessed from Route 316, crosses the Property, and passes near the historic Boston Richardson shaft and exploration decline. Smaller logging roads and trails provide good access to most areas of the Property. The elevation is nominally 70 m above sea level. The regional labour force includes experienced equipment operators, mine workers and material and equipment suppliers.

The majority of the earthworks will be realized in the preparation of the mine infrastructure, process plant and TMF infrastructures. Haulage roads on site will be built to withstand frequent heavy traffic between the proposed open pit, ROM stockpile and TMF. They will be wide enough to accommodate two trucks passing between the pits and ROM stockpile at 16.5 m with a grade no greater than 10%. The road to and from the tailing's management facility will be 11 m wide for one-way traffic by haul trucks.

In total, approximately 5,300 $m^2$ of ancillary buildings (not including the employee accommodations and process plant buildings) have been provided.

These ancillary infrastructure buildings will be pre-engineered steel structures founded on conventional spread footing foundations. Space has been provided for future buildings provided by the mining contractor or in the case of expansion during operations.

Power for the site is anticipated to be provided from a nearby Nova Scotia Power 25 kV distribution line installed along Route 316. A 1.6 km tap line would be installed along a new right of way to the mine site main substation. Nova Scotia Power would upgrade their existing distribution system as necessary to be able to provide the additional power required. Peak power demand for the site is estimated to be 10 MW, with the average demand estimated to be 7.5 MW. A network of 13.8 kV overhead distribution lines would be installed at site to provide power sourced from the main substation for the mine and surface infrastructure.

Water supply infrastructure includes:

- One intake structure.
- Two booster stations.
- One transmission watermain from Gold Brook Lake to the mill freshwater tank and to the potable water treatment unit.
- Distribution piping to supply potable water throughout the Project site (mill, ERT facility, plant office, general office, mine dry, core storage, truck shop and employee accommodation).

A transmission watermain from Gold Brook Lake to the processing plant buildings is to provide a raw water source to support mill process operations and site wide potable water, hence the watermain flowrate was estimated based on the potable and process water demands (22 $m^3/h$).

Gold Brook Lake was considered as the source water, the treatment requirements were established based on the Canadian Drinking Water Guideline, and potable water treatment was sized assuming an equal flowrate for both potable water and wastewater (16 $m^3/h$).

Two separate wastewater treatment units were developed to service employee accommodation (with 350 people) and other buildings/facilities including mill, ERT facility, plant office, general office, mine dry, core storage, truck shop (with 84 people). Sewage flow rates as well as treatment requirements were adopted from the Atlantic Canada Wastewater Guidelines Manual for Collection, Treatment and Disposal, 2006.

The MWMP and associated design measures have been developed based on the proposed feasibility level mine site arrangement with inputs from the Company and the Consultants. The MWMP will be implemented during the construction phase and will be adjusted as necessary throughout the mine operations and closure phase.

Site contact water will be managed to meet the following regulatory discharge requirements prior to discharge to the natural environment:

- MDMER Objectives
- CCME Guidelines for the Protection of Aquatic Life
- Tier 1 Nova Scotia EQS for Surface Water
- Site specific criteria (based on background)

Based on predictive water quality modelling, it is understood that the water quality at some locations may be acceptable for discharge to the environment with TSS removal as the only form of treatment (where MDMER objectives are met). Where this is not the case, contingency measures (shutoff valves and pumps etc.) will be put in place to redirect water towards the nearest WTS in case of exceedances. The primary objectives of the MWMP are as follows:

- Provide a mechanism to dewater and treat ponded water within the Project Area to allow for development and excavation of mine infrastructure (e.g., pit, waste piles, haul road etc.).
- Capture, treat and provide controlled discharge for all site contact water during construction and operations.
- Divert all off site clean water away from the mine site infrastructure to reduce the total volume of water entering the settling ponds for treatment.

### 25.9.1 Tailings Management Facility

The TMF will provide secure storage for tailings, PAG1 waste rock, and process water. Co-disposal will include management of both tailings and PAG1 waste rock in the TMF. The embankments include for adequate freeboard to provide ongoing tailings storage, PAG1 waste rock, water cover, operational water management, temporary storage of runoff resulting from the EDF and safe conveyance of runoff up to and including the IDF through a spillway. The TMF will be constructed as a paddock style, single cell facility located on a side hill northeast of Gold Brook Lake. A geomembrane lining system will be installed along the TMF basin floor and on the upstream face of the perimeter embankments to minimize seepage exiting the facility. The embankments will be raised in stages using downstream construction methods throughout the mine life.

Tailings will be pumped from the process plant to the TMF as a conventional thickened slurry via pipeline(s) and deposited into the TMF. The PAG1 waste rock will be segregated during mining operations and hauled directly to the TMF. The PAG1 waste rock pile in the TMF basin will be constructed similar to conventional waste rock piles (i.e., spread by a dozer in controlled lifts and compacted by the mine haul fleet) towards the northeast portion of the facility. The working surface of the PAG1 waste rock pile will be maintained above the elevation the tailings and supernatant throughout the mine life.

Meteoric and supernatant inflows to the TMF basin will be temporarily stored prior to reclaim by a floating pump barge in the basin to the process plant. Water reclaim, and treatment and release will be conducted such as to always maintain a 2 m minimum water cover over the deposited tailings surface. Excess water beyond the storage of the required water cover level and allowable operating range will be transferred to the TMF water treatment plant as required for treatment prior to release to the environment. The TMF will be equipped with an emergency overflow spillway in each embankment stage to accommodate flows above the EDF and up to the IDF.

### 25.9.2 Polishing Pond

A polishing pond will be constructed as an external pond to store water for the TMF WTP operations. The polishing pond will be constructed southwest of the TMF. The polishing pond has been designed to store approximately four days of TMF WTP discharge capacity plus some extra capacity for contingency. The polishing pond embankment will be constructed in one stage as zoned rockfill dam. The polishing pond basin and upstream embankment face will be lined with a smooth 80 mil HDPE geomembrane overlying a 12 oz./sq. yd. non-woven geotextile.

## 25.10 Mi'kmaq Engagement & Public Consultation

The Company recognizes the asserted Aboriginal & Treaty Rights and Title of Nova Scotia Mi'kmaq. The Company maintains ongoing engagement with KMKNO and representatives of Paqtnkek Mi'kmaw Nation. On June 2, 2019, the Company and the Assembly of Nova Scotia Mi'kmaw Chiefs signed a MOU that outlines a process that the parties may use to develop an MBA that reflects a desire to build a mutually beneficial relationship with respect to the Project. This process is ongoing. The Company maintains its commitment to work collaboratively with Nova Scotia Mi'kmaq regarding environmental and cultural priorities, as well as social and economic opportunities throughout the life of the Project. Information shared through ongoing Mi'kmaq engagement as well as completion of a Mi'kmaq Ecological Knowledge Study, has been reflected in the development of the Project to date and will continue throughout the life of the Project.

Public engagement has been ongoing with the MODG, as well as residents and property owners in the region since 2017. This includes regular meetings of Company senior executives and project consultants with the MODG Council. A CLC was established to foster environmental stewardship, and act as a conduit for transparent and ongoing communications between community, stakeholders, and the Company on all matters pertaining to potential development. The Company has held three open house meetings in Goldboro and will seek additional opportunities for community engagement throughout the life of the Project. In January 2022, the Company signed a Community Benefits Agreement with the Municipality to support sustainable social and economic benefits within the region with respect to the Project.

## 25.11 Permitting and Compliance Activities

To date, the Company has arranged access to the Property for the purpose of exploration through agreements with both private and Crown entities. Much of the Property, including all the BR, EG historical workings, is underlain by Crown Land. Similarly, access to private lands, and securing agreements with landowners has not proven to be difficult. At the effective date of this Technical Report, the Company held access agreements that specifically apply to surface core drilling. The Company has the necessary Crown Land permits for additional drilling and trenching or expects to receive them through normal exploration permitting process. The Company continues to successfully manage the Industrial Approval related to the underground Bulk Sample collected in 2018 to 2019.

The presence of past mining operation infrastructure, including several historical tailings sites associated with the past operation of the historical Boston Richardson Mine and location within the Gold Brook Lake-Seal Harbour Lake watershed, are recognized as important environmental site factors. Provincial regulators indemnified Orex in 1995 from any environmental liabilities resulting from historical mining activities, assuming that old tailings storage areas are not impacted during exploration or mining activities. A Historic Tailings Management Plan will be developed in consultation with NSECC to manage the areas that will be directly disturbed by the Project.

## 25.12 Enviromental Studies

The Project is subject to regulation under the NS Environment Act, Part IV. An EARD for the proposed project will be submitted for Class 1 Environmental Assessment in Q2 2022. The EARD will be authored by the Company and GHD and will utilize extensive baseline data collected at the Project site by the Company and its consultants since 2017, when the Company acquired the Project. Baseline studies, combined with predictive modelling, will inform project planning and provide the required information for various authorizations and permits. Mitigation measures to avoid, reduce or offset for potential effects will be developed and supported by EARD.

## 25.13 Market Studies

The Company has not completed any formal marketing studies with respect to gold production that will result from the mining and processing from the Project, which is assumed to be in the form of gold doré bars for the purposes of this FS. However, the Company was able to make reference to its existing refining contract at its Point Rousse operation with the Royal Canadian Mint to refine its gold doré bars into bullion, and a precious metals sales agreement with Auramet International LLC for the purposes of selling bullion. The Company has relied on its contracts for transportation, security, and insurance with respects to the refining of its gold doré bars.

Gold produced will likely be sold on the spot market by precious metals marketing professionals retained on behalf of the Company, at terms and conditions typical of similar contracts for the sale of refined LBMA[7] Good Delivery gold bullion. There are active and liquid gold markets throughout the world where gold can be bought and sold, and market pricing can be ascertained.

## 25.14 Capital and Operating Costs

The capital cost estimate was prepared by Nordmin and the Consultants with an expected accuracy range of ±15% weighted average accuracy of actual costs. Base pricing is in third quarter 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25. The estimate includes direct and indirect costs, (such as engineering, procurement, construction and start up of facilities) as well as owners costs and contingency associated with mine and process facilities and on site/off site infrastructure. Total LOM capital costs, including initial, sustaining and reclamation costs, are $384.5 million. The initial capital estimate is 271.1 million and includes amounts for indirect and contingency assumptions. A contingency of $31.7 million has been included in the estimate of initial capital costs, which amounts to 16% of direct initial capital costs or 11% of the total.

The operating cost estimate was prepared by Nordmin and the Consultants with an expected accuracy range ± 15% weighted average accuracy of actual costs based on third quarter 2021 Canadian dollars with no allowances for inflation or escalation beyond that time and assumes a currency exchange rate US$1.00:C$1.25, unless otherwise stated. Table 21-16 summarizes the Operating Cost Estimate for the Project. The LOM operating costs, including refining and transport cost, are estimated to be $1,064.0 million.

## 25.15 Economic Analysis

An economic model was prepared for the Project to estimate annual cash flows and assess sensitivities to certain economic parameters. The economic results of this report are based upon the services performed by:

- Nordmin for open pit mining and surface infrastructure and their associate consultants at Optimize for the pit slope stability.
- Knight Piésold for TMF.
- GHD for site water management.
- Ausenco for processing.
- Lorax for geochemistry.

- MEL for consultation and permitting.

The Company provided the inputs with respect to the tax impact of the economic model, including calculation of federal and provincial income taxes, provincial mining taxes, and available tax attributes that are applicable to the Project.

The Project includes an open pit and associated infrastructure, surface infrastructure to support the mine operations (i.e., maintenance and office facilities), water management features, ROM stockpiling area, processing facility, TMF, and employee accommodation facility.

The economic model for the Project indicates a pre-tax free cash flow of $755.1 million over a mine life of approximately 11 years, a pre-tax NPV 5% of $483.8 million and a pre-tax IRR of 31.2%. On an after-tax basis, the Project could generate free cash flow of $529.0 million, and after-tax NPV (5%) of $328.2 million and an after-tax IRR of 25.5%. The Project is most sensitive to commodity prices.

## 25.16 Opportunities and Risks

Nordmin completed a risk analysis in conjunction with Lorax, Ausenco, Knight Piésold, GHD, MEL, and the Company at the project level.

Nordmin provided each representative with a semi-quantitative risk matrix where the likelihoods and consequences were assigned numbered levels that were multiplied to generate a numerical description of risk ratings. The values assigned to the likelihoods and consequences were not related to their actual magnitude, but to the numerical value derived for risk (Figure 25-1). This approach provided for a standardized grouping and generation of indicated risk ratings. Each person/firm worked independently and reported back to Nordmin their findings which were then compiled. This compilation was discussed as a group during a workshop on November 30, and December 1, 2021, in Toronto, Ontario and subsequently summarized, including possible mitigation strategies (Appendix E).

**Likelihood x Consequence Matrix**

| Likelihood | | Consequence | | | | |
|---|---|---|---|---|---|---|
| | | Negligible | Minor | Moderate | Major | Severe |
| | *Score* | *1* | *2* | *3* | *4* | *5* |
| Rare | *1* | 1 | 2 | 3 | 4 | 5 |
| Unlikely | *2* | 2 | 4 | 6 | 8 | 10 |
| Possible | *3* | 3 | 6 | 9 | 12 | 15 |
| Likely | *4* | 4 | 8 | 12 | 16 | 20 |
| Almost Certain | *5* | 5 | 10 | 15 | 20 | 25 |

Low
Medium
High
Very High

Source: Nordmin, 2021

*Figure 25-1: Likelihood and consequence matrix*

The risk analysis defined 96 risks and their associated potential mitigation strategies (Appendix E).

Ten risks were considered a pre-response consequence of moderate or major, or severe, and a likelihood of likely or almost certain. If the action plan is initiated, the post response consequence for these high-risk items reduces to three risks outlined below.

The major group of risks identified and have an action plan assigned in Appendix E are the following:

#### 25.16.1.1 Geology, Mining and Milling

Operations – Grade control sampling and sample turnaround will be key to delivering the expected grades to the mill. Any reduction in grade will affect the overall production, milling and ounce production. To mitigate this risk complete infill drilling/sampling within the first two to three years of mining, create an appropriate reconciliation process from resource through to reserve, updating the short/long term mine plan, reserve, mine design, and mill efficiency/production. As such, extensive sampling protocols, locations and sizes will be required to improve mineralization patterns, reduce ore/waste assignments and improve the NPAG/PAG waste rock association. Extensive sampling using various sample sizes will also better define mineralization patterns.

The cyanide detoxification process primarily breaks down $CN_{WAD}$ and $CN_{TOT}$ which may or may not be directly reduced. The $CN_{TOT}$ present in the mill discharge is a function of the variable iron species present in the cyanide detox feed stream. Additional testing should be conducted to confirm the variability of $CN_{TOT}$ in the process tailings discharge water to the storage facility.

Additionally, no geotechnical information is available for the process plant area with the exception of the crushing and stockpile areas. This testing should be also completed as part of future work prior to the next phase of engineering

#### 25.16.1.2 Safety, Health, Environment and Community

While the Company is mitigating environmental impacts, is having meaningful engagement with rightsholders and stakeholders, and has the support of the municipality, there are groups and individuals that are against all forms of gold mining in Nova Scotia; therefore, there is potential for organized protests, which could cause delays. To mitigate this risk the Company will maintain ongoing opportunities for all rightsholders and stakeholders to have an occasion to raise questions and concerns about the Project. Additionally, the Company will ensure that accurate information about the Project is publicly available and will identify and correct inaccurate information in the public domain.

The Project requires the acquisition of some privately owned property. The Company has hired a third party to assess property values and negotiate sales. There may be individuals who refuse to sell regardless of the offer. If the property is essential to the Project the Company may have to pursue expropriation as a last resort, which could cause delays. To mitigate this risk the Company has engaged Turner Drake to complete property assessments, negotiate, manage, and document the process. The Company is not seeking to acquire any dwellings and will continue to work collaboratively will property owners, through its third-party service provider, for mutually agreeable outcomes. If the parties were unable to find a mutually agreeable outcome, the Company would then pursue the expropriation process to acquire the property which could cause delays, however, would unlikely cause any delays in the overall permitting process. The expropriation process is an established Provincial process used where required for land acquisition to enable development and infrastructure construction in Nova Scotia and has also been used to enable the construction and development of a now operating gold mine.

#### 25.16.1.3 Opportunities

The Deposit has been drill tested to only 500 m deep and remains open at depth; analogous geological systems, such as the Victorian Goldfields, are known to continue at depth for multiple kilometres. The most recent Mineral Resource Estimate dated November 15, 2021, demonstrated a substantial amount of underground Mineral Resources, including Measured and Indicated Resources of 1,159,000 ounces of gold (5,925,000 tonnes @ 6.09 g/t gold) and Inferred Mineral Resources of 418,000 ounces of gold (2,206,000 tonnes @ 5.89 g/t gold). A previous study on the Project has indicated that there could be potential for a long mine life that incorporated underground mining with an open pit mining scenario.

A future study will consider upgrading and expanding potentially mineable underground Mineral Resources as part of the longer-term mine development plan. Once surface mining has commenced and a significant portion of the ore body has been exposed, the company will likely undertake infill and expansion drilling, possibly from drifts off benches within the open pits, allowing for more effective and less expensive diamond drilling. Successful conversion of a major portion of the Inferred Resources to the Measured and Indicated category would allow for additional technical studies to be carried out with the intent of adding a ramp-access underground mining component to the Project, allowing for a combined open pit/underground scenario and significantly increasing the potential mine life of the Project.

As such, there are short- and longer-term opportunities for the Project such as:

- Ongoing infill drilling has the potential to reduce the strip ratio and positively impact NPV and IRR by upgrading Inferred Mineral Resources coincident with the current open pit designs based on Measured and Indicated Mineral Resources only.
- Potential for Mineral Resource expansion, particularly towards the west with further exploration of a 1.5 km long area along strike of the existing Deposit towards the past producing gold mine at Dolliver Mountain.

## 25.17 Conclusions

The results of the FS for the Company indicate that the Project has technical and financial merit using the inputs from various advanced studies. The Company anticipates filing an EARD for the Project in Q2 2022, which will focus on surface mine plan outlined in the FS over a mine life of approximately 11 years.

If a production decision is made with respects to the surface mining operation outlined in the FS, the Company will then consider further opportunity to incorporate underground mining, including the initiation of infill and expansion drilling from drifts off benches in the open pit, allowing for more effective and less expensive diamond drilling. Pending those results, the Company would then consider a supplementary study that will focus on adding an underground mining phase to the Project.

# 26. RECOMMENDATIONS

## 26.1 Introduction

The recommended program is focused on advancing technical and related studies toward an EARD being submitted for Class 1 Environmental Assessment in Q2 2022. Throughout 2022, the Company will continue with ongoing work to support further technical studies including geotechnical drilling, expanded surface water monitoring, metallurgical test programs and infill drilling. Additionally, the Company will continue exploration activities designed to target potential opportunities to expand the Project further, with emphasis on infill drilling to convert in-pit inferred mineral resources to Indicated mineral resources as well as demonstrating the continuation of the deposits along strike to the west of the existing Mineral Resource.

If a production decision is made, the Company will then commence the next phase of planning for underground mining, including infill and expansion drilling from drifts off benches in the open pit, allowing for more effective and less expensive diamond drilling. Pending those results, the Company would then consider a supplementary study that will focus on adding the underground mining phase to the Project.

Table 26-1 tabulates the budget recommendations and associated costs with advancing activities such as process, metallurgical, environmental and tailings facility studies, permitting, delineation drilling and detailed engineering.

**Table 26-1: Recommended Budget**

| Activity | Value (C$) |
|---|---:|
| Processing and Metallurgy Studies | 160,000 |
| Permitting and Environmental Studies | 2,269,000 |
| Detailed Engineering | 5,645,000 |
| Tailings Management Facility | 530,000 |
| Delineation Drilling | 2,000,000 |
| **Subtotal** | **10,604,000** |
| Contingency (10%) | 1,591,000 |
| **Total** | **12,195,000** |

The budget recommendations presented in Table 26-2 are focused on infill and expansion drilling, resource modelling and updated mine design.

**Table 26-2: Recommended Budget**

| Activity | Value (C$) |
|---|---:|
| Infill and Expansion Drilling - Existing Mine Plan | 1,000,000 |
| Growth Exploration | 1,500,000 |
| Resource Modelling and Updated Mine Design | 700,000 |
| **Total** | **3,200,000** |

### 26.1.1 Geology, Mining and Geotechnical Recommendations

- Complete further infill drilling to reduce the strip ratio and positively impact NPV and IRR by upgrading Inferred Mineral Resources coincident with the current open pit designs based on Measured and Indicated Mineral Resources only;
- Complete further infill drilling to improve the confidence within the Underground Resource

- Complete step out resource expansion drilling particularly towards the west with further exploration of a 1.5 kilometre long area along strike of the existing Goldboro Deposit towards the past producing gold mine at Dolliver Mountain;
- Complete drilling below the bottom of the current drilling which is approximately 500 m below surface.
- Complete further Geotech drilling to support further open pit and Underground studies
- Complete further detailed design within the current open pit designs to further refine the open pit mine plan
- Drill further Geotech holes in sectors 01, 1E, S4E, S5E and 5 that have limited information and reassess the sectors.
- Log additional data such as samples description and RMR parameters to improve the geotechnical database.
- Complete further triaxial cell and Brazilian tests.
- Review the current hydrogeologic model with an aim to evaluate the drawdown, as the stability of the final open slopes.
- Perform kinematic analysis to determine if failure mechanism may change. If a new sectorization is defined, then new slope stability analyses will be required.

### 26.1.2 Metallurgical and Processing Recommendations

The following further testwork is recommended as part of the detailed engineering of the process plant:

- Complete geotechnical boreholes and test pits in the mill and reagents areas to allow optimization of building foundations and anchoring.
- Complete material handling testing on crushed product.
- Complete slurry rheology confirmatory testwork for design of tailings pumps and pipeline.

### 26.1.3 Infrastructure Recommendations

This Technical Report is supported by engineering consistent with the detail required for the present level of study. The following areas are recommended to be pursued/investigated further for detailed engineering design phase:

- Continue communications with third party stakeholders in the Project area (Nova Scotia Power, Enbridge, Environmental stakeholders etc.).
- Selection of a haul truck model for detailed design of truck shop.
- Review of site personnel for detailed design of general office and plant office buildings.
- Detailed review of site fuel consumption for sizing of site fuel tanks.

It is recommended that a detailed geotechnical field investigation study be conducted to support the Project facilities locations and infrastructure design. Drilling should be completed specifically in the proposed building areas to support foundation recommendations and analysis. The geotechnical investigation should include:

- Complete foundation recommendations based on site stratigraphy and geotechnical engineering principles.
- Bearing capacity recommendation for shallow footings.
- Recommendations for slab on grade design.
- Seismic site classification.

- Optimization of infrastructure pad designs.
- Confirmation of depth to competent bedrock in specific infrastructure installation areas.
- Complete road design recommendations based on site stratigraphy and haul road engineering design principles.
- Detailed soil stratigraphy and size distributions.
- Soil chemistry in areas near historical tailings.
- Recommendations for soil cover, erosion control, and slope stability near water crossings.

#### 26.1.4 TMF Recommendations

Recommendations for the next stage of detailed design engineering to support construction of the Project are summarized below:

<u>Site Investigations</u>

- Additional site investigations should be completed as part of detailed design to namely confirm the foundation conditions below the TMF embankment, PAG1 waste rock, Polishing Pond, and the availability and suitability of potential construction materials. Additional undisturbed samples should be collected for laboratory testing based on the encountered conditions to refine the foundation material strength estimates and optimize the designs. Bulk samples of potential construction materials should be collected for suitability testing and confirmation of their compaction characteristics.

<u>TMF</u>

- Develop a Failure Modes and Effects Analysis (FMEA) for the TMF to identify and characterize potential risks associated with the TMF for the construction, operations and closure phases. The FMEA should include a risk evaluation matrix that provides a rating for the likelihood and consequence for each of the identified risks for the TMF and identify design mitigation measures to reduce the identified risks.
- Complete a dam breach analysis for the TMF to estimate the potential flood inundation zones that could result if the embankment associated with the facility were to fail. The estimated downstream inundation zone will help identify any residences, infrastructure and/or receptors that are at risk downstream of the TMF. An EPP should be developed to reduce the risk of human life loss and injury and minimize property damage in the event of the occurrence of an unusual or emergency at the TMF.
- Develop an OMS Manual for the TMF as part of the detailed design.
- Development of a full closure plan for the TMF based on the final design configuration.

<u>TMF Water Balance</u>

- Collection of site specific meteorological and hydrology data. This data will be used to refine seasonal runoff values and design storms to be used in future detailed design work.
- Calibration of the water balance should be completed, based on actual site data, during early years of operation

<u>Predictive Water Quality</u>

- Modelling of the cyanide degradation with contaminant transport software

- Modelling of the changes to ammonia, nitrate, and nitrite concentrations during cyanide degradation during operations
- Contaminant transport modelling within the TMF pond
- Gain an understanding of when the topsoil and NPAG source terms will reach equilibrium and incorporate them into post closure months.

### 26.1.5 Environment, Permitting, and Community Relations

Recommendations for the next stage of environmental studies, Mi'kmaq engagement, and community and regulator engagement for the Project:

- Continue to collect baseline data to document conditions prior to construction.
- Conduct modelling to determine the potential impact of the increased ambient sound levels (i.e., effects on wildlife) and to determine if additional mitigation measures are required to reduce the noise impacts at the nearby worst case points of reception. Develop and refine mitigation measures where potential impacts have been noted.
- Continue kinetic testing until a better understanding of acidic metal loading rates is obtained.
- Generation of ML/ARD Management and Monitoring plan to formalize procedures to confirm the effectiveness of current management strategies.
- Ongoing implementation of geochemistry data into block model to refine PAG location and tonnages.
- Consideration of new/future metallurgical test results for the tailings process water and seepage source term.
- Update and calibrate the hydrogeologic model with site specific monitoring data to predict the extent and magnitude of potential impacts to groundwater.
- Update and calibrate the hydrologic model with site specific monitoring data to predict the extent and magnitude of potential impacts to surface water.
- Complete a detailed effects assessment of fish and fish habitat in support the pending EARD submission.
- Develop a conceptual fish habitat offset plan based on expected direct and indirect impacts of the Project to support the EARD submission.
- Develop mitigations to reduce indirect effects to vascular and non-vascular species from the mining activity.
- Conduct further testing in areas of elevated archaeological potential (Low-Moderate 2 and Moderate Potential 6) to determine the presence or absence of archaeological resources in areas of Project development.
- Conduct additional delineation of historic tailings to support Phase 2 ESA predictions and to support development of a management strategy for the Historic tailings remaining in place.
- Continue engagement programs with local communities and municipal government in areas of education, training, and economic development.
- Continue engagement programs with Mi'kmaw organizations in areas of education, training, and economic development.
- Complete detailed design of all water management infrastructure to support permit applications in 2022 and 2023. Detail design is to be based on the updated modelling to be conducted for the EARD.

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# 28. GLOSSARY

The Mineral Resources and Mineral Reserves have been classified according to CIM (CIM, 2014). Accordingly, the resources have been classified as Measured, Indicated, or Inferred, the reserves have been classified as proven, and probable based on the Measured and Indicated Resources as defined below.

## 28.1 Mineral Resource

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 Project. Geological evidence is derived from the 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 Project. Geological evidence is derived from the 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 Reserve

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 prefeasibility or feasibility level as appropriate that include the 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 prefeasibility study or feasibility study.

A **Probable Mineral Reserve** is the economically mineable part of an Indicated, 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

**Table 28-1: Definition of Terms**

| Term | Definition |
|---|---|
| Assay | The chemical analysis of mineral samples to determine the metal content. |
| 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 | The initial process of reducing the 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 economical to recover its gold content by further concentration. |
| Dilution | Waste, which is unavoidably mined with ore. |
| Dip | The 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 the concentration of gold within the mineralized rock. |
| Hanging wall | The overlying side of an orebody or slope. |
| Igneous | Primary crystalline rock formed by the solidification of magma. |
| Kriging | An interpolation method of assigning values from samples to blocks that minimize the estimation error. |
| Lithological | Geological description pertaining to different rock types. |
| 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. |
| Sedimentary | Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks. |
| Stratigraphy | The study of stratified rocks in terms of time and space. |
| Strike | The direction of the line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction. |
| Sulphide | A sulphur-bearing mineral. |

| Term | Definition |
|---|---|
| Tailings | Finely ground waste rock from which valuable minerals or metals have been extracted. |
| Variogram | A statistical representation of the characteristics (usually grade). |

## 28.4 Abbreviations

The following abbreviations may be used in this Technical Report.

**Table 28-2: Abbreviations**

| Abbreviation | Unit or Term |
|---|---|
| % | percent |
| < | less than |
| > | greater than |
| ° | degree (degrees) |
| °C | degrees Celsius |
| µm | Micrometre or micron |
| AA | atomic absorption |
| AACE | Advancement of Cost Engineering |
| AAS | atomic absorption spectrometry |
| ACCDC | Atlantic Canada Conservation Data Centre |
| Ag | silver |
| Ai | abrasion index |
| Anaconda or the Company | Anaconda Mining Inc. |
| ANFO | ammonium nitrate/fuel oil |
| ARD | acid rock drainage |
| Au | gold |
| Ausenco | Ausenco Engineering Canada Inc. |
| BWi | bond ball work index |
| BHEM | borehole electromagnetic |
| BP | bedding parallel |
| BMA | Bulk Mineralogical Analysis |
| BR | Boston Richardson |
| BSE | Backscatter electron |
| CapEx | Capital Expenditures |
| CCA | capital cost allowances |
| CEPA | Canadian Environmental Protection Act |
| CGL | continuous-grind-leach |
| CIM | Canadian Institute of Mining, Metallurgy, and Petroleum |
| CLC | Community Liaison Committee |
| CMS | conventional fire assay-metallics screen methods |

| Abbreviation | Unit or Term |
|---|---|
| CNL | cyanide leaching |
| CoG | cut-off grade |
| CPD | cemented paste backfill |
| CRM | certified reference material |
| CWi | crushing work index |
| DD | diamond drilling |
| DFO | Fisheries and Oceans Canada |
| DGPS | differential global positioning system |
| dmt | dry metric tonnes |
| EARD | Environmental Assessment Registration Document |
| ECCC | Environment and Climate Change Canada |
| EDS | Environmental Design Storm |
| EEM | Environmental Effects Monitoring |
| EG | East Goldbrook |
| EM | electromagnetic |
| EMPA | electron microprobe analysis |
| EOM | end of mine |
| E-GRG | extended gravity recoverable gold |
| FA | fire assay |
| FAR | fresh air raise |
| FoS | Factor of Safety |
| ft | foot (feet) |
| $ft^2$ | square foot (feet) |
| $ft^3$ | cubic foot (feet) |
| FS | Feasibility Study |
| g | gram |
| G&A | general and administrative |
| $g/cm^3$ | grams per cubic centimetre |
| g/L | gram per litre |
| g/t | grams per tonne |
| Ga | giga-annum (1 billion years) |
| gal | gallon |
| GCNL | gravity concentration followed by cyanide leaching of tailings |
| GEMS | GEOVIA GEMS™ |
| g-mol | gram-mole |
| GMR | gross metal royalty |
| gpm | gallon per minute |

| Abbreviation | Unit or Term |
|---|---|
| GPS | global positioning system |
| GRG | gravity recoverable gold |
| ha | hectare (10,000 $m^2$) |
| HDPE | High Density Poly Ethylene |
| HMC | heavy mineral concentrate |
| HPGR | High Pressure Grinding Rolls |
| HVAC | heating, ventilation, and air conditioning |
| IA | Industrial Approval |
| ICP | induced couple plasma |
| ICP-AES | inductively coupled plasma atomic emission spectrometry |
| ICP-OES | Inductively coupled plasma optical emission spectrometry |
| ID2 | inverse distance squared |
| ID3 | inverse distance cubed |
| IDF | Inflow Design Flood |
| InnoveExplo | InnovExplo inc. |
| IP | induced polarization |
| IRA | inter-ramp angle |
| IRR | internal rate of return |
| ISR | inductive source resistivity |
| JK DWT | Julius Kruttschnitt Mineral Research Centre drop-weight test |
| JORC | Joint Ore Reserves Committee |
| kg | kilogram |
| km | kilometre |
| $km^2$ | square kilometre |
| KMKNO | Kwilmu'kw Maw-klusuaqn Negotiation Office |
| kt | thousand tonnes |
| KV | kriging variance |
| L | litre |
| lb | pound |
| LBMA | London Bullion Market Association |
| LG | Lerchs-Grossman |
| LHD | Load Haul Dump |
| LIDAR | light detection and ranging |
| LNG | Liquefied natural gas |
| LOM | life of mine |
| m | metre |
| M | million |

| Abbreviation | Unit or Term |
|---|---|
| Ma | mega annum (1 million years) |
| MCR | Mid-continent rift |
| MDMER | Metal and Diamond Mine Effluent Regulations |
| MEC | Minerals Engineering Centre |
| Mercator | Mercator Geological Services Limited |
| mg/L | milligrams/litre |
| MgO | magnesium oxide |
| MineTech | MineTech International Limited |
| ML | metal leaching |
| mm | millimetre |
| $mm^2$ | square millimetre |
| $mm^3$ | cubic millimetre |
| MMR | magnetometric resistivity |
| MODG | Municipality of the District of Guysborough |
| MOU | Memorandum of Understanding |
| Moz | million troy ounces |
| MRB | MRB & Associates Ltd. |
| MSHA | Mine Safety and Health Administration |
| MSO | Mineable Stope Optimiser |
| Mt | million tonnes |
| MT | magnetotelluric |
| MTM | Modified Transverse Mercator |
| Mtpa | million tonnes per annum |
| MUN | Memorial University of Newfoundland |
| NAD83 | North American Datum 1983 |
| NAG | Net Acid Generation |
| NAPS | National Pollutant Surveillance network |
| NI 43-101 | Canadian National Instrument 43-101 |
| NPAG | non-potentially acid generating |
| NN | nearest neighbour |
| NNW | north-northwest |
| NPRI | National Pollutant Release Inventory |
| NPV | Net Present Value |
| NSDNR | Nova Scotia Department of Natural Resources |
| NSE | Nova Scotia Environment |
| NSR | Net Smelter Return |
| OK | ordinary kriging |

| Abbreviation | Unit or Term |
|---|---|
| Onitap | Onitap Resources Inc. |
| Orex | Orex Exploration Inc. or Exploration Orex Inc. |
| ORP | oxidation reduction potential |
| OSA | overall slope angle |
| Osisko | Osisko Mining Corporation |
| OTC | Over the Counter |
| oz | troy ounce |
| P&E Mining | P&E Mining Consultants Inc. |
| PAG | Potentially Acid Generating |
| Pb | lead |
| PEA | Preliminary Economic Assessment |
| PLC | programmable logic controller |
| PMF | potential mill feed |
| ppb | parts per billion |
| ppm | parts per million |
| PV | Present Value |
| QA | quality assurance |
| QC | quality control |
| QEMSCAN | quantitative evaluation of materials by scanning electron microscopy |
| QP | Qualified Persons |
| RC | reverse circulation |
| RF | Revenue Factor |
| Rh | rhodium |
| ROM | run of mine |
| RQD | rock quality description |
| RWi | bond rod mill work index |
| S | sulphur |
| SAG | semi-autogenous grinding |
| SEC | Securities and Exchange Commission |
| SG | specific gravity |
| SMC | SAG mill comminution |
| SMU | selective mine unit |
| SSE | south-southeast |
| t | tonne (metric ton) (2,204.6 pounds) |
| t/d | tonnes per day |
| t/h | tonnes per hour |

| Abbreviation | Unit or Term |
|---|---|
| TEM | transient electromagnetic |
| the Deposit | Goldboro Gold Deposit |
| the Project | Goldboro Project |
| the Property | Goldboro Property |
| TSS | total suspended solids |
| UCS | Unconfined Compression Test |
| US | United States |
| USEPA | United States Environmental Protection Agency |
| VC | valued component |
| VL | vat leaching |
| VLF-EM | very low frequency electromagnetic |
| VTEM | vertical time domain electromagnetic |
| WMP | Waste Management Plan |
| wmt | wet metric tonnes |
| WG | West Goldbrook |

# Appendix A: QP Certificates of Authors

# Appendix B: Drill Hole Header Summary

# Appendix C: EDA Probability Plots and Histograms

# Appendix D: Visual Comparison of Block Model Elements

# Appendix E: Risk Register