N.I. 43-101 TECHNICAL REPORT & PRELIMINARY ECONOMIC ASSESSMENT OF THE GOLIATH GOLD COMPLEX

Ontario, Canada

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Goliath Project Site

Goldlund Project Site

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1 SUMMARY

1.1 Introduction

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Treasury Metals Inc. (Treasury Metals) to summarise the results of a preliminary economic assessment (PEA) of the Goliath Gold Complex. The report was prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The PEA was prepared in accordance with "N.I. 43-101 Standards of Disclosure for Mineral Projects". Readers are cautioned that the PEA report is preliminary in nature.

The N.I. 43-101 responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Treasury Metals to manage and coordinate the work related to the NI 43-101. Ausenco also developed the PEA-level design and cost estimate for the process plant and general site infrastructure.
- AGP Mining Consultants (AGP) was commissioned to complete the mineral resource estimate for the Goliath and Miller projects, and to design the open pit and underground mine plan, mine production schedule, and mine capital and operating costs.
- CGK Consulting Services (CGK) was commissioned to complete the mineral resource estimate for the Goldlund project.
- Knight-Piésold (KP) was commissioned to develop the PEA-level design and cost estimate for the tailings storage facility and site water management infrastructure.

1.2 Terms of Reference

The report supports disclosures by Treasury Metals in a news release dated February 2, 2021 entitled "Treasury Metals Announces Positive Preliminary Economic Assessment for Goliath Gold Complex".

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

The Goliath Gold Complex area contains three deposits: Goliath, Goldlund and Miller. Treasury Metals owns 100% of Goliath Gold Complex.

1.3 Property Description & Location

The Goliath Gold Complex location is presented in Figure 1-1. The Goliath property covers approximately 7,601 ha and is defined by mineral and surface rights that are 100% held by Treasury Metals. Of this total, the mineral rights cover approximately 7,511 ha..

Figure 1-1: Location of the Goliath Gold Complex

Source: Treasury Metals (2021).

The Goliath property has one deposit, the Goliath deposit, and is located as follows:

- on 1:50,000 scale NTS Mapsheets 052F/09 (Dyment), 10 (Wabigoon), 15 (Dryden), and 16 (Big Sandy Lake)
- at approximately 49°45.4' North and 92°33.0' West
- at approximately 532,441 mE; 5,511,624 mN, Zone 15U (NAD83 datum) Universal Transverse Mercator (UTM) coordinates
- in the Kenora Mining Division
- in the Dryden MNR District
- in the Zealand and Hartman Townships

The Goldlund-Miller property covers approximately 27,118 ha and is defined by mineral rights that are 100% held by Treasury Metals. Two deposits, Goldlund and Miller, comprise the Goldlund-Miller property.

The Goldlund deposit is located as follows:

- on the Goldlund-Miller property
- on 1:50,000 scale NTS Mapsheets 052F16 (Big Sandy Lake), 052K/01 (Hudson) and 052J/04 (Sioux Lookout)
- at approximately 49°54' North and 92°20.5' West
- at approximately 547000 E; 5527500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Patricia Mining Division
- in the Sioux Lookout MNR District
- in the Echo and Pickerel Townships

The Miller deposit is located as follows:

- on 1:50,000 scale NTS Mapsheet 052F16 (Big Sandy Lake)
- at approximately 49°57' North and 92°15' West
- at approximately 534000 E; 5534500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Pickerel Township

1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The Goliath Project is located in the Kenora Mining Division in northwestern Ontario, approximately 4 km northwest of the Village of Wabigoon, 20 km east of Dryden and 2 km north of the Trans-Canada Highway 17. The Goldlund and Miller Projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath Project, off Highway 72. Aerial imagery of the Goliath Project and the Goldlund Project are provided in Figures 5-1 and 5-2, respectively.

Access to the Goliath Project is north from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Anderson and Tree Nursery Roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals. Access to the Goldlund site is east off Highway 72 via Goldlund Mine Road. The Miller Project site is accessed via forestry road east off Highway 72. Access roads for the Goldlund and Miller sites are maintained by the Sustainable Forest Licence Holder (Domtar) for the area.

All major industrial services and supplies are available in Dryden and Sioux Lookout and the area is serviced by both the Dryden Airport and Sioux Lookout Airport. The Goliath Project is located 20 km from Dryden, which has a population of 5,586 according to the Statistics Canada 2016 census. The Goldlund and Miller projects are located 43 km and 35 km, respectively, south of Sioux Lookout, which has a population of 5,272. The Goliath Gold Complex is located about 300 km northwest of the City of Thunder Bay, a major economic centre along the Trans-Canada Highway and port at the northwest head of the St. Lawrence Seaway on Lake Superior.

At this time, Treasury Metals holds the sufficient surface rights necessary for any potential future mining operations including tailings storage areas, waste disposal areas, and a processing plant.

1.5 History

The first gold mining on record in the region was in Van Horne Township in the early 1900s with very limited gold production from auriferous veining in biotite schist within the regional Wabigoon fault system. Sporadic exploration was carried out along the belt throughout the 1900s with only limited documentation of exploration activity conducted on the property.

1.5.1 Goliath Property

The earliest known government report covering the larger Dryden-Sioux Lookout Belt is the Ontario Department of Mines Report and Geology Map by Satterly (1941). In 1956-57, Compton-Wabigoon conducted geological mapping, magnetometer surveys, and the completion of two diamond drill holes totalling 458 m to explore the mineral potential of the major iron formation unit located in Lots 1-4, Concession V and VI, along the northern boundary of the property. Also in 1956, G.L. Pidgeon completed surface work and one shallow drillhole (drilled south) testing a sphalerite showing in the south half of Lot 6, Concession IV (Fraser Option legacy claim 0134).

Three major mining companies conducted exploration work on the Thunder Lake gold deposit (Goliath deposit) from 1989 to 1999 (last field work 1998). These are Teck Exploration Ltd. (Teck), Corona Gold Corporation (Corona), and Laramide Resources Ltd. (Laramide). At that time, the property held by all three companies covered more than 1,300 ha. Teck held the majority of the property and all of the surface exposure.

Exploration and resource development work at Goliath was undertaken by Teck from 1989 to 1999 on what was then called the “Thunder Lake Property”. During this period, the property was divided into two properties called “Thunder Lake East” and “Thunder Lake West”. The property was optioned to Corona, previously called Continental Caretech Corporation (CCC), in which CCC could earn an interest in the project under terms of an initial agreement dated January 3, 1994. Corona funded the exploration work from 1994 to 1999, but Teck remained the project operator both designing and running all field exploration activities.

In 1998, Teck completed an underground exploration and bulk sampling program at a cost of $1,929,071. This entire underground program, from surface site preparation through final closure plan, was completed between May 15 and September 15, 1998. The underground work consisted of a 27 m long inclined trench provided a 9 m high outcrop face suitable for the construction of a portal collar. A decline was prepared at a grade of 15% with the portal located just north of Norman Road and the north boundary of the Laramide property. Four bulk samples from the Main Zone (No. 1 and No. 2 shoots) totalling 2,375 tonnes were excavated consisting of blasted muck from drift rounds and slashed and material from a 400 tonne take-down-back test mining area grading in excess of 3 g/t Au. After the underground work was completed, the portal was sealed and the area contoured, reseeded, and fully remediated in late 1999.

1.5.2 Goldlund Property

Exploration activities on the Goldlund Project date from the 1940s, where in 1941 A. Ward and R. Lundmark (two prospectors working for the Mosher group) discovered gold mineralisation in the southwestern part of Echo Township (Page, 1984). From 1946 to 1952 there were significant exploration activities carried out on the Newlund Mines Limited and Windward Gold Mines prospects. The Newlund prospect was extensively explored by 4,570 m of underground drifts and crosscuts on four levels (200 ft, 350 ft, 500 ft, and 800 ft), and 6,220 m of core drilling from a 255 m deep vertical shaft. The 200 ft level on the Newlund prospect was extended more than 3.2 km to the west to connect with the 68 m vertical shaft on the Windward prospect, crossing the entire Windward claim block (Page, 1984). From 1952 to 1973, there was only limited exploration activities carried out on the Echo Township gold prospects.

In 1974, Goldlund Mines Limited and Rayrock Mines Limited entered into an agreement and rehabilitated the surface facilities including the installation of a new headframe and hoist and dewatering the underground workings to the second level (350 ft). A program of bulk sampling, underground chip sampling, and core drilling of 41 holes totalling 4,932 ft (approximately 1,500 m) was carried out. No further activities were carried out, as the prospect was deemed uneconomic given the gold price at that time (Page, 1984).

In total, approximately 143,825 m of drilling has been completed in 808 surface drill holes, and approximately 18,624 m of drilling has been completed in 480 underground holes. Additionally, Tamaka carried out a trenching program in 2012 that included the excavation, stripping, mapping, channel sampling and a detailed structural analysis.

From mid-1982 to early 1985, Campbell Resources Inc. (Campbell Chibougamau), through its wholly owned subsidiary Goldlund Mines Limited, operated an underground mine and an open pit mine and processed material through the mill at the site. Pieterse (2005) compiled the production records that show underground mine production of 100,000 tons (approximately 90,700 tonnes) at an estimated grade of 0.15 oz/ton Au (approximately 5.14 g/t Au) and open pit production of 43,000 tons (approximately 39,000 t) at an estimated grade of 0.17 oz/ton Au (approximately 5.83 g/t Au).

1.5.3 Miller Property

There has been no historical exploration or drilling activities on the Miller deposit prior to 2018. In 2018 and 2019, First Mining completed two drill programs on Miller, as described in Section 10 of this report.

1.6 Geology Setting & Mineralisation

The Goliath Gold Project is located in the Archean Eagle-Wabigoon-Manitou greenstone belt in the Wabigoon Subprovince of the Superior Province. In the immediate area of the deposit, a 100 to 150 m thick unit of intensely deformed and variably altered, fine- to medium-grained, muscovite-sericite schist and biotite-muscovite schist with minor metasedimentary rocks hosts the most significant concentrations of gold in the Main and C Zones of the deposit.

Native gold and silver are associated with finely disseminated sulphides, coarse-grained pyrite and very narrow light grey translucent “ribbon” quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite. The alteration consists of primarily sericitisation and silicification in association with the gold mineralisation.

At Goliath, the gold-bearing zones strike from 090° to 072° with dips that are consistently between 72° and 78° south or southeast. The mineralised zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. In the Goliath deposit, higher grade gold mineralisation occurs in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west. The main area of gold, silver and sulphide mineralisation and alteration occurs up to a maximum drill-tested vertical depth of ~805 m, over a drill-tested strike-length in excess of 2,500 m. The mineralised zones remain open at depth.

The Goldlund Project is situated in northwestern Ontario approximately 60 km by road east of the town of Dryden, with a land package that covers a strike-length of over 50 km of greenstone belt in the Archean Wabigoon Subprovince. Historical gold production from the Goldlund and Windward mines is reported to be 18,000 oz of gold, with mining activities carried out between 1982 and 1985 using both open pit and underground mining methods.

Gold mineralisation is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. The stockwork zones are hosted in albite-trondhjemite to diorite (granodiorite) strata-parallel sills, which dip from vertical to -80° southward and range in thickness from 14 m to 60 m. The stockwork zones form bands within the granodiorite sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine- to medium-grained pyrite and magnetite.

The mineralised sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the project.

The gold mineralisation has been interpreted as a series of nine northeast-trending sub-parallel zone wireframes, considering a nominal 0.1 g/t Au threshold. Wireframes of Zones 1, 7, and 5 consist principally of gold mineralisation associated with the stockwork veins in the large granodiorite sills, while wireframes of Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralisation associated with stockwork veins that are hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only a minor contribution from the granodiorite sills. While the Qualified Person for this section of the report believes that the interpretation of the mineralised zone wireframes is suitable for the estimation of mineral resources, the development of a 3D model of lithology, structure, and alteration would help to improve the interpretation of the mineralised zones and the understanding of the controls on gold mineralisation.

1.7 Deposit Types

The Goliath Project hosts a hybrid deposit-type model, also known as a “Pre-orogenic Atypical Greenstone Belt Gold Model” as a promising genetic model to explain the geology, structures and mineralisation observed within the Goliath deposit. In this model, early gold-rich volcanogenic sulphide mineralisation is overprinted by subsequent deformation and alteration events which can contribute to further concentration and/or remobilising of both precious and base metals. This model also integrates potential VMS and Magmatic Hydrothermal Archean Lode Gold Deposit (“Magmatic Hydrothermal”) models in the formation of the deposit. It is likely that the Goliath deposit does not fit into any one idealised model and neither should be discounted.

The Goldlund Project hosts Archean, shear zone-hosted quartz vein mineralisation (Archean lode-gold), occurring as extensional quartz vein systems, particularly between rocks with high competency contrast. Archean lode-gold deposits occur in a broad range of structural settings, and at different crustal levels, but they share a similarity in ore fluid characteristics. Mineralisation is typically late tectonic, occurring after the main phases of regional thrusting and folding, and generally late-syn to post-peak metamorphism with most of the significant deposits in areas of greenschist facies. Many deposits are related to the reactivation of earlier structures.

Archean lode-gold occurrences are common in the Sandybeach Lake – Sioux Lookout area and are concentrated in the Southern and Central volcanic belts. Vein systems in both belts are the product of Stage 3 deformation and are related to the northeast-southwest extension associated with northwest-southeast compression and shortening; the brittle-ductile deformation near the steep, northeast-trending shear zones; and the tightening of the Stage 3 folds.

The Miller Project mineralisation fits an Archean shear-zone hosted quartz vein model (Archean lode gold). The Archean lode gold occurrences are common in the Sandy Beach Lake - Sioux Lookout area and are concentrated in the Southern and Central Volcanic Belts.

1.8 Exploration

Since 2008, Treasury Metals has focused its exploration work on the western half of the property in order to evaluate the gold potential of the Goliath deposit. During this 12-year period, exploration activities consisted of re-establishing the former Teck exploration grid, geological mapping and sampling, prospecting, the completion of structural studies, trenching and channel sampling, the completion of a ground IP geophysical survey and two airborne geophysical surveys, downhole IP and tomography surveys, metallurgical testing, mineral resource estimations of the main deposit (including Preliminary Economic Analyses in 2012 and 2017) and the completion of 18 diamond drilling programs.

1.9 Drilling

The mineralisation was sampled over the years with multiple campaigns of core drilling by Teck-Corona and Treasury Metals since the 1990s. The drill database is now a mix of historical data and more recent data collected by Treasury Metals from 2008 through to 2020. Both data types were used in the resource estimate. The mineral resource estimate for Goliath is supported by 726 surface drill holes with an aggregated length of 238,036 m and 96,912 assays.

Treasury Metals has not conducted any drill programs on the Goldlund Project since it acquired the property. Diamond drilling on the Goldlund Project has been carried out since the 1940s. There are 856 drill holes totalling 152,787.7 m of surface drilling and 480 drill holes totalling 18,626 m of underground drilling in the July 20, 2020 drill hole database, as compiled by First Mining.

The most recent drilling was carried out by First Mining in 2019 and 2020, with 14 drill holes totalling 2,506 m of drilling in 2019, and 34 holes totalling 6,452 m of drilling in 2020. The drilling was focused within and around the defined resource area at Goldlund (Main Zone), with an initial target of defining and extending mineralisation in the eastern and western portions of the deposit.

Treasury Metals has not conducted any drill programs on the Miller deposit since it acquired the property. All drilling on the Miller Project was completed by First Mining in 2018 and 2019 targeting a geophysical anomaly, with 40 drill holes totalling 7,386 m of drilling.

1.10 Sample Preparation, Analyses & Security

The analytical laboratory used by Teck-Corona prior to the 1990s is believed to be TSL Laboratory in Saskatoon. Assays from that period were recovered from historical drill logs. Treasury Metals used Accurassay Laboratory in Thunder Bay from 2008 to 2015 and then Activation Laboratory from 2016 to 2020. Accurassay was accredited by ISO/IEC 17025 and ActLab in Dryden was assessed and found to be in conformance to the ISO 9001:2015 standard.

The Treasury Metals drill core is analysed for gold on all samples and silver and trace element geochemistry on selected samples. Gold is typically analysed by fire assay with atomic absorption finish or gravimetric finish depending on the grade. Pulp metallic screen assays are routinely carried out on high grade samples.

Prior to 1997, only a few QA/QC guidelines existed, and monitoring programs were not commonly conducted by mining companies; consequently, a QA/QC program for the historical Teck-Corona drill holes is not known to exist and assumed is by AGP to be non-existent. The historical holes were validated using twin drilling. In 2008, Treasury Metals implemented a QA/QC program consisting of blanks and CRMs. In 2009 Treasury Metals added the insertion of quarter core duplicates and in 2017 added a check assay program at an umpire laboratory. The program was found to be well followed with resubmission of sample batches when a QA/QC failure occurred.

The majority of the 545 bulk density sample measurements were carried out on 10 cm core pieces submitted to the analytical laboratory. The remaining 19% were completed in house on uncoated, air-dried samples. The core at Goliath is solid with little to no pore and the in-house density measurements compared well with the laboratory values.

Core handling, core storage, and chain of custody are consistent with industry best practices.

Assays of the drillhole samples and channel samples for the Goldlund Project have been carried out between 2007 and 2020 by Accurassay and SGS Canada Inc. (SGS) in Red Lake, Ontario, Lakefield, Ontario, and Vancouver, BC. Accurassay is an accredited facility conforming to the requirements of CAN P-4E ISO/IEC 17025 and CAN-P-1579. The SGS laboratories are also accredited facilities conforming to the CAN P-4E ISO/IEC 17025:2017 requirements. ActLabs in Thunder Bay and Ancaster, Ontario carried out independent umpire check assays for the 2017-2018 drilling program samples. ActLabs is an accredited facility conforming to the CAN P-4E ISO/IEC 17025:2017 and ISO 9001:2015 requirements.

Assays of drill core samples prior to 2006 were carried out by commercial laboratories Cochenour Fire Assaying and Paul's Custom Assaying Ltd., both of Red Lake, Ontario. Both assay laboratories operated in the Red Lake area for decades. There is no description available for the sample preparation and assaying or QA/QC programs for the samples prior to 2006.

The assay laboratories that have contributed results to the drillhole database used for the estimation of mineral resources are all independent of Tamaka, First Mining and Treasury Metals. At no time were employees of Tamaka, First Mining or Treasury Metals involved in the preparation or analysis of the samples.

The chain of custody and sample security are well documented for the Tamaka 2007-2008, 2011 and 2013-2014 drilling programs and for the First Mining 2017-2018 and 2019-2020 drill programs. Both Tamaka and First Mining personnel have taken reasonable measures to ensure the samples were kept secure prior to the shipment of the samples to the respective assay laboratories for analysis.

1.11 Mineral Processing & Metallurgical Testing

Metallurgical testwork programs were conducted on Goliath samples between 2011 and 2020, and 2012 for Goldlund samples. The following sources of technical and project information were referenced in developing the process plant design for the preliminary economic assessment:

- 2011 G&T Metallurgical Services Ltd. Pre-Feasibility Metallurgical Testing Goliath Gold Project. KM2906.
- 2012 ALS Metallurgy (formerly G&T Metallurgy), Feasibility Metallurgical Testing, Treasury Metals Incorporated. KM3406.
- 2017 ALS Metallurgy, Metallurgical Test Work on Goliath Gold Samples, Treasury Metals Incorporated. KM5262.
- 2017 Base Metallurgical Laboratories, Metallurgical Testing of Goliath Project. BL0172.
- 2020 Technical Report Re-Issue, Goldlund Gold project, Sioux Lookout, Ontario.
- 2020 Metallurgical Testing of the Goliath Gold Project. BL0697.
- 2013 SGS Scoping Study and Comminution testing on samples From the Goldlund Project. 13665-001.

The parameters presented in Table 1.1 were developed from the testwork to support the development of the process design criteria.

Table 1.1: Parameters Developed From Testwork

Parameter	Unit	Value
Abrasion Index	g	0.086
Bond Ball Mill Work Index	kWh/t	15.7
Leach Feed Grind (P_{80})	µm	75
Cyanide Addition	kg/t	0.5
Lime Addition	kg/t	0.3
Gravity Gold Recovery	%	25
Leach Gold Recovery	%	91
Overall Gold Recovery	%	93.6

Source: Ausenco (2020).

These parameters are described in more detail below:

- The abrasion index is an average derived from Goliath testwork.
- The Bond ball mill work index is the 75th percentile from the Goldlund deposit representing the most competent ore for design.
- The leach feed grind size P_{80} of 75 µm was selected based on the available Goliath and Goldlund testwork. Leach tests were conducted on samples from all three deposits at the selected grind size. Goliath testwork indicates that a coarser grind (115 µm) is possible while maintaining design gold extraction.
- The cyanide and lime additions were calculated from leach tests at the selected grind target, and leach tests in which a lower cyanide concentration (0.5 g/L) was applied, as this did not display a significant reduction in gold extraction.
- The gravity recovery was estimated based on the available limited testwork and typical plant operating conditions.
- The leach and overall gold recovery was calculated using the gravity recovery discussed previously and the gold extraction in the leach tests available at the selected grind sizes.
- No metallurgical testing has been completed on Miller samples. For this study, Goldlund metallurgical characteristics have been assumed based on the two deposits having similar geology.

1.12 Mineral Resource Estimates

For Goliath, effective December 16, 2020 AGP completed an update of the July 1, 2019 estimate completed by P&E Mining Consultants Inc. The mineral resource presented herein is in conformance with the CIM Mineral Resource definitions (2014) referred to in the “N.I. 43-101 Standards of Disclosure for Mineral Projects”. The estimate takes into account all data that was available prior to October 6, 2020.

To meet the CIM definitions of reasonable prospects of economic extraction, a cut-off of 0.25 g/t Au was used for the resource amenable to open pit extraction, and a cut-off of 1.6 g/t Au was used for the material below the resource constraining shell that is considered to be amenable to underground extraction. The determination of the cut-off grade was based on a gold price of US$1,700/oz and a silver price of US$23/oz with 95.5% gold and 62.6% silver recoveries.

To further assess reasonable prospects of economic extraction, a Lerchs-Grossman optimised shell was generated to constrain the potential open pit material. Grade shells at the underground cut-off grade of 1.6 g/t Au were generated beneath the resource pit shell. The grade shells were examined by AGP's engineering team for the likelihood of being a coherent mining shape with reasonable prospect of being accessed. Those that did not meet the criteria were removed from consideration.

The mineral resource estimate presented herein is categorised as a mix of measured, indicated, and inferred resources. The reported resources are expressed in metric tonnes. Metal contents are presented as in-situ ounces.

Within the resource constraining shell, at the greater than 0.25 g/t Au cut-off grade selected, the updated model returns a total of 1.5 million measured tonnes grading at 1.90 g/t Au and 6.7 g/t Ag containing 89,800 oz of gold and 316,700 oz of silver. Indicated tonnes amounted to 27.0 Mt grading at 0.87 g/t Au and 3.0 g/t Ag containing 757,000 oz of gold and 2.6 Moz of silver. The total measured and indicated resources within the constraining shell amounted to 28.4 Mt grading at 0.93 g/t Au and 3.2 g/t silver containing 846,800 oz of gold and 2.9 Moz of silver.

Below the constraining shell and reported at a greater than 1.6 g/t Au cut-off grade, the updated model returns 98,000 tonnes of measured resources grading at 4.94 g/t Au and 20.8 g/t Ag containing 15,500 oz of gold and 65,300 oz of silver. Indicated resources amounted to 2.6 Mt grading 3.16 g/t Au and 7.6 g/t Ag containing 263,100 oz of gold and 632,700 oz of silver. The total measured and indicated resources below the constraining shell amounted to 2.7 Mt grading at 3.22 g/t Au and 8.1 g/t Ag containing 278,700 oz of gold and 698,000 oz of silver.

Inferred resources within the resource constraining shell and reported at greater than 0.25 g/t Au cut-off grade, amounted to 3.6 Mt grading at 0.65 g/t Au and 2.1g/t Ag containing 76,100 oz of gold and 247,000 oz of silver. Below the constraining shell and reported at a greater than 1.6 g/t Au cut-off grade, the updated model returned 704,000 tonnes of inferred resources grading at 2.75 g/t Au and 5.6 g/t Ag containing 62,200 oz of gold and 125,900 oz of silver.

The Goliath deposit total measured resources amounted to 1.6 Mt grading at 2.09 g/t Au and 7.58 g/t Ag containing 105,300 oz of gold and 382,000 oz of silver. Indicated resources amounted to an additional 29.5 Mt grading 1.07 g/t Au and 3.39 g/t Ag containing 1.0 Moz of gold and 3.2 Moz of silver. The total measured and indicated resources amounted to 31.1 Mt grading at 1.13 g/t Au and 3.60 g/t Ag containing 1.1 Moz of gold and 3.6 Moz of silver. Inferred resources added an additional 4.3 Mt grading 0.99 g/t Au and 2.67 g/t Ag containing 138,300 oz of gold and 372,900 oz of silver.

The Goldlund mineral resources estimate has been carried out in accordance with the CIM's "Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" (2019). The mineral resources estimate has been generated from drill hole data and the interpretation of a geological model that identifies the spatial distribution of the gold grades. The interpolation parameters have been defined based on the drill hole data and the geological interpretation and geostatistical analysis of that data.

The mineral resources have been classified by proximity to data locations and quality of the data, and have been reported in accordance with CIM's "Standards on Mineral Resources and Reserves as required by N.I. 43-101" (2014).

The mineral resources for the Goldlund Project were estimated using a 3D block model that was constructed using MineSight® 15.4 software with the block size chosen to reflect the potential selective mining unit (SMU) of 5 m x 5 m x 5 m, given the anticipated open-pit mining scenario. The block model covers an area of approximately 4.7 km by 2.5 km in plan view, and approximately 800 m vertically.

Block gold grade estimates were developed using an indicator kriging to define the proportion of high-grade material in a block and then ordinary kriging was used to estimate gold grades for the low-grade and high-grade domains separately. The final block grade is then a proportional weighted average grade of the low- and high-grade kriged estimates. This combined kriging methodology is referred to as "probability assisted kriging" or PAK.

The grade block model estimation methodology considered the domains to be the principal control, with the secondary control by the mineralised zone wireframes for the estimation of the gold grades. The density item in the block model was assigned the average density of the drill core measurements by zone.

To meet the CIM requirements of reasonable prospects of eventual economic extraction, the mineral resources amenable to open pit extraction are reported at a cut-off grade of 0.25 g/t Au inside an optimised mineral resources pit shell and mineral resources amenable to underground extraction are reported at a cut-off grade of 1.6 g/t Au inside a constraining shell that considered contiguous mineralisation. The cut-off grade was based on gold price of US$1,700/oz and a gold recovery of 89%.

The mineral resources for the Goldlund Project amenable to an open pit mining scenario, within an optimised constraining shell, at a 0.26 g/t Au cut-off grade are estimated to be 24.3 Mt of indicated material grading 1.07 g/t Au for a total of 840 koz of gold. There are additional inferred mineral resources amenable to an open pit mining scenario, which are estimated to be 14.4 Mt grading 0.56 g/t Au for a total of 260 koz of gold.

The mineral resources amenable to an underground mining scenario, for contiguous blocks below the optimised constraining shell, are estimated to be 233 kt grading 6.8 g/t Au totalling 51 koz of gold. This brings the total inferred mineral resources to be 14.6 Mt, grading 0.66 g/t Au totalling 311 koz of gold. The effective date of the Goldlund Project mineral resources is October 23, 2020.

The effective dates of the Goliath, Goldlund and Miller resource estimates are as follows:

- Goliath – December 16, 2020
- Goldlund – October 23, 2020
- Miller – October 26, 2020

Mineral resources for each are summarised in in Table 1.2.

Table 1.2: Mineral Resources for the Goliath Gold Complex

Deposit	Classification @ Cut-off Grade (g/t Au)	Tonnes (kt)	Au Grade (g/t Au)	Contained Au (koz)
Goliath	Measured @ OP 0.25 g/t Au	1,471	1.90	90
Goliath	Measured @ UG 1.60 g/t Au	98	4.94	16
Total Measured		**1,569**	**2.09**	**105**
Goliath	Indicated @ OP 0.25 G/t Au	26,956	0.87	757
Goliath	Indicated @ UG 1.60 G/t Au	2,592	3.16	263
Goldlund	Indicated @ OP 0.26 G/t Au	24,300	1.07	840
Total Indicated		**53,848**	**1.07**	**1,860**
Total Measured & Indicated		**55,417**	**1.10**	**1,965**
Goliath	Inferred @ OP 0.25 G/t Au	3,644	0.65	76
Goliath	Inferred @ UG 1.60 G/t Au	704	2.75	62
Goldlund	Inferred @ OP 0.26 G/t Au	14,400	0.56	260
Goldlund	Inferred @ UG 1.60 G/t Au	233	6.80	51
Miller	Inferred @ OP 0.26 G/t Au	1,981	1.24	79
Total Inferred		**20,962**	**0.78**	**528**

Notes: OP = open pit; UG = underground. Mineral resources are estimated in conformance with the CIM mineral resource definitions referred to in N.I. 43-101 Standards of Disclosure for Mineral Projects. This mineral resource estimate covers the Goliath deposit, the Goldlund deposit, and the Miller deposit. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The quantity and grade of the reported inferred mineral resources in this estimation are conceptual in nature and are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. For these reasons, an inferred mineral resources has a lower level of confidence than an indicated mineral resources and it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Goliath:

Mineral resources are reported within optimised constraining shell using a gold price of US$1,700/oz and a silver price of US$23/oz and recoveries of 95.5% for gold and 62.6% for silver. Grades were estimated using 1.5 m capped composites using ordinary kriging for the Main and C Zones and ID^3 for all other zones.

Goldlund:

Mineral resources are reported within an optimised constraining shell using a gold price of US$1,700/oz and gold recovery of 89%. Gold grades were estimated using 2.0 m capped composites within nine mineralised zones using ordinary kriging.

Miller:

Mineral resources are reported within an optimised constraining shell using a gold price of US$1,700/oz and gold recovery of 89%. Grades were estimated using 2.0 m capped composites within the granodiorite domain using inverse distance cubed interpolation.

Summation errors may occur due to rounding.

1.13 Mining Methods

The mine designs and schedule for both the open pit and underground utilise inferred resources as part of the analysis. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The preliminary economic assessment is preliminary in nature in that it includes inferred mineral resources that are considered too speculative to have economic considerations applied to them and should not be relied upon for that purpose.

The Goliath Gold Complex PEA is based on the mining of three deposits: Goliath, Goldlund and Miller. All three areas would be mined by open pit methods, with Goliath also being mined by underground methods beneath the open pit.

The mine schedule provides 24.0 Mt of mill feed grading 1.47 g/t gold and 1.82 g/t silver over a 13.5-year mine life after one year of pre-stripping. The open pit mining sequence begins with Goliath in pre-production and then Goldlund starts in Year 1. Miller is started in Year 6 and finishes in Year 9. At that time, open pit mining is complete. The underground mine at Goliath starts in Year 3 with first delivery of mill feed in Year 4. Underground mining continues until Year 11. The processing facility will continue to be fed from stockpiles at Goliath until the middle of Year 14.

Mill feed from Goldlund and Miller are proposed to be transported to the Goliath process plant site with highway tractors and belly dump trucks. This transport will require the use of a portion of Highway 72, as well as an upgraded road across forestry lands to reduce traffic interaction and eliminate disturbance to the nearby communities.

The PEA has three pit areas (Goliath, Goldlund and Miller) with some having multiple phases. Goliath contains four phases with Phase 4 acting as the portal for the underground mine. Goldlund has six phases: two in the main pit area and four satellite pits. Miller is a single phase to be mined near the end of the project life. These provide a total of 21.0 Mt of open pit mill feed grading 1.16 g/t gold and 0.80 g/t silver. Waste movement from these phases amounts to 82.5 Mt, giving a strip ratio of 3.93:1 (waste: mill feed).

The pits are built on 10 m benches with safety berm placement each 20 m. Ramps are at a 10% gradient and have been designed for 91 tonne haulage trucks.

The PEA schedule calls for the development of the underground mine starting in Year 3. The underground mining area is an extension of the Goliath open pit. The depth of the open pit is planned to be approximately 100 m below surface. The underground area extend to around 640 m below surface and measures a total of approximately 3 km along strike. Approximately 11% of the underground material to be processed is derived from inferred resources. The dip of the deposit varies from around 70 to around 80 degrees, averaging 75 degrees.

An elevated cut-off net value of $110/t was applied to plan stopes which approximates to a gold cut-off grade of 2.0 g/t and was calculated to provide a minimum net revenue of $20/t from all mineralisation mined. Stope width typically varies from a minimum stope width of 1.8 m to around 11 m with some pinching and swelling, but averages around 6.2 m in width. In the deposit, ground conditions are considered to be fair to good and good in the footwall and hanging wall sequences. Cablebolt installation in stope hanging walls is planned to maintain stability and minimise waste dilution.

Longhole retreat stoping will be the primary underground mining method. Where production grade is estimated to be below 4.0 g/t Au, a permanent rib pillar is planned between adjacent stopes, resulting in approximately 15% in-situ losses, and uncemented rockfill will be used. Where production grade is estimated to be above 4.0 g/t Au, there are no planned pillars; cemented rockfill will be utilised to extract this higher grade material.

Life-of-mine underground feed to the process plant is estimated to be 2.97 Mt with a gold grade of 3.67 g/t Au and 9.05 g/t Ag resulting in an estimated revenue of $200/t net of operating costs. Planned steady-state production rate is 1,400 t/d. Initial mill feed release is planned in Year 4, the second year after the commencement of underground mine development, increasing to full production by Year 6. Total production life is planned to extend slightly over seven years.

1.14 Recovery Methods

The project flowsheet and unit operations have been selected based on preliminary testwork and financial evaluations. Unit operations used to build the plant flowsheet are standard technologies widely used in gold processing plants. The basis of the selected design is presented below in Table 1.3. A process flow diagram for the project is shown in Figure 1-2.

Table 1.3: Key Process Design Criteria

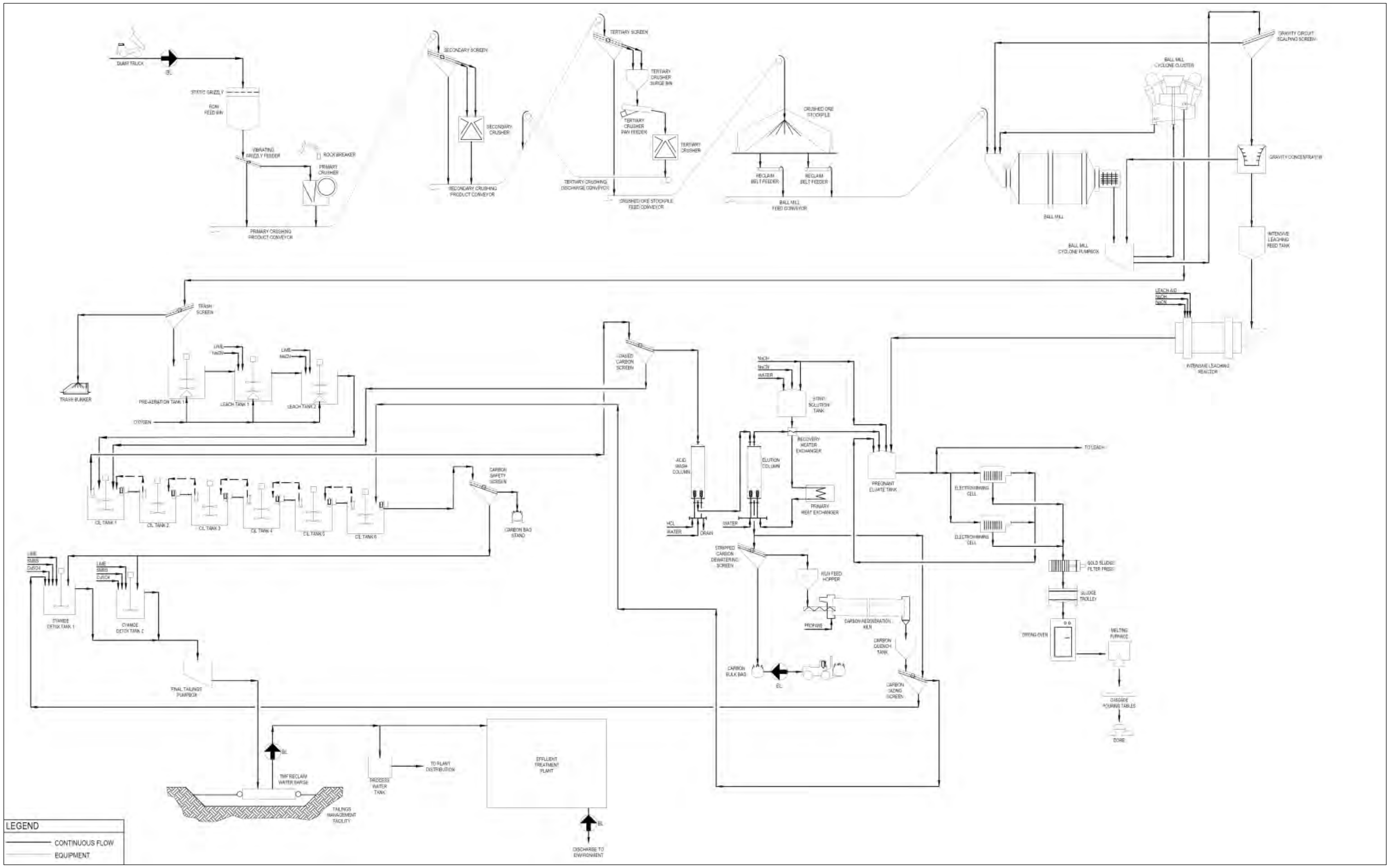
Criteria	Unit	Value
Annual Throughput (Design)	t/y	1,800,000
Daily Throughput (Design)	t/d	4,932
Operating Days per Year	d	365
Operating Availability – Crushing	h/y	5,869
Operating Availability – Grinding	h/y	8,059
Design Throughout – Crushing	t/h (dry)	311
Design Throughput – Milling	t/h (dry)	226
Crushing Feed Size, 100% Passing	mm	400
Crushing Product Size, 80% Passing	mm	8
Grinding Product Size, 80% Passing	µm	75
Ball Mill Circulating Load	%	350
Bond Ball Mill Work Index (Design)	kWh/t	15.7
ROM Head Grades Au (Average)	g/t	1.47
ROM Head Grades Ag (Average)	g/t	1.82
Recovery – Gravity Circuit	%	25.0
Recovery – CIL and Elution Circuit	%	68.6
Recovery – Overall	%	93.6
Average Annual Gold Production	oz/y	78,807

Source: Ausenco (2021).

The process plant includes the following:

- three-stage crushing of run-of-mine material
- covered crushed material stockpile to provide buffer capacity for the process plant
- ball mill with cyclone classification
- gravity recovery of ball mill discharge by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution
- trash screening
- pre-aeration, leach and carbon-in-leach adsorption
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré

Figure 1-2: Overall Process Flow Diagram



Source: Ausenco, (2021).

- carbon regeneration cyanide destruction of tailings using SO_2/air process
- carbon safety screening, and tailings disposal
- reagent storage and distribution
- water services (process water, treated water, fire water, gland water)
- potable water treatment and distribution
- air services

1.15 Project Infrastructure

Infrastructure to support the Goliath Gold Complex will consist of site civil work, buildings and facilities, water management systems, a tailings storage facility, and electrical power distribution. Mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems as required. The processing plant and tailings storage facility will be located at the Goliath property, along with most ancillary project infrastructure.

The Goliath and Goldlund-Miller properties may provide sufficient area to establish mine infrastructure (such as tailings and waste storage areas) and a processing plant site. More detailed engineering is required to confirm the suitability and sufficiency of the current property area for final mine and processing facilities, should they be constructed. The arrangement of the Goliath Gold Complex is illustrated in Figure 1-3.

1.15.1 Tailings Storage Facility & Water Management

Knight Piésold Ltd. completed a PEA-level design for the tailings storage facility at the Goliath Gold Complex. The TSF will provide secure storage for tailings and process water. The embankments include for adequate freeboard to provide ongoing tailings storage, operational water management, temporary environmental design storm storage and conveyance up to and including the inflow design flood. The TSF will be constructed as a single-cell facility northeast of the proposed process plant location. A geomembrane lining system will be installed along the TSF basin floor and on the upstream face of the perimeter embankments to minimise seepage. The embankments will be raised in stages to form a four-sided paddock-style impoundment using downstream construction methods throughout the mine life.

Tailings will be pumped from the process plant to the TSF as a conventional slurry via pipeline(s) and deposited into the TSF. Meteoric and supernatant inflows to the TSF basin will be temporarily stored prior to reclaim by a floating pump barge in the basin to the process plant. Excess water beyond the storage of the maximum water cover level will be transferred to the mine water pond. The TSF will be equipped with an overflow spillway in each embankment stage to accommodate flows above the environmental design storm and up to the inflow design flood.

Water management measures for the project will include a series of diversion berms, collection and diversion ditches, sediment basins, and water transfer pipelines to collect runoff originating within disturbed areas. The runoff will be conveyed to one of a number of catchment ponds, where the majority of the total suspended solids can settle out prior to sending the water to the mine water pond (for potential use in the mining process) or for treatment prior to releasing it to the environment.

Figure 1-3: Goliath Gold Complex Layout

1.16 Environmental Studies, Permitting & Social or Community Impact

The approach to environmental studies, permitting and approvals, and impact assessment for the Goliath Gold Complex will be to treat the Goliath, Goldlund and Miller deposits as three distinct projects. All three projects will be required to complete Regulatory Closure Plans as per the requirements of Ontario Regulation 240/00: Mine Development and Closure Under Part VII of the *Mining Act* in Ontario, prior to commencement of construction activities. Throughout the environmental baseline, permitting and approvals processes, Treasury Metals will endeavour to maximise participation with its Indigenous partners wherever possible and is committed to building and strengthening relationships, integrating traditional knowledge into decision-making frameworks, and actively communicating and sharing information in a transparent manner.

The schedule for the Goliath Gold Project is overall ahead of the schedule for the Goldlund and Miller Projects, given that a Federal Environmental Assessment (EA) has already been completed for this project. Specifically, on August 19, 2019, Treasury Metals received Federal government approval under the *Canadian Environmental Assessment Act*, 2012 (CEAA, 2012) for the Goliath Gold Project, with the Minister of Environment and Climate Change Canada concluding that the Project is not likely to cause significant adverse environmental effects. Potential benefits of the project are expected to include employment and business opportunities, as well as tax revenues at all levels of government. The Goldlund Project and Miller Project may require completion of one or more provincial environmental assessment processes pursuant to the *Ontario Environmental Assessment Act*, depending on the final project designs. Based on the current proposed design, neither the Goldlund Project nor the Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*.

The Goliath Gold Project as presented in this PEA is similar to the previous PEA, but differs in that the Goliath Gold Project processing facility is proposed to accept ore from other deposits (specifically deposits from the Goldlund and Miller properties). Pending regulatory guidance otherwise, it is not anticipated that the optimisation of the Goliath Project design would affect the current Federal EA approval of the Goliath Project, or that would trigger an Impact Assessment under the new *Impact Assessment Act* for a mining expansion. Therefore, while this engineering design change is not anticipated to have an effect on the current Federal EA approval on the Goliath Project, additional environmental data may need to be measured or modelled to support the change in the description of the assessed project. Additional environmental programs for the Goliath Project may also be required to update environmental baseline data relied on in the EA to support permitting efforts.

1.17 Capital Costs

The capital cost estimate was developed in Q4 2020 dollars based on Ausenco's in-house database of projects and studies and experience from similar operations. The estimate was developed to a level of accuracy of ±50% in accordance with the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes mining, processing, utilities, TSF and project site infrastructure.

The capital cost summary is presented in Table 21.1. The total initial capital cost for the Goliath Gold Complex is $232.6 million and LOM sustaining costs are $289.6 million. Closure costs are additional and are estimated at $28.5 million.

Table 1.4: Capital Cost Summary

WBS	WBS Description	Initial Capital (C$M)	Sustaining (C$M)	Total Capital (C$M)
1000	Mining (Goldlund and Miller)[1]	44.6	194.3	238.9
2000	Mining (Goliath)[1]			
3000	Process Plant	64.9	1.4	66.3
4000	On-site Infrastructure	49.9	70.9	120.8
5000	Off-site Infrastructure	0.6	-	0.6
	Directs	**160.0**	**266.6**	**426.6**
6000	Project Indirects	9.6	-	9.6
7000	Project Delivery	26.1	-	26.1
8000	Owner's Cost	7.1	-	6.8
9000	Provisions (Contingency)	29.8	22.9	52.7
	Total Project Cost	**232.6**	**289.6**	**522.2**

Notes: [1]Mining costs have been calculated considering shared capital expenditures among projects. Source: Ausenco (2021).

1.17.1 Mining

Open pit mining capital includes costs associated with open pit mining and haulage of mill feed from Goldlund, Miller and Goliath. The mining equipment fleet is leased, so the capital cost for equipment reflects the cost of initial down payments.

Pre-production mining occurs at Goliath first. This includes the movement of 5.7 Mt of waste and placement of 0.8 Mt of mill feed in stockpiles adjacent to the primary crusher. The mine operating costs associated with this period are expected to cost $25.2 million.

Equipment prices used current quotations from local vendors. A 20% down payment is included in the capital cost for those units leased. The remaining cost was included in operating costs.

Unique to this mine operation is a mill feed haulage fleet. This is a smaller loader (7.8 m^3) responsible for loading a fleet of highway trucks with belly dump trailers. They would transfer the mill feed from Goldlund and Miller to the Goliath plant and stockpiles. Their cost is included in the mine capital.

Underground mining capital includes those costs associated with the development of the underground at Goliath. The underground mining equipment fleet is also leased, so the capital for equipment reflects the cost of the initial down payments. The financing portion of the cost is included in the operating cost estimate. As underground develop starts in Year 3, the capital is considered under sustaining capital.

1.17.2 Process & Infrastructure

Mechanical equipment and building supply costs were based on recent and historical budget quotes from similar projects. Other material and equipment costs were developed by applying factors to the total direct cost of the mechanical equipment. The factors were based on Ausenco's historical data for similar type work and are both discipline and area specific.

Bulk earthworks for the plant site, mine ancillary buildings, tailings storage facility and water management infrastructure were developed based on semi-detailed cut-and-fill volumes based on site layout and site topographical information. Unit rates were benchmarked against recent projects within the region.

1.18 Operating Costs

The operating cost estimate was developed in Q4 2020 dollars based on Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50%. The overall life-of-mine operating cost is $975 over 13.5 years, or $40.7/t of ore milled, as summarised in Table 1.5.

Table 1.5: Operating Cost Estimate Summary

Operating Cost	Unit Cost (C$/t Processed)	Total Cost (C$M)
Mining - Open Pit	17.0	356.0
Mining - Underground	70.3	208.5
Off-site Mill Feed Haulage	5.6	83.6
Processing	11.4	272.5
Site G&A	2.3	54.7
TOTAL	**40.7**	**975.3**

Source: Ausenco (2021).

1.19 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. Cash flows have been discounted to the start of construction (January 1, 2023), assuming that the project execution decision will be made and major project financing will be carried out at this time.

The pre-tax net present value discounted at 5% (NPV5%) is C$477 million, the internal rate of return IRR is 37.3%, and payback is 1.9 years. On an after-tax basis, the NPV5% is C$328 million, the IRR is 30.2%, and the payback period is 2.2 years.

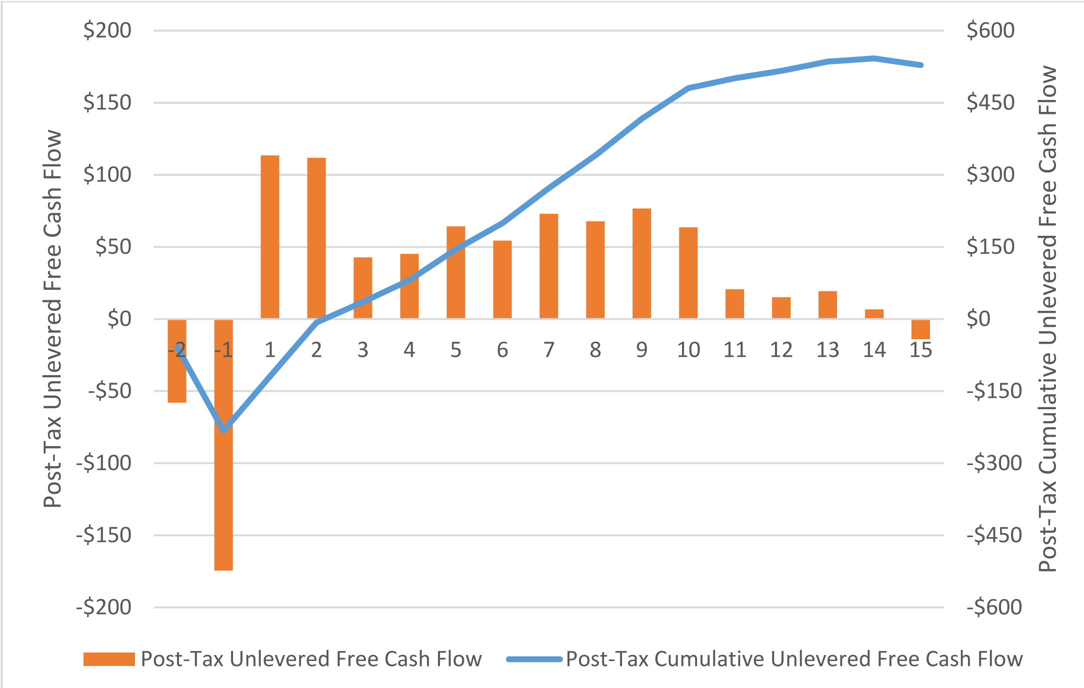
A summary of project economics is listed in Table 1.6 and shown graphically on Figure 1-1.

Table 1.6: Summary, Project LOM Cash Flow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,600
Exchange Rate (USD:CAD)	0.75
Mine Life (years)	13.5
Total Waste Tonnes Mined (kt)	82,452
Total Mill Feed Tonnes (kt)	23,966
Strip Ratio (waste:mineralisation)	3.93
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	1.47
Mill Recovery Rate (%)	93.6%
Total Mill Ounces Recovered (koz)	1,064
Average Annual Production (koz)	79
Operating Costs	**LOM Total / Avg.**
Mining Cost – Open Pit (C$/t Mined)	$3.27
Mining Cost – Open Pit (C$/t Milled)	$16.95
Mining Cost – Underground (C$/t Milled)	$70.31
Processing Cost (C$/t Milled)	$11.37
G&A Cost (C$/t Milled)	$2.28
Gold Refining (C$/oz Au)	$14.00
Silver Refining (C$/oz Ag)	$0.26
Total Operating Costs (C$/t Milled)	$40.70
Cash Costs* (US$/oz Au)	$699
All-in Sustaining Cost (AISC)** (US$/oz Au)	$911
Capital Costs	**LOM Total / Avg.**
Initial Capital (C$M)	$233
Sustaining Capital (C$M)	$290
Closure Costs (C$M)	$24
Salvage Costs (C$M)	$12
Financials - Pre Tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$477
IRR (%)	37.3%
Payback (years)	1.9
Financials - Post Tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$328
IRR (%)	30.2%
Payback (years)	2.2

Notes: *Cash costs consist of mining costs, processing costs, mine-level general & administrative expenses and refining charges and royalties. **AISC includes cash costs plus sustaining capital, closure cost and salvage value.
Source: Ausenco (2021).

Figure 1-4: Projected LOM Cash Flow



Source: Ausenco (2021).

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV, IRR and payback period of the project using the following variables: gold price, foreign exchange rate, discount rate, mill recovery, initial capital costs, and operating costs.

Table 1.7 summarises the post-tax sensitivity analysis results.

Table 1.7: Post-Tax Sensitivity Summary

Gold Price	Post-Tax NPV (5%)	Initial CAPEX		Total OPEX		FX	
US$/oz	Base Case	(-25%)	(+25%)	(-25%)	(+25%)	(-25%)	(+25%)
$1,200	$47	$101	($8)	$170	($93)	$331	($163)
$1,400	$189	$244	$134	$308	$66	$513	($15)
$1,600	$328	$383	$273	$445	$208	$694	$102
$1,850	$498	$553	$443	$615	$381	$921	$243
$2,000	$600	$655	$545	$717	$484	$1,057	$326
Gold Price	**Post-Tax IRR**	**Initial CAPEX**		**Total OPEX**		**FX**	
US$/oz	**Base Case**	**(-25%)**	**(+25%)**	**(-25%)**	**(+25%)**	**(-25%)**	**(+25%)**
$1,200	9.3%	16.9%	4.4%	19.0%	0.0%	30.4%	0.0%
$1,400	20.7%	31.0%	14.3%	28.5%	11.3%	41.5%	3.5%
$1,600	30.2%	42.7%	22.4%	37.1%	22.5%	51.4%	14.1%
$1,850	40.7%	55.6%	31.3%	46.8%	34.0%	62.7%	24.6%
$2,000	46.4%	62.6%	36.2%	52.2%	40.2%	69.2%	30.1%

1.20 Risks & Opportunities

1.20.1 Risks

1.20.1.1 Geology

- The modelling approach at Goliath assumes that the contacts between the high-grade mineralisation and the surrounding low-grade material are not sharp and visually difficult to recognise without assays. This assumption was based on drill core logging and information provided by Teck-Corona as part of their bulk sampling program completed in 1997. If the contacts are sharper and more easily identifiable than expected during mining, the deposit could return a higher grade with a corresponding lower tonnage. This risk can be mitigated in various ways. Near surface, an area within the payback period of the open pit could be selected for testing the proposed grade control program. The program can be used to de-risk the resources and increase confidence in the grade intended for the proposed mill. At depth, targeted infill drilling can provide a greater level of confidence in the estimated grade and increase confidence in the modelling approach.
- At Goliath, the silver grade presents a small risk due to the lack of assays. This risk can be mitigated by re-assaying the drill core pulps for silver.
- Drilling in the eastern portion of the deposit and around the fold nose could increase the resources.
- At Goldlund, the current geological model considers broad mineralised zones that define the trend of the mineralisation. The development of a new geological model of lithology and alteration and a new model of the high-grade mineralisation may result in a change to the mineral resources. Infill drilling is required to confirm the continuity of the high-grade mineralisation.

1.20.1.2 Mining

- Wall slopes may flatten, resulting in more waste material. This can be mitigated with additional geotechnical drilling, particularly at Goldlund and Miller where more work is required.
- Waste storage foundation study at Goldlund and Miller may require lower and large footprints or additional preparation costs. Geotechnical site investigations should help mitigate this through better understanding.
- ABA testing may indicate some of the waste material in Goldlund and Miller is potentially acid-generating and that separate storage facilities may be required to control drainage. Additional testwork will help to develop a better understanding of this issue and determine its impact on project design.

1.20.1.3 Recovery Methods & Metallurgical Testing

- No metallurgical testing has been completed on the Miller deposit. Based on geology it is assumed to be similar to the Goldlund deposit.

1.20.1.4 Tailings Storage Facility

- Non-PAG waste rock produced from the Goliath pit (up to 7% of waste rock) cannot be segregated during mining as assumed in the study and will not be available as required during construction of the TSF.
- The source of an adequate amount of suitable bulk embankment fill cannot be identified and secured from locally available borrow sources.
- There is the potential for challenging construction conditions associated with dewatering during preparation of the foundations for embankment construction and lining of the basin.
- The ability to achieve flat uniform filling of tailings via sub-aqueous deposition within the basin while maintaining the minimum required water cover over the tailings as assumed in the study.

1.20.2 Opportunities

1.20.2.1 Geology

- Drilling in the eastern portion of the Goliath deposit and around the fold nose could increase the resources.
- The additional drilling recommended for Goldlund in Zones 1, 2, 3, 4, 6, 8 and 9 could convert a portion of the inferred mineralisation to indicated mineralisation, as well as to expand the Zone 1 mineralisation to the northeast.
- Assaying of available Goldlund drill core sample rejects for silver, along with additional drilling, may generate sufficient data to allow the estimation of silver as a by product in future mineral resource estimates.

1.20.2.2 Mining

- With testing, the PAG material may represent a smaller volume of material, which may help in storage considerations at Goliath in addition to Goldlund and Miller
- The use of sorting technology may help reduce mill feed trucking tonnage, which in turn may elevate the feed grade.

1.20.2.3 Recovery Methods & Metallurgical Testing

- Optimising processing conditions related to fineness of grind and leach retention time may result in lower capital costs from employing a coarser grind and reduced retention time.
- Additional metallurgical testing will provide an opportunity to optimise reagent addition rates and grinding media wear rates.
- Further investigate the incidence of tellurides within Goldlund and Miller mill feed to optimise mill recovery factors.

1.20.2.4 Infrastructure

- Site conditions at Goldlund may be more favourable than at Goliath for siting the tailings storage facility, including closer access to large quantities of NAG waste rock for construction.

- Additional geotechnical drilling would better define the foundation conditions at the TSF and potentially reduce earthworks quantities for construction of the embankments and buttressing.

1.21 Conclusions

The total measured and indicated resources for the Goliath, Goldlund and Miller projects are estimated at 55.4 Mt at a grade of 1.10 g/t Au for an estimated 2.0 Moz of contained gold. Additional inferred resources are estimated to be 21.0 Mt at a grade of 0.78 g/t Au for a total of 0.5 Moz.

Based on the assumptions and parameters presented in this report, the PEA shows positive economics (i.e., C$328 million post-tax NPV (5%) and 30.2% post-tax IRR). The PEA supports that additional detailed studies are warranted.

1.22 Recommendations

The financial analysis of this PEA demonstrates that the Goliath Gold Complex has positive economics. It is recommended to continue developing the project through additional studies, including a pre-feasibility study. Table 1.8 summarises the proposed budget to advance the project through the pre-feasibility study stage.

Table 1.8: Proposed Budget Summary

Description	Cost $C
Geology – Goliath Work Program	5,925,000
Geology – Goldlund Work Program	8,760,000
Geology – Miller Work Program	1,830,000
Geotechnical	998,000
Mining	50,000
Metallurgy	500,000
Infrastructure	555,000
Environmental	2,100,000
PFS Study Budget	1,695,000
Total Recommended Study Budget	**$22,413,000**

1.22.1 Geology

1.22.1.1 Goliath

After reviewing the Treasury Metal data, AGP makes the recommendations outlined below for Goliath.

Goliath QA/QC

- AGP recommends that the QA/QC for silver be charted similarly to gold.
- Treasury Metals quarter core sample duplicate shows evidence of a rather strong nugget effect and AGP questions if this protocol should continue. AGP advised Treasury Metals to seek the opinion of a specialist in the QC/QA field.

Resource Modelling

- The missing silver assays represent limited risk to the resources; however, AGP recommends all recoverable drill rejects or pulps for the samples located in the mineralised horizon be assayed for silver. An estimated 6,000 pulps @ $10.00 per pulps for a total of $60,000.
- AGP also recommends that in future drilling programs, Treasury Metals should ensure that no gold assay within the mineralised horizons is missing a corresponding silver assay.
- Advance geostatistical studies (change of support and conditional simulation) should be conducted as part of future pre-feasibility or feasibility studies. These studies allow the quantification of risks to the resource. The cost for these studies is estimated at $10,000.

Drilling Recommendations

AGP recommends continuing exploration and delineation drilling at the Goliath deposit. This additional drilling should be designed to expand and improve the quality of mineral resources presented in this report and to further the understanding of the geology, specifically in the area east of the deposit where mining infrastructure may potentially be built. Drilling should also focus on infill drilling of the underground resources from surface where the potential open pit may restrict access in the future. Finally, drilling should focus on the sections of the underground mining areas that have seen reduced continuity in the current resource model when compared to previous models. If gold assays are found in these areas, there is potential to connect the high-grade wireframe and subsequently create additional areas for the proposed mining zone.

- Area “A” is designed to expand on existing resources and convert inferred blocks to indicated east of shoot 1
- Area “B” is strictly designed to convert inferred blocks to indicated in the west of shoot 2 and at depth.
- Area “C” is designed for resource expansion. This area is located at depth adjacent to the currently defined resource blocks between shoots 2 and 3.
- Area “D” is to convert the resource in the upper portion of the PEA pit from inferred to indicated. The area spans from section 526500E to 527500E.
- Area “E” is designed to explore the ground currently located under the proposed infrastructure. The area is located between sections 529750E and 529875E.
- Area “F” is designed to test a number of regional targets and follow up on several historical results that could contribute to future growth of additional “satellite pits” along strike towards the eastern boundary of the Goliath property.

AGP recommends a total of 82 drill holes totalling 36,575m of drilling for a total estimated cost of $5,925,000.

1.22.1.2 Goldlund

The following recommendations are for the Goldlund portion of the project:

- Close-spaced drilling of 6,400 m in 32 holes should be carried out in Zone 1. The drilling should target areas inside the mineral resources shell using angled core holes to confirm the grade continuity and upgrade a portion of the mineral resources for that part of Zone 1 from indicated to measured. The target area should represent the area that is likely to be mined at the start of the project.

- Infill drilling of 29,000 m should be carried out in selected areas of Zones 1, 2, 3, 4, 6, and 8 and 9 to achieve a drill hole spacing of approximately 25 m x 25 m to upgrade the inferred mineralisation to indicated and to explore for additional inferred resources. Priority should be given to areas that have inferred mineralisation inside the mineral resources shell and within or directly adjacent to proposed mining pit shells.
- Additional drilling of 7,200 m should be carried out to further explore selected areas of Zone 1 and Zone 4 and increase the confidence in the location of the mineralised zones.
- A 3D geological model of the lithology and alteration should be developed using implicit modelling software such as Leapfrog GEO® to aid in the interpretation of the granodiorite sills that host the stockwork mineralisation and the faults or other structures that might off-set the mineralised zones. These models would then be used to support a revised interpretation of the mineralised zones for the estimation of mineral resources. This modelling effort will require additional database and geological studies.
- Consideration should be given to the development of an alternative model of the gold mineralisation using a high-grade wireframe. This wireframe should be generated using a suitable gold grade threshold, such as 1.0 or 2.5 g/t Au, and implicit modelling software, such as Leapfrog GEO®. This grade-shell would then be used as an additional control to restrict the higher grades and prevent any potential smearing of the high-grade assays during block grade interpolation. This would improve the reliability of the mineral resource estimate.
- The mineral resources estimate should be updated considering the additional drilling and geological modelling of the lithology, alteration, and high-grade mineralised zone wireframes.
- Assaying of available Goldlund drill core sample rejects for silver, along with additional drilling, may generate sufficient data to allow the estimation of silver as a byproduct in future mineral resource estimates.

The estimated budget for the proposed drilling and modelling programs is approximately C$8.7 million.

1.22.1.3 Miller

AGP recommends the following exploration programs for the Miller Project. Pending positive results, further studies may be proposed.

- A review of selected completed drill holes by optical televiewer should be carried out to accurately determine vein orientations and vein sets for a better understanding of geological and structural controls of the gold mineralisation for the deposit. Optimally, this should be carried out on a variety of drill holes, that is, on angled drill holes (drilled from the northeast and southwest) and vertical drill holes.
- Infill drilling should be carried out by angled drill holes from the northwest and the southeast to reduce the current drill spacing to less than 50 m x 50 m. Drill holes should target the deposit near surface and at depth. Approximately 6,000 m of drilling is recommended. The drilling should be completed using oriented drill core if a televiewer is not employed to collect information of the vein orientations.

- Delineation drilling along strike of the known gold mineralisation to determine the extent of the deposit. Approximately 2,500 m of drilling is recommended.
- Where and if possible, stripping (trenching) and surface channel sampling across the deposit to gather geological and structural data at the surface of the deposit. An initial program of three lines of channel samples are recommended.
- Update of mineral resources based on the results of additional drilling and the geological information collected.

The estimated budget for the proposed drilling and modelling programs is approximately C$1.8 million.

1.22.2 Geotechnical

Further geotechnical and hydrogeological work are required at Goliath and new studies need to be initiated at Goldlund and Miller. The recommended work will:

- update the slope design parameters considering the current PEA design
- develop area hydrogeological models for surface and underground mining development (Goliath only) to interface with the overall project site-wide water balances
- review the underground design with focus on underground infrastructure, and required stope support (bolting)
- analyse waste and stockpile foundations with revised slope design parameters

The estimated budget for the proposed PFS geotechnical program is $998,000.

1.22.3 Mining

The following work is recommended to advance the project to a pre-feasibility study level:

1. detailed quotations on mine equipment and refined equipment selection
2. detailed mine planning on Goliath pit backfill sequence to determine if additional material could be backfilled
3. further examination of mill feed transportation options with the objective of reducing transportation cost
4. review and design of pit and underground dewatering requirements and interface with surface water management system
5. detailed design and costing of permanent water exclusion bulkheads beneath the temporary central open pit access
6. incorporation of updated geotechnical guidance on stope cablebolt designs, as the rock is currently classified as fair to good which requires this level of support
7. solicitation of contractor quotes for both open pit and underground mining to examine potential project NPV enhancements
8. update of pit slopes in all three areas based on revised geotechnical parameters resulting from additional geotechnical testwork

9. detailed design of underground infrastructure, both on surface (portals, ventilation, power interface) and underground (dewatering system, electrical, etc.)
10. complete a labour survey for salaries, benefits, and skilled worker locally available (this information would be used in pre-feasibility study costing; it may also lead to Treasury Metals assisting local colleges and workers to develop specific skill sets in anticipation of a production decision)

All of the above recommendations would be included in the normal pre-feasibility study cost estimate, with the exception of point 10. This would normally involve an outside consultant and would be expected to cost $50,000.

1.22.4 Metallurgy

The estimated budget for the recommended metallurgical testwork totals $500,000.

To progress to a pre-feasibility study level the following metallurgical testwork is recommended for the Goliath Project:

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- mineralogical studies including gold deportment analysis
- additional ore competency tests for more accurate SAG mill sizing; JK Tech SMC tests (Axb) are recommended to be conducted over a range of lithologies or zones
- ore hardness tests including Bond rod, ball and abrasion index testing to determine the variability of the lithologies or zone
- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time and cyanide addition rate
- additional cyanide destruction testing to optimise reagent addition and retention time

To progress to a pre-feasibility study level the following metallurgical testwork is recommended for the Goldlund Project:

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- addition ore competency tests for more accurate SAG mill sizing; JK Tech SMC tests (Axb) are recommended to be conducted over a range of lithologies or zones
- ore hardness tests including Bond rod, ball and abrasion index testing to determine the variability of the of lithologies or zones
- mineralogical studies including gold deportment analysis
- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time, reagent conditions (pH and cyanide concentration) for gold tellurides
- cyanide destruction testing to establish required reagent addition rates and retention time for required discharge cyanide concentrations

No previous testing has been conducted on Miller samples. The following metallurgical testwork is recommended.

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- conduct testing to identify comminution parameters including SMC tests (Axb), Bond rod, ball and abrasion index testing
- mineralogical studies including gold deportment analysis
- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time, reagent conditions (pH and cyanide concentration) for gold tellurides (if present)
- cyanide destruction testing to establish required reagent addition rates and retention time for required discharge cyanide concentrations

1.23 Sorting

Sighter-type ore sorting amenability testing is recommended. The program will establish if samples from the three deposits are amenable to particle or bulk sorting. Ore sorting could benefit the project by either upgrading mill feed with reduced quantity transported for processing or upgrading of low-grade material near the planned cut-off grades.

1.24 Infrastructure

The following activities are recommended to support infrastructure design for the pre-feasibility study phase:

1.24.1 Site Investigations

- Additional site investigations should be completed to identify suitable borrow locations, and further characterise foundations of the TSF embankments and basin.
- Cone penetration testing should be carried out in key areas to confirm strengths of the softer fine grained soils within TSF Embankment footprint and other key infrastructure, (i.e., the grey silt).
- The availability of local borrow sources for TSF embankment construction should continue to be evaluated to verify the capital cost associated with its construction based on the material available.
- The recommended budget for these items is $375,000.

1.24.2 Tailings Storage Facility

- Additional stability analyses should be carried out to refine and optimise buttress sizing requirements and embankment section (note: the analysis should take into account the potential for soil liquefaction, cyclic clay softening, and undrained strength conditions based on the updated site investigations).
- Additional seepage analyses should be performed to refine and optimise basin lining requirements and closure cover thickness.

- Potential basin lining alternatives, including geosynthetic materials (HDPE, LLDPE) and paper pulp sludge, should be evaluated.
- The recommended budget for these items is $140,000.

1.24.3 Water Management Measures

- The catchment areas contributing runoff to the process plant, open pits and waste dumps, and the amount of groundwater inflow to the open pits and underground mine with time need to be confirmed based on the ultimate mine plan.
- Site-specific meteorological and hydrology data should be collected. This data will be used to refine seasonal runoff values and design storms for future work.
- The predictive water quality model should be updated to review the requirements for water treatment and/or discharge.
- Bench-scale settling testing should be performed to characterise the required retention time for suspended solids in the runoff water.
- The recommended budget for these items is $40,000.

1.24.4 Facilities Location

The PEA was advanced with the concept of locating the process and tailings facility at the Goliath project site. This is due to the advanced nature of both the permitting and development path of the Goliath Project and previous technical studies. By adding the Goldlund and Miller properties to the overall project scope, opportunities exist that may benefit the project from a cost and environmental perspective.

Mill feed material needs to move between the various pit areas, which implies that a plant located at Goldlund would not adversely impact the operating costs of the project. The advantages of locating the plant and tailings at Goldlund should be examined and included in a detailed trade-off study that considers potential permitting delays that may accompany such changes.

It is recommended that a series of trade-off studies examining alternate locations for the plant and tailings facility be considered and included in the pre-feasibility study budget.

1.25 Environmental

The approach to environmental studies for the Goliath Gold Complex will be to treat the Goliath, Goldlund and Miller deposits as three distinct projects; therefore, each project will have a distinct set of environmental recommendations as indicated below.

Treasury Metals has an advanced understanding of the environmental baseline at the Goliath Project site having previously completed an extensive baseline investigation to support the Federal environmental assessment process for the project. Treasury Metals received Federal government approval for the Goliath Project in August 2019 under the *Canadian Environmental Assessment Act*, with the Minister of Environment and Climate Change Canada concluding that the project is not likely to cause significant adverse environmental effects. As part of the conditions on the approval of the project, Treasury Metals is obligated to notify the Federal and Provincial authorities, as well as it Indigenous partners, of any project changes, including the milling of ore from the Goldlund Project and Miller project at the Goliath property. While the engineering design change to mill ore from other sites at Goliath is not anticipated to have an effect on the current Federal EA approval on the Goliath Project, additional environmental data may need to be measured or modelled to support the change in the description of the assessed project. Additional environmental programs for the Goliath Project may also be required to update environmental baseline data relied on in the EA to support permitting efforts.

Baseline data collection for the Goldlund Project is underway and is expected to be completed within 12 months' time. Treasury Metals has not collected any baseline data from the Miller project to date; however, it is anticipated this will happen in the immediate future. Based on the current proposed design, neither the Goldlund Project nor Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*. However, baseline data for these projects will be required to support Provincial permitting and approvals processes, including potential Provincial EAs.

The cost for the above work for all three projects is estimated at $2.1 million. This is considered sufficient for a pre-feasibility level of study.

2 INTRODUCTION

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Treasury Metals Inc. (Treasury Metals) to summarise the results of a preliminary economic assessment (PEA) of the Goliath Gold Complex. The report was prepared in compliance with the Canadian disclosure requirements of National Instrument 43-101 (N.I. 43-101) and in accordance with the requirements of Form 43-101 F1.

The PEA was prepared in accordance with "N.I. 43-101 Standards of Disclosure for Mineral Projects". Readers are cautioned that the PEA report is preliminary in nature.

The N.I. 43-101 responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Treasury Metals to manage and coordinate the work related to the NI 43-101. Ausenco also developed the PEA-level design and cost estimate for the process plant and general site infrastructure.
- AGP Mining Consultants (AGP) was commissioned to complete the mineral resource estimate for the Goliath and Miller projects, and to design the open pit and underground mine plan, mine production schedule, and mine capital and operating costs.
- CGK Consulting Services (CGK) was commissioned to complete the mineral resource estimate for the Goldlund project.
- Knight-Piésold (KP) was commissioned to develop the PEA-level design and cost estimate for the tailings storage facility and site water management infrastructure.

2.1 Terms of Reference

The report supports disclosures by Treasury Metals in a news release dated February 2, 2021 entitled, "Treasury Metals Announces Positive Preliminary Economic Assessment for Goliath Gold Complex". Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (2019).

The Goliath Gold Complex area contains three deposits: Goliath, Goldlund and Miller. Treasury Metals owns 100% of the Goliath Gold Complex.

2.2 Qualified Persons

This report was prepared by the following Qualified Persons:

- Mr. Tommaso Roberto Raponi, P. Eng., Principal Metallurgist, Ausenco
- Gordon Zurowski, P.Eng., Principal Mining Engineer, AGP Mining Consultants
- Pierre Desautels, P.Geo., Principal Resource Geologist, AGP Mining Consultants
- Paul Daigle, P.Geo., Senior Resource Geologist, AGP Mining Consultants
- Chris Keech, P.Geo., Principal Geologist, CGK Consulting Services
- Reagan McIsaac, Ph.D., P.Eng., Senior Engineer, Knight Piésold
- Mackenzie Denyes, Ph.D., P.Geo., Manager of Regulatory Affairs, Treasury Metals

Information about each contributor, and the report sections for which they are responsible, is provided in Table 2.1.

Table 2.1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Treasury Metals	Date of Last Site Visit	Report Sections
Tommaso Roberto Raponi	P.Eng. (ON)	Principal Metallurgist	Ausenco Engineering Canada Inc.	Yes	No site visit	1.1, 1.2, 1.11, 1.14, 1.15 except 1.15.1, 1.17 except 1.17.1, 1.18, 1.19, 1.20.1.3, 1.20.2.3, 1.21, 1.22 except 1.22.1 to 1.22.3, 1.23, 1.24.4, 2 except 2.3.1 and 2.3.2, 13, 17, 18 except 18.1.2, 18.3.1, 18.7 and 18.7.4, 21 except 21.1.2.1 and 21.1.2.2.2 and 21.1.4 and 21.2.1, 22, 24, 25.3, 25.4, 25.7, 25.8, 25.9, 25.10.1.3, 25.11, 26.1, 26.5, 26.6, 26.8
Gordon Zurowski	P.Eng. (ON)	Principal Mining Engineer	AGP Mining Consultants Inc.	Yes	October 4, 2020	1.13, 1.17.1, 1.20.1.2, 1.20.2.2, 1.22.2, 1.22.3, 2.3.2, 15, 16, 18.3.1, 18.7.4, 21.1.2.1, 21.2.1, 25.2, 25.10.1.2, 26.3, 26.4
Pierre Desautels	P.Geo. (ON)	Principal Resource Geologist	AGP Mining Consultants Inc.	Yes	September 11, 2020	1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.12, 1.20.1.1, 1.20.2.1, 1.22.1, 2.3.1, 3, 6.1, 7.1, 7.2, 8.1, 9.1, 9.2, 10.1, 10.2, 11.1, 12.1, 14.1, 14.4, 25.1, 25.1.1, 25.10.1.1, 26.2.1
Paul Daigle	P. Geo. (ON)	Senior Resource Geologist	AGP Mining Consultants Inc.	Yes	October 13, 2020	1.3, 1.22.1.2, 4, 6.3, 7.4, 8.3, 9.4, 10.4, 11.3, 12.3, 14.3, 25.1.3, 26.2.3, and 27
Chris Keech	P.Geo. (BC)	Principal Geologist	CGK Consulting Services Inc.	Yes	October 6, 2020	1.22.1.1, 6.2, 7.3, 8.2, 9.3, 10.3, 11.2, 12.2, 14.2, 25.1.2, and 26.2.2
Reagan McIsaac	P.Eng. (ON)	Senior Engineer	Knight Piésold Ltd.	Yes	No site visit	1.15.1, 1.20.1.4, 1.20.2.4, 1.24 except 1.24.4, 18.1.2, 18.7, 21.1.2.2.2, 25.5, 25.10.1.4, 26.7
Mackenzie Denyes	P.Geo. (ON)	Manager of Regulatory Affairs	Treasury Metals Inc.	No	September 16, 2020	1.4, 1.16, 1.25, 5, 19, 20 except 20.7, 23, 25.6, 26.9
Sheila Ellen Daniel	P.Geo. (ON)	Principal Geoscientist and Discipline Lead Mining Environmental Management and Approvals	Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited	Yes	No site visit	20.7, 21.1.4

2.3 Site Visits & Scope of Professional Inspection

A summary of the site inspections by the Qualified Persons is provided in Table 2.2.

Table 2.2: Qualified Person Site Inspections

Qualified Person	Dates of Site Visit	Days on Site	Project
Gordon Zurowski, P.Eng.	Oct. 4-5, 2020	2	Goliath, Goldlund
Pierre Desautels, P. Geo.	Sep. 11-12, 2020	2	Goliath
Chris Keech, P.Geo.	Oct. 6-8, 2020	2	Goldlund
Paul Daigle, P.Geo.	Oct. 13-15, 2020	2	Goldlund, Miller
Mackenzie Denyes, P.Geo.	Sep. 16-20, 2020	4	Goliath, Goldlund

2.3.1 Geology

Mr. Desautels visited the Goliath property from September 11 to 12, 2020 to review the property geology, exploration program, drill hole collar locations, drilling program, core handling and sample protocols, and diamond drill core. Mr. Desautels was accompanied by Mr. Adam Larsen, Exploration Manager for Treasury Metals.

Mr. Keech visited the Goldlund project site from October 6 to 8 2020 to inspect the surface geology (including the historical open pit and trenched areas), core logging, sampling and core storage facilities, selected drill hole collar locations, and the core logging of selected drill core. Mr. Keech was accompanied by Treasury Metals' Mr. Adam Larsen, Exploration Manager, and Mr. Bryan Wolfe, Senior Exploration Geologist.

Mr. Daigle visited the Goldlund-Miller property from October 13 to 15, 2020. Mr. Daigle inspected drill core logging, sampling, and storage facilities at the Goldlund exploration camp. At the Miller project site, Mr. Daigle verified drill hole collar locations and reviewed drill logs against selected drill core. Mr. Daigle was accompanied on the site visit by Mr. Adam Larsen, Exploration Manager, for Treasury Metals.

2.3.2 Mining

Mr. Zurowski conducted a site visit to the Goliath Gold Complex from October 4 to 5, 2020. The Goliath Project site was inspected for two days during the site visit. Mr. Zurowski was accompanied on the site visit by Mr. Adam Larsen, Exploration Manager for Treasury Metals. While on site, Mr. Zurowski reviewed the Goliath pit area, the Goldlund pit area, proposed mill feed haulage route, core from both Goliath and Goldlund, the existing Goldlund open pit and infrastructure in the area of the project.

2.4 Effective Dates

This technical report has a number of effective dates as follows:

- Goliath mineral resource estimate: December 16, 2020
- Goldlund mineral resource estimate: October 23, 2020
- Miller mineral resource estimate: October 26, 2020
- Financial analysis: January 28, 2021

The overall effective date of this report is based on the effective date of the financial analysis, which is January 28, 2021.

2.5 Information Sources & References

This report is based on internal company reports, maps, published government reports, and public information, as listed in Section 27. It is also based on the information cited in Section 3.

2.6 Previous Technical Reports

The Goliath Gold Complex has been the subject of several previous technical reports, as summarised in Table 2.3 on the following page.

2.7 Units & Abbreviations

All measurement units used in this Report are SI units unless otherwise noted. Currency is expressed in Canadian (C) dollars (C$). A list of abbreviations is provided in Table 2.4.

Table 2.3: Summary of Previous Technical Reports

Reference	Company	Name
Goliath Project		
P&E, 2020	Treasury Metals	Amended Updated Mineral Resource Estimate for the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (August 2020)
CSA, 2017	Treasury Metals	Preliminary Economic Assessment Update on the Goliath Gold Project, Kenora Mining Division, Ontario (April 2017)
P&E, 2015	Treasury Metals	Technical Report and Updated Resource Estimate for the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (October 2015)
Roy et al., 2012	Treasury Metals	Preliminary Economic Analyses of the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (July 2012)
Roy et al., 2011	Treasury Metals	Technical Report and Mineral Resource Update on the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (November 2011)
Roy, 2010	Treasury Metals	Updated Mineral Resource Estimate for the Goliath Deposit (2010)
Roy et al., 2008	Treasury Metals	Report on the Goliath Project, Kenora Mining Division, Northwestern Ontario, Canada (December 2008)
Wetherup, 2008	Treasury Metals	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (April 2008)
Wetherup et al., 2008	Treasury Metals	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (February 2008)
Wetherup et al., 2007	Laramide Resources	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (November 2007)
Goldlund & Miller Projects		
WSP, 2020	First Mining	Technical Report Re-issue; Goldlund Gold Project, Sioux Lookout, Ontario (July 2020)
WSP, 2019	First Mining	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (April 2019)
WSP, 2017	Tamaka	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (February 2017)
WSP, 2015	Tamaka	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (March 2015)
Tetra Tech, 2014	Tamaka	Technical Report and Resource Estimate Update on the Goldlund Deposit, Goldlund Project. Sioux Lookout, Ontario (January 2014)
Tetra Tech, 2013	Tamaka	Technical Report and Resource Estimate on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (January 2013)

Reference	Company	Name
Tetra Tech, 2012	Tamaka	Technical Report and Resource Estimate for the Goldlund Gold Deposit, Sioux Lookout, Ontario (March 2012)
Wardrop, 2011	Tamaka	Technical Report on the Goldlund Property Sioux Lookout, Ontario
Wardrop, 2010	Tamaka	Technical Report and Resource Estimate on the KRP Deposit, Sioux Lookout Ontario (January 2011)
PK Geological Services, 2009	Tamaka	Technical Report and Mineral Inventory Estimate for the Goldlund Group Property, Echo Township, Northwestern Ontario (April 2009)
RPA, 2006	Tamaka	Technical Report on the Goldlund Gold Property, Ontario, Canada (June 2006)

Table 2.4: Abbreviations

Abbreviation	Meaning	Abbreviation	Meaning
µm	micron	km	kilometre
°C	degree Celsius	km^2	square kilometre
°F	degree Fahrenheit	L	litre
°	azimuth/dip in degrees	m	metre
µg	microgram	M	mega (million)
a	annum	m^2	square metre
Au	gold	m^3	cubic metre
C$ or CAD	Canadian dollars	min	minute
cal	calorie	masl	metres above sea level
cm	centimetre	mm	millimetre
d	day	oz/t, oz/st	ounce per short ton
ft	foot or feet	oz	Troy ounce (31.1035 g)
g	gram	ppb	parts per billion
G	giga (billion)	ppm	part per million
g/L	gram per litre	s	second
g/t	gram per tonne	ton, st	short ton
ha	hectare	t, tonne	metric tonne
hp	horse power	US$ or USD	United States dollar
in	inch or inches	yr	year
kg	kilogram		

Source: Ausenco (2021).

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have followed standard professional procedures in preparing the content of this report. Data used in this report has been verified where possible, and this report is based on information believed to be accurate at the time of completion considering the purpose for which the report was prepared. AGP and CGK have no reason to believe the data was not collected in a professional manner.

The Qualified Persons have referenced several sources of information on the properties, including past reports by consultants to Treasury Metals, digital geological maps, and other documents listed in Section 27 of this report. In authoring this report, the Qualified Persons have reviewed the work of the other contributors and find this work has been performed to normal and acceptable industry and professional standards.

3.1 Goliath Project & Miller Project Reliance on Other Experts

AGP has not verified the legal status or legal title to any claims and the legality of any underlying agreements that may exist concerning the Goliath and Miller Projects, as described in Section 4 of this report.

Treasury Metals has supplied the list of mineral rights and mineral claim maps presented in this report. AGP examined the Ontario Ministry of Energy, Northern Development and Mines (MENDM) online GIS website, as well as the online Mining Lands Administration System (MLAS), to selectively review, but not verify, these mineral rights. The MLAS website was most recently viewed on November 27, 2020 at the following digital location:

https://www.gisapplication.lrc.gov.on.ca/Html5Viewer261/index.html?viewer=mlas.mlas&locale=en-US

The text in Section 4 pertaining to the Goliath and Miller Projects was reviewed by Treasury Metals and was accepted on March 9, 2021 by Mr. Mark Wheeler, P. Eng., MBA Director of Project for Treasury Metals Inc.

3.2 Goldlund Project Reliance on Other Experts

CGK has not verified the legal status or legal title to any claims and the legality of any underlying agreements that may exist concerning the Goldlund Project, as described in Section 4 of this report. Treasury Metals provided a list of mineral rights and the mineral claim maps presented in this report, upon which CGK has relied in authoring its sections. CGK examined the MENDM online GIS website and MLAS website to selectively review, but not verify, these mineral rights. The MLAS website was most recently viewed on November 4, 2020 at the following digital location:

https://www.gisapplication.lrc.gov.on.ca/Html5Viewer261/index.html?viewer=mlas.mlas&locale=en-US

The text in Section 4 pertaining to the Goldlund Project was reviewed by Treasury Metals and was accepted on March 9, 2021 by Mr. Mark Wheeler, P. Eng., MBA Director of Project for Treasury Metals Inc.

4 PROPERTY DESCRIPTION & LOCATION

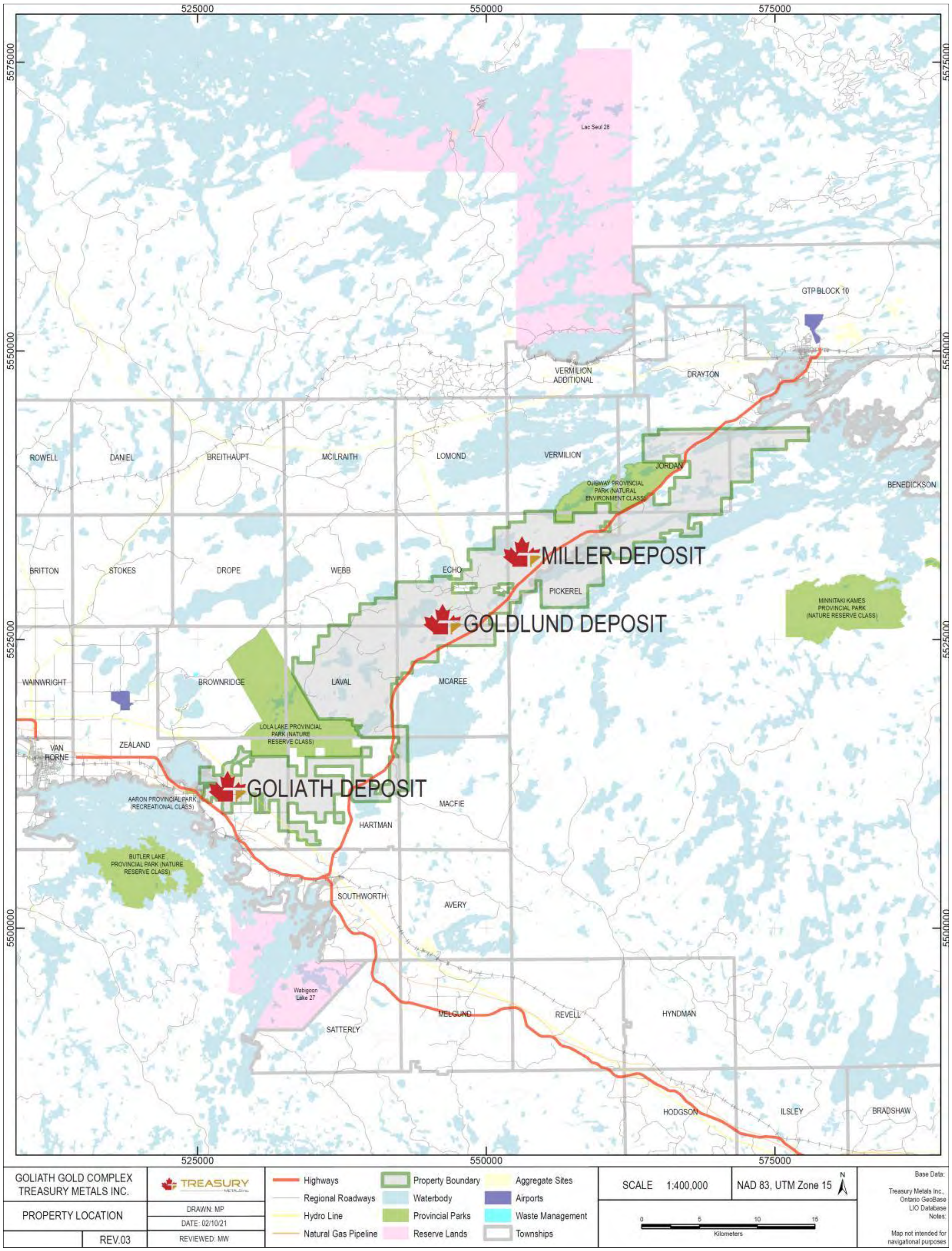
4.1 Goliath Gold Complex Location & Description

The Goliath Gold Complex is comprised of the Goliath and Goldlund-Miller properties, which together cover approximately 34,719 ha . As shown on Figure 4-1, the Goliath Gold Complex is located approximately 350 km northwest of Thunder Bay in the Northwest Ministry of Natural Resources (MNR) Region. The complex can be found on 1:250,000 scale Mapsheets National Topographic System (NTS) 052F (Dryden) and 052K (Lac Seul). Figures 4-2 and 4-3 on the following pages show the location and tenure of the properties.

Figure 4-1: Location of the Goliath Gold Complex

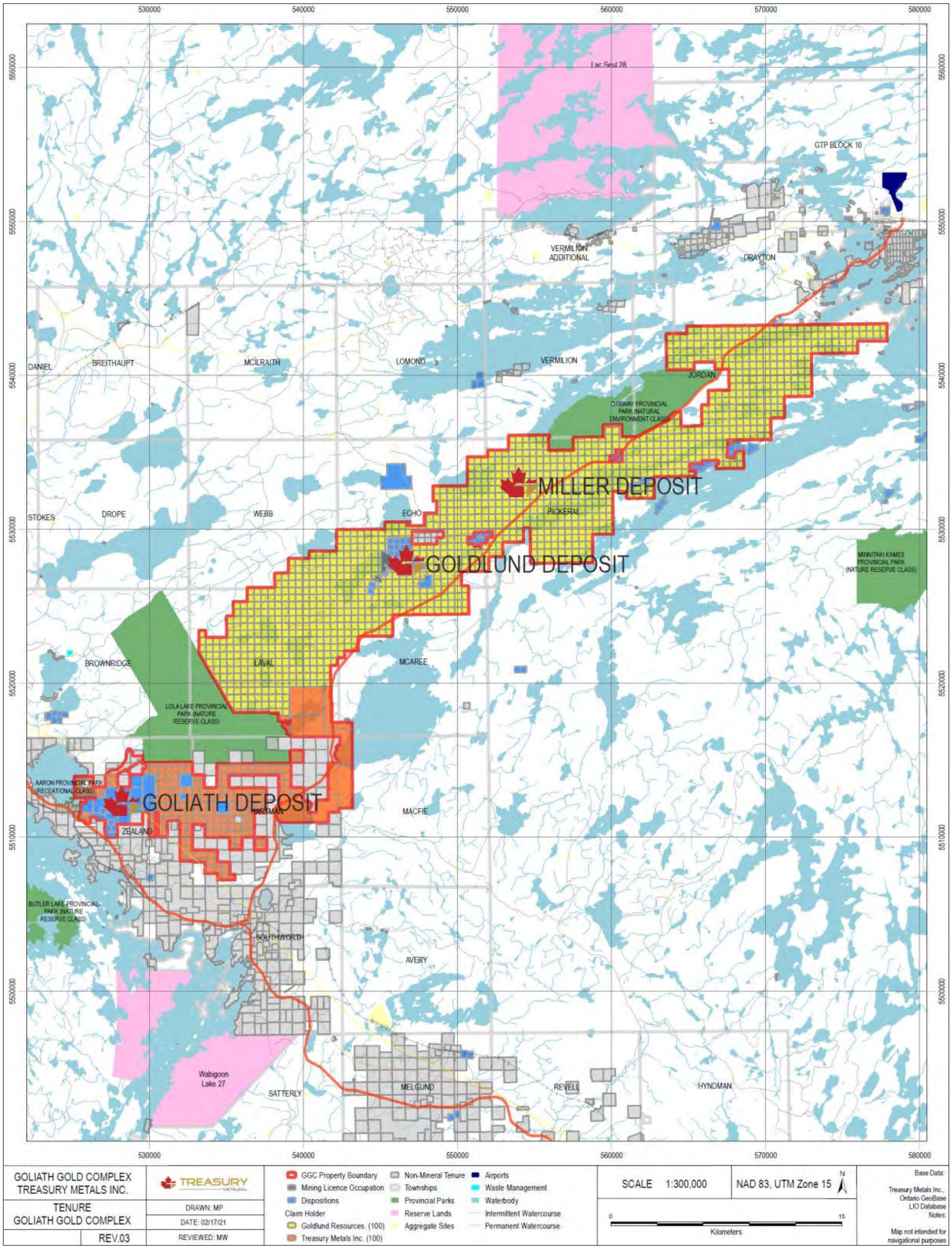
Source: Treasury Metals (2021).

Figure 4-2: Location of the Goliath & Goldlund-Miller Properties



Source: Treasury Metals (2021).

Figure 4-3: Tenure of the Goliath & Goldlund-Miller Properties



Source: Treasury Metals (2021).

The Goliath and Goldlund-Miller properties are described in more detail in Sections 4.1.1 and 4.1.2, respectively.

4.1.1 Goliath Property

4.1.1.1 Goliath Property Location

The Goliath property covers approximately 7,601 ha and is defined by mineral rights and surface rights that are 100% held by Treasury Metals. Of this total, the mineral rights cover approximately 7,511 ha.

The Goliath property has one deposit, the Goliath deposit, and is located as follows:

- on 1:50,000 scale NTS Mapsheets 052F/09 (Dyment), 10 (Wabigoon), 15 (Dryden), and 16 (Big Sandy Lake)
- at approximately 49°45.4' North and 92°33.0' West
- at approximately 532,441 mE; 5,511,624 mN, Zone 15U (NAD83 datum) Universal Transverse Mercator (UTM) coordinates
- in the Kenora Mining Division
- in the Dryden MNR District
- in the Zealand and Hartman Townships
- approximately 3.5 km north of Wabigoon
- approximately 15 km east of Dryden
- approximately 145 km east of Kenora
- approximately 2.5 km east of Aaron Provincial Park
- approximately 2.8 km southeast of Lola Lake Provincial Park
- approximately 1.5 km east of Thunder Lake

4.1.1.2 Goliath Property Description

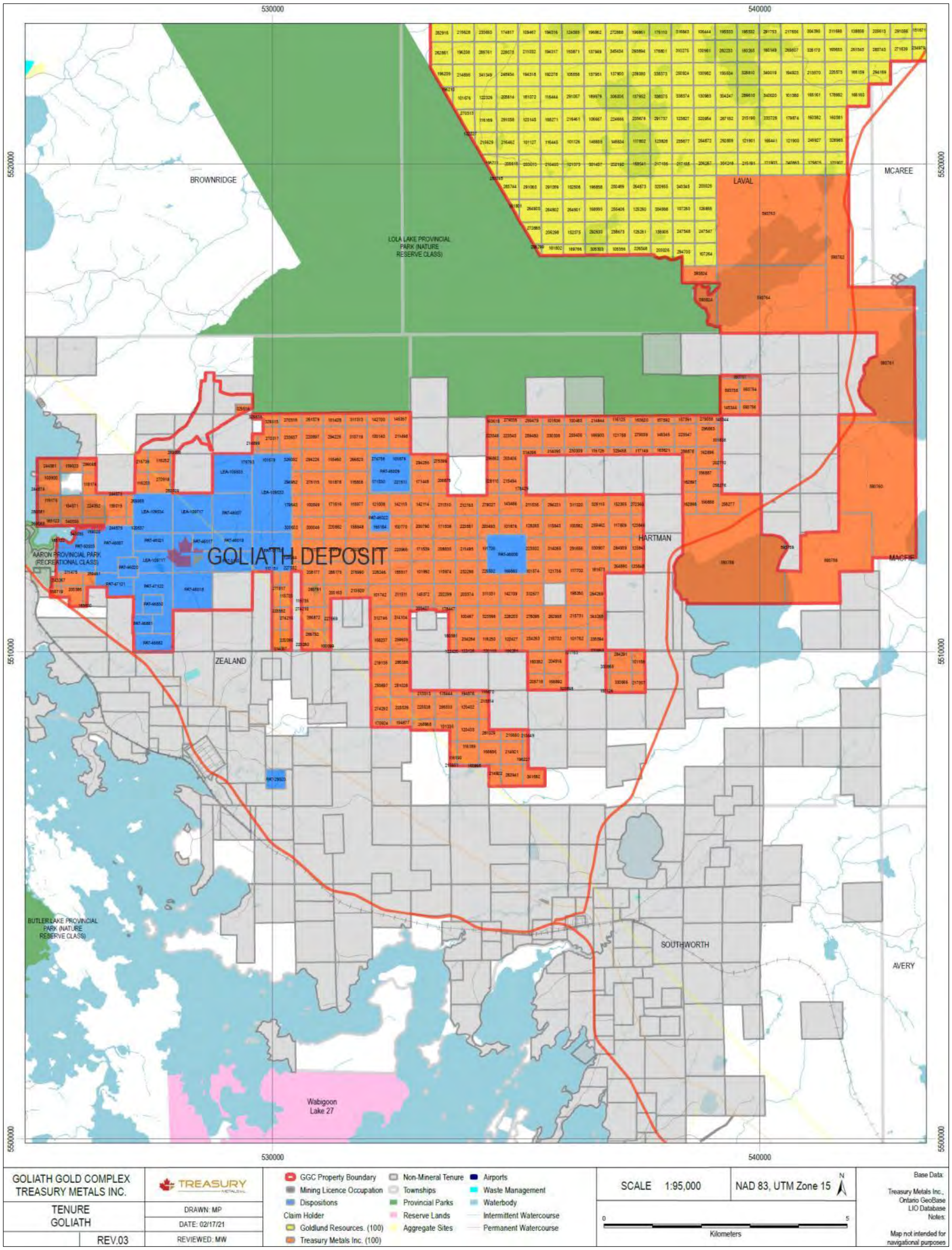
The Goliath property covers approximately 7,601 ha and consists of 284 mining claims totalling approximately 6,254 ha; four mining leases totalling 359.25 ha; and 28 land parcels (includes patented claims) totalling 1,347.189 ha. Of the 1,347.18 ha of the patents and leases, 90.2 ha are surface rights only from seven land parcels. Of the 284 mining claims, 267 are single-cell mining claims, eight are boundary cell mining claims, and nine are multi-cell mining claims. The mineral rights are 100% held by Treasury Metals and all mineral rights are in good standing.

The project is bounded by two provincial parks:

- The Lola Lake Nature Reserve is located at the northern boundary and was designated a nature reserve class park in 1985.
- Aaron Provincial Park is located at the western boundary of the property on the south shore of Thunder Lake. Aaron Provincial Park is a serviced recreation-class park established in 1958 and is operated in co-operation with the City of Dryden.

Figure 4-4 displays the Goliath property mineral and surface rights, which are summarised in Table 4.1. A full listing is provided in Table 4.2.

Figure 4-4: Goliath Property Mineral Rights Map



Source: Treasury Metals (2021).

Table 4.1: Summary of Mineral Rights & Surface Rights for the Goliath Property

Minerals Rights	Grant	PIN	Count	Expiry Date
Mineral Claims	see Table 4.2		284	see Table 4.2
Patented Claims	PA5147 PA13844 n/a n/a PA3900/PA8420 PA3900/PA8420 PA5437 PA7449 PA10196 PA14989 n/a	42090-0053 42089-0079 42089-0138 42089-0229 42089-0237*** 42089-0238*** 42089-0137 42089-0141 42089-0236 42089-0083 42089-0143	10	n/a
Patented Claims	PA12997 PA9074 n/a PA7053 n/a PA3830	42089-0139 ‡ 42089-0232 ‡ 42089-0134‡ 42089-0239 42089-0782 42089-0081	6	n/a
Lease	LEA-109532 LEA-109533 LEA-109534 LEA-109717	42089-0802 42089-0803 42089-0804 n/a	4	n/a
MNR Dryden Tree Nursery	Parcel ID 41807		1	n/a
	Subtotal Mineral Righ			
Patented Claims	PA12997 n/a n/a n/a n/a PA11706	42089-0140 ‡** 42089-0233‡** 42089-0136‡** 42089-0785 42089-0135‡** 42089-0134‡**	6	n/a
MNR Dryden Tree Nursery	Parcel 41810		1	n/a
	Subtotal Surface Rights Or			
	Total Mineral Rights and Surface Righ			

Table 4.2: Goliath Mining Claims

Tenure ID	Township / Area	Tenure Type	Anniversary Date
100099	Zealand	Single Cell Mining Claim	2021-05-21
100140	Zealand	Single Cell Mining Claim	2021-02-26
100467	Hartman	Single Cell Mining Claim	2021-07-10
100483	Hartman	Single Cell Mining Claim	2021-08-21
100549	Zealand	Single Cell Mining Claim	2021-10-26
100562	Hartman	Single Cell Mining Claim	2021-07-10
100770	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
101188	Hartman	Single Cell Mining Claim	2021-02-28
101335	Hartman	Single Cell Mining Claim	2021-02-28
101428	Zealand	Single Cell Mining Claim	2021-02-26
101574	Hartman	Single Cell Mining Claim	2021-04-02
101679	Zealand	Single Cell Mining Claim	2021-05-21
101700	Hartman	Single Cell Mining Claim	2021-02-26
101742	Zealand	Single Cell Mining Claim	2021-02-26
101762	Hartman	Single Cell Mining Claim	2021-07-10
101763	Hartman	Single Cell Mining Claim	2021-08-21
101836	Hartman	Single Cell Mining Claim	2021-08-21
101838	Hartman	Single Cell Mining Claim	2021-08-21
101876	Hartman	Single Cell Mining Claim	2021-02-26
101878	Zealand	Single Cell Mining Claim	2021-10-26
101879	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
101992	Hartman	Single Cell Mining Claim	2021-02-26
103900	Zealand	Single Cell Mining Claim	2021-07-04
103904	Zealand	Boundary Cell Mining Claim	2021-07-04
115735	Zealand	Single Cell Mining Claim	2021-09-10
115838	Zealand	Single Cell Mining Claim	2021-10-26
115843	Hartman	Single Cell Mining Claim	2021-02-26
115974	Hartman	Single Cell Mining Claim	2021-02-26
115977	Zealand	Single Cell Mining Claim	2021-10-26
116125	Hartman	Single Cell Mining Claim	2021-08-21
116126	Hartman	Single Cell Mining Claim	2021-08-21
116189	Hartman	Single Cell Mining Claim	2021-02-28
116190	Hartman	Single Cell Mining Claim	2021-02-28
116250	Hartman	Single Cell Mining Claim	2021-07-10
116252	Zealand	Single Cell Mining Claim	2021-10-10
116253	Zealand	Single Cell Mining Claim	2021-10-10
116670	Hartman	Single Cell Mining Claim	2021-02-28
117149	Hartman	Single Cell Mining Claim	2021-08-21
117151	Zealand	Single Cell Mining Claim	2021-09-10
117702	Hartman	Single Cell Mining Claim	2021-07-10
117809	Hartman	Single Cell Mining Claim	2021-07-10
119174	Zealand	Single Cell Mining Claim	2021-07-04
119175	Zealand	Single Cell Mining Claim	2021-05-21
120432	Hartman	Single Cell Mining Claim	2021-02-28
120433	Hartman	Single Cell Mining Claim	2021-02-28
120537	Zealand	Single Cell Mining Claim	2021-05-21
121008	Zealand	Single Cell Mining Claim	2021-10-26
121756	Hartman	Single Cell Mining Claim	2021-04-02
121788	Hartman	Single Cell Mining Claim	2021-08-21
122427	Hartman	Single Cell Mining Claim	2021-07-10
122428	Hartman	Single Cell Mining Claim	2021-07-10
122429	Hartman	Single Cell Mining Claim	2021-07-10
123846	Hartman	Single Cell Mining Claim	2021-07-10
123847	Hartman	Single Cell Mining Claim	2021-07-10
123848	Hartman	Single Cell Mining Claim	2021-07-10
124944	Zealand	Single Cell Mining Claim	2021-10-26

Tenure ID	Township / Area	Tenure Type	Anniversary Date
128265	Hartman	Single Cell Mining Claim	2021-02-26
142114	Hartman	Single Cell Mining Claim	2021-02-26
142115	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
142700	Zealand	Single Cell Mining Claim	2021-02-26
142709	Hartman	Single Cell Mining Claim	2021-07-10
143486	Hartman	Single Cell Mining Claim	2021-04-02
145344	Hartman	Single Cell Mining Claim	2021-08-21
145345	Hartman	Single Cell Mining Claim	2021-08-21
145357	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
145372	Hartman	Single Cell Mining Claim	2021-02-26
152355	Hartman	Single Cell Mining Claim	2021-07-10
155460	Zealand	Single Cell Mining Claim	2021-10-26
155517	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
156887	Hartman	Single Cell Mining Claim	2021-08-21
156888	Hartman	Boundary Cell Mining Claim	2021-08-21
157591	Hartman	Single Cell Mining Claim	2021-08-21
157592	Hartman	Single Cell Mining Claim	2021-08-21
158237	Zealand	Single Cell Mining Claim	2021-09-06
158719	Zealand	Single Cell Mining Claim	2021-10-13
158848	Zealand	Single Cell Mining Claim	2021-10-26
159019	Zealand	Single Cell Mining Claim	2021-05-21
159020	Zealand	Single Cell Mining Claim	2021-05-21
159023	Zealand	Single Cell Mining Claim	2021-07-04
160968	Hartman	Boundary Cell Mining Claim	2021-02-28
162896	Hartman	Single Cell Mining Claim	2021-08-21
162897	Hartman	Single Cell Mining Claim	2021-08-21
162898	Hartman	Boundary Cell Mining Claim	2021-08-21
163600	Zealand	Single Cell Mining Claim	2021-10-13
163618	Hartman	Single Cell Mining Claim	2021-08-21
163620	Hartman	Single Cell Mining Claim	2021-08-21
163621	Hartman	Single Cell Mining Claim	2021-08-21
165122	Zealand	Single Cell Mining Claim	2021-05-21
166184	Zealand	Single Cell Mining Claim	2021-10-26
166860	Hartman	Single Cell Mining Claim	2021-02-26
166903	Hartman	Single Cell Mining Claim	2021-08-21
166956	Hartman	Single Cell Mining Claim	2021-02-28
168892	Hartman	Single Cell Mining Claim	2021-08-21
170773	Zealand	Single Cell Mining Claim	2021-10-26
170924	Zealand	Single Cell Mining Claim	2021-09-06
171448	Hartman	Single Cell Mining Claim	2021-02-26
171516	Zealand	Single Cell Mining Claim	2021-10-26
171530	Zealand	Single Cell Mining Claim	2021-10-26
171538	Hartman	Single Cell Mining Claim	2021-02-26
171539	Hartman	Single Cell Mining Claim	2021-02-26
178429	Hartman	Single Cell Mining Claim	2021-04-02
178444	Hartman	Single Cell Mining Claim	2021-02-28
178447	Hartman	Single Cell Mining Claim	2021-07-10
179643	Zealand	Single Cell Mining Claim	2021-10-26
179793	Zealand	Single Cell Mining Claim	2021-05-21
180381	Hartman	Single Cell Mining Claim	2021-07-10
180382	Hartman	Single Cell Mining Claim	2021-08-21
181126	Hartman	Single Cell Mining Claim	2021-02-28
181673	Hartman	Single Cell Mining Claim	2021-07-10
184571	Zealand	Single Cell Mining Claim	2021-05-21
194876	Hartman	Single Cell Mining Claim	2021-02-28
194877	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
196227	Hartman	Single Cell Mining Claim	2021-02-28
196284	Hartman	Single Cell Mining Claim	2021-07-10

Tenure ID	Township / Area	Tenure Type	Anniversary Date
198260	Hartman	Single Cell Mining Claim	2021-07-10
200046	Zealand	Single Cell Mining Claim	2021-10-26
200163	Zealand	Single Cell Mining Claim	2021-05-21
200790	Hartman	Single Cell Mining Claim	2021-02-26
202710	Hartman	Single Cell Mining Claim	2021-08-21
203359	Zealand	Single Cell Mining Claim	2021-10-26
203374	Hartman	Single Cell Mining Claim	2021-02-26
203386	Zealand	Single Cell Mining Claim	2021-10-13
203405	Hartman	Single Cell Mining Claim	2021-08-21
203406	Hartman	Single Cell Mining Claim	2021-08-21
203427	Hartman	Single Cell Mining Claim	2021-02-26
203493	Hartman	Single Cell Mining Claim	2021-02-26
204916	Hartman	Single Cell Mining Claim	2021-08-21
205715	Hartman	Single Cell Mining Claim	2021-08-21
208177	Zealand	Single Cell Mining Claim	2021-10-26
208830	Hartman	Single Cell Mining Claim	2021-02-26
208878	Hartman	Single Cell Mining Claim	2021-02-26
209519	Zealand	Single Cell Mining Claim	2021-10-26
211475	Zealand	Single Cell Mining Claim	2021-10-13
211495	Hartman	Single Cell Mining Claim	2021-02-26
211498	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
211510	Hartman	Single Cell Mining Claim	2021-02-26
211511	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
211536	Hartman	Single Cell Mining Claim	2021-04-02
212763	Hartman	Single Cell Mining Claim	2021-02-26
213494	Hartman	Single Cell Mining Claim	2021-04-02
213513	Hartman	Single Cell Mining Claim	2021-02-28
213514	Hartman	Single Cell Mining Claim	2021-02-28
213520	Zealand	Single Cell Mining Claim	2021-02-26
214844	Hartman	Single Cell Mining Claim	2021-08-21
214899	Zealand	Single Cell Mining Claim	2021-05-21
214921	Hartman	Single Cell Mining Claim	2021-02-28
214922	Hartman	Single Cell Mining Claim	2021-02-28
215649	Hartman	Single Cell Mining Claim	2021-02-28
215650	Hartman	Single Cell Mining Claim	2021-02-28
215651	Hartman	Single Cell Mining Claim	2021-02-28
215731	Hartman	Single Cell Mining Claim	2021-07-10
215732	Hartman	Single Cell Mining Claim	2021-07-10
215736	Zealand	Single Cell Mining Claim	2021-10-10
217007	Hartman	Single Cell Mining Claim	2021-02-28
219135	Zealand	Single Cell Mining Claim	2021-09-06
220280	Zealand	Single Cell Mining Claim	2021-09-10
220882	Zealand	Single Cell Mining Claim	2021-10-26
220897	Zealand	Single Cell Mining Claim	2021-02-26
220966	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
223002	Hartman	Single Cell Mining Claim	2021-04-02
223545	Hartman	Single Cell Mining Claim	2021-08-21
223546	Hartman	Single Cell Mining Claim	2021-08-21
223547	Hartman	Single Cell Mining Claim	2021-08-21
223551	Hartman	Single Cell Mining Claim	2021-02-26
223552	Zealand	Single Cell Mining Claim	2021-09-10
224392	Zealand	Single Cell Mining Claim	2021-05-21
225528	Hartman	Single Cell Mining Claim	2021-02-28
225529	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
225532	Hartman	Single Cell Mining Claim	2021-02-26
227552	Zealand	Single Cell Mining Claim	2021-10-26
227569	Zealand	Single Cell Mining Claim	2021-05-21
227611	Hartman, Zealand	Single Cell Mining Claim	2021-02-26

Tenure ID	Township / Area	Tenure Type	Anniversary Date
228203	Hartman	Single Cell Mining Claim	2021-07-10
228246	Zealand	Single Cell Mining Claim	2021-10-26
230308	Hartman	Single Cell Mining Claim	2021-08-21
230309	Hartman	Single Cell Mining Claim	2021-08-21
232298	Hartman	Single Cell Mining Claim	2021-02-26
232299	Hartman	Single Cell Mining Claim	2021-02-26
233657	Zealand	Single Cell Mining Claim	2021-05-21
234263	Hartman	Single Cell Mining Claim	2021-07-10
234264	Hartman	Single Cell Mining Claim	2021-07-10
235594	Hartman	Single Cell Mining Claim	2021-07-10
244573	Zealand	Single Cell Mining Claim	2021-05-21
244574	Zealand	Boundary Cell Mining Claim	2021-07-04
244575	Zealand	Single Cell Mining Claim	2021-05-21
244581	Zealand	Single Cell Mining Claim	2021-07-04
258276	Hartman	Single Cell Mining Claim	2021-08-21
258277	Hartman	Boundary Cell Mining Claim	2021-08-21
259461	Zealand	Single Cell Mining Claim	2021-10-13
259462	Hartman	Single Cell Mining Claim	2021-07-10
259479	Hartman	Single Cell Mining Claim	2021-08-21
259480	Hartman	Single Cell Mining Claim	2021-08-21
259609	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
261579	Zealand	Single Cell Mining Claim	2021-02-26
261732	Zealand	Single Cell Mining Claim	2021-07-04
262955	Hartman	Single Cell Mining Claim	2021-07-10
264269	Hartman	Single Cell Mining Claim	2021-07-10
264890	Hartman	Single Cell Mining Claim	2021-07-10
266791	Zealand	Single Cell Mining Claim	2021-05-21
266792	Zealand	Single Cell Mining Claim	2021-05-21
266823	Zealand	Single Cell Mining Claim	2021-02-26
268968	Hartman	Single Cell Mining Claim	2021-02-28
269068	Zealand	Single Cell Mining Claim	2021-05-21
269069	Zealand	Single Cell Mining Claim	2021-05-21
270316	Zealand	Single Cell Mining Claim	2021-05-21
270317	Zealand	Single Cell Mining Claim	2021-05-21
270918	Zealand	Single Cell Mining Claim	2021-10-10
272360	Hartman	Single Cell Mining Claim	2021-07-10
274210	Zealand	Single Cell Mining Claim	2021-09-10
274292	Zealand	Single Cell Mining Claim	2021-09-06
274756	Zealand	Single Cell Mining Claim	2021-02-26
275399	Hartman	Single Cell Mining Claim	2021-02-26
276115	Zealand	Single Cell Mining Claim	2021-10-26
277517	Zealand	Single Cell Mining Claim	2021-09-10
278095	Hartman	Single Cell Mining Claim	2021-07-10
278990	Zealand	Single Cell Mining Claim	2021-10-26
279027	Hartman	Single Cell Mining Claim	2021-04-02
279036	Hartman	Single Cell Mining Claim	2021-08-21
279038	Hartman	Single Cell Mining Claim	2021-08-21
279039	Hartman	Single Cell Mining Claim	2021-08-21
280381	Zealand	Boundary Cell Mining Claim	2021-05-21
281028	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
281029	Hartman	Single Cell Mining Claim	2021-02-28
282941	Hartman	Single Cell Mining Claim	2021-02-28
283008	Zealand	Single Cell Mining Claim	2021-10-10
283009	Zealand	Single Cell Mining Claim	2021-10-10
284291	Hartman	Single Cell Mining Claim	2021-02-28
284939	Hartman	Single Cell Mining Claim	2021-07-10
286386	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
286872	Zealand	Single Cell Mining Claim	2021-05-21

Tenure ID	Township / Area	Tenure Type	Anniversary Date
287545	Zealand	Single Cell Mining Claim	2021-10-26
288175	Zealand	Single Cell Mining Claim	2021-10-26
288878	Hartman	Single Cell Mining Claim	2021-08-21
291656	Hartman	Single Cell Mining Claim	2021-07-10
293697	Zealand	Single Cell Mining Claim	2021-09-06
294225	Zealand	Single Cell Mining Claim	2021-02-26
294226	Zealand	Single Cell Mining Claim	2021-10-26
294231	Hartman	Single Cell Mining Claim	2021-02-26
294256	Hartman	Single Cell Mining Claim	2021-02-26
294962	Zealand	Single Cell Mining Claim	2021-10-26
296862	Hartman	Single Cell Mining Claim	2021-08-21
296863	Hartman	Single Cell Mining Claim	2021-08-21
298333	Hartman	Single Cell Mining Claim	2021-02-28
299048	Zealand	Single Cell Mining Claim	2021-07-04
310719	Zealand	Single Cell Mining Claim	2021-02-26
311313	Zealand	Single Cell Mining Claim	2021-02-26
311320	Hartman	Single Cell Mining Claim	2021-02-26
311331	Hartman	Single Cell Mining Claim	2021-02-26
312677	Hartman	Single Cell Mining Claim	2021-07-10
312746	Zealand	Single Cell Mining Claim	2021-07-10
314065	Hartman	Single Cell Mining Claim	2021-04-02
314095	Hartman	Single Cell Mining Claim	2021-08-21
314096	Hartman	Single Cell Mining Claim	2021-08-21
314097	Zealand	Single Cell Mining Claim	2021-09-10
314104	Hartman, Zealand	Single Cell Mining Claim	2021-07-10
320652	Zealand	Single Cell Mining Claim	2021-10-26
320898	Hartman	Single Cell Mining Claim	2021-08-21
323556	Hartman	Single Cell Mining Claim	2021-07-10
326092	Zealand	Single Cell Mining Claim	2021-10-26
326115	Hartman	Single Cell Mining Claim	2021-07-10
328110	Hartman	Single Cell Mining Claim	2021-04-02
329458	Hartman	Single Cell Mining Claim	2021-08-21
329515	Zealand	Single Cell Mining Claim	2021-05-21
329516	Zealand	Single Cell Mining Claim	2021-05-21
330119	Hartman	Single Cell Mining Claim	2021-07-10
330865	Hartman	Single Cell Mining Claim	2021-07-10
330866	Hartman	Single Cell Mining Claim	2021-02-28
330907	Hartman	Single Cell Mining Claim	2021-07-10
340035	Zealand	Single Cell Mining Claim	2021-05-21
341882	Hartman	Single Cell Mining Claim	2021-02-28
343265	Hartman	Single Cell Mining Claim	2021-07-10
343267	Zealand	Boundary Cell Mining Claim	2021-10-13
593754	Hartman	Single Cell Mining Claim	2022-06-03
593755	Hartman	Single Cell Mining Claim	2022-06-03
593756	Hartman	Single Cell Mining Claim	2022-06-03
593757	Hartman	Multi-cell Mining Claim	2022-06-03
593758	Hartman	Multi-cell Mining Claim	2022-06-03
593759	Hartman	Multi-cell Mining Claim	2022-06-03
593760	Hartman, MacFie	Multi-cell Mining Claim	2022-06-03
593761	Hartman, Laval, MacFie, McAree	Multi-cell Mining Claim	2022-06-03
593762	Laval, McAree	Multi-cell Mining Claim	2022-06-03
593763	Laval	Multi-cell Mining Claim	2022-06-03
593764	Laval	Multi-cell Mining Claim	2022-06-03
593824	Laval	Multi-cell Mining Claim	2022-06-04

4.1.2 Goldlund-Miller Property

4.1.2.1 Goldlund-Miller Property Location

The Goldlund-Miller property covers approximately 27,118 ha and is defined by mineral rights that are 100% held by Treasury Metals. Two deposits, Goldlund and Miller, comprise the Goldlund-Miller property, as detailed below.

The Goldlund deposit is located as follows:

- on the Goldlund-Miller property
- on 1:50,000 scale NTS Mapsheets 052F16 (Big Sandy Lake), 052K/01 (Hudson) and 052J/04 (Sioux Lookout)
- at approximately 49°54' North and 92°20.5' West
- at approximately 547000 E; 5527500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Patricia Mining Division
- in the Sioux Lookout MNR District
- in the Echo and Pickerel Townships
- approximately 40 km southeast of Sioux Lookout (42 km by road)
- approximately 40 km east of Dryden (62 km by road)
- approximately 12 km southeast of Ojibway Provincial Park
- approximately 1.2 km east of Crossecho Lake

The Miller deposit is located as follows:

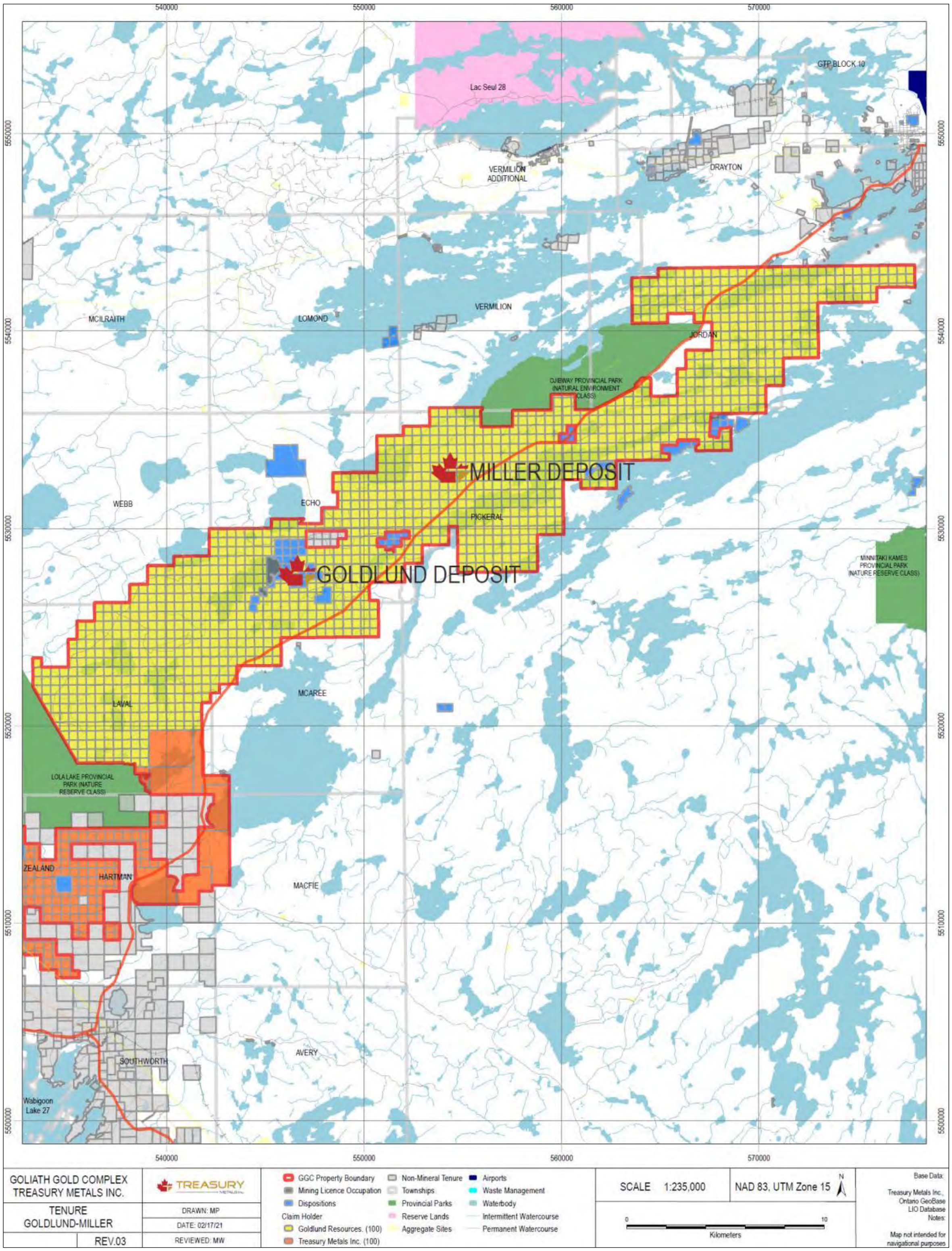
- on 1:50,000 scale NTS Mapsheet 052F16 (Big Sandy Lake)
- at approximately 49°57' North and 92°15' West
- at approximately 534000 E; 5534500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Pickerel Township
- approximately 27 km southwest of Sioux Lookout (35 km by road)
- approximately 47 km northeast of Dryden (65 km by road)
- approximately 1.7 km west of Ojibway Provincial Park
- approximately 1.2 km southwest of Little Vermilion Lake

Refer to Figures 4-2 and 4-3 above for the location of the Goldlund and Miller deposits.

4.1.2.2 Goldlund-Miller Property Description

Shown in Figure 4-5, the Goldlund-Miller property consists of 1,349 mining claims totalling approximately 26,634 ha, 26 patented claims totalling 390.97 ha, one mining lease of 48.56 ha, and one licence of occupation of 74.84 ha.

Figure 4-5: Goldlund-Miller Property Mineral Rights Map



Source: Treasury Metals (2021).

The patented claims and mining lease allow for both mineral rights and surface rights, while the Licence of Occupation allows for mineral rights only.

Table 4.3 presents a summary of the mineral rights for the Goldlund-Miller property. A full listing is provided on Table 4.4 on the following pages.

Table 4.3: Summary of Mineral Rights for the Goldlund-Miller Property

Minerals Rights	Mineral Rights Number	Count	Expiry Date	Area (ha)	Comment
Mineral Claims	see Table 4.4	1,349	see Table 4.4	~26,633.89*	
Patented Claims	PAT-6534 - 6553 PAT-41749 - 41754	20 6	n/a	360.97	Echo Twp.
Mining Lease	LEA-107464	1	31-Jul-2024	48.56	Echo Twp.
Licence of Occupation	MLO-12023	1	n/a	74.84	Echo Twp.
				~27,118.26	**Subtotal**

Notes: * approximation from property outline area less the area of Patented Claims, Mining Lease and Licence of Occupation; TWP: township . Source: Treasury Metals (2020).

Under the provincial system for mining claims, since January 2018, the 142 legacy claims have been converted into 1,342 single-cell mining claims, six boundary-cell mining claims, and one multi-cell mining claim. Dispositions for patents, leases, and licences of occupation were not converted under the new system and remain as they were.

All mineral rights are in good standing and have been granted extra time to allow for credit distributions due to the large number of claims involved.

The property was previously distributed into nine blocks to help manage exploration information. These divisions, which have been maintained by Treasury Metals, do not reflect any geological differences.

Table 4.4: Goldlund-Miller Property Mining Claims

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type
100003	Pickerel	Single Cell Mining Claim	2021-04-05	234979	McAree	Single Cell Mining C
100005	Echo	Single Cell Mining Claim	2021-04-26	235044	McAree	Single Cell Mining C
100282	Echo	Single Cell Mining Claim	2021-01-13	235052	Echo	Single Cell Mining C
100468	Pickerel	Single Cell Mining Claim	2020-08-11	235053	Echo	Single Cell Mining C
100570	Pickerel	Single Cell Mining Claim	2022-01-13	235676	Laval	Single Cell Mining C
100571	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	235677	Laval	Single Cell Mining C
100832	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	235703	Vermilion	Single Cell Mining C
100834	Drayton	Single Cell Mining Claim	2020-12-15	235727	Kabik Lake Area	Single Cell Mining C
100866	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	235728	Kabik Lake Area	Single Cell Mining C
100892	Jordan	Single Cell Mining Claim	2021-04-20	235740	Kabik Lake Area	Single Cell Mining C
100893	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	236625	McAree	Single Cell Mining C
100896	Jordan, Kabik Lake Area	Single Cell Mining Claim	2020-12-15	236626	McAree	Single Cell Mining C
100936	Jordan	Single Cell Mining Claim	2020-12-15	236636	McAree	Single Cell Mining C
100937	Jordan	Single Cell Mining Claim	2020-12-15	238673	Laval	Single Cell Mining C
100948	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	239380	Laval	Single Cell Mining C
101003	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	240268	Laval	Single Cell Mining C
101027	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2020-08-11	240310	Kabik Lake Area	Single Cell Mining C
101080	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	240311	Kabik Lake Area	Single Cell Mining C
101102	Echo	Single Cell Mining Claim	2020-08-11	240312	Kabik Lake Area	Single Cell Mining C
101103	Echo	Single Cell Mining Claim	2021-02-12	242208	Laval	Single Cell Mining C
101126	Laval	Single Cell Mining Claim	2020-09-30	242217	Laval	Single Cell Mining C
101127	Laval	Single Cell Mining Claim	2020-12-04	242696	Jordan	Single Cell Mining C
101246	Echo	Single Cell Mining Claim	2021-02-12	242697	Jordan	Single Cell Mining C
101268	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	243389	Jordan	Single Cell Mining C
101332	McAree	Single Cell Mining Claim	2020-08-05	244113	Laval	Single Cell Mining C
101336	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	245924	Parnes Lake Area	Single Cell Mining C
101359	Echo, McAree	Single Cell Mining Claim	2020-09-30	245927	Laval	Single Cell Mining C
101380	Laval	Single Cell Mining Claim	2020-08-30	246994	Laval	Single Cell Mining C
101407	Echo	Single Cell Mining Claim	2020-08-11	247547	Laval	Single Cell Mining C
101408	Echo	Single Cell Mining Claim	2020-08-11	247548	Laval	Single Cell Mining C
101498	Echo, Pickerel	Single Cell Mining Claim	2021-02-12	248253	Laval	Single Cell Mining C
101593	Jordan	Single Cell Mining Claim	2020-12-15	248934	Laval	Single Cell Mining C
101676	Laval	Single Cell Mining Claim	2020-09-30	249706	Laval	Single Cell Mining C
101738	Laval	Single Cell Mining Claim	2021-01-24	249711	Laval	Single Cell Mining C
101760	Jordan	Single Cell Mining Claim	2020-12-15	250924	Laval	Single Cell Mining C
101761	Pickerel	Single Cell Mining Claim	2021-01-13	252057	Parnes Lake Area	Single Cell Mining C
101764	Echo	Single Cell Mining Claim	2021-02-12	252192	Laval	Single Cell Mining C
101767	Laval	Single Cell Mining Claim	2020-10-31	253399	McAree	Single Cell Mining C
101775	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	253503	Echo	Single Cell Mining C
101776	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	253504	Echo	Single Cell Mining C
101837	Pickerel	Single Cell Mining Claim	2021-01-13	255406	Laval	Single Cell Mining C
101849	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	257839	Webb	Single Cell Mining C
101850	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	258271	Echo	Single Cell Mining C
101862	Jordan	Single Cell Mining Claim	2020-12-15	258272	Echo	Single Cell Mining C
101863	Jordan	Single Cell Mining Claim	2020-12-15	258933	Jordan	Single Cell Mining C
101864	Jordan	Single Cell Mining Claim	2020-12-15	258941	Echo	Single Cell Mining C
101865	Jordan	Single Cell Mining Claim	2020-12-15	258942	Echo	Single Cell Mining C
102027	Pickerel	Single Cell Mining Claim	2021-01-13	258943	Echo	Single Cell Mining C
102028	Pickerel	Single Cell Mining Claim	2021-01-13	259483	Pickerel, Vermilion	Single Cell Mining C
102053	Jordan	Single Cell Mining Claim	2020-12-15	259484	Pickerel	Single Cell Mining C
102054	Jordan	Single Cell Mining Claim	2020-12-15	259498	Pickerel, Vermilion	Single Cell Mining C
102055	Jordan	Single Cell Mining Claim	2020-12-15	259499	Pickerel	Single Cell Mining C
102092	Webb	Single Cell Mining Claim	2021-01-24	259503	Kabik Lake Area, Pickerel	Single Cell Mining C
102093	Webb	Single Cell Mining Claim	2021-01-24	259504	Kabik Lake Area, Pickerel	Single Cell Mining C
102490	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	259517	Jordan	Single Cell Mining C
102501	Vermilion	Single Cell Mining Claim	2021-01-13	259576	Jordan	Single Cell Mining C
102506	Laval	Single Cell Mining Claim	2020-09-30	260150	Drayton	Single Cell Mining C
102578	Laval	Single Cell Mining Claim	2020-09-30	260170	McAree	Single Cell Mining C
102579	Laval	Single Cell Mining Claim	2020-09-30	260171	McAree	Single Cell Mining C
102594	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260173	Echo, Pickerel	Single Cell Mining C
102934	Pickerel	Single Cell Mining Claim	2021-01-13	260179	Jordan	Single Cell Mining C
103716	McAree	Single Cell Mining Claim	2020-09-28	260180	Jordan	Single Cell Mining C
104240	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260181	Echo, McAree	Single Cell Mining C
104241	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260189	Kabik Lake Area	Single Cell Mining C

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116350	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	263536	Kabik Lake Area	Single Cell Mining C
116368	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	263539	Kabik Lake Area	Single Cell Mining C
116404	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	263549	Echo	Single Cell Mining C
116443	Laval, McAree	Single Cell Mining Claim	2020-08-05	263631	Echo, Pickerel	Single Cell Mining C
116444	Laval	Single Cell Mining Claim	2020-09-30	263632	Echo, Pickerel	Single Cell Mining C
116445	Laval	Single Cell Mining Claim	2020-09-30	264241	Echo	Single Cell Mining C
116448	Webb	Single Cell Mining Claim	2021-01-24	264270	Echo	Single Cell Mining C
116450	Jordan	Single Cell Mining Claim	2020-12-15	264286	Echo	Single Cell Mining C
116489	McAree	Single Cell Mining Claim	2020-08-05	264872	Laval	Single Cell Mining C
116490	Laval, McAree	Single Cell Mining Claim	2020-08-05	264873	Laval	Single Cell Mining C
116544	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	264901	Laval	Single Cell Mining C
116549	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	264902	Laval	Single Cell Mining C
116594	Jordan	Single Cell Mining Claim	2021-04-20	264903	Laval	Single Cell Mining C
116596	Jordan	Single Cell Mining Claim	2020-12-15	264928	Kabik Lake Area	Single Cell Mining C
116620	Jordan	Single Cell Mining Claim	2020-12-15	264929	Kabik Lake Area	Single Cell Mining C
116623	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	266146	Pickerel	Single Cell Mining C
116725	Echo	Single Cell Mining Claim	2020-08-11	266147	Pickerel	Single Cell Mining C
116791	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	266148	Kabik Lake Area, Pickerel	Single Cell Mining C
116826	Pickerel	Single Cell Mining Claim	2021-01-13	266165	Kabik Lake Area, Pickerel	Single Cell Mining C
116827	Pickerel	Single Cell Mining Claim	2021-02-12	267052	Laval	Single Cell Mining C
116912	Drayton	Single Cell Mining Claim	2020-12-15	267152	Laval	Single Cell Mining C
116937	Jordan	Single Cell Mining Claim	2020-12-15	267425	Pickerel	Single Cell Mining C
116938	Jordan	Single Cell Mining Claim	2020-12-15	268200	Jordan	Single Cell Mining C
116939	Jordan	Single Cell Mining Claim	2020-12-15	268207	Kabik Lake Area	Single Cell Mining C
117089	Jordan	Single Cell Mining Claim	2020-12-15	268208	Kabik Lake Area	Single Cell Mining C
117096	Echo	Single Cell Mining Claim	2020-08-11	268222	Pickerel	Single Cell Mining C
117097	Echo	Single Cell Mining Claim	2021-02-12	268850	Drayton	Single Cell Mining C
117098	Echo	Single Cell Mining Claim	2025-03-29	268851	Drayton, Parnes Lake Area	Single Cell Mining C
117099	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	268904	Jordan	Single Cell Mining C
117100	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	268955	Echo	Single Cell Mining C
117148	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	268987	Echo	Single Cell Mining C
117163	Pickerel	Single Cell Mining Claim	2021-01-13	269507	Laval	Single Cell Mining C
117169	Pickerel	Single Cell Mining Claim	2021-01-13	269535	Echo	Single Cell Mining C
117170	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	269536	Echo	Single Cell Mining C
117190	Jordan	Single Cell Mining Claim	2020-12-15	269580	Kabik Lake Area	Single Cell Mining C
117672	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	269622	Pickerel	Single Cell Mining C
117676	Echo	Single Cell Mining Claim	2020-08-11	269623	Pickerel	Single Cell Mining C
117701	Pickerel	Single Cell Mining Claim	2021-01-13	269658	Drayton	Single Cell Mining C
117754	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270190	Drayton	Single Cell Mining C
117755	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270309	Laval	Single Cell Mining C
117756	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270313	Laval	Single Cell Mining C
117802	Laval	Single Cell Mining Claim	2020-09-30	270318	Echo	Single Cell Mining C
117810	Vermilion	Single Cell Mining Claim	2021-01-13	270435	Echo, McAree	Single Cell Mining C
117811	Vermilion	Single Cell Mining Claim	2021-01-13	270436	McAree	Single Cell Mining C
117817	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	270437	McAree	Single Cell Mining C
117888	Laval	Single Cell Mining Claim	2021-01-24	270452	McAree	Single Cell Mining C
117889	Laval	Single Cell Mining Claim	2020-09-30	270888	Webb	Single Cell Mining C
118176	Jordan	Single Cell Mining Claim	2020-12-15	270914	Jordan	Single Cell Mining C
118244	Pickerel	Single Cell Mining Claim	2021-01-13	270915	Echo	Single Cell Mining C
120327	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	270916	Echo	Single Cell Mining C
120349	Jordan	Single Cell Mining Claim	2020-12-15	270925	Jordan	Single Cell Mining C
120350	Jordan	Single Cell Mining Claim	2020-12-15	270926	Jordan	Single Cell Mining C
120381	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270935	Echo	Single Cell Mining C
120382	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270990	Kabik Lake Area	Single Cell Mining C
120383	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270991	Kabik Lake Area	Single Cell Mining C
120429	McAree	Single Cell Mining Claim	2020-08-05	270997	Kabik Lake Area	Single Cell Mining C
121009	Pickerel	Single Cell Mining Claim	2021-04-05	271020	Pickerel	Single Cell Mining C
121010	Pickerel	Single Cell Mining Claim	2021-04-05	271024	Echo	Single Cell Mining C
121075	McAree	Single Cell Mining Claim	2020-08-05	271132	Parnes Lake Area	Single Cell Mining C
121122	Pickerel	Single Cell Mining Claim	2021-02-12	271639	McAree	Single Cell Mining C
121123	Pickerel	Single Cell Mining Claim	2021-02-12	272213	Echo	Single Cell Mining C
121124	Pickerel	Single Cell Mining Claim	2020-08-11	272234	Pickerel	Single Cell Mining C
121373	Laval	Single Cell Mining Claim	2020-09-30	272235	Echo	Single Cell Mining C
121667	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	272236	Echo	Single Cell Mining C
121746	Jordan	Single Cell Mining Claim	2020-12-15	272282	Kabik Lake Area	Single Cell Mining C
121823	Laval	Single Cell Mining Claim	2020-09-30	272283	Kabik Lake Area	Single Cell Mining C

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126884	Echo, McAree	Single Cell Mining Claim	2025-09-30	279003	Echo	Single Cell Mining C
126885	McAree	Single Cell Mining Claim	2020-09-28	279004	Echo	Single Cell Mining C
126961	McAree	Single Cell Mining Claim	2020-09-28	279005	Pickerel	Single Cell Mining C
127543	Jordan	Single Cell Mining Claim	2020-12-15	279006	Kabik Lake Area, Pickerel	Single Cell Mining C
127544	Jordan	Single Cell Mining Claim	2020-12-15	279560	Pickerel	Single Cell Mining C
127545	Jordan	Single Cell Mining Claim	2020-12-15	279561	Pickerel	Single Cell Mining C
127597	Echo	Single Cell Mining Claim	2021-01-13	279562	Pickerel	Single Cell Mining C
127598	Echo	Single Cell Mining Claim	2021-01-13	279564	Pickerel	Single Cell Mining C
127599	Echo	Single Cell Mining Claim	2020-08-11	279565	Kabik Lake Area, Pickerel	Single Cell Mining C
128305	Pickerel	Single Cell Mining Claim	2021-01-13	279579	Jordan	Single Cell Mining C
128306	Pickerel	Single Cell Mining Claim	2021-01-13	279664	Echo	Single Cell Mining C
128335	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	279692	Pickerel	Single Cell Mining C
128915	Jordan	Single Cell Mining Claim	2020-12-15	280225	Drayton	Single Cell Mining C
128977	Pickerel	Single Cell Mining Claim	2021-01-13	280255	Echo	Single Cell Mining C
129011	Drayton	Single Cell Mining Claim	2020-12-15	280257	Jordan	Single Cell Mining C
129012	Drayton	Single Cell Mining Claim	2020-12-15	280261	Kabik Lake Area	Single Cell Mining C
129508	Laval	Single Cell Mining Claim	2020-09-30	280262	Kabik Lake Area	Single Cell Mining C
129554	McAree	Single Cell Mining Claim	2020-09-30	280274	Jordan	Single Cell Mining C
129555	McAree	Single Cell Mining Claim	2020-09-28	280275	Jordan	Single Cell Mining C
129557	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	280312	Webb	Single Cell Mining C
129564	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	280316	Jordan	Single Cell Mining C
129581	Jordan	Single Cell Mining Claim	2020-12-15	280349	McAree	Single Cell Mining C
129609	Webb	Single Cell Mining Claim	2021-01-24	280892	Drayton	Single Cell Mining C
129612	Jordan	Single Cell Mining Claim	2020-12-15	280953	Jordan	Single Cell Mining C
129646	Laval, McAree	Single Cell Mining Claim	2021-01-24	280954	Jordan	Single Cell Mining C
129691	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	280986	Jordan	Single Cell Mining C
130020	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	281620	Pickerel	Single Cell Mining C
130021	Jordan	Single Cell Mining Claim	2020-12-15	281682	McAree	Single Cell Mining C
130296	Laval	Single Cell Mining Claim	2020-09-30	282013	Echo	Single Cell Mining C
130305	Laval	Single Cell Mining Claim	2020-09-30	282227	Pickerel	Single Cell Mining C
130309	Laval	Single Cell Mining Claim	2021-01-24	282233	Laval	Single Cell Mining C
130712	Jordan	Single Cell Mining Claim	2021-04-20	282333	Jordan	Single Cell Mining C
130981	Laval	Single Cell Mining Claim	2020-09-30	282334	Jordan	Single Cell Mining C
130982	Laval	Single Cell Mining Claim	2020-09-30	282335	Jordan	Single Cell Mining C
130983	Laval	Single Cell Mining Claim	2020-12-04	282916	Laval	Single Cell Mining C
131407	Jordan	Single Cell Mining Claim	2020-12-15	282917	Echo	Single Cell Mining C
131408	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	282918	Echo	Single Cell Mining C
134204	Echo	Single Cell Mining Claim	2021-01-13	282919	Echo	Single Cell Mining C
135251	McAree	Single Cell Mining Claim	2020-09-28	283028	Jordan	Single Cell Mining C
135273	McAree	Single Cell Mining Claim	2020-09-28	283029	Jordan	Single Cell Mining C
136994	Laval, Webb	Single Cell Mining Claim	2021-01-24	283036	Echo	Single Cell Mining C
137949	Laval	Single Cell Mining Claim	2020-09-30	283040	Echo	Single Cell Mining C
137950	Laval	Single Cell Mining Claim	2020-09-30	283055	Kabik Lake Area	Single Cell Mining C
137951	Laval	Single Cell Mining Claim	2020-09-30	283056	Kabik Lake Area	Single Cell Mining C
137952	Laval	Single Cell Mining Claim	2020-09-30	283057	Kabik Lake Area	Single Cell Mining C
138858	Laval	Single Cell Mining Claim	2020-08-05	283612	Kabik Lake Area	Single Cell Mining C
138905	Laval	Single Cell Mining Claim	2021-02-12	283617	Jordan	Single Cell Mining C
139221	Jordan	Single Cell Mining Claim	2021-03-28	283631	Jordan	Single Cell Mining C
139598	Laval	Single Cell Mining Claim	2020-09-30	283644	Echo	Single Cell Mining C
141432	Pickerel	Single Cell Mining Claim	2021-01-13	283709	Echo	Single Cell Mining C
141433	Pickerel	Single Cell Mining Claim	2021-01-13	283743	Laval, McAree	Single Cell Mining C
141435	Echo	Single Cell Mining Claim	2020-09-28	283744	Laval	Single Cell Mining C
141436	Echo	Single Cell Mining Claim	2020-09-28	283745	Laval	Single Cell Mining C
141714	Laval	Single Cell Mining Claim	2021-01-24	284329	Pickerel	Single Cell Mining C
142420	Laval	Single Cell Mining Claim	2021-02-12	284331	Echo, McAree	Single Cell Mining C
142682	Echo	Single Cell Mining Claim	2020-08-11	284945	Kabik Lake Area	Single Cell Mining C
143033	Laval, Webb	Single Cell Mining Claim	2021-01-24	286226	Pickerel	Single Cell Mining C
143456	Jordan	Single Cell Mining Claim	2020-12-15	286228	Echo, Webb	Single Cell Mining C
143464	Echo	Single Cell Mining Claim	2020-08-11	286229	Echo, Webb	Single Cell Mining C
143465	Echo	Single Cell Mining Claim	2021-02-12	286230	Echo	Single Cell Mining C
143466	Echo	Single Cell Mining Claim	2021-02-12	286231	Echo, Webb	Single Cell Mining C
143467	Echo	Single Cell Mining Claim	2021-02-12	286247	Echo	Single Cell Mining C
143468	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	287391	Laval	Single Cell Mining C
144756	Echo	Single Cell Mining Claim	2021-04-26	287483	Kabik Lake Area, Pickerel	Single Cell Mining C
144781	Echo	Single Cell Mining Claim	2021-01-13	287505	Kabik Lake Area, Pickerel	Single Cell Mining C
145341	Pickerel	Single Cell Mining Claim	2021-01-13	288853	Echo	Single Cell Mining C

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156838	Jordan	Single Cell Mining Claim	2020-12-15
156857	Echo	Single Cell Mining Claim	2021-04-26
157589	Pickerel	Single Cell Mining Claim	2021-01-13
157590	Pickerel	Single Cell Mining Claim	2021-01-13
157604	Pickerel	Single Cell Mining Claim	2021-01-31
158107	Pickerel	Single Cell Mining Claim	2021-01-13
158118	Jordan	Single Cell Mining Claim	2020-12-15
158119	Jordan	Single Cell Mining Claim	2020-12-15
158246	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
158789	Jordan	Single Cell Mining Claim	2020-12-15
158790	Jordan	Single Cell Mining Claim	2020-12-15
158795	Drayton	Single Cell Mining Claim	2020-12-15
158817	McAree	Single Cell Mining Claim	2020-09-28
158818	Echo, Pickerel	Single Cell Mining Claim	2020-08-11
158824	Jordan	Single Cell Mining Claim	2020-12-15
158828	McAree	Single Cell Mining Claim	2020-08-05
158829	McAree	Single Cell Mining Claim	2020-08-05
158849	Pickerel	Single Cell Mining Claim	2021-01-13
158854	Jordan	Single Cell Mining Claim	2020-12-15
158888	Webb	Single Cell Mining Claim	2021-01-24
158890	Jordan	Single Cell Mining Claim	2020-12-15
158891	Drayton, Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
159148	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
159469	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
159502	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
159503	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
159518	Jordan	Single Cell Mining Claim	2020-12-15
159528	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
159564	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
159595	McAree	Single Cell Mining Claim	2020-08-05
160125	Echo	Single Cell Mining Claim	2020-09-30
160149	Laval	Single Cell Mining Claim	2020-08-30
160166	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
160212	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
160256	Pickerel	Single Cell Mining Claim	2021-01-13
160257	Echo, Pickerel	Single Cell Mining Claim	2021-02-12
160265	Laval	Single Cell Mining Claim	2020-08-05
160271	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
160272	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
160273	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
160377	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
160381	Laval	Single Cell Mining Claim	2020-12-04
160382	Laval	Single Cell Mining Claim	2020-12-04
160816	Drayton	Single Cell Mining Claim	2020-12-15
160945	Echo	Single Cell Mining Claim	2021-02-09
161516	Laval	Single Cell Mining Claim	2021-01-24
161537	Laval	Single Cell Mining Claim	2020-09-30
161538	Laval	Single Cell Mining Claim	2020-09-30
161542	Jordan	Single Cell Mining Claim	2020-12-15
161549	Echo	Single Cell Mining Claim	2020-09-28
161561	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
161562	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
161563	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
161564	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
161616	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
161622	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28
161623	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
161630	Echo	Single Cell Mining Claim	2021-04-15
162268	McAree	Single Cell Mining Claim	2020-09-28
162269	McAree	Single Cell Mining Claim	2020-09-28
162853	Jordan	Single Cell Mining Claim	2020-12-15
162872	Echo	Single Cell Mining Claim	2021-04-26
163283	Jordan	Single Cell Mining Claim	2021-04-20
163585	Echo	Single Cell Mining Claim	2021-02-12
163586	Pickerel	Single Cell Mining Claim	2021-04-05

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292859	Laval	Single Cell Mining C
293016	Pickerel	Single Cell Mining C
293894	Laval	Single Cell Mining C
294169	Laval, McAree	Single Cell Mining C
294208	Echo	Single Cell Mining C
294705	Laval	Single Cell Mining C
294927	Echo, McAree	Single Cell Mining C
295593	Jordan	Single Cell Mining C
295611	Echo, Webb	Single Cell Mining C
295612	Echo	Single Cell Mining C
295613	Echo	Single Cell Mining C
296322	Echo	Single Cell Mining C
296323	Echo	Single Cell Mining C
296324	Echo	Single Cell Mining C
296325	Echo	Single Cell Mining C
296326	Echo	Single Cell Mining C
296327	Kabik Lake Area, Pickerel	Single Cell Mining C
296871	Pickerel	Single Cell Mining C
296872	Pickerel	Single Cell Mining C
296875	Kabik Lake Area, Pickerel	Single Cell Mining C
296880	Jordan, Kabik Lake Area	Single Cell Mining C
296881	Kabik Lake Area	Single Cell Mining C
296887	Jordan	Single Cell Mining C
296888	Jordan	Single Cell Mining C
296892	Kabik Lake Area, Pickerel	Single Cell Mining C
296984	Pickerel	Single Cell Mining C
296991	Pickerel	Single Cell Mining C
297229	Jordan, Kabik Lake Area	Single Cell Mining C
297230	Jordan, Kabik Lake Area	Single Cell Mining C
297358	Laval	Single Cell Mining C
297528	Jordan	Single Cell Mining C
297551	McAree	Single Cell Mining C
297552	McAree	Single Cell Mining C
297553	McAree	Single Cell Mining C
297554	Echo, Pickerel	Single Cell Mining C
297586	Pickerel	Single Cell Mining C
297623	Drayton, Jordan	Single Cell Mining C
297624	Drayton, Jordan	Single Cell Mining C
297625	Jordan	Single Cell Mining C
298199	Drayton, Parnes Lake Area	Single Cell Mining C
298252	Jordan	Single Cell Mining C
298253	Jordan, Kabik Lake Area	Single Cell Mining C
298288	Jordan	Single Cell Mining C
298294	Parnes Lake Area	Single Cell Mining C
298327	McAree	Single Cell Mining C
298335	Pickerel, Vermilion	Single Cell Mining C
298646	Laval	Single Cell Mining C
298909	Echo	Single Cell Mining C
298910	Echo	Single Cell Mining C
301437	Laval	Single Cell Mining C
304062	Pickerel	Single Cell Mining C
304063	Pickerel	Single Cell Mining C
304064	Pickerel	Single Cell Mining C
304247	Laval	Single Cell Mining C
304248	Laval	Single Cell Mining C
304390	Laval	Single Cell Mining C
305016	Webb	Single Cell Mining C
305303	Laval	Single Cell Mining C
306006	Laval	Single Cell Mining C
309427	Jordan	Single Cell Mining C
309568	Laval	Single Cell Mining C
309569	Laval	Single Cell Mining C
310275	Laval	Single Cell Mining C
310998	Laval	Single Cell Mining C
311329	Pickerel	Single Cell Mining C

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type
166943	Echo	Single Cell Mining Claim	2020-09-28	320955	Laval	Single Cell Mining C
167515	Drayton	Boundary Cell Mining Claim	2020-12-15	320968	Webb	Single Cell Mining C
167528	Jordan	Single Cell Mining Claim	2020-12-15	321013	Kabik Lake Area	Single Cell Mining C
167529	Jordan	Single Cell Mining Claim	2020-12-15	321047	Webb	Single Cell Mining C
167534	Laval	Single Cell Mining Claim	2020-09-30	321574	Parnes Lake Area	Single Cell Mining C
167546	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	322261	Pickerel	Single Cell Mining C
167556	Echo	Single Cell Mining Claim	2021-02-09	322262	Pickerel	Single Cell Mining C
167557	Echo	Single Cell Mining Claim	2021-02-09	322263	Pickerel	Single Cell Mining C
167627	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	322335	Jordan	Single Cell Mining C
167663	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	322809	Pickerel	Single Cell Mining C
168213	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	322810	Pickerel	Single Cell Mining C
168240	Echo	Single Cell Mining Claim	2021-02-12	322814	Echo, Webb	Single Cell Mining C
168241	Echo	Single Cell Mining Claim	2021-02-12	322827	Kabik Lake Area, Pickerel	Single Cell Mining C
168271	Laval	Single Cell Mining Claim	2020-12-04	322828	Kabik Lake Area, Pickerel	Single Cell Mining C
168313	Echo, McAree	Single Cell Mining Claim	2020-09-28	322829	Kabik Lake Area, Pickerel	Single Cell Mining C
168355	Echo	Single Cell Mining Claim	2021-02-12	323553	Kabik Lake Area, Pickerel	Single Cell Mining C
168896	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	323554	Kabik Lake Area	Single Cell Mining C
168941	Laval	Single Cell Mining Claim	2020-09-30	323555	Kabik Lake Area, Pickerel	Single Cell Mining C
168953	Webb	Single Cell Mining Claim	2021-01-24	325434	Echo	Single Cell Mining C
168971	Jordan	Single Cell Mining Claim	2021-03-28	326101	Echo	Single Cell Mining C
168984	Echo, Laval, McAree, Webb	Single Cell Mining Claim	2020-09-28	326103	Pickerel	Single Cell Mining C
168993	Laval	Single Cell Mining Claim	2020-09-30	326136	Pickerel, Vermilion	Single Cell Mining C
169567	Laval, Webb	Single Cell Mining Claim	2020-09-30	326137	Pickerel	Single Cell Mining C
169766	Laval	Single Cell Mining Claim	2021-02-12	326145	Pickerel	Single Cell Mining C
170273	Pickerel	Single Cell Mining Claim	2021-01-13	326154	Jordan, Kabik Lake Area	Single Cell Mining C
170274	Pickerel	Single Cell Mining Claim	2021-01-13	326747	Echo	Single Cell Mining C
170339	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	326780	Pickerel, Vermilion	Single Cell Mining C
170770	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	326781	Pickerel	Single Cell Mining C
170772	Echo	Single Cell Mining Claim	2020-09-28	326816	Jordan	Single Cell Mining C
170784	Kabik Lake Area	Single Cell Mining Claim	2021-01-13	326819	Drayton, Parnes Lake Area	Boundary Cell Mining
170790	Echo	Single Cell Mining Claim	2025-09-17	326820	Drayton, Parnes Lake Area	Single Cell Mining C
170791	Echo	Single Cell Mining Claim	2025-09-17	326858	McAree	Single Cell Mining C
171510	Echo	Single Cell Mining Claim	2020-08-11	326859	McAree	Single Cell Mining C
171520	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	326860	Pickerel	Single Cell Mining C
171546	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	326865	Drayton, Jordan	Single Cell Mining C
171547	Pickerel	Single Cell Mining Claim	2021-01-13	326866	Jordan	Single Cell Mining C
173418	Laval	Single Cell Mining Claim	2020-08-05	326871	Kabik Lake Area	Single Cell Mining C
173419	Laval	Single Cell Mining Claim	2020-08-05	327420	Webb	Single Cell Mining C
173634	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	327421	Webb	Single Cell Mining C
174143	Laval	Single Cell Mining Claim	2020-09-30	327422	Webb	Single Cell Mining C
174817	Laval	Single Cell Mining Claim	2020-09-30	327424	Drayton, Jordan, Parnes Lake Area	Single Cell Mining C
175970	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20	327425	Jordan, Parnes Lake Area	Single Cell Mining C
176109	Laval	Single Cell Mining Claim	2020-09-30	327536	Kabik Lake Area	Single Cell Mining C
176110	Laval	Single Cell Mining Claim	2020-09-30	328084	Parnes Lake Area	Single Cell Mining C
176113	Laval	Single Cell Mining Claim	2021-01-24	328158	Parnes Lake Area	Single Cell Mining C
176801	Laval	Single Cell Mining Claim	2020-12-04	328170	Laval	Single Cell Mining C
177364	Laval	Single Cell Mining Claim	2020-09-30	328192	Echo	Single Cell Mining C
177626	Jordan	Single Cell Mining Claim	2020-12-15	328757	McAree	Single Cell Mining C
177654	Jordan	Single Cell Mining Claim	2020-12-15	328800	Echo, Pickerel	Single Cell Mining C
177658	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	328810	Laval	Single Cell Mining C
177659	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	328814	Kabik Lake Area	Single Cell Mining C
177671	Echo	Single Cell Mining Claim	2020-09-30	328839	Drayton, Parnes Lake Area	Single Cell Mining C
177673	Pickerel	Single Cell Mining Claim	2021-01-13	328878	Drayton	Single Cell Mining C
177674	Pickerel	Single Cell Mining Claim	2021-01-13	328976	Parnes Lake Area	Single Cell Mining C
177679	Jordan	Single Cell Mining Claim	2020-12-15	328977	Parnes Lake Area	Single Cell Mining C
177717	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	328978	Parnes Lake Area	Single Cell Mining C
178320	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	328983	Laval	Single Cell Mining C
178364	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	329507	Laval	Single Cell Mining C
178365	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	329511	Echo	Single Cell Mining C
178394	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	329586	Laval, Webb	Single Cell Mining C
178408	Jordan	Single Cell Mining Claim	2021-04-20	329587	Laval, Webb	Single Cell Mining C
178416	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	329588	Laval, Webb	Single Cell Mining C
178982	Laval	Single Cell Mining Claim	2020-08-30	330115	Jordan	Single Cell Mining C
179008	Echo	Single Cell Mining Claim	2020-08-11	330117	Pickerel	Single Cell Mining C
179069	McAree	Single Cell Mining Claim	2020-08-05	330118	Pickerel	Single Cell Mining C

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182377	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	340961	Echo	Single Cell Mining C
186191	Echo	Single Cell Mining Claim	2021-01-13	341341	Laval, Webb	Single Cell Mining C
187731	Laval	Single Cell Mining Claim	2021-01-24	341349	Laval	Single Cell Mining C
187732	Laval	Single Cell Mining Claim	2021-01-24	341372	Echo	Single Cell Mining C
188977	Laval	Single Cell Mining Claim	2021-02-12	341923	Laval	Single Cell Mining C
189616	Laval, Webb	Single Cell Mining Claim	2021-01-24	341949	Jordan	Single Cell Mining C
189979	Laval	Single Cell Mining Claim	2020-09-30	341954	Echo	Single Cell Mining C
190830	Laval	Single Cell Mining Claim	2020-08-05	341963	Kabik Lake Area	Single Cell Mining C
191676	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	341994	Kabik Lake Area	Single Cell Mining C
192278	Laval	Single Cell Mining Claim	2020-09-30	341995	Kabik Lake Area	Single Cell Mining C
193322	Jordan	Single Cell Mining Claim	2021-04-20	342031	Kabik Lake Area	Single Cell Mining C
193323	Jordan	Single Cell Mining Claim	2021-04-20	342045	Jordan	Single Cell Mining C
193324	Jordan	Single Cell Mining Claim	2021-04-20	342209	McAree	Single Cell Mining C
193567	Laval	Single Cell Mining Claim	2021-01-24	342410	Laval	Single Cell Mining C
193568	Laval	Single Cell Mining Claim	2021-01-24	342426	Laval	Single Cell Mining C
194214	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	342427	Laval	Single Cell Mining C
194256	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	342553	Kabik Lake Area	Single Cell Mining C
194280	Jordan	Single Cell Mining Claim	2020-12-15	342620	Echo, Pickerel	Single Cell Mining C
194292	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	343233	McAree	Single Cell Mining C
194316	Laval	Single Cell Mining Claim	2020-09-30	343240	Echo	Single Cell Mining C
194317	Laval	Single Cell Mining Claim	2020-09-30	343310	Kabik Lake Area	Single Cell Mining C
194318	Laval	Single Cell Mining Claim	2020-09-30	343345	Laval	Single Cell Mining C
194818	Jordan	Single Cell Mining Claim	2020-12-15	343375	Jordan	Single Cell Mining C
194819	Jordan	Single Cell Mining Claim	2020-12-15	343924	Kabik Lake Area	Single Cell Mining C
194820	Jordan	Single Cell Mining Claim	2021-04-20	343964	Webb	Single Cell Mining C
194825	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	343965	Laval	Single Cell Mining C
194872	McAree	Single Cell Mining Claim	2020-08-05	344643	Pickerel	Single Cell Mining C
194873	McAree	Single Cell Mining Claim	2020-08-05	345434	Laval	Single Cell Mining C
194923	Laval	Single Cell Mining Claim	2020-08-30	545974	Kabik Lake Area	Multi-cell Mining Cl
195114	Laval	Single Cell Mining Claim	2020-09-30			
195115	Laval	Single Cell Mining Claim	2021-01-24	100003	Pickerel	Single Cell Mining C
195116	Laval	Single Cell Mining Claim	2021-01-24	100005	Echo	Single Cell Mining C
195528	Pickerel	Single Cell Mining Claim	2021-01-13	100282	Echo	Single Cell Mining C
195529	Pickerel	Single Cell Mining Claim	2021-02-12	100468	Pickerel	Single Cell Mining C
195532	Laval	Single Cell Mining Claim	2020-09-30	100570	Pickerel	Single Cell Mining C
195533	Laval	Single Cell Mining Claim	2020-09-30	100571	Kabik Lake Area, Pickerel	Single Cell Mining C
195534	Laval	Single Cell Mining Claim	2020-09-30	100832	Kabik Lake Area	Single Cell Mining C
195543	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	100834	Drayton	Single Cell Mining C
195584	Drayton	Single Cell Mining Claim	2020-12-15	100866	Kabik Lake Area	Single Cell Mining C
196208	Laval	Single Cell Mining Claim	2020-09-30	100892	Jordan	Single Cell Mining C
196209	Laval	Single Cell Mining Claim	2020-09-30	100893	Jordan, Kabik Lake Area	Single Cell Mining C
196210	Laval	Single Cell Mining Claim	2020-09-30	100896	Jordan, Kabik Lake Area	Single Cell Mining C
196211	Laval	Single Cell Mining Claim	2020-09-30	100936	Jordan	Single Cell Mining C
196269	Laval	Single Cell Mining Claim	2020-09-30	100937	Jordan	Single Cell Mining C
196270	Drayton	Boundary Cell Mining Claim	2020-12-15	100948	Parnes Lake Area	Single Cell Mining C
196280	Jordan	Single Cell Mining Claim	2020-12-15	101003	Jordan, Kabik Lake Area	Single Cell Mining C
196283	Pickerel	Single Cell Mining Claim	2021-01-13	101027	Kabik Lake Area, Pickerel	Single Cell Mining C
196298	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	101080	Echo, Pickerel	Single Cell Mining C
196301	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	101102	Echo	Single Cell Mining C
196307	Echo	Single Cell Mining Claim	2021-02-09	101103	Echo	Single Cell Mining C
196308	Echo	Single Cell Mining Claim	2021-02-09	101126	Laval	Single Cell Mining C
196309	Echo	Single Cell Mining Claim	2021-02-09	101127	Laval	Single Cell Mining C
196319	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101246	Echo	Single Cell Mining C
196320	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101268	Kabik Lake Area	Single Cell Mining C
196833	Webb	Single Cell Mining Claim	2021-01-24	101332	McAree	Single Cell Mining C
196858	Laval	Single Cell Mining Claim	2020-09-30	101336	Pickerel, Vermilion	Single Cell Mining C
196861	Laval	Single Cell Mining Claim	2020-09-30	101359	Echo, McAree	Single Cell Mining C
196862	Laval	Single Cell Mining Claim	2020-09-30	101380	Laval	Single Cell Mining C
197506	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101407	Echo	Single Cell Mining C
197507	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101408	Echo	Single Cell Mining C
197558	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	101498	Echo, Pickerel	Single Cell Mining C
197567	Echo	Single Cell Mining Claim	2025-04-15	101593	Jordan	Single Cell Mining C
197572	Jordan	Single Cell Mining Claim	2020-12-15	101676	Laval	Single Cell Mining C
197583	Echo	Single Cell Mining Claim	2021-04-05	101738	Laval	Single Cell Mining C
197662	Laval, McAree	Single Cell Mining Claim	2020-08-05	101760	Jordan	Single Cell Mining C
198227	Echo, McAree	Single Cell Mining Claim	2020-09-28	101761	Pickerel	Single Cell Mining C

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204915	Pickerel	Single Cell Mining Claim	2021-01-13	107264	Laval	Single Cell Mining C
204943	Echo	Single Cell Mining Claim	2021-02-09	109467	Laval	Single Cell Mining C
204952	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	111935	Kabik Lake Area	Single Cell Mining C
204989	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	114918	McAree	Single Cell Mining C
204990	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	114971	McAree	Single Cell Mining C
205005	Echo	Single Cell Mining Claim	2025-04-15	115046	Jordan	Single Cell Mining C
205023	Echo	Single Cell Mining Claim	2020-09-28	115070	Echo	Single Cell Mining C
205578	Echo	Single Cell Mining Claim	2021-01-13	115091	Echo	Single Cell Mining C
205579	Echo	Single Cell Mining Claim	2021-01-13	115111	Echo	Single Cell Mining C
205612	McAree	Single Cell Mining Claim	2020-08-05	115600	Pickerel	Single Cell Mining C
205613	Laval, McAree	Single Cell Mining Claim	2020-08-05	115601	Pickerel	Single Cell Mining C
205614	Laval	Single Cell Mining Claim	2020-09-30	115831	Kabik Lake Area	Single Cell Mining C
205615	Laval	Single Cell Mining Claim	2020-12-04	115859	Kabik Lake Area, Pickerel	Single Cell Mining C
205616	Laval	Single Cell Mining Claim	2020-12-04	115860	Pickerel	Single Cell Mining C
205662	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	116038	Jordan	Single Cell Mining C
206222	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	116042	McAree	Single Cell Mining C
206267	Laval	Single Cell Mining Claim	2021-02-12	116049	Kabik Lake Area	Single Cell Mining C
206273	Webb	Single Cell Mining Claim	2021-01-24	116050	Kabik Lake Area	Single Cell Mining C
206284	Jordan	Single Cell Mining Claim	2021-03-28	116105	Webb	Single Cell Mining C
206290	Vermilion	Single Cell Mining Claim	2021-01-13	116169	Laval	Single Cell Mining C
206298	Laval	Single Cell Mining Claim	2020-09-30	116171	Echo	Single Cell Mining C
206299	Laval	Single Cell Mining Claim	2020-09-30	116254	Laval	Single Cell Mining C
206319	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	116267	Kabik Lake Area	Single Cell Mining C
207561	Pickerel	Single Cell Mining Claim	2021-01-13	116268	Kabik Lake Area	Single Cell Mining C
208797	Echo	Single Cell Mining Claim	2020-08-11	116272	Jordan	Single Cell Mining C
208798	Echo	Single Cell Mining Claim	2021-01-13	116278	Echo	Single Cell Mining C
208816	Pickerel	Single Cell Mining Claim	2021-04-05	116279	Echo	Single Cell Mining C
208840	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116344	Kabik Lake Area	Single Cell Mining C
208841	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116350	Kabik Lake Area	Single Cell Mining C
208842	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116368	Echo, Pickerel	Single Cell Mining C
209121	Laval	Single Cell Mining Claim	2021-01-24	116404	Echo, Pickerel	Single Cell Mining C
210118	McAree	Single Cell Mining Claim	2020-09-28	116443	Laval, McAree	Single Cell Mining C
210220	Jordan	Single Cell Mining Claim	2020-12-15	116444	Laval	Single Cell Mining C
210768	Echo	Single Cell Mining Claim	2021-01-13	116445	Laval	Single Cell Mining C
211032	Laval	Single Cell Mining Claim	2020-09-30	116448	Webb	Single Cell Mining C
211457	Echo	Single Cell Mining Claim	2021-02-12	116450	Jordan	Single Cell Mining C
211458	Echo	Single Cell Mining Claim	2026-08-02	116489	McAree	Single Cell Mining C
211494	Pickerel	Single Cell Mining Claim	2021-01-13	116490	Laval, McAree	Single Cell Mining C
211509	Pickerel	Single Cell Mining Claim	2021-01-13	116544	Jordan, Kabik Lake Area	Single Cell Mining C
211516	Pickerel	Single Cell Mining Claim	2021-01-13	116549	Parnes Lake Area	Single Cell Mining C
211527	Jordan	Single Cell Mining Claim	2020-12-15	116594	Jordan	Single Cell Mining C
211528	Jordan	Single Cell Mining Claim	2020-12-15	116596	Jordan	Single Cell Mining C
211534	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116620	Jordan	Single Cell Mining C
211535	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116623	Parnes Lake Area	Single Cell Mining C
212170	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116725	Echo	Single Cell Mining C
212171	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116791	Jordan, Kabik Lake Area	Single Cell Mining C
212201	Drayton	Single Cell Mining Claim	2020-12-15	116826	Pickerel	Single Cell Mining C
212231	Jordan	Single Cell Mining Claim	2020-12-15	116827	Pickerel	Single Cell Mining C
212241	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116912	Drayton	Single Cell Mining C
212301	Laval	Single Cell Mining Claim	2020-09-30	116937	Jordan	Single Cell Mining C
212759	Pickerel	Single Cell Mining Claim	2021-01-13	116938	Jordan	Single Cell Mining C
212760	Pickerel	Single Cell Mining Claim	2021-01-13	116939	Jordan	Single Cell Mining C
212761	Pickerel	Single Cell Mining Claim	2021-01-13	117089	Jordan	Single Cell Mining C
212764	Jordan	Single Cell Mining Claim	2020-12-15	117096	Echo	Single Cell Mining C
212803	Webb	Single Cell Mining Claim	2021-01-24	117097	Echo	Single Cell Mining C
212804	Webb	Single Cell Mining Claim	2021-01-24	117098	Echo	Single Cell Mining C
212875	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	117099	Kabik Lake Area, Pickerel	Single Cell Mining C
212876	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	117100	Kabik Lake Area, Pickerel	Single Cell Mining C
212877	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	117148	Pickerel, Vermilion	Single Cell Mining C
213428	Jordan	Single Cell Mining Claim	2020-12-15	117163	Pickerel	Single Cell Mining C
213459	Jordan	Single Cell Mining Claim	2020-12-15	117169	Pickerel	Single Cell Mining C
213507	McAree	Single Cell Mining Claim	2020-08-05	117170	Kabik Lake Area, Pickerel	Single Cell Mining C
213518	Pickerel	Single Cell Mining Claim	2021-01-13	117190	Jordan	Single Cell Mining C
213519	Pickerel	Single Cell Mining Claim	2021-01-13	117672	Kabik Lake Area	Single Cell Mining C
213570	Laval	Single Cell Mining Claim	2020-08-30	117676	Echo	Single Cell Mining C
214104	Pickerel	Single Cell Mining Claim	2021-04-05	117701	Pickerel	Single Cell Mining C

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215772	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	122327	Laval	Single Cell Mining C
216315	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	122329	Echo	Single Cell Mining C
216316	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	122331	Echo	Single Cell Mining C
216322	Jordan	Single Cell Mining Claim	2021-03-28	122403	Laval	Single Cell Mining C
216323	Jordan	Single Cell Mining Claim	2021-03-28	122431	Laval	Single Cell Mining C
216324	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	122448	Jordan	Single Cell Mining C
216340	Jordan	Single Cell Mining Claim	2020-12-15	123023	Kabik Lake Area	Single Cell Mining C
216347	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	123024	Kabik Lake Area	Single Cell Mining C
216357	Pickerel	Single Cell Mining Claim	2020-08-11	123025	Kabik Lake Area	Single Cell Mining C
216358	Pickerel	Single Cell Mining Claim	2020-08-11	123030	Jordan	Single Cell Mining C
216399	Echo	Single Cell Mining Claim	2021-01-13	123100	Echo	Single Cell Mining C
216400	Echo	Single Cell Mining Claim	2021-01-13	123145	Laval	Single Cell Mining C
216421	Echo	Single Cell Mining Claim	2021-02-12	123738	Echo	Single Cell Mining C
216459	McAree	Single Cell Mining Claim	2020-08-05	123826	Kabik Lake Area	Single Cell Mining C
216460	McAree	Single Cell Mining Claim	2020-08-05	123827	Laval	Single Cell Mining C
216461	Laval	Single Cell Mining Claim	2020-09-30	123828	Laval	Single Cell Mining C
216462	Laval	Single Cell Mining Claim	2020-12-04	124215	Laval	Single Cell Mining C
216463	Laval	Single Cell Mining Claim	2020-09-30	124385	Laval	Single Cell Mining C
217013	Echo, McAree	Single Cell Mining Claim	2020-09-28	124401	Kabik Lake Area	Single Cell Mining C
217014	McAree	Single Cell Mining Claim	2020-09-28	124402	Kabik Lake Area	Single Cell Mining C
217015	McAree	Single Cell Mining Claim	2020-09-28	124937	Pickerel	Single Cell Mining C
217046	Pickerel	Single Cell Mining Claim	2021-01-13	124938	Pickerel	Single Cell Mining C
217047	Pickerel	Single Cell Mining Claim	2021-01-13	124942	Echo, Webb	Single Cell Mining C
217049	Echo, McAree	Single Cell Mining Claim	2020-09-28	124943	Echo	Single Cell Mining C
217065	Echo	Single Cell Mining Claim	2021-02-12	125260	Laval	Single Cell Mining C
217091	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	125261	Laval	Single Cell Mining C
217135	Laval	Single Cell Mining Claim	2021-02-12	125687	Pickerel	Single Cell Mining C
217136	Laval	Single Cell Mining Claim	2021-02-12	126858	Laval	Single Cell Mining C
217656	Laval	Single Cell Mining Claim	2020-09-30	126884	Echo, McAree	Single Cell Mining C
217699	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	126885	McAree	Single Cell Mining C
217700	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	126961	McAree	Single Cell Mining C
217701	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	127543	Jordan	Single Cell Mining C
217748	Laval	Single Cell Mining Claim	2020-09-30	127544	Jordan	Single Cell Mining C
219031	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	127545	Jordan	Single Cell Mining C
219661	Pickerel	Single Cell Mining Claim	2021-01-13	127597	Echo	Single Cell Mining C
219662	Pickerel	Single Cell Mining Claim	2021-01-13	127598	Echo	Single Cell Mining C
219663	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	127599	Echo	Single Cell Mining C
219666	Echo	Single Cell Mining Claim	2020-09-28	128305	Pickerel	Single Cell Mining C
220907	Kabik Lake Area	Single Cell Mining Claim	2021-01-13	128306	Pickerel	Single Cell Mining C
220908	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	128335	Kabik Lake Area	Single Cell Mining C
221671	McAree	Single Cell Mining Claim	2020-09-28	128915	Jordan	Single Cell Mining C
222299	Echo	Single Cell Mining Claim	2021-04-26	128977	Pickerel	Single Cell Mining C
222300	Echo, Webb	Single Cell Mining Claim	2021-04-26	129011	Drayton	Single Cell Mining C
222301	Echo	Single Cell Mining Claim	2021-04-26	129012	Drayton	Single Cell Mining C
222327	Echo	Single Cell Mining Claim	2021-01-13	129508	Laval	Single Cell Mining C
222328	Echo	Single Cell Mining Claim	2021-01-13	129554	McAree	Single Cell Mining C
222992	Jordan	Single Cell Mining Claim	2020-12-15	129555	McAree	Single Cell Mining C
223234	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	129557	Drayton, Jordan	Single Cell Mining C
223564	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	129564	Kabik Lake Area	Single Cell Mining C
223565	Pickerel	Single Cell Mining Claim	2021-01-13	129581	Jordan	Single Cell Mining C
223569	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	129609	Webb	Single Cell Mining C
223570	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	129612	Jordan	Single Cell Mining C
223579	Jordan	Single Cell Mining Claim	2020-12-15	129646	Laval, McAree	Single Cell Mining C
223927	Laval	Single Cell Mining Claim	2020-09-30	129691	Kabik Lake Area	Single Cell Mining C
223928	Laval	Single Cell Mining Claim	2020-09-30	130020	Jordan, Parnes Lake Area	Single Cell Mining C
224215	Jordan	Single Cell Mining Claim	2020-12-15	130021	Jordan	Single Cell Mining C
224217	Drayton	Single Cell Mining Claim	2020-12-15	130296	Laval	Single Cell Mining C
224241	Echo	Single Cell Mining Claim	2020-08-11	130305	Laval	Single Cell Mining C
224242	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	130309	Laval	Single Cell Mining C
224243	Pickerel	Single Cell Mining Claim	2021-01-13	130712	Jordan	Single Cell Mining C
224244	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	130981	Laval	Single Cell Mining C
224248	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	130982	Laval	Single Cell Mining C
224259	Pickerel	Single Cell Mining Claim	2021-01-13	130983	Laval	Single Cell Mining C
224666	Laval	Single Cell Mining Claim	2020-09-30	131407	Jordan	Single Cell Mining C
224944	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	131408	Jordan, Parnes Lake Area	Single Cell Mining C
225523	McAree	Single Cell Mining Claim	2020-08-05	134204	Echo	Single Cell Mining C

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type
232188	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	145492	Jordan, Kabik Lake Area, Pickerel, Vermilion	Single Cell Mining C
232221	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	145493	Kabik Lake Area, Pickerel	Single Cell Mining C
232237	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20	145500	Kabik Lake Area	Single Cell Mining C
232238	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	148834	Laval	Single Cell Mining C
232239	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	148835	Laval	Single Cell Mining C
232240	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	150149	Echo	Single Cell Mining C
232271	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	151621	Echo	Single Cell Mining C
232272	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	151622	Echo	Single Cell Mining C
232312	McAree	Single Cell Mining Claim	2020-08-05	151623	Echo	Single Cell Mining C
232875	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	151646	Echo	Single Cell Mining C
232876	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	151670	Laval, McAree	Single Cell Mining C
232942	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	151671	McAree	Single Cell Mining C
232946	McAree	Single Cell Mining Claim	2020-08-05	151721	McAree	Single Cell Mining C
232977	Pickerel	Single Cell Mining Claim	2021-02-12	151742	Pickerel	Single Cell Mining C
232990	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	152294	Kabik Lake Area	Single Cell Mining C
233653	Laval	Single Cell Mining Claim	2020-09-30	152345	Webb	Single Cell Mining C
233658	Echo	Single Cell Mining Claim	2021-02-09	152356	Vermilion	Single Cell Mining C
233727	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	152357	Vermilion	Single Cell Mining C
233728	Laval	Single Cell Mining Claim	2020-12-04	152371	Vermilion	Single Cell Mining C
233983	Echo	Single Cell Mining Claim	2021-01-13	152375	Laval	Single Cell Mining C
234234	Laval, Webb	Single Cell Mining Claim	2021-01-24	152378	Laval	Single Cell Mining C
234235	Laval	Single Cell Mining Claim	2021-01-24	152403	Kabik Lake Area	Single Cell Mining C
234249	Laval	Single Cell Mining Claim	2020-09-30	153623	Pickerel	Single Cell Mining C
234250	Drayton	Boundary Cell Mining Claim	2020-12-15	153871	Laval	Single Cell Mining C
234267	Laval	Single Cell Mining Claim	2020-09-30	154210	Pickerel	Single Cell Mining C
234272	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	154232	Echo	Single Cell Mining C
234276	Jordan	Single Cell Mining Claim	2020-12-15	155481	Kabik Lake Area, Pickerel	Single Cell Mining C
234277	Jordan	Single Cell Mining Claim	2020-12-15	156254	McAree	Single Cell Mining C
234285	Echo	Single Cell Mining Claim	2021-02-09	156838	Jordan	Single Cell Mining C
234297	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	156857	Echo	Single Cell Mining C
234345	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	157589	Pickerel	Single Cell Mining C

4.2 Property Ownership

4.2.1 Goliath Property

Treasury Metals, a former subsidiary of Laramide Resources Ltd. (Laramide), was spun out of Laramide as a dividend to Laramide's shareholders. Treasury Metals was listed and began trading on the Toronto Stock Exchange (TSX) exchange on August 19, 2008 under the trade symbol "TML".

The Goliath property consists of two historic properties that were consolidated into one: the larger Thunder Lake property, purchased from Teck and Corona, and the Laramide property. The land acquisition agreements are described below.

4.2.1.1 Thunder Lake Property Acquisition Timeline

Laramide closed its purchase transaction of the Thunder Lake property as of October 2007 (Laramide Press Release: October 4, 2007). Laramide purchased, through its former wholly owned subsidiary, Divine Lake Exploration Corp. (now "Treasury Metals Inc."), 100% of Corona's (82%) and Teck's (18%) respective interests in the Thunder Lake property. On closing, Corona received from Laramide a cash consideration of $5 million and under the terms of the agreement Corona received from Laramide aggregate cash payments of $10 million and a 10% interest in Treasury Metals after it became a public company. Teck received cash consideration of approximately $1,137,299 at closing and received from Laramide aggregate cash payment of $2,274,598 and a 2.27% interest in Treasury Metals. The balance of consideration for the properties was payable as follows:

- cash payment of $6,137,229 – 60 days after the closing date
- cash payment of $6,137,229 – 120 days after the closing date
- 12.27% of the common shares of Treasury Metals issued and outstanding on completion of a transaction pursuant to which Treasury Metals becomes a public company.

Treasury Metals announced in a press release (August 26, 2008) that it had completed the final instalment of the purchase price to Corona and Teck pursuant to the purchase agreement. In accordance with the 2007 Purchase Agreement, Corona and Teck shall receive, for no additional consideration, that number of common shares sufficient for each of Corona and Teck to maintain their respective percentage interest in the Company of 10% and 2.27% until the Company receives aggregate proceeds from the insurance of common shares of $7.5 million. This threshold has been reached. Laramide and Treasury Metals have met all of the obligations to Teck and Corona.

4.2.1.2 Laramide Property

As part of the spin-out of Treasury Metals, Laramide transferred to Treasury Metals its Goliath property (herein referred to as the Laramide property) and certain of Laramide's other non-uranium assets. As of May 2010, Laramide held approximately 13.7% of the issued and outstanding Treasury Metals common shares. Treasury Metals owns 100% of the Laramide property subject to royalties as detailed in Section 4.6.

4.2.1.3 2009 Property Expansion

In 2009, the Goliath property was expanded from its original size through the combined staking and acquisition of 18 unpatented mining claims and the signing of an option agreement pursuant to which Treasury Metals has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson property) contiguous to the Goliath Project.

4.2.1.4 Unpatented Mining Claims

In 2009, the Company acquired and/or staked 18 additional unpatented mining claims (111 units) totalling 1,776 hectares. These 18 additional claims are located in the Hartman and Zealand townships.

4.2.1.5 2009 Brisson Property

On December 11, 2009, the Company entered into an option agreement to acquire a 100% interest in the mining rights (only) of certain patented lands (40.8711 ha) from Edward Henry Brisson (the Brisson property) located immediately west and contiguous to the Goliath Project. Under the terms of the agreement, the Company made option payments totalling $100,000 and issued common shares of the Company equal to $100,000 based on the market price of the date issue. The property purchase (surface rights) was completed on March 31, 2011.

4.2.1.6 2010-2011 Property Expansion & Dryden Tree Nursery area

In 2010 and 2011 the Goliath property was further expanded by (1) acquiring the Dryden Tree Nursery; (2) staking three unpatented mining claims; and (3) making a final option payment. These expansions are described below.

On November 5, 2010, the Company acquired a 100% interest in two private land parcels consisting of mineral and surface rights (PIN 42089-0066, 100.62 ha) and the surface rights (PIN 42089-0065, 26.20 ha) formerly known as the Dryden Tree Nursery. The Dryden Tree Nursery is situated immediately northwest and contiguous to the Goliath property and covers 126.82 ha.

In 2011, the Company staked three additional unpatented mining claims (20 units) totalling 320 ha in Hartman township.

On April 12, 2011, the Company completed the final payment on the option to purchase the LeClerc surface rights (only) patent (Parcel 34303, 16.59 ha) located immediately east of the Thunder Lake deposit within the Goliath Project area.

4.2.1.7 2014 Mining Leases

Effective October 1, 2014, 11 Treasury Metals unpatented mining claims were converted to three 21-year mining leases which expire on September 30, 2035.

Mining lease 109532 has mining and surface rights covering 131.523 ha in N1/2 Lot 4, Concession 4 and S1/2 Lot 4, Concession 5 of Zealand Township and comprises all of mining claims K1119541, 1119542, K1119547, K1119548, K1119549, K1119550, K1119559 and K1119560, being all that land and land under water.

Mining lease 109533 has mining rights only covering 65.559 ha in Lot 5, Concession 5 of Zealand Township and comprises all of mining claims K1145301 and K3017938, being all that land and land under water.

Mining lease 109534 has mining rights only covering 63.940 ha in Lot 7, Concession 4 of Zealand Township and comprises of all of mining claim K1145300, being all that land and land under water.

4.2.1.8 Application for Mining Leases (Application)

In 2019, Treasury Metals made a request for a lease on 38 mining claims (in the Zealand and Hartman townships. As of the date or this report, the leasing process was still in progress.

4.2.2 Goldlund-Miller Property

4.2.2.1 Tamaka

Thirty-six claim units totalling 576 ha, were optioned from an arm's-length vendor (the Vendors), through Goldlund Resources Inc. The terms of the agreement with the Vendors stated that Tamaka Gold Corporation (Tamaka) must spend $1 million by September 5, 2009 to earn a 100% interest in the claims subject to a 1% NSR. The $1 million commitment was fulfilled, and the title of the claims was transferred by the vendors to Goldlund Resources Inc. in 2009.

4.2.2.2 First Mining

On June 17, 2016, First Mining Gold Corp. (First Mining) announced the completion of the amalgamation with Tamaka. The amalgamation resulted in Tamaka becoming a wholly-owned subsidiary of First Mining. First Mining issued 92.5 million common shares of First Mining to the shareholders of Tamaka as part of the transaction.

4.2.2.3 Treasury Metals

On June 3, 2020, Treasury Metals announced it had entered into a definitive share purchase agreement with First Mining to acquire the Goldlund-Miller property through the acquisition of Tamaka. The mineral rights to the Goldlund-Miller property are held by Goldlund Resources Inc., a wholly-owned subsidiary of Tamaka. On August 7, 2020, the acquisition was completed whereby Treasury Metals acquired all of the issued and outstanding shares of Tamaka. Under the terms of the agreement, First Mining shall receive:

- 130 million common shares (Common Shares) of Treasury Metals (the Share Consideration).
- 35 million Common Share purchase warrants of Treasury Metals (the Warrants), with each Warrant entitling the holder thereof to purchase one Common Share at an exercise price of $0.50 for a period of 36 months following the closing of the Transaction (the Warrant Consideration).

- A 1.5% net smelter returns royalty covering all of the Goldlund claims (the Goldlund Royalty), with the option for Treasury Metals to buy-back 0.5% of the Goldlund Royalty for $5 million.
- A milestone cash payment of $5 million, with 50% payable upon receipt of a final and binding mining lease under the *Mining Act* (Ontario) to extract "ore" from an open pit mine at Goldlund, and the remaining 50% payable upon the extraction of 300,000 tonnes of "ore" from a mine at Goldlund.

4.3 Royalties & Encumbrances

4.3.1 Goliath Royalties & Encumbrances

The Goliath property is held 100% by Treasury Metals, subject to certain underlying royalties and payment obligations on 13 of the 21 land parcels, totalling approximately $103,500 per year (see Table 4.5 for details).

Treasury Metals also has an option agreement pursuant to which Treasury Metals has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson property – 40.8711 hectares) located immediately west and contiguous to the Goliath Project.

The option on one patented land parcel to earn in 100% as described for the Brisson property (Section 4.2.1.5) was completed in March 2011.

Table 4.5: Options & Royalty Obligations, Patented Land Parcels – Goliath Property

Party	Parcel ID	Advance Royalty (Per Year)	Due Date	Option Amount	NSR (%)	Comments
Lundmark[1]	41941	C$50,000 **	January 1	-	2.0	
Collins[1]	17395	-	-	-	2.0	
Sheridan[1]	21374	-	-	-	1.0	
Johnson[1]	15401	-	-	-	2.0	
Hudak[1]	21609	US$3,500 *	January 1	-	2.0	
Fraser[1]	15395	C$50,000	January 1	-	2.0	
Delk[2]	24724	-	-	-	2.5	
Davenport[2]	19088	-	-	-	2.0	
Jones[3]	41215	-	-	-	2.5	
Nemeth[2]	6556	-	-	-	2.0	
Sterling[4]	4822	-	-	-	2.0	
Medlee [4]	21553	-	-	-	2.5	
Schultz[4]	13492	-	-	-	2.0	Includes 3 patents
Brisson[5]	23R2434	-	-	-	-	
Total C$		$100,000				
Total US$		$3,500				

Notes: *subject to withholding tax. (1) Thunder Lake West; (2) Thunder Lake East; (3) Jones property; (4) Laramide property; (5) surface rights.

4.3.2 Goldlund Royalties & Encumbrances

Royalties pertaining to the Goldlund-Miller property as defined in this document are as follows:

- The Goldlund Mines Limited Royalty Agreement, dated December 10, 2003, consists of six patented claims as well as the three patented claims covered by the Mining Lease. Goldlund Mines will receive a 1% NSR on any ore mined above 50 m below the existing shaft collar as of the date of the agreement. Goldlund Resources is entitled to a right of first refusal in the event Goldlund Mines wishes to dispose of its interest in the NSR. Goldlund Resources has the right but not the obligation to purchase one-half of the NSR for $500,000 at any time within three years from the date of the royalty agreement. This right has now expired.
- The Rio Algom Limited Option Agreement, dated August 28, 2014, consists of 21 patented claims. Goldlund Resources will pay a 2.5% NSR and will have the right but not the obligation to purchase the NSR in its entirety for a one-time payment of $2.5 million with a 10-day notification of intent to exercise the purchase right. Goldlund Resources is entitled to a right of first refusal in the event that Rio Algom Limited wishes to sell the NSR.
- As part of the purchase agreement of Goldlund from First Mining, First Mining holds a 1.5% net smelter returns royalty covering all of the Goldlund claims (the "Goldlund Royalty"), with the option for Treasury to buy back 0.5% of the Goldlund Royalty for $5.0 million.

Royalties pertaining to areas outside the resource as defined in this document:

- The 1074127 Ontario Limited Agreement, dated October 18, 2011, consists of 13 mining claims located in the Patricia and Kenora Mining districts of the Province of Ontario. 1074127 Ontario Limited (the 'Vendor') retains a 2% NSR in accordance with industry practice on the sale of all minerals from the property. Goldlund Resources has the sole and exclusive option to purchase 100% of the 2% NSR at any time for the sum of $1.5 million and has a right of first refusal in the event that the Vendor wishes to dispose of its interest in the NSR.

4.4 Surface Rights

4.4.1 Goliath Surface Rights

Treasury Metals holds the surface rights on 10 patents, a portion of one additional patent (PAT-46017), six land parcels and the four mining leases on the Goliath property.

4.4.2 Goldlund-Miller Surface Rights

Treasury Metals holds the surface rights on the 27 patents and one mining lease on the Goldlund-Miller property. However, for the Licence of Occupation, only mineral rights have been granted.

4.5 Ontario Mineral Tenure

4.5.1 Mining Cell Claims

In Ontario, Crown lands were available to licensed prospectors for the purposes of mineral exploration prior to 2018. Traditional claim staking in Ontario (post and blazed lines) came to an end on January 8, 2018, and on April 10, 2018 the MNDM converted all existing ground or map-staked mining claims (legacy claims) into one or more cell claims or boundary claims as part of their new provincial grid system. A cell claim was created when one or more legacy mining claims in a cell were held by the same owner. A boundary claim was created when there were multiple legacy claims in cell held by different claim holders. The provincial grid is based on latitude/longitude, and is comprised of more than 5.2 million cells ranging in size from 17.7 ha in the north up to 24 ha in the south.

A mining claim remains valid provided the claim holder properly completes and files the assessment work as required by the *Mining Act,* and the Minister approves the assessment work. A claim holder is not required to complete any assessment work within the first year of recording a mineral claim. In order to keep an unpatented mining claim current, the claim holder must perform (a minimum) $400 worth of approved assessment work per mining claim unit, per year; immediately following the initial staking date, the claim holder has two years to file one year's worth of assessment work. Mining claims are forfeited if the assessment work is not completed.

A claim holder may prospect or carry out mineral exploration on the land under the claim. However, the land covered by these claims must be converted to leases before any development work or mining can be performed.

4.5.2 Mining Lease

Mining leases grant the owner title and ownership to the land and the ability to extract and sell extracted resources. The exact rights conferred under a mining lease vary depending upon the type of lease issued (either mining rights only, surface rights only or both mining and surface rights) and will usually be described in detail, including reservations, under the lease patent document executed by the Crown. Mining leases are granted for 21 years and may be renewed for a further 21 years if the application is made within 90 days of the expiry date. Mining Leases are maintained by an annual rental fee of $3.00/ha (*Mining Act*, Ontario Regulation 45/11).

Prior to bringing a mine into production, the lessee must comply with all applicable federal and provincial legislation.

4.5.3 Licence of Occupation

Prior to 1964, Mining Licences of Occupation (MLO) were issued, in perpetuity, by the MNDM to permit the mining of minerals under the beds of bodies of water. MLOs were associated with portions of mining claims overlying adjacent land. As an MLO is held separate and apart from the related mining claim, it must be transferred separately from the transfer of the related mining claim. The transfer of an MLO requires the prior written consent of the Ministry.

MLOs are maintained by an annual rental fee of $5.00/ha (*Mining Act*, Ontario Regulation 45/11).

4.5.4 Mining Patent

Mining patents are freehold mining claims that permit the patentee to all of the Crown's title to the subject lands and to all mines and minerals relating to such lands, unless something to the contrary is stated in the patent. A mining patent can include surface and mining rights or mining rights only. Since mining patents convey freehold interest in the land subject to the patent, no consents are required for the patentee to transfer or mortgage those lands.

Mining patents were granted to perpetuity provided the taxes on these lands are paid annually.

4.6 Permits

4.6.1 Goliath Permits

Treasury Metals warrants that it possesses all permits required to execute the exploration activities it has undertaken to date on the property.

4.6.2 Goldlund Permits

Treasury Metals warrants that it possesses all permits required to execute exploration activities on the Goldlund Project.

4.6.3 Goldlund Project First Nations Agreements

Treasury Metals, has entered into three agreements with two Indigenous communities in Ontario (from Tamaka), as described in the following subsections.

4.6.3.1 Lac Seul First Nation

On September 1, 2011, Tamaka entered into a negotiation protocol with the Ojibway of Lac Seul First Nation (LSFN). The negotiation protocol establishes a committee through which Tamaka and LSFN will negotiate exploration activities on certain lands in the District of Sioux over which the LSFN asserts traditional territory rights. Under the negotiation protocol, Tamaka must also consult with LSFN from time to time in regards to its exploration activities, as well as with respect to economic and business opportunities, environmental matters and training, employment and retention programs for LSFN members mutually beneficially to the Company and LSFN and the rights, if any, asserted by other First Nations over the subject area. As consideration for LSFN's consultations, advice and assistance, Tamaka shall pay to LSFN, in connection with each drillhole conducted by Tamaka, $200 per drillhole setup and $1.50/m of drilling, and a one-time payment of 71,433 units (each unit being one Tamaka share and one warrant with an agreed value of $1.05 per unit or $75,005 in the aggregate), which were issued on execution of the agreement. As a result of the Amalgamation, these units were converted into units of First Mining.

4.6.3.2 Wabigoon Lake Ojibway Nation

On September 13, 2011, Tamaka entered into a memorandum of understanding (MOU) with Wabigoon Lake Ojibway Nation (WLON) and a community relations services agreement with Wabigoon Lake Development Corporation (WLDC). The MOU governs the Company's conduct with respect to the exploration activities it undertakes in respect of the Goldlund Project on land over which WLON asserts traditional territory rights. Pursuant to the MOU, Tamaka must notify WLDC of anticipated exploration activities, provide certain training, employment and business opportunities to the WLDC and cover costs incurred in connection with the monthly meetings of a working group established under the MOU and any community meetings held in connection with the MOU. WLDC provides ongoing advisory and consultation services with respect to Tamaka's obligations under the MOU under the community relations services agreement. As consideration for WLDC's services, Tamaka shall pay to WLDC, in connection with each drillhole conducted by Tamaka, $200 per drillhole setup and $1.50/m of drilling, and a one-time payment of 71,433 units (each unit being one Tamaka share and one warrant with an agreed value of $1.05 per unit or $75,005 in the aggregate) which were issued on execution of the agreement. As a result of the Amalgamation, these units were converted into units of First Mining shares.

Both the negotiation protocol and MOU contemplate that formal exploration agreements will be entered into once the Goldlund Project is further advanced. Treasury Metals has not entered into any exploration agreements with the Indigenous communities at this time.

4.7 Environmental Liabilities

4.7.1 Goliath Property

There are no known environmental liabilities associated with the Goliath property, other than those normally expected due to historical exploration and mining activities, and associated historical mine workings.

It has been confirmed that all closure works associated with the former bulk sample workings conducted by Teck have been completed in accordance with the Mine Rehabilitation Code and the Mine Closure Plan. As detailed, all mine hazards observed on site have been addressed in the Closure Plan and the site is consistent with the Closure Plan. Rehabilitation is proceeding as per the Closure Plan and in accordance with Part VII of the *Mining Act*, O. Reg. 240/00, and the Mine Rehabilitation Code workings.

4.7.2 Goldlund-Miller Property

CGK and AGP are unaware of any environmental liabilities associated with the Goldlund-Miller property related to the historic operation that are the responsibility of Treasury Metals. CGK and AGP are unaware of any additional environmental liabilities or other factors and risks that may affect access, title, or ability that would prevent Treasury Metals from conducting exploration activities on the property.

The Goldlund Project has two historic shafts that have been capped, an underground portal that has been blocked, a small open pit that is partially flooded, a waste rock stockpile, a mineralised material stockpile, a building housing the original mill on the property, and a small tailing containment facility. All have been overgrown with vegetation.

Treasury Metals will continue to evaluate and work collaboratively with regulators to ensure that all aspects of historical workings and their long-term implications are addressed as part of the development of the Goldlund Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

The Goliath Project is located in the Kenora Mining Division in northwestern Ontario, approximately 4 km northwest of the Village of Wabigoon, 20 km east of Dryden and 2 km north of the Trans-Canada Highway 17. The Goldlund and Miller Projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath Project, off Highway 72. Aerial imagery of the Goliath Project and the Goldlund Project are provided in Figures 5-1 and 5-2, respectively.

Access to the Goliath Project is north from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Anderson and Tree Nursery Roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals. Access to the Goldlund site is east off Highway 72 via Goldlund Mine Road. The Miller Project site is accessed via forestry road east off Highway 72. Access roads for the Goldlund and Miller sites are maintained by the Sustainable Forest Licence Holder (Domtar) for the area.

Figure 5-1: Goliath Project Office

Source: Treasury Metals, (2021).

Figure 5-2: Goldlund Project Site

Source: Treasury Metals, (2021).

5.2 Climate

Located in the west-central portion of the Boreal Shield Ecozone, the Goliath Gold Complex area experiences a continental climate generally characterised by short, mild summers and long, cold winters with relatively low precipitation. The terrain is generally flat and absent of orographic features that can block air masses or produce localised increases in precipitation. Annual temperatures range from 27°C to -26°C with an average rainfall between 60 and 80 cm and average snowfall between 1.3 and 2.3 m.

5.3 Local Resources & Infrastructure

All major industrial services and supplies are available in Dryden and Sioux Lookout and the area is serviced by both the Dryden Airport and Sioux Lookout Airport. The Goliath Project is located 20 km from Dryden, which has a population of 5,586 according to the Statistics Canada 2016 census. The Goldlund and Miller projects are located 43 km and 35 km, respectively, south of Sioux Lookout, which has a population of 5,272. The Goliath Gold Complex is located about 300 km northwest of the City of Thunder Bay, a major economic centre along the Trans-Canada Highway and port at the northwest head of the St. Lawrence Seaway on Lake Superior.

The Complex is located in an area used by the public for recreational fishing, hunting, boating, and commercial activities, including tourism. Traditional land and resource use is also practiced by a number of Indigenous communities. The local economy is largely based on forestry and tourism.

Major and minor hydro transmission lines cross portions of the Goliath Project area. The Canadian Pacific Railway line is located approximately 2 km to the southwest, parallel to Highway 17. The Trans-Canada natural gas pipeline crosses portions of the Goliath property. The closest centre of active mining operations is in the Red Lake area, approximately 155 km northwest of the project; however, northwestern Ontario generally possesses the necessary labour and infrastructure to support new exploration and mining operations.

At this time, Treasury Metals holds the sufficient surface rights necessary for any potential future mining operations including tailings storage areas, waste disposal areas, and a processing plant.

5.4 Physiography

The area is typical of glaciated terrain of the Canadian Shield. The topography overall is gently rolling, with glaciated high points seldom exceeding 50 m above local lake levels. Elevations across the Goliath Gold Complex area are generally between 370 and 430 masl. The localised topography levels range from of 390 to 400 masl in the principal deposit area at the Goliath property, from 380 masl to upwards of 430 masl at the Goldlund property, and from 390 masl to 400 masl at the Miller property.

The Goldlund deposit area contains a number of glaciated bedrock intrusions opposed to the flat till of the Goliath area. Low ground is covered by deep glacial till and frequent small lakes and/or swamps.

The Complex is located within the Ontario Shield Ecozone, which is characterised by extensive wetlands and boreal forests. Typical tree species include trembling aspen (*Populus tremuloides*), balsam poplar (*Populus balsamifera*), white and black spruces (*Picea glauca*, *Picea marina*), white birch (*Betula papyrifera*) and willow (*Salix* spp.).

6 HISTORY

A portion of the text in the following section was extracted from the P&E 2019 report and edited here for content and readability.

6.1 Goliath Property

6.1.1 Pre-1989 Exploration

The first gold mining on record in the region was in Van Horne Township in the early 1900s with very limited gold production from auriferous veining in biotite schist within the regional Wabigoon fault system. Sporadic exploration was carried out along the belt throughout the 1900s with only limited documentation of exploration activity conducted on the property.

The earliest known government report covering the larger Dryden-Sioux Lookout Belt is the Ontario Department of Mines Report and Geology Map by Satterly (1941). Ministry of Northern Development and Mines (MNDM) geologist Gary Beakhouse has written a number of reports covering the geology of the region and the Western Superior Province (Beakhouse, 2003, 2002, 2001, 2000 and 1995). Reconnaissance lake sediment geochemistry and detailed airborne geophysical surveys are also available for Thunder Lake and surrounding areas (Hornbrook and Fisk 1989, and Ontario Geological Survey, 1987).

According to Page (1991), the first reference to exploration work conducted on the property describes an “interesting contact between amphibolite, laminated grey gneisses, and beds of mica-tourmaline schists on Sheridan Option legacy claim SV200”. There is no record of further work on the property until the mid-1950s.

In 1956-57, Compton-Wabigoon conducted geological mapping, magnetometer surveys, and the completion of two diamond drillholes totalling 458 m to explore the mineral potential of the major iron formation unit located in Lots 1-4, Concession V and VI, along the northern boundary of the property. Also in 1956, G. L. Pidgeon completed surface work and one shallow drillhole (drilled south) testing a sphalerite showing in the south half of Lot 6, Concession 4 (Fraser Option legacy claim 0134). The showing and drill collar was located in the field by Teck, but subsequent surface sampling of sphalerite-rich mineralisation did not return any significant gold values (best 10 ppb). Teck determined the drillhole attempted to test the showing down-dip on the mineralisation. This showing had been previously sampled by Satterly in 1941 with similar negative results (Page, 1991).

From 1966 to 1968, Algoma Steel Corp. Ltd. conducted geological mapping and drilled five holes totalling 304 m. This program was concentrated on the main iron formation focused in the same area as Compton-Wabigoon’s work 10 years earlier (Page, 1991). Inco completed ground surveys and one drillhole (52 m) in the vicinity of Teck grid coordinates L18E, 4+00E. Teck could not locate the drill site in the field and no assays were reported in the drill log; however, the hole is located within 50 m of a strong linear (>1,000 m) VLF-EM conductor which Teck believes was the probable drill target.

6.1.2 1989-1999 Teck, Corona & Laramide Exploration

The exploration history on the property is described in a number of technical reports prepared for Treasury Metals which is summarised below (Roy et al., 2012; Roy and Trinder, 2011; Roy and Trinder, 2008; Wetherup and Kelso, 2008).

Three major mining companies conducted exploration work on the Thunder Lake gold deposit (Goliath deposit) from 1989 to 1999 (last field work 1998): Teck, Corona, and Laramide. At that time, the property held by all three companies covered more than 1,300 ha. Teck held the majority of the property and all of the surface exposure.

Exploration and resource development work at Goliath was undertaken by Teck from 1989 to 1999 on what was then called the “Thunder Lake property”. During this period, the property was divided into two properties called “Thunder Lake East” and “Thunder Lake West”. The property was optioned to Corona, previously called Continental Caretech Corporation (CCC), by which CCC could earn an interest in the project under terms of an initial agreement dated January 3, 1994. Corona funded the exploration work from 1994 to 1999, but Teck remained the project operator both designing and running all field exploration activities.

The total exploration expenditures spent on the property from 1989 to 1999 by Teck and Teck-Corona was approximately $9.7 million (Page et al., 1999a; Page et al., 1999b; Page and Waqué, 1999; Page and Waqué, 1998).

6.1.2.1 Teck Exploration Work from 1989 to 1993

It was not until 1989 that reconnaissance exploration work by Teck, in search of Hemlo-type gold mineralisation in the region as part of their Quest Project, identified a large weakly altered felsic rock unit containing sporadic anomalous values in gold, silver, zinc, and lead extending through parts of Lots 3 through 8 of Concession 4 in Zealand Township. Grab assays averaging 2.98 g/t Au, 24.7 g/t Ag, 1.20% Zn, and 0.43% Pb were reported by Page (1991). Weakly altered quartz-eye felsic rock (muscovite-sericite schist unit?) returned an assay of 630 ppb Au. This discovery was followed by land acquisition and exploratory work by Teck.

The exploration program during that period consisted of establishing a 104.7 line-km exploration grid across the property, geological mapping, prospecting, sampling, and geophysical surveying consisting of ground magnetic, induced polarisation (IP) surveys and VLF-EM surveys. Eleven samples were submitted for petrographic analyses and one outcrop was stripped using a bulldozer (on line L15+80W, 2+25N).

A short, seven-hole diamond drill program was completed to test chargeability anomalies. It is during this program that the Goliath deposit (Main Zone) Hole TL1 was discovered by Teck in the fall of 1990, which prompted resource definition and exploration work on the property throughout the 1990s.

It was determined there was a positive correlation between gold content and the presence of sphalerite and galena, but the highest gold assays were generally associated with siliceous intervals containing only 1% to 3% zinc, and 0.1% to 1.5% lead.

The whole rock geochemistry indicated the felsic schists (muscovite-sericite schist) generally represented the altered equivalents of massive to gneissic felsic (volcanic?) rocks and are moderately enriched in silica and potassium, moderately to strongly depleted in sodium, and strongly depleted in calcium and magnesium.

Drilling programs were subsequently conducted in each of the next three years (1991, 1992, 1993) with the completion of an additional 49 drillholes focused on evaluating the resource potential of the main gold deposit.

In 1993, the property was optioned to Corona. Table 6.1 summarises the exploration activities conducted by Teck from 1990 to 1993.

Table 6.1: Teck Exploration Summary from 1990 to 1993

Year	Company & Work Locations	Work Completed
1990	Thunder Lake West	Reconnaissance exploration
	Thunder Lake West	Line cutting (104.7 line-km), mapping of exploration grid
	Thunder Lake West	Geological mapping and prospecting
	Thunder Lake West	122 grab and chip samples collected (32 sent for whole rock)
	Thunder Lake West	11 petrographic samples completed; one outcrop stripped
	Independent Exploration Services	Ground magnetic, VLF-EM survey (entire grid), 31.8 line-km IP
	SAGAX Geophysique Inc.	31.8 line-km of IP
	SAGAX Geophysique Inc.	Diamond drilling program – 7 holes (TL1 to TL7) TL1 Goliath discovery hole
1991	Thunder Lake West	Diamond drilling program – 17 holes (TL8 to TL24)
1992	Thunder Lake West	Diamond drilling program – 22 holes (TL25 to TL37)
1993	Thunder Lake West	Diamond drilling program – 10 holes (TC-1 to TC-10)
	Thunder Lake West	Property optioned to Corona (funding exploration)

6.1.2.2 Teck-Corona Exploration Work from 1994 to 1999

Exploration activities conducted from 1994 to 1999 consisted of seven diamond drilling programs, re-logging and sampling of previously drillholes, mechanical stripping (22 trenches), chip and channel sampling and mapping, geological mapping (1:5,000 scale), baseline environmental studies, underground development work, bulk sampling, metallurgical testing, site remediation work, custom mill testing, and mineral resource estimation(s) (see Table 6.2 for details).

A suite of ten lithogeochemical rock samples were collected in September 1995 on legacy claims 1106349 and 1106351 in the southwestern portion of the property. None of the rock samples were found to have been subjected to significant alteration as there was no evidence of sodium, potassium, or calcium enrichment or depletions and none contained any significant gold or base metal values.

In August 1996, some mechanical stripping and sampling was completed in the northern part of legacy claim K1106349 east of East Thunder Lake Road to expose the source of an IP anomaly identified by previous Teck ground geophysical surveys (Waqué, 1996). The new exposure was chipped, channel sampled, and geologically mapped. No significant gold mineralisation or alteration was identified from the sampling and mapping program.

Table 6.2: Teck-Corona Exploration Summary from 1994 to 1999

Year	Company & Work Locations	Work Completed
1994	Teck-Corona (Teck Operator)	Diamond drill program – 69 holes (TL44 to TL110, 5 wedges)
	Teck-Corona	Re-logging core of previous holes, 12 whole rock samples
	Teck-Corona	Re-examination of existing surface exposures
1995	Teck-Corona	Diamond drilling program – 25 holes (TL-111 to TL127, 8 wedges)
	Teck-Corona	Lithogeochemical survey (10 rock samples)
1996	Teck-Corona	Diamond drilling program – re-logging 3 holes + 51 new holes (TL128 to TL142, 13 wedges; TLE11 to TLE33)
	Teck-Corona	Resource estimate completed
	Teck-Corona	Mechanical stripping, chip and channel sampling, mapping
	Teck-Corona	August (1 outcrop area, legacy claim K1106349)
	Teck-Corona	Geological mapping (1:5,000), 22 trenches/sampling No. 1 shoot (Main Zone)
	Teck-Corona	No. 1 shoot - 200 kg bulk sample (preliminary metallurgical testing)
	Teck-Corona	Prepared first resource estimate
	Teck-Corona	Geochemical analyses of core and surface samples
1997	Teck-Corona	Diamond drilling program – 65 holes (TL143 to TL206, 1 wedge)
	Teck-Corona	Baseline environmental studies, updated the 1996 Resource Estimate
	Teck-Corona	Preliminary underground program (No.1 and No. 2 shoots) designed
1998	Teck-Corona	Diamond drilling program – 71 holes (TL207 to TL277)
	J.S. Redpath Limited	Underground development – ramp and drifting
	Lakefield Research Ltd., Stock Mine Mill	Exploration, face sampling, bulk sampling, metallurgical testing
	NAR Environmental Consultants	Portal remediation work
	NAR Environmental Consultants	Updated inferred resource estimate
	Corona Gold Corporation (Jones Lot)	Diamond drilling program – 12 holes (Main Zone)
1999	St. Andrews Goldfields for Teck	2,226 t bulk sample sent by Teck to stock mill – custom mill testing

Teck completed a program of geological mapping, trenching, channel sampling, and the completion of 6,596 m of diamond drilling from May 14, 1996 to November 4, 1996 (Stewart et al., 1997). This program was undertaken to better define the alteration corridor east of the resource area, to trench the Main Zone in the No. 1 shoot area to determine controls on the gold mineralisation and obtain a bulk sample, to drill test the Main Zone at depths below previous drilling, and to test footwall zones by deepening selected holes.

Geological mapping at a scale of 1:5000 was concentrated mainly in the eastern portion of the property and 15 of the existing trenches were re-examined and chip/channel sampled. Geological mapping and sampling identified new favourable target areas for gold mineralisation in the eastern half of the property. The geology of the area was re-interpreted, and the existing geology map was updated.

A trench located on grid line L8+50W was excavated exposing the bedrock over the Main Zone No. 1 shoot. The trench was mapped and a total of 48 channel samples and two chip samples were collected and analysed for gold and multi-elements. A bulk sample of approximately 200 kg was also blasted from the No. 1 shoot for preliminary metallurgical testing (Stewart et al., 1997).

A total of 115 samples from 60 drillholes were collected primarily from the Main Zone for geochemical analyses. Additional samples were also collected from surface outcrops enlarging the surface sample database to include 500 samples in total (Stewart et al., 1997). Overall, this work indicated that higher gold values correlate with increases in lead, zinc, silver, mercury, SiO_2, and SiO_2/Al_2O_3 concentrations in the Main Zone. It was also determined that zinc and lead concentrations decrease across the zone from west to east and that mercury is a good indicator to define the alteration corridor and that the alteration zone remained untested east of the deposit for an additional strike length of at least 2,800 m.

In 1997, a baseline environmental study (water, flora, and fauna) was commissioned by Teck and preliminary engineering plans and cost estimates for an underground program, including permitting, were completed. The environmental work was completed by NAR Environmental Consultants (Sudbury, Ontario). Initial baseline water quality and biological surveys were completed in 1997 and water sampling was continued in 1998 (Page et al., 1999b).

6.1.2.2.1 Underground Development & Bulk Sampling Program

In 1998 Teck completed an underground exploration and bulk sampling program at a cost of $1,929,071. This entire underground program, from surface site preparation through final closure plan, was completed between May 15 and September 15, 1998. This program was initiated for the following reasons (Page et al., 1999b; Emdin, 1998):

- to determine the nature and continuity of gold mineralisation in the Main Zone
- to obtain a bulk sample of the Main Zone mineralisation for gold and metallurgical analyses
- to determine what structures controlled the high-grade shoots within the Main Zone by geological mapping
- to establish the true grade of the gold mineralisation

The underground work contract was awarded to J. S. Redpath Limited of North Bay, Ontario. A 27 m long inclined trench provided a 9 m high outcrop face suitable for the construction of a portal collar. A decline was prepared at a grade of 15% with the portal located just north of Norman Road and the north boundary of the Laramide property (Figure 6-1). The decline was 4.0

m high by 4.5 m wide and approximately 275 m in length extending 25 m past the Main Zone mineralised structure (Roy et al., 2012). A total of 220 m of drifting (3.0 m by 3.0 m cross-section) was completed along the Main Zone (exposing shoots 1 and 2) extending both east and west of the decline at an approximate vertical depth of 35 m (-38 m floor elevation) for a total of 496 m of underground development. The lateral development followed units of altered schists with weak to strong sulphide mineralisation. A total of 23,035 tonnes of rock was excavated.

Figure 6-1: Historic Portal/Decline Development Access to Main Zone Gold Mineralisation of the Thunder Lake Gold Deposit

Source: Historic photo circa 1998. Supplied by Treasury Metals (2015).

Geological mapping was undertaken of all drift, slash faces, and backs. Chip sampling of all drift and slash faces was completed at two elevations (Page et al., 1999b). Muck and slash round samples were collected and analysed for gold.

Four bulk sample areas from the Main Zone (No. 1 and No. 2 shoots) totalling 2,375 tonnes were excavated consisting of blasted muck from drift rounds and slashed and material from a 400 tonne take-down-back (TDB) test mining area grading in excess of 3 g/t Au. The bulk sample was processed through a crushing plant, reduced in volume through a sampling tower, and representative splits were processed and analysed for gold content at Lakefield Research Ltd.

Teck concluded that in general, rock and alteration units defined from surface mapping and surface drilling were effective for the underground mapping program. The strongest gold mineralisation was found to be localised in siliceous quartz-sericite schists containing disseminated sulphides, sulphide veins, and sulphide-mineralised quartz veins with rare coarse gold/electrum. The more significant mineralised areas are in contact with units of dark-coloured intermediate quartz porphyry. While the general distribution of alteration and mineralisation outlined by surface drilling correlated reasonably well with the results of the underground program, Teck reported there was a marked decrease in both the strike length (50 to 65 m expected down to 22 m) and gold grade (15.2 g/t Au expected down to 9.05 g/t Au) of significant mineralisation. The grade of the bulk sample (2,336 tonnes @ 9.05 g/t Au) was found to be lower than what was calculated from face and muck samples. Both the grade and the tonnage of the bulk sample was lower than what was anticipated from surface drillhole information. Teck also commented that nugget effects, while present, did not significantly increase the grade of large tonnages of mineralisation.

AGP notes that the comparison between the anticipated grade and continuity was made against the 1997 resource which was estimated via a polygonal method (likely on a longitudinal section). Polygonal resource estimation was a common method used in the 1990s. Assuming this is correct, the expected strike length of the zone would have been driven solely by the spacing between the drill intercepts and the grade would be continuous up to the edge of the adjoining polygon where it would abruptly change to the grade of the next drillhole intercept. The disappointing results may just be a reflection of the resource estimation method used. The deposit was re-estimated in 1998 and included the underground bulk sampling and new drilling using an ordinary kriging method for grade interpolation.

After the underground work was completed, the portal was sealed and the area contoured, reseeded, and fully remediated in late 1999.

6.1.2.2.2 Custom Milling of Bulk Sample

A 2,355 tonne bulk sample was shipped to the St. Andrews Goldfields' mill near Timmins, Ontario for custom milling in the fall of 1999 (Jobin-Bevans, 2007). The custom milling sample returned average recoveries of 5.63 g/t Au and 15.28 g/t Ag as calculated by St. Andrew Goldfields. The gold recovery was calculated at 96.83% and silver at 38.0%. According to Jobin-Bevans (2007), there was some disagreement as to the total recovery reported by St. Andrew Goldfields and at that time, assays of the mill feed were being reviewed by the Corona-Teck Joint Venture. Initial evaluation of the mill feed samples by an independent umpire laboratory apparently indicated the number of ounces would increase. The resolution of this dispute remains unknown at this time. The reader is directed to Section 13 for further details regarding this custom milling program.

6.1.2.2.3 Completion of Exploration Program by Teck/Corona

Work on the project was suspended by the end of 1999, largely due to the gold grade and tonnage being lower than expected when compared to the resource estimate, and also due to a downturn in the mining industry when gold prices dropped below US$300/oz.

The property was put on care and maintenance until economic circumstances changed to justify additional work to upgrade the inferred gold resource to possible minable reserve categories (Page et al., 1999a). Table 6.2 above summarises the Teck-Corona exploration activities during that period.

6.1.3 Laramide Resources Ltd. Exploration

The mineralised gold zone dipping 70° to 80° south, as established by Teck/Corona, was projected to extend onto the northern part of the Laramide property at an approximate depth of 800 m below surface.

During 1994, the historic Laramide property (then consisting of parcels 4822 and 21553 covering an area of 109.5 ha south of the Goliath deposit) was geologically mapped and a ground magnetic/IP survey was completed. Teck/Corona’s work had already established zones associated with gold mineralisation on their property were responsive to IP survey methods.

These exploration activities have been described in detail by Hogg (2002, 1996). To facilitate this work, a north-south exploration grid was cut with a baseline established along Norman Road (formally Nelson Road) and north-south oriented gridlines were cut at a line spacing of 100 m. The baseline was established along the same road used for Teck’s baseline.

The near-surface ground geophysical survey completed by Rayan Exploration Ltd. identified three zones of high to moderate chargeability, as follows:

- northern property boundary anomaly
- eastern property anomaly, 250 m south of the baseline
- southern anomaly located approximately 400 m south of the baseline

In 1996, nine trenches and ten pits were excavated, and some surface sampling was completed. Trench No. 2 and trench No. 4 exposed weakly mineralised zones hosted in biotite schist. In trench No. 2, a narrow zone of quartz veined and pyritised biotite schist returned 480 ppb Au.

A graphitic shear identified at the contact between biotite schist and mafic volcanic rocks was mapped in trench No. 8 explaining the high IP chargeability anomaly that extends across the property 400 m south of the baseline. Eight diamond drillholes were also completed; seven of these holes being collared along the north boundary of the property.

According to Hogg (2002), the exploration work indicated that the degree of silicification and frequency of occurrence of gold mineralisation on the property increased to the north. However, no economically significant gold grades were reported.

In June 2002, Laramide acquired a third parcel of land (13492) covering 57 ha to the south, giving them a contiguous land package totalling 166.5 ha in Zealand Township. During the following period of depressed gold prices, no further work was carried out, although the option agreements were kept in place and claims maintained in good standing. The Teck property was later acquired by Laramide in which Treasury Metals was originally a subsidiary company until becoming its own publicly listed company on the TSX on August 19, 2008.

A summary of exploration activities on the Laramide property is provided in Table 6.3.

Table 6.3: Laramide Property Exploration Summary

Year	Company & Work Locations	Work Completed
1994	Laramide Resources Ltd.	Exploration Grid, Geological Mapping
	Laramide Resources Ltd.	Ground Geophysics (Magnetic/IP)
1996	Laramide Resources Ltd.	9 Trenches and 10 pits (mapping and sampling)
	Laramide Resources Ltd.	Diamond Drilling – 8 holes (G1 to G8) testing the Main Zone at depth

6.1.4 Historical Drilling

6.1.4.1 Teck-Corona Drilling (1990-1999)

AGP notes that some of the historical work described above is still relevant today, since results from the Teck-Corona drilling between 1990 and 1999 support a good portion of the mineral resource estimate described in Section 14 of this report. Information on this historical drilling is described in Section 10 and analytical procedures are described in Section 11.

6.1.4.2 Laramide Resources Ltd. (Laramide Property)

Eight exploratory diamond drillholes totalling 1,622 m were completed on the Laramide property in October 1996 (Hogg, 2002). These NQ holes, numbered G-1 to G-8, were all drilled due north (grid north) at a collar inclination of -45° (see Table 6.4). Holes G-1 to G-6 were drilled on land parcel 4822, Treasury Metals patented claims PA3900 and PA8429. Drillholes G-7 and G-8 were collared on land parcel 21553, Treasury Metals patented claim PA9074. All holes were drilled on patented land acquired by Laramide in 1996 with seven of the holes collared along the north boundary of the property.

These holes tested the depth extension of the Thunder Lake gold deposit (Goliath deposit) at vertical depths ranging from 105 to 223 m from surface and were collared both south of the deposit and south of Norman Road where the exploration base line had been established.

According to Hogg (2008), some narrow intersections of biotite schist (BMS?) and felsic tuff (MSS?) were reported to contain anomalous gold and silver values. Hole G-2 returned the best intersection of 675 ppb Au over a core length of 6.0 m. Anomalous gold values were also reported from the same horizon of silicified biotite schist for Holes G-1 and G-3 located 100 m to the east and west of Hole G-2.

Table 6.4: Laramide Diamond Drilling Summary

Drill Program	Year	Holes	Dates Drilled	Hole Numbers	Metres Drilled
1	1998	8	October 1998	G-1 to G-8	1,622
Total		**8**			**1,622**

Source: Treasury Metals (2015).

Hole G-5 was collared further south to test a moderate to high chargeability ground IP anomaly. A weakly pyritised biotite schist containing possible graphitic mineralisation was interpreted to be the source of the geophysical anomaly.

6.1.5 Exploration Activity from 1999 to 2008

There was no exploration activity on the property between the end of 1999 and the commencement of Treasury Metals exploration program in 2008.

6.1.6 Historical Mineral Resource Estimates

The mineral resource estimate described in this section are now considered historical in nature. They are provided here for historical context only. Treasury Metals is not treating these historical estimates as current mineral resources or reserves and the Qualified Person has not undertaken any independent investigation of the resource estimates; therefore, the resources described below should not be relied upon. These historical resource estimates are no longer current and have been superseded by the resource estimate described in Section 14 of this report.

Three historical gold resource estimates were reported on the Thunder Lake gold deposit from 1996 to 1998 using the results from surface and annual exploration diamond drilling programs (see Table 6.5).

Table 6.5: Historical Mineral Resource Estimate by Teck-Corona

Year	Gold (oz)	"Inferred" Historical Resource Estimate
1996	854,000	3.65 Mt grading 7.28 g/t Au
1997	853,000	3.78 Mt grading 7.02 g/t Au
1998	618,700	2.974 Mt grading 6.47 g/t Au

Note: Resources are based on a cut-off grade of 3.0 g/t Au and minimum thickness of 3.0 m. Source: Wetherup and Kelso (2008).

According to Stewart (1996), all of the drilling completed to the end of February 1996 was used to prepare a preliminary inferred resource estimate of the deposit totalling 2.8 Mt averaging 9.13 g/t Au for a total of 822,000 oz Au (non-N.I. 43-101-compliant resource estimate). This resource was estimated based on 56 diamond drillholes and one wedge hole covering a strike length of 1,000 m of the deposit to a vertical depth of 500 m using a minimum horizontal thickness of 3.0 m and block cut-off grade of 3.0 g/t Au.

At the completion of the 1996 drilling campaign, an inferred resource estimate of 3.65 Mt grading 7.28 g/t Au for a total of 854,000 oz Au was estimated (see Table 6.5). In 1997, a new inferred resource estimate was completed based on diamond drilling at 25 m spacing's totalling 3.78 Mt grading 7.02 g/t Au for a total of 853,000 oz Au, as follows (Wetherup et al., 2007):

- Main Zone: 2.87 Mt, 744,000 oz Au, at 2.87 g/t Au
- C Zone: 0.91 Mt, 109,000 oz Au, at 3.75 g/t Au

According to Wetherup and Kelso (2008), these resource estimates were carried out using the polygonal method (polygons obtained by half-distances between drillholes) and were based on a cut-off grade of 3.0 g/t Au, a specific gravity of 2.7 gm/cm^3, and a minimum thickness of 3.0 m.

A final resource estimate was prepared based on all diamond drilling and surface work, including underground bulk sampling and drilling, completed to 1998 (see Table 6.5). This estimate included 678 underground samples and 219 diamond drillholes from within the resource area (Wetherup et al., 2007). This resource was estimated using computer generated threedimensional (3D) solid models of the Main Zone and C Zone muscovite-sericite-schist (MSS) units using blocks measuring 3.0 m (thickness) x 10.0 m (height) x 10.0 m (strike length) and using the ordinary kriging method for grade interpolation.

The new inferred resource estimate prepared by Teck geologists in 1998 was 2.974 Mt grading at 6.47 g/t Au (approximately 618,700 oz Au). According to Wetherup and Kelso (2007), this estimate included 2.95 Mt of 6.52 g/t Au present in the Main Zone and 49 kt grading 3.71 g/t Au in the C Zone.

Since 2008, a number of resource estimates were completed on the Goliath deposit by various consultants. These conform to the CIM best practice guidelines in effect at the time the resources were completed. Table 6.6 summarises these historical estimates along with the Teck-Corona estimates that have now been superseded by the resource estimate discussed in Section 14 of this report.

Table 6.6: Summary of Historical Resource Estimate

Company	Year	Cut-off	Measured			Indicated			Inferred			Estimation Method
			Tonnes (kt)	Au g/t	Ounces (koz)	Tonnes (kt)	Au g/t	Ounces (koz)	Tonnes (kt)	Au g/t	Ounces (koz)	
Teck-Corona	1996	3.0 g/t Au							3650	7.25	854	Polygon
Teck-Corona	1997	3.0 g/t Au							3780	7.02	853	Polygon
Teck-Corona	1998	3.0 g/t Au							2974	6.47	619	OK
A.C.A Howe International	2008	3.0 g/t Au				560	5.9	110	3,300	5.9	625	OK
A.C.A Howe International	2012	0.3 g/t Au (OP)				6,002	1.8	326	11,093	1.0	352	OK
		1.5 g/t Au (UG)				3,136	4.3	433	4,789	3.3	514	
P&E Mining Consultants	2015	0.35 g/t AuEq (OP)	1,015	1.90	62	17,174	1.22	676	1,315	1.0	43	ID^3
		1.9 g/t AuEq (UG)	103	7.32	24	2,264	4.84	352	2,120	4.2	287	
P&E Mining Consultants	2019	0.40 g/t AuEq (OP)	762	1.91	47	11,849	1.37	522	595	1.1	20	ID^3
		1.9 g/t AuEq (UG)	163	6.42	34	3,429	5.34	589	1,414	4.4	201	

Notes: (OP) = amenable to open pit extraction, (UG) = amenable to underground extraction, OK = ordinary kriging, ID^3 = inverse distance cubed.

6.2 Goldlund-Miller Property

This section has been summarised based on previous technical reports, including the 2020 Treasury Metals Technical Report.

6.2.1 Ownership

The ownership history of the Goldlund Project is complex, dating back to the 1940s. Table 6.7 shows a summary of the past ownership and the exploration and development work completed by the various companies on the property.

Table 6.7: Summary of Past Exploration & Development Work on the Property

Year	Company	Geology	Geophysics	Trenching	Surface Sampling	Diamond Drilling	Underground Development
1941-47	Lundward Gold Mines Ltd.					X	
1945, 47	Windward Gold Mines Ltd					X	
1950	Conecho Mines Ltd.					X	
1946-50	East Lund Gold Mines		X			X	X
1951-52	Newland Mines Limited						X
1971	Windfall Oil & Mines	X				X	
1976-80	Goldlund Mines Ltd.					X	
1980	Windfall Oils & Mines						
1984	Goldlund Mines Ltd.					X	
1987	Camreco Inc.		X	X	X	X	
1988	Camreco Inc.					X	X
1991-92	Noranda Exploration Ltd	X	X	X		X	
1992	Camreco Inc.						
2003	Atikwa			X	X		
2003	Quartz Crystal Dryden Inc.			X	X		
2007	Tamaka Holdings					X	
2011	Tamaka Gold	X	X		X	X	
2012	Tamaka Gold			X			
2013	Tamaka Gold					X	
2017	First Mining					X	
2018	First Mining				X	X	

Source: CGK (2020) based on July 2020 drillhole database.

6.2.2 Exploration

Exploration activities on the Goldlund Project date from the 1940s, where in 1941 A. Ward and R. Lundmark (two prospectors working for the Mosher group) discovered gold mineralisation in the southwestern part of Echo Township (Page, 1984). From 1946 to 1952 there were significant exploration activities carried out on the Newlund Mines Limited and Windward Gold Mines prospects. The Newlund prospect was extensively explored by 4,570 m of underground drifts and crosscuts on four levels (200 ft, 350 ft, 500 ft, and 800 ft), and 6,220 m of core drilling from a 255 m deep vertical shaft. The 200 ft level on the Newlund prospect was extended more than 3.2 km to the west to connect with the 68 m vertical shaft on the Windward prospect, crossing the entire Windward claim block (Page, 1984). From 1952 to 1973, there was only limited exploration activities carried out on the Echo Township gold prospects.

In 1974, Goldlund Mines Limited and Rayrock Mines Limited entered into an agreement and rehabilitated the surface facilities including the installation of a new headframe and hoist and dewatering the underground workings to the second level (350 ft). A program of bulk sampling, underground chip sampling, and core drilling of 41 holes totalling 4,932 ft (approximately 1,500 m) was carried out. No further activities were carried out, as the prospect was deemed uneconomic given the gold price at that time (Page, 1984).

In total, approximately 143,825 m of drilling has been completed in 808 surface drillholes, and approximately 18,624 m of drilling has been completed in 480 underground holes. Table 6.8 shows a summary of the surface drilling and Table 6.9 shows a summary of the underground drilling.

Table 6.8: Summary of Past Surface Drilling on the Project

Year	Company	No. Holes	Amount (ft)	Amount (m)
1941	Lunward Gold Mines Ltd.	5	1,504	459
1942	Lunward Gold Mines Ltd.	27	6,812	2,076
1945	Lunward Gold Mines Ltd.	45	3,629	1,106
1946	Lunward Gold Mines Ltd.	81	30,175	9,197
1947	Lunward Gold Mines Ltd.	10	3,776	1,151
1947	Windward Gold Mines	18	8,294	2,528
1950	Conecho Mines	15	10,020	3,054
1950	North Denison Mines	1	894	273
1976	Goldlund Mines Limited	11	4,045	1,233
1976	Selco Mining Corp	1	410	125
1977	Goldlund Mines Limited	3	922	281
1979	Goldlund Mines Limited	70	12,785	3,897
1980	Goldlund Mines Limited	21	3,780	1,152
1980	Windfall Oils and Mines	46	20,814	6,344
1982	Donald Wilkonson	1	499	152
1983	Goldlund Mines Limited	4	541	165
1984	Goldlund Mines Limited	25	12,139	3700
1987	Camreco Inc. (GML)	24	23,720	7,230
1988	Camreco Inc. (GML)	62	23,960	7,303
1989	Camreco Inc. (GML)	33	3,087	941
1991	Noranda Exploration Co Ltd	3	719	219
2007	Tamaka Holdings	43	33,077	10,082
2008	Tamaka Gold	66	62,917	19,177
2011	Tamaka Gold	31	41,936	12,782
2013	Tamaka Gold	14	17,075	5,205
2014	Tamaka Gold	10	12,457	3,797
2017	First Mining Gold Corp.	124	116,428	35,487
2018	First Mining Gold Corp.	14	15,456	4,711
Total		**808**	**471,871**	**143,826**

Source: CGK (2020) based on July 2020 drillhole database.

Table 6.9: Summary of Past Underground Drilling on the Project

Year	Company	Level (ft)	No. Holes	Amount (ft)	Amount (m)
1950	Newlund Mines Limited	200	40	6,175	1,882
1951	Newlund Mines Limited	200	8	1,686	514
1951	Windward Gold Mines	200	10	1,824	556
1952	Newlund Mines Limited	200	20	2,274	693
1952	Windward Gold Mines	200	6	1,024	312
1973	Rayrock Mines Ltd. (NEWL)	200	22	2,149	655
1979	Goldlund Mines Ltd.	200	91	11,415	3,479
1980	Goldlund Mines Ltd.	200	78	9,035	2754
1951	Newlund Mines Limited	350	15	2,103	641
1952	Newlund Mines Limited	350	3	197	60
1973	Rayrock Mines Ltd. (NEWL)	350	19	2,782	848
1980	Goldlund Mines Ltd.	350	58	6,952	2,119
1951	Newlund Mines Limited	500	20	2,441	744
1952	Newlund Mines Limited	500	13	1,125	343
1980	Goldlund Mines Ltd.	500	44	6,132	1,869
1952	Newlund Mines Limited	800	33	3,789	1,155
			480	61,104	18,624

Source: CGK (2020) based on July 2020 drillhole database.

In addition to drilling, Tamaka carried out a trenching program in 2012 that included the excavation, stripping, mapping, channel sampling and a detailed structural analysis. The structural analysis was carried out by Mr. N. Pettigrew of Fladgate Exploration Consulting Services (Pettigrew, 2012). In total, 13 trenches were excavated covering approximately 7,733.35 m^2 and a total of 1,601 channel samples were collected and submitted for assay.

Table 6.10 presents a summary compilation of the historical exploration activities conducted by various companies on the remaining portions of the Project, outside of the immediate Goldlund deposit area.

Table 6.10: Summary Compilation of Historical Work on the Property, outside of Goldlund Deposit

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Beartrack	Laval	1950	Graham Bousquet Gold Mines	Diamond drilling (12 holes - 366 m)	Bousquet North
Beartrack	Laval	1970	Canadian Nickel Company	Diamond drilling (1 hole - 56 m)	-
Beartrack	Laval	1977	Hollinger Mines	Geological mapping	-
Beartrack	Laval	1978	Hollinger Mines	Magnetic and EM surveys	-
Beartrack	Laval	1978	Selco Mining	Diamond drilling (1 hole - 73 m)	-
Beartrack	Laval	1985	Mistango Consolidated Resources	Airborne magnetic and VLF- EM surveys	-
Beartrack	Laval	1987	Camreco Inc.	Magnetic and VLF survey	-
Beartrack	Laval	1989	Robert J. Service	Trenching	Bousquet South
Beartrack	Laval	1990	A Glatz	Magnetic survey	Bousquet South
Beartrack	Laval	1991	Champion Bear Resources	Geological mapping, trenching, magnetic and VLF surveys	-
Beartrack	Laval	1992	Champion Bear Resources	Magnetic and VLF survey	-
Beartrack	Laval	1992	Champion Bear Resources	Diamond drilling (11 holes - 1,129 m)	Bousquet South
Beartrack	Laval	1996	Corona Gold	Magnetic and VLF survey	-
Beartrack	Laval	1997	Corona Gold	Diamond drilling (12 holes - 3,158 m)	Bousquet South & North
Franciscan	Echo	1950	El Pen Rey Mines	Diamond drilling (3 holes - 415 m)	El Pen Rey
Franciscan	Echo	1950	North Denison Mines	Diamond drilling (3 holes - 824 m)	El Pen Rey
Franciscan	Echo	1973	Goldlund Mines	Diamond drilling (3 holes - 110 m)	El Pen Rey
Franciscan	Echo	1979	Goldlund Mines	Diamond drilling (1 hole - 42 m)	Tarbush
Franciscan	Echo	1980	Goldlund Mines	Magnetic survey and diamond drilling (3 holes - 188 m)	Tarbush
Franciscan	Echo	1981	Goldlund Mines	Diamond drilling (2 holes - 196 m)	Tarbush
Franciscan	Echo	1984	Loydex Resources	Geological mapping	-
Franciscan	Echo	1987	Norad Resources	Magnetic survey	-
Franciscan	Echo	1988	Norad Resources	EM survey, Geological sampling	El Pen Rey
Franciscan	Echo	1995	Tri Origin Exploration	Geological mapping and prospecting	-
Franciscan	Echo	1996	Tri Origin Exploration	Magnetic survey and diamond drilling (8 holes - 1,353 m)	-

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Franciscan	Echo	1997	Tri Origin Exploration	Trenching and soil survey	-
Franciscan	Pickerel	1952	Kenwell Oil & Mines	Geological mapping and prospecting	-
Franciscan	Pickerel	1980	Cadre Corporation	Geological review	-
Franciscan	Pickerel	1982	Tarbush Lode Mining	Magnetic survey and diamond drilling (8 holes - 660 m)	Tarbush
Goldlund	Echo	1945	Lundward Gold Mines	Diamond drilling (12 holes - no drill logs available)	Goldlund
Goldlund	Echo	1947	Lundward Gold Mines	Diamond drilling (38 holes - 4,863 m)	Goldlund
Goldlund	Echo	1950	East Lund Gold Mines	Diamond drilling (2 holes - 38 m)	-
Goldlund	Echo	1950	Glenecho Mines	Diamond drilling (1 hole - 294 m)	-
Goldlund	Echo	1953	McCombe Mining & Exploration	Diamond drilling (1 hole - 109 m)	-
Goldlund	Echo	1970	Dryden Project	Diamond drilling (1 hole - 86 m) - assayed for Cu - Ni	-
Goldlund	Echo	1980	Goldlund Mines	Magnetic survey	-
Goldlund	Echo	1983	Tarbush Lode Mining	Diamond drilling (3 holes - 396 m)	-
Goldlund	Echo	1976-1979	Goldlund Mines	Diamond drilling (5 holes - 484 m)	Not Much
Goldlund	McAree	1950	Conwest Exploration	Diamond drilling (4 holes - 699 m)	Tablerock
Goldlund	McAree	1950	Porcupine Peninsular Gold Mines	Diamond drilling (8 holes - 1,718 m)	-
Goldlund	McAree	1951	Orlac Red Lake Mines	Magnetic survey	-
Goldlund	McAree	1951	Pacemaker Petroleum	Magnetic survey	-
Goldlund	McAree	1976	Donald Wilkinson	Diamond drilling (1 hole - 151 m)	-
Goldlund	McAree	1980	Tarbush Lode Mining	Diamond drilling	Tablerock
Goldlund	McAree	1981	Sulpetro Minerals	Magnetic and horizontal loop electromagnetic field (HLEM) survey	-
Goldlund	McAree	1982	Tarbush Lode Mining	Diamond drilling (3 holes - 425 m)	Tablerock
Goldlund	McAree	1982	Tarbush Lode Mining	Diamond drilling (4 holes - 370 m)	-
Goldlund	McAree	1985	Tarbush Lode Mining	Airborne magnetic and VLF- EM surveys	-
Goldlund	McAree	1988	Norad Resources	Magnetic survey	-
Goldlund	McAree	1988	Norad Resources	Geological sampling	-

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Goldlund	McAree	1988	Norad Resources	EM survey	-
Goldlund	McAree	1989	Norad Resources	Geological sampling	-
Goldlund	McAree	1991	Noranda Exploration Co Ltd.	Diamond drilling (3 holes - 201 m)	
Goldlund	McAree	2001	Tamaka Gold	Diamond drilling (27 holes - 10,667 m)	-
Goldlund	McAree	2007	Tamaka Gold	Diamond drilling (43 holes - 10,242 m)	-
Goldlund	McAree	2008	Tamaka Gold	Diamond drilling (66 holes - 18,974 m)	-
Goldlund	McAree	2013	Tamaka Gold	Diamond drilling (24 holes - 9,001 m)	-
Goldlund	McAree	2017	First Mining Gold	Diamond drilling (100 holes - 24,299 m)	
Laval	Laval	1952	Eclund Gold Mines	Diamond drilling (6 holes - 269 m)	-
Laval	Laval	1952	Floregold Red Lake Mines	Diamond drilling (2 holes - 292 m)	-
Laval	Laval	1956	Canadian Pacific Railway Company	Prospecting	-
Laval	Laval	1970	Canadian Nickel Company	Diamond drilling (2 holes - 292 m)	Troutfly
Laval	Laval	1972	Canadian Nickel Company	Diamond drilling (1 hole - 152 m)	-
Laval	Laval	1984	Mistango Consolidated Resources	Magnetic survey	-
Laval	Laval	1985	Mistango Consolidated Resources	Airborne magnetic and VLF- EM surveys	-
Laval	Laval	1986	Mistango Consolidated Resources	Diamond drilling (4 holes - 449 m)	Troutfly
Laval	Laval	1987	Camreco Inc.	Magnetic and VLF survey	-
Laval	Laval	1987	Mistango Consolidated Resources	Trenching, magnetic survey and diamond drilling (8 holes - 759 m)	Troutfly
Laval	Laval	1989	Camreco Inc.	Soil survey	-
Laval	Laval	1996	Corona Gold	Geological mapping and prospecting	-
Laval	Laval	1997	Corona Gold	Magnetic and VLF survey	-
Laval	Laval	1998	Corona Gold	Diamond drilling (40 holes - 3,826 m)	Troutfly
Laval	Laval	????	Amant Gold Mines	Diamond drilling (4 holes - 269 m)	-
Laval	McAree	1950	Porcupine Peninsular Gold Mines	Diamond drilling (2 holes - 389 m)	-
Miles	Pickerel	1950	Conwest Exploration	Geological mapping, trenching and diamond drilling (5 holes - 950 m)	Nova & Scotia
Miles	Pickerel	1950	Macho River Gold Mines	Line cutting - geological mapping	-
Miles	Pickerel	1951	Lake Fortune Gold Mines	Resistivity survey	-

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Miles	Pickerel	1981	Nahanni Mines	Diamond drilling (2 holes - 349 m)	Scotia
Miles	Pickerel	1983	Tarbush Lode Mining	VLF-EM survey and soil sampling	-
Miles	Pickerel	1984	Tarbush Lode Mining	Outcrop stripping and magnetic survey	Miles
Miles	Pickerel	1985	Tarbush Lode Mining	Outcrop stripping and diamond drilling (7 holes - 620 m)	Eaglelund
Miles	Pickerel	1985	Tarbush Lode Mining	Airborne magnetic and VLF- EM surveys	-
Miles	Pickerel	1996	Nufort Resources	Diamond drilling (2 holes - 397 m)	Scotia
Miles	Pickerel	1947-1948	Clinger Gold Mines	Line cutting, magnetic survey and geological mapping	-
Quyta	Pickerel	1950	Eagle Lund Mines & Gold Eagle Mines	Geological mapping and diamond drilling (9 holes - 707 m)	Eaglelund
Quyta	Pickerel	1950	Batch River Gold Mines	Diamond drilling (4 holes - 309 m)	Batch River
Quyta	Pickerel	1976	Albert Carruthers	Diamond drilling (3 holes - 116 m)	-
Quyta	Pickerel	1980	Nahanni Mines	Geological mapping	-
Quyta	Pickerel	1981	Nahanni Mines	Diamond drilling (10 holes - 1,930 m)	Quyta
Quyta	Pickerel	1982	Nahanni Mines	Geological mapping	-
Quyta	Pickerel	1985	Nahanni Mines	Magnetic survey	-
Quyta	Pickerel	1988	Concentrated Rare Earth Minerals	Geological mapping, electro- magnetic (EM) survey, magnetic survey	-
Quyta	Pickerel	1990	Nahanni Mines	Very low frequency- electro- magnetic (VLF-EM) and magnetic surveys	-
Quyta	Pickerel	1992	Nufort Resources	Line cutting and geological mapping	-
Quyta	Pickerel	1996	Nufort Resources	Diamond drilling (5 holes - 950 m)	Quyta
Quyta	Pickerel	1997	D. Brown & T. Darling	Prospecting and geological mapping	-
Quyta	Pickerel	1998	D. Brown & T. Darling	Prospecting and geological mapping	-

Source: WSP (2019).

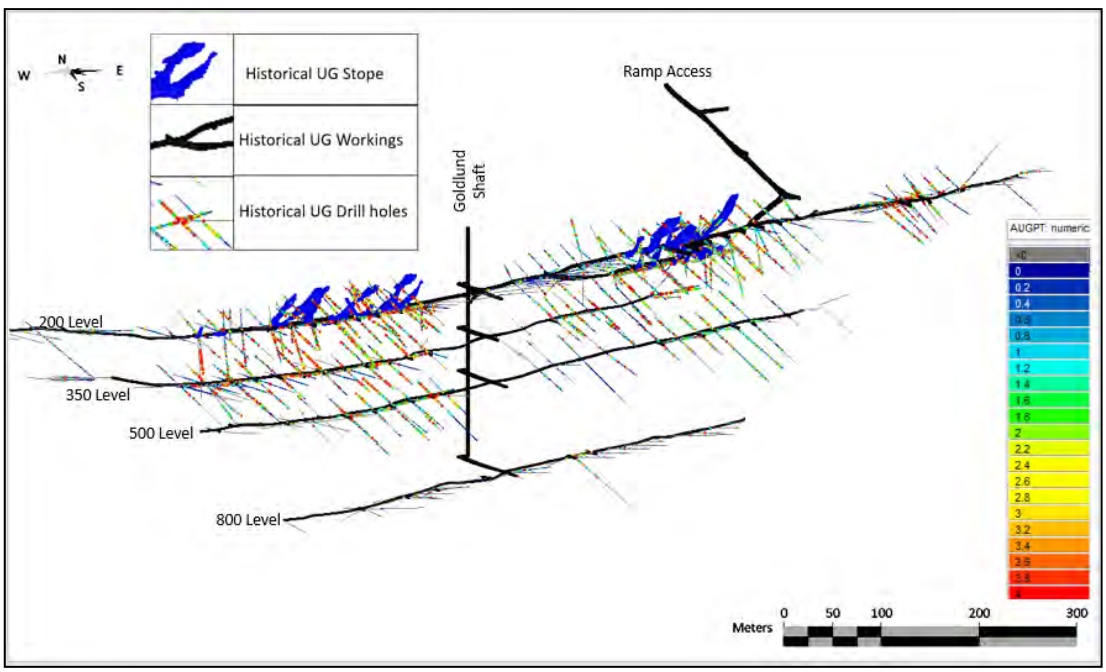
6.2.3 Historical Production

From mid 1982 to early 1985, Campbell Resources Inc. (Campbell Chibougamau), through its wholly owned subsidiary Goldlund Mines Limited, operated an underground mine and an open pit mine and processed material through the mill at the site. Pieterse (2005) compiled the production records that show underground mine production of 100,000 tons (approximately 90,700 t) at an estimated grade of 0.15 oz/ton Au (approximately 5.14 g/t Au) and open pit production of 43,000 tons (approximately 39,000 t) at an estimated grade of 0.17 oz/ton Au (approximately 5.83 g/t Au).

Plant records show that some 132,000 tons (120,000 t) were processed, from which some 18,000 oz of gold were recovered. The head grade was 0.15 oz/ton Au (approximately 5.14 g/t Au) and mill recovery of the gold was reported to be 86.6% (Pieterse, 2005). In total, some 1,050 ft (approximately 320 m) of shaft sinking, 1,385 ft (approximately 420 m) of ramp driving and 19,600 ft (approximately 6,000 m) of drifting and cross cuts were developed for the production.

Figure 6-2 displays an isometric view of the Goldlund shaft and associated underground workings and underground drilling. The historical stopes mined at Goldund are shown in blue.

Figure 6-2: Isometric View (NE) of Historical Underground Workings at Goldlund



Source: CGK (2020).

6.2.4 Previous Mineral Resources Estimates

No historical mineral resources estimates are known prior to Tamaka's ownership of the project. There were several previous mineral resources estimates completed by Tamaka prior to Goldlund being acquired by First Mining Gold Corp. There are also previous mineral resources estimates that were completed by First Mining prior to the purchase of Goldlund by Treasury Metals.

All the previous mineral resources estimates are based on prior data and reports obtained and prepared by Tamaka and First Mining. Treasury Metals has not undertaken the work required to verify these previous mineral resources estimates. Therefore, Treasury Metals is not treating any of these previous mineral resources estimates as current mineral resources estimates that should be relied upon. Table 6.11 presents a summary of the previous mineral resources estimates.

Table 6.11: Previous Resources Estimations for the Goldlund Project

Company	Year	Classification	Tonnes	Au (g/t)	Ounces
Tamaka/Goldlund	2012	Measured	3,928,950	1.86	233,690
		Indicated	2,839,200	1.57	143,355
		Measured & Indicated	**6,768,150**	**1.73**	**377,045**
		Inferred	18,905,000	1.03	627,790
Tamaka/Goldlund	2013	Measured	11,333,000	1.55	564,575
		Indicated	7,623,000	0.92	226,036
		Measured & Indicated	**18,956,000**	**1.3**	**790,611**
		Inferred	42,542,000	0.78	1,070,223
Tamaka/Goldlund	2014	Measured	8,459,000	2.1	571,450
		Indicated	10,643,000	1.82	622,800
		Measured & Indicated	**19,102,000**	**1.94**	**1,940,250**
		Inferred	25,845,000	2.51	2,085,000
FMCG/Goldlund	2017	Measured	-	-	-
		Indicated	9,324,100	1.87	560,497
		Measured & Indicated	**9,324,100**	**1.87**	**560,497**
		Inferred	40,895,000	1.33	1,754,092
FMCG/Goldlund	2019	Measured	-	-	-
		Indicated	12,860,000	1.96	809,200
		Measured & Indicated	**12,860,000**	**1.96**	**809,200**
		Inferred	18,362,000	1.49	876,954

Source: WSP (2019).

6.3 Miller Property

There has been no historical exploration or drilling activities on the Miller deposit prior to 2018. In 2018 and 2019, First Mining completed two drill programs on Miller, as described in Section 10 of this report.

7 GEOLOGICAL SETTING & MINERALISATION

The geology for the Goliath deposit was sourced from P&E (2020) and cross-referenced against the 2014-2015 Drilling and Exploration Assessment Report authored by Paul Dunbar, P.Geo. and Adam Larsen P. Geo, with edits from AGP.

The geology for the Goldlund and Miller deposits was sourced from WSP (2020) with edits from CGK and AGP.

7.1 Goliath Gold Complex – Regional Geology

The Goliath, Goldlund and Miller projects are located in the Eagle-Wabigoon-Manitou greenstone belt situated in the northeasterly projecting arm of the Wabigoon Subprovince of the Archean Age Superior Province (see Figure 7-1). This belt is situated in a 150 km wide volcano-plutonic domain with an exposed strike extent of 700 km and extends an unknown distance beneath Palaeozoic strata at either end (Beakhouse et al., 1995).

South of the property, and just north of the Village of Wabigoon, is the “Wabigoon Fault” which is a major regional fault structure. It separates a northern domain characterised by generally southward-facing alternating panels of metavolcanic and metasedimentary rocks, from a southern domain of generally northward-facing metavolcanic rocks (Beakhouse, 2000).

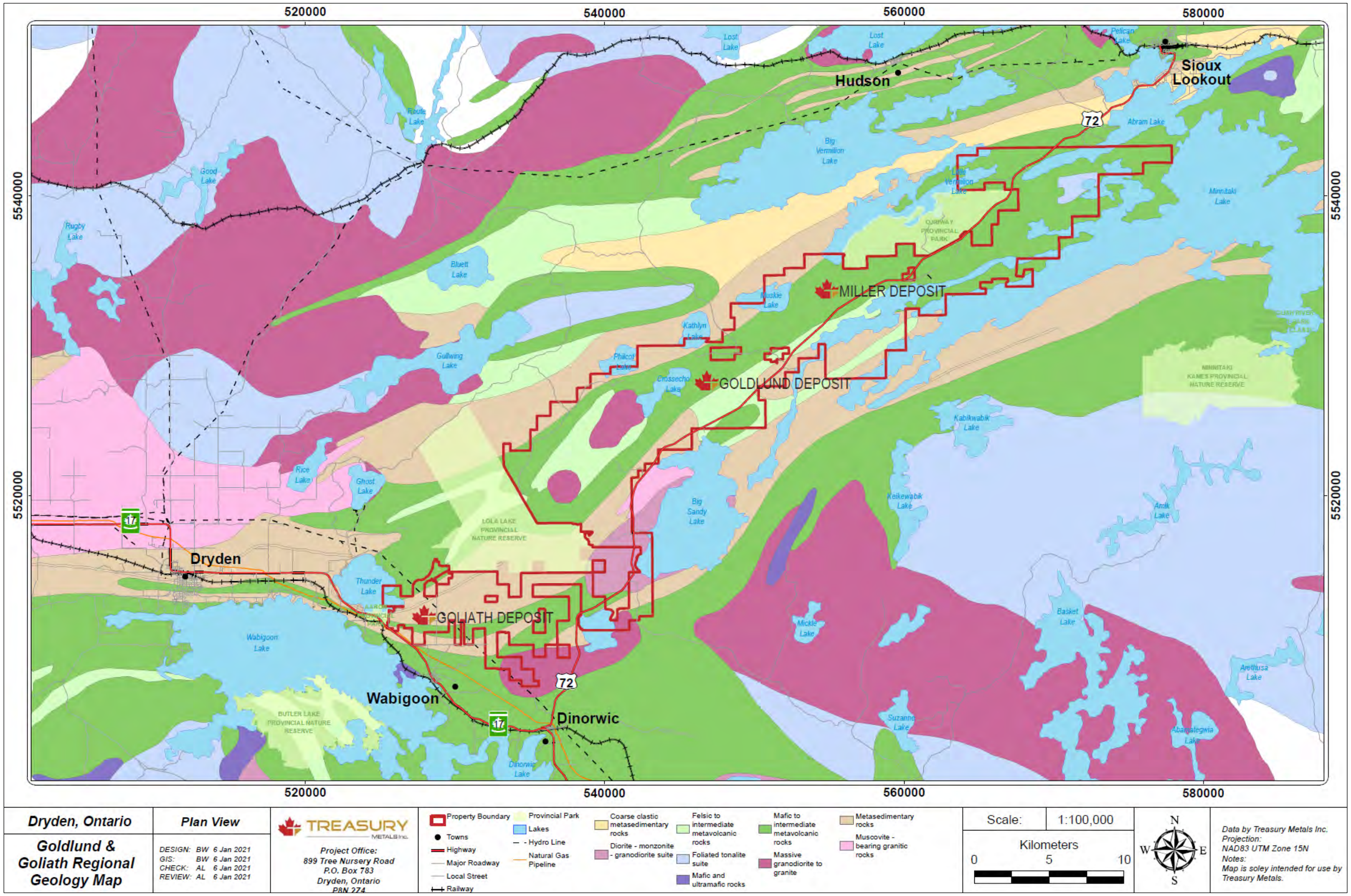
The stratigraphic assemblage has been subdivided into five principal rock groups: the Northern Volcanic Belt, the Northern Sedimentary Belt (Abram Group), the Central Volcanic Belt (Neepawa Group), the Southern Sedimentary Belt (Minnitaki Group), and the Southern Volcanic Belt. The Goliath, Goldlund and Miller Projects are located within the Central Volcanic Belt (Figure 7-2).

The greenstone belt is a volcano-plutonic complex and is one of the four-types of lithotectonic domains within the Superior Province intruded by syn-volcanic to post-tectonic granitoid plutons. The magmatic components of the greenstone belts include ultramafic to intermediate volcanics and more felsic volcanic and pyroclastic rocks.

The sedimentary component of greenstone belts includes both clastic and chemical deposits. Plutonic rocks in these domains include synvolcanic tonalitic, quartz dioritic, and granodioritic plutons, the emplacement of which is thought to have deformed the greenstone belts into arc forms. Metamorphic grade is generally green schist or sub-green schist grade except for narrow belts or the margins of larger belts which commonly display mineral assemblages typical of low-pressure amphibolite grade rocks (Percival and Easton, 2007a and 2007b).

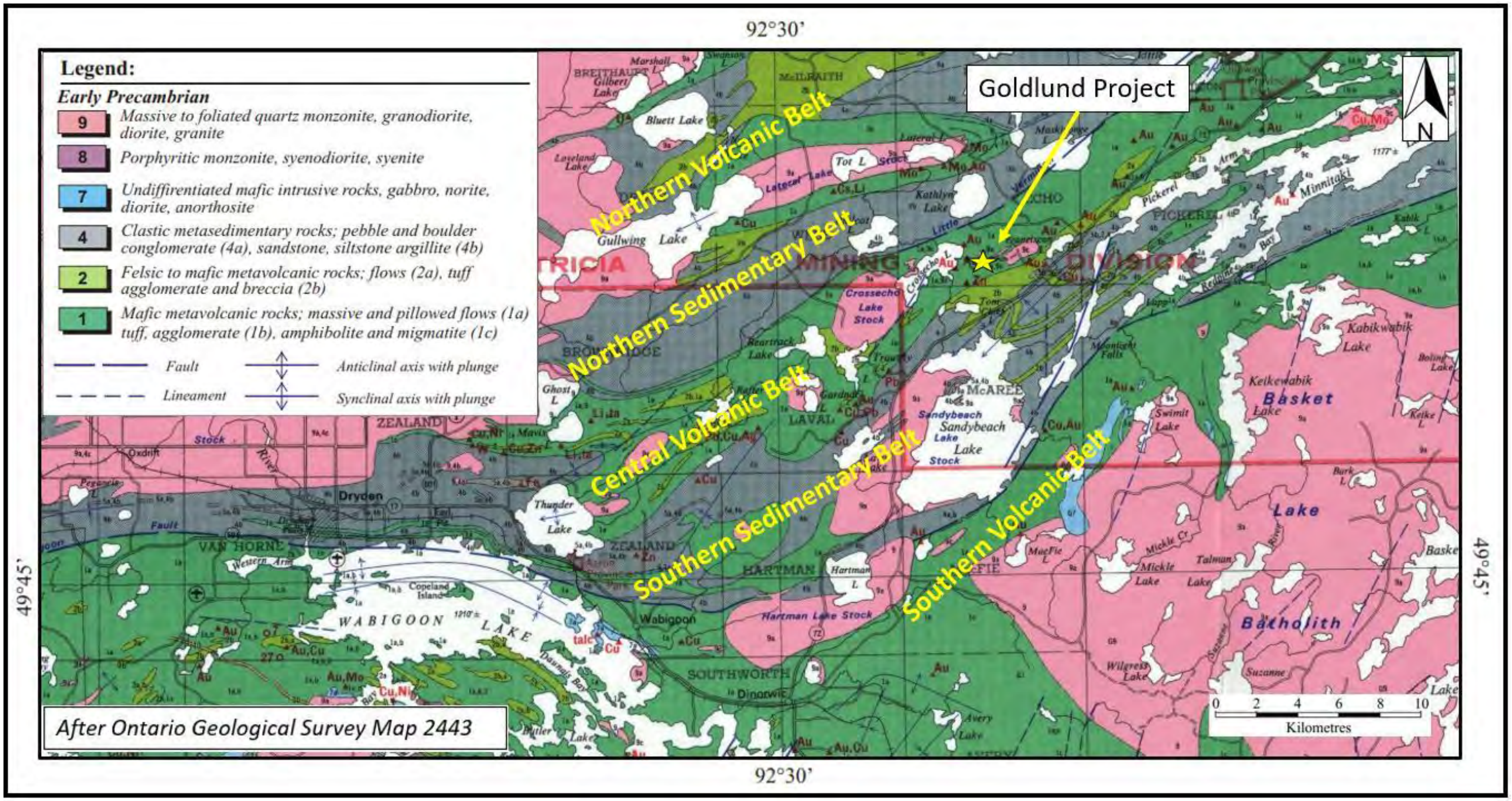
The Central Volcanic Belt (Neepawa Group) has been subdivided into a lower tholeiitic and an upper andesite-basalt division near the Goliath, Goldlund and Miller deposits. The lower division consists of tholeiitic mafic and felsic volcanic rocks with associated subvolcanic intrusions. The upper division consists of calc-alkaline, tholeiitic mafic to felsic volcanic units that crop out around the Beartrack, Troutfly, and Gardner Lakes. The Central Volcanic Belt (Neepawa Group) and the Southern Volcanic Belt are comprised of metavolcanic and metasedimentary rocks, while the Southern Sedimentary Belt (Minnitaki Group) forms an intervening belt of sedimentary units.

Figure 7-1: Regional Geology Map



Source: Treasury Metals (2021).

Figure 7-2: Regional Geology Map showing Volcanic & Sedimentary Belts



Source: CGK (2020).

The rocks of the Southern Sedimentary Belt (Minnitaki Group) are mainly greywacke and quartzo-feldspathic greywacke, with subordinate argillites and cherts, with minor mafic and felsic volcanic units. A distinctive banded chert-iron formation marks the base of the group throughout a large part of the area and displays a complex outcrop pattern, which defines the nature of the structural patterns.

7.2 Goliath Project

7.2.1 Property Geology

The earliest descriptions of the local geology were carried out by Satterly (1941) for the Ontario Department of Mines. These were later expanded with the updating of geological maps by the Ontario Geological Survey from 1995 to 2002 (Beakhouse, 2002; 2001; 2000; Beakhouse et al., 1995). A detailed geology map covering Zealand Township was published by Beakhouse and Pigeon (2003). Geology maps and descriptions of Laval and Hartman Townships were completed by Berger (1990).

The property area geology described below integrates all of the geological mapping, diamond drilling programs, and structural studies completed by Teck, Corona, CCIC and Treasury geological staff from 2008 to present (Roy et al., 2012; Roy and Trinder, 2011; Magyarosi and Peshkepia, 2011; Ilieva, 2008). The rocks have been grouped into the “Thunder Lake Assemblage” of predominantly meta-sedimentary rocks, and the “Thunder River Mafic Metavolcanic Rocks” (see Figure 7-3).

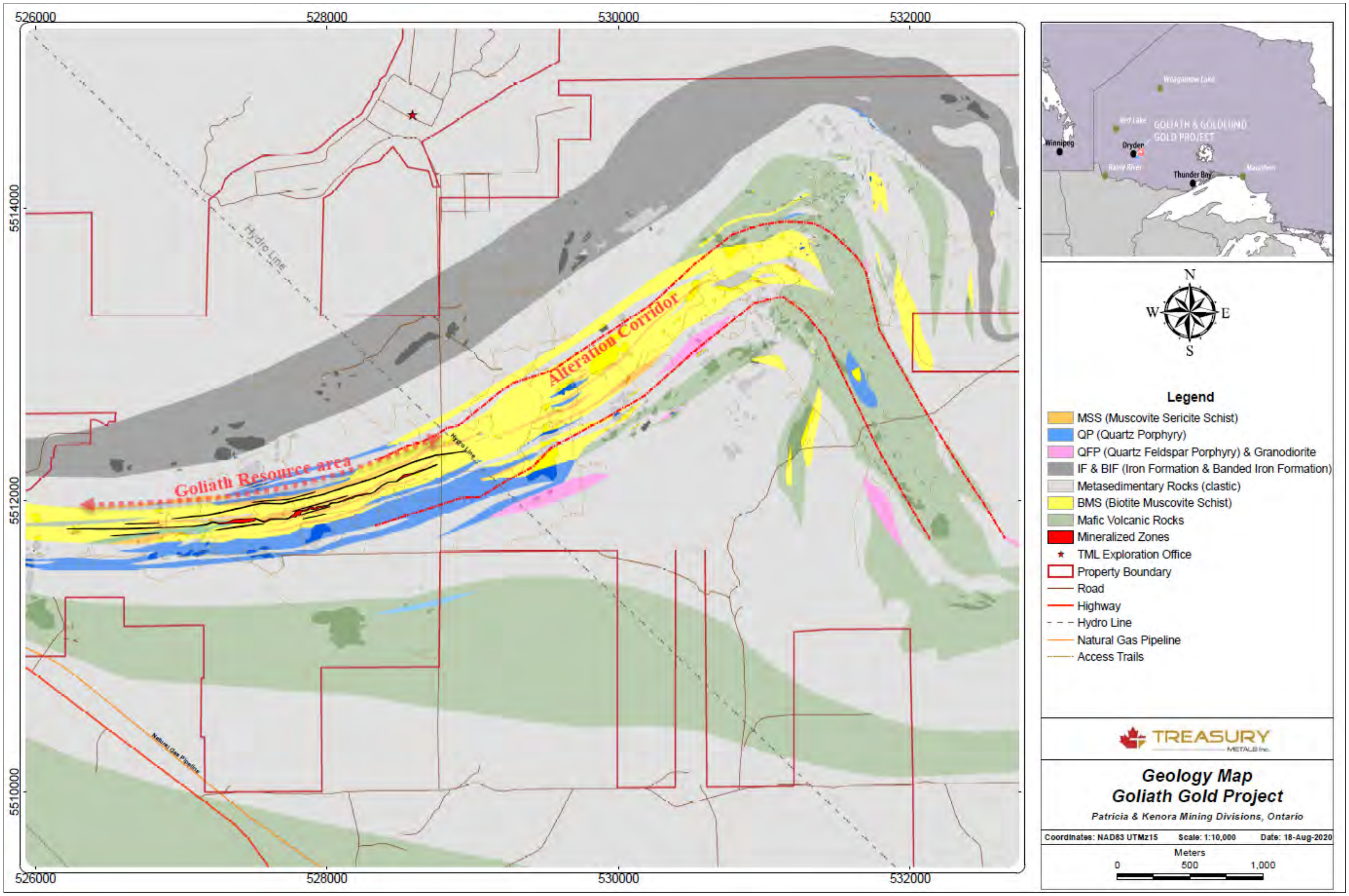
7.2.1.1 Thunder Lake Assemblage

The Thunder Lake Assemblage, an upper greenschist to lower amphibolite metamorphic grade volcanogenic-sedimentary complex, is typically separated into the “Thunder Lake Sediments” and “Thunder Lake Volcanics” (Beakhouse 2000). Underlying much of the project area, the assemblage comprises quartz-porphyritic felsic to intermediate metavolcanic rocks represented by biotite gneiss, mica schist, quartz-porphyritic mica schist, a variety of metasedimentary rocks and minor amphibolite rocks (see Figure 7-3 and Table 7.1).

Beakhouse (2001) described the Thunder Lake Sediments to be a package of rocks separated into two panels along its strike length by the Thunder Lake Volcanics. These metasedimentary rocks are dominated by biotite-muscovite and biotite schist (greywackes) with subordinate inter-layered metasedimentary rocks (probably pyroclastic siltstone and arkosic sandstone) which exhibit well-preserved primary sedimentary structures such as graded bedding, scour, and rip-up clasts unlike the nearby Zealand Sediments adjacent to the Wabigoon Fault whose primary features are contorted by a high degree of strain (Beakhouse, 2001).

The northern panel of Thunder Lake Sediments include ink blue coloured magnetite layers that are closely associated with distinctive garnet-rich layers and calc-silicate rock, described in earlier publications as iron formation (Satterly, 1941). Iron formation can be locally banded as “banded iron formation” (BIF) consisting of alternating layers of chert and magnetite. These iron formation units are the source of the prominent aeromagnetic anomaly that is folded across the western half of the property.

Figure 7-3: Local Bedrock Geology, Goliath Project, Northwestern Ontario



Source: Treasury Metals (2020).

Table 7.1: Thunder Lake Assemblage Rock Description

Rock Type	Description
Biotite Muscovite Schist (BMS)	Dark grey to grey, fine- to medium-grained mica schist. Usually, it consists of intercalated leucocratic and melanocratic bands. This unit contains a high number of grey to milky white quartz veins. Most of the veins are 1-15 cm wide, parallel, or crosscutting the foliation. Some veins are associated with highly chloritised and silicified intervals with tourmaline and sulphides.
Muscovite Sericite Schist (MSS) Interpreted as Altered Felsic Metavolcanic Rocks	Light grey to beige grey, fine- to medium-grained quartz- sericite schist. It is variably siliceous, commonly contains interbedded, dark grey biotite-muscovite bands and grey to milky white quartz veins. It is characterised by the presence of moderate to strong pervasive sericite alteration and gold- and silver-bearing disseminated sulphides.
Iron Formation (IF)	Dark greenish grey calc-silicate metamorphic rocks, which include coarse- to medium-grained gneiss, biotite schist, 10 to 15 cm wide distinctive layers enriched with garnet, chlorite, and narrow ink blue magnetite bands. The rock unit is magnetic and contains disseminated pyrite.
Metasedimentary Rocks (MSED)	Grey to dark grey-green medium-grained massive unit, which consists of biotite, feldspar, quartz, muscovite with a weak patchy potassium and sericite alteration and rare hematite (rusty brown) alteration. Foliation is poorly developed but more prominent in contact and altered areas. Quartz veins, parallel or crosscutting the foliation are very common. This unit can be distinguished by the presence of numerous "quartz eyes" or quartz porphyroblast (identified as "arkose metasediments" or "quartz feldspar porphyry" in Teck/Corona drill logs and historic reports). This unit may contain 1-5% bleb-finely disseminated pyrite and chalcopyrite.
Biotite Schist (BS)	Dark grey to black, fine- to medium-grained, slightly to well-foliated schist. Locally contains disseminated pyrite in the foliation planes and fractures. It was referred to as pelites or greywackes in the historical reports
Chloritic-Biotite Schist (Chl-BS)	Dark grey to greenish grey medium-grained, slightly to well-foliated schist. Locally it contains disseminated pyrite along foliation planes and fractures. Referred to as pelites or greywackes in the historical reports.

Source**:** Roy and Trinder (2011).

Compositional layering in metasedimentary rocks strike 90° in the western portion of the property around the Goliath deposit and dip from 70° to 80° south-southeast. The rock formational units strike northeast, east of the deposit. Schistosity is commonly developed within both the metasedimentary rocks and metavolcanic rocks and exhibits a similar orientation (Hogg, 2002). In general, the foliation and schistosity is parallel to stratigraphy.

Sandwiched between the sediments are the Thunder Lake Volcanics, a unit dominated by felsic metavolcanic rocks conformably inter-layered with wacke-siltstone. These rocks host the majority of gold mineralisation at Goliath. The lenses of metasedimentary rocks that occur within the felsic unit are similar to those making up the main sedimentary unit. All of the rocks have been subjected to folding and moderate to intense shearing with local hydrothermal alteration, quartz veining and sulphide mineralisation.

7.2.1.2 Thunder River Mafic Metavolcanics

The Thunder River Mafic Metavolcanic rocks underlie the southern part of the property between the southern panel of the Thunder Lake Sediments and the Zealand Sediments north of the Wabigoon Fault (see Table 7.2, Figure 7-2). The mafic rocks are generally massive, but are pillowed locally and include amphibolite and mafic dykes which are characterised as chlorite schists (Beakhouse, 2000). Some rocks have been described as ultramafic in character (Hogg, 2002). These ultramafic rocks have been mapped locally as soapstones.

Table 7.2: Thunder River Mafic Metavolcanic Rocks

Rock Type	Description
Mafic Dyke (MD)	Usually narrow dark green to almost black massive or slightly foliated fine- to medium-grained biotite-chlorite schist. The width of the layers can reach up to 5 m. The dykes can be either parallel to or crosscut the foliation.
Amphibolite (AMP)	Coarse- to medium-grained, dark green to black to green units, which consist mainly of 30-50% amphibole (hornblende and actinolite), 30% to 40% feldspar and pyroxene with rare post genetic quartz veins and layers of chlorite schist. It has typical “salt and pepper” appearance and nematoblastic texture.
Greenschist	Usually dark green to almost black foliated fine- to medium-grained schist, which consists mainly of chlorite, biotite, feldspar, amphibole. The width of the layers can reach up to 5 m.

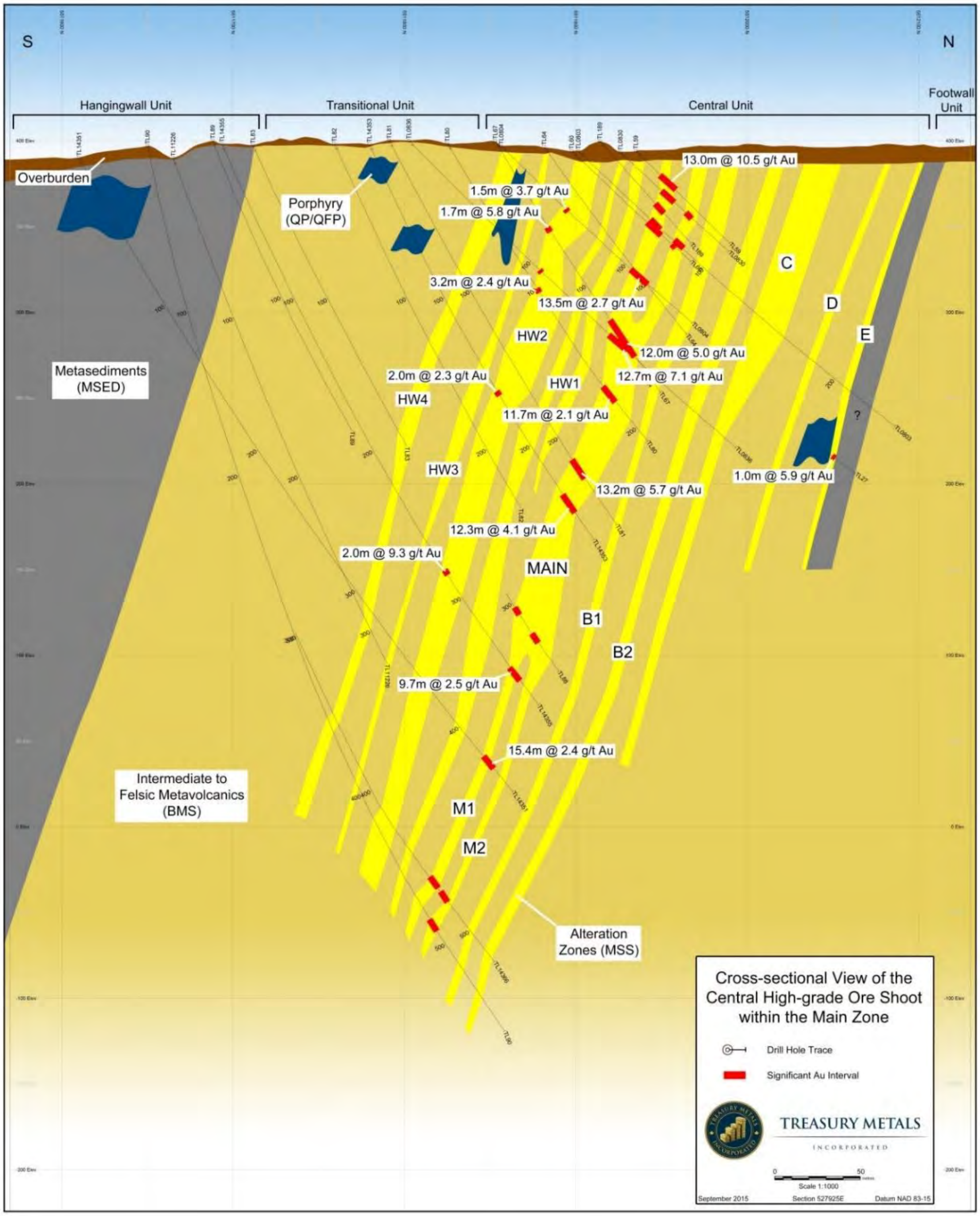
Source: Roy and Trinder (2011).

7.2.2 Deposit Geology

For the purpose of the exploration and development, the following four groupings are consistently recognised from south to north at the Goliath deposit (modified after Page, 1994; see Figure 7-4):

- “Hanging Wall Unit” of metasedimentary rocks (MSED) which share a sharp contact or may gradually grade to a biotite-muscovite schist (BMS) that have been intruded by quartz ± feldspar-porphyry intrusive rocks which may appear periodically along the strike length of the deposit
- “Transitional Unit” BMS occasionally intruded by porphyry rocks
- “Central Unit” that consists of:
  - a package of BMS, occasionally intruded by porphyry rocks, interlayered with up to four hanging wall alteration zones (HW1 to HW4) consisting of muscovite-sericite schist (MSS) that can have significant gold mineralisation that are often silicified
  - a core section of rocks, approximately 100 to 150 m true thickness, that hosts the most significant gold concentrations in the deposit (the Main and C Zones) and consists of intensely deformed and variably altered felsic, fine- to medium-grained, MSS and BMS with minor metasedimentary rocks
  - a package of rocks similar to (1) that hosts the D and E Zones in silicified MSS rocks surrounded by BMS
- “Footwall Unit” of predominantly metasedimentary rocks (MSED, BMS and weak iron formation) with some porphyritic intrusive bodies and minor felsic gneiss and schist rocks

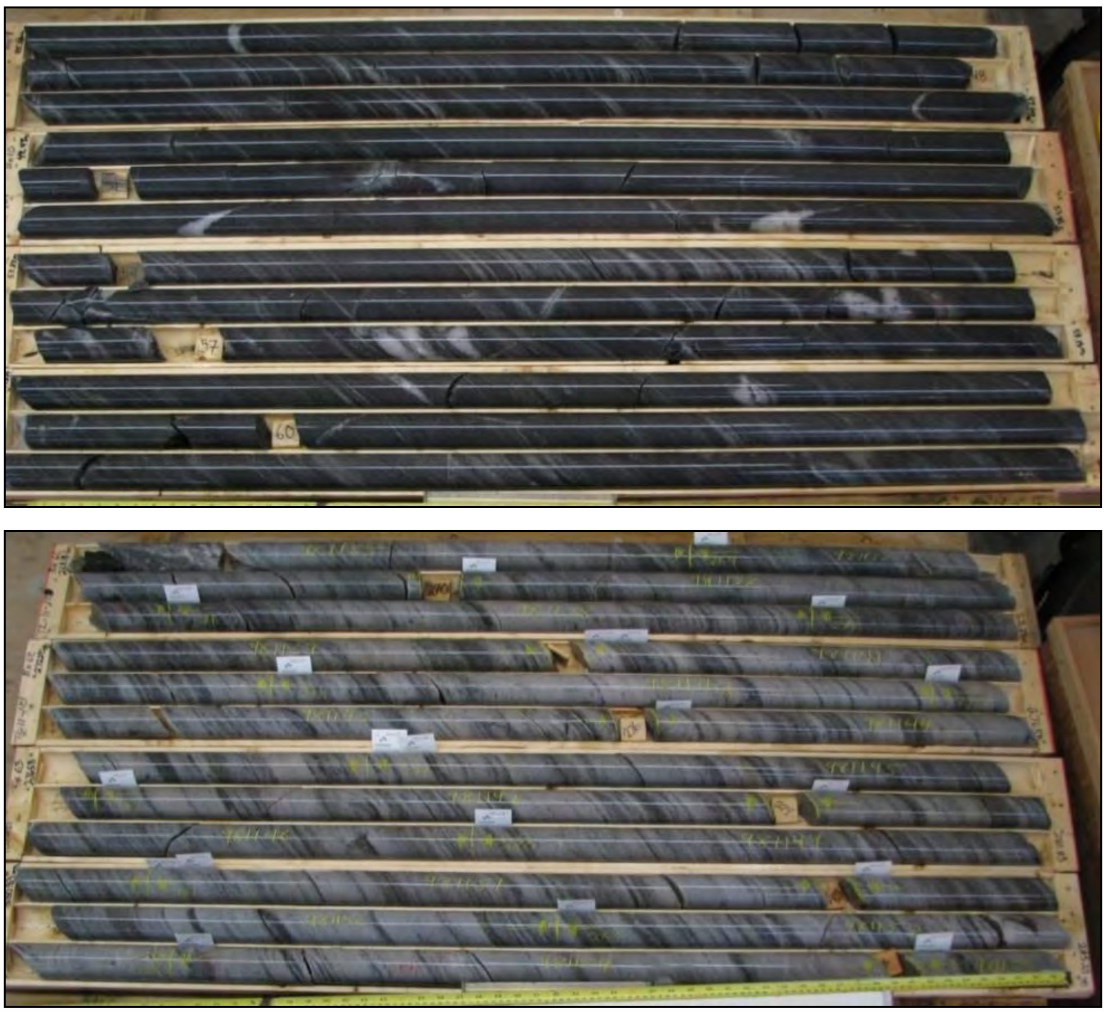
Figure 7-4: Geology of the Goliath Deposit



Source: Treasury Metals (2015).

Considering that the host rocks of the Goliath deposit are extremely altered and are now schists held together by fine-grained quartz which gives them their competency, Treasury Metals devised a system of grouping the altered schists into two distinct geological units that could be mapped across the deposit: the MSS and BMS units. These units are differentiated based on the relative modal abundance of biotite rich versus sericite rich layers, quartz (silicification) and sulphide mineral content. In general, the most altered rocks containing greater than 60% quartz-sericite felsic bands, are silicified and often contain base metal mineralisation, have been mapped as MSS (light coloured) units. Those units containing less than 60% white mica have been mapped as BMS (dark coloured). Figure 7-5 visually illustrates the difference between the two rock units. It should be noted that contacts are almost always gradational. Gold is usually associated with the MSS units in association with sphalerite and galena or occurs in smaller MSS bands hosted within the BMS units.

Figure 7-5: Diamond Drill Core Photographs showing BMS (top) & MSS (bottom) Core



Note: The top image shows BMS core; the bottom image shows MSS core. Source**:** Treasury Metals (2015).

7.2.3 Structural Geology

Page (1994), Beakhouse (2001), Ravnaas et al. (2007) and Wetherup (2008, 2010) have described and interpreted the key structural features on the property identifying three deformation events and three related generations of fold axes. Geological and trench mapping programs, as well as structural studies of bedrock and drill core, have been undertaken by Treasury Metals to obtain a better understanding of the structural geology of the property. Structures and veins observed in the area of the Goliath deposit have been interpreted within, and relative to, this basic framework.

7.2.3.1 D_0 Pre-Deformation Structures

The D_0 pre-deformation structures developed during the rock formation and are a result of possibly transposed bedding and/or alteration zones. They can be observed in core and bedrock as alternating leucocratic quartz-sericite and melanocratic biotite-feldspar layers and represent compositional layering within felsic metavolcanic and metasedimentary rocks. The width of the layers varies from 0.5 to 10 cm, but locally forms larger units interbedded with layers of metasedimentary rocks. Larger zones (< 40 m wide) of dominantly quartz-sericite schist locally contain greyish, very fine-grained layers or “ribbons” of quartz V_0 veins which are usually associated with sulphide (pyrite-sphalerite-galena-chalcopyrite) mineralisation and have the potential to host coarse gold. The association of almost pure, very fine-grained quartz layers within the centre of a larger zone of quartz-sericite schist could represent transposed and metamorphosed sericite alteration around quartz veins within the felsic metavolcanic rocks. Sulphide minerals observed in drill core commonly occur along S_1 foliation planes and appear to have been remobilised.

Contacts between the lithostratigraphic units were measured in the outcrops and in the core. Within the felsic volcanic rocks the contacts between the MSS and BMS units can range from transitional to sharp. More noticeable is the contact between the felsic volcanic rocks and the metasedimentary rocks that is usually marked by a very small angular discordance and is almost parallel to the primary bedding. The strike and dip are approximately 090°/70°S, but can change from 068°/72°S to 090°/80°S. It is interpreted that the primary syngenetic gold and silver mineralisation was deposited during this event because the mineralisation is mostly contained within the sericite schist and/or biotite-muscovite schist. Isolated concentrations of gold lying outside of these units may be related to later remobilisation or alteration and gold deposition at other parallel but different stratigraphic horizons as zones of mineralisation are all parallel to one another parallel to stratigraphy.

7.2.3.2 D1 Deformation

The D_1 deformation is represented by well-developed foliation S_1 and isoclinal folds F_1 within the felsic metavolcanic rocks (BMS, MSS) and metasedimentary rocks (biotite schist or “BS”) and iron formation). The foliation and the axes of the folds were measured in the outcrops, in the trenches and during the orientation drilling of holes TL0822 to TL0837. The foliation is approximately 074°/70°S, but it can vary from 064°/62°S to 090°/80°S. The mafic metavolcanic rock unit texture tends to be more massive as the foliation is suppressed.

F_1 folds were observed in the outcrops and in the core. The folds are isoclinal, and the fold axes are parallel to the F_1 foliation. The dip and strike of the axial planes are approximately 090°/70° but it can change from 080°/68°S to 100°/78°S. In most cases, the hinges/fold noses display evidence of distension where continuing compressional deformation has stretched the hinge and its limbs are highly attenuated and thinned. These fold noses are often completely decapitated from their limbs and generally only hook-shaped or quartz lenses remain, which suggests that some of the boudinage or quartz lenses observed in the felsic metavolcanic rocks may be related to F_1 structures. Deformed, white, coarse-grained quartz veins ± tourmaline, ± stringers or porphyroblasts of sulphides, 1 to 10 cm wide, occur dispersed throughout the felsic metavolcanic and metasedimentary rocks. White, coarse-grained quartz veins are not localised to certain pre-deformational stratigraphy and are interpreted to be syn-tectonic (V_1) as they are affected by D_1 deformation and occur in all rock types. They typically crosscut the foliation but may be parallel in some instances. The assay results show no direct correlation between the quartz veins and elevated gold and silver concentrations.

7.2.3.3 D2 Deformation

The D_2 deformation is observed as zones of disturbed foliation related to closed F_2 folds and V_2 quartz veins. Rare F_2 fold hinges are observed in the outcrops. They are several centimetres in scale and affect the position of the felsic volcanic package that hosts mineralisation on the Goliath Project. Where F_2 fold axes and fold noses occur within the gold-silver mineralised zones in the felsic metavolcanic rocks, gold and silver values are commonly 10 to 100 times higher than in the adjacent intervals (Roy et al., 2012). In some cases they contain coarse-grained visible gold (VG) or electrum, but even the very fine-grained mineralisation returns higher gold or silver concentrations. Throughout the 2008 mapping program the orientation of the F_2 fold axes were measured in the outcropping rocks. The strike of the F_2 plane is approximately 220° to 230° and dips 85° to 90° southward. In addition, the F_2 fold axes are almost vertical and the intersections of these fold axes and the mineralisation plunge steeply westward. Overall, discrete F_2 fold zones are narrow (up to 10 to 15 cm wide), widely spaced (5 to 25 m) and locally carry significant gold mineralisation. Determining where F_2 folds are likely to be located will identify areas of potential high-grade mineralisation. S and Z folded F_1 foliation, V_0 and V_1 quartz veins, and non-deformed crosscutting V_2 veins are all features attributed to the D_2 deformational event. The veins are differentiated on the basis of mineralogy, texture, and amount of strain.

7.2.3.4 D3 Deformational Event & Northwest Fault

The D_3 deformational event is represented by brittle faults and fractures filled in with quartz, chlorite, feldspar, carbonate and/or fault gouge. Local shear zones and faults are exposed in outcrops and old trenches.

The first fault system is almost vertical and strikes 220° to 240°. The system consists of almost parallel micro-faults with dextral displacement on a centimetre scale. Very often it is accompanied with a 1.0 to 1.5 m wide sericite alteration.

The second fault system, exposed in the outcrops, has almost a north-south orientation. The azimuth bearing ranges from 352° to 008° and the dips from 85° to 90°. Usually the fault zone consists of 2 to 3 micro-faults located within an interval with widths ranging from 0.5 to 1.0 m. These faults can be found in all rock types including clastic metasedimentary, felsic volcanic and mafic volcanic rocks. Commonly the rocks adjacent to the faults are highly fractured.

The most significant feature found in the drillholes that can be related to D_3 deformation is what Teck-Corona described as the Northwest Fault. This is a brittle structure which strikes west to west-northwest and dips shallowly northward and was observed in most of the deeper holes. Drill section interpretation by Teck-Corona shows very little dip-slip movement along this structure (approximately 5 to 10 m, hanging wall up). Most shallow dipping structures are dip-slip in nature, but since this is such a prevalent feature there may be a significant component of strike-slip motion, since dip-slip offset is minor.

A third generation of white, coarse-grained quartz veins (V_3) are formed during the D_3 event. These veins occur in all rock units and typically crosscut the foliation obliquely with sharp margins. No deformation appears to have occurred in these veins, which can also cut D_2 structures. V_3 veins are hematised on the surface, have been previously sampled, and do not return any significant gold or silver values. D_3 deformation is not related to the gold-silver mineralisation emplacement. However, the Northwest Fault appears to offset the mineralised zone towards the northeast of the main deposit. Wetherup (2008) demonstrated that high-grade mineralisation occurs along the steeply southwest plunging intersections of F_1-F_2 fold axes and that these shoots are offset by the northwest Fault.

7.2.4 Mineralisation

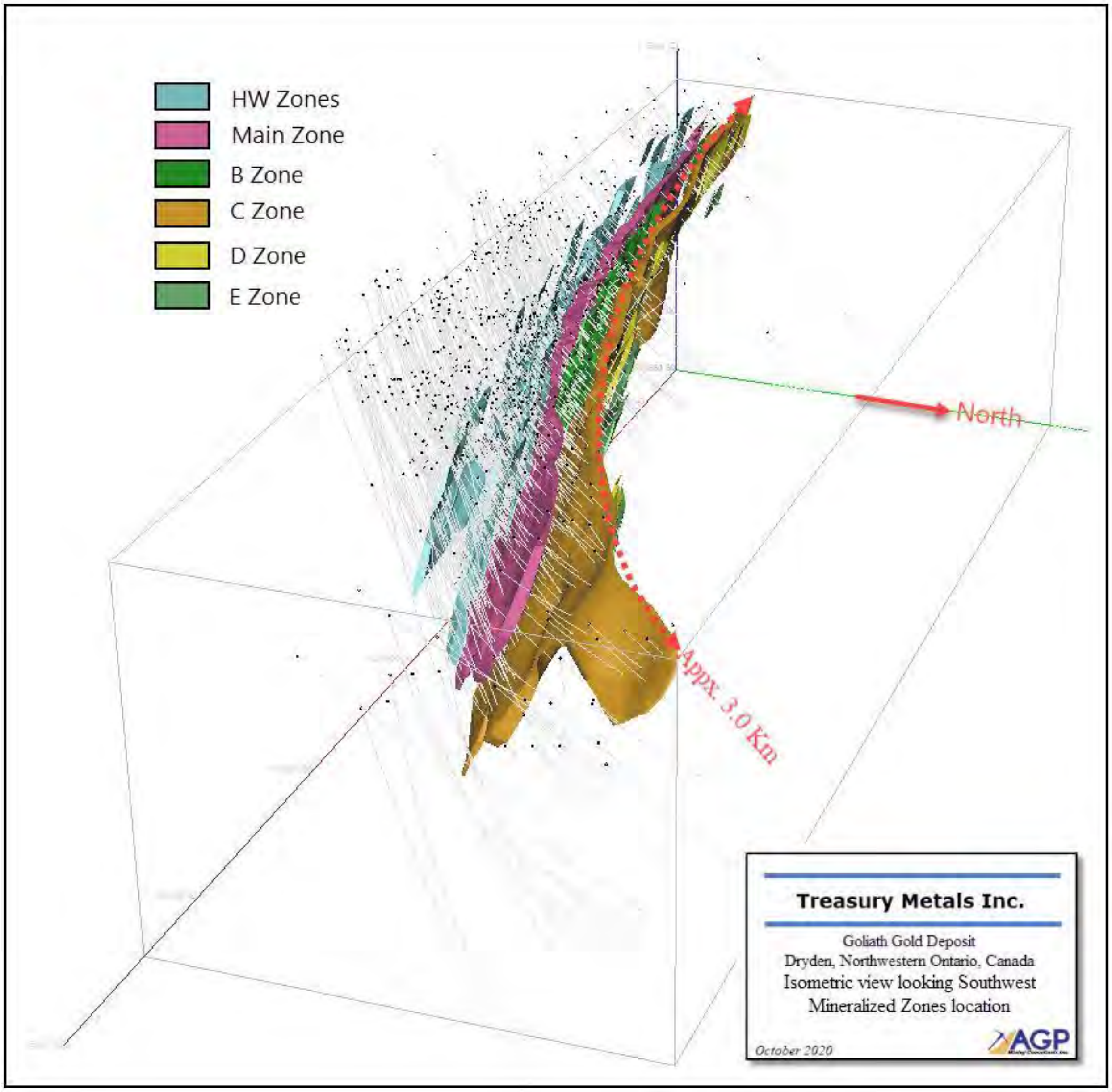
The Goliath deposit is located 250 to 300 m north of Norman Road and since 1990 the main resource area has been defined by extensive diamond drilling efforts concentrated over a strike length of over 2.0 km. To date, 12 zones containing gold and silver mineralisation have been identified within the Central Unit of the main deposit. From south to north, they are the:

- Hanging Wall Zones (HW1 to HW5 subzones), hosted in mostly BMS rock units and small amounts of metasedimentary and porphyry intrusive rocks
- Main Zone (M1 and M2 subzones), which is 5 to 40 m wide and occurs principally in silicified and sulphide mineralised (sphalerite, galena, and pyrite) MSS rocks
- B Zone, hosted in BMS rocks residing between the Main and C Zones
- C Zone (C1 and C2 subzones), hosted in silicified and sulphide-bearing (sphalerite and galena) MSS rocks
- D and E Zones, hosted in mostly a mixture of MSS and BMS rocks surrounded by significant amounts of metasedimentary rocks and minor porphyry intrusive rocks

It is noted that the BMS rocks located between the M1 and M2 and the C1 and C2 subzones often display lower grade mineralisation, which is largely due to smaller MSS bands hosted within the BMS units.

The majority of the historical gold and silver resource estimates reside in the Main Zone and C Zone (Figure 7-6). At Goliath, the gold-bearing zones all strike from 090° to 072° with dips that are consistently 72° to 78° toward the south or southeast. The main area of gold, silver and sulphide mineralisation and alteration occurs up to a maximum drill-tested vertical depth of approximately 805 m (TL135) below the surface, over a drill-tested strike-length of approximately 3,000 m within the current defined resource area. Gold mineralised zones remain open at depth. The historic Teck-Corona drilling confirmed that anomalous gold mineralisation occurs over a strike length of at least 3,500 m (Corona, 1998). Exploration work by Treasury has shown alteration zones containing intersections of gold mineralisation extend over a strike length of at least 5,000 m. Overall, rocks surrounding the principal defined target zones are often anomalous in gold mineralisation (background gold concentrations).

Figure 7-6: Perspective View of the Goliath Deposit showing Interpreted Mineralised Zones



Source: AGP (2020).

The mineralised zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. Stratigraphically, gold mineralisation is concentrated in an approximately 100 to 200 m wide Central Unit composed of intensely altered felsic metavolcanic rocks (quartz-sericite and biotite-muscovite schist) with minor argillaceous metasedimentary rocks. Higher-grade gold within the central unit is concentrated in a pyritic alteration zone consisting of MSS, quartz-eye gneiss and quartz-feldspar gneiss with lower grade gold in BMS.

To date, drilling has focused primarily on targeting the Main and C Zones. Caracle Creek International Consulting Inc. (CCIC) determined that native gold and silver (electrum) are associated with finely disseminated sulphides, coarse-grained pyrite, and very narrow light grey translucent “ribbon” quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite in decreasing order of abundance. The sulphide content ranges from 3% to 5%, but is locally up to 15%.

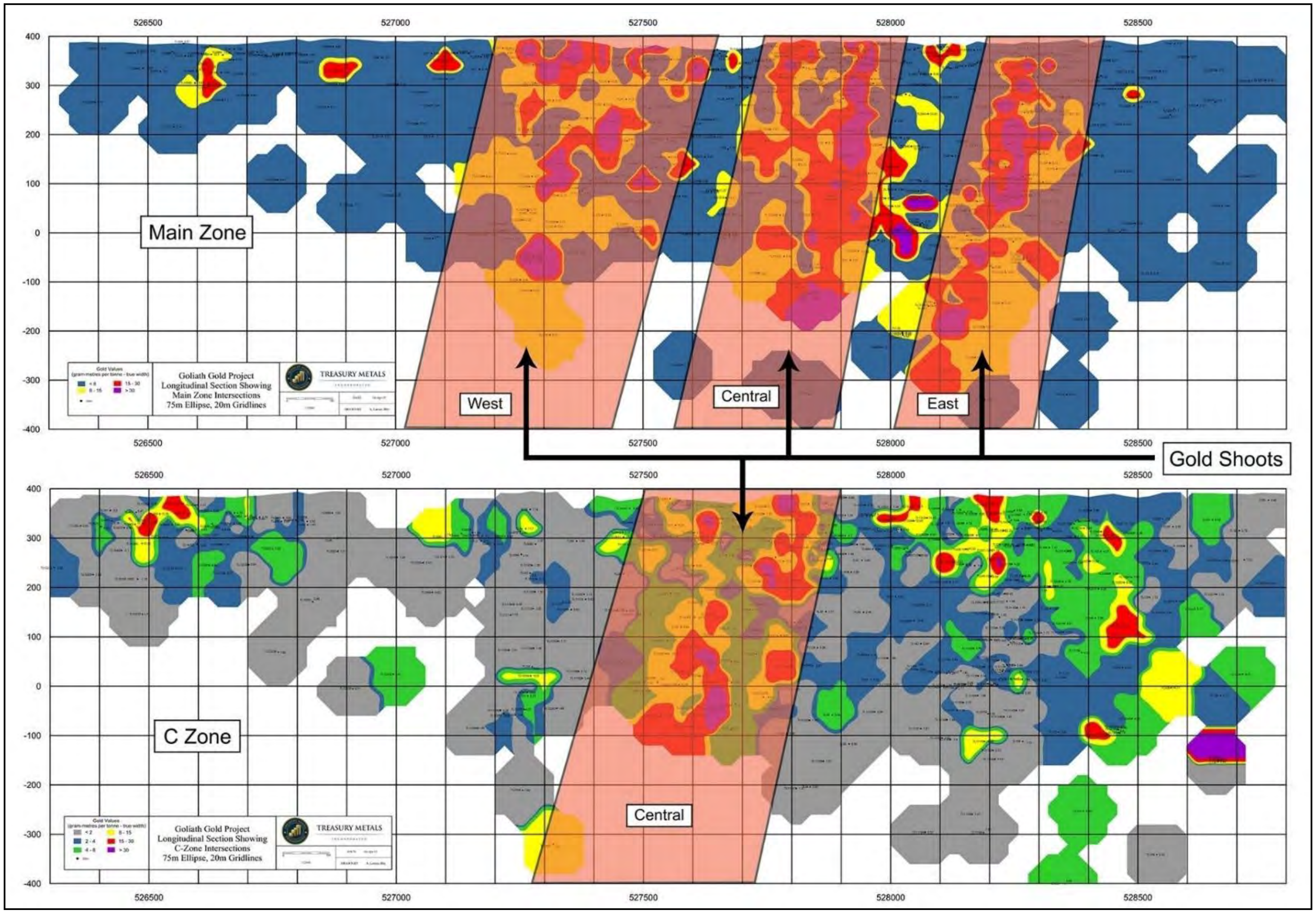
Visible gold and/or electrum are rare and occur mainly within the leucocratic bands of MSS, but can also in the melanocratic bands enriched with biotite and chlorite. In general, the highest gold and silver values occur in association with very strong pervasive quartz-sericite alteration. An increase in gold and silver correlates with an increase in pyrite and more specifically an increase in sphalerite content. The modal abundance of sphalerite usually exceeds that of galena and pyrite. Although the presence of elevated sphalerite and galena have been used as an indicator of the potential presence of gold with the deposit, there are some instances when gold is not present even through the base metals are clearly visible in drill core. In addition, an increase in chalcopyrite and galena content has a lower correlation to an increase in gold values.

Two distinct types of pyrite are recognised: disseminated fine-grained cubic euhedral crystals occurring in the foliation planes; and disseminated subhedral to irregular grains and stringers, with inclusions of galena, occurring in quartz veins and along the margins of the veins. The second type is commonly associated with other base metal sulphides. Pyrite can occur as fine-grained disseminations in the foliation planes, disseminations in the matrix, blebs, stringers and or veinlets. The base metals sulphides can be concentrated in blebs and stringers of sphalerite, cubic fine-grained galena and on occasion as chalcopyrite.

Silver-to-gold ratios are generally unpredictable and have a substantial range. Possibly during the syngenetic mineralisation event, more silver than the gold was contained in the hydrothermal solutions (ratio Ag/Au>1), but during the epigenetic mineralisation event, some of the gold was redistributed and there was enrichment in structurally induced zones of enhanced porosity and permeability. A similar relationship of gold to base metals is observed.

In the Goliath deposit, high-grade gold mineralisation and silver occur in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west (Figure 7-7). In the Main Zone, three shoots have been well defined named the “East”, “Central” and “West” shoots and a central shoot has been delineated along the C Zone. Corona (1998) interpreted the high-grade shoots to be the result of tight folding of the mineralised horizon (gold concentrated in fold noses) that appear to occur at regular intervals (Figure 7-7). The shoots have considerable down-plunge continuity and are all open and untested down dip at depth. Treasury has interpreted that these zones may be connected through a large folded anticlinal feature with a fold axis that strikes down the centre of the deposit and plunges around 10° to 20° east.

Figure 7-7: Longitudinal Section: Main Zone (Top), C Zone (Bottom)



Source: Treasury Metals (2015).

The Main Zone is comprised of one larger well-defined pyritic, and often silicified, MSS Zone or is bifurcated into two sub Zones (M1 and M2) separated by less-altered BMS rocks. C Zone gold mineralisation always occurs in the C1 and C2 subzones hosted in sulphide mineralised and silicified MSS that demonstrate excellent on strike and down dip continuity throughout the deposit.

The portion of the Central Unit of the deposit that hosts the B, C Zone and D and E Zones ranges in thickness from 75 to 150 m, but is often lower in grade than the Main Zone. It should be noted that the D and E Zones have often only been sporadically drill tested since many holes historically end before intersecting them. Since the 2011 technical report, Treasury Metals has re-entered 30 historical Teck and Treasury Metals drillholes to extend the holes in order to intersect the C, D and E Zones and have conducted an extensive infill sampling program of existing core to provide B Zone assay data to add to the mineral resource.

The Hanging Wall Zones (HW1 to HW5) are located 10 to 50 m south of the Main Zone. These zones are often narrow in width (1 to 3 m) and remain open along strike and at depth. Many of the historic Teck intersections of these zones were not consistently sampled because they were not significantly mineralised or contained no visible base metal minerals (sulphide content ranges from 3% to 5%). Gold and silver are probably included in the pyrite or around the pyrite micro grains. Only a few flakes of coarse-grained gold or electrum were visible in the core or in the grab samples. Most of the sulphides are located mainly in blebs or stringers parallel to the foliation planes. Usually blebs, stringers and veinlets of pyrite are associated with the stringers of sphalerite, cubic fine-grained galena, chalcopyrite and pyrrhotite. Very often they infill small fractures in the host rock or occur along margins of quartz veins.

7.2.5 Alteration

The Goliath deposit consists of hydrothermally altered felsic metavolcanic and metasedimentary rocks. Alteration has been traced through drilling and geological mapping for an approximate strike length of at least 5 km. The alteration consists of primarily sericitisation and silicification in association with the gold mineralisation. Chloritisation is visible in metamorphosed and altered mafic rocks in the area. Very rare flakes of aquamarine green mica (fuchsite: Cr muscovite) occur in the strongly altered sericite alteration and will sometime appear within the vicinity of high-grade gold.

Page (1995a) correlated the sericitic alteration of MSS with moderate potassium enrichment and significant sodium depletion. CCIC made the following observations from the analyses of 756 whole rock samples collected from holes TL0801, TL0802, TL0807, TL0808, and TL0823:

- The intervals with significant gold and silver mineralisation are very strongly altered.
- Very often extensive pervasive hydrothermal alteration obscures primary textural and structural features to the extent that it is not possible to identify the original rock type.
- The hydrothermal alteration commonly involves massive depletion of CaO and Na_2O and addition of H_2O, K, silica and sulphur as quartz ribbons and sericite.
- The feldspar and biotite are totally replaced by sericite, quartz and disseminated pyrite.
- Most of the mineralised zones are hosted by fine to medium-grained MSS or in patches of sericite alteration in BMS.

- The chlorite alteration is more intense in zones of fractured and brecciated host rocks. As a result of the depletion of CaO and Na_2O from the feldspar and addition of MgO and Fe_2O_3, sulphur and silica, quartz-pyrite-chlorite-tourmaline veins were formed.
- Complex, overprinting alteration and metamorphic assemblages and diverse metal associations are interpreted to be the result of an overprinting of hydrothermal and metamorphic fluids, which were focused in the zones of structurally-induced porosity/permeability.

7.3 Goldlund Project

This section has been summarised from previous technical reports, including the 2020 Treasury Metals Technical Report.

7.3.1 Property Geology

A 3 km wide belt of Precambrian mafic metavolcanic rocks strike northeast across the Goldlund Project area. These mafic metavolcanic rocks are bounded by Precambrian metasedimentary rocks to the north and to the south, with a wedge of Precambrian felsic metavolcanic rocks that occur at the southern contact between the Precambrian mafic metavolcanic rocks and the Precambrian metasedimentary rocks (see Figure 7-8).

The mafic metavolcanic rocks have a 1.5 km wide tuffaceous member to the south and a series of spherulitic basaltic flows interlayered with basaltic pillow lavas and some tuffs to the north. The basaltic metavolcanic rocks are dark green, massive in texture and weakly to strongly foliated. Other textures have also been observed, including amygdular flows, pillowed flows, lapilli tuff, feldspar crystal flows, and variolitic (or "spherulitic") flows.

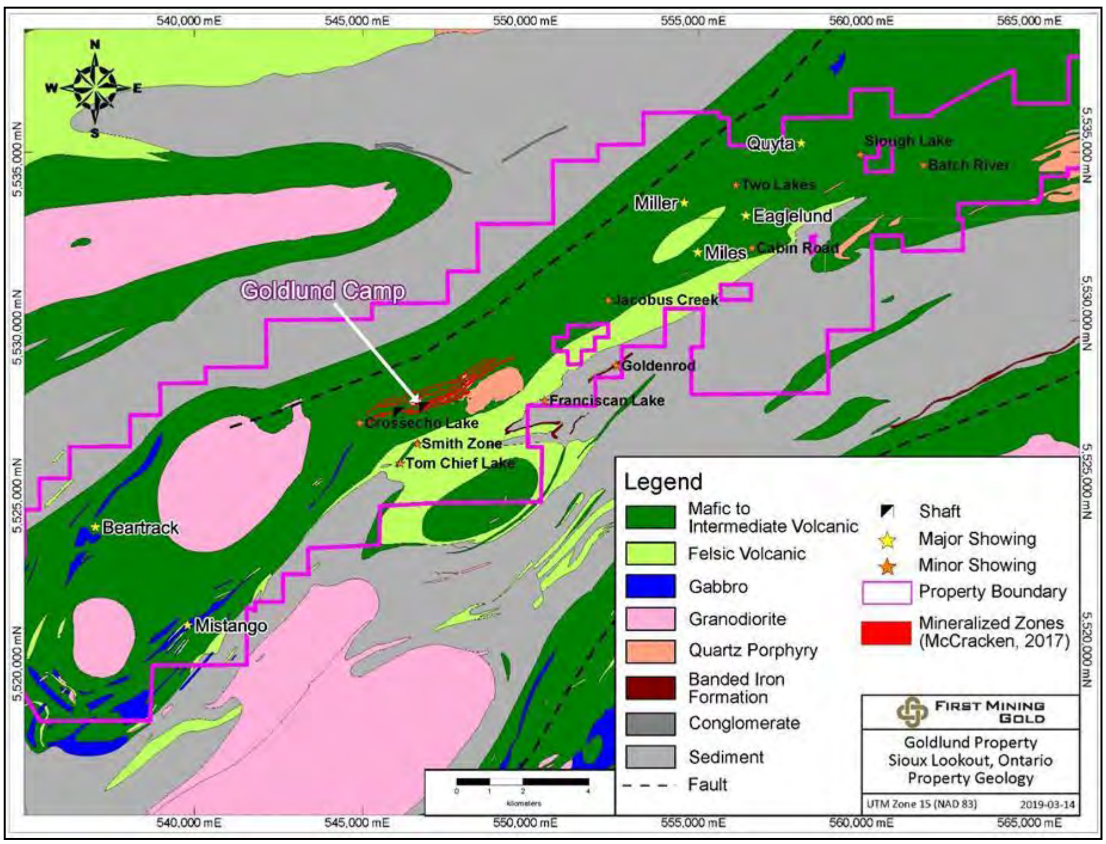
The mafic metavolcanic rocks are commonly magnetic, although significant variation in the strength of magnetism has been observed from outcrop to outcrop. In some cases, coarse magnetite crystals were observed and magnetite content up to several percent was observed. In contrast, very little pyrite or carbonate has been observed in the basaltic metavolcanic rocks in the Goldlund area. The metavolcanic rocks in the Goldlund area also lack the iron (Fe)-carbonate/sericite altered shear zones that are commonly observed in other greenstone belts.

Veining is relatively common within the mafic metavolcanic rocks. The most commonly observed veins are single, thin, sharp-walled, irregular quartz veins, containing minor chlorite and trace pyrite mineralisation. Larger veins and veinlets with minor carbonate, biotite, and chalcopyrite have also been observed and occasionally sampled. In particular, large (sometimes more than 20 cm) irregular quartz veins have been observed to form within the mafic metavolcanic rocks in close proximity to the mineralised felsic metavolcanic rocks in some places. It is unknown whether these veins carry gold. "Transverse" style veining is also observed occasionally within the mafic metavolcanic rocks, suggesting that the competency contrast between different mafic metavolcanic rock phases may be sufficient to localise veining and potentially, gold mineralisation.

Albite-trondhjemite to diorite sills ("granodiorite" in mine terminology) have intruded near the contact between the mafic metavolcanic tuffaceous rocks to the south and the spherulitic mafic metavolcanic rocks to the north. These strata-parallel sills dip from vertical to -80° southward and range from 14 m to 60 m in thickness. A subsidiary suite of sills intrudes the narrow tuffaceous metavolcanic rocks that are interbedded with the spherulitic mafic metavolcanic rocks. These strata-parallel intrusions are known to extend north-eastward well beyond the Goldlund deposit, toward the Miller deposit, and south-westward beyond Crossecho Lake where they re-appear just south of Troutfly Lake. It has been postulated that this series of intrusions may occur intermittently over a strike length of 15 km.

The albite-trondhjemite to diorite sills that host the most important zones of mineralisation at the Goldlund Project have been referred to as "grey granodiorite" due to their light colour and significant amounts of biotite and free quartz (Armstrong, 1951). Meta-gabbroic or meta-dioritic rocks in both transitional and intrusive contact with the "granodiorite", as well as crosscutting feldspar and quartz-feldspar porphyry dykes, were at times themselves referred to as "granodiorite", causing the terminology to become confused. The sills of granodiorite and/or its gabbroic counterparts to the northeast and southwest of the mineral deposit at the Goldlund Project have been considered primary exploration targets in the past.

Figure 7-8: Property Geology Map



Source: WSP (2019).

7.3.2 Structural Geology

Chorlton (1991) interpreted four-stages of deformation in the Sandy Beach Lake – Sioux Lookout area, based on the overprinting of individual structures and fabrics. These are described below.

The Stage 1 deformation is expressed by a locally preserved foliation, sub-parallel to bedding. The relatively shallow angle between bedding and foliation may be an indication of thrusting.

Stage 2 deformation is associated with the emplacement of the granitoid bodies throughout the area.

Stage 3 deformation is largely responsible for the northeast-trending structural grain of the belt. Northwest-southeast compression and sinistral rotation generated large-amplitude upright folds with steep, northeasterly-trending axial planes, together with steep northeasterly-trending shear zones. Shear zones northwest of the Beartrack – Crossecho Lakes area and southeast of the Sandy Beach Lake area tend to be sinistral-oblique, southeast-side-up, while those in the central portion of the belt tend to be sinistral and sub-horizontal.

Stage 4 deformation reflects the final phase of convergence in the belt. Large- to small-scale folds with steep, north-northeasterly-striking axial planes overprint the Stage 3 folds. Irregular belt boundaries and rigid internal stocks restricted further lateral extension and resulted in vertical displacements along the core of pre-existing shears.

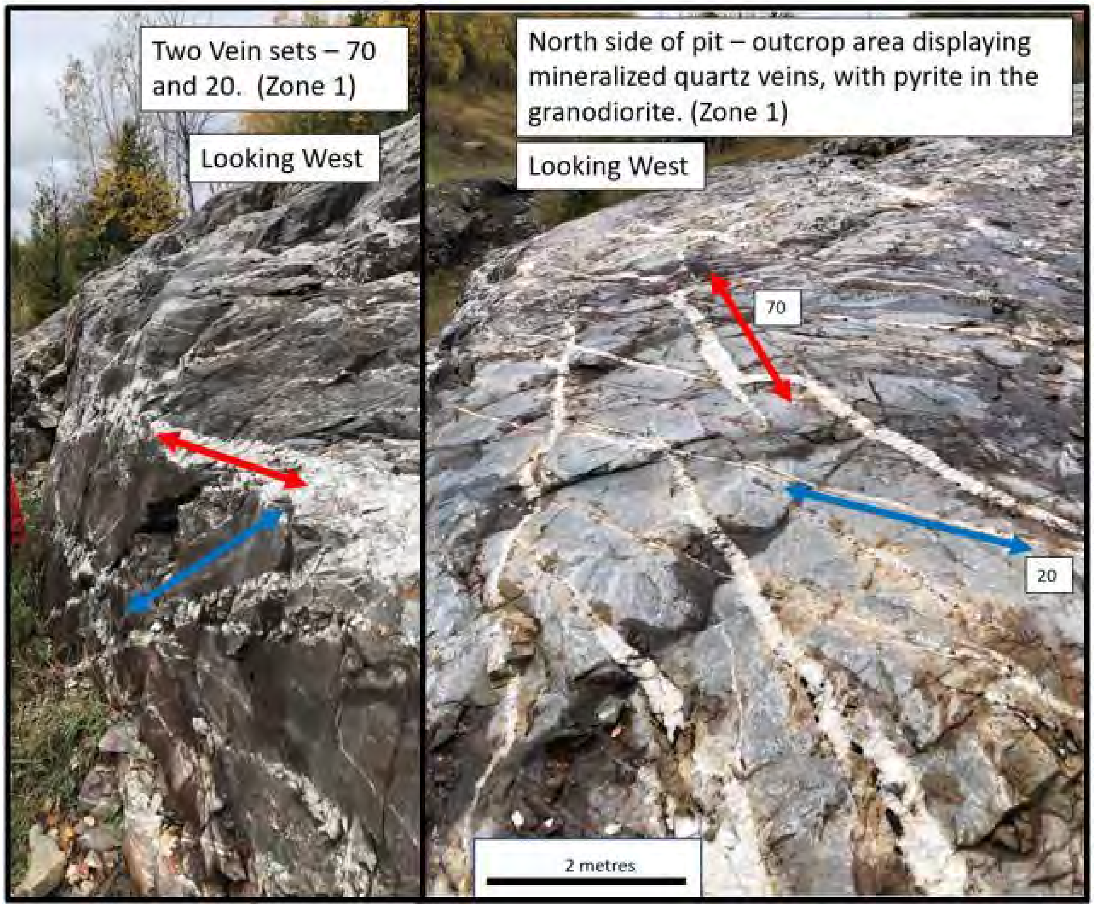
7.3.3 Mineralisation

Gold occurs in essentially two types of deposits in the Goldlund area. The most important gold mineralisation is associated with quartz vein and stock-work structures, which are found in albite-trondhjemite sills, as well as in porphyry sills and mafic metavolcanic rocks (Page, 1984). Trace to minor quantities of gold (and silver) are found in disseminated and massive sulphide deposits (copper- nickel, copper-zinc) in metavolcanic rocks.

Gold mineralisation is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. These stockwork zones form bands within the sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse- grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite.

The mineralised sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the Project. Although locally they may differ by up to ± 20° in strike and dip, overall, they are a very consistent 239°/58°N (70 set) and 189°/53°W (20 set) orientation. Figure 7-9 displays photographs of the quartz stockwork veins south of the historical open pit.

Figure 7-9: Goldlund Project Zone 1 Quartz Stockwork Mineralisation



Source: CGK (2020).

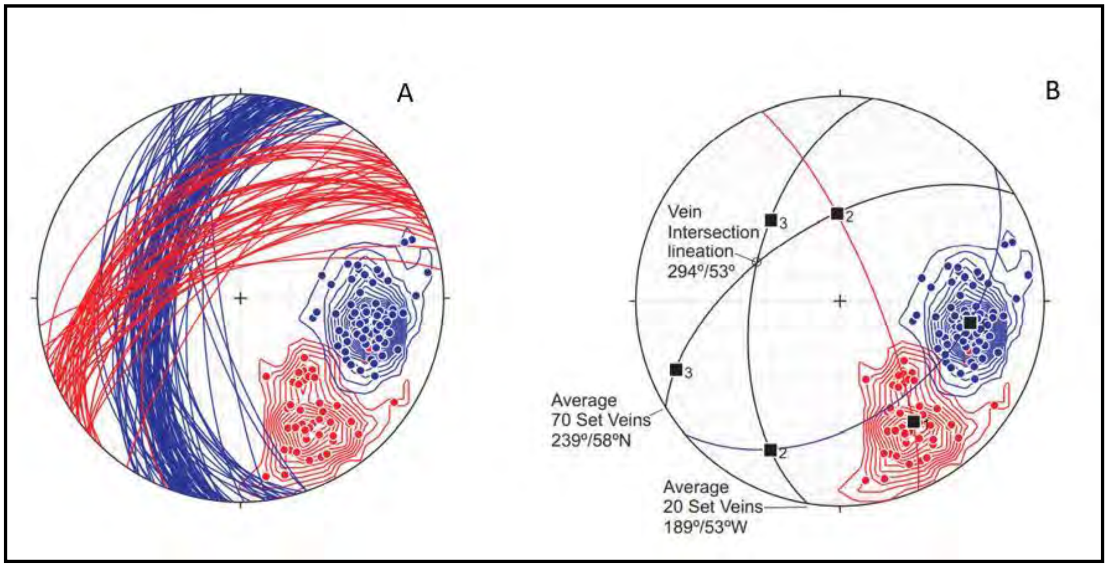
Figure 7-10 illustrates the planes and poles of the principal vein sets at Goldlund. The left-hand stereonet in Figure 7-10 (A), displays planes and contoured poles to gold-bearing 20 set (blue) and 70 set (red) veins for all 128 measurements. The planes of the two vein-sets have an intersection lineation of 294°/53°NW (Pettigrew 2012).

The right-hand stereonet in Figure 7-10 (B) displays the contoured poles with cylindrical best fit and resulting average planes of 20 set (blue) and 70 set (red) veins including the average intersection lineation between the two vein-sets. The actual angle between the average two veins sets is 42°.

The 20 and 70 set of veins are synchronous and have often been described as conjugate in their formation. This is borne out by their orientation as the acute angle between planes of the two veins sets range from ~25° to 50° with the average of all measured veins being 42°, right-hand stereonet Figure 7-10.

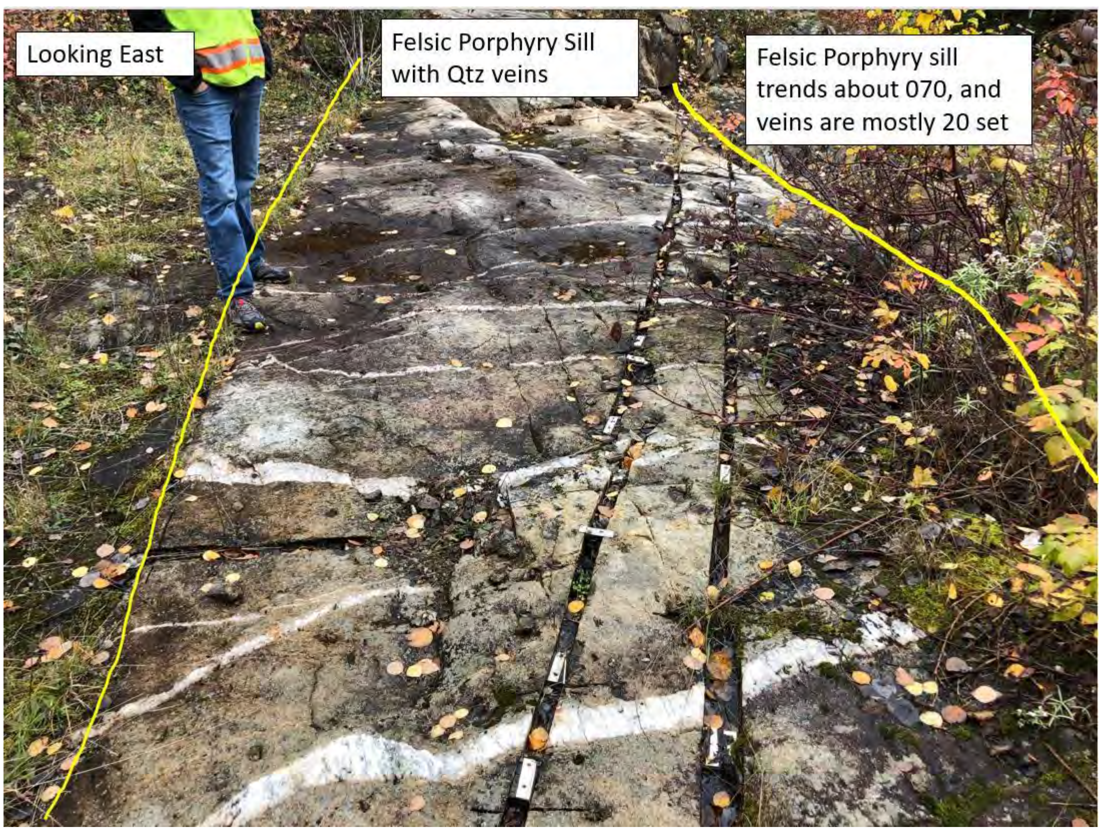
Figure 7-11 displays transverse veins (20 set) developed in a felsic porphyry sill observed in trench GDA-12-01.

Figure 7-10: Stereonets of the Planes & Poles to Gold-Bearing Veins



Source: Pettigrew (2000).

Figure 7-11: Transverse Quartz Veins developed in Felsic Porphyry Sill in Trench TR-12-01.



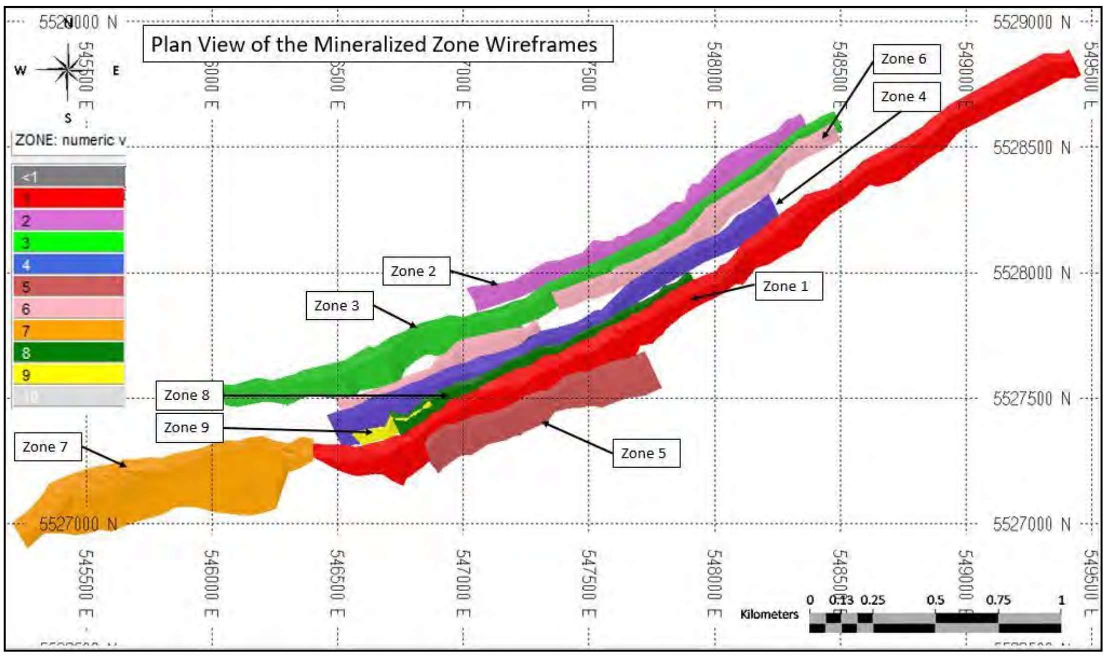
Source: CGK (2020).

The gold mineralisation has been interpreted by Miro Mytry P. Geo., of First Mining, as a series of nine northeast-trending sub-parallel zones, using a nominal 0.1 g/t Au threshold, as shown in Figure 7-12. This interpretation was prepared prior to the acquisition of Goldlund by Treasury Metals and is considered appropriate for this style of mineralisation.

Zones 1, 7, and 5 consist principally of gold mineralisation associated with the stockwork veins in the large granodiorite sills.

Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralisation that is hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only minor contribution from the granodiorite sills.

Figure 7-12: Plan View of the Goldlund Project showing Interpreted Mineralised Zones



Source: CGK (2020).

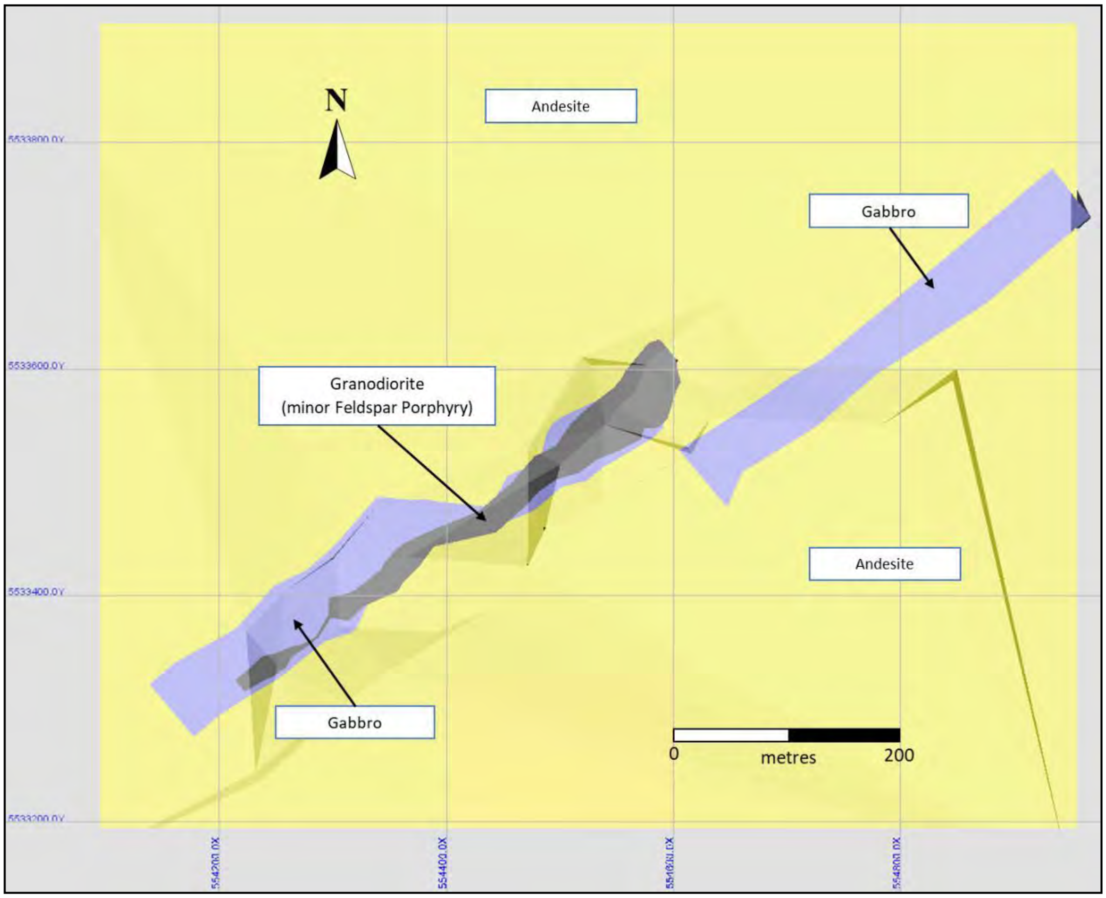
7.4 Miller Project

The Miller Project is situated approximately 8 km northeast and along strike of the Goldlund Project. The geology and gold mineralisation are similar to that of the Goldlund Project, as described in Section 7.3

The Miller deposit is at an early stage of exploration, and the geology and structural controls of the deposit and surrounding area are still under investigation.

Figure 7-13 presents a plan view of the interpreted geology.

Figure 7-13: Plan View of the Miller Project showing Interpreted Geology



Source: AGP (2020).

Similar to the Goldlund deposit, the gold mineralisation at the Miller deposit "is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. These stockwork zones form bands within the sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite" (WSP, 2020).

8 DEPOSIT TYPES

8.1 Goliath Deposit

8.1.1 Overview

In 2001, Teck-Corona originally described the Goliath deposit as a shear-hosted mesothermal gold deposit with structurally-controlled gold mineralisation related to local silica and sulphide replacements, and widespread, small, discordant to concordant quartz and sulphide veins. However, the deposit is not hosted within a shear Zone and is missing most of the critical attributes of these types of deposits. The host rocks do not contain typical iron-carbonate alteration mineral assemblages and gold is not commonly hosted by quartz veins in association with silicification (Beakhouse, 2002). Furthermore, the gold mineralisation is generally associated with highly elevated silver (locally >100 g/t Ag but varies significantly across the deposit), zinc and lead in the form of stringers and layers within felsic volcanic schist which is not common in shear-hosted mesothermal gold deposits (Page, 1995a).

Page (1995b) describes the alteration of the host rocks in the area of the deposit as being enriched in potassium and depleted in sodium, which is a diagnostic feature peculiar to volcanogenic massive sulphide (VMS) deposits. Wetherup (2008) suggested that the deposit may be part of a VMS system within a bimodal package of folded volcanic strata on the basis of this classic K-Na alteration signature along with the close association of gold with silver, zinc, and lead. No massive sulphide cap has been found to date. However, in 2012 isolated lenses of massive sulphides consisting of pyrrhotite and pyrite (no base metals) were intersected in drillholes TL12245 and TL12247 in the nose of the northeast regional fold. Although this model does not fit perfectly, it should not be dismissed as a possible mechanism in which the gold was originally introduced into the system. In addition, future exploration work should also not dismiss the possibility of perhaps finding a gold-zinc VMS deposit near surface or at depth elsewhere on the property.

Treasury favours a hybrid deposit-type model, also known as a “pre-orogenic atypical greenstone belt gold model” as a promising genetic model to explain the geology, structures and mineralisation observed within the Goliath deposit. In this model, early gold-rich volcanogenic sulphide mineralisation is overprinted by subsequent deformation and alteration events which can contribute to further concentration and/or remobilising of both precious and base metals. This model also integrates potential VMS and magmatic hydrothermal Archean lode gold deposit (magmatic hydrothermal) models in the formation of the deposit. It is likely that the Goliath deposit does not fit into any one idealised model and neither should be discounted.

8.1.2 Hybrid Deposit-Type Model of the Goliath Deposit

Hardie et al. (2012) suggested “the gold mineralisation at the Rainy River gold deposit can be interpreted as a hybrid deposit-type consisting of an early gold-rich volcanogenic sulphide mineralisation [pre-orogenic] overprinted by shear-hosted mesothermal [post-orogenic] gold mineralisation. Both styles of gold mineralisation have been progressively overprinted by deformation, whereby auriferous quartz veins post-date the sulphide stringers and veins and were emplaced during active deformation”. The presence of isoclinal folding of the pyrite-sphalerite-chalcopyrite-galena stringer veinlets gives the mineralisation a relative timing of pre- to syn-deformational. Folded mineralised stringers are found within the quartz-sericite-schist at the main deposit.

Treasury believes that there are similarities between the Rainy River deposit and Goliath and have integrated the hybrid deposit-type model into a final simplified four-stage hybrid model for the genesis of the Goliath deposit. The four stages are described below.

Stage 1: Pre-Orogenic Event. Anomalous gold, silver, zinc, and lead mineralisation is introduced as part of a VMS and/or magmatic hydrothermal system along a pre-orogenic structure consisting of stratigraphically sheared felsic volcanic (or volcanoclastic) and sedimentary rocks. If it is a VMS system, potassic alteration accompanies the mineralisation event or the felsic rocks are altered by the hydrothermal solutions moving through this conduit. Quartz and quartz-feldspar porphyries may be the heat engine, or remnants of the heat source, that drove the hydrothermal solutions as these intrusive rocks are early-stage and are folded and deformed with the rest of the rocks in subsequent deformation events. At this stage, the sericite altered weakly mineralised zone may have been several 100 m in width.

Stage 2: D_1 Deformation Event. The stratigraphic units within the deposit are isoclinally folded into an anticlinal (anticlinorium) structure whose fold axis runs east-west along the entire strike length of the deposit and plunges 10° to 20° to the east following the altered felsic volcanic rocks, which are sheared and foliated (axial planar S_1 and F_1). V1 quartz veins are formed parallel to stratigraphy.

Stage 3: D_2 Deformation Event. Northeast-striking (060°) F_2 structures intersect F_1 structures accompanied by later magmatic hydrothermal solutions which remobilise the gold, silver and base metals and re-concentrate and upgrade them within steeply west-dipping shoots that now host the “high-grade” gold and silver mineralisation. Silicification accompanies this event and V_2 quartz veins are developed. The relative abundance of base metals varies along strike depending on the original concentrations at different locations along the initial shear structure.

Stage 4: D3 Deformation Event. Brittle faults, fractures and white non-mineralised V3 quartz veins form (dip moderately north-northeast) and crosscut or follow local foliation.

8.2 Goldlund Deposit

The following has been summarised from previous technical reports, including the 2020 Treasury Metals Technical Report.

The Goldlund Project hosts Archean, shear zone-hosted quartz vein mineralisation (Archean lode-gold), occurring as extensional quartz vein systems, particularly between rocks with high competency contrast. Archean lode-gold deposits occur in a broad range of structural settings, and at different crustal levels, but they share a similarity in ore fluid characteristics. Mineralisation is typically late tectonic, occurring after the main phases of regional thrusting and folding, and generally late-syn to post-peak metamorphism with most of the significant deposits in areas of greenschist facies. Many deposits are related to the reactivation of earlier structures.

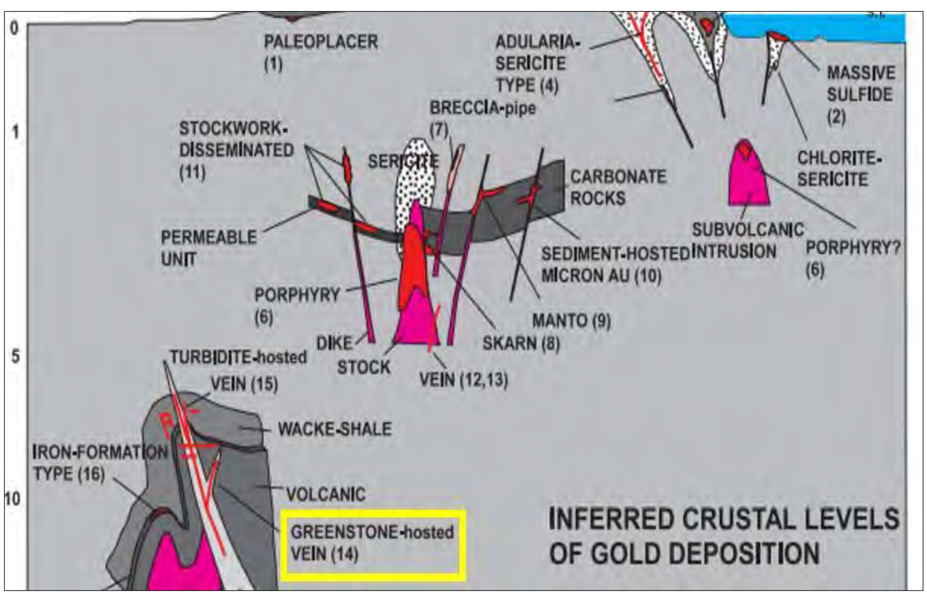
Archean lode-gold occurrences are common in the Sandybeach Lake – Sioux Lookout area and are concentrated in the Southern and Central volcanic belts. Vein systems in both belts are the product of Stage 3 deformation and are related to the northeast-southwest extension associated with northwest-southeast compression and shortening; the brittle-ductile deformation near the steep, northeast-trending shear zones; and the tightening of the Stage 3 folds.

Gold-bearing vein systems in the Southern Volcanic Belt are typically controlled by the steep, Stage 3 northeasterly-trending shears. The host mafic metavolcanic rocks are typically chlorite-ankerite schists up to several metres in width. Pyrite, with subordinate chalcopyrite, sphalerite, and galena, are the main sulphide minerals in the auriferous veins.

A few shear zone hosted gold occurrences are also present in the Central Volcanic Belt, but the dominant and economically most significant type are the transverse vein systems within competent rocks, particularly in the intermediate to mafic meta-subvolcanic intrusive sills. Vein systems occupy tensional fractures related to internal deformation of the competent units as folds tightened during Stage 3 deformation. Vein arrays could be expected to develop near fold hinges, within fold limbs, and along axial planar foliations. The orientations of individual veins within the arrays are affected by their locations within the folds.

The gold mineralisation at Goldlund has similarities to the Buffalo Gold deposit in Red Lake, Ontario and the Sigma Mine in Val-d'Or, Quebec (Pettigrew, 2012). In 1997, Robert, Poulsen, and Dube' classified the Sigma Mine as a greenstone-hosted quartz-carbonate vein deposit, that occurs within greenstone-belts spatially associated with major fault zones. The quartz-carbonate veins are associated with brittle-ductile shear zones. Figure 8-1 shows a schematic representation of the crustal levels inferred for gold deposition for the commonly recognised deposit types. The depth scale (left-hand side of the drawing) is approximate and logarithmic. The greenstone-hosted quartz-carbonate vein deposit is labelled as 14 and highlighted with a yellow box. This gold deposit type forms at a depth of approximately 10 km.

Figure 8-1: Schematic of Representation of Gold Deposit Models



Source: Robert, Poulsen & Dube (1997).'

8.3 Miller Deposit

The following is taken from WSP (2020).

The identified mineralisation fits an Archean shear Zone hosted quartz vein model (Archean lode gold). The Archean lode gold occurrences are common in the Sandy Beach Lake – Sioux Lookout area and are concentrated in the Southern and Central volcanic belts. Vein systems in both belts are the product of Stage 3 deformation and are related to:

- northeast-southwest extension associated with northwest-southeast compression and shortening
- ductile-brittle deformation near steep northeast-trending shear zones
- tightening of Stage 3 folds

Vein systems in the Southern Volcanic Belt are typically controlled by the steep, Stage 3 northeasterly-trending shears. Host mafic rocks are chlorite-ankerite schists up to several metres in width. Pyrite, with subordinate chalcopyrite, sphalerite, and galena are the main sulphide minerals in auriferous veins.

A few shear Zone hosted occurrences are also present in the Central Volcanic Belt, but the dominant, and economically most significant type, are transverse vein arrays within competent rocks and particularly the intermediate to mafic sub-volcanic intrusive sheets. Vein systems occupy tensional fractures related to internal deformation of the competent units as folds tightened during Stage 3 deformation. Vein arrays could be expected to develop near fold hinges, within fold limbs, and along axial planar foliations. The orientations of individual veins within the arrays are affected by their locations within folds.

9 EXPLORATION

9.1 Overview

The text for this section was sourced primarily from Treasury Metal's "Drilling and Exploration Assessment Report" (P. Dunbar and A. Larsen, 2014-2015); P&E's 2019 NI 43-101 Technical Report; and various Treasury Metals press releases, with edits from AGP.

9.2 Goliath Deposit

Since 2008, Treasury Metals has focused its exploration work on the western half of the property in order to evaluate the gold potential of the Goliath deposit. During this 12-year period, exploration activities consisted of re-establishing the former Teck exploration grid, geological mapping and sampling, prospecting, the completion of structural studies, trenching and channel sampling, the completion of a ground IP geophysical survey and two airborne geophysical surveys, downhole IP and tomography surveys, metallurgical testing, mineral resource estimations of the main deposit (including Preliminary Economic Analyses in 2012 and 2017) and the completion of 18 diamond drilling programs (see Table 9.1).

The 2008, 2009 and 2010 exploration programs were conducted and managed by Caracle Creek International Consulting Inc. (CCIC) of Toronto, Ontario. Treasury Metals personnel assumed field management all exploration activities as of February 2011.

The exploration work completed on the property has been documented in a number of independent technical reports prepared for the Company and is summarised below (Puritch et al., 2015; Roy et al., 2012; Roy and Trinder, 2011; Roy and Trinder, 2008). Assessment reports filed with the Ministry of Northern Development and Mines ("MNDM") provides additional information on their exploration activities. The reader is directed to Section 10 for details regarding the diamond drilling programs completed by Treasury Metals from 2008 to 2020. Table 9.1 provides a summary of the exploration work conducted by Treasury Metals from 2008 to 2020.

9.2.1 2008 Exploration Activities

9.2.1.1 Historic Core Reclamation

In 2008, all historical Teck drill core was in a locked, long-term storage compound behind a chain-link fence across from the Pine Grove Motel in the town of Wabigoon (approximately 20 km east of Dryden, Ontario). According to Wetherup and Kelso (2008), approximately 8,000 boxes (one third of the core) were stored outside on metal racks and open to the elements (sun, rain, snow, etc.). These boxes were in poor condition and required re-boxing before they could be moved or re-examined. The remaining core boxes (around 16,000) were cross-stacked onto wooden pallets with approximately 100 core boxes per pallet. These boxes were in various states of decay from moderate to nearly completely rotted through.

Whatever core could be salvaged was moved to Treasury Metals' core office facility at the former Tree Nursery, where it is now in long-term storage outside at their core farm.

Table 9.1: Exploration Activities from 2008 to 2020

Year	Company	Work Completed
2008	CCIC	Core reclamation: exploration grid cut (65.9 line-km)
	CCIC	Geological mapping (1:5,000 Scale), 32 grab samples collected including 17 whole rock and REE analyses
	CCIC	Diamond drilling program – 55 holes (TL0801 to TL0855)
2008	CCIC	Structural study on 2008 drill core
	CCIC	One Main Zone trench, 10 Channels, 29 samples, channel sampling iron formation (3 channels, 25 samples) + mapping
	Firefly Aviation Ltd.	Aeromagnetic (HRAM) survey, 309 line km covering 3,064 ha
	JVX Geophysical Surveys & Consulting	Ground IP/resistivity survey, 29.6 line-km covering 230 ha
	A.C.A. Howe International Limited	Mineral resource estimate (N.I. 43-101 compliant)
2009	CCIC	Prospecting, sampling and mapping program covering nine legacy claims; outcrop sampling (5 grabs) and channel sampling (34 channels, 115 channel samples)
	CCIC	Diamond drilling program – 31 holes (TL0956 to TL0986)
2010	CCIC	Downhole DCIP/resistivity EarthProbe survey; 60 holes profiled; 94 hole-to-hole tomography imaging; 4-line, 21 surface-to-hole tomography pairings; petrographic/ SEM Study (Beakhouse, 2010); SCIP core testing
	CCIC	3 phase diamond drilling program – 32 holes (TL1087 to TL10118)
	CCIC	Trenching of Main Zone, mapped and channel sampled, 47 channel samples, 2 duplicate channels, 4 geological units mapped
	A.C.A. Howe International Limited	Updated resource estimate & preliminary economic analyses
	Ministry of Northern Development and Mines	Petrographic and scanning electron microscopy
2011	Treasury Metals	Diamond drilling program – 111 holes (TL11119 to TL11229)
	G & T Metallurgical Services Limited, B.C.	Preliminary metallurgical test program, 59 kg composite sample; grindability, gravity and cyanidation testing
	Fugro Airborne Surveys	DIGHEM EM & magnetic survey (July), helicopter, 582.62 line-km
	A.C.A. Howe International Limited	Updated resource estimate (N.I. 43-101)
2012	G & T Metallurgical Services Limited, B.C.	2 Tests: gravity + cyanidation and just cyanidation (48 hours); Sample size 398.5 kg, ½ diamond core, 163 samples
	Treasury Metals	2 phase diamond drilling program – 81 holes (TL12278 to TL12295; 15 re-entry holes)
	Treasury	Goliath 3D inversion study (Ellis, 2012); petrographic work
	A.C.A. Howe International Limited	Preliminary economic analyses (using 2011 Resource Estimate)
	Vancouver Petrographic	This section study on mineralised drill core
2013	Treasury Metals	Diamond drilling program – 48 holes (TL13296 to TL13336; 7 Re-entry holes)
2014	Treasury Metals	2 Phase diamond drilling program – 48 holes (TL14337 to TL14377; 5 re-entry holes, 3 wedges and 1 abandoned hole)
	Treasury Metals	Soil mobile metal ion survey (MMI) – property-wide survey
	Gekko Systems Pty Ltd (Australia)	Leach optimisation testwork and bulk concentrate production; cyanide detox testwork; high-grade and medium-grade ore testwork (gravity, flotation, cyanide leach recovery)
2015	Treasury Metals	Diamond drilling program – 27 holes (TL14378B, TL15379 to TL15402; 2 re-entry holes); infill core sampling program (95 holes, 2,091 samples); cyanide bottle roll testing program (19 holes, 391 samples).

Year	Company	Work Completed
	P & E Mining Consultants Inc.	Updated mineral resource estimate (N.I. 43-101)
2016	Treasury Metals	Diamond drill program – 19 holes (TL16403 to TL16420), 1 wedge hole (TL16-415W1)
	Treasury Metals	Condemnation field mapping program – 146 grab samples (G156001 to G156146), 15 channel samples (C156351 to C156365), 7 coarse blanks and 7 standards (CDN-CM-26) were used during the sampling. Covers an area of approximately 1.4 km^2.
	Treasury Metals	Eastern alteration corridor mapping and sampling program
	Treasury Metals	Gossan showing mapping and sampling program
2017	Treasury Metals	2 phase diamond drill program – 43 holes (TL17421A to TL17463)
	Treasury Metals	Iron formation mapping program – 36 grab samples, in addition to 2 coarse blanks and 2 standards (CDN-CM-26 & CDN-GS-1P5K) were used during the sampling. Covers an area of approximately 5 km^2
	Treasury Metals	Outcrop mapping program (western map area, northwest map area, Central map area, Eastern map area)
	Treasury Metals	Infill sampling program – 5256 Samples (across 142 drillholes), including 525 blanks and standards.
2018	Treasury Metals	Diamond drill program – 38 holes (TL18464 to TL18501)
	Treasury Metals	Soil gas hydrocarbon sampling program – 845 soil samples. Covers an area of approximately 9.88 km^2
	Knight Piésold Consulting Ltd.	Geotechnical drill program – 20 holes. Covers an area of approximately 2 km^2.
2019	Golden Mallar Corp	Hole to hole spectral induced polarisation/resistivity survey
	Treasury Metals	Soil gas hydrocarbon sampling program – 1,040 soil samples. Covers an area of approximately 10.25 km^2
	Treasury Metals	Diamond drill program – 12 holes (TL19502 to TL19513)
2020	Treasury Metals	Diamond drill program – 15 holes (TL20514 to TL20528)
	Axiom Exploration	Soil gas hydrocarbon sampling program – 1,260 Soil Samples. Covers an area of approximately 12.50 km^2

Source: P&E (2015), AGP (2020).

Records show that CCIC recovered around 65% or 13,723 boxes out of a possible 21,070 boxes of historical Teck drill core. After moving the core to the Exploration Office in 2011, it was found that the drill core was in such a poor condition that it precluded a resampling program. Some photographs were re-covered of a few drillholes (TL1, TL4 and TL46) documenting the condition of the core, but no other information is available. There is no information on the state of the Laramide drill core (holes G-1 to G-8) or if that core was also recovered.

9.2.1.2 Geological Mapping Program

An exploration grid was cut in January 2008 to facilitate geological mapping, sampling, ground geophysical surveys, trenching and diamond drilling programs. A total of 69.5 line-km were cut with the base line established along Norman Road which represented the former border between the old Laramide and Teck properties. Grid lines were cut at 50 m intervals perpendicular to the baseline in an attempt to establish or mimic the former Teck grid. The grid consisted of 30 lines at approximately 1,500 m length, 11 lines at 1,225 m, and five lines at 1,025 m.

Geological mapping, at a scale of 1:50,000, was completed between June and August 2008. Major lithological units were identified, structures interpreted, and a new geological map of the property was completed (see Figure 9-1 overleaf). Thirty-two representative and grab samples were taken (Ilieva and McKenzie, 2009), and 17 samples were sent to Accurassay Laboratory in Thunder Bay for fire assay, whole rock and rare-earth element (REE) analyses. None of the samples returned any significant precious or base metal assays.

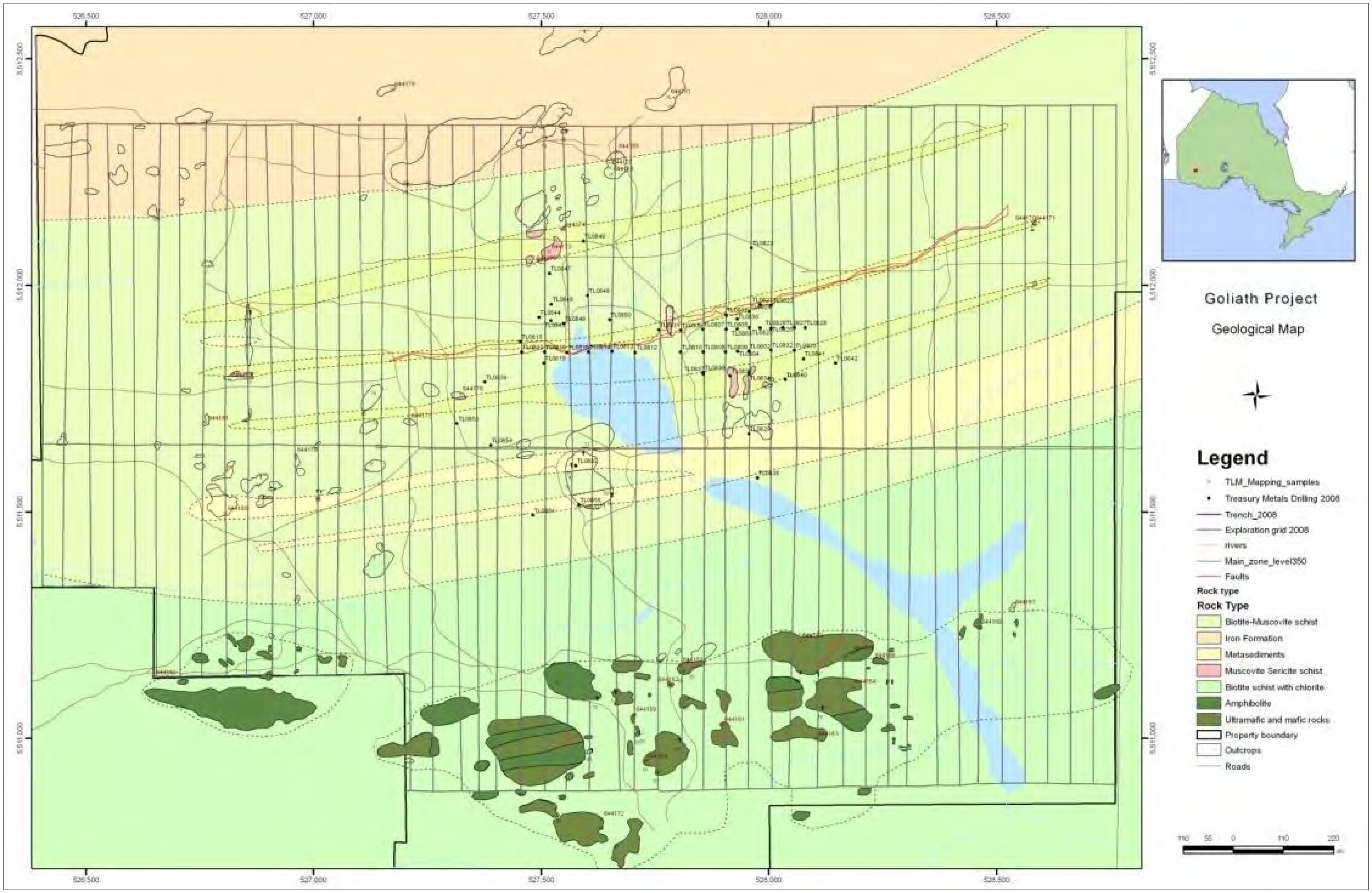
9.2.1.3 Structural Geology Study

CCIC was retained by Treasury Metals to review both the geological and structural data on its Thunder Lake property (now the Goliath property) and prepared a report containing a structural description and interpretation of the geology (Wetherup, 2008). Three different generations of folds and deformational events were described (see Table 9.2).

Oriented core was used during the 2008 diamond drilling program for the first time to collect additional structural data (Roy et al., 2012). Core from drillholes TL0822 to TL0837 was used for this study. Foliation, geological contacts, fault lines and fold axes were measured using an Ezy-Mark™ core orientation tool provided by BoreInfo Ltd. (BoreInfo). The purpose of this program was to clarify the spatial relationships between the structural features and their influence on the mineralisation.

CCIC observed that the F_2 folds (axial planes) upgrade gold mineralisation within the Main Zone and that gold is focused in shoots where F_1 and F_2 structures intersect and where F_2 structures are concentrated (in the shoots). Shoot structures are steeply plunging (west as observed on current Treasury Metals longitudinal sections of the Main and C Zones). In addition, it should be noted that the zones of alteration and gold mineralisation strike more northerly and assume a northeast strike east of the deposit and are nearly parallel to the strike of the F_2 axial planes. Therefore, it was concluded that it might be more difficult for exploration drilling to locate and intersect gold-bearing shoots in this region.

Figure 9-1: Geological Grid Map (Goliath Deposit Outlined in Red)



Source: Treasury Metals (2015).

Table 9.2: Summary of Structural Features Observed on the Goliath Deposit

Vent	Structure	Description	Veins	Description
D_0	S_0	Compositional layering of meta-volcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	Greyish, highly deformed, S_1 foliation parallel quartz-sulphide ribbons and silicification surrounded by quartz-sericite schist
D_1	F_1 S_1	Isoclinal folding F_1 axial planar and layer parallel foliation/schistosity	V_1	White deformed, locally crosscutting quartz+/-tourmaline+/-sulphide veins
D_2	F_2	Closed (60º) folds; axial planes ~045/90º; discrete, 50-40 m spaced, axial planes	V_2	Weakly deformed white quartz+/-sulphide veins along F_2 axial planes & at 45° to F_2 axial planes.
D_3	NW Fault	Brittle faults/fractures dip moderately NNE	V_3	Un-deformed white, non-planar quartz veins dip moderately NNE and follow foliation locally

Source: Treasury Metals (2015).

9.2.1.4 Exploration Trench & Channel Sampling

9.2.1.4.1 Main Zone

A 1,005 m long trench, oriented north-south, was excavated in September 2008 to expose auriferous “Main Zone” mineralisation intersected by diamond drilling within the Goliath deposit (Ilieva, 2009). The trench, located at UTM 527782E, 5511893N (NAD 83, Zone 15N), is an elongated oval shape and measured at surface 46 m in length, 14 to 15 m wide and 5 m deep. A decline was added at the southern end of the trench for easier access.

Two outcrops were exposed and geologically mapped at a scale of 1:200 and channel sampled perpendicular to strike. The bedrock geology was described as strongly altered (sericitised) volcanic rocks. Ten channel samples (designated Channel 1 to 10) were cut across the two exposures and 29 samples were collected. Each channel is approximately 4 to 5 cm wide and 5 to 6 cm deep (Roy and Trinder, 2008). A blank or standard was inserted in alternating order at every tenth sample. All samples were dispatched to Accurassay for gold and base metal analyses.

Two zones of mineralisation were exposed in Channel 3 and Channel 5 located about 2.5 m to the south. Channel 3 (Sample 644112) returned the highest gold value of 27.55 g/t Au and 2.19 g/t Ag over a sample length of 0.65 m (see Table 9.3). A 1.5 m lower-grade mineralised interval was also sampled in Channel 5 where samples 644115, 644116 and 644117, each 0.5 m in length, returned 1.75 g/t Au, 2.74 g/t Au and 1.03 g/t Au, respectively.

Table 9.3: Channel Sampling (2008) Significant Assay Results

Channel	Sample Number	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
3	644112	0.65	27.55	2.19	43	98	34
5	644115	0.50	1.75	3.70	145	280	351
5	644116	0.50	2.74	3.78	48	346	386
5	644117	0.50	1.03	1.97	39	92	87

Source: Treasury Metals (2015).

9.2.1.4.2 Iron Formation: Tree Nursery Road

Three channels were cut across a bedrock exposure of iron formation that outcrops on either side of Tree Nursery Road located at in Zealand Township (UTM 528767E, 5513144N; UTM 528803E, 5513165N; UTM 528802E 5513155N, NAD83, Zone15N). Twenty-five channel samples were collected and dispatched to Accurassay in Thunder Bay for analyses for gold, base metals, and trace element geochemistry (31 element package). Only one sample returned gold value in excess of 0.2 g/t Au.

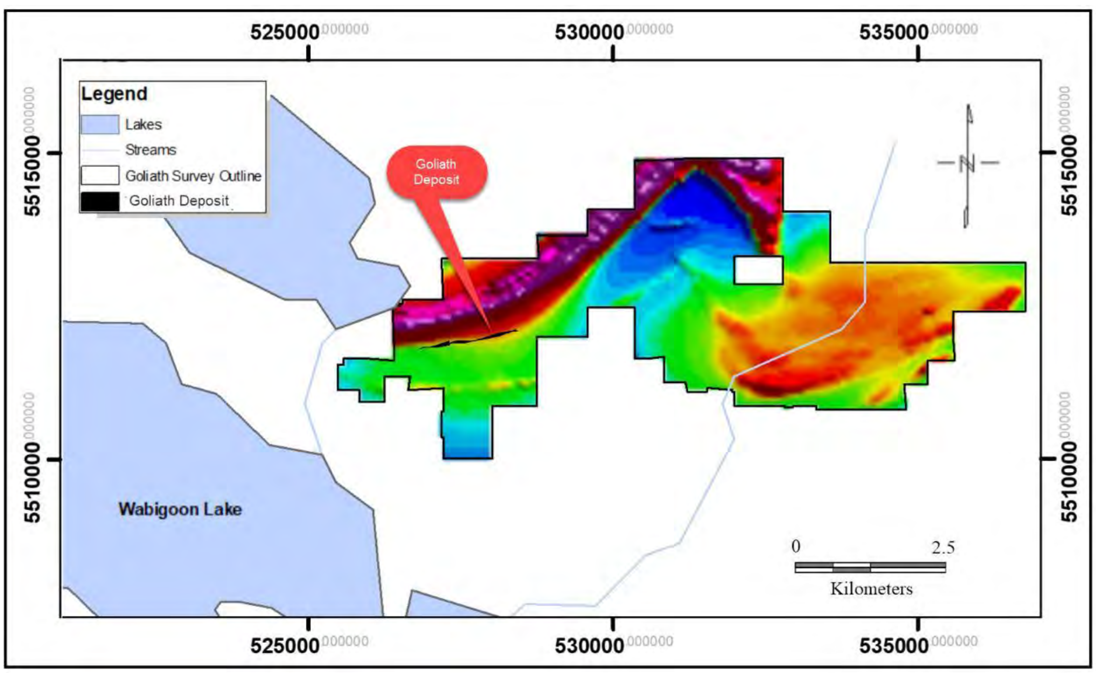
9.2.1.5 Geophysical Surveys

9.2.1.5.1 Aeromagnetic (HRAM) Survey

Considering that approximately 70% of the Goliath property is covered by glaciofluvial outwash, and that overburden can range in thickness from a few metres to over 40 m thick, CCIC concluded that an airborne magnetic survey was required to identify the regional bedrock geology and structure.

A high-resolution aeromagnetic survey (HRAM) was completed by Firefly Aviation Ltd. (Firefly) during March 2008. A total of 2,165 line km were flown by fixed wing aircraft covering an area of 180 km^2 (see Figure 9-2) North-south survey lines were flown at 100 m spacing and east-west tie lines flown every 500 m covering a large area of Zealand and Hartman Townships and the southern portions of Brownridge and Laval Townships (Evans, 2008). Standard and enhanced gridding filters were applied to the Goliath survey data based on the calculated international geomagnetic reference model (IGRF). This survey was conducted using a NAD83, Zone 15 projection and datum.

Figure 9-2: 2008 Firefly Geophysics Total Magnetic Field Intensity Map



Source: Modified from McKenzie (2008)

According to McKenzie (2008), the data was subsequently interpreted by Balch Exploration Consulting Inc. (BECI). The bedrock underlying the survey area reflects the typical magnetic signature of a regional greenstone belt which is expressed as a large arcuate high/low sequence reflecting the magnetite precipitated during and after formation along with subsequent tectonic deformation. However, the Goliath deposit is not detected on the airborne magnetic survey and actually occurs in a magnetic low. The property is underlain by large scale synclinal and anticlinal folded structures and it was concluded that the magnetic data provides a better understanding of the F_1 fold architecture. Secondary F_2 structures, believed to be responsible for upgrading concentrations of both gold and silver at Goliath, are not identified by the survey results. A regional thrust fault is mapped throughout the southern extent of the survey. This is coincident with a string of discrete magnetic bodies occurring along the trace of the fault.

9.2.1.5.2 Ground Induced Polarisation/Resistivity Survey

JVX Geophysical Surveys and Consulting (JVX) was contracted by Treasury Metals to conduct 29.6 line-km spectral IP/resistivity survey on the Goliath project grid from March 31 to May 1, 2008. The maximum vertical depth of penetration of this survey was approximately 60 m (Palich, 2010b). This grid covered the main resource area for a strike length of approximately 2.0 km. The exploration grid consisted of 21 north-south oriented lines at 100 m spacing plus two line segments from stations 750S to 750N. The survey instrumentation consisted of a Scintrex IPC-7 (2.5 kW) transmitter and Scintrex IPR-12 receivers. Surveys were completed in time domain with a pole-dipole array (‘a’ =25 m, n=1 to 8).

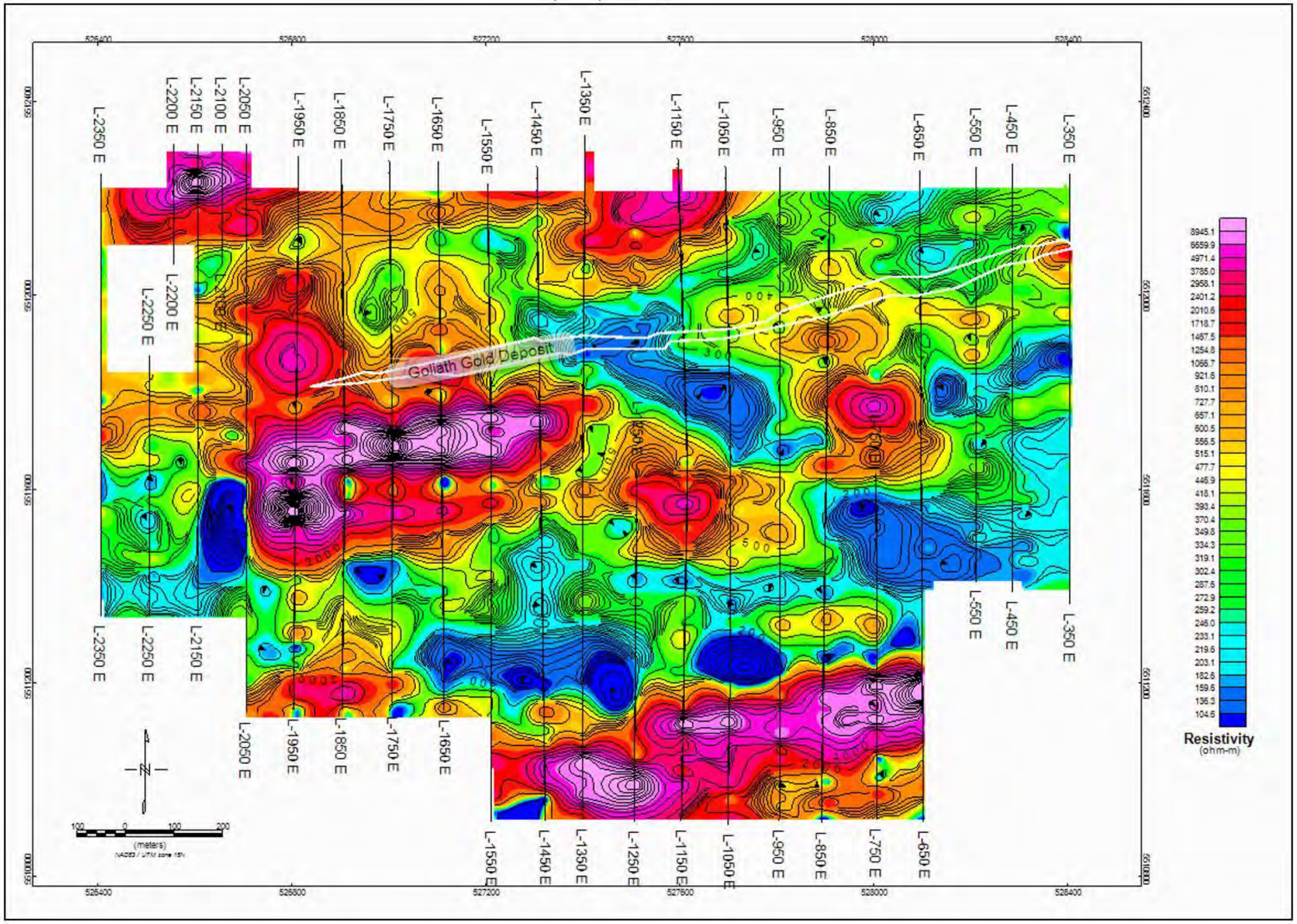
The contract stated that ground magnetic data would also be collected. However, due to time constraints, including poor weather, the deep IP and ground magnetic surveys were not completed (McKenzie, 2008). Plan maps at the scale of 1:5,000 resistivity (n=2) are presented in Figure 9-3.

It was determined that much of the survey area is covered by extensive surficial overburden with 43% of the survey area at 250 Ωm or less. Conductive overburden can mask chargeable bodies thus requiring a high percentage of sulphide mineralisation to overcome this problem. However, JVX noted that despite the presence of conductive overburden, the conductivity was not as high as initially anticipated (Johnson and Webster, 2008).

The Goliath deposit is marked by weak resistivity highs in an area of predominantly low resistivity. Overall, the main gold deposit has a weak and uncertain IP/resistivity expression. It appears to be defined by three marginal IP anomalies associated with relative resistivity highs. This signature does not improve to the east or west of the deposit. South of the deposit, there is a coinciding chargeability and resistivity anomaly in the western portion of the deposit between lines L1950 to L450. A possible northwest-trending fault was also identified by the survey.

A series of pseudo-sections were also generated at the scale of 1:2,500 and can be found in the JVX report. Examination of these sections identified a possible northwest-trending fault and low values of chargeability which was interpreted to possibly displace the mineralisation in a west-northwest direction (Ilieva and McKenzie, 2009). Seven IP anomalies were defined for possible follow-up exploration work and CCIC recommended that the data be inverted for proper 3D interpretation of the IP survey results (see Table 9.4).

Figure 9-3: 2008 JVX Ltd. Resistivity (n=2) Map, Goliath Property*



Source: Ilieva and McKenzie (2009).

Table 9.4: 2008 IP Survey Targets Selected for Further Investigation

Anomaly ID	Easting*	Northing	Comments
TL_0001	526661	5512237	Cluster of strong IP anomalies at north end of lines 2050W, 2200W; Shallow; N1 resistivities are moderate to high; Short time constants - response of fine-grained disseminated sulphides (+gold)
TL_0002	526908	5511224	Very strong, shallow IP anomalies 0 part of 300 m long IP zone with weaker end members that may define an east/west IP zone that crosses entire grid; Coincident lower resistivities at depth may indicate a partial cause by bedrock conductors; Strong IP anomalies noted - masked by conductive cover - short time constants upgrading for gold target
TL_0003	527010	5511629	Stronger of two IP anomalies - lower resistivity at depth - possible bedrock conductor - time constant uniformly long
TL_0004	527009	5511705	Part of 400 m long IP zone - may be on strike with Thunder Lake gold deposit; Moderate resistivity noted - possible bedrock conductor
TL_0005	527507	5512155	Two nearby strong, shallow IP anomalies 250 m north of Thunder Lake. N1 resistivities are moderate. Some outcrop/subcrop and a prospecting history are likely. Time constants are long or mixed
TL-0006	528006	5511247	One of two strong IP anomalies south of the Thunder Lake deposit; Part of east-west-trending IP/resistivity zones; Interpreted as probable bedrock conductors; This anomaly portion has short time constants and high resistivities - interesting for gold; N1 resistivity is high suggesting thin overburden
TL_0007	528006	5511021	One of two strong IP anomalies south of the Thunder Lake deposit; Part of east-west-trending IP/resistivity zones; Interpreted as probable bedrock conductors

Note: *Coordinates: UTM NAD83, Zone 15N Datum. Source: Treasury Metals (2015).

9.2.2 2009 Exploration Activities

In 2009, general reconnaissance prospecting and some focused stripping and channel sampling, was completed by CCIC from July 6, 2009 to August 4, 2009. A small grid was cut and geologically mapped on the Collins Patent and the remaining work was concentrated on the Jones, Johnson Patent and 12 legacy claims.

Five grab samples were collected during the prospecting exercise, 22 channel samples collected from three stripped outcrops on legacy claim 1119562 and 93 channels collected from two stripped outcrops located just east of Tree Nursery Road near the power lines on the Johnson Patent (Parcel 15401) in Zealand Township, Lot 5, Concession 4.

Three samples returned significant gold assays from this program. The best gold assay was obtained from sample 59109 that assayed 20.519 g/t Au over a channel sample length of

1.0 m on the Johnson Patent. The host rock is described as a biotite-muscovite schist containing 1% to 2% sulphides and is identified by Treasury Metals as an outcrop exposure of Zone D just east of Tree Nursery Road, west of the hydro line. A second channel was cut directly adjacent to sample 59109 over a sample length of 1.0 m. That sample was subsequently cut into five 20 cm samples to isolate where the gold was concentrated. One of these samples C59139 returned 3.296 g/t Au over a sample length of 0.20 m. One grab sample from the reconnaissance prospecting program returned 2.14 g/t Au. However, the location of this sample was not disclosed in the memo-style report.

During the month of July, three and a half days were spent completing general reconnaissance prospecting, outcrop sampling and a channel sampling program to generate future exploration targets for geological mapping and sampling on legacy claim 4211252 (CCIC 2009b). Work was focused in Lot 1, Concession II within the southern portion of Zealand Township.

A detailed grid was set up over an outcrop area where five outcrops were exposed, and a 100 m long east oriented baseline and cross lines were established, and the lines were mapped at a scale of 1:500. The area was found to be underlain by predominantly meta-sedimentary rocks with lesser amounts of felsic volcanic/quartz porphyry rocks. A total of 24 channel samples, ranging from 0.3 to 1.0 m in length, were taken from five distinct outcrops with interesting mineralisation (quartz veins with elevated pyrite) and dispatched to Accurassay for gold analyses. There are no individual sample descriptions of the mineralisation. None of the samples returned any significant gold assays (best 5 ppb Au).

In July 2009, a reconnaissance prospecting program was conducted to ascertain the geology underlying legacy claim 4211250 (CCIC, 2009c). A total of 1.5 line-km were traversed in Lot 9, Concession II within the southern portion of Zealand Township. Only one large outcrop ridge was encountered on the traverse which appeared to be an unmineralised granitoid intrusive rock. No samples were taken.

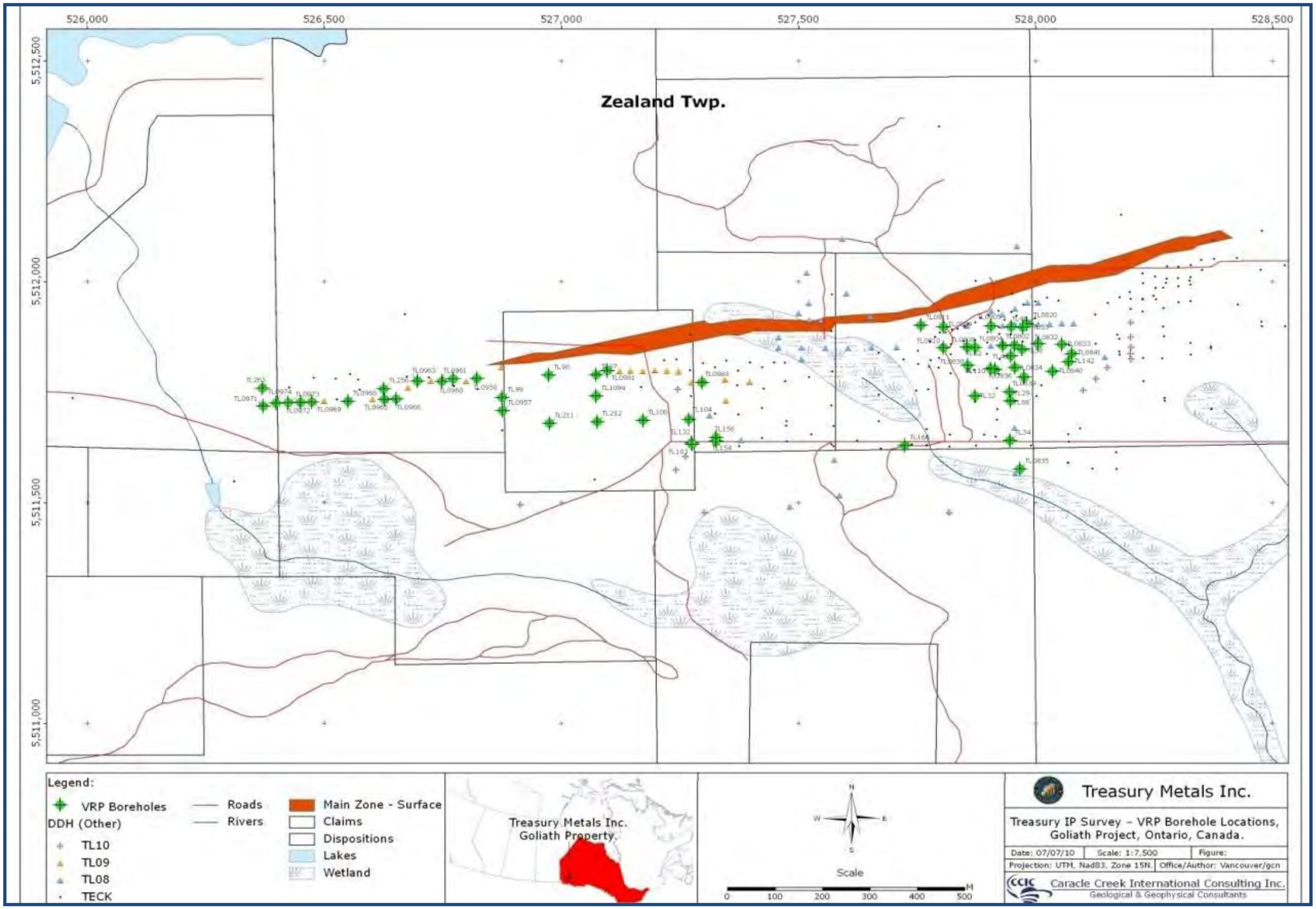
9.2.3 2010 Exploration Activities

9.2.3.1 Ground Geophysical Surveys

A downhole direct current induced polarisation (DCIP/resistivity) survey was completed by CCIC over a 24-day period in the spring of 2010 (Palich, 2010). The survey consisted of 60 holes profiled for vertical resistivity/chargeability and 94 hole-to-hole tomography images between holes up to 150 m separation (see Figure 9-4). Four surface lines with 21 surface-to hole tomography pairings were also completed. The survey was designed to:

- characterise the resistivity/chargeability signatures of rock types and ore zones
- determine if zones containing significant concentrations of gold can be isolated with distinct geophysical signatures
- test if a new CCIC IP/resistivity technology called EarthProbeTM was capable of imaging between drillholes

Figure 9-4: Vertical Resistivity Probe & Tomography Drillhole Locations



Source: Treasury Metals (2010).

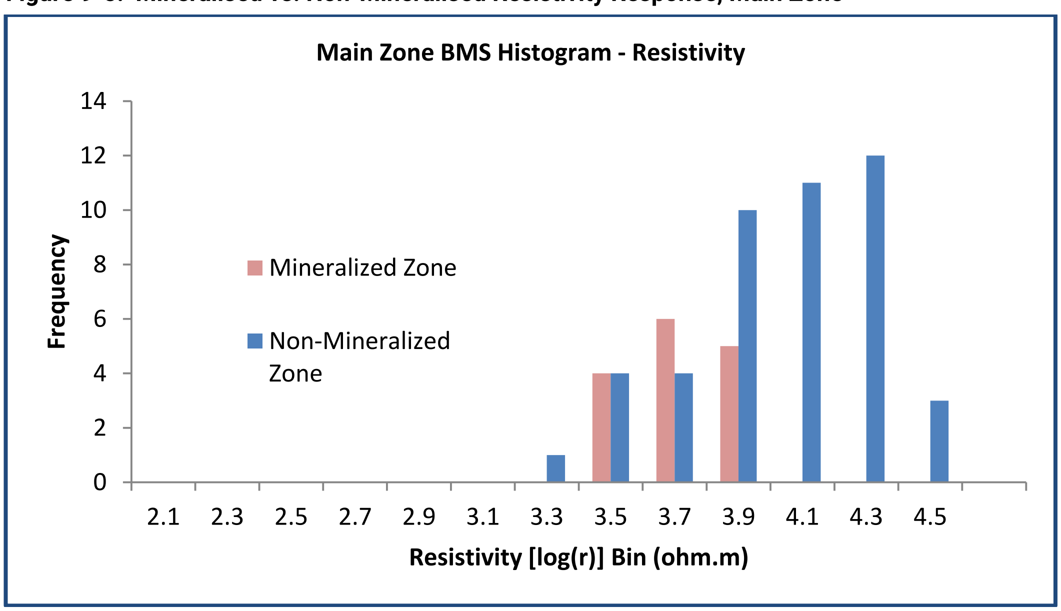
The EarthProbe™ survey method utilises closely spaced electrode at 5 m separation distances to a centralised data acquisition system that enables arbitrary selection of current and potential electrodes through relays (Roy and Trinder, 2011). Rapid data acquisition and signal processing techniques allow for efficient use of conventional and non-conventional arrays and the removal of natural and cultural noise. The result is a high resolution DCIP system able to delineate both large resistivity/chargeability anomalies and narrow structural features down to depths of approximately 240 m (Roy et al., 2012).

9.2.3.2 Resistivity/Chargeability Correlations

CCIC identified seven distinct resistivity/chargeability correlations from the DCIP survey (Palich, 2010), as follows:

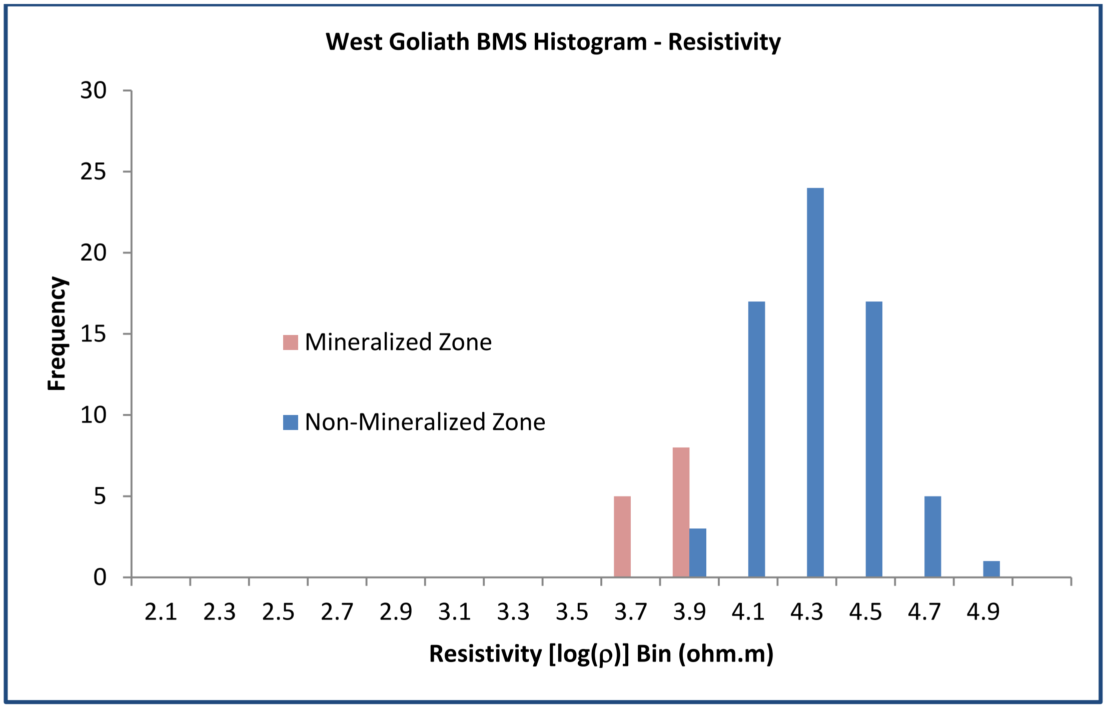
- mineralised zones exhibit low resistivity and high chargeability
- different DCIP signatures between Main Zone and West Goliath extensional area
- resistivity responses greater than 7,900 Ω.m (3.9 log Ω.m) reflect non-mineralised zones (see Figures 9-5 and 9-6)
- resistivity responses less than 5,000 Ω.m (3.7 log Ω.m) reflect mineralised zones (Figures 9-5 and 9-6)
- chargeability responses less than 30 mV/V in the Main Zone and less than 50 mV/V in the West Goliath extensional area reflect non-mineralised zones
- chargeability responses greater than 50 mV/V reflect mineralised zones
- there is overlap of resistivity and chargeability response between the mineralised and non-mineralised zones in the Main Zone, suggesting that the occurrence of gold may be controlled by multiple factors (e.g., several alteration types) each having a unique IP signature

Figure 9-5: Mineralised vs. Non-Mineralised Resistivity Response, Main Zone



Source: Treasury Metals (2015).

Figure 9-6: Mineralised vs. Non-Mineralised Resistivity Response, West Goliath Extension



Source: Treasury Metals (2015).

9.2.3.3 Mineralisation Response Signature

CCIC characterised the following three mineralisation responses from the survey (Palich, 2010):

- Anomalous resistivity responses occur in association with mineralised zones that are greater than 4.0 m thick and exhibit a gold grade greater than 2 ppm.
- An anomalous resistivity response does not occur if the thickness of the mineralised zone is less than 2.0 m unless the intersection is in close proximity (less than 5.0 m) to a thicker mineralised zone.
- An anomalous resistivity response typically does not occur if the thickness of the mineralised zone is less than 4 m unless the gold grade exceeds 2 ppm and zinc exceeds 2,000 ppm.

9.2.3.4 Anomaly Summary

CCIC summarised the anomaly findings as follows (Palich, 2010):

- Numerous in-hole and off-hole low resistivity responses were identified.
- Main Zone: A high level of electrical continuity existed between known gold intersections suggesting that mineralisation is continuous.
- West Goliath extensional exploration area: Vertical resistivity probe and tomography results were well correlated with known mineralisation zones showing limited additional extent

from previously drilled intersections. A shallow conductor (50-70 m) was identified near drillholes TL0965, TL0966, TL0968, TL0969 and TL0972.

- Four low resistivity anomalies were identified from the surface survey. At least one of these anomalies is beyond the western extent of existing drilling

The DCIP survey was not correlated to the sericite alteration zones. CCIC recommended completing that correlation as well as characterising the bulk resistivity/chargeability using the entire vertical resistivity probe and drillhole assay dataset (Palich, 2010). They also recommended compiling the special resolution of the resistivity responses into a format that could be overlain with the existing 3D model of the deposit and drilling four IP anomalies identified in the West Goliath extensional exploration area.

9.2.3.5 SCIP Core Testing

CCIC collected 79 sample core induced polarisation (SCIP) readings on limited intervals of mineralised core form three 2008 drillholes in early August 2010 (Palich, 2010b). They also compared the results of the 2010 EarthProbe™ IP survey to the 2008 JVX traditional IP survey. The results of this work are summarised below. SCIP core test readings were collected using a GDD SCIP Rx 8-32 unit as follows:

- Hole TL0802: 38 reading were taken of mineralised BMS between 121.1 and 128.9 m
- Hole TL0803: 26 readings were collected in mineralised MSS between 62.0 to 70.2 m
- Hole TL0836A: 15 readings were taken from mineralised MSS occurring from 165.07 to 168.08 m

The SCIP could not identify any clear correlations between chargeability and resistivity with gold mineralisation or gold assays observed in these drill cores. However, both resistivity and chargeability values within the mineralised zones were consistent with the bulk resistivity and chargeability values obtained in the mineralised zones during the EarthProbe™ drillhole surveys.

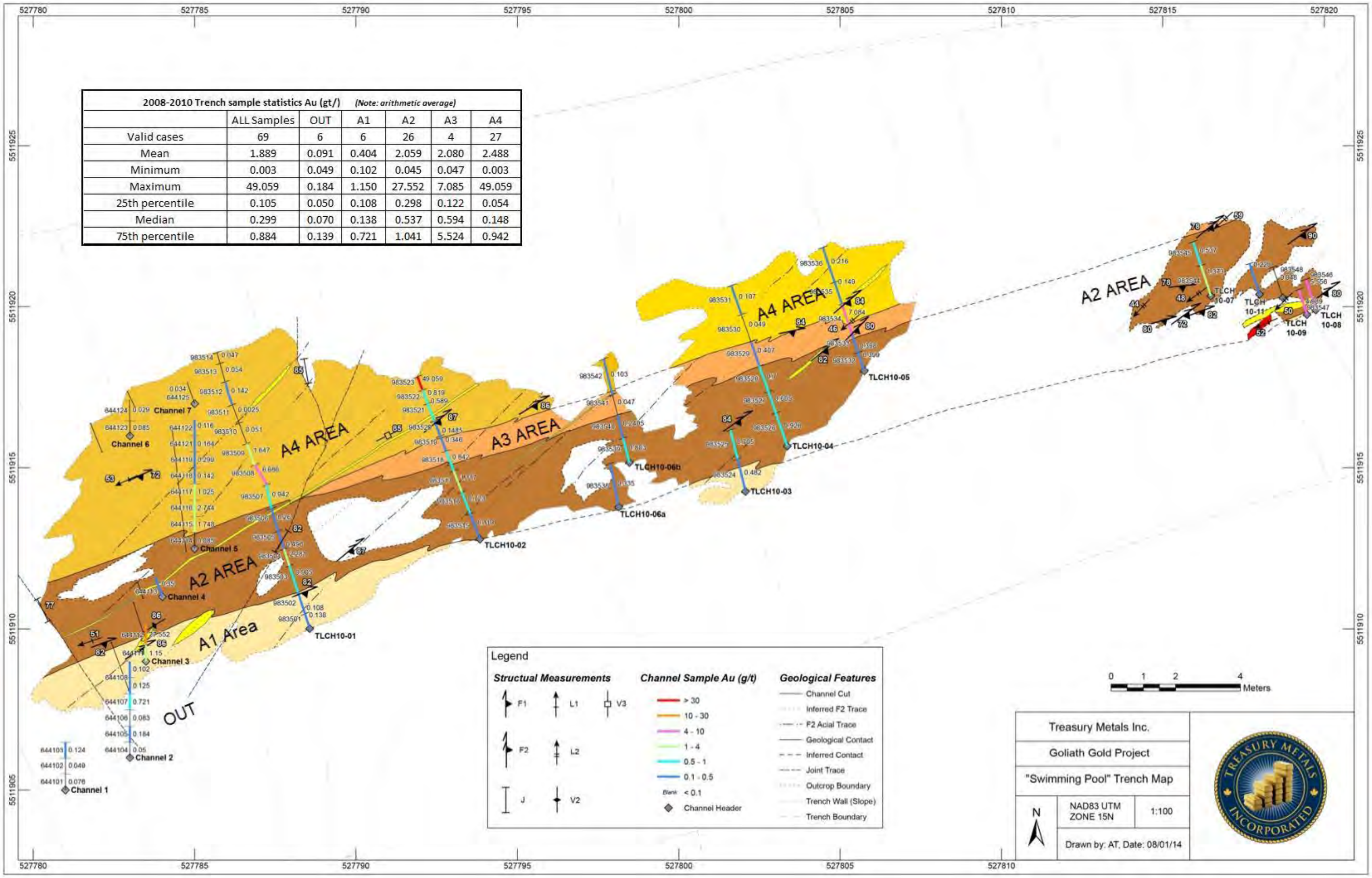
Although the vertical depth of penetration for the EarthProbe™ survey is deeper (250 m), compared to the JVX survey, which could only reach a vertical depth of around 60 m, CCIC was not able to define any new ground geophysical anomalies that were not already identified by the 2008 JVX IP survey.

9.2.3.6 Trenching Program

The 2008 trench was extended by CCIC to expose mineralised bedrock of the Main Zone for an approximate strike length of 42 m in the summer of 2010. This trench exposes the central shoot of the Main Zone and is located around drill section 527800E. It was geologically and structurally mapped at a scale of 1:100 and then systematically channel sampled (see Figure 9-7).

Table 9.5 summarises the structures mapped in the Main Zone trench.

Figure 9-7: Geology & Structural Map of the Main Zone Trench with Gold Channel Sample Assay Results (2008-2010)



2008-2010 Trench sample statistics Au (g/t) (Note: arithmetic average)

	ALL Samples	OUT	A1	A2	A3	A4
Valid cases	69	6	6	26	4	27
Mean	1.889	0.091	0.404	2.059	2.080	2.488
Minimum	0.003	0.049	0.102	0.045	0.047	0.003
Maximum	49.059	0.184	1.150	27.552	7.085	49.059
25th percentile	0.105	0.050	0.108	0.298	0.122	0.054
Median	0.299	0.070	0.138	0.537	0.594	0.148
75th percentile	0.884	0.139	0.721	1.041	5.524	0.942

Source: Treasury Metals (2014), modified by AGP (2020).

Table 9.5: Summary of Structures Mapped in the 2010 Main Zone Trench

Event	Structure	Description	Veins	Description
D_0	S_0	Compositional layering of meta-volcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	White to grey, highly deformed, S_1 foliation parallel very fine-grained quartz-sulphide ribbons and silicification with narrow sericite lamellae
D_1	F_1	Isoclinal folding	V_1	White coarse-grained deformed, foliation parallel distended quartz lenses (rare)
	S_1	F_1 axial planar and layer parallel foliation/schistosity ~073/80°		
	L_1	Stretching lineation, axis to isoclinal fold hinges; trend ~248 °, plunge 52°		
D_2	F_2	Closed (interlimb angle 60 °) folds; axial planes ~052/83°; discrete, 20 cm to 1.5 m spacing	V_2	Weakly deformed white quartz+/-sulphide lenses along F_2 axial planes.
	L_2	F_2 fold axes trend 228° and plunge 49°		
D_3	J (?)	Brittle joints oriented ~162/81° and 032/82° ; possibly related to NW Fault	V_3	White un-deformed, planar crosscutting quartz-tourmaline+/-sulphide veins near vertical WSW striking.

Source: Wetherup (2010).

Overall, CCIC concluded that the best potential for the highest gold concentrations are likely to occur near the F_1-F_2 intersections and in areas where there is an increased intensity of F_2 structures in the formation of high-grade shoots. It was also noted that concentrations of sulphide minerals also increased where F_2 fold hinges cut the Main Zone. They also recommended that future drilling programs should be focused along these westward plunging shoots.

A total of 47 channel samples plus two duplicates was collected for the trench covering all four geological units (see Table 9.6). Six of the samples collected assayed in excess of 3.0 g/t Au. Table 9.6 lists the samples assaying above 1.0 g/t Au.

Table 9.6: 2010 Channel Samples Excess of 1.0 g/t Au

Channel	Sample Number	Length (m)	Unit	Au (uncapped) (g/t)	Ag (g/t)
TLCH10-02	983523	0.55	4	49.059	
TLCH10-01	983508	0.75	4	6.686	
TLCH10-05	644115	0.50	4	1.748	3.70
TLCH10-01	983509	0.85	4	1.647	
TLCH10-05	983534	1.00	3	7.084	217.14
TLCH10-03	644112	0.65	2	27.552	2.19
TLCH10-08	983546	1.00	2	5.556	
TLCH10-09	983547	0.60	2	4.989	133.43
TLCH10-01	983504	0.50	2	2.281	
TLCH10-07	983544	0.90	2	1.373	
TLCH10-02	983517	0.65	2	1.117	

Source: Treasury Metals (2015).

Overall, samples from Unit 1 (three samples taken) were generally low with the highest of 1.15 g/t Au over a channel sample length of 0.5 m (sample 644111). Unit 2 (22 samples), which contained the most sulphide mineralisation, returned three high-grade samples of 27.55 g/t Au over a sample length of 0.65 m, 5.56 g/t Au over 1.0 m and 4.99 g/t Au over 0.60 m. The latter sample also returned 133.43 g/t Ag over the 0.6 m channel length. A metallic screen fire assay of sample 644112 returned 12.98 g/t Au. Unit 3, with a total of five samples, averaged 2.11 g/t Au with a high of 7.08 g/t Au and 217.14 g/t Ag over a sample length of 1.0 m (sample 983534). Seventeen samples were collected from Unit 4 and averaged 2.99 g/t and returned the highest gold assay grade of the program of 49.06 g/t over a sample length of 0.55 m hosted in the MSS rocks.

Overall, the 69 trench samples from the 2008 and 2010 work program returned an average of 1.889 g/t Au with a median grade of 0.299 g/t Au. AGP notes that the high-grade assays in excess of 3.0 g/t Au are sporadic and do not form a continuous zone at that location.

9.2.3.7 Petrographic & Scanning Electron Microscope Study

Two polished sections of two samples collected from diamond drillhole TL0814 for petrographic examination (Beakhouse, 2010). The samples were analysed by Gary Beakhouse of the Ministry of Northern Development of Mines (MNDM) under plan polarised, cross-polarised and reflected light as well as on the OGL scanning electron microscope (SEM). The following observations were reported:

- Minor amounts of gold were present in both thin sections; small grains infilling pyrite in association with galena, between sphalerite grains or between larger pyrite crystals.
- It was unclear if the gold occurred in the sulphides or whether the association observed is representative and accounts for the high gold assay results (38.63 g/t Au and 44.62 g/t Ag).
- Gold is spatially associated with galena and sphalerite and appears to be paragenetically late.
- Galena and sphalerite exhibit a paragenetically late timing relative to other sulphides occurring as overgrowths around, and veins within, pyrite and minor amounts of arsenopyrite.
- The timing relationship of chalcopyrite is unclear.
- Silicate mineralogy consists of quartz, feldspar, white mica, and calcium alumosilicate (stilpnomelane?).

Mineralogical observations are supported by 14 photomicrographs identifying the various mineral phases and relationships.

9.2.4 2011 Exploration Activities

A DIGHEM electromagnetic and magnetic helicopter supported airborne geophysical was carried out for Treasury Metals over the Goliath property between July 14 and July 16, 2011 (Fugro Airborne Surveys, 2011). A total of 531.46 line-km of traverse lines (oriented north-south) were flown with a spacing of 100 m and 54.16 km of tie lines with a spacing of 1,000 m for a total of 585.6 km for the complete survey.

Fugro created the following set of maps: (1) horizontal gradient enhanced total magnetic intensity; (2) calculated vertical magnetic gradient; (3) apparent resistivity (56,000 Hz); (4) apparent resistivity (7,200 MHz); and (5) DIGHEM EM anomaly maps. All final maps were created at a scale of 1:20,000 with the Universal Transverse Mercator (UTM Zone 15N) coordinate system, NAD83 Datum. The results of the Fugro airborne survey are summarised below from the technical report by Roy et al. (2012):

- Magnetic calculated vertical gradient (CVG) and horizontal gradient enhanced total magnetic intensity maps clearly define geological rock contacts throughout the property.
- An iron formation with high magnetic responses (BIF) is defined in the western part of the property.
- The Thunder Lake Assemblage of meta-volcanic and meta-sedimentary rocks also show strong magnetic intensity in the southern parts of the property.
- A combination of magnetic and resistivity parameters has outlined a few interesting magnetic lows that coincide with resistivity highs that might reflect alteration zones or siliceous caps warranting further investigation.
- Deep conductive units are potentially capped by superficial resistive units.
- Several low resistivity zones where values are less than 100 ohm-m likely represent conductive clays or graphitic shales which some of the more discrete responses might be caused by conductive sulphide content or clay-altered shears.
- The survey identified 987 EM anomalies with nearly 69% of those linked to conductive overburden or metasedimentary rocks, about 7.5% are due to cultural sources and approximately 23.5% are due to possible or probably bedrock sources.

9.2.5 2012 Exploration Activities

9.2.5.1 Goliath 3D Inversion Study of Aeromagnetic Survey Data

In 2012, a 3D inversion modelling study was completed by Ellis (2012) using the Fugro airborne magnetic survey data (assuming they used the Fugro dataset). This study was initiated to (1) attempt to identify the aeromagnetic signature of the Goliath deposit, (2) determine the possible explanation for the apparent termination of the zone east of the main deposit and (3) define possible easterly extensions of the gold-bearing zone again east and northeast of the deposit.

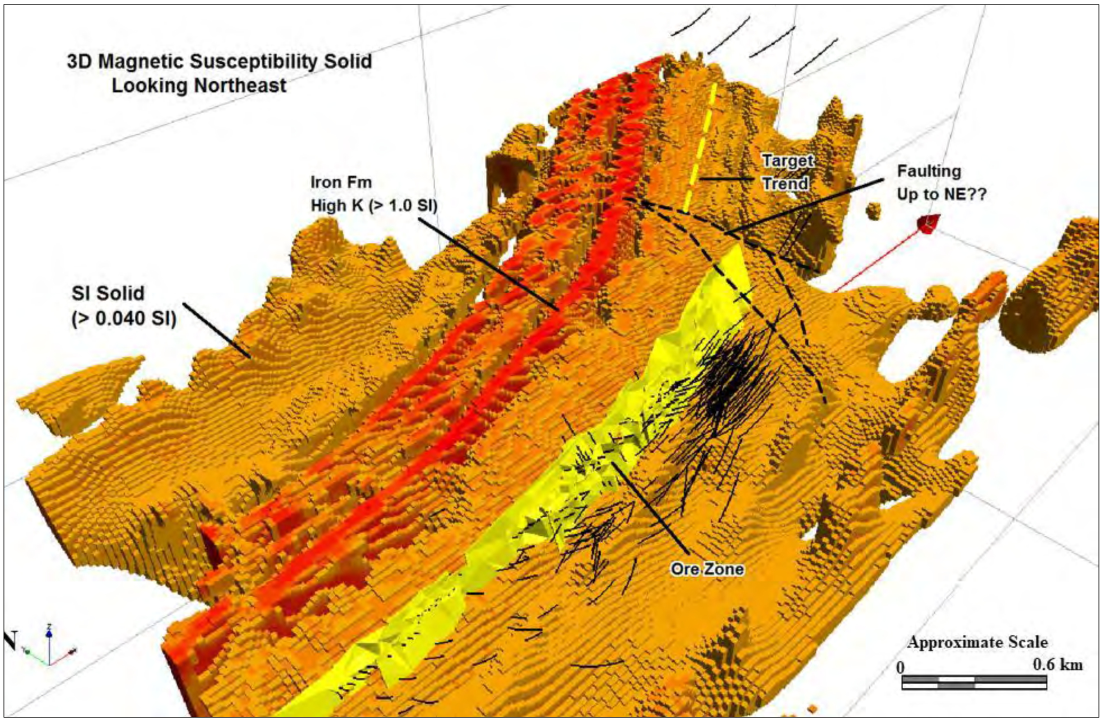
The 3D inversion modelling of the aeromagnetic data generates a solid of magnetic susceptibility that will fit the raw magnetic data within a predefined error tolerance. A series of 3D susceptibility solid maps at 350 m elevation, including cross-sections of the model, were prepared to compare with both known zones of gold mineralisation at Goliath and local geology.

The models clearly define a north-trending normal fault that displays left lateral motion disrupting the main deposit in the east and shifting the main zone north of its present location (see Figure 9-8). The red areas on the map in the north represent the iron formation. Ellis (2012) also had the following additional observations:

- There is a bend in the iron formation to the north that is consistent with shifting of the target trend of mineralisation to the north by the fault.
- The stratigraphy hosting the gold mineralisation is not always concordant with mineralisation (structurally controlled).

The 3D inversion modelling was able to demonstrate where the gold mineralised zone resumes east of the fault for future drill targeting of the Main Zone (eastern alteration corridor, east of the fault).

Figure 9-8: 3D Magnetic Susceptibility Solid Map, 350 m Elevation



Source**:** Treasury Metals (2015).

9.2.5.2 Thin Section Study of Mineralised Drill Core

Eighteen samples collected from nine diamond drillholes were submitted to Vancouver Petrographics Ltd. in Langley, BC, in 2012 for petrographic thin section work (see Table 9.7). An examination of these samples concluded that (Leitch, 2012):

- Seven samples likely represented either felsic to intermediate meta-volcanic rocks (samples TLTS-3 to 7, TLTS-10 and TLTS-12).
- Seven samples represented exhalative rocks containing massive or semi-massive sulphides with some local significant occurrences of visible gold (samples TLTS-11, TLTS-14 to 18, and TLTS-13).
- Two samples likely represented mafic meta-volcanic rocks (samples TLTS-2, TLTS-9).
- One possibly a meta-microdiorite rock (sample TLTS-8).
- One was a possible anhydrite-quartz-amphibole-green biotite vein hosted in felsic to intermediate meta-volcanic rock (sample TLTS-1).

Detailed petrographic descriptions and photomicrographs were included with the final report.

Table 9.7: 2012 Petrographic Study Results

Sample Number	Drillhole	Depth (m)	Comments
TLTS-1	TL11229	224.75	F_l/chlorite veining
TLTS-2	TL11229	148.60	Orange porphyroblasts and cordierite (?)
TLTS-3	TL11223	527.00	Green silicate band with silicification and some sulphide mineralisation
TLTS-4	TL11229	234.42	MSS (east), northeast exploration area, no mineralisation
TLTS-5	TL11135	321.20	MSS (west), silicified, no mineralisation
TLTS-6	TL11222	358.00	BMS (east), northeast exploration area
TLTS-7	TL11209A	129.15	BMS (west) from western zone
TLTS-8	TL11222	363.15	Massive, less foliated BMS with quartz eyes
TLTS-9	TL11187	179.95	Mafic dyke
TLTS-10	TL11209A	129.80	F2 fold
TLTS-11	TL11148	55.35	Massive fuchsite/chlorite with black tourmaline/amphibole
TLTS-12	TL11193	377.10	Mineralised zone with coarse pyrite, chalcopyrite, sphalerite, and garnet; sample no.1076645 (0.25 g/t)
TLTS-13	TL11121	266.70	Semi-massive sulphide band; sample #981132 (19.63 g/t)
TLTS-14	TL11121	268.15	Deformed quartz veins (no VG); Scattered sulphides, no VG, sample #981135 (Trace)
TLTS-15	TL11122	270.70	Low grade (1-2 g/t); sample #981248 (1.24 g/t)
TLTS-16	TL11152	239.20	Medium to high grade; stringers adjacent to quartz veins, sample #1007597 (18.6 g/t)
TLTS-17	TL11135	325.95	Medium to high grade; edge of semi-massive sulphide band, increased Pb, sample #983067 (10.3 g/t)
TLTS-18	TL11130	341.30	Deformed & boudinage quartz veins (with VG); several VG flecks with quartz, sample#981797 (89.2 g/t)

Source: Treasury Metals (2015).

9.2.6 2014 Exploration Activities

A mobile metal ion (MMI) soil sampling program was conducted on selected target area throughout the Goliath project area from July to October 2014. A total of 1,850 samples were collected during this period by two Treasury Metals field sampling teams. Target grids were located over numerous areas targeting airborne EM, magnetic, ground IP and geological units of interest including iron formation to the north and the strike extension of the Goliath deposit. All samples were collected following sampling procedures outlined by SGS Minerals Services (SGS, 2013a, 2013b).

An orientation survey identified the optimal sampling depth of 10 to 25 cm below the surface. No grid lines were physically cut, but samples were collected using a GPS at line spacings of 200 m and samples taken at 25 m stations. Additional infill lines at 100 m were added to higher priority target areas after the survey results were made available.

Samples were analysed at SGS and response ratios for gold and multi-elements, including base metals copper, lead, and zinc were calculated by Treasury Metals and the results plotted on a regional plan map of the property (see Figure 9-9, overleaf). Five high-priority targets for groundtruthing and further field investigation were identified from this survey, as follows:

- Anomaly P – Iron formation possibly intercepted by F2 gold-bearing structures northeast of the Goliath deposit; moderate to strong linear Au/Cu/Sb/W and weak Ag and as response ratios (RRs); highest Au RR of 60. This anomaly was drill tested in 2015 by holes TL15401 and TL15402 with no significant gold intersections.
- Anomaly N – Nose of regional fold structure (iron formation and eastern strike extension zone of the Goliath deposit). High magnetic anomaly, moderate to strong Au/Ag/Cu, weak Pb and Zn RRs in close proximity to historical Teck holes that intersected some significant gold mineralisation.
- Anomaly O – Corresponding magnetic and EM linear anomaly, moderate to strong Ag/As/Pb/As and weak Au/Bi/Cu/Sb/W RRs.
- Anomaly G – EM anomaly following a magnetic trend. Moderate to strong Cu/Pb/Zn and weak Sb/W RR.
- Anomaly D – Strong tungsten/zinc, moderate to strong Ag/Cu/Sb, weak As in close proximity to 2012 Treasury Metals drilling fence where one hole intersected 2.0 g/t Au over a core length of 2.0 m (hole TL12266) in a 70 m wide MSS unit located in the far east of the property (represents extreme east extension of the Goliath gold zone).

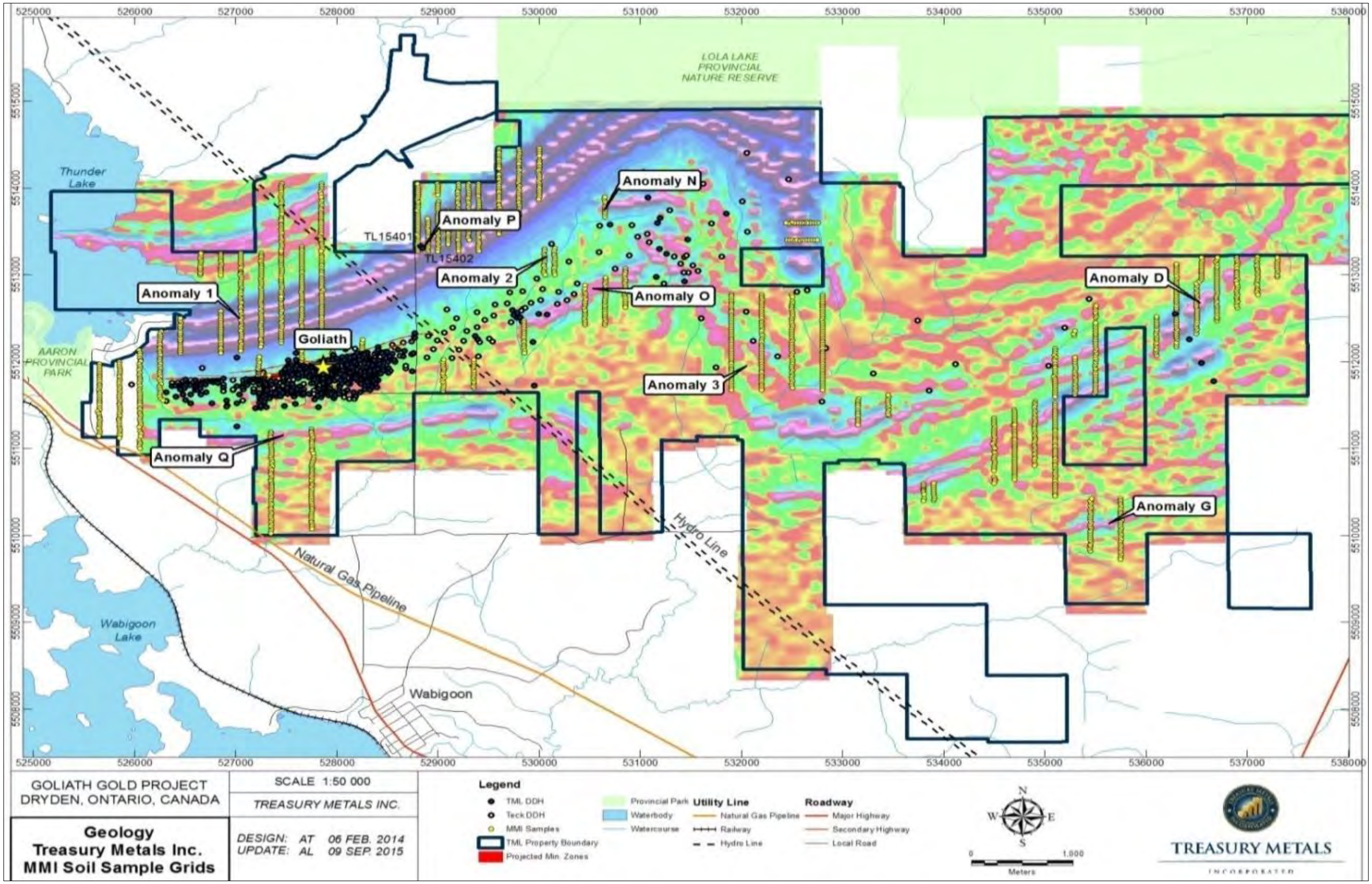
With the exception of Anomaly P, which was drill tested during the 2015 drilling program, the remaining anomalies need to be investigated in the field to determine if the source of the anomalies can be explained at surface.

9.2.7 2015 Exploration Activities

An infill core sampling program was completed at the conclusion of the 2015 diamond drilling program to further evaluate the gold potential of the B Zone and test other zones throughout the deposit known to contain significant gold mineralisation, but were never previously sampled or assayed (Treasury Metals drill core only). This program covered untested areas of either extensions or potential new zones of previously unsampled drill core focusing on identifying zones that would reside in a potential open pit located from surface to a vertical depth of around 200 m. The boxes containing the target intervals of drill core were retrieved from the core farm located on site, examined, and logged by the geologist, and samples were marked up for splitting. Canadian Standards and blanks were submitted for each hole. Split core samples were then dispatched to Accurassay Laboratories for gold analyses.

A total of 2,090 new split core samples were collected from 95 drillholes. The program was successful in identifying new zones of gold mineralisation in half (56) of the 110 new target zones that were identified for inspection. Gold assay intersections in excess of 1.0 g/t are summarised on Table 9.8. A near-surface hole and a newly tested Hanging Wall Zone both reported significant results: hole TL10116 returned 6.08 g/t Au over 6.0 m at a vertical depth of 17 m from surface and TL0853 returned 4.53 g/t Au over a sample length of 5.0 m at a depth of 160 m. Visible gold was observed in some of the drill core. Table 9.8 lists gold assay intersections above 1 g/t Au.

Figure 9-9: 2014 MMI Sample Grid Location Map



Source: Treasury Metals (2015).

Table 9.8: Gold Assay Intersections above 1 g/t Au – Core Infill Sampling Program

Hole Number	Section	From (m)	To (m)	Length (m)	Au (g/t)	Target Zone Description
TL13301	528300	105.00	106.00	1.00	7.15	D Zone, Visible Gold
TL10116	527825	14.00	20.00	6.00	6.08	Hanging Wall 1 (In-Pit)
TL11184	527225	203.00	204.00	1.00	5.77	B2 Zone
TL11210	527700	360.00	361.00	1.00	5.62	B1 Zone
TL0853	527300	177.00	182.00	5.00	4.53	Main Zone (In-Pit)
TL11206A	527225	411.00	412.03	1.03	3.37	Main Zone
TL12278	527600	306.00	307.00	1.00	2.81	B1 Zone
TL0852-12RE	527575	354.00	355.00	1.00	2.58	Main Zone
TL12283	527325	433.00	434.00	1.00	2.52	B1 Zone
TL15385B	527675	375.00	377.00	2.00	2.25	B1 Zone
TL14358	528000	180.00	183.00	3.00	2.29	Main Zone
TL11128	528150	443.00	444.00	1.00	1.90	C Zone

Source: P&E (2015).

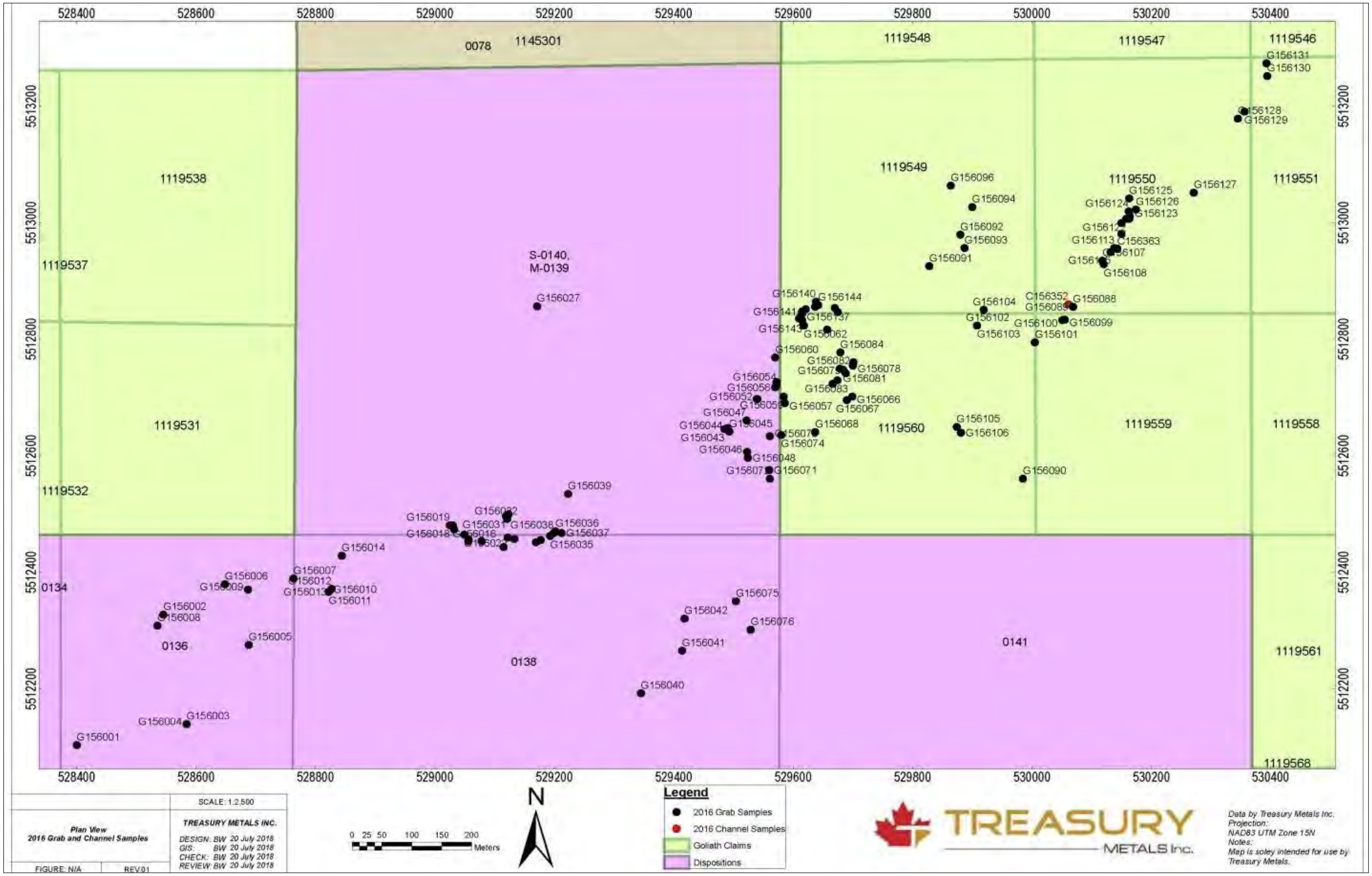
9.2.8 2016 Exploration Activities

9.2.8.1 Field Mapping & Sampling Program

In 2016, Treasury Metals completed a field mapping and sampling program conducted by geologist Cheyenne Sica. This program consisted of a total of 134 grab samples and 13 channel samples (not including seven coarse blanks and seven standards CDN-CM-26) that were collected and dispatched to ActLabs in Dryden, Ontario, for gold assay and multi-element analysis. A total of 65 samples were taken over three separate patented claims and an additional 69 samples were taken from five unpatented legacy claims (see Figure 9-10). The samples were mainly located along strike of the known resource over a distance of approximately 2.4 km and covering an area of approximately 1.4 km^2. The purpose of this program was to:

- map the terrain and geology of the proposed mine infrastructure sites
- locate and GPS survey historical drill collars locations
- groundtruth the surface mineralisation locations of gold chutes interpreted by Exploration Manager Paul Dunbar from historical drillhole compilation and newly prepared longitudinal sections of the eastern alteration corridor (EAC)
- map and sample the eastern strike extension of Goliath deposit Main Zone, C Zone, and parallel zones (D-G) along the EAC
- further investigate, prospect, and sample the Gossan showing
- follow up on MMI anomalies observed from the previous year's MMI sampling program
- identify new exploration drill targets to potentially increase gold ounces outside of the currently defined resource area

Figure 9-10: 2016 Grab & Channel Sampling Program (Goliath Project)



Source: Treasury Metals (2019).

9.2.8.2 Proposed Mine Infrastructure Sites Mapping & Sampling

From September 17, 2016 to October 26, 2016 the locations of the proposed mine infrastructure sites were surveyed to explore for outcrops with potential gold mineralisation.

Infrastructure to the west of Tree Nursery Road, north of the proposed open pit, is located in old slash from previous logging activities with new growth of small alder trees. Several outcrops of MSS and BMS were mapped and sampled in this area with no detectable gold mineralisation.

The location of the tailings pond is dominantly in muskeg lowland. In the northern portion of the proposed tailings pond location, scattered outcrops of iron formation are present with no evidence of alteration, deformation, or mineralisation. The southeastern portion of the tailings pond covers the strike extension of the Goliath deposit Main Zone and C Zone extensions. In this area mineralised BMS and MSS rocks were sampled and mapped returning assays from 0.42 g/t Au to 1.42 g/t Au (see Table 9.9).

The proposed site of the polishing pond is located in a mixture of old slash with new growth alders, muskeg lowland, Jackpine forest high-ground with sandy soil, and a swamp surrounding a small creek. Outcrops of mineralised BMS and MSS rocks were mapped in this area returning assays of 0.37 g/t Au to 0.41 g/t Au (Table 9.9).

Figure 9-11 shows the surface terrain and comparison of locations of resurveyed Teck diamond drillholes. The red units denote the surface projection of mineralised gold chutes interpreted by Paul Dunbar using the best intersections from historical Teck and Treasury Metals drillhole data.

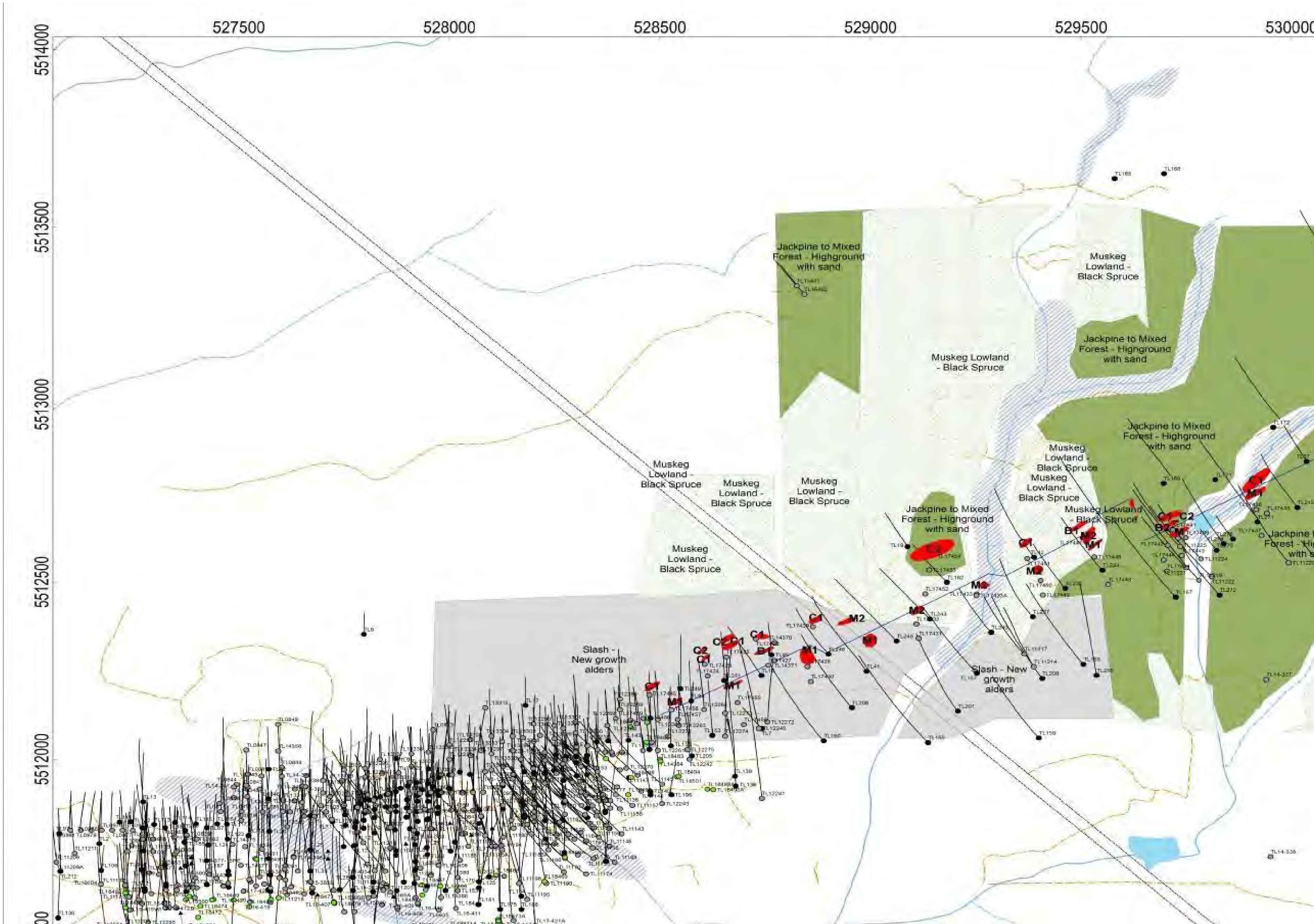
AGP notes that the condemnation mapping program returned grades above the open pit cut-off presented in this study.

Table 9.9: Significant Assay Results – 2016 Condemnation Field Mapping Program

Sample Number	Easting	Northing	Sample Description	Au (ppm)
G156061	529615	5512847	BMS: Strongly silica flooded with weak to moderate shear fabric; pink colour (pervasive hematite?); some chlorite bands with ≤1-% biotite bands; altered porphyry protolith? Pyrite fine grains – coarse grains throughout (up to 10%)	1.42
C156361	530143	5512954	BMS: moderate silica flooding; ~5% MSS bands; 5-7% blue quartz eyes; 1% pyrite seams and 1-2% fine grains pyrite disseminations; 2% chlorite bands containing locally up to 5% pyrite ; smoky grey quartz vein with oxidised staining hosting 1% galena + sphalerite seams; strongly sheared around quartz with increased sericite alteration	0.98 (over 0.7 m)
C156362	530143	5512954	BMS: moderate silica flooding; 2-3% MSS bands; 2% blue quartz eyes; mm-wide seam of galena; mm-scale massive pyrite seams; 2% very fine grains pyrite mineralisation	0.76 (over 1.0 m)
G156110	530132	5512949	BMS; massive pyrite (~15% of sample); 15% chlorite; 60% glassy dark silica flooding;	0.754
G156058	529584	5512701	BMS: 60% biotite, 5% sericite the rest is silica flooded; moderate foliation; coarse grain pyrite seams up to 5%	0.471
G156054	529572	5512721	MSS: ~60% sericite + muscovite, ~30% silica; oxidised on foliation planes; trace chalcopyrite and arsenopyrite; up to 4% pyrite along foliation planes as blebs and fine grains dissemination	0.417
G156031	529119	5512496	BMS: moderate silica flooding; ≤1% black quartz eyes; balk quartz veinlets parallel to foliation (~30% of sample); ≤ 3% fine grains pyrite	0.41
G156029	529123	5512499	MSS: sericite + muscovite + silica; oxidised staining throughout; up to 3% pyrite as seams along foliation planes	0.373
G156114	530138	5512956	MSS: strong silica flooded with galena seam and 2% pyrite seams	0.333

Source: Treasury Metals (2019).

Figure 9-11: 2016 Surface Features & Diamond Drillhole Locations



9.2.8.3 Eastern Alteration Corridor

Geological mapping and sampling of the eastern alteration corridor (EAC) was conducted from September 17, 2016 to October 13, 2016 by geologist Cheyenne Sica. Mapping and sampling targeted the strike extension of Goliath gold-bearing MSS and BMS geological units. A high priority of the 2016 field program was to groundtruth and explore for signs of mineralisation at the locations where gold chutes were projected to outcrop, utilising the new longitudinal sections that had been constructed along the entire strike length of the EAC (red dotted line shown on Figure 9-12). Unfortunately, no outcrops were found at the exact locations of the interpreted high-grade chutes.

Grid geology mapping completed by Teck in the 1990s mapped a large package of felsic metavolcanic rocks throughout the central portion of the EAC bound to the north and south by metasedimentary rocks. The surface projection of geology from historical drillhole data reveals a more complicated stratigraphy consisting of quartz-feldspar porphyry, quartz-porphyry, strongly altered BMS and MSS rock units. The BMS and MSS units are on strike with identical rock units that host the high-grade gold mineralisation at the Goliath deposit. Pervasive alteration, metamorphism and deformation make it difficult to distinguish these units throughout the EAC as definitive felsic volcaniclastic rocks, such as a silicified felsic tuff, and the MSS and BMS rocks typically have a porphyritic texture and could be interpreted as felsic intrusive porphyry rocks.

A final geology compilation map has been generated integrating (1) the geological mapping from the 2016 field program, (2) the newly interpreted drill sections utilising all historic drillholes along the EAC, (3) the newly interpreted longitudinal sections, (4) geology from the old Teck grid mapping programs, and (5) all existing ground and airborne geophysical data.

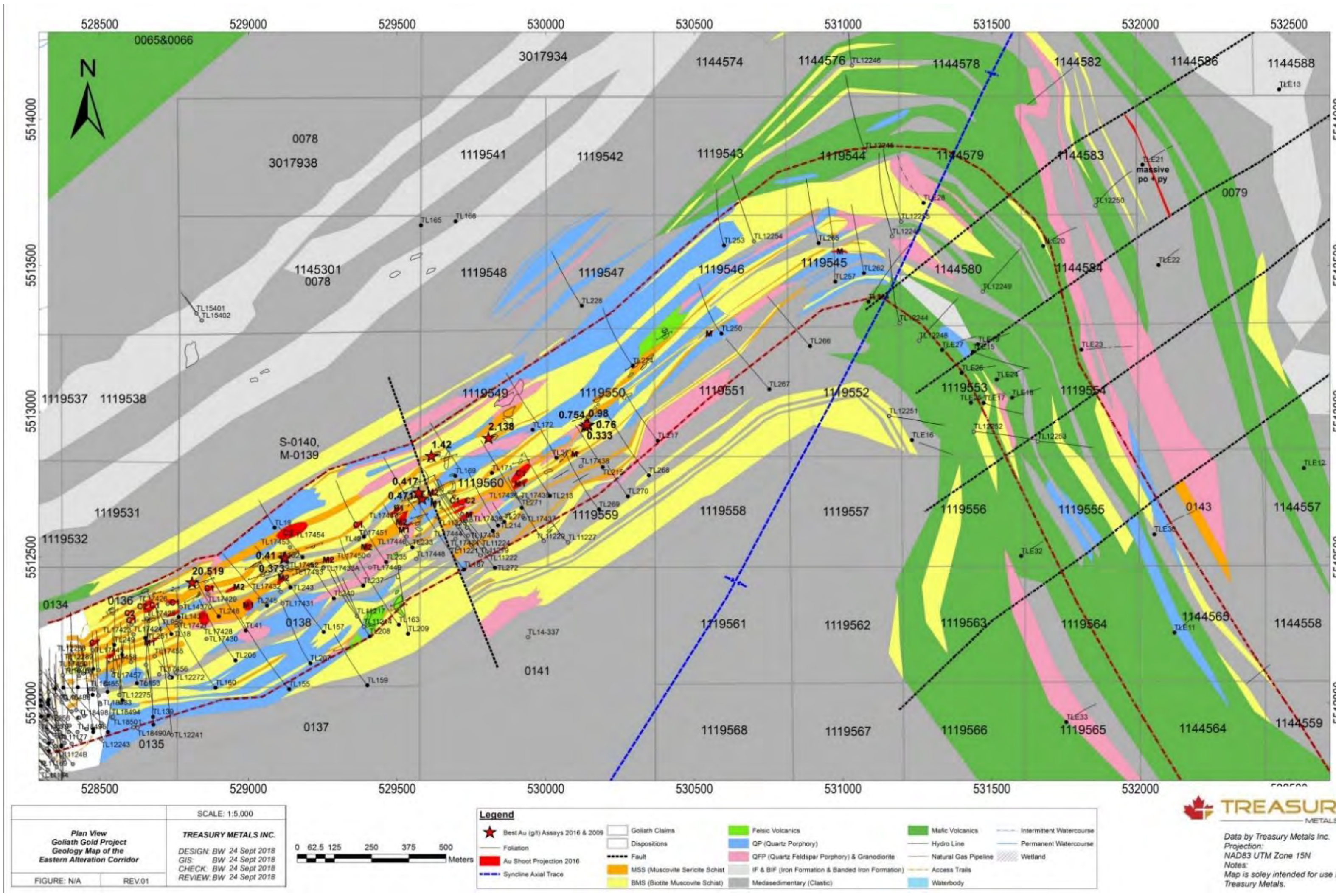
Of the 134 grab samples collected from the EAC, 110 samples contained anomalous gold returning assays of > 0.005 g/t Au. Thirteen channel samples were taken, all of which returned > 0.10 g/t Au. The dominant sulphide phase observed was pyrite occurring as fine to coarse-grained disseminations with some massive pyrite seams. Galena and sphalerite were also observed as stringers concentrated at contacts between BMS rocks and smoky grey quartz veinlets.

Grab samples with gold assays of > 0.3 g/t Au have been plotted on the geology map as red stars (Figure 9-12). Each of the showings have been described below starting in the western portion of the map area:

- Two grab samples returning 0.373 g/t Au (G156029) and 0.41 g/t Au (G156031) are located at the merging point of the C and B Zones. These samples contain low concentrations of Ag and were anomalous in Pb and Zn.
- Assays returns of 0.417 g/t Au (G156054) and 0.471 g/t Au (G156058) were obtained from grab samples collected from the B1 and Main Zones just west of the fault. Sample G156058 contained 4.6 g/t Ag and anomalous Pb (155 ppm) and Zn (175 ppm).
- Sample G156061 returned 1.42 g/t Au and 1.10 g/t Ag and was collected north of the best-known mineralised MSS zones.

In all three gold occurrences described above, gold is associated with silicification and pyrite and is hosted by both MSS and BMS rock units.

Figure 9-12: Geology Map of the Eastern Alteration Corridor (Goliath Deposit)



Source: Treasury Metals (2019).

Samples C156361, C156361, G156110 and G156114 were collected from the C Zone and are located 2 km along strike from the main Goliath deposit. C Zone grab samples returned 0.333 g/t Au (G156114) and 0.754 g/t Au (G156110). Sample G156110 also contained 7.5 g/t Ag (the highest silver assay of the 2016 program) in association with 409 ppm Cu, 560 ppm Pb and 1,060 ppm Zn. Channel sample C156361 assayed 0.98 g/t Au and 6.8 g/t Ag over a sample length of 0.7 m and also contained elevated base metals (259 ppm Cu), as well as the highest Pb and Zn assays of the program of 1,760 ppm Pb and 3,020 ppm Zn, respectively. BMS rocks host the three highest gold assays. This C Zone showing occurs further east than anticipated from historical drill best assay intersections suggesting the potential for additional chutes to occur east along strike.

Upon completion of the field sampling program, several new exploration targets were identified along strike of the main resource situated in the EAC near the nose of a regional fold structure (folded syncline) which is considered a very high priority target area. Anomalous gold assays found within the grab and channel samples warrant an additional soil sampling program (soil gas hydrocarbon) or follow up with exploratory drilling to further test the potential of the select locations.

9.2.8.4 Gossan Showing

On November 4, 2016, geologists visited the Gossan Showing exposed by the construction of a new logging road that was initially described and sampled by geologist Adam Larsen in October 2015. The access logging road has since been extended westward 700 m following the strongly gossaned (oxidised) shear zone.

The Gossan Showing coincides with a ~1 km long east-west-trending airborne EM and magnetic geophysical anomaly which also occurs in association with Pb, Zn and Cu MMI anomalies identified during the 2014 soil sampling program. During the field program, the strike extension of the Gossan Showing was traversed and sampled over a strike length of almost one kilometre.

The gossan zone itself is hosted in a moderately to strongly sheared mafic volcanic package with strong chlorite + amphibole alteration and is typically strongly oxidised along foliation planes. The strike of the zone is 260° and it dips north from 80° to 85°. Portions of the showing are weakly silicified. Pyrite was observed as semi-massive bands and fine-grained disseminations concentrated along foliation planes. Pyrrhotite (up to 2%) was also observed along foliation planes.

The intense gossan alteration zone extends for at least 1.0 km in strike length and is contacted by mafic volcanic rocks to the north and south. On certain outcrops, pink felsic dykes with irregular contacts are injected into the southern mafic volcanic rocks. South of the southern mafic volcanic unit is a large body of strongly silicified felsic intrusive rocks (a possible quartz feldspar porphyry unit).

Five samples were taken along strike length of the gossaned unit. None of the samples returned any significant gold values. Samples were found to be anomalous in Cu (up to 244 ppm), Zn (up to 184 ppm), Pb (up to 39 ppm) and Mn (up to 4,130 ppm).

9.2.9 2017 Exploration Activities

9.2.9.1 Iron Formation Mapping & Sampling Program

In 2017, Treasury Metals completed an outcrop mapping and sampling program with focus on the iron formation lithological unit. The program consisted of 36 grab samples including two coarse blanks and two standards. The sampling program occurred over 12 unpatented mining claims and one patented mining claim. The samples were mainly located along the iron formation, as well as from within the nose of the regional fold structure. The samples covered an area of approximately 5 km^2. The purpose of this program was to:

- further investigate and sample the iron formation
- map and sample newly exposed outcrops that had recently been exposed by logging activity in the area of the EAC
- identify new exploration drill targets in the nose of the regional fold to potentially increase gold ounces outside of the currently defined resource

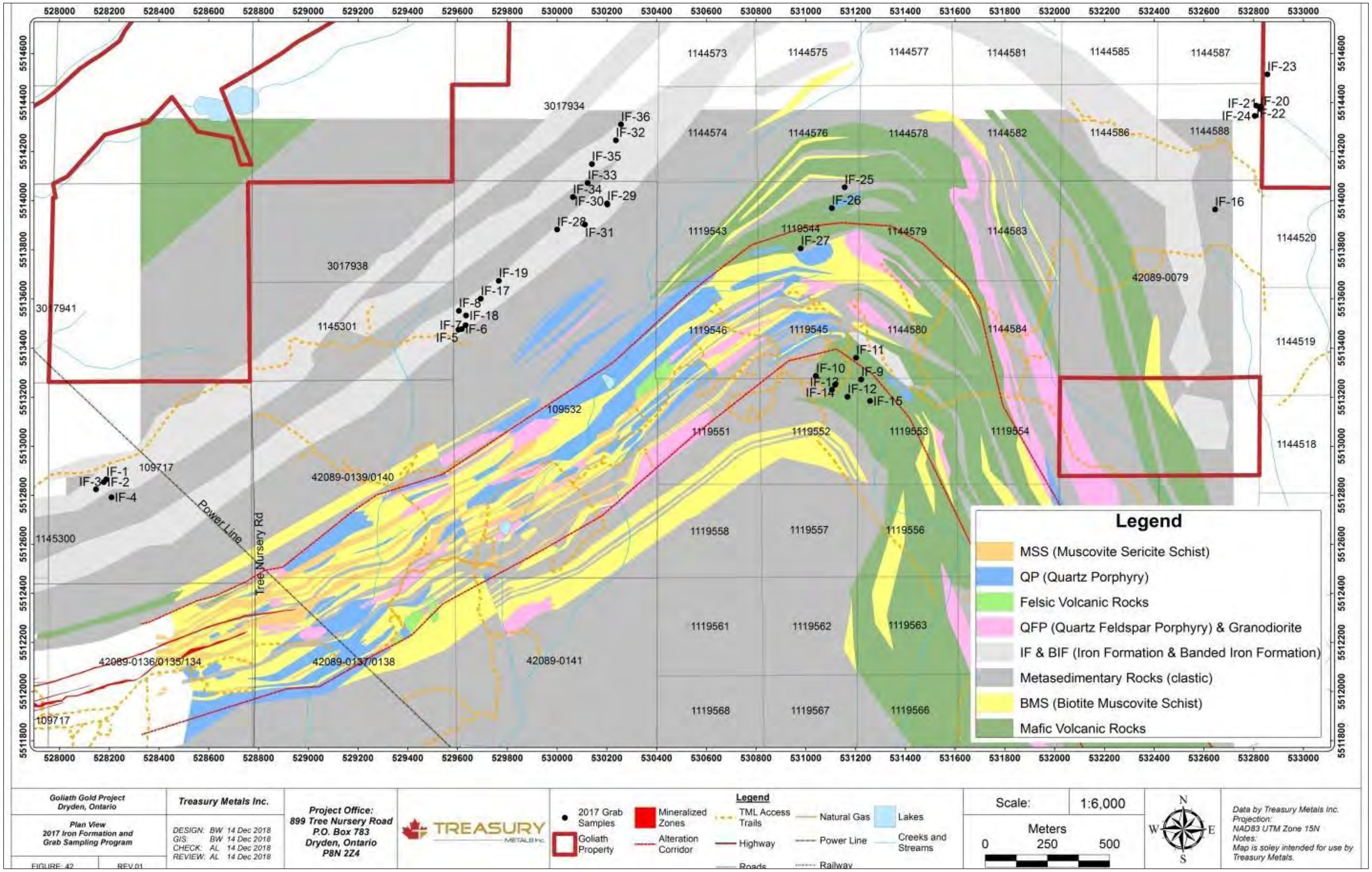
9.2.9.2 Sampling Program

Sampling of the iron formation and the EAC was conducted from August 30, 2017 to November 6, 2017 by geologists Bryan Wolfe and Eldon Phillips. A total of 36 grab samples were collected and dispatched to ActLabs in Dryden, Ontario for a fire assay gold analysis (see Figure 9-13). The areas of interest were accessed using trucks, all-terrain vehicles (ATVs), and on foot.

Mapping and sampling targeted the strike extension of the iron formation along the limbs of the regional fold structure as well as the hinge of the regional fold structure in close proximity to the EAC. Of the 26 grab samples collected from the iron formation only one sample (IF-33, sample number 303986) returned an anomalous gold value of 0.01 g/t Au. The remainder of the iron formation samples returned no significant assay results. Of the 10 grab samples collected from the nose of the regional fold structure, seven samples returned anomalous gold values ranging from 0.01 g/t Au to 0.12 g/t Au. The remaining three samples collected from the nose of the fold returned no significant assay values as summarised below.

Upon completion of the sampling program it was determined that the program was unsuccessful at identifying any new prospective exploration targets within the iron formation. The hinge of the regional fold structure covers a large area (approximately 2 km^2) with an abundance of new outcrop exposure and was not extensively sampled at the time of the program. This area still warrants further investigation as the program was modest in size and previously identified anomalies in the 2014 MMI sampling study require follow up. Although no substantial gold assays were returned, the nose of the regional fold and the EAC still remain high priority targets with the potential to add additional gold ounces along strike of the main resource.

Figure 9-13: 2017 Iron Formation & Grab Sampling Program (Goliath Property)



Source: Treasury Metals (2019).

9.2.9.3 Mapping of the Iron Formation & Eastern Alteration Corridor

In addition to sampling, a brief geological mapping program of the iron formation and the EAC was conducted from August 30, 2017 to November 6, 2017 by geologists Bryan Wolfe and Eldon Phillips to further add to the current outcrop database.

The purpose of this program was focused on mapping the extent and continuity of the iron formation as well as mapping newly exposed outcrops due to recent logging activity in the nose of the regional fold structure situated within the EAC. Only a small amount of the exposed outcrops was mapped in detail in this program and an extensive mapping program should be undertaken to further explore the continuity of the lithologies and the structural elements that constrain them.

9.2.9.4 The Eastern Alteration Corridor

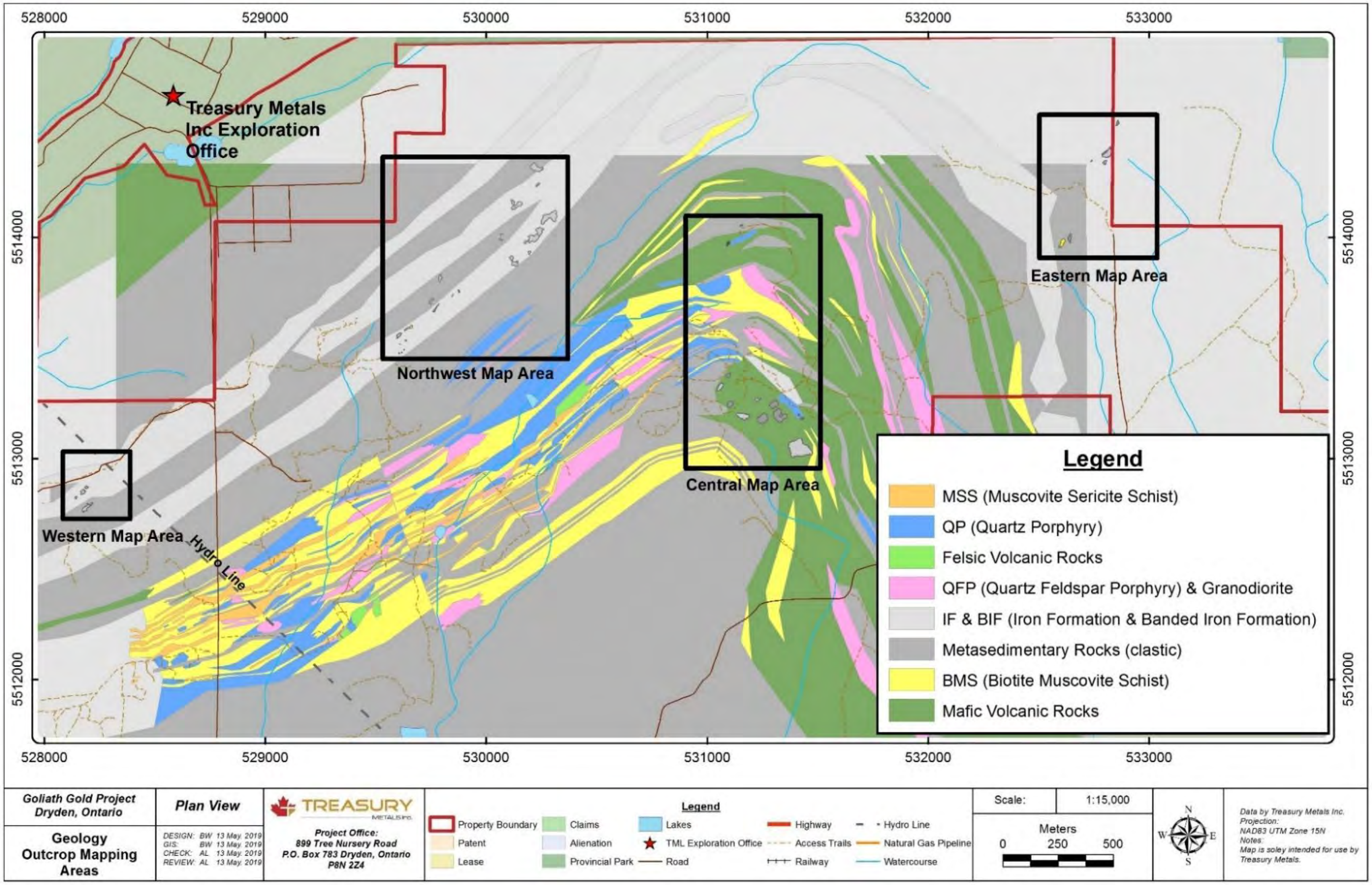
Recent timber logging of the Goliath property has exposed an extensive amount of new outcrop showings at the nose of the fold within the EAC. The area is primarily clear cut and can easily be navigated through use of an ATV. In this area several new outcrops were observed and mapped by using a Trimble geo-explorer 600 series handheld GPS unit and traversing the circumference of the surface feature. Within the nose of the fold, in close proximity to the EAC, Treasury Metals personnel located and identified five new outcrops of BIF, six new MSED outcrops, and one outcrop of BMS. Towards the end of the mapping program the geologists were successful in locating one new outcrop of MSS. Since the MSS is the primary gold bearing lithology within the Goliath deposit, further mapping of the area will be required to establish where the mineralised zones may be projected to the surface.

9.2.9.5 Outcrop Mapping Program

The primary purpose of the mapping and sampling program was focused on trying to establish the extents of the Iron formation along the northern limb of the regional fold structure, as well as to identify any new prospective exploration targets. Treasury Metals personnel was successful in identifying several new outcrop showings in the southwest and northeast (see Figure 9-14).

The iron formation is thought to extend all the way to the most northern tip of the Goliath property, but was not easily accessible. In order to reach the northern tip of the regional fold the geologists used an ATV on an old drill trail. Once reaching the end of the existing trail the geologists had to follow a small ridge of outcrops and had to be traversed by foot. On the northern limb of the fold Treasury Metals was able to identify and map 25 new outcrops of BIF, five outcrops of BMS and two new MSED outcrops. Due to the short nature of the program, Treasury Metals personnel were unable to map the northern most extent of the iron formation in the nose of the fold.

Figure 9-14: 2017 Outcrop Mapping Program (Goliath Property)



Source: Treasury Metals (2019).

9.2.9.6 Infill Core Sampling Program

From April 7 to June 14, 2017, Treasury Metals initiated a second infill sampling program intended to assay previously drilled but unsampled drill core. The program was designed to cover all mineralised zones while prioritising intervals within and near the proposed open pit. A total of 5,256 samples were submitted including 525 blanks and standards and covered 142 separate drillholes. The three main objectives for the infill sampling program were to (1) add new gold ounces to be included in the next mineral resource estimate; (2) extend existing gold mineralisation; and (3) uncover any potential new zones. Table 9.10 lists significant assay intersections greater than 1.0 g/t Au

Table 9.10: Significant Assay Intersections Greater than 1.0 g/t Au

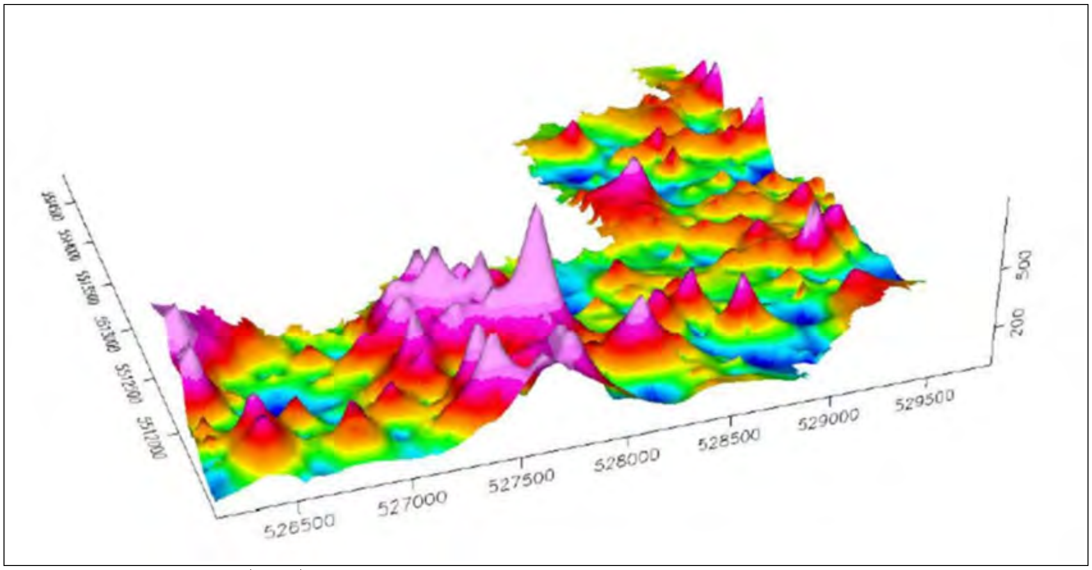
Diamond Drillhole	Section	From (m)	To (m)	Intercept (m)	Au (g/t)	Ag (g/t)	Target
TL0849	527600E	100.00	104.00	3.00	1.61	15.97	E Zone
TL1096	527250E	206.80	211.00	4.20	11.37	P	D Zone
including		208.00	209.30	1.30	34.80	P	
TL10108	527475E	250.00	253.00	3.00	31.38	21.63	HW Zone
including		252.00	253.00	1.00	93.40	64.10	
TL11145	528500E	49.50	52.00	2.50	1.36	9.90	BMS HW
TL11167	527275E	134.30	137.00	2.70	4.52	5.18	
including		134.30	135.00	0.70	15.90	11.70	HW Zone
TL11171	527225E	279.57	284.00	4.43	4.97	1.16	B Zone
including		283.00	284.00	1.00	18.20	0.70	
TL11209A	527075E	43.00	47.00	4.00	8.61	0.99	HW Zone
including		44.00	45.00	1.00	29.80	2.20	
TL12287	527275E	292.00	294.00	2.00	4.12	2.09	HW Zone
including		292.70	294.00	1.30	6.07	2.40	
TL13306	527850E	86.00	90.00	4.00	1.12	1.65	C Zone
TL15387	527550E	143.00	145.00	2.00	3.70	5.38	HW Zone
TL164-12RE	527625E	417.00	419.30	2.25	3.01	N/A	B Zone

The next steps for the program will be to re-visit the portions of the geological model wherein these new results are located to better understand their impact and develop a follow-up program that may include additional infill core sampling and new drillholes.

9.2.10 2018 Exploration Activities

A soil gas hydrocarbon (SGH) orientation survey was carr4ied out consisting of 845 soil samples. The survey can be described as two grids (defined as “eastern” and “western” grids) with sample spacing of approximately 50 m and approximately 200 m between transects. One was conducted across the Goliath deposit and the other near the regional fold nose northeast of the deposit. The survey identified strong redox and gold pathfinder anomalies (see Figure 9-15) on and around the deposit area believed to be caused by gold mineralisation with a high level of confidence (5.5 out of 6 SGH signature rating). With the capability of this surface sampling technique to detect the Goliath deposit, it is recommended to conduct additional sampling across the remaining strike length.

Figure 9-15: 3D View of Western Gold Pathfinder Class map from 2018 Orientation Survey



Source: Treasury Metals (2019).

9.2.11 2019 Exploration Activities

9.2.11.1 Hole-to-Hole Induced Polarisation (IP) Survey

A downhole spectral IP / resistivity survey was completed by Golden Mallard Corporation. Using 15 existing drillholes spanning 1.2 km along strike, this program was designed to outline the chargeability signature of Goliath, to test the high-grade down-dip extension potential below the current resource (400 m below), and to outline new drill targets and detect any previously unknown nearby mineralised concentrations.

The IP survey confirmed the project's gold-bearing zones correlate with high to moderate resistivity and chargeability high and high-low contacts. This is believed to be associated with strong silicification and an increase in disseminated sulphides, both of which are found in the Goliath gold zones. The inversion model suggests that these zones are located on a major structure and has outlined signatures of the high-grade gold shoots including the confirmation of the down-dip extension potential below the current resource to approximately a depth of 800 m below surface. The survey also identified the continuation of the resistivity and chargeability responses on the east and west sides of the resource area, indicating that the zones that host the gold extend along strike of the deposit in both directions.

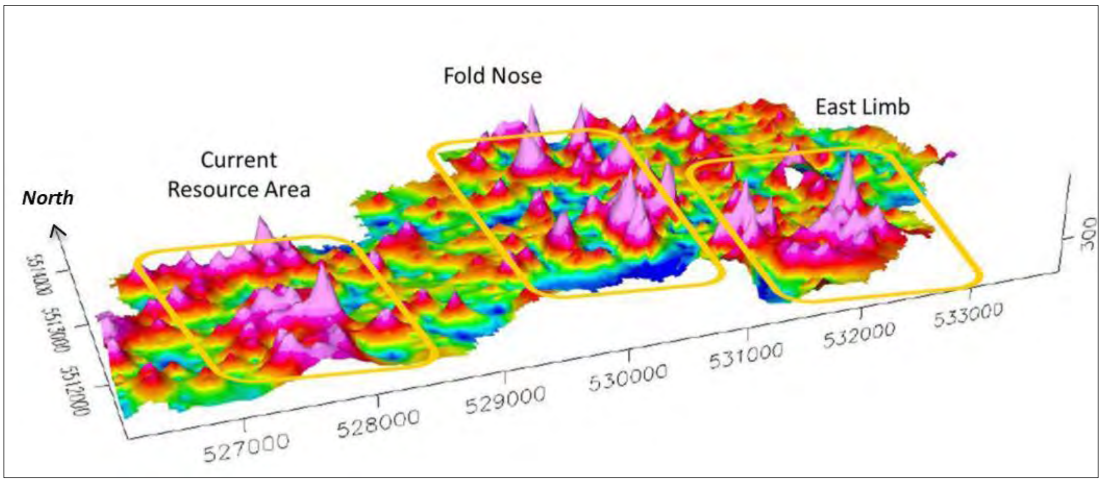
The completed holes have shown positive results and strong correlation to the currently defined resource. The IP results indicate a new valuable use of this technology and will provide Treasury Metals with the ability to define additional high-priority drill targets.

9.2.11.2 Soil Gas Hydrocarbon Sampling Follow-up Program

A follow-up program commenced in 2019 to expand sample coverage along strike length to the east of the Goliath deposit, as well as a number of other areas of interest including highly prospective areas both north of the deposit and on the eastern side of the property. Approximately 1,040 additional samples have been collected, maintaining the 50 m sample spacing and 200 m transects from the orientation survey and covering approximately 10.25 km^2.

The Activation Laboratories SGH interpretation highlighted areas of interest (see Figure 9-16), analogous to the results found across the current resource area and given a confidence rating of 4.0 out of 6.0 (the resource area survey scored 5.0 out of 6.0). Most notably are the anomalies around the nose of a large regional fold which also occur near several potential redox cells. Recommendations for future work include surface investigations of these areas of interest as well as continuing to sample the remaining strike length on the eastern half of the property.

Figure 9-16: 3D View of Main Gold Pathfinder Class map from 2019 SGH Program



Source: Treasury Metals (2019).

9.2.12 2020 Exploration Activities

An additional SGH sampling program was executed by Axiom Exploration in order to complete the sample coverage along the strike length of the Goliath deposit on the eastern half of the property. Approximately 1,260 additional samples have been collected, maintaining the 50 m sample spacing and 200 m transects from the previous surveys and covering approximately 12.50 km^2. The Activation Laboratories SGH interpretation report has been received and an internal review is underway to determine if infill sampling of identified anomalies is required and to assist in the planning of future field programs.

9.3 Goldlund Deposit

Treasury Metals has not conducted any surface exploration on the deposit since the acquisition of the property. Exploration conducted by previous owners is summarised in Section 6 of this report.

9.4 Miller Deposit

Treasury Metals has not conducted any exploration activities on the deposit since the acquisition of the property.

10 DRILLING

10.1 Overview

Much of the information regarding the various Goliath drill programs was sourced from technical reports prepared by Puritch et al. (2015, 2020), Roy et al. (2012) and Roy and Trinder (2011) as well as a number of drilling reports that have been filed for assessment credits by CCIC and Treasury Metals with the MNDM with edits from AGP.

The Goldlund-Miller property was acquired by Treasury Metals in July 2020. The information was sourced from various technical reports prepared by T. McCracken of Wardrop (2010 to 2011), T. McCracken of Tetra Tech (2012 to 2013), S. Zellerer of Tetra Tech (2014) and by T. McCracken of WSP (2015 to 2020).

10.2 Goliath Deposit

The drill programs can be divided in two parts between the historical Teck-Corona exploration drilling carried out between 1990 to 1998 and more recent Treasury Metals drilling carried since 2008. The historical drilling was added to this section since it is considered highly relevant to the resource estimate discussed in section 14 of this report. The following sub sections summarise the various drill programs.

10.2.1 Teck Exploration & Teck-Corona, 1990 to 1998

Thirteen drilling campaigns were undertaken by Teck Exploration and Teck-Corona over an eight-year period from 1990 to 1998. During this period, 340 diamond drillholes were completed for a total of 97,514 m of drilling (see Table 10.1, Figure 10-1).

10.2.1.1 Teck & Teck-Corona Core Handling Procedures

Several different drilling companies were used including Bradley Bros. Limited, Forage St. Lambert Ltd., Boart Longyear Inc. and St. Lambert Drilling Co. Ltd. Drill core size was predominantly BQ in the early years (1990 to 1996) and NQ in the later years. A majority of the drill logs record that the casing was left in the hole upon completion and the hole was capped. Downhole surveys for azimuth and dip were taken normally at 50 m intervals using initially Wel-Nav single shoot instruments and in the latter years using a Sperry-Sun Single Shot downhole instrument supplemented by acid tests when necessary. Usually, the first reading was taken immediately below the casing to ensure the hole was on course. Transit surveys of all drillhole casings within the resource area was completed by W.J. Bowman Ltd.

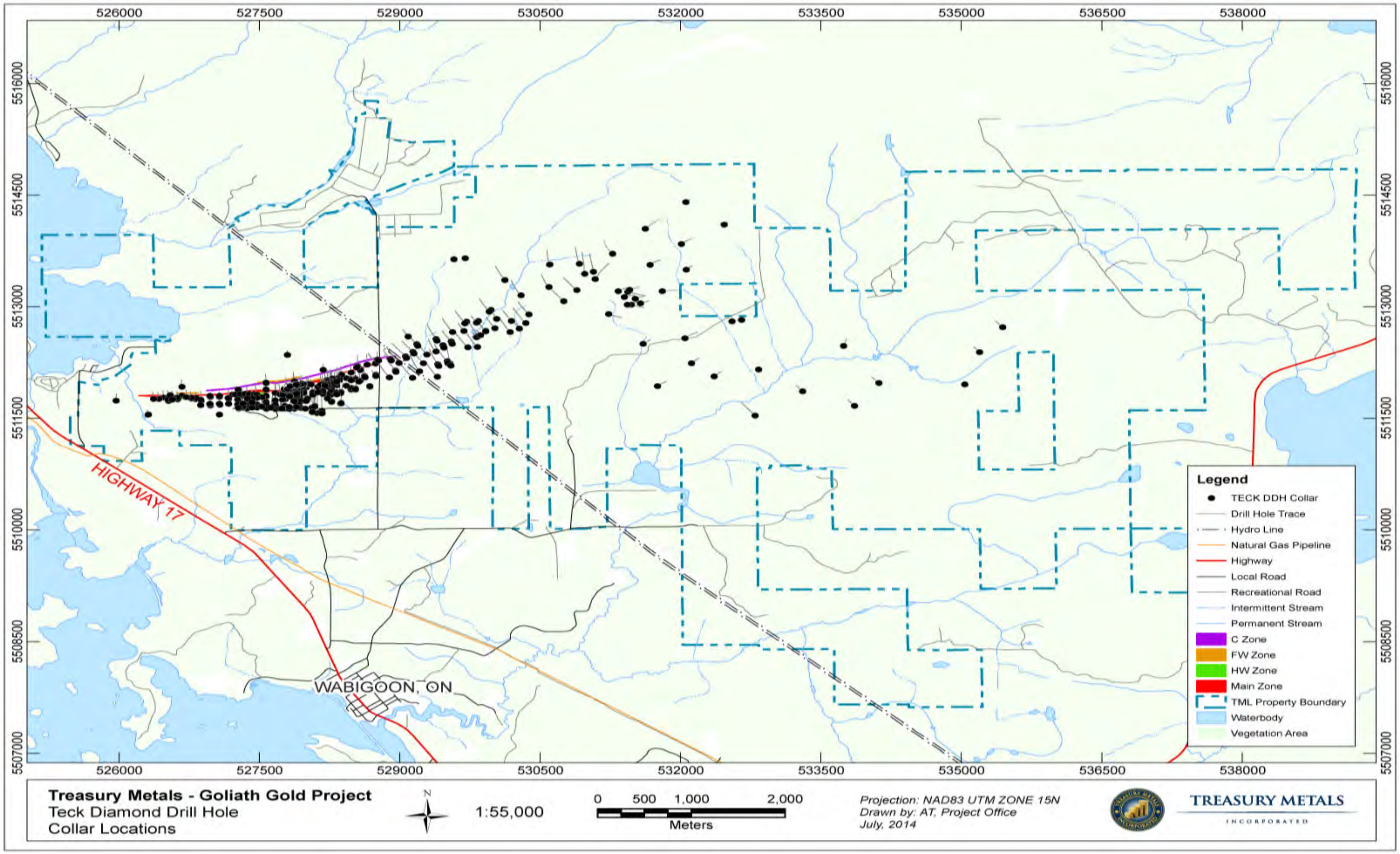
Upon the daily receipt of the drill core at the cores logging facility, the core was logged, marked, and tagged for assay by the geologist. The typed standard “Teck” core logs in PDF format are all available for inspection. For the major intervals, the logs record the rock type name along with a long description that typically contain the rock descriptions, mineralisation, and alteration for the entire interval. The long description was often split in minor intervals describing zones of particular interest.

Table 10.1: Summary of Teck & Teck-Corona Diamond Drilling Program

Drill Program	Year	Holes	Dates Drilled	Hole Numbers	Metres Drilled
1	1990	7	October 28 to November 30, 1990	TL1 to TL7	1,096
2	1991	17	April 18 to May 15, 1991	TL8 to TL24	3,368
3	1992	13	May 13 to June 30, 1992	TL25 to TL37	4,373
4		9	October 16 to November 23, 1992	TL38 to TL43 TL43W1 to TL43W3	3,800
5	1993	10	August 14 to September 10, 1993	TC-1 to TC-10	1,747
6	1994	72	January 18 to November 16, 1994	TL44 to TL110 TL44W1 to TL44W3 TL88W, TL96W	15,998
7	1995	14	January 27 to February 27, 1995	TL111 to TL124	1,814
8		11	November 28 to December 25, 1995	TL125 to TL127 TL125W1, TL125W2 TL126W1 to TL126W3 TL127W1 to TL127W3	5,668
9	1996	18	January 7 to February 8, 1996	TL128 to TL132 TL128W1 toTL128W3 TL129W1 to TL129W3 TLE11 to TLE17	6,250
1		33	June 12 to October 31, 1996	TL133 to TL142 TL133W1 to TL133W3 TL136W1, TL136W2 TL137W1, TL137W2 TLE18 to TLE33	14,598
11	1997	65	January 15 to December 31, 1997	TL143 to TL206 TL170W1	23,232
12	1998	6	May 19 to July 1, 1998	TL207 to TL212	2,831
13		65	September 3 to December 5, 1998	TL213 to TL277	12,739
Total		340			97,514

Source: Treasury Metals (2015).

Figure 10-1: Goliath Drillhole Location Map, Teck Exploration & Teck-Corona, 1990 - 1998



Source: Treasury Metals (2014).

The legend used by Teck-Corona was different than the current legend used by Treasury Metals and the exact date of conversion between the Teck-Corona Legend to the Treasury Metals legend is not known. A significant amount of work was carried out by CCIC to recover the mineralisation, veining, and alteration information from the long description of the logs and populate the appropriate database tables now used by Treasury Metals.

The samples were then sawn in half using a Target masonry saw with a 14″ diamond blade. All samples were shipped to the primary laboratory by Gardwine and Porter transport firms. The primary laboratory used was TSL Laboratories of Saskatoon, Saskatchewan with XRAL Laboratories and Interteck Testing Services used for assay verification work or whole rock analyses.

All core samples were submitted for gold and sporadically assays for silver, copper, lead, and zinc when the gold grade was expected to be high.

In 2008, all recoverable historical Teck and Teck-Corona drill core that was in long-term storage in the town of Wabigoon was moved to Treasury Metals' core farm located on the former tree nursery site. The core is in very poor condition and Treasury Metals was not able to resample the core as part of its resource evaluation work.

The highlights of the various drilling programs completed by Teck during the 1990s have been summarised below. AGP would like to point out that Teck-Corona was looking for high-grade zones in excess of 3.0 g/t and because of that, did not typically report assays that were considered sub-economic at the time but well above a modern open pit cut-off grade.

The Qualified Person also comment that the drilling and core handling procedure described above is consistent with the core handling procedure in place in the 1990s while employed at the David Bell Mine owned by Teck-Corona operating corporation.

10.2.1.2 1990 to 1993 Teck Drilling Programs

Teck's very first diamond drilling program on the Thunder Lake deposit commenced October 28, 1990 to November 30, 1990 with the completion of seven BQ holes (TL1 to TL7) totalling 1,096 m. The discovery hole (TL1) on the Main Zone of the deposit intersected three significant zones of polymetallic disseminated sulphide mineralisation containing gold (Page, 1991):

- Zone A returned 2.23 g/t Au, 18.9 g/t Au, 0.63% Zn over 6.1 m (80.0 to 86.1 m) including 5.25 g/t Au, 16.8 g/t Ag, 0.28% Zn over 1.9 m
- Zone B intersected 0.97 g/t Au over 10.4 m (107.4 to 117.8 m) in a pyritic alteration zone
- Zone C assayed 7.99 g/t Au, 16.5 g/t Ag and 0.61% Zn over 6.1 m (196.7 to 202.8 m) including 17.49 g/t Au, 33.6 g/t Ag and 0.42% Zn over 2.6 m

This hole was drilled to test a "high priority" IP chargeability anomaly determining that this exploration method was very useful in defining potential future drill targets within and on-strike with the Goliath deposit.

Following this discovery, much of the remaining historic exploration on the Thunder Lake property centred on diamond drilling programs with the most drilling having been completed in the area north of the Laramide property in the Thunder Lake West portion; there was minimal drilling on the Thunder Lake East portion in Hartman Township.

Teck completed 17 BQ diamond drillholes for a total of 3,368 m in 1991. Hole TL9 intersected an isolated high of 45.96 g/t Au over a sample length of 0.5 m (44.8 to 45.3 m) in a section of biotite-muscovite schist in the Main Zone. Holes TL21 and TL23, drilled on the same drill section, intersected three sections of high-grade gold mineralisation corresponding to the Main Zone Hanging Wall, Main Zone and C Zone.

Two diamond drilling programs were completed in 1992 with Phase I initiated during the months of May and June and Phase two in the Fall in October and November. A total of 22 BQ holes were drilled (TL25 to TL43) and three wedges were turned off of hole TL43 (TL43W1, TL43W2 and TL43W3) for a total of 8,173 m of diamond drilling. Drillhole TL39 was abandoned due to excessive flattening of the hole and restarted as new hole TL39A.

In 1993, 10 BQ diamond drillholes totalling 1,747 m were drilled to test a series of ground IP geophysical anomalies located in the extreme eastern portion of the property in Hart Township (east of UTM 532400E). The holes were numbered TC1 to TC10. Hole TC6 was a failed hole ending at 135 m and no samples were taken for assay. None of the holes returned any significant gold assays (all less than 0.09 g/t Au). However, many of the IP anomalies were attributed to either the presence of graphite, elevated pyritised rocks or sulphide iron rich metasedimentary rocks.

10.2.1.3 1994 Teck-Corona Drill Program

In 1994 a total of 72 diamond drillholes totalling 15,998 m, including five wedge holes and one abandoned hole, were completed. These drillholes were numbered TL44 to TL110, TL44W1 to W3, TL88W and TL96W and were drilled using both NQ and BQ size rods.

From January to February 1994, Teck completed a 4,846 m diamond drilling program. A total of 34 holes were drilled of which 20 were NQ and 14 were BQ sized core numbered TL44 to TL77. Twelve samples were collected from hole TL44 and dispatched to X-Ray Laboratories in Don Mills, Ontario for whole rock analyses. The best gold assay intersections were obtained from the Main Zone and the most significant drillhole intersection was from TL49 that returned 21.2 g/t Au over a sample length of 8.5 m from 178.0 to 186.5 m. The better auriferous intersections in the Main Zone were characterised by (Page, 1994):

- quartz-sericite schist host rock
- rocks containing 1% to 5% disseminated pyrite with local concentrations of 5% to 20% pyrite
- trace to locally 3% to 5% disseminated and stringer sphalerite accompanied by lesser amounts of galena (trace to 2%), chalcopyrite (trace to 1%) and rare occurrences of arsenopyrite
- intense silicification containing 5% to 25% total sulphides
- rare pinpoint to mm grains of native gold and electrum

Teck also completed a re-logging and sampling program of earlier drillholes, and also re-examined surface exposures and carried out metallic screen fire assaying of most core intersections through the Main Zone (Page, 1995a).

Pulp metallic screen fire assaying determined that there were significant nugget effects present in the deposit reflected in both the assay results and the observed distribution of native gold and electrum (Page, 1995a). Roughly two-thirds (64%) of the 210 samples revealed gold assay results that compared well between the 30 gm fire assay and pulp metallic methods. Just over one-tenth (12%) of the samples returned initial assays much larger than the pulp metallic and around one-quarter (24%) of the samples yielded pulp metallic gold assays much larger than the initial gold fire assay results. It was determined that, although more expensive, utilising pulp metallic screen fire assaying proved to be most useful in defining the overall character and geometry of the deposit.

Highlights of gold assay (> 3.0 g/t Au) returns from the remaining holes drilled in 1994 include holes:

- TL80: 3.53 g/t Au over a core length of 5.6 m (174.7 to 180.3 m) including 10.50 g/t Au over a sample length of 1.5 m (178.8 to 180.3 m)
- TL81: 5.67 g/t Au over 13.2 m (215.0 to 228.2 m)
- TL82: 18.89 g/t Au over 3.7 m (266.5 to 270.2 m)
- TL84: 3.54 g/t Au over 11.0 m (48.4 to 59.4 m)
- TL96: 3.29 g/t Au over 5.4 m (375.4 to 380.8 m)
- TL44W3: 5.64 g/t over 7.9 m (535.5 to 543.4 m)

10.2.1.4 1995 Teck-Corona Drill Program

Fourteen BQ holes totalling 1,814 m, numbered TL111 to TL124, were completed in the early part of 1995. These holes were drilled to delineate a shallow gold resource in the “West Alteration Zone” (TL11 to TL117) to vertical depths of around 80 m and to partially define the west and east edges of the No. 2 shoot to depths of -50 to -85 m (TL119, TL120) and west edge of the No. 1 shoot (TL121, TL122 to a vertical depths of -140 m and -110 m, respectively). Holes TL114, TL117 and TL118 were abandoned prematurely due to drilling difficulties (Stewart, 1995).

Hole TL114 intersected the Main Zone returning 15.81 g/t Au over a core length of 3.0 m (60.2 to 63.2 m) and hole TL118 returned a Hanging Wall/Main Zone intersection of 14.73 g/t Au over a core length of 5.5 m, which includes a single 53.24 g/t Au assay over a core length of 1.5 m (87.2 to 88.7 m).

10.2.1.5 1996 Teck-Corona Drill Program

A winter drilling was program completed from November 1995 to February 1996. A total of eight deep BQ holes, numbered TL125 to TL132, were drilled for a total of 4,142 m to test the Main Zone at a vertical depth of between 400 m and 500 m to the east and west of the No. 1 and No. 2 shoot area (Stewart, 1996).

Drilling resulted in extending the Main Zone in the area of the “West Alteration Zone” in the main deposit to a vertical depth of around 450 m. Hole TL-129 intersected the Main Zone from 433.5 m to 474.0 m grading 2.31 g/t au over a 40.5 m core length which includes grades of up to 16.96 g/t Au over 2.0 m (452.5 to 454.5 m) and 15.47 g/t Au over a sample length of 1.0 m (470.0 to 471.0 m). The Main Zone in the area of the “East Alteration Zone” was extended to a vertical depth of approximately 500 m.

During the winter program, seven BQ holes were drilled (TLE11 to TLE17) for a total of 1,126 m. These were regional exploration holes in the eastern portion of the property, an area called Thunder Creek East, to test a series of both IP and VLF-EM anomalies. Most of these holes encountered amphibolite, garnet amphibolite, and meta-sedimentary rocks (argillites, conglomerates, greywacke, and chert-magnetic bearing iron formation). Geophysical target anomalies were attributed to the presence of graphite and elevated sulphides in the metasedimentary rocks. The best drillhole TLE15 intersected 11.60 g/t Au over a core length of 4.2 m (119.4 to 123.6 m) including 46.74 g/t Au over 1.0 m (122.6 to 123.6 m). Hole TLE16 returned 3.58 g/t Au over a sample length of 1.0 m (57.2 to 58.2 m).

A second phase of diamond drilling was completed from June to the end of October 1996. Ten NQ holes, numbered TL133 to TL142, 20 BQ wedges in 7 holes (2-3 wedges per hole) and nine previous drillholes were extended for a cumulative total of 1,482 m (Stewart et al., 1997). There was also a program of partial re-logging of holes TL41, TL42 and TL59.

The most significant results of the Phase II drilling program was the intersection of high-grade gold mineralisation in hole TL141 and two additional intersections of lower grade mineralisation at the eastern and depth extent of the resource areas (holes TL135 and TL136). In addition, the East Alteration Zone was extended eastward for another 150 m and to a vertical depth of 550 m.

Sixteen exploration holes (BQ) were drilled in the eastern portion of the property to follow-up the high-grade gold intersection by hole TLE15 earlier that year and to test additional IP and VLF-EM anomalies as well as local stratigraphy. These holes were numbered TLE18 to TLE33 totalling 3,359 m. Drilling encountered predominantly amphibolite and metasedimentary rocks (greywacke, biotite schist, mafic schist, graphitic argillites, some iron formation and garnetiferous metasedimentary rocks) some of which were intruded by quartz-feldspar and feldspar porphyry bodies. Hole TLE18 returned 2.38 g/t Au over 0.8 m (81.4 to 82.2 m) and hole TLE27 assayed 1.94 g/t Au over a core length of 1.0 m (168 to 169 m). In each case, gold mineralisation was hosted in amphibolite rocks containing elevated sulphides including sphalerite.

10.2.1.6 1997 Teck-Corona Drill Program

A 64 hole diamond drilling program was completed between January 15, 1997 to December 31, 1997 (Page and Waqué, 1998). The holes, numbered TL143 to TL206, totalled 23,232 m of NQ drilling. Reconnaissance (step-out) drilling program following the eastern extension of the Thunder Lake alteration corridor, east of the deposit, included the completion of 13 drillholes covering 1,400 m of strike length. Drilling east of the resource area was disappointing with only geochemically anomalous gold values being intersected over significant to narrow widths. The best assay intersection was obtained from drillhole TL95 that returned 2.01 g/t Au over a core length of 1.2 m (77.9 to 79.1 m).

The majority of the drilling consisted of resource exploration and delineation of the No. 3 shoot (formally called the “East Alteration Zone”) in the eastern resource area and the West Alteration Zone. A total of 44 new drillholes (and one wedge cut) were completed within the resource area. Nine drillholes defined the high to moderate grade portion of the No. 3 shoot: TL144, 145, 150, 151, 174, 175, 176, 180 and TL181 (Page and Waqué, 1998). Hole TL151 returned 9.49 g/t Au over a sample length of 23.3 m (432.9 to 456.2 m) and hole TL144 intersected 11.81 g/t Au over a core length of 10.5 m (69.0 to 79.5 m).

Seven short holes drilled in the area of the No. 1 and No. 2 shoots confirmed the presence of a “dead zone” between the shoots and erratic gold distribution within the No. 2 shoot. Hole TL190 intersected the best gold intersection returning 26.04 g/t Au over a sample length of 2.3 m (52.2 to 54.5 m). Closely-spaced definition drilling at 12.5 m centres in the area confirmed some nugget effects in both the No. 1 and No. 2 shoots (Page and Waqué, 1998). For example, higher grade intersections in the No. 2 shoot did not appear to correlate well beyond two or three drillholes. The No. 1 shoot demonstrated better grade continuity both along strike and down dip.

10.2.1.7 1998 Teck-Corona Drill Program

In 1998, a total of 71 BQ diamond drillholes totalling 15,570 m numbered TL207 to TL277 were completed in a two-phased program. Previous diamond drilling programs focused on defining gold mineralisation within the Main Zone alteration corridor over a strike length of about 1,800 m to vertical depths of 400 m to 500 m with only a few holes to depths of 700 m to 800 m below surface. Drilling had consisted mostly of closely-spaced (25 m centres) shallow holes for resource definition, multiple wedge cuts to evaluate nugget effects, widely-spaced deeper drilling and reconnaissance drillholes located up to 1,500 m east of the main resource deposit (Page et al., 1999a).

The 1998 drilling program consisted of infill definition drilling plus reconnaissance surface diamond drilling and was completed from (1) May 19, 1998 to July 1, 1998 and (2) September 3, 1998 to December 5, 1998. Drilling was dispersed over a large area of the property and included 25 closely-spaced (25 m to 50 m centres) infill holes within the gold resource area, three holes in the western portion of the property, four deep holes and seven shallow holes in the area adjacent (east) of the gold resource, and 21 reconnaissance to 100 m spaced infill holes covering an additional 2,000 m of strike length in the eastern portion of the property.

In the resource area, 23 holes tested the No. 3 shoot (Main Zone) and two holes tested for the up-dip extension of the C Zone. The C Zone holes (TL249 and TL251) returned only anomalous gold values. Four intersections of greater than 3.0 g/t Au over 3.0 m were returned from the No. 3 shoot drilling (holes TL225, TL234, TL238 and TL244).

Drillholes located west and east, and less than 1,000 m along strike of the resources did not return any significant intersections. Hole TL212 returned 1.33 g/t Au over a core length of 5.5 m (219.0 to 224.5 m) in strongly altered Main Zone rocks.

Fifteen holes totalling 3,737 m were drilled to test the alteration corridor over an additional 1,100 m strike length from grid line L14+00 E to L25+00E. These widely spaced reconnaissance and infill drillholes returned anomalous gold values with rare assays exceeding 3.0 g/t Au. Hole TL271 returned 17.36 g/t Au and 754.5 g/t Ag over a core length of 1.6 m from 59.2 to 60.8 m in a weakly sericitic zone containing abundant silver-rich electrum. However, two follow-up holes, drilled 25 m on either side of TL271, did not return any significant gold values in the target locations. These two holes returned best assays of less than 0.10 g/t Au in TL275 and 0.8 g/t Au over 1.0 m from 60.5 to 61.5 m in TL276. Hole TL208 contained an isolated stringer of visible gold yielding a high-grade single assay of 43.3 g/t Au over a core length of 1.5 m (532.5 to 534.0 m) obtained from a zone located 40 m above what is interpreted to be the Main Zone in this area. Drillhole TL272 returned a single high-grade assay of 9.47 g/t Au over a sample length of 1.1 m from 187.7 to 188.8 m.

Six holes totalling 2,013 m were also drilled in the vicinity of the regional-scale synformal fold hinge (an area called the fold nose). This program was designed to test a number of anomalous sericite schist and sulphide showings, several IP anomalies, and interpreted structures. All drillholes in the fold nose returned multiple short intervals of anomalous gold hosted in virtually all rock types in this area usually associated with quartz veining and/or increased sulphide content. While two of the holes returned single high-grade assays in excess of 3.0 g/t Au, Teck could not define any localised structure or rock type that would have allowed focussing of alteration and mineralisation in the fold nose area.

10.2.1.8 1998 Corona Gold Corporation (Jones Property/Lot)

Corona Gold Corporation (Corona) conducted a small diamond drilling program on its 100% owned Jones property (or “Lot”), land Parcel PA3830, from early October to early December 1998 (Page and Waqué, 1999). This parcel is located in the south part of Lot 8, Concession IV in Zealand Township. A total of 12 shallow NQ drillholes totalling 1,452 m were drilled at close spacing’s (50 m centres) to intercept the western Main Zone extension targeting the zone at vertical depths of 25 m to 85 m from surface. The holes were numbered TL252, TL254 to TL256, TL258 to TL261, TL263, TL273, TL274 and TL277. Drilling was undertaken to follow-up on favourable gold intersection obtained from the first-pass drillholes which covered the full strike length of the claim package. The initial nine drillholes (TL252 to TL263) tested 500 m of strike length along the Main Zone.

According to Page and Waqué (1999), the results of this drilling program were disappointing. In this area, the Main Zone is only weakly mineralised with sericitic alteration of variable intensity and silicification, quartz and sulphide veining as well as intense deformation fabrics was found to be generally lacking. Overall, the assay results from all drillholes completed during this program were consistent with the character of a weakened mineralised system. Hole TL274 intersected the best mineralisation returning 4.30 g/t Au over a sample length of 2.6 m (29.0 to 31.6 m). The highest grade was returned from hole TL259 that intersected 5.81 g/t Au over a core length of 1.4 m (61.0 to 62.4 m).

It was concluded that the potential for gold mineralisation decreases significantly further west of the main resource area along the Main Zone structure and it was recommended that no further work be completed on the Jones property.

10.2.2 Treasury Metals, 2008 to 2020

Treasury Metals has conducted 18 diamond drilling campaigns on the Goliath property since 2008. A total of 180,269 m has been drilled by Treasury Metals on the property since 2008 including 528 newly collared holes, 30 re-entry holes and 4 wedge holes (see Table 10.2, Figures 10-2 and 10-3).

Table 10.2: Treasury Metals Drill Program

Drill Program	Year	Dates Drilled	Hole Numbers	Metres Drilled
1	2008	February 18 to September 21, 2008	TL0801 to TL0855	13,121
2	2009	October 20 to December 15, 2009	TL0956 to TL0986	4,589
3	2010	February 20 to March 29, 2010	TL1087 to TL1094	5,211
4		May 2 to June 2, 2010	TL1095 to TL10112	5,153
5		December 2 to December 19, 2010	TL10113 to TL10118	1,818
6	2011	January 17 to September 1, 2011	TL11119 to TL11229	48,538
7	2012	January 25 to June 6, 2012	TL12230 to TL12277	16,110
			TL220-12RE, TL234-12RE, TL231-12RE, TL219-12RE, TL216-12RE	
8		October 22 to December 14, 2012	TL12278 to TL12295	6,540
			TL164-12RE, TL0852-12RE, TL230-12RE	
			TL227-12RE, TL226-12RE, TL238-12RE	
			TL242-12RE, TL148-12RE, TL225-12RE	
			TL0826-12RE	
9	2013	January 7 to February 26, 2013	TL13296 to TL13336	7,772
			TL176-13RE, TL180-13RE, TL223-13RE	
			TL1095-13RE, TL10107-13RE	
			TL0827-13RE, TL10113-13RE	
10	2014	January 23 to June 23, 2014	TL14337 to TL14371	10,749
			TL0855W2b, TL166-14RE, TL161-14RE	
			TL0851-14RE, TL10109-14RE	
			TL0855W1, TL0855W2, TL0855W2b	
11		November 27 to December 19, 2014	TL14372 to TL14377	1,614
12	2015	January 8 to March 17, 2015	TL14378B to TL15402	7,263
			TL14373-15RE, TL14377-15RE	
13	2016	August 24 to January 15, 2017	TL16403 to TL16420	12,154
			TL16415W1	
14	2017	January 10 to March 16, 2017	TL17421 to TL17445	4,022
15		June 22 to October 31, 2017	TL17446 to TL17463	4,494
16	2018	January 8 to June 22, 2018	TL18464 to TL18501	20,987
17	2019	November 15 to Dec 14, 2019	TL19502 to TL19513	4,468
18	2020	January 4 to March 6, 2020	TL20514 to TL20528	5,667
Total				180,269

Figure 10-2: Goliath Drillhole Location Map, Treasury Metals, Western Goliath Property

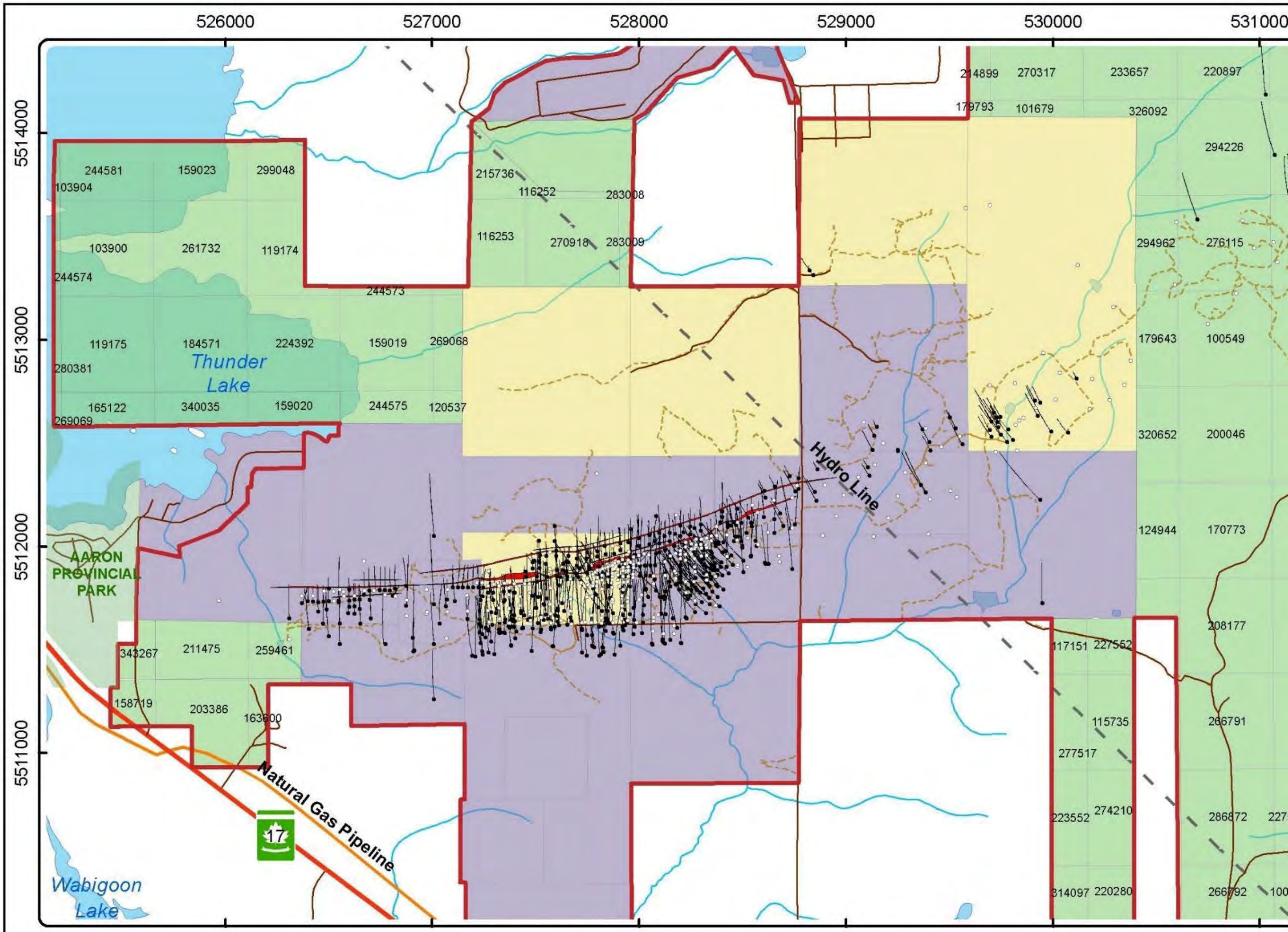
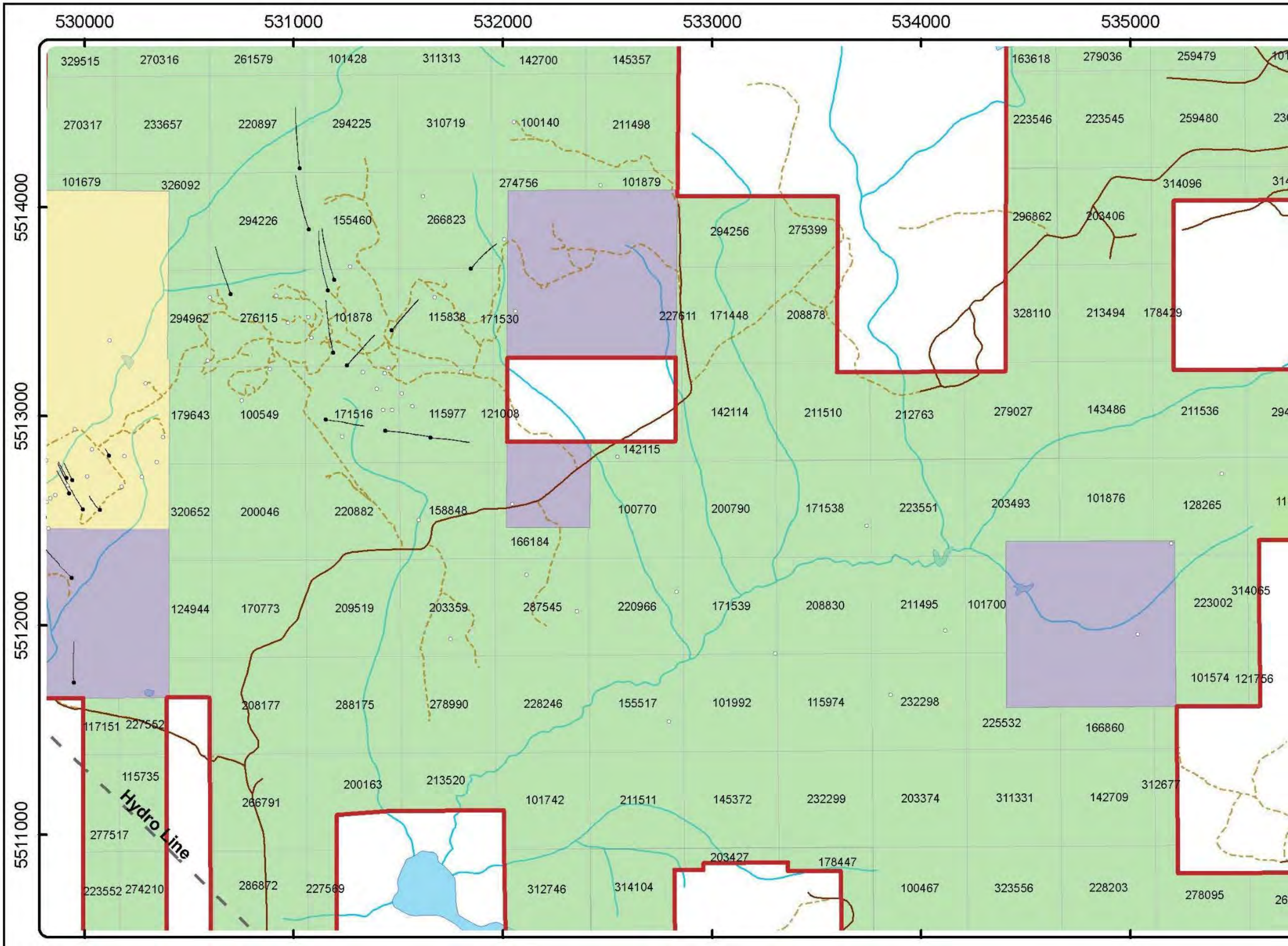


Figure 10-3: Goliath Drillhole Location Map, Treasury Metals, Eastern Goliath Property



10.2.2.1 Treasury Metals, Core Handling Procedures

Caracle Creek International Consulting Inc. (CCIC) designed and supervised all of the drilling programs from 2008 to 2010. In February 2011, Treasury Metals geological staff took over the direct supervision of all Goliath exploration activities.

Over the last 12 years, Treasury Metals has used four different drilling contractors to complete the drilling programs (see Table 10.3). The majority of the drill contracts were awarded to Distinctive Drilling Services Inc. of Westbank, BC, from 2009 to 2013 and George Downing Estate Drilling Ltd. (Downing Drilling) of Grenville-sur-la-Rouge, QC, from 2014 to 2020. Other contractors include G & O Diamond Drilling Contractors Ltd. (G&O) of Hay Lakes, AB, which drilled the first 37 holes of the 2008 drilling campaign and North Star Drilling Limited of Thunder Bay, ON, in 2014. All holes were drilled with NQ or NQTK (NQ2) size core which have a nominal diameter of 47.6 mm and 50.7 mm, respectively.

Table 10.3: Drilling Contractors by year

Drilling Years	Drill Contractor Name
2008	G & O Diamond Drilling Contractors Ltd.
	North Star Drilling Limited (Thunder Bay)
2009 to 2013, 2018	Distinctive Drilling Services Inc. (B.C.)
2014 (January to June)	North Star Drilling Limited (Thunder Bay)
2014 to 2020	George Downing Estate Drilling Ltd.

Source: Treasury Metals (2020).

Each drill contractor constructed drill access trails and drill pads for each setup with water supplied by pump from local beaver ponds, creeks, and streams. A Reflex single-shoot down-hole survey tool is used to survey the holes with readings taken at 50 m intervals. The drill casing is left in each hole and the hole capped to allow for future downhole geophysical testing and/or deepening of the hole.

Each hole is initially surveyed with a GPS handheld instrument in UTM coordinates (NAD83 Zone 15N) and upon completion holes are surveyed using a high precision Trimble survey instrument for higher accuracy. Oriented core drilling was implemented for holes TL0822 to TL0837 using an EzyMark tool provided by Boreinfo Ltd. The objective of this oriented core drilling was to clarify the spatial relationships between structural features and their influence on the mineralisation (Roy et. al, 2012).

The drill core was logged, split, and stored at the exploration field office and core logging facility in Dryden under the supervision of the CCIC staff from 2008 to 2010. Once Treasury Metals staff took over the project management, they moved their operations to the former 136 ha Tree Nursery facility located at the end of Tree Nursery Road which they purchased in 2011 (building and surface rights). This facility includes a large office building with a core logging and core cutting room, additional large warehouses which are used for storing pulps, rejects and drill core and there is also a core farm on site. A gate has been set up on the road at the pond restricting access to the site and the main office building is monitored by a security alarm system.

As the core boxes arrive at the core logging facility from the drill, the meterage in each box is recorded and verified by a technician and hole number and meterage interval label tags are made using a dymo gun or handwritten on an aluminium tag and stapled to the end of each box. Rock-quality designation (RQD) is also determined for each hole. Overall, core recovery has been excellent.

The geologist then logs and marks out samples for assaying. Treasury Metals uses DHlogger™ and log directly into the software. Sample lengths are adjusted as necessary to reflect geological and/or mineralisation contacts. Sample assay tags are placed in the box by the geologist. In general, samples range in width from 0.2 to 1.5 m with the majority of sampling being 1.0 m or 1.5 m in length. Longer sample lengths have occasionally been collected of strongly sheared core sections with poor core recoveries. All drill core boxes are photographed after they have been logged and sampled.

Samples are spilt using a core saw to retain half of the sampled sections for future verification and metallurgical testing (if required). Sample tags are placed in the bags and the sample number is written on the bag using a black permanent marker pen. Samples are then sealed in plastic sample bags using zip-straps, placed in sealed and numbered rice bags. Samples were originally shipped by courier to Accurassay in Thunder Bay. In 2016, Treasury Metals started to use the ActLabs facility in Dryden, Ontario and the samples were then delivered by company personal. Laboratory and assaying procedures are discussed in detail in Section 11 of this report. Core boxes are placed in long-term storage on site at the core farm.

Samples are analysed for gold (fire assay), silver, zinc, lead, and trace element geochemistry (ICP) as discussed in Section 11. Digital assay files provided by the laboratories are merged directly into the Datamine digital database using DHlogger and DHexplorer software to avoid errors in transferring data.

The majority (81%) of the 545 bulk density sample measurements were carried out on 10 cm core pieces submitted to the analytical laboratory. The remaining 19% were completed in-house on un-coated, air dry samples. The core at Goliath is solid with little to no pore and the in-house density measurements compare well with the laboratory figures.

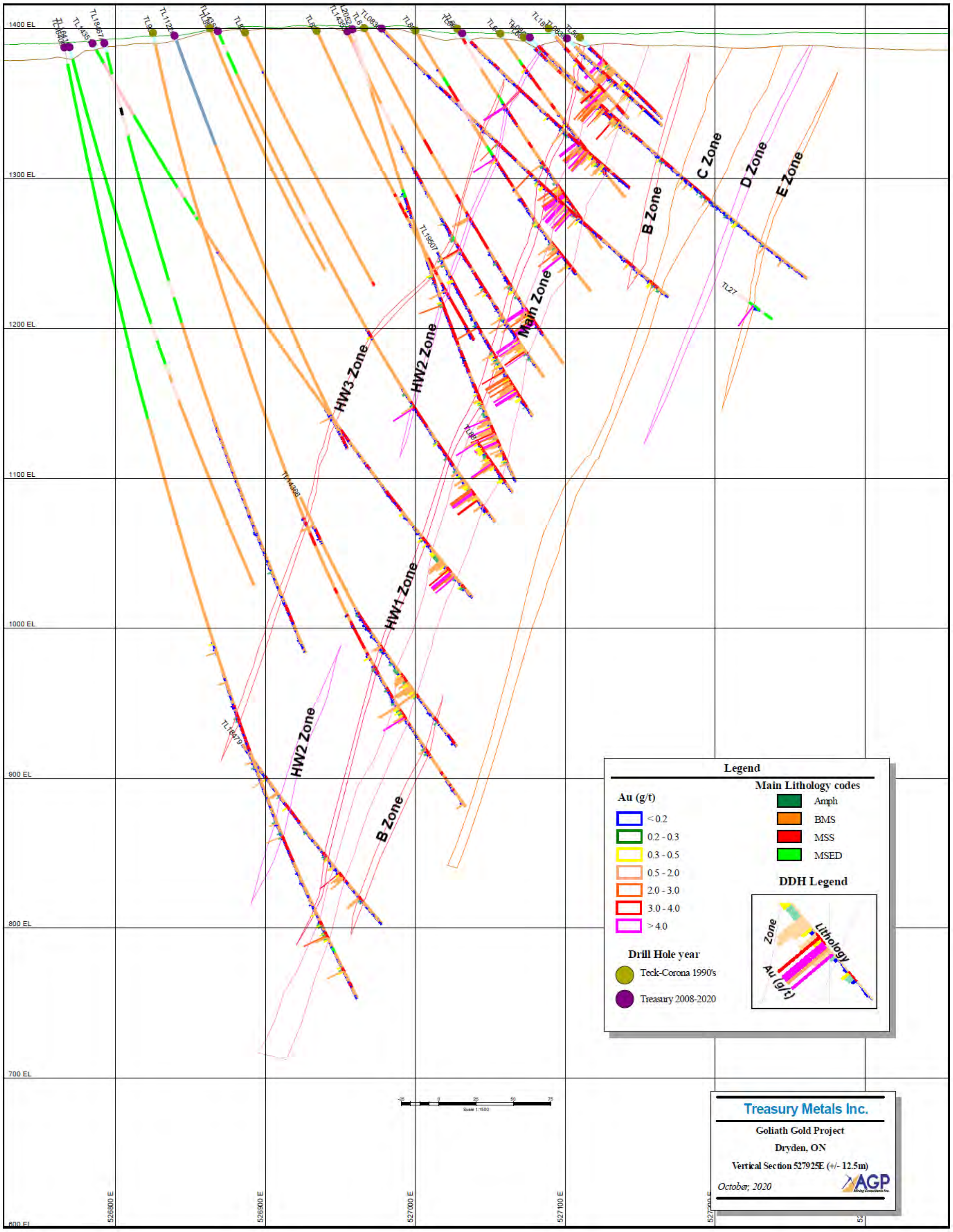
Figures 10-4 and 10-5 display representative cross-sections of the Goliath deposit showing the Teck-Corona drillholes along with the Treasury Metals drilling. Each of the various drilling campaigns completed by the Company over the last ten years is summarised below.

10.2.2.2 2008 Diamond Drilling Program

Fifty-five NQ2 diamond drillholes were drilled on 21 drill sections for 13,121 m from February 15, 2008 to September 22, 2008. This program targeted the Main Zone over a strike length of 1,700 m within the resource-defined area to a maximum vertical depth of around 695 m (hole TL0835). The drill contracts were awarded to G&O who drilled the first 37 holes and North Star completed the remainder. The objective of this program was three-fold (Ilieva, 2009):

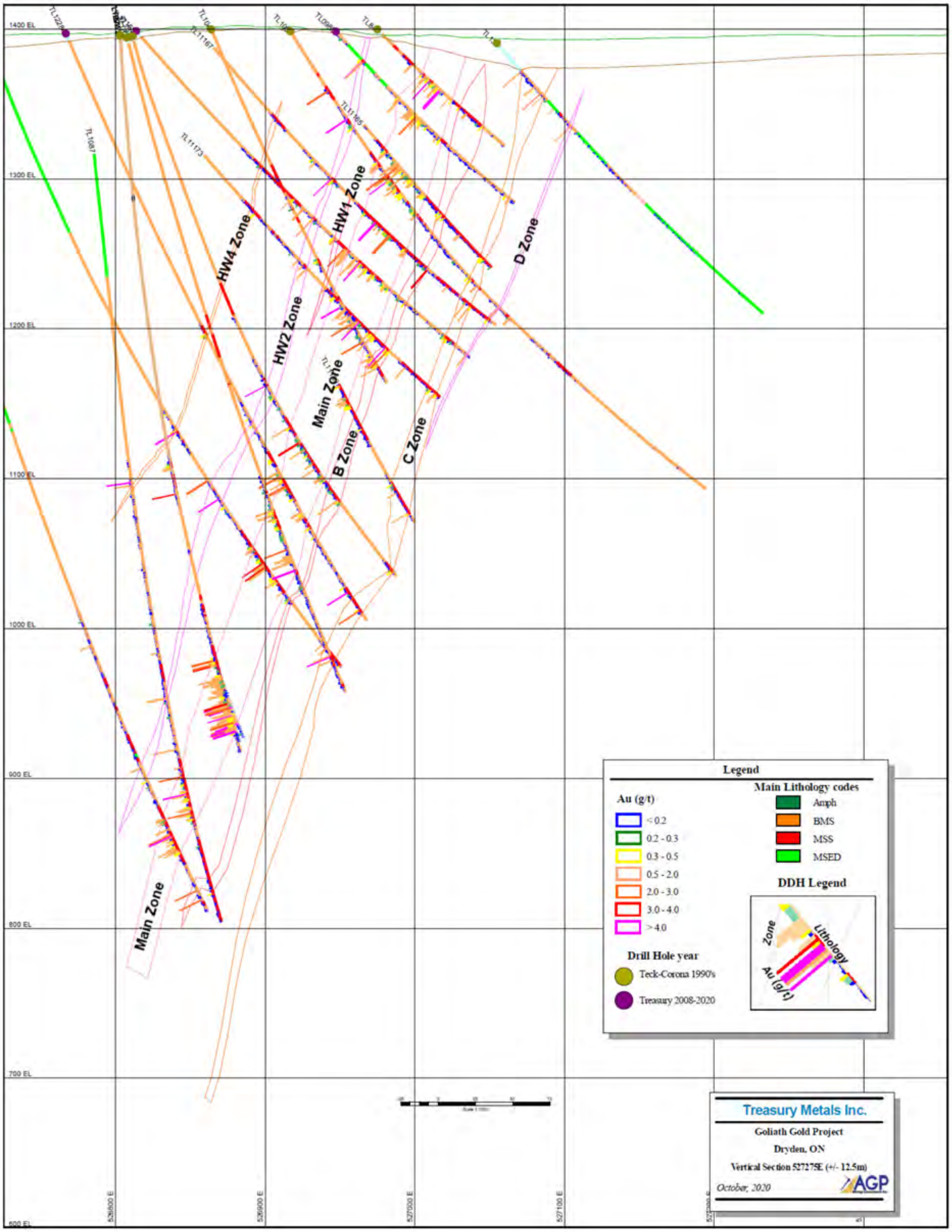
- to confirm and add potential gold ounces to the historical inferred mineral resource of the Thunder Lake deposit (now referred as the “Goliath deposit”)
- to include not only gold but also silver, zinc, and lead assays to eventually prepare a new resource estimate of the deposit
- to target deeper (>400 m) down dip extensions of known gold mineralised shoots

Figure 10-4: Goliath Cross-Section 527925E Looking West



Source: AGP (2020).

Figure 10-5: Goliath Cross-Section 527275E Looking West



Source: AGP (2020).

This new drilling data was integrated into the mineral resource estimate prepared by A.C.A. Howe International Limited PEA in December 2008 (Roy, 2008).

Holes were drilled at azimuths of 360° or 180° with the inclination of each hole set at -45° or -60°. The first 10 holes (TL0801 to TL0810) were drilled in close proximity to former Teck drillholes along the deposit to confirm historical gold assays as well as testing the areas that were not previously sampled Teck. Drillholes TL0801 to TL0837 were completely sampled from top to bottom. Once it was confirmed by CCIC that the gold mineralisation was associated with the MSS unit and visible occurrences of sphalerite and galena, sampling was focused mostly on these targets and the Main Zone. Magnetic susceptibility readings were collected from 7,430.1 m of drill core using a handheld KT-9 Kappameter instrument.

All of the diamond drillholes intersected and tested the Main Zone which consisted of the Hanging Wall (M1) and Footwall (M2) sub Zones. Intersection core lengths of this zone ranged from 5.0 to 30.4 m (hole TL0836). Mineralised intervals were often narrow (up to 0.5 m) zones enriched with 3% to 5% visible sulphide, locally up to 15% (Ilieva, 2009). The main sulphide mineral phases identified were pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite, arsenopyrite and dark grey needles of stibnite in decreasing order of abundance. These sulphides occur as disseminations, blebs, and stringer as well as cubic in the case of galena.

Visible occurrences of gold and electrum (gold-silver) are rare and are observed mostly in the MSS units and in leucocratic sericite-rich bands. For example, very rare specks of visible gold were found in holes TL0815 and TL0817 and downhole depths of 50.8 m and 129.2 m, respectively.

All of the holes intersected gold-bearing sulphide mineralisation many returning significant assay results for gold silver, zinc, and lead.

Gold concentrations were found to be independent of pyrite content. However, an increase in the pyrite (especially coarser grained pyrite) and sphalerite content corresponded to increases in both gold and silver grades. Grade wise, it was determined that an increase in chalcopyrite and galena did not seem to affect the overall gold content or grade.

CCIC concluded that "low-grade gold-silver mineralisation is pervasive throughout the Main Zone, but the high-grade gold (>3.0 g/t Au) is concentrated in steeply west-plunging "shoots" with relatively short strike-lengths up to 50 m, good down-plunge continuity and remained open at depth". Very rare flakes aquamarine green mica (fuchsite- Cr muscovite) were found to occur in the strongly altered sericite alteration in association with high-grade gold.

10.2.2.3 2009-2010 Diamond Drilling Program

Four phases of drilling were completed from October 2009 to the end of 2010. The purpose of this drilling program was to (1) follow-up on the results of the 2008 drilling program with "infill" drilling to better define the resource in and around the Main Zone and expand it at depth and along strike, and (2) to conduct exploration drilling to expand the known resource along strike to the west and to the east and at depth (Magyarosi and Peshkepia, 2011).

Sixty-three NQ holes were drilled on 28 drill sections for a total of 16,672 m testing the gold potential of the main deposit over a strike length of around 2.0 km. The drill contract was awarded to Distinctive Drilling Services Inc. (Distinctive Drilling). All holes were drilled approximately perpendicular to the mineralisation with azimuths of 360°and 320° and dips ranging between -45° and -87°.

Drilling was conducted in four phases over the 14-month period. Phase 1 was carried out in the fall of 2009 with 31 holes drilled for a total of 4,590 m (TL0956 to TL0986) with most of this work being concentrated in the western portion of the deposit. Phase 2 was completed in the spring of 2010 and includes eight holes numbered TL1087 to TL1094 for a total of 5,111 m. Phase 3 was initiated in the summer of 2010 where 18 holes were drilled for a total of 5,153 m (TL1095 to TL10112). The final phase of drilling was carried out in December 2010 with the completion of six holes totalling 1,818 m numbered TL10113 to TL10118. The majority of the 2010 drill program tested primarily the eastern flank of the Main Zone as well as its down-dip gold potential.

The drilling program was successful by extending the known mineralisation and alteration corridor an additional 650 m to the west, 200 m to the east and tested the gold potential of the Main Zone to a vertical depth of 720 m.

10.2.2.4 2011-2012 Diamond Drilling Program

10.2.2.4.1 Overview

Treasury Metals completed three diamond drilling programs from January 17, 2011 to December 13, 2012 with the completion of 192 NQ2 drillholes totalling 70,775 m (Table 10.1). This drilling included 15 re-entry holes to extend historical Teck drillholes (Krocker and Wolfe, 2013). The objective of this drilling was three-fold:

- to confirm and increase the confidence level of indicated gold resources at Goliath
- to locate additional gold mineral resources at depths no more than 400 m from surface in and around the Main Zone focusing on the western shoot and on the eastern flank of the Main Zone; several former Teck holes were re-entered in 2012 to test the gold potential of the C Zone
- to test new exploration targets that reside on strike with the Goliath deposit to the northeast following the known alteration corridor and other potential targets elsewhere on the property (reconnaissance exploration drilling)

Drilling was contracted to Distinctive Drilling. The 2011 holes were drilled approximately perpendicular to the mineralised zone with azimuths ranging from 312° to 005° and dips ranging from -50° to -87°. Most of the 2011 drilling was concentrated in the eastern portion of the main resource deposit. The new drilling data collected from the main deposit was integrated into the new mineral resource estimate prepared by A.C.A. Howe International Limited PEA in 2010 and updated resource estimate in 2011 (Roy and Trinder, 2011; Roy, 2010).

According to Krocker and Wolfe (2013), compilation work indicated that there was approximately 11.5 km of potential strike length of the alteration corridor that hosts the Goliath deposit heading east throughout the remainder of the property to the far northeast corner of the property claim block. The folded stratigraphy (nose area) is clearly illustrated by the Fugro airborne magnetic survey data.

The 2012 drilling program including further drilling of the main resource deposit and exploration the gold potential of this 11.5 km proposed alteration corridor. A reconnaissance exploration drilling program was initiated to:

- drill test the northeast strike extension of the main deposit in areas where Teck had previously intersected some high gold assay values (Parcel 0138 and legacy claims 1119559 and 1119560)
- drill test the large fold nose centred around claim 1144580 where F_2 folds were thought to possibly concentrate gold mineralisation (holes TL12244 to TL12254)
- explore for similar Goliath deposit geology utilising a north-northwest-trending fence of four holes (covering legacy claims 3017880 and 1144553) to test 1,200 m of prospective stratigraphy to a vertical depth of 300 m where alteration and gold mineralisation was anticipated to occur (holes TL12266, TL12262, TL12271 and TL12277)

Hole azimuths for the 2012 drilling ranged from 320° to 360° with hole dips ranging between -45° and -70°.

During September-November 2011, ACA Howe International Limited prepared a new mineral resource estimate for the Goliath deposit using the historical and Treasury Metals drillholes completed up to hole TL11228 (Roy and Trinder, 2011). That mineral resource estimate was used to prepare the preliminary economic analyses of the gold deposit in 2012 (Roy et. al, 2012).

10.2.2.4.2 2012 Drilling Results

Highlights of the 2012 drilling include the following:

- Hole TL12245 intercepted 2.27 g/t Au and 2.5 g/t Ag over a sample length of 3.0 m (51.0 to 54.0 m)
- Hole TL12235 drilled to test the westernmost strike extension of the main resource area mineralisation returned 1.05 g/t Au and 1.25 g/t Ag over a sample length of 3.32 m (199.18 to 202.5 m) within the C Zone
- Re-entry hole TL148-12RE assayed 17.13 g/t Au and 9.0 g/t Ag over 1.5 m from 201.0 to 202.5 m within a lower grade C Zone of 1.05 g/t Au and 1.2 g/t Ag from 172.5 to 202.5 m
- Hole TL164-12RE intersected 5.87 g/t Au and 9.26 g/t Ag over a sample length of 17.13 m (485.31 to 502.44 m) including 18.64 g/t Au and 26.94 g/t Ag over 5.2 m (485.31 to 490.50 m) with visible gold
- Hole TL12293 returned 2.47 g/t Au and 2.70 g/t Ag over a core length of 10.65 m (33.25 to 43.90 m) including 6.65 g/t Au and 7.0 g/t Ag over 2.25 m (33.25 to 35.50 m) near surface in the C Zone

The most northwest exploration fence hole TL12266 on legacy claim 1144553 returned 2.62 g/t Au and 2.48 g/t Ag over a core length of 2.1 m (336.16 to 338.25 m), including 3.67 g/t Au over 1.0 m (337.25 to 338.25 m), hosted in an MSS unit surrounded by BMS rocks in association with elevated pyrite and trace chalcopyrite. The other three holes to the south did not return any significant assays. These results clearly demonstrate that the alteration corridor hosting gold mineralisation is still present in the eastern portion of the Goliath property.

Two exploration drillholes (TL12247 and TL12255) intersected several massive to semi-massive sulphides, mostly consisting of pyrrhotite and pyrite bands up to 30 cm wide hosted in mafic volcanoclastic amphibolite rocks with minor meta-sedimentary rocks. These holes were collared on claim 1119545 in the nose of the regional fold structure. Hole TL12247 intersected several 20 to 30 cm wide semi-massive sulphide intervals containing predominantly pyrrhotite with lesser amounts of pyrite from 291.0 to 343.0 m. The second hole intersected seams and stringers of massive sulphides hosted in biotite schist and amphibolite rocks within seams 1 to 10 cm thick. The sulphide enriched units did not contain any significant base metal mineralisation. However, hole TL12247 returned 17.52 g/t Au and 2.0 g/t Ag over a sample length of 1.5 m (22.5 to 24.0 m) in a metasedimentary rock and 4.86 g/t Au and 2.0 g/t Ag over 1.0 m (103.0 to 104.0 m) in a biotite mica schist.

10.2.3 2013 Drilling

From January 7, 2013 to February 26, 2013, Treasury Metals completed 48 NQ2 diamond drillholes totalling 7,773 m. This program consisted of 41 holes numbered TL13296 to TL13336 and seven re-entry holes on former Teck drillholes.

The primary objective of the drilling program was to further delineate the C Zone within the proposed open pit to convert inferred gold resources to the indicated resource category and to add ounces to the open pit. Drilling was focused along the main deposit over a strike length of 1.5 km. Additional exploration work focused on the C Zone high-grade gold shoot discovered in the central part of the Goliath deposit intersected approximately 50 m after the Main Zone mineralisation. A re-interpretation of the geology concluded that the re-entry holes were required in order to extend the Teck holes past the Main Zone to test the gold potential of the C Zone that was largely unknown during the Teck drilling programs in the 1990s. The C Zone mineralisation within MSS rocks usually starts downhole around 30 to 60 m past the Main Zone.

This drill contract was awarded to Distinctive Drilling. Holes were drilled north with azimuths ranging from 355° to 045° with the exception of hole TL13315 that was drilled south at 190°. Collar dips ranged from -45° to -80°.

A number of significant C Zone intersections were reported on company press releases. It was concluded that drilling of the proposed open pit mine shell was successful in providing significant gold intersections of the central shoot of the C Zone and in adding ounces to the resource inventory and reducing overall waste to potential ore stripping ratios, especially in the eastern portion of the deposit. The hole extensions also lead to the discovery of several new mineralised zones, including the B Zone intercepts hosted in the BMS unit located between the Main Zone and C Zone.

At the completion of the program, Treasury Metals performed a gap analysis to determine what further diamond drilling would be required for future resource conversion from inferred to the indicated classification within the proposed open pit design focusing on the Main and C Zones to propose an expanded 2014 infill diamond drilling program.

10.2.4 2014 Drilling, Phase I

In 2014, Treasury Metals completed Phase 1 diamond drilling program from January 23, 2014 to June 23, 2014. A total of 42 NQ2 holes were drilled for a total of 10,294 m. This drilling consisted of 35 holes numbered TL14337 to TL14371, five re-entry holes of both Teck and Treasury Metals historical holes and three wedge holes turned off of Treasury Metals hole

TL0855 previously drilled in 2008 (Table 10.1). Drillhole TL14363 was abandoned at a depth of 50 m. None of the core in that hole was mineralised.

This program consisted of infill and expansion drilling of the Main and C Zones, further delineation of the new high-grade zone discovered in the central portion of the C Zone and exploration drill testing of targets on its Norman property acquisition, located east of the deposit, which Treasury Metals purchased the surface rights to in 2014 (holes TL14337 and TL14338). The Norman property is contiguous to and located along strike and down dip of the eastern end of the mineral resource at Goliath. Prior to that purchase, Treasury Metals was not allowed surface easement on that property. The new acquisition allowed for the first-time access for drilling on an additional 1.6 km of potential deposit strike length given that the resources defined at that time were interpreted to project towards the northeast portion of this new ground.

This program focused considerably on both exploring and developing the C Zone target both near surface and at depth to add to potential open pit and underground resources. The purpose of the re-entry holes was to extend drillholes to evaluate the C Zone where these original holes were initially terminated after the Main Zone. Further delineation efforts of the Main Zone were also implemented to tighten grades and extend limits of known mineralisation within the westward plunging shoots, which included additional infill drilling.

The drill contract was awarded to North Star Drilling. The majority of the holes along the main deposit were drilled north with azimuths ranging from 320° to 005° with the exception of hole TL14356 that was drilled southeast at 145° in the central portion of the deposit. Collar dips ranged from -49° to -77°.

Highlights of the drilling program include the following notable intersections of the C Zone:

- TL14343: 4.32 g/t Au and 32.50 g/t Ag over 3.0 m (16.3 to 19.3 m) in the western portion of the C Zone
- TL14346A: 4.69 g/t Au and 6.67 g/t Ag over 6.4 m (317.0 to 323.4 m) including 27.23 g/t Au and 29.0 g/t Ag over 1.0 m (319.4 to 320.4 m) in the western area of the C Zone
- TL14349: 2.2 g/t Au and 3.48 g/t Ag over 9.3 m (112.7 to 122.0 m) approximately 30 m below hole TL14350
- TL14350: 5.39 g/t Au and 14.59 g/t Ag over 6.7 m (79.33 to 86.00 m) including 28.41 g/t Au and 93.0 g/t Ag over 1.0 m (81.33 to 82.33 m) was intersected in the C Zone at a vertical depth of 60 m from surface
- TL14356: 2.69 g/t Au and 8.87 g/t Ag over 13.5 m (111.5 to 125.0 m) in the C Zone that was drilled down dip on the mineralisation
- Wedge hole TL0855W2b: a step-out exploration hole that that intersected 3.64 g/t Au and 2.5 g/t Ag over 5.75 m (561.50 to 567.25 m) in the C2 sub Zone with visible gold located 36 m west of previous C Zone hole TL164-12RE (18.64 g/t Au over 5.2 m reported above)
- TL161-14RE: 4.94 g/t Au and 44.0 g/t Ag over a sample length of 4.0 m (485.0 to 489.0 m)

At the conclusion of drilling program, Treasury Metals determined that the C Zone remained a "high priority" exploration target that remained open to the west and down plunge at depth.

Two exploration holes were drilled on the Norman ground collared on land Parcel 0141 with only one gold assay intersection. Hole TL14337 was targeting an EM anomaly identified from the Fugro airborne geophysical survey as well as testing the potential to intercept down dip MSS mineralisation intersected by nearby Teck hole TL272 that returned 9.47 g/t Au over a sample length of 1.1 m. However, this hole did return an isolated assay of 2.79 g/t Au over a sample length of 1.0 m (444.5 to 445.5 m) hosted in a BMS unit with patches of moderate to strong sericite alteration in associated with elevated concentrations of copper (98 ppm Cu) and zinc (761 ppm Zn). It is possible that this hole just intersected the fringe of alteration located just south of the main alteration corridor that hosts the Goliath mineralisation. A strong magnetic iron formation containing both magnetite and pyrrhotite were intersected from 118.0 to 120.0 m and the core was determined to be very conductive using a multi-meter resistivity instrument which was most likely the source of the EM target.

A second drillhole TL14338 drilled further to the south was found to be meta-sedimentary rocks with a small iron formation unit intersected from 74.0 to 89.0 m containing patches of blebby pyrrhotite and pyrite. This hole did not return any significant gold assays.

10.2.5 2014-2015 Drilling

The Phase II drilling program on the Goliath property was completed between November 27, 2014 and March 17, 2015. A total of 31 NQ2 holes were drilled for a total of 8,769 m. Twenty-nine new holes were drilled numbered TL14372 to TL15402 and two re-entry holes (TL14373-15RE and TL14377-15RE) were extended to evaluate the gold potential of the C Zone (Table 10.1).

This drilling program was initiated for the purpose of resource category conversion and expanding known gold mineralisation by drill testing high-grade gold intercepts down plunge and along the perimeter of the gold-bearing shoots outside of the main shoots to complete the current mineral resource update. The program focused on further developing and expanding the resource potential of the C Zone and Main Zone mineralisation and Western shoots at depth in areas that had not been previously drill tested. A short two-hole exploration drilling program was also completed to test one of the best gold MMI anomalies defined by the 2014 soil sampling program.

The drill contract was awarded to Downing Drilling. In February 2015, a second drill was added accelerate the drilling program. This program focused predominantly along a 1.6 km strike length of the main resource deposit with holes drilled north at azimuths ranging from 325° to 002°. Collar dips ranged from -45° to -79°.

Significant Main Zone Intersections consisted of the following:

- TL14372 returned an interval of 3.86 g/t Au and 1.67 g/t Ag over 4.5 m (267.0 to 271.5 m) through the western Main Zone shoot.
- TL14374 intersected the western Main Zone shoot containing an interval with visible gold that assayed 199.75 g/t Au and 13.25 g/t Ag over 2.0 m (234.5 to 236.5 m). This hole was drilled around 41.0 m down plunge of the same zone tested by hole TL11204A that returned 17.83 g/t Au over a sample length of 6.0 m (223.5 to 229.5 m).
- TL14375 returned 4.87 g/t Au over 3.5 m in a Hanging Wall Zone from 133.0 to 136.5 m and then intersected 3.81 g/t Au and 8.38 g/t Ag over 8.0 m (185.0 to 193.0 m) through the Main Zone (western shoot).

- TL15396 intersected a well mineralised and quartz veined unit that returned 7.93 g/t Au and 43.57 g/t Ag over a sample length of 2.74 m (45.00 to 47.74 m) at a depth of just 36.0 m vertically from surface in the Main Central Zone. This result is within the proposed reserve pit and came from an area considered to contain low gold concentration.

In an area located 400 metres west of the main proposed pit, Treasury Metals completed seven infill holes to discover and potentially delineate additional shallow open pit-able resources. The program was following up on TL 14367, which intersected 12.8 m at 2.71 g/t (68.0 to 75.0 m) in the Main Zone at a vertical depth of 52 m identified by the 2014 Phase 1 program. Hole TL15400 returned 6.68 g/t Au and 1.97 g/t Ag over a sample length of 3.6 m (23.4 to 27.0 m) in a Hanging Wall (HW) Zone at a depth of 21.0 m from surface. Main Zone intersections included holes TL15395 that returned 1.43 g/t Au and 1.44 g/t Ag over 8.0 m (107.0 to 115.0 m), and hole TL15397 that assayed 2.44 g/t Au and 0.5 g/t Ag over 4.6 m (M1: 109.4 to 114.0 m) followed by 6.20 g/t Au and 0.5 g/t Ag over 2.0 m (M2: 120.0 to 122.0 m). The latter hole also returned the best C Zone (C2) intersection of 2.07 g/t Au and 0.5 g/t Ag over a sample length of 2.0 m (189.0 to 191.0 m).

The B Zone has been previously intersected by other historical holes throughout the deposit that have also returned significant gold assays. This program, including the 2015 infill core sampling program, further emphasized the importance of the B Zone located in the BMS rocks situated between the Main Zone and C Zone and their potential to add additional gold ounces to the Goliath deposit. In the 2015 drilling, hole TL15-390B intersected the B Zone in BMS rocks with no significant base metal mineralisation but containing coarse visible gold on the selvage edge of a well mineralised grey glassy quartz vein.

Two exploration holes numbered TL15401 and TL15402 were drilled just northeast of Tree Nursery Road on claim 1145301 to test the gold potential of a “high priority” mobile metal ion (MMI) Anomaly P in iron formation. This was a moderately strong gold (RR=60) and copper anomaly that occurred in association with weak silver and arsenic RR’s. Treasury Metals interpreted that F2 structures at the main resource deposit could be possibly extrapolated northeast to potentially intersect this target anomaly.

Both holes were drilled as a fence across the target anomaly and they intersected a series of iron formational units separated by strong to moderately garnetiferous metasedimentary rocks (MSED) that were locally weakly magnetic. Small sections of chert-magnetite banded iron formation (BIF) were also recorded. The iron formation was periodically intercalated with chloritised amphibolite rocks, which could represent mafic volcanoclastic rocks or inter-pyroclastic flows.

A bleached silicified and possibly weakly sericite altered zone was intersected by both drillholes at the point where the gold MMI high was centred. All cores were split for assay. None of the samples returned any significant gold or base metal assays.

10.2.6 2016 Drilling

A single phase diamond drill program on the Goliath property was completed from August 24, 2016 to January 15, 2017. A total of 28 NQ2 holes were drilled for a total of 12,154 m. Eighteen new holes were drilled numbered TL16403 to TL16420, including one wedged hole (TL16415W1) in order to recover 2 m of lost core in the main zone of mineralisation. In this program, ten drillholes were abandoned due to bad ground conditions causing the drill to deviate from the planned pierce points.

The objective of this drilling program was to:

- convert and increase indicated gold resources at Goliath through means of infill drilling
- locate and identify additional gold resources at depth with focus on the down plunge potential of the eastern, western, and central high-grade chutes of the Main Zone as well as the C Zone chute
- further delineation of the new high-grade zone discovered in the central portion of the C Zone
- to continue drill testing high-grade gold intercepts down plunge to depth's up to 723.0 m (TL16404D) to potentially add to underground resources

Drilling was contracted to George Downing Estate Drilling Ltd**.** The 2016 holes were drilled approximately perpendicular to the mineralised zone with azimuths ranging from 345° to 357° and dips ranging from -67° to -83°. Most of the drilling was concentrated along the peripherals of known high-grade chutes of the Main Zone and the C Zone to further delineate the chutes and convert mineral resources from inferred to indicated. The remainder of the drilling focused on testing the down dip potential of the high-grade chutes to add additional ounces of gold to the current resource. The average core recoveries were excellent and the RQD was good.

Drilling of the proposed underground mineral resource was successful in providing significant gold intersections of the central chute of the Main Zone and the C Zone. In addition to the Main Zone there was also significant gold intercepts occurring in the Hanging Wall and B Zones. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from the inferred to the indicated category and assist in further delineating the high-grade chutes of the Main Zone and C Zone.

Out of a total of 5,078 individual samples the highest gold assay obtained from the 2016 drilling program was from drillhole TL16405 that returned 63.1 g/t Au over a sample length of 1.0 m. Additional significant intervals from the 2016 drill program include:

- TL16403B intersected 5.44 g/t Au and 5.90 g/t Ag over 1.0 m (529.0 to 530.0 m) followed by 3.94 g/t Au and 4.28 g/t Ag over 4.0 m (541.0 to 545.0 m) as well as 14.3 g/t Au and 6.60 g/t Ag over a 1.0 m sample in the Main Zone that contained visible gold. This hole is located in the main zone central chute approximately 475 m from surface.
- TL16405 encountered several specks of visible gold in the B Zone returning 13.3 g/t Au and 6.68 g/t Ag over a sample length of 5.15 m (582.85 to 588.0 m) including 19.27 g/t Au and 9.51 g/t Ag over 3.45 m (582.85 to 586.3 m).
- TL16410 returned 10.95 g/t Au and 12.44 g/t Ag over a sample length of 7.0 m (544.0 to 551.0 m) including 24.47 g/t Au and 22.7 g/t Ag over 3.0 m (547.0 to 550.0 m). Visible gold was observed within this interval which was centrally located in the M2 portion of the Main Zone.
- TL16413 returned 6.54 g/t Au and 7.04 g/t Ag over a sample length of 11.50 m (657.0 to 668.5 m) including 11.32 g/t Au and 9.38 g/t Ag over 5.5 m (663.0 to 668.5 m) in the M2 footwall of the Main Zone. This hole was drilled to a depth of 717.0 m to test the down plunge potential of the eastern chute.

At the conclusion of the drilling program, and given the excellent gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the

C Zone remained a high priority exploration target that remained open to the west and down plunge at depth.

10.2.7 2017 Drilling

Treasury Metals conducted a diamond drill program from January 10, 2017 through to October 31, 2017. A total of 43 NQ2 drillholes totalling 8,516 m was completed, not including two holes that were abandoned due to poor sub-surface conditions. A total of 6,176 samples were taken over the span of the year, not including a total of 686 blanks and standards. The objectives of the drilling program were:

- to conduct condemnation and exploration drilling of areas where proposed mining infrastructure will be situated, including milling, tailings storage facility and mining operations
- to convert and increase indicated gold resources at Goliath property through infill and expansion drilling
- to locate and identify additional gold resources at depth with focus on the down plunge potential of the eastern, western, and central high-grade chutes of the Main Zone as well as the C Zone chute

This drilling program consisted of condemnation/exploration drilling along strike of the main resource as well as infill and expansion drilling of the Main and C Zone, further delineating the extents of the high-grade chutes. The program also included drill testing high-grade gold intercepts down plunge to depths up to 774.0 m (TL17412A) to potentially add to underground resources.

Drilling was contracted to George Downing Estate Drilling Ltd. The holes were drilled approximately perpendicular to the mineralised zone with azimuths ranging from 319.2° to 355° and dips ranging from -48.6° to -82.9°. The condemnation/exploration drilling was concentrated along strike, northeast of the known resource and outside of the current proposed open-pit. The purpose of the condemnation/exploration program was to drill test areas along strike of the main resource where proposed mining infrastructure is to be located, including milling, tailings storage facility and mining operations. In addition, to test locations of potential gold chutes interpreted by Exploration Manager Paul Dunbar from historical drillhole compilation and newly prepared longitudinal sections of the EAC. The condemnation/exploration drilling was comprised of a series of shallow drillholes ranging in depth from 57.0 m to 204.0 m.

Treasury Metals spent the remainder of the drilling in 2017 focused on infill and resource conversion around the perimeter of known high-grade chutes of the Main Zone and the C Zone. The purpose of this program was to further delineate the chutes and convert resources from the inferred to indicated classification, while also testing the down dip extents of the mineralised chutes. The average core recoveries were excellent and the RQD was good.

Infill drilling of the proposed underground resource was successful in providing significant gold intersections of the Main Zone and C Zone. In addition to the Main Zone and C Zone there was also significant gold intercepts occurring in the Hanging Wall, D Zone and E Zone. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from the inferred to the indicated classification and assist in further delineating the high-grade chutes of the Main Zone and C Zone.

Two infill/resource conversion drillholes numbered TL17422 and TL17460 were drilled on mining patent 47122 and mining lease 109717. The purpose of these infill holes was to test the gold potential of the eastern most edge of the western high-grade chute of the Main Zone and to test the down dip potential of the central chute of the Main Zone. TL17422 intersected good gold assays within the Main, C, and B Zones, therefore successfully expanding the Main Zone's western chute to the east and warranting further drilling to test the continuity. TL17460 also intersected good gold grades within the Main Zone and was able to expand the continuity of the high-grade central chute further down dip. It was determined from TL17460 that the central chute of the Main Zone remains open down dip and remains to be a high priority target for future drill programs. Infill holes TL17445 and TL17459 were drilled just southwest of where the main hydro line intersects with Tree Nursery Road on mining patent 46017. These are both near surface holes with the goal of identifying and expanding the C Zone resource along strike to the east of the known deposit. TL17445 returned a number of anomalous gold assays within the C zone and the highest gold sample of the program within the D Zone. TL17459 intersected a single high-grade gold assay in the C Zone. Both of these holes display that there is some continuity down dip between the high-grade lenses within the eastern side of the C Zone, but more investigation is required to determine their trend and extent.

The condemnation and exploration drilling took place along strike of the main resource area, stepping out to the North East over a distance of approximately 1.4 km from the current known resource. Low-grade gold intersections were encountered to the northeast of the proposed tailings pond in what was previously a sparsely drilled portion of the property. Hole TL17442 and TL17443 intersected discontinuous low-grade mineralisation confirming the grade observed in the 2011 drill program. The near surface mineralisation appears to be in two poorly define zones extending 200 m below surface with a short strike length on 50 to 60 metres.

Out of a total of 6,176 individual samples the highest gold assay obtained from the 2017 drilling program was from drillhole TL17445 that returned a single assay of 33.3 g/t Au over a sample length of 1.0 m corresponding to the D Zone. Additional significant intervals from the 2017 program include:

- TL17422 intercepted 3.67 g/t Au and 3.58 g/t Ag over a sample length of 4.0 m (348.0-352.0 m) in the Main Zone. This hole also intersected 7.13 g/t Au and 6.20 g/t Ag over a sample length of 0.9 m (392.0-392.9 m) in the B Zone. In the C Zone this hole intersected 4.10 g/t Au and 26.46 g/t Ag over a sample length of 5.0 m (457.0-462.0 m), including 18.2 g/t Au and 119.0 g/t Ag over a sample length of 1.0 m (459.0-460.0 m) which contained several specks of visible gold and electrum.
- TL17445 was targeting the C and D Zones and returned several high-grade assay values. In the C Zone this hole found 9.92 g/t Au and 3.60 g/t Ag over a sample length of 1.0 m (47.0-48.0 m) with several specks of visible gold within a wider zone grading 2.67 g/t Au and 4.49 g/t Ag (43.17-48 m). In the D Zone this hole intersected 16.79 g/t Au and 1.90 g/t Ag over a sample length of 2.0 m (68.0-70.0 m), including 33.30 g/t Au and 2.10 g/t Ag over a 1.0 m sample length (69.0-70.0 m). No visible gold was noted in this interval.
- TL17459 intercepted 13.8 g/t Au and 19.90 g/t Ag over a sample length of 1.0 m (122.0-123.0 m) in the C Zone within a wider zone grading 3.88 g/t Au and 6.58 g/t Ag over 4.0 m (122-126 m).

- TL17460 intersected 4.53 g/t Au and 29.90 g/t Ag over 1.0 m (576.0-577.0 m) in the Hanging Wall. Followed by 3.34 g/t Au and 5.94 g/t Ag over 5.0 m (641.0-646.0 m), including 4.80 g/t Au and 8.83 g/t Ag over 3.0 m (643.0-646.0 m) and 3.41 g/t Au and 56.50 g/t Ag over 2.0 m (663.0-665.0 m), including 6.47 g/t Au and 80.10 g/t Ag over 1.0 m (664.0-665.0 m).

Upon completion of the drilling program and given the excellent gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the C Zone remained a high priority exploration target that remained open to the east and west as well as down plunge at depth.

10.2.8 2018 Drilling

Treasury Metals conducted a diamond drill program on the Goliath property from January 8, 2018 through to June 22, 2018, totalling 20,987 m. This consisted of 38 new holes drilled (TL18464 to TL18501), not including 14 holes that were abandoned due to bad ground conditions causing deviation from the intended target. A total of 10,251 samples and 1,139 blanks and standards were tested over the span of the year. The objective of the drilling program was to:

- convert and increase indicated gold resources in the Main and C Zones of the Goliath property, through means of infill and expansion drilling
- investigate the extent of high-grade mineralisation found in historic Teck drillholes in the East C Zone

This drilling program consisted of infill and resource conversion drilling within the Main and C Zones and further delineation of the high-grade chutes of each. The program included drill testing of high-grade gold intercepts down plunge of the Main Zone to depths up to 762.0 m (TL18471A) to potentially add to underground resources.

Drilling was contracted to George Downing Estate Drilling and Distinctive Drilling. The 2018 holes were drilled approximately perpendicular to the mineralised zone with azimuths ranging from 350° to 0° and dips ranging from -58° to -78°. This program consisted of drilling at depths ranging from 195.0 m (TL18480) to 831.0 m (TL18473A) and targeted areas along the outer edges and down plunge of the high-grade chutes in the central, western, and eastern chutes of the Main Zone as well as the C Zone. Additionally, 5,000 m of drilling was conducted on the East C zone area where historic Teck drillholes intercepted moderate to high-grade mineralisation. The average core recoveries were excellent and the RQD rock mass quality was good.

Drilling of the underground resource was successful in providing significant gold intersections in both the Main and C Zone. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from inferred to the indicated classification and assist in further delineating the high-grade chutes of the Main Zone and C Zones.

Out of a total of 10,251 individual samples the highest gold assay obtained from the 2018 drilling program was from drillhole TL18494 that returned 111 g/t Au over a sample length of 1.0 m within a large zone of lower grade mineralisation grading 6.28 g/t Au and 1.71 g/t Ag over 19.0 m (425-444 m). This was drilled as a follow up to Teck drillhole TL205 which intersected 1.0 g/t Au over 23.5 m and is located near the eastern most extent of drilling in the C Zone. Additional significant intervals from the 2018 program include:

- TL18469 intersected 14.88 g/t Au and 5.33 g/t Ag over 6.0 m (558.0-564.0 m), including 79.6 g/t Au and 3.8 g/t Ag over 1.0 m (559.0-560.0 m) in the Main Zone. This hole is situated along the eastern edge of the east chute. Three small specks of visible gold (< 1 mm grain size) was observed between 559.75 m to 559.58 m.
- TL18474 intersected 10.35 g/t Au and 5.89 g/t Ag over a sample length of 7.0 m (445.0-452.0 m), including 64.5 g/t Au and 1.8 g/t Ag over 1.0 m (451.0-452.0 m). This hole was drilled along the eastern edge of the west chute in the Main Zone.
- TL18489 intersected 48.71 g/t Au and 310.67 g/t Ag over a sample length of 3.0 m (542.0-545.0 m), including 145.00 g/t Au and 921.00 g/t Ag over 1.0 m (543.0-544.0 m) in the C Zone in addition to 5.28 g/t Au and 143.00 g/t Ag over 1.0 m (528.4-529.4 m). This hole was drilled at the deepest extent of the C Zone and was successful in confirming the continuity of gold mineralisation down plunge. Minor visible gold was observed between 528.4-529.4 m depth and approximately 20 specks of visible gold ranging in size from 1-5 mm was observed between 543.2-543.3 m depth.
- TL18494 intersected 25.20 g/t Au and 3.98 g/t Ag over a 4.50 m sample length (426.0-430.5 m), including 1.0 m (426.0-427.0 m) at 111.00 g/t Au and 11.10 g/t Ag. This drillhole was drilled to investigate nearby Teck drillhole TL205 which intersected 1.0 g/t Au over 23.5 m and is located near the eastern most extent of drilling in the C Zone.
- TL18499A intersected 3.81 g/t Au and 34.65 g/t Ag over 13.0 m (516.0-529.0 m), including 10.17 g/t Au and 120.47 g/t Ag over 3.0 m (516.0-519.0 m) in the Main Zone. This hole was drilled as a follow up to TL18469 and is located on the eastern edge of the east chute within the Main Zone. Visible gold was observed in four small specks (< 1 mm grain size) between 518.4-518.5 m depth.

Upon completion of the drilling program and given the gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the C Zone remained a high priority exploration target that remained open to the east as well as down plunge.

10.2.9 2019-2020 Drilling

Treasury Metals conducted a diamond drill program on the Goliath property from November 15, 2019 through to March 7, 2020, totalling 10,135 m. This program consisted of 27 new holes drilled (TL19502 to TL20528), not including six holes that were abandoned due to bad ground conditions causing deviation from the intended target. A total of 6,468 core samples and 680 blanks and standards were tested over the span of the program. The three objectives of the drilling program were to:

- convert and increase measured gold resources in the Main Zone east and central shoots of the Goliath deposit for inclusion as potential estimate ounces for the initial mine life years and for grade control purposes through infill drilling
- convert and increase indicated and inferred gold resources in the C East area through infill and expansion drilling
- investigate and expand the Main Zone east shoot at depth on the eastern side through exploration and expansion drilling

Drilling was contracted to George Downing Estate Drilling Ltd. The 2019 and 2020 holes were drilled approximately perpendicular to the mineralised zone with azimuths ranging from 350° to 0° and dips ranging from -60° to -78°. This program consisted of drilling intersections at depths from surface ranging from 120 m (TL20528) to 625 m (TL20517). Out of the total program, 3,816 m were for Main Zone measured infill, 4,176 m targeted the C East area, and 2,143 m at depth adjacent to the Main Zone eastern shoot. The average core recoveries were excellent and the RQD rock mass quality was good.

Out of a total of 6,468 individual samples the highest gold assay obtained from the 2019-2020 drilling program was from drillhole TL20520 that returned 152.0 g/t Au over a sample length of 1.0 m within an intersection grading 51.5 g/t Au over 3.0 m (523.5-526.5 m). This was drilled in the C East area 100 m down dip from hole TL18494 which returned 25.2 g/t Au over 4.5 m including 111.0 g/t Au over 1.0 m. Additional significant intervals from the program include:

- TL19503, also in the C East area, intersected 17.1 g/t Au over 7.0 m including 117.0 g/t Au over 1.0 m
- TL19505, located in the Main Zone central shoot, intersected 9.2 g/t Au over 6.3 m including 13.0 g/t Au over 4.0 m
- TL20517, drilled at depth adjacent to the Main Zone eastern shoot, intersected 4.6 g/t Au over 4.4 m including 13.2 g/t Au over 1.0 m in the Main Zone and 2.4 g/t Au over 6.0 m including 10.6 g/t Au over 1.0 m in a Hanging wall Zone

Highlights of the program are summarised in Table 10.4. In the table, duplicate samples were averaged together to calculate intersection grade; all grades are reported uncut and interval lengths were reported at core length. AGP notes that true width at the Goliath deposit typically range between 74% to 90% of the sample length, but can occasionally reach as low of 44% and a high of 96%.

10.2.10 Qualified Person Opinion on the Goliath Drill Programs

The drillhole orientation was found to be appropriate for the deposit style and the orientation of the mineralisation. Drill spacing in the most densely drilled areas is less than 25 m and is deemed sufficient to adequately define the grade of the mineralisation and the spatial grade distribution.

Drill core logging is appropriate for the mineralisation style and carried out to industry standards.

AGP would like to point out that high-grade intersections above 10 g/t Au are rare and are sometime isolated (> 99th percentile in the MSS lithology). Intercepts above 3 g/t Au occur more frequently (> 98th percentile in the MSS lithology) and have documented continuity of at least 20 m strike length in the Teck bulk sample area.

Drill core handling, surveying, and chain of custody from the rig to the core logging facility was found to meet or exceed industry standards.

Table 10.4: Highlights of the 2019 & 2020 Program

Drillhole	Target	Zone	From (m)	To (m)	Sample Length (m)	Au g/t
TL19503	C	Main	356.0	361.0	5.0	2.0*
		C	449.0	456.0	7.0	17.08*
		including	449.0	450.0	1.0	117.00*
TL19505	Main	Main	214.7	221.0	6.3	9.23*
		including	217.0	221.0	4.0	13.02
TL20515	C	Main	348.0	352.0	4.0	5.38*
		including	348.0	349.0	1.0	20.90*
		C	446.0	462.4	16.4	0.52
		C	477.0	483.1	6.1	0.52
TL20517	Main	HW	454.0	460.0	6.0	2.42*
		including	454.0	455.0	1.0	10.60*
		Main	658.6	663.0	4.4	4.64*
		including	658.6	659.6	1.0	13.20*
TL20518	C	HW	129.0	140.0	11.0	0.45
		C	403.1	417.1	14.0	0.67*
		including	413.1	417.1	4.0	1.21*
		C	432.7	438.2	5.5	0.70
TL20519	C	HW	55.7	58.3	2.6	1.32
		Main	308.8	310.5	1.7	0.43
		C	419.4	428.0	8.6	1.30*
		including	427.0	428.0	1.0	7.10*
		C	448.7	452.3	3.6	0.68
TL20520	C	C	495.0	509.7	14.7	1.19*
		including	507.0	508.0	1.0	8.27*
		C	523.5	526.5	3.0	51.60*
		including	524.5	525.5	1.0	152.00*
TL20521	Main	Main	205.0	231.0	26.0	1.00*
		including	211.0	215.0	4.0	1.44*
		and including	222.0	223.0	1.0	9.89*
TL20522	Main	Main	265.0	280.0	15.0	1.63*
		including	267.0	271.0	4.0	3.95*
		including	269.0	270.0	1.0	9.72*
	Main	Main	285.1	290.0	4.9	2.70*
TL20523	Main	HW	140.0	142.5	2.5	1.47
		Main	221.0	240.5	19.5	4.04*
		including	222.0	234.0	12.0	6.02*
		including	222.0	224.0	2.0	27.30*
TL20525	Main	Main	157.5	166.5	9.0	6.04*
		including	162.5	166.5	4.0	12.92*
		Main	174.0	176.0	2.0	1.69
TL20527	Main	Main	197.0	207.0	10.0	3.59*
		including	204.0	205.0	1.0	18.00*
		Main	219.0	226.0	7.0	7.03
		including	224.0	225.0	1.0	40.6
TL20528	Main	Main	118.4	122.0	3.6	0.84

Note: * Includes metallic screen assays. Source: Treasury Metals (2020).

10.3 Goldlund Deposit

Diamond drilling on the Goldlund Project has been carried out since the 1940s. There is a total of 856 drillholes totalling 152,787.7 m of surface drilling and 480 drillholes totalling 18,626 m of underground drilling in the July 20, 2020 drillhole database as compiled by First Mining Gold Corp. At the time of the site visit by CGK there was no drilling in progress, so the following summary is based on the descriptions of the procedures that were used at the time the work was carried out taken from previous Technical Reports including the 2020 Treasury Metals report prepared by WSP.

The most recent drilling was carried out by First Mining Gold Corp. in 2019 and 2020, with a total of 14 drillholes totalling 2,506 m of drilling in 2019, and 34 holes totalling 6,452 m of drilling in 2020. The drilling was focused within and around the defined resource area at Goldlund (Main Zone), with an initial target of defining and extending mineralisation in the eastern and western portions of the deposit. The procedures and results for the 2019 and 2020 drill program are summarised below.

Drilling procedures, sampling methodology, and results prior to 2019 have been presented in detail in previous Technical Reports including the 2020 Technical Report prepared by WSP for Treasury Metals and will only be summarised here.

10.3.1 First Mining, 2019-2020

10.3.1.1 Drilling

The drilling was conducted by Rodren Drilling of Manitoba with HQ sized core. Casings were left in place and capped. The drillhole collar locations were initially surveyed using a handheld Garmin GPS, then after drilling was completed, collars were surveyed by differential GPS. After Treasury Metals acquired the Goldlund Project they independently surveyed the collar locations using a Trimble GeoExplorer 6000 Series, Model: 88950 GPS and found that all but one drillhole (GL-19-034) had similar locations to the planned drillhole collars. The location of drillhole GL-19-034 has been corrected in the drillhole database used for the mineral resources estimate.

Down-the-hole surveying was done using an EZ Gyro survey tool to determine the deviation of inclined drill paths. The path of the drillhole was surveyed upon completion of the hole, with readings taken approximately every 30 m. There were optimised readings (consisting of three consecutive readings taken at the same interval and averaged together), taken at the top and bottom of the drillhole.

The core logging methodology and QA/QC procedures were overseen by Mr. Miro Mytny, P.Geo., Senior Exploration Manager for First Mining. The logging procedures applied during the 2019 and 2020 drilling programs at Goldlund are summarised below. Figure 10-6 displays photographs of the core logging area on the left-hand side and the core sampling area with a diamond saw on the right-hand side of the figure.

Figure 10-6: Core Logging & Sampling Facilities at the Goldlund Exploration Camp

Source: CGK (2020).

10.3.1.2 Logging

The HQ diameter (63.5 mm) drill core was cleaned, and the run blocks checked. After this, the runs were measured for recovery. The recovery percentage was then used to mark off the adjusted metres within the run. The core was logged for lithology, alteration, minerology, veining, and structure, and entered into the DH Logger® software, which synchronises with First Mining's central Fusion® SQL drilling database. The RQD was measured and recorded in an Excel sheet, for importing into the Datamine DH Logger® software. The core was photographed twice, both dry and wet.

10.3.1.3 Drill Core Sampling

One-metre sample intervals were marked off, except at lithological contacts and in zones of poor recovery, where sample size was adjusted accordingly. The core was sawn in half on site, with one half bagged and labelled to be sent for assay. The remaining half core was placed in core boxes which were stored in a secure on-site facility to serve as a permanent record.

For field duplicates, the core was quartered and one quarter was sent for regular assay, while the other quarter was sent as a duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the pulverised sample to run a duplicate assay.

Standards and blanks were inserted in the sample stream at the required intervals. Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates.

The sample bags were placed in zip-tied rice bags and shipped to SGS Laboratory facilities either in Red Lake, Ontario or Burnaby, British Columbia for fire assay. Intact pieces of drill core were selected and measured for specific gravity using the buoyancy methodology.

The SGS laboratories returned all coarse rejects and pulps to First Mining for permanent and secure storage on site at the Goldlund Project. The remaining drill core is securely stored in open core racks or in core racks inside temporary structures.

10.3.1.4 Sample Recovery

The drill core recovery is good, with an average core recovery of approximately 100%, and only 0.6% of the core intervals had less than 90% core recovery.

10.3.1.5 Database

The drillhole data is stored in a SQL database that is managed using appropriate access controls. The database stores all appropriate data including geological and assay data associated with the drill core.

10.3.1.6 Results

The 2019-2020 drill program at the Main Zone consists of 48 drillholes for a total of 8,958 m, focused primarily on Zones 2 and 3. Drilling was completed at an approximate 50 m spacing. The goal of this drill program is to define and extend mineralisation in the eastern and western portions of the Main Zone area.

Hole GL-19-008 intersected 21 m of 5.36 g/t Au within highly mineralised granodiorite and porphyry units, as well as within andesite, and was successful in confirming the high grades within Zone 2 that were encountered in historical drilling.

Hole GL-19-010 was drilled to intersect the area between the known mineralised areas at Zones 2 and 3 and encountered significant gold mineralisation hosted within andesite (15.0 m at 1.68 g/t Au), before intersecting the mineralised granodiorite and porphyries of Zone 2 towards the end of the hole.

Hole GL-20-004 intersected both Zones 2 and 3 in their southwest extension. Hole GL-20-016 targeted the southwest extension of Zone 4. Most of the mineralisation intercepted by these holes is associated with altered porphyry units. Minor mineralisation is also associated with gabbro and basalt/andesite, and mainly occurs in close proximity to contacts with a porphyry intrusion. The highest-grade gold mineralisation was intersected in holes GL-20-010 (44 m at 1.20 g/t Au) and GL-20-006 (13 m at 2.10 g/t Au), which were drilled vertically through Zone 3.

The gold mineralisation encountered in holes GL-20-030, 031, 032, 033, and 034 occurs within locally silicified and sheared variolitic andesite, as well as gabbro, and altered porphyry intrusions. These results support the concept that the gold mineralisation can occur in settings other than that associated with the granodiorite, which is the principal host for the gold mineralisation in Zones 1 and 7.

Table 10.5 presents a list of the drillhole collar locations for the holes drilled in the 2019 and 2020 drill program. Table 10.6 presents a list of the significant drillhole intercepts encountered in the Main Zone for Zone 2 and Zone 3. Figures 10-7 and 10-8 display a plan view and cross-section of the 2019 and 2020 northeast area drilling. Figures 10-9 and 10-10 display a plan view and cross-section of the 2019 and 2020 drilling for the southwest area drilling.

Table 10.5: Goldlund Drillhole Collar Information

Hole ID	Collar UTM East	Collar UTM North	Hole Azimuth (⁰)	Hole Dip (⁰)	Final Depth (m)	Target
GL-19-001	546,604	5,527,909	0	-45	161	New Target
GL-19-002	546,604	5,527,909	34	-45	170	New Target
GL-19-003	547,642	5,528,090	335	-74	197	Main Zone (Zone 2)
GL-19-004	547,642	5,528,090	335	-84	257	Main Zone (Zone 2)
GL-19-005	547,702	5,528,089	335	-77	242	Main Zone (Zone 2)
GL-19-006	547,702	5,528,089	335	-86	167	Main Zone (Zone 2)
GL-19-008	547,722	5,528,154	335	-85	125	Main Zone (Zone 2)
GL-19-010	547,746	5,528,102	335	-77	176	Main Zone (Zone 2,3)
GL-19-012	547,774	5,528,162	335	-73	182	Main Zone (Zone 2)
GL-19-013	547,774	5,528,162	335	-62	101	Main Zone (Zone 2)
GL-19-014	547,774	5,528,162	335	-45	95	Main Zone (Zone 2)
GL-19-021	546,100	5,527,506	0	-90	386	Main Zone (Zone 3)
GL-19-022	546,100	5,527,506	0	-50	173	Main Zone (Zone 2, 3)
GL-19-034*	547,644	5,528,082	335	-45	74	Main Zone (Zone 2)
GL-20-001	546,099	5,527,538	0	-90	314	Main Zone (Zone 3)
GL-20-002	546,150	5,527,548	0	-90	293	Main Zone (Zone 3)
GL-20-003	546,150	5,527,576	0	-90	206	Main Zone (Zone 3)
GL-20-004	546,150	5,527,576	0	-55	194	Main Zone (Zone 2)
GL-20-005	546,200	5,527,550	0	-90	188	Main Zone (Zone 3)
GL-20-006	546,200	5,527,525	0	-90	221	Main Zone (Zone 3)
GL-20-007	546,250	5,527,530	0	-90	83	Main Zone (Zone 3)
GL-20-008	546,250	5,527,530	0	-90	203	Main Zone (Zone 3)
GL-20-009	546,250	5,527,555	0	-90	125	Main Zone (Zone 3)
GL-20-010	546,295	5,527,527	0	-90	218	Main Zone (Zone 3)
GL-20-011	546,295	5,527,527	0	-75	164	Main Zone (Zone 3)
GL-20-012	546,350	5,527,559	0	-90	263	Main Zone (Zone 3)
GL-20-013	546,350	5,527,559	0	-55	122	Main Zone (Zone 3)
GL-20-014	546,400	5,527,566	0	-90	188	Main Zone (Zone 3)
GL-20-015	546,388	5,527,591	0	-90	191	Main Zone (Zone 3)
GL-20-016	546,338	5,527,332	0	-70	206	Main Zone (Zone 4)

Hole ID	Collar UTM East	Collar UTM North	Hole Azimuth (°)	Hole Dip (°)	Final Depth (m)	Target
GL-20-017	547,702	5,528,089	155	-65	179	Main Zone (Zone 3)
GL-20-018	547,648	5,528,051	155	-70	200	Main Zone (Zone 3)
GL-20-019	547,648	5,528,051	335	-60	182	Main Zone (Zone 3)
GL-20-020	547,752	5,528,112	335	-50	140	Main Zone (Zone 2)
GL-20-021	547,750	5,528,119	155	-60	161	Main Zone (Zone 3)
GL-20-022	547,774	5,528,159	155	-60	164	Main Zone (Zone 3)
GL-20-023	547,847	5,528,149	335	-70	200	Main Zone (Zone 2,3)
GL-20-024	547,847	5,528,149	155	-70	200	Main Zone (Zone 3)
GL-20-025	547,873	5,528,180	335	-68	182	Main Zone (Zone 2,3)
GL-20-026	547,865	5,528,182	145	-60	179	Main Zone (Zone 3)
GL-20-027	547,932	5,528,278	335	-70	143	Main Zone (Zone 2)
GL-20-028	547,932	5,528,278	335	-50	104	Main Zone (Zone 2)
GL-20-029	547,992	5,528,465	155	-45	203	Main Zone (Zone 2)
GL-20-030	547,992	5,528,465	155	-57	218	Main Zone (Zone 2)
GL-20-031	548,072	5,528,499	155	-50	218	Main Zone (Zone 2)
GL-20-032	548,138	5,528,554	155	-45	230	Main Zone (Zone 2)
GL-20-033	548,028	5,528,491	155	-45	200	Main Zone (Zone 2)
GL-20-034	548,028	5,528,491	155	-59	170	Main Zone (Zone 2)

Source: First Mining Press Releases (2020), * corrected coordinates

Table 10.6: Summary of Significant Drill Intercepts for Zones 2, 3

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
GL-19-003	23.57	25.00	1.43	14.85	10.91	10.91	Main Zone (Zone 2)
and	44.00	47.00	3.00	0.62	n/a	0.62	
including	45.00	46.00	1.00	1.51	n/a	1.51	
and	72.10	74.10	2.00	0.25	n/a	0.25	
and	102.40	107.46	5.06	0.95	n/a	0.95	
including	106.62	107.46	0.84	4.57	n/a	4.57	
GL-19-004	32.86	36.21	3.35	1.28	n/a	1.28	Main Zone (Zone 2)
and	51.90	55.05	3.15	0.60	n/a	0.60	
and	149.91	155.00	5.09	1.72	n/a	1.72	
including	149.91	151.00	1.09	4.73	n/a	4.73	
and	166.00	172.00	6.00	1.57	n/a	1.57	
including	166.00	167.00	1.00	3.03	n/a	3.03	
and incl.	170.00	172.00	2.00	2.47	n/a	2.47	
GL-19-005	58.90	64.00	5.10	0.33	n/a	0.33	Main Zone (Zone 2)
and	83.00	84.00	1.00	0.30	n/a	0.30	
and	133.00	135.00	2.00	1.98	n/a	1.98	
including	133.00	134.00	1.00	3.58	n/a	3.58	
and	169.30	174.00	4.70	1.05	n/a	1.05	
including	172.21	174.00	1.79	2.40	n/a	2.40	
and	189.00	195.00	6.00	0.52	n/a	0.52	
GL-19-006	82.00	86.00	4.00	3.08	n/a	3.08	Main Zone (Zone 2)
including	83.00	85.00	2.00	5.72	n/a	5.72	
and incl.	83.00	83.67	0.67	9.53	n/a	9.53	
and	107.00	114.00	7.00	0.96	n/a	0.96	
including	107.00	108.00	1.00	3.14	n/a	3.14	
and incl.	112.00	114.00	2.00	1.63	n/a	1.63	
and	134.00	134.50	0.50	1.80	n/a	1.80	
and	137.54	137.85	0.31	5.13	n/a	5.13	
and	147.76	148.09	0.33	48.03	n/a	48.03	
GL-19-008	1.40	25.00	23.60	0.33	n/a	0.33	Main Zone (Zone 2)

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
including	10.00	16.00	6.00	1.06	n/a	1.06	
and incl.	13.00	15.00	2.00	1.90	n/a	1.90	
and	57.00	66.00	9.00	0.82	n/a	0.82	
and	83.00	104.00	21.00	6.49	5.36	5.36	
including	88.00	89.00	1.00	5.49	n/a	5.49	
and incl.	96.00	97.00	1.00	113.43	89.60	89.60	
GL-19-010	69.00	84.00	15.00	1.68	n/a	1.68	Main Zone (Zone 2, 3)
including	69.00	70.00	1.00	8.02	n/a	8.02	
and incl.	71.00	72.00	1.00	4.86	n/a	4.86	
and incl.	80.00	81.00	1.00	4.89	n/a	4.89	
and	143.00	148.00	5.00	1.26	n/a	1.26	
including	147.00	148.00	1.00	5.24	n/a	5.24	
and	167.00	175.00	8.00	0.97	n/a	0.97	
GL-19-012	9.40	9.71	0.31	0.69	n/a	0.69	Main Zone (Zone 2)
and	48.00	65.00	17.00	1.11	n/a	1.11	
including	48.00	53.00	5.00	2.27	n/a	2.27	
and incl.	48.00	49.00	1.00	4.14	n/a	4.14	
and	86.00	87.00	1.00	3.59	n/a	3.59	
and	96.00	97.00	1.00	0.98	n/a	0.98	
and	103.00	104.00	1.00	0.74	n/a	0.74	
GL-19-013	32.00	34.00	2.00	0.66	n/a	0.66	Main Zone (Zone 2)
and	63.00	77.00	14.00	1.15	n/a	1.15	
including	70.00	77.00	7.00	2.20	n/a	2.20	
and incl.	70.00	71.00	1.00	5.32	n/a	5.32	
and incl.	75.00	76.00	1.00	9.42	n/a	9.42	
GL-19-014	25.00	27.00	2.00	0.75	n/a	0.75	Main Zone (Zone 2)
and	36.00	37.00	1.00	4.07	n/a	4.07	
and	56.00	58.00	2.00	0.71	n/a	0.71	
GL-19-021	139.00	140.00	1.00	9.19	n/a	9.19	Main Zone (Zone 3)
and	188.00	191.00	3.00	3.20	n/a	3.20	
including	188.00	189.00	1.00	6.54	n/a	6.54	

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
and	286.00	288.61	2.61	1.97	n/a	1.97	
including	286.00	286.70	0.70	6.64	n/a	6.64	
GL-19-034	25.94	27.17	1.23	8.63	n/a	8.63	Main Zone (Zone 2)
and	30.72	31.20	0.48	1.81	n/a	1.81	
and	53.00	55.00	2.00	1.46	n/a	1.46	
and	60.00	62.00	2.00	3.40	n/a	3.40	
GL-20-005	52.13	57.07	4.94	0.38	n/a	0.38	Main Zone (Zone 3)
and	60.00	94.57	34.57	0.28	n/a	0.28	
GL-20-006	153.00	211.00	58.00	0.88	0.83	0.83	Main Zone (Zone 3)
including	153.00	166.00	13.00	2.10	n/a	2.10	
and incl.	161.00	162.00	1.00	12.07	n/a	12.07	
and incl.	165.00	166.00	1.00	5.10	n/a	5.10	
and incl.	202.00	211.00	9.00	1.94	1.67	1.67	
and incl.	208.00	209.00	1.00	11.37	9.00	9.00	
GL-20-008	94.00	95.00	1.00	2.74	n/a	2.74	Main Zone (Zone 3)
and	123.00	167.00	44.00	0.27	n/a	0.27	
including	147.00	166.00	19.00	0.47	n/a	0.47	
and incl.	147.00	148.00	1.00	1.64	n/a	1.64	
and incl.	165.00	166.00	1.00	2.19	n/a	2.19	
and	175.00	176.00	1.00	1.33	n/a	1.33	
GL-20-009	37.00	100.00	63.00	0.33	0.27	0.27	Main Zone (Zone 3)
including	80.00	100.00	20.00	0.70	0.52	0.52	
and incl.	99.00	100.00	1.00	11.36	7.90	7.90	
GL-20-010	119.00	122.00	3.00	3.06	n/a	3.06	Main Zone (Zone 3)
including	120.00	121.00	1.00	7.86	n/a	7.86	
and	148.00	192.00	44.00	1.26	1.20	1.20	
including	152.00	153.00	1.00	6.70	n/a	6.70	
and incl.	166.00	183.00	17.00	2.08	1.94	1.94	
and incl.	182.00	183.00	1.00	18.28	15.90	15.90	
and	199.00	210.00	11.00	0.26	n/a	0.26	
including	209.00	210.00	1.00	1.72	n/a	1.72	

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
GL-20-011	88.00	130.00	42.00	0.26	n/a	0.26	Main Zone (Zone 3)
including	88.00	107.00	19.00	0.54	n/a	0.54	
and incl.	93.00	99.00	6.00	1.03	n/a	1.03	
GL-20-012	12.00	102.00	90.00	0.31	n/a	0.31	Main Zone (Zone 3)
including	19.00	23.00	4.00	1.10	n/a	1.10	
and	175.00	225.00	50.00	0.14	n/a	0.14	
GL-20-013	17.00	61.00	44.00	0.27	n/a	0.27	Main Zone (Zone 3)
including	20.00	21.00	1.00	1.21	n/a	1.21	
and incl.	54.00	58.00	4.00	0.67	n/a	0.67	
GL-20-014	1.15	29.00	27.85	0.42	n/a	0.42	Main Zone (Zone 3)
including	2.00	3.00	1.00	2.92	n/a	2.92	
and incl.	16.00	17.00	1.00	1.75	n/a	1.75	
and incl.	27.00	28.00	1.00	1.52	n/a	1.52	
and	41.00	123.00	82.00	0.10	n/a	0.10	
and	131.00	140.00	9.00	0.25	n/a	0.25	
and	158.00	166.00	8.00	0.32	n/a	0.32	
GL-20-015	10.00	171.00	161.00	0.12	n/a	0.12	Main Zone (Zone 3)
including	87.00	88.00	1.00	1.29	n/a	1.29	
and incl.	102.00	115.00	13.00	0.20	n/a	0.20	
and incl.	140.00	171.00	31.00	0.22	n/a	0.22	
and incl.	164.00	171.00	7.00	0.52	n/a	0.52	
and incl.	164.00	165.00	1.00	2.57	n/a	2.57	
GL-20-017	87.00	93.00	6.00	1.67	n/a	1.67	Main Zone (Zone 3)
including	88.00	89.00	1.00	8.49	n/a	8.49	
and	130.00	141.00	11.00	0.15	n/a	0.15	
including	130.00	134.00	4.00	0.33	n/a	0.33	
GL-20-018	45.00	76.00	31.00	0.14	n/a	0.14	Main Zone (Zone 3)
including	71.00	72.00	1.00	2.14	n/a	2.14	
and	87.00	88.00	1.00	1.11	n/a	1.11	
and	126.00	136.00	10.00	5.42	n/a	5.42	
including	129.00	131.00	2.00	22.03	n/a	22.03	

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
and incl.	135.00	136.00	1.00	5.10	n/a	5.10	
GL-20-019	102.00	104.00	2.00	0.95	n/a	0.95	Main Zone (Zone 2)
and	128.43	130.17	1.74	0.78	n/a	0.78	
and	145.27	146.33	1.06	0.19	n/a	0.19	
GL-20-020	34.89	35.76	0.87	0.60	n/a	0.60	Main Zone (Zone 2)
and	87.00	109.00	22.00	1.25	n/a	1.25	
including	103.00	109.00	6.00	2.71	n/a	2.71	
and incl.	103.00	104.00	1.00	5.46	n/a	5.46	
and incl.	107.00	108.00	1.00	6.37	n/a	6.37	
GL-20-021	82.50	83.50	1.00	0.39	n/a	0.39	Main Zone (Zone 3)
and	116.00	117.00	1.00	0.88	n/a	0.88	
and	121.00	122.00	1.00	0.43	n/a	0.43	
and	141.00	142.00	1.00	1.75	n/a	1.75	
GL-20-022	15.00	18.40	3.40	0.59	n/a	0.59	Main Zone (Zone 3)
and	30.00	32.00	2.00	0.18	n/a	0.18	
and	108.00	116.00	8.00	0.35	n/a	0.35	
including	108.00	109.00	1.00	1.21	n/a	1.21	
and incl.	112.00	113.00	1.00	1.14	n/a	1.14	
GL-20-023	14.50	15.50	1.00	0.75	n/a	0.75	Main Zone (Zone 2, 3)
and	52.00	62.00	10.00	1.42	n/a	1.42	
including	52.00	54.54	2.54	5.24	n/a	5.24	
and	131.86	139.00	7.14	1.05	n/a	1.05	
including	131.86	132.86	1.00	2.90	n/a	2.90	
and incl.	138.00	139.00	1.00	2.72	n/a	2.72	
and	146.00	147.00	1.00	0.81	n/a	0.81	
and	157.00	158.00	1.00	0.41	n/a	0.41	
and	173.00	193.00	20.00	0.50	n/a	0.50	
including	173.00	185.00	12.00	0.77	n/a	0.77	
and incl.	184.00	185.00	1.00	6.95	n/a	6.95	
GL-20-024	26.00	27.00	1.00	0.32	n/a	0.32	Main Zone (Zone 3)
and	107.00	129.00	22.00	0.48	n/a	0.48	

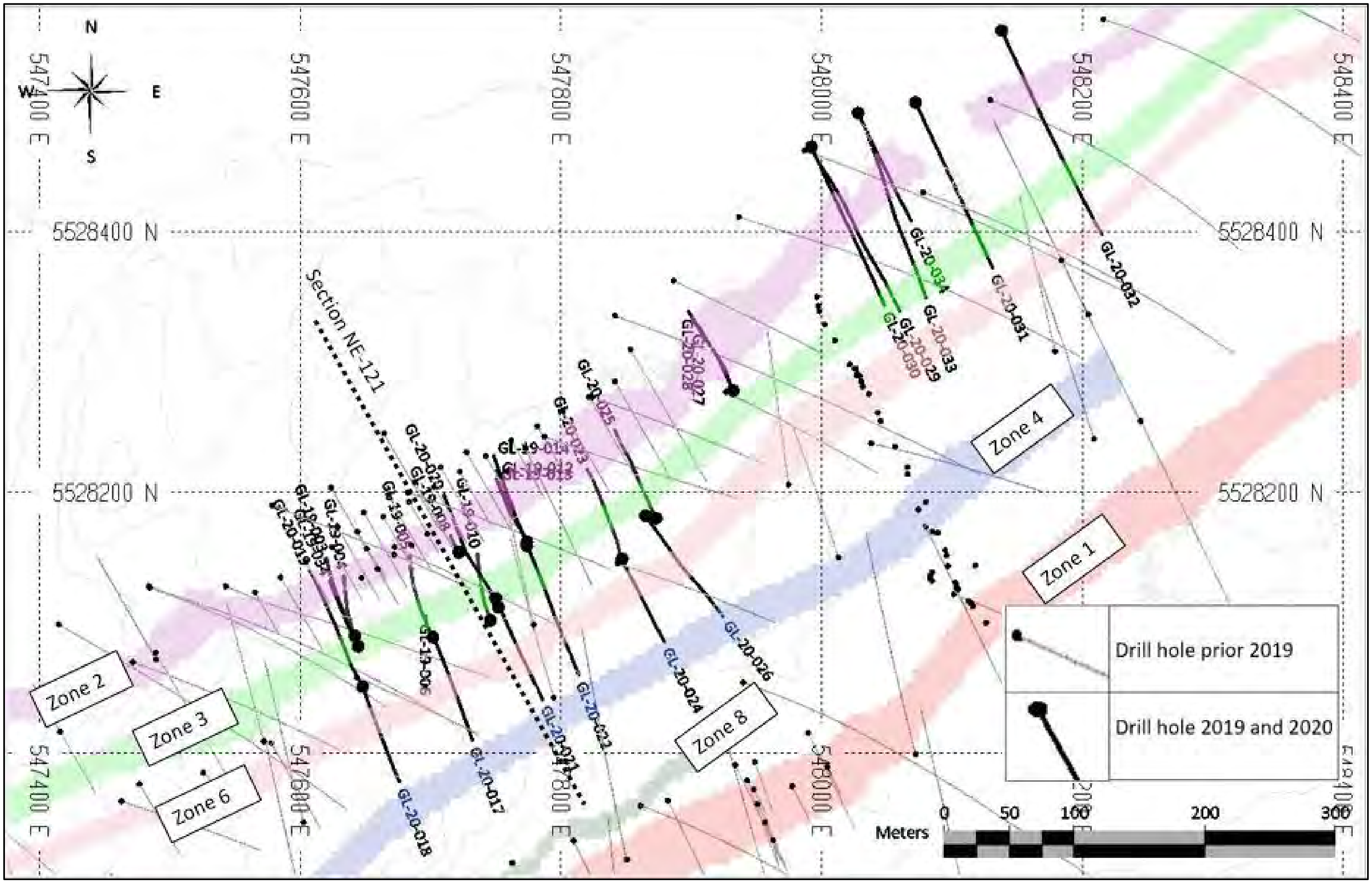
Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
including	107.00	114.00	7.00	1.22	n/a	1.22	
and incl.	107.00	109.00	2.00	3.36	n/a	3.36	
and	156.00	157.00	1.00	0.24	n/a	0.24	
and	159.00	160.00	1.00	0.26	n/a	0.26	
and	182.00	183.00	1.00	3.03	n/a	3.03	
GL-20-025	23.00	54.18	31.18	1.82	n/a	1.82	Main Zone (Zone 2, 3)
including	23.00	39.00	16.00	3.08	n/a	3.08	
and incl.	24.00	25.00	1.00	20.12	n/a	20.12	
and incl.	33.05	33.65	0.60	7.58	n/a	7.58	
and incl.	35.00	36.00	1.00	6.03	n/a	6.03	
and	118.00	134.00	16.00	1.54	n/a	1.54	
including	126.00	134.00	8.00	2.95	n/a	2.95	
and	145.78	167.27	21.49	0.56	n/a	0.56	
including	150.00	160.00	10.00	0.84	n/a	0.84	
and incl.	159.00	160.00	1.00	3.77	n/a	3.77	
and incl.	166.00	167.27	1.27	2.18	n/a	2.18	
GL-20-026	5.00	6.00	1.00	0.18	n/a	0.18	Main Zone (Zone 3)
and	31.00	32.00	1.00	6.22	n/a	6.22	
and	38.00	39.00	1.00	0.12	n/a	0.12	
and	43.00	44.00	1.00	0.10	n/a	0.10	
and	55.00	56.00	1.00	0.28	n/a	0.28	
and	78.00	79.00	1.00	0.14	n/a	0.14	
and	97.00	119.00	22.00	0.17	n/a	0.17	
including	97.00	98.00	1.00	2.16	n/a	2.16	
and	137.00	137.76	0.76	0.25	n/a	0.25	
GL-20-027	28.00	66.71	38.71	1.39	n/a	1.39	Main Zone (Zone 2)
including	31.00	32.61	1.61	5.22	n/a	5.22	
and incl.	35.67	37.01	1.34	19.54	n/a	19.54	
and incl.	37.01	38.00	0.99	3.01	n/a	3.01	
and incl.	55.45	57.00	1.55	4.42	n/a	4.42	
and	83.00	98.00	15.00	0.33	n/a	0.33	

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
GL-20-028	16.03	38.00	21.97	2.51	n/a	2.51	Main Zone (Zone 2)
including	20.00	35.00	15.00	3.58	n/a	3.58	
and incl.	20.00	29.55	9.55	5.46	n/a	5.46	
and incl.	28.00	29.55	1.55	24.08	n/a	24.08	
and	46.00	59.00	13.00	0.55	n/a	0.55	
including	54.00	59.00	5.00	1.15	n/a	1.15	
and	64.14	65.00	0.86	1.17	n/a	1.17	
and	72.00	77.00	5.00	0.97	n/a	0.97	
including	73.00	74.00	1.00	3.87	n/a	3.87	
GL-20-029	73.00	80.00	7.00	0.21	n/a	0.21	Main Zone (Zone 2)
and	92.00	93.00	1.00	3.54	n/a	3.54	
and	123.00	124.00	1.00	1.81	n/a	1.81	
and	133.00	151.00	18.00	1.69	n/a	1.69	
including	141.00	151.00	10.00	2.98	n/a	2.98	
and incl.	150.00	151.00	1.00	19.93	n/a	19.93	
and	175.00	176.00	1.00	0.96	n/a	0.96	
GL-20-030	97.00	101.00	4.00	0.15	n/a	0.15	Main Zone (Zone 2)
and	152.00	154.00	2.00	0.29	n/a	0.29	
and	169.00	180.00	11.00	0.42	n/a	0.42	
including	175.00	179.00	4.00	0.72	n/a	0.72	
GL-20-031	30.00	38.00	8.00	0.49	n/a	0.49	Main Zone (Zone 2)
including	37.00	38.00	1.00	1.73	n/a	1.73	
and	71.00	94.00	23.00	0.28	n/a	0.28	
including	73.00	89.00	16.00	0.36	n/a	0.36	
and incl.	85.00	86.00	1.00	1.28	n/a	1.28	
GL-20-032	57.00	58.00	1.00	0.42	n/a	0.42	Main Zone (Zone 2)
and	125.00	126.00	1.00	0.22	n/a	0.22	
and	138.00	139.00	1.00	0.47	n/a	0.47	
and	172.00	173.00	1.00	1.89	n/a	1.89	
and	200.71	202.21	1.50	0.26	n/a	0.26	
GL-20-033	61.00	66.00	5.00	0.63	n/a	0.63	Main Zone (Zone 2)

Hole ID	From (m)	To (m)	Length (m)	Au g/t Fire Assay	Au g/t with Metallics	Au g/t Final*	Target
and	73.00	74.00	1.00	173.80	n/a	173.80	
and	99.00	100.00	1.00	0.34	n/a	0.34	
and	119.00	122.00	3.00	0.62	n/a	0.62	
and	197.00	198.00	1.00	0.57	n/a	0.57	
GL-20-034	36.50	37.50	1.00	0.21	n/a	0.21	Main Zone (Zone 2)
and	101.00	136.00	35.00	0.32	n/a	0.32	
including	104.00	111.00	7.00	1.14	n/a	1.14	
and incl.	104.00	105.00	1.00	5.10	n/a	5.10	
and incl.	110.00	111.00	1.00	1.65	n/a	1.65	

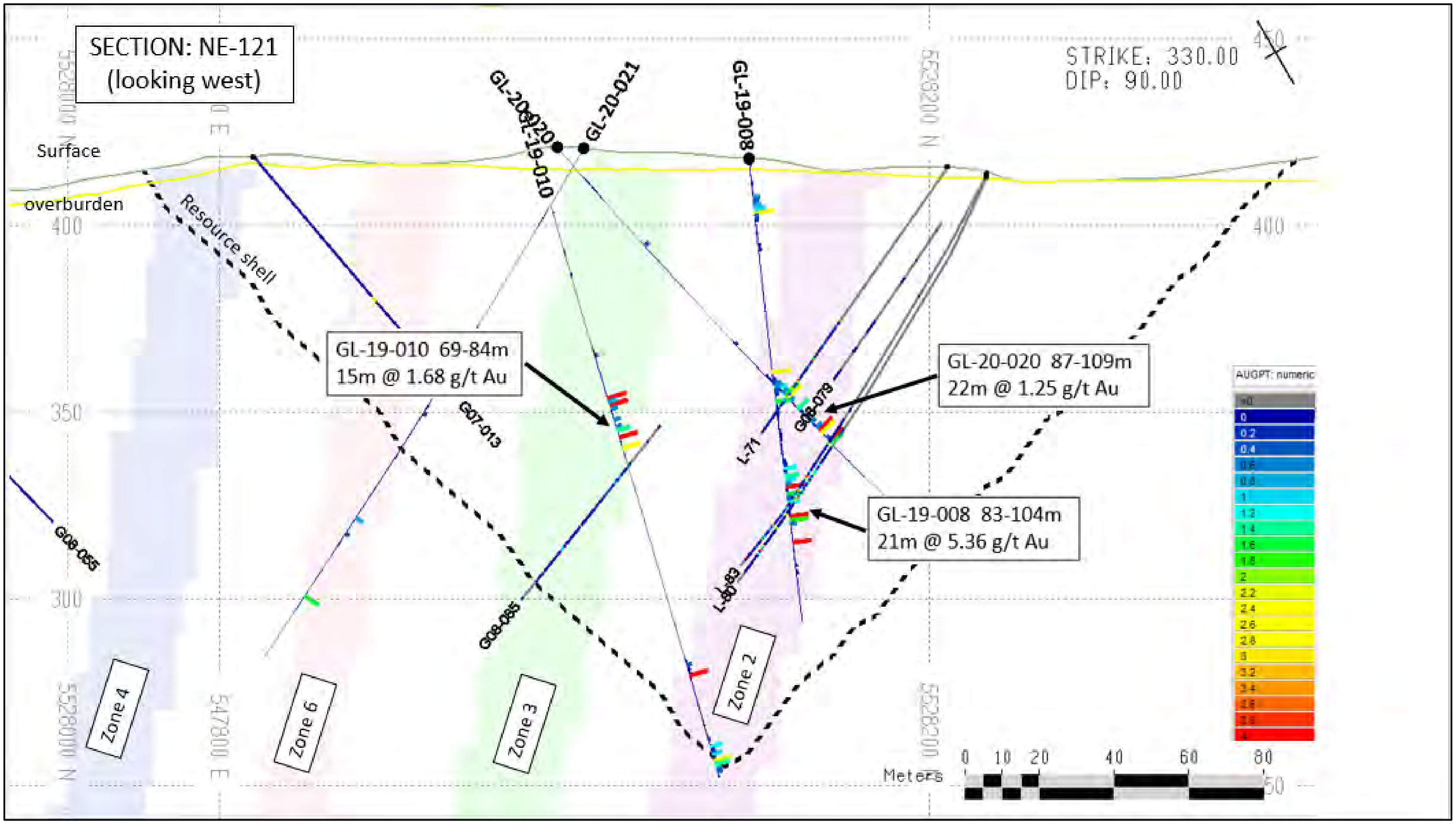
Notes: Assaying for the Main Zone drill program was completed by SGS at their laboratories in Red Lake, Ontario and Vancouver, BC. Prepared 50 g samples were analysed for gold by lead fusion fire assay with an atomic absorption spectrometry (AAS) finish. Multi-element analysis was also completed on selected holes by two-acid aqua regia digestion with ICP-MS and AES finish. Reported widths are drilled core lengths; true widths are unknown at this time. Assay values are uncut. Final collar coordinates surveyed by differential GPS. Intervals for the Goldlund Main Zone holes GL-19-003, GL-19-008, GL-20-006, GL-20-009 and GL-20-010 include results of selected assay repeats. These repeats were done by screened metallic fire assay on 1 kg size samples at the SGS laboratories in Lakefield and Vancouver. Final gold grades include results of metallic screen fire assay reruns ("metallics"), where completed. Source: First Mining Gold Corp Press Releases (2020).

Figure 10-7: Goldlund Drillhole Location Map, 2019-2020, Northeast Main Zone (Zone 2, 3)



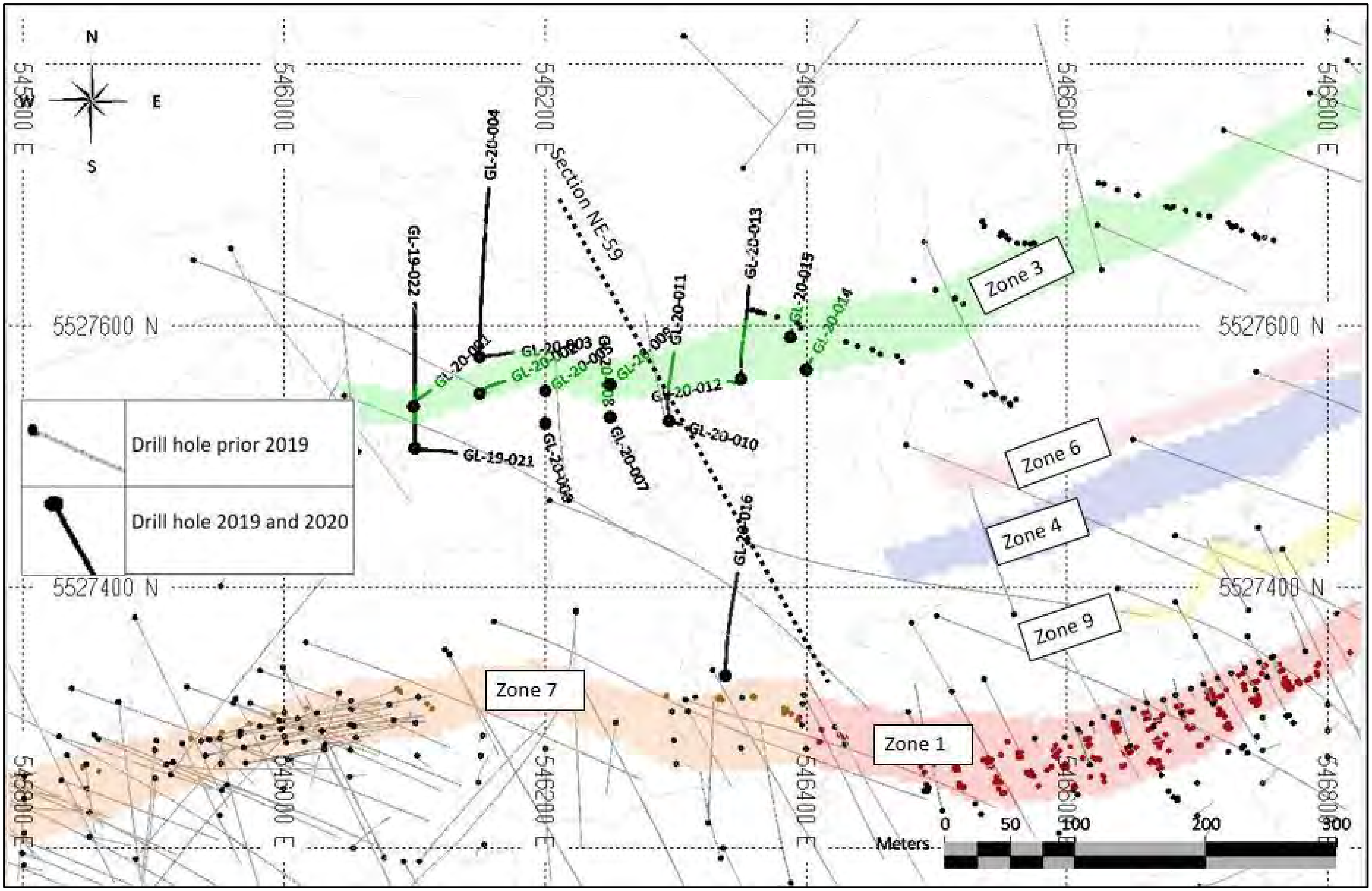
Source: CGK (2020).

Figure 10-8: Goldlund Cross-Section NE-121, 2019-2020, Northeast on Main Zone (Zone 2, 3)



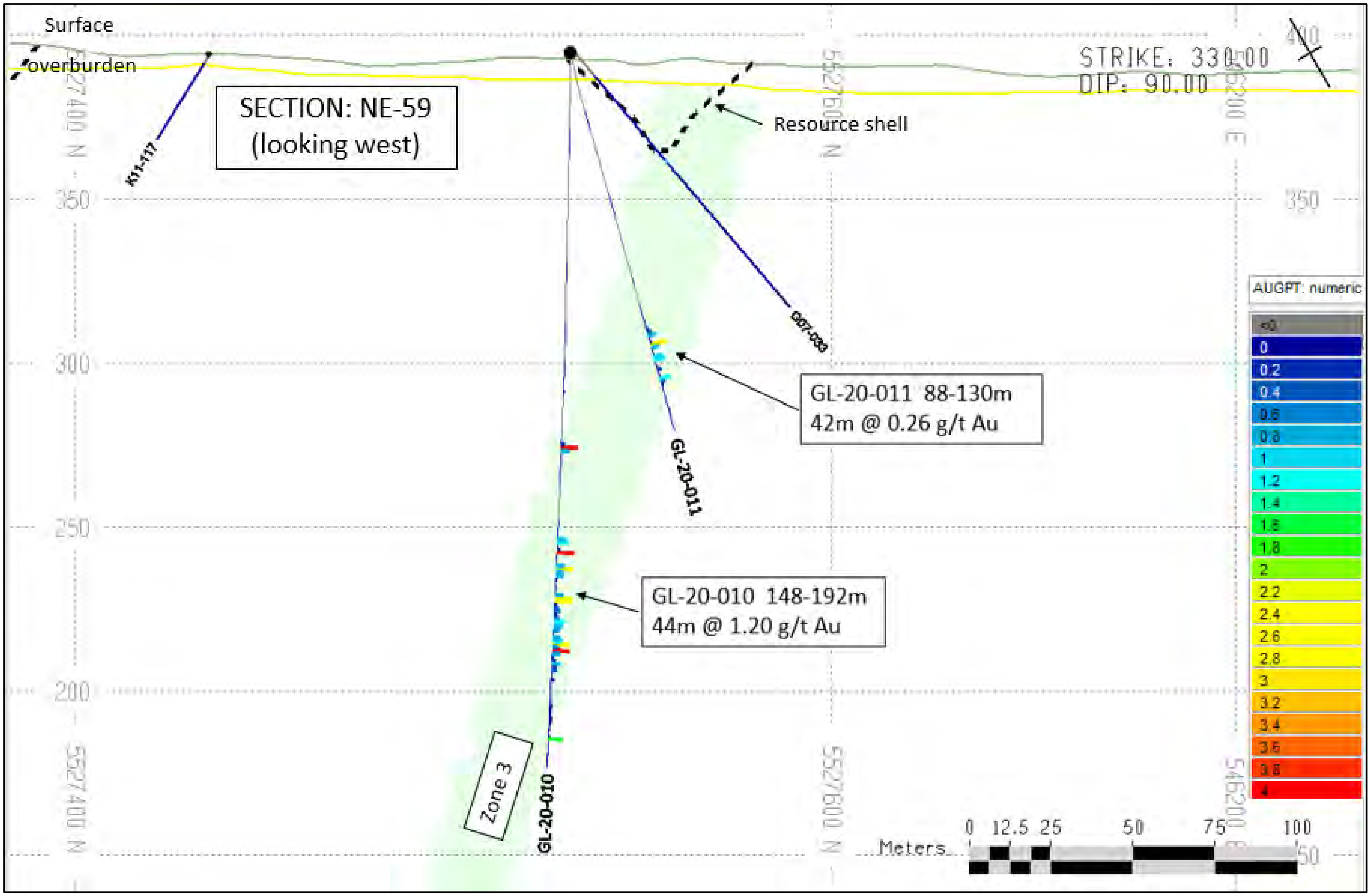
Source: CGK (2020).

Figure 10-9: Goldlund Drillhole Location Map, 2019-2020, Southwest on Main Zone (Zone 3)



Source: CGK (2020).

Figure 10-10: Goldlund Cross-Section NE-59, 2019-2020, Northeast on Main Zone (Zone 3)



Source: CGK (2020).

10.3.2 Tamaka Holdings, Pre-2019

Drilling prior to 2019 has been carried out by various companies including Tamaka Holdings Inc. through its wholly-owned subsidiary, Goldlund Resources Inc., (Goldlund Resources) from 2007 to 2013, and by First Mining Gold Corp. from 2017 to 2018. The drilling procedures prior to 2019 for First Mining are similar to those described in the previous section and will not be repeated here.

10.3.2.1 2007 & 2008 Drilling

In 2007 and 2008, Tamaka carried out a drilling program of 109 holes totalling 29,259 m of surface drilling on the Project. The drilling was completed by Bradley Brothers of Timmins. All holes were drilled NQ (47.6 mm) and NQ2 (50.6 mm) and all drilling runs were in 10 ft intervals (3 m). The collars were initially spotted with a hand-held GPS and the final completed collars were surveyed by a land surveyor from Dryden. Downhole surveys were completed using the Reflex Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore system is not affected by the magnetic influence in the surrounding environment.

The NQ or NQ2 core was received at the logging facility and the run lengths were measured to confirm the block markers. The core recovery and RQD were measured and then entered into a Microsoft Excel template. Magnetic susceptibility measurements were taken at 0.5 m intervals using a hand-held unit. The core was photographed both wet and dry. Logging of the lithology, structure, alteration, and sulphide content were recorded in an Excel spreadsheet template. Sample lengths were marked and range from 0.20 to 1.5 m, but do not cross lithological boundaries.

The samples were taken continuously from collar to the end of the hole. The drill core was sawn in half, with one half placed in a plastic sample bag, and the other half returned to the core box. One of the sample tags was placed in the sample bag, while the other tag was stapled into the core box. The sample bags were then sealed with fibre tape. QA/QC samples were inserted into the sample stream and the samples were placed in rice bags, then sealed and stored in the secure logging facility until shipment. The samples were delivered by a Tamaka employee to Manitoulin Transport in Dryden, Ontario for delivery to the Accurassay Laboratory in Thunder Bay, Ontario. The laboratory returned all coarse rejects and pulps to Tamaka for safe and secure storage at the Project.

10.3.2.2 2011 Drilling

In 2011, Tamaka carried out a drilling program of 31 holes totalling 12,782 m of surface drilling. The drilling was completed by C3 Drilling of Ithaca, New York. All holes were drilled NQ (47.6 mm) and all drilling runs were in 10 ft intervals (3 m). The drilling program was managed independently by geologists employed by Fladgate Exploration based in Thunder Bay and monitored by the Vice President of Exploration for Tamaka.

The collars were initially spotted using a hand-held GPS and the final completed collars were surveyed with a handheld GPS. Downhole surveys were completed using the Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore® system is not affected by magnetic influence in the surrounding environment.

Drill core was delivered by C3 Drilling to the Tamaka core logging facility located on site and the run block measurements were checked. The core recovery and RQD were recorded and magnetic susceptibility measurements were made using a hand-held instrument for each 3 m length of core. Drillholes K11-110 to K11-120 were logged into Microsoft Excel spreadsheets, while from K11-121 onwards, holes were logged into a Gemcom© Gemslogger (Gemslogger) Microsoft Access database. A geologist logged the core, recording lithology, alteration, structure, and mineralisation in Gemslogger on the spreadsheet, marking the intervals with a grease pen. Sample lengths range between 0.2 and 2.6 m in length, with an average sampling length of around 0.7 m. The samples did not cross lithological boundaries and at least two shoulder samples are taken on either side of the mineralisation. Core was photographed after logging and sampling was completed, both wet and dry.

The core was sawn using a top-mounted diamond saw blade. Half of the core was placed in a sample bag while the other half was replaced in the core box. The QA/QC samples consisting of standard reference material (SRM), blanks and duplicates were inserted into the sample stream. For field duplicates, the remaining half of the core was quarter split and placed in a sample bag. For coarse duplicates, a sample tag was placed in an empty sample bag. The sample tag was stapled to the inside of the sample bag and the sample bag is stapled sealed. The samples were placed into rice bags and stored in crates awaiting shipment. Crates were shipped every week to Accurassay in Thunder Bay by Manitoulin Transport. The laboratory returned all course rejects and pulps to Tamaka for storage at the Project.

10.3.2.3 2013-2014 Drilling

In 2013 to 2014, Tamaka carried out a drilling program of 24 holes totalling 9,000 m of surface drilling. The drilling was completed by C3 Drilling of Ithaca NY and North Star Drilling of Thunder Bay. All holes were drilled NQ (47.6 mm) and all drilling runs were in 10 ft (3 m) intervals. The drilling program was managed independently by geologists employed by Fladgate Exploration based in Thunder Bay and monitored by the Tamaka employees.

The collars were initially spotted with a hand-held GPS and the final completed collars were surveyed with a differential GPS. The downhole surveys were completed using the Reflex Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore® system is not affected by magnetic influence in the surrounding environment.

The NQ core was received at the logging facility and the run lengths were measured to confirm the block markers. The core recovery and RQD were measured and then entered into a Microsoft Excel template. Magnetic susceptibility measurements were taken at 0.5 m intervals using a hand-held unit. The core was photographed both wet and dry. Logging of the lithology, structure, alteration, and sulphide content were recorded directly into a Microsoft Excel spreadsheet template. Sample lengths are variable and range from 0.20 to 1.5 m; however, the samples do not cross lithological boundaries.

The drill core selected to be sampled was sawn in half with one half placed in a plastic sample bag, with the other half returned to the core box. One of the sample tags was placed in the sample bag while the other tag was stapled into the core box. The sample bags were then sealed with fibre tape. QA/QC samples were inserted into the sample stream and the samples were placed in rice bags, then sealed and stored in the secure logging facility until shipment.

The samples were delivered by a Tamaka employee to Manitoulin Transport in Dryden, Ontario for delivery to the Accurassay Laboratory in Thunder Bay, Ontario. The laboratory returned all coarse rejects and pulps to Tamaka for safe and secure storage at the Project.

10.3.2.4 Historical Drilling

Prior to 2006, considerable surface and underground drilling had been completed on the Project by various operators since the 1940s. Drill logs, assay summaries, and assay certificates for the majority of these historical drillholes are available and were compiled into a digital format to support the mineral resources estimate. A summary of the historical work is described in Section 6.

The procedures of the various historical drilling programs are not documented. Sampling details for the historical programs prior to 2006 have not been verified by the Qualified Person for this section of the report. No QA/QC programs are believed to have been conducted at that time. The legible quality of the diamond drill logs, and assay certificates has allowed for the construction and validation of the historical drilling, sampling, and assay results in the drillhole database.

10.3.3 Qualified Person Opinion

The Qualified Person responsible for this section of the report believes, based on its review of selected drill core and the description of the logging and sampling methodology provided in various technical reports, that the drilling and sampling was undertaken in accordance with industry standards and best practices. The Qualified Person also believes that the data is sufficiently accurate to be reliable and is therefore suitable for use in the estimation of mineral resources.

10.4 Miller Deposit

Treasury Metals has not conducted any drilling programs on the Miller deposit since it acquired the property. All drilling on the Miller deposit was completed by First Mining in 2018 and 2019 targeting a geophysical anomaly.

10.4.1 First Mining, 2018-2019

10.4.1.1 Drilling

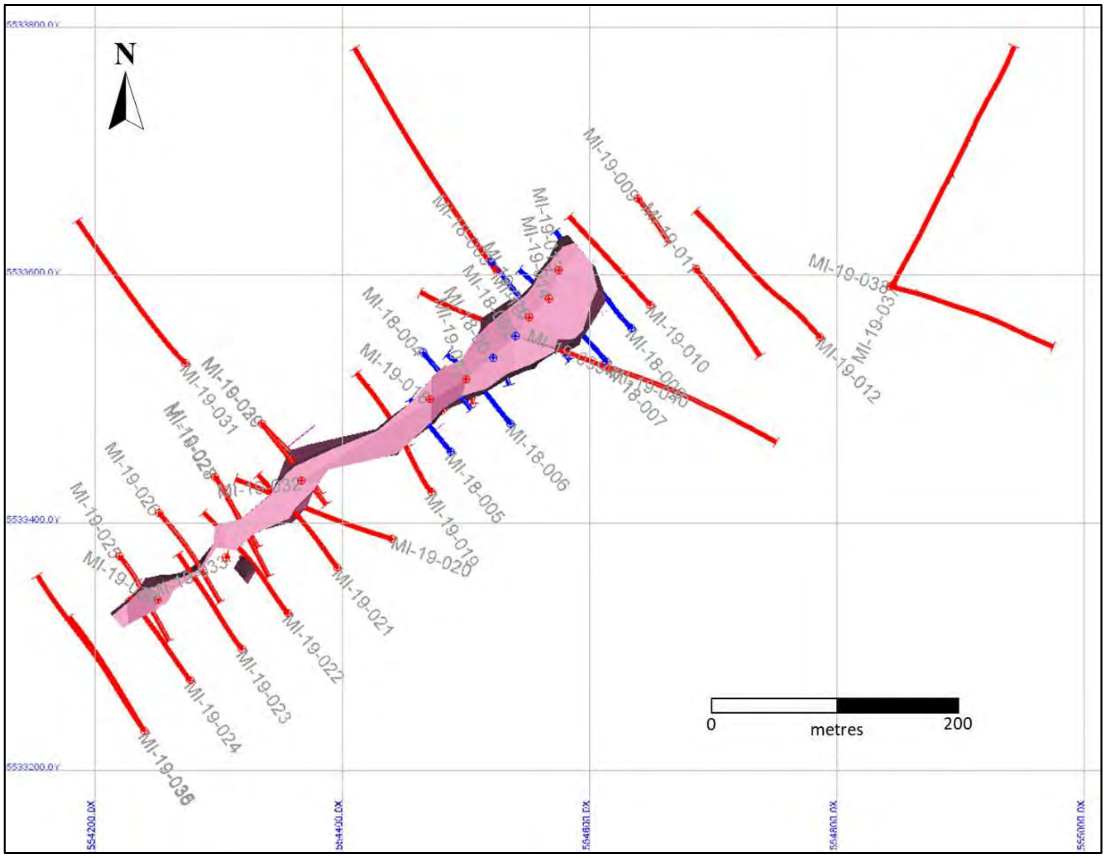
In 2018, First Mining completed several diamond drill programs on three targets on the Miller property intended to test the potential to host gold mineralisation similar to that at the Goldlund Project. These targets included Miller, Eaglelund and Miles. The Miller deposit was subject to an initial program of eight drillholes totalling 1,255.5 m that targeted a geophysical anomaly. A follow-up drill program in 2019 was conducted along strike of the 2018 drillholes based on significant gold intercept results from the initial drill program.

Table 10.7 presents a summary of the drill programs on the Miller deposit. Figure 10-11 presents a drill location map on the Miller deposit.

Table 10.7: Summary of Drill Programs – Miller Deposit

Year	Contractor	Core Size	No. Drillholes	No. Metres
2018	Rodren Drilling	HQ	8	1255.5
2019	Rodren Drilling	NQ	32	6130.0
Totals			40	7385.5

Figure 10-11: Miller Drillhole Location Map



Notes: Blue: 2018 drillholes; Red: 2019 drillholes. Source: AGP (2020).

Drilling was completed by Rodren Drilling Ltd., based in Winnipeg, Manitoba. Drill core size was HQ (63.5 mm) from the 2018 drilling program and NQ (47.6 mm) from the 2019 drilling program.

Drillholes were surveyed downhole using a Reflex or EZ Shot device. The downhole survey was carried out at approximately 30 m to 60 m intervals. Drillholes were initially located in the field using either a differential or handheld GPS.

Drill core was transported to the Goldlund exploration camp for logging and sampling.

Table 10.8 lists the drillholes completed at Miller.

Table 10.8: Miller Drillhole Collar Information

Hole ID	Collar UTM East	Collar UTM North	Hole Azimuth ⁰	Hole Dip ⁰	Final Depth (m)	Target
MI-18-001	554522	5533533.2	140	-80	140.5	Miller
MI-18-002	554540	5533550.8	140	-85	200	Miller
MI-18-003	554520.6	5533610.7	140	-55	170	Miller
MI-18-004	554465.4	5533537.4	140	-55	101	Miller
MI-18-005	554487.5	5533457.7	320	-65	110	Miller
MI-18-006	554535.9	5533480.2	320	-65	170	Miller
MI-18-007	554613.5	5533530.3	320	-60	182	Miller
MI-18-008	554633.7	5533557.3	315	-60	182	Miller
MI-19-009	554639	5533662	140	-75	167	
MI-19-010	554649	5533576	315	-60	170	
MI-19-011	554686	5533605	140	-60	161	
MI-19-012	554786	5533550	320	-60	236	
MI-19-013	554575	5533604	140	-85	251	Miller
MI-19-014	554567	5533581	140	-85	245	Miller
MI-19-015	554551	5533566	140	-85	224	Miller
MI-19-016	554525	5533603	320	-45	278	
MI-19-017	554500	5533516	140	-85	242	Miller
MI-19-018	554471	5533500	120	-85	212	Miller
MI-19-019	554472	5533425	320	-55	176	Miller
MI-19-020	554440	5533387	290	-55	215	Miller
MI-19-021	554396	5533364	320	-60	173	Miller
MI-19-022	554356	5533327	320	-60	167	Miller
MI-19-023	554319	5533298	320	-60	164	Miller
MI-19-024	554277	5533273	320	-60	146	Miller
MI-19-025	554220	5533373	140	-65	176	Miller
MI-19-026	554252	5533408	140	-60	161	Miller
MI-19-027	554297	5533437	140	-60	128	Miller
MI-19-028	554297	5533437	140	-45	125	Miller
MI-19-029	554335	5533480	135	-70	203	Miller
MI-19-030	554335	5533480	140	-45	113	Miller
MI-19-031	554273	5533529	315	-45	185	
MI-19-032	554367	5533434	0	-90	212	Miller
MI-19-033	554306	5533372	140	-90	155	
MI-19-034	554251	5533338	113	-90	179	Miller
MI-19-035	554240	5533232	325	-45	200	
MI-19-036	554240	5533232	325	-65	197	
MI-19-037	554845	5533592	27	-45	287	
MI-19-038	554843	5533591	106	-45	185	
MI-19-039	554614	5533526	108	-45	185	
MI-19-040	554616	5533525	287	-45	212	Miller

10.4.1.2 Core Logging & Sampling

The following was taken from WSP (2020).

The core logging methodology and QA/QC procedures were overseen by Mr. Miro Mytny, P.Geo, Senior Exploration Manager for First Mining. The logging procedures applied during the Miller drill programs were as follows:

- Drill core was cleaned, and the run (meterage) blocks checked. After this, the runs were measured for recovery. The recovery percentage was then used to mark-off the adjusted metres within the run. The RQD was measured and recorded in an Excel® spreadsheet, for importing into Datamine DH Logger software.
- The core was logged for lithology, alteration, minerology, veining, and structure directly into DH Logger, which synchronises with First Mining's central Fusion SQL drilling database.
- One-metre sample intervals were marked-off, except at lithological contacts, and in zones of poor recovery, where sample size could be adjusted accordingly.
- Standards and blanks were inserted in the sample stream at the required intervals.
- Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates.
- Core pieces were selected and measured for specific gravity.
- The core was photographed twice, both dry and wet.
- The core was sawn in half on site, with one half bagged and labelled to be sent for assay. For field duplicates, the core was quartered, and one quarter was sent for the regular assay and the other quarter was sent for the duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the coarse reject or the pulverised sample to run a duplicate assay.
- The remaining half core was placed in core boxes which are stored in a secure on-site facility to serve as a permanent record.
- Sample bags were placed in zip-tied rice bags and shipped to SGS Laboratory facilities in Red Lake, Ontario and Lakefield, Ontario for fire assay analysis.

10.4.1.3 Results

The 2018 and 2019 drill programs at the Miller consists of 40 drillholes where 28 drillholes intersected the core of the deposit. Drilling was completed over the Miller deposit at approximately 50 m to 100 m spacing and covers an area approximately 500 m x 100 m. Where the 2018 drill program discovered a core of gold mineralisation at Miller, the 2019 drill program defined the extension of the mineralisation along strike, mainly to the southwest.

Table 10.9 lists selected drillhole intercepts in the Miller deposit with significant gold values. The results demonstrate the presence of a core of gold mineralisation the deposit is still open at depth and along strike to the southwest. The northeast end, currently, appears truncated by a regional structure. Figure 10-12 shows a selected cross-section of the Miller deposit.

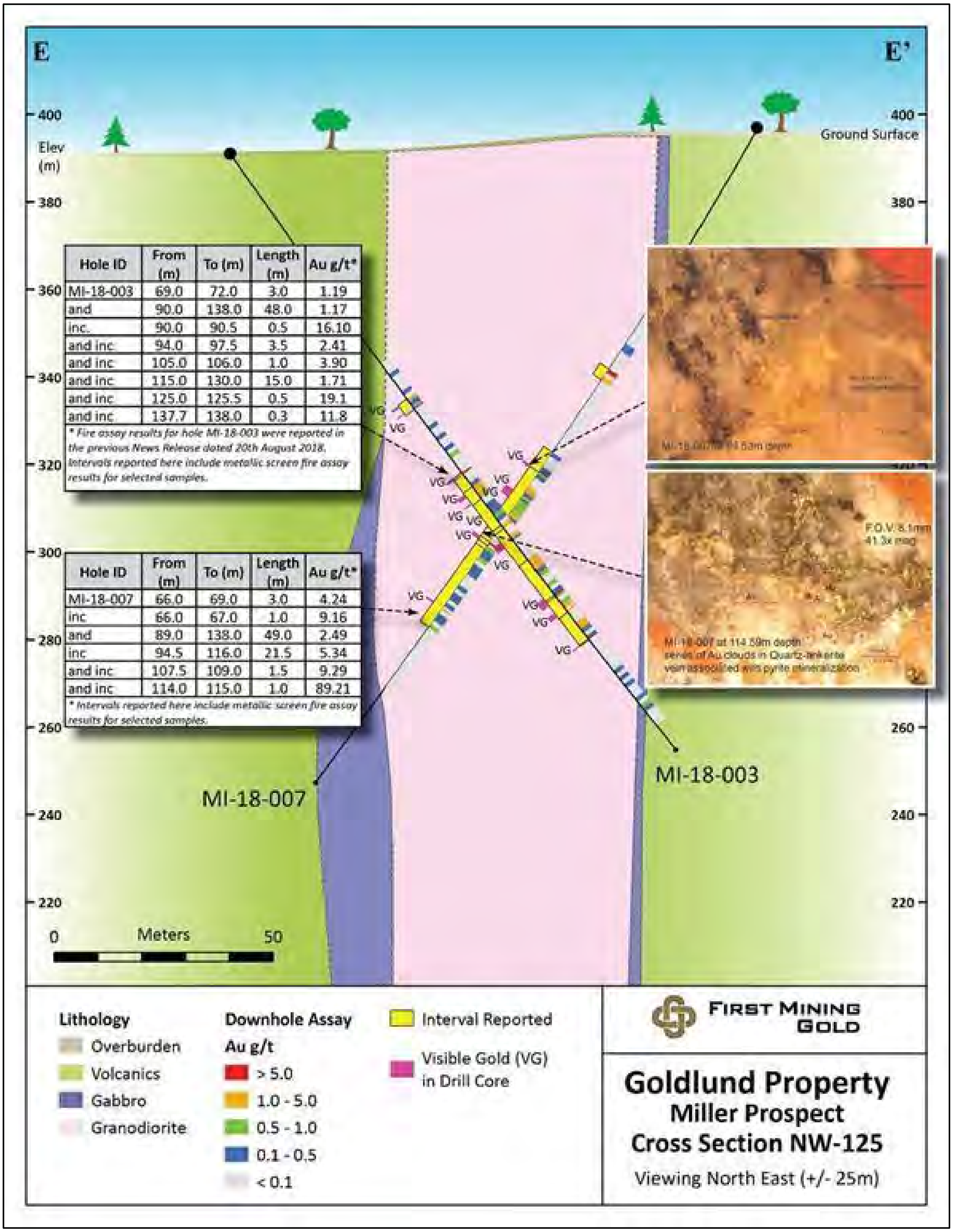
Table 10.9: Summary of Significant Drillhole Intercepts – Miller Deposit

DH No.		From (m)	To (m)	Width (m)	No. Metres
MI-18-001		7	114.6	107.6	0.33
	including	15	88.6	73.6	0.41
	including	16	18.3	2.3	1.93
	including	18	18.3	0.3	8.59
	including	23.3	29.6	6.3	0.91
	including	27.3	27.6	0.3	8.67
	including	77.6	88.6	11	1.17
	including	87.6	88.6	1	6.27
MI-18-002		0.42	142.5	142.08	1.9
	including	1.5	109.5	108	2.44
	including	57.5	88.5	31	4.44
	including	75.5	82.5	7	14.67
	including	81.5	82.5	1	88.8
	including	102.5	109.5	7	9.6
	including	108.5	109.5	1	54.47
MI-18-003		69	72	3	1.12
	and	90	138	48	1.07
	including	90	90.5	0.5	17.23
	including	94	97.5	3.5	2.28
	including	105	106	1	3.9
	including	115	130	15	1.41
	including	125	125.5	0.5	10.55
	including	137.7	138	0.3	9.87
MI-18-004		34	57.8	23.8	0.54
	including	34	35	1	2.56
	including		52	57.8	5.8
	including		55	56	1
MI-18-005		46	47	1	4.18
	including	68	78	10	0.43
	including		72	74	2
	including	109	110	1	1
MI-18-006		76	77	1	1.38
	and	102	124	22	0.69
	including	103	109.4	6.4	2.09
	including	103.62	104	0.38	21.66
	including	109	109.4	0.4	4.69
	and	145	147	2	1.48
	and	169	170	1	3.01
MI-18-007		66	69	3	4.24
	including	66	67	1	9.16
	and	89	138	49	2.53
	including	94.5	116	21.5	5.43
	including	107.5	109	1.5	8.83
	including	114	115	1	91.41
MI-18-008		135	149	14	0.58
	including	135.5	138	2.5	1.59
	including	146	147	1	2.14
MI-19-013		46	228	182	1.09
	including	46	50	4	9.15

	including	47	48	1	35.19
	including	88	109	21	2.73
	including	107	113	6	3.95
	including	134	147	13	2.67
MI-19-014		3	210	207	1.57
	including	42	91	49	2.34
	including	56	70	14	4.53
	including	60	61	1	26.43
	including	142	183	41	4.07
	including	168	182	14	7.38
	including	168	169	1	55.28
MI-19-015		1	168	167	1.01
	including	1	26	25	1.62
	including	5	8	3	5.4
	including	108	141	33	1.84
	including	120	122	2	5.82
MI-19-017		6	7	1	1.48
	and	32	201	169	0.88
	including	56	93	37	3.42
	including	79	93	14	7.27
	including	83	84	1	65.97
	including	85	86	1	11
MI-19-018		18	141	123	0.86
	including	67	141	74	1.18
	including	100	134	34	2.08
	including	105	106	1	6.49
	including	113	114	1	12.91
	including	129	130	1	23.96
	and	168	169	1	4.24
MI-19-019		65	101	36	0.41
	including	68	69	1	2.78
	including	83	85	2	2.09
	including	100	101	1	1.62
MI-19-020		133	139	6	1.77
	including	134	135	1	8.15
MI-19-021		111	118	7	0.99
	including	112	113	1	4.78
MI-19-022		115	122	7	0.82
	including	119	120	1	1.56
	including	121	122	1	2.58
MI-19-032		39	143	104	0.25
	including	60	80	20	0.40
	including	79	80	1	3.56
	and	107	143	36	0.38
	including	126	127	1	5.50
MI-19-040		60	119	59	1.35
	including	60	62	2	5.91
	including	78	83	15	3.88
	including	80.88	81.88	1	6.83
	including	86.88	87.88	1	44.07

Source: WSP (2020), First Mining Press Releases (2018, 2019, 2020).

Figure 10-12: Miller Deposit Cross-Section Looking Northeast 050°Az



Source: First Mining Press Release (2018).

10.4.2 Qualified Person Opinion

The Qualified Person responsible for this section of the report reviewed selected drill core from the Miller deposit to verify the logging and sampling procedures were in accordance with industry standards. The Qualified Person compared the selected drill core to the drill logs to verify that the descriptions, lithological and sampling intervals were correctly described. The Qualified Person believes the data is sufficiently accurate to be reliable and is suitable for use in the estimation of mineral resources.

11 SAMPLE PREPARATION, ANALYSES & SECURITY

11.1 Goliath Project

11.1.1 Teck-Corona Sample Preparation & Analysis, 1990-1998

Teck-Corona samples were typically 0.5, 1.0 and 1.5 metres, but could range between a low of 0.3 m to 2.5 m with very few exceptions. All samples were shipped to the primary laboratory by Gardwine and Porter transport firms. The primary laboratory used was TSL Laboratories (TSL) of Saskatoon, Saskatchewan. XRAL Laboratories and Interteck Testing Services were used for assay verification work or whole rock analyses.

Not much detail is available on sample preparation and analysis procedures during that period. The following was extracted from the Teck bulk sample program and it is assumed that the analytical procedure at the TSL Laboratory in Saskatoon for the face and muck samples was similar to what was use for the drill core submitted to that laboratory.

The samples were prepared by crushing the whole samples 90% passing -10 mesh and then splitting into 250 g sub-sample. The pulverised sub-sample was then analysed by fire assays with either atomic absorption (FA-AA) or gravimetric (FA-GRAV) finish. Silver was analysed by dissolution (aqua regia digestion?) and atomic absorption spectrometry (AAS). High-grade samples were known to have be analysed by 1000 g pulp metallics.

11.1.2 Teck-Corona Quality Control & Quality Assurance (QA/QC), 1990-1998

No details were available with regard to the quality control and quality assurance (QA/QC) program during that period. AGP notes that the insertion of blanks, and analytical standards were rarely done in the 1990s, but check assays at an umpire laboratory was fairly common.

11.1.3 Treasury Metals Sample Preparation & Analysis

As described in Section 10 of this report, the drill core for the Goliath Project was logged and split with a core saw lengthwise, with the majority of samples ranging from 1.0 m to 1.5 m in length. Half of the core was retained for future verification and the other half was sent to the analytical laboratories. A two primary laboratories were used between the 2008 and 2020 drill campaign.

11.1.3.1 Accurassay Laboratory, 2008-2015

Accurassay Laboratory (Accurassay) was used by Treasury Metals from 2008 to 2015. Once the rock samples were received at the Accurassay's facilities in Thunder Bay, Ontario, they were entered into the Laboratories Local Information System (LIMS).

The samples were prepare using procedure code ALP1. Samples were dried then jaw crushed to 8 mesh size. A 500 g split was then pulverised to approximately 90% passing -150 mesh and then matted to ensure homogeneity. Silica abrasive sand was used to clean out the pulverising dishes between each sample to prevent cross contamination. Some certificates listed ALP2 procedure code which is similar to the ALP1 but crushing at 90% passing -8 mesh and collecting a 1000 g split instead of the 500 g. Once prepared, the samples were then sent to the fire assay laboratory or the wet chemistry laboratory depending on the required analysis.

For gold, all samples were assayed using code "ALFA1", denoting a 30 g fire assay with an AAS finish.

Starting during the 2009 drill program, samples grading above 5 g/t Au were re-assayed using the code "ALFA7", which indicated a gold fire assay with a gravimetric finish. This was altered to all samples grading above 3 g/t Au for the 2010-2012 drill programs. It reverted to samples above 5 g/t Au for drill programs occurring between 2013-2015.

From 2008 to 2015, samples returning values in excess of 5.0 g/t Au were analysed with the pulp metallic method code "ALPM1". The 2015 drilling program used 6.0 g/t Au as the threshold limit. Accurassay described the pulp metallic method as a procedure that is able to overcome the "nugget effect" of gold by increasing the sub-sample size to 1,000 g and physically collecting the free gold within the system using a 150 mesh (106 µm) sieve. This procedure is most effective when the whole sample is used for the analysis. The sub-sample is pulverised to ~90% - 150 mesh (106 µm) and subsequently sieved through a 150-mesh (106 µm) screen. The entire +150 metallics portion is assayed along with two duplicate sub-samples of the -150 pulp portion. Results are reported as a weighted average of gold in the entire sample.

Geochemistry for silver and a suite of six or nine additional elements from 2008 to the beginning of the 2010 drill campaign. Late in 2010 through to 2015 Treasury Metals ran geochemistry for silver and 29 other elements using procedure code "ALMA1", which is described as a multi-acid digestion with an inductively coupled plasma with optical emission spectrometry (ICP-OES) finish.

A certificate was produced from the LIMS laboratory database system. The laboratory manager checks the data, validates the certificates, and issues the results as a PDF file and a Microsoft Excel file.

Accurassay was accredited by ISO/IEC 17025 was accountable to the Standards Council of Canada for its quality management at the time the samples were processed. Accurassay filed for bankruptcy on May 16, 2017.

11.1.3.2 Activation Laboratories (ActLabs), 2016-2020

Starting in 2016 Treasury Metals submitted samples to the Activation Laboratory Ltd. (ActLabs) in Dryden. At the ActLabs facility, the samples were processed using procedure code RX1, which is described as crushing up to 80% passing 2 mm, riffle splitting a sub-sample of 250 g, and pulverising to 95% passing 105 µm.

Sample pulps were then assayed using procedure code 1A2-50, which is a 50 g fire assay with AA finish. Samples grading above 3 g/t Au were re-assayed with code 1A3-50, which is a 50 g fire assay with gravimetric finish.

High-grade samples in excess of 5 g/t Au were assayed using procedure code 1A4-1000, which is a metallic screen assay. For this type of assay, a representative 500 g split (1,000 g for 1A4-1000) is sieved at 100 mesh (149 µm) with fire assays performed on the entire +100 mesh and 2 splits on the -100 mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fraction is weighed for assay reconciliation.

Starting in 2016, Treasury Metals assayed the sample for silver and an additional 37 elements on selected samples within the mineralised zones only. The samples are analysed using ActLabs code 1E3, which is described as a partial digestion by aqua regia with an ICP-OES for the analysis. The method quantitatively dissolves base metals for the majority of geological materials, but major rock-forming elements and more resistive metals are only partially dissolved. As such, the leach should be considered partial for most elements.

ActLabs in Dryden was assessed by TRC Inc. and found to be in conformance to the ISO 9001:2015 standard (Certificate number TRC 01028).

11.1.4 Treasury Metals QA/QC Program

Treasury Metals implemented and monitored a thorough QA/QC program for the diamond drilling and sampling undertaken at the Goliath property from 2008 through 2020. QC protocol included the insertion of control samples into every batch sent off for analysis. The QA/QC protocols were altered somewhat over the program, as described in the following sections by year.

A number of certified reference materials (CRMs) were used throughout the years. During the 2008 drill program, CRMs were supplied by Accurassay and CDN Resource Laboratories Ltd of Delta, BC, and ORE Pty Ltd (now OREAS). The CDN Laboratory CRMs were found to be more reliable and Treasury Metals exclusively used the CRMs supplied by CDN Laboratory for the subsequent years. Table 11.1 summarised the various CRMs used throughout the years.

The discussion in this section will focus on diamond drill core. AGP notes that QA/QC samples were also inserted in exploration samples (soil, rock, trench, channel) not used in the resource estimate.

Table 11.1: Summary of CRM Used Throughout the Years

Standard (CRM)	Recommended Value Au(ppm)	Standard Deviation Au (ppm)	Supplier	Drill Program Year												
				2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
AuQ1	1.33	0.114	Accurassay	X												
Au43	12.686	0.859	Accurassay	X												
CDN-FCM4 *	0.97	0.04	CDN	X												
AuG1	1.019	0.04	Accurassay	X												
AuG2	1.013	0.02	Accurassay	X												
OREAS_61D *	4.76	0.14	OREAS	X	X	X	X									
Au48	16.15	0.964	Accurassay	X												
CDN-GS-5D	5.06	0.125	CDN	X	X	X	X									
CDN-SE-2 *	0.242	0.009	CDN	X	X	X	X									
CDN-GS-1D	1.05	0.05	CDN		X											
CDN-GS-1F	0.242	0.009	CDN			X	X									
CDN-CGS-13	1.01	0.055	CDN			X	X									
CDN-CM6 **	1.43	0.045	CDN			X	X									
CDN-ME-6 *	0.27	0.014	CDN			X	X									
CDN-GS-P2A	0.229	0.015	CDN					X	X							
CDN-CM-26 **	0.372	0.024	CDN					X	X	X	X	X	X	X		
CDN-GS-2K	1.97	0.09	CDN					X	X	X	X					
CDN-GS-5J *	4.96	0.21	CDN					X	X							
CDN-GS-1P5K	1.44	0.065	CDN							X	X	X	X	X		
CDN-GS-5P *	4.78	0.155	CDN							X	X	X		X		
CDN-GS-1P5P	1.59	0.075	CDN									X	X	X		
CDN-GS-5T *	4.76	0.105	CDN									X	X	X	X	
CDN-CM-26 **	0.372	0.024	CDN										X		X	
CDN-GS-1P5Q	1.329	0.05	CDN											X	X	
CDN-CM-43	0.309	0.02	CDN												X	X
CDN-GS-1P5R	1.81	0.07	CDN												X	X
CDN-GS-4H	5.01	0.15	CDN												X	X

Notes: *Denotes CRM is also certified for silver. ** Denotes CRM with a provisional or indicated silver value.

11.1.4.1 2008 QA/QC Program

To monitor accuracy, CRMs (or standards) and blanks were inserted into the sample stream by Treasury Metals at a rate of at least 1 in every 20 samples submitted.

A total of nine CRMs were utilised to monitor gold results over the course of the 2008 drill program including the AuQ1, Au43, CDN-FCM4, AuG1, AuH2, OREAS_61D, Au48, CDN-GS-5D, CDN-SE2. Treasury Metals selected a mixture of low-, medium-, and high-grade CRMs to monitor lab accuracy. A summary of the standards used is given in Table 11.1

Treasury Metals uses a mean ±3x standard deviation as control limit and mean ± 2x standard deviation as warning limit. Any single standard analysis beyond the upper and lower control limit is considered a "failure". Treasury Metals also consider a failure when three successive standard analyses are outside the upper and lower warning limits on the same side of the mean.

11.1.4.1.1 Performance of Certified Reference Materials

It was reported that most standard failures occur at the beginning of the drill program. Oreas61D and Au48 retuned erratic results and were replaced by CCIC with more reliable standards. Failure of a standard within the mineralised horizon prompted the resubmission of the pulps for the entire batch.

Most of the CRMs monitoring accuracy within the mineralised zone returned values within three standard deviations from the mean. The CDN_GS-5D mean value is low when compared to the certify mean and likely a matrix match issue. There were 20 failures within the mineralised zone and the pulp samples from all 20 batches were re-analysed at Accurassay to confirm results.

11.1.4.1.2 Performance of Blank Material

The blank material used for the QC monitoring was a prepared blank supplied by Accurassay that was pulverised to -200 mesh, blended and packaged in 60 gram packets. The blank was inserted at a rate of at least one in 20 samples and has a gold concentration of less than 15 ppb. A tolerance limit of 45 ppb was set by the Company to evaluate for contamination. AGP note that the blank material used in 2008 is unsuitable to monitor cross contamination at the crushing stage of the sample preparation.

Sixteen samples of 636 returned results greater than the 45 ppb tolerance limit, and of those 16, three lay within the mineralised zone. Two were sample misallocations, where a standard was used instead of a blank, and the remaining sample is not considered by the author to be of significant impact to the resource.

11.1.4.1.3 Performance of Duplicate Samples

Field duplicates often consist of one-quarter core duplicate, coarse laboratory rejects and laboratory pulp duplicates. Re-inserting coarse rejects and pulp duplicates in the sample stream is an additional protocol seen at some operations, but it is not common. Quarter-core duplicates are common, but in high nugget deposits they are not always reliable. The problem is also compounded by the smaller volume of sample submitted to the laboratory. During the 2008 drill campaign, Treasury Metals did not insert any duplicate samples into the sample stream.

The Accurassay laboratory pulp duplicates were monitored by Treasury Metals by graphing the results and after reviewing the chart provided, AGP agrees with A.C.A. Howe International Limited (A.C.A. Howe) that the 1,318 lab pulp duplicates showed good correlation between the original samples and duplicates.

11.1.4.1.4 Check at Umpire Laboratory

In many QA/QC programs, pulp duplicates are also submitted for external check analyses at an umpire laboratory to provide an independent check of relative bias and accuracy. The submission rate is usually 5% of the pulps. Treasury Metals did not submit check samples to a second laboratory during the 2008 exploration program.

11.1.4.2 2009 QA/QC Program

Treasury Metals undertook a similar QA/QC program throughout the 2009 drill program, with every tenth sample being either a low- or medium-grade CRM or blank. The insertion of quarter-core (field) duplicates was implemented for this program. Insertion rates are summarised in Table 11.2.

Table 11.2: Insertion Rate for the 2009 Drill Program

Insertion Rate	QA/QC Sample Type
10 samples	
Insert	Low-grade CRM
10 samples	
Insert	Blank
5 samples	
Collect	Quarter-core duplicate
5 samples	
Insert	Medium-grade CRM
10 samples	
Insert	Blank
5 samples	
Collect	Quarter-core duplicate
5 samples	
Insert	High-grade CRM
10 samples	
Insert	Blank

Source: Treasury Metals (2009).

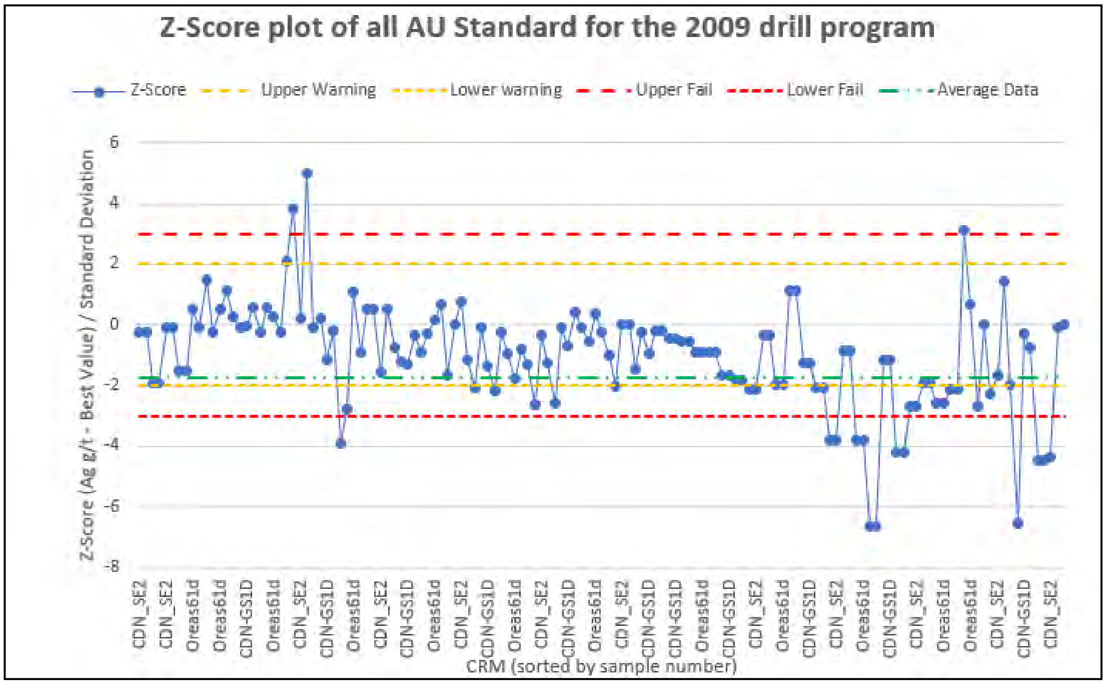
11.1.4.2.1 Performance of Certified Reference Materials

The Company utilised five CRMs to monitor gold results for the 2009 drill program, including the OREAS 61D, CDN-GS-1D, CDN-GS-5D and the CDN-SE2 (Table 11.1).

CRM results were monitored the same as in the 2008 drill program. The majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean. There were eight failures within the mineralised zone, so Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent six samples in the batch to confirm results.

AGP could not locate the CRM charts and consequently plotted a Z-Score chart of the CRM results (Figure 11-1). The chart reveals a low failure rate, but also a degradation in precision with higher sample number near the end of the program.

Figure 11-1: Z-Score Chart for 2009 CRM



Source: AGP (2020).

11.1.4.2.2 Performance of Blank Material

The same blank material that was used for the 2008 QC monitoring was used again in 2009. A tolerance limit of 45 ppb was set by the Company to evaluate for contamination.

There were 184 data points for the blank material and all results were below three times the detection limit of the analysis type (15 ppb).

11.1.4.2.3 Performance of Quarter-Core Duplicate

Details on the performance of the quarter-core duplicate could not be located.

11.1.4.3 2010-2011 QA/QC Program

Treasury Metals continued their QA/QC program in similar fashion throughout the 2010 and 2011 drill program.

11.1.4.3.1 Performance of Certified Reference Materials

The Company utilised seven CRMs to monitor gold results for the 2010 and 2011 drill programs, including the CDN-SE-2, CDN-GS-1F, CDN-GS-5D, OREAS 61D, CDN-CGS-13, CDN-CM-6, and CDN-ME-6 (Table 11.1).

CRM monitoring continued in the same fashion as the previous years and the majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean. Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent five samples for any batches where failures occurred within the mineralised zone (if any samples were greater than 5.0 g/t Au) to confirm results. Some failures were considered to be misallocated blanks or standards and the records were changed accordingly. AGP reviewed the chart provided in a report authored by Julie Selway and the chart reported by A.C.E. Howe Ltd (2012) and concurred with the finding described above. CRM Oreas-61D was showing a low bias first quarter of the program then a then a high bias in the last three quarters of the program.

11.1.4.3.2 Performance of Blank Material

The same blank material that was used for the 2008 and 2009 QC monitoring was used again in 2010. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were a total of 291 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (15 ppb). One result was considered a misallocated CDN-CM-6 standard and the other sample (at 0.045 g/t Au) was considered the only failure.

11.1.4.3.3 Performance of Duplicate Samples

Treasury Metals submitted 970 quarter-core duplicate samples into the 2010 and 2011 drill programs. A.C.A. Howe (2012) reported the results of the field duplicate data and a plot of the original versus duplicate material. The data shows acceptable correlation between the original samples and quarter-core duplicates. Most deviation can be attributed to the nugget effect. A.C.A. Howe reported that very few high-grade samples were submitted and recommended that Treasury Metals collect additional quarter-core duplicates from the mineralised zones. AGP noted that about one-third of the samples collected graded in excess of 0.2 g/t with approximately nine samples above 3 g/t.

11.1.4.4 2012-2013 QA/QC Program

The 2012-2013 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low- or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample.

11.1.4.4.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2012 and 2013 drill programs, including the CDN-GS-P2A, CDN-CM-26, CDN-GS-2K and CDN-GS-5J (Table 11.1).

A slightly higher rate of failures was noted by the Company at the commencement of the 2012-2013 drill program for the CDN-GS-2K standard, with four out of 18 failures in total. Overall, 28 standards failed, where results were greater than three standard deviations away from the CRM mean value. Out of these 28 failures, 22 samples were selectively chosen to retest due to their proximity to mineralised zones and magnitude of failure.

11.1.4.4.2 Performance of Blank Material

The same blank material was continued to be used for the 2012-2013 QC monitoring. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 197 data points for the blank material, and all results except three were below three times the detection limit of the analysis type (15 ppb). None of these failures were considered to be of significant impact to the resource.

11.1.4.4.3 Performance of Duplicate Samples

The Company submitted 750 quarter-core duplicate samples for assaying during the 2012-2013 drill program. The results of the original and duplicate data display poor precision as is to be expected for these coarse level duplicates. AGP re-plotted the data with seven outliers removed and found the regression showed a R2 value of 0.86 with a slope of regression of 0.81and agreed with the poor precision due to nugget.

11.1.4.4.4 Re-assay Comparison

Treasury Metals re-assayed 742 samples due to failure of control samples. The re-assay results were charted in a QA/QC report. AGP reviewed the data and found that the re-assayed sampled for 2012 and 2013 compared well with the original assay as evidenced by a R^2 value of 0.92 and a slope of regression of 1.08 which indicates virtually no bias.

11.1.4.4.5 Pulp Re-submitted to ALS Chemex

Treasury Metals re-submitted pulps analysed at Accurassay for five drillholes (TL13316, TL13318, TL13319, TL13322, TL13323) to be assayed at ALS labs in response to the high failure rate at the beginning of the 2013 drill program. ALS's service was prompt and had zero failed standards throughout the five holes.

11.1.4.5 2014-2015 QA/QC Program

The 2014-2015 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low- or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample.

11.1.4.5.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2014-2015 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-2K and CDN-GS-5P (Table 11.1).

CRMs were monitored in the same fashion as the previous years and the majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean. Starting in 2015, Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent five samples for any batches where failures occurred within the mineralised zone to confirm results as oppose to re-submit the entire batch.

Overall, 21 standards failed out of a total of 274, where results were greater than three standard deviations away from the CRM mean value. Out of these 21 failures, 10 samples were selectively chosen to retest due to their proximity to mineralised zones and magnitude of failure. Additional failures were considered to be misallocated blanks or standards and the records were altered accordingly. AGP deems that the performance of the CRM's during this drill program was excellent with no systemic bias shown on any of the charts that were inspected.

11.1.4.5.2 Performance of Blank Material

The same blank material was continued to be used for the 2014-2015 QC monitoring. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were a total of 277 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (15 ppb). None of these failures were considered to be of significant impact to the resource.

11.1.4.5.3 Performance of Duplicate Samples

The Company submitted quarter-core duplicate samples only for assaying during the 2014-2015 drilling program. The results of the original and duplicate data were plotted on a scatter plot and poor (but acceptable) correlation is displayed for these coarse level duplicates.

Treasury Metals did not insert any other duplicate samples into the sample stream; however, Accurassay's pulp duplicates and crusher replicate samples were available for analysis. All data was analysed for gold and the pulp duplicates displayed excellent precision.

11.1.4.5.4 Laboratory change (Accurassay – ActLabs) & Assay Verification

For the 2016 drill program, Treasury Metals started using the Activation Laboratories (ActLabs) in Dryden due to the closure of the Accurassay facility in Thunder Bay.

In order to validate the analytical results from both laboratory, Treasury Metals submitted 328 pulp samples from Accurassay Laboratory for check assaying to ActLabs Laboratory in Thunder Bay. Pulp samples were taken from 29 drillholes, drilled over the 2014 to 2015 period. Samples were sent in two batches of 134 and 194 pulp samples.

Scatter plots and line graphs of the ActLabs results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

AGP reviewed 194 paired samples from the second batch previously assayed by Accurassay and re-assayed by ActLabs. The results indicated a good correlation as evidenced by a R^2 of 0.99. The slope of regression was 0.95 which indicate a slight negative bias.

11.1.4.6 2016 QA/QC Program

The 2016 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample. The laboratory derived blank material was replaced by a crushable blank material in 2016 which is suitable to monitor contamination at the sample preparation stage.

11.1.4.6.1 Performance of Certified Reference Materials

Five CRMs were used to monitor gold results for the 2016 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P, CDN-GS-5T and CDN-GS-5P (Table 11.1).

CRMs were monitored in a similar fashion as the previous years and the majority of the CRMs within the mineralised zone returned values within the acceptable limits of three standard deviations from the mean. Overall, 11 standards failed out of a total of 276, where results were greater than three standard deviations away from the CRM mean value. A slightly higher rate of failures was noted by the Company at the commencement of the 2016 drill program for the CDN-CM-26 standard, which accounted for five out of the 11 failures in total. There was also a slightly elevated failure rate for the CDN-GS-5T standard, which accounted for three out of the 11 failures. None of these failures were considered to be of significant impact to the resource.

AGP noted that the CDN-GS-1P5K showed a slight positive bias and CDN-GS-5T displayed a slight negative bias and resulted in the higher failure rate.

11.1.4.6.2 Performance of Blank Material

In 2016, a coarse blank made from bags of crushed granite replaced the packaged blank (CDN-BL-10) used in previous years. A total of 10 test samples were sent to the lab to ensure that the material was suitable for use. All test samples returned values below detection limit. A tolerance limit of 15 ppb was maintained by Treasury Metals to evaluate for contamination.

There were 281 samples of blank material and all results, except one, were below three times the detection limit of the analysis type (5 ppb).

11.1.4.6.3 Performance of Duplicate Samples

The Company submitted 278 quarter-core duplicate samples for assaying during the 2016 drill program. The results of the original and duplicate data were plotted on a scatter plot and show acceptable correlation for these coarse level duplicates.

11.1.4.7 2017 QA/QC Program

The 2017 QA/QC program carried out by Treasury Metals followed the same protocol as 2016 with the addition of checks samples submitted at Agat Laboratory.

11.1.4.7.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2017 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P and CDN-GS-5T (Table 11.1).

CRMs were monitored in the same fashion as the previous years and the majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean value. Overall, 12 standards failed out of a total of 343, where results were greater than three standard deviations away from the CRM mean value. A slightly higher rate of failures was noted by the Company at the commencement of the 2017 drill program for the CDN-CM-26 standard, with five out of twelve failures in total. There was also an elevated failure rate for the CDN-GS-5T standard with six out of twelve failures. Out of these 12 failures, 11 were actual failures not selected for retesting as the failures were considered to have minimal impact to the resource. The remaining sample flagged for failure, was not an actual failure but a misallocated standard that fell within acceptable limits.

AGP noted that the CDN-GS-5T showed a slight negative bias during the first half of the drill program then a slight positive bias in the second half of the program. This suggest that a change occur at the laboratory.

11.1.4.7.2 Performance of Blank Material

The same blank material was again used for the 2017 QC monitoring (coarse crushed granite). A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 343 samples of blank material and all results, except five, were below three times the detection limit of the analysis type (5 ppb). None of these failures were considered to be of significant impact to the resource.

11.1.4.7.3 Performance of Duplicate Samples

The company submitted 341 quarter-core duplicate samples for assaying during the 2017 drill program. The results of the original and duplicate data were plotted on a scatter plot and show acceptable correlation for these coarse level duplicates.

11.1.4.7.4 Agat Laboratory Check Samples

Treasury Metals submitted 172 pulp samples to AGAT Laboratory located in Mississauga, Ontario for check assaying from ActLabs Laboratory in Thunder Bay. Pulp samples were taken from 10 drillholes drilled during 2017.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

AGP reviewed the results of this new program and the regression produced for the review indicated a R^2 of 0.95 with a slope of regression of 1.06 with 3 outliers removed from the data set. The average differences between the assays were 0.001 g/t Au.

11.1.4.8 2018 QA/QC Program

The 2018 QA/QC program carried out by Treasury Metals followed the same protocol implemented in 2016 including the use of an umpire laboratory.

11.1.4.8.1 Performance of Certified Reference Materials

Six CRMs were used to monitor gold results for the 2018 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P, CDN-GS-1P5Q, CDN-GS-5P and CDN-GS-5T (Table 11.1).

CRMs were monitored in the same fashion as previous years and the majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean. Overall, 23 standards failed out of a total of 569, where results were greater than three standard deviations away from the CRM mean value. A higher rate of failures was noted by the Company for the CDN-GS-5T standard, accounting for 11 out of the 23 failures. Of these failures, 8 standards were selected for retesting due to their proximity to significant mineralisation. All standards selected for retesting have fallen within acceptable limits and no further action is deemed necessary. The remaining failed standards were not considered to be of significant impact to the resource.

AGP noted that the CDN-CM-26 showed a slight positive bias and the negative bias displayed by the CDN-GS-5T is no longer visually noticeable on the chart presented.

11.1.4.8.2 Performance of Blank Material

The Company continued to use the coarse crushed granite blank material for the 2018 QC monitoring program. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 569 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (5 ppb). Neither of the two failures was considered to be of significant impact to the resource

11.1.4.8.3 Performance of Duplicate Samples

The Company submitted 569 quarter-core duplicate samples for assaying during the 2018 drill program. The results of the original and duplicate data were plotted on a scatter plot and acceptable correlation is displayed for these coarse level duplicates.

11.1.4.8.4 Agat Laboratory Check Samples

Treasury Metals submitted 560 pulp samples to AGAT Laboratory for check assaying to ActLabs Laboratory in Thunder Bay in 2018. Pulp samples were taken from 25 drillholes, drilled over the 2018 period.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

11.1.4.9 2019-2020 QA/QC Program

The 2019-2020 QA/QC program carried out by Treasury Metals followed the same protocol implemented in 2016 including the use of an umpire laboratory.

11.1.4.9.1 Performance of Certified Reference Materials

Six CRMs were used to monitor gold results for the 2019 - 2020 drill program, including the CDN-CM-26, CDM-CM-43, CDN-GS-1P5Q, CDN-GS-1P5R, CDN-GS-4H, and CDN-GS-5T (Table 11.1).

CRMs were monitored in the same fashion as previous years and the majority of the CRMs within the mineralised zone returned values within three standard deviations from the mean. Overall, 23 standards failed out of a total of 357, where results were greater than three standard deviations away from the CRM mean value. A higher rate of failures was noted by the Company for the CDN-GS-4H standard, accounting for 14 out of the 23 failures. Of these failures, three standards were selected for retest due to their proximity to significant mineralisation. All standards selected for retesting have fallen within acceptable limits and no further action is deemed necessary. The remaining failed standards were not considered to be of significant impact to the resource.

11.1.4.9.2 Performance of Blank Material

The company continued to use the coarse crushed granite blank material for the 2019-2020 QC monitoring program. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 354 data points for the blank material and all results, except one, were below three times the detection limit of the analysis type (5 ppb). The failure was not considered to be of significant impact to the resource.

11.1.4.9.3 Performance of Duplicate Samples

The Company submitted 357 quarter-core duplicate samples for assaying during the 2019-2020 drill program. The results of the original and duplicate data were plotted on a scatter plot and acceptable correlation is displayed for these coarse level duplicates.

11.1.4.9.4 Agat Laboratory Check Samples

Treasury Metals submitted 323 pulp samples to AGAT Laboratory for check assaying to ActLabs Laboratory in Thunder Bay in 2018. Pulp samples were taken from 14 drillholes, drilled over the 2019-2020 period.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

11.1.4.10 Qualified Person Opinion

Treasury Metals routinely charts all QA/QC samples. If a trend exists or samples deviate from the norm, either the entire batches or a number of samples surrounding the “failure” are re-submitted to the laboratory for check assays.

The data shows no evidence of systemic contamination during the assaying process and since a crushable blank material was utilised, the data show no evidence of systemic cross-contamination between samples at the sample preparation facility.

The Treasury Metals quarter-core sample duplicate shows evidence of a rather strong nugget effect and AGP question if this protocol should continue. AGP advice Treasury Metals to seek the opinion of a specialist in the QC/QA field.

The Qualified Person reviewed the sample preparation, analytical and security procedures, as well as the insertion rates and performance of blanks, CRM and duplicates from the data provided and concluded that the observed failure rates are within the expected ranges and that no significant assay biases are present.

AGP noted a lack of QA/QC follow up on the silver assays. All charts and figures presented by Treasury Metals focussed on gold. Blanks and CRM that have a certified silver value should also be charted.

Based upon the evaluation of the QA/QC program undertaken by Treasury, it is AGP’s opinion that the results are acceptable for use in the current mineral resource estimate.

11.2 Goldlund Project

Treasury Metals has not conducted any drill programs on the Goldlund Project since it acquired the property. The following is a summary of the detailed presentation of the sample preparation, analysis, and security presented in previous technical reports, including the “Treasury Metals 2020 Technical Report” prepared by WSP. The data files used for the statistical analysis of the QA/QC results were prepared by First Mining and provided to the Qualified Person for this section of the report as a series of Microsoft Excel spreadsheet files.

Assays of the drillhole samples and channel samples for the Goldlund Project have been carried out between 2007 and 2020 by Accurassay and SGS Canada Inc. (SGS) in Red Lake, Ontario, Lakefield, Ontario, and Vancouver, BC. Accurassay is an accredited facility conforming to the requirements of CAN P-4E ISO/IEC 17025 and CAN-P-1579. The SGS laboratories are also accredited facilities conforming to the CAN P-4E ISO/IEC 17025:2017 requirements. ActLabs in Thunder Bay and Ancaster, Ontario carried out independent umpire check assays for the 2017-2018 drilling program samples. ActLabs is an accredited facility conforming to the CAN P-4E ISO/IEC 17025:2017 and ISO 9001:2015 requirements.

Assays of drill core samples prior to 2006 were carried out by commercial laboratories Cochenour Fire Assaying and Paul's Custom Assaying Ltd., both of Red Lake, Ontario. Both assay laboratories operated in the Red Lake area for decades. There is no description available for the sample preparation and assaying or QA/QC programs for the samples prior to 2006.

The assay laboratories that have contributed results to the drillhole database used for the estimation of mineral resources are all independent of Tamaka, First Mining and Treasury Metals. At no time were employees of Tamaka, First Mining or Treasury Metals involved in the preparation or analysis of the samples.

11.2.1 Chain of Custody

Chain of custody and sample security are documented for the Tamaka (2007-2008, 2011, 2013-2014) drilling programs. For these drilling and sampling programs, the sample bags were sealed and kept secure by Tamaka in the Goldlund logging and sampling facility until they were transported to Accurassay in Thunder Bay, Ontario.

Chain of custody and sample security are also documented for the First Mining (2017-2018, 2019-2020) drill programs. For these drilling and sampling programs, the sample bags were sealed and kept secure by First Mining in the Goldlund logging and sampling facility until they were transported to the SGS Laboratories in either Red Lake, Ontario or Vancouver, BC.

The chain of custody for the drilling and sampling programs prior to 2006 is not documented.

From these descriptions, the Qualified Person responsible for this report section believes that both Tamaka and First Mining personnel have taken reasonable measures to ensure the samples were kept secure prior to the shipment of the samples to the respective assay laboratories for analysis.

11.2.2 Sample Preparation

11.2.2.1 Tamaka 2007 & 2008 Sample Preparation

Samples for the Tamaka 2007 and 2008 drilling program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverisers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverised to 90% passing -150 mesh (0.104 mm) and sent for fire assaying.

11.2.2.2 Tamaka 2011 Sample Preparation

Samples for the Tamaka 2011 drilling program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverisers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverised to 90% passing -150 mesh (0.104 mm) and sent for fire assaying.

11.2.2.3 Tamaka 2012 Trenching Sample Preparation

Samples for the Tamaka 2012 trenching program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverisers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverised to 90% passing -150 mesh (0.104 mm) and sent for fire assaying.

11.2.2.4 Tamaka 2013 & 2014 Sample Preparation

Samples for the Tamaka 2013 and 2014 drilling program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverisers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverised to 90% passing -150 mesh (0.104 mm) and sent for fire assaying.

11.2.2.5 First Mining 2017 & 2018 Sample Preparation

Two different sample preparation and analytical procedures were used for the samples from the 2017 and 2018 drilling program.

Samples of mineralised granodiorite material, including the standard, duplicate, and blank samples, were shipped to the SGS in Vancouver for bulk leach extractable gold (BLEG) analysis. The complete half-core sample was crushed to 80% passing -10 mesh (1.68 mm), and then 3,000 g was pulverised in three separate batches of 1 kg each to 85% passing -200 mesh (0.074 mm). The samples were recombined and blended for homogeneity and re-split into three separate 1 kg batches. One of the 1 kg splits was sent for BLEG analysis, while the other two were retained for future testing.

Samples of material other than granodiorite, including the standard, duplicate, and blank samples, were shipped to SGS in Red Lake or Lakefield where they were prepared for fire assay analysis. Samples were dried, crushed to 75% passing -8 mesh (2 mm). A split of 250 g was taken and pulverised to 85% passing -150 mesh (0.104 mm), and sent for fire assaying.

11.2.2.6 First Mining 2019 & 2020 Sample Preparation

Samples from the 2019 and 2020 drilling program at Goldlund, including the standard, duplicate and blank samples, were shipped to SGS laboratories in Red Lake or Vancouver and prepared for fire assay analysis. Samples were dried, crushed to 75% passing -8 mesh (2 mm), and a 250 g split was taken and pulverised to 85% passing -150 mesh (0.104 mm) and sent for fire assaying.

11.2.3 Analysis

11.2.3.1 Tamaka 2007 & 2008 Analysis

The samples from the 2007 and 2008 drilling program were analysed by Accurassay in Thunder Bay for gold and silver using a 50 g aliquot from a 500 g pulp by lead fusion fire assay with an inductively coupled plasma mass spectrometry (ICP-MS) finish.

11.2.3.2 Tamaka 2011 Analysis

The samples from the 2011 drilling program were analysed by Accurassay in Thunder Bay. For samples from drillholes K11-110 to K11-118, a 30 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish. For the samples from drillholes K11-119 to K11-2-140, a 50 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish for gold and silver. For samples more than 10 g/t Au, a second lead fusion fire assay was carried out for gold using either a 30 or 50 g aliquot from a second 500 g pulp with a gravimetric finish.

11.2.3.3 Tamaka 2012 Analysis

The samples from the 2012 trenching program were analysed by Accurassay in Thunder Bay. A 50 g aliquot was taken from a 500 g pulp and analysed by conventional lead fusion fire assay with an AAS finish for gold and silver. For samples that assayed more than 10 g/t Au, a second lead fusion fire assay was carried out for gold using a 50 g aliquot from a second 500 g pulp with a gravimetric finish.

11.2.3.4 Tamaka 2013 & 2014 Analysis

The samples from the 2013 and 2014 drilling program were analysed by Accurassay in Thunder Bay. A 50 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish. For samples assaying more than 10 g/t Au a second lead fusion fire assay was carried out for gold using a second 50 g aliquot from the 500 g pulp with a gravimetric finish.

11.2.3.5 First Mining 2017 & 2018 Analysis

The samples from the 2017 and 2018 drilling program were analysed for gold at either the SGS laboratory in Vancouver, using a BLEG methodology, or the SGS laboratory in Red Lake, using a lead fusion fire assay methodology.

The BLEG methodology uses a large sample (1,000 g) that is digested, or leached, with a cold cyanide solution (LeachWell™ CN) for two hours. The gold in the sample is dissolved as cyanide complexes. The leachate is then concentrated in a solvent exchange type procedure and analysed by AAS or ICP. The large sample sizes and solvent extraction technology used in bulk leach extractable gold analysis provides detection limits as low as 0.1 ppb. The precision of BLEG test results is high due to the large sample size. However, this methodology is not a total assay, so a fire assay of the residual material is also required. This methodology was considered to improve the reproducibility of the gold assays for the “nuggety” Goldlund mineralisation.

The pulverised sample material was weighed and placed into labelled bottles and the cyanide reagent was added. The bottles were agitated using a bottle roll with a leach time of two hours to homogenise the sample with the cyanide solution. Once settled, a layer of clear solution is available for analysis by AAS. The residue sample is then filtered and washed to remove the cyanide solution. The residue is dried, homogenised and a 200 g split is collected, with a 50 g aliquot taken and analysed for gold by a lead fusion fire assay. The final assay is then a combination of the cyanide leachable gold and the residual fire assay gold.

In addition to the gold assay, a 50 g split from each sample was sent for ICP multi-element analysis by two-acid aqua regia digestion with an ICP-MS and atomic emission spectroscopy (AES) finish.

The samples that were sent to the SGS laboratory in Red Lake were assayed for gold using either a 30 g or a 50 g aliquot for lead fusion gold fire assay with an AAS finish.

11.2.3.6 First Mining 2019 & 2020 Analysis

The samples from the 2019 and 2020 drilling program were analysed by SGS laboratories in Red Lake or Vancouver. A 50 g aliquot was taken from a 250 g pulp and analysed for gold by conventional lead fusion fire assay with an AAS finish. For drillholes GL-19-003, GL-19-008, GL-20-006, GL-20-009, and GL-20-010 selected assay repeats were done for gold by screen "metallics" lead fusion fire assay on 1 kg size samples at the SGS laboratories in Lakefield and Vancouver.

11.2.4 QA/QC Results, 2007-2020

Both Tamaka (2007 to 2014) and First Mining (2017 to 2020) carried out QA/QC programs that consisted of the insertion and analysis of blanks, CRMs (or standards), and duplicate samples to monitor the precision and accuracy or the reliability of the assay results from their drilling and sampling programs. This is in addition to the quality control samples that are inserted by the respective assay laboratories that would consist of blanks, standards, and duplicates.

For samples prior to 2006, it is not known if any QA/QC programs were carried out, other than those inserted by the respective assay laboratories at the time.

The QA/QC results are analysed in detail in previous technical reports—including the "Treasury Metals 2020 Technical Report" prepared by WSP—and will only be summarised here.

11.2.4.1 Tamaka, 2007-2008

Tamaka's 2007 and 2008 QA/QC program consisted of the insertion of blanks and CRM samples or "standards" into the sample stream at specified intervals. The standards were inserted every 20th sample, or 5% of the samples, while blanks were inserted every 30th sample, or 3% of the samples. Tamaka did not include any field duplicates in the QA/QC program. In addition to the Tamaka field-inserted QA/QC program, Accurassay operates its own QA/QC protocols. The laboratory inserts quality control materials, blanks, and duplicates with each analytical batch.

The blanks were obtained from ALS Chemex as pre-packaged samples. There were 741 results for the blanks in the QA/QC data files with 40 failures, a failure rate of 5.4%. These blanks have assayed more than 0.022 g/t, the upper control limit. This was a concern for Tamaka, and they replaced this standard with the Nelson granite in future QA/QC programs.

There were 10 different CRM samples incorporated into the samples for assay for the 2007-2008 drilling program. All 10 standards were purchased from Rocklabs (part of Scott Automation or SCOTT® since 2008), and range in expected value from 2.645 g/t Au to 30.104 g/t Au. Table 11.3 lists the standards with their expected values and standard deviation, along with the number of assay results and the average grade of the assays. Those assays that were outside the limit of ±3 standard deviations were considered failures.

There were 1,355 assays of the various standards with 27 being outside the acceptance criteria, or an overall failure rate of approximately 2%. Failure rates for the individual standards range from 0.0% up to 6.7%, with only one being more than 3%, as shown in Table 11.3. The average assayed grade of the standards is typically below the expected value for all the standards. These results confirm that Accurassay was producing sufficiently accurate and precise results such that these assays can be considered reliable.

Table 11.3: Summary of Standards for 2007-2008 Drilling Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2007-2008	Accurassay	FAAU	Rocklabs	OXP39	14.890	0.090	184	13.468	4	2.2%
	Accurassay	FAAU	Rocklabs	OXP61	14.920	0.130	174	13.916	3	1.7%
	Accurassay	FAAU	Rocklabs	SJ32	2.645	0.027	45	2.439	3	6.7%
	Accurassay	FAAU	Rocklabs	SL34	5.893	0.057	136	5.555	0	0.0%
	Accurassay	FAAU	Rocklabs	SL46	5.867	0.066	209	5.549	6	2.9%
	Accurassay	FAAU	Rocklabs	SN26	8.543	0.072	92	8.168	2	2.2%
	Accurassay	FAAU	Rocklabs	SP27	18.100	0.270	87	17.637	2	2.3%
	Accurassay	FAAU	Rocklabs	SP37	18.140	0.150	124	16.555	3	2.4%
	Accurassay	FAAU	Rocklabs	SQ36	30.040	0.240	133	28.766	1	0.8%
	Accurassay	FAAU	Rocklabs	SQ28	30.104	0.300	171	28.573	3	1.8%

11.2.4.2 Tamaka, 2011 & 2012

Tamaka's 2011 and 2012 QA/QC programs consisted of the insertion of blanks, CRM samples or "standards", field duplicates of one-quarter core and coarse duplicates from coarse reject material into the sample stream at specified intervals. The standards were inserted every 20th sample, while blanks were inserted every 30th sample. Field and coarse duplicates were inserted into the sample stream only for the latter portion of the 2011 drilling campaign with a frequency of one field duplicate every 30th sample, and one coarse duplicate every 30th sample. In addition to the Tamaka field-inserted QA/QC program, Accurassay conducts their own QA/QC protocols consisting of quality control materials, blanks, and duplicates with each analytical batch.

The blank sample material was obtained from the Nelson granite quarry near Vermillion Bay, in Northwestern Ontario. There were 400 assays of blank material in the QA/QC data files with only 10 failures, which are blanks that assayed more than the upper control limit of 0.013 g/t Au. This failure rate is considered as acceptable.

The CRMs were obtained from RockLabs (part of Scott Automation or SCOTT® since 2008), and from Geostats Pty Ltd. A total of 11 different standards were used during the 2011 and 2012 sampling campaigns with three in use at any one time. Table 11.4 lists the different standards and a summary of the results, including the number of failures and the percentage of failures. There is a total of 568 assays for the standard material with only 11 failures, which are samples that are outside the ± 3 standard deviations. This is a failure rate of approximately 2%, which is acceptable. The failure rates for the individual standards is shown in Table 11.4 and they range from 0.0% to 11.4%. The failure rate for standard G907-2 is high, but there are only 35 assay results for that standard. The performance of the other standards is acceptable.

Table 11.4: Summary of Standards for 2011-2012 Drilling Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2011-2012	Accurassay	FAAU	Geostats	G907-2	0.890	0.060	35	0.944	4	11.4%
	Accurassay	FAAU	Geostats	G302-6	0.990	0.050	50	1.023	0	0.0%
	Accurassay	FAAU	RockLabs	SH55	1.375	0.014	42	1.314	2	4.8%
	Accurassay	FAAU	RockLabs	SJ53	2.637	0.016	42	2.548	0	0.0%
	Accurassay	FAAU	Geostats	G301-10	5.570	0.210	85	5.591	3	3.5%
	Accurassay	FAAU	RockLabs	SL46	5.867	0.066	60	5.584	2	3.3%
	Accurassay	FAAU	Geostats	G308-5	13.300	0.56	30	13.417	0	0.0%
	Accurassay	FAAU	Geostats	G904-3	13.660	0.620	52	13.491	0	0.0%
	Accurassay	FAAU	RockLabs	OxP76	14.980	0.080	56	14.554	0	0.0%
	Accurassay	FAAU	RockLabs	SP37	18.140	0.15	58	16.799	0	0.0%
	Accurassay	FAAU	RockLabs	SP49	18.340	0.12	58	16.799	0	0.0%

The field duplicate and coarse duplicate results are summarised in Table 11.5. As this program was carried out in the latter part of the 2011 drilling program, there are a limited number of results. The failure rates of 13.5% for the field duplicates and 15.8% for the coarse duplicates, as shown in Table 11.5, are higher than is typical for this style of gold mineralisation. However, these high failure rates may be due to the limited number of assay results used for this statistical analysis.

Table 11.5: Summary of Duplicate Results for the 2011-2012 Drilling Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2011	Accurassay	FAAU	Field Dup.	37	0.930	3.155	0.992	30%	5	13.5%
2011	Accurassay	FAAU	Field Dup.	38	0.497	0.519	0.773	20%	6	15.8%

Considering the good results observed for the blanks and standards, and considering the poorer results from the duplicates, on the balance of probabilities, it appears that Accurassay, which assayed the samples for the 2011 and 2012 sampling campaigns, has produced sufficiently accurate and precise results such that these results can be considered reliable.

11.2.4.3 Tamaka, 2013-2014

The 2013-2014 QA/QC program consisted of the insertion of CRMs, blanks, field duplicates, and coarse duplicates into the sample stream at specified intervals. QA/QC samples were inserted every 30th sample such that for each group of 30 samples there was one of three standards: one blank, one field duplicate, and one coarse duplicate. This gives an overall insertion rate for the QA/QC samples of approximately 12%, which is believed to be sufficient to determine the reliability of the assay results.

The blank sample material was obtained from the Nelson granite quarry near Vermillion Bay, in Northwestern Ontario. There were 238 assays of blank material in the QA/QC data files with no failures, which are blanks that assayed more than the upper control limit of 0.010 g/t Au. Accurassay's results for the blank samples are considered as good.

The CRMs or "standards" were obtained from Geostats Pty Ltd. Three different standards were used during the 2013-2014 QA/QC program. Table 11.6 lists the different standards and provides a summary of the results, including the number of failures and the percentage of failures. There is a total of 274 assays for the standard material with only 11 failures, which are samples that are outside the ±3 standard deviation acceptance criteria. This is a failure rate of approximately 4%, which is acceptable. The failure rates for the individual standards, shown in Table 11.6, range from 3.4% to 5.2%. The performance of the standards for the Accurassay laboratory results is considered as acceptable.

The field duplicate and coarse duplicate results are summarised in Table 11.7. The results for the coarse duplicates are good, with a failure rate of 8 out of 268 (3%). The average grades are also similar, and the linear correlation is strong at 0.96. The results for the field duplicates are acceptable, with a failure rate of 17 out of 268, or 5.6%. This higher failure rate for the field duplicates is likely due to the nature of the "nuggety" gold mineralisation at Goldlund.

Table 11.6: Summary of Standards for 2013-2014 Drilling Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Average Assay Au (g/t)	No. of Failures	% Failures
2013-2014	Accurassay	FAAU	Geostats	G907-2	0.890	0.060	89	0.905	3	3.4%
	Accurassay	FAAU	Geostats	G301-10	5.570	0.210	89	5.418	3	3.4%
	Accurassay	FAAU	Geostats	G308-5	13.300	0.560	96	13.170	5	5.2%

Table 11.7: Summary of Duplicate Results for the 2013-2014 Drilling Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2013-2014	Accurassay	FAAU	Field Dups	303	0.037	0.072	0.503	30%	17	5.6%
	Accurassay	FAAU	Coarse Dups	268	0.060	0.059	0.962	20%	8	3.0%

The results of the statistical analysis of the 2013-2014 QA/QC samples confirms that the Accurassay laboratory was producing sufficiently accurate and precise results such that these assays can be considered as reliable.

11.2.4.4 First Mining, 2017-2018

The First Mining 2017-2018 QA/QC program consisted of the insertion of CRMs or “standards”, blanks, field duplicate samples and coarse duplicate samples at specified intervals. Blanks and standards were inserted at a rate of one standard for every 20 samples (5% of the total), and one blank for every 30 samples (3% of the total). Field duplicates from quartered core, as well as coarse duplicates taken from 1 kg crushed rejects, were also inserted at regular intervals with an insertion rate of 4% for field duplicates, 4% for coarse duplicates and 4% for pulp duplicates. As well, selected samples were sent to Activation Laboratories (ActLabs) in Thunder Bay and Ancaster, Ontario, for independent umpire check assay.

In addition to the QA/QC program implemented by First Mining, the SGS laboratories each operate their own internal QA/QC protocols, inserting quality control materials, blanks, laboratory replicates and laboratory duplicates for each analytical batch. Blank samples of barren “garden rock” purchased from a local hardware store were used. An upper control limit of 0.020 g/t Au was used to determine if there was a blank failure, indicating potential contamination between samples. Any assays above this threshold were reviewed on a case-by-case basis to determine if any corrective action was required at that laboratory.

As a general rule, for samples of granodiorite being assayed at the SGS laboratory in Vancouver, BC, if a single blank or standard was deemed to have failed, that QA/QC sample plus five samples either side in the same batch were sent for re-analysis. If a blank/standard plus one or more consecutive standards were deemed to have failed, then the failed samples plus ten samples to either side and all the samples in between were sent for re-analysis. For samples of non-granodiorite material, which were sent for fire assay at the SGS Red Lake, Ontario laboratory, if only a single standard failed within a batch where the other standards or blanks passed, the entire batch was deemed to have passed and no corrective action was taken.

A total of 600 blanks were submitted for assay for the 2017-2018 program. Two blanks from the SGS Vancouver, BC, laboratory and three from the SGS Red Lake, Ontario, laboratory exceeded the upper control limit, and a portion of those batches were re-run in accordance with the corrective action protocols detailed above. Table 11.8 shows a summary of results for the blanks from the SGS laboratories. Overall, the SGS laboratories performed well.

Table 11.8: Summary of Assay Results for Blanks for the 2017-2018 Drilling Program

Year	Assay Laboratory	Method-ology	Source	Type	No of Assays	Average Assay Au (g/t)	No. of Failures	% Failures
2017-2018	SGS Red Lake	FAAU	"Garden Rock"	blank	100	0.005	3	3.0%
	SGS Vancouver	BLEG	"Garden Rock"	blank	500	0.006	2	0.4%

There were essentially eight different standards used in the 2017-2019 drilling program and all were supplied by CDN Resource Laboratories Ltd. (CDN) of Langley, BC. While there were four other standards considered, they were used only 1 to 3 times so there are insufficient results for statistical analysis and their results will not be presented here. The range in expected value of the eight standards is 0.968 g/t Au to 9.0 g/t Au. A standard was deemed as a failure if the result fell outside 3 standard deviations from its expected value as defined by the standard's certificate. Any assay results outside this acceptance criteria were reviewed on a case-by-case basis to determine if any corrective action was required.

Table 11.9 presents a summary of the standards that includes the expected value and associated standard deviation, along with the number of assays, the average assay grade, the number of failures and the percentage of failures. For the SGS Red Lake, Ontario laboratory there are 101 assays of standard material and there are no failures. For the SGS Vancouver, BC laboratory there are 698 assays of standard material and there are 18 failures, or a failure rate of 2.6%, which is considered acceptable. The individual standard percentage failure rates for the SGS Vancouver, BC laboratory results ranges from 0% up to 4.4%, which is also considered acceptable.

Table 11.9: Summary of Standards for 2017-2018 Drilling Program

Year	Assay Lab	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2017-2018	SGS Red Lake	FAAU	CRL	GS-1U	0.968	0.086	46	0.986	0	0.00%
		FAAU	CRL	GS-1M	1.070	0.090	40	1.079	0	0.00%
		FAAU	CRL	GS-2S	2.380	0.160	15	2.344	0	0.00%
	SGS Van-couver	BLEG	CRL	GS-1U	0.968	0.086	54	0.961	1	1.85%
		BLEG	CRL	GS-1M	1.070	0.090	159	1.042	7	4.40%
		BLEG	CRL	GS-2P	1.990	0.150	68	1.980	2	2.94%
		BLEG	CRL	GS-2R	2.030	0.140	39	1.975	1	2.56%
		BLEG	CRL	GS-2S	2.380	0.160	24	2.316	0	0.00%
		BLEG	CRL	GS-3P	3.060	0.180	152	2.956	1	0.66%
		BLEG	CRL	GS-5M	3.880	0.380	145	3.893	5	3.45%
		BLEG	CRL	GS-9B	9.020	0.750	57	8.722	1	1.75%

Note: CRL = CDN Resource Labs.

Table 11.10 presents a summary of the duplicate assay results for the 2017-2018 drilling program. Duplicate samples, regardless of whether they were BLEG duplicates, metallic screens, or check duplicates for the umpire laboratory, utilised 1 kg splits from the original 3 kg pulverised sample. The only exception to this in the BLEG QA/QC program were the field duplicates which were done on separately prepared, quarter-core samples.

There are 420 duplicate samples assayed by the SGS Red Lake, Ontario laboratory with 20 failures, or a failure rate of approximately 5%. The field duplicate and the coarse duplicate results have low failure rates, while the pulp re-runs failure rate is higher than would be expected. However, overall, the duplicate results for the SGS Red Lake, Ontario laboratory are considered acceptable.

Table 11.10: Summary of Duplicate Results for the 2017-2018 Drilling Program

Year	Assay Laboratory	Method	Type	No. of Assays	Avg. 1	Avg. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2017-2018	SGS Red Lake	FAAU	Field Dups	125	0.099	0.190	0.782	30%	3	2.4%
		FAAU	Coarse Dups	116	0.080	0.061	0.811	20%	3	2.6%
		FAAU	Re-Run Pulp Dups	179	0.164	0.093	0.241	10%	14	7.8%
	SGS Vancouver	BLEG	Field Dups	647	0.438	0.346	0.835	30%	105	16.2%
		BLEG	Coarse Dups	74	0.215	0.225	0.954	20%	6	8.1%
		BLEG vs. Metallics	Check Assays	294	6.433	6.758	0.992	20%	28	9.5%
		BLEG	Pulp Dups	514	0.335	0.336	0.951	10%	50	9.7%
		BLEG	Re-Run Pulp Dups	234	0.516	0.498	0.997	10%	6	2.6%
	SGS vs. ActLabs	BLEG	Check Assays	326	2.131	1.908	0.987	20%	18	5.5%

The duplicate results for the SGS Vancouver, BC laboratory consist of five different types of samples: field duplicates, coarse duplicates, pulp duplicates, re-run of pulp duplicates and check assays, as shown in Table 11.10.

The SGS Vancouver results for the field duplicates (647) shows a high failure rate, which is an indication of the high "nugget effect" in this style of gold mineralisation. The failure rate for the coarse duplicate samples (74) and pulp duplicate samples (514) are somewhat higher than preferred at 8.1% and 9.7%, respectively. However, these results are still considered acceptable, as the failure rate is less than 10%. The duplicate results for the re-run on the pulps (234 assays) shows a good failure rate of only 2.6%.

The comparison between the BLEG methodology and the screen fire assays or "metallics" assay methodology shows that 28 assays of the 294 were failures, for a failure rate of 9.5%. While this failure rate is higher than preferred, it is still considered acceptable for the comparison of two different methodologies.

The last comparison of duplicate sample results is for the SGS Vancouver versus Activation Laboratories BLEG assays. There are 326 results with 18 failures for a failure rate of 5.5%, which is considered acceptable. There is a bias in the mean of approximately 10%, with the SGS Vancouver, BC assays having a higher average grade of 2.13 g/t Au, compared to the ActLabs average grade of 1.91 g/t Au. This difference is expected given the high nugget effect observed for the Goldlund mineralisation.

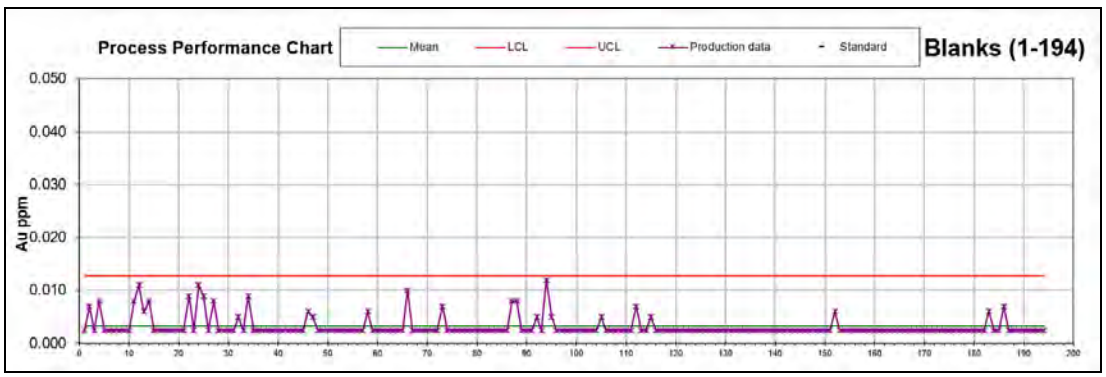
The statistical analysis of the 2017-2018 QA/QC sample assays indicates that both the SGS Vancouver and the SGS Red Lake laboratories are producing results that are sufficiently accurate and precise, such that these results can be considered as reliable.

11.2.4.5 First Mining, 2019-2020

The QA/QC employed by First Mining for the 2019-2020 drilling program to assess the quality of the drilling results consisted of the submission of CRMs (or standards) at an insertion rate of 5%, a sample of blank material at an insertion rate of 3%, a field duplicate from quartered drill core at an insertion rate of 4%, a coarse duplicate taken from a second split of the crushed material at an insertion rate of 4% and pulp duplicates taken from pulverised material with an insertion rate of 4%. In addition to the QA/QC program carried out by First Mining, SGS also uses an internal laboratory QA/QC program consisting of CRMs, blanks, laboratory repeats and laboratory duplicates for each analytical batch.

Blanks are made from barren decorative stone purchased from a local hardware store, "garden rock". Figure 11-2 displays a control chart of the 194 assay results for the blanks inserted into the 2019-2020 sample stream, with an upper control limit of 0.013 g/t Au that is determined as 4 times the average grade of the blanks. There are no failures for the blank samples.

Figure 11-2: Control Chart of Blanks Sample Results for the 2019-2020 Samples



Source: CGK (2020).

There were five different commercial CRMs incorporated into the 2019-2020 drillhole sample program. All five standards were prepared by CDN Resource Laboratories Ltd. of Langley, BC, and range in grade from 0.562 g/t Au up to 9.02 g/t Au. Table 11.11 presents a listing of the five standards, including the expected value and standard deviation at a 95% confidence limit, the number of assays of each of the standards, the average assay by SGS and the number and percentage of failures.

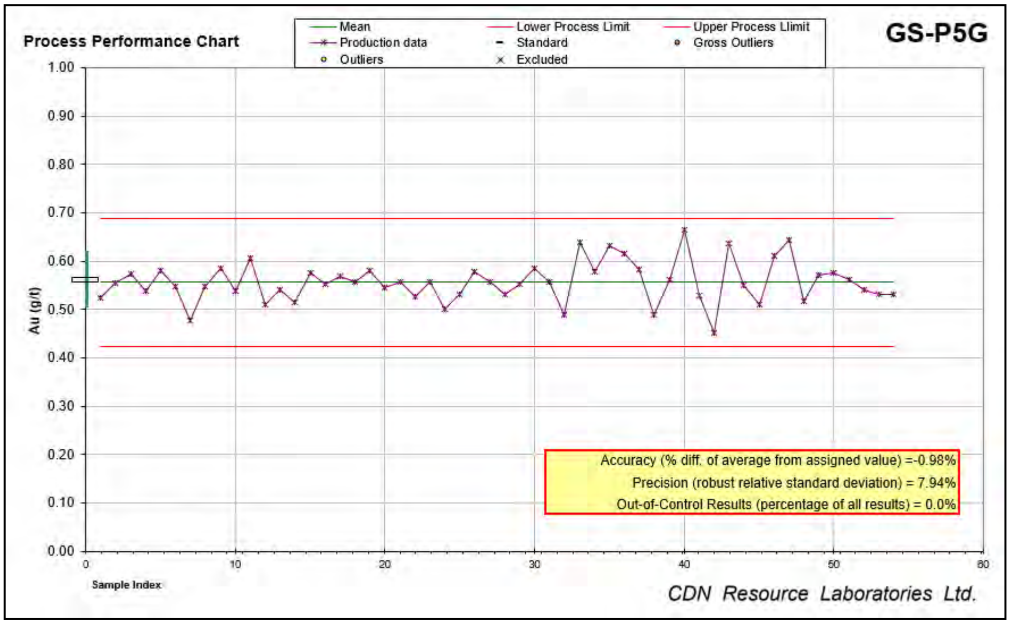
Table 11.11: Summary of Standards for 2019-2020 Drilling Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2019-2020	SGS	FAAU	CRL	GS-1W	1.063	0.076	63	1.055	0	0.00%
	SGS	FAAU	CRL	GS-2U	2.120	0.130	75	2.114	0	0.00%
	SGS	FAAU	CRL	GS-4F	3.830	0.240	62	3.856	2	3.23%
	SGS	FAAU	CRL	GS-9B	9.020	0.750	39	9.033	1	2.56%
	SGS	FAAU	CRL	GS-P5G	0.562	0.054	54	0.557	0	0.00%

Note: CRL = CDN Resource Labs.

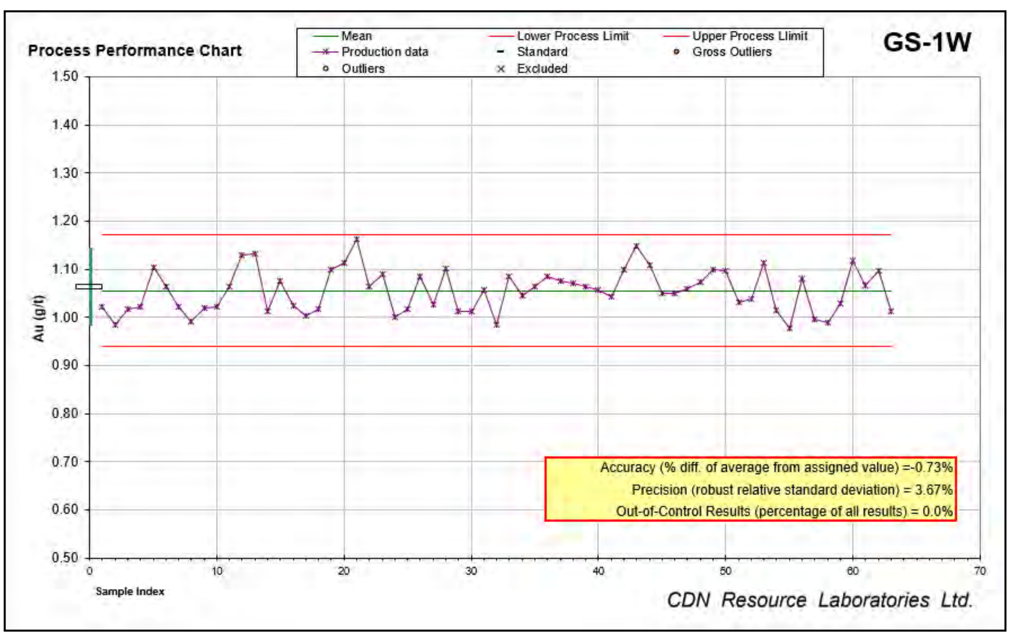
Figures 11-3 to 11-7 display control charts of the five different standards, with the upper and lower process limits (UPL and LPL) shown as ± 3 standard deviations. The results for these standards show that SGS laboratory assays are similar to the expected values for each of the standards. There are two failures observed for GS-4F and one failure observed for GS-9B.

Figure 11-3: Control Chart of GS-P5G Standard Results for the 2019-2020 Samples



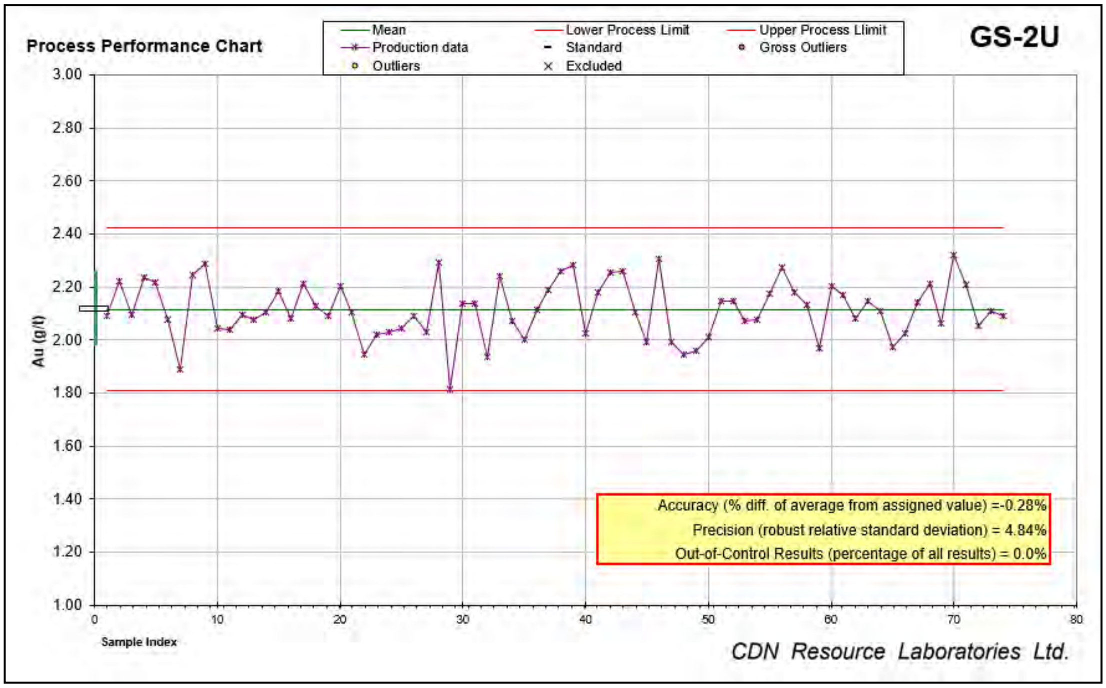
Source: CGK (2020).

Figure 11-4: Control Chart of GS-1W Standard Results for the 2019-2020 Samples



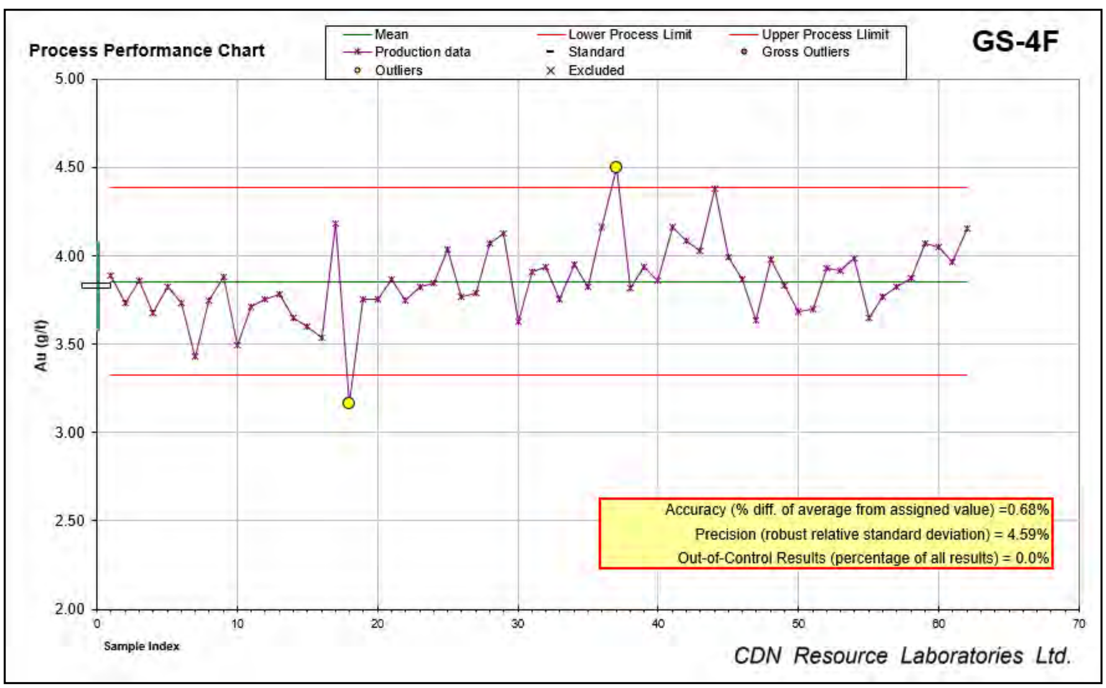
Source: CGK (2020).

Figure 11-5: Control Chart of GS-2U Standard Results for the 2019-2020 Samples



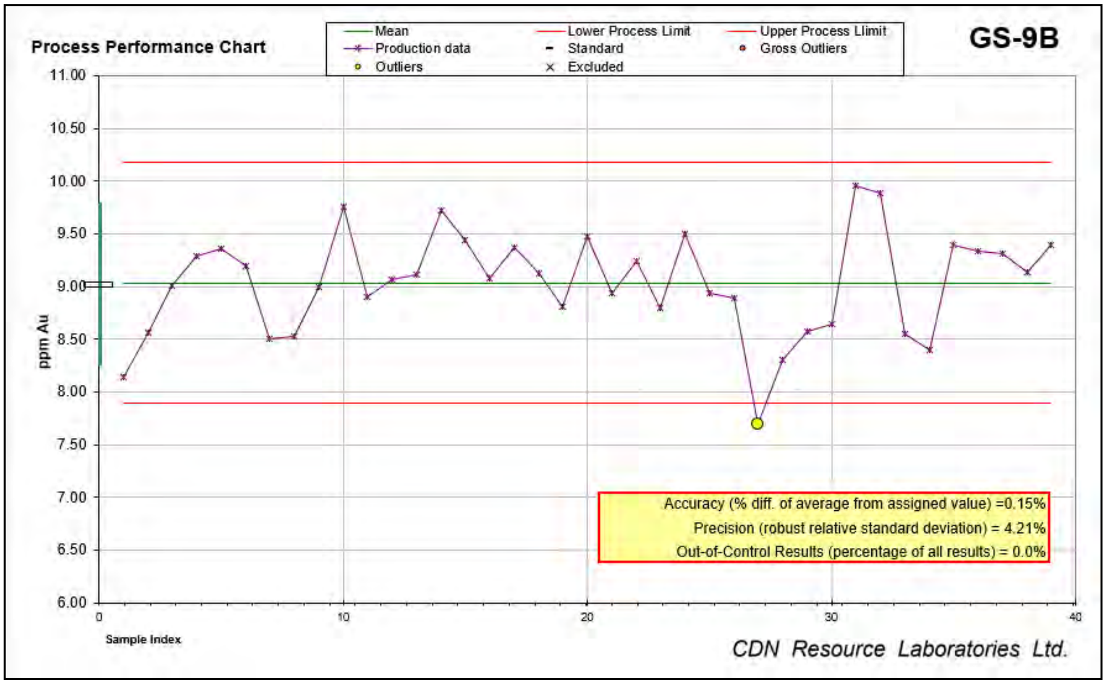
Source: CGK (2020).

Figure 11-6: Control Chart of GS-4F Standard Results for the 2019-2020 Samples



Source: CGK (2020).

Figure 11-7: Control Chart of GS-9B Standard Results for the 2019-2020 Samples



Source: CGK (2020).

The summary results shown in Table 11.11 and in Figures 11-3 to 11-7 show that the SGS laboratories are reproducing the grade of the expected values for each of the standards and are therefore producing reliable assay results.

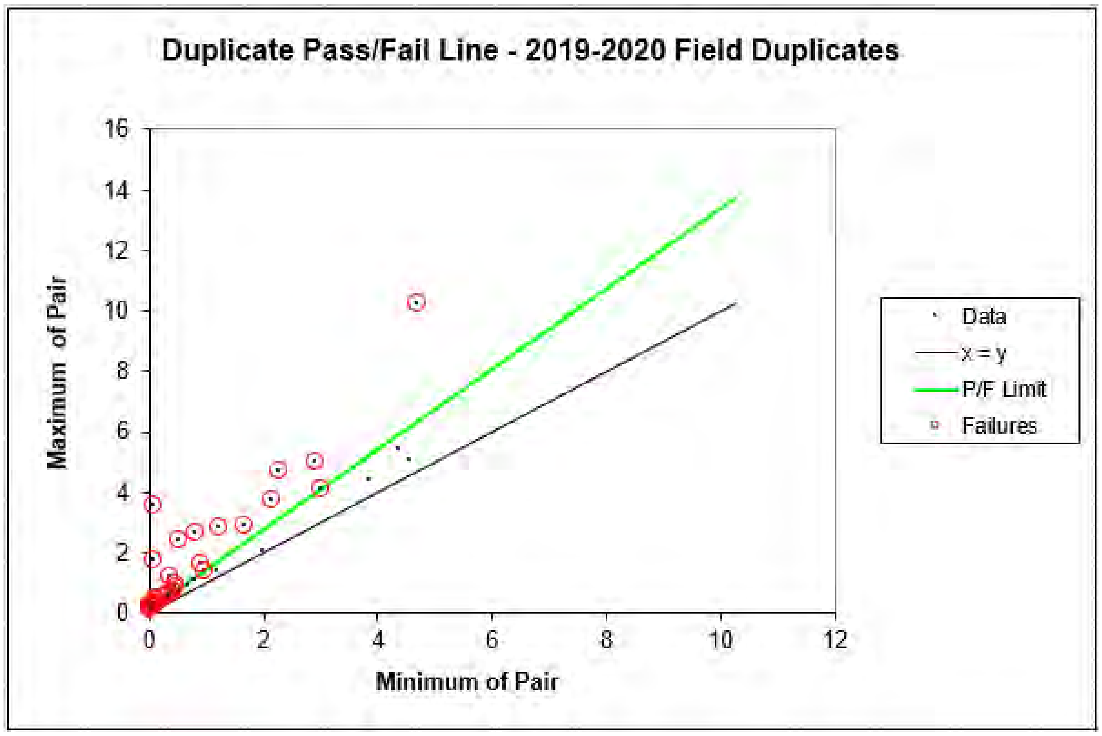
The 2019 and 2020 QA/QC program also included duplicate analysis using field duplicates, coarse duplicates, and pulp duplicates. Table 11.12 presents a summary of the results for the three different duplicate samples for the 2019-2020 Drilling Program. This summary includes the average grade of the original assay (Avg. 1) and duplicate assay (Avg. 2) results, along with the linear correlation coefficient. The pass/fail criteria are ±30% for field duplicates, ±20% for coarse duplicates and ±10% for the pulp duplicates. The number of failures and the percentage of failures is also provided in Table 11.12.

Table 11.12: Summary of Duplicate Results for the 2019-2020 Drilling Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2019-2020	SGS	FAAU	Field Dups	238	0.285	0.288	0.818	30%	30	12.6%
2019-2020	SGS	FAAU	Coarse Dups	149	0.568	0.547	0.984	20%	5	3.4%
2019-2020	SGS	FAAU	Pulp Dups	119	0.573	0.602	0.995	10%	8	6.7%

Figure 11-8 displays a scatter plot with the pass/fail line for the half-core field duplicates. There is a total of 30 failures out of 238 assay results. This is higher than the desired maximum of 10% and is an indication of the amount of variability due, largely, to the “nuggety” gold mineralisation found at Goldlund. The samples with the red circles are those that are considered failures.

Figure 11-8: Scatter Plot with Pass/Fail line for Field Duplicates of the 2019-2020 Samples



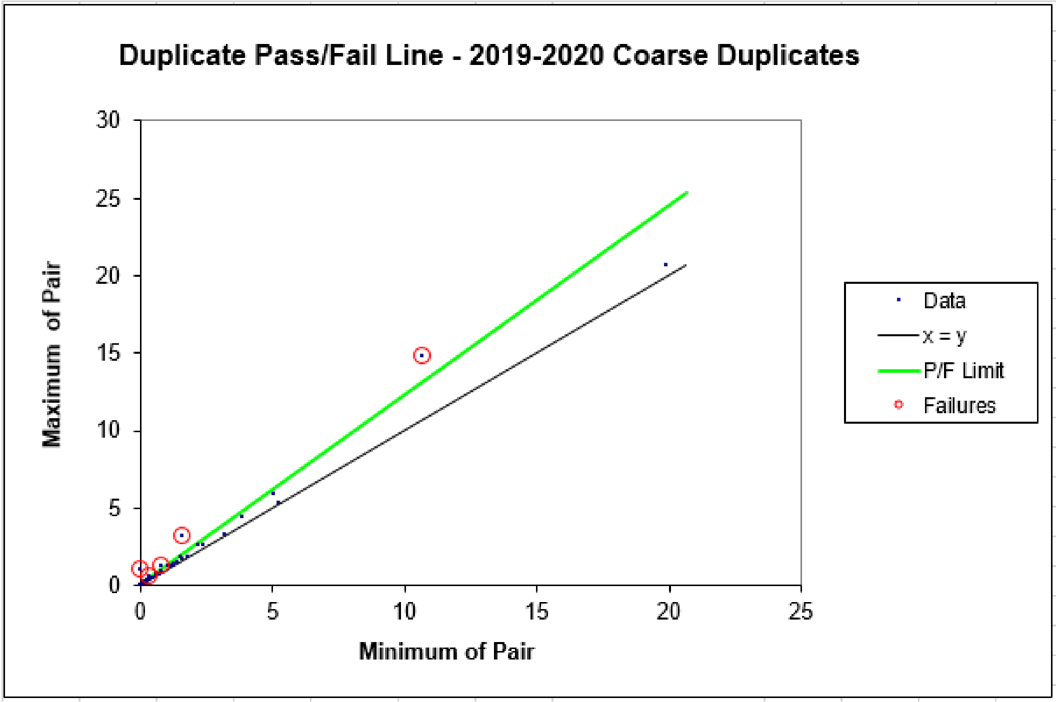
Source: CGK (2020).

Figure 11-9 displays a scatter plot with the pass/fail line for the coarse duplicates. There are only 5 failures out of 149 assay results. This is a failure rate of only 3.4%, which is considered acceptable.

Figure 11-10 displays a scatter plot with the pass/fail line for the pulp duplicates. There are 8 failures out of 119 assay results. This is a failure rate of only 6.7% which is also considered acceptable, because it is less than the 10% failure limit.

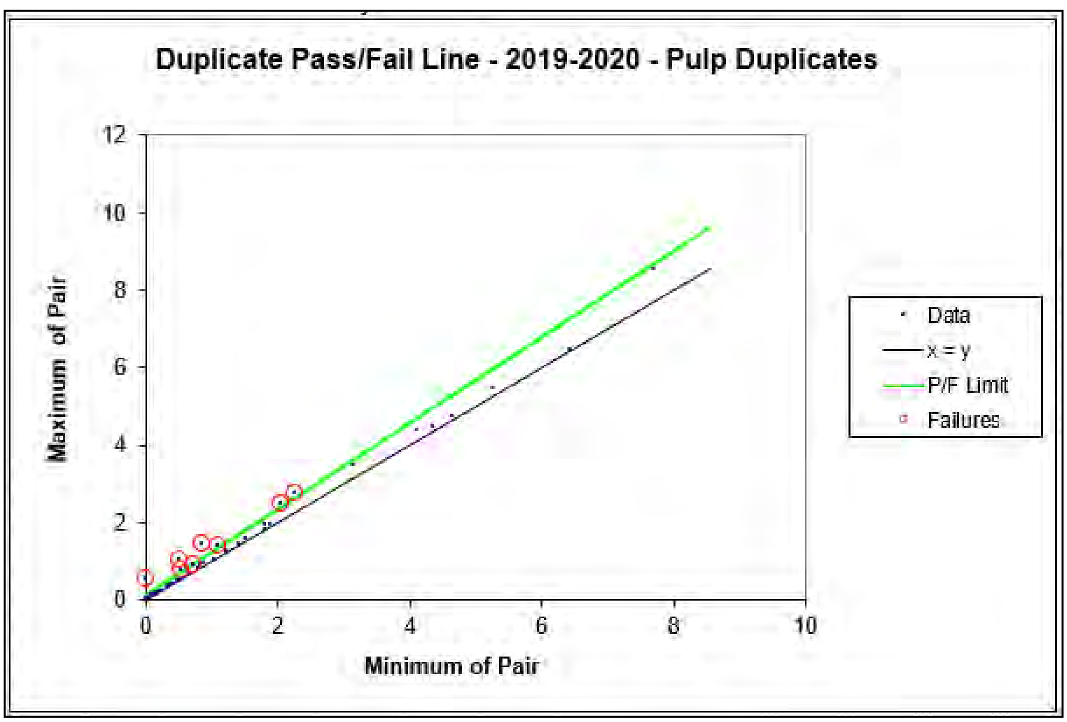
The statistical analysis of the QA/QC sample results shows that the SGS laboratories that assayed the 2019 and 2020 drillhole samples are producing sufficiently accurate and precise results such that the 2019 and 2020 assays can be considered as reliable.

Figure 11-9: Scatter Plot with Pass/Fail line for Coarse Duplicates of the 2019-2020 Samples



Source: CGK (2020).

Figure 11-10: Scatter Plot with Pass/Fail line for Pulp Duplicates of the 2019-2020 Samples



Source: CGK (2020).

11.2.5 Qualified Person Opinion

The Qualified Person for this section of the report believes that the preparation and analyses of the samples are satisfactory for this type of deposit and style of gold mineralisation and that the sample handling and chain of custody, as documented, meet standard industry practice.

The Qualified Person for this section of the report has reviewed the QA/QC program and deems it to be in accordance with standard industry practice and CIM's "Exploration Best Practice Guidelines". Both Tamaka and First Mining personnel have taken reasonable measures to ensure the sample analysis completed is sufficiently accurate and precise such that the assays can be considered as reliable.

Therefore, the Qualified Person for this section of the report, based on the statistical analysis of the QA/QC results, considers that the assay results and database are suitable for use in the estimation of mineral resources.

11.3 Miller Project

Treasury Metals has not conducted any drilling programs on the Miller deposit since it acquired the property. The following information is taken from WSP (2020) for the sample preparation, analysis, and security; density; and the QA/QC on the Miller drilling results.

11.3.1 Sample Preparation

Samples from the 2018 drilling at Miller were shipped to SGS Laboratories in Red Lake, Ontario or Lakefield, Ontario for sample preparation. Samples received by the laboratory were processed as follows:

- dry and crush sample (less than 3 kg) where 75% pass -8 mesh (2 mm)
- split to 250 g
- pulverise to 85% passing -150 mesh (106 µm) for the 2018 drill program; pulverise to 85% passing -200 mesh (75 µm) for the 2019 drill program

11.3.2 Sample Analysis

Samples from the 2018 and 2019 drill programs at the Miller deposit were analysed at the SGS laboratories in Red Lake or Lakefield, Ontario or Burnaby, BC by 50 g fire assay and atomic absorption (AA) finish (SGS Code: GE_FAA515). Additionally, a 51 multi-element analysis (SGS Code: ZMS_ICM14B) was completed on the first eight drillholes, but was discontinued for the remained for the drill program.

Due to the frequent occurrence of visible gold in the drillholes and the coarse, nuggety nature of the gold mineralisation, analyses were followed up on selected samples with a more definitive assay protocol of metallic screen fire assay using a 1,000 g sample size to minimise the high nugget effect (SGS Code: GO_FAS30M).

11.3.3 Density

Density measurements were collected by First Mining on selected drill core samples from all 40 drillholes and all lithologies using the water immersion (wet/dry) method. A total of 97 measurements were collected during the 2018 drill program and an additional 292 measurements were collected during the 2019 drill program. The SG measurements were collected by a hanging a wire cage below the scale (Acculab VIC-612) on the lower hook and the scale was zeroed. Core samples were placed within the cage and the dry weight taken. A bucket of water was raised below the hanging samples until the rock was fully submerged and not touching the bucket, the wet weight was then taken (WSP, 2020).

The wet and dry values were entered into the following formula.

$$Specific\ Gravity = \frac{weight_{dry}}{(weight_{dry} - weight_{wet})}$$

11.3.4 QA/QC, 2018, 2019

The QA/QC program consisted of submitting duplicate samples and inserting CRMs (or standards) at regular intervals. Blanks and CRMs were inserted at a rate of one CRM for every 20 samples, and one blank for every 30 samples. Field duplicates from quartered core, as well as alternating pulp and coarse duplicates (taken from coarse reject materials or pulverised splits) were also inserted at regular intervals, with an insertion rate of 4% for field duplicates, and 4% for pulp and coarse duplicates. Check assays were submitted to a second independent laboratory. Table 11.13 summarises the control samples.

Table 11.13: Summary of Control Samples – Miller Deposit

Description	2018	2019
Total Number of Samples	951	2955
Number of Control Samples	180 (19%)	571 (19%)
Distribution		
Blanks	34 (4%)	116 (4%)
Standards	54 (6%)	158 (5%)
CDN-GS-5M	10	
CDN-GS-9B	3	18
CDN-GS-1U	3	52
CDN-GS-2S	17	
CDN-GS-P4E	11	
CDN-GS-P4G	10	
CDN-GS-1W		1
CDN-GS-2U		38
CDN-GS-4F		26
CDN-GS-P5G		23
Duplicates	92 (10%)	297 (10%)
Field Duplicates	44	141
Coarse Duplicates	22	81
Pulp Duplicates	26	75

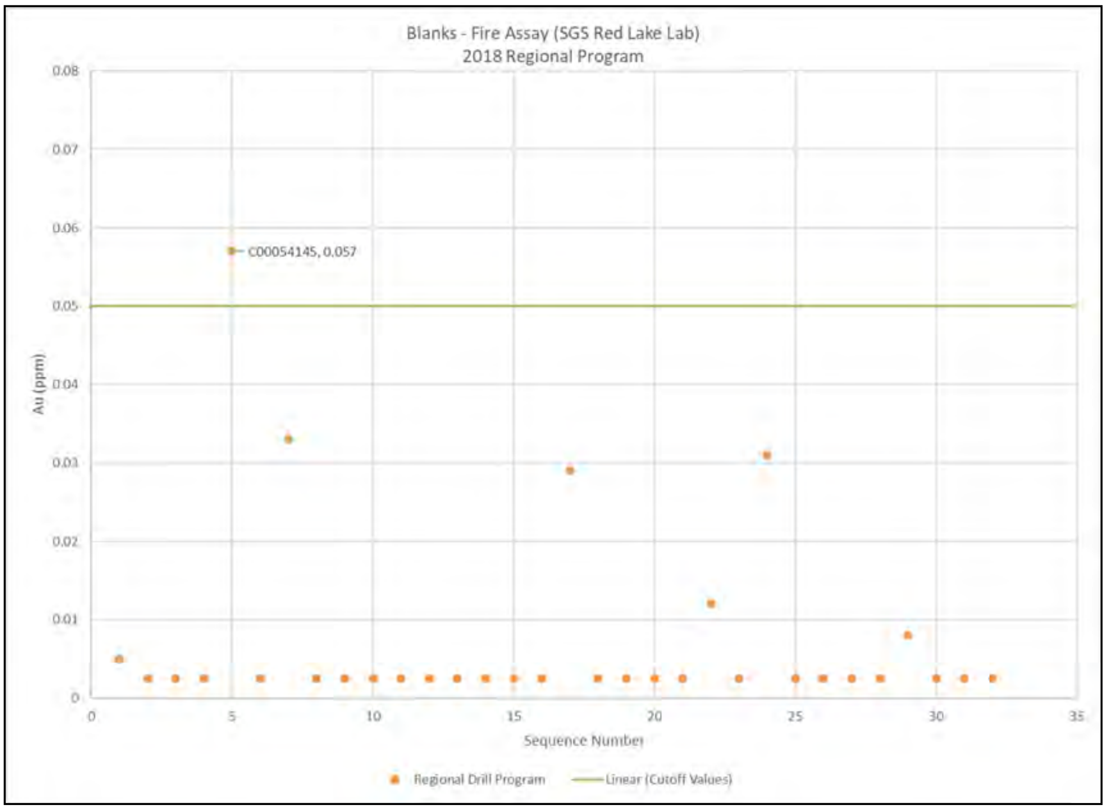
11.3.4.1 Blanks

Coarse blanks for the Miller drill program were taken from barren garden rocks purchased from a local hardware store. A threshold of ten times the lower detection limit (LDL) was used as a guide to determine potential contamination. Any assays above this threshold were reviewed on a case-by-case basis to determine if any corrective action was required at that laboratory.

As a general rule, if a single blank was deemed to have failed, that QA/QC sample plus five samples on either side in the same batch were sent for reanalysis. If a blank/standard plus one or more consecutive standards were deemed to have failed, then the failed samples plus ten samples on either side and all the samples in between were sent for re-analysis.

In 2018, one sample failed the threshold limit but was no action was taken as it occurred within unmineralised host rock (see Figure 11-11). In 2019, no failures were recorded.

Figure 11-11: Control Plot – Blanks, 2018 Drill Program



Source: Treasury Metals (2020).

11.3.4.2 Standards

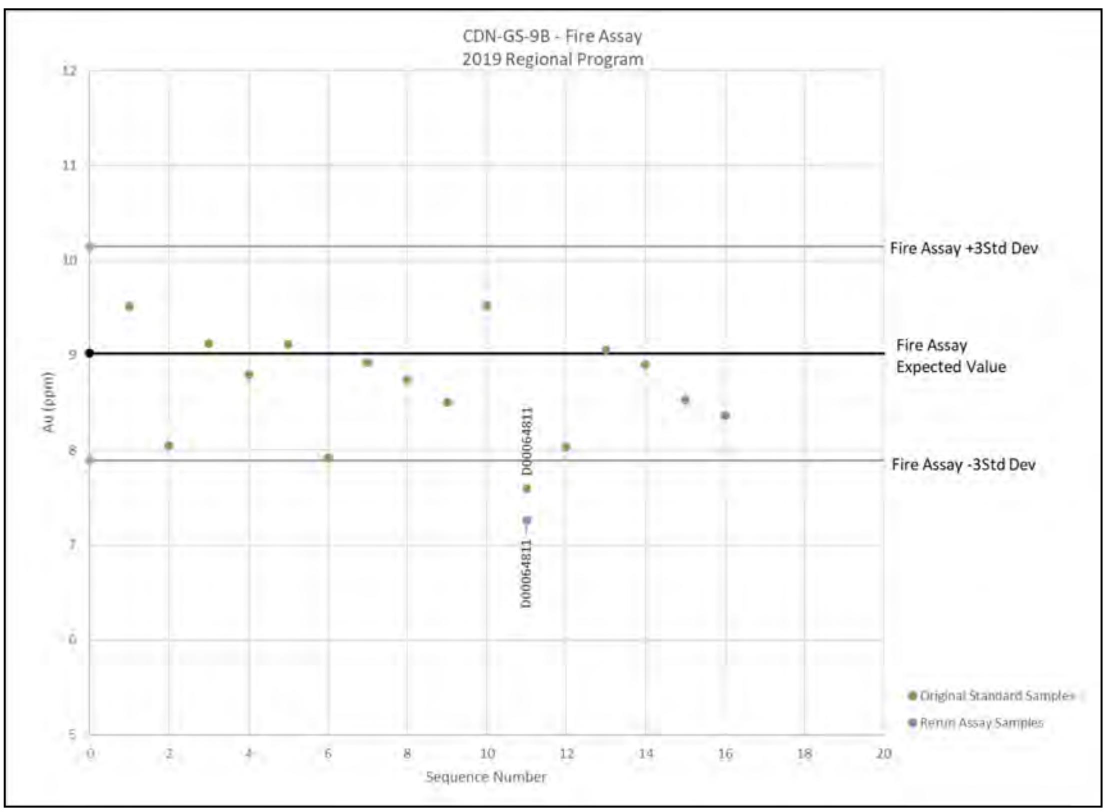
Ten different standards were used for the QA/QC program. The standards were supplied by CDN Resource Laboratories Ltd. of Vancouver, BC. A standard was deemed suspect as a failure if the result fell outside three standard deviations (± 3STDEV) from its expected value as defined by the standard's certificate. Any assays outside of this threshold were reviewed on a case-by-case basis to determine if any corrective action was required. Table 14.14 presents a summary of failures and those resolved by reanalysis or where no further action was taken due to the occurrence within the unmineralised host rock.

Table 11.14: Summary of Standards – Miller Deposit

Standards	2018	Failures	Action	2019	Failures	Action
CDN-GS-5M	10	0				
CDN-GS-9B	3	0		18	2	Reanalysis
CDN-GS-1U	3	0		52	2	No Action
CDN-GS-2S	17	1	No action			
CDN-GS-P4E	11	1	Reanalysis			
CDN-GS-P4G	10	0				
CDN-GS-1W				1	0	
CDN-GS-2U				38	0	
CDN-GS-4F				26	4	Reanalysis
CDN-GS-P5G				23	2	Reanalysis

Figure 11-12 presents, as an example, the control chart for the standard CDN-GS-9B for the 2019 drill results showing the two failures that were sent for reanalysis.

Figure 11-12: Control Plot – Standard CDN-GS-9B, 2019 Drill Program



Source: Treasury Metals (2020).

11.3.4.3 Duplicates

Three types of duplicate samples were used as part of the QA/QC program for the Miller drill programs: field duplicates, coarse and pulp duplicates, and check assay duplicates.

Field duplicate samples were produced by quarter-splitting the core and placing the quartered core into separate sample bags with sequential sample numbers. A field duplicate assay was taken approximately every 30 samples. A total of 185 field duplicates were assayed as part of the Miller QA/QC program. Alternating coarse and pulp duplicates were carried at every 25 samples in the sample stream. An empty sample bag containing the duplicate's sample tag was provided in the rice bag of samples shipped to the laboratory. A total of 103 coarse duplicates and 101 pulp duplicates were assayed as part of the Miller QA/QC program. Only one major departure was found in the duplicates.

The duplicate data shows expected similarities in grades; however, due to the nuggety nature of the gold mineralisation, some samples are difficult to reproduce and often show differences greater than 20% difference between samples.

11.3.5 Qualified Person Opinion

AGP reviewed the sample preparation, analytical and security procedures used by First Mining for the Miller drill core. AGP also reviewed the insertion rates and performance of blanks, CRM and duplicates from the data provided and concluded that the observed failure rates are within the expected ranges and that no significant assay biases are present.

AGP is of the opinion that the QA/QC program employed by First Mining personnel was undertaken in accordance with industry standards and best practices and have taken reasonable measures to ensure that the assays are accurate and may be relied upon.

12 DATA VERIFICATION

12.1 Goliath

The data verification description in Section 12.1 pertains to the procedures implemented by Treasury Metals as observed by the Qualified Person during site visits.

12.1.1 Drillhole Database

Following the site visit, and prior to the resource evaluation, AGP carried out an internal validation of the drillholes databases.

12.1.1.1 Downhole Survey Validation

AGP reviewed the down-hole deviation visually in GEMS and did not find any holes that displayed extreme deviation due to erroneous entry.

12.1.1.2 Assay Validation

Assays in the database were validated using information derived from assay values written in historical drillhole logs and original laboratory assay certificates in Excel and pdf formats.

AGP randomly selected a suite of drill logs and assay certificates. A total of 3,094 assays were validated by AGP, amounting to 29% of the assay database (Table 12.1). Most of the discrepancies noted between the GEMS database and the certificates originated from re-assays and these were mostly all resolved once the correct certificate was located. The small amount of remaining discrepancies (68) are likely related to the same issue, but the original certificate could not be located. AGP considers the true error rate is very low.

Table 12.1: Assay Validation Rate

Year	# of Assays in GEMS Database	Validated	Percent Validated	# Assays within Mineralised Domains	Validated within Mineralised Wireframes	Percent Validated
1990 - 1998	25,421	1,808	7%	7,515	765	10%
2008 - 2020	84,977	6,405	8%	22,518	1,986	9%
Total validated	110,398	8,213	7%	30,033	2,751	9%
Total number of discrepancies found and not resolved (> 0.015 g/t)						68
Total percentage of discrepancies found and not resolved (> 0.015 g/t) vs. validated						0.8%

During the validation process, AGP found that assays and its duplicate for hole TL041 were averaged together and the average value populated the GEMS assay database. The remainder of assays that were validated used a conventional 'first pass the post' approach for the treatment of duplicate values. This created some inconsistencies in the dataset. AGP recognises this is a minor issue and that the total number of samples affected is unknown. AGP recommends Treasury Metals review the data set and revert all assays to a 'first pass the post' approach when time becomes available.

Treasury Metals ran several cyanide bottle roll re-assays. For those assays encountered in the data that was validated, the calculated head grade value populated the GEMS assay database. The head grade value was calculated on the laboratory certificate by adding the gold content of the solid portion plus the gold content in solution, and then dividing that value by the sample mass.

AGP also noted that a significant portion (30.8%) of the gold assays within the mineralised zones are missing a corresponding silver assay. While silver does not contribute significantly to the resource, it is nevertheless carried as an estimated grade element and as such, every effort should be made to ensure the material within the mineralised horizon is fully assayed for both gold and silver if the core rejects or pulps are available.

12.1.2 Twin Drillhole Assessment

Resources for the Goliath deposit are supported mainly by the historical Teck Exploration drilling carried out in the 1990s and newer drilling completed between 2008 and 2020.

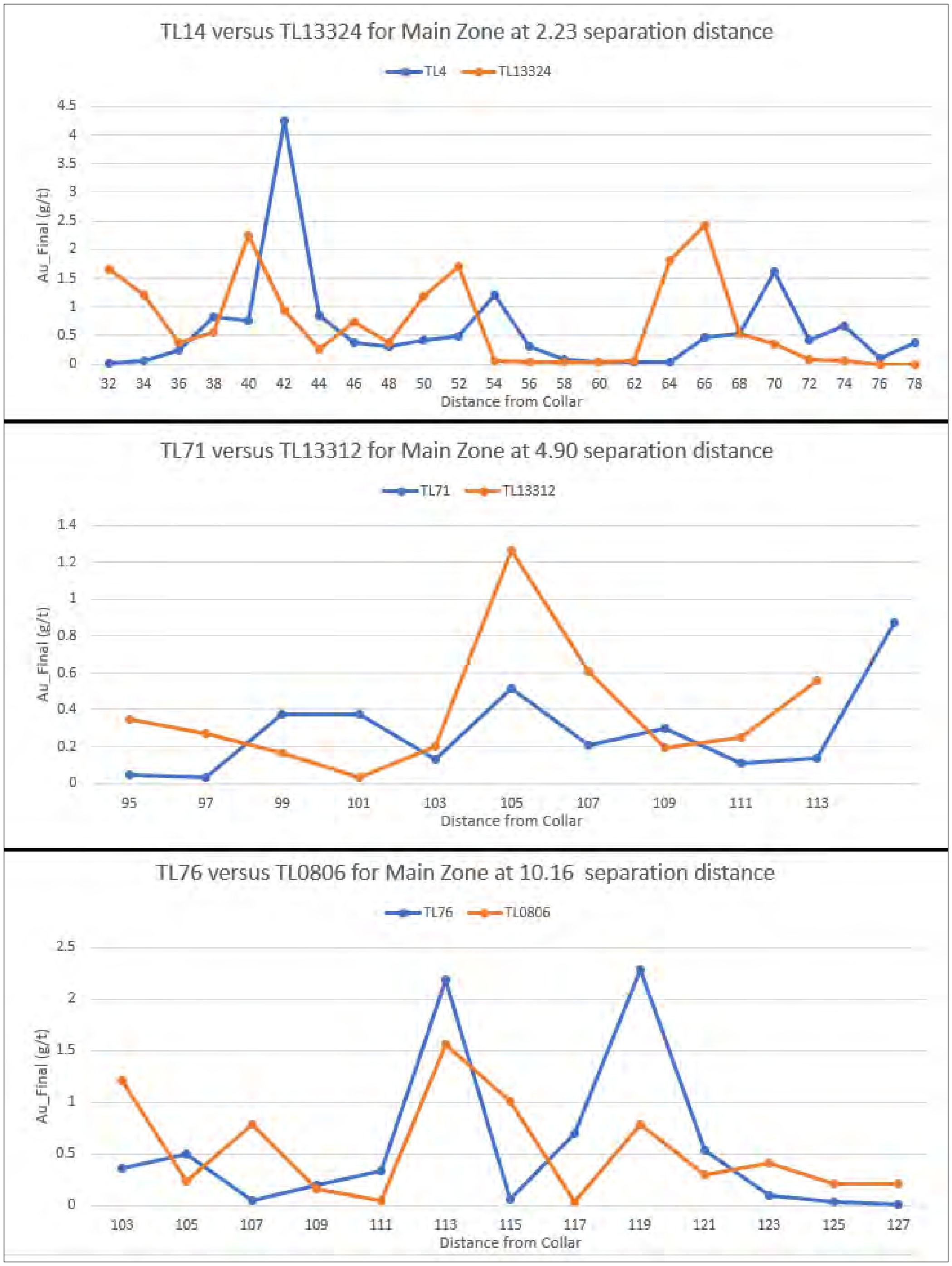
Treasury Metals did not officially conduct a twin drilling campaign; however, numerous newer holes have been drilled as infill to the Teck exploration drillholes since 2008. To validate the Teck assays, AGP randomly selected three drillhole pairs collared close to each other that were drilled with similar azimuth and dip.

To compare the drillholes grade, the raw assays were composited in 2 m intervals from top to bottom and the composites were adjusted so that the hanging wall location of the Main Zone and C Zone match in both drillholes. The composites were then paired for each of the mineralised zones intersected.

12.1.2.1 Main Zone

AGP found the composited assays for the Main Zone compared relatively well between the paired holes. The higher-grade spikes and lower grade sections are generally well reflected in both drillholes (Figure 12-1).

Figure 12-1: Teck Exploration vs. Newer Drilling – Main Zone

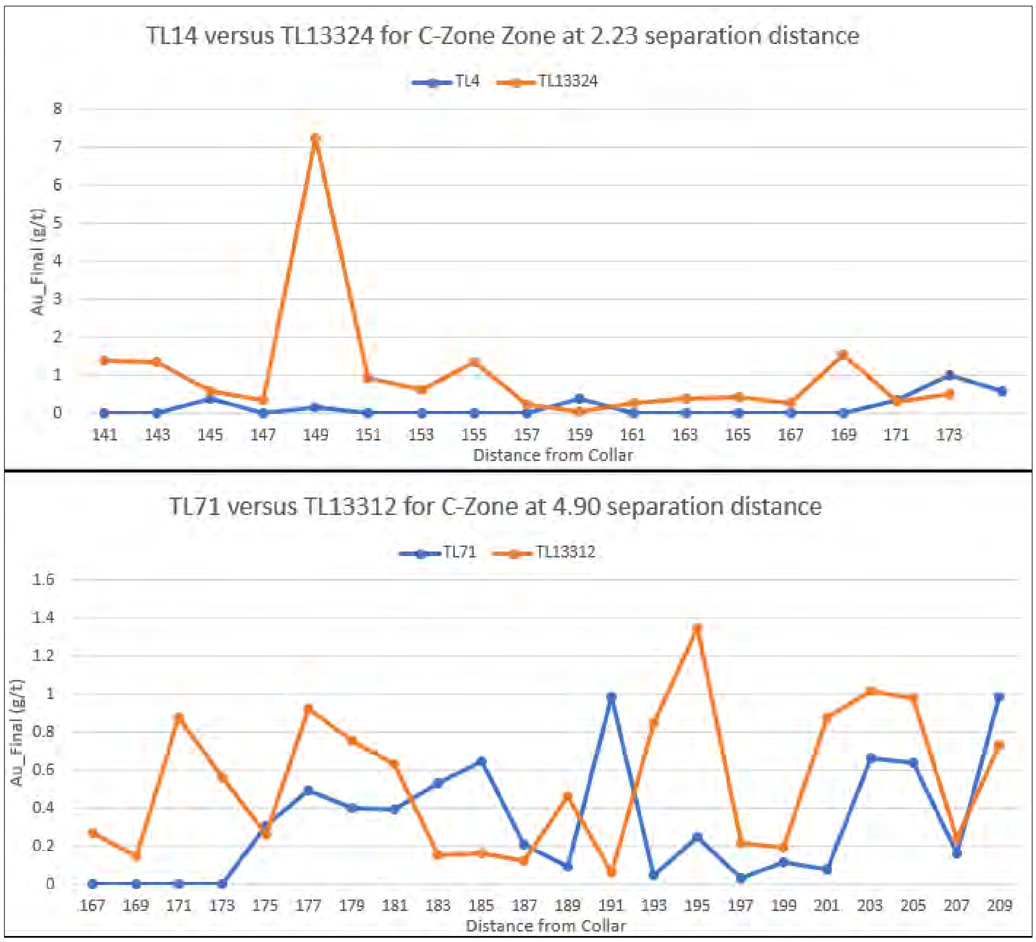


Source: AGP (2020).

12.1.2.2 C Zone

Hole TL76 was not drilled deep enough to reach the C Zone and could not be compared with Hole TL0806. Holes TL4 only showed partial assays for the C Zone and as a result, the comparison with hole TL13324 is not deemed representative. AGP recommends Treasury Metals review all drillholes that can be considered near twin and eliminate the hole from the dataset if the data shows lack of sampling. Holes TL71 and TL13312 were well sampled within the C Zone and the composite grades were found to be comparable (Figure 12-2).

Figure 12-2: Teck Exploration vs. Newer Drilling – C Zone



Source: AGP (2020).

12.1.3 Qualified Person Site Inspection

Mr. Pierre Desautels, P.Geo., visited the Goliath deposit on September 11 through September 12, 2020, and was accompanied by Mr. Adam Larsen, P.Geo., Exploration Manager. No drill program was in progress during the site visit.

The 2020 site visit entailed brief reviews of the following:

- overview of the geology and exploration history of the property

- management of the exploration program on the property
- drillhole collar locations
- description of the drill rig procedures, including core handling
- sample collection protocols at the core logging facility
- discussion of sample transportation, chain of custody, and security
- core recovery
- QA/QC program (insertion of standards, blanks, and duplicates)
- monitoring of the QA/QC program
- review of diamond drill core, core logging sheets and procedures which included commentary on typical lithologies, alteration and mineralisation styles, and contact relationships at the various lithological boundaries
- SG sample collection

Independent characterisation samples were collected during the first site visit. AGP packaged the samples, which were subsequently shipped via FedEx to ActLabs, Ancaster, Ontario. The sample analysis allowed an independent laboratory to confirm the presence of gold and silver in the deposit and assess differences in terms of grade ranges. Samples were analysed for gold and silver with fire assay with gravimetric finish (procedure code 1A3-Ag). One sample was analysed for gold using metallic screen (1A4-1000). The remaining 36 other elements, which include silver, were analysed using total digestion ICP-OES (1F2). The procedure used for the characterisation samples matches the procedure used by Goliath. Core SG was also requested on all samples.

Table 12.2 shows the analytical results of the AGP samples.

Table 12.2: Independent Characterisation Sample Results vs. Goliath Assays

Hole-ID	From	To	Treasury Metal Assays			AGP Check Sample Assays					
			Sample Nb (Treasury)	Au (g/t)	Ag (g/t)	Sample Nb (AGP)	Au-MeT (g/t)	Au-Grav (g/t)	Ag-ICP (ppm)	Ag-Grav (g/t)	Sg (g/cm^3)
TL13306	80.6	81.60	1368225	1.16	0.50	83672		3.02	2.30	< 3	2.73
TL14355	344.0	345.50	199956	0.64	0.50	83673		1.38	1.20	< 3	2.71
TL11182	268.8	269.83	1005641	0.33	4.00	83674		0.34	2.70	< 3	2.74
TL17430	156.0	156.97	272775	0.71		83675		0.60	12.70	12.0	2.75
TL16403B	521.9	523.00	153181	0.14		83676		0.20	1.20	< 3	2.73
TL18494	426.0	427.00	476921	111.00	11.10	83677	1040.00		2.00	3.0	2.75

Assay results on the AGP check samples also revealed five other elements with elevated values, as indicated in Table 12.3.

The independent check samples collected by AGP prove the presence of the metal of interest at the Goliath deposit and the values obtained by the independent laboratory correlate well with the analytical results from Goliath. AGP notes that the higher grade samples show more variability due to nugget effect and one sample (272775), not assayed for silver, returned a value of 12.70 g/t Ag in the check sample.

Table 12.3: Elements with Elevated Values

Hole-ID	From	To	Treasury	AGP Check Sample Assays					
			Sample Nb (Treasury)	Sample Nb (AGP)	Cu-ICP (ppm)	Fe-ICP (%)	Pb-ICP (ppm)	S-ICP (%)	Zn-ICP (ppm)
TL13306	80.6	81.60	1368225	83672	15.00	1.90	282.00	1.0	117.00
TL14355	344.0	345.50	199956	83673	32.00	1.87	68.00	0.8	122.00
TL11182	268.8	269.83	1005641	83674	41.00	1.06	968.00	0.7	631.00
TL17430	156.0	156.97	272775	83675	31.00	1.73	147.00	1.1	559.00
TL16403B	521.9	523.00	153181	83676	65.00	2.47	26.00	0.9	112.00
TL18494	426.0	427.00	476921	83677	29.00	2.20	88.00	0.7	478.00

Drill rigs owned by Downing Drilling were left on site at the end of the 2019 drill program. All drillholes were accessible by gravel road and four-wheel drive vehicles, with no requirement for helicopter support.

Goliath reported that the drill core is picked up twice daily at the drill rig. Once brought into the core logging facility, the core boxes are opened, the core is laid out on the core logging table and then measured and marked by a technician. Treasury Metals uses a Reflex EZ-SHOT instrument for down-the-hole surveys.

The core is logged in the core logging facility directly into DH Logger software. Items logged are lithology, mineralisation, alteration, structure, texture, and vein intensity. The geotechnical information collected consists of RQD and core recovery. Lithology is logged using a standardised legend. Once logged, the geologist inserts the control samples, consisting of standards, blanks, and duplicates. Treasury Metals uses a crushable blank material (granite) suitable for monitoring cross-contamination during sample preparation. Treasury Metals uses purchased standards from CDN Resource Laboratories. After a review of the control sample assays and charting, AGP found the QA/QC program was well followed.

All core is photographed wet before cutting, using the same camera set-up, which ensures consistency in the photos.

The drillholes inspected by AGP show the core was properly marked. Sampling intervals averaged 1.0 m in the drillhole inspected by AGP.

Bulk density samples measuring approximately 10 cm were submitted to the same analytical laboratory. Treasury Metals does not carry out in-house bulk density measurements. The bulk density from the AGP check samples averaged 2.74 g/cm^3, which corresponds well with Treasury Metals' average density for the Main Zone of 2.75 g/ cm^3.

The core is cut longitudinally with a modified 5 hp electric-powered Husqvarna diamond core saw. The saw is equipped with a custom-made dust and mist extraction system, blade is cooled continuously by fresh water and on suspected high-grade samples, Treasury Metals cleans the diamond blade with an iron rebar. The cuttings are decanted in a three-tank set-up to avoid environmental contamination. This core cutting facility was one of the best inspected by AGP in terms of cleanliness and protection of the workers.

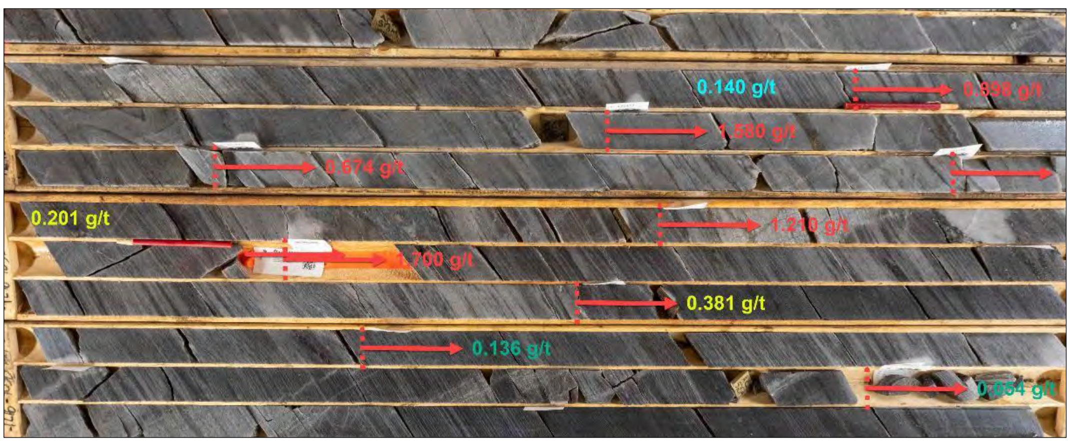
Treasury Metals indicated the samples are bagged in 6 mil plastic bags, sealed with tie wraps, and inserted into polypropylene rice bags for shipping. The samples are transported to the ActLabs facility in Dryden, Ontario.

The Ontario Ministry of Natural Resources and Forestry (MNRF) established a tree nursery facility north of the mineral deposit that was sold to Treasury Metals in 2011. The facility houses the project office core logging facility and core storage. Most of the core is stored in racks either outside or inside the various warehouses built by MNRF and is easily accessible for inspection. Some of the core remains cross-stacked on pallets. Laboratory rejects and pulps are stored indoors. The facility is secured by a gate on the main access road leading to the office. A portion of the old Teck exploration drill core, while available for inspection, can no longer be used for detailed inspection.

AGP inspected selected sections of holes TL13306, TL14355, TL11182, TL17430, TL16403B, and TL18494. High-grade mineralisation typically occurs in altered felsic zones within the bleached white MSS units. The increased presence of sphalerite, galena, and pyrite is a good indicator of high grade. The core was of particularly good quality, with high rock quality designation (RQD).

AGP found that high-grade zones are not easily identified visually in the core inspected, especially in the grade range amenable to open pit extraction. Figure 12-3 illustrates the transition between high-grade and low-grade material in hole TL16403B. Figure 12-4 shows details of the grade transition in hole TL14355.

Figure 12-3: High-Grade & Low-Grade Material in Hole TL16403B from 414 to 524 m



Source: AGP (2020).

Figure 12-4: High-Grade & Low-Grade Material in Hole TL14355 @ 349.35

Source: AGP (2020).

In the field, the new drillholes are clearly marked with an aluminum casing cap engraved with the drillhole name. Older Teck Exploration holes were difficult to find. Hole TL77 could only be identified by the aluminum tag stapled on a wooden stick since no metal drill casing was visible. Figure 12-5 shows a few photographs taken during the site visit by AGP.

Figure 12-5: 2020 Site Visit Photographs by AGP

Hole TL18469 Casing

Outcrop with Channel Sampling Lines

TL77 Teck Exploration Marker

Teck Exploration Underground Portal Vent Tube

Indoor Core Storage

Core Saw

Source: AGP (2020).

Overall, AGP concludes the logging, sampling, sample preparation, security, and chain of custody procedures reviewed during the site visit are to industry standards and adequate to support the resource estimate.

All holes drilled by Treasury Metals were surveyed once completed, using a Trimble Nomad high-precision GPS device.

Collar coordinates were validated by AGP in the field with the aid of a hand-held Garmin GPSMAP 60CSx. Collars were randomly selected from various drill campaigns and their GPS position recorded in UTM WGS84. The differences with the GEMS database were calculated in an X-Y 2D plane using the following formula:

$$X - Y\ difference = \sqrt{(\Delta East)^2 + (\Delta North)^2}$$

As shown in Table 12.4, results indicated an average difference in the X-Y plane of 2.9 m. On the Z plane, an average difference of -7.4 m was recorded. These differences are well within the precision of the Garmin 60CSx used for the validation.

12.1.4 Qualified Person Opinion

The Qualified Person identified no major material issues during the review of the drill data and gold assays. AGP found some minor inconsistencies in the treatment of duplicate assays (in one drillhole) and recommends the dataset be reviewed, and the inconsistencies addressed.

The missing silver assays represent limited risk to the resources and AGP recommends all recoverable drill rejects or pulps for the samples located in the mineralised horizon be assayed for silver. AGP also recommends the near twin drillholes (within 10 m separation) be manually inspected and if one of the holes is poorly sampled, it should be flagged to ensure it is not used in future resource estimations. For the resource model discussed in Section 14.1 of this report, all drillholes intersecting the mineralised zones were used and unsampled gold intervals were assigned a conservative grade of 0 g/t.

The data collected by Treasury Metals adequately represents the style of mineralisation present. The error rate in the drill database, for the data that was validated by the Qualified Person, was found to be very low to non-existent.

Table 12.4: Collar Coordinate Verification

Point-ID	Gems Database Entry			GPS Points Recorded During Site Visit			Differences between GEMS & GPS	
	East	North	El. (+1000)	East	North	El. (+1000)	X-Y Plane (m)	Z Plane (m)
TL10115	527800.81	5511855	1395	527801	5511854	1394	1.02	1
TL11195	528184.54	5511604.8	1394.87	528186	5511608	1392	3.55	2.87
TL15389	528019.4	5511646.2	1391.06	528023	5511647	1397	3.70	-5.94
TL16413	528126.51	5511529.8	1385.44	528122	5511532	1392	5.03	-6.56
TL18469	528225.61	5511655.7	1391.1	528226	5511657	1393	1.37	-1.9
TL18476B	528118.64	5511532.9	1385.93	528118	5511537	1381	4.18	4.93
TL77	527800.01	5511849	1390.25	527801	5511850	1392	1.44	-1.75
Average Difference							2.90	-7.35

12.2 Goldlund

12.2.1 Drillhole Database

Drillhole data is stored in a central Fusion® SQL database and accessed using DH Logger®. The drillhole database for the Goldlund Project contains drillholes, underground channel samples, and surface trench channel samples. The underground channel samples and surface trench channel samples have been incorporated into the database as pseudo drillholes.

In the block model area for Goldlund there are a total of 1,771 drillholes in the July 20, 2020 database (FMG_Goldlund_Drill Database_20th July 2020.accdb) totalling 176,498.3 m of drilling, with a total of 114,102 gold assays. The drilling in the project area spans a period from 1941 to 2020, with drilling carried out by 11 different companies, and assays carried out by five different laboratories. The database was compiled from historical records including plan maps, drill logs, and assay certificates by Tamaka in 2010. Both Tamaka, and later First Mining, have added additional drilling and corrected errors in the database that were provided for this mineral resource estimate.

The First Mining drillhole database has been reported to have gone through several validation efforts, including those carried out by Wardrop (a Tetra Tech Company) in 2010, 2011, 2012, 2013 and 2014; and also by WSP in 2017 and 2018.

The Qualified Person for this section has carried out a series of validation and verification assessments, including a review of the collar elevations, a review of the down-hole surveys for extreme deviations, a team validation of selected assays using signed assay certificates, and statistical comparisons between assays from the different assay laboratories to assess the reliability of the assay data. As well, statistical comparisons were also used to verify the historical assay data with recent assay data that are supported by well documented QA/QC programs.

12.2.1.1 Review of Drill Collar Elevations

To assess the quality of the drillhole collar elevations, the drillhole collars were compared with the high-resolution Bare Earth LiDAR digital terrain model developed from the survey by Airborne Imaging of Calgary, Alberta in 2012. There were 92 drillhole collar elevations out of 856 surface drillholes that were adjusted to be consistent with the digital terrain model. Most of the adjustments in the collar elevations for these 92 holes were less than 4 m. The adjusted drillhole collar elevations, which are consistent with the digital terrain model, were used for the estimation of the mineral resources.

To assess the quality of the location of the underground drillholes and channel samples they were compared to the location of the digitised underground workings in 3D using MineSight® software. There was acceptable agreement between the underground workings and the underground drillhole locations. There was also acceptable agreement between the location of the underground channel samples and the digitised underground workings.

12.2.1.2 Review of Down-hole Survey Data

To assess the reliability of the location of the drillholes, the down-hole survey data was reviewed to examine the drillhole paths for excessive deviation by calculating the deviation between consecutive down-hole survey measurements. There are ten holes out of the 1,771 holes that have a down-hole survey that shows excessive deviation, with seven of the holes located in mineralised Zone 7, and two of the holes located in mineralised Zone 1. One hole was situated in unmineralised material. A review of these holes in 3D using MineSight® software indicates that these results are not considered to be material to the estimation of the mineral resources due to the nature of the interpreted broad mineralised zones, and that for Zone 7, there has been extensive drilling with high-quality down-hole surveys (Reflex EZ Gyro®) that have been used to define the location of the mineralised zone.

12.2.1.3 Validation of Assays

The Goldlund drillhole database has been compiled from historical data including drill logs, assay certificates, drillhole location plans and geological maps. As such, it is important to ensure the historical data is as accurate as possible. Validation of assays with scanned images of the signed assay certificates was carried out for selected drillholes using team checking. The selection of holes was based on those that have the largest gold grade times drilled length (grade * thickness) and a designated assay laboratory. The number of holes required for validation is based on taking 5% of the total number of drillholes inside the block model area.

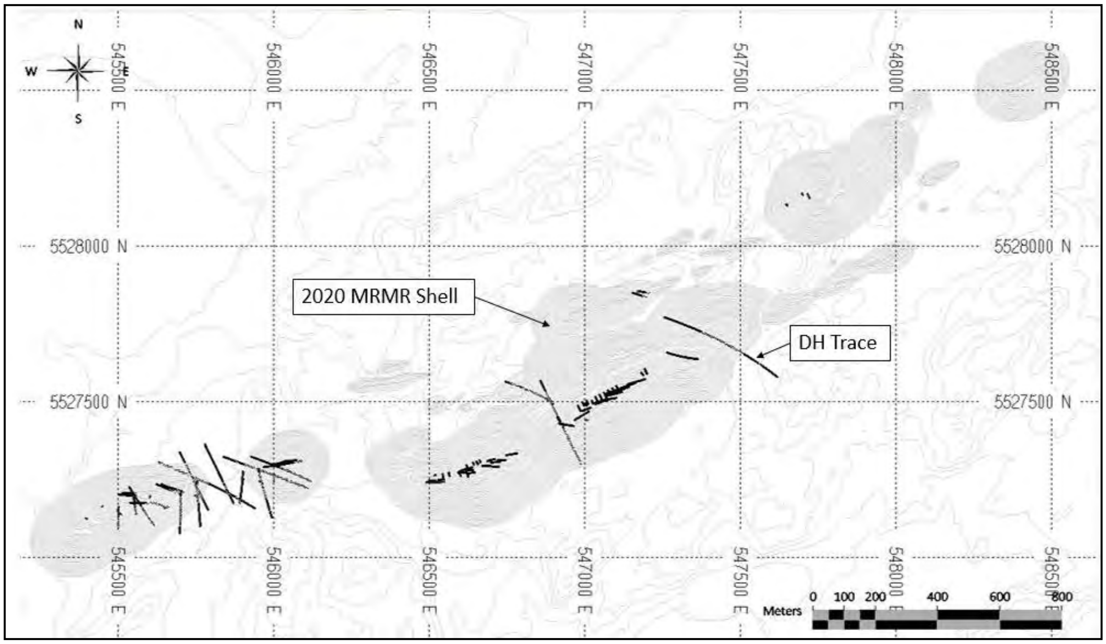
There are 1,717 drillholes inside the block model area that have samples that were assayed. There are an additional 54 holes that appear in the drillhole collar file and have geological information, but do not have any sample intervals. The number of holes selected for validation is 5% of 1,717, or approximately 86 drillholes, which has been rounded up to 90 holes. Figure 12-6 (overleaf) displays a plan view of the holes selected for validation using scanned images of the signed assay certificates.

The drillhole database contains a total of 114,102 assays in the block model area. Five percent of these assays would be 5,705. The selected 90 drillholes contain a total of 9,937 assays, which exceeds the minimum number required for this validation study. Table 12.5 shows a breakdown of the number of holes and number of assays by laboratory.

Table 12.5: Number of Assays in the Selected 90 Holes by Assay Laboratory

Assay Laboratory	No. of Holes	Sum of DH Depth (m)	No. of Au Assays	% of Assays by Laboratory
Accurassay Laboratories	7	2,838.4	2,688	27.1%
Cochenour Fire Assaying	54	3,119.7	3,640	36.6%
Paul's Custom Fire Assaying Ltd.	9	1,181.3	514	5.2%
Randy Farmer - GML	3	666.9	351	3.5%
SGS Laboratories	17	5,828.0	2,744	27.6%
Total	**90**	**13,634.3**	**9,937**	100%

Figure 12-6: Plan View of the Selected 90 Holes for Verification



Source: CGK (2020).

A total of 9,266 assay records were validated using scanned images of the signed assay certificates by team checking. There were 28 instances of differences observed, a difference rate of 0.3%, which is considered acceptable for this database. The results of this validation analysis indicates that the drillhole assays in the database match the assay certificates. Therefore, the assays have been accurately transcribed into the database and the database is suitable to be used for the estimation of mineral resources.

12.2.1.4 Verification of Historical Au Assays using Paired Data

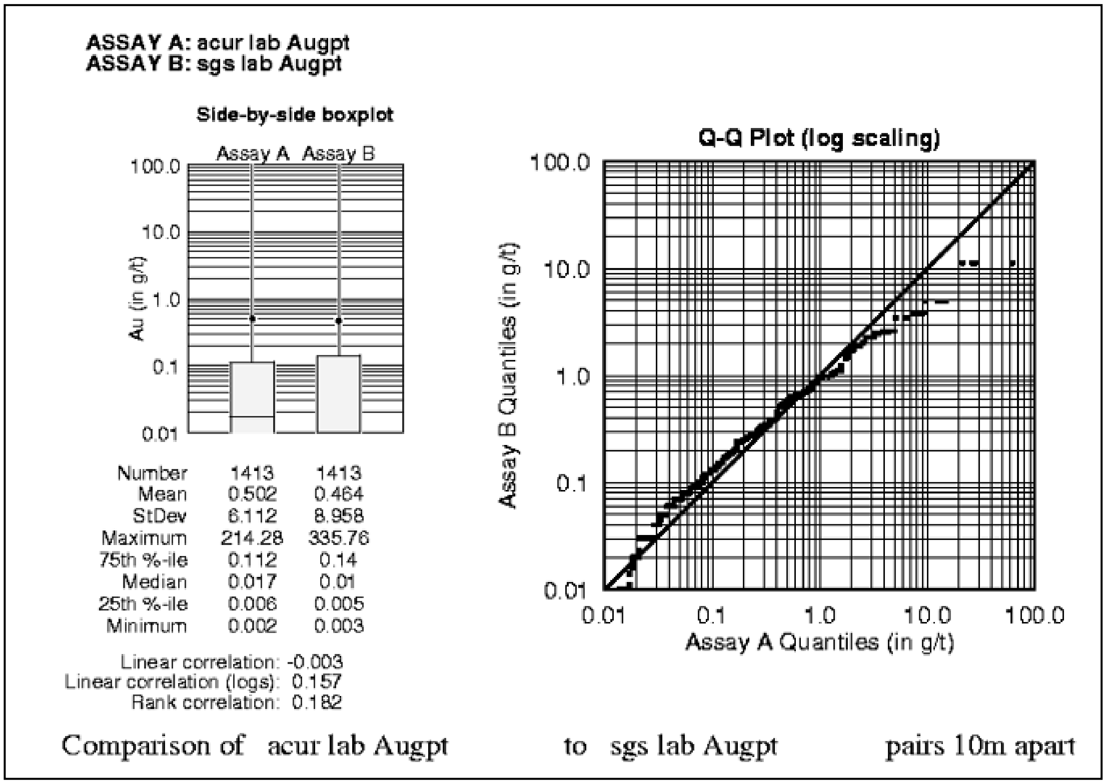
The Goldlund drillhole database contains a total of 1,771 drillholes, including underground and surface channel samples treated as drillholes, of which, 1,232 holes, or approximately 70% of the number of drillholes, have been collected prior to the introduction of the N.I. 43-101 regulations on February 1, 2001. It is therefore important to confirm the reliability of the assay data and determine if the assay data from different sources are sufficiently similar that they can be pooled together for the estimation of mineral resources. It is also important to verify that the historical data is sufficiently accurate so that it can be used for the estimation of mineral resources. A total of 539 drill holes have been drilled by Tamaka and First Mining that can be used to verify the historical drillhole data. The assessment of reliability and the verification of the historical assays was carried out by making comparisons of the sample assays between the different assay laboratories and drilling campaigns that have contributed assay data to the drillhole database. This verification analysis was done by finding pairs of samples at a specified distance for each of the different assay laboratories and drilling campaigns and assessing their similarity using statistical and graphical summaries. This approach is like the verification of historical assay data by twinning drillholes, except that it uses assay pairs from multiple drillholes.

If the metres of drilling are considered rather than the number of drillholes, the 2007 to 2020 drilling totals 101,640 m out of a total of 176,498 m. That is approximately 60% of the drilling, by metres, has been carried out since 2007. The drilling and assaying between 2007 to 2020 has well documented QA/QC programs such that these recent assays can be used to assess the data reliability and verify the historical assay results. The 2007 to 2020 drillhole samples have been assayed for gold by Accurassay and SGS, both are accredited commercial laboratories that are independent of Tamaka, First Mining and Treasury Metals.

The statistical and graphical summaries were examined to determine if the results are sufficiently similar such that they could be pooled together and used for mineral resources estimation. As these are sample assays within a specified distance tolerance, the graphical and statistical summaries can show a wide variation due to the nature of the “nuggety” gold mineralisation. This potential wide variation has been considered in developing the acceptance criteria.

Figure 12-7 displays an example plot of a comparison between assays from Accurassay and SGS for pairs of sample assays up to 10 m apart. There is a side-by-side boxplot, with summary statistics below the boxplots, on the left-hand side in Figure 12-7 as well as a quantile-quantile plot on the right-hand side of the figure. The aim is for the assays from the two different laboratories to have similar average grades and a similar distribution of the grades. In this example, there is reasonable agreement between the assay pairs. The average grade for the Accurassay gold assays is 0.502 g/t Au and the average grade for the SGS gold assays is 0.464 g/t Au. The line of points on the quantile-quantile plot falls approximately along the 1:1 line, indicating a similar distribution. Therefore, the assays from Accurassay and SGS are considered to be sufficiently similar that they can be pooled together for the estimation of mineral resources.

Figure 12-7: Comparison of Assay Pairs for Accurassay & SGS up to 10 m Apart

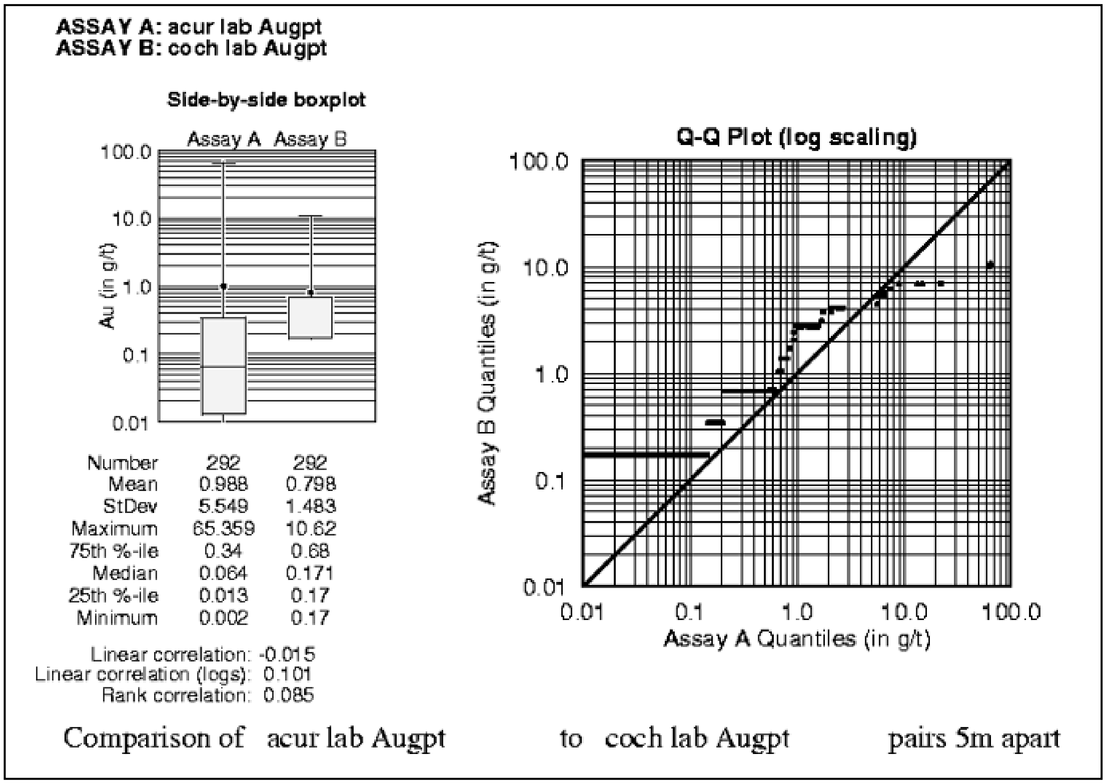


Source: CGK (2020).

Figure 12-8 displays a comparison between Accurassay gold assays and the Cochenour gold assays from the historical drilling for distances up to 5 m apart. The average gold grades and the overall distribution are similar. Figure 12-9 displays a comparison between the Cochenour gold assays from the historical drilling and Paul's Custom assays also from the historical drilling. Again, the average grades and overall distribution are considered similar. While there were some comparisons that did show a bias between certain laboratories, it is believed this bias is due to a trend in the gold mineralisation and does not decrease the reliability of the gold assays.

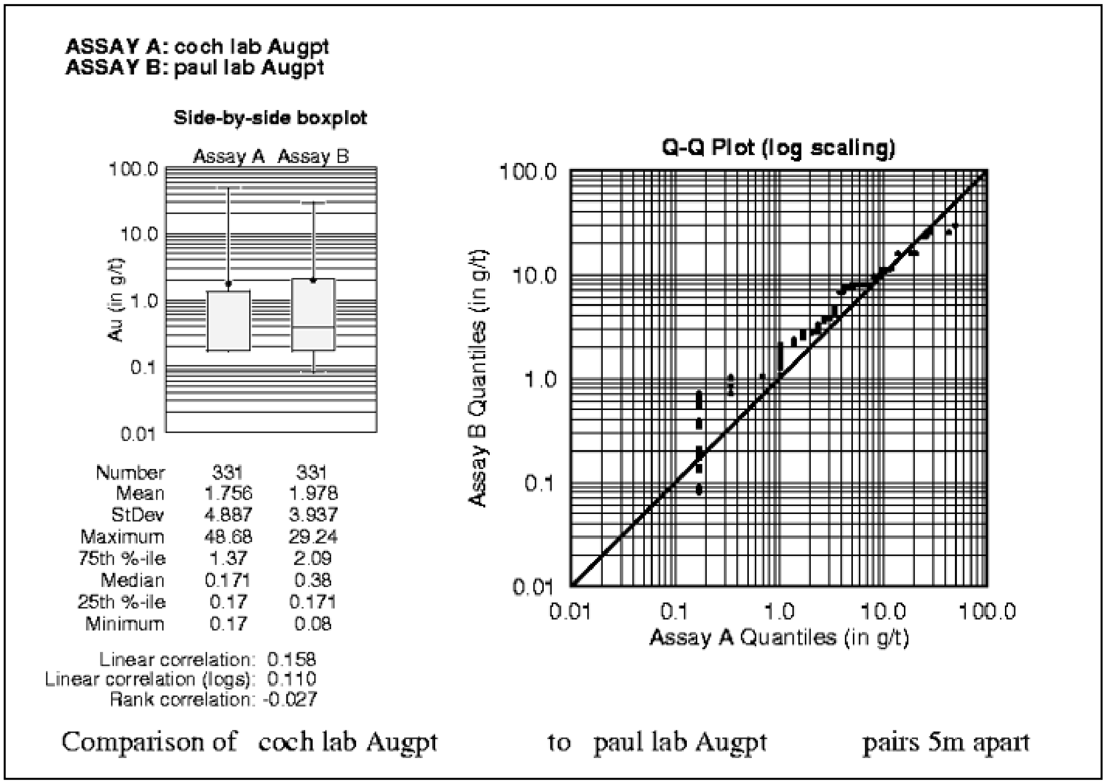
After reviewing all the various comparisons carried out between the five different laboratories, the different drilling campaigns and the three different distance criteria of 3, 5, and 10 m, the Qualified Person for this section of the report is of the opinion that the assay results from the various laboratories, and drilling campaigns, considering the natural geological variability that is inherent in an Archean lode-gold deposit, are sufficiently similar and sufficiently accurate that they can be pooled together for the estimation of mineral resources.

Figure 12-8: Comparison of Assay Pairs for Accurassay & Cochenour up to 5 m Apart



Source: CGK (2020).

Figure 12-9: Comparison of Assay Pairs for Cochenour & Paul's Custom Assay up to 5 m Apart



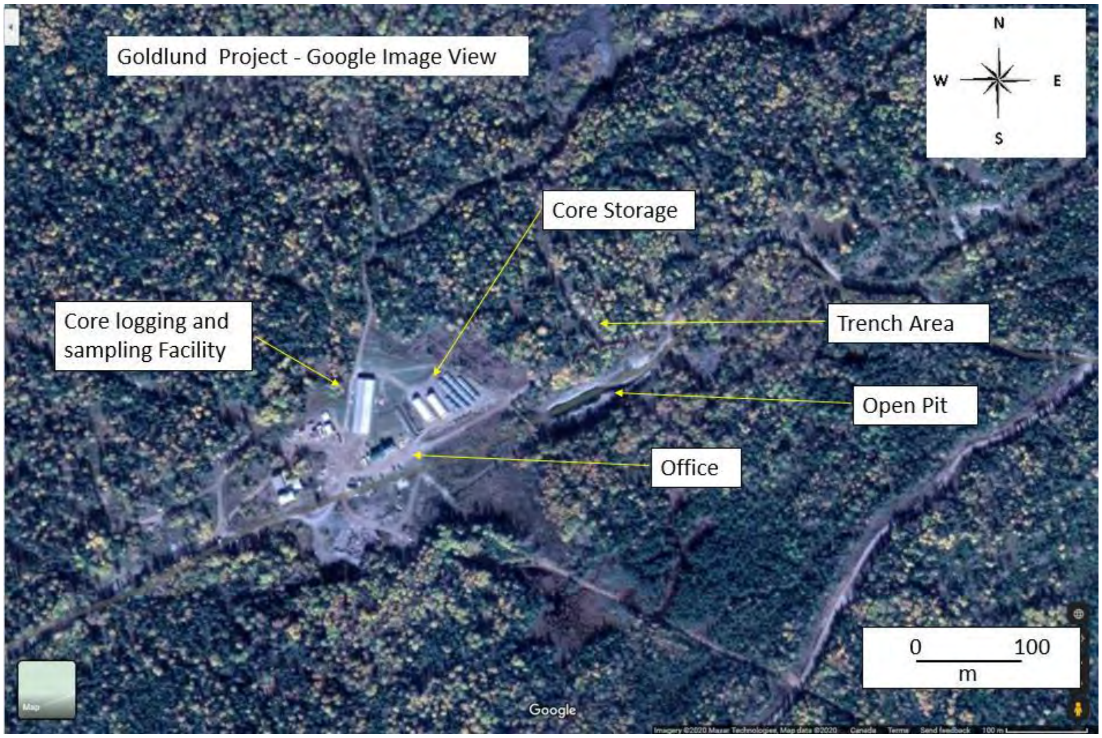
Source: CGK (2020).

12.2.2 Qualified Person Site Inspection

A site visit to the Goldlund Project was conducted by the Qualified Person for this section of the report on October 6 and 7, 2020. There was no active exploration being conducted at the time of the site visit. The Qualified Person was accompanied by Mr. Adam Larsen, Exploration Manager, and Mr. Bryan Wolfe, Senior Project Geologist, both of Treasury Metals. The site visit included an inspection of the surface geology including the historical open pit and trenched areas, the core logging, sampling and core storage facilities, selected drillhole collar locations, and a review of the core logging of selected drill core.

There is an exploration office, a warehouse, and core logging and sampling facilities located on the property, as shown in Figure 12-10. The office facility contains the current and historical data organised in filing cabinets and under drafting tables. The core logging and sampling facility also has storage for sample material returned from the various assay laboratories. The core storage facility includes both covered and uncovered core racks for core storage. Figure 12-11 shows the core storage facilities looking to the west.

Figure 12-10: Aerial View of the Goldlund Project Site



Source: CGK (2020).

Figure 12-11: View of the Core Storage Facility at the Goldlund Project (Looking West)

Source: CGK (2020).

12.2.2.1 Surface Geology

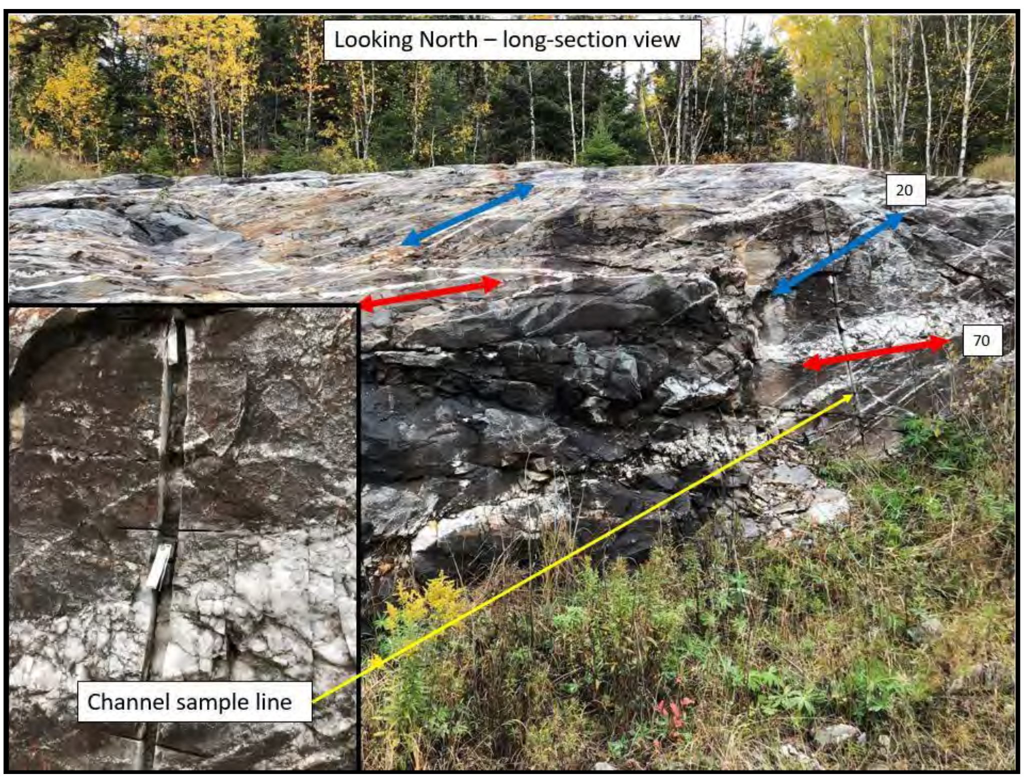
Inspection of the surface geology consisted of examining the surface exposure near the historical open pit and the trenched area 'GDA-12-01', situated to the north of the historical open pit (see Figure 12-10). Figure 12-12 displays a view of the historical pit on the left, and the trenched area on the right. The outcrop exposure adjacent to the historical open pit shows abundant pyrite-bearing quartz veins trending 010° and 070° hosted in granodiorite.

Figure 12-12: View of the Historical Open Pit & the Trench 1 Areas

Source: CGK (2020).

Figure 12-13 displays a north-looking or long-section view of the mineralised quartz veins associated with Zone 1 in the Main Zone. The channel line on the face of the outcrop exposure shows the methodology of sampling the trenched areas using a hand-held diamond saw. The strike, dip, and intersection of the 010° and 070° quartz veins can also be seen.

Figure 12-13: View of the Historical Open Pit Area (Looking North)



Source: CGK (2020).

12.2.2.2 Drillhole Collar Locations

The Qualified Person for this section of the report examined the collars of seven drillholes in the field located in the southwestern area of the 2020 drilling program, which tested the Zone 3 and Zone 4 targets in the Main Zone. Drill collars for Holes G07-033, GL-20-007, GL-20-008, GL-20-009, GL-20-010, GL-20-011, and GL-20-016 were observed. The location of collar GL-20-016 was measured in the field using a hand-held GPS device (Trimble GeoExplorer 6000 Series, Model: 88950) and using the NAD 83 datum, the same datum that is used by First Mining and Treasury Metals for the Goldlund Project.

The drillhole collars are capped by an aluminium screw cap that is engraved with the year, the drillhole number, and the depth of hole. The drillhole is marked with a metal rod topped with orange flagging tape. These holes are well marked in the field in keeping with good industry practice. Treasury Metals independently located and measured 43 drillhole collar locations in the field for the 2019-2020 drilling program. All but one (GL-19-034) were similar to the drillhole collars recorded in the July 20, 2020 database, with an average deviation of less than 4 m. The collar coordinates for Hole GL-19-034 were corrected in the drillhole database used for the estimation of the mineral resources.

Figure 12-14 displays an example of the engraved cap, metal rod, and the GPS coordinates measured by First Mining, Treasury Metals, and CGK.

Figure 12-14: Drill Collar of Hole GL-20-016 showing the Capped Casing & Coordinates

Source: CGK (2020).

The Qualified Person for this section of the report is of the opinion that the drillhole collar coordinates are acceptable, given the accuracy of the handheld GPS that was used to measure the drillhole collar locations.

12.2.2.3 Drill Core Log Review

The site visit included a comparison of the core logging as recorded in the drillhole database to selected drill core intervals for 12 holes that represent the mineralisation for Zones 1, 2, 3, 4, and 7 of the Main Zone. The majority of the selected intervals are situated inside the 2020 MRMR pit shell. The lithology, alteration and mineralisation descriptions, and sample intervals in the drillhole database were consistent with the drill core intervals reviewed. Visible gold was observed in a quartz vein with pyrite in Hole GL-19-008 at 96 m. The gold assay for this interval was 95.3 g/t Au. Table 12.6 presents a listing of the drillhole intervals that were reviewed.

Table 12.6: Selected Drillhole Intervals Examined

Mineralised Zone	Drillhole	From	To	Interval	Comments
Zone 1	GL-17-096	0	60	60	inside pit
	GL-17-117	223	328	105	below pit; example mineralisation
Zone 2	GL-19-008	50	105	55	inside pit
	GL-20-020	80	100	20	inside pit
Zone 3	GL-19-010	56	86	30	inside pit
	GL-20-025	20	60	40	inside pit
Zone 4	GL-20-024	104	131	27	outside pit; example mineralisation
	GL-20-017	75	96	21	outside pit; example mineralisation
Zone 7	GL-17-032	50	115	65	inside pit - west side
	GL-17-073	30	90	60	inside pit - west side
	GL-17-084	50	90	40	inside pit - west side
	GL-17-063	40	72	32	inside pit - east side
# of holes	12			555	

12.2.3 Independent Sample Analysis

The collection of independent samples is meant to demonstrate that mineralisation exists on the property in similar ranges as reported by the issuer. These samples are not intended to act as duplicate samples.

Independent samples were collected by Roscoe Postle Associates (RPA) in 2006. In total, 11 samples of surface material and drill core were collected and sent for analysis to SGS Laboratories in Toronto, Ontario. The samples returned gold grades ranging from 0.07 g/t Au to 28.7 g/t Au.

In 2010, Wardrop (a TetraTech Company), collected 29 independent samples of mineralised split drill core (one-quarter core) and these samples were submitted to Activation Laboratories of Thunder Bay, Ontario. The samples returned gold grades ranging from 0.009 g/t Au to 46.8 g/t Au.

In 2013, WSP collected 30 independent samples of drill core that were sent to Accurassay laboratories in Thunder Bay, Ontario. The samples returned gold grades ranging from <0.005 g/t Au to 3.76 g/t Au.

The Goldlund Project is a historical mining property that has produced a reported 18,000 oz of gold. There has been a total of 70 independent samples collected by previous Qualified Persons, and these samples have confirmed gold mineralisation on the project. The Qualified Person for this section of the report observed visible native gold in drillhole GL-19-008 at 96 m, and that sample assayed 95.3 g/t Au. Therefore, the Qualified Person for this section of the report did not collect any independent samples as there is sufficient evidence that gold mineralisation exists at the Goldlund Project in the ranges reported by the issuer.

12.2.4 Qualified Person Opinion

The Qualified Person is of the opinion that the core descriptions, sampling procedures, and data entries were conducted in accordance with industry standards, and that the data is sufficiently accurate to be reliable. The Qualified Person is also of the opinion that the database is representative and adequate to support the estimation of mineral resources for the Goldlund Project.

12.3 Miller

Treasury Metals has not conducted any drill programs on the Miller deposit since it acquired the property. All drill programs on the Miller deposit were completed by First Mining in 2018 and 2019 targeting a geophysical anomaly.

12.3.1 Drillhole Database

AGP received the data for the Miller deposit as exported CSV files for collar, downhole survey, assay values, and lithology from Datamine DH Logger. The data was formatted and imported into GEMS and verified using the GEMS validation tool to determine whether there were missing and/or overlapping intervals. The drillholes were also checked visually for any misplaced drillhole collars. No errors were found.

The Miller assay values in the database were compared against the assay laboratory certificates provided by Treasury Metals. AGP verified approximately 38% of the Miller assay values (1,471 out of 3,906). No errors were found. It was noted that the laboratory certificate number did not match the assay value in 19 instances. This was either due to a re-run of several samples or an average of several values. This is not considered an error, however for consistency, it is recommended that these laboratory certificate numbers be reviewed and updated.

12.3.2 Qualified Person Site Inspection

The site visit to the Miller project area was conducted by Mr. Paul Daigle, Qualified Person for the Miller deposit from October 13 to 15, 2020. There was no exploration or drilling activity on the project site at the time of the visit. The author was accompanied on the site visit by Mr. Adam Larsen, Exploration Manager for Treasury Metals.

The site visit included an inspection of the project site to review drillhole collars and collar coordinates; an inspection of the core logging, sampling and storage facilities situated at the Goldlund Exploration Camp; and a review of randomly selected Miller drill core compared to the drill log descriptions.

12.3.2.1 Drillhole Collar Locations

AGP located 10 drillhole collars at the Miller deposit. The locations of diamond drillhole collars were measured in the field using a hand-held GPS device (Garmin GPS map 62s) and using the NAD 83 datum, the same datum used by Treasury Metals. Drillhole collars are capped by an aluminium screw cap that is punched with the drillhole number, date, and total depth. The drillhole is marked by a threaded metal rod in the centre of the cap so it can be seen above the level of snow (Figure 12-15).

Figure 12-15: Miller Drillhole Collar & Cap for MI-19-35

Note: Top image: drillhole collar for MI-19-35; bottom shows cap.
Source: AGP (2020).

The collar coordinates measured by AGP fell within a 5 m tolerance of those reported in the drillhole database. It is the Qualified Person's opinion the coordinates are acceptable, given the accuracy of the handheld GPS used to review the drillhole collar locations. Table 12.7 presents the comparison of the AGP and Treasury Metals drillhole database coordinates for verified drillholes.

Table 12.7: Comparison of Collar Coordinates at the Miller Deposit; NAD83 Zone 15U

Drillholes	TML* Easting (m UTM)	TML* Northing (m UTM)	AGP Easting (m UTM)	AGP Northing (m UTM)	Δ Easting (m)	Δ Northing (m)
MI-19-035	554,277	5,533,273	554,277	5,533,273	0	-5
MI-19-036	554,277	5,533,273	554,277	5,533,273	0	-5
MI-19-024	554,319	5,533,298	554,319	5,533,298	1	-1
MI-19-023	554,356	5,533,327	554,356	5,533,327	1	-1
MI-19-020	554,440	5,533,387	554,440	5,533,387	-2	-1
MI-19-018	554,471	5,533,500	554,471	5,533,500	-2	0
MI-19-017	554,500	5,533,516	554,500	5,533,516	-1	0
MI-18-001	554,522	5,533,533	554,522	5,533,533	3	-3
MI-18-002	554,540	5,533,551	554,540	5,533,551	-5	2
MI-19-015	554,551	5,533,566	554,551	5,533,566	-2	-2

Note: *TML: Treasury Metals.

12.3.2.2 Drill Core Log Review

The site visit included a review of the drill logs and a comparison to selected drill core intervals. The lithology descriptions and sample intervals in the drill logs were consistent with the drill core intervals reviewed. Table 12.8 lists the selected drill core intervals examined during the site visit.

Table 12.8: Selected Drill Core & Core Logs Examined

Drillholes	From (m)	To (m)	Core Boxes
MI-18-002	48.23	81.84	18 - 29
MI-18-003	89.91	112.06	28 - 35
MI-19-015	50.20	80.90	12 - 18
MI-19-017	54.56	98.22	13 - 22
MI-19-025	56.00	81.50	11 - 16
MI-19-028	59.30	80.60	13 - 17
MI-19-040	77.4	111.7	19 - 26

12.3.2.3 Drill Core Logging & Sampling & Core Storage Facilities

The Miller drill core was logged and sampled and is stored at the Goldlund Exploration Camp. This was done at the rear of a permanent warehouse building where the front serves as a garage and an exploration office is situated in front of the warehouse (Figures 12-16 to 12-19). Outside the warehouse, drill core is stored in two Quonset-style domed tents and inside outdoor, covered, core racks. The core boxes are not covered and are in good condition.

Figure 12-16: Drill Core Logging & Sampling Facility & Exploration Office

Note: Drill core logging & sampling facility (background) and exploration office (left). Source: AGP (2020).

Figure 12-17: Drill Core Storage Facility, Tents & Core Racks

Source: AGP (2020).

Figure 12-18: Drill Core Logging Tables

Source: AGP (2020).

Figure 12-19: Drill Core Sample Saw

Source: AGP (2020).

The interior the core logging and sampling facility is kept clean and well-maintained. All field and sampling supplies, CRMs, and blanks material are kept orderly and organised on shelves and tables in the facility.

12.3.3 Independent Sample Analysis

The collection of independent samples is meant to demonstrate that mineralisation exists on the property in similar ranges as reported by the issuer. These samples are not intended to act as duplicate samples. AGP collected three samples selected from the available drill core during the site visit. The sample intervals were selected from the 2018 and the 2019 drill programs in the northeast core of the Miller deposit. The samples were collected from the same sample intervals as those of in the database for a direct comparison.

AGP supervised the quartering of the selected samples by rock saw and placed each sample in a marked sample bag, sealed with a zip tie. A sample tag was stapled in the core box at the location of the AGP sample. Collected samples were transported by AGP to Toronto and couriered to ActLabs in Ancaster, Ontario for assay analysis.

Once received at ActLabs, samples were prepared by crushing the sample to 80% passing 10 mesh and then a split of 250 g was pulverised to 85% passing 200 mesh (ActLabs code: RX1).

Gold was analysed separately by fire assay (using a 50 g charge) and atomic absorption and gravimetric methods (ActLabs Code 1A2B-50 and 1A3-50). Samples were analysed for 63 elements by Aqua Regia digestion and ICPMS method (ActLabs code ICP-MS ultratrace1). The list of independent samples is shown in Table 12.9 and the comparison of gold results is presented in Table 12.10.

Table 12.9: Summary of Independent Samples – Miller Deposit

AGP Sample No.	TML Sample No.	Drillhole	Core Box	Sample Interval (m)
163962	C00054457	MI-18-003	29	94.5 – 95.0
163963	C00063048	MI-19-015	14	62.0 – 63.0
163964	C00065462	MI-19-040	22	92.0 – 93.0

Table 12.10: Independent Sample Results – Miller Deposit

Company Name	Sample No.	Drillhole	FA-AA Au (g/t)	Gravimetric Au (g/t)
AGP	163962	MI-18-003	>5	48.2
	163963	MI-19-015	0.19	0.20
	163964	MI-19-040	28.00	<0.02
TML	C00054457	MI-18-003	2.150	2.150
	C00063048	MI-19-015	0.111	0.111
	C00065462	MI-19-040	2.696	2.696
Difference		MI-18-003	-	-46.05
		MI-19-015	-0.08	-0.09
		MI-19-040	-25.30	-

The results of the independent samples have demonstrated the presence of mineralisation on the property and have also demonstrated the variability between samples, and analysis methods may have an impact on results. AGP interprets the difference of the gold grades in the independent samples to be due to the degree of variability of the gold mineralisation.

Despite the samples collected by AGP were one-quarter core in lieu of one-half core, the noted differences are significant to warrant a review of the analysis methods of the Miller mineralised drill core.

12.3.4 Qualified Person Opinion

The Qualified Person is of the opinion the core descriptions, sampling procedures, and data entries were conducted in accordance with industry standards.

The database shows a few inconsistencies in the laboratory certificate numbers and the corresponding assay values. The mislabels of certificate numbers are not considered errors; however, the laboratory certificates should be compared to those in the database and updated where necessary.

The Qualified Person is of the opinion the database is representative and adequate to support the resource estimates for the Miller deposit for a preliminary mineral resource estimate. However, based on the difference between the independent assay analyses and the assay values in the database, there is wide variability between analysis methods. It is recommended that a review of the analysis methods on the Miller mineralised drill core be completed prior to a subsequent mineral resource update.

13 MINERAL PROCESSING & METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork programs were conducted on Goliath samples between 2011 and 2020, and on Goldlund samples in 2012 and 2020.

The Goliath and Goldlund testwork programs examined the following:

- head analysis
- mineralogy
- comminution testing
- gravity separation and cyanidation of gravity tailing
- gravity separation and flotation of gravity tailing followed by cyanidation of the flotation concentrate
- cyanide detoxification
- preliminary static solid-liquid separation tests

The Goldlund testwork program examined the following:

- head analysis
- gold deportment
- comminution testing
- gravity separation and flotation of gravity tailing followed by cyanidation of the flotation concentrate
- gravity separation and cyanidation of gravity tailing

In 2020, cyanide leach tests were conducted on two composite samples from existing Goliath and Goldlund samples. A combined composite sample was used for a bulk leach test and cyanide detoxification testing. Flotation testing was conducted on the detoxified tailings to generate a sulphide concentrate and a low sulphide tailing. The resulting tailings solution and solids samples were submitted for environmental testing.

No metallurgical testing has been conducted on the Miller deposit. The geology of the Miller deposit is similar to the Goldlund deposit, so recoveries derived from the Goldlund metallurgical testing were applied to the Miller deposit.

13.2 Source of Testwork Information

The following sources of technical and project information were referenced in developing the process plant design for the preliminary economic assessment:

- 2011 G&T Metallurgical Services Ltd. Pre-Feasibility Metallurgical Testing Goliath Gold Project. KM2906.

- 2012 ALS Metallurgy (formerly G&T Metallurgy), Feasibility Metallurgical Testing, Treasury Metals Incorporated. KM3406.
- 2017 ALS Metallurgy, Metallurgical Test Work on Goliath Gold Samples, Treasury Metals Incorporated. KM5262.
- 2017 Base Metallurgical Laboratories, Metallurgical Testing of Goliath Project. BL0172.
- 2020 Technical Report Re-Issue, Goldlund Gold project, Sioux Lookout, Ontario.
- 2020 Metallurgical Testing of the Goliath Gold Project. BL0697.
- 2013 SGS Scoping Study and Comminution testing on samples From the Goldlund Project. 13665-001.

13.3 Goliath Metallurgical Testing

13.3.1 2011 Mineral Processing Testwork (KM2906)

Metallurgical testing was carried out in 2011 at G&T Metallurgy in Kamloops, BC, on a single master composite. The master composite was assembled from 30 discrete samples. Head analysis is summarised in Table 13.2. The gold head grade is the assayed head grade. Calculated head grades for individual tests were not provided. The sulphide sulphur ($S^{=}$) did not result in excess cyanide consumption or impact recovery.

Table 13.1: Head Analysis, 2012

Sample	Au (g/t)	Ag (g/t)	Cu (%)	$S^{=}$ (%)	Pb (%)	Zn (%)	Fe (%)
Master Composite	3.4	25	0.017	1.41	0.04	0.08	1.33

Source: Ausenco (2021).

A QEMSCAN bulk mineral analysis on master composite 1 indicated that the major minerals are quartz (55.6%) and micas (22.2%) and that the main sulphide mineral is pyrite (1.65%). Pyrrhotite, sphalerite, galena and copper sulphides are also present.

The master composite was submitted for a single Bond ball mill grindability test at a closing size at of 106 µm and had a work index of 11.1 kWh/t. The result is considered as medium hardness.

The two flowsheets investigated included gravity concentration followed by cyanidation of the gravity tailing and gravity concentration followed by flotation of the gravity tailing followed by cyanidation of the flotation concentrate. The gravity plus cyanidation of gravity tailing produced overall gold recoveries of between 96% to 97% across a range of conditions tested.

Gold extractions appeared relatively insensitive to primary grind sizing (80% passing (P_{80}) 144 to 68 µm) and target sodium cyanide concentration in the range tested for these variables. At a target sodium cyanide concentration of 500 mg/L, sodium cyanide consumption was 0.2 kg/t and lime consumption was 0.6 kg/t.

The gravity flotation/concentrate cyanidation flowsheet produced about 6% lower recovery at about 90%. Overall silver recoveries were higher with this flowsheet due to higher silver extractions from the flotation concentrate.

A preliminary static solid-liquid separation test was completed on one of the leach residues, which showed good settling properties at low flocculant dosage.

13.3.2 2012 Mineral Processing Testwork (KM3406)

Metallurgical testing was carried out in 2012 at ALS Metallurgy in Kamloops, BC, on two master composite samples and 10 variability composites. The two master composites were assembled from 163 discrete samples of half core. The testwork program included head analysis, comminution, gravity separation and cyanidation.

The head analysis results are summarised in Table 13.2. Gold head grades are assayed head grades. The reported gold head grade for master composite 2 was higher than expected. Additional sample intervals were added to this sample to reduce the grade to the planned head grade. Master composite 3 was used for the balance of the testing. The gold head grades of the variability composites ranged from 0.36 to 15.4 g/t Au, which extends beyond the minimum and maximum of the mine plan. Problematic elements determined from the master composites, such as mercury, antimony and arsenic, are at low concentrations and will not pose any metallurgical issues. The sulphide sulphur did not result in excess cyanide consumption or impact recoveries.

Table 13.2: Head Analysis Results, 2012

Sample	Au (g/t)	Ag (g/t)	As (%)	S (%)	$S^{=}$ (%)	C (%)	TOC (%)	Hg (g/t)	Sb (%)
Master Composite 2	5.89	11	0.02	1.24	1.22	0.03	0.02	1	0.003
Master Composite 3	2.15	8	0.004	1.27	1.24	0.02	0.01	1	0.003

Source: Ausenco (2021).

The comminution testing consisted of seven Bond ball mill grindability tests and a single SMC test. The SMC (SAG mill comminution) tests on master composite 2 had an Axb of 50, which is considered as moderately hard. The ball mill work index tests were performed on master composite 2 and six variability composites and ranged from 9.2 to 13.9 kWh/t at a closing size of 106 µm. These results are considered as soft to medium hardness.

The gravity tests were performed on 2 kg samples with a Knelson laboratory concentrator and recovered 70% and 77% of gold from master composite 2 and 3, respectively.

The leach test results for 48 hours leaching for master composites 2 and 3 are shown in Table 13.3. The overall gold extraction indicates that the master composite samples are amenable to gravity and cyanidation of the gravity tailings. The kinetics show that leaching is complete by 24 hours retention time. Using air instead of oxygen in test No. 14 had no impact on recovery.

The variability composites cyanidation test results are presented in Table 13.4. All tests were run with a target grind of 80% passing 100 µm, pH 10.5 to 11, sodium cyanide concentration of 1000 mg/L and leach time of 48 hours with air sparging. The variability composites displayed similar overall gold extractions and reagents consumptions as the master composites. All samples showed very high gravity gold recoveries and high overall recoveries.

Table 13.3: Master Composite Leach Test Results

Sample	Test No.	Cyanide Conc. (mg/L)	Sparging Gas	Grind P_{80} (µm)	Reagent Cons. (kg/t)		Gold Recovery (%)	
					NaCN	Lime	Gravity	Total
Master Composite 2	2	1,000	Oxygen	114	1.1	0.3	72	98
	3	1,000	Oxygen	114	1.0	0.3	-	98
Master Composite 3	4	1,000	Oxygen	94	0.8	0.2	69	96
	5	1,000	Oxygen	94	0.6	0.2	-	95
	7	1,000	Oxygen	147	0.3	0.3	70	94
	8	1,000	Oxygen	73	0.6	0.4	74	96
	9	1,000	Oxygen	60	1.3	0.4	73	96
	10	2,000	Oxygen	94	0.8	0.3	61	94
	11	750	Oxygen	94	0.4	0.3	63	93
	12	500	Oxygen	94	0.3	0.3	68	93
	13	250	Oxygen	94	0.3	0.3	71	95
	14	1000	Air	94	2.1	0.7	85	97

Source: Ausenco (2021).

Table 13.4: Variability Leach Test Results

Sample	Test No.	Reagent Cons. (kg/t)		Gold Recovery (%)	
		NaCN	Lime	Gravity	Total
Var 1	15	1.5	0.2	76	97
Var 2	16	0.6	0.4	95	99
Var 3	17	0.5	0.4	79	97
Var 5	18	0.6	0.4	84	96
Var 6	19	0.5	0.4	67	92
Var 7	20	0.5	0.4	74	95
Var 8	21	0.7	0.4	66	92
Var 9	22	0.6	0.3	91	98
Var 10	23	0.4	0.3	77	98

Source: Ausenco (2021).

13.3.3 ALS Testing KM5262 – Goliath

A testing campaign involving head analysis and comminution testing was conducted at ALS Metallurgy in Kamloops, BC, on master and variability composites. The composites were created from 159 discrete samples of half core. The head analysis is presented in Table 13.5.

Table 13.5: Head Analysis, 2017 – KM 5262

Sample	Au (g/t)	Ag (g/t)	Cu (%)	S (%)	$S^=$ (%)	C (%)	Pb (%)	Zn (%)	Fe (%)
Minimum	0.52	<1	0.002	0.67	0.64	0.01	0.01	0.05	1.30
Maximum	2.31	8	0.03	2.62	2.60	0.03	0.18	0.39	2.98
Average	1.08	4.5	0.01	1.68	1.66	0.02	0.07	0.18	1.83

Source: Ausenco (2021).

The gold head grades ranged from 0.52 to 2.31 g/t Au. Future testing should include samples representative of underground production grades. Problematic elements, such as copper, antimony and arsenic are at low concentrations and will not pose any significant metallurgical issues.

The comminution tests results are presented Table 13.6.

Table 13.6: Comminution Testing, 2017

Sample	Axb	Abrasion Index (g)	Rod Mill Work Index (kWh/t)	Ball Mill Work Index (kWh/t)
VS11_MSS_MZ_C_UG_FR	41	0.086	11.9	8.9
VS12_MSS_MZ_C_UG_HR	43	0.093	11.5	8.9
VS13_MSS_MZ_W_UG_HR	38	0.072	13.0	11.0
VS14_MSS_MCZ_W_UG_F	37	0.086	12.7	10.1
VS15_MSS_MZ_W_UG_MW	38	0.066	13.2	10.5
VS16_MSS_CZ_UG_HR	39	0.048	12.0	10.7
VS17_MSS_CZ_UG_MWR	39	0.072	12.8	11.9
VS18_MSS_MCZ_W_OP_M	39	0.068	13.5	11.1
VS19_MSS_MZ_WC_OP_H	35	0.069	12.2	8.5
VS20_BMS_MZ_OP_HR	33	0.085	12.9	9.4

Source: Ausenco (2021).

The Axb value for the SMC test on the samples ranged from 33 to 43, which is considered moderately hard. The Bond rod mill work index ranges from 11.5 to 13.5 kWh/t, which is considered medium hardness. The Bond ball mill work index ranges from 8.5 to 11.9 kWh/t, at a closing size of 150 µm, which is considered soft to medium hardness. The bond abrasion test results ranged from 0.048 to 0.104 g, which is reflective of low abrasion wear.

The Axb values ranged from 33 to 43. Work indexes ranges indicate a SABC or ABC circuit would be applicable to treat this hardness based on the selected throughput tonnage.

13.3.4 Base Metallurgical Laboratories Testing BL0172

A testing campaign involving cyanidation followed by cyanide detoxification using SO_2/air technology was conducted at Base Metallurgical Laboratories in Kamloops, BC, on a master composite. The material used to construct the master composite was from four of the variability composites (VS12-MSS-MZ-C-UG-HR, VS15-MSS-MZ-W-UG-MWR, VS17-MSS-CZ-UG-MWR and VS21-BMS-MZ-UG-FR) used in testing shown in Section 13.3.3. The average head analysis of the master composite is presented in Table 13.7.

Table 13.7: Head Analysis, 2017 – BL0172

Sample	Au (g/t)	Ag (g/t)	Cu (g/t)	S (%)	C (%)	Pb (g/t)	Zn (g/t)	Fe (%)
Master Composite	0.96	5.9	143	1.83	0.02	1370	3325	2.0

Source: Ausenco (2021).

The average gold head grade was 0.96 g/t gold, which is below the yearly feed grade for gold based on the most recent mine plan. Problematic elements determined from the master composites, such as mercury, antimony and arsenic, are at low concentrations and will not pose any metallurgical issues. Air sparging was used for the bulk leach tests.

Two 24-hour whole ore carbon-in-leach (CIL) tests were conducted; the results are presented in Table 13.8.

Table 13.8: Carbon-In-Leach Tests

Test No.	pH	Cyanide Conc. (mg/L)	Slurry Density (%)	Grind Target P_{80} (µm)	Reagent Cons. (kg/t)		Recovery (%)	
					NaCN	Lime	Au	Ag
T01	11.0	500	45	100	0.37	0.79	88	45
Y02	11.0	1000	45	100	0.60	0.73	89	53

Source: Ausenco (2021).

These tests were completed to provide feed for cyanide detoxification testing. The results show gold extractions of 88% and 89% for tests 01 and 02 respectively. Gold extraction was not significantly improved with the higher concentration of cyanide. A lower cyanide consumption was achieved using 500 mg/L sodium cyanide concentration.

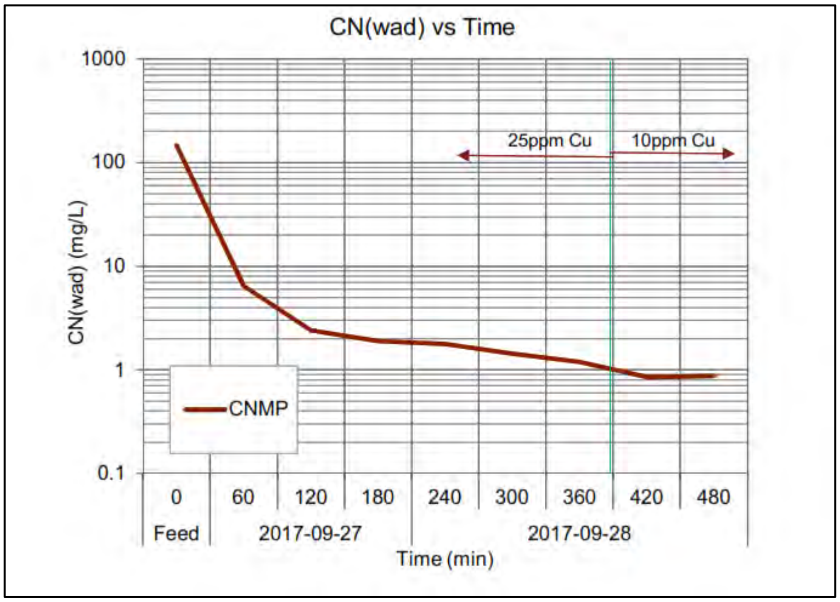
A series of batch detoxification tests were completed to determine optimal conditions for the continuous tests. Cyanide detoxification reduces weak acid dissociable cyanide (CN_{WAD}), which is toxic to aquatic species and measured by potentiometric titration and referred to as CNMP. Parameters examined include:

- SO_2:CN_{WAD} addition rates (sodium metabisulphate solution used as the SO_2 source) in the range of 3:1 to 6:1 on weight basis
- copper (as copper sulphate solution) addition rates in the range of 0 to 50 mg/L
- pH in the range of 7 to 8.5 using caustic soda and lime
- retention time up to 180 minutes
- slurry density at 35% and 45% solids

The results showed optimum conditions of SO_2:CN_{WAD} ratio of 5:1, 60 minutes retention time, copper addition of 25 mg/L at pH 8.5 and 40% solids.

The continuous test results are presented graphically in Figure 13-1. The initial CN_{WAD} concentration was 146 mg/L. The results show that after 180 minutes operation, the CN_{WAD} concentration was reduced to 2 mg/L. The copper addition was reduced to 10 mg/L and the CN_{WAD} concentration was further reduced to 0.9 mg/L. The results demonstrate that Goliath leached samples are amenable to SO_2/air cyanide detoxification and achieve low concentrations of CN_{WAD}.

Figure 13-1: Continuous Detoxification



Source: BaseMet Laboratories (2017).

13.4 Goldlund Metallurgical Testing

In 2013 SGS (Lakefield) performed head analysis, gold deportment study, comminution testing, gravity separation, rougher flotation and cyanidation testing on 15 samples from five different Zones (1, 2, 3, 4 and 7). The samples were a combination of drill core and crusher rejects.

Various methods of head analyses are summarised in Table 13.9. Direct assays are by screened metallics at 105 µm. The other head grades are calculated from large-scale (10 kg and 30 kg samples) gravity concentration followed by cyanidation of gravity tailings. The gold head grade from 0.48 to 16.3 g/t gold (ignoring one sample at detection limit of <0.02 g/t), which extends beyond the minimum and maximum of the mine plan. A selection of eight samples were also submitted for mercury assays, all of which were all below the detection limit of 0.3 g/t.

The sample grades ranged from 0.48 to 3.17 g/t Au. Sample 5 from Zone 7 had a very high head grade that was confirmed through the calculated head grade from testing. The head grade of sample 3 from Zone 3 was below detection limit and the sample was not used for testing.

Bulk mineralogy on three samples indicated the major minerals present are plagioclase, followed by quartz, then mica, with minor amount of calcite, chlorite and iron oxides as well as trace minerals.

The gold deportment study on 4 samples by SEM-EDS indicated the gold minerals are native gold with composition ranging from 87.1% to 93.0% gold and 5.3% to 10.6% silver. The presence of gold tellurides was also identified. The tellurium head assays ranged from 4 to 13 g/t for the samples examined, which is considered moderate to abundant.

Table 13.9: Head Analysis & Calculated Head Grades, 2013

Sample	Zone	Direct (g/t)	10 kg Gravity & Cyanidation (g/t)	30 kg Gravity & Cyanidation (g/t)
Sample 1	1	0.81		
Sample 4		1.02		
Sample 13		1.36		1.07
Sample 16		1.08	0.89	1.15
Sample 11	2	0.65	0.73	0.72
Sample 19	2 & 3	0.60		0.72
Sample 3	3	<0.02		
Sample 10		0.84		1.02
Sample 12		0.66		0.57
Sample 14	4	0.82	0.81	0.80
Sample 15		0.59		0.55
Sample 17	1 & 4	0.66		0.62
Sample 18	1 & 4	0.48	0.50	0.50
Sample 5	7	16.3		
Sample 6		3.17		
Sample 7		1.64		0.93 & 1.31

Source: Ausenco (2021).

The comminution testing included SAG power index (SPI) tests and Bond ball mill grindability tests. The Bond ball mill results are presented in Table 13.10.

Table 13.10: Goldlund Comminution Tests

Sample	Zone	Ball Mill Work Index (kWh/t)	
		(105 µm)	(75 µm)
Sample 1	1	13.4	14.0
Sample 4		-	14.0
Sample 3	3	-	20.7
Sample 6	7	13.7	14.0

Source: Ausenco (2021).

The SPI is a measure of the hardness of the sample as it relates to semi-autogenous grinding. The results from samples tested from Zone 1 (samples 1 and 4) and zone 7 (sample 6) are considered to be hard.

The Bond ball mill work index (BWI) was around 13 to 14 kWh/t for the 105 µm tests, classified as medium hardness. The 75 µm test results ranged from 14 to 20.7 kWh/t, classified as hard. The test results from the smaller closing size of 75 µm would only be relevant if a grind of about 53 µm was selected.

A summary of the Goldlund testing is shown in Table 13.11. Higher grade samples from Zone 1 (samples 1 and 4) and Zone 7 (samples 5 and 6) were tested with gravity concentration followed by flotation of gravity tailings and concentrate leach. The samples had high gravity and flotation recoveries but low gold extractions from flotation concentrate, ranging from 36% to 76%. Overall gold recoveries for the Zone 1 samples were 62% and72% and for Zone 7 was 55% and 74%.

Table 13.11: Goldlund Test Results

Zone	Sample	Grind Target P_{80} (µm)		Gravity Rec. (% Au)	Flot. Rec. (% Au)	Leach Extraction (% Au)		Overall Recovery (% Au)		Head Grade (g/t Au)
		Feed	Conc.			48 h	72 h	Gravity/Leach	Gravity/ Flot./ Leach	
1	1	75	50	33	75	-	59	-	62	0.77
	4	75	34	30	79	-	76	-	72	1.10
	13	76	-	33	-	83	-	89	-	1.07
	16	93	-	29	-	83	-	88	-	1.15
	16	47	-	36	-	80	-	87	-	0.89
2	11	68	-	36	-	80	-	87	-	1.15
		43	-	30	-	82	-	88	-	0.72
2&3	19	77	-	25	-	80	-	85	-	0.73
3	10	91	-	31	-	78	-	85	-	1.02
	12	77	-	33	-	78	-	85	-	0.57
4	14	66	-	22	-	85	-	88	-	0.80
		43	-	19	-	88	-	91	-	0.81
	15	72	-	12	-	87	-	88	-	0.55
1&4	17	82	-	12	-	84	-	86	-	0.62
	18	67	-	12	-	88	-	89	--	0.50
	18	39	-	30	-	89	-	92	-	0.50
7	5	75	20	30	92	-	39	-	55	17.7
	6	72	25	32	89	-	70	-	74	1.42
	7	76	-	53	-	86	-	93	-	0.93
		40	-	69	-	89	-	96	-	1.31

Source: Ausenco (2021).

The balance of the samples were tested via gravity concentration followed by leaching of gravity tailings. Leach conditions included:

- grind size of 80% passing 75 to 93 µm, with 40 µm for selected samples
- slurry density = 40% solids
- pH 10.5 to 11.0
- cyanide concentration of 0.5 g/L NaCN
- retention time of 48 hours with air sparging

The results were variable with recoveries ranging from 85% to 96%. Finer grinds were evaluated on several samples with inconclusive results. The identification of gold tellurides in the mineralogical evaluation is a possible reason for the lower recoveries in some of the samples.

13.5 Goliath & Goldlund Metallurgical Testing

In 2020, samples from previous Goliath and Goldlund testing programs were utilised for a program to generate tailings samples for environmental testing. The program included:

- Grinding was carried out to 80% passing 75 µm and leaching for 48 hours with 0.5 g/L NaCN.
- The Goliath sample was leached at pH 10.5 and the Goldlund sample at pH 12 to counteract potential tellurides. The combined composite was leached at pH 11.
- The combined sample was tested for SO_2/air cyanide destruction.
- Oxygen uptake test on the combined sample.
- The detoxified sample was tested for sulphide flotation to generate a sulphide concentrate and low sulphide tailings for environmental testing. Test conditions included 12 minutes flotation time with standard pyrite flotation reagents (potassium amyl xanthate and MIBC).

The results included:

- Calculated gold head grades for the samples included Goliath 1.40 g/t, Goldlund 0.95 g/t and blend 0.81 g/t.
- Leach extractions of 93%, 92% and 92% gold from the Goliath, Goldlund and combined samples, respectively. The results are summarized in Table 13.12.
- Oxygen uptake tests demonstrated low oxygen demand with the majority of oxygen consumed in the first hour of the 24-hour test.
- The cyanide destruction testing reduced the CN_{WAD} concentration from 124 mg/L to below 1 mg/L using typical conditions (SO_2:CN_{WAD} addition of 5:1, 25 mg/L copper addition and 60 minutes retention time).
- Flotation test results included 3.3% mass and 85% sulphur recovery to the concentrate. The flotation tailings sulphur grade was 0.12% S compared to the flotation feed grade of 0.72% S.

Table 13.12: 2020 Goliath-Goldlund Test Results

Composite	Test	Gold Extraction – Percent Cumulative					Consumption (kg/t)	
		2	6	8	24	48	NaCN	Lime
Goliath	1	60.2	72.9	77.0	91.6	93.2	0.29	0.60
Goldlund	2	55.1	74.8	79.3	85.0	92.1	0.18	3.45
Blend	3	57.2	63.2	66.3	75.4	92.0	0.07	0.06

Source: Ausenco (2021).

14 MINERAL RESOURCE ESTIMATES

14.1 Goliath

AGP completed an updated mineral resource estimate of the Goliath deposit held by Treasury Metals. The project is located 20 km east of Dryden, Northwestern, Ontario. Geovia GEMS Version 6.8™ software was used for the resource estimate. The metals of interest at the Goliath deposit are gold with minor quantities of silver.

14.1.1 Data

On August 8, 2020 Treasury Metals provided AGP with a project database consisting of collar data, down-the-hole survey, logged lithology, assays, and density. The data set was supplemented with assay certificates, the 2019 resource model and associated wireframes, QA/QC data, and topography consisting of LiDAR 50 cm contour lines. The drill data was updated on August 18, 2020 to provide more information such as geochemistry, density, mineralisation, alteration, structure, texture, and veining. Final assays for the 2019-2020 drill program were made available on October 6, 2020.

Data was fully validated before being used in the resource estimate, as described in Section 12 of this report. As a final step, drill data were checked for overlapping, missing, and negative length intervals. No erroneous data was detected affecting the primary database table used in the resource estimation.

No further additions were made to the database after October 6, 2020, which is the official data cut-off date for this resource estimate. For the Goliath deposit, 837 core holes exist in the database. Of these, 726 core holes contributed to the grade estimation.

Table 14.1 shows a summary of the number of holes and assays used in the resource estimate.

Table 14.1: Summary of Number of Holes used in the Resource Estimate

Zone	Type	Number of Holes	Total Length (m)	Number of Assays	Comment
Holes intersecting the mineralised wireframes					
Goliath	Core hole	726	238,036	96,912	
Holes not intersecting the mineralised wireframe					
Goliath	Core hole	4	1,631	1,107	Exploration drill holes > 7 km east
Goliath	Core hole	107	27,746	12,379	Hole with no wireframe intersection
Subtotal		111	29,377	13,486	
Total in Database					
Grand Total		837	267,413	110,398	

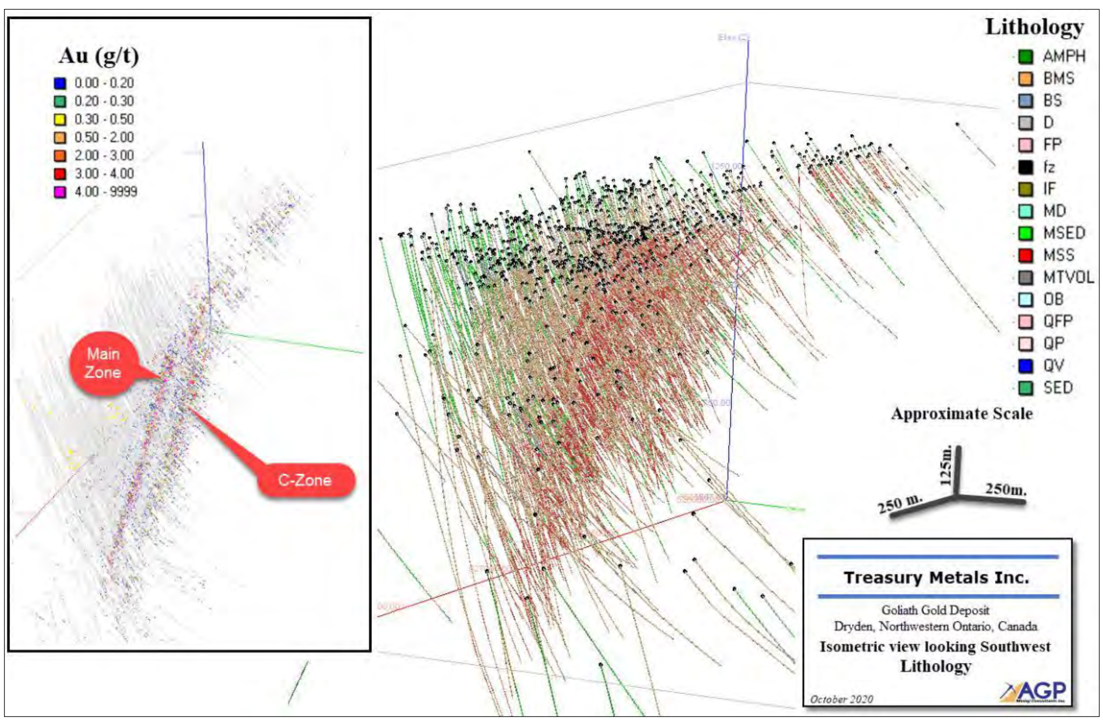
An additional 46 drill holes were omitted from the GEMS database. These were abandoned due to excessive deviation, misalignment, and drill issues.

All data in the GEMS database was elevated by 1,000 m to avoid negative elevation at depth.

14.1.2 Geological Model

At Goliath, the bulk of the mineralisation occurs in higher grade pyritic muscovite-sericite schist horizons (MSS) intercalated between lower grade to waste biotite-muscovite schists (BMS) with minor metasedimentary rocks (MSED). Alteration consist mainly of sericitisation and silicification associated with gold mineralisation. The BMS/MSS horizons are variable in thickness and the logged intervals of MSS can be interpreted as containing “mostly” MSS with possibly some BMS. The bulk of the mineralisation is located in two principal mineralised corridors namely the Main Zone and the C Zone. Other minor zones of mineralisation exist on the hanging wall and footwall but are not as well developed, are lower grade, and more discontinuous. Figure 14-1 illustrates the distribution of the MSS and BMS lithologies along with the gold grade distribution in the inset image.

Figure 14-1: Lithology & Assays



Source: AGP (2020).

The logged alteration code and the estimated pyrite content in the log failed to provide additional support to the model. The quartz vein intensity showed a clustering in the vicinity of the Main Zone and C Zone; however, the clustering was not specific enough to aid the model.

The 3D lithological wireframes, developed to control the grade interpolation of the resource model, were based primarily on the gold grades above 0.2 g/t and on the previous interpretation. Procedures used in the development of these wireframes are as follows:

14.1.2.1 Main Zone & C Zone

For the Main Zone and C Zone, the mineralisation is located within two wide mineralised corridors as defined with assays above 0.2 g/t. Grade tends to be the highest in proximity to the corridor edges within a few metres from the "contact". Internally there is often a waste/low grade zone, but that can be variable from hole to hole. On some sections, the waste/low grade is well defined but on other sections, that pattern is broken by high-grade holes. The position of the mineralised corridors is predictable as evidenced by the numerous infill holes which intersected the mineralisation at the expected location with similar tenors. In order to capture these features, the hanging wall and footwall extent of the corridors were selected as points along the drill hole trace. The selection included internal waste/low grade which will be separated out using a probabilistic model, as described in Section 14.1.2.4. Hanging wall and footwall surfaces were created and stitched together to form a 3D solid which was then clipped to the extent of the drilling.

14.1.2.2 B, D, E & H Zones

On the hanging wall of the Main Zone, the H series zones comprised of H1 Zone, H2 Zone, H3 Zone, H4 Zone, and H5 Zone. The B Zone is located between the Main Zone and C Zone and on the footwall of the C Zone there are two more zones namely the D Zone and the E Zone. All these zones were modelled conventionally using sectional polylines joined by tie lines guided by the light table option in GEMS. A probabilistic approach was attempted but unsuccessful. Wireframes were generally created using a minimum of two intercepts on one section and a minimum of one intercept on another section. A minimum of two assays on each individual drill hole were used which approximate a 1.5 to 2.0 m minimum mining width. Zones were occasionally extended through assayed waste intervals and un-sampled drill hole intervals in order to maintain zonal continuity.

AGP notes that while the completed zones resemble discrete veins, they are not. The lower grade mineralisation is not readily identifiable in the core and the only clue to the continuity of these zones was offered by the higher-grade assays occurring at similar position across the sections and along the drill hole traces. Zones located in proximity to the Main Zone and C Zone (H1, H2, B, and D) are more readily identifiable and thus easier to model, while the H3, H4, H5, and E Zones were not as well supported by drilling and as a result, were more difficult to model.

14.1.2.3 Domain Model

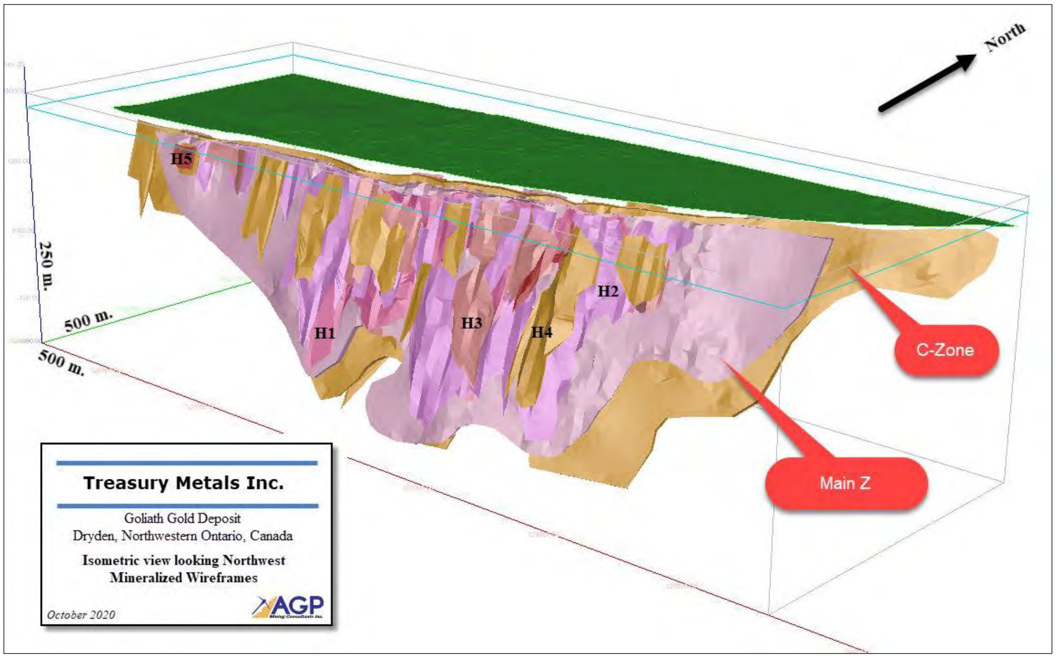
A total of 47 individual wireframes were developed for the resource model. Some of these were combined resulting in ten mineralised domains. The Main Zone and C Zone occupy 82.57% of the total volume (see Table 14.2).

Figure 14-2 illustrates the location of the mineralised wireframes within the model.

Table 14.2: Mineralised Wireframes Clipped to Overburden & Volume

GEMS Name 1	GEMS Name 2	GEMS Name 3	Domain Name	Domain Code	Analytical Volume (M3)	Percent of Total Volume
HW	H5a	CLIP	H5	5500	53,217	0.09%
HW	H5b	CLIP	H5	5500	9,696	0.02%
HW	H4a	CLIP	H4	5400	121,742	0.22%
HW	H4b	CLIP	H4	5400	453,623	0.80%
HW	H4c	CLIP	H4	5400	26,385	0.05%
HW	H4d	CLIP	H4	5400	27,692	0.05%
HW	H4e	CLIP	H4	5400	95,083	0.17%
HW	H4f	CLIP	H4	5400	77,946	0.14%
HW	H4g	CLIP	H4	5400	220,928	0.39%
HW	H4h	CLIP	H4	5400	27,546	0.05%
HW	H4i	CLIP	H4	5400	117,671	0.21%
HW	H3a	CLIP	H3	5300	53,472	0.09%
HW	H3b	CLIP	H3	5300	51,513	0.09%
HW	H3c	CLIP	H3	5300	207,166	0.37%
HW	H3d	CLIP	H3	5300	154,752	0.27%
HW	H3e	CLIP	H3	5300	39,404	0.07%
HW	H2a	CLIPM	H2	5200	309,042	0.55%
HW	H2b	CLIPM	H2	5200	794,349	1.41%
HW	H2c	CLIPM	H2	5200	96,067	0.17%
HW	H2d	CLIPM	H2	5200	437,138	0.77%
HW	H2e	CLIPM	H2	5200	23,750	0.04%
HW	H2f	CLIPM	H2	5200	111,690	0.20%
HW	H1	CLIPM	H1	5100	862,344	1.53%
HW	H1a	CLIPM	H1	5100	140,990	0.25%
HW	H1b	CLIPM	H1	5100	322,930	0.57%
HW	H1c	CLIPM	H1	5100	45,148	0.08%
MAIN	M Zone	CLIP	M Zone	1000	23,411,658	41.49%
FW	Ba	CLIPMC	B Zone	2000	501,784	0.89%
FW	Bb	CLIPMC	B Zone	2000	17,975	0.03%
FW	Bc	CLIPMC	B Zone	2000	1,042,823	1.85%
FW	Bd	CLIPMC	B Zone	2000	861,533	1.53%
FW	Be	CLIPMC	B Zone	2000	30,657	0.05%
FW	Bf	CLIPMC	B Zone	2000	35,785	0.06%
FW	B1a	CLIPMC	B Zone	2000	240,733	0.43%
C ZONE	C ZONE	CLIP	C Zone	3000	23,172,901	41.07%
FW	Da	CLIPC	D	6000	194,385	0.34%
FW	Db	CLIPC	D	6000	30,002	0.05%
FW	Dc	CLIPC	D	6000	33,360	0.06%
FW	Dd	CLIPC	D	6000	1,178,674	2.09%
FW	De	CLIPC	D	6000	74,071	0.13%
FW	Df	CLIPC	D	6000	105,259	0.19%
FW	Ea	CLIP	E	7000	55,378	0.10%
FW	Eb	CLIP	E	7000	71,340	0.13%
FW	Ec	CLIP	E	7000	396,788	0.70%
FW	Ed	CLIP	E	7000	67,550	0.12%
FW	Ee	CLIP	E	7000	14,849	0.03%
FW	Ef	CLIP	E	7000	10,450	0.02%
Total Analytical Volume					56,429,239	

Figure 14-2: Mineralised Wireframes



Source: AGP (2020).

14.1.2.4 Probabilistic Model for Main Zone & C Zone

Within the Main Zone and C Zone wireframes, there is significant mixing of grade population. Low grade/waste zones are often intercalated between short high-grade zones mixed within medium grade zones. Interpolating the model without further grade separation would results in significant grade smearing.

While the hanging wall and footwall contacts of the Main Zone and C Zone are well defined, the internal grade subdivisions can be erratic from section to section and between drill holes which render the development of conventionally modelled wireframes challenging. In order to address this issue two probabilistic models were generated to separate the population.

14.1.2.4.1 Low-Grade/Waste Delineation

A probabilistic model was first completed to separate the low grade/waste from the medium to higher-grade mineralisation. The raw assays within the wireframes were composited at 0.5 m since the high-grade assays are often smaller than 0.5 m in length. The composited raw assays were flagged with a 1 for assays < 0.2 g/t and 0 for values ≥ 0.2 g/t gold. The 0.2 g/t gold corresponds to the first inflection on probability charts. Ten orientation sub-domains for the Main Zone were created and seven orientation sub-domains were created for the C Zone in order to follow the predominant strike and dip of the deposit from east to west. The orientation sub-domains are discussed in more detail in Section 14.1.13 below. The sample search ellipsoid range was set to 150 m for the Major axis, 60 m for the semi-major axis, and 15 m for the minor axis to reflect the steep westerly plunge observed in the mineralisation. The probabilistic model was interpolated in one pass with a minimum of 6 composites, 15 maximum and a maximum of 3 composites per hole. The resulting interpolated model bears a value between 0 and 1 where the higher values indicates a higher probability of the blocks being low-grade/waste.

The interpolated model was visually validated on sections and plan. Selecting a threshold value to separate the grade population as cleanly as possible is a critical step in this procedure. A threshold value of 0.675 probability was selected because:

- the threshold value maximises the average grade for the high-grade mineralisation
- the threshold value also maximises the grade differential between the high-grade – low-grade/waste portion of the model
- the coefficient of variation was found to be marginally better for the Main Zone and unchanged for the C Zone

The last step in the process is to tag the blocks in the model showing a probability in excess of 0.675 with a code of 1. The final model was groomed with an aggressive algorithm which applied more weight to the blocks oriented along the strike of the deposit. This grooming process eliminated a good portion of the isolated blocks.

14.1.2.4.2 High-Grade Delineation

The procedure to create the high-grade probabilistic model was similar. The 0.5 m assay composites were flagged with a 1 for assays ≥ 2.0 g/t gold, 0.5 for values between 0.5 g/t and 2.0 g/t gold and 0 for values < 1.0 g/t gold. The 2.0 g/t gold corresponds to a slight inflection on probability charts and the 0.5 flag allowed for more weight to be applied to the shoulder assays which after experimentation, yielded improvements in zonal continuity.

The high-grade probabilistic model was interpolated in two passes with a minimum of 7 composites, 15 maximum and a maximum of 4 composites per hole for Pass 1 and a minimum of 2 composites, 15 maximum and a maximum of 5 composites per hole for Pass 2. The two-pass approach was used to quantify data support and adjust the class model in the resource amenable to underground extraction. Pass 1 range was set to 75 m along the major axis, 30 m along the semi-major axis and 10 m along the minor axis. These ranges were increased to 130 m, 50 m, 20 m respectively for Pass 2.

The resulting interpolated model bears a value between 0 and 1 where the higher values indicates a higher probability of the blocks being high grade. The model was manually adjusted between section 528,000E and 527,875E (below 1000 elevation) via the use of a helper wireframe. The probabilistic value for the blocks within the wireframe envelope was boosted 1.5 times its interpolated value to account for an area in the model that was not representative of the trend in the mineralisation.

The interpolated model was visually inspected on sections and plan. Blocks in excess of 0.34 probability were converted to a code of 1 and then groomed to eliminate the isolated blocks.

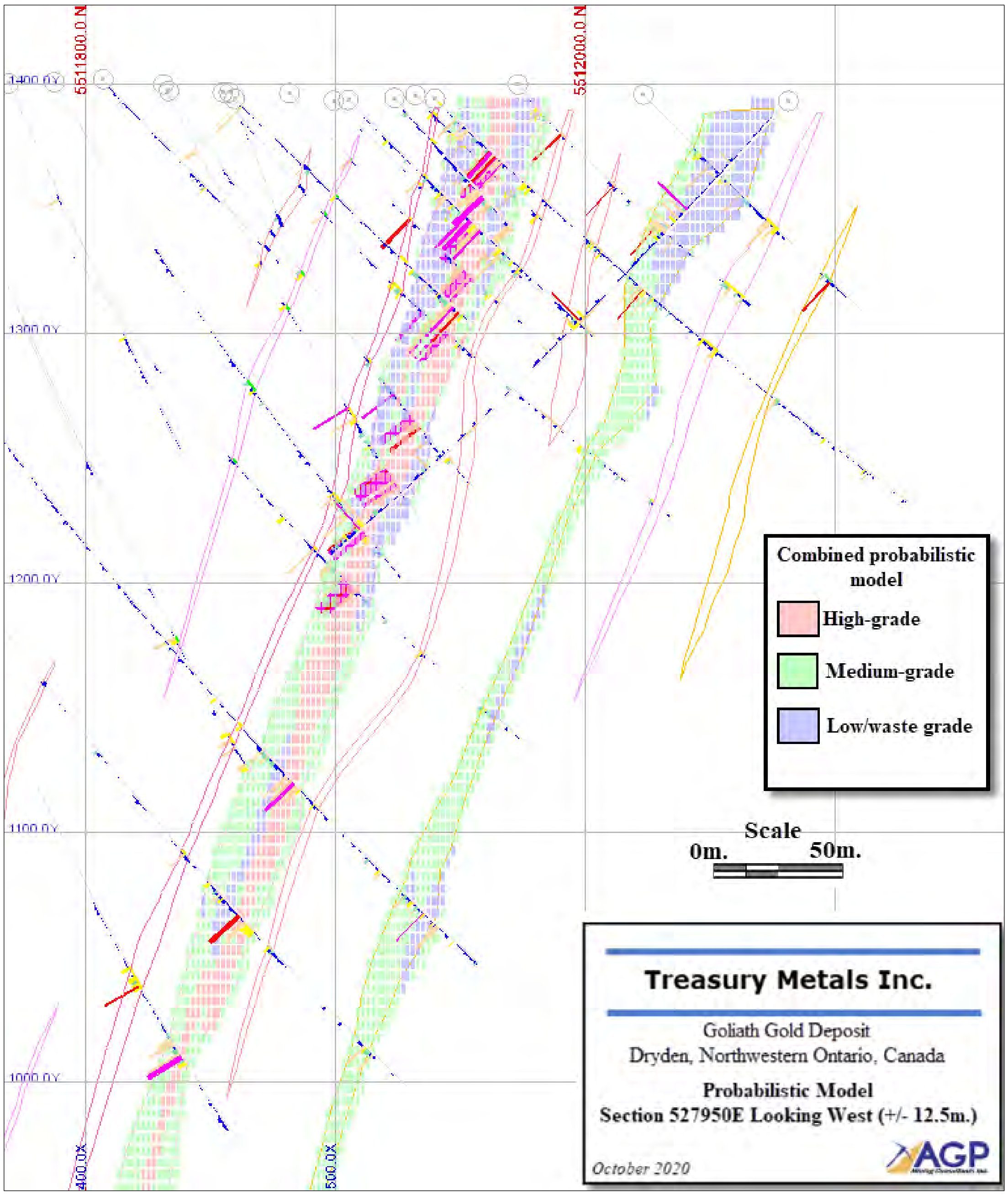
14.1.2.4.3 Final Model

The low-grade/waste probabilistic model and the high-grade probabilistic model were re-combined in a single model via a GEMS™ Cypress-enabled script which assigned a code of 1, for the probable high-grade zones, a code of 2 for the medium grade zones and a code of 3 for the probable low-grade/waste areas.

The final model can easily be validated for reasonableness by comparing the coded blocks with the drill hole grade.

The probabilistic approach reasonably separated the high-grade, medium-grade and the low-grade within the Main Zone and C Zone as illustrated in Figure 14-3.

Figure 14-3: High-grade/Medium grade/Low-grade Separation on Section 527950E



Source: AGP (2020).

14.1.2.5 Topography

The topography was created by merging the Government of Canada 1:50000 topography from the CanVec series with the LiDAR 50 cm contour lines provided by Treasury Metals.

14.1.2.6 Overburden

The overburden was created by generating an isopach map based on drill hole information. The thickness of the overburden was then subtracted from the topography and the final overburden surface was completed. The overburden shows a number of “dimples” related to the drill hole collar located slightly above or below the LiDAR topo. Since the holes were surveyed using a high precision surveying instrument, the drill data was considered correct and no drill holes were adjusted.

14.1.3 Exploration Data Analysis

Exploratory data analysis is the application of various statistical tools to characterise the statistical behaviour or grade distributions of the data set. In this case, the objective is to understand the population distribution of the grade elements in the various domains using such tools as histograms, descriptive statistics, and probability plots.

14.1.3.1 Assays

The raw assay statistics were evaluated, grouping all assays intersecting the mineralised wireframes. The assays from Main Zone and C Zone were back tagged from the combination probabilistic model to separate the high-grade versus medium and low-grade/waste components. Box and whisker plots on the H1, H2, H3, H4, and H5 indicated the gold distribution was sufficiently similar to allow the grouping of those zones for statistical evaluation.

Table 14.3 provides descriptive statistics for raw, uncapped, gold values.

The Main Zone and C Zone high grade bears the highest gold grade averaging 3.89 g/t Au and 2.21 g/t Au, respectively. Grade above 1 g/t Au does not occur in the other zones before the 90th percentile of the population. The coefficient of variation (CV) indicates high variability in the assay distribution. From the CV values observed in the table, one can deduct that capping of outliers will be required and will likely have a significant impact on the total ounces.

The Spearman correlation between gold and silver shows an R-square of 0.46. A linear regression was attempted and found to be poor with an R-Square of 0.08, with a slope of regression of 0.97. By eliminating 70 outliers from the data set, the regression R-Square improves to 0.13 and the slope of regression remains unchanged. The binned gold grade shows a positive relationship to the silver assays. AGP concludes that while individual assay pairs (Au-Ag) are variable, the grouped gold values shows a definite trend of increasing silver values with higher gold grade. From this work, AGP concludes the gold domains could be used for silver for the purpose of this study, however for more advance studies, more silver analysis should be completed, and additional work should be conducted to evaluate the silver distribution within the model.

Table 14.4 provides descriptive statistics for raw, assayed uncapped, silver values.

Table 14.3: Gold Descriptive Statistics

Domain	All	H1-H5 Zones	Main Zone			B Zone	C Zone			D Zone	E Zone
			High Grade	Medium Grade	Low Grade/Waste		High Grade	Medium Grade	Low Grade/Waste		
Valid cases	30033	3281	2737	5811	6334	1846	829	3572	4727	633	263
Mean	0.87	0.53	3.89	0.65	0.37	0.72	2.21	0.59	0.36	0.68	0.48
Variance	49.7	3.4	416.7	11.1	3.9	53.3	52.5	13.3	5.8	7.8	1.0
Standard Deviation	7.1	1.8	20.4	3.3	2.0	7.3	7.2	3.7	2.4	2.8	1.0
Variation Coefficient	8.1	3.5	5.3	5.1	5.4	10.2	3.3	6.1	6.6	4.1	2.1
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.01	0.01
Maximum	747.26	49.18	747.26	126.30	68.49	286.23	122.19	152.00	145.00	45.17	8.74
1st Percentile	0.01	0.01	0.02	0.01	0.01	0.01	0.02	0.01	0.01	0.01	0.01
5th Percentile	0.02	0.01	0.07	0.03	0.01	0.01	0.06	0.03	0.01	0.02	0.02
10th Percentile	0.03	0.02	0.14	0.06	0.02	0.02	0.11	0.06	0.02	0.04	0.03
25th Percentile	0.07	0.05	0.32	0.15	0.04	0.06	0.25	0.14	0.04	0.08	0.07
Median	0.21	0.19	0.92	0.31	0.09	0.15	0.57	0.29	0.10	0.25	0.19
75th Percentile	0.49	0.41	2.43	0.58	0.22	0.34	2.12	0.53	0.25	0.55	0.44
90th Percentile	1.26	1.00	6.14	1.03	0.59	0.72	4.12	0.98	0.62	1.13	1.04
95th Percentile	2.43	1.89	12.33	1.59	1.13	1.46	7.88	1.40	1.10	1.68	1.92
99th Percentile	10.27	5.83	56.13	4.56	4.34	9.81	22.88	4.03	4.22	8.11	6.57

Table 14.4: Silver Descriptive Statistics

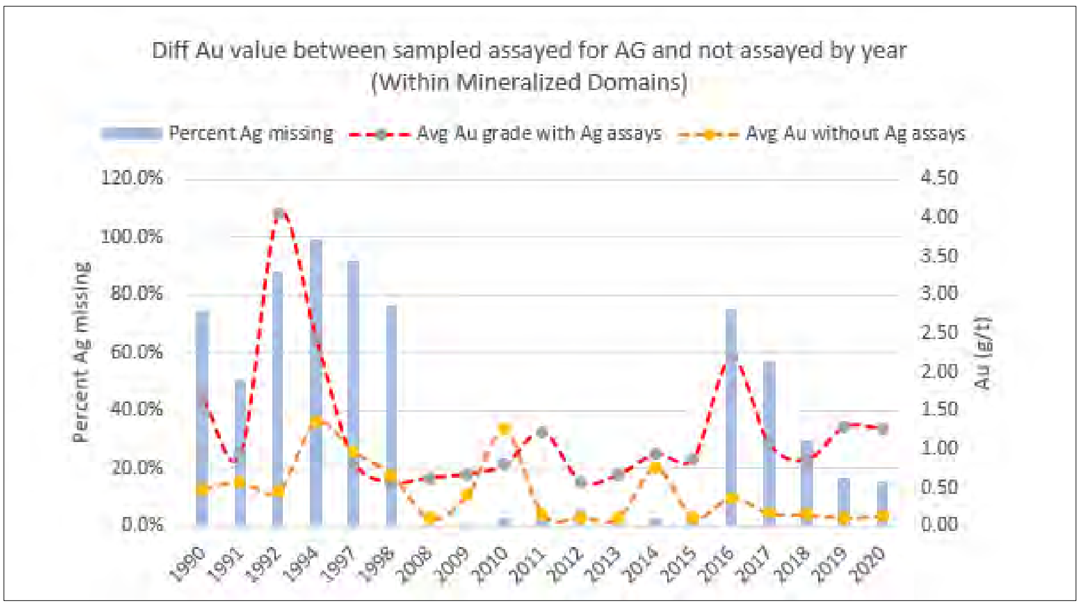
Domain	All	H1-H5 Zones	Main Zone			B Zone	C Zone			D Zone	E Zone
			High Grade	Medium Grade	Low Grade/Waste		High Grade	Medium Grade	Low Grade/Waste		
Valid cases	20989	1940	1871	3773	3969	1391	694	2998	3676	471	206
Mean	4.6	3.4	14.7	4.0	3.1	7.8	8.2	2.7	2.4	2.3	2.2
Variance	707.8	264.5	3869.2	202.5	364.3	2205.3	396.0	44.0	306.3	79.5	12.2
Standard Deviation	26.6	16.3	62.2	14.2	19.1	47.0	19.9	6.6	17.5	8.9	3.5
Variation Coefficient	5.8	4.8	4.2	3.5	6.2	6.0	2.4	2.5	7.1	4.0	1.6
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0	0.1	0.2
Maximum	1300	565	1214	344	923	1300	205	128	921	186	26
1st Percentile	0.1	0.0	0.4	0.1	0.0	0.0	0.4	0.2	0.0	0.1	0.3
5th Percentile	0.5	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.4	0.3	0.5
10th Percentile	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
25th Percentile	0.5	0.5	1.0	0.7	0.5	0.5	0.5	0.5	0.5	0.5	1.0
Median	1.0	1.0	3.0	1.3	1.0	1.0	2.0	1.0	1.0	1.0	1.0
75th Percentile	2.9	2.0	9.2	3.0	2.0	2.2	6.0	2.6	1.9	2.0	2.0
90th Percentile	7.0	5.0	26.0	7.6	4.0	10.1	21.0	5.0	3.3	4.0	5.0
95th Percentile	14.0	11.0	48.4	12.7	8.0	31.5	36.4	9.0	6.2	6.0	8.1
99th Percentile	58.0	42.6	207.3	45.7	36.9	93.2	100.9	21.0	28.0	14.6	21.8

The Main Zone, C Zone (High-grade) and the B Zone bears the highest silver grade, averaging 14.7, 8.2 and 7.8 g/t Ag, respectively. Except for the two high-grade domains, silver values are below 100 g/t even in the 99th percentile of the population. Similar to gold the CV is also well above 2.0 and outlier capping will be required.

14.1.3.2 Missing Silver Assays

An assessment of the missing silver assays shows that, on average, 30.8% of the gold assays have missing silver assays. This is due to limited assaying for silver throughout the years. The silver assaying practice changed over the years. The best data is between 2008 and 2015 where most gold assays also have a silver assay (see Figure 14-4). Figure 14-4 also shows the average gold values are higher when there is a silver value indicating the core was often only assayed for silver when there were visual clues that the core is mineralised.

Figure 14-4: Missing Assays per Drill Campaign



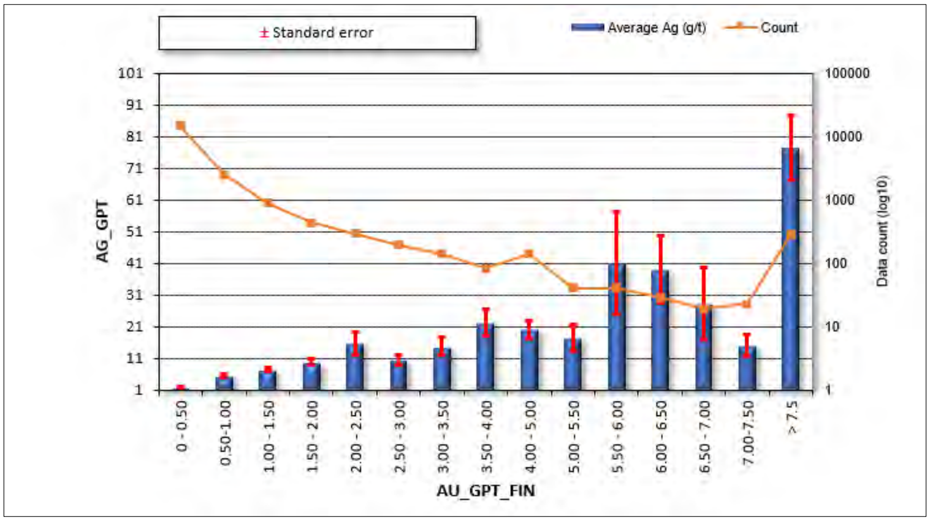
Source: AGP (2020).

Figure 14-5 shows the relationships of gold versus silver. The graph indicates that generally the silver shows a positive relationship with gold. AGP attempted to construct a linear regression and found the silver grade is too variable and the correlation with gold is not strong enough.

From the statistical work carried out by AGP the following three conclusions can be deducted:

- The selective assaying for silver when the gold grade is high can promote a higher silver grade in the model, Ignoring the missing silver assays during compositing will only promote an even higher silver grade model.
- assigning zero grade for the unsampled interval will un-justly penalised the silver grade since the grade of the missing assays is likely higher
- the problem is also acute due to the large number of missing assays

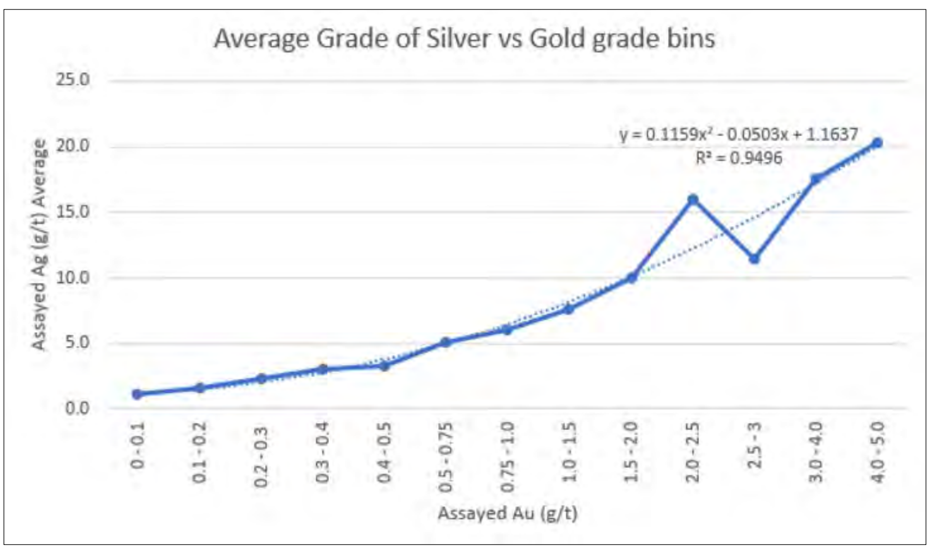
Figure 14-5: Relationship Gold – Silver by Bins



Source: AGP (2020).

One option is to calculate a silver grade for the missing assays which would likely be better than ignoring or assigning zero grade. However, due to the poor Au-Ag correlation on individual assays AGP elected to calculate the silver values based on the gold bin average (Figure 14-6).

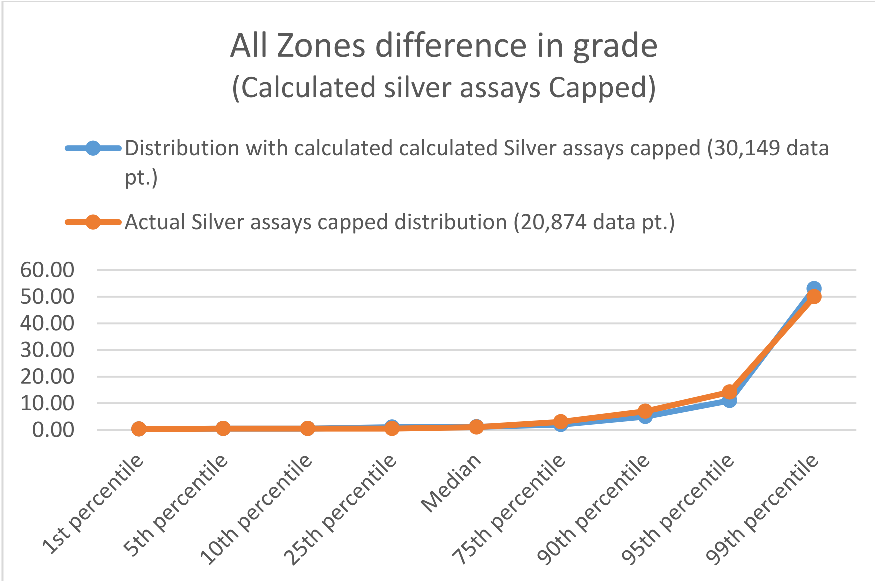
Figure 14-6: Formula to Calculate the Missing Silver Values



Source: AGP (2020).

The goal of the formula was to calculate the silver missing assays while not affecting the silver distribution and the overall silver average. Using this methodology, it is impossible to back calculate the silver value to test the equation. AGP also found the formula can generate outrageous silver assays on high-grade gold assays. The best comparison was obtained by using capped silver assays. Figure 14-7 shows the distribution of the calculated capped assays versus the actual silver assays.

Figure 14-7: Calculated Missing Silver Assays Capped vs. Actual Assays



Source: AGP (2020).

Table 14.5 shows the difference between the laboratory silver assay distribution versus the laboratory + calculated capped missing silver assays distribution. The data shows the original distribution is well maintained up to the 75th percentile. Above that, the dataset with the calculated sliver assays generally return a lower grade except for assays in the 99th percentile. While AGP understands this is not ideal, the methodology implemented provides a reasonable compromise between ignoring the missing assays or assigning zero grade. AGP suggests Treasury Metals re-assay additional drill core pulps located within the mineralised wireframes for silver in order to mitigate this issue in future models.

Table 14.5: Differences in the Distribution of Silver by Zone (Measured Only vs. Measured + Calculated)

Zones	ALL	B Zone	C Zone	D Zone	E Zone	H Zones	M Zone	Avg. Diff.
Number calculated values	9275	481	1822	165	57	1390	5360	
Percent of total calculated	31%	26%	20%	26%	21%	42%	36%	
Mean difference	-0.40	-1.98	-0.22	-0.62	-0.28	0.10	-0.30	-0.53
1st Percentile	0.00	-0.02	0.00	0.00	-0.06	0.00	-0.10	-0.03
5th Percentile	0.00	0.00	0.00	-0.19	0.00	0.00	0.00	-0.03
10th Percentile	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
25th Percentile	-0.50	-0.50	0.00	0.00	0.00	-0.50	-0.50	-0.29
Median	-0.16	-0.16	-0.10	-0.16	0.00	-0.16	-0.07	-0.12
75th Percentile	1.00	0.45	0.00	0.15	0.66	0.78	1.00	0.58
90th Percentile	2.00	4.33	0.40	0.40	1.00	1.91	2.88	1.84
95th Percentile	3.19	9.78	1.73	0.58	-2.00	5.00	4.80	3.30
99th Percentile	-3.00	-42.96	4.00	-6.54	-14.54	5.00	-3.01	-8.72

14.1.3.3 Outlier Control

A combination of probability plots and degradation analysis was used to determine the potential risk of grade distortion from higher grade assays. A decile analysis was not performed since the goal of the study is to assess if grade capping is warranted. Due to the high CV in the raw assays, it was obvious that outlier control is required for this model. After conducting a careful examination of the data set, AGP elected to use a two-fold approach as noted below:

- apply a high hard cap on the raw assay prior to compositing to reduce extreme high-grade assays
- impose a sample search restriction on the "mild" outlier's population to control the range of influence

The grade capping strategy used has the benefit of limiting grade distortion from extreme outliers while restricting the range of influence of the "mild" high-grade outliers to prevent them from influencing blocks further away than the first pass search ellipsoid.

14.1.3.3.1 Raw Assay Capping

Tables 14.6 and 14.7 show a summary of the treatment of high-grade outliers. The cap value selected for gold and silver was generally above the 99th percentile of the raw assay distribution. The raw assay capping scenario for gold reduced the CV by approximately 50% on average for the Main Zone and C Zone. The CV of the gold and silver capped raw assays remains high for linear interpolation methods. Once that data was composited at 1.5 m (as described in Section 14.1.10) the CV was further reduced. A search restriction on mild outliers for silver was not implemented.

Table 14.6: Cap Levels for Gold & Search Restriction Grade Threshold by Domains

Domain (Domain Code)	Cap Level Au (g/t)	Total Number of Assay Affected	Total Number of Assays	Percent of Assays Affected (%)	Composite Grade Threshold Au (g/t)	Number of Composite Affected	Total Number of Composites	Percent of Composite Affected (%)
M Zone HG (1000)	125	8	1,193	0.67%	45	12	1,032	1.16%
M Zone MG (1500)	50	11	6,570	0.17%	20	14	4,807	0.29%
M Zone LG (1900)	20	11	7,119	0.15%	9	9	5,472	0.16%
C Zone HG (3000)	30	3	263	1.14%	11	8	3,08	2.60%
C Zone MG (3500)	20	5	3,698	0.14%	8	7	2,602	0.27%
C Zone LG (3900)	15	6	5,167	0.12%	6	14	4,038	0.35%
B Zone (2000)	13	13	1,846	0.70%	8	10	1,358	0.74%
D Zone (6000)	4	12	633	1.90%	Not Implemented			
E Zone (7000)	4	4	263	1.52%	Not Implemented			
H1-5 Zones (5x00)	7	19	3,281	0.58%	Not Implemented			
Total		116	30,033	0.39%				

Table 14.7: Cap Levels for Silver

Domain (Domain Code)	Cap Level Ag (g/t)	Total Number of Assay Affected	Total Number of Assays	Percent of Assays Affected (%)
M Zone HG (1000)	240	23	1,193	1.9%
M Zone MG (1500)	100	23	6,570	0.4%
M Zone LG (1900)	75	19	7,119	0.3%
C Zone HG (3000)	60	12	263	4.6%
C Zone MG (3500)	50	21	3,698	0.6%
C Zone LG (3900)	40	18	5,167	0.3%
B Zone (2000)	50	44	1,846	2.4%
D Zone (6000)	7	20	633	3.2%
E Zone (7000)	6	19	263	7.2%
H1-5 Zones (5x00)	40	21	3,281	0.6%
Total		255	30,033	0.8%

14.1.3.3.2 Search Restriction Threshold Grade & Range

The search restriction for mild gold outliers was applied to domains where the composite CV was above 2.0. For silver, a search restriction was deemed unnecessary.

The threshold grade used was selected based on degradation analysis of the composite data. The values used are shown in Table 14.8. The maximum range of influence for composites above the threshold was 70 m in the major direction and 20 m in the semi-major direction.

Table 14.8: CV Tracking between Assays & Composites by Domain for Gold & Silver

	Gold			Silver		
Domain (Domain Code)	CV Before Assay Capping	CV After Assay Capping	CV After Compositing	CV Before Assay Capping	CV After Assay capping (with Calc AG)	CV After Compositing
M Zone HG (1000)	4.6	2.7	2.1	3.9	2.6	2.3
M Zone MG (1500)	4.1	3.2	2.9	3.6	2.6	2.3
M Zone LG (1900)	5.4	3.6	2.7	6.1	2.7	2.0
B Zone (2000)	10.2	3.0	2.6	6.0	2.4	2.4
C Zone HG (3000)	2.6	1.7	1.4	2.5	1.7	1.5
C Zone MG (3500)	5.4	2.2	1.5	2.6	2.0	1.5
C Zone LG (3900)	6.6	3.2	2.4	6.9	2.1	1.6
H1 Zone (5100)	3.6	2.1	1.8	2.4	2.0	1.8
H2 Zone (5200)	2.9	2.1	1.9	2.7	2.2	1.7
H3 Zone (5300)	4.7	2.2	2.0	5.5	2.3	2.4
H4 Zone (5400)	3.2	1.9	2.0	3.7	2.0	2.1
H5 Zone (5500)	1.8	1.8	1.9	2.0	2.0	1.9
D Zone (6000)	4.1	1.5	1.3	4.0	1.0	0.9
E Zone (7000)	2.1	1.7	1.3	1.6	0.9	0.8

14.1.3.3.3 Total Metal Affected by the Treatment of Outliers

The total metal affected by the treatment of outliers was evaluated for gold in the final model. At the 0.2% Au cut-off, the outlier control strategy removed 16% of the gold ounces for the blocks within all mineralised wireframes in the measured and indicated categories (see Table 14.9). AGP notes only a very small percentage of the assays and composites were affected by the treatment of outliers, yet the amount of metal removed is deemed quite substantial.

Table 14.9: Metal Removed by Capping Strategy (Entire Model Measured + Indicated)

Grade Cut-off Bins Au (g/t)	Cumulative Gold Ounces Removed % Change
>1.9	-340,000 / -35%
>0.5	-334,000 / -20%
>0.2	-329,000 / -16%

14.1.3.4 Composites

The drill core was preferentially sampled in either 0.5, 1.0, or 1.5 m intervals. Within the mineralised domains, the core length average was 1.08 m.

From the sampling length statistics, AGP elected to use a composite length of 1.5 m. The composite size selected is above the third quartile to allow grade variations to be represented while reducing the variance.

Assays were length-weight averaged, and any grade capping was applied to the raw assay data prior to compositing. True gaps in sampling were composited at zero grade.

The 1.5 m composite intervals were created moving downward from the collar of the holes toward the hole bottoms. Composite lengths are automatically adjusted by the software to leave no remnants. The adjustment resulted in composite lengths ranging between 0.77 and 2.25 m, with mean and median of 1.5 m, and a standard deviation of 0.08. Tables 14.10 and 14.11 show the descriptive statistics for gold and silver capped composites within the various domains.

Composites were extracted to a point file and for the high-grade, medium grade and low-grade/waste component of the Main Zone and C Zone the composites domain code were back tagged from the probabilistic model.

Table 14.10: Gold Composite Statistics by Domains

Domain	ALL	H1-H5 Zones	Main Zone			B Zone	C Zone			D Zone	E Zone
			HG	MG	Low Grade /Waste		HG	MG	Low Grade /Waste		
Valid cases	22,605	2,843	1,956	4,305	4,785	1,342	608	2,524	3,614	457	171
Mean	0.67	0.37	3.10	0.58	0.29	0.42	1.73	0.47	0.27	0.41	0.38
Variance	6.27	0.49	52.89	2.60	0.60	1.20	7.15	0.45	0.45	0.28	0.25
Standard Deviation	2.50	0.70	7.27	1.61	0.77	1.10	2.67	0.67	0.67	0.53	0.50
Variation Coefficient	3.76	1.92	2.34	2.77	2.66	2.60	1.54	1.42	2.46	1.27	1.30
Minimum	0.00	0.00	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	88.86	7.00	88.86	49.66	15.61	13.00	24.67	10.63	12.49	4.00	2.83
1st Percentile	0.00	0.00	0.04	0.01	0.00	0.00	0.04	0.01	0.00	0.00	0.00
5th Percentile	0.01	0.00	0.15	0.05	0.01	0.00	0.11	0.04	0.00	0.00	0.01
10th Percentile	0.03	0.00	0.24	0.09	0.02	0.02	0.18	0.09	0.02	0.03	0.04
25th Percentile	0.08	0.03	0.53	0.20	0.05	0.07	0.37	0.20	0.05	0.10	0.09
Median	0.23	0.16	1.28	0.34	0.11	0.17	1.00	0.33	0.11	0.26	0.22
75th Percentile	0.52	0.37	2.48	0.60	0.24	0.35	1.89	0.54	0.26	0.51	0.45
90th Percentile	1.22	0.83	5.68	1.00	0.58	0.75	3.96	0.88	0.56	0.94	1.07
95th Percentile	2.18	1.48	11.13	1.37	0.99	1.30	6.14	1.22	0.94	1.26	1.47
99th Percentile	7.54	4.10	41.72	3.71	3.31	6.67	16.65	2.92	2.63	3.10	2.72

Table 14.11: Silver Composite Statistics by Domains

Domain	ALL	H1-H5 Zones	Main Zone			B Zone	C Zone			D Zone	E Zone
			HG	MG	Low Grade /Waste		HG	MG	Low Grade /Waste		
Valid cases	22,605	2,843	1,956	4,305	4,785	1,342	608	2,524	3,614	457	171
Mean	2.79	1.63	9.47	2.66	1.85	3.30	5.47	2.14	1.55	1.37	1.49
Variance	68.47	10.34	528.66	31.20	14.14	62.30	72.58	9.04	6.55	1.41	1.35
Standard Deviation	8.27	3.22	22.99	5.59	3.76	7.89	8.52	3.01	2.56	1.19	1.16
Variation Coefficient	2.96	1.97	2.43	2.10	2.03	2.39	1.56	1.40	1.65	0.87	0.78
Minimum	0.00	0.00	0.07	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	240.00	40.00	240.00	99.32	74.47	50.00	60.00	49.59	44.22	6.60	6.00
1st Percentile	0.00	0.00	0.50	0.48	0.00	0.00	0.39	0.23	0.00	0.00	0.00
5th Percentile	0.39	0.00	0.50	0.50	0.43	0.36	0.50	0.50	0.11	0.00	0.50
10th Percentile	0.50	0.00	0.98	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
25th Percentile	0.94	0.50	1.23	1.16	1.00	0.94	1.15	0.77	0.50	0.50	1.00
Median	1.16	1.16	2.35	1.17	1.16	1.16	2.00	1.17	1.01	1.16	1.16
75th Percentile	2.00	1.19	7.42	2.15	1.38	1.68	5.62	2.31	1.30	1.61	1.82
90th Percentile	4.78	2.73	20.51	5.03	3.00	5.53	14.44	4.37	2.83	3.16	3.00
95th Percentile	9.10	5.06	38.58	8.21	4.92	17.11	24.69	6.64	4.67	3.94	4.28
99th Percentile	32.62	17.35	140.57	27.97	18.61	48.85	41.42	14.51	14.07	5.75	6.00

14.1.3.5 Bulk Density

Treasury Metals provided 545 bulk density measurements. Core samples typically measuring 10 cm were analysed at the same laboratory used for the assays.

The 545 samples averaged 2.76 g/cm^3 with a median value of 2.75 g/cm3. There was a slight increase in density with the average gold grade, but it is very minor. In 3D the bulk density data is well distributed throughout the entire deposit. AGP elected to assign a base bulk density for each domain and then interpolate a bulk density to honour local variations. The density was interpolated only within the mineralised wireframes. Table 14.12 shows the base bulk density assigned to the domains. The interpolated bulk density relied on an inverse distance squared (ID^2) methodology carried out in a single pass using a minimum of two samples and maximum of 15 samples, and a maximum of three samples originating from a single drill hole. There was an insufficient amount of data points to interpolate the density in any other zone except the Main Zone and C Zone. The sample search ellipsoid was identical to the search ellipsoid for the third pass gold estimate. Using these parameters, 351,442 blocks were interpolated out of a total of 1,595,116 representing 22% of the blocks within the mineralised domains.

Table 14.12: Bulk Density by Domains

Domain (Domain Code)	Bulk Density (g/cm^3)
Main Zone HG (1000)	2.75
Main Zone MG (1500)	2.76
Main Zone LG (1900)	2.76
C Zone HG (3000)	2.77
C Zone MG (3500)	2.78
C Zone LG (3900)	2.76
B Zone (2000)	2.76
All Other Zones (4000-5000, 6000 and 7000)	2.78
Waste outside the Wireframes	2.75
Overburden	1.75

14.1.3.6 Spatial Analysis – Variography

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity, of a variable with separation distance and direction. If we compare samples that are close together, it is common to observe their values are quite similar. As the distance between samples increases, there is likely to be less similarity in the values. The experimental variogram mathematically describes this process. It is commonly represented as a graph that shows the variance in measurements with distance between all pairs of sampled locations.

In all semi-variograms, the distance where the model first flattens out is known as the range. Sample locations separated by distances closer than the range are believed to be spatially auto-correlated. The sill is the value on the Y-axis where the model attains the range, while the nugget is the value at the location where the model intercepts the Y-axis. The nugget typically represents variation at a micro scale that can be attributed to measurement errors, sources of variation at distances smaller than the sampling interval, or both. Therefore, the shape of the semi-variogram describes the pattern of spatial continuity. A very rapid decrease near the origin indicates short-scale variability. A more gradual decrease moving away from the origin suggests longer-scale continuity.

Various semi-variogram types exist. Using SAGE™ software, experimental correlograms for gold and silver were computed for the various domains.

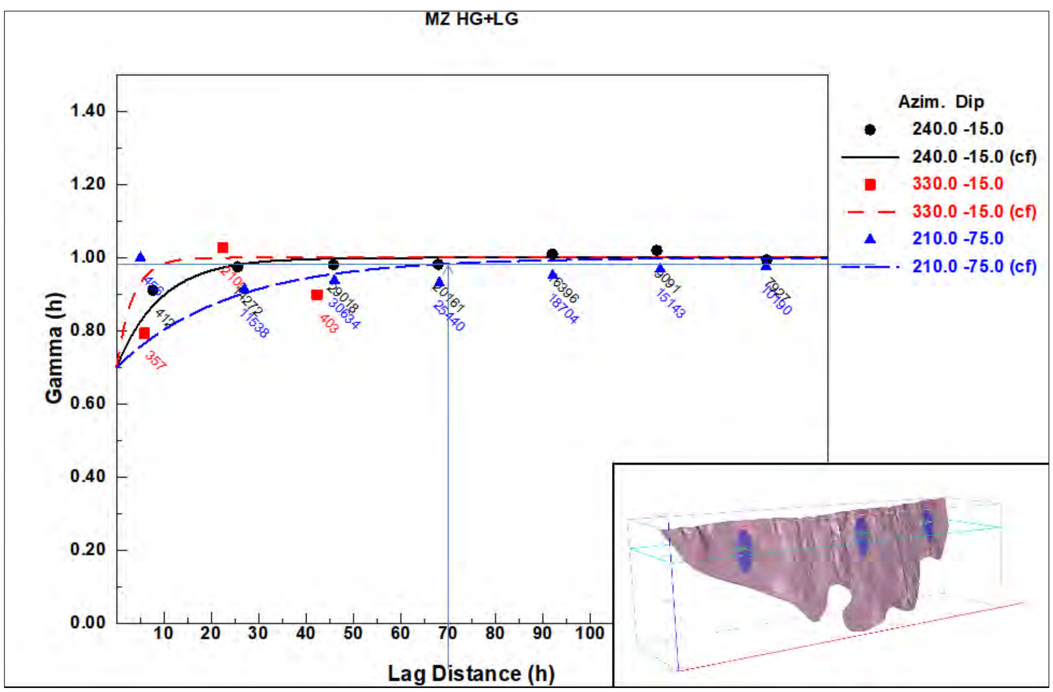
The resulting anisotropy models generated were visually inspected in GEMS to ensure the ellipsoid model corresponded well with the expected orientation of the deposit.

For gold, the effective range at 97% of the sill along the apparent plunge of the mineralisation is approximately 70 m. The nugget is very high, between 70% to 84% of the sill value. At 100% of the sill, the maximum range is estimated to be between 150 and 170 m. The definition of the variogram, near the origin, is good when the lag distances are adjusted to the drill angle. Figure 14-8 illustrates one example of a final variogram model for the Main Zone, along with a plan view of the ellipsoid generated by GEMS software (see Figure 14-8 inset image). The direction and plunge represented by the variogram coincide with the known interpreted plunge of the mineralisation. The variography is considered representative of the trend of the mineralisation. As a result, AGP elected to interpolate the grade model using ordinary kriging.

Variograms for the H1-5 Zones, B Zone, D Zone, and E Zone were attempted, but not used due to lack of data pairs.

Silver variograms for the Main Zone and C Zone show a similar range.

Figure 14-8: Example Main Zone Variogram



Source: AGP (2020).

Table 14.13 lists the variogram parameters used in the model for gold and silver. The variograms were fitted using the GEMS "Z-X-Z" rotation method which is dependent of the block model orientation. The GEMS Z,X,Z rotation convention uses a right hand rule for all three axis. The C1 traditional exponential range in Table 14.13 is defined as Gam(3R)=0.95 *sill as defined by the first edition of GSLIB (Deutsch and Journel).

Table 14.13: Variogram Parameters

Domain Code	Element	Model	Nugget	C1	ZXZ (degree)	C1 Range (m)
Main Zone (1000, 1900)	Gold	Exponential	0.70	0.30	13, 72, -8	9.5, 58.1, 3.3
C Zone (3000, 3900)	Gold	Exponential	0.845	0.155	19, 80, 59	51.2, 18.1, 3.0
Main Zone (1000,1900)	Silver	Exponential	0.794	0.206	2, 72, -6	44.5, 143.3, 13.1
C Zone (3000,3900)	Silver	Exponential	0.807	0.193	12, 85, -68	59.1, 214.5, 20.5

14.1.4 Block Model

14.1.4.1 Search Ellipsoid Dimension & Orientation

While it is common to use the variogram model as a guide to set the search ellipsoids' ranges and attitudes, the geologist modelling the deposit must consider the strike and dip of the mineralised horizon, and the drill hole spacing and distribution. For this model, AGP used the overall geometry as confirmed by the variography as a guiding principle to set the search ellipsoid orientation. AGP also took into consideration the change in azimuth from east to west.

The first pass maximum range was sized to reach at least the next drill section. The maximum range for the second interpolation pass was set slightly below the maximum range of the variogram. Lastly, the third interpolation pass typically exceeded the maximum range displayed by the variograms. The ratio between the major and semi-major axis was kept within reasonable limits from the information derived from variography.

The search ellipsoid orientation was adjusted via the use of subdomains in order to optimise the alignment of the search volume with the mineralisation. Table 14.14 lists the final values used in the resource model for the range of the major, semi-major, and minor axes. Rotation angles are based on the GEMS ZXZ methodology, which uses a conventional right-hand rule and Table 14.15 lists the rotation angles for the various sub-domains.

The search ellipsoids dimension and orientation applied for both the gold and silver interpolation plan.

Table 14.14: Search Ellipsoid Dimensions & Orientation

Domain Code	ZXZ (degrees)	Pass 1 (m)	Pass 2 (m)	Pass 3 (m)
Main Zone (1000-1900)	Based on Sub-domains	75, 31, 10	135, 55, 15	229, 94, 25
C Zone (3000, 3900)	Based on Sub-domains	77, 23, 10	123, 37, 15	197, 59, 25
All Other zones	Based on Sub-domains	76, 27, 10	129, 46, 15	213, 77, 25

Table 14.15: Search Ellipsoid Orientation Sub-Domains

Sub-Domains	Main Zone GEMS (Z,X,Z)	C Zone GEMS (Z,X,Z)	All Other Zones GEMS (Z,X,Z)
Sub1	9, 70, 75	40, 65, 64	40, 65, 64
Sub2	20, 73, 75	29, 67, 64	29, 67, 64
Sub3	3, 72, 75	20, 69, 64	20, 69, 64
Sub4	7, 74, 75	14, 69, 64	14, 69, 64
Sub5	7, 69, 75	8, 71, 64	8, 71, 64
Sub6	8, 71, 75	7, 72, 65	7, 72, 65
Sub7	12, 76, 75	2, 69, 65	2, 69, 65
Sub8	14, 72, 75	N/A	N/A
Sub9	23, 71, 75	N/A	N/A
Sub10	4, 77, 75	N/A	N/A

14.1.4.2 Block Model Matrix

The block model was constructed using GEMS 6.8™. An elongated block size of 5 m along the strike of the deposit (horizontally) x 5 m vertically x 2 m across was selected based on mining selectivity considerations and the density of the dataset. This block matrix size assumed a mid-size open pit operation that would also be suitable for long hole underground operation.

The block model was defined on the project coordinate system with a 0-degree rotation.

Table 14.16 lists the upper southeast corner of the model and is defined on the block edge.

The final domain codes controlling the interpolation were coded by adding the lithological code with the HG, MG, and low-grade/waste probabilistic code and then the sub-domain code. For example, a block in Main Zone with a probability of being low grade or waste in the middle of the deposit would receive the code 1000 (Main Zone) + 900 (Low Grade/Waste Probability) + 4 (Orientation Sub-Domain Code).

Table 14.16: Block Model Definition (Block Edge)

Resource Model Items	Parameters
Easting	526,050
Northing	5,511,500
Top relative elevation (true elevation + 1,000 m)	1,500
Rotation angle (counter clockwise)	0
Block size (X, Y, Z in metres)	5 x 2 x 5
Number of blocks in the X direction	638
Number of blocks in the Y direction	618
Number of blocks in the Z direction	182

14.1.4.3 Interpolation Plan

The resource model was created in GEMS 6.8™ with a single folder setup, using ordinary kriging (OK) for interpolating the gold and silver grade for Main Zone and C Zone. The remaining zones were interpolated using ID^2 with anisotropic weighting. A nearest neighbour

(NN) model and inverse distance to the power of three (ID^3) were also interpolated to be used for validation. The interpolation was carried out in a multi-pass approach, with an increasing search dimension coupled with decreasing sample restrictions as noted below:

- Pass 1 used an ellipsoid search with 8 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of three holes to be used in the search.
- Pass 2 used an ellipsoid search with 6 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of two holes to be used in the search.
- Pass 3 used an ellipsoid search with 3 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, allowing a block to be interpolated with composites originating from one hole.

The wireframe boundaries were considered hard for all zones. The internal subdivisions between the HG, MG and LG/Waste for the Main Zone and C Zone are considered by AGP as soft due to the probabilistic approach that was used to separate the material, not because composites from one domain could use the composite from the adjoining domain.

The mineralised wireframes for the D Zone, E Zone, B Zone, and H1 to 5 Zones are split in two to six pieces. The interpolation plan did not differentiate between the various pieces since they are coded with a single rock code. This means for example, that composites from the D Zone (a) a portion could be used for the estimation of the D Zone, (b) a portion of the search ellipsoid was large enough to include composites from both parts.

14.1.4.4 Block Model Validation

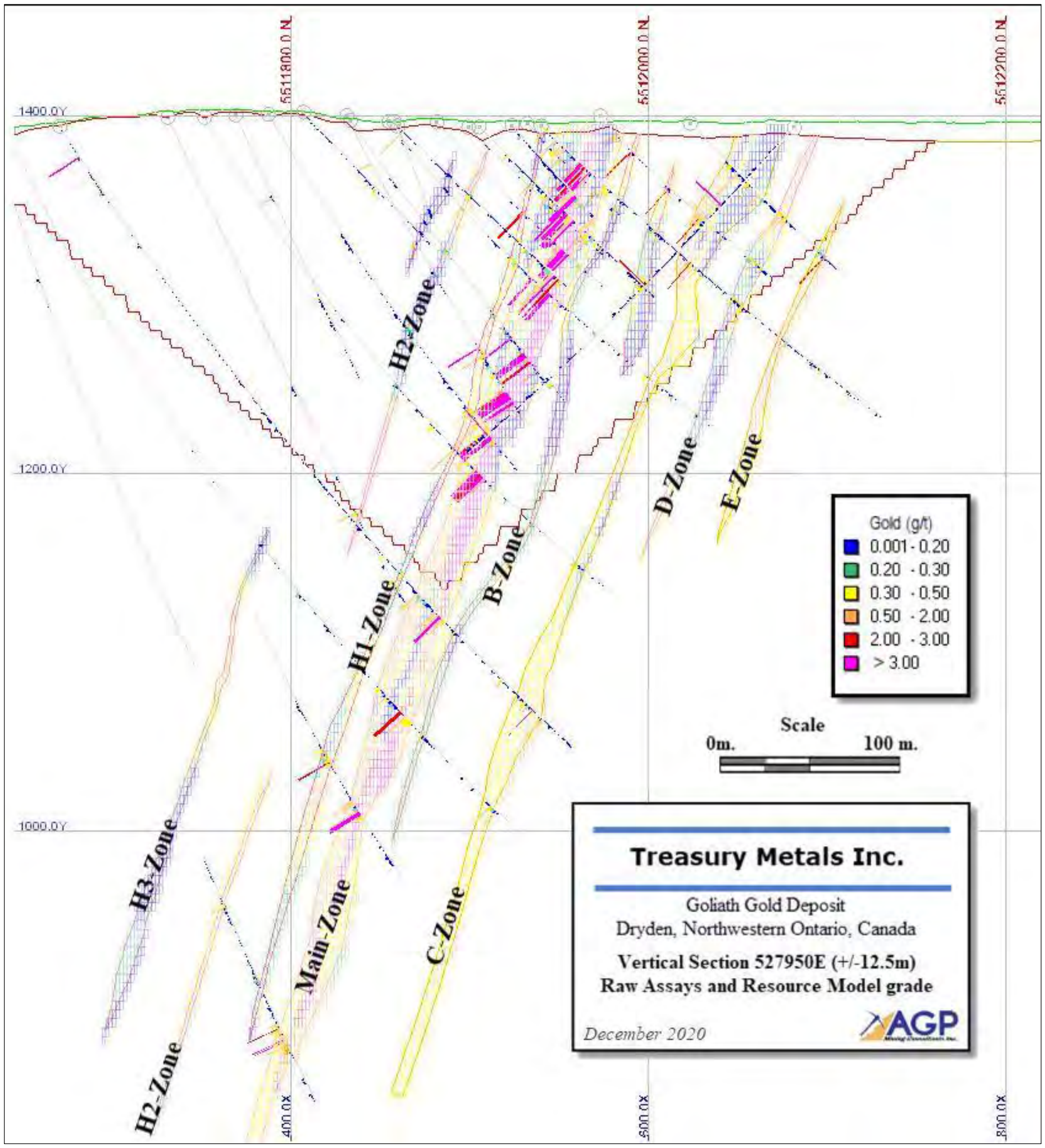
The Goliath deposit grade models were validated by five methods:

- visual comparison of colour-coded block model grades with composite grades on sections and plans
- comparison of the global mean block grades for OK/ID^2, ID^3, NN models, composite, and raw assay grades
- comparison using grade profiles to investigate local bias in the estimate
- naïve cross-validation tests with composite grade versus block model grade
- comparing the resource estimate against the Teck Exploration underground bulk sample

14.1.4.4.1 Visual Comparison

The visual comparison of block model grades on sections and plans indicated a good correlation between drill hole grade and resource model grade (see Figure 14-9), especially in the high-grade portion of the model near the old Teck bulk sampling area.

Figure 14-9: Gold Grade Model Distribution



Source: AGP (2020).

14.1.4.4.2 Global Comparison

Table 14.17 shows the grade statistics for the raw assays, composites, NN, ID^3, and OK/ID^2 models for all zones in the measured, indicated, and inferred categories. Statistics for the gold and silver composite mean grades compare well to the raw assay grades, with a normal reduction in values due to smoothing, related to volume variance. The block model mean grade, when compared against the composites, showed a normal reduction in values. More importantly, the grade of the NN, ID^3, and OK/ID^2 models are less than 2% of each other for gold and less than 5% of each other for silver, indicating the methodology used did not introduce a bias into the estimate.

Table 14.17: Global Comparisons (Measured, Indicated & Inferred)

Methodology	Au (g/t) @ > 0.0 cut-off (Class 1-3)	Ag (g/t) @ > 0.0 cut-off (Class 1-3)
Raw assays uncapped at 0.0 Cut-off (clustered/declustered)	0.868 / 0.769	3.2 / 3.3
Composite capped at 0.0 Cut-off (clustered/declustered)	0.666 / 0.604	2.8 / 2.3
Nearest neighbour (NN)	0.597	2.4
Inverse distance squared using true distance (ID^3)	0.591	2.3
Ordinary kriged (Main and C) + ID^2 (All others)	0.594	2.3

14.1.4.4.3 Local Comparison – Grade Profiles

Comparison of the grade profiles (swath plots) of the raw assay, composites, and estimated grades allow for a visual verification of an over or under estimation of the block grades at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of three swath plots, generated at 50 m intervals in the X direction, 50 m in the Y direction, and 25 m vertically.

The OK/ID^2 and ID^3 estimates should be smoother than the NN estimate; the NN estimate should fluctuate around the OK/ID^2 and ID^3 estimates on the plots or display a slightly higher grade. The composite line is generally located between the assay and the interpolated grade. A model with good composite distribution should show very few crossovers between the composite and the interpolated grade line on the plots. In the fringes of the deposit, as composite data points become sparse, crossovers are often unavoidable. The swath size also controls this effect to a certain extent; if the swaths are too small, fewer composites will be encountered, which usually results in erratic lines on the plots.

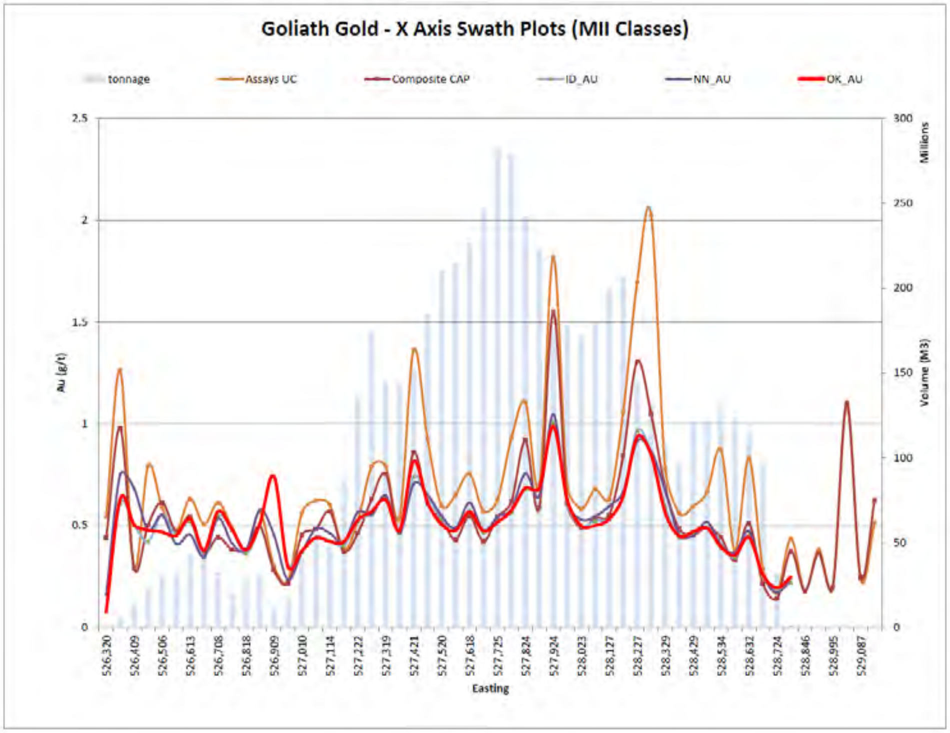
The peaks and valleys on the assay and composite lines are well represented, but more subdued in the resource model due to smoothing. The effect of capping the assays is readily visible in the plots, and the search restrictions on the mild outliers appear to have normalised the grade.

In general, the swath plots show good agreement, with the three methodologies showing no major local bias. The Z-Chart shows some minor overestimation below 760 m elevation where the OK/ID^2 and the ID^3 check model shows an over estimation of the grade when compare to the raw assays and composites. The area in question was investigated and was found to have low volume and poor composite support. The area is supported by only 271 composites in 22 drill holes compared to the 22,334 composites in 720 drill holes above that elevation. Ninety percent of the blocks below 760 m elevation are either coded as inferred or were interpolated but not bearing a resource classification (Code 4). AGP concluded that the chart only highlights the need for additional drilling at depth.

Charts for the silver grade shows the same general pattern although the area on the Z-chart below 760 m has not been less affected due to the lower silver variability of the composites.

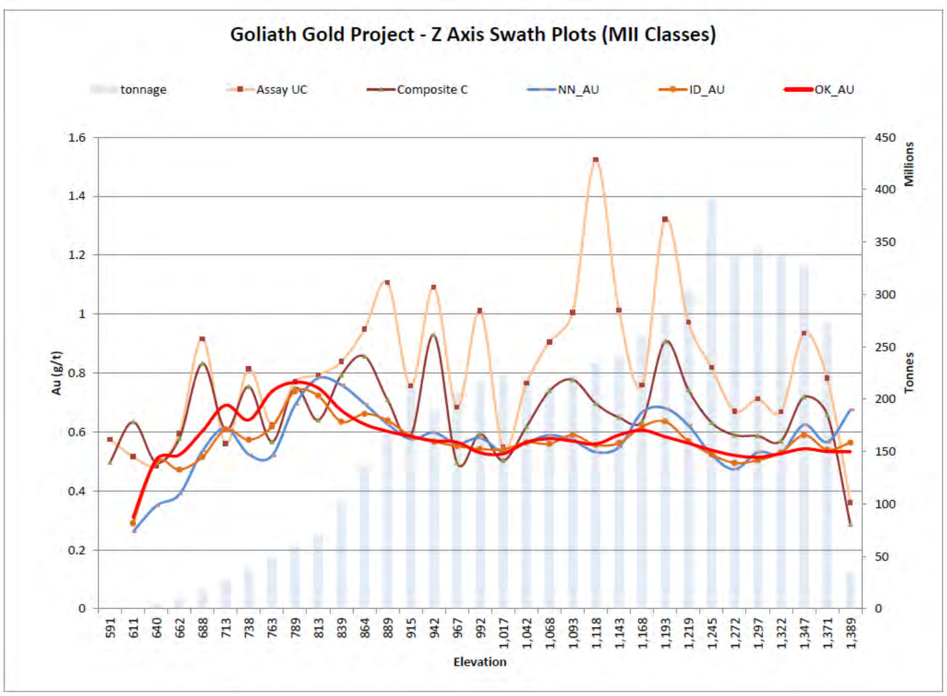
Grade profiles for gold are presented in Figures 14-10 and 14-11. The profile for the Y chart was omitted because it is viewing down the strike of the deposit and is not the best viewing direction.

Figure 14-10: X-Axis Grade Profile



Source: AGP (2020).

Figure 14-11: Z-Axis Grade Profile



Source: AGP (2020).

14.1.4.4.4 Naïve Cross-Validation Test

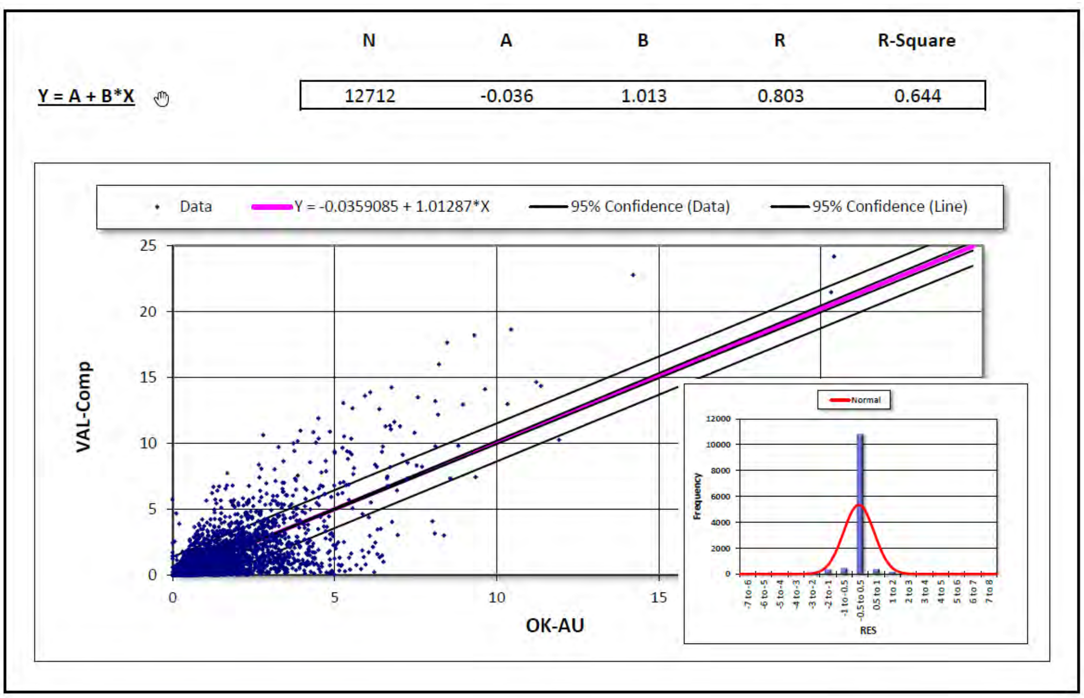
A comparison of the average grade of the composites within a block with the estimated grade of that block provides an assessment of the estimation process close to measured data. Pairing of these grades on a scatter plot gives a statistical valuation of the estimates. This methodology is distinct from “jack-knifing” which replaces a composite with a pseudo-block at the same location and evaluates and compares the estimated grade of the pseudo-block against that of the composite grade.

With the naïve cross validation test, it is anticipated the estimated block grades should be similar (while not exactly the same value) to the composited grades within the block. This is especially true with deposits bearing a higher nugget component.

A high correlation coefficient (R2) indicates satisfactory interpolation process results, while a medium to low correlation coefficient indicates larger differences in the estimates, or a low data density, which would suggest a further review of the interpolation process. Results from the pairing of the composited and estimated grades within blocks pierced by a drill hole are presented in Figure 14-12. Following the removal of 84 outliers (out of 12,796 pairs in the measured, indicated, and inferred categories), the R2 value is considered moderate for a gold deposit, at 0.64 R2.

The regression residuals are the differences, on a case-by-case basis, between the actual Y values and the values calculated by the best-fit equation. These can be evaluated for normality and randomness. The inset image in Figure 14-12 shows the residual distribution. The chart shows a normal distribution with very small positive bias.

Figure 14-12: Naïve Cross Validation Test Results (MII blocks)



Source: AGP (2020).

14.1.4.5 Comparing the Resource Estimate against the Teck Underground Bulk Sample

Teck conducted an underground exploration and bulk sampling program in 1998. The result of the program was documented in a report titled "Report on the 1998 underground exploration and bulk sampling program, Thunder Lake West Project, Zealand Township, Northwestern Ontario" authored by R. Page, R. Stewart, P. Waque, C. Galway and dated April 21, 1999.

Teck collected the bulk samples in areas that exceeded 3.0 g/t gold. On the west side of the ramp, three areas were included in the bulk sample. On the east side, three low-grade areas, one high-grade area, and one take-down-back were included in the bulk sample. Table 14.18 shows a summary of the bulks sample tonnage and grade recovered from Teck.

Table 14.18: Teck Bulk Sample

Drift & Rounds	Area (m^2)	Height (m)	Calculated Volumes (m^3)	Avg. SG	Calculated Tonnes	Face Sample Grade PMA Au (g/t)	Actual Measured Tonnes	Final Grade Au (g/t)
B-West	65.5	3.30	216	2.75	594	4.61	636	3.57
A-East LG	95.4	3.05	291	2.88	837	6.30	865	7.46
A-East HG	51.9	3.30	171	2.85	488	35.10	447	16.80
A-East TDB	61.7	1.80	111	3.00	333	23.00	388	12.70
Total			789	2.85	2252	14.12	2336	9.05

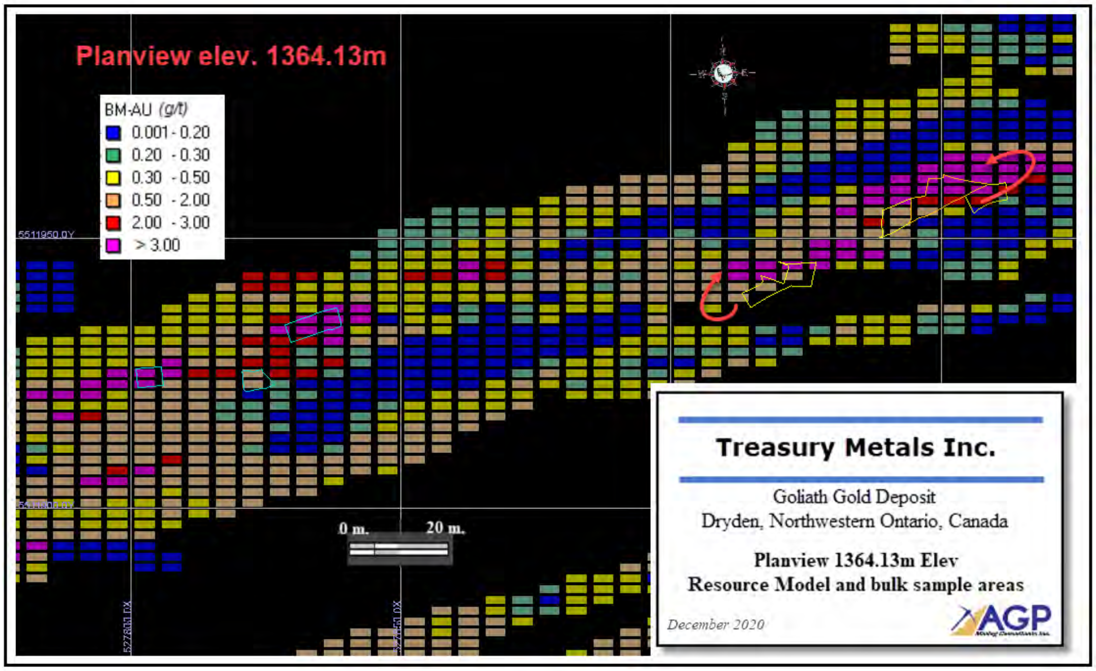
The excavated bulk sample areas were digitised in GEMS and evaluated against the resource model. During this process, AGP found that the location of the high-grade zones in the resource model for the east areas were offset by approximately 3-4 m north from the actual location of the underground excavation as digitised in GEMS. This may be due to the orientation (strike and dip) of the sample search ellipsoids controlling both the grade interpolation and the probabilistic model defining the high-grade and low-grade/waste domains. The sample search ellipsoids can only describe an average direction within the orientation sub-domain. Alternately, there could be a rotation issue of the underground excavations (see Figure 14-13).

In order to validate the grade in the excavations provided, a new set of shapes were created for the east drift. The shapes were displaced by 4 m north for the East LG #2, #3, East HG and TDB. The East LG #1 shape was displaced by 3 m. The grade of the resource model was then re-evaluated with the new shapes. Table 14.19 shows the resource within the bulk sample excavated shapes.

From Table 14.19, the following conclusions could be drawn:

- The digitised bulk sample shapes indicated a total of 771 m^3 of material versus 789 m^3 excavated by Teck, suggesting the digitised shapes are slightly smaller than actual.
- The total tonnage from the shapes in GEMS is 2,129 tonnes vs. 2,336 tonnes as reported by Teck. This partially due to the smaller shapes and the bulk density.
- Average density in the GEMS resource model indicated 2.76 t/m^3 versus an average of 2.85 t/m^3 in the Teck report. This suggests more density measurements should be collected.

Figure 14-13: Plan View at Elevation 1364.13



Source: AGP (2020).

Table 14.19: Resource Model within Bulk Sample Shapes

Excavation	Volume (M**3)	Density (t/m**3)	Tonnage (t)	AU (g/t)	Gold (oz)	AG (g/t)	Silver (oz)
BULK-AREA A-EAST-LG 3-(Adjusted)	72	2.74	197	9.81	62	29.9	190
BULK_AREA A-EAST-HG 1-(Adjusted)	172	2.74	470	18.95	286	42.5	641
BULK_AREA A-EAST-LG 1-(Adjusted)	128	2.77	356	7.72	88	12.6	144
BULK_AREA A-EAST-LG 2-(Adjusted)	70	2.77	193	8.21	51	19.9	123
BULK_AREA B-WEST 1 (as digitised)	52	2.82	145	3.38	16	8.3	39
BULK_AREA B-WEST 2 (as digitised)	47	2.86	134	1.17	5	4.1	18
BULK_AREA B-WEST 3 (as digitised)	106	2.75	291	4.03	38	6.7	63
BULK_AREA TDB 1 (as digitised)	125	2.74	343	13.47	148	31.7	349
Total	**771**	**2.76**	**2,129**	**10.15**	**695**	**22.9**	**1,567**

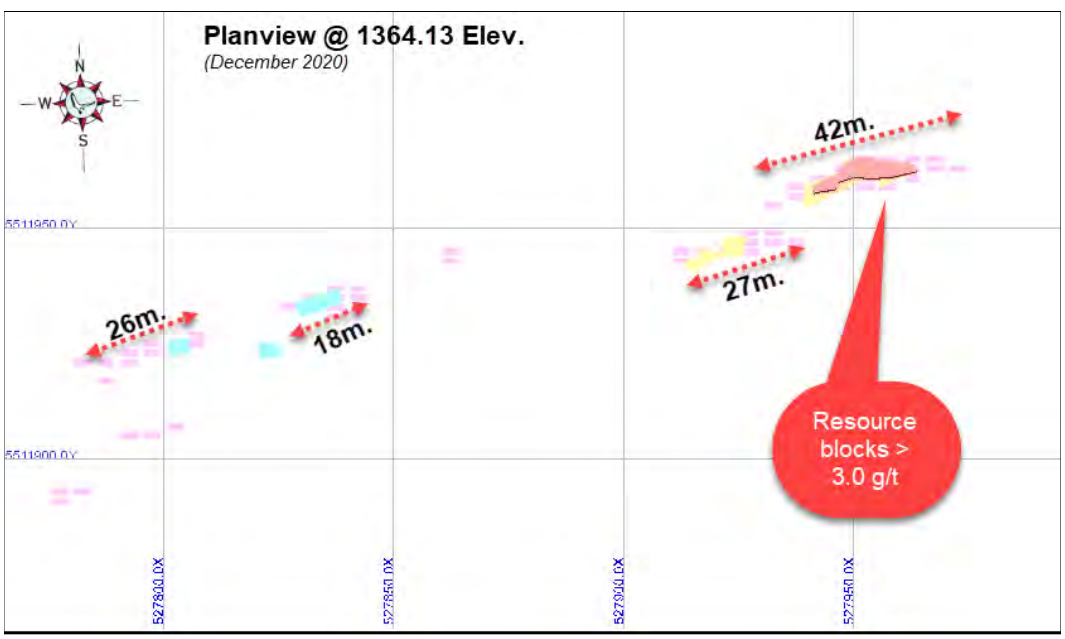
- The resource model reported 2,129 tonnes grading at 10.15 g/t Au and Teck reported 2,336 tonnes grading 9.05 g/t Au for all excavated shapes.
- The resource model reported 813 tonnes grading 16.64 g/t Au for the east high-grade and TDB areas and Teck reported 835 tonnes grading 14.89 g/t Au for the same areas.
- According to the Teck report, the average grade of the face samples within the bulk mining shaped averaged 14.12 g/t Au which is higher than the bulk sample grade of 9.05 g/t indicating a certain amount of dilution may have been sent with the bulk sample.

Teck reported that the strongest gold mineralisation is localised in siliceous quartz-sericite schist containing disseminated sulphides, sulphide veins, and sulphide-mineralised quartz veins with rare coarse gold/electrum. Teck also added the more significant mineralised areas are in contact with units of dark coloured intermediate quartz porphyry.

Teck reported the distribution of alteration and mineralisation, outlined by surface drilling, correlated reasonably well with the results of the underground program. However, Teck found the strike length of the best continuous mineralisation decreases from the expected 50 to 60 m down to 20 m. It was also noted the grade of the bulk sample extracted was lower than what was calculated from face and muck samples. This represents a risk in the model since the average drill spacing is about 23 m, therefore high-grade mineralised zones could conceivably be hidden between two sections or alternately, the predicted length of the high-grade zone could be exaggerated by the wireframe describing the mineralisation since they are interpreted on a 25 m section set.

AGP notes the length of the mineralised zones (blocks > 3.0 g/t) in the area of the bulk samples from the resource model indicated zones reaching 42 m in length as illustrated by Figure 14-14.

Figure 14-14: High-Grade Mineralised Zones in Bulk Sample Areas



Source: AGP (2020).

14.1.5 Mineral Resources – Goliath

Effective December 16, 2020, AGP completed an update of the July 1, 2019 estimate completed by P&E Mining Consultants Inc. for the Goliath deposit located near the municipalities of Dryden, Ontario. The mineral resource presented herein is in conformance with the CIM mineral resource definitions referred to in N.I. 43-101 Standards of Disclosure for Mineral Projects. The estimate takes into account all data that was available prior to October 6, 2020.

The resource estimate consists of a combination of measured, indicated, and inferred resources. Based on current exploration drilling data, the bulk of the mineralisation is carried by the Main Zone and C Zone. The Main Zone and C Zone are two elongated zones striking in excess of 2.4 km. Within the Main Zone, exploration drilling identified three high-grade shoots displaying a steep plunge to the west. Within the C Zone only one shoot was identified.

From the geometry described, the deposit is amenable to open pit extraction followed by a potential underground operation, likely using a long-hole or modified Avoca mining method, with or without backfill.

14.1.5.1 Classification of Mineral Resources

Several factors are considered in the definition of a resource classification:

- Canadian Institute of Mining (CIM) requirements and guidelines
- experience with similar deposits
- spatial continuity
- confidence limit analysis
- geology

No environmental, permitting, legal, title, taxation, socioeconomic, marketing, or other relevant issues are known to the author that may currently affect the estimate of mineral resources. Mineral reserves can only be estimated based on an economic evaluation used in a pre-feasibility or feasibility study of a mineral project. Thus, no reserves have been estimated. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.

Typically, the confidence level in the block model is reduced with the increase in the search ellipsoid size, along with the diminishing restriction on the number of samples used for the grade interpolation. This is essentially controlled by the pass number of the interpolation plan. A common technique is to categorise a model based on the pass number and the average distance to the composites.

Table 14.20 lists the parameters used to code the classification model. For the Goliath deposit, in addition to using the pass number and the average distance to the composites, AGP adjusted the classification based on several other factors such as kriging efficiency and proximity to surface/underground exposures and areas that were considered sufficiently drilled as define by a cluster of overlapping drill hole traces expanded to 25 m diameter.

Table 14.20: Primary Classification Parameters

Pass Number	Retained As	Downgraded To
Pass 1	Measured if average distance to the composites < 25 m	Indicated if the average distance to composites is ≥ 25 m and < 70 m
Pass 2	Indicated if average distance to composites is < 70 m	Inferred if average distance to composites is ≥ 70 m and <120 m
Pass 3	Inferred if average distance to composites is < 120 m	Code 4 if average distance to composites is ≥120 m.

The following adjustments were made to the primary classification:

- To ensure no measured blocks existed outside the tightly drilled areas; these were downgraded to indicated.
- To ensure all blocks within the core area received a category, Code 4 blocks were upgraded to inferred. Outside the core areas indicated blocks were downgraded to inferred and inferred blocks were downgraded to Code 4.
- To account for reduce data support, manually assigned inferred category to H3, H4, H5, and E Zones.
- To ensure that blocks within the helper wireframe received an inferred category; measured and indicated blocks were downgraded.
- The model was then groomed to eliminate isolated blocks. This was accomplished by using a GEMS™ Cypress-enabled script that adjusts, or grooms, the classifications of isolated blocks by upgrading or downgrading the classification of the mid-block depending on the classifications of the 26 surrounding blocks.

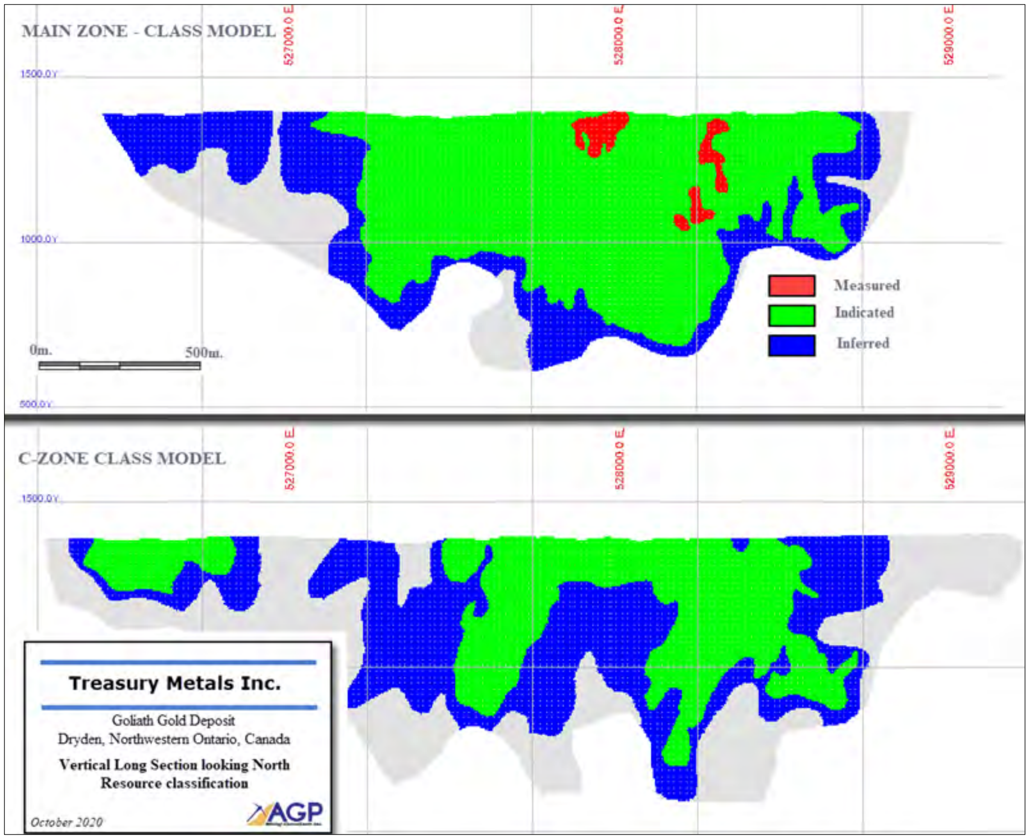
AGP validated the final block classification visually and did a final manual adjustment for Main Zone and C Zone by creating on incline sections a measured, indicated, and inferred resource shape that was based on the output from the groomed model.

Three confidence categories now exist in the model. The CIM guideline classes of measured, indicated, and inferred are coded 1, 2 and 3, respectively. A special Code 4 represents mineralisation that was considered too far away from the existing drilling to be classified as an inferred resource. The Code 4 blocks have been left in the classification model solely to assist Treasury Metals in its exploration activity. Figure 14-15 illustrates the block model classification of the Goliath deposit for Main Zone and C Zone.

14.1.5.2 Marginal Cut-off Grade for Mineral Resources

Under CIM definitions, mineral resources should have a reasonable prospect of economic extraction. A gold price of US$1,700/oz and a silver price of US$23/oz was used for the cut-off determination. For open pit resources, a cut-off of 0.25 g/t gold was used. Resources below the open pit shell used a cut-off of 1.60 g/t gold to define possible underground resources. The economic calculation to support this estimate is provided in Table 14.21 below.

Figure 14-15: Model Classification



Source: AGP (2020).

Table 14.21: Breakeven Cut-off Grade for Resource

Goliath Project	Unit	Gold	Silver
World Price	US$/oz	$1.700	$23.00
Payables	%	99.8%	97%
Refining, transportation	US/oz	$5.00	$0.00
Royalty	%	1.5%	1.5%
Net Price	US$/oz	$1,645.91	$21.81
	C$/oz	$2,189.05	$29.01
	Unit	**Open Pit**	**Underground**
Mining	C$/t moved base cost at 410 level	$2.14	$77.00
	C$/t moved increment/5 m bench below 410 level	$0.03	-
Milling	C$/t mill feed	$14.92	$14.92
G&A	C$/t mill feed	$1.62	$1.62
Process Recovery			
Gold	%	95.5%	95.5%
Silver	%	62.6%	62.6%
Dilution considered for cut-off	%	0%	15%
Breakeven Cut-off	g/t Au	0.25	1.60

14.1.5.3 Reasonable Prospects for Eventual Economic Extraction

14.1.5.3.1 Mineral Resources Amenable to Open Pit Extraction

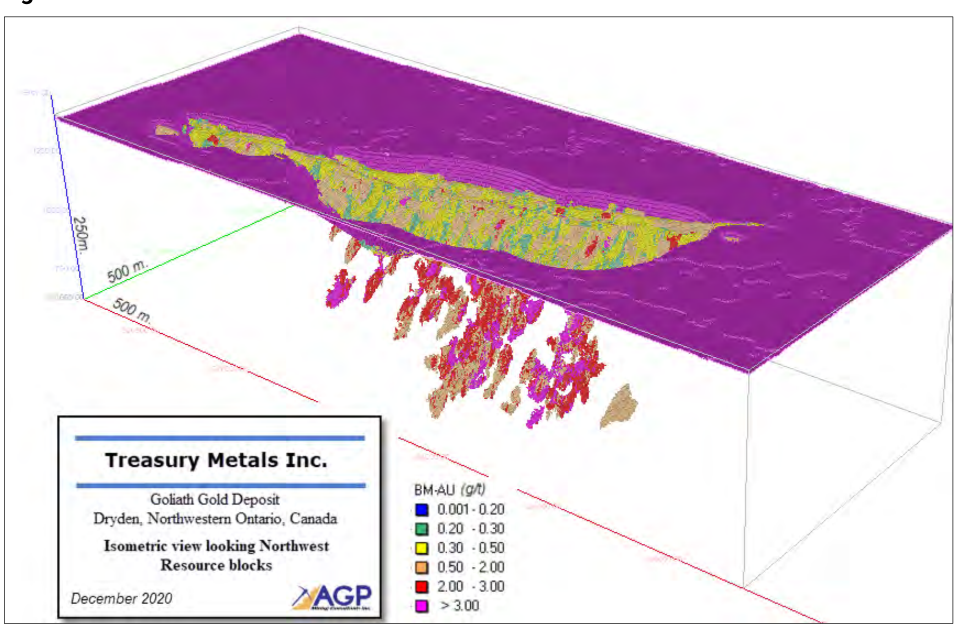
To further assess reasonable prospects of economic extraction, a Lerchs-Grossman optimised shell was generated to constrain the potential open pit material. Parameters used to generate this shell included:

- average of 43.6° overall slopes for the pit shell
- operating costs of:
  - mining: C$2.14/t mined with a base level of 410 masl and an incremental cost of C$0.03/t below the 410 level
  - milling: C$14.92/t mill feed
  - general & administrative: C$1.62/t mill feed
  - assumes a 5000 t/d operation
- recoveries: gold 95.5%, silver 62.6%
- metal prices: gold $1,700/oz, silver $23/oz
- above criteria was applied to measured, indicated, and inferred materials

14.1.5.3.2 Mineral Resources Amenable to Underground Extraction

Grade shells at the underground cut-off of 1.6 g/t Au were generated beneath the resource pit shell. The grade shells were examined for the likelihood of being a coherent mining shape with reasonable prospect of being accessed. Those that did not meet the criteria were removed from consideration. Only those shapes considered reasonable to form underground stopes were included in the resource estimate (Figure 14-16).

Figure 14-16: Resource Blocks



Source: AGP (2020).

14.1.5.4 Mineral Resources Tabulation

Within the resource constraining shell, at the greater than 0.25 g/t Au cut-off selected, the updated model returns a total of 1.5 million measured tonnes grading at 1.90 g/t Au and 6.7 g/t Ag containing 89,800 oz of gold and 316,700 oz of silver. Indicated tonnes amounted to 27.0 Mt grading at 0.87 g/t Au and 3.0 g/t Ag containing 757,000 oz of gold and 2.6 Moz of silver. The total measured and indicated resources within the constraining shell amounted to 28.4 Mt grading at 0.93 g/t Au and 3.2 g/t silver containing 846,800 oz of gold and 2.9 Moz of silver.

Below the constraining shell and reported at a greater than 1.6 g/t Au, the updated model returns 98,000 tonnes of measured resources grading at 4.94 g/t Au and 20.8 g/t Ag containing 15,500 oz of gold and 65,300 oz of silver. Indicated resources amounted to 2.6 Mt grading 3.16 g/t Au and 7.6 g/t Ag containing 263,100 oz of gold and 632,700 oz of silver. The total measured and indicated resources below the constraining shell amounted to 2.7 Mt grading at 3.22 g/t Au and 8.1 g/t Ag containing 278,700 oz of gold and 698,000 oz of silver.

Inferred resources within the resource constraining shell and reported at greater than 0.25 g/t Au, amounted to 3.6 Mt grading at 0.65 g/t Au and 2.1g/t Ag containing 76,100 oz of gold and 247,000 oz of silver.

Below the constraining shell and reported at a greater than 1.6 g/t Au cut-off, the updated model returned 704,000 tonnes of inferred resources grading at 2.75 g/t Au and 5.6 g/t Ag containing 62,200 oz of gold and 125,900 oz of silver.

14.1.5.4.1 Goliath Total Mineral Resources

The Goliath deposit total measured resources amounted to 1.6 Mt grading at 2.09 g/t Au and 7.58 g/t Ag containing 105,300 oz of gold and 382,000 oz of silver. Indicated resources amounted to an additional 29.5 Mt grading 1.07 g/t Au and 3.39 g/t Ag containing 1.0 Moz of gold and 3.2 Moz of silver. The total measured and indicated resources amounted to 31.1 Mt grading at 1.13 g/t Au and 3.60 g/t Ag containing 1.1 Moz of gold and 3.6 Moz of silver. Inferred resources added an additional 4.3 Mt grading 0.99 g/t Au and 2.67 g/t Ag containing 138,300 oz of gold and 372,900 oz of silver (see Table 14.22).

Table 14.22: Goliath Resource Statement Effective December 16, 2020

Area	Category	Cut-off (g/t Au)	Tonnes	Au (g/t)	Gold (oz Au)	Ag (g/t)	Silver (oz Ag)
Resource amenable to open pit extraction	Measured	0.25	1,471,000	1.90	89,800	6.7	316,700
	Indicated		26,956,000	0.87	757,000	3.0	2,591,400
	Measured + Indicated		28,426,000	0.93	846,800	3.2	2,908,100
	Inferred		3,644,000	0.65	76,100	2.1	247,000
Resource amenable to underground extraction	Measured	1.6	98,000	4.94	15,500	20.8	65,300
	Indicated		2,592,000	3.16	263,100	7.6	632,700
	Measured + Indicated		2,690,000	3.22	278,700	8.1	698,000
	Inferred		704,000	2.75	62,200	5.6	125,900
Total Resources	Measured	OP 0.25 and UG 1.6	1,569,000	2.09	105,300	7.58	382,000
	Indicated		29,548,000	1.07	1,020,100	3.39	3,224,100
	Measured + Indicated		31,116,000	1.13	1,125,500	3.60	3,606,100
	Inferred		4,348,000	0.99	138,300	2.67	372,900

AGP is required to inform the public that the quantity and grade of reported inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. Geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an inferred resource has a lower level of confidence than an indicated resource. It is reasonably expected that most of the inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. It is also noted that mineral resources, which are not mineral reserves, do not have demonstrated economic viability. Lastly, rounding of values as required by the reporting guidelines may result in apparent differences between tonnes, grades, and metal contents.

14.1.5.5 Grade Sensitivity

Table 14.23 shows the sensitivity of the model to changes in cut-off within the resource constraining shell. Table 14.24 shows the sensitivity of the model to changes in cut-off for the material amenable to underground extraction. The base case cut-off of 0.25 g/t Au and 1.6 g/t Au is highlighted in the tables.

Table 14.23: Model Sensitivity to Cut-off within the Resource Constraining Shell

Area	Category	Cut-off (g/t Au)	Tonnes	Au (g/t)	Gold (oz Au)	Ag (g/t)	Silver (oz Ag)
Resource amenable to open pit extraction	Measured	> 1.0	560,000	4.20	75,600	13.43	241,900
		> 0.5	925,000	2.80	83,100	9.33	277,300
		> 0.4	1,165,000	2.31	86,600	7.93	297,100
		> 0.3	1,373,000	2.01	89,000	7.04	310,900
		> 0.25	1,471,000	1.90	89,800	6.70	316,700
		> 0.2	1,581,000	1.78	90,600	6.35	322,700
	Indicated	> 1.0	4,926,000	2.68	423,900	6.92	1,096,000
		> 0.5	12,561,000	1.44	583,500	4.21	1,699,300
		> 0.4	18,259,000	1.13	665,600	3.57	2,094,300
		> 0.3	24,225,000	0.94	732,800	3.14	2,444,700
		> 0.25	26,956,000	0.87	757,000	2.99	2,591,400
		> 0.2	29,710,000	0.81	776,900	2.87	2,744,600
	Measured + Indicated	> 1.0	5,486,000	2.83	500,000	7.58	1,340,000
		> 0.5	13,486,000	1.54	670,000	4.56	1,980,000
		> 0.4	19,424,000	1.20	750,000	3.83	2,390,000
		> 0.3	25,598,000	1.00	820,000	3.35	2,760,000
		> 0.25	28,426,000	0.93	850,000	3.18	2,910,000
		> 0.2	31,291,000	0.86	870,000	3.05	3,070,000
	Inferred	> 1.0	460,000	2.00	30,000	3.22	50,000
		> 0.5	1,411,000	1.09	50,000	2.59	120,000
		> 0.4	2,207,000	0.86	60,000	2.30	160,000
		> 0.3	3,237,000	0.70	70,000	2.12	220,000
		> 0.25	3,644,000	0.65	80,000	2.11	250,000
		> 0.2	4,072,000	0.61	80,000	2.05	270,000

Table 14.24: Model Sensitivity to Cut-off below the Resource Constraining Shell

Area	Category	Cut-off (G/t Au)	Tonnes	Au (g/t)	Gold (Oz)	Ag (g/t)	Silver (Oz)
Resource amenable to underground extraction	Measured	> 10	8,000	12.44	3,300	35.68	9,500
		> 5	35,000	7.87	8,700	27.54	30,600
		> 2.5	85,000	5.35	14,700	21.77	59,800
		> 1.9	94,000	5.08	15,300	21.18	63,800
		> 1.6	98,000	4.94	15,500	20.78	65,300
		> 1.0	99,000	4.91	15,600	20.68	65,600
	Indicated	> 10	24,000	14.05	10,900	25.99	20,200
		> 5	270,000	7.48	64,900	13.13	114,000
		> 2.5	1,415,000	4.07	185,400	8.89	404,600
		> 1.9	2,218,000	3.39	242,100	8.00	570,500
		> 1.6	2,592,000	3.16	263,100	7.59	632,700
		> 1.0	2,652,000	3.12	265,800	7.51	640,400
	Measured + Indicated	> 10	32,000	13.64	14,200	28.46	29,700
		> 5	304,000	7.52	73,600	14.77	144,600
		> 2.5	1,501,000	4.15	200,100	9.63	464,400
		> 1.9	2,312,000	3.46	257,400	8.53	634,300
		> 1.6	2,690,000	3.22	278,700	8.07	698,000
		> 1.0	2,751,000	3.18	281,400	7.98	706,000
	Inferred	> 10	1,000	10.35	500	21.50	900
		> 5	66,000	6.29	13,300	8.18	17,300
		> 2.5	251,000	4.22	34,000	7.40	59,700
		> 1.9	475,000	3.23	49,200	6.22	94,900
		> 1.6	704,000	2.75	62,200	5.56	125,900
		> 1.0	725,000	2.71	63,200	5.52	128,600

14.1.6 Comparison with 2019 Mineral Resource Estimate

Comparing this new resource estimate against the last resource model effective July 1, 2019 and authored by P&E, reveal an increase of 92% in the measured and indicated tonnes. Despite the reduction in grade from 2.29 g/t Au in the P&E study to 1.13 g/t Au in this study, the resource yields a decrease of 6% in gold ounces in the measured and indicated category.

The change in the inferred resource amounted to 116% more tonnes. Grade is significantly lower from 3.43 g/t Au to down to 0.99 g/t Au; consequently, the total gold ounces decreased by 38% (see Table 14.25).

AGP notes that the July 1, 2019 estimate reports the resources using a Gold equivalent cut-off which was higher than the AGP gold only cut-off. This change will produce lower tonnes at a higher grade in the P&E model when compared to the current AGP estimate.

Table 14.25: Resources Statement compared with Previous Estimate

	AGP December 16, 2020			P&E July 1, 2019					
Cut-off	> 0.25 g/t OP & > 1.6 g/t Au UG			> 0.4 g/t OP & > 1.9 g/t AuEq					
Classification	Tonnage	Au	Gold	Tonnage	Au	Gold	Tonnage	Grade	Ounces
	(t)	(g/t)	(oz)	(t)	(g/t)	(oz)	% Diff.	Diff (g/t)	% Diff
Measured	1,569,000	2.09	105,300	925,000	2.7	80,000	70%	-0.61	32%
Indicated	29,548,000	1.07	1,020,100	15,227,000	2.26	1,111,000	94%	-1.19	-8%
Meas. + Ind.	31,116,000	1.13	1,125,500	16,202,000	2.29	1,192,000	92%	-1.16	-6%
Inferred	4,348,000	0.99	138,300	2,009,000	3.43	222,000	116%	-2.44	-38%

To show a fairer comparison, the new resource models were reported within and below the same resource constraining shell used in the P&E resource estimate and reported at the same 0.4 g/t AuEq cut-off within the shell and at 1.9 g/t AuEq below the shell.

P&E described the AuEq calculation based on gold and silver prices of US$1,275 and US$16.50, respectively, and recoveries of 95.5% for gold and 62.6% for silver. In the P&E report, AuEQ equals gold grade + (silver grade/112.17). AGP notes these prices and recovery are no longer valid, and they were used here only to offer a fair comparison with the previous resource estimate. AGP would also like to point out that for this comparison, the material below the shell on the AGP model was strictly selected based on the cut-off within the AGP mineralised envelope.

Results from this comparison show the new resource (within and below the P&E resource shell) returns 47% more tonnes in the measured and indicated categories. The grade is lower from 2.29 g/t Au in the P&E estimate to 1.55 g/t Au in this resource. The higher tonnage was not sufficient to offset the lower grade to return a small 1% decrease in gold ounces.

In the inferred category, the new resource returns 47% more tonnes. The inferred grade is much lower from 3.43 g/t Au in the P&E estimate down to 1.67 g/t Au in this resource. The increase tonnage was not sufficient to offset the grade difference therefore there was a 36% reduction in the inferred gold ounces.

Table 14.26 shows the model comparison within and below the P&E July 1, 2019 resource constraining shell at the greater than 0.4 g/t AuEq cut-off for the open pit portion and at 1.9 g/t AuEq for the material below the P&E shell.

Table 14.26: Percent Difference – 2019 Resource vs. 2020 Resource within & below P&E Shell

Classification	0.40 g/t AuEq within the P&E Resource Constraining Shell			1.9 g/t AuEq Below the P&E Resource Constraining Shell			0.40 g/t & 1.9 g/t AuEq Above & Below the P&E Resource Constraining Shell		
	Tonnage	Grade	Ounces	Tonnage	Grade	Ounces	Tonnage	Grade	Ounces
	% Diff.	Diff (g/t)	% Diff	% Diff.	Diff (g/t)	% Diff	% Diff.	Diff (g/t)	% Diff
Measured	54%	0.20	68%	53%	-2.11	3%	54%	-0.21	43%
Indicated	52%	-0.32	16%	26%	-2.03	-22%	47%	-0.77	-4%
Meas. + Ind.	52%	-0.29	20%	28%	-2.03	-20%	47%	-0.74	-1%
Inferred	227%	0.00	145%	-29%	-1.58	-54%	47%	-1.76	-36%

The major contributor to the changes in the resources was mostly related to the Main Zone and C Zone wireframes. The P&E open pit portion of the wireframes (down to 1,150 m elevation) were modelled using a 0.4 AuEq grade threshold. For the mineralisation deemed amenable to underground extraction the grade threshold was 1.9 g/t AuEq. AGP elected to model the mineralised corridors (Main Zone and C Zone) at 0.2 g/t gold threshold for the external hanging wall and footwall contacts and then to internally subdivide zones of high-grade, medium-grade, and low-grade/waste via a probabilistic approach. This minimises grade smearing and ensures the high-grade portion of the deposit is honoured as much as possible. This approach is less rigorous and typically produces more tonnes at a lower grade than a discrete wireframe which tend to box in the higher grade and leave the lower grade material (< 0.4 AuEq) un-estimated. AGP believes that this methodology returns a resource model that is more representative of the in-situ grade distribution.

In the current model, the methodology did not change at the 1,150 m elevation and AGP carried the mineralised corridors as far as the drill holes displayed evidence of mineralisation. This resulted in the large grade difference in the material below the resource constraining shell where the high-grade intercepts can be more isolated within large zones of lower grade material.

Additional factor was to interpolate the Main Zone and C Zone using OK which tends to smooth the grade more than the ID^3 method used by P&E.

14.2 Goldlund

First Mining and later Treasury Metals engaged CGK to carry out an update of the mineral resources for their Goldlund Archean lode gold deposit in northwestern Ontario, Canada. CGK has relied upon drill hole assay data and the geological interpretation of the mineralised zones as provided by First Mining.

The mineral resource modelling was carried out by Chris Keech, P. Geo., Principal Geologist of CGK using MineSight® version 15.4 software for the development of the block model gold grade estimates and SAGE2001® for variography analysis of the composite gold grades. Mr. Keech is a Qualified Person and is independent of Treasury Metals as defined by N.I. 43-101.

The Goldlund mineral resources estimate has been carried out in accordance with the CIM's "Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines" (November 2019). The mineral resources estimate has been generated from drill hole data and the interpretation of a geological model that identifies the spatial distribution of the gold grades. The interpolation parameters have been defined based on the drill hole data and the geological interpretation and geostatistical analysis of that data.

The mineral resources have been classified by proximity to data locations and the quality of the data, and have been reported in accordance with CIM's "Standards on Mineral Resources and Reserves" (May 2014) as required by N.I. 43-101.

14.2.1 Drill Hole Database

The database for the Goldlund Project consists of drill holes, underground channel samples, and surface trenches. The underground channel samples and surface trenches have been incorporated into the database as pseudo drill holes. In the project area, there are 1,771 drill holes in the July 20, 2020 database, ("FMG_Goldlund_Drill Database_20th July 2020.accdb") totalling 176,498.3 m of drilling, with 114,102 gold assays. The drilling in the project area spans from 1941 to 2020, with the drilling carried out by 11 different companies, and with assays carried out by five different assay laboratories. The database was compiled from historical records including plan maps, drill logs, and assay certificates by Tamaka in 2010. Both Tamaka and later First Mining have added additional drilling and corrected errors in the database that was provided for this mineral resource estimate.

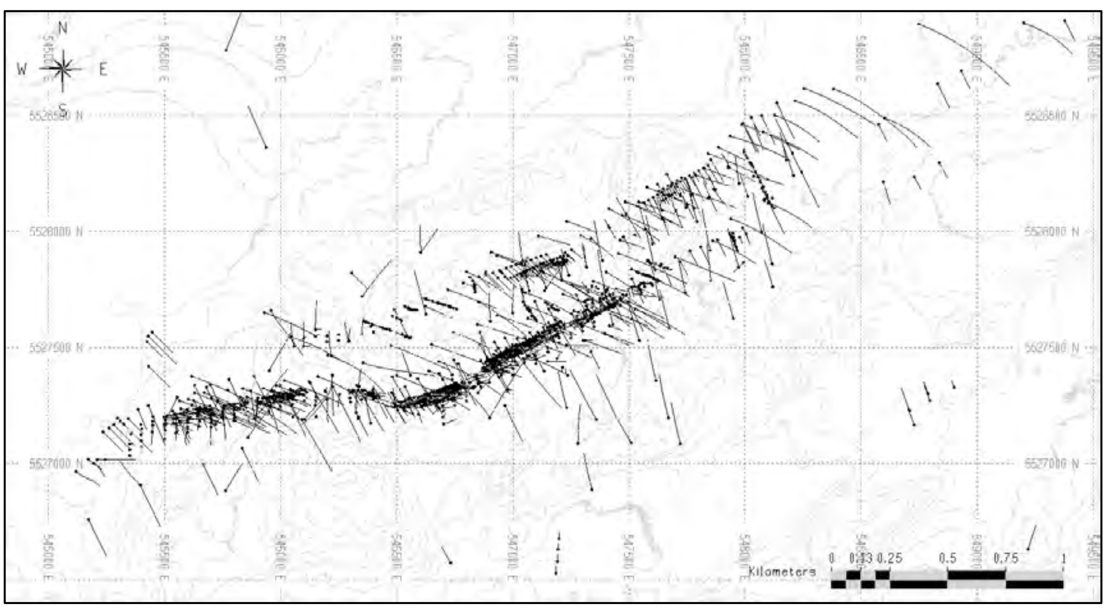
Table 14.27 displays a breakdown of the drill hole and other sample data in the database. "DDH" represents surface core drilling of various diameters; "TR" represents surface channel samples, "UG" represents underground drill holes, and "UGWC" represents underground wall channel samples. The majority of the assays come from the surface and underground drill holes.

Table 14.27: Summary of the Goldlund Project Drill hole Data by Sampling Method

Type	No. of Holes	Metres	No. of DH Surveys	No. of Assays	No. of Lithology Intervals
DDH	856	152,794.7	18,495	94,480	9,183
TR	189	1,441.7	375	1,601	1,601
UG	480	18,626.0	480	14,650	1,227
UGWC	246	3,645.9	289	3,370	250
	1,771	176,498.3	19,639	114,102	12,266

Figure 14-17 displays a plan view of the drill hole collar locations and drill hole traces. The trend in the drilling follows the strike of northeast-trending granodiorite sills, which hosts much of the gold mineralisation.

Figure 14-17: Goldlund Project – Drill Hole Plan with Surface Contours

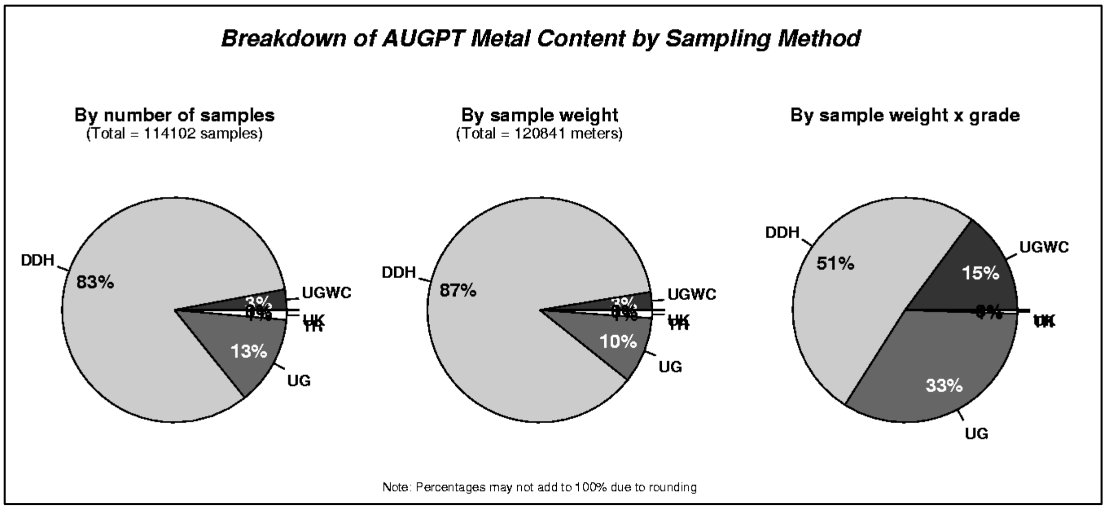


Source: CGK (2020).

Figure 14-18 displays three pie graphs showing the percentage of samples by count in the left-hand pie graph, the percentage of samples by sample length in the middle pie graph and the percentage of metal (gold grade * sample length) in the right-hand pie graph. From these graphs it can be seen that the DDH and UG contribute 96% of the number of sample assays, 97% of the total length of sample assays and 84% of the metal.

The UGWC samples only contribute 3% of the number and length of samples; however, they do contribute 15% of the overall metal. That is, the UGWC have been collected in high-grade areas. Trench channel samples are not that important overall as they contribute only a small percentage of samples and a small percentage of overall metal.

Figure 14-18: Breakdown of Au (g/t) Metal Content by Sampling Method



Source: CGK (2020).

Table 14.28 displays summary statistics for the gold (g/t) sample assays by assay laboratory. There are four commercial laboratories, ACUR, COCH, PAUL and SGS. It appears that RAND was the name of the chief assayer at Goldlund Mines Ltd and is not considered a commercial laboratory. However, there are signed assay certificates by Mr. Randy Farmer, chief assayer, that were used to validate assays in the database.

Table 14.28: Summary Statistics for Sample Gold Assays by Laboratory

Assay Lab.	Description	Count of AUGPT	Average of AUGPT	Min. of AUGPT	Max. of AUGPT
ACUR	Accurassay Laboratories, Thunder Bay, ON	54,946	0.249	0.001	433.0
COCH	Cochenour Fire Assaying, Cochenour, ON	16,440	2.846	0.000	1,402.3
PAUL	Paul's Custom Fire Assaying Ltd., Red Lake, ON	3,225	2.094	0.080	1,413.3
RAND	Randy Farmer Assay - Goldlund Mines Ltd.	217	3.427	0.000	238.1
SGS	SGS Laboratories, Burnaby, BC	21,103	0.267	0.000	367.0
UK	Unknown, not identified in the database, assays from drill logs	18,171	2.440	0.000	880.5
Total		114,102	1.034	0.000	1,413.3

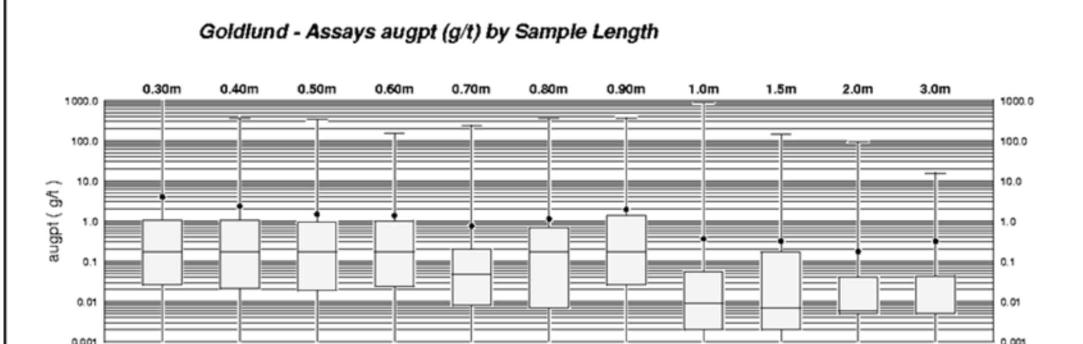
Spot checks of a selected 9,000 assays from 90 drill holes was carried out using team verification with an error rate of less than 0.3%, which is considered acceptable. As well, comparisons of the assays between the historical drill holes and recent drill holes were carried out to ensure that the historical assays are sufficiently accurate to be reliable. From these comparisons, the Qualified Person for this section of the report is of the opinion that the database contains data that is sufficiently accurate to be reliable and is therefore suitable for the estimation of mineral resources.

The historical drill hole data has been sampled on a selective basis, using variable sample lengths ranging from 0.3 m (~1 ft) up to 1.0 m (~3 ft). This has resulted in large sections of drill core that are unsampled.

It is believed that the sampling of these historical holes reflects visual guides to the gold mineralisation. A decision not to sample an interval is interpreted to mean no visual indicators of gold mineralisation were present. Much of the recent drilling has been routinely sampled at either 1 m or 2 m intervals.

Figure 14-19 shows side-by-side boxplots of the gold grade (g/t) by sample length. The 0.30 m samples have the highest average gold grade as shown by the filled circle on the left-hand box plot, with an average grade of 3.96 g/t Au. The average gold grade essentially decreases from left to right as the sample lengths become longer. The 0.80 m and 0.90 m sample length intervals break the trend in the average grade from left to right. This is thought to be the result of the implementation of a more consistent sample length in the underground drill holes and underground channels samples, collected from 1979 to 1983.

Figure 14-19: Side-by-Side Boxplots of Au (g/t) by Sample Length



	0.30m	0.40m	0.50m	0.60m	0.70m	0.80m	0.90m	1.0m	1.5m	2.0m	3.0m	
Number of data	6973	4483	6644	6957	4751	8590	8085	26864	14774	19799	366	Number of data
Mean	3.961	2.36	1.47	1.36	0.747	1.133	1.931	0.36	0.318	0.176	0.315	Mean
Maximum	1402.29	352.052	335.76	149.829	234.17	345.6	356.571	864.686	141.25	91.63	15.326	Maximum
Upper quartile	1.029	1.029	0.937	1.02	0.201	0.686	1.34	0.055	0.17	0.04	0.043	Upper quartile
Median	0.171	0.171	0.171	0.171	0.048	0.17	0.171	0.009	0.007	0.006	0.005	Median
Lower quartile	0.026	0.021	0.019	0.024	0.008	0.007	0.026	0.002	0.002	0.005	0.005	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum
Variance	1000.85	198.291	57.7864	28.4276	17.6777	41.6922	100.307	47.0493	4.3111	2.0807	1.6162	Variance
CV	7.988	5.966	5.171	3.919	5.627	5.698	5.188	19.052	6.525	8.209	4.041	CV
Skewness	24.041	16.157	23.06	13.234	37.223	31.765	19.619	98.214	31.687	34.406	7.611	Skewness

Source: CGK (2020).

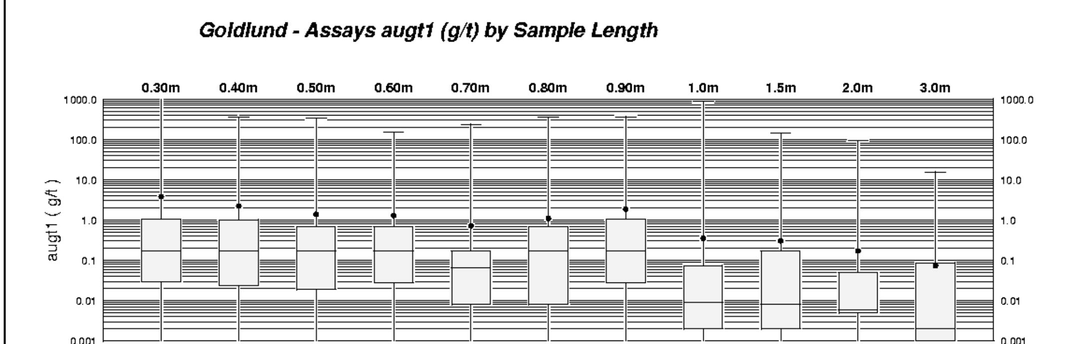
The fact that the highest average grade comes from the shortest sample length is interpreted to mean that the geologists at the time could visually identify alteration and mineralisation associated with gold mineralisation and they selectively sampled high-grade mineralisation. Therefore, the unsampled intervals in the historical drill core likely do not show any significant alteration or mineralisation and are expected to be low-grade material.

A visual inspection of a series of cross-sections displaying drill holes that have been completely assayed close to drill holes that have unsampled intervals typically show that the completely sampled holes display low-grade gold assays. This supports the view that the unsampled intervals are, more likely than not, to be low-grade material. Therefore, assigning the unsampled intervals a low-grade gold assay is a prudent approach to treating these unsampled intervals.

The treatment of the unsampled intervals is very important and must be done prior to compositing the assays to a regular length. The approach adopted to the treatment of the unsampled intervals was to determine a low-grade background value that was consistent with the detection limits available at the time the sampling and assaying was done for the sampled intervals. This led to the assignment of one-quarter to one-half the detection limit based on the drilling year and the assay laboratory for the unsampled intervals. In general, the earlier drilling was assigned a background value of 0.086 g/t Au, while later drilling has an assigned background value of 0.0025 g/t Au.

Figure 14-20 displays side-by-side boxplots of the gold grades that have the unsampled intervals assigned background values based on one-quarter to one-half the detection limit, (AUGT1). The overall trend in the average gold grade is similar to that shown in Figure 14-19. The 0.3 m sample lengths still have the highest average grade, but it is now lower at 3.76 g/t Au and there are now 379 more intervals with grades.

Figure 14-20: Side-by-Side Boxplots of Au1 (g/t) by Sample Length



	0.30m	0.40m	0.50m	0.60m	0.70m	0.80m	0.90m	1.0m	1.5m	2.0m	3.0m	
Number of data	7352	4759	7062	7409	4985	8940	8434	27636	15763	20448	2230	Number of data
Mean	3.762	2.229	1.388	1.282	0.716	1.092	1.854	0.352	0.302	0.172	0.074	Mean
Maximum	1402.29	352.052	335.76	149.829	234.17	345.6	356.571	864.686	141.25	91.63	15.326	Maximum
Upper quartile	1.029	1.02	0.686	0.686	0.171	0.686	1.029	0.073	0.17	0.05	0.086	Upper quartile
Median	0.171	0.17	0.17	0.17	0.064	0.17	0.171	0.009	0.008	0.006	0.002	Median
Lower quartile	0.03	0.024	0.019	0.028	0.008	0.008	0.028	0.002	0.002	0.005	0.001	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum
Variance	950.329	187.143	54.4978	26.7931	16.8709	40.0977	96.3171	45.6561	4.043	2.0154	0.2606	Variance
CV	8.195	6.138	5.32	4.037	5.74	5.801	5.293	19.223	6.652	8.24	6.883	CV
Skewness	24.672	16.634	23.731	13.628	38.08	32.375	20.016	99.699	32.713	34.959	19.454	Skewness

Source: CGK (2020).

Table 14.29 displays summary statistics (unweighted) for the AUGPT (original assays) and AUGT1 (unsampled intervals assigned one-quarter to one-half the detection limit). There is a total of 10,095 unsampled intervals that have been assigned background gold grades. The impact of assigning background gold grades to the unsampled intervals is to reduce the average gold grade from 1.034 g/t Au (unweighted) to 0.955 g/t Au (unweighted). This is a reduction in the global average gold grade of about 8%.

The drill hole database also contains a total of 2,096 specific gravity measurements that were made by both Tamaka and First Mining on representative pieces of drill core. The core samples were weighted in air and then in water, the buoyancy method, using an Acculab VIC-612 electronic balance, with a maximum weight of 610 g and an accuracy of 0.01 g.

Table 14.29: Summary Statistics for Sample Gold Assays by Zone

Zone	Count of AUGPT	Average of AUGPT	Min. of AUGPT	Max. of AUGPT	Count of AUGT1	No. of Infilled Intervals	Average of AUGT1	Min. of AUGT1	Max. of AUGT1
1	33,056	2.305	0.000	1,402.3	35,378	2,322	2.159	0.000	1,402.3
2	4,688	1.008	0.000	880.5	4,954	266	0.956	0.000	880.5
3	8,755	0.933	0.000	514.3	9,134	379	0.897	0.000	514.3
4	3,994	0.562	0.000	159.6	4,043	49	0.555	0.000	159.6
5	2,489	0.259	0.000	36.8	2,561	72	0.253	0.000	36.8
6	2,068	0.355	0.003	55.7	2,137	69	0.344	0.001	55.7
7	19,913	1.073	0.000	1,413.3	23,610	3,697	0.918	0.000	1,413.3
8	887	0.123	0.000	10.6	932	45	0.119	0.000	10.6
9	336	0.353	0.000	6.9	345	9	0.345	0.000	6.9
10	37,916	0.096	0.000	261.7	41,103	3,187	0.091	0.000	261.7
Total	114,102	1.034	0.000	1,413.3	124,197	10,095	0.955	0.000	1,413.3

The procedure described for the Tamaka measurements is:

- a core sample was placed within the cage and the dry weight taken
- a bucket of water was raised below the hanging sample until the rock was fully submerged and not touching the bucket, the wet weight was then taken
- the wet and dry values were entered into the following formula to calculate the specific gravity

$$Specific\ Gravity = \frac{Weight_{dry}}{Weight_{dry} - Weight_{wet}}$$

Figure 14-21 displays photographs of the specific gravity methodology employed by Tamaka, taken by Mr. T. McCracken P.Geo of WSP.

Figure 14-21: Specific Gravity Measurement Equipment Setup

Source: (WSP) 2019.

Table 14.30 presents summary statistics for the specific gravity measurements on drill core by both Tamaka and First Mining. The mineralised zones 1 to 9 have an average specific gravity of 2.73, while zone 10, unmineralised material, has a higher specific gravity of 2.83.

Table 14.30: Summary Statistics for Specific Gravity Measurements by Zone

Company	Zone	Count of SG	Average of SG	Min. of SG	Max. of SG
Tamaka	2	11	2.737	2.580	2.850
	3	4	2.730	2.650	2.820
	4	62	2.731	2.320	3.020
	5	9	2.713	2.630	2.760
	6	1	2.650	2.650	2.650
	7	1	2.860	2.860	2.860
	10	37	2.790	2.160	2.970
	Subtotal	125	2.748	2.160	3.020
First Mining	1	246	2.734	2.390	3.100
	2	80	2.758	2.630	2.900
	3	92	2.765	2.540	3.030
	5	43	2.783	2.520	3.040
	6	17	2.796	2.710	2.850
	7	532	2.735	2.380	2.950
	8	1	2.830	2.830	2.830
	10	960	2.836	2.410	3.300
	Subtotal	1,971	2.788	2.380	3.300
	Total	2,096	2.786	2.160	3.300

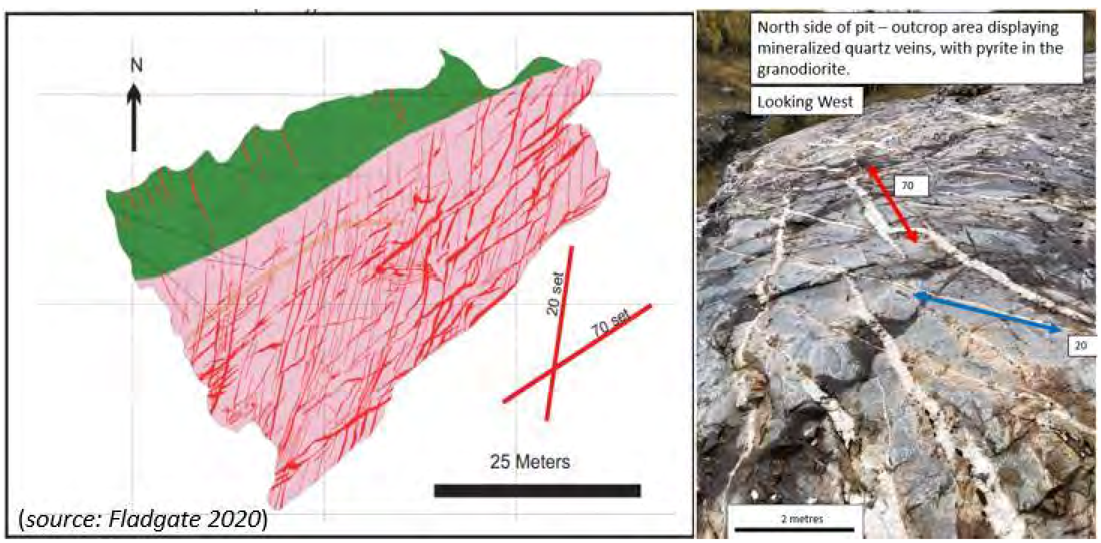
14.2.2 Geological Model

Gold mineralisation at Goldlund is interpreted as an Archean shear hosted quartz vein lode gold deposit. The predominant gold mineralisation is associated with quartz vein and stock-work structures situated inside northeast-trending albite-trondhjemite dykes (granodiorite), with lesser amounts in porphyry dykes and metavolcanic rocks.

The mineralised dykes generally strike (065°) and dip steeply to the southeast. The gold-bearing quartz stockwork veins consist of two synchronous sets of veins, referred to as the 20° set (trending 189°/53°W) and 70° set (trending 239°/58°N). The veins structures have developed preferentially in the granodiorite dykes, as they were the most competent (brittle) rock type; however, vein structures do propagate into the surrounding metavolcanic rocks, most often as brittle-ductile, biotite-carbonate-rich shears.

Figure 14-22 displays a map of the historical open pit area showing the 20° set and 70° set veins (red). The veins are hosted in a fine-grained granodiorite (pink), with the footwall gabbroic rocks shown in green, and late “tension veins” shown in orange (Fladgate 2012). The photograph on the right-hand side of Figure 14-22 displays the 20° set and 70° set vein stockwork.

Figure 14-22: Pit Trench Example of Stockwork Mineralisation (Pettigrew, Fladgate 2012)

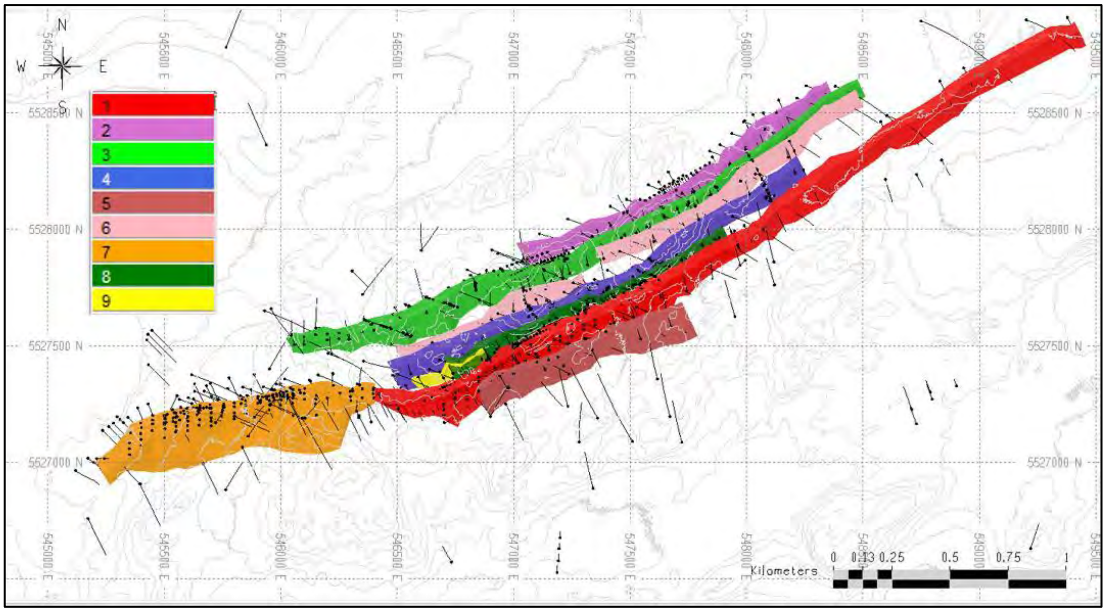


Source: CGK (2020).

The interpretation of the mineralised zones was completed by Mr. Miro Mytry P.Geo. of First Mining. The mineralised zones represent corridors of mineralisation that consider a nominal 0.1 g/t Au cut-off and principally follow the trend of the granodiorite dykes, but there are also mineralised trends within the porphyry dykes and mafic metavolcanic rocks.

Figure 14-23 displays the mineralised wireframe zones trending to the northeast, along with the traces of the drill holes.

Figure 14-23: Mineralised Zone Wireframes



Source: CGK (2020).

Table 14.31 presents a listing of the Zone codes and the wireframe names. Zone 1 and Zone 7 have the largest volumes. Zone 8 and Zone 9 overlap as the interpretation from 2019 was adjusted by First Mining in this area. Where the two overlap, Zone 9 takes priority over Zone 8, although they both represent the same mineralised trend. Zone 6 is defined by wireframe Zone35_2020 and this mineralised zone is situated between Zone 3 and Zone 4. Note that Zone 8 is based on the WSP wireframe model from 2019.

Zone 10, that is not listed in the table below, is the background or “waste” material code assigned to all assays not inside the Zone 1 to Zone 9 wireframes.

Table 14.31: Summary statistics for 2020 Mineralised Zones

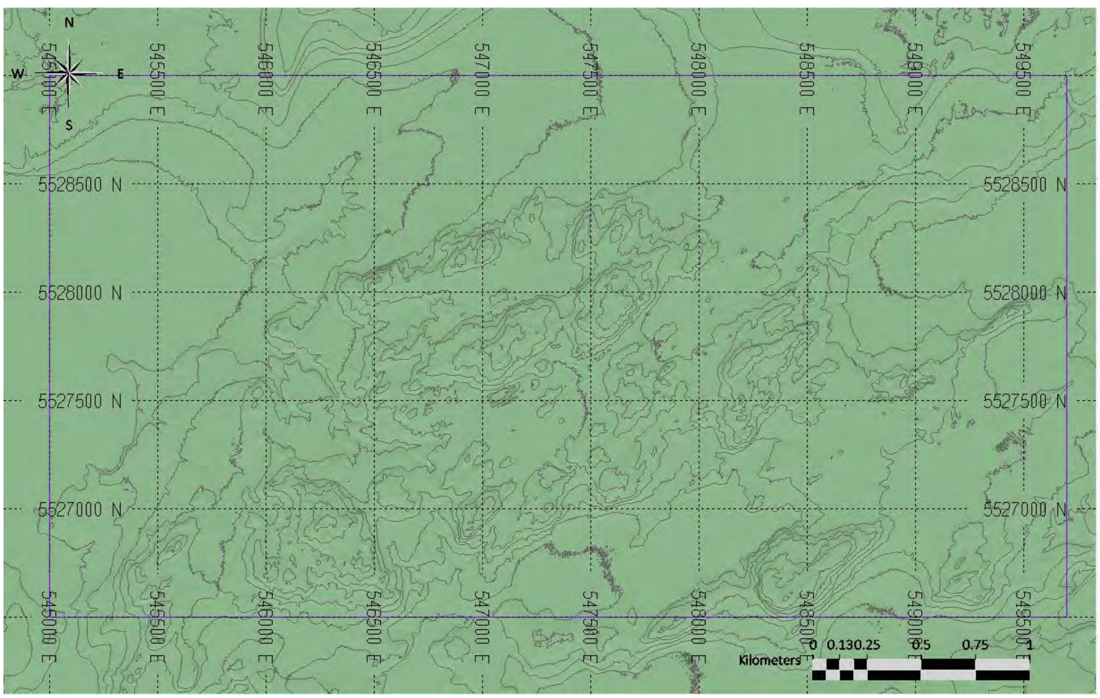
Zone Code	Zone Name	Wireframe Name	Volume (m^3)	tonnes
1	1	Zone1_2020tr.dm.msr	56,019,490	151,252,625
2	2	Zone2_2020tr.dm.msr	13,028,502	35,176,956
3	3	Zone3_2020tr.dm.msr	26,865,807	72,537,681
4	4	Zone4_2020tr.dm.msr	23,547,096	63,577,159
5	5	Zone5_2020tr.dm.msr	14,162,485	38,238,709
6	35	Zone35_2020tr.dm.msr	16,855,884	42,922,208
7	7	Zone7_2020tr.dm	37,794,350	102,044,747
8	8	zone8_2019tr.dm.msr	11,192,791	30,220,536
9	80	Zone80_2020tr.dm.msr	1,227,816	3,315,102

The wireframes of the mineralised zones were provided by First Mining as Datamine® files that were converted to MineSight® files and validated to ensure they were closed solids. The wireframes were plotted in plan and section and compared against the drill hole assays to ensure that they honoured the drill hole data. From this review, the Qualified Person for this section of the report is of the opinion that the interpretation is suitable to be used for the estimation of mineral resources.

14.2.2.1 Topography

A topographic surface model was provided as a Datamine file by First Mining and is based on a high-resolution Bare Earth LiDAR survey by Airborne Imaging of Calgary Alberta in 2012. The surface drill hole collars were compared against the topographic survey. Some drill hole collar elevations were adjusted to be consistent with this topographic surface. Figure 14-24 shows a plan view of the resolution topography with 5 m contours. The blue outline shows the limits of the block model.

Figure 14-24: Bare Earth LiDAR Topography Surface



Source: CGK (2020).

14.2.2.2 Overburden Surface Model

The Goldlund Project has intermittent overburden across the project area. An overburden surface was developed using the drill hole lithological codes of OVB (overburden) or CAS (casing). The surface was interpolated in MineSight® software using an inverse distance algorithm with a minimum of 4 and a maximum of 12 drill holes to produce a smoothed surface. The resulting surface of the bottom of the overburden was normalised against the topography surface to ensure there was consistency between the two surfaces and that the bottom of the overburden was not above the surface topography. The estimated overburden thickness across the mineralised zones ranges from 0 m to 10 m, with an average of 4.8 m, which is less than one bench in the block model.

14.2.3 Exploratory Data Analysis

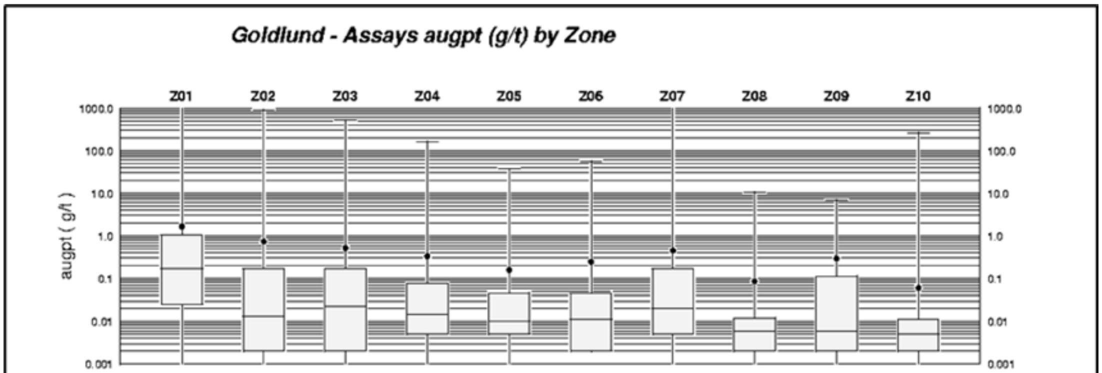
Statistical and graphical summaries of the gold grades were produced to understand the distribution of the grades in the deposit. The statistical and graphical summaries include histograms, log-probability plots, side-by-side boxplots, scatterplots, decile analysis plots and contact plots. The assays were separated by mineralised zone to examine the distribution of the gold assay grades. The results of this analysis were used to develop the estimation parameters.

14.2.3.1 Assays

The sample lengths vary from 0.1 m to 3.0 m with the most common being a 2 m length. Typically, the sample lengths prior to 1977 were less than 1.0 m, while after this period the samples were more than 1.0 m. Therefore, the following summary statistics will be weighted by sample length.

Figure 14-25 displays side-by-side boxplots and summary statistics (weighted by sample length) for AUGPT (assay grades without treating the unsampled intervals) separated by mineralised zone. The highest mean average grade comes from Zone 1, with an average of 1.658 g/t Au and a coefficient of variation (CV=Std. Dev. / Average) of 7.056. This high CV indicates a large variability in this mineralised zone that is caused by the mixing of low- and high-grade gold assays.

Figure 14-25: Side-by-Side Boxplots of AUGPT Assays by Zone (Length Weighted)



	Z01	Z02	Z03	Z04	Z05	Z06	Z07	Z08	Z09	Z10	
Number of data	33056	4688	8755	3994	2489	2068	19913	887	336	37916	Number of data
Mean	1.658	0.734	0.517	0.332	0.158	0.246	0.454	0.086	0.295	0.06	Mean
Maximum	1402.29	880.46	514.279	159.575	36.753	55.738	1413.26	10.62	6.85	261.733	Maximum
Upper quartile	1.029	0.17	0.17	0.075	0.046	0.045	0.17	0.012	0.113	0.011	Upper quartile
Median	0.171	0.013	0.022	0.015	0.01	0.011	0.02	0.006	0.006	0.005	Median
Lower quartile	0.025	0.002	0.002	0.005	0.005	0.002	0.005	0.002	0.002	0.002	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.002	0.0	0.0	0.0	0.0	Minimum
Variance	136.828	97.9671	30.7318	8.0273	0.7441	3.1285	47.1801	0.3678	0.7582	1.3385	Variance
CV	7.056	13.477	10.721	8.527	5.466	7.186	15.136	7.055	2.952	19.315	CV
Skewness	49.694	59.849	53.95	25.769	17.127	17.693	102.785	12.411	4.368	131.479	Skewness

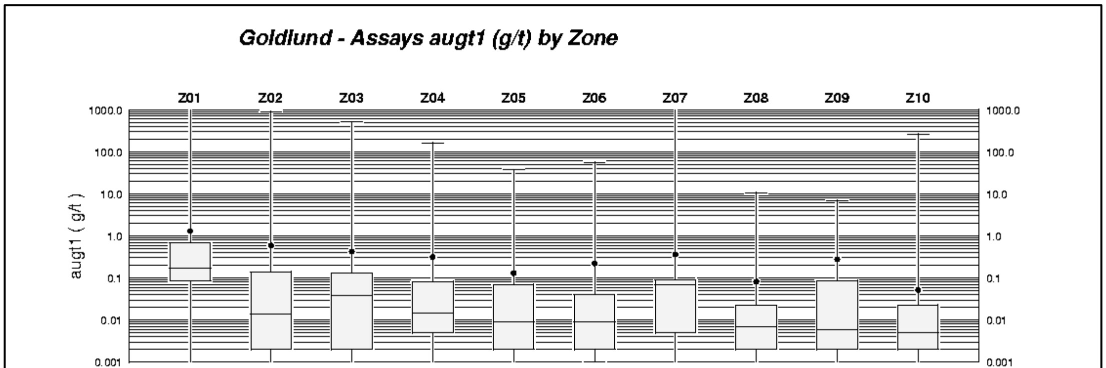
Source: CGK (2020).

Figure 14-26 displays side-by-side box plots and summary statistics (weighted by sample length) for AU1GT (unsampled intervals assigned background gold grades) separated by mineralised zone. The highest mean average grade still comes from Zone 1, with an average of 1.298 g/t Au and a coefficient of variation of 7.931. There is a 23% decrease in the average grade for Zone 1 between the AUGPT and AUGPT1 grades when considering a length-weighted average, i.e. (1.298 g/t Au/ 1.658 g/t Au).

The impact of assigning background grades to the unsampled intervals using one-quarter to one-half the detection limit is to decrease the average grade for each of the zones.

The other item of note is the coefficient of variation (CV) which is very high for all zones, ranging from 3.05 for Zone 9 up to 16.537 for Zone 7. A high CV indicates a highly skewed distribution of gold grades where there is mixing of both low-grade and high-grade assays. This is an indication that a further separation of the low gold grades from the high gold grades may be required, possibly using sub Zones or “domains”.

Figure 14-26: Side-by-Side Box Plots of AUGT1 Assays by Zone (Length Weighted)



	Z01	Z02	Z03	Z04	Z05	Z06	Z07	Z08	Z09	Z10	
Number of data	35378	4954	9134	4043	2561	2137	23610	932	345	41103	Number of data
Mean	1.298	0.592	0.422	0.315	0.13	0.22	0.36	0.082	0.276	0.052	Mean
Maximum	1402.29	880.46	514.279	159.575	36.753	55.738	1413.26	10.62	6.85	261.733	Maximum
Upper quartile	0.686	0.139	0.129	0.083	0.07	0.04	0.09	0.022	0.086	0.023	Upper quartile
Median	0.17	0.014	0.038	0.015	0.009	0.009	0.07	0.007	0.006	0.005	Median
Lower quartile	0.086	0.002	0.002	0.005	0.002	0.002	0.005	0.002	0.002	0.002	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.001	0.0	0.0	0.0	0.0	Minimum
Variance	106.048	77.6994	24.2953	7.5544	0.5846	2.7841	35.4341	0.2958	0.7067	0.8402	Variance
CV	7.931	14.899	11.692	8.719	5.863	7.582	16.537	6.639	3.05	17.522	CV
Skewness	56.318	67.184	60.649	26.566	19.331	18.765	118.564	13.846	4.56	165.834	Skewness

Source: CGK (2020).

14.2.3.2 Composites

The drill hole data was composited to 2 m down-the-hole composites, without consideration of any geological boundaries. The 2 m composite length was selected because it is approximately half of the block height used for the block model and is the next most common sample length used (18,851 samples), with only the 1 m sampling length being used more frequently (20,329 samples).

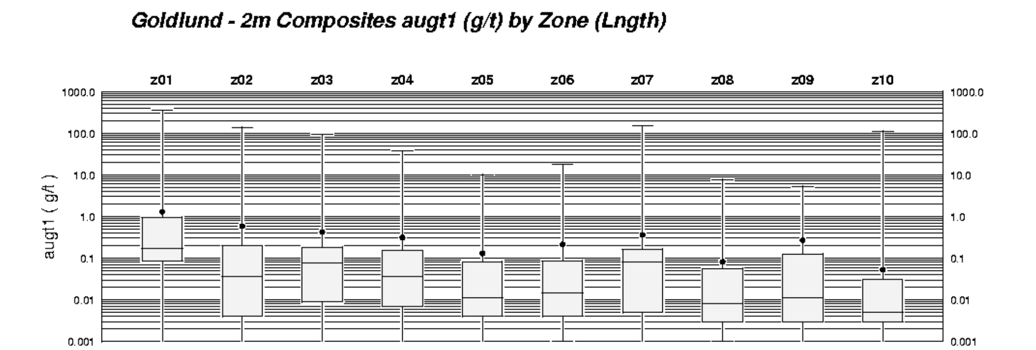
Recall that the unsampled intervals were assigned background gold grades prior to compositing. Some of these unsampled intervals were long and this would generate multiple 2 m composites. Therefore, the treatment of the unsampled intervals will have a greater impact on the number of composites and on the average grade of those composites.

Figure 14-27 shows side-by-side boxplots and length weighted summary statistics for the 2 m composites based on sample assays that have the unsampled intervals assigned a background grade. The average grades for each mineralised zone are like those average grades of the length weighted assays shown in Figure 14-26.

Compositing the assays to 2 m composites has not changed the average but has reduced the variability as the CV is reduced for each of the zones. For example, the Zone 1 assays have a

CV of 7.93, as shown in Figure 14-26, while the 2 m composites have a CV of 4.36, as shown in Figure 14-27, which is a reduction in variability of almost half. The maximum gold grade for the Zone 1 assays is 1,402.29 g/t Au, while the 2 m composites for Zone 1 have a maximum grade of 354.57 g/t Au. While the CV for each of the zones has been reduced by compositing, they are still very high, due to some high-grade outliers. Adjustment of these outliers by grade capping is required.

Figure 14-27: Side-by-Side Boxplots of AUGT1 2 m Composites by Zone (Length Weighted)



	z01	z02	z03	z04	z05	z06	z07	z08	z09	z10	
Number of data	19959	2678	4636	1664	1604	1107	17570	682	200	34794	Number of data
Mean	1.292	0.587	0.422	0.314	0.13	0.214	0.361	0.082	0.269	0.053	Mean
Maximum	354.572	137.659	91.64	36.96	10.075	17.526	148.529	7.632	5.234	111.42	Maximum
Upper quartile	0.958	0.196	0.182	0.15	0.083	0.086	0.163	0.057	0.127	0.031	Upper quartile
Median	0.171	0.037	0.075	0.036	0.011	0.015	0.083	0.008	0.011	0.005	Median
Lower quartile	0.086	0.004	0.009	0.007	0.004	0.004	0.005	0.003	0.003	0.003	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.001	0.0	0.001	0.0	0.0	Minimum
Variance	31.7202	23.5197	4.9072	2.166	0.2535	1.0073	6.5145	0.1751	0.479	0.2592	Variance
CV	4.361	8.261	5.252	4.694	3.885	4.701	7.073	5.134	2.577	9.678	CV
Skewness	25.604	23.056	22.554	14.806	11.864	11.236	28.389	12.91	4.012	62.575	Skewness

Source: CGK (2020).

14.2.3.3 Grade Capping

Grade capping is often carried out prior to compositing to limit any possible smearing of high-grade assays inside a composite. Statistical analysis and grade capping for Goldlund was carried out on the 2 m composites due to the use of variable sample length intervals and due to the shorter samples being taken in the strongly altered and mineralised drill core. This creates a selection bias in the sampling and capping the assays would result in a reduction of too much metal from the composite data.

The side-by-side boxplots in Figure 14-27 show that for each zone there are some very high-grade gold assays even after compositing to a regular length. The highest average grade can be found in Zone 1, at 354.6 g/t Au. This appears to be an extreme grade that requires adjustment or grade capping prior to block model grade estimation.

To determine if a composite grade was an outlier and should be capped, a series of graphical and statistical summaries were considered, including log-probability plots, cutting statistics plots (after M. Srivastava 1993) and decile analysis (I. Parish 1997).

Table 14.32 displays the summary statistics for the uncapped (AUGT1) and capped (CAP01) gold grades. Capping of the 2 m composites has reduced the metal in the composites by approximately 6% with a capping of 159 composites. The capping grades range from 2 g/t Au for Zone 10 up to 90 g/t Au for Zone 1.

Capping the outlier grades has also reduced the variability with a change in the global CV from 7.04 for the uncapped 2 m composites to 5.58 for the capped 2 m composites. While capping has helped reduce the CV values, they are still very high, ranging from 2.39 for Zone 9 to 5.73 for Zone 7. These high CV values indicate that the 2 m composites for the mineralised zones should be separated into more stable statistical groups and reduce the CV values for each zone prior to block grade estimation.

Table 14.32: Summary Statistics for 2 m Composites Uncapped & Capped Gold Grades

Zone	No of 2 m AUGT1	Ave. of AUGT1	CV of AUGT1	No. of 2 m CAP01	Ave of CAP01	CV of CAP01	Capping Grade	No. Capped	Mean Ratio	CV Ratio
1	19,959	1.292	4.361	19,959	1.252	3.476	90.0	11	0.969	0.823
2	2,678	0.587	8.261	2,689	0.431	3.644	20.0	5	0.734	0.601
3	4,636	0.422	5.252	4,628	0.385	3.664	18.0	10	0.912	0.765
4	1,664	0.314	4.694	1,664	0.275	3.245	9.0	6	0.876	0.789
5	1,604	0.130	3.885	1,604	0.123	3.258	5.0	3	0.946	0.886
6	1,107	0.214	4.701	1,065	0.201	4.118	10.0	3	0.939	0.933
7	17,570	0.361	7.073	17,570	0.344	5.731	50.0	8	0.953	0.850
8	682	0.082	5.134	682	0.074	4.143	4.0	2	0.902	0.894
9	200	0.269	2.577	200	0.253	2.388	3.0	2	0.941	0.985
10	34,794	0.053	9.678	34,833	0.041	3.591	2.0	109	0.774	0.480
Total	84,894	0.454	7.039	84,894	0.429	5.575		159	0.943	0.839

14.2.3.4 CV Partitioning

The high CV values observed for each of the mineralised zones requires some additional effort to separate the 2 m composite grades into more stable statistical groups. One approach is to use a CV partitioning methodology to separate the composite grades.

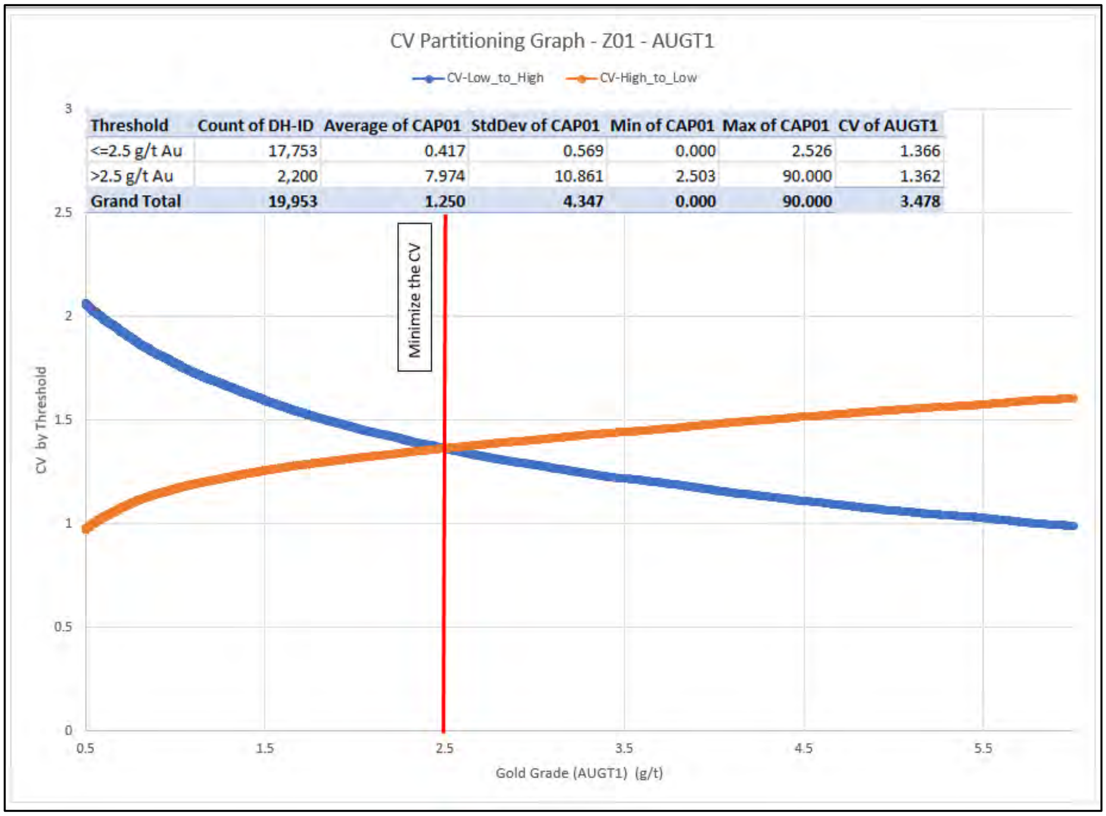
The concept of CV partitioning is to find a grade threshold that can separate the composites into two groups with the lowest CV. This concept is based on a paper by H. Parker, “Statistical Treatment of Outlier Data in Epithermal Gold Deposit Reserve Estimation” (1991). The concept is to calculate the CV of the composite grades starting with all the data, and then leave one out at a time to examine how the CV changes as composite grades are excluded. A graph can be generated that displays the change in the CV of the composites, with the results sorted from high to low and from low to high. There will be a cross-over point where the CV will be the minimum, and that composite grade can be used as the grade threshold to separate the composites into two groups that are more statistically stable and that have a minimum CV for each group.

Figure 14-28 displays an example of the CV partitioning methodology for the Zone 1 composites. The cross-over point for the minimum CV is at 2.5 g/t Au, as shown by the red-line. The CV for the low-grade and high-grade domains is now 1.36, down from 3.48 for all the Zone 1 composites.

Table 14.33 shows the thresholds used to divide the Zone 1 gold grades into separate domains that have minimum CV values. This separation creates more statistically stable groups of composite data that are suitable for block grade estimation.

Zone 9 had too few composites to allow CV Partitioning, so it will be treated as a single domain for block grade estimation.

Figure 14-28: CV Partitioning for Zone 1 – AUGT1



Source: CGK (2020).

Table 14.33: CV Partitioning of Gold Grades by Zone & Domain

Zone	LG Domain	Thresh-old	No. of LG Data	LG Ave CAP01	LG CV	HG Domain	Threshold	No. of HG Data	HG Ave. CAP01.	HG CV
z01	10	≤ 2.500	17,757	0.416	1.366	11	>2.500	2,202	7.969	1.363
z02	20	≤ 0.456	2,206	0.062	1.494	21	>0.456	472	2.164	1.488
z03	30	≤ 0.580	4,085	0.090	1.300	31	>0.580	551	2.567	1.302
z04	40	≤ 0.290	1,390	0.051	1.285	41	>0.290	274	1.409	1.282
z05	50	≤ 0.120	1,328	0.024	1.340	51	>0.120	276	0.600	1.349
z06	60	≤ 0.256	971	0.037	1.479	61	>0.256	136	1.374	1.462
z07	70	≤ 1.260	16,790	0.126	1.584	71	>1.260	780	5.037	1.582
z08	80	≤ 0.1048	626	0.022	1.416	81	>0.1048	49	0.735	1.249
z09	90	none	200	0.253	2.398		none	-	-	-
z10	100	≤ 0.086	32,620	0.020	1.501	101	>0.086	2,174	0.357	1.360

14.2.3.5 Contact Analysis

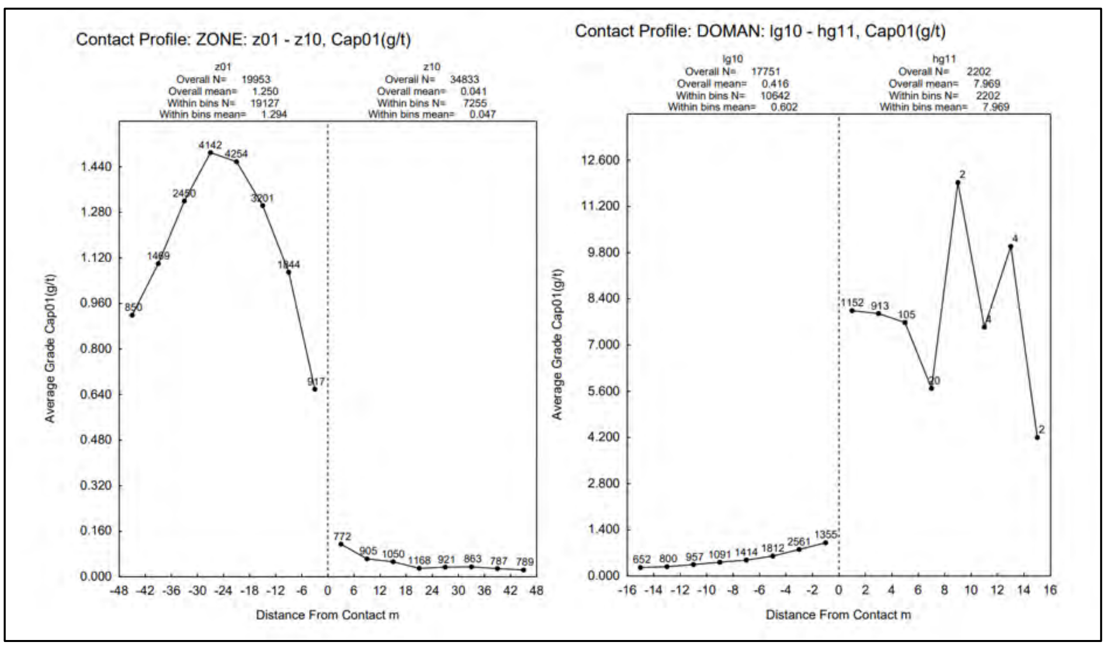
Contact analysis plots are used to assess the behaviour of the gold grades across the boundary of two different geological types. For Goldlund, it is the boundaries between the mineralised zones, Zone 1 to Zone 9, and the unmineralised material Zone 10, as well as between the high-grade and low-grade domains.

Figure 14-29 displays contact plots for the boundary between Zone 1 (mineralised) and Zone 10 (unmineralised) for the capped 2 m composite grades on the left, and the Zone 1 domains LG-10 and HG-11 for the capped 2 m composite grades on the right.

The left-hand plot in Figure 14-29 shows a well-defined break between the 2 m composites inside Zone 1 and with those in Zone 10. This indicates that there should be no sharing of composites across this boundary for block grade estimation.

The right-hand plot in Figure 14-29 also shows a well-defined break between the LG-10 and HG-11 composites. This also confirms that there should be no sharing of composites across the domain boundary. This is true for all the various comparisons examined using contact plots. The boundaries will all be treated as “hard-boundaries”, and there will be no sharing of composites across the Zone or Domain boundaries.

Figure 14-29: Contact Plots of 2 m Composites for Capped Gold Grades (CAP01) by Zone, Domain



Source: CGK (2020).

14.2.3.6 Spatial Analysis – Variography

Variography is a study of the spatial continuity of an attribute. The variography study for Goldlund consisted of two parts: indicator semi-variograms to estimate the proportion of high-grade in a block, and correlograms of the gold grades for the estimation of low-grade (LG) and high-grade (HG) block grade estimates.

14.2.3.6.1 Indicator Variography

For each zone, down-the-hole indicator semi-variograms were computed to determine the nugget effect and directional indicator semi-variograms were computed in multiple directions using SAGE2001® software. The indicator thresholds are listed in Table 14-32. The indicator semi-variograms were then fitted using an automatic fitting methodology that considered a nugget effect and two spherical structures. The fitted models were adjusted, if required, to better match the controls on the gold mineralisation, which is in part interpreted from the historical stopes in Zone 1 that display a low-angle plunge to the west.

Table 14.34 shows a listing of the indicator semi-variogram models used for kriging the HG proportion (HGIND) by zone for blocks in the model.

Table 14.34: Indicator Semi-Variogram Models for Kriging the HG Proportion in the Block Model

Zone	z01	z02	z03	z04	z05	z06	z07	z08	z10
c0	0.409	0.46	0.5	0.58	0.35	0.679	0.52	0.3	0.402
c1	0.49	0.371	0.17	0.11	0.471	0.251	0.446	0.586	0.427
c2	0.101	0.169	0.33	0.31	0.179	0.07	0.034	0.114	0.171
Range 11 - Y	10.4	25.3	5.8	9.7	8.2	6.6	5.6	8.2	6.7
Range 12 - X	6	3.9	14.7	8.7	27.6	9.5	4.2	7.5	21.4
Range 13 - Z	6.1	7.9	4	20.9	7	7.6	4	7	62
Rotation 1	70	48	-10	125	-30	56	111	63	-32
Rotation 2	0	21	-19	-63	-10	16	-66	15	-21
Rotation 3	5	13	17	48	11	18	32	-80	1
Range 21 - Y	173.5	77.4	13.9	62.8	60	160.4	99.3	180	47.8
Range 22 - X	32.4	26.4	128.2	13.8	104.4	16.7	26.7	64	66.4
Range 23 - Z	113.9	194	60.2	132.7	215.9	65.2	172.1	30.7	282.9
Model type 1-spherical	1	1	1	1	1	1	1	1	1

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.2.3.6.2 Gold Grade Variography

The spatial continuity of the 2 m composite gold grades were assessed using SAGE2001® software. The correlogram was selected as the methodology to study the spatial continuity of the gold grades and to build models of the spatial continuity for block grade estimation using ordinary kriging.

Down-the-hole correlograms were calculated to determine the nugget effect for each zone and directional correlograms were computed in multiple directions for each zone to determine the anisotropy of the gold mineralisation. The experimental correlogram of the gold grades were then fitted using an automatic fitting methodology that considered a nugget effect and two spherical structures. The fitted models were adjusted, if required, to better match the controls on the gold mineralisation, which is in part interpreted from the historical stopes in Zone 1 that display a low-angle plunge to the west.

Table 14.35 presents a summary of the model parameters used for kriging the LG gold and the HG gold grades into the blocks in the model. The nugget effects are typically between 0.4 and 0.5, which indicates a significant level of short-scale variability that is typical for Archean lode-gold deposits. As well, the first structure ranges of the spatial models are short, with ranges of 10 m to 20 m. This is a further indication of the significant level of short-scale variability. The second structure ranges are much longer, 200 m to 300 m, which supports the interpretation of the mineralised zone wireframes to the northeast.

Table 14.35: Models fitted to Correlograms of Gold Grade Composites

Zone	z01	z02	z03	z04	z05	z06	z07	z08	z09	z10
c0	0.407	0.500	0.403	0.549	0.500	0.687	0.610	0.483	0.483	0.229
c1	0.419	0.379	0.298	0.340	0.350	0.203	0.337	0.263	0.263	0.466
c2	0.174	0.121	0.298	0.111	0.150	0.110	0.053	0.254	0.254	0.305
Range 11 - Y	8.4	12.9	12.6	12.2	6.1	4.4	19.7	6.3	6.3	37.8
Range 12 - X	15.5	4.9	10.8	8.5	5.3	10.5	12.4	62.7	62.7	40.9
Range 13 - Z	17.7	14.6	4.0	61.5	7.4	5.1	7.4	25.4	25.4	160.0
Rotation 1	-30	52	66	120	-30	56	-12	-27	-27	63
Rotation 2	-15	18	14	-63	-10	16	70	-7	-7	-1
Rotation 3	-6	18	12	48	11	18	90	12	12	23
Range 21 - Y	86.0	114.0	133.4	97.5	79.7	190.0	195.0	125.0	125.0	490.0
Range 22 - X	256.0	48.0	23.2	52.5	142.5	27.0	55.2	294.7	294.7	89.5
Range 23 - Z	302.0	320.0	69.8	181.7	233.8	110.0	240.2	202.7	202.7	320.0
Model type 1-spherical	1	1	1	1	1	1	1	1	1	1

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.2.4 Block Model

A 3D block model was constructed using MineSight® 15.4 software with the dimensions shown in Table 14.36. The block size was chosen to reflect a potential selective mining unit (SMU) of 5 m x 5 m x 5 m, given the anticipated open-pit mining scenario. The block model covers an area of approximately 4.7 km by 2.5 km in plan view, and approximately 800 m vertically. The block model coordinates are in the NAD83 UTM Zone 15 grid system.

Table 14.36: Block Model Definition

	Minimum (m)	Maximum (m)	Distance (m)	Block Size (m)	No. of Blocks
Easting	545,000	549,700	4,700	5	940
Northing	5,526,500	5,529,000	2,500	5	500
Elevation	-350	460	810	5	162

Block gold grade estimates were developed using an indicator kriging to define the proportion of high-grade in a block and then ordinary kriging to estimate gold grades for the low-grade and high-grade domains separately. The final block grade is then a proportional weighted average grade of the low-grade and high-grade kriged estimates. This combined kriging methodology will be referred to as probability assisted kriging or PAK.

There are five steps to the PAK procedure:

- define the HG/LG threshold using CV partitioning to find the lowest CV for the HG/LG domains for each Zone
- indicator analysis and kriging of the indicator to define the proportion of HG/LG in each block in the model for each Zone
- ordinary kriging using the LG composites for each block in the model for each zone
- ordinary kriging using the HG composites for each block in the model for each zone
- combination of the LG and HG block grade estimates using the indicator proportion to build the final block grade estimates

The approach described above is based on two papers by Dr. Isobel Clark, "Practical Reserve Estimation in a Shear-Hosted Gold Deposit, Zimbabwe" (1993) and in "Geostatistical Modelling for Realistic Mine Planning" (1999).

The grade block model estimation methodology considered the domains to be the principal control, with the secondary control by the mineralised zone wireframes for the estimation of the gold grades. The density item in the block model was assigned the average density of the drill core measurements by Zone.

Block model gold grades were also estimated using NN, ID^2, and OK with each zone estimated independently using hard boundaries; that is, there was no sharing of composites across zone boundaries. These three additional models were used to validate the PAK methodology and to ensure that it was working as intended.

14.2.4.1 IK Estimation Parameters

The following is a summary of the parameters used to estimate the high-grade proportion in the block model (HGIND):

- Two metre composites, assigned with an indicator value using the indicator thresholds listed in Table 14-32, were used for ordinary kriging of the high-grade proportion in the blocks for Zones 1, 2, 3, 4, 5, 6, 7, 8, and 10.
- Geological zone boundaries based on the mineralised zone wireframes were used to control the selection of the 2 m composites and the blocks to be estimated in the model. There was no sharing of composite grades across the zone boundaries.
- Spatial 3D mathematical models were fitted to the experimental indicator semi-variograms for each of the zones and used for ordinary kriging of the indicator variable for each of the zones in the model (see Table 14.34).
- Ordinary kriging was used to interpolate the high-grade block indicator proportions using a block discretisation of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the semi-variogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (see Table 14.37).
- A minimum of four and maximum of 16 composites were required to make a block estimate, with a maximum of four composites allowed from a single drill hole (see Table 14.37).

Table 14.37: Kriging Plan for HG Indicator Proportion (HGIND)

Zone	z01	z02	z03	z04	z05	z06	z07	z08	z10
MIN-COMP	4	4	4	4	4	4	4	4	4
MAX-COMP	16	16	16	16	16	16	16	16	16
MAX-PER-DH	4	4	4	4	4	4	4	4	4
MAJOR-SRCH	230	100	30	120	60	300	130	290	50
MINOR-SRCH	40	30	240	30	100	30	30	100	280
VERT-SRCH	150	250	110	250	220	120	220	50	280
ROTATION1	70	48	-10	125	-30	56	111	63	-32
ROTATION2	5	21	-19	-63	-10	16	-66	15	-21
ROTATION3	4	13	17	48	11	18	32	-80	1
SEMI-VAR-MODEL	z01ind	z02ind	z03ind	z04ind	z05ind	z06ind	z07ind	z08ind	z10ind
BLK-CODE Zone	1	2	3	4	5	6	7	8	10
CMP Zone-CODE	1	2	3	4	5	6	7	8	10

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.2.4.2 Gold Grade Estimation Parameters LG Domains

The following is a summary of the parameters used to estimate the LG domain block gold grades in the block model (LGZN1):

- Capped gold grade composites (CAP01) of 2 m were used for ordinary kriging of the LG gold composites into the blocks in the model for Zones 1, 2, 3, 4, 5, 6, 7, 8, 9, and 10. The selection of the composites was controlled by the LG domains 10, 20, 30, 40, 50, 60, 70, 80, 90 and 100.
- Geological zone boundaries based on the mineralised zone wireframes and the domain codes were used to control the selection of the 2 m composites and the blocks to be estimated in the model. There was no sharing of composite grades across the zone or the domain boundaries.
- Spatial 3D mathematical models were fitted to the experimental correlograms of capped gold composites for each of the zones and used for ordinary kriging of the block estimates in the model (see Table 14.35).
- Ordinary kriging was used to interpolate the LG gold grade estimates using a block discretisation of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the correlogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (see Table 14.38).
- A minimum of four and maximum of 12 composites were required to make a block estimate, with a maximum of four composites allowed from a single drill hole (see Table 14.38).

Table 14.38: Kriging Plan for LG Gold Grades (LGZN1)

Zone	z01	z02	z03	z04	z05	z06	z07	z08	z09	z10
MIN-COMP	4	4	4	4	4	4	4	4	4	4
MAX-COMP	12	12	12	12	12	12	12	12	12	12
MAX-PER-DH	4	4	4	4	4	4	4	4	4	4
MAJOR-SRCH-LG	90	150	330	150	90	340	210	130	130	490
MINOR-SRCH-LG	260	60	60	80	160	50	60	290	290	90
VERT-SRCH-LG	300	420	170	270	260	200	260	200	200	320
ROTATION1-LG	-30	52	66	120	-30	56	-12	-27	-27	63
ROTATION2-LG	-15	18	14	-63	-10	16	70	-7	-7	-1
ROTATION3-LG	-6	18	12	48	11	18	90	12	12	23
SPATIAL-MODEL	z01cor	z02cor	z03cor	z04cor	z05cor	z06cor	z07cor	z08cor	z09cor	z10cor
BLK-CODE Zone	1	2	3	4	5	6	7	8	9	10
CMP Zone-CODE	1	2	3	4	5	6	7	8	9	10
LG-DOMAN	10	20	30	40	50	60	70	80	90	100

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.2.4.3 Gold Grade Estimation Parameters HG Domains

The following is a summary of the parameters used to estimate the HG domain block gold grades in the block model (HGZN1):

- Capped gold grade composites (CAP01) of 2 m were used for ordinary kriging of the HG gold composites into the blocks in the model for Zones 1, 2, 3, 4, 5, 6, 7, 8, and 10. The selection of the composites was controlled by the HG domains 11, 21, 31, 41, 51, 61, 71, 81 and 101.
- Geological zone boundaries based on the mineralised zone wireframes and the domain codes were used to control the selection of the 2 m composites and the blocks in the model for grade estimation. There was no sharing of composite grades across the zone or domain boundaries.
- Spatial 3D mathematical models were fitted to the experimental correlograms of capped gold 2 m composites for each of the zones and used for ordinary kriging of the block estimates in the model (see Table 14.35).
- Ordinary kriging was used to interpolate the HG gold grade estimates using a block discretisation of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the semi-variogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (see Table 14.39).
- A minimum of four and maximum of 12 composites were required to make a block estimate, with a maximum of four composites allowed from a single drill hole (see Table 14.39).

Table 14.39: Kriging Plan for HG Gold grades (HGZN1)

Zone	z01	z02	z03	z04	z05	z06	z07	z08	z10
MIN-COMP	4	4	4	4	4	4	4	4	4
MAX-COMP	12	12	12	12	12	12	12	12	12
MAX-PER-DH	4	4	4	4	4	4	4	4	4
MAJOR-SRCH-HG	90	150	330	150	90	340	210	130	490
MINOR-SRCH-HG	260	60	60	80	160	50	60	290	90
VERT-SRCH-HG	300	420	170	270	260	200	260	200	320
ROTATION1-HG	-30	52	66	120	-30	56	-12	-27	63
ROTATION2-HG	-15	18	14	-63	-10	16	70	-7	-1
ROTATION3-HG	-6	18	12	48	11	18	90	12	23
SPATIAL-MODEL	z01cor	z02cor	z03cor	z04cor	z05cor	z06cor	z07cor	z08cor	z10cor
BLK-CODE Zone-HG	1	2	3	4	5	6	7	8	10
CMP-ZOND-CODE-HG	1	2	3	4	5	6	7	8	10
HG-DOMAN	11	21	31	41	51	61	71	81	101

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.2.4.4 Block Gold Grade Estimation

The final block gold grade estimate (BKAU1) is a probability weighted combination of the LGZN1 kriged gold grade estimates with the HGZN1 kriged gold grade estimates using the HGIND as the proportion of high-grade in each block. If a block does not have a HGZN1 estimate or the probability of high-grade is zero, then the block is assigned the LGZN1 grade.

The equation used to calculate the block gold grades (BKAU1) using the HGIND block proportion and the two kriged gold grades is shown below.

$$BKAU1 = (HGZN1 * HGIND) + (LGZN1 * (1 - HGIND))$$

14.2.4.5 Other Gold Grade Models

To ensure that the PAK methodology was working as intended, NN, ID^2, and OK methodologies were used to generate block gold grade estimates in the model. All these methodologies used the same search strategies as the PAK methodology.

Both the ID^2 and OK methodologies gave similar results to the PAK methodology, which indicates that the PAK procedure is working as intended.

The NN block grade estimates were used to determine the declustered global average grade for each zone. The NN block grade estimates are used to ensure that the PAK estimates are globally unbiased. For the NN block grade estimates, 5 m composites were used to estimate the block grades, rather than the 2 m composites to help ensure that all the composites were used in the nearest neighbour estimation. Using the smaller 2 m composites could result in some composites not being considered in the NN estimation, as only the closest composite is assigned to a block.

14.2.4.6 Density Model

An average density was assigned to the blocks in the block model based on measurements of drill core. Two average grades were developed: the mineralised zones 1 to 9 were assigned an average density of 2.73 t/m^3, with Zone 10 (unmineralised) assigned an average density of 2.83 t/m^3. The heavier density for Zone 10 is due to the amount of andesite and gabbro lithologies that are unmineralised.

As no measurements have been made of the overburden material, it was assigned a default value of 2.20 t/m^3, based on McKinstry (1948), gravel plus 10%, pg. 65.

14.2.4.7 Block Model Validation

The block model gold grade estimates (BKAU1) were validated using a series of statistical and graphical methods. These include a check of the global average using the NN model, a comparison with the ID^2 and OK block grade estimates, a check of the global trends using swath plots, and visual validation in plan and section to confirm that the estimates honoured the composite grades, domain and zone boundary conditions and the kriging plan.

Table 14.40 shows summary statistics for the NNAU1 and BKAU1 block grade estimates. Overall, the average block grade estimates are similar: 0.221 g/t Au for NNAU1 and 0.212 g/t for BKAU1. While there are differences for certain zones between NNAU1 and BKAU1 average grades, the most important zones, Zone 1, and Zone 7, show good agreement.

Figure 14-30 displays grade-tonnes curves that compare the BKAU1 block grade estimates and the OKAU1 and IDAU1 block grade estimates. Overall, the BKAU1 estimates (shown in light blue) are less variable than the other two estimates. That is, for the range of likely mining cut-offs, the BKAU1 block grade estimate predicts more tonnes at a lower grade than the other two methodologies. From these grade-tonnes curves it appears that there is sufficient variance reduction incorporated into the BKAU1 block grade estimates to match the proposed 5 m x 5 m x 5 m selective mining unit.

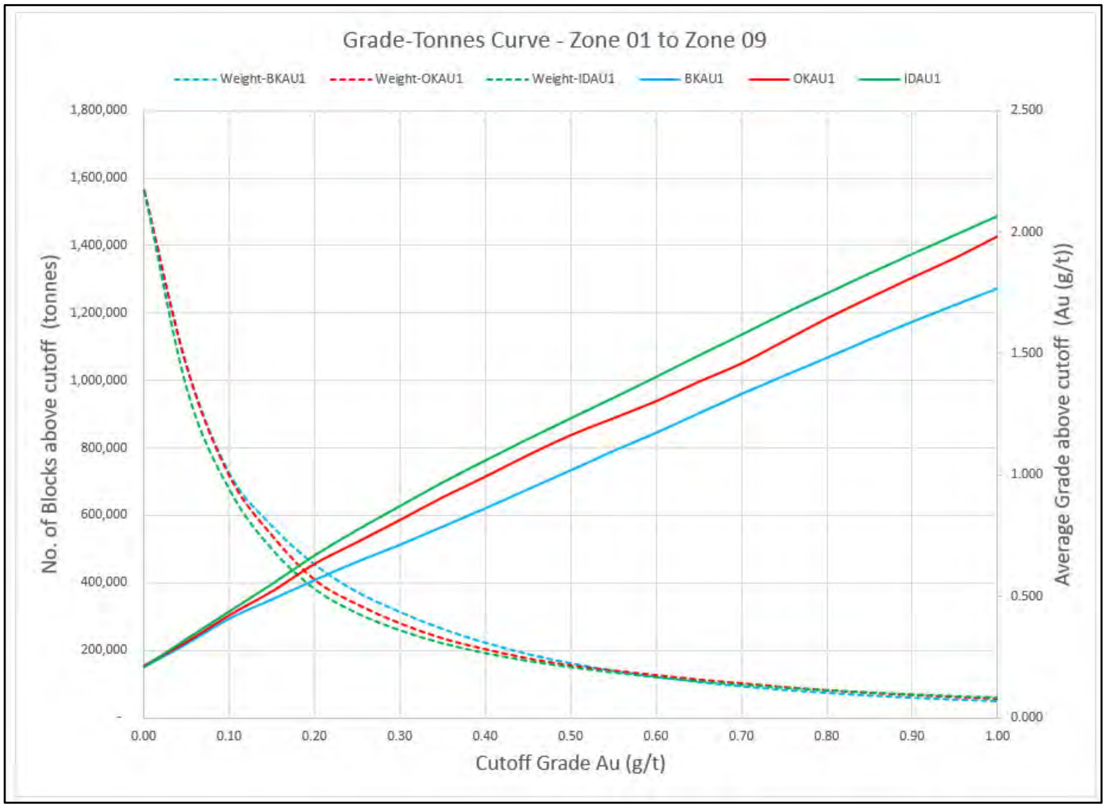
Table 14.40: Summary Statistics for Block Grade Estimates NNAU1 & BKAU1

Zone	No. of Blocks NNAU1	Ave. NNAU1	Std. Dev. NNAU1	No. of Blocks BKAU1	Ave. BKAU1	Std. Dev. BKAU1
1	443,124	0.253	1.376	443,124	0.249	0.568
2	101,557	0.254	1.139	101,557	0.275	0.339
3	201,418	0.235	0.992	201,418	0.235	0.287
4	183,474	0.212	0.715	183,474	0.182	0.217
5	114,344	0.102	0.378	114,344	0.112	0.137
6	127,453	0.203	0.615	127,453	0.197	0.220
7	298,811	0.256	1.189	298,811	0.232	0.388
8	80,821	0.055	0.172	80,821	0.046	0.077
9	9,996	0.157	0.398	9,996	0.182	0.260
Total z01-z09	1,560,998	0.221	1.061	1,560,998	0.212	0.391

Swath plots were generated to determine if the block model gold grade estimates honoured the local trends in gold grade. A swath is the average of the NNAU1 and BKAU1 block grade estimates for collections of blocks. The swath width is 25 m or five blocks in easting, 25 m or

5 blocks in northing, and 10 m or two blocks in elevation. The average swath grade is then plotted versus the easting, northing, and elevation coordinates. There should be reasonable agreement between the trends of the two block grade estimates.

Figure 14-30: Grade-Tonnes Curve for BKAU1, OKAU1 & IDAU1 Block Grade Estimates



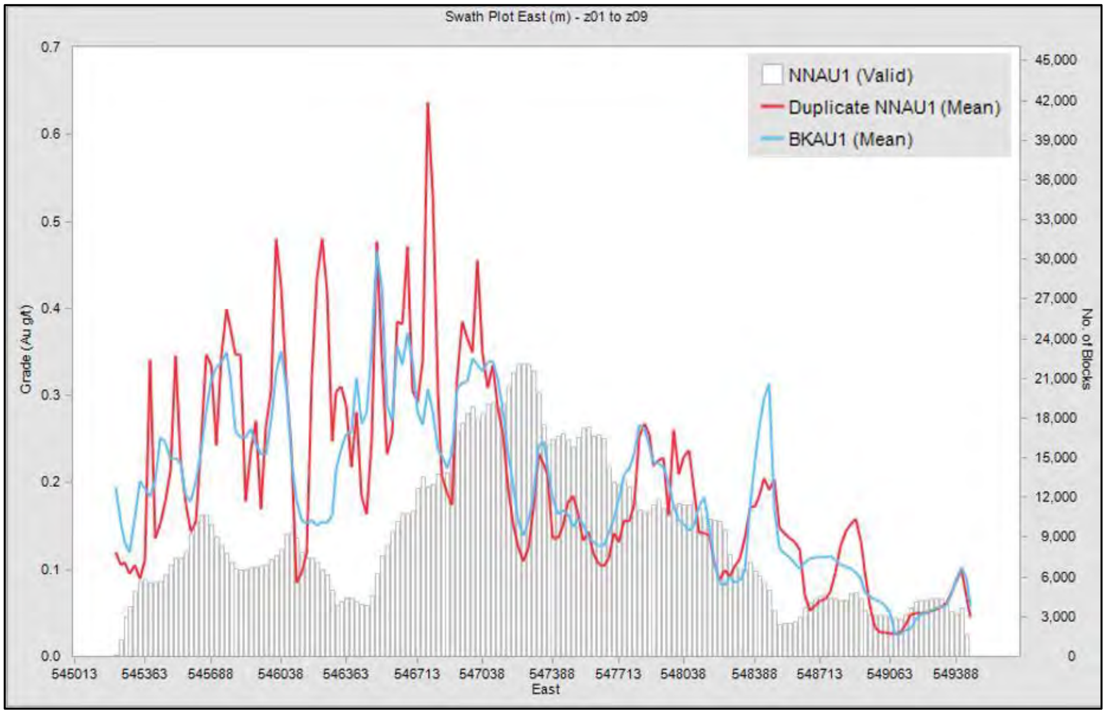
Source: CGK (2020).

Figures 14-31, 14-32 and 14-33 display swath plots of the NNAU1 and BKAU1 block grade estimates. For all three directions, there is acceptable agreement between the two block grade estimation results. That is, the BKAU1 block grade estimates honour the gold grade trends as modelled by the NNAU1 block grade estimates.

Detailed visual inspection of the block grade estimates (BKAU1) were conducted in both plan and section, to ensure that the interpolation results honoured the geological boundaries and the drill hole data. This validation included confirmation of the proper coding of blocks for each of the mineralised zones and the distribution of block gold grade estimates relative to the 2 m drill hole composites, to ensure that the drill hole data were properly represented in the model.

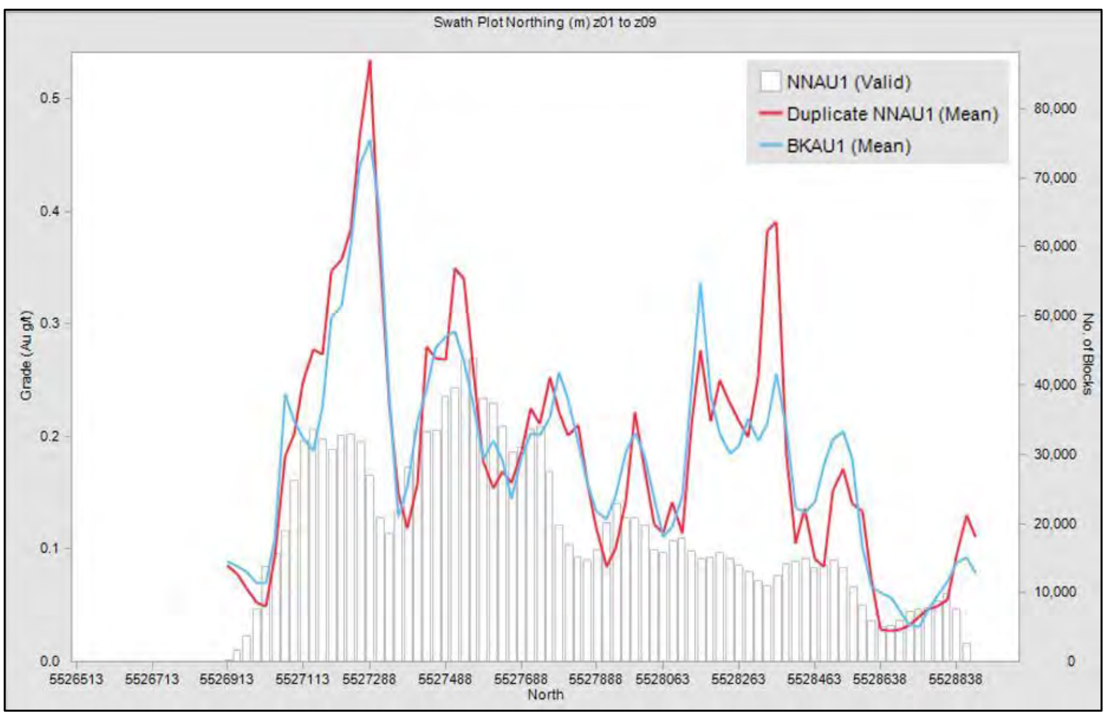
Figure 14-34 displays a cross-section of the block gold grade estimates (BKAU1) for Zone 1 at 547,100 E, looking to the west. There appears to be good agreement between the 2 m composite gold grades and the estimated block model gold grade estimates. There is a marked break between the block model gold grade estimates inside Zone 1 and those outside of Zone 1, which confirms the use of “hard” boundary requirements in the kriging plan.

Figure 14-31: Swath plot in Easting – BKAU1 & NNAU1 Estimates



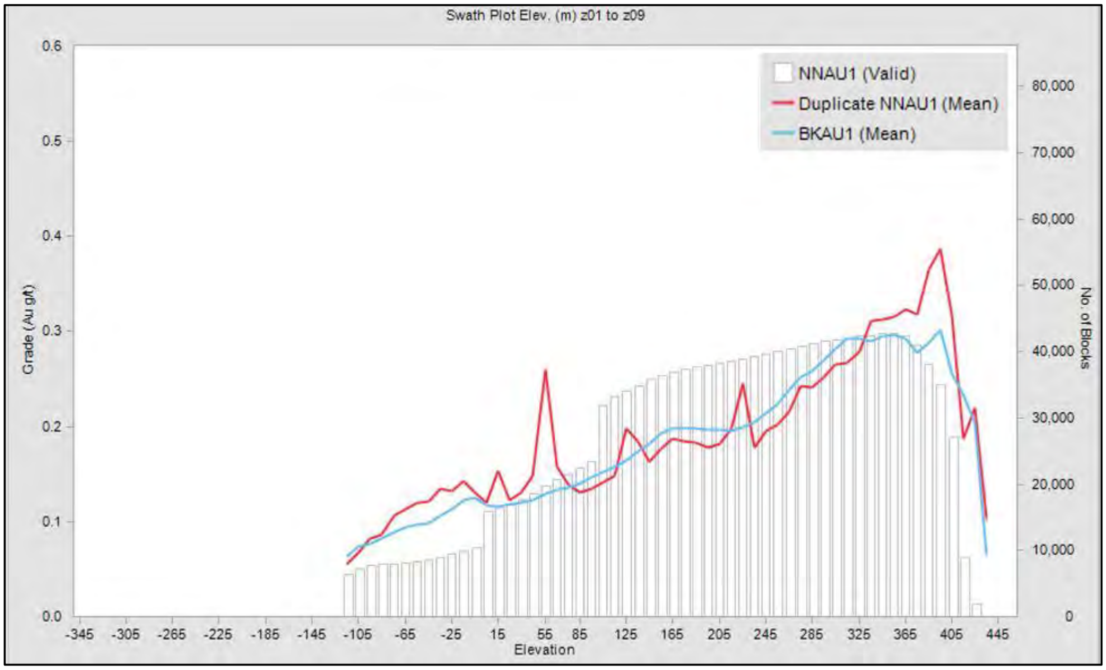
Source: CGK (2020).

Figure 14-32: Swath plot in Northing – BKAU1 & NNAU1 Estimates



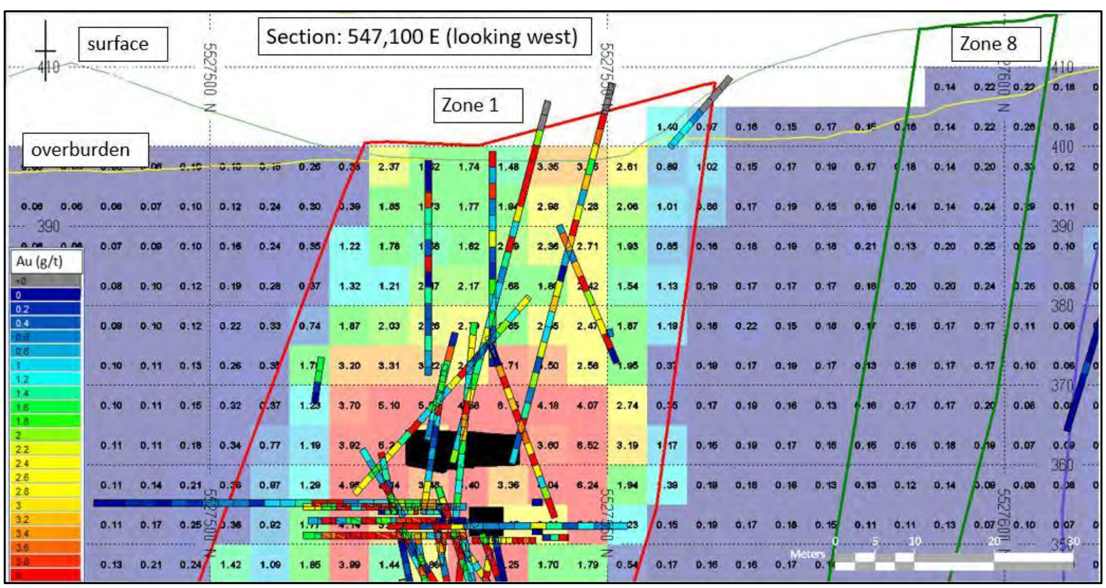
Source: CGK (2020).

Figure 14-33: Swath plot in Elevation – BKAU1 & NNAU1 Estimates



Source: CGK (2020).

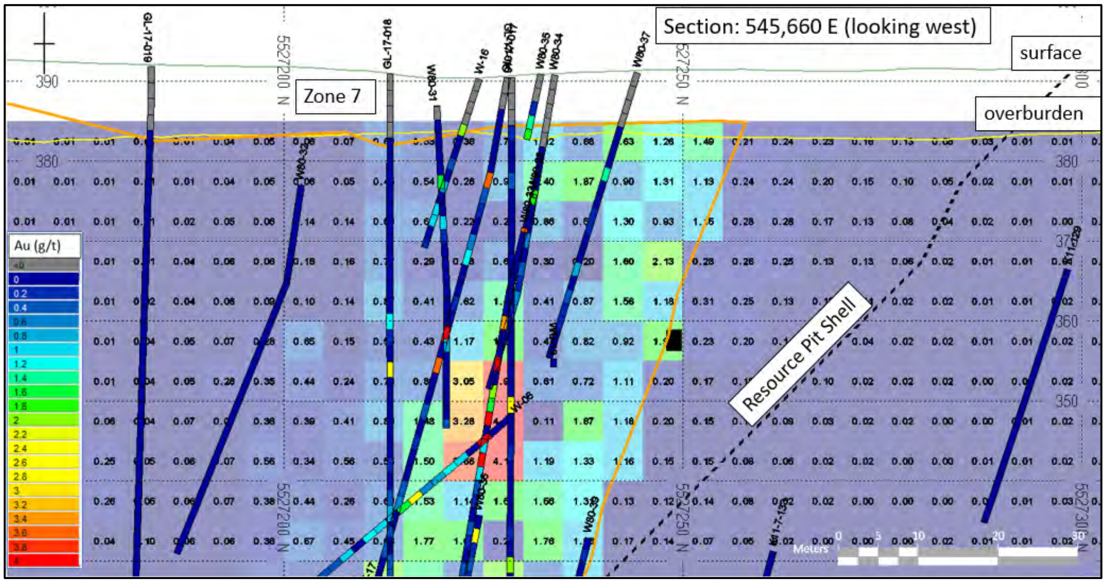
Figure 14-34: Section 547,100 E of Gold Grade Estimates (BKAU1) – Zone 1



Source: CGK (2020).

Figure 14-35 displays a cross-section of the block gold grade estimates (BKAU1) for Zone 7 at 545,660 E, looking to the west. There also appears to be good agreement between the 2 m composite gold grades and the estimated block model gold grade estimates for this zone. There is also a marked break between the block model gold grade estimates inside Zone 7 and those outside of Zone 7, especially near the footwall contact, which again confirms the use of “hard” boundary requirements in the kriging plan.

Figure 14-35: Section 545,660 E of Gold Grade Estimates (BKAU1) – Zone 7



Source: CGK (2020).

The results of the various validation statistical and graphical summaries show that the kriging plan and block model gold grade estimates are working as intended. Based on the validation results, the Qualified Person for this section of the report believes that the block model grade estimates (BKAU1) are suitable for the estimation of mineral resources at Goldlund.

14.2.5 Mineral Resources – Goldlund

14.2.5.1 Classification of Mineral Resources

The classification of the current mineral resources estimate for the Goldlund Project has been carried out in accordance with the May 2014 CIM standards and definitions, as required under N.I. 43-101 regulations. The CIM standards and definitions are described below.

Mineral resources are sub-divided, in order of increasing geological confidence, into “inferred”, “indicated” and “measured” categories. An inferred mineral resource has a lower level of confidence than that applied to an indicated mineral resource. An indicated mineral resource has a higher level of confidence than an inferred mineral resource but has a lower level of confidence than a measured 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, 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.

Material of economic interest refers to diamonds, natural solid inorganic material, or natural solid fossilised organic material including base and precious metals, coal, and industrial minerals.

The term “mineral resource” covers mineralisation and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which mineral reserves may subsequently be defined by the consideration and application of modifying factors. The phrase “reasonable prospects for eventual economic extraction” implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction.

Interpretation of the word “eventual” in this context may vary depending on the commodity or mineral involved. For example, for some coal, iron, potash deposits and other bulk minerals or commodities, it may be reasonable to envisage ‘eventual economic extraction’ as covering periods in excess of 50 years. However, for many gold deposits, application of the concept would normally be restricted to perhaps 10 to 15 years, and frequently to much shorter periods.

The definitions below for the classification of mineral resources were adopted by the CIM Council on May 10, 2014.

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 an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

An “indicated mineral resource” is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit.

Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An indicated mineral resource has a lower level of confidence than that applying to a measured mineral resource and may only be converted to a probable mineral reserve.

A “measured mineral resource” is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit.

Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

A measured mineral resource has a higher level of confidence than that applying to either an indicated mineral resource or an inferred mineral resource. It may be converted to a proven mineral reserve or to a probable mineral reserve.

“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.

The classification parameters considered for the mineral resources at Goldlund are the proximity to and quantity of data used to make the block gold grade estimates. The distance parameters used for the classification were based on a geostatistical method proposed by Davis (1997) that defines confidence limits using large sample normal theory. The confidence limit analysis considers the drill hole spacing, the variability of the data from the correlogram model and the planned production rate. For this study, measured material is considered known within ±15% 90% of the time for a quarterly production period, and indicated material is considered known within ±15% 90% of the time for an annual production period.

The methodology considers an idealised block representing a one-month production period, and a series of grids of different drill hole spacings are used to krig the block to calculate the kriging variance. The nominal one-month production period is approximately a 110 m x 110 m x 5 m panel. The kriging variance needs to be adjusted by the square of the CV to obtain a relative variance as correlogram models were used to krig the panel.

Table 14.41 summarises the confidence limits for different drill hole spacings and the two production periods. A 10 m x 10 m spacing would be sufficient to predict the block grade estimates within ±15% 90% of the time on a quarterly basis. This material would be considered as measured. A 25 m x 25 m spacing would be sufficient to predict the block grade estimates within ±15% 90% of the time on an annual basis. This material would be considered as indicated.

Table 14.41: Drill Hole Spacing used for Mineral Resource Classification

Drill Hole Spacing	3 Months	12 Months
10 m x 10 m - Measured	15.07%	7.53%
25 m x 25 m - Indicated	28.84%	14.42%

The drill hole spacing distances listed in Table 14-40 were then used to code the blocks in the block model using the following conditions to define the indicated resources:

- three holes used to estimate a block value must occur within the required drill hole spacing plus ten percent (25 m + 10%) to account for irregular spacing of the composites; the closest of these samples must be within half the diagonal distance in the grid plus 10% to account for irregularities in the composite spacing;
- or two holes used to estimate a block must occur within the required drill hole spacing plus 10%; the closest composite must be within half the diagonal drill hole spacing plus 10%;
- or one hole used to estimate a block must occur within one-third the required drill hole spacing

To classify measured resources, only the three-hole condition is allowed so that the block is surrounded by data and considers the grid spacing required to classify measured resources.

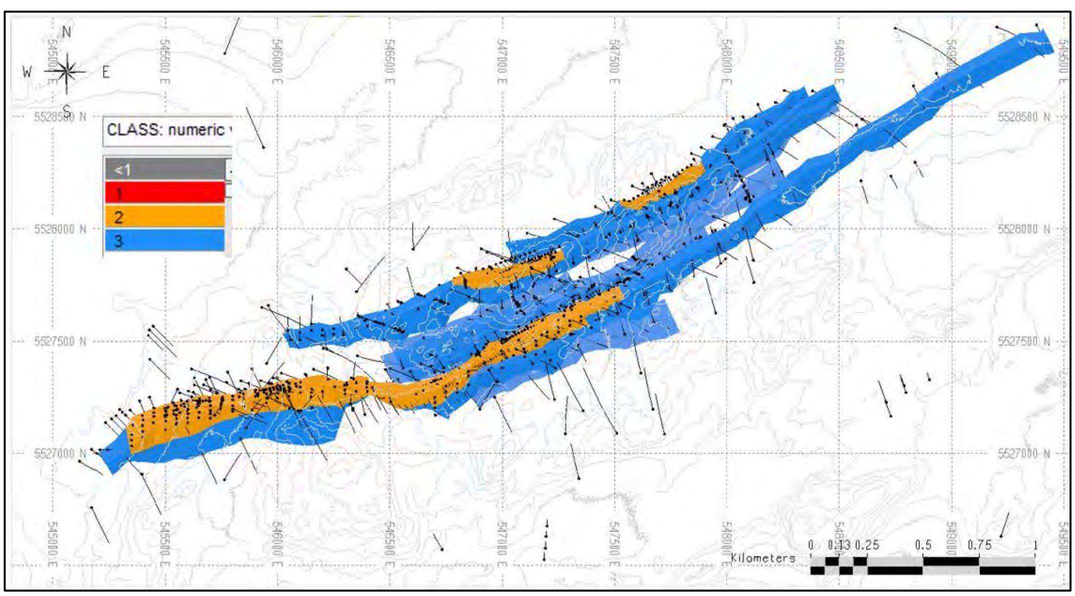
All remaining estimated blocks are classified as inferred. The results of the codification of the blocks using the conditions listed above were then reviewed in 3D using MineSight® software. Then simplified wireframes representing indicated and inferred material were developed in long-section view to smooth the automated codification.

Table 14.42 presents a listing of the simplified classification wireframes along with the classification code assigned to the blocks in the block model and the volume of the wireframes. Figure 14-36 displays a plan view of the simplified classification wireframes, with indicated material shown in orange and inferred material shown in blue for Zones 1 to 9. Not shown are the block gold grade estimates in Zone 10, "unmineralised". All block grade estimates in Zone 10 are considered as inferred material and assigned a classification code of 3.

Table 14.42: Summary of Simplified Classification Wireframes

Wireframe Name	CLASS code	Volume m³
z01_indicated_solid.msr	2	11,736,779
z01_inferred_solid.msr	3	44,282,710.30
z02_indicated_solid.msr	2	1,331,914.77
z02_inferred_solid.msr	3	11,696,586.88
z03_indicated_solid.msr	2	2,472,359.62
z03_inferred_solid.msr	3	24,393,447.79
z04_inferred_solid.msr	3	23,547,095.58
z05_inferred_solid.msr	3	14,162,484.71
z06_inferred_solid.msr	3	15,897,113.73
z07_indicated_solid.msr	2	17,083,484.43
z07_inferred_solid.msr	3	20,710,865.52
z08_inferred_solid.msr	3	10,355,165.17
z09_inferred_solid.msr	3	1,227,815.69

Figure 14-36: Plan of Mineral Resource Classification Simplified Wireframes



Source: CGK (2020).

While there are some areas in Zone 1 that appear to have sufficient drill hole data to support measured resources, these areas have been downgraded to indicated material due to the historical nature of the drill hole data. Therefore, there are only indicated and inferred mineral resources for Goldlund. Currently, there are no measured mineral resources.

A statistical check of the classification criteria was carried out by computing summary statistics for block model attributes that represent the proximity to, and quantity of data used to interpolate the block model estimates. These include averages of the following: number of drill holes, number of composites, average distance of the composites used to estimate a block, the distance of the nearest composite to estimate a block, the distance to the farthest composite used to estimate a block, the kriging slope of regression and the kriging efficiency. The kriging slope of regression provides an assessment of the accuracy of the estimate, while kriging efficiency provides an assessment of the precision of the estimate.

Table 14.43 presents summary statistics for the various attributes used to check the classification criteria. There are 262,278 blocks in the model that have been classified as indicated, while the remaining 1,298,720 have been classified as inferred. That is, approximately 17% of the block gold grade estimates for Zones 1 to 9 have been classified as indicated.

Table 14.43: Summary Statistics of Block Model Attributes Used to Check the Classification

Class	No. of Blocks	Ave. NDIST	Ave. ADIST	Ave. FDIST	Ave. NHOLS	Ave. NCMPS	Ave. KVAR	Ave. KSLP	Ave. KE
2-Indicated	262,278	14.4	22.0	28.8	3	12	0.329	0.4351	-0.088
3-Inferred	1,298,720	56.7	78.6	99.7	3	11	0.436	0.2542	-0.360
	1,560,998	49.6	69.0	87.8	3	11	0.418	0.285	-0.314

From the information in Table 14.43, it can be seen that a typical indicated block gold grade estimate will have been estimated using 12 composites from three drill holes with an average distance to the composites of 22.0 m, with the farthest composite being 28.8 m away, on average, and the closest composite being 14.4 m away, on average. The average kriging slope is 0.44, which means there is some conditional bias to the block grade estimates. The kriging slope of regression is strongly influenced by the large nugget effects that were used to model the experimental gold grade correlograms. The average kriging efficiency is also low, -0.088, which confirms that an indicated classification is appropriate for these block gold grade estimates.

A typical inferred block gold grade estimate will have been estimated using 11 composites from three drill holes, with an average distance to the composites of 79.0 m, with the farthest composite being 100 m away, on average, and the closest composite being 57 m away, on average. The average kriging slope is 0.25, which means there is greater conditional bias for the inferred block grade estimates than for the indicated block grade estimates. The average kriging efficiency for the inferred block grade estimates is also lower than that for indicated block grade estimates at -0.314, which confirms that an inferred classification is appropriate for these block gold grade estimates.

14.2.5.2 Reasonable Prospects of Eventual Economic Extraction

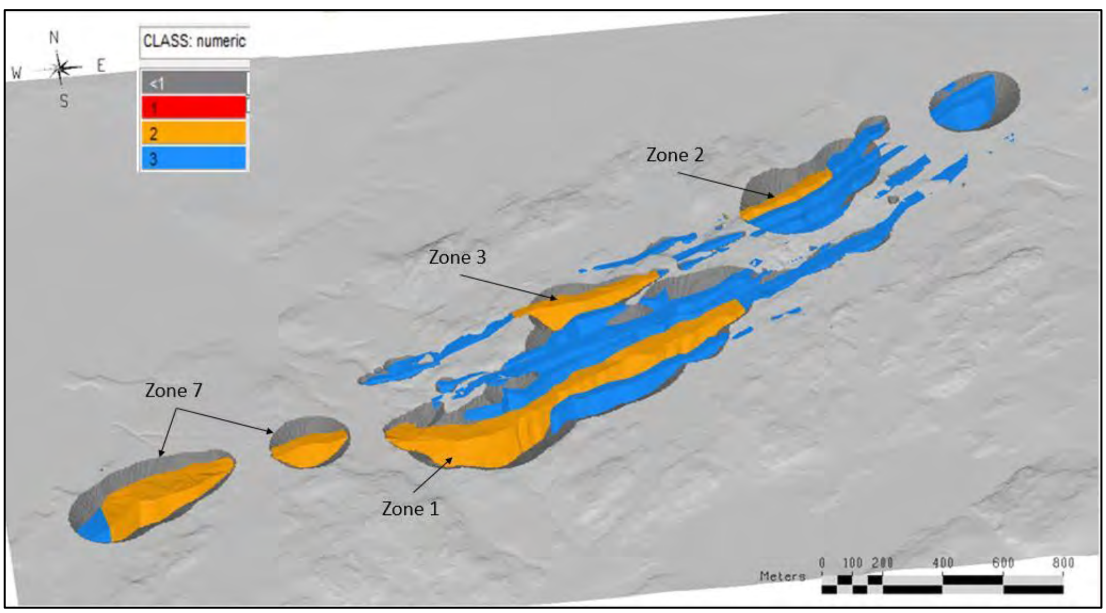
To meet the CIM requirements of reasonable prospects of eventual economic extraction, an optimised pit shell was used to limit the mineral resources estimate at depth. A mineral resource pit shell limit was built by AGP using MineSight software by means of a Lerchs-Grossmann pit design method using the parameters listed in Table 14.44. The pit ("PIT01 TIN 1700Au RSC Goldlund.msr") was optimised using material classified as indicated and inferred.

Table 14.44: Summary of Parameters for Open Pit & UG Mineral Resource Shells

Description	Open Pit Shell	Underground Shell
Mining Cost (C$/t)	$2.48	$77.00
Processing Cost + G&A (C$/t)	$16.03	$16.53
Base Mill Feed Cost (C$/t)	$2.71	
Au Price (US$/oz)	$1,700	$1,700
Foreign Exchange (CAD:USD)	$1.33	$1.33
Gold Recovery Factor (%)	89%	89%
Pit Slope – Constant	48°	
Breakeven Cut-off Grade (g/t Au)	0.26	1.60

Figure 14-37 displays an isometric view looking NE, displaying the mineral resource pit shell (dark grey) and the simplified mineral resource classification wireframes.

Figure 14-37: Isometric View of Mineral Resource Classification Simplified Wireframes & Limiting Pit Shell

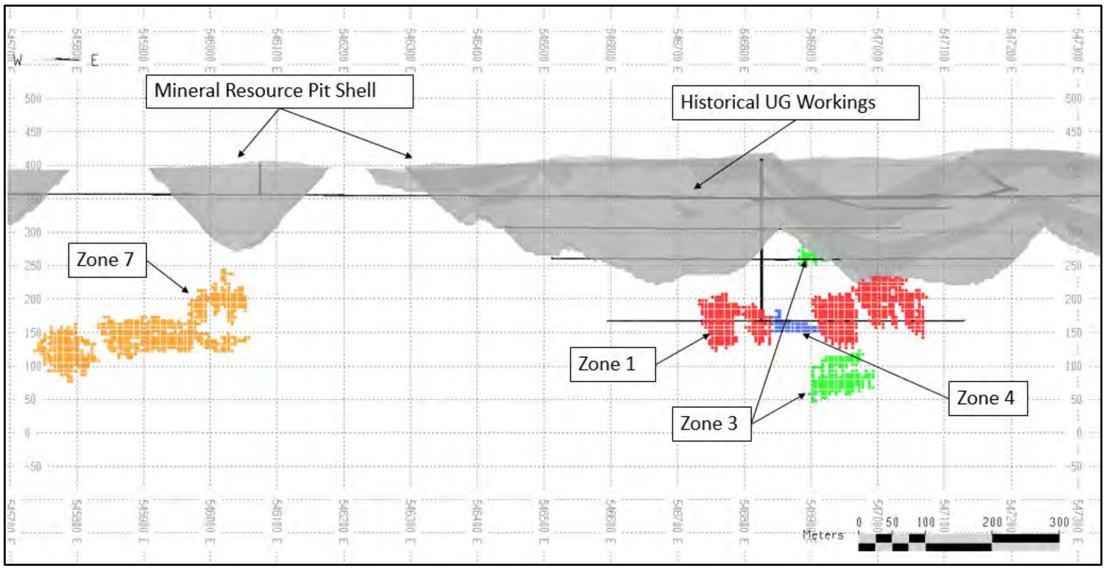


Source: CGK (2020).

The block gold grade estimates that are situated beneath the mineral resource pit shell represent mineralisation that may have reasonable prospects of eventual economic extraction by underground mining methods. To assess the potential for underground mineral resources, a grade shell was generated by AGP using the block model gold grade estimates with a 1.6 g/t Au cut-off, see Table 14-43. Then the grade shell wireframe was edited to remove any isolated blocks so that only large continuous clusters of blocks were kept. Then the edited wireframe grade shell ("Grade shell 1.6 g/t Au MII below 1700 RSC pit shell for UG.msr") was used to constrain the underground mineral resources.

Figure 14-38 displays a long-section view, looking north, of the clusters that were kept as those blocks that have reasonable prospects of eventual economic extraction.

Figure 14-38: Long-section View of the Underground Mineral Resources Shells by Zone



Source: CGK (2020).

14.2.5.3 Mineral Resources Tabulation

The mineral resource estimate was completed by Mr. Chris Keech, P.Geo., a Qualified Person who is independent of Treasury Metals, The limiting pit shell for the resources estimate was developed by Mr. W. Hamilton of AGP. The open pit mineral resources are stated within the mineral resource pit shell described previously and below the overburden surface. The underground mineral resources are stated within the simplified 1.6 g/t Au grade shell, also developed by Mr. Hamilton. The historical underground workings have been removed from the mineral resources estimate.

The indicated mineral resources are inclusive of those resources modified to produce mineral reserves. The mineral resource figures have been rounded to reflect that they are estimates.

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, socio-political, marketing, or other relevant issues. The Qualified Person for this section of the report is not aware of any issues that would materially affect the estimate of the mineral resources as of the date of this report.

There has been insufficient exploration to define the inferred resources as an indicated or measured mineral resources. It is uncertain if further exploration will result in upgrading them to an indicated or measured mineral resources category.

Table 14.45 presents a summary of the open pit mineral resources inside the mineral resource pit shell at a series of cut-offs. The marginal gold cut-off grade is calculated by AGP to be 0.26 g/t Au. Table 14.46 presents a summary of the mineral resources inside the mineral resource pit shell at the 0.26 g/t Au cut-off by mineralised zone.

Table 14.47 presents a summary of the underground mineral resources inside the limiting grade shell envelope at 1.6 g/t Au cut-off.

Table 14.45: Mineral Resources Estimate (Open Pit) by Cut-off (Effective Date: October 23, 2020)

Indicated				Inferred			
Cut-off (g/t Au)	Tonnes (kt)	Grade (g/t Au)	Contained Au (koz Au)	Cut-off (g/t Au)	Tonnes (kt)	Grade (g/t Au)	Contained Au (koz Au)
≥ 0.20	26,600	1.00	860	≥ 0.20	18,500	0.48	290
≥ 0.26	**24,300**	**1.07**	**840**	**≥ 0.26**	**14,400**	**0.56**	**260**
≥ 0.30	23,000	1.12	830	≥ 0.30	12,400	0.60	240
≥ 0.40	20,100	1.23	800	≥ 0.40	8,400	0.72	200
≥ 0.50	17,500	1.35	760	≥ 0.50	5,700	0.86	160
≥ 0.60	15,200	1.47	720	≥ 0.60	4,100	0.98	130
≥ 0.70	13,200	1.59	680	≥ 0.70	2,900	1.12	100
≥ 0.80	11,500	1.72	640	≥ 0.80	2,200	1.24	90
≥ 0.90	10,100	1.84	600	≥ 0.90	1,700	1.35	70

Note: The numbers may not add up due to rounding. The indicated mineral resources are inclusive of those mineral resources modified to produce the mineral reserves. The open pit mineral resources are stated within the mineral resource pit shell “PIT01 TIN 1700Au RSC Goldlund.msr” and below the overburden surface.

Table 14.46: Mineral Resources Estimate (Open Pit) by Zone (Effective Date: October 23, 2020)

Zone	Indicated (≥ 0.26 g/t Au)			Inferred (≥ 0.26 g/t Au)		
	Tonnes (kt)	Grade (g/t Au)	Contained Au (koz Au)	Tonnes (kt)	Grade (g/t Au)	Contained Au (koz Au)
1	14,600	1.28	600	800	0.48	10
2	2,000	0.71	50	1,500	0.75	40
3	1,300	0.83	40	1,900	0.69	40
4	0	0.00	0	5,500	0.51	90
5	0	0.00	0	300	0.36	0
6	0	0.00	0	1,300	0.73	30
7	6,400	0.77	160	400	0.46	10
8	0	0.00	0	200	0.30	0
9	0	0.00	0	800	0.55	10
10	0	0.40	0	1,700	0.40	20
Total	**24,300**	**1.07**	**840**	**14,400**	**0.56**	**260**

Note: The numbers may not add up due to rounding. The indicated mineral resources are inclusive of those mineral resources modified to produce the mineral reserves. The open pit mineral resources are stated within the mineral resource pit shell “PIT01 TIN 1700Au RSC Goldlund.msr” and below the overburden surface.

Table 14.47: Inferred Mineral Resources Estimate (Underground) (Effective Date: October 23, 2020)

Zone	Cut-off (g/t Au)	Tonnes (t)	Grade (g/t Au)	Contained Au (oz Au)
1	≥ 1.6	59,700	11.90	22,900
3	≥ 1.6	50,200	3.80	6,100
4	≥ 1.6	24,600	2.20	1,700
7	≥ 1.6	98,800	6.40	20,400
Total	**≥ 1.6**	**233,200**	**6.80**	**51,100**

Note: The numbers may not add up due to rounding. The indicated mineral resources are inclusive of those mineral resources modified to produce the mineral reserves. The underground mineral resources are stated within the mineral resource shell "Grade shell 1.6 g/t Au MII below 1700RSC pit shell for UG.msr"". The tabulation considers only the kriged high-grade block grade estimates (HGZN1) and the proportion of the high-grade in a block (HGIND) to tabulate the tonnes and grade of the inferred mineral resources.

The open pit mineral resources for the Goldlund Project are estimated to be 24.3 Mt of indicated material grading 1.07 g/t Au for a total of 840 koz of gold. There are additional inferred open pit mineral resources, which are estimated to be 14.4 Mt grading 0.56 g/t Au for a total of 260 koz of gold.

There are also additional inferred underground mineral resources, which are estimated to be 233 kt grading 6.8 g/t Au totalling 51,000 oz of gold. This brings the total inferred mineral resources to be 14.6 Mt, grading 0.66 g/t Au totalling 311 koz of gold.

14.2.6 Comparison with 2019 Mineral Resources Estimate

In 2019, mineral resources were estimated to be 12.9 Mt grading 1.96 g/t Au for a total of 809 koz of gold at a 0.4 g/t Au cut-off and inside the mineral resources pit shell. There were also additional inferred mineral resources that were estimated to be 18.4 Mt grading 1.49 g/t Au for a total of 877 koz at a 0.4 g/t Au cut-off and inside the mineral resources pit shell.

The current 2020 indicated mineral resources estimate has similar ounces of gold, 840 koz compared to the 2019 indicated mineral resources estimate of 809 koz of gold. However, the cut-off grade used for the current mineral resources estimate results in an increase in the tonnes and a decrease in the average grade above cut-off.

There is a large difference in the inferred mineral resources between the 2019 Mineral Resource Estimate and the current mineral resources estimate. The 2019 mineral resources inferred gold ounces were estimated to be 877 koz, compared to the current mineral resources estimate of 260 koz. This is a reduction of more than 600 thousand inferred ounces of gold.

The change in the estimated indicated and inferred mineral resources for Goldlund is due to several factors, including:

- the inclusion of 48 additional drill holes from the 2019-2020 drilling program
- an updated geological interpretation that models nine mineralised zones compared to the seven mineralised zones used for the 2019 Mineral Resources Estimate
- the treatment of unsampled intervals to ensure all unsampled intervals are assigned a background grade prior to compositing, compared to the 2019 approach of leaving many unsampled intervals as unassigned

- capping of the gold grades after compositing because of the variable sample lengths used to sample the drill core, compared to the 2019 approach of capping the assays prior to compositing
- the use of new spatial mathematical models based on the correlogram, that have high nugget effects, compared to the very low nugget effects used for the 2019 variograms
- the use of the probability assisted kriging methodology to better control the high-grade gold assays and to separate out the composite gold grades into more stable statistical domains compared to the 2019 approach that used ordinary kriging inside the mineralised wireframes
- updated parameters for the mineral resources pit shell including a change in the cut-off grade from 0.40 g/t Au used in 2019 to the current cut-off of 0.26 g/t Au

While it is difficult to assign a specific degree of impact to any particular factor, it is believed that the treatment of the unsampled intervals and the probability assisted kriging methodology are the two most significant factors that are responsible for the reduction of the inferred mineral resources.

14.2.7 Peer Review of Mineral Resources Estimate

The decrease in the amount of inferred mineral resources prompted Treasury Metals to have a peer review of the 2020 Mineral Resources Estimate in keeping with Section 6.13 of the CIM Best Practice Guidelines adopted by the CIM Council on November 29, 2019.

Dr. Gilles Arseneau, P.Geo. of Arseneau Consulting Services (ACS) carried out a peer review of the mineral resources estimate prepared by C. Keech of CGK in 2020 and that prepared by WSP in 2019. The following is a summary of the important conclusions and recommendations presented by Mr. Arseneau in his memorandum of October 22, 2020.

ACS provided the following conclusions:

- The two mineral resource estimates provide different interpretations of the same deposit. Both models perform similarly in areas of dense drilling, but the CGK model is more restrictive in areas where the drill hole spacing is wider.
- ACS reviewed the treatment of the missing intervals and, for the most part, agrees with the insertion of very low-grade values for most missing intervals.
- The principal differences between the two mineral resources estimates can be attributed to the change in estimation methodology from ordinary kriging to the PAK and in the numbers of holes used to estimate a block grade specifically for the blocks classified as inferred.
- The geological model (wireframes) could be improved by preparing a high-grade internal model to prevent the smearing of high-grade values.
- The CGK mineral resources estimate probably offers a better representation of the contained metal at Goldlund than the WSP mineral resources estimate. The CGK estimate appears to provide a better estimate of the highly-skewed gold distribution than the WSP estimate, but additional close-spaced drilling is required to verify which of the two models provides a better estimate of the grade continuity at Goldlund.

ACS provided the following recommendations:

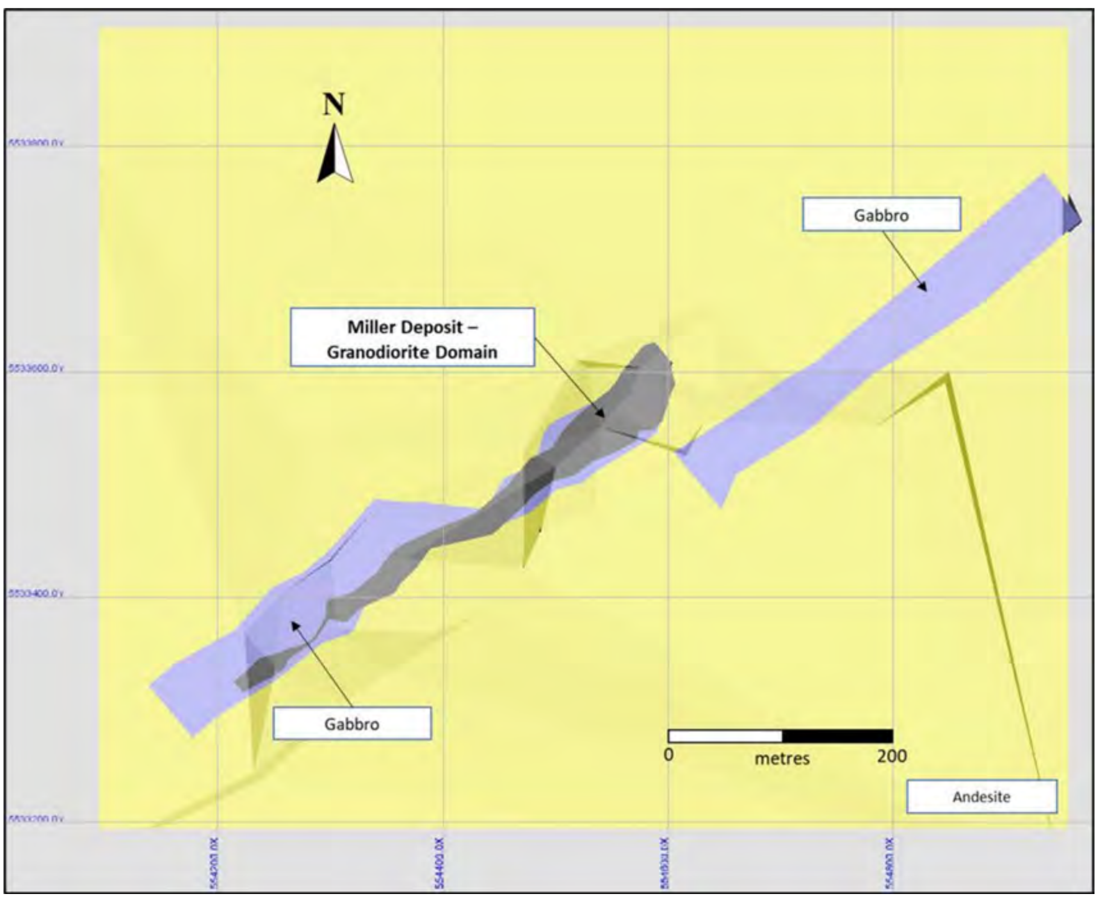
- Treasury Metals consider conducting a close-space drilling program to evaluate which of the two models provides a better estimate of the continuity of grade at Goldlund.
- Treasury Metals consider developing a higher-grade shell using a 1.0 or a 2.5 g/t cut-off generated using LeapfrogGEO® software and that the model be re-estimated using both the broad 0.1 g/t and the higher 2.5 g/t shells to restrict the higher grades within the deposit and prevent the smearing of high-grade values.

14.3 Miller

This section discloses the mineral resources for the Miller deposit, prepared and disclosed in accordance with the CIM's "Standards and Definitions for Mineral Resources and Mineral Reserves" (2014). The Qualified Person responsible for these resource estimates is Mr. Paul Daigle, P.Geo., Senior Resource Geologist for AGP. The effective date of this mineral resource is October 26, 2020.

The resource estimate has been prepared using an interpreted mineralised domain which corresponds to the granodiorite and feldspar porphyry lithologies (Figure 14-39).

Figure 14-39: Plan View of the Miller Deposit



Source: AGP (2020).

The Miller resource estimate was completed using Geovia GEMS™ 6.8.3 resource estimation software. The coordinate system used a UTM coordinates (NAD 83). The resource estimate used a block matrix of 5 m x 5 m x 5 m and the block model was not rotated. The blocks model grades were estimated using inverse distance cubed (ID^3) interpolation method using 2 m capped composite values within a mineralisation envelope which roughly corresponds to a granodiorite and feldspar porphyry lithology. Gold grades were capped prior to compositing.

The mineral resources amenable to open pit extraction are reported within optimised constraining shell at a 0.26 g/t Au cut off grade. The optimised constraining shell was developed by AGP using Hexagon Mining MineSight 3D software and incorporates an assumed metal recovery, geotechnical parameters, and assumed costs from the neighbouring Goldlund deposit, situated approximately 8 km southwest of the Miller deposit. The mineral resources are classified as inferred resources in accordance with the CIM Definitions of Mineral Resources and Mineral Reserves (2014).

Table 14.48 presents the mineral resources for the mineral resources amenable to open pit extraction for the Miller deposit.

Table 14.48: Mineral Resources for the Miller Deposit; within constraining shell

Classification	Cut-off Grade (g/t Au)	Tonnes (kt)	Au Grade (g/t Au)	Contained Au (oz)
Inferred	0.26	1,981	1.24	79,000

Notes: 1. Mineral resources that are not mineral reserves do not have demonstrated economic viability. 2. Summation errors may occur due to rounding. 3. Mineral resources are reported within optimised constraining shell. 4. Block matrix is 5 m x 5 m x 5 m (no rotation). 5. Blocks were estimated using ID^3 on capped 2 m composite values. 6. Capping of grades was at 35.00 g/t Au within the granodiorite domain. 7. The density for the granodiorite domain is 2.82 g/cm^3.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an inferred resource has a lower level of confidence than an indicated resource. It is reasonably expected that most of the inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.3.1 Database

The Treasury Metals drill hole database included 96 drill holes from four target areas. Of these, 40 surface diamond drill holes, totalling 7,385,5 m, were drilled within the Miller deposit area. Of these 40 drill holes, a total 28 drill holes were used in the estimation of mineral resources. Table 14.49 presents a summary of drill holes in the database for the Miller deposit and drill holes used in the estimation of mineral resources.

AGP received the drill hole database as comma delimited format (CSV) and included tables for collar, survey, assay, and lithology. The drill hole database was imported into GEMS and verified using the GEMS validation tool to check for overlapping intervals. No errors were found.

All data received was in the NAD83 UTM grid coordinate system.

Table 14.49: Summary of Drill Hole Database for the Miller Deposit

Year	Total Drill Holes	Total Metres (m)	Drill Holes in Resource	Total Metres (m)
2018	8	1255.5	8	1255.5
2019	32	6130.0	20	3724.0
Totals	40	7385.5	28	4979.5

AGP reviewed approximately 38% of the assay database (1,549 assays out of 3,906) distributed over the deposit comparing the results from the assay certificates issued by the laboratory. Only one typographic error was found and corrected in the database.

During the check on the database review, 40 assay values were labelled with the incorrect laboratory certificate number, or numbers where assay values were an average of analyses. These are not considered as errors; however, for consistency, it is recommended that the database be updated with the correct laboratory certificate or certificates.

It should also be noted, in drill hole MI-19-040, the lithology is logged as a gabbro yet intersects the core granodiorite domain. During the site visit, this intersection was compared to MI-18-002 and MI-19-015, situated to the southwest and northeast, respectively. The lithology in MI-19-040 matches that in the surrounding drill holes therefore, for purposes of interpretation, the gabbro in MI-19-040 is treated as granodiorite. It is recommended that these three drill holes be revisited and, where necessary, re-logged.

The author is of the opinion that the database is adequate for the purposes of mineral resource estimation for the project.

14.3.2 Geological Model

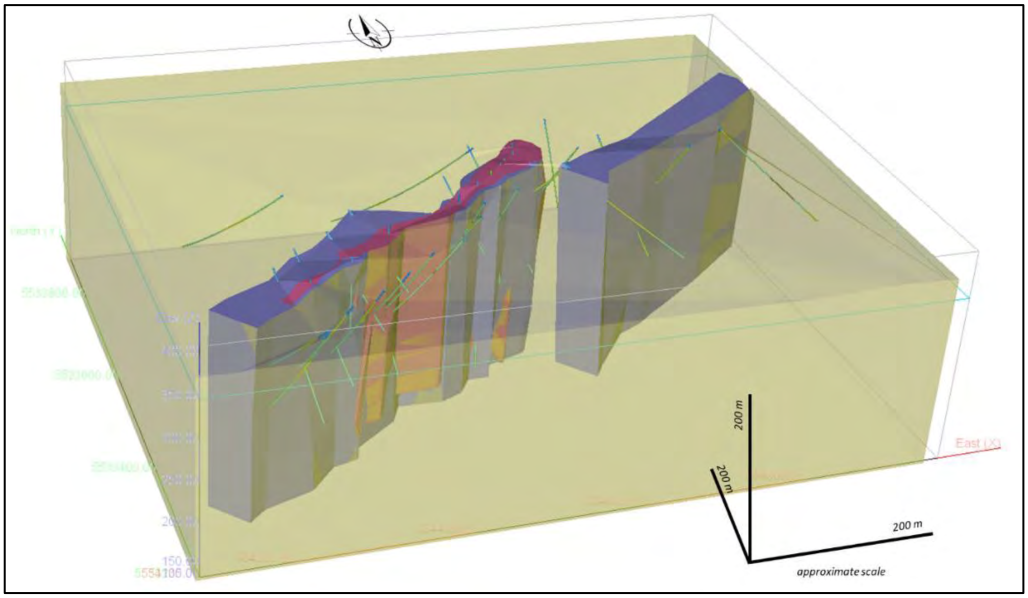
The mineralised granodiorite domain at Miller deposit was created using conventional polylines on vertical sections defined along 10 m to 25 m spaced sections. The polylines capture the mineralised granodiorite and includes few feldspar porphyry units. This domain is host to the gold mineralisation.

The gabbro and andesite lithologies were modelled as separate domains and represent the surrounding country rock. Minor intercepts of dacite, tuff and diorite were incorporated into the andesite wireframe. The gabbro and andesite wireframes are considered as waste and were not used in the resource estimation.

A topographic surface was created from drill hole collars and an overburden surface was created based on the logged overburden or casing. An overburden wireframe was created between these tow surfaces. The granodiorite, gabbro and andesite wireframes were clipped to the overburden solid wireframe. The four wireframes were validated, and no errors were found.

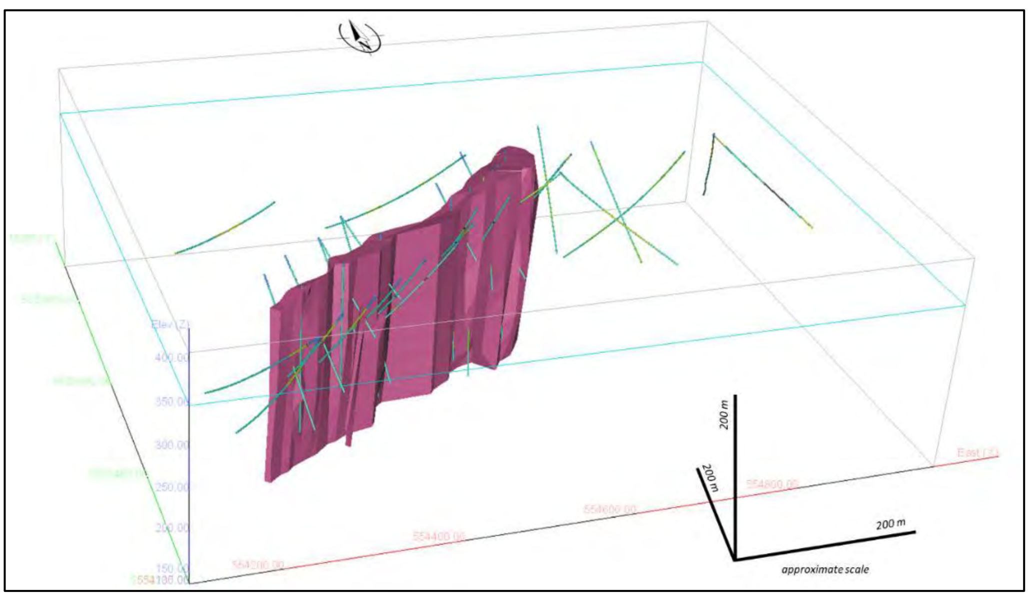
Figure 14-40 shows the granodiorite, gabbro, and andesite wireframes for the Miller deposit with intersecting drill holes. Figure 14-41 shows the mineralised granodiorite wireframe with intersecting drill holes.

Figure 14-40: Geological Domains of the Miller Deposit Looking North-Northeast



Source: AGP (2020).

Figure 14-41: Granodiorite Domain of the Miller Deposit Looking North-Northeast



Source: AGP (2020).

Table 14.50 lists the mineralised domain wireframes and subdomains for the Miller deposit.

Table 14.50: Domains & Subdomains – Miller Deposit

Domain	Rock Code	Rock Type	Comment
Granodiorite + Feldspar Porphyry	GRD	400	Mineralised Lithology
Andesite	AND	200	Waste Rock / Country Rock
Gabbro	GAB	210	Waste Rock / Country Rock
Overburden	OVB	99	
Air	AIR	0	

14.3.3 Exploratory Data Analysis

14.3.3.1 Raw Assays

The drill hole database for Miller deposit data consists of 40 drill holes and 3,906 assay values for gold. Any assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned a zero. Of this total, 28 drill holes and 2,551 assay values were used in the resource estimation. Table 14.51 presents the descriptive statistics for the Miller database. Table 14.52 presents the descriptive statistics for gold assays values for the 28 drill holes used in the resource estimate by domain.

Table 14.51: Descriptive Statistics – Miller Deposit

Statistic	Au (g/t)	Length (m)
Count	3906	3906
Minimum	0.0025	0.29
Maximum	137.00	1.88
Mean	0.52	0.99
Median	0.006	1.00
Standard Deviation	3.67	0.10
CV	7.06	0.10

Table 14.52: Descriptive Statistics by Domain – Miller Deposit

Domain	Granodiorite (400)	Gabbro (210)	Andesite (200)
Count	2551	388	658
Minimum	0.0025	0.0025	0.0025
Maximum	137.00	15.33	4.18
Mean	0.77	0.11	0.03
Median	0.03	0.002	0.002
Standard Deviation	4.50	0.86	0.23
CV	5.85	8.09	9.14

14.3.3.2 Capping Analysis

Capping analysis was carried out on gold values within the mineralised granodiorite domain using histogram and disintegration plots. Table 14.53 presents the capping level for gold in the granodiorite domain for the Miller deposit. Table 14.54 presents the descriptive statistics for uncapped and capped gold assay values in the granodiorite domain.

Table 14.53: Capping Level – Granodiorite Domain

Domain	Au (g/t Au)	No. of Values Affected	% Loss
Granodiorite	35.00	8	12.0

Table 14.54: Descriptive Statistics for Uncapped & Capped Gold Assays – Granodiorite Domain

Domain	Uncapped Au (g/t Au)	Capped Au (g/t Au)	Length (m)
Count	2551	2551	2551
Minimum	0.0025	0.0025	0.30
Maximum	137.00	35.00	1.88
Mean	0.77	0.67	0.99
Median	0.03	0.03	1.00
Standard Deviation	4.50	2.72	0.10
CV	5.85	4.06	0.10

14.3.3.3 Composites

Composites were created after capping of assay values. The assay intervals situated within the granodiorite domain were composited to two metre lengths within each mineralised domain wireframe where the composite lengths were adjusted across the intersection of the granodiorite. Table 14.55 presents the descriptive statistics for the uncapped and capped 2 m composite values for gold.

Table 14.55: Descriptive Statistics for Uncapped & Capped Gold Assays – Granodiorite Domain

Domain	Uncapped Au (g/t Au)	Capped Au (g/t Au)	Length (m)
Count	1280	1280	1280
Minimum	0.00	0.00	1.84
Maximum	47.50	19.90	2.20
Mean	0.68	0.61	2.00
Median	0.08	0.08	2.00
Standard Deviation	2.56	1.70	0.02
CV	3.74	2.80	0.01

14.3.3.4 Bulk Density

Density test work completed by First Mining was collected from 389 core samples across all 40 drill holes in the Miller deposit. The core samples tested were generally whole core pieces ranging in length from approximately 10 to 15 cm. Core samples were then weighed in air and in water. The mean value was assigned to the three interpreted domains. Overburden was assigned a density of 2.2. Table 14.56 shows the descriptive statistics for density used in the Miller deposit by domain.

Table 14.56: Descriptive Statistics for Bulk Density – by Domain

Domain	Granodiorite	Gabbro	Andesite	Overburden
Count	148	88	153	
Minimum	2.62	2.74	2.71	
Maximum	3.03	3.12	2.08	
Mean	2.82	2.93	2.83	2.20 (assigned)
Median	2.83	2.91	2.83	
Standard Deviation	0.07	0.08	0.08	
CV	0.02	0.03	0.03	

14.3.3.5 Spatial Analysis - Variography

Spatial analysis was performed on 2 m composites from the northeast half of the granodiorite domain where the majority of data is situated. Due to the high variability of data and low sample support, orientations of the variograms appear forced in the direction of the vertical drill holes. Definitive variograms could not be prepared for the Miller deposit at this time.

14.3.4 Block Mode Interpolation

14.3.4.1 Block Model

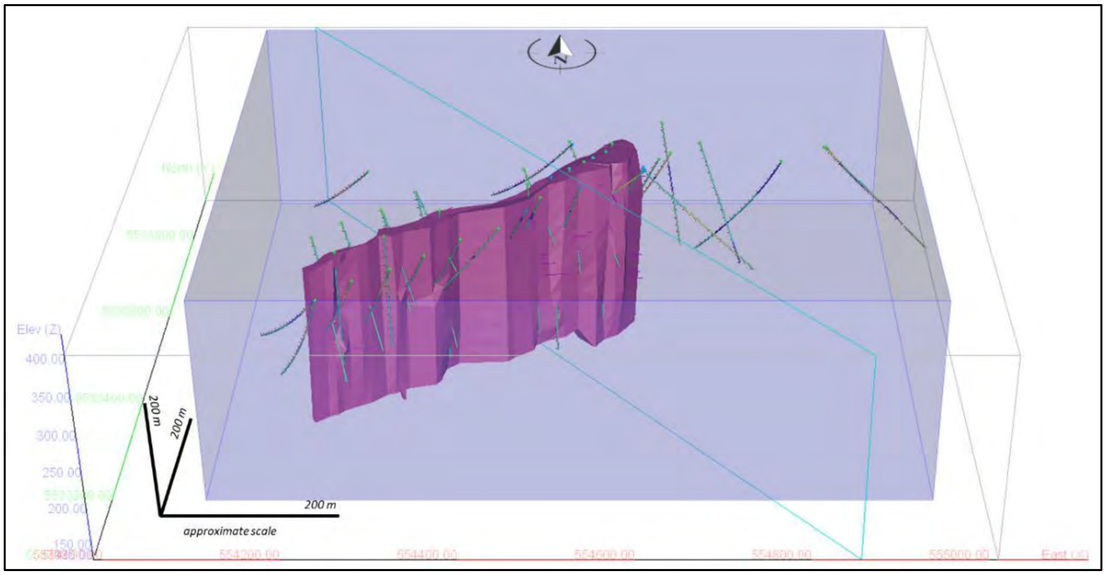
The block model for Miller deposit was created with a block matrix of 5 m long by 5 m wide by 5 m high and is not rotated. The block matrix was selected as appropriate based on the drill spacing and the block height and in consideration of an open pit scenario. The datum for UTM coordinates used are in NAD83.

Table 14.57 summarises the block model parameters and Figure 14-42 illustrates the block model over the interpreted mineralised domains for the Miller deposit.

Table 14.57: Block Model Parameters – Miller Deposit

Parameter	Minimum	Maximum	No. of Blocks
Easting	554100	554950	170
Northing	5533200	5533900	140
Elevation	135	410	55
Rotation Angle	No rotation°		
Block Size (X, Y, Z in metres)	5 m x 5 m x 5 m		

Figure 14-42: Block Model of the Miller Deposit Looking North



Source: AGP (2020).

The block model is a whole block model where blocks are assigned a specific rock type code. Any block with greater than 50% within the mineralised domain wireframe was assigned that code. The volume of the coded blocks was compared to the analytical volume and was found to be within 0.2%.

Block model attributes in the block model include:

- rock type
- density
- capped and uncapped gold grades
- classification
- distance to the nearest composite
- number of composites used in estimation of block
- number of drill holes used for estimation of block
- pass number

14.3.4.2 Estimation/Interpolation Methods

The gold grades were interpolated in two passes using the 2 m capped composite values by the ID^3 interpolation method. OK, ID^2, and NN interpolations were also run for validation purposes. Grades were interpolated within the mineralised granodiorite domain. Table 14.58 shows estimation parameters for each pass used to estimate gold grades. Table 14.59 shows the search ellipse parameters for the Miller deposit.

Table 14.58: Block Model Parameters – Miller Deposit

Pass	Min. No. Composites	Max. No. Composites	Max. Composites	Min. No. of Drill Holes
Pass 1	4	20	3	2
Pass 2	3	20	3	1

Table 14.59: Block Model Parameters – Miller Deposit

Pass	Anisotropy	Azimuth (°)	Dip (°)	Azimuth (°)	Range X (m)	Range Y (m)	Range Z (m)	Search
Pass 1	Az,Dip,Az	167.59	-51.69	146.22	30	30	15.06	Ellipsoidal
Pass 2	Az,Dip,Az	167.59	-51.69	146.22	50	50	25	Ellipsoidal

14.3.4.3 Block Model Validation

Various methods to validate the block model included:

- statistical comparison of resource assay and block grade distributions
- visual inspection and comparison of block grades with composite and assay grades
- inspection of swath plots with composites and block grades elevations and northings

The block grades were compared with the composite grades on sections and plans and found good overall visual correlation. Grades are most consistent nearest the drill holes as expected.

Table 14.60 presents a comparison of the mean interpolated gold grades and 2 m composite values. The comparison between interpolated block values and composite values does not appear to show a bias in the estimation. The mean gold grades from the different interpolations are negligible.

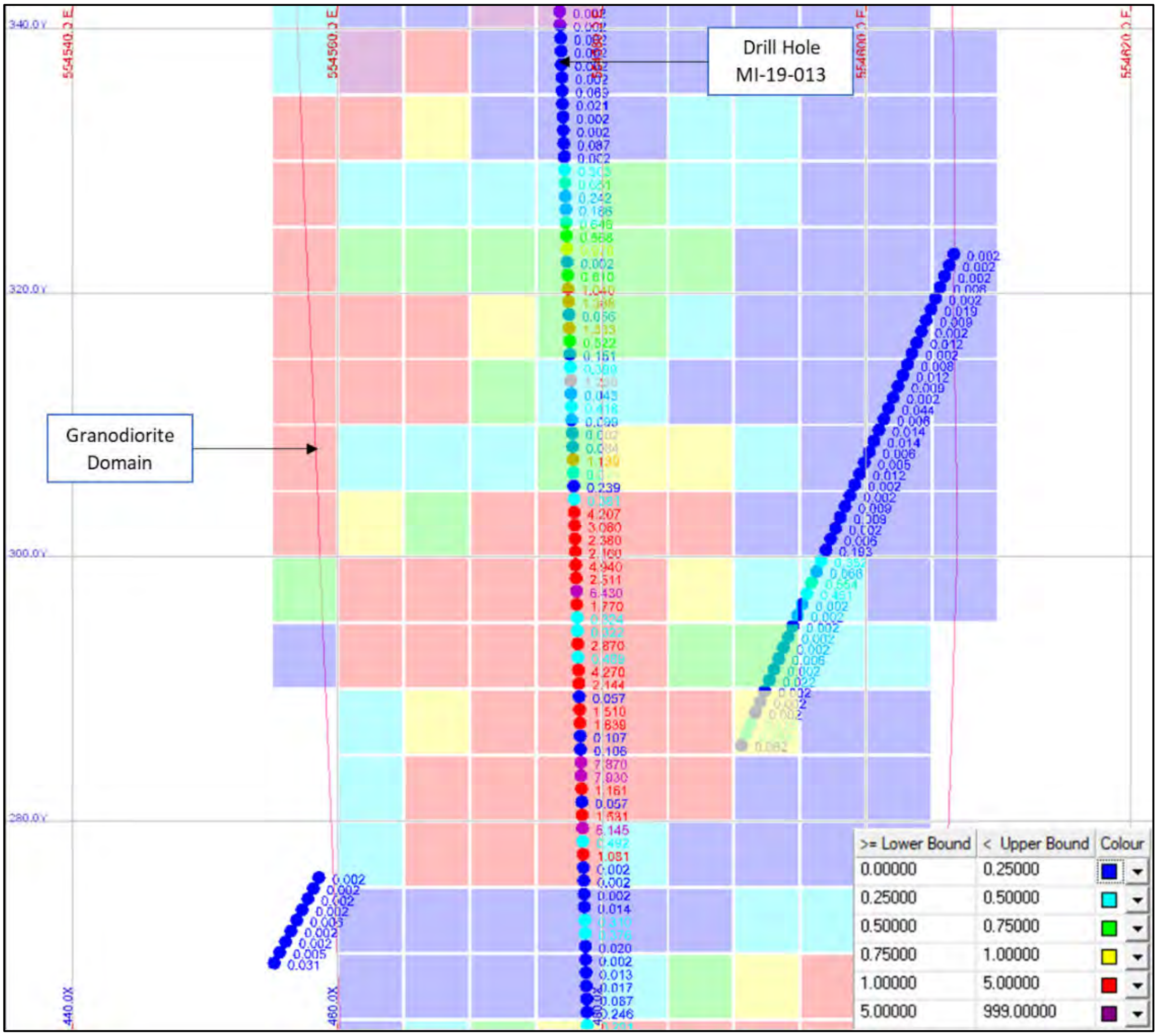
Table 14.60: Block Model Parameters – Miller Deposit

Interpolation	Au (g/t)
ID^3	0.526
ID^2	0.525
OK	0.525
NN	0.540
2 m Composite	0.625

Visual comparison of the ID^3 block grades and the 2 m composite values do not show an apparent bias. Figures 14-43 and 14-44 present cross-section views for the Miller looking north and looking west, respectively.

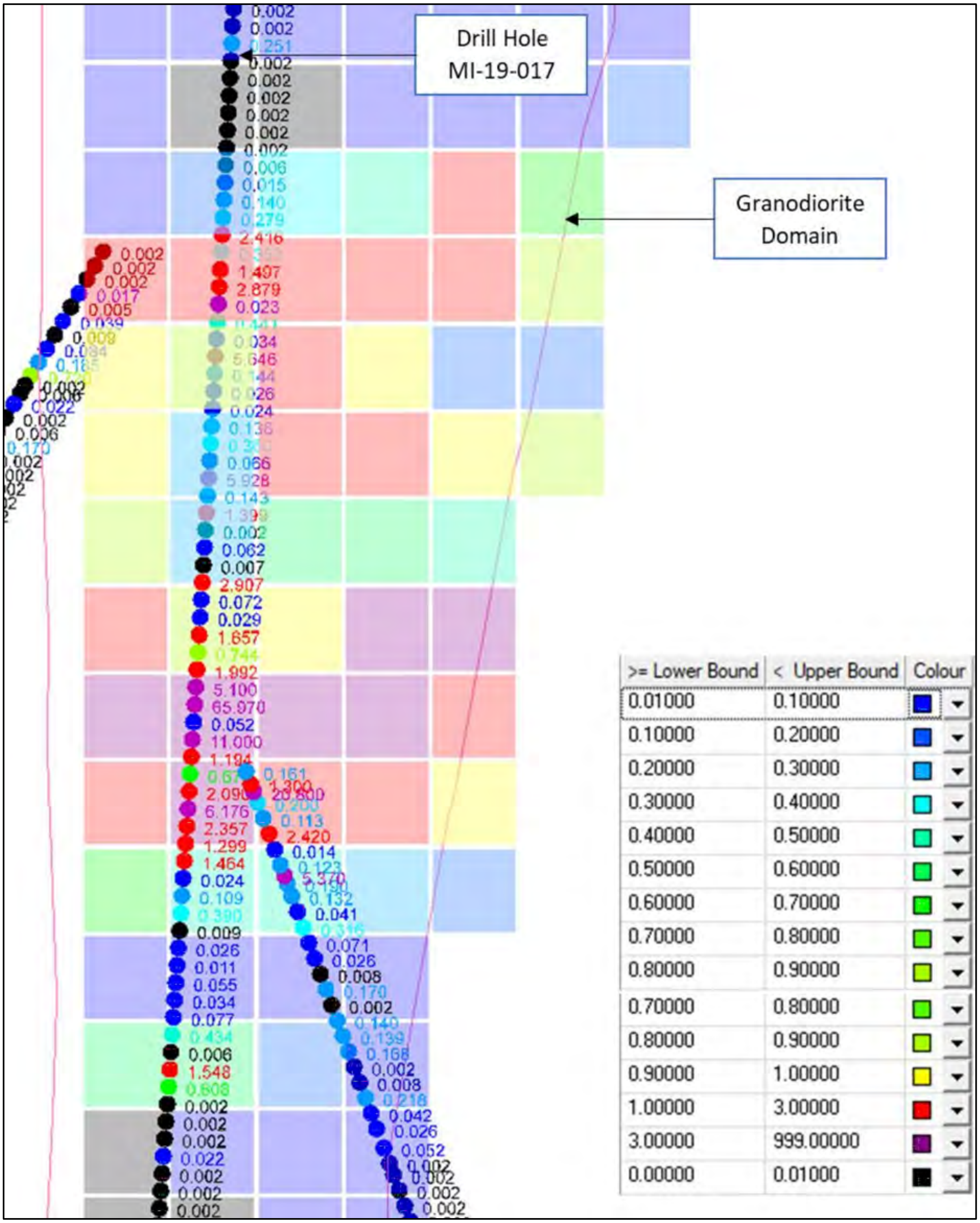
Swath plots by northing, easting and by elevation were reviewed for the Miller deposit. The distribution of gold composites and interpolated block grades show no issues were found with the distribution of interpolated grades. Figures 14-45, 14-46, and 14-47 present the swath plots by easting, northing, and elevation for gold, respectively.

Figure 14-43: Miller Deposit – Cross-section 5533600mN; Looking North



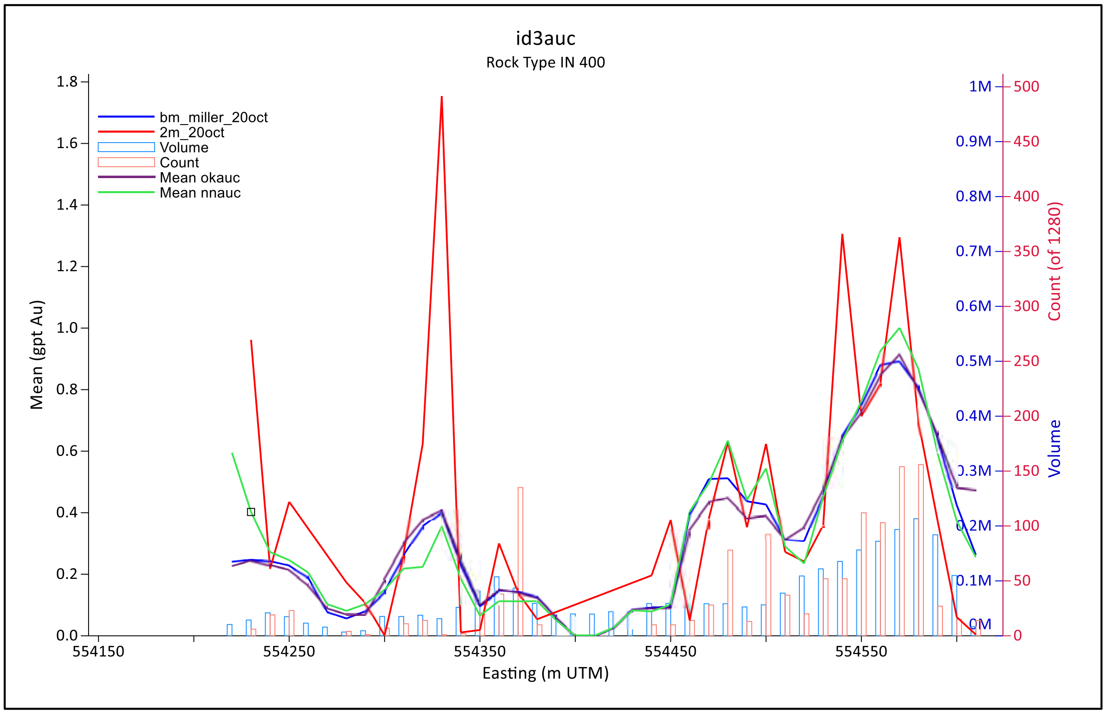
Note: Blocks are 5 m x 5 m. Source: AGP (2020).

Figure 14-44: Miller Deposit– Cross-section 554500mE; Looking West



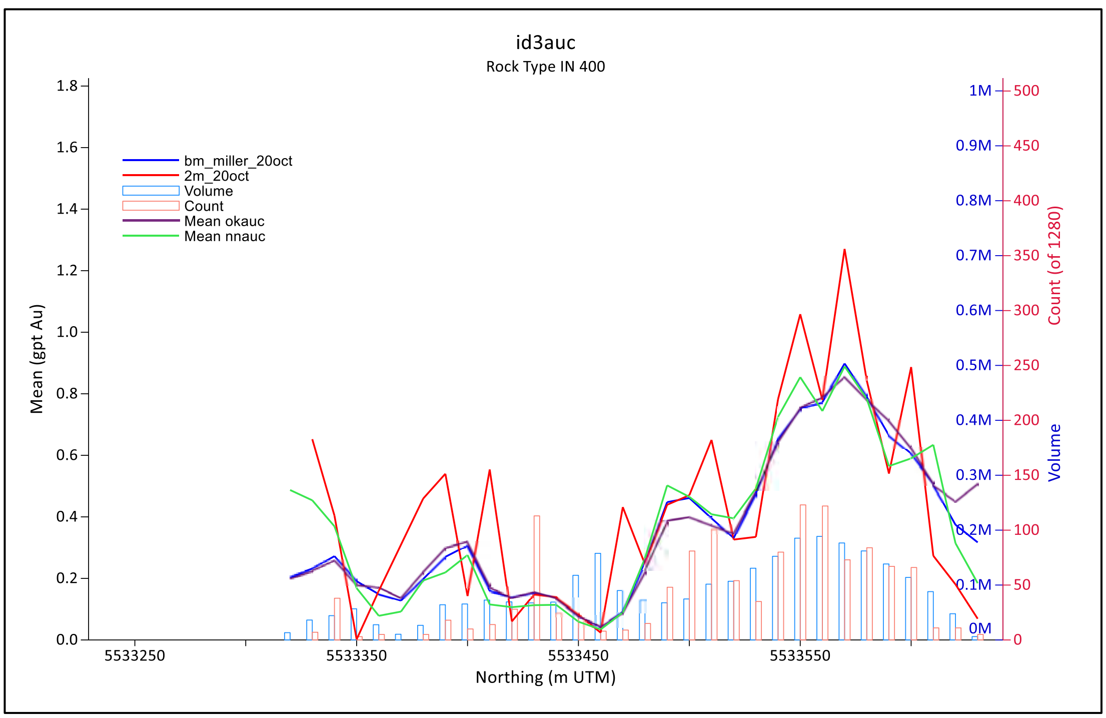
Note: blocks are 5 m x 5 m. Source: AGP (2020).

Figure 14-45: Gold Swath Plot by Easting



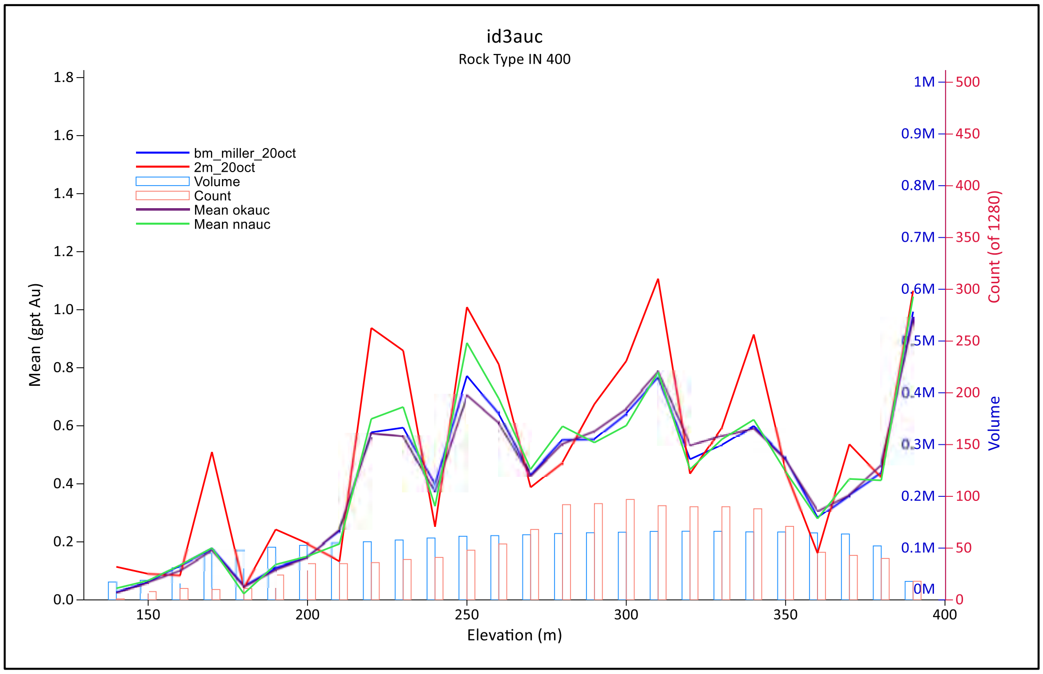
Source: AGP (2020).

Figure 14-46: Gold Swath Plot by Northing



Source: AGP (2020).

Figure 14-47: Gold Swath Plot by Elevation



Source: AGP (2020).

14.3.5 Mineral Resources – Miller

14.3.5.1 Classification of Mineral Resources

Definitions for mineral resource categories used in this report are consistent with those defined by CIM (2014) and referenced by N.I. 43-101. In the CIM classification, a mineral resource is defined as "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 mineral resources for the Miller deposit were classified as inferred mineral resources. Blocks interpolated with a minimum of 2 drill holes and nominally within 50 m from the nearest drill hole were classified as inferred.

It should be noted that the core of the deposit in the northeast of the granodiorite may be upgraded based on current drilling once the direction, or directions, of mineralisation is confirmed via televiewer. Future drilling should include oriented drill core to ascertain vein sets and directions. With this information, a higher confidence in mineralisation direction should render a better representation of the mineralisation at the Miller deposit.

14.3.5.2 Cut-off Grade for Mineral Resources

AGP has determined a resource cut-off grade of 0.26 g/t Au to be used for reporting of the mineral resources within constraining shells for the material amenable to open pit extraction. The cut-off grades are based on the parameters defined below.

14.3.5.3 Reasonable Prospects of Eventual Economic Extraction

To meet the CIM requirements of reasonable prospects of eventual economic extraction, an optimised constraining shell was used to report mineral resources amenable to open pit extraction. The constraining shell was built by AGP on the Miller block model using MineSight software (Lerchs-Grossman method). Table 14.61 shows the economic assumptions made to constrain the reported mineral resources.

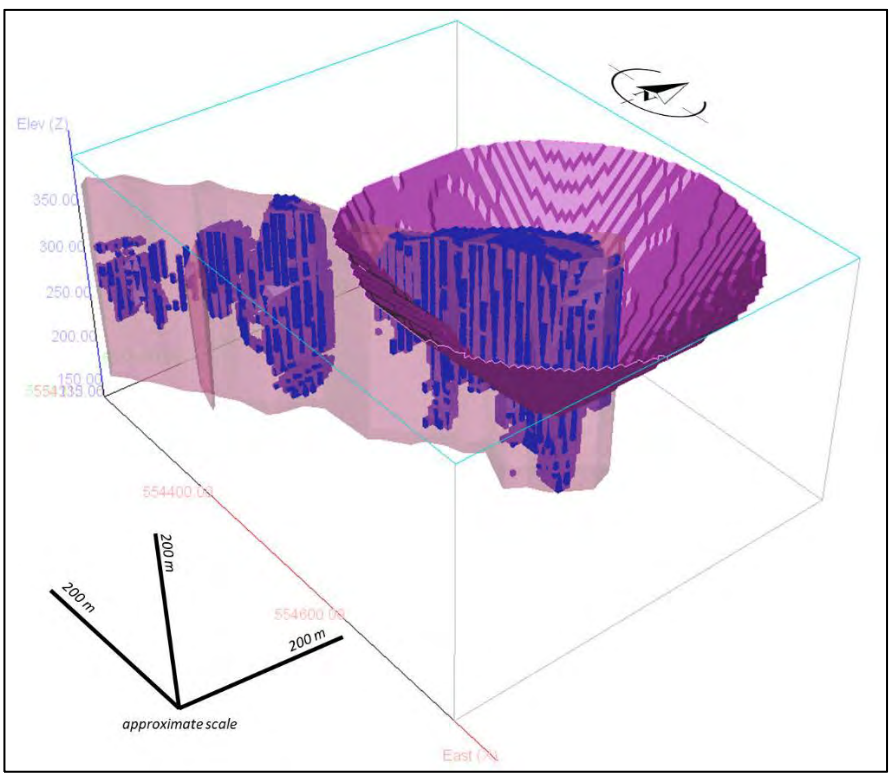
Table 14.61: Assumed Parameters for the Constraining Shell

Parameter	Units	Value
Gold Price	US$/oz Au	1700.00
Gold Recovery	%	89
Exchange Rate	CAD:USD	1.33
Mining Rate – Open Pit	t/d	5,000
Mining Cost – Open Pit	US$/t	2.48
Base Mill Feed Cost	US$/t	2.71
Processing and G&A Cost	US$/t	16.03
Pit Slope	degrees	48

Note: G&A = general and Administration.

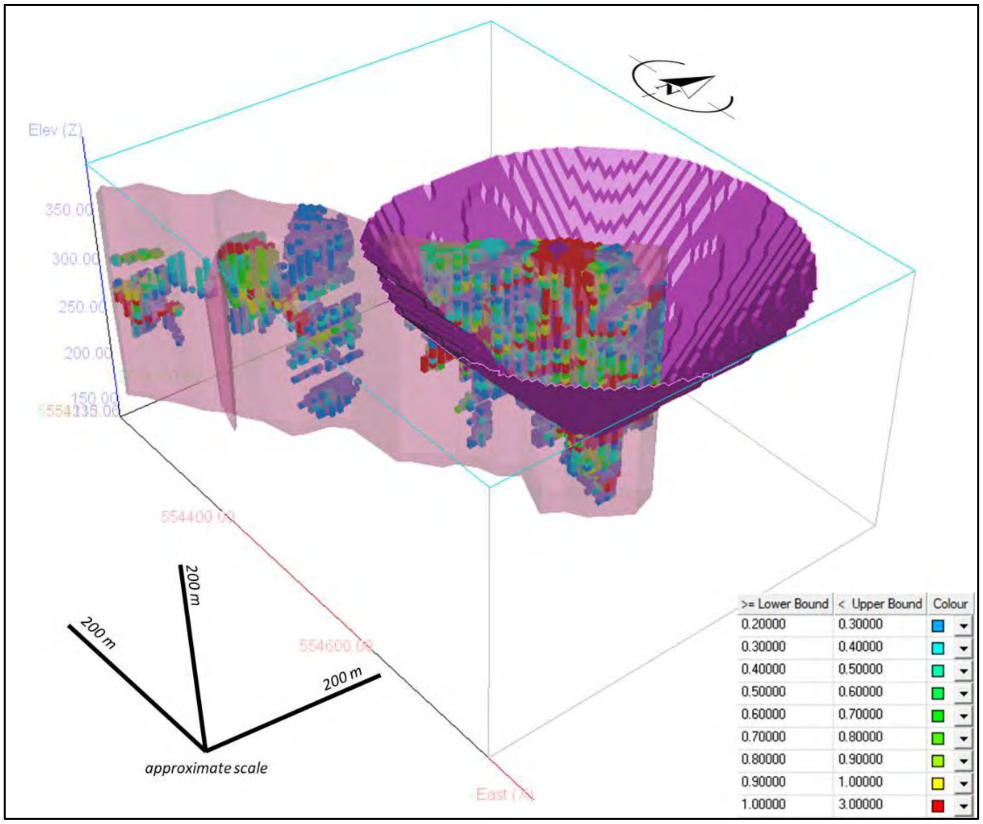
Figure 14-48 and Figure 14-49 present the Miller deposit with classified inferred blocks and grade blocks, respectively, with the constraining shell.

Figure 14-48: Miller Deposit Inferred Blocks with Constraining Shell – Perspective View Looking Northwest



Note: Inferred blocks are shown in blue. Source: AGP (2020).

Figure 14-49: Miller Deposit Grade Blocks with Constraining Shell – Perspective View Looking Northwest



Note: Showing blocks ≥ 0.26 g/t Au mineral resource statement. Source: AGP (2020).

14.3.5.4 Mineral Resources Tabulation

The mineral resources for the Miller deposit at a 0.26 g/t Au cut-off grade are inferred resources of 2.0 Mt at 1.24 g/t Au. The effective date of the mineral resources is October 26, 2020.

Table 14.62 presents the mineral resources for the mineral resources amenable to open pit extraction for the Miller deposit.

Table 14.62: Mineral Resources for the Miller Deposit; within constraining shell

Classification	Cut-off Grade (g/t Au)	Tonnes (kt)	Au Grade (g/t Au)	Contained Au (oz)
Inferred	0.26	1,981	1.24	79,000

Notes: 1. Mineral resources that are not mineral reserves do not have demonstrated economic viability. 2. Summation errors may occur due to rounding. 3. Mineral resources are reported within optimised constraining shell. 4. Block matrix is 5 m x 5 m x 5 m (no rotation). 5. Blocks were estimated using ID^3 on capped 2 m composite values. 6. Capping of grades was at 35.00 g/t Au within the granodiorite domain. 7. The density for the granodiorite domain is 2.82 g/cm^3.

AGP is not aware of any information not already discussed in this report, which would affect their interpretation or conclusions regarding the subject property. AGP is required to inform the public that the quantity and grade of reported inferred resources in this estimation must be regarded as conceptual in nature and are based on limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, geological grade or quality of continuity. For these reasons, an inferred resource has a lower level of confidence than an indicated resource. It is reasonably expected that most of the inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. The rounding of values, as required by the reporting guidelines, may result in apparent differences between tonnes, grade, and metal content.

Mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.3.5.5 Grade Sensitivity

The mineral resources for the project are reported below to demonstrate the sensitivity to various gold equivalent cut-off grades for each zone.

Table 14.63 presents the deposit sensitivity to various gold cut-off grades in Miller deposit for the open pit mineral resources.

Table 14.63: Open Pit Inferred Mineral Resources for the Miller Deposit at Various Cut-off Grades

Cut-off Grade (g/t Au)	Tonnes (kt)	Au Grade (g/t Au)	Contained Au (oz)
0.6	1,272	1.70	70,000
0.5	1,461	1.55	73,000
0.4	1,657	1.42	76,000
0.3	1,878	1.29	78,000
0.26	1,981	1.24	79,000
0.2	2,130	1.17	80,000

14.3.6 Factors that May Affect the Mineral Resource Estimate

The mineralisation is currently interpreted as a stockwork of veins within the granodiorite lithological unit, with minor feldspar porphyry units. The stockwork of veins seem to appear in a various sets, perhaps similar to Goldlund, where veins are seen in the drilling at shallow angles to near parallel. For greater geological and grade control and sample support further drilling is recommended. Where the Miller deposit outcrops at surface, it is possible to get a better understanding of the vein stockworks and orientations directly by stripping back some of the overburden to expose some of the vein sets where the granodiorite outcrops at surface.

Current drilling is concentrated in the northeast end of the granodiorite domain and is intersected by 14 drill holes: seven vertical drillholes and seven drill holes drilled at an angle from the northwest or the southeast flank. These angled drill holes target an elevation approximately 60 m to 80 m below surface. While the sample support is adequate at this elevation, it leaves less support both near surface and depth and therefore spatial analysis may be biased due to the localised sample support. It is therefore necessary that sufficient drilling is completed above and below this elevation for a more complete sample database and support.

14.4 Mineral Resources

The mineral resources for the Goliath, Goldlund and Miller projects are presented in Table 14.64.

Table 14.64: Mineral Resources for the Goliath Gold Complex

Deposit	Classification @ Cut-off Grade (g/t Au)	Tonnes (kt)	Au Grade (g/t Au)	Contained Au (koz)
Goliath	Measured @ OP 0.25 g/t Au	1,471	1.90	90
Goliath	Measured @ UG 1.60 g/t Au	98	4.94	16
Total Measured		**1,569**	**2.09**	**105**
Goliath	Indicated @ OP 0.25 G/t Au	26,956	0.87	757
Goliath	Indicated @ UG 1.60 G/t Au	2,592	3.16	263
Goldlund	Indicated @ OP 0.26 G/t Au	24,300	1.07	840
Total Indicated		**53,848**	**1.07**	**1,860**
Total Measured & Indicated		**55,417**	**1.10**	**1,965**
Goliath	Inferred @ OP 0.25 G/t Au	3,644	0.65	76
Goliath	Inferred @ UG 1.60 G/t Au	704	2.75	62
Goldlund	Inferred @ OP 0.26 G/t Au	14,400	0.56	260
Goldlund	Inferred @ UG 1.60 G/t Au	233	6.80	51
Miller	Inferred @ OP 0.26 G/t Au	1,981	1.24	79
Total Inferred		**20,962**	**0.78**	**528**

Notes: OP = open pit; UG = underground. Mineral resources are estimated in conformance with the CIM mineral resource definitions referred to in N.I. 43-101 Standards of Disclosure for Mineral Projects. This mineral resource estimate covers the Goliath deposit, the Goldlund deposit, and the Miller deposit. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The quantity and grade of the reported inferred mineral resources in this estimation are conceptual in nature and are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. For these reasons, an inferred mineral resources has a lower level of confidence than an indicated mineral resources and it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Goliath:

Mineral resources are reported within optimised constraining shell using a gold price of US$1,700/oz and a silver price of US$23/oz and recoveries of 95.5% for gold and 62.6% for silver. Grades were estimated using 1.5 m capped composites using ordinary kriging for the Main and C Zones and ID^3 for all other zones.

Goldlund:

Mineral resources are reported within an optimised constraining shell using a gold price of US$1,700/oz and gold recovery of 89%. Gold grades were estimated using 2.0 m capped composites within nine mineralised zones using ordinary kriging.

Miller:

Mineral resources are reported within an optimised constraining shell using a gold price of US$1,700/oz and gold recovery of 89%. Grades were estimated using 2.0 m capped composites within the granodiorite domain using inverse distance cubed interpolation.

Summation errors may occur due to rounding.

15 MINERAL RESERVE ESTIMATES

This section is not relevant to this technical report.

16 MINING METHODS

16.1 Overview

AGP was retained by Treasury Metals to prepare a preliminary economic assessment level mining study (PEA) of the Goliath, Goldlund and Miller deposits. AGP's opinion is that with current metal pricing levels and knowledge of the mineralisation, a combination of open pit and underground mining offers the most reasonable approach for the development of the three deposits. Goliath is planned to be mined by open pit and underground methods, while the Goldlund and Miller deposits will be mined by open pit.

No prior mining activities have occurred on the Miller site while underground mining has occurred in the past at Goldlund and an underground bulk sample was obtained at Goliath.

The Goliath project is located approximately 20 km east of the municipality of Dryden in Ontario, while Goldlund is 24 km northeast of Goliath and Miller is an additional 10 km northeast of Goldlund.

16.2 Mining Geotechnical

The quantity of geotechnical work completed on the three pit areas varies with the level of studies completed to date. Goliath is at a more advanced stage and has had multiple studies completed. Miller is the least studied and no geotechnical documentation was provided for the study. The work by various consultants has been summarised below.

Additional geotechnical work is recommended as the project advances forward with the current design as guidance to focus the efforts and priorities.

For all pit areas the overburden slope was based on an overall angle of 27.3° with a 55° face angle. The area with the most significant depth of overburden is Goliath.

16.3 Goliath Mining Geotechnical

Work at Goliath is further advanced than the other areas. Analyses have been completed based on geotechnical information collected from oriented drill core, including kinematic, empirical and numerical analyses.

A geotechnical borehole investigation was completed in 2013-2014. The primary unit encountered was biotite-muscovite-schist (BMS) 64% of the time; muscovite-sericite-schist (MSS) was encountered 22% of the time. The muscovite-sericite schist is the mill feed bearing unit. These units rock quality is considered as fair (Q ranging between 4.7 and 10) and mean UCS of 67.3-76.3 MPa.

Two other rock types were encountered in the drilling program. This includes the metasediments (MSED) with 8% occurrence and the quartz-feldspar porphyry (QFP) with 4% occurrence.

Regional structures which have been delineated include:

- two sub-horizontal thrust faults passing through the central portion of deposit delineated at depths between 200 and 300 m
- the CZ fault, which is a sub-vertical fault that runs approximately along strike of the proposed pits
- the NW fault, which is a brittle structure striking west to west-northwest dipping shallowly to the north

Structural conditions were observed to be consistent across the site, dominated by foliation jointing (which strikes parallel to the mineralisation) and a sub-horizontal joint set. Additional sets were noted in some lithological units but were not observed throughout the site. A summary of the joint set orientations is shown in Table 16.1

Table 16.1: Identified Joint Set Orientations

Joint Set	All Data	BMS	MSS
Foliation	74/165 (14)	74/164 (14)	74/167 (14)
J1	4/315 (18)	2/115 (17)	12/290 (15)

Note: Orientation recorded as dip/dip direction (degrees), first standard deviation quoted in brackets.

From the analyses, the open pit has the following recommendations:

- inter-ramp angle between 50° and 55°
- bench face angle of 82°
- minimum berm width of 7.3 m

The identified fault structures are generally favourably oriented; however, the NW fault may cause instability along a portion of the southwest pit wall. The potential for bench scale toppling and planar failures has been identified from joint data in north and south dipping walls. Further investigation of conditions which control the toppling failure mode is recommended with additional study and preliminary excavations.

Previous underground geologic models considered very narrow high-grade veins that would be mined with thin pillars between the stopes. This type of mining was analysed by RockEng, formerly Mine Design Engineering (MDEng), and the design geotechnical parameters provided in a report for review. As the geologic model interpretation has changed, some of this work is not entirely applicable, but the base analysis behind that remains valid.

Potentially economic stoping areas were identified on each mining level throughout the deposit by manually designing stope outlines in section whilst recognising the level spacing, minimum mining width requirement as well as the minimum inter-lens pillar width requirements in the relatively few instances of closely-spaced parallel stope arrangements. In-vein development was designed in much the same manner.

The PEA underground design incorporated these previous geotechnical recommendations provided. RockEng subsequently reviewed the current underground stope designs for the PEA and found they honoured their earlier work, and no alterations were required.

The underground geotechnical design parameters for a future pre-feasibility study would require additional study to confirm and refine the design parameters.

16.4 Goldlund Mining Geotechnical

Limited geotechnical study work for an open pit has been completed for the Goldlund project site. A review of the existing information and small open pit was completed to provide PEA-level slope guidance. The data reviewed indicated that no hydrogeological model was available, so the assumption was made that the pits will be dewatered and surface diversion ditches established.

RQD had been collected and was part of the drillhole database. This information was reviewed and the rock at Goldlund is considered to be excellent quality rock with a lower bound (20th percentile) RQD of 85 and a mean RQD value (50th percentile) of 95. Using an empirical factor of safety target of 1.5, the maximum overall slopes by wall height was provided to AGP for use in the design and are shown in Table 16.2.

Table 16.2: Goldlund Recommended Slope Parameters

Rock Slope Height	Overall Angle (degrees)
< 100 metres	50
100 – 150 metre	47
150 – 200 metres	45

16.5 Miller Mining Geotechnical

No geotechnical work has been completed to date on the Miller site. The assumptions provided for Goldlund were employed and considered sufficient for PEA-level work.

16.6 Geological Model Import

The three 2020 final resource models were received between October and December 2020. GEMS® software was used for the estimation of resource block model values in the Miller and Goliath models, while Hexagon MinePlan® was used for Goldlund. All resource models were provided in CSV format for open pit mine planning. Goliath grade values were estimated into a single mineralisation percentage model, while the Miller and Goldlund models were estimated into whole block models. The grades in each block of the resource models were considered as undiluted.

Model framework details of the different deposits are provided in Table 16.3. Resource model item descriptions are shown in Tables 16.4 to 16.6. The mining models were created by AGP to include additional items for mine planning purposes. Hexagon's MinePlan® software was used for the mine planning portion of the PEA, using their Lerchs-Grossmann (LG) economic pit shell generation, pit and WRSF design and mine scheduling tools. The PEA mine plan is based on measured, indicated and inferred mineral resources.

A model capture check was completed for each of the three resource models. Global tonnages and grades at various gold cut-off grade values were compared between the resource model reports and the captured models in Hexagon MinePlan® software. Excellent agreement was achieved for all deposits with differences in tonnes, grade, and ounces less than 0.2% in all cases.

Table 16.3: Open Pit Model Framework by Deposit

Framework Description	Deposit		
	Goliath	Goldlund	Miller
MineSight® file 10 (control file)	goth10.dat	gld10.dat	mill10.dat
MineSight® file 15 (resource model file)	goth15.da4	gld15.da2	mill15.dat
MineSight® file 15 (mine planning model file)	goth.mp4	gld15.mp2	mill15.mp1
X origin (m)	526,050	545,000	554,100
Y origin (m)	5,511,500	5,526,500	5,533,200
Z origin (m) (max)	-410	-350	135
Rotation (degrees clockwise)	0	0	0
Number of blocks in X direction	638	940	170
Number of blocks in Y direction	618	500	140
Number of blocks in Z direction	182	162	55
X block size (m)	5	5	5
Y block size (m)	2	5	5
Z block size (m)	5	5	5

Table 16.4: Resource Model Item Description for Goliath

Field Name	Min	Max	Precision	Units	Comments
DOM	0	9999	1	-	Domain codes for interpolated blocks
DEN	0	9.999	0.001	t/m^3	Final density for interpolated blocks
AU	0	99	0.001	g/t	Gold capped grade model for resource (kriged + ID^2 mix)
AG	0	99	0.001	g/t	Silver capped grade model for resource (kriged + ID^2 mix)
CLASS	0	9	1	-	Classification, where 1 = measured, 2 = indicated, 3 = inferred
ORE%	0	100	0.001	%	Final percent ORE model
DOMRT	0	9999	1	-	Domain codes without the probabilistic model
PROB	0	3	0.001	-	Final PROBABILITY model 1 = HG, 2 = MG, 3=LG/WASTE

Table 16.5: Resource Model Item Description for Goldlund

Field Name	Min.	Max.	Precision	Units	Comments
TOPO%	0	100	0.01	%	Topography percentage, 100%=rock below surface
OVB%	0	100	0.01	%	Overburden percentage, 100%=rock below overburden
ZONE	0	99	1	-	Mineralised zones z01-z09 "ore", z10="waste"
SG	0	5	0.001	t/m^3	Density from drill core sample measurements
CLASS	0	9	1	-	Classification, where 1=Measured, 2=indicated, 3=inferred (no measured)
UG%	-1	100	0.001	%	Mined percentage, 100% is completely mined - no rock left
ORE%	-1	100	0.001	%	Percentage below OB and outside of mined out underground. Blocks are whole blocks (100%) away from contacts.
AUGPT	0	99	0.001	g/t	Kriged gold grade (g/t) using a probability methodology (IK for proportion of HG/LG domains, OK for gold grade)
LCODE	0	999	1	-	Simplified lithology model

Table 16.6: Resource Model Item Description for Miller

Field Name	Min.	Max.	Precision	Units	Comments
TOPO%	0	100	0.01	%	Topography percentage, 100%=rock below surface
ROCK	0	400	1	-	Rock Type
DEN	0	5	0.001	t/m^3	Density assigned by lithology
CLASS	0	9	1	-	Classification, where 3=inferred, 4=potential (no measured or indicated)
AU	0	99	0.001	g/t	Gold grade (ID^3)

16.7 Open Pit Mining

16.8 Economic Pit Shell Development

For each of the project areas, the open pit ultimate size and phasing opportunities were completed with various input parameters including estimates of the expected mining, processing, and G&A costs, as well as metallurgical recoveries, pit slopes, and reasonable long-term metal price assumptions. AGP worked together with Treasury Metals personnel and the other contractors to select appropriate operating cost parameters for the Goliath, Goldlund, and Miller open pits.

The mining costs are estimates based on cost estimates for equipment from vendors and previous studies completed by AGP. The costs represent what is expected as a blended cost over the life of the mine for all material types to the various destinations with an estimate made for incremental haulage at depth. Process costs and G&A costs were provided by Ausenco based on their benchmarking of other relevant studies and test results.

The parameters used for each of the project areas are shown in Table 16.7 and Table 16.8. The revenue values are in United States dollars unless otherwise noted. Costs were developed in Canadian dollars and revenues are converted to Canadian dollars for use in pit shell determination. The mining cost estimates are based on the use of 91 tonne trucks using an approximate waste dump configuration to determine incremental hauls for mill feed and waste. The refining terms and recovery assumptions are based on creating a gold and silver doré.

Table 16.7: Mining & Processing Economic Pit Shell Parameters

Description	Units	Goliath	Goldlund	Miller
Mining Cost*				
Waste Base Rate - 410 m Elevation	C$/t mined	2.14	2.48	2.48
Incremental Rate - Above	C$/t/5 m bench			
Incremental Rate - Below	C$/t/5 m bench	0.030	0.027	0.027
Mill Base Rate - 410 m Elevation	C$/t mined	2.52	2.71	2.71
Incremental Rate - above	C$/t/5 m bench			
Incremental Rate - below	C$/t/5 m bench	0.016	0.017	0.017
Processing**				
Processing Cost	C$/t feed	12.92	12.26	12.26
Mill Feed Haulage Cost	C$/t feed	-	5.61	5.61
Tailings and Water Management	C$/t feed	2.00	2.00	2.00
Total Processing Cost	C$/t feed	14.92	19.87	19.87
General & Administrative Cost				
G&A Cost	C$/t feed	1.62	1.77	1.77
Total Process & G&A				
Process + G&A	C$/t feed	16.54	21.64	21.64

Notes: *Mining costs based on using 91 tonne haul trucks. **Process costs based on 5000 kt/d dry throughput at Goliath mine site.

Table 16.8: Common Economic Pit Shell Parameters (US Dollars unless otherwise noted)

Description	Units	Value	Goliath	Goliath	Goldlund	Miller
Exchange rates						
CAD	US$	1.33				
Resource Model						
Block classification used		M+I+I				
Block Model Height		5				
Mining Bench Height		5				
Metal Prices			**Gold**	**Silver**	**Gold**	**Gold**
Price	US$/oz		1,475	20	1,475	1,475
Royalty	%		1.5%	1.5%	1.5%	1.5%
Smelting, Refining, Transportation Terms						
Payable	%		99.0%	97.0%	99.0%	99.0%
Minimum Deduction	unit, g/dmt		0	0	0	0
Refining Charge	US$/oz		5	0	5	5
Net Price Calculation						
Payability	%		99.0%	97.0%	99.0%	99.0%
Transportation & Refining Charge	US$/oz		12.03	0.17	12.03	12.03
Subtotal Price	US$/oz FOB mine		1,448.22	19.23	1,448.22	1,448.22
less Royalty	US$/oz FOB mine		21.72	0.29	21.72	21.72
Net Price	US$/oz FOB mine		1,426.50	18.95	1,426.50	1,426.50
	US$/g FOB mine		45.86	0.61	45.86	45.86
	C$/g FOB mine		61.00	0.81	61.00	61.00
Metallurgical Information						
Recovery	%		95.5%	60.0%	89.0%	89.0%
Power Cost						
Cost of power	C$/kWh	0.08				
Fuel Cost						
Diesel Fuel Cost to site	C$/L	0.79				

Wall slopes for pit optimisation were based on the previous geotechnical studies. Allowances were made for ramps in the slopes to determine an overall angle for use in the LG routine. The overall slope angle calculations are shown in Table 16.9.

Table 16.9: Overall Slopes for Economic Pit Shells

Area	Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Haul roads in slope	Slope height (m)	Overall Slope (degrees)
Goliath	OB	1	27.3	0.0	12.0	27.3
	Rock	2	52.6	3.0	200.0	39.9
Goldlund	OB	1	27.3	1.0	10.0	11.7
	Rock (<100 m)	2	52.6	1.0	100.0	43.6
	Rock (100-150 m)	2	52.6	2.0	50.0	41.1
	Rock (150-200 m)	2	52.4	2.0	50.0	43.4
Miller	OB	1	27.3	0.0	12.0	27.3
	Rock	2	52.6	3.0	200.0	39.9

Note: 28.7 m haul road width

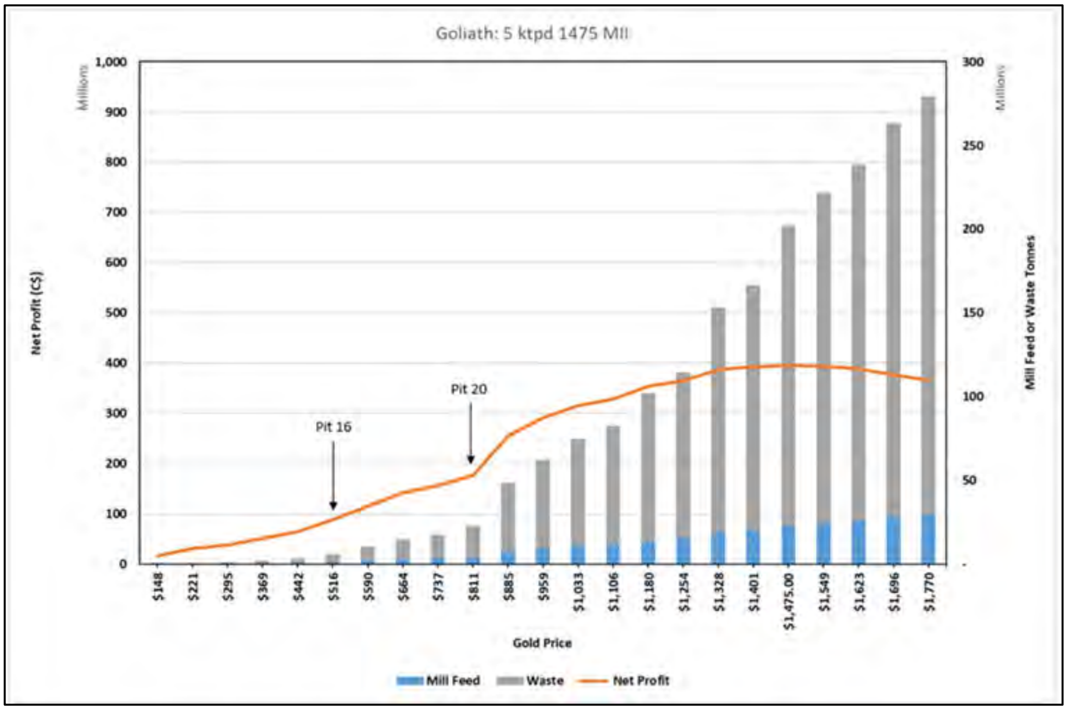
A set of nested LG pit shells were generated for each of the project areas to examine sensitivity to the gold and silver prices with base case prices of US$1,475/oz Au and US$20.00/oz Ag. This was to gain an understanding of the deposits and highlight potential opportunities in the design process to follow. Undiluted measured, indicated and inferred mineral resources were used in the analysis. The net prices were varied by applying revenue factors of 0.10 to 1.20 at 0.05 increments, to generate a set of nested LG shells. The chosen set of revenue factors result in an equivalent gold price varying from US$148/oz up to US$1,770/oz. All other parameters were fixed. The resulting nested pit shells assist in visualising natural breakpoints in the deposit and selecting shells to act as design guidance for phase design. The net profit before capital for each pit was calculated on an undiscounted basis for each pit shell using US$1,475/oz Au and US$20.00/oz Ag. Mill feed/waste tonnages and net profit were plotted against gold price and are displayed in Figures 16-1, 16-2 and 16-3.

Figures 16-1, 16-2 and 16-3 illustrate various break points on the pit shells. With each incremental pit shell, the waste tonnage, mill tonnage, and undiscounted net profit also increased up to the base price of US$1,475/oz Au.

In the case of Goliath (Figure 16-1), the first break point shown at US$516/oz Au (Pit 16), the cumulative waste tonnage is 4.69 Mt, with a corresponding mill feed tonnage of 1.2 Mt or a strip ratio of 3.9:1. The net profit also increased beyond this point showing that there was still value to be obtained by going with a higher metal price or an additional phase. This break point represented 22% of the net value of a $1,475/oz Au pit but with only 3% of the waste of the larger pit shell. This pit shell was used to guide the design of the first phase of the Goliath pit.

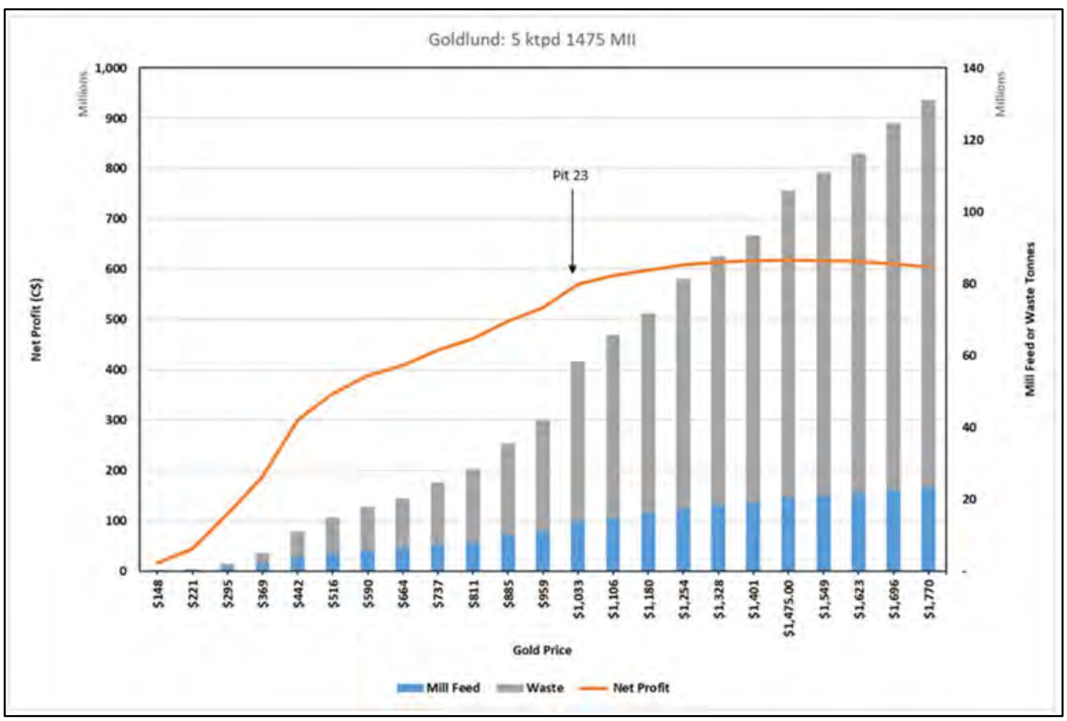
The next selected break point was at US$811/oz Au (Pit 20)(Figure 16-1). The incremental waste tonnage from the first break point is 14 Mt, with a corresponding increase in mill feed tonnage 2.8 Mt or a strip ratio of 5:1. The cumulative net value of the first two break points was 45% of the US$1,475/oz Au pit shell but with only 10% of the waste movement of the larger pit required. This pit shell was used for the pit design of the ultimate phase in the Goliath pit.

Figure 16-1: Goliath Profit vs. Gold Price by Pit Shell



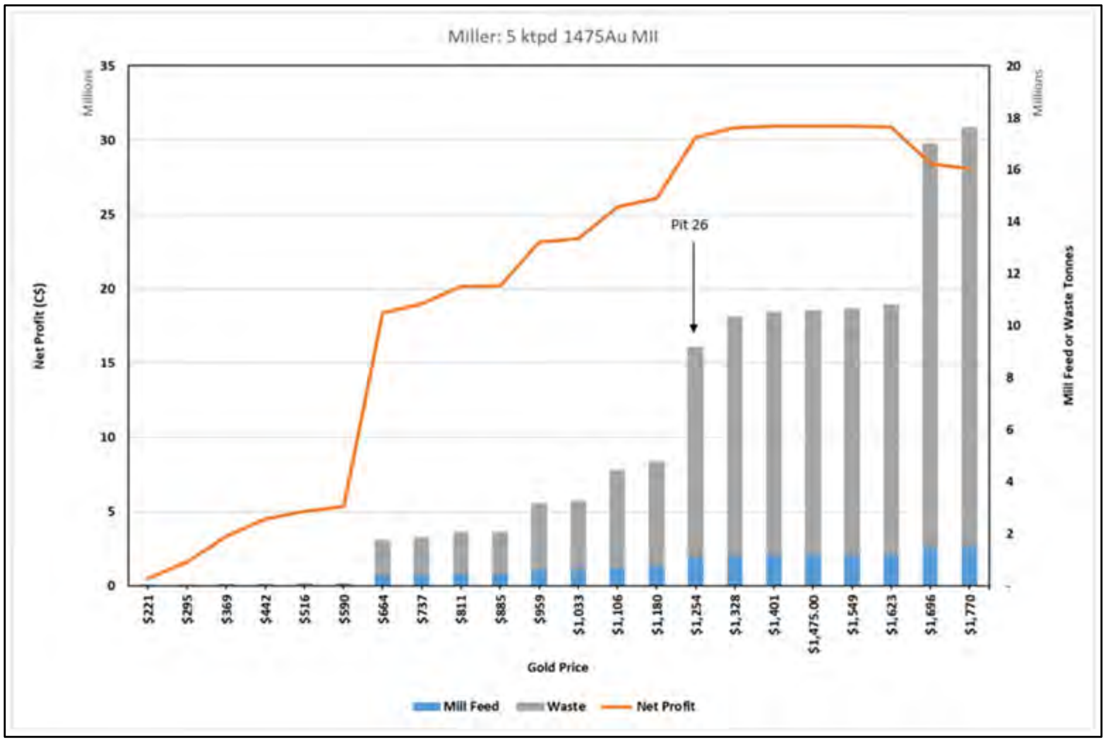
Source: AGP (2021).

Figure 16-2: Goldlund Profit vs. Gold Price by Pit Shell



Source: AGP (2021).

Figure 16-3: Miller Profit vs. Price by Pit Shell



Source: AGP (2021).

At the request of Treasury Metals, a physical limit was placed on the footprint and volume of waste rock produced. The pit shells considered reflect this physical limit when they were selected for detailed design.

In the case of Goldlund (Figure 16-2), the break point shown at US$1,033/oz Au (Pit 23), the cumulative waste tonnage is 44.5 Mt, with a corresponding mill feed tonnage of 13.7 Mt or a strip ratio of 3.3:1. This break point represented 92% of the net value of a $1,475/oz Au pit but with only 52% of the waste of the larger pit shell. This pit shell was used to guide the design of the Goldlund pits. The incremental net value is not considered significant for the next higher pit prices, particularly when discounting is accounted for.

In the case of Miller (Figure 16-3), the break point shown at US$1254/oz Au (Pit 26), the cumulative waste tonnage is 8.1 Mt, with a corresponding mill feed tonnage of 1.1 Mt or a strip ratio of 7.6:1. This break point represented 98% of the net value of a $1,475/oz Au pit but with only 86% of the waste of the larger pit shell. This pit shell was used to guide the design of the Millar pits. The incremental net value is not considered significant for the next higher pit prices, particularly when discounting is accounted for.

16.9 Open Pit Dilution

The open pit resource models were provided as undiluted. The Goliath model is a percentage type model. This means the (grades or volumes) from the wireframes were coded into separate percentage parcels of mill feed and waste in each block. Goldlund and Miller models are whole block models which means that for any given block, it is routed as either mill feed or waste.

To account for mining dilution, AGP modelled contact dilution into the in-situ resource blocks for all three of the project areas. To estimate the amount of dilution and the grade of the dilution, the size of the block in the model was examined. The block size within the Goliath model is 5 m x 2 m in plan view, and 5 m high. While the block size within the Goldlund and Miller models is 5 m x 5 m in plan view, and 5 m high. Mining would be completed on 10 m lifts for waste and 5 m lifts for mill feed if required and the equipment selected is capable of mining in that manner.

The percentage of dilution is calculated for each contact side using an assumed 0.5 m contact dilution thickness for Goliath, and 1.0 m contact thickness for Goldlund and Miller. This dilution skin thickness was selected by considering the block dimensions, the spatial nature of the mineralisation, proposed grade control methods, GPS assisted digging accuracy, and blast heave.

For example, if one side of a mineralised block at Goldlund above cut-off is in contact with a waste block, then it is estimated that dilution of 20% (1 m / 5 m) would result. If two sides are contacting a waste block, dilution would rise to 40%. Three or four sides in contact with a waste block would result in 60% and 80% dilution, respectively. Four sides represent an isolated block of mill feed. Likewise, if the block is surrounded by no waste blocks (surrounded by mill feed), then 0% dilution has been assumed.

During the pit optimisation, the net value per tonne was stored in each of the block for each of the models. This net value per tonne figure acts as the grade for cut-off application as that net value per tonne is inclusive of all on-site operation costs except for mining. Applying a C$0.01/t cut-off represents the marginal cut-off grade to flag initial feed and waste blocks.

AGP applied different dilution skin methods to estimate the diluted grades and percentage for each of the resource models as they were all modelled slightly differently.

The Goliath model is "percentage model" and AGP used a percentage method on it. The Goldlund model is a "whole block model" however AGP used a percentage method on it as the model was built with a percentage item to indicate the proportion of each block that was below the overburden surface and had not been mined by underground methods. The Miller model is a "whole block model" and AGP used a whole block method on it. A description of the methods are as follows:

In all three methods, dilutions skins are considered around a mill feed block if the mill feed block is in contact with neighbouring waste blocks.

The first step in all three methods is to identify the mill feed and waste block in the model. This is done by using a net value per tonne cut-off grade. Positive value blocks are considered mill feed while zero and negative blocks are considered waste blocks. In the case of Goliath, routing was coded into three groups – mill feed, mineralised waste blocks, and waste blocks.

And in the case of Miller, a gold cut-off grade of 0.40 g/t Au and resource class of inferred and higher was used.

The second step in all three methods is to process each of the mill feed blocks to determine the number of waste neighbour contacts. For each waste neighbour a dilution volume is applied to the mill feed block based on the specified dilution skin thickness.

In the case of the percent method, mass is effectively added to the mill feed block by increasing the mill feed routing percent item and adjusting the proportionate grade lower. The diluting material is considered to have zero grade as this is the waste material within the mill feed block itself. A review of the geologic model and against the drillholes noted that some mineralisation was present. This was estimated to have the potential to boost the grade by 2%. This factor was applied to Goliath only to mimic the expected mineralised dilution. In the PFS, modelling of the waste grades needs to be expanded upon and included to properly assess this dilution. In the other areas though that diluting mineralised block material in contact with the mill feed block is included in the overall dilution at the grade of the mineralised block, but it is accounted for in the mineralised waste block.

In the case of Goldlund, this method was modified to leave the mill feed block percentage and grades unchanged as the percentage item reflected the portion of the block that was below the overburden surface and unmined by underground mining methods – not a true “ore percent” model.

In the case of the whole block method, mass is added to the mill feed block by increasing the density of the block and adjusting the grades based on the amount of diluting material and grade. The diluting material is considered at its grade.

The third step in all three methods is to process each of the waste blocks to determine the number of mill feed neighbour contacts. For each mill feed neighbour, the mass of material routed as waste is reduced.

In the case of the percent method, a portion of the mineralised waste block is routed as mill feed at the grade of the mineralised block, and remaining portion of the block is routed as waste. Waste blocks outside of the mineralisation are entirely routed as waste. In the case of Goldlund, a portion of the waste blocks are routed as mill feed with the remaining of the block being routed as waste. Mineralised waste is not considered in the Goldlund model.

For the whole block method, the mass of the waste blocks is reduced by adjusting the density lower. The grade of the waste block is not adjusted.

Refer to Table 16.10 for a comparison of in-situ to diluted values for each of the project areas. In the case of Goldlund and Miller and only a minor amount with Goliath, the grade dilution percentage is lower than the feed tonnage percentage since the mineralised waste blocks included some grade. AGP considers these dilution percentages to be reasonable considering the nature of the mineralisation.

Table 16.10: Comparison of In-situ to Diluted Value Summary

Goliath	In-situ				Diluted				Delta		
		Tonnes	Au g/t	Contained Oz		Tonnes	Audil g/t	Contained Oz	Tonnes	Au g/t	Oz
	Mill Feed	5,276,253	1.09	184,903	Mill Feed	6,048,910	0.97	190,038	15%	-12%	3%
	Waste	30,133,631			Waste	29,360,975			-3%		
	Total	35,409,884			Total	35,409,884			0%		
	COG	VLT4>$0.01									
Goldlund	**In-situ**				**Diluted**				**Delta**		
		Tonnes	**Au g/t**	**Oz**		**Tonnes**	**Audil g/t**	**Oz**	**Tonnes**	**Au g/t**	**Oz**
	Mill Feed	19,533,771	1.53	960,879	Mill Feed	23,725,694	1.31	999,266	21%	-14%	4%
	Waste	78,132,631			Waste	73,940,709			-5%		
	Total	97,666,403			Total	97,666,403			0%		
	COG	VLT4>$0.01									
Miller	**In-situ**				**Diluted**				**Delta**		
		Tonnes	**Au g/t**	**Oz**		**Tonnes**	**Audil g/t**	**Oz**	**Tonnes**	**Au g/t**	**Oz**
	Mill Feed	1,130,159	1.32	47,963	Mill Feed	1,324,087	1.15	48,956	17%	-13%	2%
	Waste	11,931,624			Waste	11,739,800			-2%		
	Total	13,061,783			Total	13,063,887			0%		
	COG	Au>0.04 g/t			Total	13,063,887					

16.10 Pit Designs

16.10.1 Goliath Project Pit Designs

For the Goliath project area, pit designs were developed for a main pit as well as one satellite pit immediately to the east. The main pit was further subdivided into four phases. The designs used a 10 m bench height. The pit optimisation shells used to guide the ultimate pit were also used to outline areas of higher value for targeted early mining and phase development.

Geotechnical parameters discussed in Section 16.2 were applied to pit designs as shown in Table 16.11.

Table 16.11: Pit Design Slope Criteria – Goliath

Area	Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Slope height (m)
Goliath	OB	1	27.3	10.0
	Rock	2	52.6	200.0

Note: 10 m bench heights during mining.

Equipment sizing for ramps and working benches is based on the use of 91 tonne rigid frame haul trucks. The operating width used for the truck is 6.7 m. This means that single-lane access is 22 m (two times the operating width plus berm and ditch) and double lane widths are 28.7 m (three times the operating width plus berm and ditch). Ramp gradients are 10% in the pit and dump for uphill gradients. Working benches were designed for 35 to 40 m minimum mining width on pushbacks.

Tonnes and grade for the designed pit phases are reported in Table 16.12 using the diluted tonnes and grade from the resource model. Positive marginal block values from the optimisation run were used to delineate mill feed from waste.

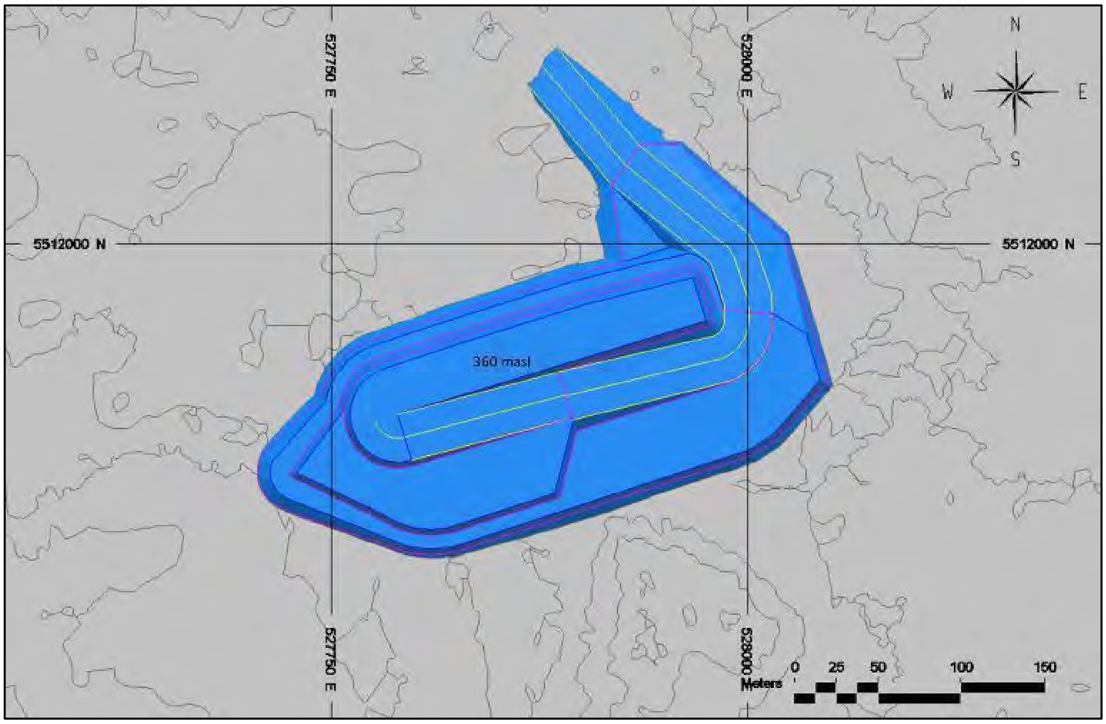
Table 16.12: Pit Phase Tonnage & Grades – Goliath

Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
1A	467,034	1.48	4.04	1,925,877	2,392,911	4.12
1B	2,874,055	0.94	2.90	13,119,095	15,993,150	4.56
2	1,975,187	0.91	2.24	9,654,799	11,629,986	4.89
3	672,901	0.98	3.00	4,195,187	4,868,088	6.23
4	109,790	0.64	2.13	456,133	565,923	4.15
Total	**6,098,967**	**0.97**	**2.77**	**29,351,091**	**35,450,058**	**4.81**

16.10.1.1 Phase 1A

Phase 1A is the first phase mined in the schedule and comprises the small pit in the middle of the deposit. The phase will be mined down to the 360 masl elevation. All waste and mill feed access will be on the north side of the pit in a slot to ramp configuration (see Figure 16-4).

Figure 16-4: Phase 1A Design – Goliath



Source: AGP (2021).

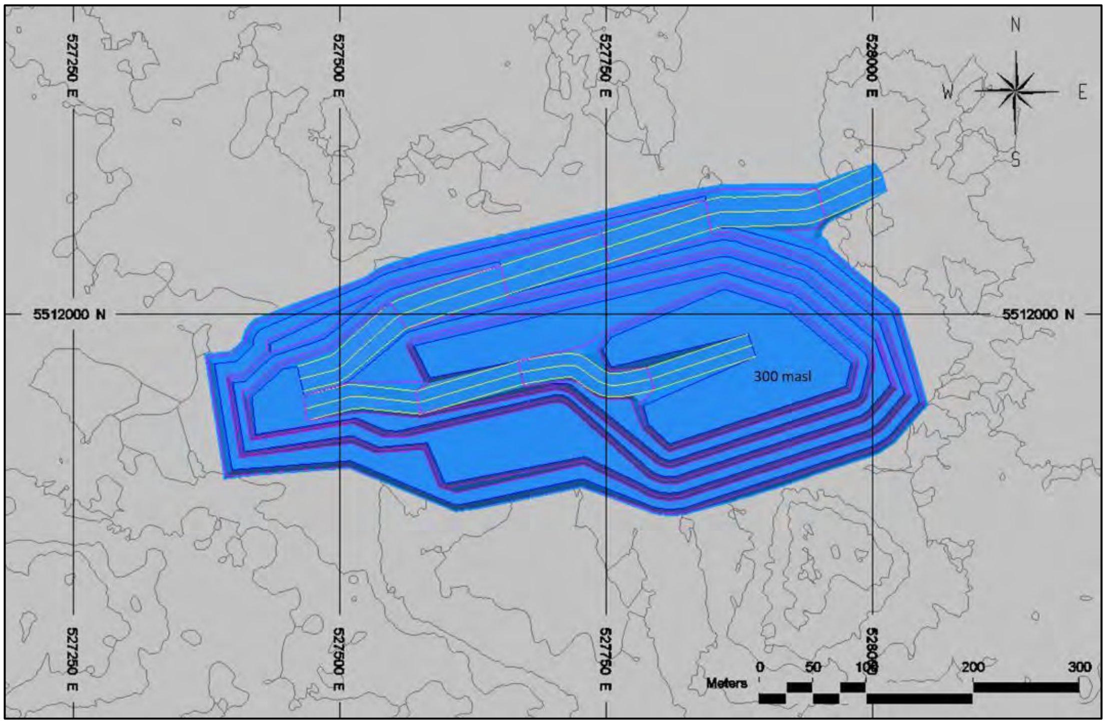
16.10.1.2 Phase 1B

Phase 1B is the second phase mined in the schedule and targets the area north and west of Phase 1A. This phase is mined down to the final pit level in this area, 300 masl elevation. Mine access is provided by the main ramp which is developed on the north side of the pit descending to the bottom of phase 1B forming the main ramp of the ultimate pit (see Figure 16-5). The bottom of this phase will be used by the underground for a year to develop a second access to the development of underground, advancing production.

16.10.1.3 Phase 2

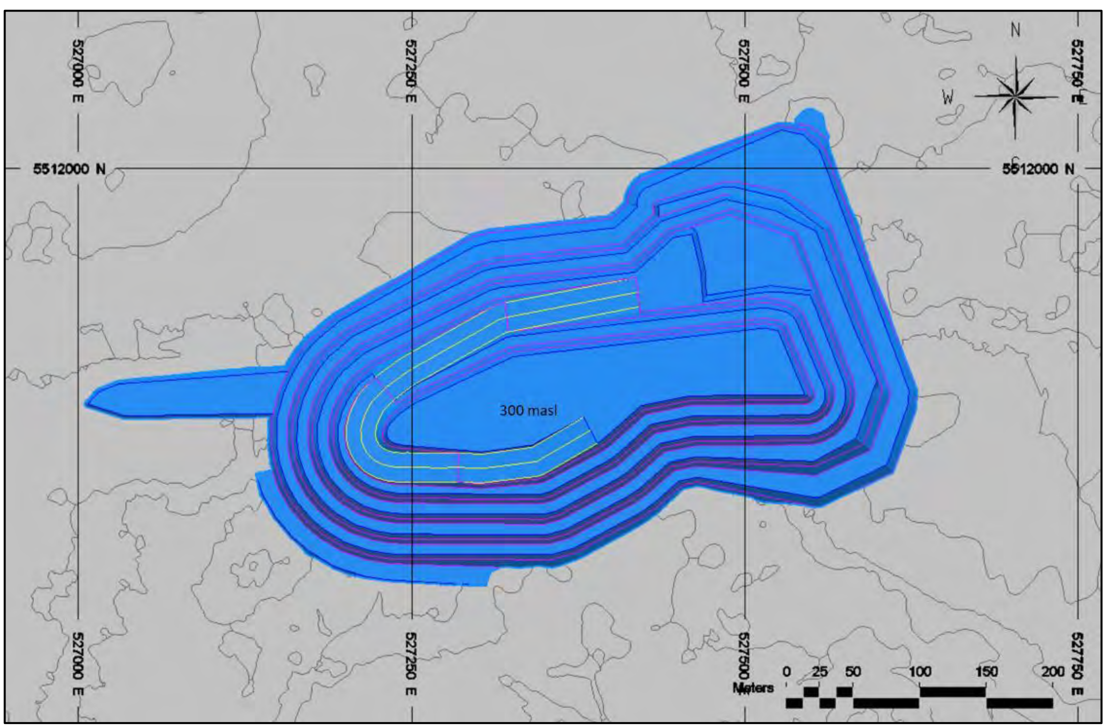
Phase 2 is the third phase mined in the schedule and targets the area west of phase1B. This phase is mined down to the 300 masl elevation. The bottom of the phase is accessed by a ramp that ties into the main ramp at the 340 masl elevation and wraps around the west side of the phase to the bottom (see Figure 16-6).

Figure 16-5: Phase 1B Design – Goliath



Source: AGP (2021).

Figure 16-6: Phase 2 Design – Goliath

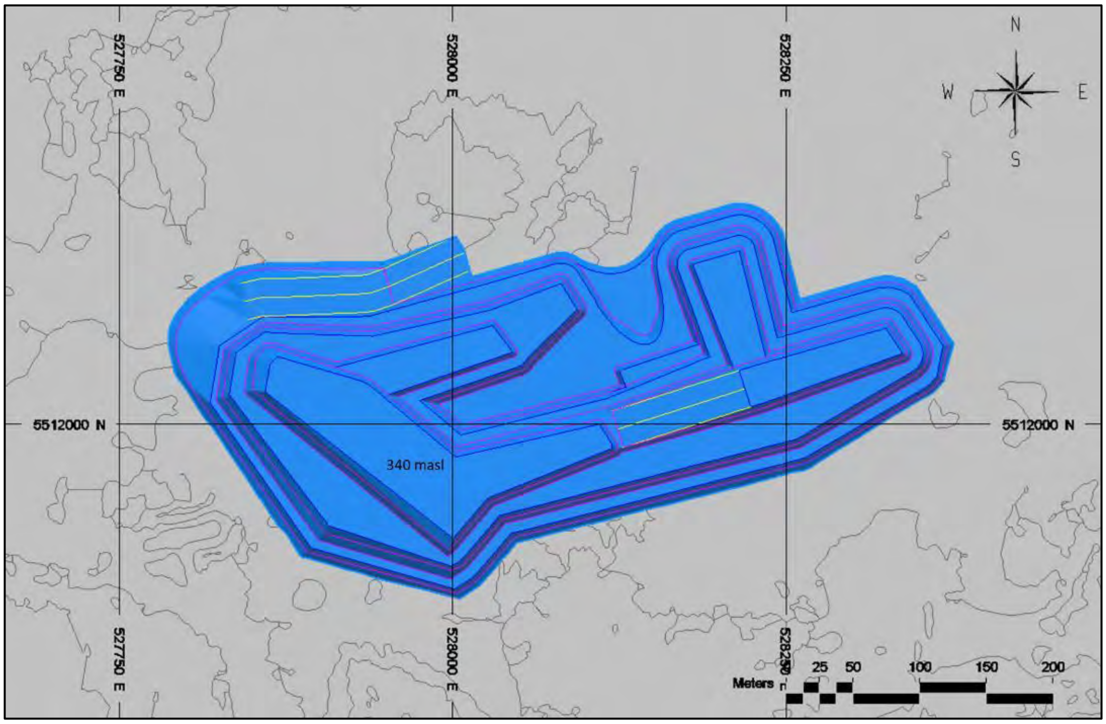


Source: AGP (2021).

16.10.1.4 Phase 3

Phase 3 is the fourth phase mined in the schedule and targets the area east of Phases 1A and 1B. This phase is mined down to the 340 masl elevation. The upper bench of this phase is accessed by an internal ramp that is slotted in from the east (see Figure 16-7).

Figure 16-7: Phase 3 Design – Goliath

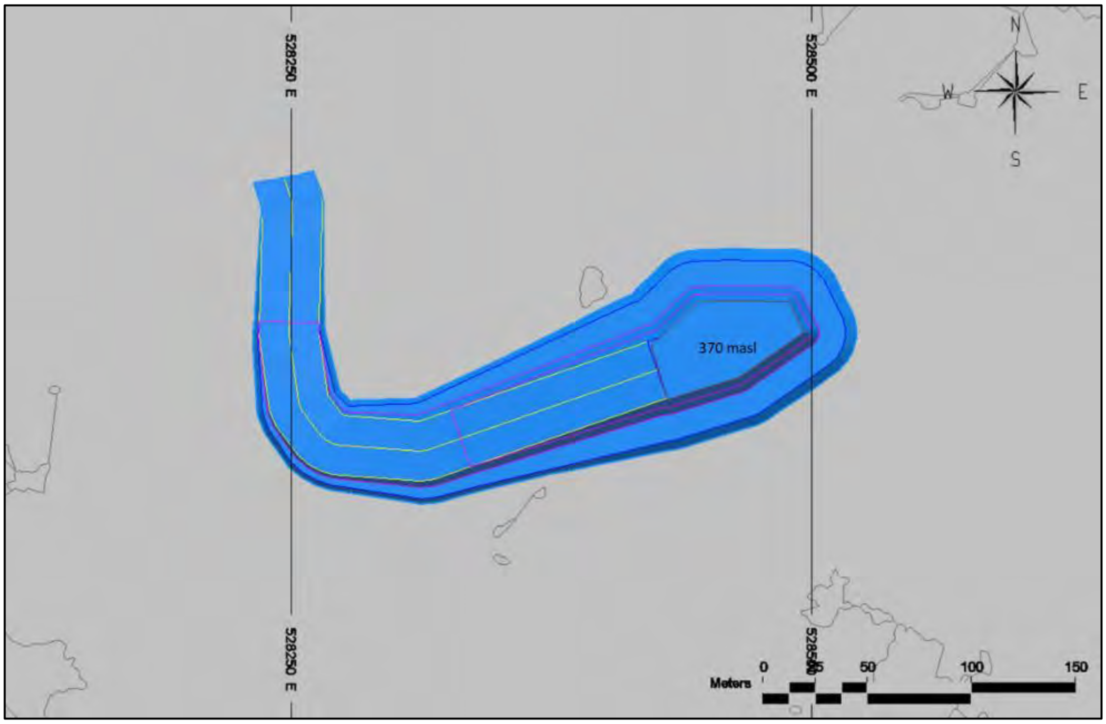


Source: AGP (2021).

16.10.1.5 Phase 4

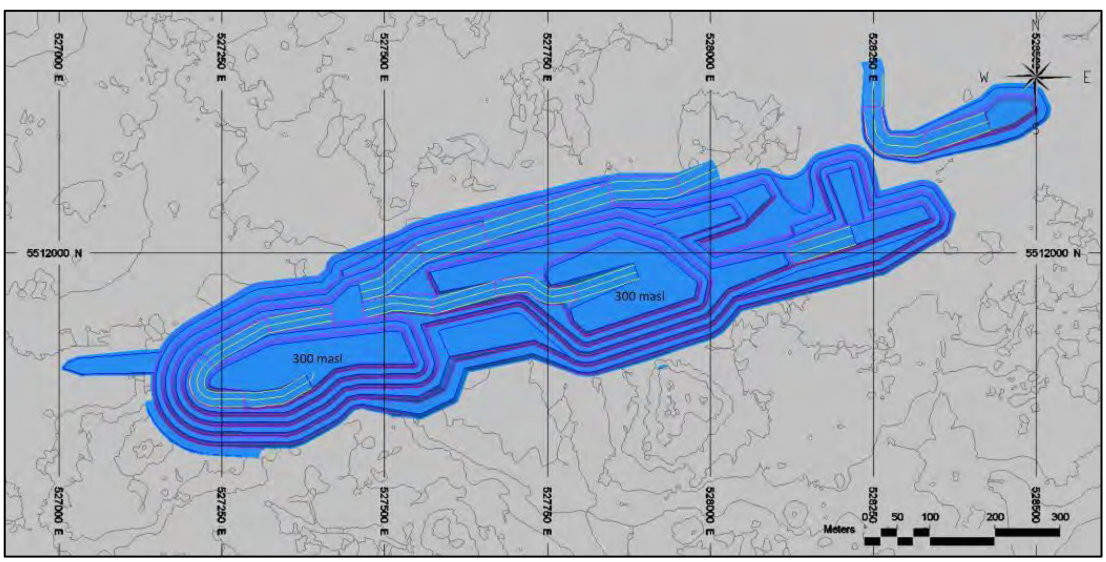
Phase 4 is a small external pit that is accessed by a slot coming from the north and swings east. This phase targets a portion of the deposit just to the east of the main pit. This phase is mined in the early part of the schedule along with Phases 1A and 1B and will be used as the prime portal location for the underground development (see Figures 16-8 and 16-9).

Figure 16-8: Phase 4 Design – Goliath



Source: AGP (2021).

Figure 16-9: Goliath Ultimate Pit



Source: AGP (2021).

16.10.2 Goldlund Project Pit Designs

For the Goldlund project area, pit designs were developed for a main pit with two phases and four satellite pits – two to the north east and two to the west along the trend of the deposit. The designs used a 10 m bench height.

Geotechnical parameters discussed in Section 16.2 were applied to pit designs, as shown in Table 16.13.

Table 16.13: Pit Design Slope Criteria – Goldlund

Area	Design Sector	Zone Code	Inter-Ramp Angle (degrees)	Slope height (m)
Goldlund	OB	1	27.3	10.0
	Rock (<100 m)	2	52.6	100.0
	Rock (100-150 m)	2	52.6	50.0
	Rock (150-200 m)	2	52.4	50.0

Equipment sizing for ramps and working benches is based on the use of 91 tonne rigid frame haul trucks like the Goliath designs. The operating width used for the truck is 6.7 m. This means that single lane access is 22 m (two times the operating width plus berm and ditch) and double lane widths are 28.7 m (three times the operating width plus berm and ditch). Ramp gradients are 10% in the pit and dump for uphill gradients. Working benches were designed for 35 to 40 m minimum mining width on pushbacks.

Tonnes and grade for the designed pit phases are reported in Table 16.14 using the diluted tonnes and grade from the resource model. Positive marginal block values from the optimisation run were used to delineate mill feed from waste.

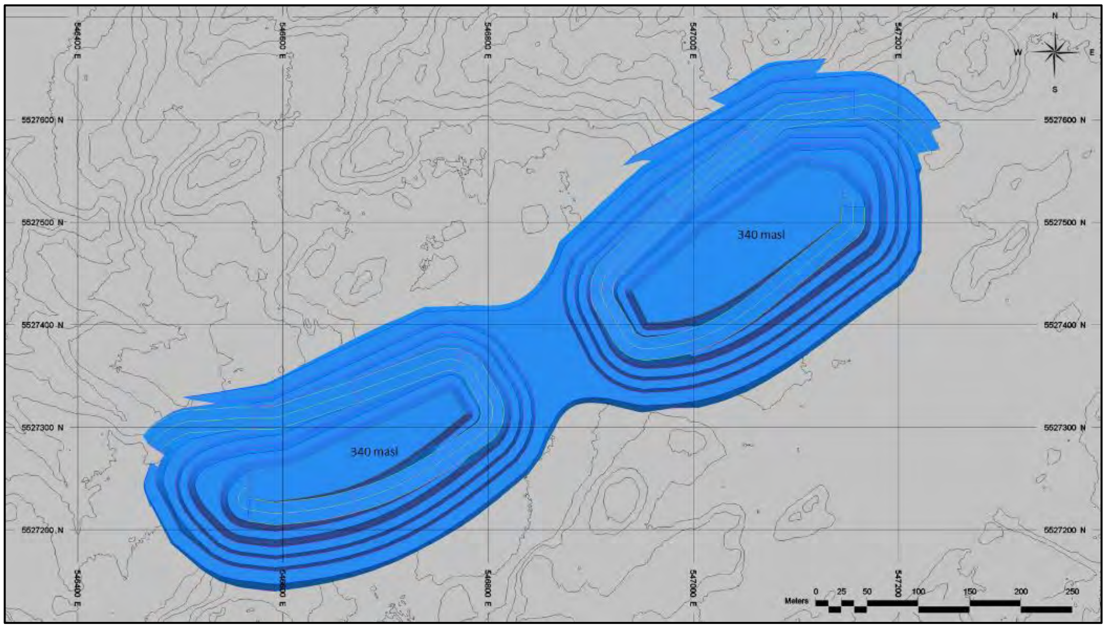
Table 16.14: Pit Phase Tonnage & Grades – Goldlund

Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
1	3,367,180	1.64	0	9,474,531	12,841,711	2.81
2	6,768,484	1.25	0	23,117,012	29,885,496	3.42
3	186,890	1.09	0	430,921	617,811	2.31
4	938,010	1.06	0	2,450,450	3,388,460	2.61
5	246,467	0.78	0	1,274,846	1,521,313	5.17
6	2,082,995	0.81	0	4,601,404	6,684,399	2.21
Total	**13,590,026**	**1.26**	**0**	**41,349,164**	**54,939,190**	**3.04**

16.10.2.1 Phase 1

Phase 1 is the first phase mined in the schedule for Goldlund and comprises the first phase of the main pit at Goldlund. The phase has two pit bottoms and will be mined down to the 340 masl elevation in both. All waste and mill feed access will be from two ramps: the east side exiting to the south and the west side exiting to the west (see Figure 16-10).

Figure 16-10: Phase 1 Design – Goldlund



Source: AGP (2021).

16.10.2.2 Phase 2

Phase 2 is the second phase mined at Goldlund in the schedule and comprises the second phase of the main pit. The phase again has two pit bottom and will be mined down to the 280 masl elevation on the east side and 290 on the west side. A single ramp starting on the southwest end and wrapping north and east provides access to both sides to the 340 masl elevation. Two separate ramps then access the bottoms on both sides from the saddle between the two pit bottoms (see Figure 16-11).

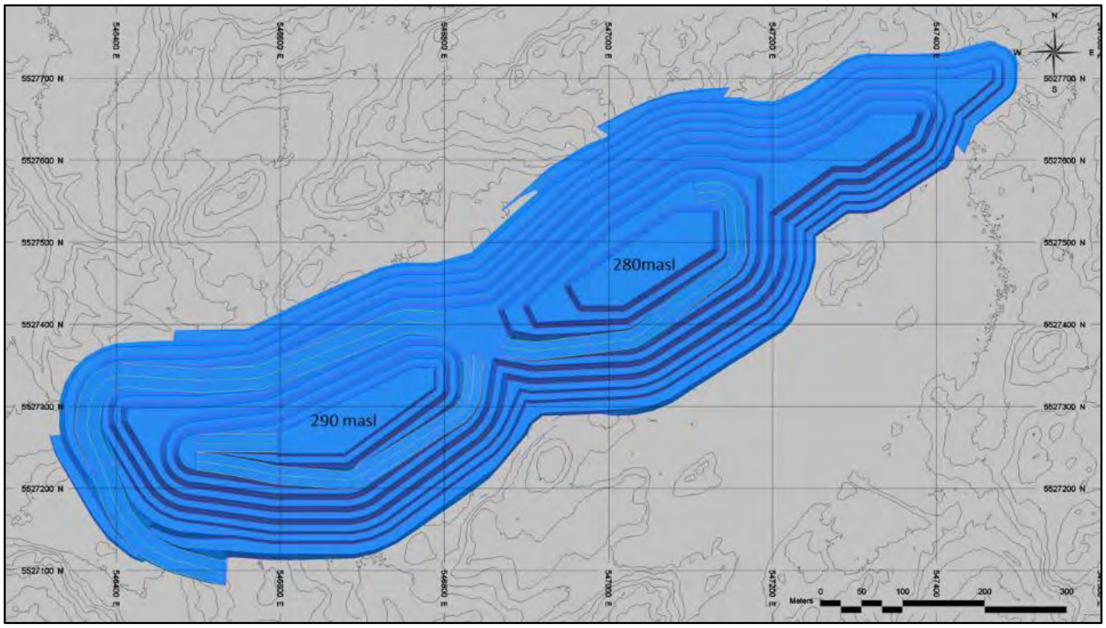
16.10.2.3 Phase 3

Phase 3 is the third phase mined in the schedule and is a small pit located to the north of the main pit. The phase will be mined down to the 400 masl elevation and is accessed by a slot ramp coming from the north and swinging east (see Figure 16-12).

16.10.2.4 Phase 4

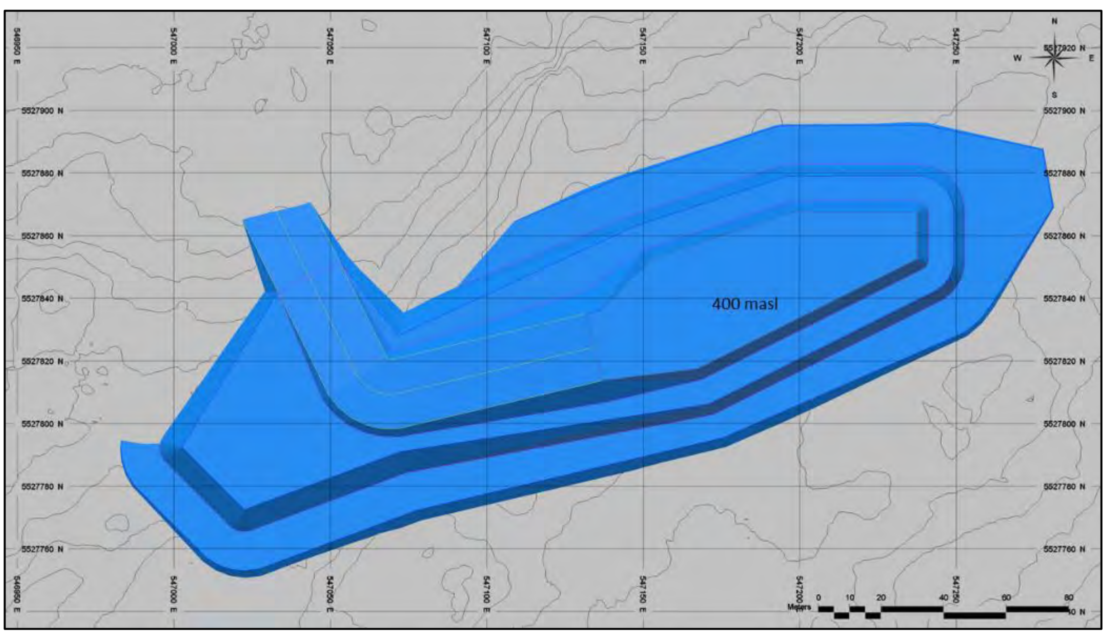
Phase 4 is the fourth phase mined in the schedule and is a pit located to the northeast of phase 2. The phase will be mined down to the 360 masl elevation. The phase is accessed by a ramp that starts on the south side and swings northeast and wraps the east end of the pit (see Figure 16-13).

Figure 16-11: Phase 2 Design – Goldlund



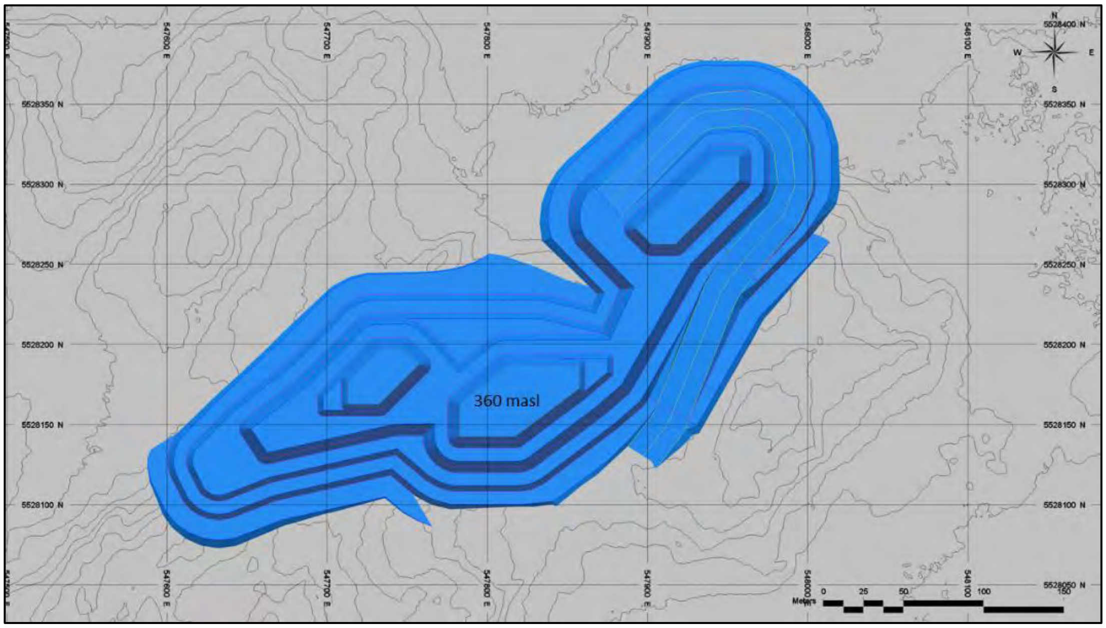
Source: AGP (2021).

Figure 16-12: Phase 3 Design – Goldlund



Source: AGP (2021).

Figure 16-13: Phase 4 Design – Goldlund



Source: AGP (2021).

16.10.2.5 Phase 5

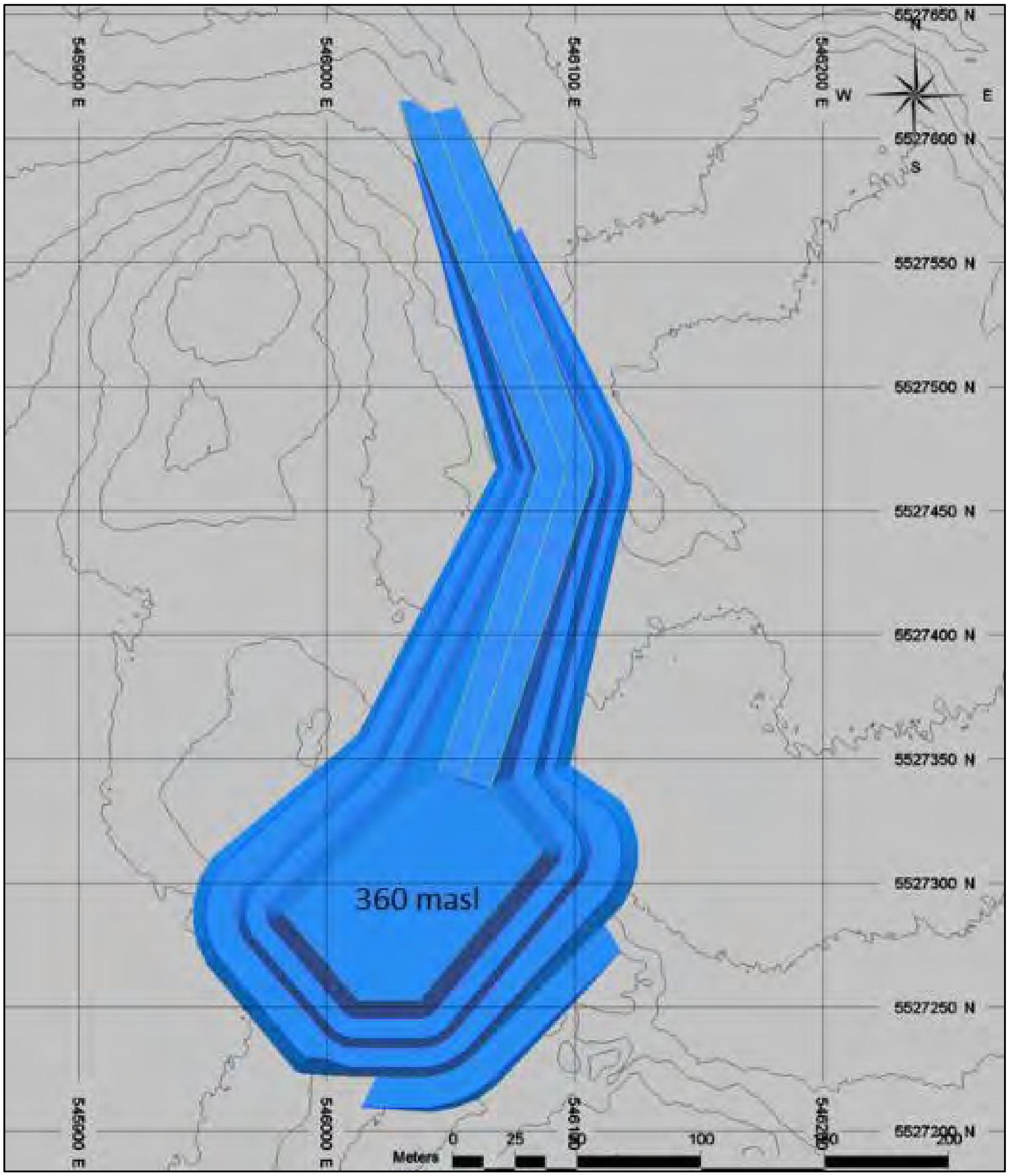
Phase 5 is the sixth phase mined in the schedule and is a small pit located to the northwest of the main pit. The phase will be mined down to the 360 masl elevation. The phase is accessed by a slot ramp that start on the north side and swings southeast (see Figure 16-14).

16.10.2.6 Phase 6

Phase 6 is the fifth phase mined in the schedule and is a pit located to the southwest of phase 5. The phase will be mined down to the 310 masl elevation. The phase is accessed by a ramp that starts on the south side spiralling counter-clockwise to reach the bottom (see Figure 16-15).

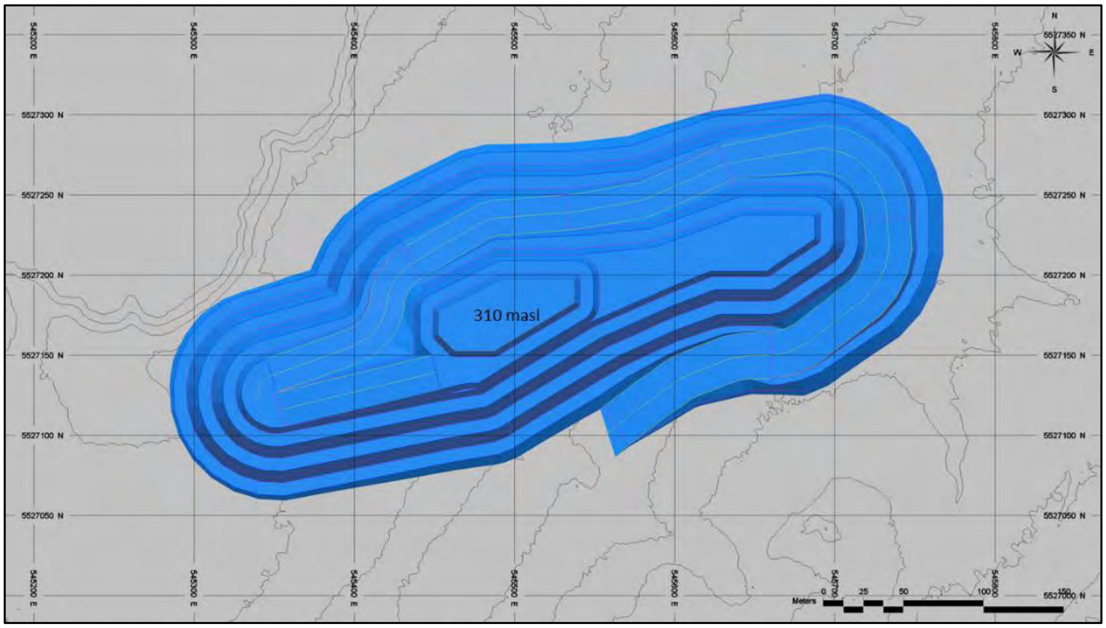
Also refer to Figure 16-16 Goldlund Phases as part of the total Goldlund pit phases.

Figure 16-14: Phase 5 Design – Goldlund



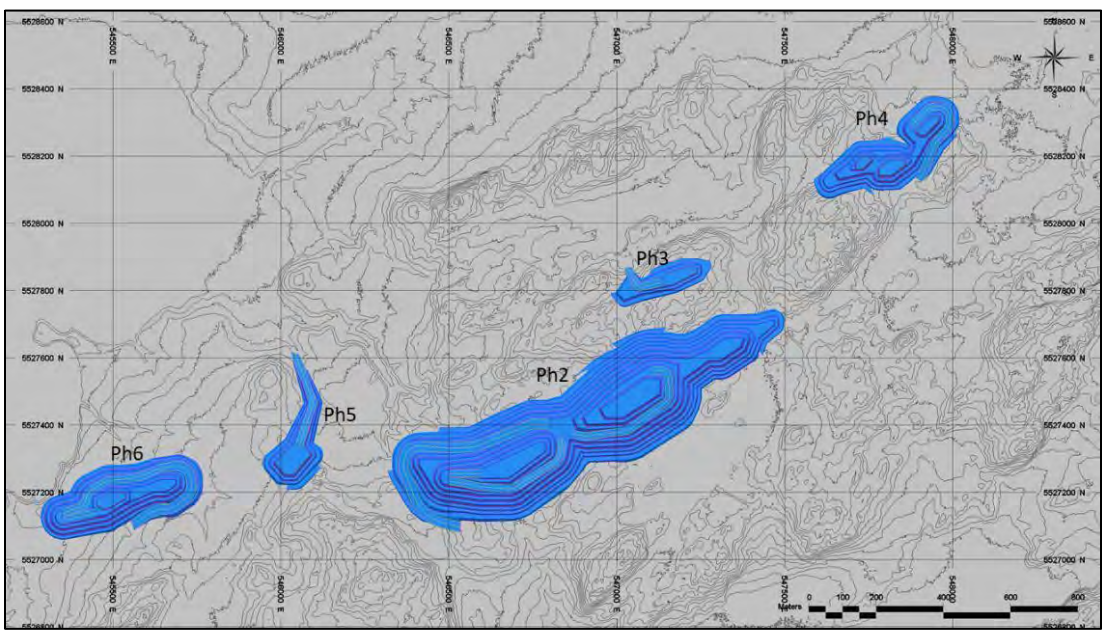
Source: AGP (2021).

Figure 16-15: Phase 6 Design – Goldlund



Source: AGP (2021).

Figure 16-16: Ultimate Goldlund Phases



Source: AGP (2021).

16.10.3 Miller Project Pit Designs

For the Miller project area, a single pit and phase design was developed using a 10 m bench height. Geotechnical parameters discussed in Section 16.2 were applied to the pit design as shown in Table 16.15.

Table 16.15: Pit Design Slope Criteria – Miller

Area	Design Sector	Zone Code	Inter-Ramp Angle (Degrees)	Slope height (m)
Miller	OB	1	27.3	12.0
	Rock	2	52.6	200.0

Equipment sizing for ramps and working benches is based on the use of 91 tonne rigid frame haul trucks like the Goliath and Goldlund designs. The operating width used for the truck is 6.7 m. This means that single lane access is 22 m (two times the operating width plus berm and ditch) and double lane widths are 28.7 m (three times the operating width plus berm and ditch). Ramp gradients are 10% in the pit and dump for uphill gradients. Working benches were designed for 35 to 40 m minimum mining width.

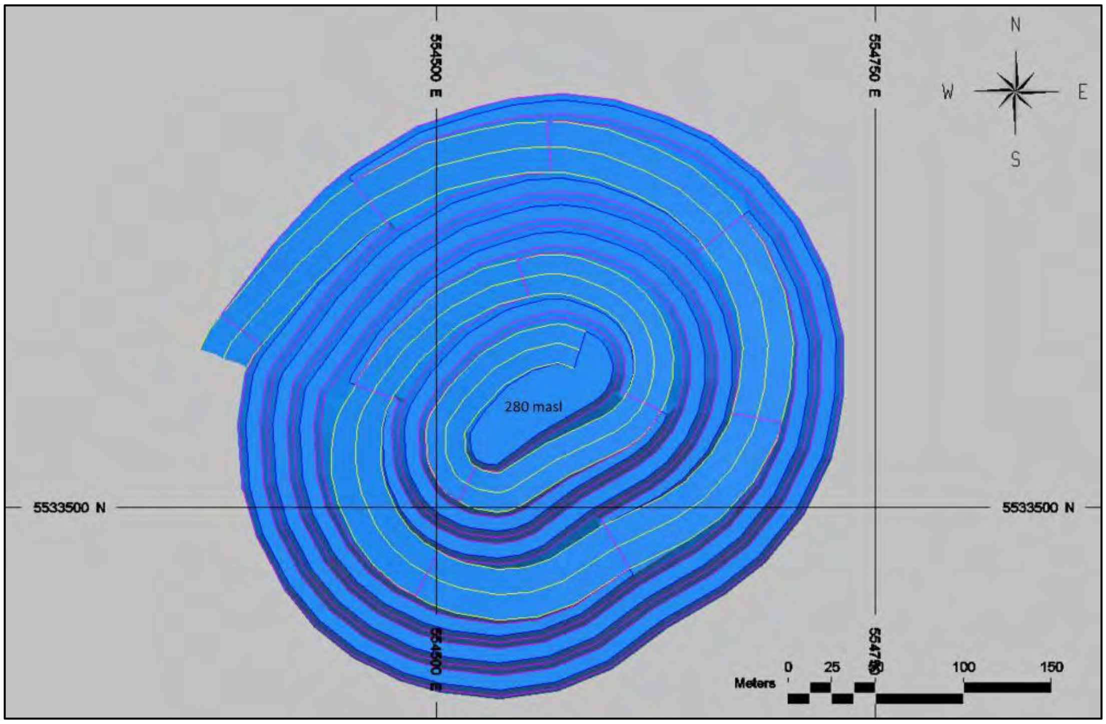
Tonnes and grade for the designed pit phases are reported in Table 16.16 using the diluted tonnes and grade from the resource model. Positive marginal block values from the optimisation run were used to delineate mill feed from waste.

Table 16.16: Pit Tonnage & Grades – Miller

Phase	Mill Feed (Mt)	Au (g/t)	Ag (g/t)	Waste (Mt)	Total (Mt)	Strip Ratio
Total	1,312,127	1.16	0	11,751,760	13,063,887	8.96

The Miller pit is the last phase mined in the overall schedule and comprises a single phase. The pit will be mined down to the 280 masl elevation. All waste and mill feed access will be single spiral ramp that starts on the north side and descends counter-clockwise (see Figure 16-17).

Figure 16-17: Pit Design – Miller



Source: AGP (2021).

16.11 Waste Rock Storage Facility Design

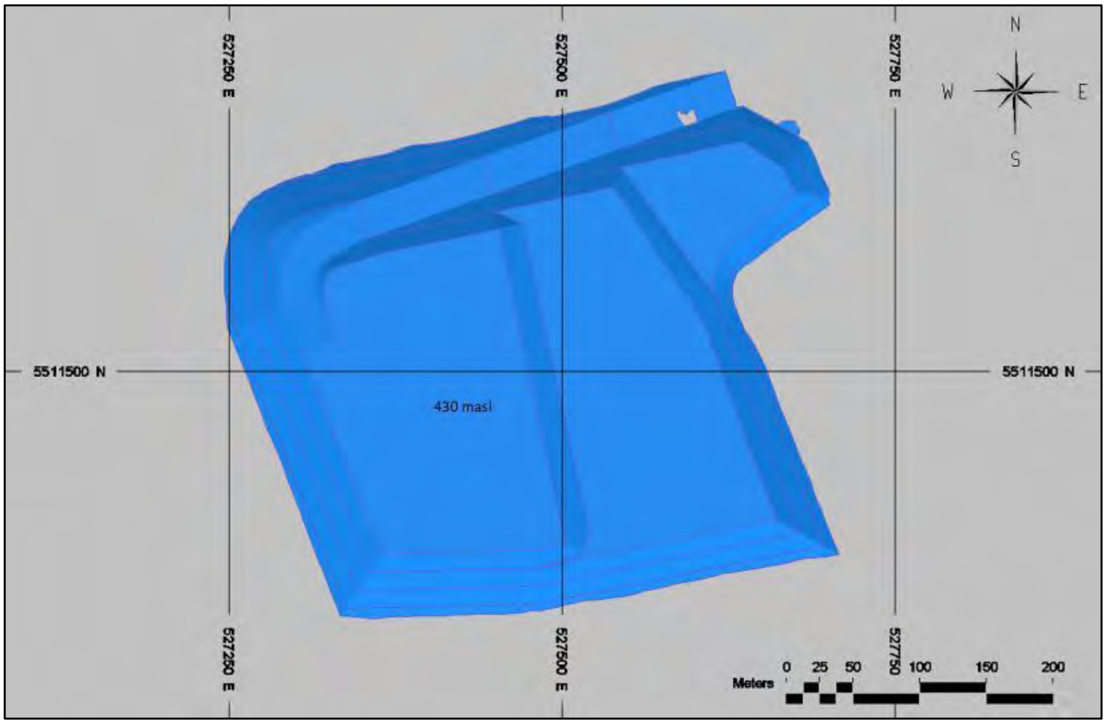
16.11.1 Goliath Waste Rock Storage

Overburden and waste rock are stored in two separate facilities adjacent to the main pit. Overburden will be stored in a stockpile located south of the west end of the main pit. It has a storage capacity of 3 Mm^3 and reaches a crest elevation of 430 masl.

The waste rock storage facility will be located directly north of the main pit. It has a capacity of 6.4 Mm^3 and reaches a crest elevation of 440 masl. The overburden stockpile is designed with 33° slopes and considers a 20% swell factor. The waste rock storage facility is designed with 20° slopes and considers a 30% swell (see Figures 16-18 and 16-19).

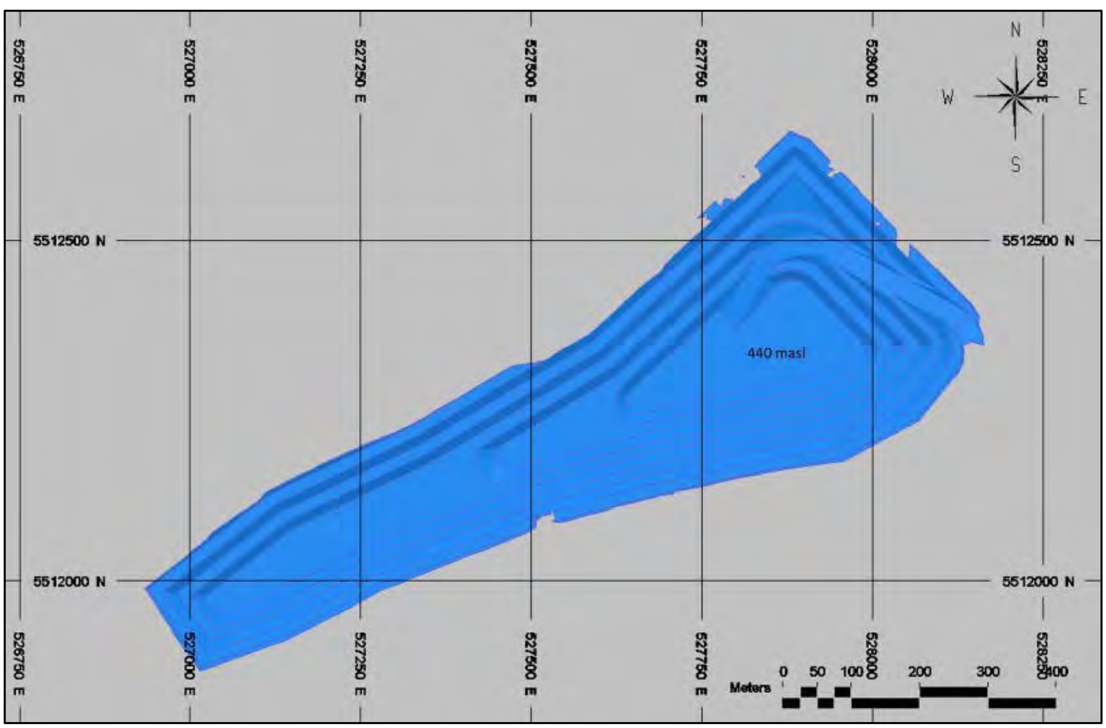
A total of 4.5 Mm^3 is stored in the Goliath pit by direct placement from mining after Phase 1A and 1B have been mined out. This comes from Phase 2 and 3.

Figure 16-18: Overburden Stockpile – Goliath



Source: AGP (2021).

Figure 16-19: Waste Rock Storage – Goliath

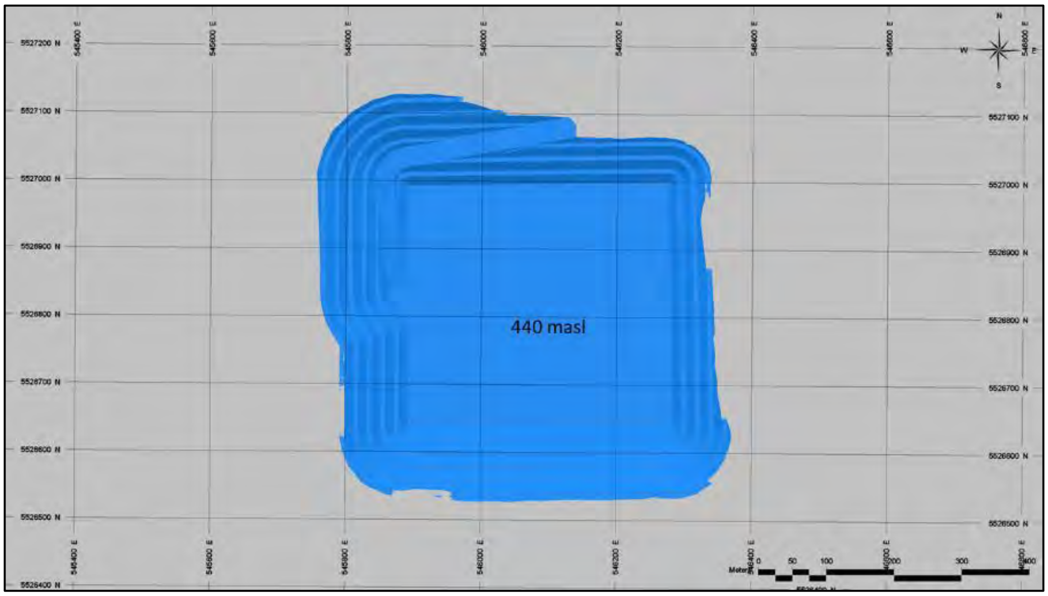


Source: AGP (2021).

16.11.2 Goldlund Waste Rock Storage

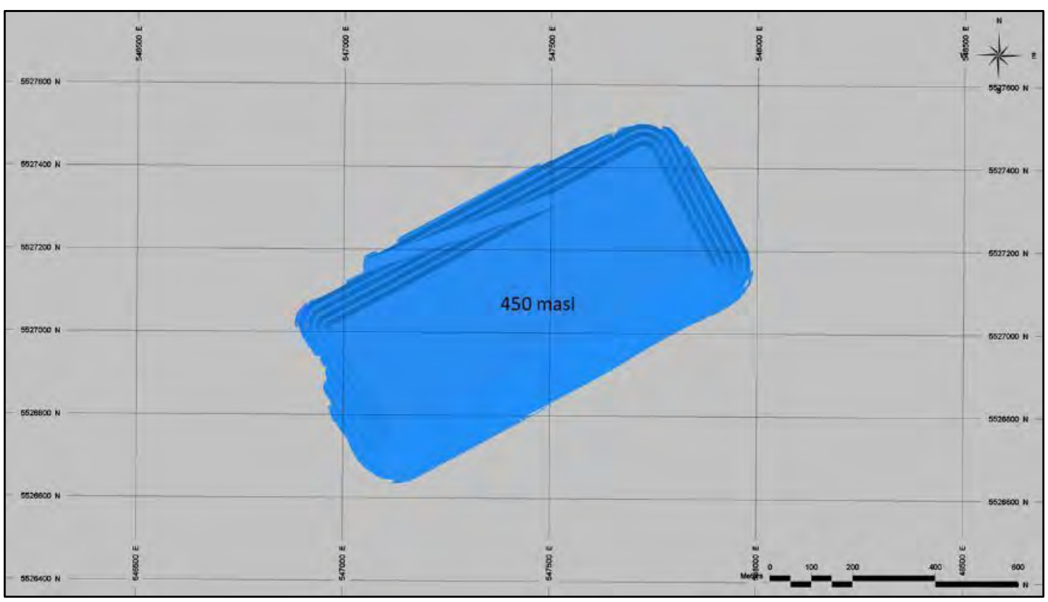
At the Goldlund area, overburden and waste rock are stored in two separate facilities adjacent to the main pit. Overburden will be stored in a stockpile located south of the southwest corner of the main pit. It has a storage capacity of 7.4 Mm^3 and reaches a crest elevation of 440 masl. The waste rock storage facility will be located directly south of the main pit. It has a capacity of 16.9 Mm^3 and reaches a crest elevation of 450 masl (see Figures 16-20 and 16-21).

Figure 16-20: Overburden Stockpile – Goldlund



Source: AGP (2021).

Figure 16-21: Waste Rock Storage – Goldlund



Source: AGP (2021).

16.11.3 Miller Waste Rock Storage

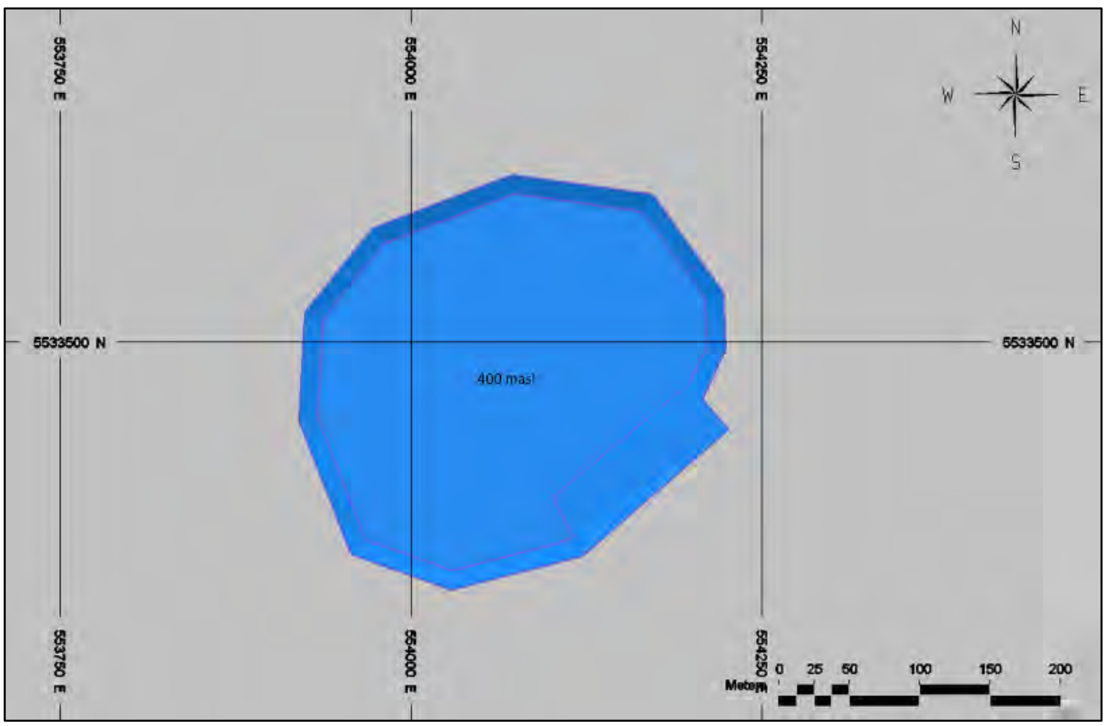
At the Miller area, overburden is stored in a stockpile west of the pit and waste rock is stored in a facility south of pit. The overburden stockpile has a capacity of 0.6 Mm^3 and can reach and elevation of 400 masl. The rock storage facility has a capacity to store 5.8 Mm^3 and can reach an elevation of 430 masl (see Figures 16-22 and 16-23).

Table 16.17 below provides a summary of soil stockpile and waste storage capacities for the project.

Table 16.17: Overburden Stockpile & Waste Storage Capacity

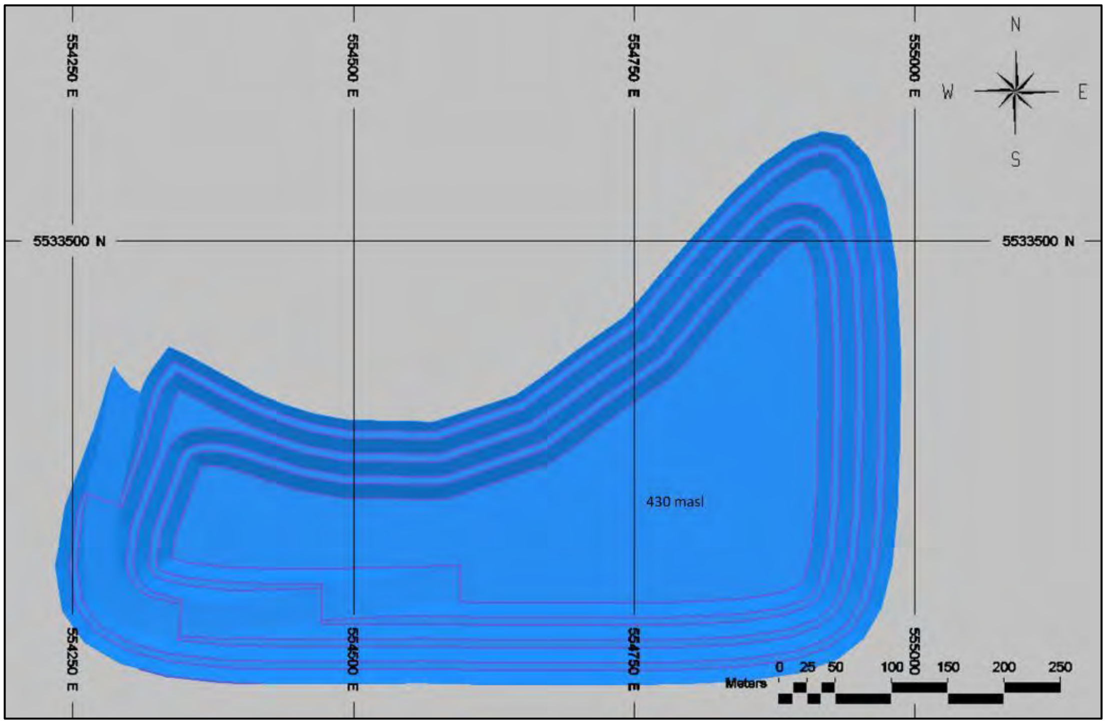
Facility	Waste Storage Capacity (Mm^3)		Maximum Elevation (masl)	
	Overburden	Rock Storage	Overburden	Rock Storage
Goliath	3.0	6.4	430	440
Goldlund	7.4	16.9	440	450
Miller	0.6	5.8	400	430

Figure 16-22: Overburden Stockpile – Miller



Source: AGP (2021).

Figure 16-23: Waste Rock Storage – Miller



Source: AGP (2021).

16.11.4 Blasting & Explosives

Blast patterns be the same for feed and waste to assist in mine productivity. Mill feed and waste patterns will be 4.8 m x 4.2 m (spacing x burden). Holes will be 10 m plus an additional 0.8 m sub-drill for a total 10.8 m using a 140 mm bit.

Mill feed powder factors are 0.28 kg/t, and the waste powder factor is 0.27 kg/t. Only emulsion explosives are currently estimated to be used.

Pre-shear holes will be 10 m deep, spaced 1.70 m apart and be separated from production blasts by 1.9 m. The powder charge will be 19 kg per hole in a decoupled manner.

16.11.5 Open Pit Mining Equipment

The mining equipment selected to meet the required production schedule is conventional mining equipment, with additional support equipment for snow removal and surface ditching maintenance.

Drilling will be completed with down the hole (DTH) hammer drills with a 140 mm bit. This provides the capability to drill 10 metre bench heights but will require an additional steel to be added from its carousal.

The primary loading units will be 13 m^3 front-end loaders. Additional loading will be completed by 6.7 m^3 hydraulic excavators. It is expected that one of the loaders will be at the primary crusher for the majority of its operating time. The haulage trucks will be conventional 91 tonne rigid body trucks.

Mill feed will be hauled from Goldlund and Miller using heavy Kenworth trucks pulling 36 tonne belly dump trailers. They will travel partly on the highway and partly on an upgraded backroad between Goliath and Goldlund. This is to avoid travel in the local communities.

The support equipment fleet will be responsible for the usual road, pit, and dump maintenance requirements. But due to the extra haulage roads required for hauling mill feed from Goldlund and Miller, they will have a larger role in snow removal and water management. Snowplows and additional graders have been included in the fleet. In addition, smaller road maintenance equipment is included to keep drainage ditches open and sedimentation ponds functional.

16.11.6 Open Pit Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralisation taking samples each metre. The holes will be inclined at 60°.

In areas of low-grade mineralisation or waste the pattern spacing will be 20 m x 10 m with sampling over 6 m. These holes will be used to find undiscovered veinlets or pockets of mineralisation.

These grade control holes serve to define the mill feed grade and mineralisation contacts.

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Blasthole sampling while not included in the cost may become part of the program should a gold deportment study show that the information would be reliable for grade control. For the PEA no blasthole sampling is considered.

16.12 Underground Mining

16.12.1 Mineral Resources for Underground Mining

The diluted and recovered mineral resources utilised in the underground mine plans for the Goliath deposit are shown by resource category in Table 16.18.

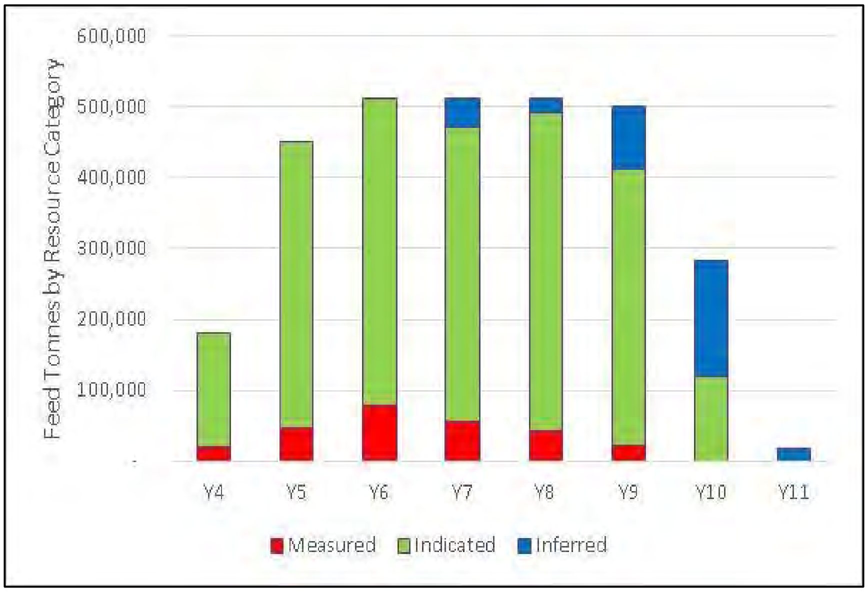
Table 16.18: Mineral Resources Utilised in the Goliath Underground Mine Plan

Description	Tonnes (kt)	Grade (g/t Au)	Grade (g/t Ag)
Measured Resources	271	3.7	9.1
Indicated Resources	2,381	3.7	9.3
Inferred Resources	313	3.3	6.9
Total	**2,965**	**3.7**	**9.0**

Note: Totals may be affected by rounding.

The recovery of resources by classification is shown in Figure 16-24.

Figure 16-24: Recovery of Resource Tonnes



Source: AGP (2021).

16.12.2 Mining Method Selection

Longhole stoping was selected as the preferred mining method due to the fair to good classification of the orebody and country rock ground conditions and the dip and thickness of the deposit.

The regular, steep dipping geometry of the mineralisation indicates a long hole stoping method to be appropriate. The width of the stopes varies from a minimum stope width of 1.8 m to around 11 m with some pinching and swelling exhibited but averaging 6.2 m in width.

A vertical level interval of 25 m was planned, apart from occasional stopes with reduced height where blind up-hole drilling is planned, as well as a few levels in the interface area with the open pit. In higher grade areas above 4.0 g/t Au, where cemented rockfill is economically justified to minimise resource losses, a continuous retreat system is planned with production pausing every 28 m along strike for backfilling and cement curing. Mining of the adjacent stope will recommence from the cemented fill wall, without the need for a rib pillar to separate adjoining stopes. In lower grade areas below 4.0 g/t Au, where uncemented rockfill will be utilised, unrecoverable rib pillars (2/3 the width of the ore zone) are planned every 28 m along strike. This width is a minimum of 3 m. The rib pillars will constrain the uncemented rockfill, thus eliminating rockfill dilution in the stope muck. In those mining areas employing unrecoverable rib pillars, the pillars represent a 13% to 15% loss of recoverable in-situ resources.

The overall stoping arrangement is comprised of three main stoping zones (zones A, B, and C), and two minor zones (D and E) as illustrated in Figure 16-25. The minor stoping zones are noted to be of lesser economic unit value than the main stoping zones, and as such have been incorporated in the mine schedule towards the end of life-of-mine.

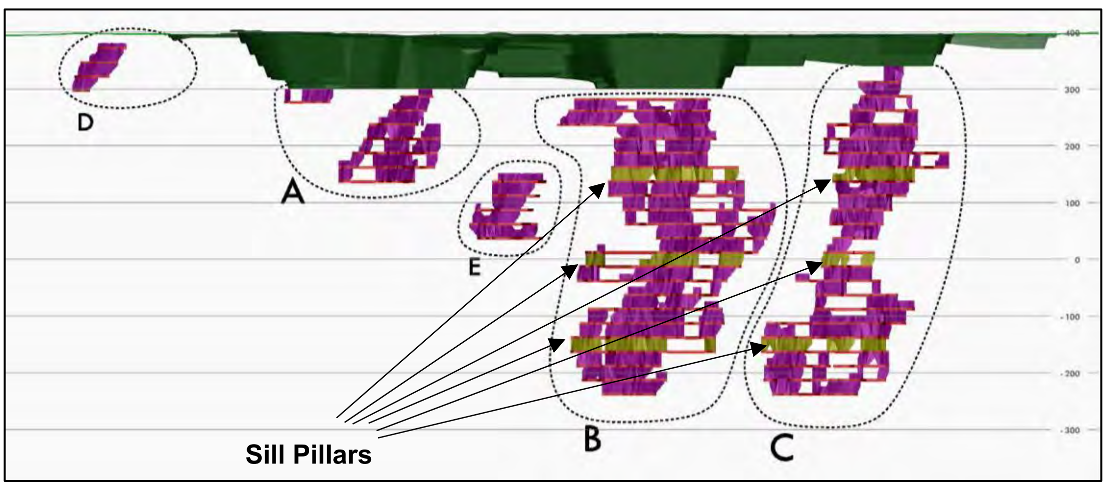
Each of the three main stoping zones extend to the floor of the ultimate open pit. The relative timing between the open pit and underground operations, as well as the need to utilise a portion of the open pit for in-pit waste storage, prevent the extraction of some underground stopes up to the open pit floor. Underground stopes in stoping zones A and C can be mined up to the open pit floor in those areas, whilst those in stoping zone B have not been due to open pit waste placement.

For the main stoping zones B and C that are comprised of a continuous unbroken sequence of down-dip stoping levels to depth, 25 m high partially recoverable sill pillars will be used to create several separate stope production areas within these zones. Each stope production area is comprised of from four to six stoping levels, included the associated sill pillar if applicable. In this way, multiple production areas will be created in stoping zones B and C which can be mined concurrent with each other if desired. Fifty percent of the sill pillar tonnage will be recovered by blind up-hole stoping methods from the level below.

Stope production will commence at the bottom of each production area, immediately above the associated sill pillar level. The stopes are mined from the extents on each level to the central access (on retreat) employing downhole drilling techniques. Stoping will progress upwards through the production area, on a level by level basis. Stopes on subsequent levels will utilise the rockfill floor, comprised of either cemented of uncemented rockfill, created by backfilling the stope below.

The stoping zones and planned sill pillar locations are illustrated in Figure 16-25.

Figure 16-25: Goliath Mining Zones & Sill Pillars



Source: AGP (2021).

16.12.3 Cut-off Grade

A preliminary cut-off analysis was undertaken at the commencement of the study using broadly estimated costs and factors to determine an appropriate breakeven combined gold and silver net revenue of $89.38/t, equivalent to an in-situ gold cut-off grade of 1.63 g/t. Treasury Metals elected to utilise an elevated cut-off grade strategy for stope design purposes by applying an additional $20/t minimum return to the breakeven calculation. A combined gold and silver net revenue of $109.69/t (rounded to $110/t) cut-off value was therefore used to identify stopes, equivalent to an in-situ gold cut-off grade of 2.0 g/t. Table 16.19 shows a summary of these analyses.

Table 16.19: Underground Cut-off Value/Grade Analyses

Description	Value	Unit	Breakeven Analysis	$20/t Min. Return
In-situ Cut-Off Grade				
Gold Grade		g/t	1.63	2.00
Silver Grade		g/t	4.84	5.94
Effective Mining Dilution @ 20% Feed Grade		%	15.0%	15.0%
Ore to Process Plant				
Gold Grade		g/t Au	1.45	1.78
Silver Grade		g/t Ag	4.32	5.30
Metal Prices	1.33	CAD:USD		
Gold	$1,475.00	US$/oz		
Gold Refining	5.00	US$/oz Au		
Net Gold Price	**$1,955.10**	C$/oz		
Silver	$20.00	US$/oz		
Net Silver Price	**$26.60**	C$/oz		
Revenues				
Gold Metallurgical Recovery	95.5%			
Payable Gold	99.8%			
Silver Metallurgical Recovery	62.6%			
Payable Silver	97.0%			
Net Revenue Gold		C$/t	87.14	106.93
Net Revenue Silver		C$/t	2.24	2.75
Net Revenue		**C$/t**	**$89.38**	**$109.69**
Operating Costs				
U/G Mining Operating Costs Used		C$/t processed	65.00	65.00
U/G Mine Sustaining Capital Cost	10%	C$/t processed	6.50	6.50
Process		C$/t processed	14.92	14.92
G&A		C$/t processed	1.62	1.62
Royalty Gold	1.5%	% net revenue	1.31	1.60
Royalty Silver	1.5%	% net revenue	0.03	0.04
Minimum Return		C$/t processed	-	20.00
Total Operating Cost		**C$/t processed**	**89.38**	**109.69**

Near to the end of the mine planning process Treasury Metals relaxed the $20/t minimum return requirement which allowed limited additional, slightly lower grade material – largely contained within stoping zone E and the upper portion of zone B - to be included in the mill feed estimate towards the end of mine life.

16.12.4 Application of Modifying Factors to Estimate ROM Feed

The methodology to modify in-situ stope resources to estimate mill feed is described in Table 16.20. The resulting estimates of mill feed by mining zone and elevation are shown in Table 16.21.

Table 16.20: Modifying Factors

Description	Methodology	Average Factor
Sill Pillars	Designed as part of the stope in-situ resources 25 m high	
Rib Pillars	Uncemented fill areas: % of in-situ stope resources.	13% to 15%
	Cemented fill areas: % of in-situ stope resources.	0%
External (Unplanned) Dilution	Development	5%
	Stopes	16%
	(Applied at 20% to 30% of in-situ grade)	
	Average Dilution	14.2%
Tonnage Recovery	Development	100%
	Stopes	98%
	Rib Pillars	0%
	Sill Pillars	50%

Table 16.21: Estimated Mill Feed by Zone & Elevation

Elevation	Zone A			Zone B			Zone C			Zone D			Zone E		
	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag	kt	g/t Au	g/t Ag
370 L										5.2	3.2	5.8			
345 L										17.4	3.1	5.6			
320 L										17.9	3.0	5.5			
295 L										15.6	3.2	5.7			
275L	18.7	2.9	1.9												
285L	4.5	4.2	3.7	-			41.1	2.8	7.1						
260 L	37.2	6.5	4.3	68.3	4.4	11.4	23.8	3.4	12.9						
235 L	42.9	8.3	4.8	110.9	3.9	9.7	27.3	3.8	19.9						
210 L	54.3	7.1	4.6	90.8	4.6	11.4	49.8	3.2	18.6						
185 L	44.2	5.8	4.9	97.6	4.7	12.2	67.3	2.8	17.6						
160 L	55.7	6.2	5.2	103.3	3.8	11.3	95.3	2.7	15.5						
135 L	64.8	5.2	5.2	35.9	3.2	9.7	28.3	3.0	15.2				13.0	2.3	6.3
110 L				63.2	2.6	6.5	78.3	3.2	13.2				30.3	2.5	5.6
85 L				42.4	2.3	5.3	80.5	3.8	13.6				43.2	2.7	5.3
60 L				52.6	2.3	7.2	69.5	4.5	16.8				57.3	2.3	6.7
35 L				55.8	2.5	8.2	47.3	4.5	19.9				67.3	2.1	8.8
10 L				72.7	2.6	6.8	21.1	5.6	20.9						
-15 L				34.9	2.8	6.1	6.3	4.2	7.1						
-40 L				81.9	2.8	5.5	28.8	3.6	8.4						
-65 L				41.6	2.9	4.4	28.9	5.0	10.3						
-90 L				47.6	2.6	3.1	31.8	4.0	9.4						
-115 L				66.2	3.1	4.5	48.4	2.8	9.6						
-140 L				93.2	3.5	5.7	76.2	3.6	6.8						
-165 L				39.2	3.6	6.9	21.4	4.1	8.0						
-190 L				56.5	3.7	6.6	52.4	4.0	8.4						
-215 L				45.5	3.0	5.5	50.3	4.0	7.7						
-240 L				44.5	3.2	4.8	58.3	3.5	7.2						
Total	322.4	6.2	4.7	1,344.6	3.4	7.8	1,032.2	3.6	12.7	56.1	3.1	5.6	211.0	2.4	6.9
Grand Total	2,965.3	3.7	9.0												

Note: Surface elevation is approximately +400 m.

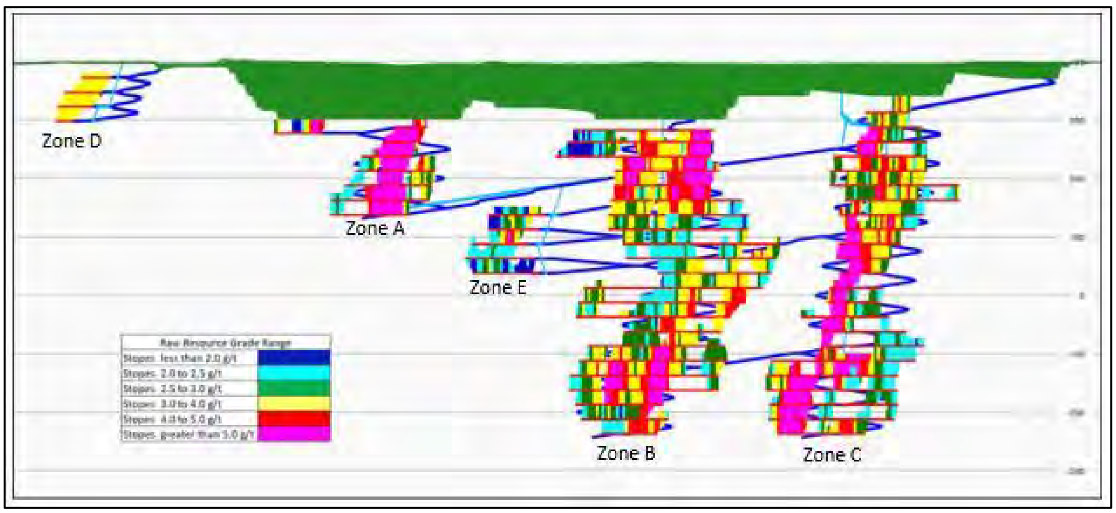
16.12.5 Mine Design

16.12.5.1 Stope Designs

Potentially economic stoping areas were identified on each mining level throughout the deposit by manually designing stope outlines in section whilst recognising the level spacing and minimum mining width requirements.

Stopes outlines were designed on a 5 m section spacing. The stopes, coloured by in-situ resource grade ranges, are shown in Figure 16-26.

Figure 16-26: Goliath Final Stope Slices Coloured by In-situ Resource Grade Range

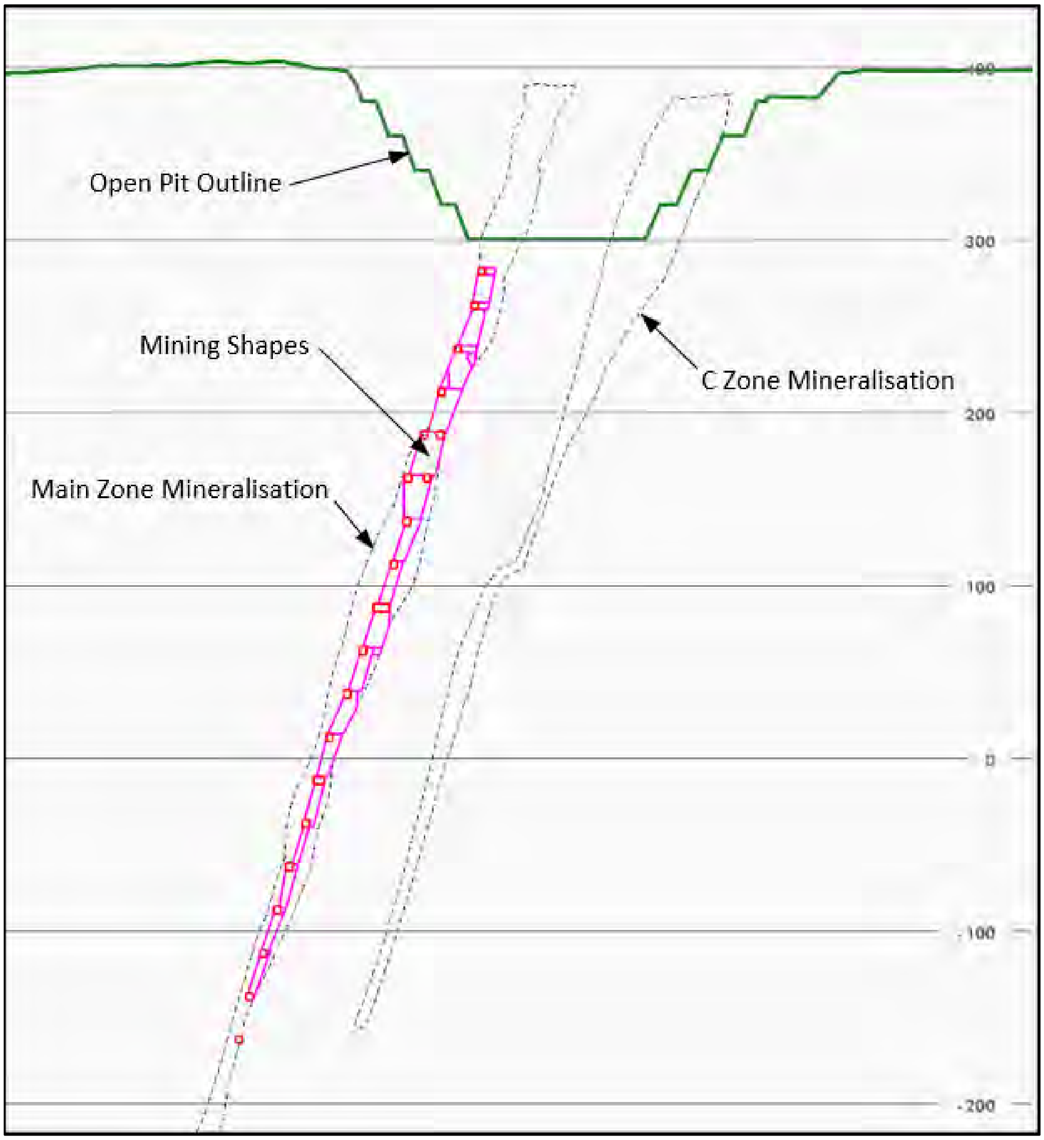


Source: AGP (2021).

In general, the Goliath stopes are located within two main mineralised zones with a total strike extent of about 3 km within which designed stope strike length was about 1.5 km. Many smaller mineralised zones are also present in and around the main mineralised zones. The majority of the stope hanging wall and footwall contacts are wholly contained within the mineralised zones, thus stope dilution is often mineralised.

A section through the most continuous portion of stoping zone B showing the relationship between the main mineralised zones and the stopes in this area is illustrated in Figure 16-27.

Figure 16-27: Stoping Section through Stoping Zone B Showing Main Mineralised Zones



Source: AGP (2021).

16.12.5.2 Mine Access Design

The stoping arrangement of the three main stoping zones enabled a simple mine access layout consisting of a centrally located level access and return air raise (RAR) arrangement to be adopted for each of the main stoping zones. Two main ramp layout options were evaluated during the course of the study:

- a single shared main ramp located between stoping zones B and C creating in effect a single B/C stoping zone, and a main ramp for stoping zone C
- separate main ramps for stoping zones A, B and C

The two ramp design options were fully designed to incorporate the other minor stoping zones and life-of-mine production schedules generated from each design option. Option 2 was selected for the study due to the improved project economics resulting from the higher production rate, earlier start and quicker build up to full production, as well as the realisation of higher grade stope production in the early years resulting from this option.

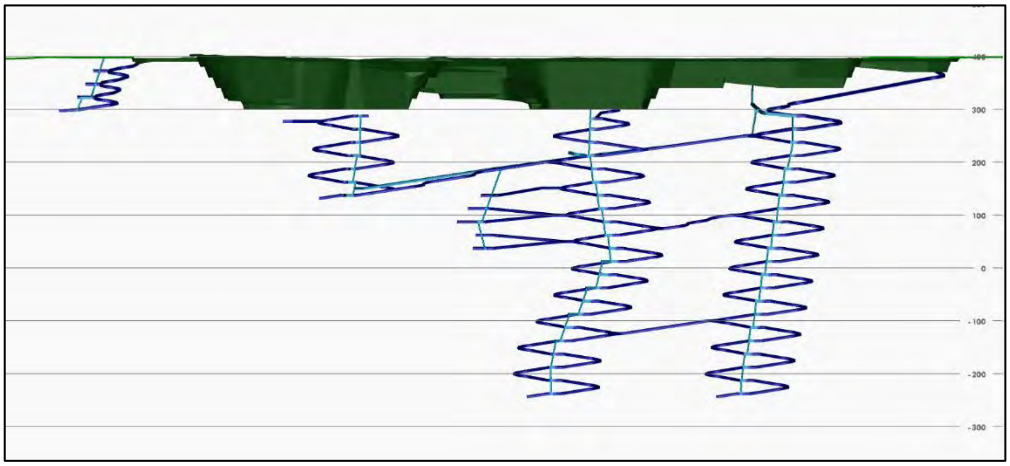
Additionally, recognising the opportunity for a 15-month delay in placing open pit waste into the mined-out central portion of the Goliath open pit allowed a second development crew to commence mine development from the open pit floor, resulting in a further decrease in both the underground pre-production period and the underground production build-up time, and improving project economics.

The underground mine access design is illustrated in Figures 16-28 to 16-31, and can be described as follows:

- The underground mine is accessed from a single ramp to surface, with the ramp portal located within a shallow open pit (Goliath Phase 4) to the east of the main Goliath open pit.
- A single return air raise to surface is located near the top of stoping zone C and incorporates a horizontal drift to locate the ventilation raise collar outside of the ultimate open pit rim.
- The zone C ramp spirals down in a 'racecourse' configuration following the level accesses for stoping zone C. The ventilation arrangement for stoping zone C is comprised of a series of short raises driven between each level and also serves as an alternate means of egress for the stoping zone.
- A main ramp connection is made between the upper portions of stoping zone C and B, and spirals down zone B in a similar fashion to that of Zone C. The zone B ramp also spirals up to the top of the stoping zone and the temporary open pit floor access previously described. An engineered mass concrete plug will be constructed in the pit floor ramp access to separate the open pit environs from the underground mine. Several ramp connections are provided between zones B and C to facilitate mobile equipment access between these two mine stoping zones. The ventilation arrangement for zone B is the same as that of zone C, but with the addition of a lateral ventilation connection drift to connect to the RAR to surface. The stoping levels for the minor stoping zone E are accessed from the zone B ramp.
- A main ramp connection is also made between the upper portion of stoping zone B and A, and spirals up zone A in similar fashion to the other stoping zones. The ventilation arrangement is also similar to the other zones, incorporating an extension of lateral ventilation connection developed for zone B.
- Stoping zone D is accessed independent from the other stoping zones from a shallow boxcut and portal located at the western extent of the Goliath open pit. Ventilation and emergency egress is provided by a separate ventilation raise system to surface, with the raise collar located in an area of approximately 1.5 m of overburden.
- Each stoping level in each stoping zone incorporates either a storage bay or electrical bay, and a local sump. Additionally, each level is provided with two large bays to allow either production truck loading of ROM material from each active level and backfill handling during the stope filling cycle.
- Main ramp dimensions are 5.0 m x 5.0 m (H), except for the portion of main ramp from surface to the upper portion of the stoping zone C ramp where the ramp bifurcates to access stoping zone B. This portion of the ramp is 6.0 m x 6.0 m (H) in order to limit the airflow velocity in this portion of the ramp system.

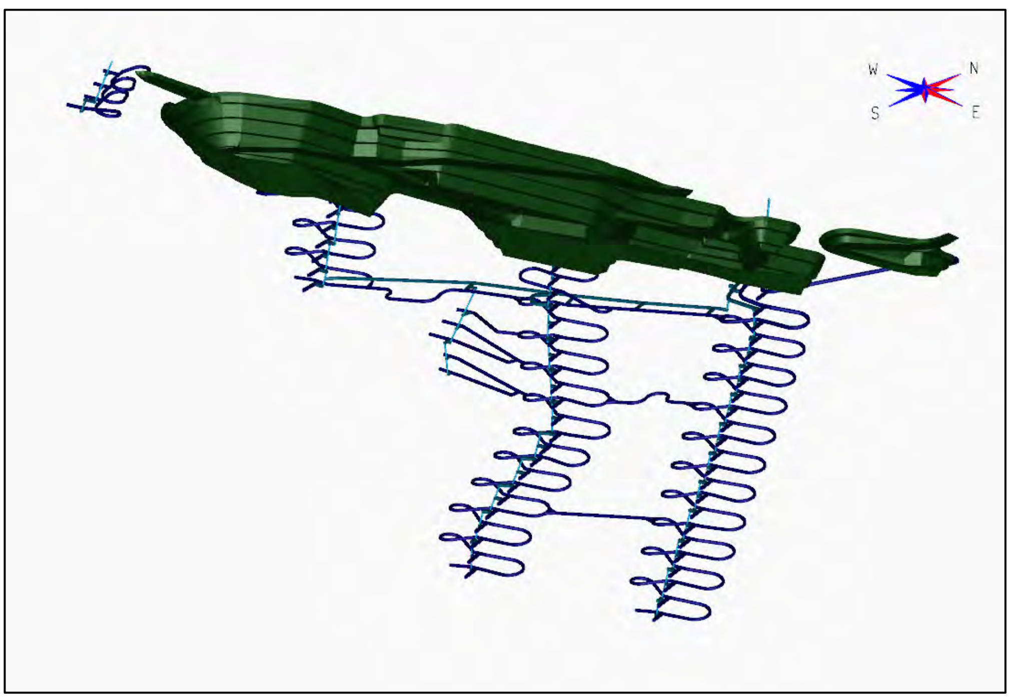
- Ventilation raise dimensions for all internal raises are 3.0 m x 3.5 m, whilst the 87 m-long raise to surface is 4.5 m x 4.5 m.
- Pump stations are located at the bottom of stoping zones A, B and C as well as at mid elevation of stoping zones B and C, for a total of five pump stations.
- A small underground maintenance facility is located at mid elevation of stoping zone B to provide for servicing and light repairs to the mobile equipment fleet.

Figure 16-28: Long Section of Underground Ramp & Ventilation Arrangement



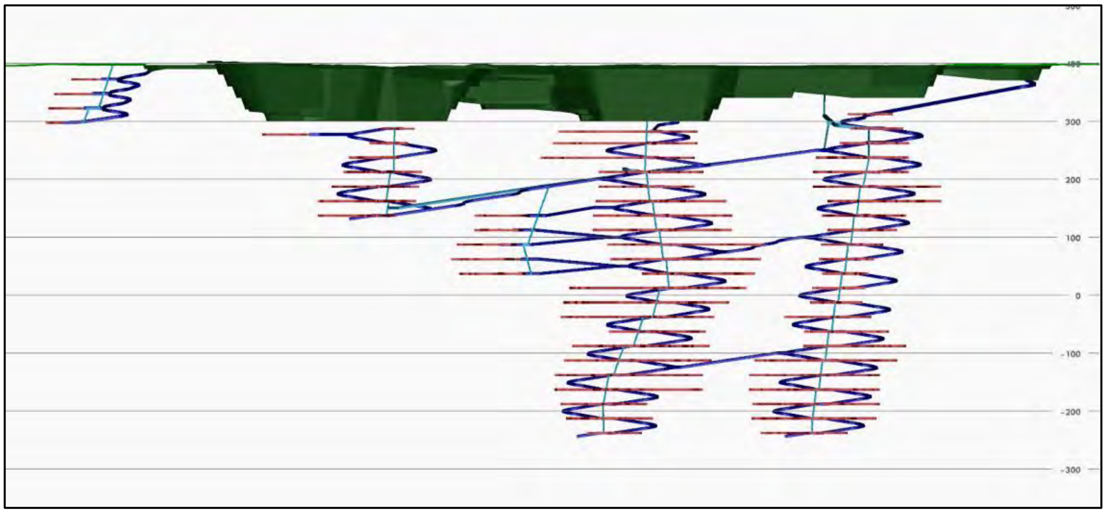
Source: AGP (2021).

Figure 16-29: Isometric of Underground Ramp & Ventilation Arrangement



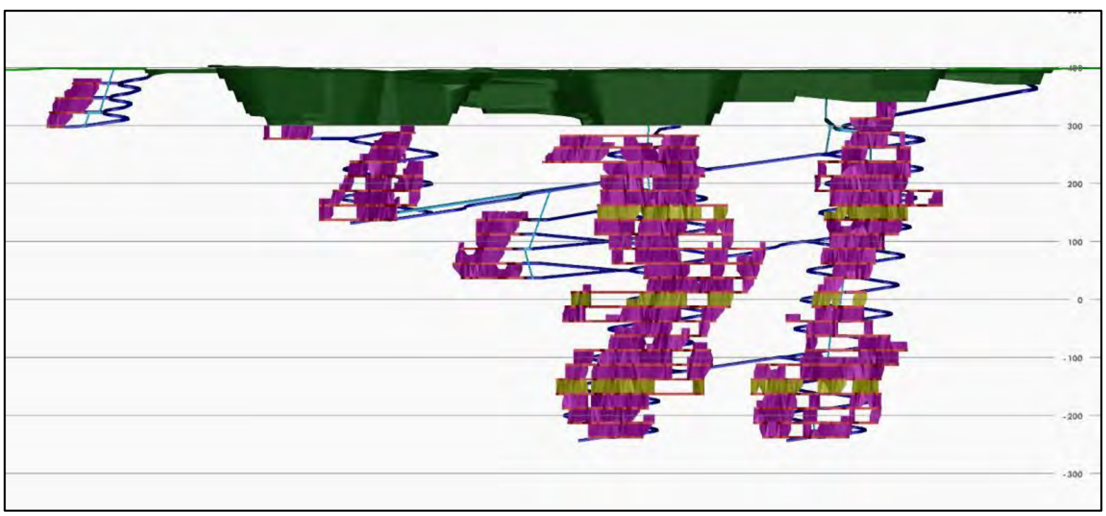
Source: AGP (2021).

Figure 16-30: Long Section of Underground Ramp, Ventilation & In-Vein Development



Source: AGP (2021).

Figure 16-31: Long Section of Mine Layout Design, Stope & Sill Pillar Arrangement



Source: AGP (2021).

16.12.6 Development & Production Scheduling

The mine development and production schedule were manually scheduled in Excel® by quarter using the following process:

- account for all required mine development based on the mine layout design
- schedule mine development using appropriate mine development advance rates
- schedule mine production using appropriate average stope productivity rates based on average stope widths, LHD tramming distances, rockfilling requirements and other relevant factors

The various development types in the mine layout were categorised and individually factored to account for additional associated development not specifically designed, but known to be required during development, such as remuck bays, electrical equipment bays, and over-height excavations. Development drift types, associated development factors, and planned drift dimensions are summarised in Table 16.22.

Table 16.22: Development Factors & Drift Dimensions by Development Type

Development Type	Development Factors	Drift Width (m)	Drift Height (m)
Ramp - Upper	1.13	6.0	6.0
Ramp - Lower	1.03	5.0	5.0
Level Access	40 m	5.0	5.0
Vent	1.03	5.0	5.0
Ore Access	1.00	4.0	4.5
Pump Station/Workshop	1.02	5.0	5.0

The resultant total waste development by development type are summarised in Table 16.23.

Table 16.23: Total Mine Development in Waste

Development Type	Length (m)	Proportion
Ramp	14,467	57%
Level Access	4,974	20%
Ventilation	2,433	10%
In-Vein Waste	2,669	11%
Pump Station/Workshop	459	2%
Total	**25,002**	**100%**

Mine development advance rates based on the number of available headings per scheduling period are summarised in Table 16.24. In-vein development rates are generally lower than similar sized waste development headings due to being developed under geology control for heading size and direction, as well as delays due to sampling requirements, etc.

Table 16.24: Quarterly Development Advance Rates by Heading Type & Availability

Heading Type	Single Heading	Double Heading	Multiple Heading
Waste Development Heading	450 m	630 m	780 m
In-vein Development Heading	383 m	536 m	663 m

Mine development rates for the first three months of each mine development crew were de-rated to 75% of the full rate to reflect initial development crew inefficiencies.

The manual scheduling of mine development allowed recognition the number of available headings in each forthcoming scheduling period and tailor the mine development advance rate to development heading availability.

Average stope productivities were determined for the individual 28 m long stopes using a first-principles approach that recognised the mechanics of stope drilling and blasting activities, average LHD tramming distances, backfilling method, and various delays between stope activities. The resultant average stope productivity rates determined by this process are summarised in Table 16.25.

Table 16.25: Average Stope Productivity Rates on a Tonnes per Day Basis

Stoping Zone	Cemented Backfill Stopes	Uncemented Backfill Stopes
Stoping Zones A, B, D and E	145 t/d	205 t/d
Stoping Zone C	150 t/d	215 t/d

Two underground development crews are able to commence development simultaneously with one development crew commencing from the open pit floor and the other crew commencing main ramp development from the eastern portal. The 15-month open pit window before backfilling allows for 12 months of underground mine development from the open pit floor followed by an additional 3 months to install the mass concrete plug in the open pit ramp access.

The initial focus of the underground mine development schedule is for the two initial development crews to make connection (for access and ventilation purposes) prior to the loss of the open pit window, which the crews are comfortably able to achieve. Following the breakthrough, one development crew begins pre-production development of stoping zone B and the other crew focuses on stoping zone C. Additionally, the breakthrough of the two crews also allows for a third mine development crew to commence pre-production development to stoping zone A, thus expediting pre-production development for all three of the main stoping zones.

The steady state ROM production rate of 1,400 t/d is reached in 3.75 years following the commencement of underground mine development which is Q4 Year 5 of the overall combined open pit/underground mine project schedule. The Goliath underground mine development and production schedule is summarised by year in Table 16.26.

Table 16.26: Underground Mine Schedule Summary by Project Year

Item	Totals	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Lateral Waste Development (m)	22,324	3,637	4,927	5,582	3,981	2,451	1,380	366		
Lateral Waste Development (t)	1,536,659	275,287	332,573	376,785	268,718	165,443	93,150	24,705		
In-Vein Waste Development (m)	2,669		434	299	332	179	784	517	124	
In-Vein Waste Development (t)	129,713		21,092	14,531	16,135	8,699	38,102	25,126	6,026	
Stope Development (m)	9,961		1,654	1,688	2,009	1,111	1,723	1,424	352	
Stope Development (t)	514,761		84,564	86,318	104,245	56,848	92,547	72,423	17,817	
Block Value ($/t)	161		175	174	173	154	136	148	162	
Gold (g/t)	2.9		3.2	3.2	3.2	2.8	2.5	2.7	3.0	
Silver (g/t)	8.3		9.7	8.9	6.6	7.0	11.2	6.2	7.5	
Stope (t)	2,450,588		96,296	363,456	406,756	454,153	418,453	428,452	265,375	17,647
Block Value ($/t)	208		188	204	244	229	191	184	196	267
Gold (g/t)	3.8		3.4	3.7	4.5	4.2	3.5	3.4	3.6	4.9
Silver (g/t)	9.2		12.4	10.6	10.0	9.6	9.1	7.8	6.6	8.4
Total Mill Feed (t)	2,965,349		180,860	449,774	511,001	511,001	511,000	500,875	283,192	17,647
Block Value ($/t)	200		182	198	230	220	181	179	193	267
Gold (g/t)	3.7		3.3	3.6	4.2	4.0	3.3	3.3	3.6	4.9
Silver (g/t)	9.0		11.1	10.3	9.3	9.3	9.5	7.6	6.6	8.4
Tonnes per Day			661	1,232	1,400	1,400	1,400	1,372	776	446

16.12.7 Backfill

For the purposes of this study, it was assumed that completed stopes would be filled by rockfill from underground development or open pit waste. An average of 6% cement addition to sized rockfill product was confined to higher grade areas to eliminate the need for rib pillars. Current waste development arisings and uncemented surface sized product will be utilised in lower grade areas.

It was assumed that rockfill will be backhauled by the 45 tonne production trucks to a bay adjacent to the access ramp, from where it will be transported and tipped into the stope using a 10 tonne LHD. Measured cement quantities will be added to the rockfill when needed using a transportable high-shear colloidal mixer. It was assumed that two such units will be required and that these can be dismantled and transported from location to location within the mine as needed. Such a unit is illustrated in Figure 16-32.

Figure 16-32: Illustration of a High-Shear Colloidal Cement Mixer

Source: AGP, 2021.

It was assumed that once stopes are available for uncemented rockfilling, 50% of the waste development arisings would be trucked directly to the rockfill bay for transfer to the stope. The remainder of the development waste would be trucked to surface. Waste material for remaining uncemented rockfilling and for cemented rockfilling would be crushed and screened on surface to a 75 mm product size with controlled fines content. A cost allowance for this work was included in cost estimation. Production trucks returning underground will be loaded using a surface loader and diverted to the rockfill bay in use, prior to proceeding to its mineral loading level. Incremental trucking cost for this diversion and reduced truck speeds hauling down-ramp were included in the estimates.

16.12.8 Ventilation

The Goliath mine will be designed as a pull-type ventilation system mine. The return air raise (RAR) system will be designed as a two-fan parallel arrangement; each fan will be rated at ~400 kW, exhausting up to 125 m^3/s each. These fans will be equipped with variable speed drives to regulate the flow as required, based on production rates and diesel equipment requirements throughout the life of mine.

A utilisation factor has been applied to each piece of diesel equipment to estimate the total volume requirement. The total airflow requirements will be ~250 m^3/s, as shown in Table 16.27.

Table 16.27: Total Airflow Requirements

Equipment Type	Number	kW	Utilisation	m^3/s	Total m^3/s
6.7 t Scoop	1	150	80%	7.2	7.2
10 t Scoop	5	235	80%	11.3	56.4
45 t Diesel Truck	5	450	100%	27.0	135.0
Two-Boom Development Jumbo	3	110	10%	0.7	2.0
Rockbolter	3	93	25%	1.4	5.0
Boom Truck	1	111	25%	1.7	1.7
Fuel/Lube	1	111	25%	1.7	1.7
Shotcrete	1	111	25%	1.7	1.7
Eight-Person Carrier	1	60	25%	0.9	0.9
Scissors	3	111	25%	1.7	5.0
Transmixer	5	111	25%	1.7	8.3
Emulsion Loader	1	111	25%	1.7	1.7
Grader	4	118	25%	1.8	7.1
Toyota Runaround	1	96	25%	1.4	1.4
Mechanics Runaround	8	96	25%	1.4	11.5
Rescue/First Aid	1	96	25%	1.4	1.4
Telehandler	1	75	25%	1.1	1.1
Sanitation	1	69	25%	1.0	1.0
				Total Airflow Requirements (m^3/s)	**250**

This airflow requirement is based upon the *Ontario Health and Safety Act*, Regulation 854 Section 183.1 (3) "The flow of air must be at least 0.06 cubic metres per second for each kilowatt of power of the diesel-powered equipment operating in the workplace".

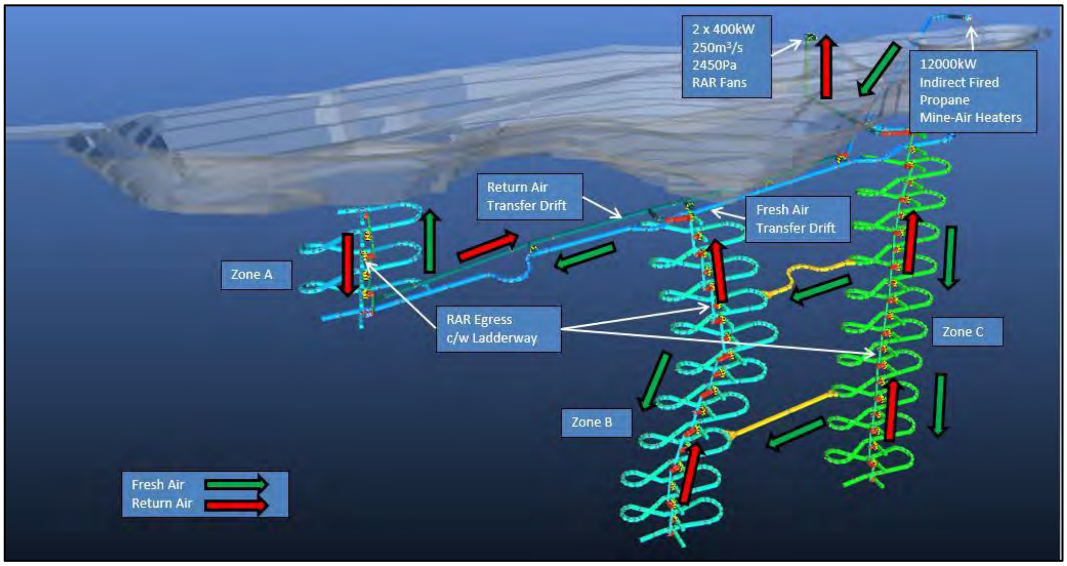
To provide redundancy in the event of a fan break-down, a single fan can supply up to 60% of the total required airflow (150 m^3/s). The ladder-equipped secondary mine-egress will be located within the mine RAR system.

In the final ventilation design (Figure 16-33), heated fresh air will enter the mine though the portal equipped with a propane heating system. A small fan integrated with the heaters will provide air over the burners for combustion purposes. This fresh air heater system will heat the air to +2° C. In addition, a stench warning system will be located at the portal intake.

The fan requirements for LOM are shown in Table 16.28.

Airflow will be matched to the projected mine production and development tonnages as the mine development progresses through the A, B, and C Zones. Figure 16-34 indicates airflow versus production tonnages over the life of mine.

Figure 16-33: Overall Mine Ventilation Schematic

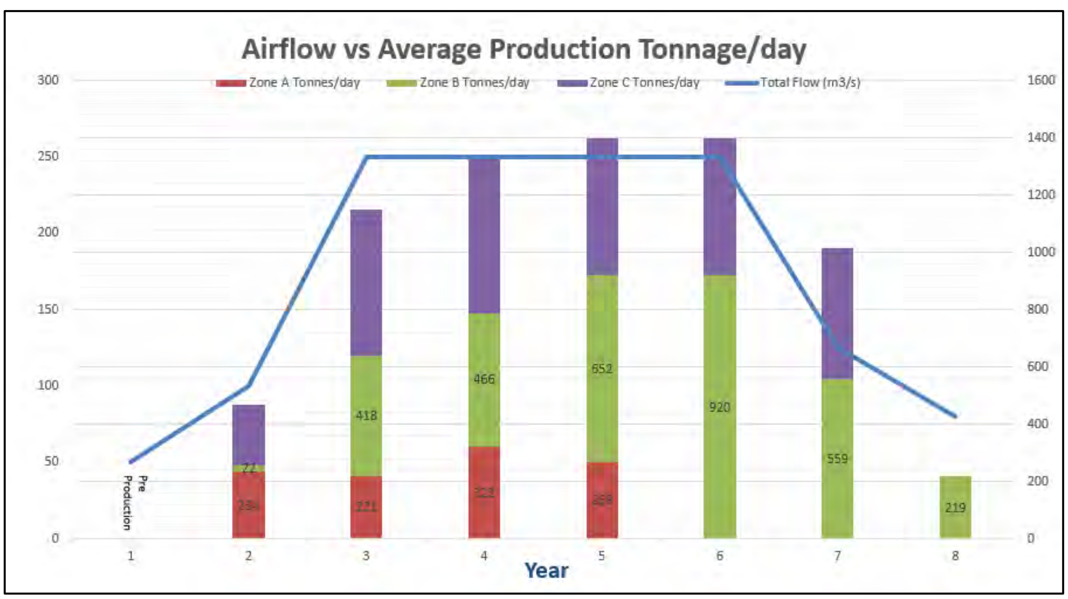


Source: AGP (2021).

Table 16.28: Overall Fan Requirements

Fan Purpose	Number	Power per Fan (kW)	Flow per Fan (m^3/s)	Pressure (Pa)
Main RAR	2	400	125	2450
Heater (for combustion)	1	70	100	
Decline Development	6	100	45	
Level/Production	12	50	30	

Figure 16-34: Airflow vs. Production Tonnage



Note: Year 1 equates to Year 3 of the consolidated open pit and underground production plan. Source: AGP (2021).

16.12.9 Decline Development

Initial decline development will be reliant on auxiliary ventilation until the first ventilation loop is established and the RAR is in place. This auxiliary ventilation will be provided by 100 kW fans supplying up to 45 m^3/s of air to the face through ventilation ducting. A temporary mine-air heating system will be installed at the portal during this period until the permanent heating system is constructed.

Once the RAR has been completed, one of the permanent RAR fans will be commissioned, exhausting up to 125 m^3/s of air out the mine, providing fresh air into the mine through the portal. One side of the surface mine-air heating plant will be constructed in this same period, running at half-capacity of ~6,000 kW.

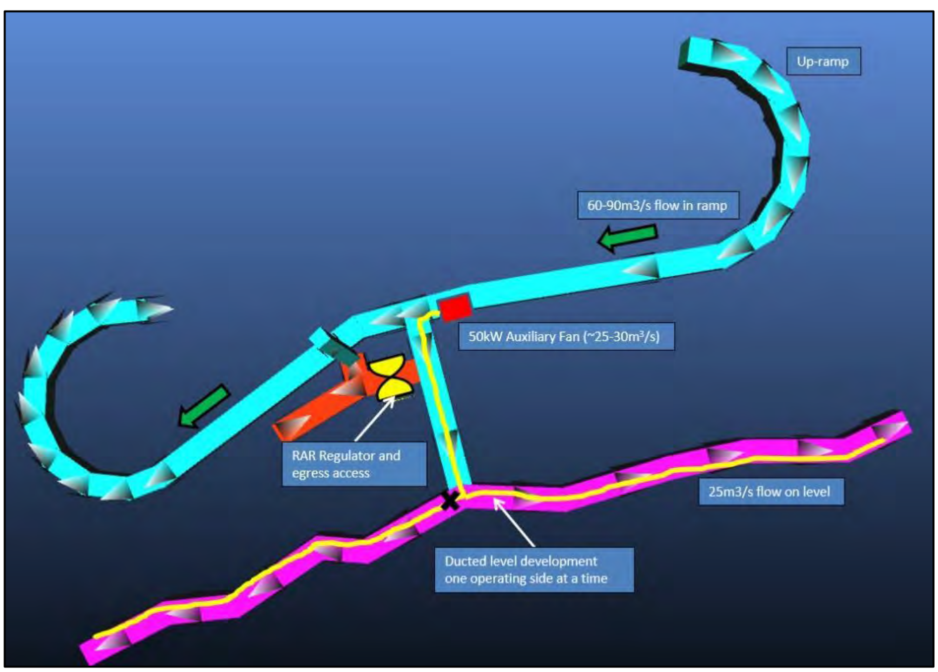
The development will progress across the three zones from C to A. After three years the final ventilation system will be constructed and regulated over the LOM based on production and equipment requirements, providing the full 250 m^3/s of fresh air and 12,000 kW of heated air.

This decline development will continue in a leap-frog manner extending the ventilation loop downward and across the mine as internal RAR raises are driven. This decline methodology will be similar for all zones with reliance on auxiliary ventilation until additional ventilation loops are completed.

16.12.10 Production & Level Development

The production level development will use ~50 kW fans suppling up to 30 m^3/s to each level as shown in Figure 16-35. Regulators located in the level access (LA) will regulate the flow of air from the ramp providing the required quantity of air to the truck loading area.

Figure 16-35: Typical Level Development



Source: AGP (2021).

16.12.11 Hydrogeology

No hydrogeological study regarding the Goliath underground deposit is yet available. It will be necessary to prevent as much water as possible from entering the open pit by means of diversion channels around the perimeter. Water entering the pit will be collected in suitably located sumps and pumped directly to surface, but it appears likely that surface water will enter the underground via geological structure and general seepage. Exploration drilling will be required in order to identify the locations of any underground aquifers.

In general, the hydrogeological regime at the site is unknown at this time. A suitable system of collection sumps and pumps will be planned to capture and discharge this water from the underground workings.

AGP's preliminary mine designs include the provision of five pump stations midway and at the bottom of each main production area. It was assumed that these pump stations will be commissioned as development and production extends downwards. Water entering the mining areas will need to be pumped up either the access ramp or ventilation raise.

Allowances were provided in the cost estimation, but definitive design of the system will be required in future studies once additional data are available.

16.12.12 Mine Safety

16.12.12.1 Fire Prevention

All diesel equipment (light vehicles and heavy-duty mobile equipment) will be equipped with automatic fire suppression systems and hand-held fire extinguishers. Hand-held fire extinguishers will be located throughout the mine at refuelling bays, workshops, explosive and detonator magazines, refuge bays, and lunchrooms. Refuelling bays, workshops, explosive and detonator magazines will be equipped with automatic deluge systems.

A mine-wide stench gas system will be installed at the fresh air intakes to alert underground workers in the event of an emergency.

16.12.12.2 Mine Rescue

A mine rescue team will consist of members selected and trained from the workforce and at least two teams would be available on rotation for rescue efforts. Surface and underground training facilities will be necessary for ongoing employee training and refresher training programs. Dedicated mine rescue equipment including a rescue vehicle and all supporting testing and maintenance equipment for mine rescue purposes will be available and specific underground mine rescue equipment would include self-contained breathing apparatus.

16.12.12.3 Refuge Stations

Lunchrooms in each main production area will automatically be designed and designated as a refuge station. Portable refuge chambers accommodating from 12 to 16 people are planned for the initial development period and for subsequent isolated areas. The refuge stations will be equipped with CO and CO_2 scrubbers, medical oxygen cylinders, oxygen candle, air conditioning, first aid kit, radio, telephone, and a toilet. They would provide protection for at least 36 hours as a self-contained unit. Access will be through an air lock door and they will be equipped with compressed air line cylinders and sealant to hermetically seal the doors in order to prevent the entry of harmful gases.

16.12.12.4 Emergency Egress

Independent emergency egress to surface from each sublevel is provided by ladders installed in the return air raises. In the event of fire underground preventing safe retreat through the return air raises the workforce will report to the refuge stations.

16.12.12.5 Dust Control

Spray nozzles operated by the mobile equipment drivers will be installed at all material loading points for dust control.

16.12.13 Cost Estimation

Costs for the development and production plan were estimated quarterly throughout the life of mine from first principles. All costs are reported in Q4 2020 Canadian Dollars.

A series of unit cost models was adapted to reflect the direct activities at the mine. Each of the models was developed reflecting the mine design criteria and other general engineering estimates of performance. The mine was assumed to work on two 12-hour shifts per day, 365 days per year. All activities will be undertaken by owners' crews apart from raising and delineation drilling which will be completed by a contractor. The cost models included ground support assumptions provided by the geotechnical study. The unit rates were applied to the scheduled quantities in order to estimate the direct costs.

Additional models were designed to reflect overhead-type activities at the mine:

- mine services and fixed plant (including labour, supplies, and equipment for construction, materials transport, road maintenance, and sanitation); diesel maintenance labour costs are also included
- owners mine management and technical (including mine supervision, mine technical and safety staff)
- mine air heating (based on local weather station data and estimated annual air flow requirements)
- mine power (developed from aggregation of mine loads and estimated usage)

Overheads were estimated on a quarterly basis and applied as a fixed daily cost. The overheads for each period were split between operating and capital development estimates in the ratio of the respective direct costs.

Consumable and material unit pricing for underground mining activities, were applied using data from other recent AGP mining projects. Where necessary pricing data was escalated from the date of information at the rate of 3% per annum to Q4 2020 values.

Details of the main material costs used in the analysis are provided in Table 16.29.

Table 16.29: Major Material & Consumables Cost Assumptions

Description	Unit	C$
Diesel	L	0.79
Power	kWh	0.08
Emulsion Explosive (Bulk)	kg	1.76
Trim Product	kg	3.02
NONEL LP detonator 5 m	ea.	3.54
NONEL MS detonator 18 m (60 ft)	ea.	12.46
1.5 m Rebar (complete)	ea.	20.94
1.8 m Rebar (complete)	ea.	22.96
2.4 m Rebar (complete)	ea.	25.63
Welded Mesh	m^2	8.37
6 m Cablebolt (complete)	ea.	58.45
9 m Cablebolt (complete)	ea.	63.02
Fibrecrete	m^3	253.45
45 mm Development Face Drillhole (Consumables)	m	1.52
33 mm Development Support Drillhole (Consumables)	m	0.96
51 mm Development Cablebolt Drilling	m	2.01
64 mm Stoping Long Hole (Consumables)	m	2.49
42″ Ventilation Wire Reinforced FlexiDuct Installed	m	128.13
36″ Ventilation FlexiDuct Installed	m	37.00
100 mm 4″ Pipes HDPE	m	55.96
150 mm 6″ Pipes HDPE	m	99.57
6″ x 10 ft Ultratech Schedule 80	m	129.48
48 Strand Fibre Optic Cable	m	8.71

Local labour rates were for the study were sourced from Canadian Mine Labour Survey for Ontario, Canada.

The models were also used to track labour and equipment hours to identify annual requirements in each labour category and equipment type.

A summary of the unit costs derived during the modelling process is shown in Table 16.30.

Table 16.30: Summary of Underground Unit & Overhead Costs

Model Description	Unit	C$
Ramp - 6.0 m wide x 6.0 m high	m	3,049
Ramp - 5.0 m wide x 5.0 m high	m	2,712
Level Waste Drift - 5.0 m wide x 5.0 m high	m	2,352
Vent/Other Drift - 5.0 m wide x 5.0 m high	m	2,123
Workshop/Pumps - 5.0 m wide x 4.5 m high	m	3,031
In Vein Waste/Low Grade - 4.0 m wide x 4.5 m high	m	1,870
Ore Drift - 4.0 m wide x 4.5 m high	m	3,094
Conventional Alimak Raise - 3.5 m wide x 3.5 m high with Ladder (Contractor)	m	7,294
Raise Bore (4.5 m) (Contractor)	m	10,667
Longhole Open Stope Drilling & Blasting - 5.0 m thick orebody	t	6.47
Scoop Mucking - LHD From Stope to Remuck Zone A	t	2.63
Scoop Mucking - LHD From Stope to Remuck Zone B/C	t	3.92
Ore/Waste Trucking to Surface		
50 m Vertical Haul	t	2.96
150 m Vertical Haul	t	4.89
250 m Vertical Haul	t	6.83
350 m Vertical Haul	t	7.84
450 m Vertical Haul	t	9.57
550 m Vertical Haul	t	11.30
650 m Vertical Haul	t	13.04
Rockfill Crush & Screen	t Fill	2.00
Incremental Rockfill Haul		
A Zone Cemented Fill	t Fill	14.96
150 m Uncemented Vertical Haul	t Fill	2.65
250 m Uncemented Vertical Haul	t Fill	3.17
350 m Uncemented Vertical Haul	t Fill	3.69
450 m Uncemented Vertical Haul	t Fill	4.22
550 m Uncemented Vertical Haul	t Fill	4.89
650 m Uncemented Vertical Haul	t Fill	5.41
Rockfill - LHD From Rockfill Storage to Stope Zone A	t Fill	1.02
Rockfill - LHD From Rockfill Storage to Stope Zone B/C	t Fill	2.40
Contract Diamond Drilling (Delineation)	Stope t	1.95
Mine Services, Fixed Plant & Mobile Equipment Maintenance Labour	Day	21,368
Owners Mine Supervision & Technical	Day	14,961
Mine Air Heating (Propane)	Day	3,752
Power	Day	6,269

Note: Overheads provided as typical daily averages. Modelled estimate overheads vary in each estimate period depending on mine activity.

16.12.14 Equipment

Equipment type and duty chosen for the evaluation are described in Table 16.31.

Table 16.31: Duty & Equipment Type

Duty	Equipment Type
Development Drilling	Two-boom Jumbo
Ground Support	Mechanised Bolter Shotcreter
Development & Production LHD	10 tonne LHD
Truck	45 tonne Diesel
Longhole Drilling (64 mm)	Longhole Jumbo
Explosives Loader	Emulsion Loader
Underground Utility Vehicles:	
Services & Construction	Scissors Lift
	Transmixer
	7 t Auxiliary LHD
Materials	Flatbed with Crane
	Telehandler
Road Maintenance	Grader
Equipment Operation	Fuel/Lube Truck
Personnel Transport	Eight-Person Carrier
Mobile Maintenance	Fitters Vehicle
Mobile Supervision	Runaround
Sanitation	Sanitation Vehicle
Ambulance/Rescue	Underground Ambulance/Rescue

Modelled equipment requirements are based on operational hours. Recent quotations for other AGP projects were used for the equipment types selected. Mechanical availability and operational life were estimated by AGP for each equipment type and the hourly operating costs were assessed. A mid-life 50% rebuild was provided in order to achieve the indicated equipment life. Table 16.32 shows the equipment data used for modelling.

The mine will contain a significant diesel fleet, which poses a fire hazard. All vehicles will be fitted with on-board detection and suppression systems and in addition, a mine-wide fire detection system is recommended. The underground diesel workshop will be ventilated directly via the main return raise.

As the activities vary the equipment fleet requirements change. Table 16.33 shows the fleet requirements for example periods in the development and production program.

For the purpose of this study it was assumed that mobile equipment will be leased. The lease terms applied were 20% upfront capital purchase followed by a five-year lease on the remaining 80% balance at the rate of 1.9% per month. The cost of mid-life rebuilds was included in the capital estimate. No replacement purchases were necessary during the limited mine life.

Table 16.32: Equipment Costs & Operational Data

Description	Base Cost ($k)	Mechanical Availability	Useful Life (h)	Hourly Cost ($/h)
6.7 t Scoop	864	82.9%	28,000	80.65
10 t Scoop	1,066	82.9%	28,000	134.27
45 t Diesel Truck	1,455	82.9%	28,000	160.27
Two-boom Development Jumbo	1,397	77.9%	25,000	74.10
Longhole Drill	1,098	77.9%	25,000	61.97
Rockbolter	1,129	77.9%	25,000	64.66
Boom Truck	549	82.9%	25,000	40.44
Fuel/Lube	576	82.9%	25,000	41.39
Shotcrete	993	77.9%	25,000	59.31
Eight-Person Carrier	113	82.9%	15,000	19.94
Scissors	563	82.9%	25,000	44.83
Transmixer	728	82.9%	25,000	46.84
Emulsion Loader	629	82.9%	25,000	47.37
Grader	647	82.9%	25,000	44.98
Toyota Runaround	94	82.9%	15,000	26.58
Mechanics Runaround	200	82.9%	15,000	30.49
Rescue/First Aid	123	82.9%	15,000	23.38
Telehandler	281	82.9%	25,000	24.74
Sanitation	568	82.9%	25,000	34.87

Table 16.33: Modelled Equipment Requirements

Fleet Type	Y4 Q1	Y6 Q1	Y8 Q1	Y10 Q1
6.7 t Scoop		1	1	1
10 t Scoop	2	5	5	4
45 t Diesel Truck	2	5	5	4
Two-Boom Development Jumbo	2	3	2	1
Longhole Drill		2	2	2
Rockbolter	2	3	2	1
Boom Truck	1	1	1	1
Fuel/Lube	1	1	1	1
Shotcrete	1	1	1	1
Eight-Person Carrier	2	3	3	3
Scissors	3	4	3	2
Transmixer	1	1	1	1
Emulsion Loader	2	3	3	2
Grader		1	1	1
Toyota Runaround	6	8	8	6
Mechanics Runaround	1	1	1	1
Rescue/First Aid	1	1	1	1
Telehandler	1	1	1	1
Sanitation	1	1	1	1

16.12.15 Labour

Annual labour force plans were developed to support the life of mine plan and the activities scheduled to meet production objectives. The labour force tables provided reflect the owner's underground workforce required to support development and production activities.

Details of the basic unit labour rates for production and mine supervision/technical staff salaries used in the analysis are provided in Tables 16.34 and 16.35. The rates include bonus, benefits and burden and reflect employment cost to company. The rates exclude any camp costs and transport to and from site.

Hourly paid employees will workday and night shifts, each of 12 hours, on a rotating 7 days on and 7 days off schedule for 365 mine-operating days per year. Each hourly paid employee will work 2,147 h/a. To provide for continuous operations there will be two employees per position.

The estimate of effective working hours in each shift is shown in Table 16.36.

Senior manager, supervisor, and training staff will work 2,000 hr/a. At an operational level there are two employees per position. For more senior and technical positions there will be one employee per position with work based on five 8-hour working days per week.

Job categories will be staffed one or two shifts per day basis depending on the position.

Table 16.34: Hourly Paid Labour Cost Assumptions

Job	Total $/h
Development Miner	89.54
Longhole Driller	83.75
Stope Blasting	77.40
Scoop Driver	67.43
Construction	60.46
Truck Driver	60.06
Materials/Pumps	59.38
Labourer	48.73
Mechanic/Diesel/Electrician I	73.24
Mechanic/Diesel/Electrician II	65.08

Table 16.35: Owner Staff Cost Assumptions

Position	Total ($/year)
Mine Superintendent	209,500
Mine Captain	170,300
Shift Boss	137,550
Mine Dry/Lamps/Bits	78,600
Safety	137,550
Secretary/Clerk/Stores	78,600
Senior Geologist	144,100
Mine Geologist	117,900
Geology Technician/Grade Control	91,700
Senior Mine Engineer	157,200
Mine Engineer	137,550
Mine Technician	91,700
Surveyor	91,700
Survey Helper	85,150
Ventilation / Samplers / Rock Mechanics Assistant	85,150
Maintenance Supt	196,500
Maintenance General Foreman	170,300
Maintenance Planner	124,450
Maintenance Foreman	137,550
Portal Attendant	91,700

Table 16.36: Hourly Paid Effective Working Hours

Factor	Unit	Value
Shift length	hours	12.0
Travel time	hours	1.00
Safety huddle	hours	0.25
Breaks	hours	0.75
Efficiency Factor (50 min-hour)	%	83.3
Effective Hours	hours per shift	8.3

Table 16.37 shows the estimate of personnel employed by the owner for selected periods during the development and production phases of the life-of-mine.

Table 16.37: Employed Labour

	Y4 Q1	Y6 Q1	Y8 Q1	Y10 Q1
Hourly Paid				
Longhole Drilling	-	5	5	4
Development Miner	17	19	13	5
Scoop Driver	7	17	16	12
Stope Blasting	-	3	3	3
Construction	7	7	7	5
Materials	4	4	4	4
Truck Driver	6	20	17	12
Labourer	13	19	15	8
Pumps	-	4	4	4
Mechanic I	2	2	2	2
Mechanic II	-	2	2	2
Electrician I	2	2	2	2
Electrician II	-	2	2	2
Diesel Mechanic I	6	9	8	6
Diesel Mechanic II	6	9	8	6
Diesel Mechanic III	6	9	8	6
Total Hourly Paid	76	133	116	83
Staff				
Maintenance Superintendent	-	0.5	0.5	-
Maintenance Foreman	1	3	3	2
Maintenance General Foreman	-	1	1	-
Maintenance Planner	-	0.5	0.5	-
Mine Superintendent	-	0.5	0.5	-
Mine Captain	1	1	1	1
Shift Boss	8	8	8	4
Mine Dry/Lamps/Bits	-	2	2	2
Secretary/Clerk/Stores	2	2	2	1
Safety	1	1	1	1
Senior Mine Engineer	1	1	1	1
Senior Geologist	1	1	1	1
Mine Geologist	1	2	2	1
Mine Technician	1	2	2	1
Geology Technician/Grade Control	1	2	2	1
Mine Engineer	1	2	2	1
Surveyor	2	2	2	2
Survey Helper	4	4	4	2
Portal Attendant	4	4	4	4
Ventilation / Samplers / Rock Mechanics Assistant	4	4	4	4
Total Staff	33	44	44	29
Total Employed Labour	109	177	160	112

16.12.16 Power

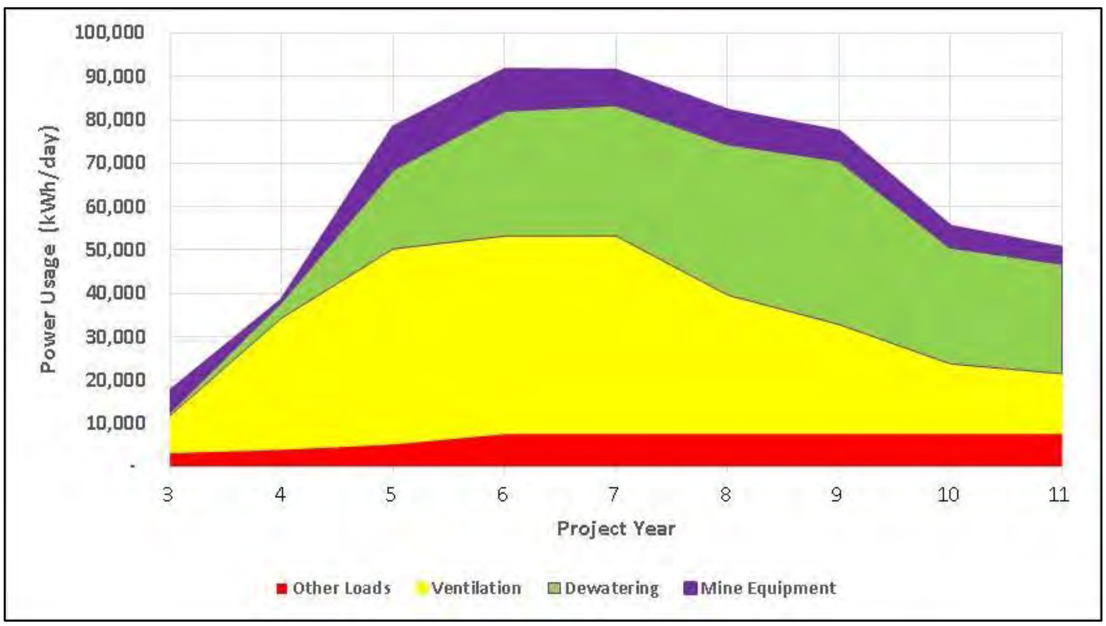
A load list was estimated for the underground mine, which is summarised in Table 16.38.

Table 16.38: Summary Total Mine Power Requirements

Load	Installed (kW)
Ventilation	2,070
Rockfill	150
Dewatering	2,299
Mine Equipment	931
Other Loads	631
Total	6,081

The load list was examined by period to estimate the power usage in the mine plan, as shown in Figure 16-36.

Figure 16-36: Power Usage



Source: AGP (2021).

16.13 PEA Mine Schedule

The mine production schedule consists of 24 Mt of mill feed grading 1.47 g/t gold and 1.82 g/t silver of the 13.5 year of mine life. The processing rate is 1.8 Mt per year. Open pit overburden and rock waste tonnage totals 82.5 Mt and will be placed the rock storage facilities. The overall pit strip ratio 3.9:1.

The mining production schedule includes one year of pre-stripping, nine years of mining, and 4.5 years of processing stockpiled material. The plant feed is composed of material from the

Goliath, Goldlund, and Miller open pits, and from the Goliath underground mine. In Year 9, tonnage in the pits is complete and the underground is complete in Year 11. Processing material will continue from the stockpiles until the middle of Year 14.

Mill feed is stockpiled during the pre-production year and throughout the production schedule as required. Three stockpiles located Goliath east of the main pit are used: high grade, medium grade, and low grade. A low-grade stockpile is used at Goldlund and Miller. All medium- and high-grade material is transported to the Goliath site as mined.

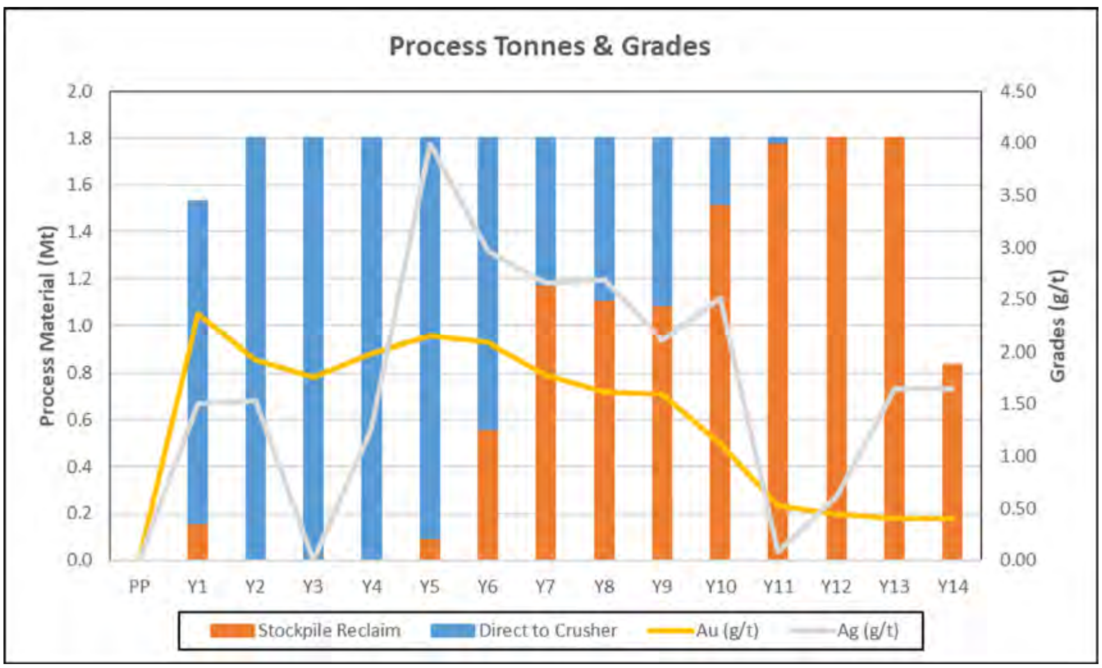
Refer to Table 16.39 for the stockpile diluted gold cut-off grades.

Table 16.39: Stockpile Cut-off Grades

Stockpile Grade	Goliath Au (g/t)	Goldlund Au (g/t)	Miller Au (g/t)
Low Grade	0.35	0.30	0.30
Medium Grade	0.60	0.60	0.60
High Grade	1.00	1.00	1.00

A peak stockpile capacity of 10.5 Mt is reached near the end of Year 6 (see Table 16.40 on the following page and Figure 16-37 below).

Figure 16-37: Process Tonnage & Gold Grade



Source: AGP (2021).

The annual mining rate starts at 6.5 Mt/a in the pre-production year and reaches at peak of 17.5 Mt/a in Years 2 and 3. In the open pits, a maximum descent rate of seven benches per year per phase is applied to ensure that reasonable mining operations and ore control will occur (see Table 16.41 and Figure 16-38).

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Table 16.40: Goliath Project Mine Schedule

		PP	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8
Mining Summary										
Open Pit Total	Waste tonnes	5,740,840	11,942,429	13,427,901	14,045,053	12,669,153	11,958,573	5,742,909	4,437,720	2,01
	Mill Feed tonnes	759,160	2,900,013	4,066,645	3,454,948	3,649,987	3,698,033	1,402,144	315,189	40
	Au (g/t)	0.92	1.40	1.17	1.24	1.24	0.95	0.90	1.06	
	Ag (g/t)	2.68	1.08	1.26	0.00	0.25	1.46	0.00	0.00	
	Total Tonnes	6,500,000	14,842,442	17,494,546	17,500,001	16,319,140	15,656,606	7,145,053	4,752,909	2,42
Underground Total	Waste tonnes	0	0	0	0	0	0	0	0	
	Mill Feed tonnes	0	0	0	0	180,860	449,774	511,001	511,001	51
	Au (g/t)	0.00	0.00	0.00	0.00	3.31	3.62	4.22	4.04	
	Ag (g/t)	0.00	0.00	0.00	0.00	11.14	10.27	9.32	9.34	
	Total Tonnes	0	0	0	0	180,860	449,774	511,001	511,001	51
Goliath - Open Pit	Waste tonnes	5,740,840	5,224,279	4,535,985	68	5,468,301	8,381,618	0	0	
	Mill Feed tonnes	759,160	1,111,189	1,580,532	0	367,323	2,280,764	0	0	
	Au (g/t)	0.92	0.92	1.06	0.00	0.74	0.93	0.00	0.00	
	Ag (g/t)	2.68	2.81	3.23	0.00	2.46	2.37	0.00	0.00	
	Total Tonnes	6,500,000	6,335,468	6,116,517	68	5,835,624	10,662,382	0	0	
Goliath - Underground	Waste tonnes	0	0	0	0	0	0	0	0	
	Mill Feed tonnes	0	0	0	0	180,860	449,774	511,001	511,001	51
	Au (g/t)	0.00	0.00	0.00	0.00	3.31	3.62	4.22	4.04	
	Ag (g/t)	0.00	0.00	0.00	0.00	11.14	10.27	9.32	9.34	
	Total Tonnes	0	0	0	0	180,860	449,774	511,001	511,001	51
Goldlund - Open Pit	Waste tonnes	0	6,718,150	8,891,916	14,044,985	7,200,852	3,576,955	916,306	0	
	Mill Feed tonnes	0	1,788,824	2,486,113	3,454,948	3,282,664	1,417,269	1,160,209	0	
	Au (g/t)	0.00	1.69	1.24	1.24	1.30	0.97	0.87	0.00	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Total Tonnes	0	8,506,974	11,378,029	17,499,933	10,483,516	4,994,224	2,076,515	0	
Miller - Open Pit	Waste tonnes	0	0	0	0	0	0	4,826,603	4,437,720	2,01
	Mill Feed tonnes	0	0	0	0	0	0	241,935	315,189	40
	Au (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	1.05	1.06	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	Total Tonnes	0	0	0	0	0	0	5,068,538	4,752,909	2,42

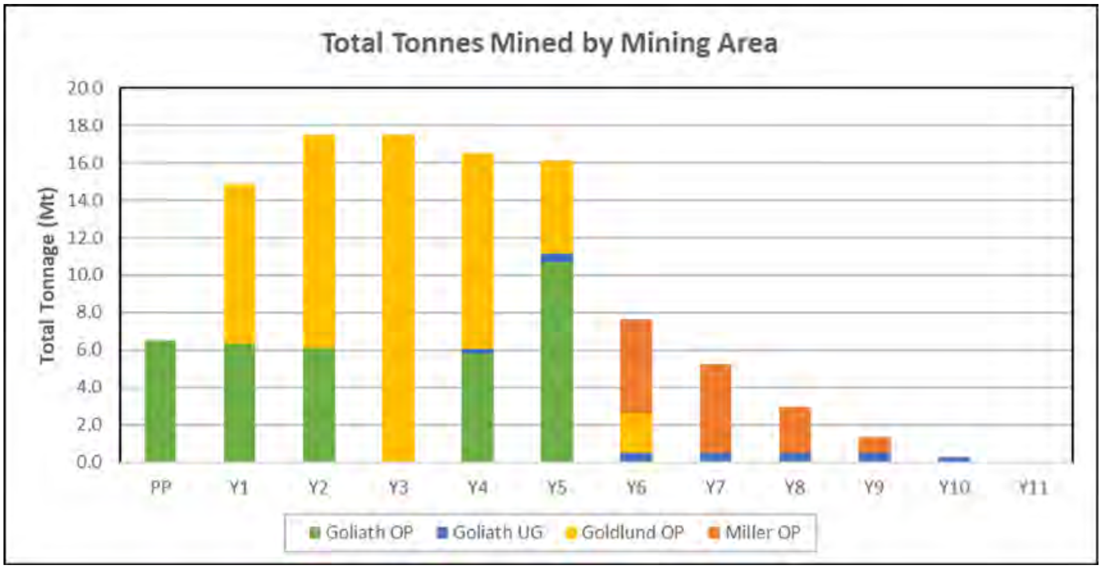
Processed Material										
Total	Mill Feed tonnes	0	1,530,000	1,800,000	1,800,000	1,800,001	1,800,000	1,800,001	1,800,001	1,80
	Au (g/t)	0.00	2.36	1.92	1.76	1.98	2.15	2.09	1.78	
	Ag (g/t)	0.00	1.51	1.54	0.00	1.27	3.99	2.96	2.65	
Goliath	Mill Feed tonnes	0	301,171	353,383	0	206,892	1,051,365	707,969	511,001	51
	Au (g/t)	0.00	3.15	2.92	0.00	3.23	2.79	3.41	4.04	
	Ag (g/t)	0.00	7.65	7.82	0.00	11.03	6.84	7.54	9.34	
Goldlund	Mill Feed tonnes	0	1,228,829	1,446,617	1,800,000	1,593,109	748,635	925,364	1,171,345	1,11
	Au (g/t)	0.00	2.16	1.68	1.76	1.82	1.24	1.22	0.78	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
Miller	Mill Feed tonnes	0	0	0	0	0	0	166,668	117,655	17
	Au (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	1.32	1.91	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
Stockpile Balance	LG tonnes	403,381	1,053,741	1,880,449	1,880,449	2,103,826	3,306,066	3,306,066	3,306,066	3,30
Goliath	Au (g/t)	0.37	0.38	0.40	0.40	0.40	0.40	0.40	0.40	
	Ag (g/t)	1.58	1.65	1.70	1.70	1.69	1.65	1.65	1.65	
	MG tonnes	198,220	447,380	823,447	823,447	871,592	1,313,757	1,313,757	1,313,757	1,31
	Au (g/t)	0.63	0.64	0.66	0.66	0.65	0.65	0.65	0.65	
	Ag (g/t)	2.08	2.06	2.09	2.09	2.06	2.01	2.01	2.01	
	HG tonnes	157,559	68,057	92,431	92,431	162,200	196,968	0	0	
	Au (g/t)	2.70	1.22	1.26	1.26	1.30	1.30	0.00	0.00	
	Ag (g/t)	6.24	2.77	2.90	2.90	2.97	2.91	0.00	0.00	
Goldlund	LG tonnes	0	220,777	711,647	1,358,076	1,917,829	2,198,166	2,445,202	2,445,202	2,44
	Au (g/t)	0.00	0.47	0.48	0.48	0.48	0.48	0.48	0.48	
	Ag (g/t)	0	0	0	0	0	0	0	0	
	MG tonnes	0	333,764	886,951	1,843,755	2,678,459	3,055,761	3,402,271	2,230,926	1,12
	Au (g/t)	0.00	0.79	0.78	0.78	0.78	0.78	0.78	0.78	
	Ag (g/t)	0	0	0	0	0	0	0	0	
	HG tonnes	0	5,454	893	52,608	347,706	358,701	0	0	
	Au (g/t)	0.00	1.12	1.05	1.16	1.41	1.42	0.00	0.00	
	Ag (g/t)	0	0	0	0	0	0	0	0	
Miller	LG tonnes	0	0	0	0	0	0	74,114	214,836	37
	Au (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.46	0.46	
	Ag (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
	MG tonnes	0	0	0	0	0	0	1,153	57,965	12
	Au (g/t)	0.00	0.00	0.00	0.00	0.00	0.00	0.72	0.76	

Total Stockpile Reclaim										
Total	tonnes	0	157,559	5,454	0	0	93,621	560,852	1,171,345	1,11
Goliath	tonnes	0	157,559	0	0	0	0	196,968	0	
Goldlund	tonnes	0	0	5,454	0	0	93,621	363,884	1,171,345	1,11
Miller	tonnes	0	0	0	0	0	0	0	0	
Total Stockpile Balance										
Total	tonnes	759,160	2,129,173	4,395,818	6,050,766	8,081,612	10,429,419	10,542,563	9,568,752	8,68
Goliath	tonnes	759,160	1,569,178	2,796,327	2,796,327	3,137,618	4,816,791	4,619,823	4,619,823	4,61
Goldlund	tonnes	0	559,995	1,599,491	3,254,439	4,943,994	5,612,628	5,847,473	4,676,128	3,56
Miller	tonnes	0	0	0	0	0	0	75,267	272,801	50
Total Material Movement										
Total	tonnes	6,500,000	15,000,001	17,500,000	17,500,001	16,319,140	15,750,227	7,705,905	5,924,254	3,53
Goliath	tonnes	6,500,000	6,493,027	6,116,517	68	5,835,624	10,662,382	196,968	0	
Goldlund	tonnes	0	8,506,974	11,383,483	17,499,933	10,483,516	5,087,845	2,440,399	1,171,345	1,11
Miller	tonnes	0	0	0	0	0	0	5,068,538	4,752,909	2,42

Table 16.41: Total Tonnes Mined by Deposit

Mining Area	Total Tonnage (Mt)						
	PP	Y1	Y2	Y3	Y4	Y5	Y6
Goliath Open Pit	6.5	6.3	6.1	-	5.8	10.7	-
Goliath Underground	-	-	-	-	0.2	0.4	0.5
Goldlund Open Pit	-	8.5	11.4	17.5	10.5	5.0	2.1
Miller Open Pit	-	-	-	-	-	-	5.1
Total	**6.5**	**14.8**	**17.5**	**17.5**	**16.5**	**16.1**	**7.7**

Figure 16-38: Total Tonnes Mined by Deposit



Source: AGP (2021).

Table 16.42 displays a summary of the resource classification for the mill feed.

Table 16.42: Resource Classification in Mill Feed

Area	Resource Class	Mill Feed	Grade		Contained Ounces	
		(kt)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
Goliath	Measured	804	1.96	5.75	50.6	148.2
	Indicated	5,194	0.82	2.34	138.2	390.2
	Inferred	101	0.36	1.63	1.2	5.1
	Total	**6,099**	**0.97**	**2.77**	**190.0**	**543.5**
Goldlund	Measured	-	-	-	-	-
	Indicated	11,818	1.35	-	511.8	-
	Inferred	1,772	0.64	-	36.1	-
	Total	**13,590**	**1.25**	**-**	**547.9**	**-**
Miller	Measured	-	-	-	-	-
	Indicated	-	-	-	-	-
	Inferred	1,312	1.16	-	48.9	-
	Total	**1,312**	**1.16**	**-**	**48.9**	**-**
Goliath UG	Measured	271	3.66	9.09	32.0	79.3
	Indicated	2,381	3.71	9.33	284.3	713.8
	Inferred	313	3.31	6.88	33.3	69.2
	Total	**2,965**	**3.67**	**9.05**	**349.6**	**862.3**

16.14 Mine Plan Sequence – Open Pit & Underground

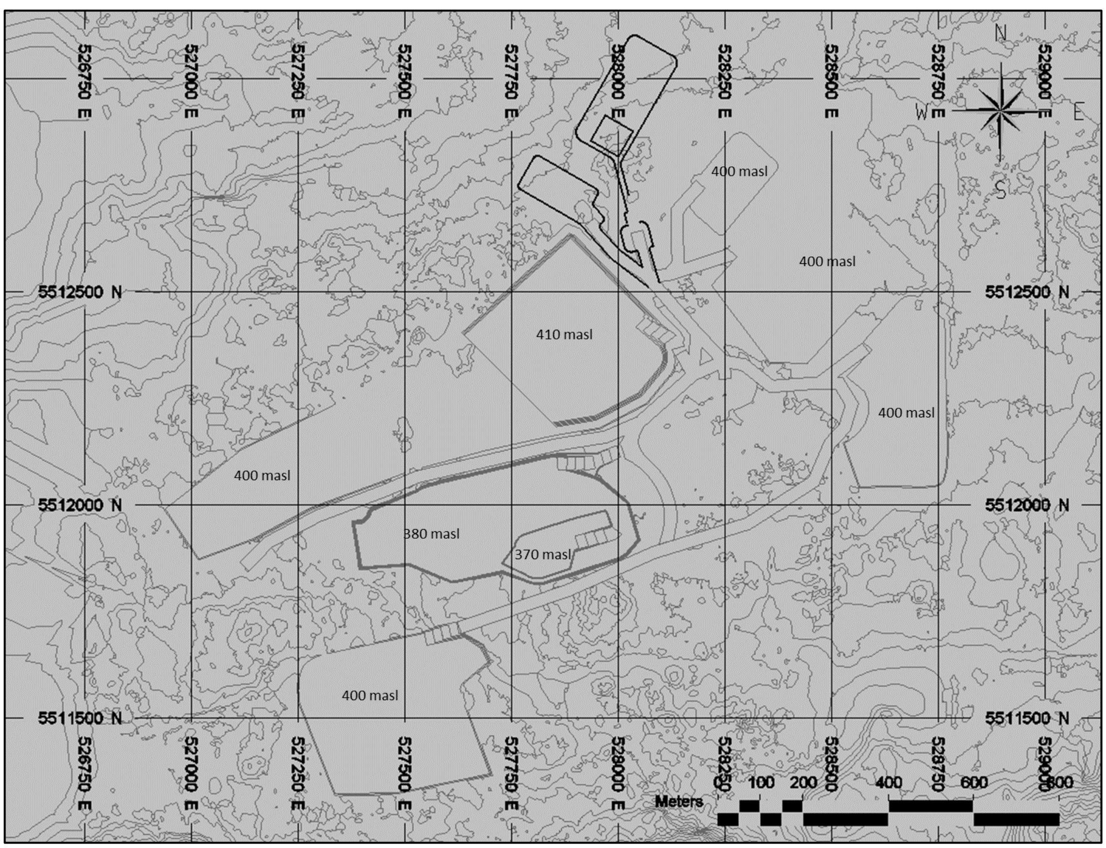
The annual periods are displayed from end of pre-production to the end of processing in Figures 16-39 to 16-73 in this section, along with a short description.

Progression of PEA underground development and stoping by year end is also shown. The ultimate open pit is shown for reference. Figures are based on the mine development, stopes, and production schedule. Stoping levels are shown in the year that stoping commences on the level.

16.14.1 Pre-Production

Site construction is undertaken at Goliath which includes access roads, surface haul roads, and tailings dam. Clearing and soil removal is done to prepare the bases of the rock storage facilities and mill feed stockpiles. Pre-stripping is initiated at Goliath phases 1A and 1B with overburden being stored in an overburden stockpile. 5.74 Mt of waste is mined and 0.76 Mt of mill feed grading 0.94 g/t Au, and 2.73 g/t Ag is stockpiled, as the processing plant is not yet operational. Phase 1A is nearly depleted to the 360 masl elevation and Phase 1B is mined down to the 370 masl elevation. The overburden stockpile reaches a lift elevation of 400 masl elevation, and the rock storage facility reaches a lift elevation of 410 masl (see Figure 16-39).

Figure 16-39: End of Pre-Production – Goliath

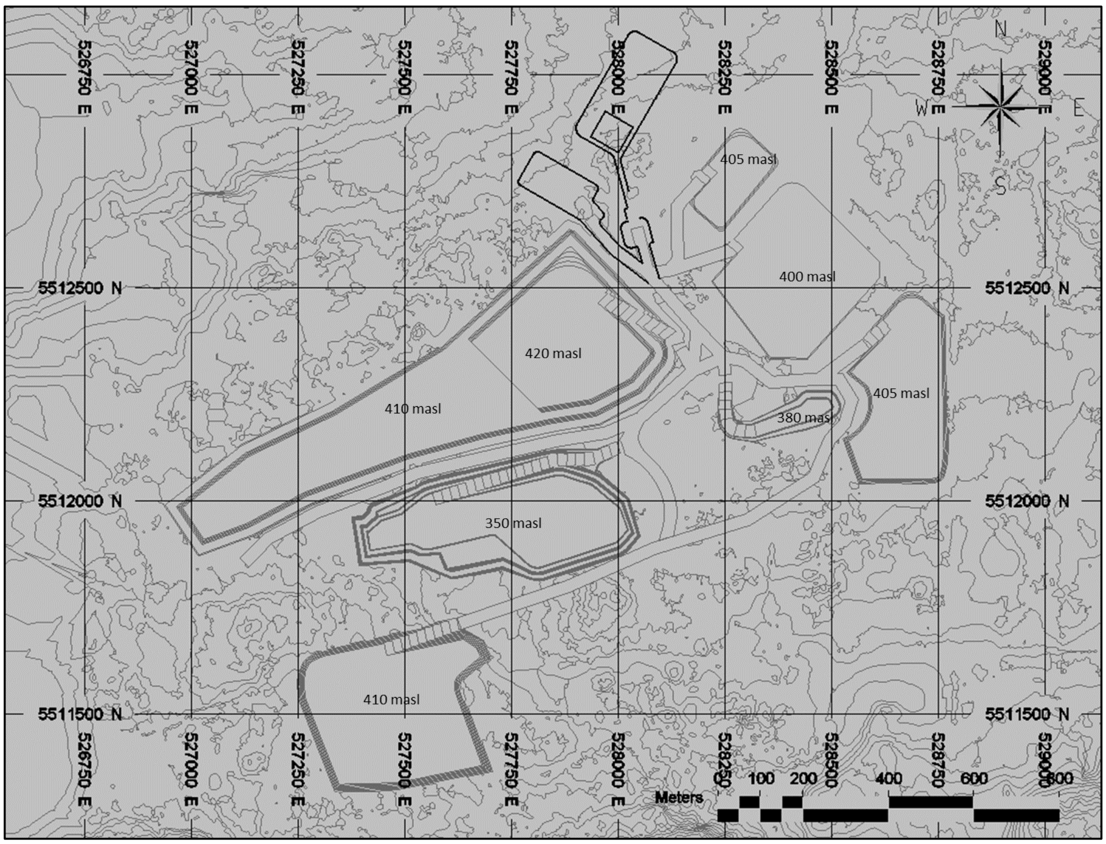


Source: AGP (2021).

16.14.2 Year 1

Mining continues at Goliath to deplete Phase 1A and to mine Phase 1B to the 340 masl elevation (Figure 16-40). The external phase 4 pit is initiated and is mined to the 380 masl elevation. Mined material amounted to 5.22 Mt of waste rock, and 1.11 Mt of mill feed grading 0.94 g/t Au and 2.86 g/t Ag. Processed material amounted to 0.30 Mt grading 3.22 g/t Au and 7.80 g/t Ag. Material stockpiled balance increased to 1.57 Mt. Overburden stockpile reaches a lift elevation of 410 masl. Rock storage facility reaches a lift elevation of 420 masl.

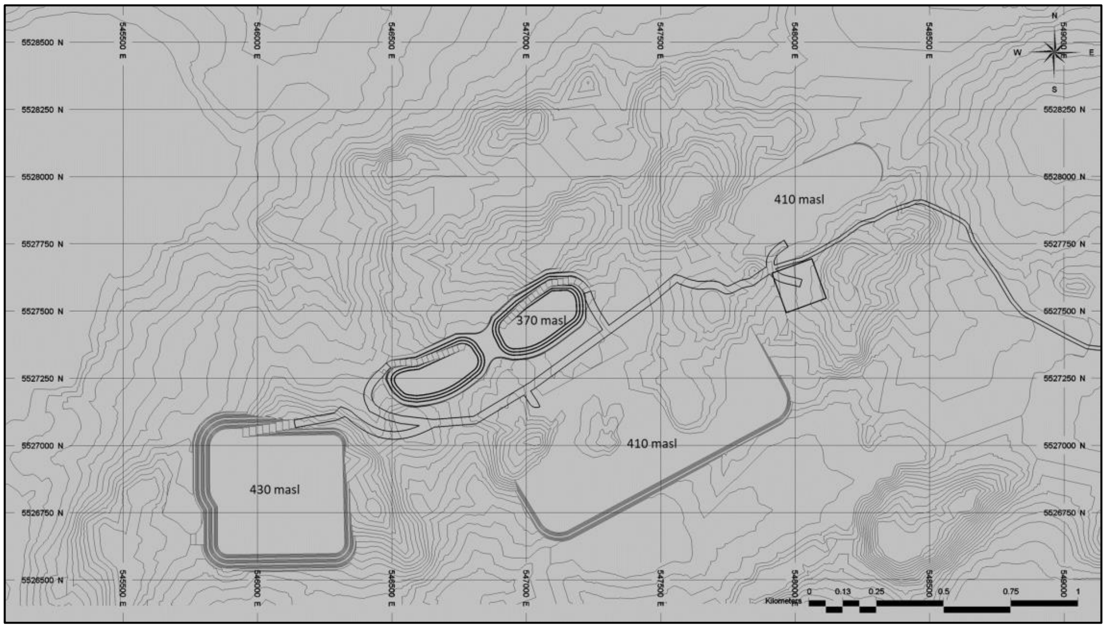
Figure 16-40: End of Year 1 – Goliath



Source: AGP (2021).

Mining is initiated at the Goldlund site in Phase 1 of the main pit with overburden being stored in an overburden stockpile (Figure 16-41). Phase 1 is mined down to the 370 masl elevation. Mined material amounted to 6.72 Mt of waste rock, and 1.79 Mt of mill feed grading 1.69 g/t Au. Processed material amounted to 1.23 Mt grading 2.16 g/t Au. Material stockpiled balance increased to 0.56 Mt. The overburden stockpile reaches a lift elevation of 430 masl and the rock storage facility reaches a lift elevation of 410 masl.

Figure 16-41: End of Year 1 – Goldlund



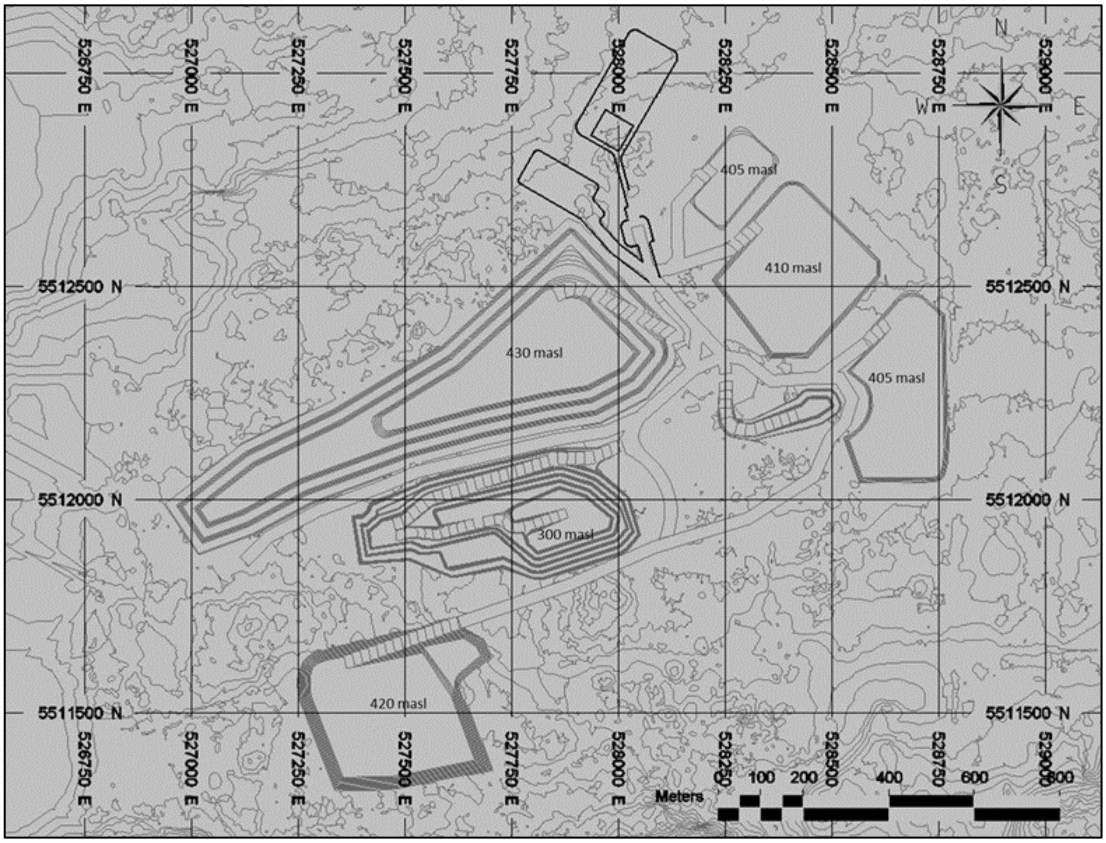
Source: AGP (2021).

16.14.3 Year 2

Mining (Phase 1B) continues at Goliath down to the 300 masl elevation before depletion (Figure 16-42). Phase 4 is mined down to the 370 masl elevation and is also depleted. Mined material amounted to 4.54 Mt of waste rock, and 1.58 Mt of mill feed grading 1.08 g/t Au and 3.30 g/t Ag. Processed material amounted to 0.35 Mt grading 2.98 g/t Au and 7.98 g/t Ag. Material stockpiled balance increased to 2.80 Mt. The overburden stockpile reaches a lift elevation of 420 masl and the rock storage facility reaches a lift elevation of 430 masl.

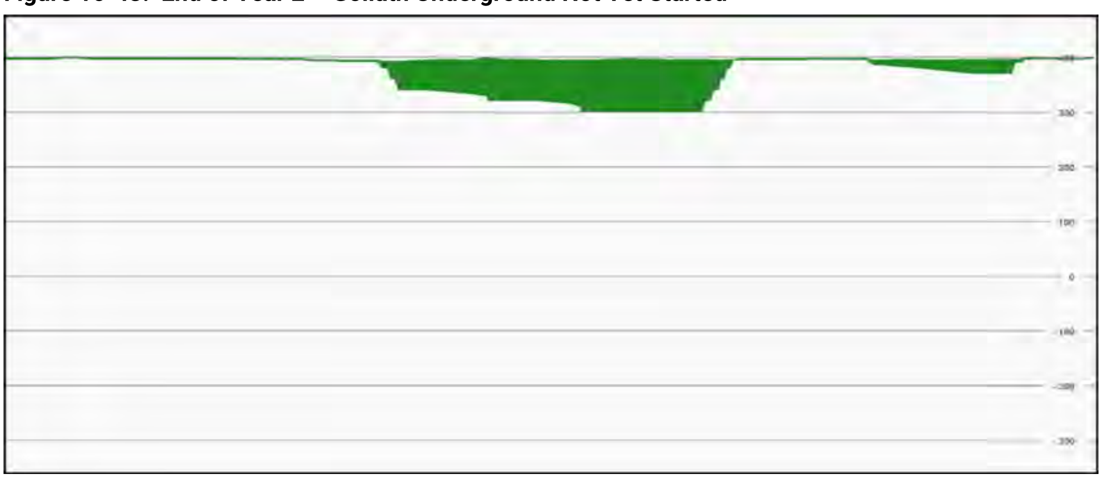
Mining continues at Goldlund with Phase 1 being mined down to the 340 masl elevation, and Phase 2 is initialised and mined to the 380 masl elevation (Figure 16-44). Mined material amounted to 8.89 Mt of waste rock, and 2.49 Mt of mill feed grading 1.24 g/t Au. Processed material amounted to 1.45 Mt grading 1.68 g/t Au. Material stockpiled balance increased to 1.60 Mt. Rock storage facility reaches a lift elevation of 420 masl.

Figure 16-42: End of Year 2 – Goliath



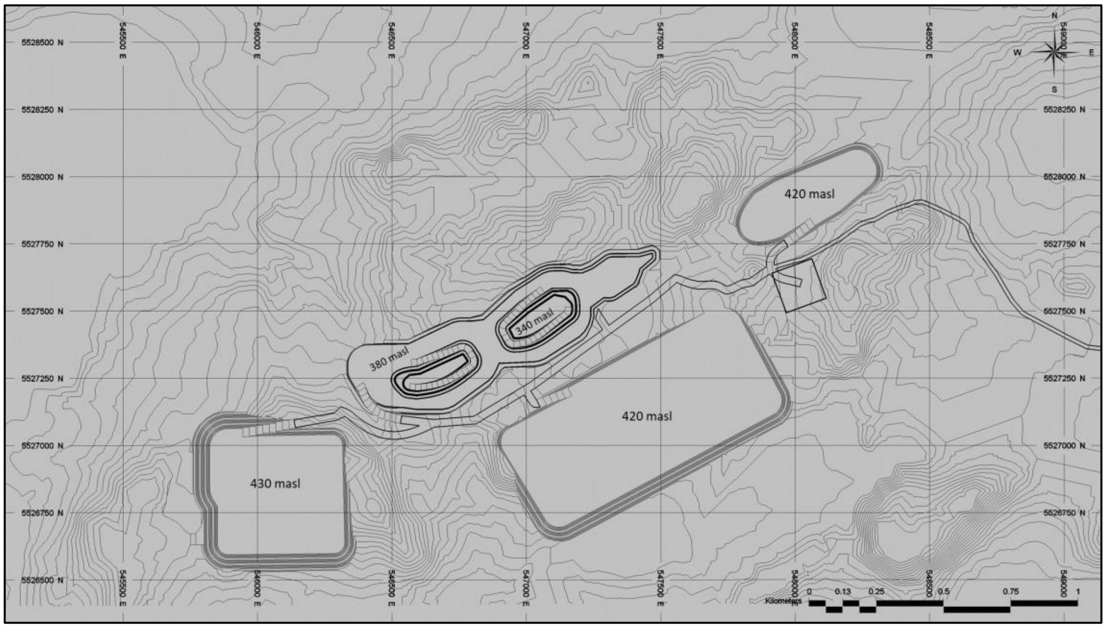
Source: AGP (2021).

Figure 16-43: End of Year 2 – Goliath Underground Not Yet Started



Source: AGP (2021).

Figure 16-44: End of Year 2 – Goldlund



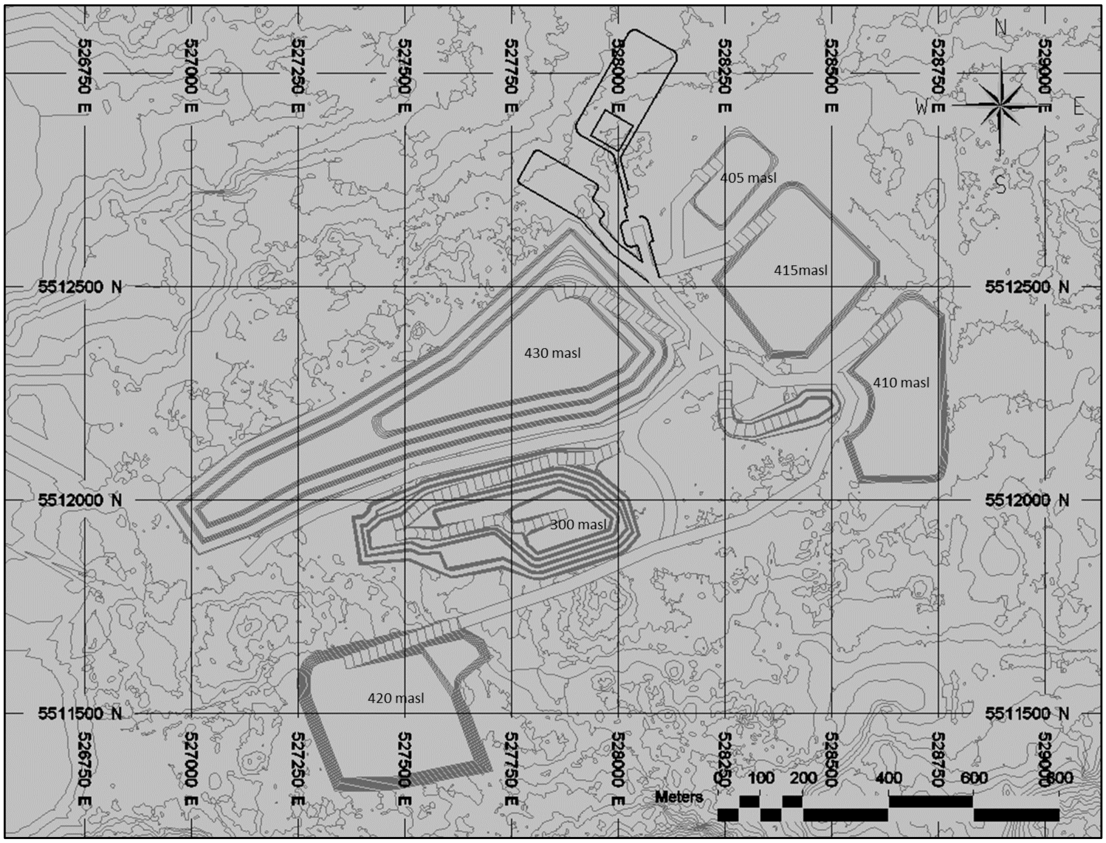
Source: AGP (2021).

16.14.4 Year 3

Mining pauses at Goliath for the year and all mining occurs at Goldlund. This allows the underground to portal in the bottom of Phase 1B and start a second face of development to advance the underground (Figures 16-45 and 16-46).

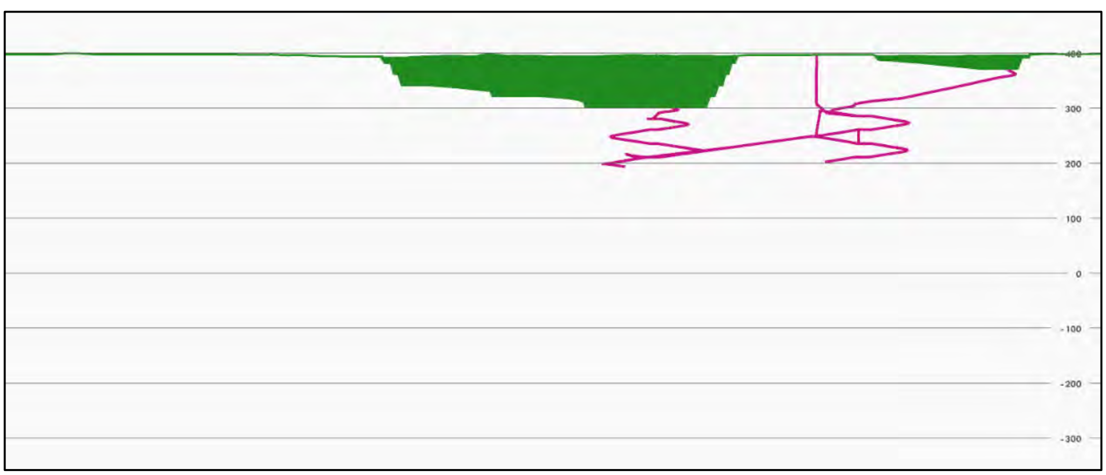
Mining continues at Goldlund with Phase 1 being depleted at the 340 masl elevation, and Phase 2 being mined to the 320 masl elevation (Figure 16-47). Mined material amounted to 14.04 Mt of waste rock, and 3.45 Mt of mill feed grading 1.24 g/t Au. Processed material amounted to 1.80 Mt grading 1.76 g/t Au. Material stockpiled balance increased to 3.25 Mt. Rock storage facility reaches a lift elevation of 440 masl.

Figure 16-45: End of Year 3 – Goliath



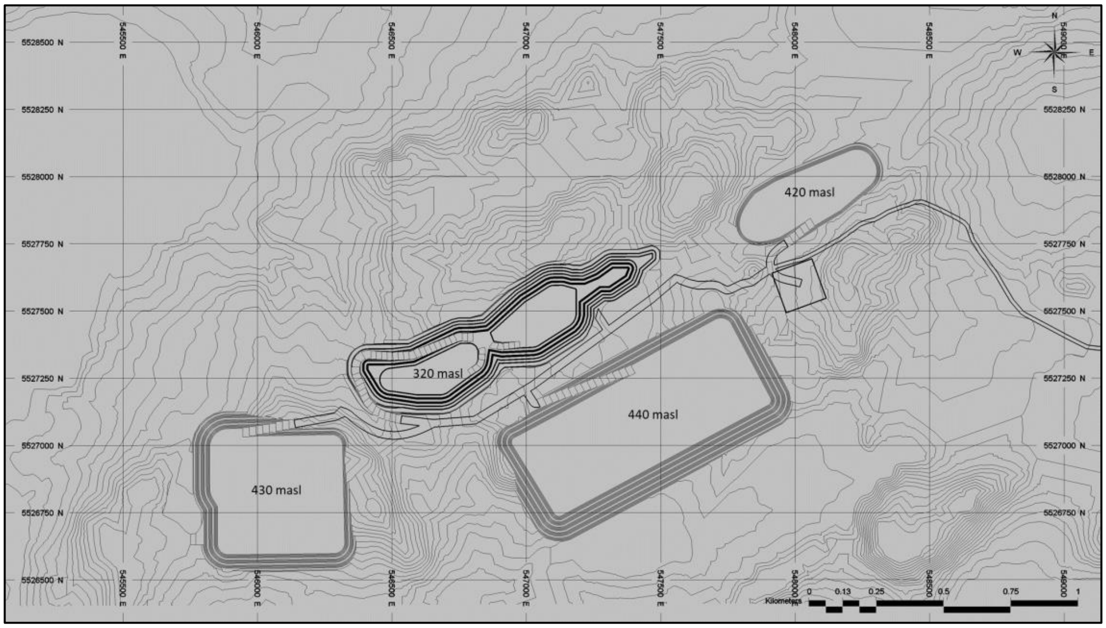
Source: AGP (2021).

Figure 16-46: End of Year 3 – Goliath Underground



Source: AGP (2021).

Figure 16-47: End of Year 3 – Goldlund



Source: AGP (2021).

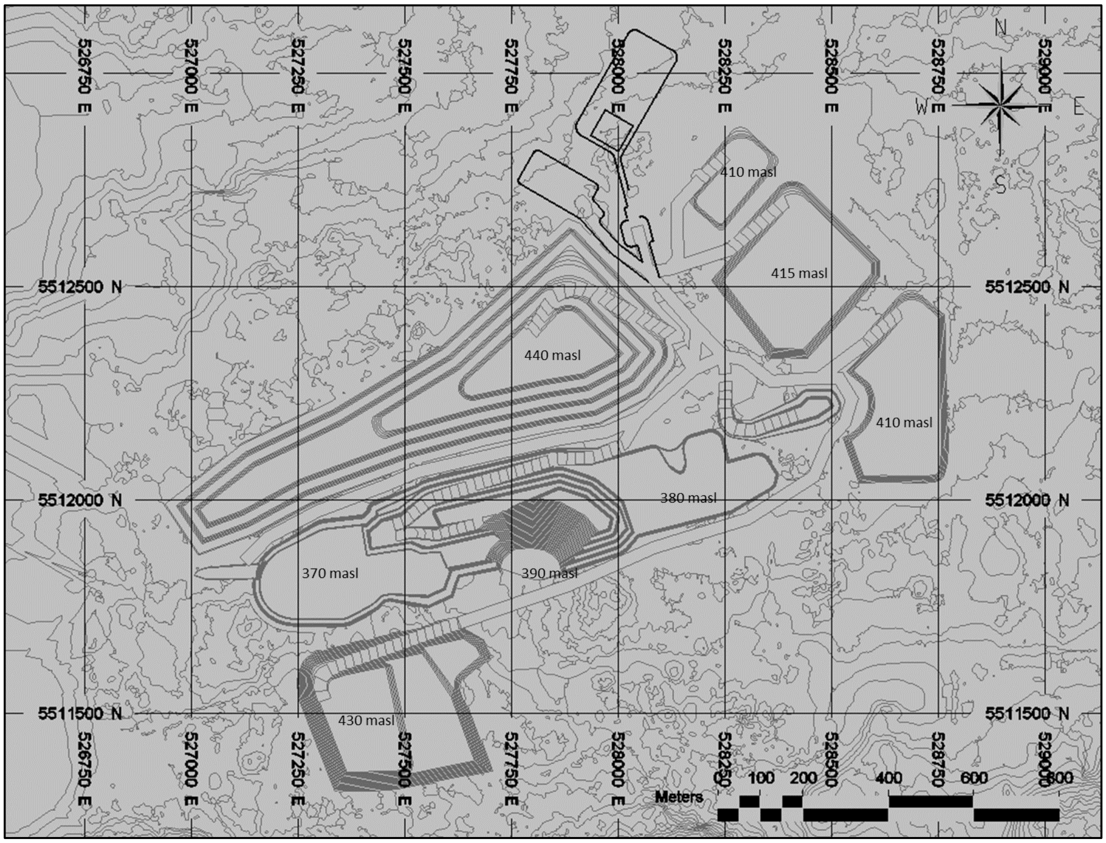
16.14.5 Year 4

Open pit mining continues at Goliath with Phase 2 being mined down to the 370 masl elevation, and Phase 3 being initiated and mined down to the 370 masl elevation (Figure 16-48). Mined material amounted to 5.47 Mt of waste rock, and 0.37 Mt of mill feed grading 0.76 g/t Au and 2.51 g/t Ag.

Underground mining is initiated (Figure 16-49). 0.18 Mt of mill feed is mined grading 3.31 g/t Au and 11.14 g/t Ag. Processed material amounted to 0.21 Mt grading 3.24 g/t Au and 11.06 g/t Ag. Material stockpiled balance increased to 3.14 Mt. Overburden stockpile reaches a lift elevation of 430 masl. Rock storage facility reaches a lift elevation of 440 masl. In pit dumping begins in Phase 1B up to the 390 masl lift elevation.

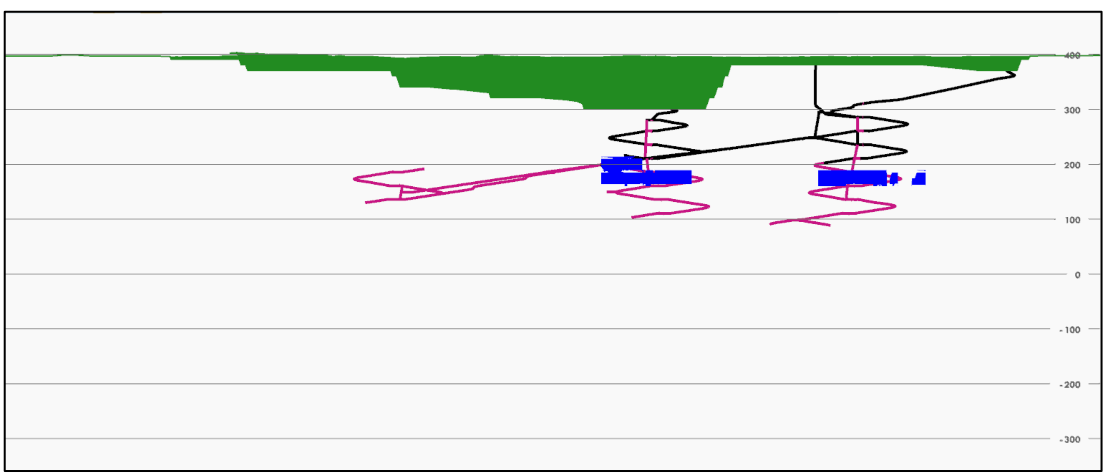
Mining continues at Goldlund with Phase 2 being mined to the 280 masl elevation, Phase 3 being initiated and mined to the 400 masl elevation, Phase 4 being initiated, mined to the 410 masl elevation, and Phase 6 being mined to the 350 masl elevation (Figure 16-50). Mined material amounted to 7.20 Mt of waste rock, and 3.28 Mt of mill feed grading 1.30 g/t Au. Processed material amounted to 1.59 Mt grading 1.82 g/t Au. Material stockpiled balance increased to 4.94 Mt. Rock storage facility reaches a lift elevation of 440 masl and the overburden stockpile reaches a lift elevation of 440 masl.

Figure 16-48: End of Year 4 – Goliath



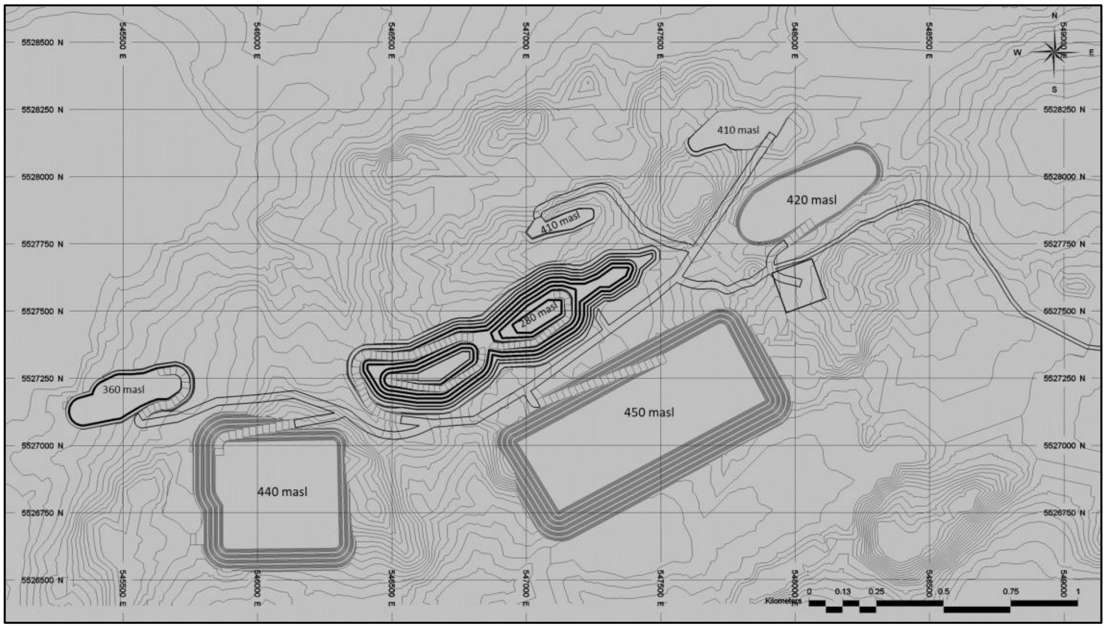
Source: AGP (2021).

Figure 16-49: End of Year 4 – Goliath Underground



Source: AGP (2021).

Figure 16-50: End of Year 4 – Goldlund



Source: AGP (2021).

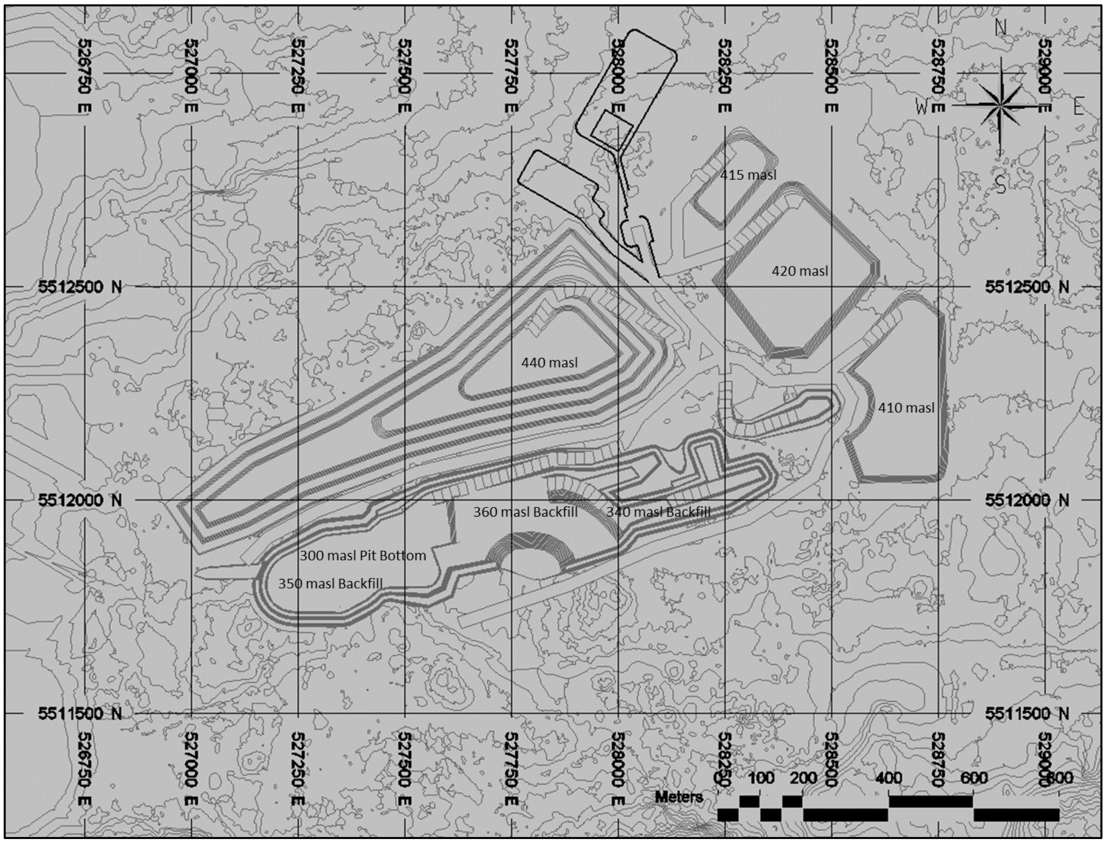
16.14.6 Year 5

Mining continues at Goliath with Phase 2 being depleted at the 300 masl elevation, and Phase 3 being depleted as well at the 340 masl elevation (Figure 16-51). The Goliath open pits are all depleted in this year. Mined material amounted to 8.38 Mt of waste rock, and 2.28 Mt of mill feed grading 0.95 g/t Au and 2.42 g/t Ag. In pit dumping occurs in Phases 1B and 2 up to the 360 masl and 350 masl elevations, respectively.

Underground mining continues at Goliath (Figure 16-52). Mined material amounted 0.45 Mt of mill feed grading 3.62 g/t Au and 10.27 g/t Ag. Processed material amounted to 1.05 Mt grading 2.82 g/t Au and 6.89 g/t Ag. Material stockpiled balance increased to 4.82 Mt.

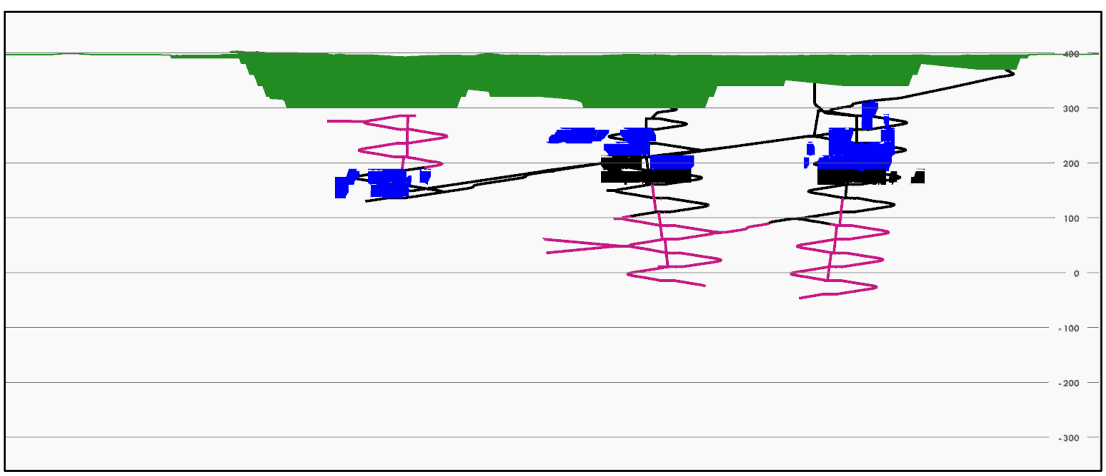
Mining continues at Goldlund with Phase 2 being depleted at the 280 masl elevation, Phase 3 depleted at the 400 masl elevation, Phase 4 depleted at the 360 masl elevation, and Phase 5 being initiated and mined down to the 370 masl elevation (Figure 16-53). Phase 6 is mined the 350 masl elevation. Mined material amounted to 3.58 Mt of waste rock, and 1.42 Mt of mill feed grading 0.97 g/t Au. Processed material amounted to 0.75 Mt grading 1.24 g/t Au. Material stockpiled balance increased to 5.61 Mt. Rock storage facility continues with a lift elevation of 450 masl.

Figure 16-51: End of Year 5 – Goliath



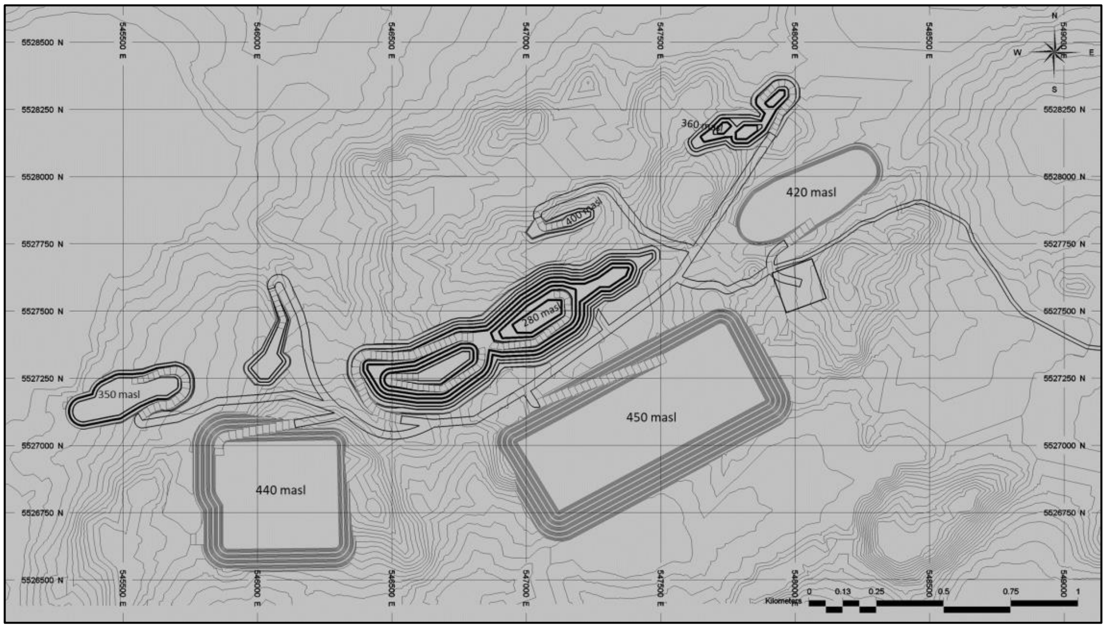
Source: AGP (2021).

Figure 16-52: End of Year 5 – Goliath Underground



Source: AGP (2021).

Figure 16-53: End of Year 5 – Goldlund



Source: AGP (2021).

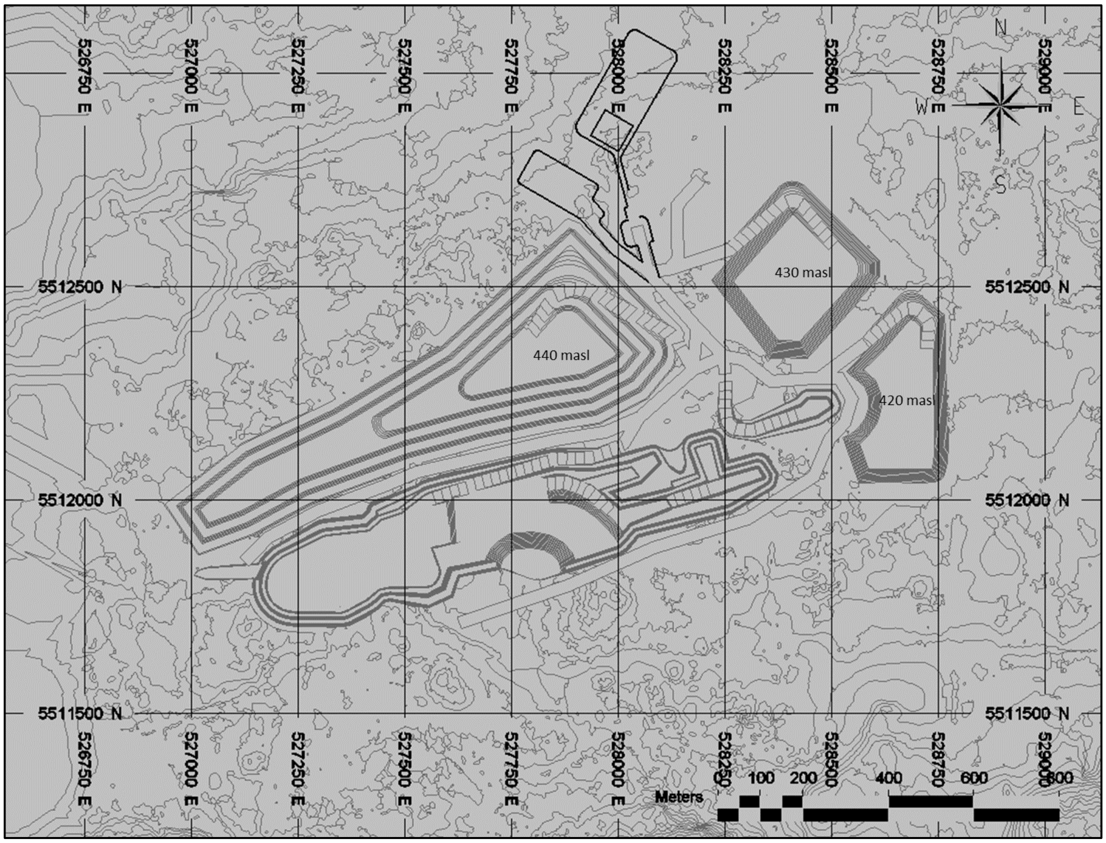
16.14.7 Year 6

Underground mining continues at Goliath (Figures 16-54 and 16-55). Mined material amounted to 0.51 Mt of mill feed grading 4.22 g/t Au and 9.32 g/t Ag. Processed material amounted to 0.71 Mt grading 3.41 g/t Au and 7.55 g/t Ag. Material stockpiled balance decreased to 4.62 Mt.

Mining continues at Goldlund with Phase 5 being depleted at the 360 masl elevation, and Phase 6 depleted at the 310 masl elevation (Figure 16-56). All Goldlund pits are depleted in the year. Mined material amounted to 0.92 Mt of waste rock, and 1.16 Mt of mill feed grading 0.87 g/t Au. Processed material amounted to 0.93 Mt grading 1.22 g/t Au. Material stockpiled balance increased to 5.85 Mt.

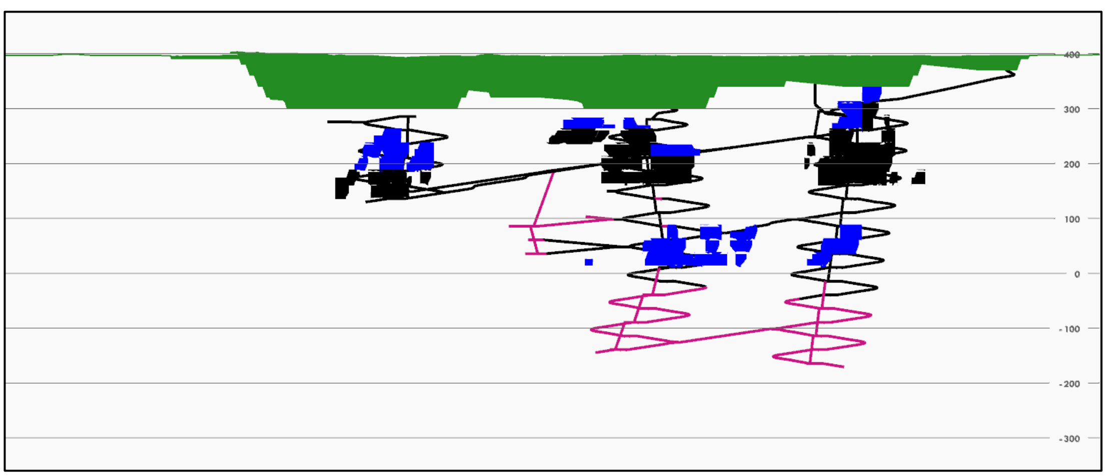
Mining is initiated at Miller and it is mined down to the 370 masl elevation (Figure 16-57). Mined material amounted to 4.83 Mt of waste rock, and 0.24 Mt of mill feed grading 1.05 g/t Au. Processed material amounted to 0.17 Mt grading 1.32 g/t Au. Material stockpiled balance increased to 0.08 Mt. Overburden stockpile and rock storage facility reach a lift elevation of 400 masl.

Figure 16-54: End of Year 6 – Goliath



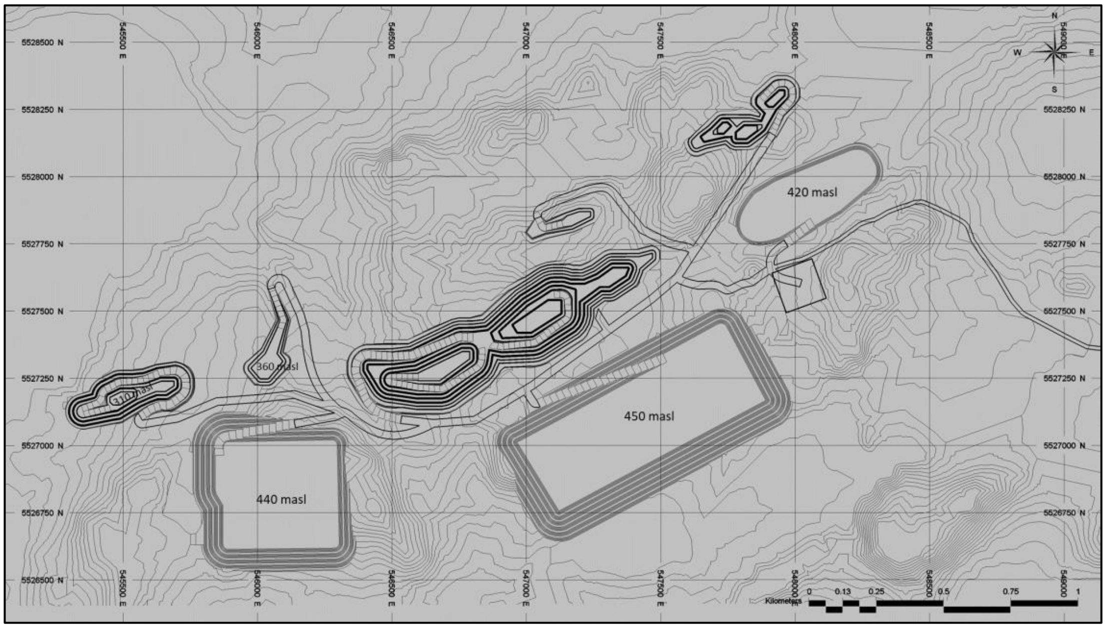
Source: AGP (2021).

Figure 16-55: End of Year 6 – Goliath Underground



Source: AGP (2021).

Figure 16-56: End of Year 6 – Goldlund



Source: AGP (2021).

Figure 16-57: End of Year 6 – Miller

Source: AGP (2021).

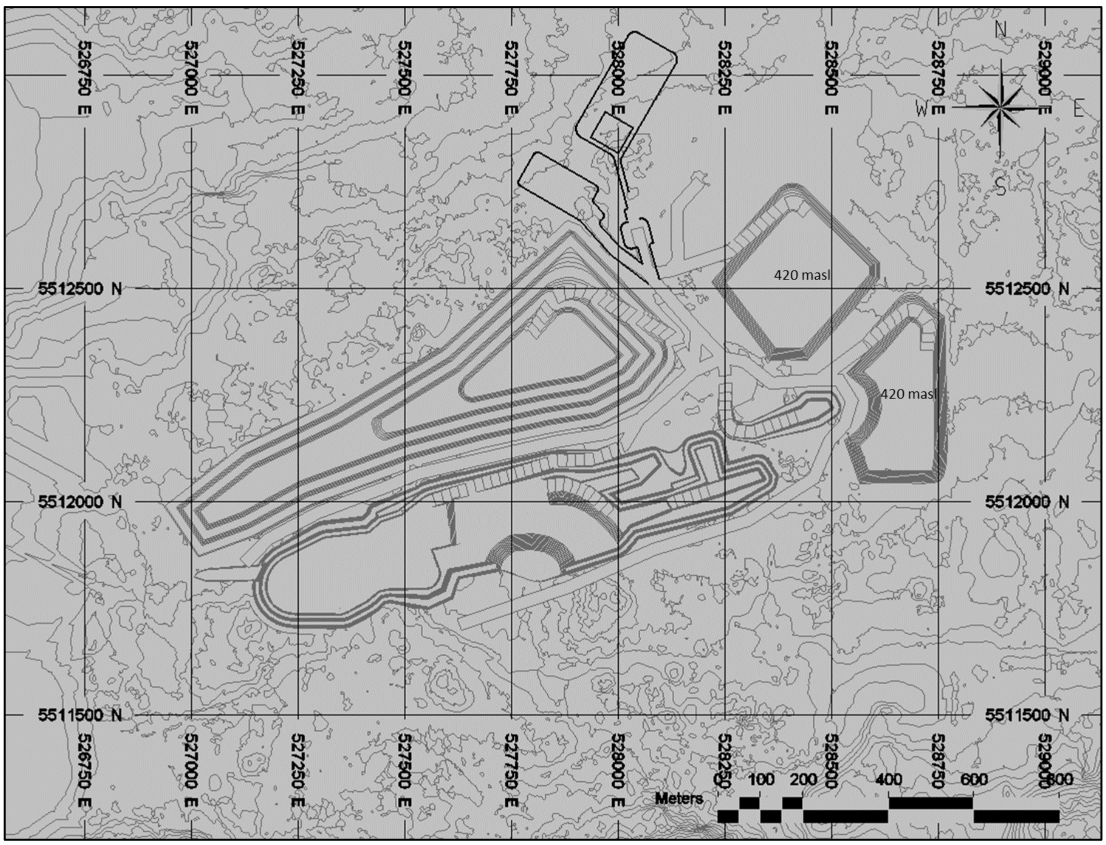
16.14.8 Year 7

Underground mining continues at Goliath (Figures 16-58 and 16-59). Mined and processed material amounted to 0.51 Mt of mill feed grading 4.04 g/t Au and 9.34 g/t Ag.

Processed material from Goldlund amounted to 1.17 Mt grading 0.78 g/t Au. Material stockpiled balance at Goldlund decrease to 4.68 Mt.

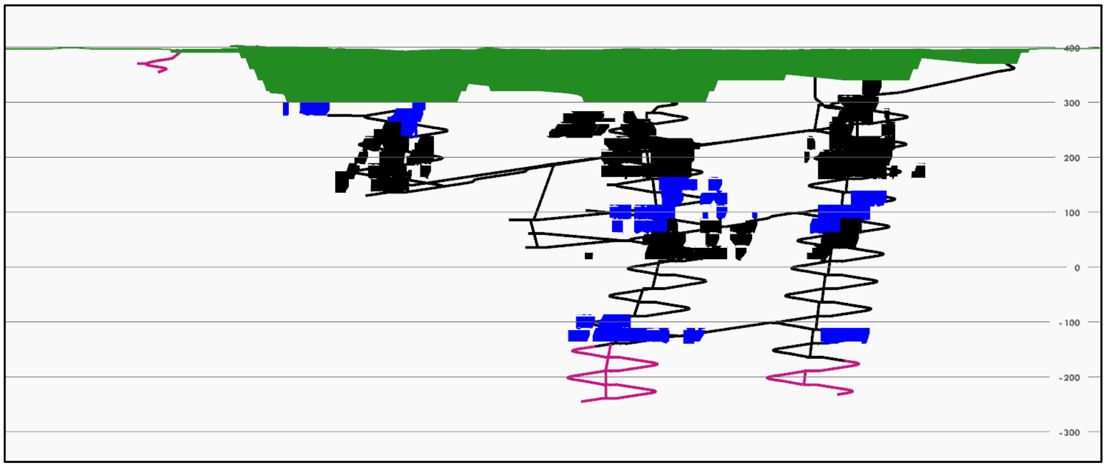
Mining continues at Miller and it is mined down to the 340 masl elevation (Figure 16-60). Mined material amounted to 4.44 Mt of waste rock, and 0.32 Mt of mill feed grading 1.06 g/t Au. Processed material amounted to 0.12 Mt grading 1.91 g/t Au. Material stockpiled balance increased to 0.27 Mt. Rock storage facility reaches a lift elevation of 410 masl.

Figure 16-58: End of Year 7 – Goliath



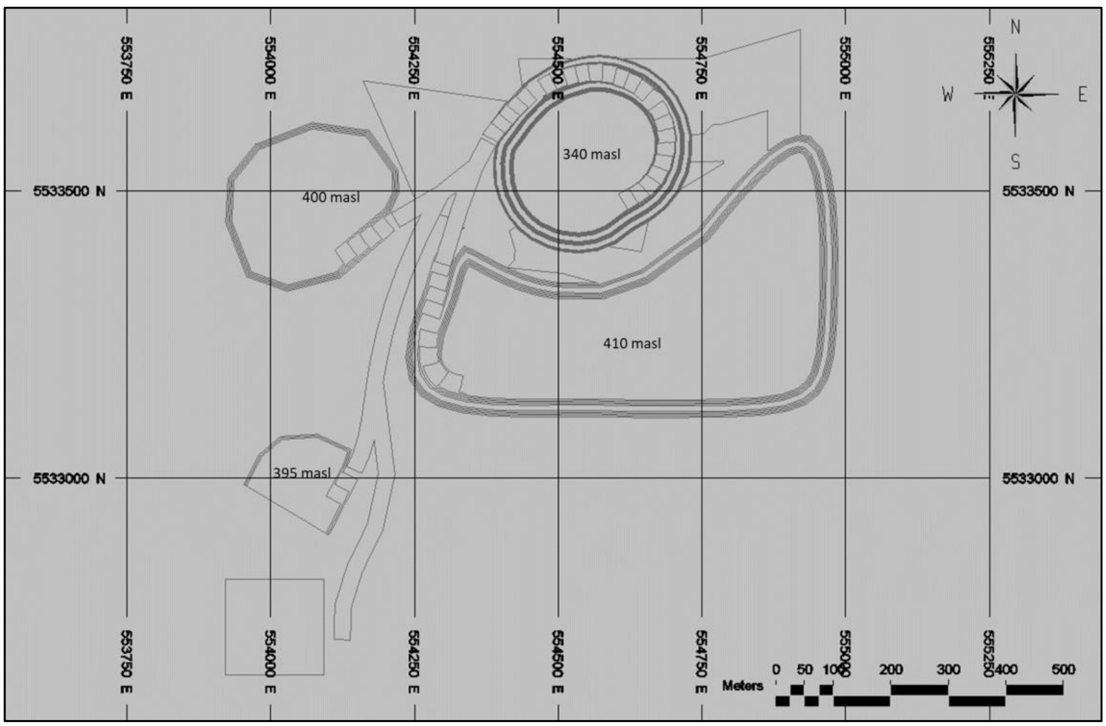
Source: AGP (2021).

Figure 16-59: End of Year 7 – Goliath Underground



Source: AGP (2021).

Figure 16-60: End of Year 7 – Miller

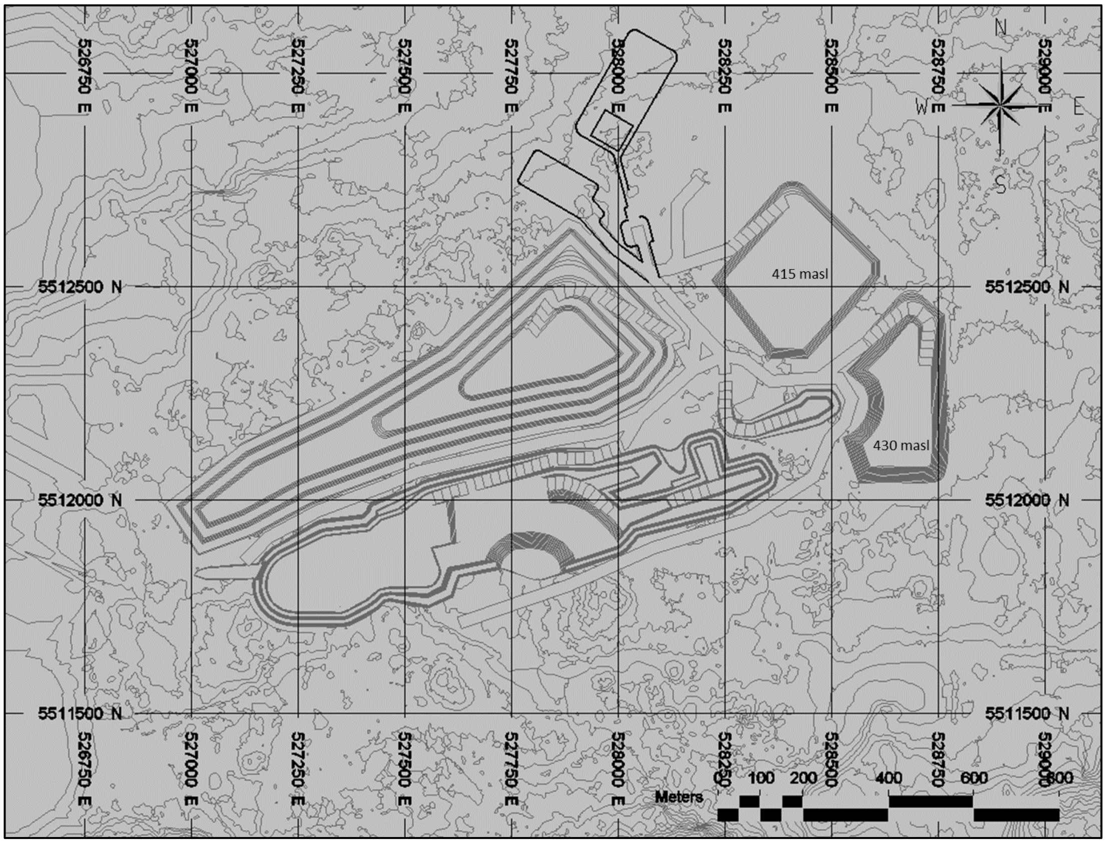


Source: AGP (2021).

16.14.9 Year 8

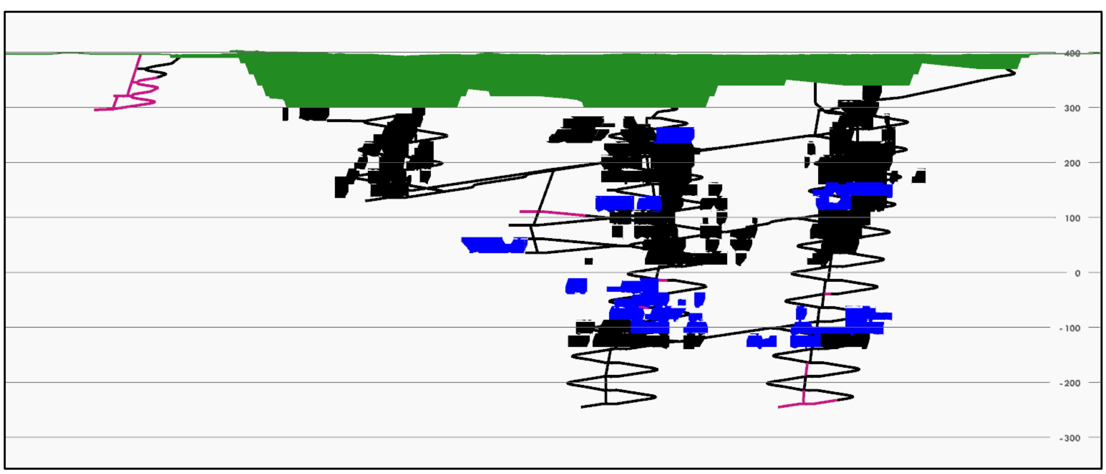
Underground mining continues at Goliath. Mined and processed material amounted to 0.51 Mt of mill feed grading 3.32 g/t Au and 9.45 g/t Ag (Figures 16-61 and 16-62). Processed material from Goldlund amounted to 1.11 Mt grading 0.78 g/t Au. Material stockpiled balance at Goldlund decrease to 3.57 Mt.

Figure 16-61: End of Year 8 – Goliath



Source: AGP (2021).

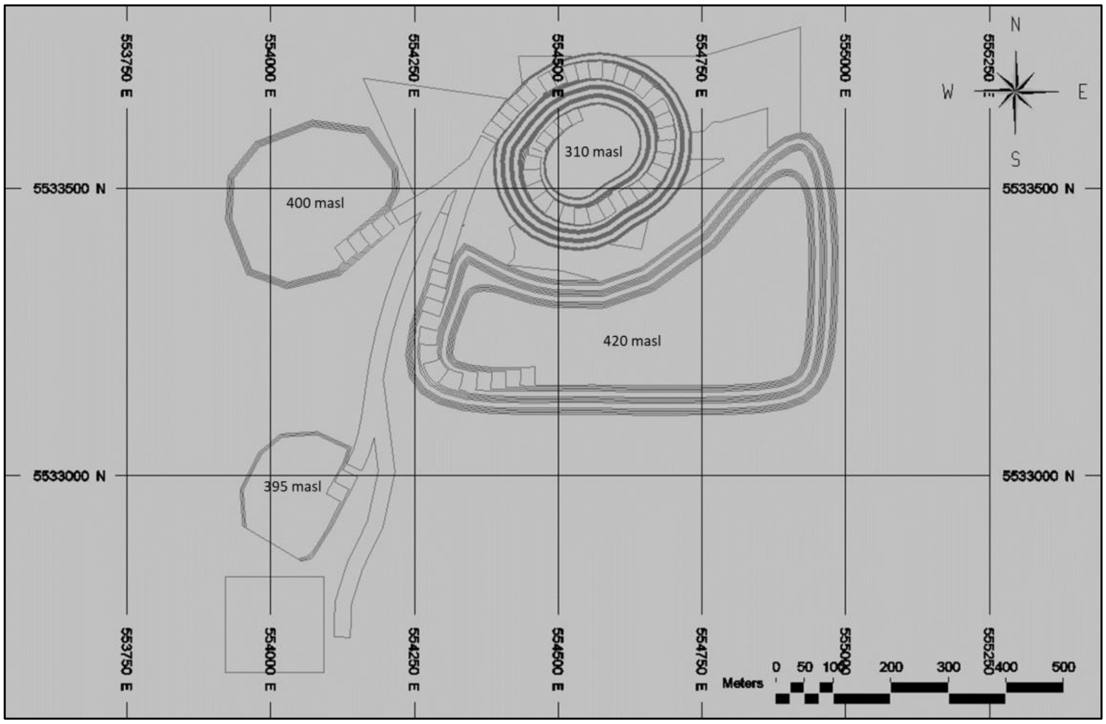
Figure 16-62: End of Year 8 – Goliath Underground



Source: AGP (2021).

Mining continues at Miller and it is mined down to the 310 masl elevation (Figure 16-63). Mined material amounted to 2.02 Mt of waste rock, and 0.41 Mt of mill feed grading 1.15 g/t Au. Processed material amounted to 0.18 Mt grading 1.92 g/t Au. Material stockpiled balance increased to 0.5 Mt. Rock storage facility reaches a lift elevation of 420 masl.

Figure 16-63: End of Year 8 – Miller



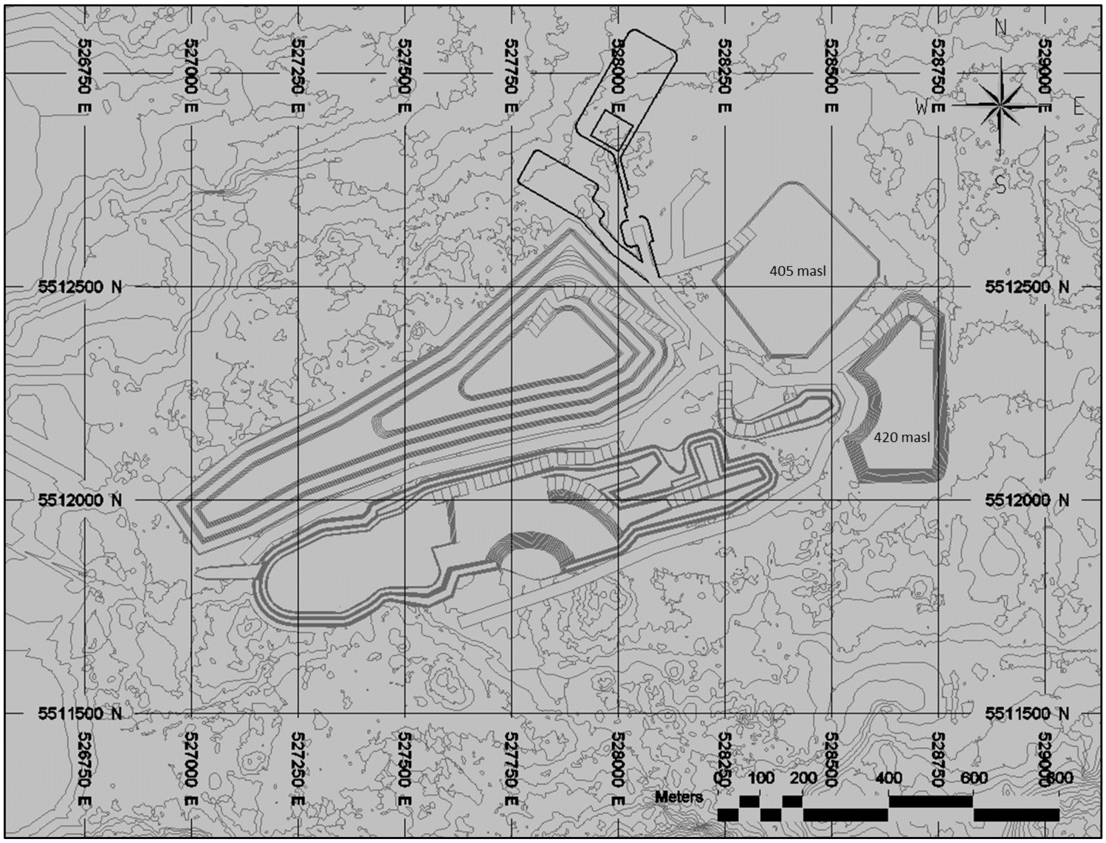
Source: AGP (2021).

16.14.10 Year 9

Underground mining continues at Goliath (Figures 16-64 and 16-65). Mined and processed material amounted to 0.50 Mt of mill feed grading 3.27 g/t Au and 7.60 g/t Ag. Processed material from Goldlund amounted to 1.08 Mt grading 0.78 g/t Au. Material stockpiled balance at Goldlund decrease to 2.48 Mt.

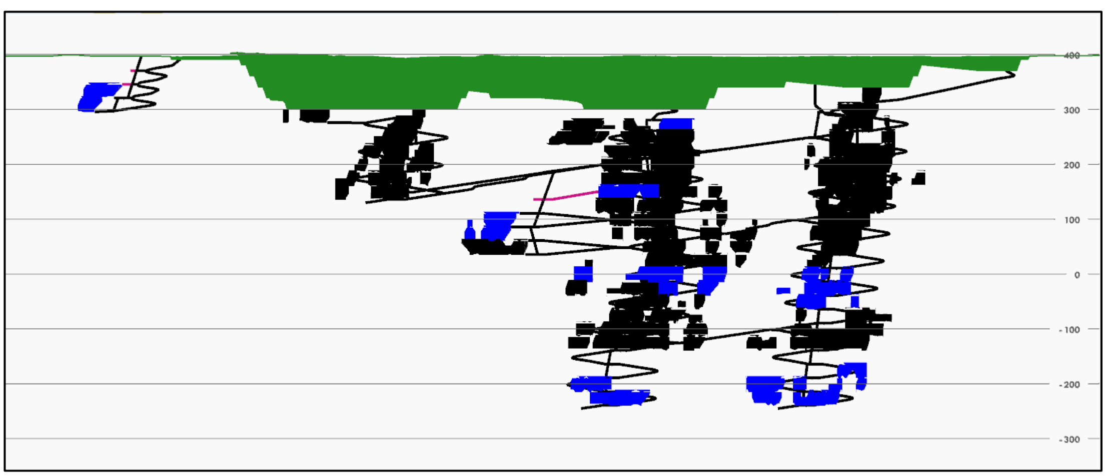
Mining continues at Miller and the pit is depleted at the 280 masl elevation (Figure 16-66). Mined material amounted to 0.47 Mt of waste rock, and 0.35 Mt of mill feed grading 1.36 g/t Au. Processed material amounted to 0.21 Mt grading 1.85 g/t Au. Material stockpiled balance increased to 0.64 Mt. Rock storage facility reaches a lift elevation of 430 masl.

Figure 16-64: End of Year 9 – Goliath



Source: AGP (2021).

Figure 16-65: End of Year 9 – Goliath Underground



Source: AGP (2021).

Figure 16-66: End of Year 9 – Miller

Source: AGP (2021).

16.14.11 Year 10

Underground mining continues at Goliath (Figures 16-67 and 16-68). Mined material amounted to 0.28 Mt of mill feed grading 3.56 g/t Au and 6.61 g/t Ag. Processed material amounted to 1.60 Mt grading 1.18 g/t Au and 2.86 g/t Ag. Material stockpiled balance decrease to 3.31 Mt. Processed material from Goldlund amounted to 0.04 Mt grading 0.77 g/t Au.

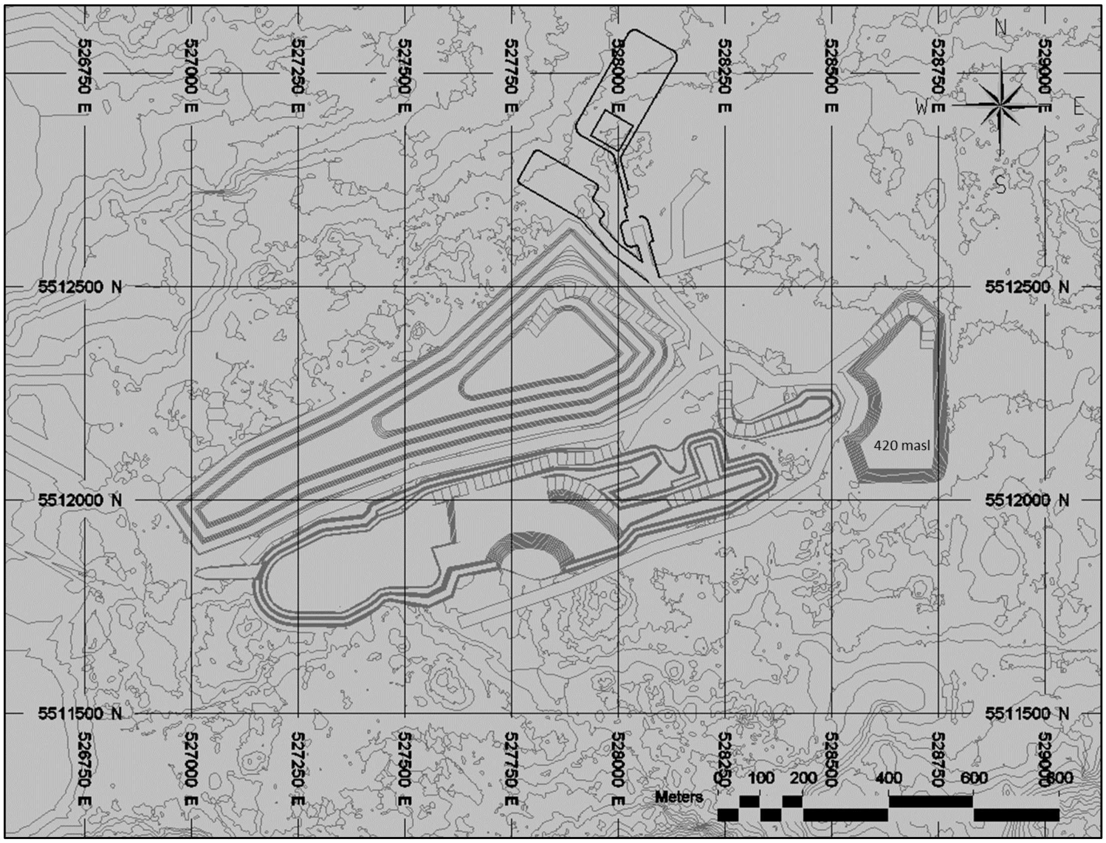
Material stockpiled balance at Goldlund decrease to 2.44 Mt.

Processed material from Miller amounted to 0.17 Mt grading 0.76 g/t Au (Figure 16-69). Material stockpiled balance at Miller decrease to 0.47 Mt.

16.14.12 Year 11

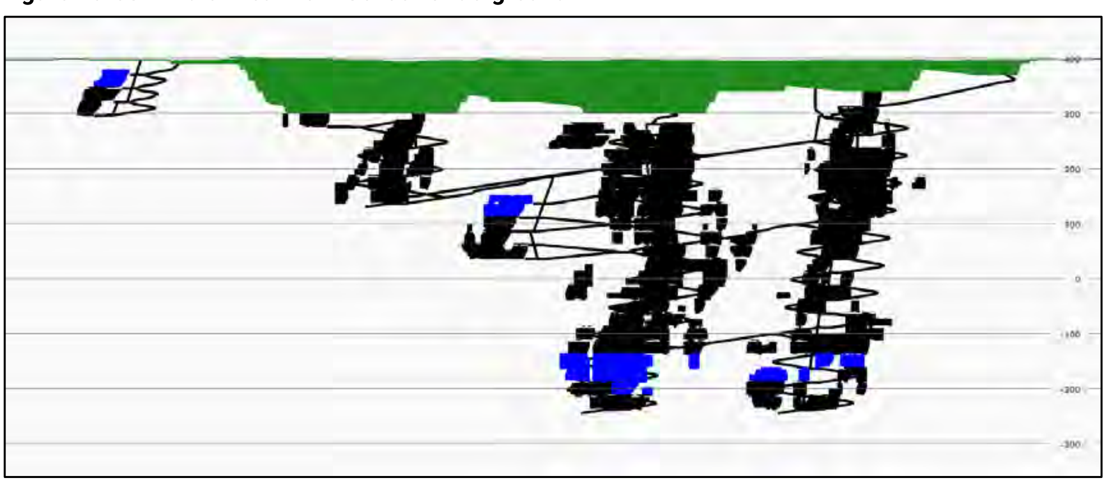
Underground mining is depleted Goliath at this year (Figures 16-70 and 16-71). Mined and processed material amounted to 0.02 Mt of mill feed grading 4.92 g/t Au and 8.37 g/t Ag. Processed material from Goldlund amounted to 1.78 Mt grading 0.48 g/t Au. Material stockpiled balance at Goldlund decrease to 0.66 Mt.

Figure 16-67: End of Year 10 – Goliath



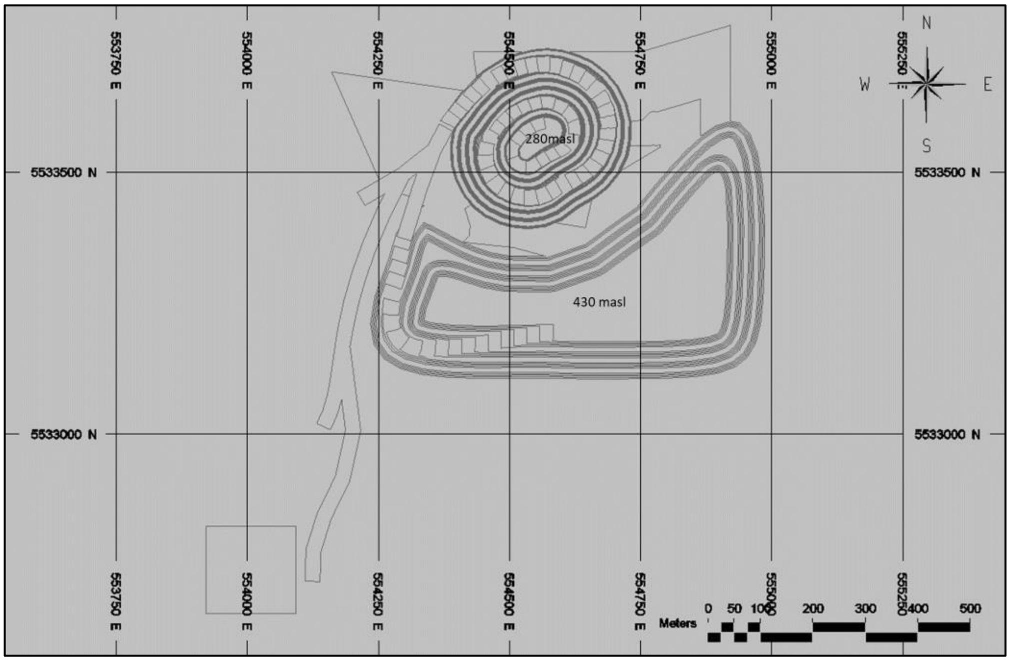
Source: AGP (2021).

Figure 16-68: End of Year 10 – Goliath Underground



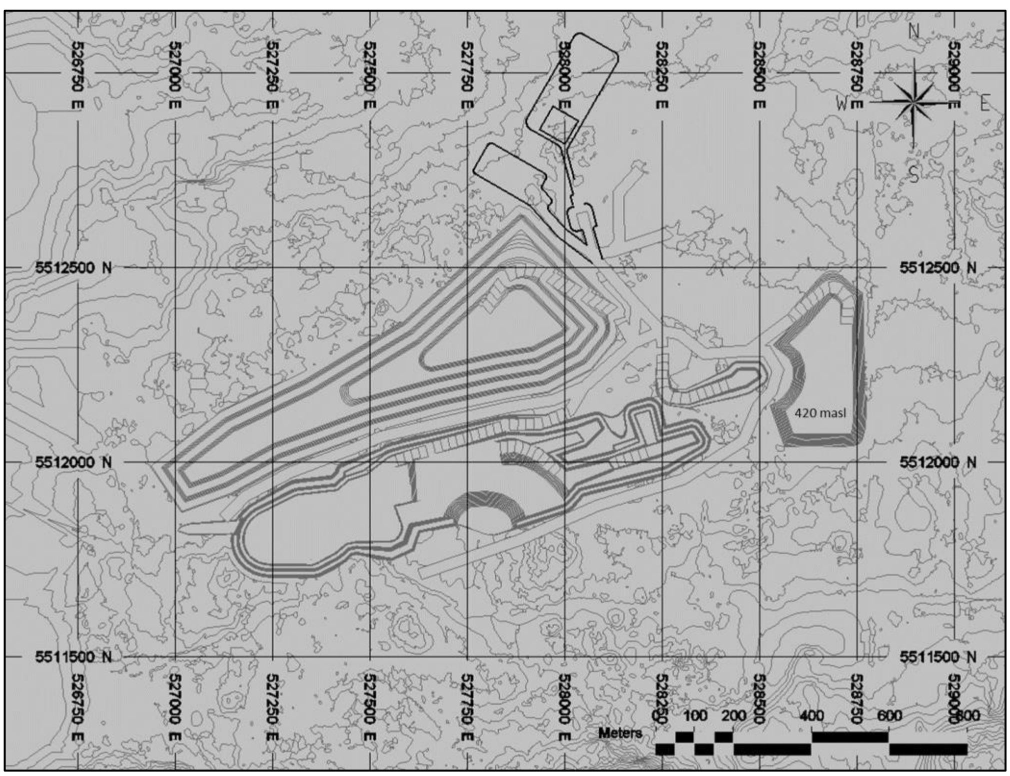
Source: AGP (2021).

Figure 16-69: End of Year 10 – Miller



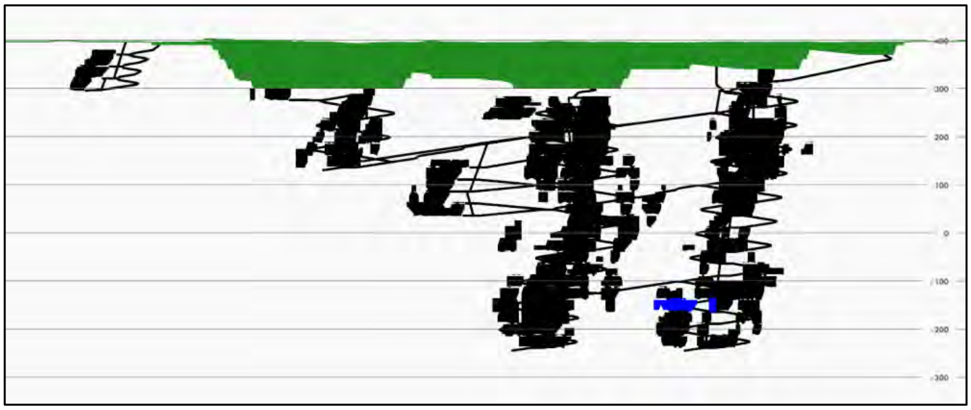
Source: AGP (2021).

Figure 16-70: End of Year 11 – Goliath Open Pit – Stockpile Rehandle



Source: AGP (2021).

Figure 16-71: End of Year 11 – Goliath Underground – Underground Complete



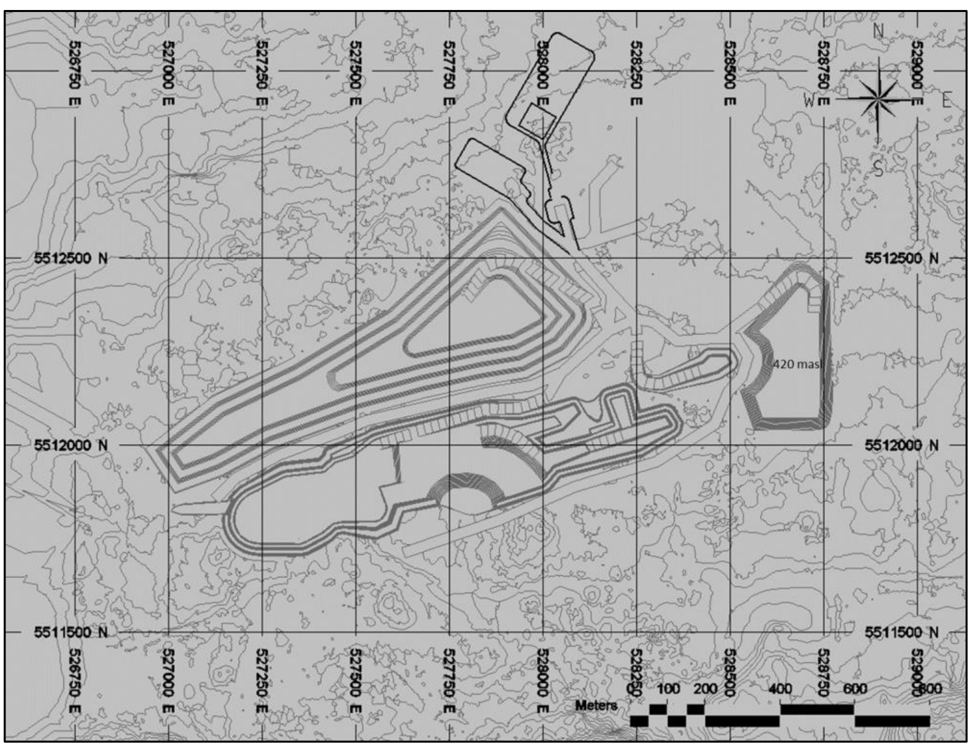
Source: AGP (2021).

16.14.13 Year 12

Processed material from Goliath amounted to 0.67 Mt grading 0.41 g/t Au and 1.68 g/t Ag (Figure 16-72). Material stockpiled balance at Goliath decrease to 2.64 Mt.

Processed material from Goldlund amounted to 0.66 Mt grading 0.48 g/t Au. Material stockpiled balance at Goldlund is depleted. Processed material from Miller amounted to 0.47 Mt grading 0.46 g/t Au. Material stockpiled balance at Miller is depleted.

Figure 16-72: End of Year 12 – Goliath Open Pit – Stockpile Rehandle



Source: AGP (2021).

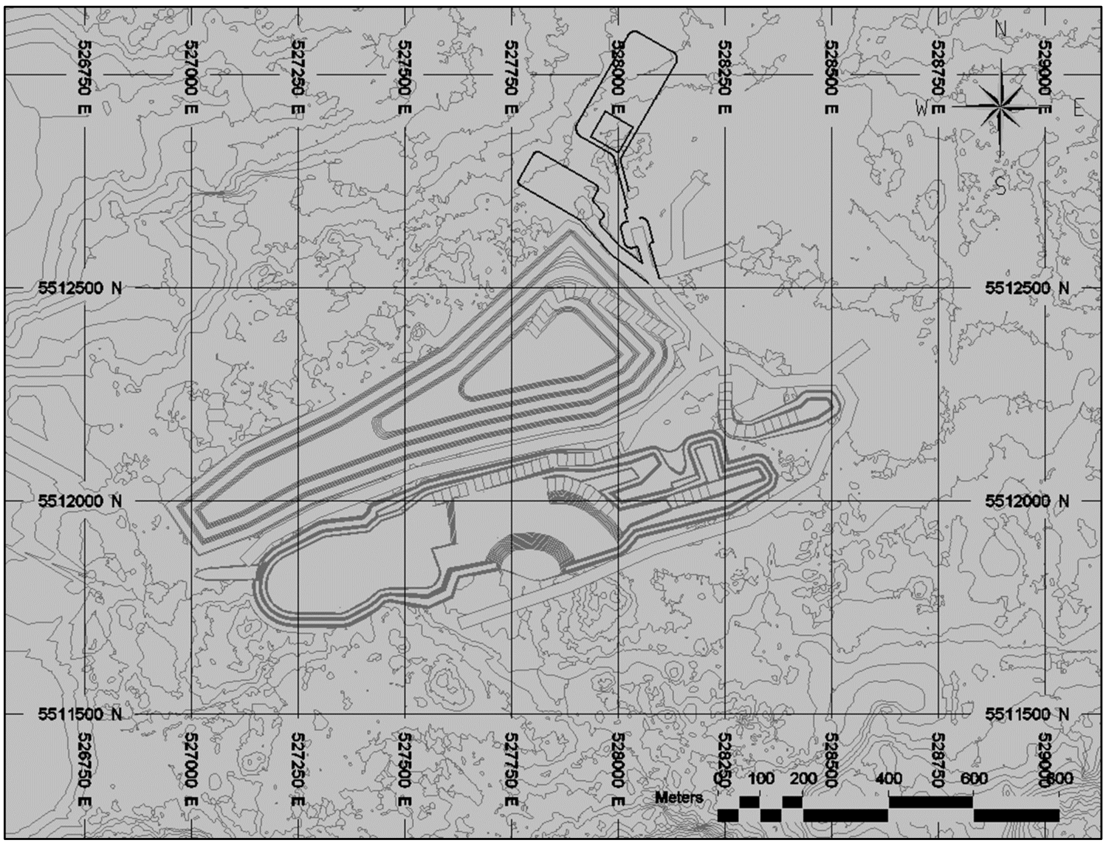
16.14.14 Year 13

Processed material from Goliath amounted to 1.80 Mt grading 0.41 g/t Au and 1.68 g/t Ag (Figure 16-73). Material stockpiled balance at Goliath decrease to 0.84 Mt.

16.14.15 Year 14

Processed material from Goliath amounted to 0.84 Mt grading 0.41 g/t Au and 1.68 g/t Ag (Figure 16-73). Material stockpiled balance at Goliath is depleted.

Figure 16-73: End of Year 13/14 Goliath Open Pit – Stockpile Removal Completed



Source: AGP (2021).

17 RECOVERY METHODS

17.1 Introduction

The project flowsheet has been selected based on recovery methods that would be required for processing Goliath and Goldlund production separately, supported by preliminary testwork and financial evaluations. Recovery characteristics for the Miller deposit are assumed to be the same as Goldlund, based on similar geology. The basis of the selected design is presented in Section 17.2. A process flow diagram and mechanical equipment list have been developed.

The process plant includes the following:

- three-stage crushing of run-of-mine material
- covered crushed material stockpile to provide buffer capacity for the process plant
- ball mill with cyclone classification
- gravity recovery of ball mill discharge by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution
- trash screening
- pre-aeration, leach and carbon-in-leach adsorption
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- carbon regeneration cyanide destruction of tailings using SO_2/air process
- carbon safety screening, and tailings disposal
- reagent storage and distribution
- water services (process water, treated water, fire water, gland water)
- potable water treatment and distribution
- air services

17.2 Process Design Criteria

The process plant has been designed to treat material from the Goliath, Goldlund and Miller deposits. Production from the three deposits will be processed as a blended feed to the plant according to the mining schedule. The key process design criteria for the mineral processing facility are listed in Table 17.1, which also summarises the grade and recovery data.

Table 17.1: Key Process Design Criteria

Criteria	Unit	Value
Annual Throughput (Design)	t/y	1,800,000
Daily Throughput (Design)	t/d	4,932
Operating Days per Year	d	365
Operating Availability – Crushing	h/y	5,869
Operating Availability – Grinding	h/y	8,059
Design Throughout – Crushing	t/h (dry)	311
Design Throughput – Milling	t/h (dry)	226
Crushing Feed Size, 100% Passing	mm	400
Crushing Product Size, 80% Passing	mm	8
Grinding Product Size, 80% Passing	µm	75
Ball Mill Circulating Load	%	350
Bond Ball Mill Work Index (Design)	kWh/t	15.7
ROM Head Grades Au (Average)	g/t	1.47
ROM Head Grades Ag (Average)	g/t	1.82
Recovery – Gravity Circuit	%	25.0
Recovery – CIL and Elution Circuit	%	68.6
Recovery – Overall	%	93.6
Average Annual Gold Production	oz/y	78,807

Source: Ausenco (2021).

17.3 Process Description

17.3.1 Crushing Circuit

ROM production is delivered by haul truck to the ROM feed bin where production from the Goliath, Goldlund and Miller deposits will feed the crushing circuit. ROM stockpiles can be blended as required to stabilise plant feed grade and material hardness when deposits are being mined simultaneously.

ROM material is fed into the crushing circuit to a vibrating grizzly screen. Grizzly screen oversize will then feed the primary jaw crusher, while grizzly undersize will bypass primary crushing. The material will then be reduced for secondary and tertiary screening and crushing before reaching the mill feed stockpile. Crushing circuit product is designed to be 80% passing size of 8 mm.

17.3.2 Mill Feed Stockpile

Crushing circuit product is conveyed to a covered mill feed stockpile. The stockpile is designed to have a live capacity of 24 hours and live retention 5,360 tonnes. The stockpile ensures the processing plant operates independently of the mining and crushing activities, providing constant feed to the grinding circuit.

17.3.3 Grinding & Gravity Circuits

Mill feed is reclaimed from the mill feed stockpile by two apron feeders. From the apron feeders, the crushed mineralised rock is fed to a ball mill by a conveyor. Process water is added to the conveyor discharge to create to slurry the crushed mineralisation in the ball mill for particle size reduction. The mill also receives oversized material from the gravity scalping screen and the underflow from the hydrocyclone cluster pack. The mill is operated in closed circuit where the product is discharged into a common pumpbox for both the hydrocyclone cluster pack and the gravity circuit, material that is too coarse at either unit operations will report back for further size reduction. This three-stage crushing and ball mill reduction circuit is known as a 3CB comminution circuit.

Ball mill product feeds the ball mill pumpbox where slurry is pumped to a hydrocyclone cluster as well as a gravity circuit. The hydrocyclones classify ball mill discharge to achieve the particle size required for leaching. Optimal leaching can be done at a hydrocyclone overflow 80% passing size of 75 microns. The hydrocyclone underflow containing larger particles returns to the ball mill for further size reduction, the circulating load within the ball mill hydrocyclone circuit is expected to be 350%.

The Goldlund deposit will require a finer grind for leaching. Particles leaving the grinding circuit through the hydrocyclone overflow will be required to have an 80% passing size of 75 microns.

Ball mill discharge slurry is pumped to both a hydrocyclone cluster and a gravity circuit. The gravity circuit is fed 100% equivalent of the fresh feed received from the ball mill from the mill feed stockpile. The gravity circuit consists of a gravity circuit screen and concentrator unit. The oversize screen will prevent any oversized material from entering the gravity concentrator and blocking fluidisation openings. Oversized material is returned to the ball mill feed chute. This equipment uses centrifugal forces to separate the liberated coarse gold material from the unliberated mineralised material. It operates on a semi-continuous basis. Gravity concentrator tailings is returned to the ball mill pumpbox for further liberation while free gold will accumulate on the walls of the concentrator during the concentration cycle. The concentrated free gold is flushed with water and gravitates to the Intensive Leach Reactor module.

17.3.4 Leaching & CIL

Hydrocyclone overflow gravitates to the Leach and Carbon-In-Leach (CIL) area via a trash screen. The trash screen will remove any oversized material from the slurry before leaching. This will ensure that minimise blockage with the intertank carbon screens.

The leach and CIL circuit used consists of nine tanks: one pre-aeration tank, two leach tanks and six CIL tanks. The total residence time required for leaching and adsorption is 27 hours. Following this criteria, all tanks have been sized for to achieve a volume of 1,149 m^3 each.

Air is injected to the bottom of the tanks to ensure the target dissolved oxygen level is maintained. Hydrated lime slurry is added to the pre-aeration and leach tanks to ensure the slurry pH remains above 10.5 in the circuit. This is a critical operation, as dangerous hydrogen cyanide gases will form if the pH drops below this level. Sodium cyanide is added to the first leach tank, and first CIL tank. Cyanide will dissolve the gold in the feed slurry. The pH of the leach circuit will be higher than typical gold leaching circuits in order to facilitate the breakdown of telluride minerals present in the Goldlund deposit. This will result in a higher consumption of lime. CIL tanks contain activated carbon which adsorbs and concentrates dissolved gold from solution. Carbon is pumped from the last CIL tank to the first CIL tank, counter-currently to the slurry flow. As carbon enters a tank slurry is displaced and flows to the following tank in the CIL circuit. The carbon in the seventh CIL tank will be mostly barren. Gold-loaded carbon is pumped from the first CIL tank to the elution circuit when an elution batch is initiated.

Barren slurry gravitates from the seventh CIL tank to the tailings circuit via safety screen. This safety screen ensures that any fine carbon that will contain gold is not sent to the tailings storage facility. The screen oversize is bagged and will be reprocessed. Screen undersize gravitates to cyanide detoxification and tailings management.

17.3.5 Elution & Carbon Regeneration

Gold laden carbon is pumped to the elution circuit for gold recovery. The selected elution circuit is of the AARL type. A 4-tonne acid wash column and a 4-tonne elution carbon column have been selected. Gold is stripped from carbon using a strong solution of sodium cyanide and sodium hydroxide. Pregnant solution flows into the pregnant solution tank for use in the electrowinning circuit. When an elution cycle is complete, the circuit is ready to initiate a new acid wash and elution cycle.

At the end of an elution cycle, the barren carbon is transferred to the carbon regeneration circuit. This circuit consists of a rotary kiln that will heat the carbon to about 700 ˚C, re-activating the surfaces of the carbon. Regenerated carbon is then cooled with water and mixed with fresh carbon as needed and returned to the CIL circuit.

17.3.6 Intensive Leaching

A separate leaching circuit is used to treat the free gold concentrate produced by the gravity concentrator. In the intensive leach reactor (ILR), free gold concentrate is leached into solution using sodium hydroxide, sodium cyanide and hydrogen peroxide. The ILR unit operates on a batch cycle producing both pregnant solution and concentrate tailings. Tailings slurry is returned to the grinding circuit for further liberation. Pregnant solution is stored in a pregnant solution tank for use in the electrowinning circuit.

17.3.7 Electrowinning & Gold Room

The electrowinning circuit consist of two independent cells, one dedicated to the elution pregnant solution and one dedicated to the ILR pregnant solution. An electric current is applied across the cells, causing gold to deposit on the surface of the cathodes. After an electrowinning cycle, the deposited gold is washed off the cathodes and dewatered in a manually operated filter press. The dewatered gold is dried in an oven and then mixed with the flux. Finally, the mixture is fed to a furnace where the gold is melted and poured in bars.

After the electrowinning process, barren solution from the elution circuit is returned to the elution circuit for processing during the next elution cycle. A portion of this solution will be purged to the CIL circuit to prevent the build-up of contaminants. The ILR circuit barren solution is pumped to the CIL circuit.

17.3.8 Cyanide Detoxification & Tailings Management

Barren slurry passing through the carbon safety screen is discharged into a cyanide detoxification tank. Cyanide detoxification will take place using the SO_2/ air process. In this process, sodium metabisulphite (SMBS) and air are used to detoxify the contained free cyanide and weak acid dissociable cyanide (CN_{WAD}) to below specific environmental discharge limits. This reaction is typically carried out at a pH of 8.5 and makes use of copper sulphate as a catalyst. Lime is used to increase the pH of the reaction as this reagent is already available on site. The cyanide detoxification tank has been sized base on a total residence time of 2.0 hours giving the tank a 383 m^3 live volume. After cyanide detoxification, slurry is pumped to a final tailings storage facility where water can be reclaimed and used as process water within the plant.

17.3.9 Consumables & Reagents

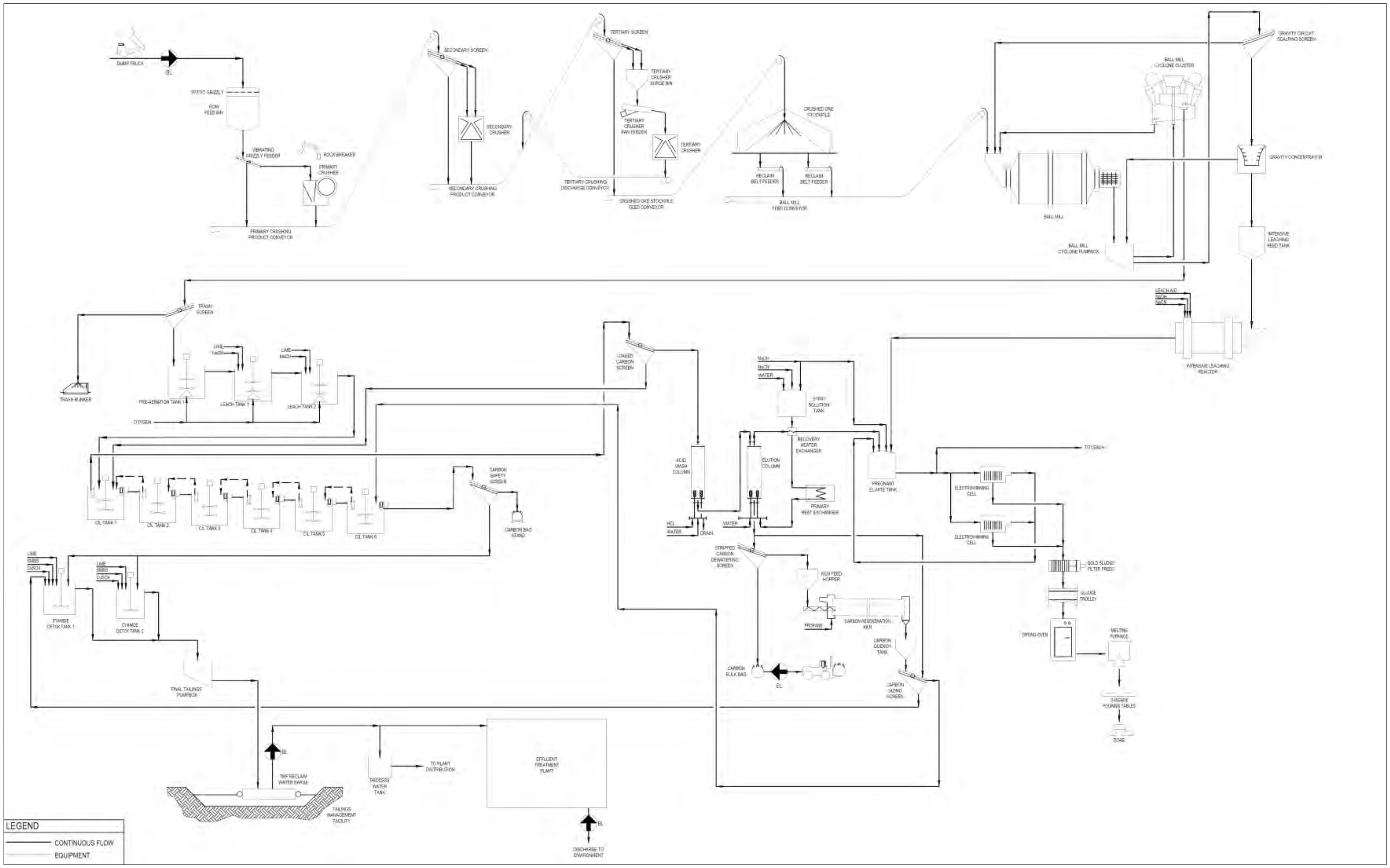
The consumables and reagents required for the mechanical and chemical treatment of the ROM can be summarised as follows:

- Hydrated Lime (Ca(OH)2) – used to control the pH in the leach, CIL and detox circuit
- Sodium Cyanide (NaCN) – used as the main gold leaching reagent in the CIL circuit and in the ILR circuit. It is also used to prepare the barren liquor in the gold desorption (elution) circuit
- Sodium Hydroxide (NaOH) – used to control pH in the elution, IRL and cyanide preparation circuits
- Hydrochloric Acid (HCl) – used in the acid wash circuit to remove scale formation on the carbon
- Activated carbon – used in the CIL circuit to adsorb dissolved gold
- Flocculant – used as a thickening aid in the pre-leach thickener
- Leach aid – used in the ILR circuit to improve the free gold leaching process
- Antiscalant – to reduce the formation of scale in the elution and electrowinning circuits equipment, and on the activated carbon itself
- Flux – used as a cleaning agent during gold smelting
- Ball mill media – grinding media required in the ball mill
- crusher and grinding mills liners

17.4 Process Flowsheet

An overall process flow diagram is presented in Figure 17-1.

Figure 17-1: Overall Process Flow Diagram



Source: Ausenco, (2021).

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Infrastructure to support the Goliath Gold Complex will consist of site civil work, buildings and facilities, water management systems, a tailings storage facility, and electrical power distribution. Mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems as required. The Goliath Gold Complex layout is shown in Figure 18-1.

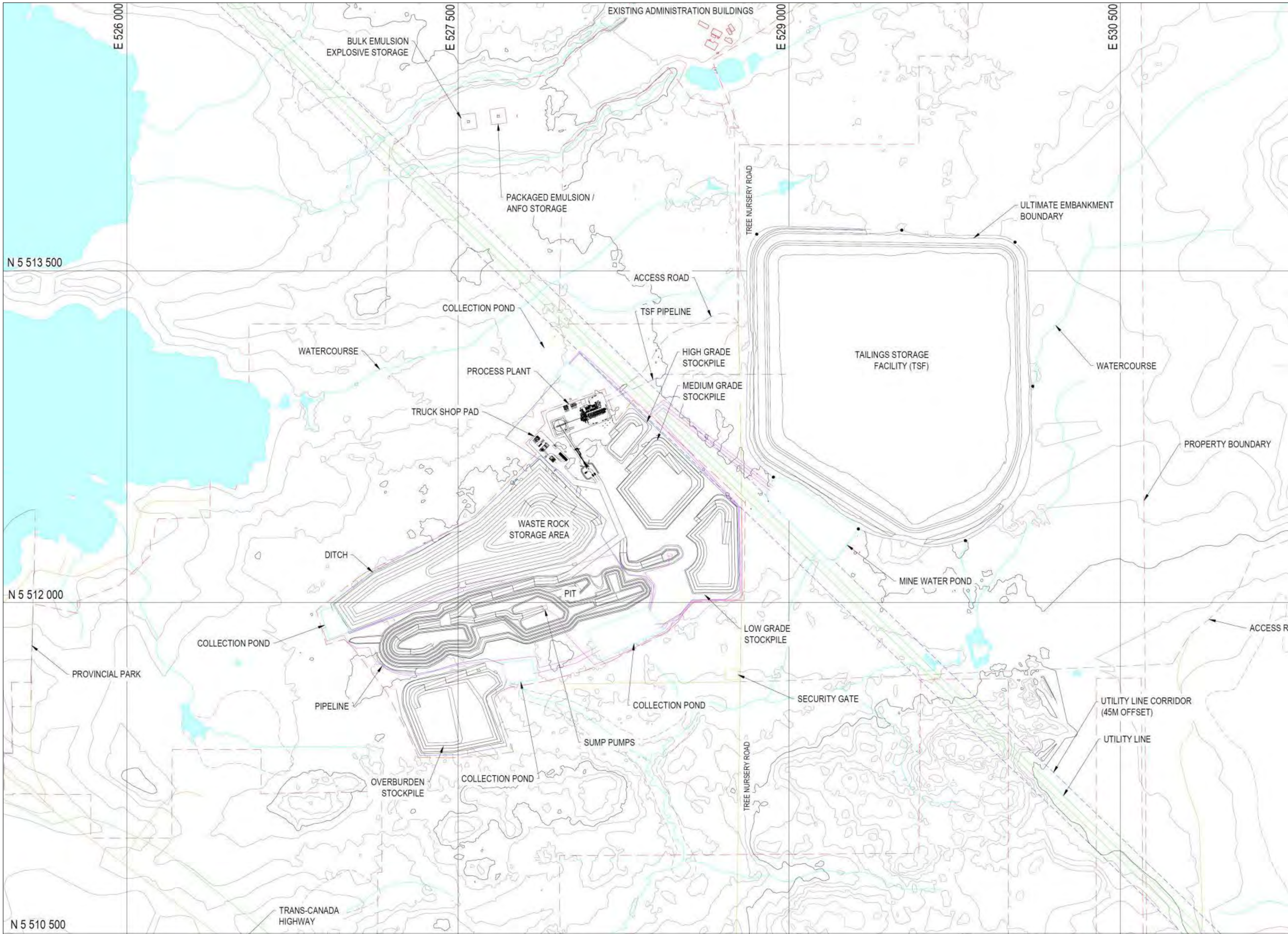
The processing plant and tailings storage facility will be located at the Goliath property, along with most ancillary project infrastructure. Infrastructure on the Goliath property will include:

- light vehicle and heavy equipment roads
- plant access road
- overburden stockpiles
- low-, medium- and high-grade stockpiles
- mine facility platforms and process facility platforms
- water management ditches and collection ponds
- tailings storage facility (TSF)
- process plant, including crushing, stockpile, mill, gold room and reagent storage buildings
- effluent water treatment plant
- mine dewatering pond
- waste storage facilities
- mine dewatering pumps and pipelines
- incoming power high voltage substation and site-wide electrical distribution
- assay laboratory
- mine and process administration offices and change rooms
- mine truck shop, truck wash and refuelling station
- workshop and warehouse facilities

The Goldlund and Miller properties will each have the following infrastructure:

- heavy equipment roads
- overburden stockpiles
- low-grade stockpiles
- rock storage facilities
- mill feed transfer pads
- mine dewatering pumps and pipelines

Figure 18-1: Goliath Gold Complex Layout



18.1.1 Layout Development

Locating the site facilities was based on the following considerations:

- within the claim boundary
- no requirement to divert the Tree Nursery Road
- suitable geotechnical conditions (discussed in Section 18.1.2)
- outside the battery limits of the potential future high-voltage power corridor expansion
- stockpiles are near mine pits to reduce haul distances
- process plant is in an area safe from flooding
- administration, processing plant and offices are in close proximity to limit travel distances
- process plant and other buildings are outside the 500 m blast zone radius

18.1.2 Site Geotechnical

Geotechnical site investigations were completed for the Goliath Project in 2014, 2017 and 2018 to characterise subsurface conditions at the TSF, plant site, waste rock dumps, collection ponds and overburden stockpiles (KP, 2018). Geotechnical site investigations have not been performed for the Goldlund or Miller projects.

The geotechnical materials encountered vary across the Goliath site, with areas of near-surface bedrock to areas with over 18 m of fine-grained soil overlying bedrock. The overall site was divided into three generalised geotechnical areas (KP, 2018), as follows:

- Near-surface Bedrock – Includes areas where surficial soil depth is less than 3 m. Where present, the near-surface bedrock features are oriented northeast-southwest and are located at the north and south ends of the project site.
- Sand and Silt – Includes areas where the surficial soil depth is greater than 3 m and the primary soil units comprise sands and silts typically overlying the silts and clays.
- Silt and Clay – Includes areas where the surficial soil depth is greater than 3 m and the primary soil units comprise clays and silts.

Five generalised geotechnical units were observed during the investigations. The units as observed (from surface to depth) are summarised as follows.

- top soil
- sand and silt
- brown silt and clay
- grey silt
- sand
- bedrock

The following key details were noted from the completed investigations:

- In-situ strength measurements in the grey silt (which is predominate over the majority of the "sand and silt" and "silt and clay" areas) show peak strengths ranging between very soft (8 kPa) to stiff (54 kPa) with moisture content typically at or above the liquid limit. This material is expected to be a controlling layer for static stability and bearing capacity.
- "Sand and silt" geotechnical conditions are predominant in the foundation of the proposed TSF area. The sand and silt material is expected to have relatively high permeability and is not considered favourable from a seepage control perspective.
- The investigations indicated that the sand and silt material present in the "sand and silt" unit is generally poorly graded, loose to compact, and moist to saturated. These are high-level indicators that the material under certain loading conditions can be susceptible to liquefaction and must be evaluated on a case-by-case basis.
- Depth to bedrock surface was found to be highly variable with depths ranging from outcrop to 18 m depth.

18.1.3 Site Preparation

Forest clearing and topsoil removal will be required for the processing plant, mining pits, stockpiling areas, and other buildings and facilities.

Existing roads connected to the project site enable access to the properties. Typical method of clearing and topsoil removal, excavation, drains, safety bunds and aggregates will be employed to construct additional roads and upgrade existing roads as required.

18.2 Roads & Site Access

18.2.1 Existing Roads

The Goliath Project is located in western Ontario, 20 km east from the Dryden, and is accessible year-round north from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Anderson and Tree Nursery Roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals.

The Goldlund and Miller projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath Project, off Highway 72. The Goldlund site is accessed via private gravel roads connected to Ontario Provincial Highway 72, approximately 60 km from Dryden. The Miller site is accessed via a forestry road east off Highway 72. Access roads for the Goldlund and Miller sites are maintained by Domtar, which is the sustainable forest licence holder for the area.

18.2.2 Planned Upgrades

A new road connecting the process plant pad to the existing Tree Nursery Road will be constructed, as well as interior roads within the Goliath property connecting the process plant, crushing area, mining operation structures and all mill buildings. New light vehicle roads will be approximately 6 m in width and will be designed to include draining and safety berms/bunds where appropriate. Upgrades of existing roads will be required for heavy equipment access to the Goldlund and Miller projects, discussed in Section 18.3.1.

18.3 Mining Infrastructure

18.3.1 Haul Roads

Haul roads will be approximately 29 m in width and constructed as new roads or as upgrades to existing roads prior to the start of mining activity. New and upgraded roads will include:

- construction of a haul road between the ROM pad and Goliath pit
- upgrading the existing 3.7 km road connecting the Goldlund transfer pad to Highway 72 to allow for mill feed haulage and equipment traffic
- upgrading of the existing 12.8 km road connecting Highway 72 to the Goliath pit area to allow for mill feed haulage and equipment traffic

18.3.2 Truck Shop & Truck Wash

The truck shop and truck wash building will provide maintenance to the truck fleet, and be located on a separate pad near the process plant. The buildings types and sizes are described in Table 18.1.

Table 18.1: Warehouse, Office & Workshops Building Descriptions

WBS Code	Building Description	Building Construction	L (m)	W (m)	H (m)	Area (m^2)	Vol (m^3)
4400	Truck Shop	Fabric	46	45	12	2,070	24,480
4400	Truck Wash	Fabric	23	19	12	463	5,390
4400	Truck Shop Warehouse	Fabric	18	11	8	193	1,463
4400	Administration Offices & Dry Facilities	Modular	19	19	3	361	1,083
4400	Mine Office & Change Room	Modular	23	19	3	437	1,311
4400	Maintenance Workshop & Warehouse	Fabric	28	17	9	469	4,317
4400	Security Gatehouse	Modular	12	4	4	43	173
4400	Assay & Metallurgical Laboratory	Modular	18	12	2	223	536
4400	Reagent Offloading & Storage	Fabric	16	20	4	320	1,280

Source: Ausenco (2021).

18.3.3 Explosive Storage Facility

A suitable location for explosive storage west of the old tree nursery was identified based on minimum allowable distances defined by Natural Resources Canada. Regular deliveries will minimise the amount of explosives stored on site.

18.4 Process Plant Buildings

The process plant will be located on the Goliath property. Process plant buildings are summarised in Table 18.2, and described in the following sections.

Table 18.2: Processing Plant Infrastructure

WBS Code	Description	Building Construction	L (m)	W (m)	H (m)	Area (m^2)	Vol (m^3)
3100	Primary Crushing Building	Pre-engineered	25	15	14	375	5,250
3100	Secondary Screen & Crusher Building	Pre-engineered	30	14	16	476	7,616
3100	Tertiary Screen & Crusher Building	Pre-engineered	24	16	16	384	6,144
3200	Stockpile Cover	Fabric	54	54	28	2,916	81,648
3300	Mill Building	Pre-engineered	38	25	30	2,091	28,500
3500	Gold Room	Pre-engineered	17	12	10	204	2,040
3700	Main Reagent Building	Pre-engineered	55	24	14	1,320	17,820

Source: Ausenco (2021).

18.4.1 Crushing Area Buildings

Crushing area buildings will be of modular design and equipped with dust collection systems.

The primary crushing building will house the ROM hopper equipped with a static grizzly, vibrating grizzly feeder, primary jaw crusher, chutes and additional platework. The rock breaker will also be within the building. In addition, access platforms and reinforced concrete will be utilised for the pad to support the primary jaw crusher.

Additional screening and crushing will also be completed prior to the mill feed stockpile. The secondary screen and crusher will be housed in a dedicated building, while the tertiary screen and crusher will be located in a separate building. Conveyors and feeders will be used to control the movement of material between the buildings.

Conveyors are used in the nominal operation to move the crushed material through the crushing circuit and do not rely on regular use of mobile equipment.

A fabric building cover and concrete reclaim tunnel will be used for the mill feed stockpile.

18.4.2 Processing Plant Buildings

The process plant complex is comprised of the following separate buildings:

- mill building (grinding and gravity)
- main reagents building
- gold room

Large-scale buildings will be constructed from pre-engineered metal, supported on reinforced concrete footings and are complete with concrete slabs and pedestals. To account for winter conditions, buildings will be built with insulated metal panel (IMP) roof and wall cladding. Area cranes will be available for equipment servicing in the process plant.

The mill building includes a ground floor, one elevated concrete floor. The various equipment will be accessed by purpose-built mezzanine platforms for maintenance, service and sampling. The grinding and gravity 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 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, flocculant, oxygen and antiscalant. Where possible totes of reagents will be used directly, to conserve space and tankage.

The gold room will house the pregnant solution tank, electrowinning cells, sludge filters, furnace, drying oven and vault. The building will be a two-storey concrete wall structure.

External parts of the processing plant include a pre-aeration tank, as well as two leach and six carbon-in-leach tanks, all of which are 11 m in diameter, and two detoxification tanks that are 7.9 m diameter. The tanks will be accessed by a purpose-built mezzanine platform and walkway to allow servicing, sampling and maintenance. An area crane will provide access to screens, tanks, pumps and agitators. The tailings will report to a pumpbox before being pumped to the tailings storage facility.

18.5 Project Support Infrastructure

18.5.1 Buildings

Plant ancillary buildings located on the Goliath property are described in the following sections. Refer to Table 18.1 above for a building description summary.

18.5.1.1 Administrative

New administrative offices will be located near the process plant. Buildings will be a single-storey, prefabricated modular design placed on precast concrete footings. The administrative building will include offices, meeting rooms, lunchroom, washrooms, men's and women's dry, lockers, first-aid and showers, and will be equipped with heating, ventilation and air conditioning (HVAC).

Existing structures originally served as the former Ontario Ministry of Natural Resources and Forestry facility and are currently used for office space and warehousing. These are located approximately 4 km north of the future process plant and are accessed via Tree Nursery Road. These structures may be retained as auxiliary space, but are not critical to the project.

18.5.1.2 Security Gatehouse

The security gatehouse will be a small, prefabricated building with a single boom gate, located south of the process plant near the junction of Tree Nursery Road and Normans Road. Site inductions for visitors and new employees can be conducted at this point.

18.5.1.3 Laboratory

The laboratory will be a prefabricated, single-storey, modular building on precast concrete blocks.

18.5.1.4 Maintenance Shop & Warehouse Building

The plant maintenance shop and storage building will be located close to the process plant. Buildings will have a reinforced concrete raft foundation and fabric.

18.5.2 Power Supply & Electrical

The processing plant and mining activities will require significant power supply. Two power transmission lines operated by Hydro One, a 115 kV and a 230 kV line, cut diagonally across the Goliath Property and are assumed to be sufficient for the project. The process plant and mine will be powered by means of a new incoming substation with a tie-in to the 115 kV powerline, located on the Goliath property, and property-wide reticulation. Peak demand is estimated at 13.4 MW. Emergency power for the Goliath property will be provided by emergency diesel generators.

The Goldlund and Miller projects will not require permanent electrical infrastructure.

18.6 Tailings Storage Facility

Knight Piésold Ltd. (KP) completed a PEA-level design for the TSF at the Goliath Gold Complex. The TSF will provide secure storage for tailings and process water, and protect groundwater and surface waters during operations and post closure. The PEA level design is based on a projected 13.5-year mine life at a nominal processing rate of approximately 4,875 t/d. The TSF has been sized to permanently store approximately 24.0 Mt of tailings, or 18.4 Mm^3 at an average settled dry density of 1.3 t/m^3.

The design basis for the TSF has been developed based on input from Treasury Metals, industry-accepted best practices, previous project studies (including site geotechnical investigations), and anticipated mine site conditions. The TSF embankment concepts have been developed to meet local and international standards for the design of mining facilities (CDA, 2019; MAC, 2019).

The embankments include for adequate freeboard to provide ongoing tailings storage, operational water management (water cover), temporary environmental design storm storage and conveyance up to and including the inflow design flood. The tailings and waste rock produced from the Goliath property is expected to be potentially acid generating (PAG), while the Goldlund tailings are anticipated to be non-PAG. Miller tailings were assumed to be non-PAG.

Tailings produced from all deposits will be jointly processed and stored in a single TSF located at the Goliath project site. Permitting commitments include a 2 m water cover during operations, a basin and embankments lined with high-density polyethylene (HDPE), embankments constructed from non-PAG material, and a minimum 2 m non-PAG tailings cover at closure.

The TSF will be constructed as a single cell facility. A geomembrane lining system consisting of a 80-mil HDPE geomembrane underlain by a 12 oz/yd^2 (407 g/m^2) non-woven geotextile cushion layer will be installed along the TSF basin floor and on the upstream face of the perimeter embankments. The lining system will be installed on prepared subgrade within the TSF basin floor. The prepared subgrade will have organics and unsuitable materials removed, with localised regrading of existing overburden soils to develop a relatively smooth surface for liner placement.

The embankments will be raised to form a four-sided paddock style impoundment. The TSF design will include an initial starter embankment (Stage 1) occupying a smaller footprint than subsequent stages. Stages 2 through 4 of the TSF will operate as one large basin and will be raised using downstream construction methods throughout the mine life.

The bulk fill within the embankment will be constructed with non-PAG material, sourced from locally available borrow sources. Select zones of the TSF embankment will include processed locally borrowed fill materials. The zoned embankment will be constructed with filter graded materials consisting of an upstream filter zone (sand and gravel), followed by transition zone (sand, gravel and cobbles) and a downstream general fill (rockfill and/or glacial till borrow). The embankment will be constructed on a prepared subgrade with organics and unsuitable materials removed from the embankment footprint. The embankment will be constructed with 2.5H:1V upstream and downstream slopes, a 10 m wide embankment crest, and a downstream buttress (in select locations) to meet stability requirements. A collection drain will be installed below the embankment to collect potential seepage and mitigate the development of excess pore water pressures in any finer grained zones of the general fill.

Instrumentation consisting of vibrating wire piezometers, survey monuments and slope inclinometers will be installed within the foundation and embankment fill materials. The instrumentation will be monitored to verify embankment performance.

Tailings will be pumped as a conventional slurry tailings (typically 35% to 50% solids content by weight) from the process plant to the TSF via pipeline(s). Tailings will be deposited from multiple locations around the perimeter of the TSF basin from the upstream face of the TSF embankments and from a floating pipeline (as required) to facilitate sub-aqueous deposition. The tailings deposition strategy will allow for even filling of the basin, maintain a water cover over the tailings, and maximise tailings storage within the impoundment.

Meteoric and supernatant inflows to the TSF basin will be temporarily stored prior to reclaim by a floating pump barge in the basin to the process plant. Adequate freeboard allowances for temporary storage of the environmental design storm have been included within the proposed staging plan. Excess water beyond the storage of the maximum water cover level will be transferred to the mine water pond. The TSF will be equipped with an overflow spillway in each embankment stage to accommodate flows above the environmental design storm and up to the inflow design flood.

18.7 Water Management

18.7.1 Site Water Balance

In general, runoff from the various catchment areas within the project footprint will be collected in sediment collection ponds and pumped to the mine water pond. Groundwater inflow into the open pit and underground mine will also be pumped to the mine water pond. Excess water beyond the required storage associated with the water cover level in the TSF and reclaim requirements, will be transferred to the adjacent mine water pond. Water from the following areas will be transferred to the mine water pond on an ongoing basis throughout the mine life:

- waste rock storage area
- overburden stockpile
- open pit and underground mine
- low-, medium- and high-grade stockpiles
- process plant
- TSF

Reclaim water for use in the process will be taken from the TSF, when available. Fresh water and make-up reclaim water (i.e., reclaim water that is not available from the TSF) will be taken from the mine water pond.

Any excess water from the mine water pond will be treated (as required) in the effluent water treatment plant and discharged to the environment.

A monthly average water balance was completed to estimate the expected water reporting to each facility. The water balance has shown that there will be enough water available from the combined TSF and mine water pond for use as fresh and reclaim water in the process (with no make-up water required) and to maintain a 2 m water cover over the tailings during operations.

18.7.2 Surface Water Management

Site water management measures have been developed based on the PEA site arrangement, operational requirements, and environmental site conditions. Following a precipitation event, the runoff will be managed to reduce the total suspended solids prior to discharge to the environment. This is a requirement of the project's operating conditions. Water management measures for the project will include a series of diversion berms, collection and diversion ditches, sediment basins, and water transfer pipelines to collect runoff originating within disturbed areas. The runoff will be conveyed to one of a number of catchment ponds, where the majority of the total suspended solids can settle out prior to sending the water to the mine water pond (for potential use in the mining process) or for treatment prior to releasing it to the environment.

18.7.3 Mine Water Pond

A mine water pond will be constructed to the south of the TSF. The mine water pond will be constructed during the initial construction utilising the same general fill as the TSF embankments. The facility will be lined with a geomembrane lining system consisting of 80 mil HDPE geomembrane underlain by a 12 oz/yd^2 (407 g/m^2) non-woven geotextile cushion layer placed overtop of a prepared subgrade. The facility will be equipped with an overflow spillway, freeboard allowances, instrumentation, and a collection drain.

18.7.4 Mine Dewatering

The dewatering system includes pumps and piping required to maintain dry working conditions in the mine area. At Goliath, the pumps are electric and will lift the water to the pit rim and then pump it horizontally to the mine water pond. For Goldlund and Miller, the pumps are diesel, but they follow the same principle of pumping the water to the pit rim and then horizontally to the settling ponds.

19 MARKET STUDIES & CONTRACTS

Treasury Metals has not conducted a market study in relation to the gold metal that will be produced from the project. Gold is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. Prices are usually quoted in USD dollars per troy ounce. The gold doré refining agreement will be negotiated once the project is approved for construction.

20 ENVIRONMENTAL STUDIES, PERMITTING & SOCIAL COMMUNITY IMPACTS

20.1 Summary

Treasury Metals has engaged a number of technical consultants to collect baseline environmental data for the Goliath, Goldlund and Miller projects, collectively referred to as the "Goliath Gold Complex". The objective of the work completed, underway or planned is to characterise the existing physical, biological, and human environment at each of the three project locations, expanding on existing information where available. In all cases, the work has/will apply standard field protocols and scientific methodologies, and will address the anticipated information needs of regulatory agencies for the approval of Ontario mining projects.

The approach to environmental studies, permitting and approvals, and impact assessment for the Goliath Gold Complex will be to treat the Goliath, Goldlund and Miller deposits as three distinct projects. The overall schedule for the Goliath Project is ahead of the Goldlund and Miller project schedules, given that a Federal Environmental Assessment (EA) has already been completed for Goliath. Specifically, on August 19, 2019, Treasury Metals received federal government approval under the *Canadian Environmental Assessment Act, 2012* (CEAA, 2012) for the Goliath Project, with the Minister of Environment and Climate Change Canada concluding that the project is not likely to cause significant adverse environmental effects. Potential benefits of the project are expected to include employment and business opportunities, as well as tax revenues at all levels of government.

The Goliath Project presented in this PEA is similar to the previous PEA, but differs in that the Goliath Project processing facility is proposed to accept ore from other deposits (specifically deposits from the Goldlund and Miller properties). Pending regulatory guidance otherwise, it is not anticipated that the optimisation of the Goliath Project design would affect the current Federal EA approval of the Goliath Project or trigger an Impact Assessment under the *Impact Assessment Act* for a mining expansion. Therefore, while this engineering design change is not anticipated to have an effect on the current Federal EA approval on the Goliath Project, additional environmental data may need to be measured or modelled to support the change in the description of the assessed project. Additional environmental programs for the Goliath Project may also be required to update environmental baseline data relied upon in the EA to support permitting efforts.

Baseline data collection for the Goldlund Project is underway and is expected to be completed within 12 months. Treasury Metals has not collected any baseline data from the Miller Project to date; however, it is assumed this will begin in the immediate near future. Based on the currently proposed design, neither the Goldlund Project nor the Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*. However, baseline data for these projects will be required to support provincial permitting and approvals processes, including potential provincial EAs.

20.2 Environmental Setting

The Goliath Project is located in the Kenora Mining Division in northwestern Ontario, approximately 4 km northwest of the Village of Wabigoon, 20 km east of Dryden and 2 km north of the Trans-Canada Highway 17. The Goldund and Miller projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath Project, just off Highway 72.

The projects are located within the Treaty 3 area of Ontario and it has been shared with Treasury Metals that there are areas within the Goliath Gold Complex property boundaries that are used by Indigenous communities for traditional land and resource use.

The Goliath Gold Complex sites are located within the Ontario Shield Ecozone and the Lake Nipigon Ecoregion. The area is generally characterised as a Black Spruce forest, with dominant woody vegetation that includes White Spruce, Balsam Fir, Trembling Aspen, White Birch, Tamarack and Jack Pine. The ecoregion is also characterised by abundant wetlands, ponds, lakes and rivers.

Additional details on the environmental setting of the Goliath Project that were used to support the Federal EA process are provided in Section 20.5.

20.3 Regulatory Framework

20.3.1 Environmental / Impact Assessment

Most mining projects in Canada are reviewed under one or more EA processes whereby design choices, environmental impacts and proposed mitigation measures are compared and reviewed to determine how best to proceed through the environmental approvals and permitting stages. Entities involved in the review process normally include government agencies, municipalities, Indigenous groups, various interested parties, and the public.

The Goliath Project has already completed a Federal EA approval process. On August 19, 2019, Treasury Metals received approval from the federal government under CEAA (2012) for the Goliath Project, with the Minister of Environment and Climate Change Canada concluding that the project is not likely to cause significant adverse environmental effects. Additional details of the EA process and approval are provided below. Based on current proposed design, neither the Goldlund Project nor the Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*; however, EA approvals under Provincial EA frameworks are anticipated to be required.

20.3.1.1 Federal Environmental Assessment of the Goliath Project

Treasury Metals submitted a final Environmental Impact Statement (EIS) for the Goliath Project to the Canadian Environmental Assessment Agency (the Agency) in April 2018 for approval consideration under CEAA (2012). The Goliath Project was defined in the EIS as an open pit and underground gold mine and associated infrastructure, with an ore input capacity of 3,240 t/d and an anticipated mine and mill life of 12 years.

The Goliath Project was subject to CEAA (2012) regulation because it involves activities described as follows:

- Item 16 (c): a new rare earth element mine or gold mine, other than a placer mine, with an ore production capacity of 600 tonnes per day or more.

With support from Treasury Metals and their consultants, Indigenous communities, and technical reviewers, the Agency identified key mitigation and follow-up program measures that would prevent or reduce potential adverse effects, verify the accuracy of the environmental assessment predictions, and verify the effectiveness of mitigation measures. The Agency concluded that the Goliath Project is not likely to cause significant adverse environmental effects, taking into account the implementation of key mitigation measures. The mitigation measures and EA report were provided to the Minister of Environment and Climate Change (the Minister) who established conditions as part of the Decision Statement. The positive Decision Statement and associated conditions indicating that the Goliath Project may proceed were issued to Treasury Metals on August 19, 2019.

While the Agency carried out a Federal EA of the Goliath Project in accordance with CEAA (2012) and the project was not subject to a Provincial Class EA, the following provincial ministries provided support to the Agency upon request on areas within their expertise and within the scope of their regulatory roles: Ministry of Natural Resources and Forestry (MNRF); Ministry of the Environment, Conservation and Parks (MECP); Ministry of Tourism, Culture and Sport; and Ministry of Energy, Northern Development and Mines (ENDM).

The conditions outlined in the Minister's Decision Statement dated August 19, 2019 are legally binding and have been considered as part of the decision-making framework relied on within this PEA. Additionally, Treasury Metals is aware it is the Agency's expectation that all of the commitments Treasury Made as part of the EA process will be implemented, such that the Goliath Project can be executed in a careful and precautionary manner.

20.3.1.2 Recent Changes to the Federal Environmental Assessment Process

Shortly after the Minister issued the Decision Statement approving the Goliath Project to proceed, the Government of Canada enacted a new *Impact Assessment Act* that superseded CEAA, 2012. The Goliath Project received its Federal EA approval under CEAA, 2012 on August 19, 2019, and the new *Impact Assessment Act* came into effect on August 28, 2019.

It is noted that the Goliath Project, as described in this current PEA, would no longer meet the requirements of the Federal EA / Impact Assessment process under the new *Impact Assessment Act*, as the new Act states:

- Item 18 (d): a new metal mill, other than a uranium mill, with an ore input capacity of 5,000 t/day or more.

Furthermore, it is noted that the Goliath Project, as described in this current PEA, does not meet the requirements of requiring an Impact Assessment for an expansion as per the new *Impact Assessment Act*, as the new Act states:

- Item 19 (d): in the case of an existing metal mill, other than a uranium mill, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore input capacity would be 5,000 t/day or more after the expansion.

Finally, based on the requirements of the new *Impact Assessment Act*, and the current engineering design outlined within the PEA, a Federal impact Assessment is not expected to be required for the Goldlund or Miller projects, as the ore input threshold (5,000 t/d) will not be met. Instead, EA approvals under Provincial EA frameworks are anticipated to be needed.

20.3.1.3 Potential Provincial EA Process for the Goldlund & Miller Projects

The Goldlund Project and Miller Project may require completion of one or more Provincial environmental assessment processes pursuant to the *Ontario Environmental Assessment Act*, depending on the final project designs. Based on the preliminary designs, it is anticipated that there would only be a requirement for a Class Environmental Assessment(s) for Resource Stewardship and Facility Development Projects, subject to regulatory confirmation.

Based on the current proposed design, neither the Goldlund Project nor the Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*.

20.3.2 Federal / Provincial Approvals

Three primary provincial agencies will be involved with additional permitting and approvals for the Goliath Gold Complex: MNRF, MECP and ENDM. Additional agencies that may be involved in permitting include the Ontario Energy Board (OEB), Ministry of Transportation, and Ministry of Tourism, Culture and Sport (MTCS).

Provincial environmental approvals that are expected to be required to construct and operate each of the three projects include (but are not limited to) those shown in Table 20.1 on the following page.

20.3.3 Regulatory Schedule

As previously stated, the overall schedule for the Goliath Project is ahead of the other projects, given that a Federal EA has already been completed. The approach to environmental studies, permitting and approvals, and impact assessment for the Goliath Gold Complex will has been to treat the Goliath, Goldlund and Miller deposits as three distinct projects to reflect that each of the three project sites are currently at different stages of environmental baseline completeness. This approach is intended to ensure that the most efficient environmental approvals and permits schedule can be achieved for each project.

Table 20.1: Expected Provincial Environmental Approvals & Permits

Agency	Permit/ Approval	Act	Relevant Components
MNRF	Various Work Permits for Construction	*Lakes and Rivers Improvement Act/Public Lands Act*	For work/construction on Crown land. May be required as part of construction of transmission lines if required for Miller/Goldlund
MNRF	*Lakes and Rivers Improvement Act* (LRIA) Permit	*Lakes and Rivers Improvement Act*	Construction of a dam in/near any lake or river in circumstances set out in the regulations requires a written approval for location of the dam and its plans and specifications.
MNRF	Forest Resource License	*Crown Forest Sustainability Act*	For clearing of Crown merchantable timber. May be required for a number of project features
MNRF	Aggregate Permit	*Aggregate Resources Act*	For extraction of aggregate (e.g., sand/gravel/ rock for tailings dam or other site construction).
MNRF	Endangered Species Permit	*Endangered Species Act*	For any activity that could adversely affect species or their habitat identified as 'Endangered' or 'Threatened' in the various schedules of the Act
MECP	Environmental Compliance Approval – Industrial Sewage Works	*Ontario Water Resources Act*	For constructing a mine/mill water treatment system(s) discharging to the environment, such as for tailings, pit water, site stormwater and mine rock pile runoff.
MECP	Permits to Take Water	*Ontario Water Resources Act*	For taking of ground or surface water (in excess of 50 m^3/day), such as for potable needs and pit dewatering. During construction, a permit(s) may be required for dam and/or mill construction to keep excavations dry
MECP	Environmental Compliance Approval – Air and Noise	*Environmental Protection Act*	For discharge of air emissions and noise, such as from mill processes, on-site laboratory and haul trucks (road dust).
MECP	Environmental Compliance Approval – Waste Disposal Site	*Environmental Protection Act*	For operation of a landfill and/or waste transfer site.
MECP	Environmental Compliance Approval	*Environmental Protection Act*	For establishment and operation of a domestic sewage treatment plant, industrial sewage treatment facility (such as minewater pond, tailings storage facility) and domestic landfill, and management of air emissions.
ENDM	Closure Plan	*Mining Act*	For mine construction/production and closure, including financial assurance, inclusive of on land dams, such as for a tailings storage facility.
MTCS	Clearance Letters	*Heritage Act*	For confirmation that appropriate archaeological studies and mitigation, if required, have been completed.
OEB	Leave to Construct	*Ontario Energy Board Act*	For approval to construct a transmission line.

20.4 Community Relations & Engagement

20.4.1 Indigenous Communities/Partners

Treasury Metals is committed to working collaboratively with Indigenous and regional communities to ensure informed and engaged dialogue throughout the life of the project. To date, Treasury Metals has participated in meaningful consultation and engagement activities with the following Indigenous communities:

- Wabigoon Lake Ojibway Nation
- Eagle Lake First Nation
- Asubpeeschoseewagong Netum Anishinabek (Grassy Narrows First Nation)
- Métis Nation of Ontario
- Naotkamegwanning First Nation
- Wabauskang First Nation
- Lac Des Mille Lacs First Nation
- Lac Seul First Nation
- Aboriginal People of Wabigoon
- Grand Council Treaty #3

Treasury Metals will endeavour to maximise participation with its Indigenous partners wherever possible. Treasury Metals is focused on building and strengthening relationships, integrating traditional knowledge into its decision-making frameworks, and actively communicating and sharing information in a transparent manner via phone calls, meetings, letters, delivery of reports and presentations. As part of the Federal EA Approval on the Goliath Project, Treasury Metals made several firm commitments to its Indigenous partners regarding consultation and engagement, which may also be extended to the Goldlund Project and Miller Project.

20.4.2 Non-Indigenous Stakeholders

In addition to the Indigenous partners list above, non-Indigenous public interest groups have been identified as part of past, present and future consultation and engagement efforts. This includes the Village of Wabigoon, City of Dryden, Town of Sioux Lookout, and other regional industrial partners and stakeholders.

20.5 Environmental Studies / Description of the Environment

The following description of the environment summarises baseline studies conducted to date with an emphasis on the Goliath Project. Final reports from environmental baseline programs at the Goldlund Project are not yet available, as these studies are currently underway. Additionally, Treasury Metals intends to initiate baseline environmental work at the Miller Project in 2021. All environmental baseline studies will continue in consultation with interested stakeholders and Indigenous communities. The following sub-sections outline the results from the studies of the Goliath Project to date.

20.5.1 Hydrology

The project is located east of Thunder Lake and northeast of Wabigoon Lake, and sits within sub-watersheds that drain to either Thunder Lake or Wabigoon Lake. Thunder Lake ultimately discharges to Wabigoon Lake via Thunder Creek. The sub-watersheds surrounding the project Site include Thunder Lake Tributaries 2 and 3, Hoffstrom's Bay Tributary, and Little Creek in the Thunder Lake watershed, and Blackwater Creek in the Wabigoon Lake watershed.

A perimeter runoff and seepage collection system will be constructed around the operations area at the start of the site preparation and construction phase to collect runoff and seepage. As a result, runoff from portions of the Hoffstrom's Bay Tributary and Little Creek catchments will no longer drain to Thunder Lake, but will be collected, used in the process, and ultimately treated and discharged to Blackwater Creek. Blackwater Creek and its tributaries provide low-gradient stream habitat punctuated by active and inactive beaver dams and ponds.

The measurement of hydrolytic flow at some stations along Blackwater Creek were indicative of the challenges associated with accurately measuring continuous streamflow in small, low-gradient runoff- dominated systems that experience frequent beaver impoundments. During operations, fresh water required in the process will be withdrawn from the pre-existing ponds located on Thunder Lake Tributary 2 and Thunder Lake Tributary 3. Both of these tributaries are located within the Thunder Lake Tributary 2 catchment area that eventually drains to Thunder Lake.

20.5.2 Hydrogeology

Groundwater levels at the Goliath Project are relatively close to surface and approximately follow the topography. Groundwater flow from the project site follows the surface drainage with flow both to the west towards Thunder Lake and to the south towards Wabigoon Lake. Groundwater provides minimal baseflow to creeks in the immediate vicinity of the project site and for much of the project area. The creeks in the area of the proposed project are runoff dominated. Groundwater baseflow represents a small proportion of the total flow in the surface watercourses near the project. Most of the groundwater flow that occurs around the project site is expected to follow the topography with greatest flows along the contact between the upper weathered and fractured bedrock and the basal sand. Rates of groundwater flow are expected to be much lower in the deeper bedrock.

20.5.3 Air & Noise

The Goliath Project is located in a mostly forested area between the communities of Dryden and Wabigoon and north of Highway 17. The site is at least 10 km from any existing sources of significant air emissions. There are several aggregate operations on the east side of Airport Road in Dryden. The town of Dryden, located approximately 15 km to the west, is home to a kraft pulp mill operated by Domtar, which would contribute to the background air quality in the area, primarily due to emissions from the natural gas and wood-waste fired boilers, recovery boiler, and lime kiln. Due to the distance between sources at the Domtar pulp mill, the aggregate operations and the project site, significant interaction between these sources and the expected emissions from the project are expected to be minimal.

Noise levels in the vicinity of the Goliath Project site reflect a rural sound environment, and are generally characterised by sounds of nature and minimal road traffic.

20.5.4 Geochemistry

The geochemical characterisation program competed as part of the baseline programs in support of the Federal EA included a suite of static and kinetic tests to evaluate short-term static conditions and long-term potential for acid rock drainage and metal leaching. Characterisation methods for the various project components (i.e., overburden, waste rock, ore) included static and kinetic geochemical characterisation tests. This includes acid-base accounting, whole rock metals, shake flask extraction, humidity cell tests and field cell tests. The geochemical program for the Goliath Project is currently being updated to satisfy the conditions on the approval of the Goliath Project.

For the purposes of the Federal EA and determination of mitigation/management measures, it was assumed both waste rock and tailings were potentially acid generating and hat the time to acid onset was quite rapid (< 2 years). Additional geochemistry baseline work is currently being completed to evaluate the changes in ARD and ML potential in the tailings should the Goliath Project process ore from the Goldlund and Miller deposits.

20.5.5 Surface Water & Groundwater Quality

Baseline surface water and groundwater quality at the Goliath site generally meets Ontario Provincial Water Quality Objectives for the protection of aquatic life with occasional exceedances of a few parameters, including total iron. Such exceedances are not unusual due to the metal-rich nature of the bedrock of the Canadian Shield region. Groundwater flows generally southwesterly, from the elevated wetland to the north, then splitting off in the general vicinity of the project study area to the south towards Wabigoon Lake and to the west towards Thunder Lake.

On a regional level, mercury concentrations in the English-Wabigoon river system (downstream of the Goliath Project) are elevated, and the production of methylmercury has caused adverse effects on human health to members of Asubpeeschoseewagong Netum Anishinabek (Grassy Narrows First Nation). Treasury Metals is mindful of the sensitivities regarding mercury in the regional area and has made firm commitments regarding effluent discharge as part of the Federal EA approval for the Goliath Project to ensure that the environment and human health are protected.

20.5.6 Biological Environment

The Goliath Project is located within the Canadian Shield in the west-central portion of a hydrological basin containing low to moderate relief topographic features, including low-lying wetlands and marsh type lands, exposed bedrock ridges and a range of boreal forest types. Among avian species in the area are the olive-sided flycatcher, the bald eagle and the Canada warbler. Several large mammals and furbearers also characterise the area, including moose, white-tailed deer, black bear, American beaver, red fox and snowshoe hare. Thunder Lake is a cold-water lake that supports a fish community including Lake Trout, Lake Whitefish, Walleye, Northern Pike and Smallmouth Bass. It has several areas of spawning habitat for Lake Whitefish and Lake Trout. Thunder Lake supports both recreational and commercial fishing. Wabigoon Lake is a cool-water lake. In particular, there are two fish sanctuaries on Wabigoon Lake created to protect spawning Walleye and Sauger. Wabigoon Lake supports an active sport fishery focused on Walleye and Muskellunge angling.

20.5.7 Human Environment

The Goliath Project is located only 20 km east of the City of Dryden, Ontario, which has a population of approximately 8,000 people. The project is located in an area used by the public for recreational fishing, hunting, boating, and commercial activities including tourism, fishing, trapping, and wild rice and bait harvesting. For example, Thunder Lake is popular for fishing and hiking trails, and snowmobile trails exist in the area.

20.5.8 Traditional Land & Resource Use

As previously mentioned, the Goliath Project is located within the Treaty 3 (1873) area of Ontario, which affords hunting, trapping and fishing rights and protections to its signatories throughout the Treaty territory. The Indigenous communities nearest to the project are Eagle Lake First Nation, Wabigoon Lake Ojibway Nation and Lac Suel First Nation.

Information regarding traditional land and resource use that was shared with Treasury Metals throughout the EA process was included wherever possible in the Federal EA for the project. Formal traditional knowledge and traditional land and resource use studies for the project were provided to Treasury Metals by Eagle Lake First Nation the Métis Nation of Ontario. While the specific details of these studies are confidential, it can be confirmed that there is overlap of the impacts and effects of the Goliath Project with areas currently used by members of Indigenous communities for hunting large game, non-commercial fishing and gathering of plant material.

20.6 Environmental Monitoring

Based on the Federal EA process, Treasury Metals has established, and committed to, a preliminary environmental monitoring program for the Goliath Project site. The existing environmental baseline monitoring programs conducted to date provide the basis for the monitoring frameworks and may be modified to meet compliance and reporting requirements as the project moves through the permitting phase. The proposed monitoring programs will apply to the construction, operation, closure and post-closure phases of the project, as appropriate, and will allow for compliance of activities with anticipated environmental approvals and permits, while providing information to determine the effectiveness of proposed mitigation measures. Environmental monitoring programs will also be established for the Goldlund Project and Miller Project, commensurate with the scale of those projects and potential environmental effects.

20.7 Preliminary Closure Planning

All three projects in the Goliath Gold Complex will be required to complete Regulatory Closure Plans as per the requirements of Ontario Regulation 240/00: Mine Development and Closure Under Part VII of the *Ontario Mining Act*, prior to commencement of construction activities. The *Ontario Mining Act* requires that a closure plan be filed for any mining project before the construction of the project is initiated. At the same time, it also requires that financial assurance be submitted in the form of sufficient funds to support the activities required by the closure plan.

Table 20.2 summarises a preliminary closure approach for the Goliath Gold Complex. Reclamation work will be completed progressively during operations as reasonable, which is considered an industry best management practice.

Table 20.2: Summary of Preliminary Closure

Project Element	Preliminary Approach
Open Pits (Goliath, Goldlund and Miller)	• Remove all infrastructure and equipment within the open pit. • Revegetate overburden slopes to a stable condition. • Allow pit to fill naturally by means of seepage and runoff inputs from the local area. • Ramps will be barricaded and a safety berm established around the perimeter. • Channels will be constructed from open pits for future passive overflow if needed.
Underground Mine (Goliath)	• Remove all infrastructure and equipment with value within the underground workings. • Allow workings to flood naturally via groundwater seepage. • Seal the entrances to underground.
Buildings, Machinery, Equipment and Infrastructure (primarily Goliath, but also Goldlund and Miller)	• Salvageable machinery, equipment and other materials will be dismantled and taken off site (sale or reuse). • Remaining items managed according to regulatory requirements at the time, either within an approved landfill at the Goliath site or at a licensed facility off site. • Above grade concrete structures will be broken and reduced to near grade with rebar and will be cut flush with the surface. • Concrete structures will be infilled with clean mine rock, if needed.
Chemicals (primarily Goliath, but also Goldlund and Miller)	• All petroleum products, chemicals and explosives remaining will be removed from the site and transported to a licensed facility for disposal. • Soil found to exceed acceptable criteria will be handled otherwise according to regulatory requirements.
On-site Infrastructure (primarily Goliath, but also Goldlund and Miller)	• Roads not needed in the longer term will be scarified and seeded. • On-site power lines and associated materials without value will be dismantled and deposited in an approved landfill.
Tailings Storage Facility and Ponds (Goliath)	• Tailings surface will be covered with a permanent cover that limits oxygen • Overflow spillways within the dams will be deepened, if needed. • Mine water pond will be breached and stabilised.
Overburden / Non-potentially Acid-Generating Mine Rock Stockpiles (Goldlund and Miller)	• Stockpiled overburden will be used for reclamation on site, as needed, including as a cap over the potentially acid generating mine rock. • Remaining stockpile will be revegetated once no longer required. • Flat surfaces of the non-acid generating rock will be covered with a growth material and revegetated.
Potentially Acid-Generating Mine Rock Stockpile (Goliath)	• Drainage will continue to be directed to the open pit. • Stockpile will be encapsulated within a low permeability cover. • The cover will be seeded.

21 CAPITAL & OPERATING COST

The capital and operating cost estimates presented for the Goliath Gold Complex are based on open pit and underground mining of the Goliath deposit, open pit mining of the Goldlund deposit, open pit mining of the Miller deposit, and the construction of a process plant, tailings storage facility, and requisite supporting infrastructure. The processing plant design point daily throughput is 4,932 t/d, with a mine life of 13.5 years.

The purpose of the capital estimate is to provide substantiated costs which can be used to assess the preliminary economics of the project. All capital and operating cost estimates are reported in Canadian dollars for this PEA.

21.1 Capital Costs

The capital cost estimate was developed in Q4 2020 from Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50% in accordance with the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes mining, processing, utilities, TSF and project site infrastructure.

The capital cost summary is presented in Table 21.1. The total initial capital cost for the Goliath Gold Complex is $232.6 million and LOM sustaining costs are $289.6 million. Closure costs are additional and are estimated at $28.5 million.

Table 21.1: Capital Cost Summary

WBS	WBS Description	Initial Capital (C$M)	Sustaining (C$M)	Total Capital (C$M)
1000	Mining (Goldlund and Miller)[1]	44.6	194.3	238.9
2000	Mining (Goliath)[1]			
3000	Process Plant	64.9	1.4	66.3
4000	On-site Infrastructure	49.9	70.9	120.8
5000	Off-site Infrastructure	0.6	-	0.6
	Directs	**160.0**	**266.6**	**426.6**
6000	Project Indirects	9.6	-	9.6
7000	Project Delivery	26.1	-	26.1
8000	Owner's Cost	7.1	-	6.8
9000	Provisions (Contingency)	29.8	22.9	52.7
	Total Project Cost	**232.3**	**289.6**	**522.2**

Notes: [1]Mining costs have been calculated considering shared capital expenditures among projects. Source: Ausenco (2021).

The cost estimate is based on an engineering, procurement and construction management (EPCM) implementation approach. The following basic information pertains to the estimate:

- base date is Q4 2020
- expressed in Canadian dollars (C$ or CAD)
- currency exchange rate US$0.75:C$1.00

- accuracy is ±50%
- no allowance has been made for exchange rate fluctuations

Data for the estimates have been obtained from numerous sources, including:

- mining schedule
- conceptual engineering design by Ausenco, Knight-Piésold Consulting, and AGP Mining Consultants
- historical pricing data from similar projects in the Eastern Canada region
- in-house benchmarking data from similar projects in the Eastern Canada region
- topographical information

21.1.1 Capital Cost Estimate Responsibilities

The capital cost estimate was developed in accordance with the responsibility breakdown presented in Table 21.2.

Table 21.2: Estimate Responsibility Summary

WBS	Description	Responsible Company / Consultant
1000	Mining (Goldlund and Miller)	AGP Mining
2000	Mining (Goliath)	AGP Mining
3000	Processing	Ausenco
4000	On-site Infrastructure	Ausenco[1]
5000	Off-site Infrastructure	Ausenco
6000	Project Indirects	Ausenco
7000	Project Delivery	Ausenco
8000	Owner's Costs	Ausenco
9000	Provisions: Contingency	Ausenco
	Site Closure and Monitoring	Wood

Notes: [1]WBS 4700 Tailings Storage Facilities by Knight Piésold, Source: Ausenco (2021).

21.1.2 Basis of Capital Cost

21.1.2.1 Direct Costs – Mining (WBS 1000 & 2000)

The mining capital cost estimate is grouped into two main categories: open pit capital costs and underground capital costs. These costs are summarised in Table 21.3.

Table 21.3: Mining Capital Cost Estimate

Mining Capital Category	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Open Pit Capital	44.6	14.4	59.0
Underground Mining Capital	-	179.9	179.9
Total	**44.6**	**194.3**	**238.9**

21.1.2.1.1 Mining Capital Costs – Open Pit

Open pit mining capital includes costs associated with open pit mining and haulage of mill feed from Goldlund, Miller and Goliath. The mining equipment fleet is leased, so the capital cost for equipment reflects the cost of initial down payments. The financing portion of the cost is included in the operating cost estimate.

The open pit capital cost estimate is subdivided into three main categories:

- pre-production stripping costs
- mining equipment capital
- miscellaneous mine capital

The cost breakdown of these categories is shown in Table 21.4.

Table 21.4: Mining Capital Cost Estimate

Mining Capital Category	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Pre-Production Stripping	25.2	-	25.2
Mine Equipment Capital	14.6	9.9	24.5
Miscellaneous Mine Capital	4.8	4.5	9.3
Total	**44.6**	**14.4**	**59.0**

Pre-Production Stripping

Mining activity commences in advance of the process plant achieving commercial production. Pre-production mining occurs at Goliath first with the movement of 5.7 Mt of waste and placement of 0.8 Mt of mill feed in stockpiles adjacent to the primary crusher. Mine operating costs associated with this period are included in the capital cost estimate and are expected to be $25.2 million. This amount covers all associated management, dewatering, drilling, blasting, loading, hauling, support, engineering and geology departments labour, grade control costs and financing costs for that period.

Mining activities during this period include developing the initial phase, widening/improving haul roads, initiating the waste dump, and stockpiling mill feed.

Mining Equipment Capital

Mining equipment capital costs reflect the cost of financing major equipment and some support equipment. Equipment prices are based on current quotations from local vendors. A 20% down payment is included in the capital cost for those units financed; the remaining cost is included in operating costs, discussed in Section 21.2.1.1.10.

Base costs provided by the vendors are included for each unit cost calculation and options. The capital cost, cost of financing, and down payment amounts are shown in Table 21.5. Note that the trailer units for the mill feed haulage fleet were capitalised only.

Table 21.5: Major Mine Equipment – Capital Cost, Full Finance Cost & Down Payment

Equipment	Unit	Capacity	Capital Cost (C$)	Full Finance Cost (C$)	Down Payment (C$)
Production Drill	mm	140	1,143,000	1,204,000	235,000
Production/Crusher Loader	m^3	13	2,516,000	2,651,000	503,000
Production/Support Excavator	m^3	6.7	2,323,000	2,448,000	465,000
Haulage Truck	t	91	1,799,000	1,895,000	360,000
Track Dozer	kW	474	1,721,000	1,813,000	344,000
Grader	kW	163	468,000	493,000	94,000
Mill Feed - Loader	m^3	7.8	1,088,000	1,146,000	218,000
Mill Feed – Tractor Unit	kW	380	220,000	232,000	44,000
Mill Feed – Belly Dump Trailer	t	40	153,000	-	-

The cost of spare truck boxes and loader buckets is included in the major equipment capital cost to take into consideration spares that will be required due to wear.

The distribution of capital costs is completed using the number of units required within a period. If new or replacement units are needed, that number of units, by the unit cost (20% of that for major equipment) is applied to the capital cost for that period. There is no allowance for escalation in any of these costs.

The balancing of equipment units based on operating hours is completed for each major piece of mine equipment. The smaller equipment was based on an assumed number of units required from operational experience. This includes such things as pickup trucks (dependent on the field crews), lighting plants, mechanics trucks, etc. The project considered additional support equipment for road maintenance as the mill feed haulage road would be the mine's responsibility.

The most significant piece of major mine equipment is the haulage trucks. At the peak of mining activities from Year 2 onwards, it is estimated that 11 units will be necessary to maintain mine production at the various sites. The maximum number of hours per truck per year is set at 6,000. There are periods where the maximum hours per unit are below what the maximum possible can be. In those situations, increasing the maximum on the number of trucks still leaves residual hours required to complete the material movement; therefore, the number of total trucks is unchanged. In these cases, the hours required are distributed evenly across the number of trucks on site and available.

As open pit mining starts winding down in Year 5, some of the trucks are sold and the money is brought into the cash flow. Three trucks will be sold in Year 5 and another three trucks in Year 6. That will leave a fleet of five trucks.

The other major mine equipment is determined in the same manner. In some instances the loaders have a longer period of life (same number of hours between replacements) due to the sharing of hours with the other units in the fleet.

The support equipment is usually replaced after a certain number of years. For example, pickup trucks are replaced every three years, with the older units possibly being passed down to other departments on the mine site, but for capital cost estimating, new units are considered for mine operations, engineering, and geology.

It is considered that equipment would be moved between pit areas in reasonable blocks of time. The larger equipment could be scheduled to drive down the highway with permission at predetermined times. The smaller equipment (dozers, drills) would be hauled with a low bed over the highway to site.

An allowance in equipment numbers has been made to have dozers present at each site for the minimum of activities. They would also be used for reclamation.

Unique to this mine operation is a mill feed haulage fleet. This includes a smaller loader (7.8 m^3) responsible for loading a fleet of highway trucks with belly dump trailers. They would transfer the mill feed from Goldlund and Miller to the Goliath plant and stockpiles. The intent is they would travel the highway for a portion of the trip and then traverse the countryside in a more direct route to the Goliath site to avoid disturbance to the various communities. The primary crusher would have an area adjacent to the main truck dump for unloading these units to then feed into the primary crusher with a small conveyor.

The amount of major equipment required by year is shown in Table 21.6.

Table 21.6: Mine Equipment on Site

Equipment	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10	Y 11	Y 12	Y 13	Y 14
Production Drill	2	3	3	3	3	3	4	4	3	1	-	-	-	-	-
Production Loader	1	3	3	3	3	3	3	3	3	3	3	3	3	1	1
Production Excavator	1	2	2	2	2	2	2	2	2	2	-	-	-	-	-
Haulage Truck	4	8	11	11	11	8	5	5	5	5	5	5	5	5	5
Track Dozer	3	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Grader	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Mill Feed – Loader	-	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Mill Feed – Tractor Unit	-	11	11	11	11	11	11	11	11	11	-	-	-	-	-
Mill Feed – Belly Dump Trailer	-	12	12	12	12	12	12	12	12	12	-	-	-	-	-

One full-time production loader will be positioned at the primary crusher when the plant commences operation. Its role will be to tram/load material from stockpiles and manage the blending of various mill feed types. It also serves as a backup loader for the Goliath pit.

The support excavator is a larger unit meant to clean mill feed hanging wall and footwall waste and windrow the material for loading by the production loader or load the trucks as needed.

The predicted lifespan of the major equipment is provided below:

Production drill .... 25,000 hours

Production loader .... 35,000 hours

Production excavator .... 35,000 hours

Haulage truck .... 72,000 hours

Track dozer .... 35,000 hours

Grader .... 25,000 hours

Mill feed loader........ 5 years

Mill feed tractor .... 5 years

Mill feed trailer.. 5 years

The replacement schedule for support equipment varies according to the duty of each piece of equipment. For example, light plants are replaced every four years, while the integrated tool carrier for site support is only purchased once at the start of the project and does not need further replacement.

The mine also purchases a lowbed and tractor to move drills, dozers and other small equipment between the pit areas.

Miscellaneous Mine Capital

Miscellaneous mine capital includes various separate line items in the cost estimate. These are shown in Table 21.7.

Table 21.7: Miscellaneous Mine Capital

Miscellaneous Mining Capital	Initial Cost (C$)	Sustaining Cost (C$)	Total Capital Cost (C$)
Engineering Office Equipment	600,000	600,000	1,200,000
Dispatch System	572,000	-	572,000
Communications	200,000	-	200,000
Goliath Dewatering System – Pumps/Pipe	-	1,086,000	1,086,000
Goldlund Dewatering System – Pumps/Pipe	-	1,784,000	1,784,000
Goliath Initial road to pit	40,000	-	40,000
Mill Feed Haul Road – Goldlund to Highway	111,000	-	111,000
Mill Feed Haul Road – Highway to Goliath	577,000	-	577,000
Goliath Pit Area – Clear/Grub	473,000	-	473,000
Goliath Dump Area – Clear/Grub	591,000	-	591,000
Goldlund Pit Area – Clear/Grub	264,000	264,000	528,000
Goldlund Dump Area – Clear/Grub	382,500	382,500	765,000
Miller Pit Area – Clear/Grub	-	103,000	103,000
Miller Dump Area – Clear/Grub	-	260,000	260,000
Total	**3,810,500**	**4,479,500**	**8,290,000**

Engineering office equipment includes desktop computers, plotter, mining and geology software, and survey equipment with associated peripherals. This cost is estimated at $1.2 million in the early years of the mine start-up with the majority being the mining/geology software.

The dispatch system will utilise an iPad-based system with Wi-Fi in the pit area. This system will provide checklists and truck routing in addition to data collection.

The communication system involves establishing radio/cell coverage complete with lightning protection in the pit areas for use by mine engineering and operations.

The dewatering system includes pumps and piping required to maintain dry working conditions in the mine area. At Goliath, the pumps are electric and will lift the water to the pit rim and then pump it horizontally to the mine water pond. For Goldlund and Miller, the pumps are diesel, but they follow the same principle of pumping the water to the pit rim and then horizontally to the settling ponds.

Various roads will need to be constructed prior to the start of mining. For Goliath, a pioneer road needs to be built to the pit which will be expanded with mine material. Other roads are associated with hauling mill feed material from Goldlund to Goliath. The road from the transfer pad area to Highway 72 is 3.7 km long. This will need to be upgraded to handle the increased haul truck traffic expected to move mill feed and equipment between Goliath and Goldlund.

The other portion of the road construction/upgrade is from Highway 72 to the Goliath pit. This road is 12.8 km long and will require upgrading the existing road and placing culverts over low-lying areas. The road will go through previously logged zones and will utilise existing roads that will be upgraded.

The pit and waste dump locations need to be cleared by removing merchantable timber, grubbing, and removing/stockpiling the topsoil. These activities have been estimated to cost approximately $8,000/ha.

21.1.2.1.2 Underground Mining Capital

Underground mining capital comprises costs associated with underground development at Goliath. The underground mining equipment fleet will be leased, so the capital estimate reflects the cost of initial down payments. The financing portion of the cost is included in the operating cost estimate.

The underground capital cost estimate is subdivided into three main categories:

- capital development
- mobile equipment fleet
- underground mine infrastructure

As the underground is developed in Year 3, its associated capital cost is considered to be a sustaining capital cost for the project. The cost breakdown is shown in Table 21.8.

Table 21.8: Underground Mine Capital

Underground Mining Capital	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Capital Development	-	136.1	136.1
Mobile Equipment Fleet	-	15.8	15.8
Underground Mine Infrastructure	-	28.0	28.0
Total	**-**	**179.9**	**179.9**

Capital Development

For the study, capital development is defined as all lateral and raise development in waste. Vein development, including non-mineralised areas, was classified as operating development. Capital development starts in Year 3 and continues until Year 9 with the bulk of the cost being expended in the first four years.

Waste development in metres and kilotonnes is summarised in Table 21.9.

Table 21.9: Waste Development in Metres & Kilotonnes

Activity	Units	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11
Waste Development	m	3,777	4,787	5,582	3,981	2,451	1,380	366	-	-
Waste Development	kt	284.7	323.1	376.8	268.7	165.4	93.2	24.7	-	-

Mobile Equipment Fleet

The mobile equipment fleet requirements are based on operational hours. Recent quotations for other AGP projects were used for the equipment types selected. Mechanical availability and operational life were estimated by AGP for each equipment type and the hourly operating costs were assessed. A mid-life 50% rebuild was provided to achieve the indicated equipment life. No replacement of mobile equipment is required during the mine life. The equipment fleet selected and its associated base cost are shown in Table 21.10.

Table 21.10: Underground Mining Fleet

Equipment	Base Cost ($k)	Useful Life (h)
Scoop - 6.7 t	864	28,000
Scoop – 10 t	1,066	28,000
Diesel Truck – 45 t	1,455	28,000
Development Jumbo – Two Boom	1,397	25,000
Longhole Drill	1,098	25,000
Rock Bolter	1,129	25,000
Boom Truck	549	25,000
Fuel/Lube Truck	576	25,000
Shotcrete	993	25,000
Personnel Carrier – 8 person	113	15,000
Scissor Lift	563	25,000
Transmixer	728	25,000
Emulsion Loader	629	25,000
Grader	647	25,000
Toyota Runaround	94	15,000
Mechanics Runaround	200	15,000
Rescue/First Aid	123	15,000
Telehandler	281	25,000
Sanitation	568	25,000

The capital costs are comprised of an initial 20% down payment on the items purchased. Equipment leasing costs were divided between the capital development and operating cost estimates.

As the activities underground vary, the equipment fleet requirements also change. The fleet requirements for some example periods in the development and production portion of the underground mine are shown in Table 21.11 in the first quarter of the year.

Table 21.11: Underground Equipment Requirements – Select Periods

Equipment	Year 4	Year 6	Year 8	Year 10
Scoop - 6.7 t	-	1	1	1
Scoop – 10 t	2	5	5	4
Diesel Truck – 45 t	2	5	5	4
Development Jumbo – two Boom	2	3	2	1
Longhole Drill	-	2	2	2
Rock Bolter	2	3	2	1
Boom Truck	1	1	1	1
Fuel/Lube Truck	1	1	1	1
Shotcrete	1	1	1	1
Personnel Carrier – 8 person	2	3	3	3
Scissor Lift	3	4	3	2
Transmixer	1	1	1	1
Emulsion Loader	2	3	3	2
Grader	-	1	1	1
Toyota Run Around	6	8	8	6
Mechanics Run Around	1	1	1	1
Rescue/First Aid	1	1	1	1
Telehandler	1	1	1	1
Sanitation	1	1	1	1

Underground Mine Infrastructure

Underground mine infrastructure includes both underground and surface infrastructure. Various allowances and estimates for the infrastructure were made and include the following:

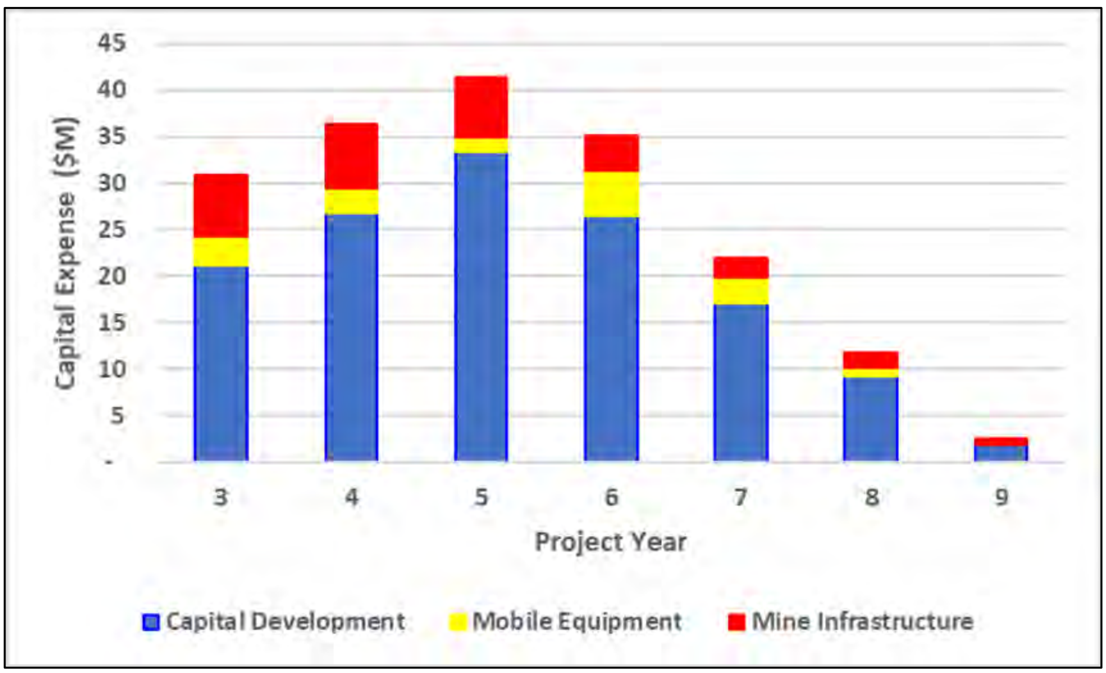
- Underground Infrastructure
  - power distribution
  - portals
  - additional support near portals
  - ventilation raise support and ladders
  - underground lighting
  - underground service water system
  - dewatering pumps and controls
  - face pumps
  - underground fuel storage
  - workshop concrete and equipment
  - rockfill crushing and screening system
  - rockfill cement addition
  - leaky feeder/communications/automation controls
  - ventilation doors, seals, and regulators

- auxiliary fans
- temporary portable 16-person refuges
- mobile compressors
- underground explosive and detonator storage
- permanent refuge stations/lunchroom/shift boss's office and toilets

- Surface Infrastructure
  - decline fans and air heaters
  - return air raise fans
  - stench gas system
  - handheld drills
  - cap lamps and self-rescuers
  - mine rescue equipment
  - portal office
  - infrastructure EPCM and indirects
  - infrastructure replacement and rehabilitation

The distribution of underground capital is shown in Figure 21-1.

Figure 21-1: Underground Mine Capital by Year



Source: AGP (2021).

21.1.2.2 Direct Costs – Process & Infrastructure

Process and infrastructure costs are summarised in Table 21.12 and described in the following sections. Direct costs include all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

Table 21.12: Goliath Gold Complex Process & Infrastructure Capital Cost Summary

WBS	WBS Description	Initial (C$M)	Sustaining (C$M)	Total (C$M)
	Processing	**64.9**		**66.3**
3100	Crushing	14.5	-	14.5
3200	Stockpiling & Reclaim	7.8	-	7.8
3300	Grinding / Gravity Gold	21.2	1.4	22.6
3400	Gravity Tails / Leach Adsorption	9.8	-	9.8
3500	Elution / Carbon Regeneration / Gold Room	4.2	-	4.2
3600	Cyanide Detox / Tailings Disposal	2.2	-	2.2
3700	Reagents Offloading and Storage	2.9	-	2.9
3800	Air / Water Services	2.1	-	2.1
3900	Oxygen Plant	0.2	-	0.2
	On-site Infrastructure	**49.9**		**109.1**
4100	Bulk Earthworks	2.7	-	2.7
4200	Power Supply	4.4	-	4.4
4300	Plant Ancillaries	1.7	-	1.7
4400	Warehousing, Office and Workshops	7.2	-	7.2
4500	Site Water Services	4.8	-	4.8
4600	Site Water Management	4.2	-	4.2
4700	Tailings Storage and Management Facilities	25.0	59.1	84.1
	Off-site Infrastructure	**0.6**		**0.6**
5100	Main Plant Access Road	0.2	-	0.2
5200	High-voltage Power Supply	0.4	-	0.4
Total Process & Infrastructure Cost		**115.4**	**60.5**	**175.9**

Source: Ausenco (2021).

21.1.2.2.1 Process Plant (WBS 3000)

The definition of process equipment requirements was based on conceptual process flowsheets and process design criteria (refer to Section 17). Each major process area has been built up with costs by separately addressing the following disciplines:

- concrete (C)
- structural steel (E)
- architectural and building (F)
- mechanical platework and tanks (L)
- mechanical equipment (M)
- piping (P)

- electrical equipment (Q)
- conduit and cable tray (R)
- wire and cable (S)
- instrumentation (T)

Mechanical equipment and building (inclusive of HVAC and lighting) supply costs were based on recent and historical budget quotes from similar projects, adjusted to reflect the Goliath Gold Complex sizing. Building costs are presented in Table 21.13.

The oxygen plant cost estimate assumes a vendor oxygen supply contract, with a plant down-payment included in the initial capital ($0.2 million) and oxygen supply costs accounted for under operating costs.

The materials and equipment total direct costs for other disciplines were developed by applying factors (percentages) to the total direct cost (supply and install) of the mechanical equipment. The factors are based on Ausenco's historical data for similar type work, and are specific to both discipline and area. The overall process plant area costs by discipline are presented in Table 21.14.

Table 21.13: WBS 3000 & 4000 Building Costs

<table>
<tr><th>WBS</th><th>WBS Description</th><th>Building Description</th><th>Building Type</th><th>Total Initial Capital (C$M)</th></tr>
<tr><td rowspan="3">3100</td><td rowspan="3">Crushing</td><td>Secondary & Tertiary Screen Building</td><td>Pre-Engineered</td><td>0.75</td></tr>
<tr><td>Secondary & Tertiary Crusher Building</td><td>Pre-Engineered</td><td>0.75</td></tr>
<tr><td>Primary Crushing Building</td><td>Pre-Engineered</td><td>0.75</td></tr>
<tr><td>3200</td><td>Stockpiling / Reclaim</td><td>Stockpile Cover</td><td>Fabric</td><td>1.5</td></tr>
<tr><td>3300</td><td>Grinding / Gravity Gold</td><td>Grinding Building</td><td>Pre-Engineered</td><td>2.5</td></tr>
<tr><td>3500</td><td>Elution / Carbon Regen / Gold Room</td><td>Gold Room</td><td>Pre-Engineered</td><td>0.26</td></tr>
<tr><td>3700</td><td>Reagents Offloading and Storage</td><td>Reagent Building</td><td>Pre-Engineered</td><td>1.5</td></tr>
<tr><td rowspan="9">4400</td><td rowspan="9">Warehousing, Office and Workshops</td><td>Truck Shop</td><td>Fabric</td><td>2.5</td></tr>
<tr><td>Laboratory</td><td>Modular</td><td>0.75</td></tr>
<tr><td>Administration Building</td><td>Modular</td><td>0.75</td></tr>
<tr><td>Truck Wash</td><td>Fabric</td><td>0.75</td></tr>
<tr><td>Mine Office & Change Room</td><td>Modular</td><td>0.75</td></tr>
<tr><td>Reagent Storage Building</td><td>Fabric</td><td>0.5</td></tr>
<tr><td>Truck Shop Warehouse</td><td>Fabric</td><td>0.5</td></tr>
<tr><td>Plant Warehouse and Maintenance Bld.</td><td>Fabric</td><td>0.5</td></tr>
<tr><td>Gatehouse</td><td>Modular</td><td>0.15</td></tr>
</table>

Source: Ausenco (2021).

Table 21.14: WBS Area 3000 (Process Plant) Total Initial Capital by Discipline

Discipline	Total Initial Capital (C$M)	% of Mechanical Equipment Total Direct Cost
Concrete (C)	7.23	28
Structural Steel (E)	3.19	12
Architectural and Building (F)	8.01	31
Mechanical Platework and Tanks (L)	5.77	22
Mechanical Equipment (M)	26.15	100
Piping (P)	4.08	16
Electrical Equipment (Q)	6.54	25
Conduit and Cable tray (R)	1.30	5
Wire and Cable (S)	1.30	5
Instrumentation (T)	1.30	5

Source: Ausenco (2021).

21.1.2.2.2 Tailings Storage Facility (WBS 4700)

The estimated capital expenditures have been developed based on the PEA-level TSF concept, 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 typical sections and details, neat line AutoCAD modelling, unit rate development, and contractor quotes from similar projects. Lump sum placeholder estimates have been applied were necessary.

The estimated capital cost includes for the following main items:

- mobilisation/demobilisation of contractors, equipment procurement, and access road construction
- earthworks costs associated with foundation preparation, material processing and embankment construction for the TSF
- earthworks costs for the mine water pond, and miscellaneous infrastructure required for the TSF operations
- installation of a seepage collection drain, collection ditches, seepage collection sumps, and pump back systems to collect potential embankment seepage and contact runoff from the embankments
- supply and installation of geotechnical instrumentation to monitor embankment performance during operations
- indirect costs associated with local borrow development and site investigations to support detailed design

The cost estimate assumes that 7% of the waste rock produced from Goliath Pit:

- is non-PAG
- can be segregated during mining
- is suitable for construction in the embankments
- will be delivered to the TSF during construction as part of the mining operations (i.e., an overhaul)
- will be available as required during the construction of the TSF

The cost estimate also assumes that general fill for the construction of the embankments will be sourced from existing pits or adjacent properties with limited sorting required.

The estimate does not include:

- cost for fish offset/compensation requirements
- mechanical systems
- engineering and permitting support
- construction management

Initial and sustaining capital costs are summarised in Table 21.15.

Table 21.15: WBS 4700 (TSF) Initial & Sustaining Capital Costs

Description	Initial Capital (C$M)	Sustaining Capital (C$M)	Total (C$M)
Mobilisation/Demobilisation	1.3	3.1	4.4
Earthworks	13.4	48.7	62.1
Geosynthetics	8.3	7.1	15.4
Instrumentation	0.3	0.2	0.5
Other Indirects	1.6	0.0	1.6
Total	**25.1**	**59.1**	**84.1**

Source: Ausenco (2021).

21.1.2.2.3 Other On-Site Infrastructure (WBS 4000)

On-site infrastructure costs were developed based on Ausenco's in-house database of costs and labour rates and include:

- WBS 4100: bulk earthworks including:
  - ROM pad earthworks
  - Truck shop pad earthworks
  - process plant pad earthworks
- WBS 4200: power supply including:
  - incoming power substation
  - site-wide distribution
  - emergency generator
  - telecommunications infrastructure and equipment
- WBS 4300: plant ancillaries including:
  - process plant mobile equipment
  - fuel storage
- WBS 4400: buildings described in Table 21.13

- WBS 4500: site water services including:
  - tails slurry pipeline
  - reclaim water pipeline
  - freshwater intake pipelines
  - effluent water treatment plant
- WBS 4600: site water management including:
  - earthworks (ditching and collection ponds)
  - pumps and sumps
  - geosynthetics

Bulk earthworks and site water management infrastructure costs were developed using semi-detailed cut-and-fill volumes based on site layout and site topographical information. Unit rates were benchmarked based on recent projects in the Eastern Canada region.

21.1.2.2.4 Off-Site Infrastructure (WBS 5000)

Off-site infrastructure costs were developed based on in-house database of costs and labour rates and include:

- WBS 5100: A 750 m light vehicle road connecting the process plant pad to Tree Nursery Road
- WBS 5200: A high-voltage overhead power line to connect the incoming power substation and tie-in to the existing 115 kV powerline crossing the Goliath site

Road works volumes were developed based on the site layout and planned road alignment, existing conditions, and site topographical information. Unit rates were benchmarked based on recent projects in the Eastern Canada region.

21.1.2.3 Other Costs

Other costs are summarised in Table 21.16 and described in the following sections.

Table 21.16: Other Costs Summary

WBS	Description	Initial Capital (C$M)	Sustaining Capital (C$M)	Total (C$M)
6000	Project Indirects	9.6	-	9.6
7000	Project Delivery	26.1	-	26.1
8000	Owners Costs	7.1	-	6.8
9000	Provisions (Contingency)	29.8	22.9	52.7
Total Indirect Capital Costs		**72.3**	**72.6**	**95.5**

Source: Ausenco (2021).

21.1.2.3.1 Project Indirects (WBS 6000)

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, initial and insurance)
- first fills and initial charges
- freight and logistics

The project indirects have been based on Ausenco's historical project costs of similar nature and have been included at a rate of 6% of the total direct cost.

21.1.2.3.2 Project Delivery (WBS 7000)

The project delivery cost has been calculated at 13% of total direct costs based on Ausenco's historical project costs of similar nature. This includes:

- Engineering, procurement and construction management services (EPCM)
- commissioning services

In addition to the 13% of total direct costs, allowances totalling C$5.3 million have been made for the following based on estimated costs by Treasury Metals:

- environment services and permitting
- a pre-feasibility study
- exploratory drilling programs on the Goliath and Goldlund-Miller properties

21.1.2.3.3 Owner's Costs (WBS 8000)

Owner's costs were factored from total directs and are 4.25% of total direct costs, or C$6.8 million, and include the following:

- project staffing and miscellaneous expenses
- pre-production labour
- home office project management
- home office finance, legal and insurance

In addition to the above, select NSR royalty advance buy-out options were considered in Owner's Costs as an additional C$0.3 million, for a total initial capital of C$7.1 million.

21.1.2.3.4 Provisions: Contingency (WBS 9000)

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.

Contingency percentages by area are summarised as follows:

- WBS 1000/2000 mining:
  - open pit capital purchases: 5%
  - underground capital development and mobile equipment: contingency of 10%
  - underground mine infrastructure: 25%
- WBS 3000/4000/5000 process plant, on-site infrastructure, off-site infrastructure:
  - initial capital total direct costs: 25%
  - sustaining capital total direct costs: 20%

The total estimated contingency is C$29.8 million for the initial capital cost estimate and C$34.7 million for the sustaining capital cost estimate. The contingency estimated over the life of the project is shown by area in Table 21.17.

Table 21.17: Project Contingency Estimate

<table>
<tr><th>WBS</th><th>Description</th><th>Initial Cost (C$M)</th><th>Sustaining Cost (C$M)</th><th>Total Capital Cost (C$M)</th></tr>
<tr><td rowspan="2">1000 / 2000</td><td>Open Pit Contingency</td><td>1.0</td><td>0.7</td><td>1.7</td></tr>
<tr><td>Underground Contingency</td><td>-</td><td>22.2</td><td>22.2</td></tr>
<tr><td>3000</td><td>Process Plant</td><td rowspan="3">28.9</td><td rowspan="3">11.8</td><td rowspan="3">28.9</td></tr>
<tr><td>4000</td><td>On-site Infrastructure</td></tr>
<tr><td>5000</td><td>Off-site Infrastructure</td></tr>
<tr><td colspan="2">TOTAL</td><td>29.8</td><td>34.7</td><td>64.5</td></tr>
</table>

Source: Ausenco (2021).

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

21.1.3 Exclusions

The following costs and scope were excluded from the capital cost estimate:

- land acquisitions

- taxes not listed in the financial analysis
- sales taxes
- operating costs
- scope changes and project schedule change 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
- any costs for demolition or decontamination for the current site
- third party costs

21.1.4 Closure Costs

Preliminary closure costs were estimated for the Goliath and Goldlund-Miller properties by Wood. The estimate considers reclamation, administrative and associated monitoring activities for each property, while recognising there are some reclamation-related interrelationships. It is assumed that a demolition landfill will be established on the Goliath property to accept demolition waste from all three projects. As is the industry standard in Ontario, the estimate assumes no salvage cost and may be considered conservative, as it generally assumes third-party costs.

The estimated total reclamation and closure costs, exclusive of taxes and contingency, for the Goliath, Goldlund and Miller projects are $12.93 million, $3.50 million and $2.07 million, respectively. Monitoring activities for the Goliath, Goldlund and Miller sites are $7.03 million, $1.49 million, and $1.44 million, respectively. Wood recommended a contingency of at least 20% be added to these costs to reflect uncertainties.

These costs have been based primarily on the closure costing that has been accepted by the provincial government recently for other mining projects, as well as a demolition cost estimate provided to Wood for a similar project by Ausenco. The cost of closure has been accounted for in the cash flow at the end of the mine life.

21.2 Operating Costs

The operating cost estimate was developed in Q4 2020 dollars based on Ausenco's in-house database of projects and studies and experience from similar operations to a level of accuracy of ±50%.

The overall life-of-mine operating cost is $975 million over 13.5 years, or $40.7/t of ore milled, as summarised in Table 21.18, and detailed in the following sections.

Table 21.18: Operating Cost Estimate Summary

Operating Cost	Unit Cost (C$/t Mined)	Unit Cost (C$/t Processed)	Total Cost (C$M)
Mining - Open Pit	3.27	17.0	356.0
Mining - Underground	-	70.3	208.5
Off-site Mill Feed Haulage		5.6	83.6

Operating Cost	Unit Cost (C$/t Mined)	Unit Cost (C$/t Processed)	Total Cost (C$M)
Processing	-	11.4	272.5
Site G&A	-	2.3	54.7
TOTAL		**40.7**	**975.3**

Source: Ausenco (2021).

21.2.1 Operating Costs – Mining

The Goliath Gold Complex mine operating costs have been estimated from first principles with vendor quotations for repair and maintenance costs and other suppliers for consumables. Key inputs to the mine cost are fuel and labour. The diesel price provided for the project by local vendors was $0.79 per litre delivered to the site. The mine fleet is entirely diesel powered except for the electric dewatering pumps at Goliath.

21.2.1.1 Open Pit Mining Costs

21.2.1.1.1 Open Pit Labour

Labour costs for the various job classifications were obtained from salary surveys in Ontario. A burden rate of 31% was applied to the various rates based on nearby project information. Labour was estimated for both staff and hourly personnel based on 12-hour shifts and utilising a rotation of either two weeks on and two weeks off, or five days on and two days off. Mine positions and salaries are shown in Table 21.19.

Table 21.19: Mine Staffing Requirements & Annual Employee Salaries (Year 5)

Position	Employees	Loaded Annual Salary (C$/a)	(C$M/a)
Maintenance Shift Foreman	4	137,550	0.55
Maintenance Planner/Contract Administration	2	124,450	0.25
Clerk	1	78,600	0.08
Mine Maintenance Subtotal	**7**		**0.88**
Mine General Foreman	0.5	183,400	0.09
Senior Shift Foreman	4	137,550	0.55
Mine Operations Subtotal	**4.5**		**0.64**
Chief Engineer	1	183,400	0.18
Senior Engineer	1	157,200	0.16
Open Pit Planning Engineer	2	137,550	0.28
Blasting/Geotechnical Technician	1	91,700	0.09
Surveyor/Mining Technician	2	91,700	0.18
Surveyor/Mining Technician Helper	2	85,150	0.17
Mine Engineering Subtotal	**9**		**1.06**
Chief Geologist	1	170,300	0.17
Senior Geologist	1	144,100	0.14
Grade Control Geologist/Modeller	2	117,900	0.24
Sampling/Geology Technician	2	91,700	0.18
Geology Subtotal	**6**		**0.73**
TOTAL	**26.5**		**3.31**

The mine staff labour remains constant from Year 1 until Year 5 when positions are removed as the mine winds down. From Year 3 until Year 6, a general foreman position is added to mine operations to help with the multiple pit areas and underground, which starts in Year 3. As the open pits wind down this general foreman position will no longer be required.

Hourly employee labour force levels in mine operations and maintenance fluctuate with production requirements. The peak occurs in Years 2 and 3, and then the levels start to diminish as the strip ratios drop. The Year 5 hourly labour requirements are shown in Table 21.20.

Table 21.20: Hourly Manpower Requirements & Annual Salaries (Year 5)

Position	Employees	Loaded Annual Salary (C$/a)	(C$M/a)
General Equipment Operator	4	83,400	0.33
Road/Pump Crew	2	83,400	0.17
General Mine Labourer	8	82,900	0.66
Trainee	4	73,800	0.30
Light Duty Mechanic	1	141,200	0.14
Tire Technician	1	107,600	0.11
Lube Truck Driver	4	83,400	0.33
Mine General Subtotal	**24**		**2.04**
Driller	12	105,600	1.27
Blaster	1	105,600	0.11
Blast Helper	2	82,900	0.17
Loader Operator	8	111,300	0.89
Hydraulic Excavator Operator	4	111,300	0.45
Haul Truck Driver	32	95,400	2.35
Dozer Operator	9	105,600	0.95
Grader Operator	5	105,600	0.53
Crusher Loader Operator	2	105,600	0.21
Snow Plow/Water Truck	3	83,400	0.25
Mine Operations Subtotal	**78**		**7.17**
Heavy Duty Mechanic	18	141,200	2.54
Welder	12	141,200	1.69
Electrician	2	141,200	0.21
Apprentice	5	101,900	0.51
Mine Maintenance Subtotal	**38**		**4.95**
Total Hourly	**139**		**14.16**

Labour costs are based on an owner-operated scenario with Treasury Metals responsible for the maintenance of the equipment with its own employees.

Overseeing all the mine operations, maintenance, engineering, and geology functions is the mine manager (covered under the G&A expenses). The chief engineer, chief geologist and shift foremen (mine operations and maintenance) would report directly to the mine manager.

The shift foremen report directly to the mine general foreman later in the mine life when underground mining starts.

The mine has four mine operations crews, each with a senior shift foremen and a lead hand to assist.

The chief engineer manages one senior engineer and two open pit engineers. The blasting/geotechnical technician is included in the short-range planning group and works with the planning engineers to cover aspects of the wall slopes and waste dumps.

The short-range planning group in engineering also has two surveyor/mine technicians and two surveyors/mine helpers. These individuals assist in the field with staking, surveying, and sample collection with the geology group.

In the geology department, there is one senior geologist reporting to the chief geologist. There are also two grade control geologists/modellers; one for short range and grade control drilling, and the other for long range/reserves. There are also two grade control/sampling technicians.

The four mine maintenance shift foremen will report to the mine manager. As well, there are two maintenance planners/contract administrators and a clerk.

The hourly labour force includes positions for the light duty mechanic, tire mechanics, and lube truck drivers. These positions all report to maintenance. There is generally one lube truck driver per crew. Other general labour includes general mine labourers (two per crew) and trainees (one per crew) plus two road/pump crew personnel designated for water management/snow removal.

The drilling labour force is based on one operator per drill per crew while operating. This peaks at 12 drillers in Year 1 and stays steady until Year 5, after which it declines over time as the drilling hours are diminished.

Loader operators peak at 8 in Year 1 and hold until Year 7. This does not include the transfer loader operator. Haulage truck drivers peak at 44 in Years 2 and 3 and then taper off to the end of the mine life.

Maintenance factors are used to determine the number of heavy-duty mechanics, welders and electricians required and are based on the number of equipment operators. Heavy-duty mechanics work out to 0.25 mechanics required for each drill operator for example. Welders are 0.25 per operator and electricians are 0.05 per operator.

The number of loader, truck and support equipment operators is estimated using the projected equipment operating hours. The maximum number of employees is four per unit to match the mine crews.

21.2.1.1.2 Equipment Operating Costs

The vendors provided repair and maintenance (R&M) costs for each piece of equipment selected for the Goliath PEA. Fuel consumption rates were estimated from the supplied information and knowledge of the working conditions. The costs for the R&M are expressed in dollars per hour.

Tire costs were collected from various vendors based on the expected sizes required. Estimates of tire life are based on AGP's experience. The operating cost of the tires is expressed in dollars per hour. The life of the haulage truck tires is estimated at 5,500 hours per tire with proper rotation from front to back. Each truck tire costs $14,000, so the cost per hour for tires is $15.27/h for the truck using six tires in the calculation.

The cost for ground-engaging tools (GET) is estimated from other projects and can be more fine-tuned once the project is operational.

Drill consumables are estimated as a complete drill string using the parts list and component lives provided by the vendor. Drill productivity is estimated at 24.4 m/h for mill feed and waste. The equipment costs used in the estimate are shown in Table 21.21.

Table 21.21: Major Equipment Operating Costs – No Labour ($/h)

Equipment	Fuel	Lube/Oil	Tires/ Undercarriage	Repair & Maintenance	GET/ Consumables	Total
Production Drill	39.50	3.95	-	107.59	73.04	224.08
Production Loader	63.20	6.32	27.52	76.91	7.00	180.95
Production Excavator	47.40	9.48	-	60.55	6.00	123.43
Haulage Truck	55.30	5.53	15.27	52.59	3.00	131.69
Track Dozer	59.25	5.93	10.00	64.04	5.00	144.22
Grader	17.38	1.74	4.00	18.58	5.00	46.70

21.2.1.1.3 Drilling

Drilling in the open pit will use down-the-hole hammer drill rigs with 140 mm bits. The pattern size is the same for both mill feed and waste and is blasted in recognition of the equipment being used. The material will be smaller and finer to improve productivity and reduce maintenance costs as well as improve plant performance. The drilling pattern parameters are shown in Table 21.22.

Table 21.22: Drill Pattern Specifications

Specification	Unit	Mill Feed	Waste
Bench Height	m	10	10
Sub-Drill	m	0.8	0.8
Blasthole Diameter	mm	140	140
Pattern Spacing - Staggered	m	4.8	4.8
Pattern Burden - Staggered	m	4.2	4.2
Hole Depth	m	10.8	10.8

The sub-drill is included to allow for caving of the holes in weaker zones, reducing re-drill requirements or short holes that would affect bench floor conditions. The extra sub-drill is above what is normally required.

The parameters used to estimate drill productivity are shown in Table 21.23.

Table 21.23: Drill Productivity Criteria

Drill Activity	Unit	Mill Feed	Waste
Pure Penetration Rate	m/min	0.50	0.50
Hole Depth	m	10.8	10.8
Drill Time	min	21.60	21.60
Move, Spot and Collar Hole	min	3.00	3.00
Level Drill	min	0.50	0.50
Add Steel	min	0.50	0.50
Pull Drill Rods	min	1.00	1.00
Total Setup/Breakdown Time	min	5.00	5.00
Total Drill Time per Hole	min	26.60	26.60
Drill Productivity	m/h	24.4	24.4

21.2.1.1.4 Blasting

An emulsion product will be used for blasting to provide water protection. With the Goliath pit being in a local depression, it was deemed that a water-resistant explosive will be required. The powder factors used in the explosives calculation are shown in Table 21.24.

Table 21.24: Design Powder Factors

Description	Unit	Mill Feed	Waste
Powder Factor	kg/m^3	0.75	0.75
Powder Factor	kg/t	0.276	0.267

The blasting cost is estimated using quotations from a local explosives vendor. The emulsion price is $89.00 per 100 kg. The mine is responsible for guiding the loading process, including placing the boosters/Nonels, and stemming and firing the shot.

The explosives vendor also leases the explosives and accessories for a monthly cost. Additionally, a service charge for the vendors pickup trucks, pumps, labour and cost of the explosives plant is included. The total monthly cost is $57,300 per month.

21.2.1.1.5 Loading

Loading costs for both mill feed and waste are based on the use of front-end loaders with support from hydraulic excavators. The loaders are the primary diggers with the hydraulic excavators as backup/support units. The average percentage of each material type that the various loading units are responsible for is shown in Table 21.25. This highlights the focus of the loaders over the excavator.

Table 21.25: Loading Parameters – Year 5

Item	Unit	Front-end loader	Hydraulic Excavator
Bucket Capacity	m^3	13	6.7
Truck Capacity Loaded	t	91	91
Waste Tonnage Loaded	%	85	15
Mill Feed Tonnage Loaded	%	85	15
Bucket Fill Factor	%	95	95
Cycle Time	sec	40	38
Trucks Present at Loading Unit	%	80	80
Loading Time	min	2.70	5.13

21.2.1.1.6 Hauling

Haulage profiles were determined for each pit phase for the primary crusher or the waste rock facility destinations. Cycle times were generated for the appropriate period tonnage by destination and phase to estimate the haulage costs. Maximum speed on the trucks is limited to 50 km/h for tire life and safety reasons, although few locations in the mine plan appeared to offer the truck the opportunity to accelerate to that velocity. Calculation speeds for various segments are shown in Table 21.26.

Table 21.26: Haulage Cycle Speeds

Road Segment	Loaded (km/h)	Empty (km/h)
Flat – Outside of pit (0%)	50	50
Flat – In pit, Crusher or Dump (0%)	40	40
Slope – Up 5%	18	45
Slope – Up 8%	16	35
Slope – Up 10%	12.1	25
Slope – Down 5%	35	40
Slope – Down 8%	30	35
Slope – Down 10%	30	35

21.2.1.1.7 Support Equipment

Support equipment hours and costs are determined on factors applied to various major pieces of equipment. For the PEA some of the factors used are shown below in Table 21.27.

These factors resulted in the need for five track dozers, two graders, and one support backhoe. Their tasks include clean-up of the loader faces, roads, dumps, and blast patterns. The graders will maintain the crusher, waste haul routes and route for mill feed trucks. In addition, snowplow/water trucks have the responsibility for patrolling the haul roads for snow removal and controlling fugitive dust for safety and environmental reasons. The small backhoe and road crew dump trucks will be responsible for cleaning out sedimentation ponds and water ditch repairs.

Table 21.27: Support Equipment Operating Factors

Mine Equipment	Factor	Factor Units
Track Dozer	30%	of haulage hours to maximum of 5 dozers
Grader	15%	of haulage hours to maximum of 2 graders
Crusher Loader	15%	of loading hours to maximum of 1 loader
Snowplow/Water Truck	7%	of haulage hours to maximum of 2 trucks
Pit Support Backhoe	10%	of loading hours to maximum of 1 backhoe
Road Crew Backhoe	2	hours/day/unit
Road Crew Dump Truck	2	hours/day/unit
Road Crew Loader	2	hours/day/unit
Lube/Fuel Truck	6	hours/day/unit
Mechanics Truck	10	hours/day/unit
Blasting Loader	8	hours/day/unit
Blaster’s Truck	8	hours/day/unit
Integrated Tool Carrier	3	hours/day/unit
Light Plants	12	hours/day/unit
Pickup Trucks	10	hours/day/unit

The hours generated in this manner are applied to the individual operating costs for each piece of equipment. Many of these units are support equipment so no direct labour is allocated to them due to their variable function. The operators come from the general equipment operator pool.

21.2.1.1.8 Grade Control

Grade control will be completed with a separate fleet of reverse circulation (RC) drill rigs. They will drill the deposit off on a 10 m x 5 m pattern in areas of known mineralisation taking samples each metre. The holes will be inclined at 60 degrees.

In areas of low-grade mineralisation or waste the pattern spacing will be 20 m x 10 m with sampling over 6 m. These holes will be used to find undiscovered veinlets or pockets of mineralisation. Over the life of the mine, a total of 158,000 m of drilling are expected to be completed for grade control work. A total of 177,000 samples will be assayed from that drilling.

These grade control holes serve two purposes:

1) Define the mill feed grade and contacts in all pit areas

2) Locate previous underground infrastructure prior to blasthole rigs drilling at Goldlund

Samples collected will be sent to the assay laboratory and assayed for use in the short-range mining model.

Additional costing for blasthole sampling has not been included. A gold deportment study should be completed to determine the best sampling protocol for blastholes and evaluate whether it has any merit.

Costs associated with this separate drill program are tracked as a distinct line item for the mining cost. The drill crew is one driller and two helpers with oversight by the mine geology department. The cost of this drilling is expected to be slightly more than $2 million per year.

21.2.1.1.9 Dewatering

Pit dewatering is an important part of mining of the Goliath Gold Complex. The Goliath project will require significant volumes need to be pumped initially due to its location in a geographic low. Goldlund, while on higher ground initially, will require the small, previous pit to be dewatered as well as the previous underground workings.

Reviewing past data collected and comparing this to the proposed mining area allowed AGP to make an estimate at a PEA level for the water volume required to be pumped. Initial pumping in Year -1 for Goliath is expected to be 365,000 m^3. That rate is maintained annually while the mine is in operation. The peak expected would be 1,500 m^3/d and the pumps required have been included in the capital and operating cost estimate. With Goliath's proximity to electrical power, electric pumps are considered. One pump in the pit and one pump on the surface have been included in the estimate with a spare for each on site should they be required.

Goldlund and Goliath were estimated at this stage using the same information used for Goliath. Due to the more remote location, diesel pumps were used in the estimate rather than electric pumps. This results in a slightly higher operating cost over the time the pumping is necessary.

Additional dewatering in the form of horizontal drill holes are part of the dewatering costs. These holes will be campaigned and will be part of the sustaining mine capital.

Dewatering is expected to cost $2.9 million over the mine life.

21.2.1.1.10 Leasing

Leasing the mine fleet is considered a viable option to reduce initial capital cost. Various vendors offer this as an option to help select their equipment. Both Caterpillar and Komatsu have the ability, and desire, to lease equipment from their product lines.

Indicative terms for leasing provided by the vendors are:

- Down payment = 20% of equipment cost
- Term Length = 3 to 5 years (depending on equipment)
- Interest Rate = LIBOR plus a percentage
- Residual = $0

The proposed interest rate is used to calculate a multiplier on the amount being leased. The multiplier is 1.067 to equate to the rate. It does not consider a declining balance on the interest, but rather the full amount of interest paid over the term, equally distributed over those years. The calculation is as follows:

Annual Lease Cost = {[(Initial Capital Cost) x 80%] x 1.067} / term in years

The initial capital, down payments, and annual leasing costs were shown previously in the capital cost area of this section.

The support equipment fleet is calculated in the same manner as the major mining equipment.

All of the major mine equipment, and the majority of the support equipment where it was considered reasonable, was leased. If the equipment has a life greater than the lease term length, then the following years onward of the lease do not have a lease payment applied. In the case of the mine trucks, with an approximate 10-year working life, the lease would be complete, and the trucks would simply incur operating costs after the lease was complete. For this reason, the operating cost would vary annually depending on the equipment replacement schedule and timing of the leases.

Utilising the leasing option adds $0.40/t to the mine operating cost over the life of the mine. On a cost per tonne of feed basis, it was $2.07/t mill feed.

21.2.1.1.11 Total Open Pit Mine Costs

The total life of mine operating costs per tonne of material moved and per tonne of mill feed processed are shown in Tables 21.28 and 21.29, respectively.

The cost associated with an owner-operated crushing plant to make stemming material and road crush is included under general mine engineering. That cost is approximately $1.3 million per year.

Table 21.28: Open Pit Mine Operating Costs – with Leasing ($/t Total Mined)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine and Engineering	$/t mined	0.46	0.40	0.45	0.45
Drilling	$/t mined	0.26	0.31	0.32	0.28
Blasting	$/t mined	0.40	0.46	0.45	0.40
Loading	$/t mined	0.30	0.27	0.29	0.34
Hauling	$/t mined	0.61	0.77	0.61	0.71
Support	$/t mined	0.52	0.55	0.50	0.53
Grade Control	$/t mined	0.13	0.12	0.14	0.15
Leasing Costs	$/t mined	0.62	0.48	0.35	0.40
Dewatering	$/t mined	0.03	0.02	0.01	0.03
Total	**$/t mined**	**3.33**	**3.39**	**3.12**	**3.27**

Table 21.29: Open Pit Mine Operating Costs – with Leasing ($/t Processed)

Open Pit Category	Unit	Year 1	Year 3	Year 5	LOM Average
General Mine & Engineering	$/t mill feed	4.44	3.92	3.91	2.33
Drilling	$/t mill feed	2.49	3.02	2.75	1.44
Blasting	$/t mill feed	3.89	4.51	3.91	2.06
Loading	$/t mill feed	2.90	2.63	2.55	1.77
Hauling	$/t mill feed	5.94	7.52	5.32	3.66
Support	$/t mill feed	5.07	5.39	4.34	2.73
Grade Control	$/t mill feed	1.27	1.15	1.18	0.77
Leasing Costs	$/t mill feed	6.06	4.63	3.02	2.07
Dewatering	$/t mill feed	0.24	0.21	0.13	0.14
Total	**$/t mill feed**	**32.30**	**32.97**	**27.10**	**16.95**

21.2.1.2 Underground Mining Costs

This section summarises underground mining cost information. For a discussion of underground mining, see Section 16.12.

While underground development starts in Year 3, the underground operating costs commence with the delineation drilling and in-vein development work in early Year 4 extending to the end of the mine life in early Year 11.

The mine was assumed to work on two 12-hour shifts per day, 365 days per year. All activities are completed by owner's crews except for the raising and delineation drilling which will be completed by a contractor. The costing includes ground support assumptions provided by the geotechnical study.

The major material and consumable cost assumptions are shown in Table 21.30. The various unit rates applied to the scheduled quantities in order to estimate the direct costs are shown in Table 21.31.

Table 21.30: Underground Major Material & Consumables Cost Assumptions

Description	Units	Cost (C$)
Diesel	L	0.79
Emulsion Explosive (Bulk)	kg	1.76
Trim Product	kg	3.02
NONEL LP detonator 5 m	ea.	3.54
NONEL MS detonator 18 m (60 ft)	ea.	12.46
1.5 m Rebar (complete)	ea.	20.94
1.8 m Rebar (Complete)	ea.	22.96
2.4 m Rebar (Complete)	ea.	25.63
Welded Mesh	m^2	8.37
6 m Cablebolt (Complete)	ea.	58.45
9 m Cablebolt (Complete)	ea.	63.02
Fibrecrete	m^3	253.45
45 mm Development Face Drill Hole (Consumables)	m	1.52
33 mm Development Support Drill Hole (Consumables)	m	0.96
51 mm Development Cablebolt Drilling	m	2.01
64 mm Stoping Long Hole (Consumables)	m	2.49
42″ Ventilation Wire Reinforced FlexiDuct Installed	m	128.13
36″ Ventilation FlexiDuct Installed	m	37.00
100 mm 4″ Pipes HDPE	m	55.96
150 mm 6″ Pipes HDPE	m	99.57
6″ x 10ft Ultratech Schedule 80	m	129.48
48-Strand Fibre Optic Cable	m	8.71

Table 21.31: Underground Unit & Overhead Costs

Cost Model Description	Units	Cost (C$)
Ramp - 6.0 m wide x 6.0 m high	m	3,049
Ramp - 5.0 m wide x 5.0 m high	m	2,712
Level Waste Drift - 5.0 m wide x 5.0 m high	m	2,352
Vent/Other Drift - 5.0 m wide x 5.0 m high	m	2,123
Workshop/Pumps - 5.0 m wide x 4.5 m high	m	3,031
In Vein Waste/Low Grade - 4.0 m wide x 4.5 m high	m	1,870
Ore Drift - 4.0 m wide x 4.5 m high	m	3,094
Conventional Alimak Raise - 3.5 m wide x 3.5 m high with Ladder (Contractor)	m	7,294
Raise Bore (4.5 m) (Contractor)	m	10,667
Longhole Open Stope Drilling & Blasting - 5.0 m Thick Orebody	t	6.47
Scoop Mucking - LHD From Stope to Remuck Zone A	t	2.63
Scoop Mucking - LHD From Stope to Remuck Zone B/C	t	3.92
Mill Feed/Waste Trucking to Surface		
50 m Vertical Haul	t	2.96
150 m Vertical Haul	t	4.89
250 m Vertical Haul	t	6.83
350 m Vertical Haul	t	7.84
450 m Vertical Haul	t	9.57
550 m Vertical Haul	t	11.30
650 m Vertical Haul	t	13.04
Rockfill Crush & Screen	t fill	2.00
Incremental Rockfill Haul		
A Zone Cemented Fill	t fill	14.96
150 m Uncemented Vertical Haul	t fill	2.65
250 m Uncemented Vertical Haul	t fill	3.17
350 m Uncemented Vertical Haul	t fill	3.69
450 m Uncemented Vertical Haul	t fill	4.22
550 m Uncemented Vertical Haul	t fill	4.89
650 m Uncemented Vertical Haul	t fill	5.41
Rockfill - LHD From Rockfill Storage to Stope Zone A	t fill	1.02
Rockfill - LHD From Rockfill Storage to Stope Zone B/C	t fill	2.40
Contract Diamond Drilling (Delineation)	stope t	1.95
Mine Services, Fixed Plant & Mobile Equipment Maintenance Labour	day	21,368
Owners Mine Supervision & Technical	day	14,961
Mine Air Heating (Propane)	day	3,752
Power	day	6,269

The cost model included additional components to reflect overhead-type activities at the mine. These include:

- Mine Services and Fixed Plant
  - labour
  - supplies
  - equipment for construction
  - materials transportation
  - road maintenance
  - sanitation
  - diesel maintenance labour costs
- Owners Mine Management and Technical
  - mine supervision, mine technical and safety staff
- Mine Air Heating
  - based on the local weather station data
  - uses the estimated annual air flow requirements from the ventilation estimate
- Mine Power
  - developed from aggregation of mine loads and estimated usage

The overheads were estimated on a quarterly basis and applied as a fixed daily cost.

21.2.1.2.1 Underground Labour

Underground labour was estimated to be sufficient to support the production objectives of the life of mine plan. The fully burdened rates for the labour costing are shown in Table 21.32 and Table 21.33.

Hourly paid employees will work both day and night shifts of 12 hours per shift on a rotating schedule of 7 days on and 7 days off.

Labour levels for the first quarter of the listed year are shown in Table 21.34; annual costs are listed in Table 21.35.

Table 21.32: Underground Mining Staff Labour Rates

Position	Loaded Annual Salary C$/a
Mine Superintendent	209,500
Mine Captain	170,300
Shift Boss	137,550
Mine Dry/Lamps/Bits	78,600
Safety	137,550
Secretary/Clerk/Stores	78,600
Senior Geologist	144,100
Mine Geologist	117,900
Geology Technician - Grade Control	91,700
Senior Mine Engineer	157,200
Mine Engineer	137,550
Mine Technician	91,700
Surveyor	91,700
Survey Helper	85,150
Ventilation / Samplers / Rock Mechanics Assistant	85,150
Maintenance Supt	196,500
Maintenance General Foreman	170,300
Maintenance Planner	124,450
Maintenance Foreman	137,550
Portal Attendant	91,700

Table 21.33: Underground Hourly Labour Rates

Position	Loaded Annual Salary C$/h
Development Miner	89.54
Longhole Driller	83.75
Stope Blasting	77.40
Scoop Driver	67.43
Construction	60.46
Truck Driver	60.06
Materials/Pumps	59.38
Labourer	48.73
Mechanic/Diesel/Electrician I	73.24
Mechanic/Diesel/Electrician II	65.08

Table 21.34: Underground Labour Levels

Position	Year 4	Year 6	Year 8	Year 10
Longhole Drilling	-	5	5	4
Development Miner	17	19	13	5
Scoop Driver	7	17	16	12
Stope Blasting	-	3	3	3
Construction	7	7	7	5
Materials	4	4	4	4
Truck Driver	6	20	17	12
Labourer	13	19	15	8
Pumps	-	4	4	4
Mechanic I	2	2	2	2
Mechanic II	-	2	2	2
Electrician I	2	2	2	2
Electrician II	-	2	2	2
Diesel Mechanic I	6	9	8	6
Diesel Mechanic II	6	9	8	6
Diesel Mechanic III	6	9	8	6
Subtotal Hourly	**76**	**133**	**116**	**83**
Maintenance Superintendent	-	0.5	0.5	-
Maintenance Foreman	1	3	3	2
Maintenance General Foreman	-	1	1	-
Maintenance Planner	-	0.5	0.5	-
Mine Superintendent	-	0.5	0.5	-
Mine Captain	1	1	1	1
Shift Boss	8	8	8	4
Mine Dry/Lamps/Bits	-	2	2	2
Secretary/Clerk/Stores	2	2	2	1
Safety	1	1	1	1
Senior Mine Engineer	1	1	1	1
Senior Geologist	1	1	1	1
Mine Geologist	1	2	2	1
Mine Technician	1	2	2	1
Geology Technician/Grade Control	1	2	2	1
Mine Engineer	1	2	2	1
Surveyor	2	2	2	2
Survey Helper	4	4	4	2
Portal Attendant	4	4	4	4
Ventilation / Samplers / Rock Mechanics Assistance	4	4	4	4
Subtotal Staff	**33**	**44**	**44**	**29**
Total Underground Labour	**109**	**177**	**160**	**112**

Table 21.35: Underground Labour Annual Cost (C$M/a)

Position	Year 4	Year 6	Year 8	Year 10
Longhole Drilling	0.15	0.67	0.67	0.42
Development Miner	3.64	3.44	2.19	0.30
Scoop Driver	1.19	2.19	2.01	1.01
Stope Blasting	0.09	0.41	0.41	0.26
Construction	0.82	0.82	0.81	0.53
Materials	0.52	0.52	0.52	0.52
Truck Driver	0.89	2.29	1.86	0.94
Labourer	1.70	1.93	1.39	0.39
Pumps	0.26	0.52	0.52	0.52
Mechanic I	0.32	0.32	0.32	0.32
Mechanic II	0.14	0.29	0.29	0.29
Electrician I	0.32	0.32	0.32	0.32
Electrician II	0.14	0.29	0.29	0.29
Diesel Mechanic I	1.04	1.39	1.18	0.67
Diesel Mechanic II	0.93	1.24	1.05	0.59
Diesel Mechanic III	0.88	1.17	0.99	0.56
Subtotal Hourly	**13.05**	**17.79**	**14.81**	**7.93**
Maintenance Superintendent	0.07	0.10	0.10	-
Maintenance Foreman	0.24	0.41	0.41	0.28
Maintenance General Foreman	0.13	0.17	0.17	-
Maintenance Planner	0.05	0.06	0.06	-
Mine Superintendent	0.08	0.10	0.10	-
Mine Captain	0.17	0.17	0.17	0.17
Shift Boss	1.10	1.10	1.10	0.55
Mine Dry/Lamps/Bits	-	0.16	0.16	0.16
Secretary/Clerk/Stores	0.16	0.16	0.16	0.08
Safety	0.14	0.14	0.14	0.14
Senior Mine Engineer	0.16	0.16	0.16	0.16
Senior Geologist	0.14	0.14	0.14	0.14
Mine Geologist	0.18	0.24	0.24	0.12
Mine Technician	0.14	0.18	0.18	0.09
Geology Technician/Grade Control	0.14	0.18	0.18	0.09
Mine Engineer	0.21	0.28	0.28	0.14
Surveyor	0.18	0.18	0.18	0.18
Survey Helper	0.34	0.34	0.34	0.17
Portal Attendant	0.37	0.37	0.37	0.37
Ventilation / Samplers / Rock Mechanics Assistance	0.34	0.34	0.34	0.34
Subtotal Staff	**4.32**	**4.98**	**4.98**	**3.17**
Total Underground Labour	**17.37**	**22.78**	**19.79**	**11.10**

21.2.1.2.2 Underground Power

The load list of installed power for the underground mine is shown in Table 21.36.

Table 21.36: Underground Mine Power Requirements

Mine Activity	Installed kW
Ventilation	2,070
Rockfill	150
Dewatering	2,299
Mine Equipment	931
Other Loads	631
Total	**6,081**

Power costs over the mine life are shown in Figure 21-2.

Figure 21-2: Underground Power Usage by Year

Source: AGP (2021).

21.2.1.2.3 Total Underground Mine Costs

A summary of the underground mining cost estimate by element and activity are shown in Tables 21.37 and 21.38.

Table 21.37: Underground Mining Cost Summary – By Element

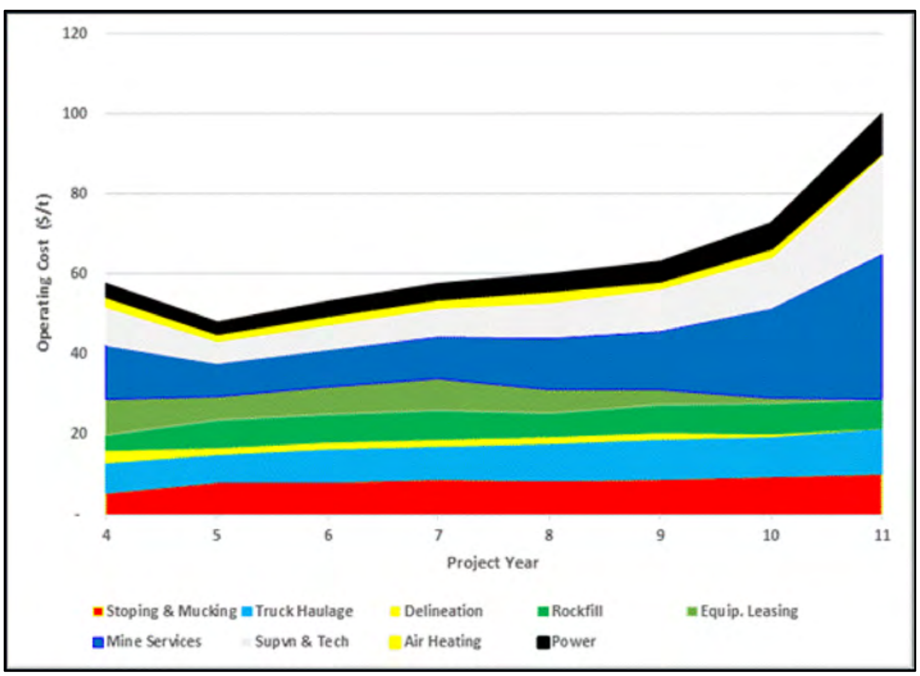
Element	LOM (C$k)	Cost C$/t Processed
Labour	89,395	30.15
Supplies	44,741	15.09
Equipment	48,827	16.47
Fuel	14,616	4.93
Power	10,924	3.68
Total	**208,503**	**70.31**

Table 21.38: Underground Mining Cost Summary – By Activity

Activity	LOM (C$k)	Cost C$/t Processed
Development	35,937	12.12
Stoping and Mucking	24,580	8.29
Truck Haulage	26,152	8.82
Delineation Drilling	4,800	1.62
Rockfill	19,983	6.74
Mobile Equipment Leasing	17,041	5.75
Mine Services	36,841	12.42
Supervision and Technical	25,902	8.73
Mine Air Heating	6,343	2.14
Power	10,924	3.68
Total	**208,503**	**70.31**

Figure 21-3 shows the underground mine operating cost by activity below.

Figure 21-3: Underground Life of Mine Operating Cost – By Activity



Source: AGP (2021).

21.2.1.3 Mill Feed Haulage Costs

The haulage of mill feed from Goldlund and Miller is carried as a separate operating cost. The capital for the fleet is carried under mine capital, but the cost of transportation is a separate line item. The cost considers the loading of material at either Goldlund or Miller from a transfer pad into the truck and trailer haulage fleet.

Mill feed haulage is planned to be a 24-hour operation, 365 days per year. Loading would be with a 4.4 m^3 loader working with Kenworth W900B heavy duty tractors pulling 40 tonne belly dump trailers. A total of 11 tractor and 12 trailer units are required to meet the full production requirement of 1.8 Mt per year of mill feed. Due to stockpile rehandling at Goliath and underground production, this full annual requirement is only required in Year 3 and tends to be in the 1.3 Mt per year range.

Local vendors provided quotations for the fleet capital and operating costs and consumables (tires, fuel). The costs associated with transfer pad clean-up and haul road maintenance (Goldlund to the highway; highway to Goliath) are carried under the normal mine activities as they are extensions of the pit areas.

The estimated mill feed haulage cost is $5.61/t based on the following cost components:

- loading = $0.51/t mill feed
- haulage = $5.10/t mill feed

21.2.2 Operating Costs – Process Plant & G&A

21.2.2.1 Summary

The process plant and infrastructure operating cost estimates are summarised in Table 21.39. These are derived from benchmarking against existing gold processing plants located in Eastern Canada as well as in-house data.

Table 21.39: Operating Cost Summary

Process & G&A Category	C$M/a	C$/t Processed
Labour	$5.30	$3.10
Power	$5.89	$3.27
Reagents	$4.70	$2.61
Consumable	$2.78	$1.54
Maintenance	$1.20	$0.66
Lab Services	$0.33	$0.19
Subtotal	**$20.19**	**$11.37**
General & Administration	$2.84	$1.66
Effluent Treatment Plant	$0.74	$0.41
Mobile Equipment	$0.39	$0.22
Total Costs	**$24.16**	**$13.66**

Source: Ausenco (2021).

Common to all plant operating cost estimates are the following assumptions:

- For material sourced or benchmarked in US dollars, an exchange rate of 0.75 US dollar per Canadian dollar was assumed.
- Diesel costs used are C$0.90/L and gasoline costs are C$1.05/L.
- The annual power costs were calculated using a unit price of C$0.09/kWh.
- The majority of the labour requirement is assumed to come from neighbouring municipalities.
- Processing unit operations were benchmarked against similar or comparable processing plants.
- Equipment and materials will be purchased as new.
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- General and administration (G&A) costs were baselined against previous project experience.
- Grinding media consumption rates have been estimated based on the material characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork.
- The mobile equipment cost provides for fuel and maintenance.

21.2.2.2 Basis of Processing Operating Costs

21.2.2.2.1 Labour

The labour estimate was determined from benchmarking against similar projects with comparable unit processes, as well burden costs/employee bonuses. A burden rate of 31% was applied to all rates.

An organisational roster outlining the labour requirement for the process plant is shown in Table 21.40 (note: loaded salaries include burden).

21.2.2.2.2 Power

The power cost is calculated from the overall plant power draw determined from the mechanical equipment list. This cost was calculated to be C$0.063/kWh using an assumption by Ausenco of 72.5% installed power utilisation at a delivered power cost of C$0.09/kWh.

21.2.2.2.3 Reagents

The regent profile was developed from the testwork review in Section 13. The testwork enabled an estimate to be made of the addition and consumption rates of particular reagents. Where testwork was not available, benchmarking against currently operating unit technologies was carried out. Costs for each reagent were identified from other projects in eastern Canada. The details are presented in Table 21.41.

Section 17 discusses the use of processing reagents in detail. Reagents for the intensive leach reactor (ILR) and the elution process have been grouped together.

Table 21.40: Processing Plant Shift Roster Summary

Labour / Contractor Summary	Employees	Loaded Salary (C$/a)	C$M/a
Mill Superintendent	1	111,350	0.11
Maintenance Superintendent	1	111,350	0.11
Plant Metallurgist	2	98,250	0.20
Process Management Total	**4**		**0.42**
Mine General Manager	1	183,400	0.18
Manager - Procurement/Contracts	1	85,150	0.09
Manager – Human Resources	1	85,150	0.09
Administrative Assistant	1	65,500	0.07
Warehouse Attendant	3	62,225	0.19
Administrative Total	**7**		**0.61**
General Foreman	4	98,250	0.39
Crusher Operator	8	85,150	0.68
Grinding/Gravity Operator	4	85,150	0.34
Leach/Elution/Detoxification/Reagents Operator	4	85,150	0.34
Reagents/Swing Operator	4	85,150	0.34
Gold Refining Foreman/Operator	4	85,150	0.34
Mill Operations Total	**28**		**2.44**
Lab Managers	1	111,350	0.11
Lead Hands	2	98,250	0.20
Laboratory Technicians	2	72,050	0.14
Laboratory Total	**5**		**0.45**
Maintenance Planner	1	85,150	0.12
Millwrights	8	85,150	0.48
Electricians	8	85,150	0.48
Process Control/Instrument Technician	2	85,150	0.12
Apprentices/Welders	2	62,225	0.24
Mill Maintenance Total	**21**		**1.74**
Contract Allowance			0.25
Contract Labour Total			**0.25**
Overall Total	**58**		**5.30**

Source: Ausenco (2021).

Table 21.41: Reagent Cost Summary

Description	Reagent (C$/t)	Consumption (t/a)	C$M/a
Caustic	1059	7.6	0.008
Cyanide	2629	510.9	1.34
Lime	312	2250.2	0.70
Carbon	4000	40.0	0.16
Hydrochloric Acid	629	5.8	0.004
Smelting Reagents	-	-	0.10
SMBS	733	2575.9	1.89
Oxygen	100	2690.6	0.27
Copper Sulphate	2687	82.6	0.22
Total Reagent Cost			**4.70**

Source: Ausenco (2021)

21.2.2.2.4 Consumables

Consumables are identified as non-regent requirements/replacements that are related to the crushing and grinding circuit. The following items have been included under consumables:

- ball mill grinding media
- ball bill liners
- propane
- trommel and screen panels
- jaw and cone crusher sets

Annual grinding media costs were estimated at C$1.14 million, and liner and miscellaneous costs at C$1.64 million. The costs have been developed from Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. The consumption rates were calculated internally.

21.2.2.2.5 Maintenance

The process plant annual maintenance costs was derived from the total installed mechanical cost determined from the mechanical equipment list (Ausenco, 2021) using a factor of 4.8%.

21.2.2.2.6 Laboratory Services

The operating cost estimate for the laboratory and assay activities were based on the number of daily and annual assays required, as well as assay types. These assay costs arise from monitoring grade and recovery for unit operations to permit optimisation of the process plant, environmental analysis, and metallurgical accounting.

21.2.2.2.7 Effluent Treatment

The water treatment plant size and cost are influenced by the site water balance and TSF. The cost estimate, which is presented in Table 21.42, includes maintenance, labour, power and consumables.

Table 21.42: Effluent Treatment Plant Operating Cost Summary

Description	C$/t Processed	C$M/a
Labour	0.30	0.52
Others (including consumables)	0.01	0.02
Power	0.05	0.98
Plant Maintenance	0.05	0.99
Total	**0.41**	**0.74**

Source: Ausenco (2021).

21.2.2.2.8 General and Administration

General and administrative (G&A) costs were developed with Ausenco's in-house data on existing Canadian operations. The costs were estimated including the following items:

- human resources (including recruiting, training, and community relations)
- infrastructure power (HVAC and administrative buildings)

- site administration, maintenance and security (including subscriptions, professional memberships and dues, first aid, office equipment, garbage disposal, bank and payroll fees)
- assets operation (including non-operation-related vehicles)
- health and safety (including personal protective equipment)
- environmental (including sampling and TSF operation)
- IT and telecommunications (including hardware)
- contract services (including insurance, consulting, sanitation, auditing, licences, freight, and legal fees)

21.2.2.2.9 Mobile Equipment

The process plant mobile equipment operating costs shown in Table 21.43 were developed from the number of light vehicles and mobile equipment, maintenance, spares and tires for different plant services. Fuel (diesel, gasoline) was included under mobile equipment.

Table 21.43: Process Plant Mobile Equipment Operating Costs

Cost Centre	C$/t Processed	C$M/a
G&A Vehicles	0.11	0.19
Processing Vehicles	0.04	0.08
Maintenance Vehicles	0.07	0.12
Total	**0.22**	**0.39**

Source: Ausenco (2021).

22 ECONOMIC ANALYSIS

22.1 Cautionary Statement

The results of the economic analyses discussed in this section represent forward- looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented herein. Information that is forward-looking includes:

- mineral resource estimates
- assumed commodity prices and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions as to mining dilution and the ability to mine in areas previously exploited using the mining methods envisaged
- sustaining costs and proposed operating costs
- assumptions as to closure costs and closure requirements
- assumptions as to environmental, permitting and social risks

Additional risks to the forward-looking information include:

- changes to costs of production from what is assumed
- unrecognised environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralised material, grade or recovery rates
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social licence to operate
- accidents, labour disputes and other risks of the mining industry
- changes to interest rates
- changes to tax rates

The mine plan is partly based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA based on these mineral resources will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations, and approval for development to be provided by Treasury Metals' Board.

22.2 Methodology Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess impact of variations in metal prices, head grades, operating costs and capital costs. The capital and operating cost estimates were developed specifically for this project and are summarised in Section 21 in 2020 dollars. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- Commercial production will start up in July 1, 2024
- Construction will take 1.5 years, beginning January 1, 2023.
- The life of the mine will be 13.5 years.
- A base case gold price of US$1,600/oz and silver price of US$20/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.
- A United States to Canadian dollar exchange rate of 0.75 (USD/CAD) was used.
- Cost estimates are provided in constant Q4 2020 Canadian dollars with no inflation or escalation factors considered.
- Results are based on 100% ownership with an average 0.04% net smelter return (NSR) for Goliath, 2.12% NSR Goldlund, and 0% NSR for Miller.
- Capital costs will be funded with 100% equity (i.e., no financing costs assumed).
- All cash flows have been discounted to the start of construction (January 1, 2023)
- All metal products will be sold in the same year they are produced.
- Project revenue is derived from the sale of gold doré.
- No contractual arrangements for refining currently exist.

22.3.1 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was compiled by Treasury Metals with assistance from third-party taxation professionals. The calculations are based on the tax regime as of the date of the PEA study and include estimates for Treasury Metal's expenditures and related impacts to various tax pool balances between the PEA study and the assumed construction start date.

At the effective date of this report, the project was assumed to be subject to the following tax regime:

- Canadian corporate income tax system and provincial income tax
- mining tax calculated in accordance with the *Ontario Mining Tax Act*
- total undiscounted tax payments of C$216 M over the life of mine

The tax evaluation was completed by applying the following assumptions:

- A 10% Ontario Provincial Income Tax Rate (applicable to manufacturing and processing) would be applied to this project.
- All royalty payments are deductible for income tax purposes and non-deductible for Ontario Mining Tax purposes.
- Operating expenses and Refining charges are fully deductible for income and mining tax purposes.
- The opening balance of non-capital losses carry forward corresponds to the closing balance as per the T2 filed for Treasury Metals Inc. for fiscal year 2019 ("FY 2019 T2"). An assumption has been considered that the entire pool available to the legal entity can be utilised to offset taxable income generated by this particular project. The cumulative non-capital losses as of December 31, 2019 expire at various years from 2026 onwards. The projections currently indicate the existing non-capital losses will be fully utilised.
- The opening balance of Canadian Exploration Expenses (CEE) corresponds to the closing balance of regular CEE as per the FY 2019 T2. An assumption has been considered that the entire pool available to the legal entity can be utilised to offset taxable income generated by this particular project.
- The opening balance of Canadian Development Expenses (CDE) corresponds to the closing balance of regular CDE as per the FY 2019 T2. An assumption has been considered that the entire pool available to the legal entity can be utilised to offset taxable income generated by this particular project.
- The opening balance of Input Tax Credits (ITCs) corresponds to the closing balance as per the FY 2019 T2. Since the balance on the return is nil, the projections currently indicate no available ITC's.
- The opening balance of Undepreciated Capital Cost (UCC) corresponds to the closing balance as per the FY 2019 T2. An assumption has been considered that the entire pool available to the legal entity can be utilised to offset taxable income generated by this particular project.
- For simplicity purposes, accelerated deductions of CDE and UCC under the Accelerated Investment Incentive have not been considered, as it would result in a timing impact only.
- Property, Plant and Equipment (PP&E) is expected to be disposed of by the end of the life of the mine and salvage proceeds are expected to be received. As a result, a terminal loss is being considered for all Capital Cost Allowance (CCA) classes in 2037. The salvage proceeds were added back in the taxable income calculation.
- The model currently indicated that non-capital losses will be generated in FY 2038 (last year of operation). Such losses can be carried back to the three previous taxation years. The model reflects a loss carry back.
- The opening balance of the exploration and development pool corresponds to the CEE and CDE pools for income tax purposes as per the 2019 T2 filed by Treasury Metals Inc. An assumption has been considered that the entire CEE and CDE pool available to the legal entity qualify as exploration and development expenditures incurred in Ontario that

can be utilised to offset Ontario Mining Tax taxable income generated by this particular project.

- The opening balance of the mining assets pool has been assumed to be nil as there are no class 41 assets reported on schedule 8 of the FY 2019 T2.
- For a 36-month period, the first $10 million of profits generated by a new non-remote mine or major expansion of an existing non-remote mine is exempt from tax ($10 million exemption). It has been assumed the Goliath Gold Complex will eventually qualify as a non-remote mine and the template has been set up to enable the $10 million exemption deduction.
- The *Ontario Mining Tax Act* indicates the tax depreciation of fixed assets of mining, transportation and processing asset should be determined based on straight-line method (30% for mining assets and 15% for processing and transportation assets). No minimum depreciation is required to be claimed, except during the period the $10 million exemption is being claimed, when 30% must be taken for mining assets and 15% must be taken for processing and transportation assets. The template considers a tax depreciation based on declining balance method.
- It has been assumed that the processing assets indicated in the capital cost analysis would qualify as "Concentrators, smelters and refineries in Canada but not in Northern Ontario", which would be subject to a 16% rate. The model assumes no disposals of processing assets during the life of the mine.
- Based on an administrative policy from the Ontario Ministry, it has been assumed the closing costs expected to be incurred in the last forecasted year could be carried back for one year.

22.3.2 Working Capital

An estimation of working capital has been incorporated into the cash flow based on accounts receivable (30 days), inventories (0 days) and accounts payable (30 days).

22.3.3 Closure Costs & Salvage Value

Closure costs of C$24 million were applied at the end of the life of mine. The costs include site closure for Goliath, Goldlund and Miller, as well as the requisite post-closure monitoring and inspection programs for a 50-year period discounted to the time of closure.

A salvage value of C$9.7 million was applied at the end of the life of mine, as well as sales of mining equipment in Year 5 (C$1.2 million) and Year 6 (C$0.9 million).

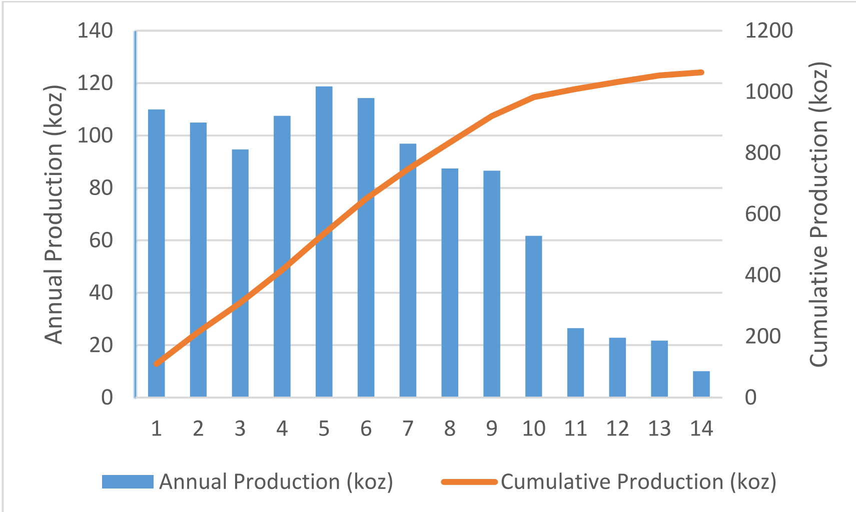
22.4 Royalties

Based on the agreements in place, 0.04% NSR for Goliath and 2.12% NSR Goldlund, and 0.00% NSR for Miller has been assumed for the Project, resulting in approximately C$23 million in undiscounted royalty payments over life of mine.

22.5 Metal Production

Over the life of mine, a total of 1,064 koz of gold (average annual: 78,807 oz) and 844 koz of silver will be produced. Gold production for the life of mine is shown on Figure 22-1.

Figure 22-1: Gold Production Profile over the Life of Mine



Source: Ausenco (2021).

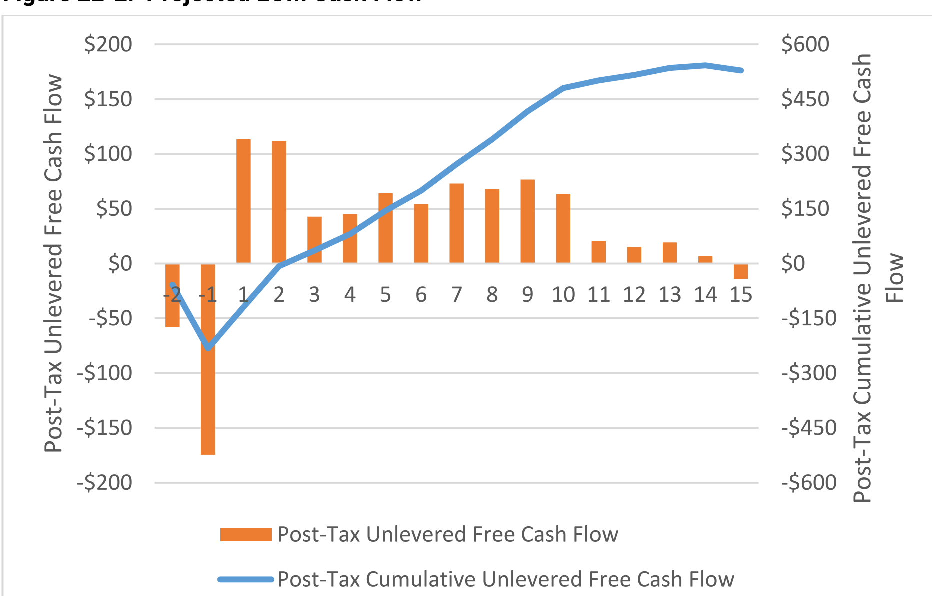
22.6 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. Cash flows have been discounted to the start of construction (January 1, 2023), assuming the project execution decision will be made and major project financing would be carried out at this time.

The pre-tax net present value discounted at 5% (NPV5%) is C$477 million, the internal rate of return IRR is 37.3%, and payback is 1.9 years. On an after-tax basis, the NPV5% is C$328 million, the IRR is 30.2%, and the payback period is 2.2 years.

A summary of project economics is shown graphically on Figure 22-1 and listed in Table 22-1. The cash flow on an annualised basis is provided in Table 22-2.

Figure 22-2: Projected LOM Cash Flow



Source: Ausenco (2021).

Table 22.1: Summary, Project LOM Cash Flow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,600
Exchange Rate (USD:CAD)	0.75
Mine Life (years)	13.5
Total Waste Tonnes Mined (kt)	82.452
Total Mill Feed Tonnes (kt)	23,966
Strip Ratio (waste:mineralisation)	3.93
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	1.47
Mill Recovery Rate (%)	93.6%
Total Mill Ounces Recovered (koz)	1,064
Average Annual Production (koz)	79
Operating Costs	**LOM Total / Avg.**
Mining Cost – Open Pit (C$/t Mined)	$3.27
Mining Cost – Open Pit (C$/t Milled)	$16.95
Mining Cost – Underground (C$/t Milled)	$70.31
Processing Cost (C$/t Milled)	$11.37
G&A Cost (C$/t Milled)	$2.28
Gold Refining (C$/oz Au)	$14.00
Silver Refining (C$/oz Ag)	$0.26
Total Operating Costs (C$/t Milled)	$40.70
Cash Costs* (US$/oz Au)	$699
All-in Sustaining Cost (AISC)** (US$/oz Au)	$911
Capital Costs	**LOM Total / Avg.**
Initial Capital (C$M)	$233
Sustaining Capital (C$M)	$290
Closure Costs (C$M)	$24
Salvage Costs (C$M)	$12
Financials - Pre Tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$477
IRR (%)	37.3%
Payback (years)	1.9
Financials - Post Tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$328
IRR (%)	30.2%
Payback (years)	2.2

Notes: *Cash costs consist of mining costs, processing costs, mine-level general & administrative expenses and refining charges and royalties. **AISC includes cash costs plus sustaining capital, closure cost and salvage value. Source: Ausenco (2021).

Table 22.2: Project LOM Post-Tax Unlevered Free Cash Flow

	Year	Total/Avg.	-2	-1	1	2	3	4	5	6	7
	Unit		2023	2024	2025	2026	2027	2028	2029	2030	2031
Production Summary											
Resource Sent to Mill	kt	23,966	--	--	1,530	1,800	1,800	1,800	1,800	1,800	1,800
Head Grade (Au Diluted)	g/t	1.47	--	--	2.37	1.93	1.76	1.98	2.16	2.09	1.78
Head Grade (Ag Diluted)	g/t	1.82	--	--	1.54	1.57	--	1.27	4.02	2.97	2.65
Gold Recovered	koz	1,064	--	--	109.9	104.9	94.7	107.5	118.8	114.3	96.9
Silver Recovered	koz	844	--	--	45.3	54.4	--	44.1	139.7	103.1	92.1
Gold Payable	koz	1,064	--	--	109.9	104.9	94.7	107.5	118.8	114.3	96.9
Silver Payable	koz	844	--	--	45.3	54.4	--	44.1	139.7	103.1	92.1
Revenue											
Gold Price	US$/oz	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600	$1,600
Silver Price	US$/oz	$20	$20	$20	$20	$20	$20	$20	$20	$20	$20
Exchange Rate	USD:CAD	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Gross Revenue	C$M	$2,292	--	--	$236	$225	$202	$231	$257	$247	$209
Operating Costs											
Mine Operating Costs	C$M	($648)	--	--	($56)	($67)	($69)	($80)	($80)	($69)	($60)
Mill Processing Costs	C$M	($273)	--	--	($18)	($20)	($20)	($20)	($20)	($20)	($20)
G&A Costs	C$M	($55)	--	--	($4)	($4)	($4)	($4)	($4)	($4)	($4)
Refining & Royalties											
Refining	C$M	($15)	--	--	($1.6)	($1.5)	($1.3)	($1.5)	($1.7)	($1.6)	($1.4)
Royalties	C$M	($23)	--	--	($4)	($3)	($4)	($4)	($1)	($2)	($1)
Capital Expenditures											
Initial Capital	C$M	($233)	($58)	($175)	--	--	--	--	--	--	--
Sustaining Capital	C$M	($290)	--	--	($26)	($2)	($44)	($52)	($51)	($73)	($25)
Closure Cost	C$M	($24)	--	--	--	--	--	--	--	--	--
Salvage Value	C$M	$12	--	--	--	--	--	--	$1	$1	--
Change in Working Capital											
Change in Working Capital	C$M	--	--	--	($13)	$2	$2	($1)	($2)	($0)	$2
Pre-Tax Unlevered Free Cash Flow											
Pre-Tax Unlevered Free Cash Flow	C$M	$745	($58)	($175)	$114	$130	$61	$68	$98	$78	$100
Pre-Tax Cumulative Unlevered Free Cash Flow	C$M		($58)	($233)	($119)	$11	$71	$139	$237	$315	$415
Taxes											
Unlevered Cash Taxes	C$M	($216)	--	--	--	($18)	($18)	($23)	($33)	($24)	($27)
Post-Tax Unlevered Free Cash Flow											
Post-Tax Unlevered Free Cash Flow	C$M	$528	($58)	($175)	$114	$112	$43	$45	$64	$54	$73
Post-Tax Cumulative Unlevered Free Cash Flow	C$M		($58)	($233)	($119)	($7)	$36	$81	$145	$199	$272

22.7 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV, IRR and payback period of the project using the following variables: gold price, foreign exchange rate, discount rate, mill recovery, initial capital costs, and operating costs.

Table 22-3 summarises the post-tax sensitivity analysis results; Table 22-4 shows the pre-tax sensitivity analysis findings; and Table 22-5 shows the results post-tax.

As shown in Figures 22-3 and 22-4, the sensitivity analysis revealed that the project is most sensitive to changes in gold price, foreign exchange rate and recovery, and then to a lesser extent, operating costs and capital costs.

Table 22.3: Post-Tax Sensitivity Summary

Gold Price	Post-Tax NPV (5%)	Initial CAPEX		Total OPEX		FX	
US$/oz	Base Case	(-25%)	(+25%)	(-25%)	(+25%)	(-25%)	(+25%)
$1,200	$47	$101	($8)	$170	($93)	$331	($163)
$1,400	$189	$244	$134	$308	$66	$513	($15)
$1,600	$328	$383	$273	$445	$208	$694	$102
$1,850	$498	$553	$443	$615	$381	$921	$243
$2,000	$600	$655	$545	$717	$484	$1,057	$326
Gold Price	**Post-Tax IRR**	**Initial CAPEX**		**Total OPEX**		**FX**	
US$/oz	**Base Case**	**(-25%)**	**(+25%)**	**(-25%)**	**(+25%)**	**(-25%)**	**(+25%)**
$1,200	9.3%	16.9%	4.4%	19.0%	0.0%	30.4%	0.0%
$1,400	20.7%	31.0%	14.3%	28.5%	11.3%	41.5%	3.5%
$1,600	30.2%	42.7%	22.4%	37.1%	22.5%	51.4%	14.1%
$1,850	40.7%	55.6%	31.3%	46.8%	34.0%	62.7%	24.6%
$2,000	46.4%	62.6%	36.2%	52.2%	40.2%	69.2%	30.1%

Table 22.4: Pre-Tax Sensitivity Analysis

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
1.0%	$155	$416	$678	$808	$939
3.0%	$114	$341	$569	$682	$796
5.0%	$79	$278	$477	$577	$676
8.0%	$38	$203	$368	$450	$533
10.0%	$16	$162	$309	$382	$455

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
1.0%	11.6%	25.5%	37.3%	42.8%	48.0%
3.0%	11.6%	25.5%	37.3%	42.8%	48.0%
5.0%	11.7%	25.5%	37.3%	42.8%	48.0%
8.0%	11.7%	25.5%	37.3%	42.8%	48.0%
10.0%	11.7%	25.5%	37.3%	42.8%	48.0%

Pre-Tax Payback Sensitivity To Discount Rate

Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
1.0%	6.5	3.2	1.9	1.7	1.6
3.0%	6.5	3.2	1.9	1.7	1.6
5.0%	6.5	3.2	1.9	1.7	1.6
8.0%	6.5	3.2	1.9	1.7	1.6
10.0%	6.5	3.2	1.9	1.7	1.6

Pre-Tax NPV Sensitivity To FX

FX	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	$382	$630	$879	$1,004	$1,128
(10.0%)	$214	$435	$656	$766	$877
--	$79	$278	$477	$577	$676
10.0%	($31)	$150	$331	$422	$512
20.0%	($122)	$44	$209	$292	$375

Pre-Tax IRR Sensitivity To FX

FX	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	31.8%	45.6%	58.1%	64.0%	69.7%
(10.0%)	21.3%	34.9%	47.0%	52.6%	58.0%
--	11.7%	25.5%	37.3%	42.8%	48.0%
10.0%	2.1%	17.0%	28.8%	34.2%	39.3%
20.0%	0.0%	8.8%	21.1%	26.4%	31.5%

Pre-Tax Payback Sensitivity To FX

FX	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	2.3	1.7	1.4	1.2	1.1
(10.0%)	4.1	2.0	1.6	1.5	1.4
--	6.5	3.2	1.9	1.7	1.6
10.0%	9.1	4.8	2.7	2.1	1.8
20.0%	-	7.2	4.1	3.0	2.3

Pre-Tax NPV Sensitivity To Opex

Opex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	$215	$415	$614	$713	$813
(10.0%)	$147	$346	$545	$645	$744
--	$79	$278	$477	$577	$676
10.0%	$11	$210	$409	$509	$608
20.0%	($57)	$142	$341	$441	$540

Pre-Tax IRR Sensitivity To Opex

Opex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	21.1%	33.4%	44.3%	49.5%	54.4%
(10.0%)	16.6%	29.5%	40.9%	46.2%	51.3%
--	11.7%	25.5%	37.3%	42.8%	48.0%
10.0%	6.0%	21.2%	33.6%	39.3%	44.7%
20.0%	0.0%	16.6%	29.8%	35.7%	41.2%

Pre-Tax Payback Sensitivity To Opex

Opex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	4.1	2.2	1.7	1.6	1.4
(10.0%)	4.8	2.6	1.8	1.6	1.5
--	6.5	3.2	1.9	1.7	1.6
10.0%	8.1	4.1	2.1	1.8	1.7
20.0%	9.8	4.9	2.5	2.0	1.8

Pre-Tax NPV Sensitivity To Capex

InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	$123	$322	$521	$621	$720
(10.0%)	$101	$300	$499	$599	$698
--	$79	$278	$477	$577	$676
10.0%	$57	$256	$455	$555	$654
20.0%	$35	$235	$434	$533	$633

Pre-Tax IRR Sensitivity To Capex

InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	17.4%	33.5%	47.3%	53.7%	59.9%
(10.0%)	14.3%	29.1%	41.8%	47.7%	53.4%
--	11.7%	25.5%	37.3%	42.8%	48.0%
10.0%	9.5%	22.5%	33.6%	38.7%	43.6%
20.0%	7.6%	20.0%	30.4%	35.2%	39.8%

Pre-Tax Payback Sensitivity To Capex

InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(20.0%)	4.8	2.0	1.6	1.4	1.3
(10.0%)	5.9	2.6	1.7	1.6	1.4
--	6.5	3.2	1.9	1.7	1.6
10.0%	6.9	3.8	2.2	1.9	1.7
20.0%	7.5	4.2	2.6	2.1	1.9

Pre-Tax NPV Sensitivity To Recovery Mill

Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(-)2.0%	$53	$248	$443	$540	$638
(-)1.0%	$66	$263	$460	$559	$657
--	$79	$278	$477	$577	$676
(+)1.0%	$92	$293	$494	$595	$696
(+)2.0%	$105	$308	$512	$613	$715

Pre-Tax IRR Sensitivity To Recovery Mill

Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(2.0%)	9.6%	23.6%	35.4%	40.8%	46.1%
(1.0%)	10.7%	24.6%	36.4%	41.8%	47.0%
--	11.7%	25.5%	37.3%	42.8%	48.0%
1.0%	12.7%	26.5%	38.3%	43.8%	49.0%
2.0%	13.6%	27.4%	39.2%	44.7%	50.0%

Pre-Tax Payback Sensitivity To Recovery Mill

Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
	Gold Price (US$/oz)				
(2.0%)	7.0	3.5	2.0	1.8	1.6
(1.0%)	6.7	3.4	2.0	1.8	1.6
--	6.5	3.2	1.9	1.7	1.6
1.0%	6.2	3.0	1.9	1.7	1.6
2.0%	6.0	2.9	1.9	1.7	1.5

Table 22.5: Post-Tax Sensitivity Analysis

Post-Tax NPV Sensitivity To Discount Rate

	Gold Price (US$/oz)				
Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
1.0%	$108	$294	$477	$567	$656
3.0%	$75	$237	$396	$474	$552
5.0%	$47	$189	$328	$396	$464
8.0%	$13	$131	$246	$302	$359
10.0%	($5)	$100	$202	$252	$303

Post-Tax IRR Sensitivity To Discount Rate

	Gold Price (US$/oz)				
Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
1.0%	9.2%	20.7%	30.2%	34.5%	38.6%
3.0%	9.3%	20.7%	30.2%	34.5%	38.6%
5.0%	9.3%	20.7%	30.2%	34.5%	38.6%
8.0%	9.4%	20.8%	30.2%	34.5%	38.7%
10.0%	9.4%	20.8%	30.2%	34.5%	38.7%

Post-Tax Payback Sensitivity To Discount Rate

	Gold Price (US$/oz)				
Discount Rate	$1,200	$1,400	$1,600	$1,700	$1,800
1.0%	6.9	3.7	2.2	1.9	1.8
3.0%	6.9	3.7	2.2	1.9	1.8
5.0%	6.9	3.7	2.2	1.9	1.8
8.0%	6.9	3.7	2.2	1.9	1.8
10.0%	6.9	3.7	2.2	1.9	1.8

Post-Tax NPV Sensitivity To FX

	Gold Price (US$/oz)				
FX	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	$261	$433	$603	$688	$773
(10.0%)	$143	$298	$450	$526	$601
--	$47	$189	$328	$396	$464
10.0%	($37)	$98	$226	$289	$352
20.0%	($124)	$20	$141	$199	$257

Post-Tax IRR Sensitivity To FX

	Gold Price (US$/oz)				
FX	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	25.8%	36.7%	46.5%	51.0%	55.4%
(10.0%)	17.3%	28.3%	37.8%	42.2%	46.4%
--	9.3%	20.7%	30.2%	34.5%	38.6%
10.0%	1.4%	13.8%	23.4%	27.7%	31.7%
20.0%	0.0%	6.9%	17.1%	21.5%	25.5%

Post-Tax Payback Sensitivity To FX

	Gold Price (US$/oz)				
FX	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	2.7	1.8	1.5	1.4	1.3
(10.0%)	4.5	2.4	1.8	1.7	1.5
--	6.9	3.7	2.2	1.9	1.8
10.0%	9.3	5.5	3.1	2.5	2.0
20.0%	-	7.6	4.5	3.5	2.8

Post-Tax NPV Sensitivity To Opex

	Gold Price (US$/oz)				
Opex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	$146	$285	$421	$489	$557
(10.0%)	$97	$237	$375	$443	$511
--	$47	$189	$328	$396	$464
10.0%	($5)	$140	$280	$349	$417
20.0%	($60)	$91	$232	$302	$370

Post-Tax IRR Sensitivity To Opex

	Gold Price (US$/oz)				
Opex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	17.2%	27.1%	35.7%	39.8%	43.7%
(10.0%)	13.5%	24.0%	33.0%	37.2%	41.2%
--	9.3%	20.7%	30.2%	34.5%	38.6%
10.0%	4.5%	17.2%	27.2%	31.7%	36.0%
20.0%	0.0%	13.4%	24.1%	28.8%	33.2%

Post-Tax Payback Sensitivity To Opex

	Gold Price (US$/oz)				
Opex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	4.5	2.6	1.9	1.7	1.6
(10.0%)	5.6	3.0	2.0	1.8	1.7
--	6.9	3.7	2.2	1.9	1.8
10.0%	8.3	4.5	2.5	2.0	1.9
20.0%	10.0	5.6	3.0	2.3	1.9

Post-Tax NPV Sensitivity To Capex

	Gold Price (US$/oz)				
InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	$90	$233	$372	$440	$508
(10.0%)	$69	$211	$350	$418	$486
--	$47	$189	$328	$396	$464
10.0%	$25	$167	$306	$374	$442
20.0%	$3	$145	$284	$352	$420

Post-Tax IRR Sensitivity To Capex

	Gold Price (US$/oz)				
InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	15.1%	28.5%	39.6%	44.7%	49.6%
(10.0%)	11.9%	24.2%	34.4%	39.1%	43.5%
--	9.3%	20.7%	30.2%	34.5%	38.6%
10.0%	7.1%	17.9%	26.7%	30.7%	34.6%
20.0%	5.2%	15.4%	23.7%	27.5%	31.1%

Post-Tax Payback Sensitivity To Capex

	Gold Price (US$/oz)				
InitialCapex	$1,200	$1,400	$1,600	$1,700	$1,800
(20.0%)	5.0	2.0	1.6	1.5	1.4
(10.0%)	6.2	2.8	1.9	1.7	1.6
--	6.9	3.7	2.2	1.9	1.8
10.0%	7.5	4.4	2.7	2.2	2.0
20.0%	8.1	4.9	3.2	2.7	2.3

Post-Tax NPV Sensitivity To Recovery Mill

	Gold Price (US$/oz)				
Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
(-)2.0%	$28	$168	$304	$371	$438
(-)1.0%	$37	$178	$316	$384	$451
--	$47	$189	$328	$396	$464
(+)1.0%	$56	$199	$340	$408	$477
(+)2.0%	$66	$210	$351	$421	$490

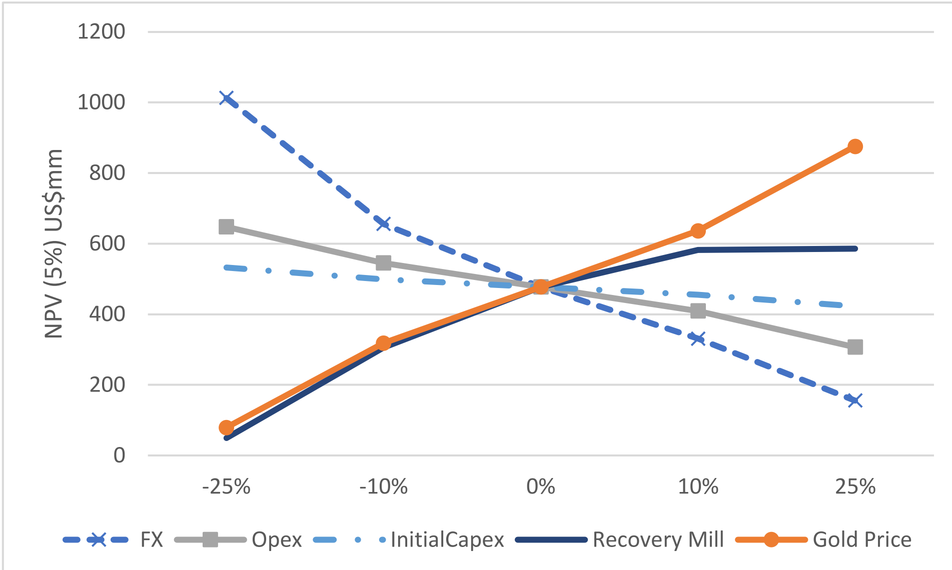
Post-Tax IRR Sensitivity To Recovery Mill

	Gold Price (US$/oz)				
Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
(2.0%)	7.6%	19.2%	28.7%	33.0%	37.1%
(1.0%)	8.5%	20.0%	29.4%	33.7%	37.9%
--	9.3%	20.7%	30.2%	34.5%	38.6%
1.0%	10.2%	21.5%	31.0%	35.3%	39.4%
2.0%	11.0%	22.2%	31.7%	36.0%	40.2%

Post-Tax Payback Sensitivity To Recovery Mill

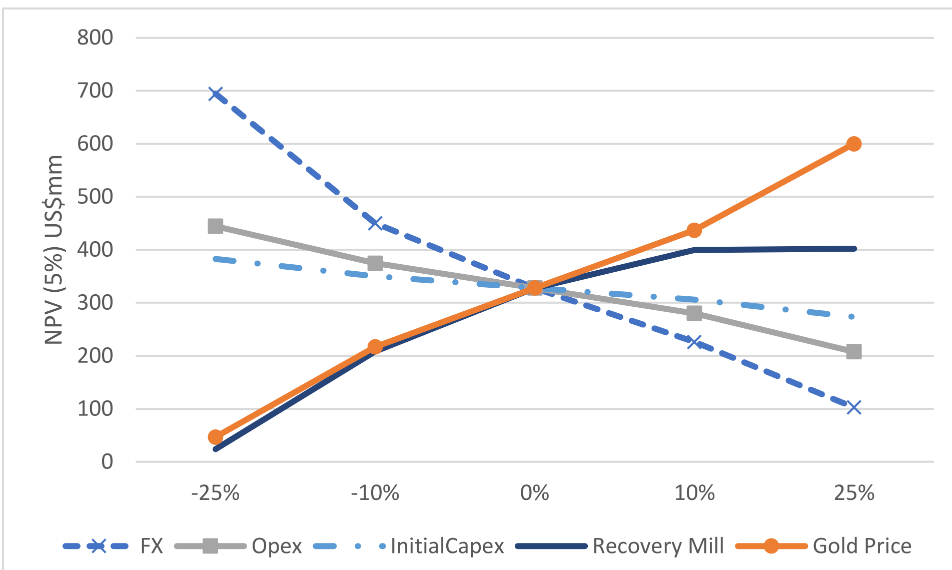
	Gold Price (US$/oz)				
Recovery Mill	$1,200	$1,400	$1,600	$1,700	$1,800
(2.0%)	7.4	4.1	2.3	2.0	1.8
(1.0%)	7.1	3.9	2.2	1.9	1.8
--	6.9	3.7	2.2	1.9	1.8
1.0%	6.6	3.5	2.1	1.9	1.7
2.0%	6.4	3.4	2.0	1.9	1.7

Figure 22-3: Pre-Tax Sensitivity – Spider Chart



Source: Ausenco (2021).

Figure 22-4: Post-Tax Sensitivity – Spider Chart



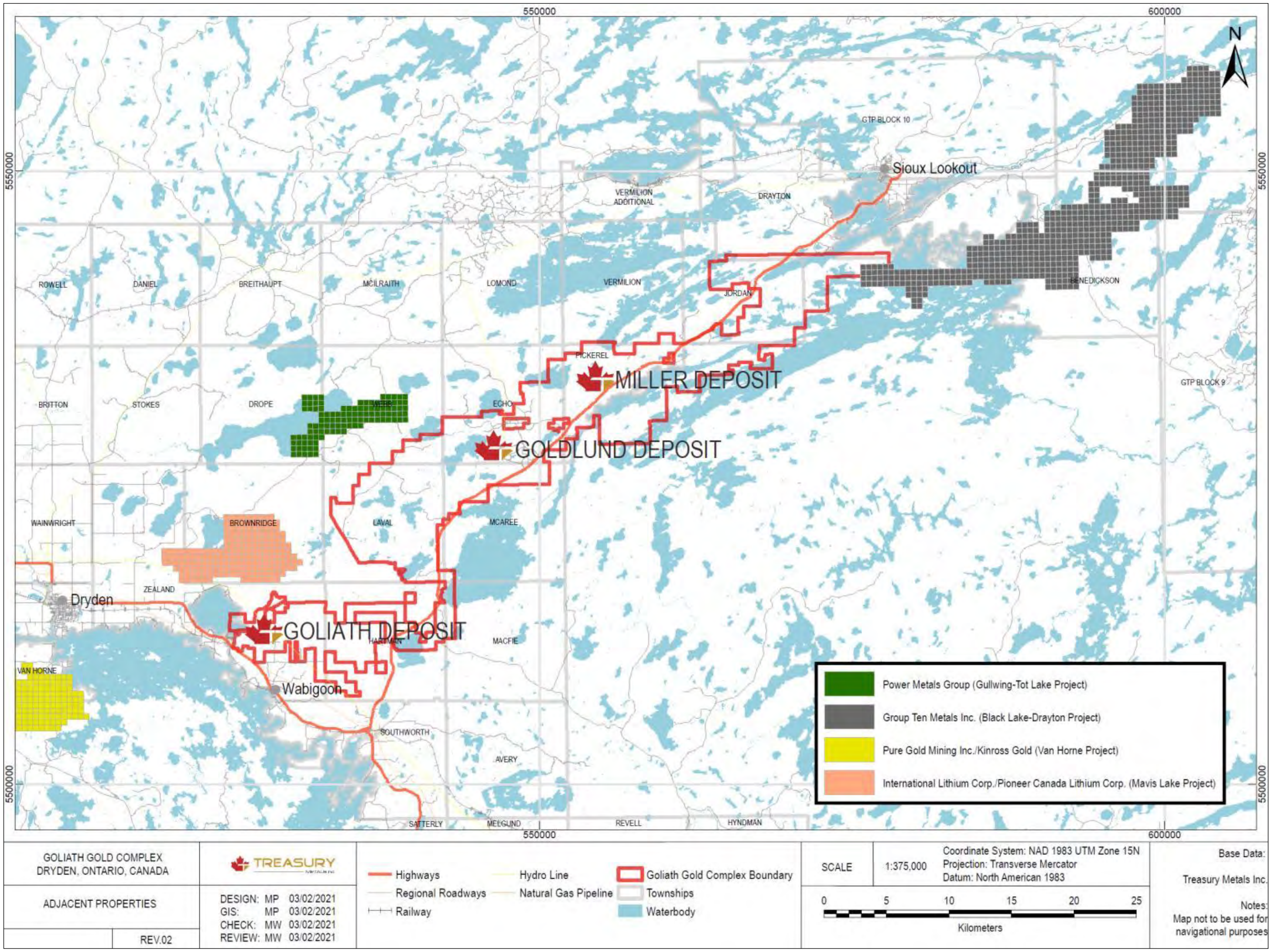
Source: Ausenco (2021).

23 ADJACENT PROPERTIES

Treasury Metals does not hold or control any additional properties in the immediate vicinity. Treasury Metals does hold additional claims at its Goldrock property held under its subsidiary holding of Goldeye Exploration Inc.; these are not the subject of this report and are not included in any analysis.

Various companies have several projects in the region that are at different stages of exploration and development, and are within the vicinity of the Goliath Gold Complex. Figure 23-1 shows the location of known mineral projects in the vicinity of the Goliath Gold Complex. Projects of note include the joint venture project between Kinross and Pure Gold known as the Van Horne property and Group 10's Black Lake-Drayton Project. Accompanying these gold-focused projects are a number of smaller holdings, and those focused on lithium and battery metals. These projects include the Gull Wing-Tot Lake Project held by Power Metals and Mavis Lake Project held by International Lithium Corporation.

Figure 23-1: Adjacent Properties



Source: Treasury Metals (2021).

24 OTHER RELEVANT DATA & INFORMATION

There is no other relevant data or information to add to this report.

25 INTERPRETATION & CONCLUSIONS

25.1 Geology

The quantity and grade of inferred resources reported in this section are conceptual in nature and are estimated on the basis of limited geological evidence and sampling. The geological evidence is sufficient to imply, but not verify, the geological, grade or quality and continuity of the gold mineralisation. For these reasons, an inferred mineral resource has a lower level of confidence than an indicated mineral resource and it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The Qualified Persons for this section of the report are unaware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the mineral resource estimate.

25.1.1 Goliath

The Goliath Project is located 20 km east of the City of Dryden in northwestern Ontario, within the Townships of Zealand and Hartman in the Kenora Mining Division. The property is centred at approximately 532,441mE and 5,511,624mN UTM NAD83 Zone 15N or longitude W92°32'58" and latitude N49°45'22".

The Goliath Project is located in the Archean Eagle-Wabigoon-Manitou greenstone belt in the Wabigoon Subprovince of the Superior Province. In the immediate area of the deposit, a 100 to 150 m thick unit of intensely deformed and variably altered, fine- to medium-grained, quartz-feldspar-sericite schist and biotite-quartz-feldspar-sericite schist with minor metasedimentary rocks hosts the most significant gold concentrations of gold in the Main and C Zones of the deposit.

Native gold and silver are associated with finely disseminated sulphides, coarse-grained pyrite and very narrow light grey translucent “ribbon” quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite. The alteration consists of primarily sericitisation and silicification in association with the gold mineralisation.

At Goliath, the gold-bearing zones strike from 090° to 072° with dips that are consistently between 70° and 80° south or southeast. The mineralised zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. In the Goliath deposit, higher grade gold mineralisation occurs in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west. The main area of gold, silver and sulphide mineralisation and alteration occurs up to a maximum drill-tested vertical depth of ~805 m over a drill-tested strike-length in excess of 2,500 m. Gold mineralised zones remain open at depth.

Based on the review of the QA/QC, data validation, and statistical analysis the following conclusions were made:

- The regional geology, lithological, and structural controls on the mineralisation at the Goliath deposit are well understood by Treasury Metals' exploration team.
- AGP has reviewed the methods and procedures to collect and compile geological, geotechnical, and assaying information for the Goliath deposit and found them to be suitable for the style of mineralisation found on the property and meet accepted industry standards.
- The mineralisation was sampled over the years with multiple campaigns of core drilling by Teck-Corona and Treasury Metals since the 1990s. The drill database is now a mix of historical data and more recent data collected by Treasury Metals from 2008 and 2020. Both data types were used in the resource estimate.
- The analytical laboratory used by Teck-Corona prior to the 1990s is believed to be TSL Laboratory in Saskatoon. Assays from that period were recovered from historical drill logs. Treasury Metals used Accurassay Laboratory located in Thunder Bay from 2008 to 2015 and then Activation Laboratory from 2016 to 2020. Accurassay was accredited by ISO/IEC 17025 and ActLab in Dryden was assessed and found to be in conformance to the ISO 9001:2015 standard.
- Treasury Metals' drill core are analysed for gold on all samples and silver and trace element geochemistry on selected samples. Gold is typically analysed by fire assay with AA finish or gravimetric finish depending on the grade. Pulp metallic screen assays are routinely carried out on high-grade samples. Silver and trace elements are typically assayed using aqua regia digest followed by ICP-OES.
- AGP also noted that a significant portion (30.8%) of the gold assays within the mineralised zones are missing a corresponding silver assay. While silver does not contribute significantly to the resource, it is nevertheless carried as an estimated grade element and as such, every effort should be made to ensure that the material within the mineralised horizon is fully assayed for both gold and silver if the core rejects or pulps are available.
- Prior to 1997, only a few QA/QC guidelines existed, and monitoring programs were not commonly conducted by mining companies. Consequently, a QA/QC program for the historical Teck-Corona drill holes is not known to exist and it was assumed by AGP to be non-existent. In 2008 Treasury Metals implemented a QA/QC program consisting of blanks and CRMs. In 2009 Treasury Metals added the insertion of quarter core duplicates, and in 2017, added a check assay program at an umpire laboratory. The program was found to be well-followed with resubmission of sample batches when a QA/QC failure occurred.
- Submission rates meet the industry-accepted practice for each of the QA/QC type of samples. The sampling procedures, analytical methods, and QA/QC procedures undertaken by Treasury Metals indicate reasonable accuracy of the sample data and no systemic cross-contamination at the sample preparation or analytical level. Based upon the evaluation of the QA/QC program undertaken by Treasury Metals, it is AGP's opinion that the results are acceptable for use in the current mineral resource estimate.
- Historical drill holes were validated via twin drilling. For the holes that were examined by AGP, it was found that the twin drill hole compared relatively well between the paired holes. The higher-grade spikes and lower-grade sections are generally well reflected in both drill holes.
- The majority (81%) of the 545 bulk density sample measurements were carried out on 10 cm core pieces submitted to the analytical laboratory. The remaining 19% were completed in-

house on uncoated, air dry samples. The core at Goliath is solid with little to no pore, and the in-house density measurements compared well with the laboratory.

- Through site visits in 2020, AGP verified data, collected independent character samples, and audited the database. The drill database was found to be virtually error-free and suitable to be used for a resource estimate.
- Core handling, core storage, and chain of custody are consistent with industry best practices.

Based on the above conclusions and effective as of December 16, 2020, AGP completed an update of the July 1, 2019 estimate completed by P&E Mining Consultants Inc. The mineral resource estimate (MRE) presented herein is in conformance with the CIM's mineral resource definitions (2014) referred to in the "N.I. 43-101 Standards of Disclosure for Mineral Projects". The estimate takes into account all data that was available prior to October 6, 2020.

The MRE is supported by 726 surface drill holes with an aggregated length of 238,036 m and 96,912 assays. The estimate was completed based on the concept of a medium open pit and underground operation.

To meet the CIM definitions of reasonable prospects of economic extraction, a cut-off of 0.25 g/t Au for the resource amenable to open pit extraction and a cut-off of 1.6 g/t Au was used for the material below the resource constraining shell that are considered to be amenable to underground extraction. The determination of the cut-off grade was based on a gold price of US$1,700/oz and a silver price of US$23/oz with 95.5% gold and 62.6% silver recoveries.

To further assess reasonable prospects of economic extraction, a Lerchs-Grossman optimised shell was generated to constrain the potential open pit material. Grade shells at the underground cut-off grade of 1.6 g/t Au were generated beneath the resource pit shell. The grade shells were examined by AGP's engineering team for the likelihood of being a coherent mining shape with reasonable prospect of being accessed. Those that did not meet the criteria were removed from consideration.

The MRE presented herein is categorised as a mix of measured, indicated, and inferred resources. The reported resources are expressed in metric tonnes. Metal contents are presented as in-situ ounces.

Within the resource constraining shell at the greater than 0.25 g/t Au cut-off grade selected, the updated model returns a total of 1.5 million measured tonnes grading at 1.90 g/t Au and 6.7 g/t Ag containing 89,800 oz of gold and 316,700 oz of silver. Indicated tonnes amounted to 27.0 Mt grading at 0.87 g/t Au and 3.0 g/t Ag containing 757,000 oz of gold and 2.6 Moz of silver. The total measured and indicated resources within the constraining shell amounted to 28.4 Mt grading at 0.93 g/t Au and 3.2 g/t silver containing 846,800 oz of gold and 2.9 Moz of silver.

Below the constraining shell and reported at a greater than 1.6 g/t Au cut-off grade, the updated model returns 98,000 tonnes of measured resources grading at 4.94 g/t Au and 20.8 g/t Ag containing 15,500 oz of gold and 65,300 oz of silver. Indicated resources amounted to 2.6 Mt grading 3.16 g/t Au and 7.6 g/t Ag containing 263,100 oz of gold and 632,700 oz of silver. The total measured and indicated resources below the constraining shell amounted to 2.7 Mt grading at 3.22 g/t Au and 8.1 g/t Ag containing 278,700 oz of gold and 698,000 oz of silver.

Inferred resources within the resource constraining shell and reported at greater than 0.25 g/t Au cut-off grade, amounted to 3.6 Mt grading at 0.65 g/t Au and 2.1g/t Ag containing 76,100 oz of gold and 247,000 oz of silver. Below the constraining shell and reported at a greater than 1.6 g/t Au cut-off grade, the updated model returned 704,000 tonnes of inferred resources grading at 2.75 g/t Au and 5.6 g/t Ag containing 62,200 oz of gold and 125,900 oz of silver.

The Goliath deposit total measured resources amounted to 1.6 Mt grading at 2.09 g/t Au and 7.58 g/t Ag containing 105,300 oz of gold and 382,000 oz of silver. Indicated resources amounted to an additional 29.5 Mt grading 1.07 g/t Au and 3.39 g/t Ag containing 1.0 Moz of gold and 3.2 Moz of silver. The total measured and indicated resources amounted to 31.1 Mt grading at 1.13 g/t Au and 3.60 g/t Ag containing 1.1 Moz of gold and 3.6 Moz of silver. Inferred resources added an additional 4.3 Mt grading 0.99 g/t Au and 2.67 g/t Ag containing 138,300 gold oz and 372,900 oz of silver.

25.1.2 Goldlund

The Goldlund Project is situated in northwestern Ontario approximately 60 km by road east of the town of Dryden, with a land package that covers a strike-length of over 50 km of greenstone belt in the Archean Wabigoon Subprovince. Historical gold production from the Goldlund and Windward mines is reported to be 18,000 oz of gold, with mining activities carried out between 1982 and 1985 using both open pit and underground mining methods.

Gold mineralisation is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. The stockwork zones are hosted in albite-trondhjemite to diorite (granodiorite) strata-parallel sills, which dip from vertical to -80° southward and range in thickness from 14 m to 60 m. The stockwork zones form bands within the granodiorite sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite.

The mineralised sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the project.

The gold mineralisation has been interpreted as a series of nine northeast-trending sub-parallel zone wireframes, considering a nominal 0.1 g/t Au threshold. Wireframes of Zones 1, 7, and 5 consist principally of gold mineralisation associated with the stockwork veins in the large granodiorite sills, while wireframes of Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralisation associated with stockwork veins that are hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only a minor contribution from the granodiorite sills. While the Qualified Person for this section of the report believes that the interpretation of the mineralised zone wireframes is suitable for the estimation of mineral resources, the development of a 3D model of lithology, structure, and alteration would help to improve the interpretation of the mineralised zones and the understanding of the controls on gold mineralisation.

Drilling on the Goldlund Project spans a period from 1941 to 2020, with the drilling carried out by 11 different companies, and with assays carried out by five different assay laboratories.

The database was compiled from historical records including plan maps, drill logs, and assay certificates by Tamaka in 2010. Both Tamaka and later First Mining have added additional drilling to the database. There is a total of 1,771 drill holes in the database, totalling 176,498.3 m of drilling, with a total of 114,102 gold assays.

From 2007 to 2020, Tamaka and First Mining conducted several drilling programs in support of the estimation of mineral resources. The Qualified Person for this section of the report believes that the drilling, sampling, and the preparation and analyses have been carried out in accordance with industry standards and are appropriate for this style of gold mineralisation. As well, comparisons of the historical drill hole data with the more recent drilling from 2007 to 2020 indicate that the historical data is sufficiently similar such that the historical data can be pooled together with the recent data and used for the estimation of mineral resources.

The mineral resources for the Goldlund Project amenable to an open pit mining scenario, within an optimised constraining shell at a 0.26 g/t Au cut-off grade, are estimated to be 24.3 Mt of indicated material grading 1.07 g/t Au for a total of 840 koz of gold. There are additional inferred mineral resources amenable to an open pit mining scenario, which are estimated to be 14.4 Mt grading 0.56 g/t Au for a total of 260 koz of gold.

The mineral resources amenable to an underground mining scenario, for contiguous blocks below the optimised constraining shell, are estimated to be 233 kt grading 6.8 g/t Au totalling 51 koz of gold. This brings the total inferred mineral resources to be 14.6 Mt, grading 0.66 g/t Au totalling 311 koz of gold. The effective date of the Goldlund Project Mineral Resource Estimate is October 23, 2020.

All mineral resources estimates have a degree of uncertainty. For the Goldlund Project, these uncertainties are principally due to the historical drill hole data, the unsampled intervals, the extreme outlier gold grades, and the geological interpretation and geological model. There are several procedures that have been carried out to mitigate or minimise these uncertainties. The historical drill hole data has been verified using historical records and recent drilling to confirm that the data is sufficiently accurate such that it can be considered reliable and therefore suitable for the use in mineral resources estimation. All the unsampled intervals have been assigned a low-grade value prior to compositing and this strategy is consistent with the statistical and visual assessments using the results from recent drilling. The extreme gold grades have been capped to ensure these grades do not generate an overestimation of the mineral resources. The use of the CV partitioning and the development of the more statistically stable estimation domains to help improve the geological model used for the estimation of mineral resources. The use of the PAK estimation methodology to ensure that the block grade estimates honour the likely distribution of the selective mining unit and estimate an appropriate grade and tonnes at the range of suitable economic cut-off grades. Finally, the Goldlund Project mineral resources have been classified as only indicated and inferred resources. Currently, there are no measured resources at the Goldlund Project.

The mineralisation at the Goldlund Project remains open, both along strike to the northeast and at depth. The Qualified Person for this section of the report concludes that further exploration is warranted and recommended for the Goldlund Project.

Silver has been assayed in some drill core samples within the Goldlund Project. Currently there are insufficient silver assay results to model the silver mineralisation. It is recommended that the available drill core sample rejects be assayed for silver, and along with additional drilling, this may generate sufficient data to allow the estimation of silver as a byproduct in future mineral resource estimates.

The recommended exploration consists of both infill drilling to confirm the continuity of high-grade mineralisation and drilling to convert inferred to indicated mineralisation. Exploration drilling is also recommended to expand the Zone 01 mineralisation along strike to the northeast. Additional geological modelling is recommended that includes the development of a lithology and alteration model as well as a model of the high-grade mineralisation. This modelling will be supported by database and geological studies. Assaying of available drill core sample rejects for silver, along with additional drilling, may generate sufficient data to allow the estimation of silver as a by product in future mineral resource estimates.

25.1.3 Miller

The Miller Project is located on the Goldlund property and is situated approximately 10 km northeast and along strike of the Goldlund deposit, less than 3 km from Highway 72. The Miller deposit was discovered by First Mining in 2018 during a regional drill campaign and is relatively undeveloped in comparison to the Goldlund deposit.

The Miller deposit is analogous to the Goldlund deposit in that the gold mineralisation is hosted within stockworks of veins and veinlets that occur within a granodiorite and feldspar porphyry lithology. The granodiorite is hosted within a sequence of regional andesite and gabbro lithologies. The deposit has been outlined along a 500 m long strike length, with a width up to 50 m, and appears to be open at depth.

Drilling on Miller, by First Mining in 2018 and 2019, was completed by targeting a geophysical anomaly. A total of 40 drill holes, approximately 7,385 m, were completed in the area where 28 drill holes, approximately 4,980 m, intersected significant gold mineralisation within the Miller deposit.

The mineral resource estimate for the Miller deposit at a 0.26 g/t Au cut-off grade is 2.0 Mt of inferred resources at 1.24 g/t Au. The mineral resources for the Miller deposit are amenable to an open pit mining scenario and are reported within an optimised constraining shell. The effective date of the mineral resources is October 26, 2020.

AGP concludes that further exploration and development is warranted and recommended.

25.2 Mining

The mine designs and schedule, both for the open pit and underground, utilise inferred resources as part of the analysis. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The preliminary economic assessment is preliminary in nature in that it includes inferred mineral resources that are considered too speculative to have economic considerations applied to them and should not be relied upon for that purpose.

The Goliath Gold Complex PEA is based on the mining of three deposits, Goliath, Goldlund and Miller. All three areas would be mined by open pit methods, with Goliath also being mined underground, beneath the open pit. AGP's opinion is that with the current metal pricing levels and knowledge of the mineralisation, these methods of extraction offer the most reasonable approach for development.

The mine schedule provides 24.0 Mt of mill feed grading 1.47 g/t gold and 1.82 g/t silver over a 13.5-year mine life after one year of pre-stripping. The open pit mining sequence begins with Goliath in pre-production and then Goldlund starts in Year 1. Miller is started in Year 6 and finishes in Year 9. At that time, open pit mining is complete. The underground mine at Goliath starts in Year 3 with first delivery of mill feed in Year 4. Underground mining continues until Year 11. The processing facility will continue to be fed from stockpiles at Goliath until the middle of Year 14.

Mill feed from Goldlund and Miller are proposed to be transported to the Goliath process plant site with highway tractors and belly dump trucks. This transport will require the use of a portion of Highway 72, as well as an upgraded road across forestry lands to reduce traffic interaction and eliminate disturbance to the nearby communities.

25.2.1 Open Pit

The PEA has three pit areas (Goliath, Goldlund and Miller) with some having multiple phases. Goliath contains four phases with Phase 4 acting as the portal for the underground mine. Goldlund has six phases: two in the main pit area and four satellite pits. Miller is a single phase to be mined near the end of the project life. These provide a total of 21.0 Mt of open pit mill feed grading 1.16 g/t gold and 0.80 g/t silver. Waste movement from these phases amounts to 82.5 Mt, giving a strip ratio of 3.93:1 (waste: mill feed).

The mill feed cut-off is based on a value per tonne which is often referred to as the milling cut-off. This was determined to be approximately 0.28 g/t gold for Goliath and 0.40 g/t gold for Goldlund and Miller.

The feed to the plant has been diluted. The calculation varies by pit area due to different modelling techniques, but is based on a dilution skin thickness of 0.5 m at Goliath and 1.0 m at Goldlund and Miller. The result of the dilution calculation was a 15% increase in feed tonnage at Goliath and a 12% grade drop. For Goldlund the tonnage increase was 21% and a 14% lower feed grade. Miller was between the two with a tonnage increase of 17% and a grade drop of 13%.

The open pits are scheduled to provide mill feed over the nine-year operating mine life after one year of pre-production stripping. Initially this will be the full 1.8 Mt/a the process plant requires, but as the underground mine comes online, the component of the lower grade open pit material will be reduced. The pits are sequenced to minimise initial stripping and provide higher feed grades in the early years of the mine life. This is accomplished by stockpiling the lower grade material for processing at the end of mine life.

The pits are built on 10-m benches with safety berm placement each 20 m. Minimum mining widths of 35 to 40 m were maintained in the design. Ramps are at 10% gradient and vary in width from 22.0 m (single-lane width) to 38.7 m (double-lane width). They have been designed for 91 tonne haulage trucks.

The mine equipment fleet is anticipated to be leased to lower capital requirements. The fleet will be comprised of three 140 mm rotary drills, two 13 m^3 front-end loaders and two 6.7 m^3 hydraulic excavators. The truck fleet will peak at 11 trucks in Year 2 due to the higher initial strip ratios, but the truck requirements will drop from Year 5 onwards when some of the fleet will be sold. The usual assortment of dozers, graders, small backhoes and other support equipment is considered in the equipment costing. Additional support equipment in the form of snowplows and small excavators are part of the fleet to maintain operations year-round. An additional front-end loader (13 m^3) will be at the primary crusher full time and tramming/loading material from the stockpile as required.

The Goliath waste dump will contain material from the initial two phases of the open pit. The last two phases will backfill the first phase which reduces waste haulage costs, minimises the site footprint and helps in the storage of PAG material. The waste dumps at Goldlund and Goliath will be single structures adjacent to the main pits. A total rock storage volume of 6.4 Mm^3 has been designed at Goliath, 16.9 Mm^3 at Goldlund, and 5.8 Mm^3 at Miller. This is sufficient for the mine needs with the pit backfill considered.

Material from the Goliath mine has been assumed to be potentially acid-generating (PAG). To handle this, all drainage from the waste facility will be collected in ditches, pumped to the settling ponds and treated as required. Additional work on the exact nature of the material from a PAG perspective is to be defined in later studies.

The LOM operating cost is estimated at $3.27/t of material moved. This includes equipment leasing of $0.40/t of material moved.

Pre-production stripping costs of $25.2 million are capitalised. Initial mine equipment capital is $14.6 million with sustaining capital of $9.9 million.

Additional capital in the mining category includes initial road development, dewatering pumps and pipelines, and engineering office equipment. The initial cost of this is $4.8 million and $4.5 million for sustaining.

25.2.2 Underground

The PEA plan calls for the development of the underground mine starting in Year 3. Underground production will be mined concurrently with lower grade open pit material, thereby enhancing mill grade.

The planned underground mining area is an extension of the Goliath open pit. The depth of the open pit is planned to be approximately 100 m below surface. The currently identified measured, indicated and inferred resources for the underground area extend to around 640 m below surface and measure a total of approximately 3 km along strike. Approximately 11% of the underground material to be processed is derived from inferred resources. The dip of the deposit varies from around 70 degrees to around 80 degrees, averaging 75 degrees.

An elevated cut-off net value of $110/t was applied to plan stopes. This value approximates to a gold cut-off grade of 2.0 g/t and was calculated to provide a minimum net revenue of $20/t from all mineralisation mined. Some material identified using slightly lower cut-off criteria was added towards the end of mine life where appropriate. Application of the cut-off criteria resulted in the identification of five mining zones, below and adjacent to the planned open pit. Stope width typically varies from a minimum stope width of 1.8 m to around 11 m with some pinching and swelling, but averages around 6.2 m width.

In the deposit, ground conditions are considered to be fair to good and good in the footwall and hanging wall sequences. Cablebolt installation in stope hanging walls is planned to maintain stability and minimise waste dilution.

Longhole retreat stoping will be the primary underground mining method. Where production grade is estimated to be below 4.0 g/t Au, a permanent rib pillar is planned between adjacent stopes, resulting in approximately 15% in-situ losses, and uncemented rockfill will be used. Where production grade is estimated to be above 4.0 g/t Au, there are no planned pillars; cemented rockfill will be utilised to extract all this higher grade material.

Life-of-mine underground feed to the process plant is estimated to be 2.97 Mt with a gold grade of 3.67 g/t Au and 9.05 g/t Ag resulting in an estimated revenue of $200/t net of operating costs. Planned steady-state production rate is 1,400 t/d. Initial mill feed release is planned in Year 4, the second year after the commencement of underground mine development, increasing to full production by Year 6. Total production life is planned to extend slightly over seven years. Life-of-mine underground mine capital expenditure are estimated to be $202.1 million, which is classified as sustaining capital in project analyses and includes $22.2 million in contingency. Owner crews will undertake all mine activities apart from raising and deposit delineation. Life-of-mine underground operating costs are estimated to average $70.31/t of mill feed.

Large-scale mobile equipment types were assumed to maximise productivity. The mobile equipment fleet will be leased.

Potential issues or risks noted by AGP during the study may be summarised as follows:

- As future mine planning proceeds, additional cut-off grade analysis may further optimise underground economics.
- Dilution estimates should be confirmed by more rigorous analysis.
- Hydrogeological conditions require confirmation. The underground mine is located below the planned open pit. Backfilling of the open pit with broken waste is planned. There is potential for the build-up of water or saturated waste at times, particularly after the spring freshet. More detailed analysis of potential water inflows, dewatering requirement assumptions and system designs will be required in future. It will be important to ensure that crown pillar procedures protect the underground mine.

25.3 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.

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. No historical metallurgical testing has been completed on the Miller deposit. Metallurgical characteristics for the Miller deposit are assumed to be similar to the Goldlund deposit based on similar geology.

Based on the testwork results completed between 2011 and 2020 on composite samples representative of the Goliath and Goldlund deposits, doré can be produced at a high recovery of gold.

25.4 Process Plant

The plant will process material at a rate of 1.8 Mt/a with an average head grade of 1.47 g/t Au to produce doré.

The process plant flowsheet designs were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimising 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.

25.5 Infrastructure

Infrastructure to support the Goliath Gold Complex will consist of site civil work, buildings and facilities, water management systems, a tailings storage facility (TSF), and electrical power substation and distribution. Mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems as required. The processing plant and TSF will be located at the Goliath property, as well as most ancillary project infrastructure.

Knight Piésold completed a PEA-level design for the TSF at the Goliath Gold Complex. The TSF will provide secure storage for tailings and process water. The embankments include for adequate freeboard to provide ongoing tailings storage, operational water management, temporary environmental design storm storage and conveyance up to and including the inflow design flood. The TSF will be constructed in a phases as a single-cell facility northeast of the proposed process plant location. A geomembrane lining system will be installed along the TSF basin floor and on the upstream face of the perimeter embankments to minimise seepage. The embankments will be raised in stages to form a four sided paddock style impoundment using downstream construction methods throughout the mine life.

Tailings will be pumped from the process plant to the TSF as a conventional slurry via pipeline(s) and deposited into the TSF.

Meteoric and supernatant inflows to the TSF basin will be temporarily stored prior to reclaim to the process plant by a floating pump barge in the basin. Excess water beyond the required storage associated with the maximum water cover level will be transferred to the adjacent mine water pond. The TSF will be equipped with an overflow spillway in each embankment stage to accommodate flows above the environmental design storm and up to the inflow design flood.

Water management measures for the project will include a series of diversion berms, collection and diversion ditches, sediment basins, and water transfer pipelines to collect runoff originating within disturbed areas. The runoff will be conveyed to one of a number of catchment ponds where the majority of the total suspended solids can settle out prior to sending the water to the mine water pond (for potential use in the mining process) or to treatment prior to releasing it to the environment.

25.6 Environmental, Permitting & Social Considerations

The approach to environmental studies, permitting and approvals, and impact assessment for the Goliath Gold Complex will be to treat the Goliath, Goldlund and Miller deposits as three distinct projects. All three Goliath Gold Complex projects will also be required to complete Regulatory Closure Plans as per the requirements of Ontario Regulation 240/00: Mine Development and Closure Under Part VII of the *Mining Act* in Ontario, prior to commencement of construction activities. Throughout the environmental baseline, permitting and approvals processes, Treasury Metals will endeavour to maximise participation with Indigenous partners wherever possible and is committed to building and strengthening relationships, integrating traditional knowledge into decision-making frameworks, and actively communicating and sharing information in a transparent manner.

The overall schedule for the Goliath Project is ahead of the Goldlund and Miller project schedules, given that a Federal Environmental Assessment (EA) has already been completed for Goliath. Specifically, on August 19, 2019, Treasury Metals received federal government approval under the *Canadian Environmental Assessment Act, 2012* (CEAA, 2012) for the Goliath Project, with the Minister of Environment and Climate Change Canada concluding that the project is not likely to cause significant adverse environmental effects. Potential benefits of the project are expected to include employment and business opportunities, as well as tax revenues at all levels of government. The Goldlund Project and Miller Project may require completion of one or more Provincial environmental assessment processes pursuant to the *Ontario Environmental Assessment Act*, depending on the final project designs. Based on the current proposed design, neither the Goldlund Project nor the Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*.

The Goliath Project as presented in this PEA is similar to the previous PEA, but differs in that the processing facility for the Goliath Project is proposed to accept ore from other deposits (specifically from the Goldlund and Miller properties). Pending regulatory guidance otherwise, it is not anticipated that the optimisation of the Goliath Project design would affect the current Federal EA approval of the Goliath Project, or that it would trigger an Impact Assessment under the new *Impact Assessment Act* for a mining expansion. Therefore, while this engineering design change is not anticipated to have an effect on the current Federal EA approval on the Goliath Project, additional environmental data may need to be measured or modelled to support the change in the description of the assessed project. Additional environmental programs for the Goliath Project may also be required to update environmental baseline data relied on in the EA to support permitting efforts.

25.7 Capital Cost Estimates

The capital cost estimate is presented at a ±50% accuracy, using a base date of Q4 2020 and an exchange rate of US$0.75:C$1.00.

The total initial capital cost for the Goliath Gold Complex is $232.6 million and life-of-mine sustaining costs are $289.6 million. Closure costs are additional and estimated at $28.5 million.

25.8 Operating Cost Estimates

The operating cost estimate is presented at a ±50% accuracy using a base date of Q4 2020 and an exchange rate of US$0.75:C$1.00.

The overall life-of-mine operating cost is $975 million over 13.5 years, or $40.7/t of ore milled.

25.9 Economic Analysis

An engineering economic model was developed to estimate the project's annual pre-tax and post-tax cash flows and sensitivities based on a 5% discount rate.

The analysis uses the following key inputs:

- mine life of 13.5 years
- base case gold price of US$1,600/oz and silver price of US$20/oz
- United States to Canadian dollar exchange rate of 0.75 (USD:CAD)
- cost estimates in constant Q4 2020 Canadian dollars with no inflation or escalation factors considered
- results are based on 100% ownership with an average 0.04% NSR for Goliath, 2.12% NSR Goldlund, and 0% NSR for Miller
- capital costs funded with 100% equity (i.e., no financing costs assumed)

The pre-tax net present value discounted at 5% (NPV5%) is C$477 million, the IRR is 37.3%, and the payback period is 1.9 years. On an after-tax basis, the NPV5% is C$328 million, the IRR is 30.2%, and the payback period is 2.2 years.

A sensitivity analysis was conducted on the base case pre-tax and after-tax NPV and IRR of the project, using the following variables: gold price, discount rate, grade, capital costs, and operating costs. Analysis revealed that the project is most sensitive to changes in gold price, foreign exchange rate, head grade and recovery, and then to a lesser extent, operating costs and capital costs.

25.10 Risks & Opportunities

25.10.1 Risks

25.10.1.1 Geology

- The modelling approach at Goliath assumes that the contacts between the high-grade mineralisation and the surrounding low-grade material are not sharp and visually difficult to recognise without assays. This assumption was based on drill core logging and information provided by Teck-Corona as part of their bulk sampling program completed in 1997. If the contacts are sharper and more easily identifiable than expected during mining, the deposit could return a higher grade with a corresponding lower tonnage. This risk can be mitigated in various ways. Near surface, an area within the payback period of the open pit could be selected for testing the proposed grade control program. The program can be used to de-risk the resources and increase confidence in the grade intended for the proposed mill. At depth, targeted infill drilling can provide a greater level of confidence in the estimated grade and increase confidence in the modelling approach.
- At Goliath, the silver grade presents a small risk due to the lack of assays. This risk can be mitigated by re-assaying the drill core pulps for silver.

- At Goldlund, the current geological model considers broad mineralised zones that define the trend of the mineralisation. The development of a new geological model of lithology and alteration and a new model of the high-grade mineralisation may result in a change to the mineral resources. Infill drilling is required to confirm the continuity of the high-grade mineralisation.

25.10.1.2 Mining

- Wall slopes may flatten, resulting in more waste material. This can be mitigated with additional geotechnical drilling, particularly at Goldlund and Miller where more work is required.
- Waste storage foundation study at Goldlund and Miller may require lower and large footprints or additional preparation costs. Geotechnical site investigations should help mitigate this through better understanding.
- ABA testing may indicate some of the waste material in Goldlund and Miller is potentially acid-generating and that separate storage facilities may be required to control drainage. Additional testwork will help to develop a better understanding of this issue and determine its impact on project design.

25.10.1.3 Recovery Methods & Metallurgical Testing

- No metallurgical testing has been completed on the Miller deposit. Based on geology it is assumed to be similar to the Goldlund deposit.

25.10.1.4 Tailings Storage Facility

- Non-PAG waste rock produced from the Goliath pit (up to 7% of waste rock) cannot be segregated during mining as assumed in the study and will not be available as required during construction of the TSF.
- The source of an adequate amount of suitable bulk embankment fill cannot be identified and secured from locally available borrow sources.
- There is the potential for challenging construction conditions associated with dewatering during preparation of the foundations for embankment construction and lining of the basin.
- The ability to achieve flat uniform filling of tailings via sub-aqueous deposition within the basin while maintaining the minimum required water cover over the tailings as assumed in the study.

25.10.2 Opportunities

25.10.2.1 Geology

- Drilling in the eastern portion of the Goliath deposit and around the fold nose could increase the resources.
- The additional drilling recommended for Goldlund in Zones 1, 2, 3, 4, 6, 8 and 9 could convert a portion of the inferred mineralisation to indicated mineralisation, as well as to expand the Zone 1 mineralisation to the northeast.

- Assaying of available Goldlund drill core sample rejects for silver, along with additional drilling, may generate sufficient data to allow the estimation of silver as a byproduct in future mineral resource estimates.

25.10.2.2 Mining

- With testing, the PAG material may represent a smaller volume of material, which may help in storage considerations at Goliath in addition to Goldlund and Miller
- The use of sorting technology may help reduce mill feed trucking tonnage, which in turn may elevate the feed grade.

25.10.2.3 Recovery Methods & Metallurgical Testing

- Optimising processing conditions related to fineness of grind and leach retention time may result in lower capital costs from employing a coarser grind and reduced retention time.
- Additional metallurgical testing will provide an opportunity to optimise reagent addition rates and grinding media wear rates.
- Further investigate the incidence of tellurides within Goldlund and Miller mill feed to optimise mill recovery factors.

25.10.2.4 Tailings Storage Facility

- Site conditions at Goldlund may be more favourable than at Goliath for siting the tailings storage facility, including providing closer access to large quantities of NAG waste rock for construction.
- Additional geotechnical drilling would better define the foundation conditions at the TSF and potentially reduce earthworks quantities for construction of the embankments and buttressing.

25.11 Interpretations & Conclusions

The total measured and indicated resources for the Goliath, Goldlund and Miller projects are estimated at 55.4 Mt at a grade of 1.10 g/t Au for an estimated 2.0 Moz of contained gold. Additional inferred resources are estimated to be 21.0 Mt at a grade of 0.78 g/t Au for a total of 0.5 Moz.

Based on the assumptions and parameters presented in this report, the PEA shows positive economics (i.e., C$328 million post-tax NPV (5%) and 30.2% post-tax IRR). The PEA supports that additional detailed studies are warranted.

26 RECOMMENDATIONS

26.1 Overall

The financial analysis of this PEA demonstrates that the Goliath Gold Complex has positive economics. It is recommended to continue developing the project through additional studies, including a pre-feasibility study. Table 26.1 summarises the proposed budget to advance the project through the pre-feasibility study stage.

Table 26.1: Proposed Budget Summary

Description	Cost C$
Geology – Goliath Work Program	5,925,000
Geology – Goldlund Work Program	8,760,000
Geology – Miller Work Program	1,830,000
Geotechnical	998,000
Mining	50,000
Metallurgy	500,000
Infrastructure	555,000
Environmental	2,100,000
PFS Study Budget	1,695,000
Total Recommended Study Budget	**22,413,000**

26.2 Geology

26.2.1 Goliath Project

After reviewing the Treasury Metals data, AGP makes the recommendations outlined below.

Goliath QA/QC

- AGP recommends that the QA/QC for silver be charted similarly to gold.
- Treasury Metals quarter core sample duplicate shows evidence of a rather strong nugget effect and AGP questions if this protocol should continue. AGP advised Treasury Metals to seek the opinion of a specialist in the QC/QA field.

Resource Modelling

- The missing silver assays represent limited risk to the resources; however, AGP recommends all recoverable drill rejects or pulps for the samples located in the mineralised horizon be assayed for silver. An estimated 6,000 pulps @ $10.00 per pulps for a total of $60,000.
- AGP also recommends that in future drilling programs, Treasury Metals should ensure that no gold assay within the mineralised horizons is missing a corresponding silver assay.
- Advance geostatistical studies (change of support and conditional simulation) should be conducted as part of future pre-feasibility or feasibility studies. These studies allow the quantification of risks to the resource. The cost for these studies is estimated at $10,000.

Drilling Recommendations

AGP recommends continuing exploration and delineation drilling at the Goliath deposit. This additional drilling should be designed to expand and improve the quality of mineral resources presented in this report and to further the understanding of the geology, specifically in the area east of the deposit where mining infrastructure may potentially be built. Drilling should also focus on infill drilling of the underground resources from surface where the potential open pit may restrict access in the future. Finally, drilling should focus on the sections of the underground mining areas that have seen reduced continuity in the current resource model when compared to previous models. If gold assays are found in these areas, there is potential to connect the high-grade wireframe and subsequently create additional areas for the proposed mining zone.

- AREA "A" is designed to expand on existing resources and convert inferred blocks to indicated east of shoot 1
- AREA "B" is strictly designed to convert inferred blocks to indicated in the west of shoot 2 and at depth.
- AREA "C" is designed for resource expansion. This area is located at depth adjacent to the currently defined resource blocks between shoots 2 and 3.
- AREA "D" is to convert the resource in the upper portion of the PEA pit from inferred to indicated. The area spans from section 526500E to 527500E.
- AREA "E" is designed to explore the ground currently located under the proposed infrastructure. The area is located between sections 529750E and 529875E.
- AREA "F" is designed to test a number of regional targets and follow up on several historical results that could contribute to future growth of additional "satellite pits" along strike towards the eastern boundary of the Goliath property.

AGP recommends a total of 82 drill holes totalling 36,575 m for a total estimated cost of $5,925,000 (see Table 26.2).

Table 26.2: Goliath Proposed Drill Program

Area	Purpose	Number of Holes	Total Length (m)	Unit Cost (C$/m)	Approximate Cost (C$)
A	Expand resources and convert inferred to indicated	20	13,100	150	1,965,000
B	Convert inferred to indicated	17	12,300	150	1,845,000
C	Low-priority target - resource expansion	5	2,400	150	360,000
D	Infill top western portion of the PEA pit and convert material inferred to indicated	32	6,100	150	915,000
E	Resource expansion between sections 529750E and 529875E	8	2,600	150	390,000
F	Regional exploration and historic drilling follow-up	20	3,000	150	450,000
	Total Cost of Proposed Drill Program	**82**	**39,500**		**5,925,000**

26.2.2 Goldlund Project

The Qualified Person for this section of the report makes the following recommendations for the Goldlund Project:

- Close-spaced drilling of 6,400 m in 32 holes should be carried out in Zone 1. The drilling should target areas inside the mineral resources shell using angled core holes to confirm the grade continuity and upgrade a portion of the mineral resources for that part of Zone 1 from indicated to measured. The target area should represent the area that is likely to be mined at the start of the project.
- Infill drilling of 29,000 m should be carried out in selected areas of Zones 1, 2, 3, 4, 6, and 8 and 9 to achieve a drill hole spacing of approximately 25 m x 25 m to upgrade the inferred mineralisation to indicated and to explore for additional inferred resources. Priority should be given to areas that have inferred mineralisation inside the mineral resources shell and within or directly adjacent to proposed mining pit shells.
- Additional drilling of 7,200 m should be carried out to further explore selected areas of Zone 1 and Zone 4 and increase the confidence in the location of the mineralised zones.
- A 3D geological model of the lithology and alteration should be developed using implicit modelling software such as Leapfrog GEO® to aid in the interpretation of the granodiorite sills that host the stockwork mineralisation and the faults or other structures that might off-set the mineralised zones. These models would then be used to support a revised interpretation of the mineralised zones for the estimation of mineral resources. This modelling effort will require additional database and geological studies.
- Consideration should be given to the development of an alternative model of the gold mineralisation using a high-grade wireframe. This wireframe should be generated using a suitable gold grade threshold, such as 1.0 or 2.5 g/t Au, and implicit modelling software, such as Leapfrog GEO®. This grade-shell would then be used as an additional control to restrict the higher grades and prevent any potential smearing of the high-grade assays during block grade interpolation. This would improve the reliability of the mineral resource estimate.
- The mineral resources estimate should be updated considering the additional drilling and geological modelling of the lithology, alteration, and high-grade mineralised zone wireframes.
- Silver has been assayed in some drill core samples within the Goldlund Project. Currently, there are insufficient silver assay results to model the silver mineralisation. It is recommended that the available drill core sample rejects be assayed for silver, and along with additional drilling, this may generate sufficient data to allow the estimation of silver as a byproduct in future mineral resource estimates.

The estimated budget for the proposed drilling and modelling programs is approximately C$8.7 million. A breakdown of this cost is shown in Table 26.3.

Table 26.3: Estimated Budget for Goldlund Proposed Work

Proposed Work	Number of Holes	Total Length (m)	Unit Cost (C$)	Approximate Cost (C$)
Zone 1 – Close-Spaced Drilling	32	6,400	$200/m	1,280,000
Zones 2, 3, 4, 6, 8, 9 – Infill Drilling	134	29,000	$200/m	5,800,000
Zones 1, 4 – Additional Drilling	36	7,200	$200/m	1,440,000
Database and Geological Studies			$200/h	96,000
Lithological and Alteration – 3D Geological Model	-	-	$160/h	32,000
High-grade Mineralisation – 3D Geological Model	-	-	$160/h	16,000
Update of the Mineral Resources Estimate	-	-	$200/h	96,000
Total Cost of Proposed Work Program				**8,760,000**

26.2.3 Miller Project

AGP recommends the following exploration programs for the Miller Project. Pending positive results, further studies may be proposed.

- A review of selected completed drill holes by optical televiewer should be carried out to accurately determine vein orientations and vein sets for a better understanding of geological and structural controls of the gold mineralisation for the deposit. Optimally, this should be carried out on a variety of drill holes, that is, on angled drill holes (drilled from the northeast and southwest) and vertical drill holes.
- Infill drilling should be carried out by angled drill holes from the northwest and the southeast to reduce the current drill spacing to less than 50 m x 50 m. Drill holes should target the deposit near surface and at depth. Approximately 6,000 m of drilling is recommended. The drilling should be completed using oriented drill core if a televiewer is not employed to collect information of the vein orientations.
- Delineation drilling along strike of the known gold mineralisation to determine the extent of the deposit. Approximately 2,500 m of drilling is recommended.
- Where and if possible, stripping (trenching) and surface channel sampling across the deposit to gather geological and structural data at the surface of the deposit. An initial program of three lines of channel samples are recommended.
- Update of mineral resources based on the results of additional drilling and the geological information collected.

The estimated budget for the proposed drilling and modelling programs is approximately C$1.8 million. A breakdown of this cost is shown in Table 26.4.

Table 26.4: Estimate Budget for Miller Proposed Work

Proposed Work	No of Holes	Total Length (m)	Unit Cost (C$)	Approximate Cost (C$)
Miller – Infill Drilling	32	6,000	$200/m	1,200,000
Miller – Delineation Drilling	10	2,500	$200/m	500,000
Miller – Stripping and Channel Sampling	-	-	-	80,000
Miller – Mineral Resource Update	-	-	$200/h	50,000
Total Cost of Proposed Work Program				**1,830,000**

26.3 Geotechnical

Further geotechnical and hydrogeological work are required at Goliath and new studies need to be initiated at Goldlund and Miller. The recommended work will:

- update the slope design parameters considering the current PEA design
- develop area hydrogeological models for surface and underground mining development (Goliath only) to interface with the overall project site-wide water balances
- review the underground design with focus on underground infrastructure, and required stope support (bolting)
- analyse waste and stockpile foundations with revised slope design parameters

The budget for the proposed PFS geotechnical program is shown in Table 26.5.

Table 26.5: Geotechnical Program Estimate

Proposed Work	Unit Cost (C$/m)	Approximate Cost (CAD)
Goldlund – Geotechnical Holes – 6 x 300 m deep	300	540,000
Miller – Geotechnical Holes – 4 x 150 m deep	300	180,000
Laboratory Work – 3 area samples		100,000
Hydrogeological Investigation – each area		100,000
Hydrogeological Models – each area		50,000
Foundation Drillhole Program – 4 holes/area x 10 m deep	150	18,000
Stockpile and Dump Design and Foundation Analysis		10,000
Total Cost of Proposed Work Program		**998,000**

26.4 Mining

The following work is recommended to advance the project to a pre-feasibility study level:

1. detailed quotations on mine equipment and refined equipment selection
2. detailed mine planning on Goliath pit backfill sequence to determine if additional material could be backfilled
3. further examination of mill feed transportation options with the objective of reducing transportation cost
4. review and design of pit and underground dewatering requirements and interface with surface water management system
5. detailed design and costing of permanent water exclusion bulkheads beneath the temporary central open pit access
6. incorporation of updated geotechnical guidance on stope cablebolt designs, as the rock is currently classified as fair to good which requires this level of support
7. solicitation of contractor quotes for both open pit and underground mining to examine potential project NPV enhancements

8. update of pit slopes in all three areas based on revised geotechnical parameters resulting from additional geotechnical testwork
9. detailed design of underground infrastructure, both on surface (portals, ventilation, power interface) and underground (dewatering system, electrical, etc.)
10. complete a labour survey for salaries, benefits, and skilled worker locally available (this information would be used in pre-feasibility study costing; it may also lead to Treasury Metals assisting local colleges and workers to develop specific skill sets in anticipation of a production decision)

All of the above recommendations would be included in the normal pre-feasibility study cost estimate, with the exception of point 10. This would normally involve an outside consultant and would be expected to cost $50,000.

26.5 Metallurgy

Recommendations for additional metallurgical testwork for all three projects are provided below. The budget for this work is estimated at $500,000.

26.5.1 Goliath Project

The following metallurgical testwork is recommended in the next project phase:

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- mineralogical studies including gold deportment analysis
- additional ore competency tests for more accurate SAG mill sizing; JK Tech SMC tests (Axb) are recommended to be conducted over a range of lithologies or zones
- ore hardness tests including Bond rod, ball and abrasion index testing to determine the variability of the lithologies or zone
- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time and cyanide addition rate
- additional cyanide destruction testing to optimise reagent addition and retention time

26.5.2 Goldlund Project

The following metallurgical testwork is recommended in the next project phase:

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- addition ore competency tests for more accurate SAG mill sizing; JK Tech SMC tests (Axb) are recommended to be conducted over a range of lithologies or zones
- ore hardness tests including Bond rod, ball and abrasion index testing to determine the variability of the of lithologies or zones
- mineralogical studies including gold deportment analysis

- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time, reagent conditions (pH and cyanide concentration) for gold tellurides
- cyanide destruction testing to establish required reagent addition rates and retention time for required discharge cyanide concentrations

26.5.3 Miller Project

No previous testing has been conducted on Miller samples. The following metallurgical testwork is recommended:

- identify samples required to provide geo-metallurgical representation of the deposit sufficient for a pre-feasibility study requirement
- conduct testing to identify comminution parameters including SMC tests (Axb), Bond rod, ball and abrasion index testing
- mineralogical studies including gold deportment analysis
- extended gravity recoverable gold (E-GRG) testing
- cyanidation testing on major lithologies examining grind size, retention time, reagent conditions (pH and cyanide concentration) for gold tellurides (if present)
- cyanide destruction testing to establish required reagent addition rates and retention time for required discharge cyanide concentrations

26.6 Sorting

Sighter-type ore sorting amenability testing is recommended. The program will establish if samples from the three deposits are amenable to particle or bulk sorting. Ore sorting could benefit the project by either upgrading mill feed with reduced quantity transported for processing or upgrading of low-grade material near the planned cut-off grades.

26.7 Infrastructure

The following activities are recommended to support infrastructure design for the pre-feasibility study phase:

26.7.1 Site Investigations

- Additional site investigations should be completed to identify suitable borrow locations, and further characterise foundations of the TSF embankments and basin.
- Cone penetration testing should be carried out in key areas to confirm strengths of the softer fine grained soils within TSF Embankment footprint and other key infrastructure, (i.e., the grey silt).
- The availability of local borrow sources for TSF embankment construction should continue to be evaluated to verify the capital cost associated with its construction based on the material available.
- The recommended budget for these items is $375,000.

26.7.2 Tailings Storage Facility

- Additional stability analyses should be carried out to refine and optimise buttress sizing requirements and embankment section (note: the analysis should take into account the potential for soil liquefaction, cyclic clay softening, and undrained strength conditions based on the updated site investigations).
- Additional seepage analyses should be performed to refine and optimise basin lining requirements and closure cover thickness.
- Potential basin lining alternatives, including geosynthetic materials (HDPE, LLDPE) and paper pulp sludge, should be evaluated.
- The recommended budget for these items is $140,000.

26.7.3 Water Management Measures

- The catchment areas contributing runoff to the process plant, open pits and waste dumps, and the amount of groundwater inflow to the open pits and underground mine with time need to be confirmed based on the ultimate mine plan.
- Site-specific meteorological and hydrology data should be collected. This data will be used to refine seasonal runoff values and design storms for future work.
- The predictive water quality model should be updated to review the requirements for water treatment and/or discharge.
- Bench-scale settling testing should be performed to characterise the required retention time for suspended solids in the runoff water.
- The recommended budget for these items is $40,000.

26.8 Facilities Location

The PEA was advanced with the concept of locating the process and tailings facility at the Goliath project site. This is due to the advanced nature of both the permitting and development path of the Goliath Project and previous technical studies. By adding the Goldlund and Miller properties to the overall project scope, opportunities exist that may benefit the project from a cost and environmental perspective.

Mill feed material needs to move between the various pit areas, which implies that a plant located at Goldlund would not adversely impact the operating costs of the project. The advantages of locating the plant and tailings at Goldlund should be examined and included in a detailed trade-off study that considers potential permitting delays that may accompany such changes.

It is recommended that a series of trade-off studies examining alternate locations for the plant and tailings facility be considered and included in the pre-feasibility study budget.

26.9 Environmental

The approach to environmental studies for the Goliath Gold Complex will be to treat the Goliath, Goldlund and Miller deposits as three distinct projects; therefore, each project will have a distinct set of environmental recommendations as indicated below.

Treasury Metals has an advanced understanding of the environmental baseline at the Goliath Project site having previously completed an extensive baseline investigation to support the Federal environmental assessment process for the project. Treasury Metals received Federal government approval for the Goliath Project in August 2019 under the *Canadian Environmental Assessment Act*, with the Minister of Environment and Climate Change Canada concluding that the project is not likely to cause significant adverse environmental effects. As part of the conditions on the approval of the project, Treasury Metals is obligated to notify the Federal and Provincial authorities, as well as it Indigenous partners, of any project changes, including the milling of ore from the Goldlund Project and Miller project at the Goliath property. While the engineering design change to mill ore from other sites at Goliath is not anticipated to have an effect on the current Federal EA approval on the Goliath Project, additional environmental data may need to be measured or modelled to support the change in the description of the assessed project. Additional environmental programs for the Goliath Project may also be required to update environmental baseline data relied on in the EA to support permitting efforts.

Baseline data collection for the Goldlund Project is underway and is expected to be completed within 12 months' time. Treasury Metals has not collected any baseline data from the Miller project to date; however, it is anticipated this will happen in the immediate future. Based on the current proposed design, neither the Goldlund Project nor Miller Project is expected to require completion of a Federal Impact Assessment under the new *Impact Assessment Act*. However, baseline data for these projects will be required to support Provincial permitting and approvals processes, including potential Provincial EAs.

The cost for the above work for all three projects is estimated at $2.1 million. This is considered sufficient for a pre-feasibility level of study.

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Tetra Tech, 2013, Technical Report and Resource Estimate Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario, Tetra Tech Inc. (QP: McCracken, T.)

Tetra Tech., 2014, Technical Report and Resource Estimate Update on the Goldlund Deposit, Goldlund Project. Sioux Lookout, Ontario, Tetra Tech Inc. (QP: Zellerer, S.)

Wardrop, 2010, Technical Report and Resource Estimate on the KRP Deposit, KRP Project, Sioux Lookout, Ontario, Wardrop Engineering Inc. (QP: McCracken, T.)

Wardrop 2011, Technical Report on the Goldlund Property, Sioux Lookout, Ontario, Wardrop Engineering Inc. (QP: McCracken, T.)

WSP, 2015, Technical Report and Resource Estimation Update on the Goldlund Goldlund Project, Patricia, and Kenora Mining Division Ontario, WSP Canada Inc. (QP: McCracken, T.)

WSP, T. 2017, Technical Report and Resource Estimation Update on the Goldlund Goldlund Project, Patricia, and Kenora Mining Division Ontario, WSP Canada Inc. (QP: McCracken, T.)

WSP, 2019, Technical Report and Resource Estimate Update, Goldlund Gold Project, Sioux Lookout, Ontario, WSP Canada Inc. (QP: McCracken, T.)

WSP, 2020, Technical Report Re-Issue, Goldlund Gold Project, Sioux Lookout, Ontario, WSP Canada Inc. (QP: McCracken, T.)

27.3 Websites

International Lithium Corp.:
http://internationallithium.com/s/mavis_fair_service.asp

Canadian Impact Assessment Registry website, Impact Assessment Agency of Canada:
https://ceaa-acee.gc.ca/050/evaluations/document/132190?culture=en-CA

Ontario Ministry of Energy, Northern Development and Mines online GIS website, Mining Lands Administration System (MLAS):
https://www.gisapplication.lrc.gov.on.ca/Html5Viewer261/index.html?viewer=mlas.mlas&locale=en-US

Treasury Metals Inc., Press Releases:
https://www.treasurymetals.com/news/

Ontario Ministry of Energy, Northern Development and Mines online GIS website, Mining Lands Administration System (MLAS):
https://www.gisapplication.lrc.gov.on.ca/Html5Viewer261/index.html?viewer=mlas.mlas&locale=en-US

First Mining Gold Corp., Press Releases:
https://www.firstmininggold.com/news/

27.4 News Releases

News Release 20 September 2018 (Miller Drill Results)

News Release 27 March 2019 (Miller Drill Results)

News Release 19 November 2019 (Miller Drill Results)

News Release 11 February 2020 (Miller Drill Results)

News Release 2 March 2020

News Release 6 May 2020

News Release 4 August 2020

News Release 7 August 2020

Treasury Metals Inc., Press Releases:
https://www.treasurymetals.com/news/ (most recently viewed on 18 November 2020)

News Release 3 June 2020

News Release 4 August 2020

News Release 5 August 2020

News Release 7 August 2020

News Release 27 October 2020

ATTACHMENT 1 – QUALIFIED PERSONS CERTIFICATES

CERTIFICATE OF QUALIFIED PERSON

I, Tommaso Roberto Raponi, P. Eng., as a co-author of the Technical Report, do hereby certify that:

I am a Principal Metallurgist at Ausenco Engineering Canada Inc., 11 King St West, Suite 1550, Toronto, ON, M5H 4C7.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a Bachelor of Applied Science in Geological Engineering from University of Toronto in 1984.
2. I am registered as a Professional Engineer in Ontario (License No. 90225970), Nunavut/NWT (License No. L4508) and British Columbia (License No. 23536). I have worked for more than 36 years in the mining industry in various positions continuously since my graduation from university.
3. I have worked for more than 36 years in the mining industry in various positions continuously since my graduation from university. I have worked as an independent consultant since 2016.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (N.I. 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I have not visited the property.
6. I am the co-author of this report and responsible for Sections 1.1, 1.2, 1.11, 1.14, 1.15 except 1.15.1, 1.17 except 1.17.1, 1.18, 1.19, 1.20.1.3, 1.20.2.3, 1.21, 1.22 except 1.22.1 to 1.22.3, 1.23, 1.24.4, 2 except 2.3.1 and 2.3.2, 13, 17, 18 except 18.1.2, 18.3.1, 18.7 and 18.7.4, 21 except 21.1.2.1 and 21.1.2.2.2 and 21.1.4 and 21.2.1, 22, 24, 25.3, 25.4, 25.7, 25.8, 25.9, 25.10.1.3, 25.11, 26.1, 26.5, 26.6, 26.8.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Tommaso Roberto Raponi, P. Eng.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

I, Gordon Zurowski, P.Eng., do hereby certify that:

1. I am a Principal Mine Engineer with AGP Mining Consultants Inc., with a business address at #246-132K Commerce Park Dr., Barrie, Ontario L4N 0Z7, Canada.
2. I graduated from the University of Saskatchewan with a B.Sc. in Geological Engineering in 1988.
3. I am a member in good standing of the Professional Engineers of Ontario (License No. 100077750).
4. I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 30 years in mineral resource and reserve estimations and feasibility studies in Canada, the United States, Central and South America, Europe, Asia, Africa, and Australia.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects ("N.I. 43-101") and certify that by virtue of my education, affiliation with a professional association (as defined in N.I. 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of N.I. 43-101.
6. I am independent of the issuer, Treasury Metals Inc., as defined in Section 1.5 of N.I. 43-101.
7. I am responsible for Sections 1.13, 1.17.1, 1.20.1.2, 1.20.2.2, 1.22.2, 1.22.3, 2.3.2, 15, 16, 18.3.1, 18.7.4, 21.1.2.1, 21.2.1, 25.2, 25.10.1.2, 26.3, 26.4 of the Technical Report and accept professional responsibility for those sections.
8. I have no previous involvement with Goliath Gold Complex.
9. My most recent site visit to the Goliath Gold Complex was from October 4 to 5, 2020.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
11. I have read N.I. 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with N.I. 43-101 and Form 43-101F1.

Dated:	**March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Gordon Zurowski, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Joseph Rosaire Pierre Desautels, P.Geo, as a co-author of the Technical Report, do hereby certify that:

I am the Principal Resource Geologist for AGP Mining Consultants Inc. #246-132 Commerce Park Dr., Unit K Barrie, ON L4N 0Z7.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. in Geology from the University of Ottawa in 1978.
2. I am a member in good standing of the Association of Professional Geoscientists of Ontario (Member No. 1362).
3. I have worked as a geologist for 43 years. My experience in the mining sector includes geological databases, mine geology, grade control, and resource modelling. I have been involved in numerous projects around the world in both base metals and precious metals deposits. My most recent relevant gold resource estimates includes Euro Sun Mining, Rovina Valley Project (2019), Roxgold Yaramoko Deposit (2012, 2013). I also have relevant experience in underground gold mines (Camshib Resources' Henderson Mine and Teck-Corona's David Bell Mine).
4. I have read the definition of Qualified Person set out in N.I. 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I visited the property on September 10 and 11, 2020.
6. I am the co-author of this report and responsible for Sections 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.12, 1.20.1.1, 1.20.2.1, 1.22.1, 2.3.1, 3, 6.1, 7.1, 7.2, 8.1, 9.1, 9.2, 10.1, 10.2, 11.1, 12.1, 14.1, 14.4, 25.1, 25.1.1, 25.10.1.1, 26.2.1.
7. I am independent of the issuer applying all of the tests in Section 1.5 of N.I. 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:	**March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Joseph Rosaire Pierre Desautels, P. Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Paul Daigle, P.Geo., as a co-author of the Technical Report, do hereby certify that:

I am an Associate Senior Resource Geologist with AGP Mining Consultants Inc., located at #246-132 Commerce Park Dr., Unit K Barrie, Ontario.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. in Geology from Concordia University in 1989.
2. I am a registered Professional Geologist, in good standing, with the Professional Geoscientists of Ontario (Member No. 1592).
3. I have worked as a geologist continuously for 31 years. My experience in the mining and exploration sectors include mineral exploration and diamond drill programs, managing data, and estimating resources. I have been involved in numerous precious metal projects in similar deposits within Archean/Proterozoic greenstone belts. My most recent experience includes the Troilus Gold-Copper Project, Quebec; Boto Gold Project, Senegal; and the Detour Gold Project, Ontario.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (N.I. 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I visited the Goldlund property on October 13 to 15, 2021, for two days.
6. I am the co-author of this report and responsible for Sections 1.3, 1.22.1.2, 4, 6.3, 7.4, 8.3, 9.4, 10.4, 11.3, 12.3, 14.3, 25.1.3, 26.2.3, and 27.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read N.I. 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated:	**March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Paul Daigle, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

I, Christopher Keech, P.Geo., as a co-author of the Technical Report, do hereby certify that:

I am employed as a Principal Geologist with CGK Consulting Services Inc. located at 15410 95A Ave., Surrey, BC, V3R 7S7.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a B.Sc. in Geology from McMaster University in 1980.
2. I am a member in good standing of Engineers and Geoscientists of British Columbia (Membership No. 27185).
3. I have worked as a professional geologist in the mining sector continuously for 40 years, including 11 years as an exploration geologist, 19 years as a resource geologist with several mining companies and 10 years as a geological consultant specialising in the estimation of mineral resources. My relevant experience for the purpose of this Technical Report includes the estimation of gold mineral resources for greenstone-hosted, and epithermal gold deposits at development properties and operating mines in Canada, USA, Brazil, Honduras, Venezuela, and Tanzania. In addition, I hold a Citation in Geostatistics from the University of Alberta (2004).
4. I have read the definition of the Qualified Person set out in National Instrument 43-101 (N.I. 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I visited the Goldlund Project property between October 6 and 8, 2020.
6. I am the co-author of this report and responsible for Section 1.22.1.1, 6.2, 7.3, 8.2, 9.3, 10.3, 11.2, 12.2, 14.2, 25.1.2, and 26.2.2.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read N.I. 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Christopher Keech, P.Geo.

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Reagan McIsaac, Ph.D., P.Eng., as a co-author of the Technical Report, do hereby certify that:

I am professional engineer with Knight Piésold Ltd., located at 1650 Main Street, West, North Bay, Ontario, Canada.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated from the University of Waterloo, Ontario, Canada, in 1997 with a Bachelor of Applied Science in Geological Engineering.
2. I am a registered Professional Engineer of Ontario (Member No. 100074049).
3. I have worked as a geotechnical engineer continuously for 14 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (N.I. 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I have not visited the property.
6. I am the co-author of this report and responsible for Sections 1.15.1, 1.20.1.4, 1.20.2.4, 1.24 except 1.24.4, 18.1.2, 18.7, 21.1.2.2.2, 25.5, 25.10.1.4, 26.7.
7. I am independent of the issuer.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read N.I. 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Reagan McIsaac, Ph.D., P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Mackenzie Denyes, Ph.D., QP_{RA}, P.Geo., as a co-author of the Technical Report, do hereby certify that:

I am the Manager of Regulatory Affairs at Treasury Metals Inc., located at The Exchange Tower, 130 King St W Suite 3680, Toronto, ON M5X 1B1.

This certificate applies to the technical report entitled, "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" with an effective date of January 28, 2021 and a report date of March 10, 2021 (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a Doctorate degree in Chemistry and Chemical Engineering from the Royal Military College of Canada in 2015. I also graduated with a Bachelor (Honours) degree in Environmental Science from the University of Ottawa in 2010, and a Bachelor degree in Biochemistry from the University of Ottawa in 2010.
2. I am a registered Professional Geoscientist of Ontario (Member No. 3006).
3. I have worked as an environmental geoscientist continuously for 10 years.
4. I have read the definition of Qualified Person set out in National Instrument 43-101 (N.I. 43-101) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a Qualified Person for the purposes of N.I. 43-101.
5. I visited the property on September 2020
6. I am the co-author of this report and responsible for Section 1.4, 1.16, 1.25, 5, 19, 20 except 20.7, 23, 25.6, 26.9.
7. I am not independent of the issuer.
8. I have prior involvement (specifically the environmental assessment) with the property that is the subject of the Technical Report.
9. I have read N.I. 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
10. As of the date of the certificate, to the best of my knowledge, information and belief, the Technical Report contains all material scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: **March 10, 2021**

Signature: *– Original signed and sealed –*

Name: Mackenzie Denyes, Ph.D., QP_{RA}, P.Geo.

Wood Environment & Infrastructure Americas
a Division of Wood Canada Limited
2020 Winston Park Drive, Suite 600
Oakville, Ontario, Canada
L6H 6X7
T: 905.568.2929

CERTIFICATE OF QUALIFIED PERSON

I, Sheila Ellen Daniel, P.Geo., am employed as a Principal Geoscientist, Discipline Lead Mining Environmental Management Approvals at Wood Environment & Infrastructure Americas, a Division of Wood Canada Limited, located at 2020 Winston Park Drive, Suite 600, Oakville, Ontario, Canada, L6H 6X7

This certificate applies to the technical report titled "N.I. 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex" that has an effective date of January 28, 2021 and a report date of March 10, 2021, (the "Technical Report").

My qualifications and relevant experiences are that:

1. I graduated with a M.Sc. from McMaster University in 1990 and with a B.Sc. (Honours) from the University of Western Ontario in 1988.
2. I am a registered Professional Geoscientist of Ontario (#0151).
3. I have worked as a mining environmental consultant continuously for 30 years experience.
4. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).
5. I have not visited the property.
6. I am the co-author of this report and responsible for Sections 20.7 and 21.1.4.
7. I am independent of Treasury Metals Inc. as independence is described by Section 1.5 of NI 43–101.
8. I have I have previous involvement with the environmental assessment and approvals process for the that is the subject of the Technical Report.
9. I have read National Instrument 43-101 and those parts of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.
10. As of the date of this Certificate, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: March 10, 2021

"Signed and sealed"

Sheila Daniel, M.Sc., P.Geo.
Principal Geoscientist
Discipline Lead, Mining Environmental Management and Approvals