Arctic Project

S-K 1300 Technical Report Summary

Ambler Mining District, Alaska

Report Date: November 30, 2022

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This technical report summary (the TRS), entitled "Technical Report Summary on the Arctic Project, Ambler Mining District, Alaska" is current as at November 30, 2022 and has been prepared by:

Qualified Person or Consulting Firm	Responsible for the following sections	Signature	Date
Ausenco Engineering Canada Inc	1.1, 1.3, 1.4, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.3, 1.14.4, 1.15, 1.17, 1.18, 1.19, 1.20.8, 1.20.9, 1.20.10, 2, 3.1, 3.2, 3.10, 4, 5.3.8, 10, 14, 15.1, 15.2, 15.3, 15.4, 15.5, 15.6, 15.7, 15.10.8, 15.12, 15.13, 15.14, 15.15, 16, 18.1.1, 18.1.2, 18.1.3, 18.1.4, 18.1.5, 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.1.12, 18.1.13, 18.1.14, 18.2.1, 18.2.3, 18.2.4, 18.2.5, 18.2.6, 19, 21, 22.1, 22.5, 22.9, 22.10, 22.12, 22.13, 22.14, 22.15, 22.16, 22.17.8, 22.17.9, 23.1, 23.9, 23.10, 23.11, 24, 25.1, 25.3, 25.5	"signed"	Feb 13, 2023
Brown & Caldwell	1.14.9, 1.20.7, 2.3, 15.9, 17.3.4, 17.3.6.2, 18.1.9.1, 18.2.6, 23.8, 25.2	"signed"	Feb 13, 2023
SRK Consulting (Canada) Inc.	1.14.6, 1.14.7, 1.14.8, 1.16.3, 1.20.3, 1.20.4, 1.20.5, 1.20.6, 2.3, 2.4, 3.7, 3.8, 3.9, 3.10, 5.3.7, 7.2.4, 8.1.2, 8.2.5, 13.9, 13.10.1, 13.11, 15.8, 15.10.1, 15.10.2, 15.10.3, 15.10.4, 15.10.5, 15.10.6, 15.10.7, 15.11, 17.3.1, 17.3.2, 17.3.3, 17.3.5, 17.6, 22.17.3, 22.17.4, 22.17.5, 22.17.6, 22.17.7, 23.4, 23.5, 23.6, 23.7, 25.2, 25.4	"signed"	Feb 13, 2023
Wood Canada Limited	1.2.1, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11, 1.12, 1.14.5, 1.20.1, 1.20.2, 2.3, 2.4, 2.5, 2.6, 3.3, 3.4, 3.5, 3.6, 3.10, 6, 7.1, 7.2.1, 7.2.2, 7.2.3, 7.2.5, 7.2.6, 7.2.7, 7.2.8, 8.1.1, 8.1.3, 8.1.4, 8.1.5, 8.1.6, 8.2.1, 8.2.2, 8.2.3, 8.2.4, 8.3, 9, 11, 12, 13.1, 13.2, 13.3, 13.4, 13.5, 13.6, 13.7, 13.8, 13.10, 17.1, 17.2, 18.1.6, 18.2.2, 22.2, 22.3, 22.4, 22.6, 22.7, 22.8, 22.17.1, 22.17.2, 23.2, 23.3, 25.2, 25.4, 25.5	"signed"	Feb 13, 2023

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1 EXECUTIVE SUMMARY

1.1 Introduction

Trilogy Metals Inc. (Trilogy or Trilogy Metals) is listed on the Toronto Stock Exchange (TSX) and the New York Stock Exchange (NYSE). As a result, Trilogy is a reporting issuer in Canada and must comply with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and is a registrant with the United States Securities and Exchange Commission (SEC) and must also comply with subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations of Regulation S-K (S-K 1300).

Trilogy commissioned Ausenco Engineering Canada Inc. (Ausenco) to manage the update of the 2020 Arctic Feasibility Study Technical Report (2020 FS) prepared in accordance with NI 43-101 into a prefeasibility-level study (the Arctic Project) and summarize into a S-K 1300 Technical Report Summary (the Report) on the Arctic deposit in the Ambler Mining District of northwest Alaska.

This Report was prepared by Ausenco, Brown & Caldwell (B&C), SRK Consulting (Canada) Inc. (SRK or SRK Canada), and Wood Canada Limited (Wood) for Trilogy to support disclosures in its Annual Report on Form 10-K for the fiscal year ended November 30, 2022.

The Report supports Mineral Resources and Mineral Reserves using the standards and definitions in S-K 1300.

The Report presents Mineral Resource and Mineral Reserve estimates for the Project, and an economic assessment based on open pit mining operations and a conventional processing circuit that would produce copper, zinc, and lead concentrates.

All units of measurement in this Report are metric, unless otherwise stated. Monetary units are in US dollars, unless otherwise stated.

1.2 Property Description

The Arctic property is located in the Ambler mining district (Ambler Mining District) of the southern Brooks Range, in the Northwest Arctic Borough (NWAB) of Alaska. The property is geographically isolated with no current road access or nearby power infrastructure. The property is located 270 km east of the town of Kotzebue, 37 km northeast of the village of Kobuk, and 260 km west of the Dalton Highway, an all-weather state-maintained highway and centred around geographic coordinates N67.17° latitude and W156.39° longitude.

1.2.1 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

NovaGold Resources Inc. (NovaGold) acquired the Arctic Project from Kennecott Exploration Company and Kennecott Arctic Company (collectively, Kennecott) in 2004. In 2011, NovaGold transferred all copper projects to NovaCopper Inc. and spun-out NovaCopper to its then existing shareholders in 2012. NovaCopper Inc. subsequently underwent a name change to Trilogy Metals Inc. in 2016. Under the Kennecott Purchase and Termination Agreement, Kennecott retained a 1% net smelter return (NSR) royalty that has been subsequently sold by Kennecott. The 1% NSR runs with the lands and is purchasable at any time from the royalty holder for a one-time payment of $10 million.

The Project is directly held by Ambler Metals LLC (Ambler Metals), in a 50/50 joint venture formed between South32 Limited (South32) and Trilogy Metals in February 2020. Upon the formation of the joint venture, Trilogy Metals contributed all of its Alaskan assets, including the Project and the agreement with NANA (see below), to Ambler Metals in exchange for a 50% membership interest and at the same time, South32 contributed $145 million in cash for a 50% membership interest.

The UKMP consists of an approximately 448,217-acre land package containing state, patented and native lands within an area of interest. There are two discrete mineralized belts within the UKMP – the Devonian Ambler Schist Belt and the Devonian Bornite Carbonate Sequence. The Project is located within the Ambler Schist Belt which comprises approximately 231,008 acres (93,446 ha) of State of Alaska mining claims and US Federal patented mining claims in the Kotzebue Recording District. Exclusive of native lands, the UKMP land tenure consists of 2,136 contiguous State claims totalling 230,736 acres (93,336 ha), including 905 40-acre claims, 1,231 160-acre claims, and 18 Federal patented claims comprising 271.9 acres (110 ha) held in the name of Ambler Metals LLC. Surface use of the private land held as Federal patented claims is limited only by reservations in the patents and by generally applicable environmental laws. Surface use of State claims allows the owner of the mining claim to make such use of the surface as is "necessary for prospecting for, extraction of, or basic processing of minerals."

The UKMP also consists of lands owned by NANA Regional Corporation, Inc. (NANA), who controls lands granted under the Alaska Native Claims Settlement Act (ANCSA) to the south of the Project boundary. Ambler Metals and NANA are parties to an agreement dated October 19, 2011 (the NANA Agreement) that consolidates the parties' land holdings into an approximately 190,929 ha land package and provides a framework for the exploration and development of the area. The NANA Agreement has a term of 20 years, with an option in favour of Ambler Metals to extend the term for an additional 10 years. If, following receipt of a feasibility study and the release for public comment of a related draft environmental impact statement, a decision is made to proceed with construction of a mine on the lands subject to the NANA Agreement, NANA will have 120 days to elect to either (a) exercise a non-transferrable back-in-right to acquire between 16% and 25% (as specified by NANA) of that specific project; or (b) not exercise its back-in-right, and instead receive a net proceeds royalty equal to 15% of the net proceeds realized from such project. In the event that NANA elects to exercise its back-in-right, the parties will, as soon as reasonably practicable, form a joint venture with NANA electing to participate between 16% to 25%, and Ambler Metals owning the balance of the interest in the joint venture. If Ambler Metals decides to proceed with construction of a mine on its own lands subject to the NANA Agreement, NANA will enter into a surface use agreement which will afford Ambler Metals access to the project along routes approved by NANA. In consideration for the grant of such surface use rights, NANA will receive a 1% net smelter royalty on production and provide an annual payment on a per acre basis.

1.3 Accessibility, Climate, Local Resources, and Infrastructure

Primary access to the Project is by air, using both fixed wing aircraft and helicopters.

There are four well-maintained, approximately 1,500 m-long gravel airstrips located near the Project, capable of accommodating charter fixed wing aircraft. These airstrips are located 64 km west at Ambler, 46 km southwest at Shungnak, 37 km southwest at Kobuk, and 34 km southwest at Dahl Creek. There is daily commercial air service from Kotzebue to the village of Kobuk, the closest community to the Project. During the summer months, the Dahl Creek Camp airstrip is suitable for larger aircraft, such as a C-130 and DC-6.

In addition to the four 1,500 m airstrips, there is a 700 m airstrip located at the Bornite Camp. The airstrip at Bornite is suited to smaller aircraft, which support the Bornite Camp with personnel and supplies. There is also a 450 m airstrip (Arctic airstrip) located at the base of Arctic Ridge that can support smaller aircraft.

A winter trail and a one-lane dirt track suitable for high-clearance vehicles or construction equipment links the Arctic Project's main camp located at Bornite to the Dahl Creek airstrip southwest of the Arctic deposit. An unimproved gravel track connects the Arctic airstrip with the Arctic deposit.

The climate in the region is typical of a sub-arctic environment. Weather conditions on the Project can vary significantly from year to year and can change suddenly. During the summer exploration season, average maximum temperatures range from 10 °C to 20 °C, while average lows range from -2°C to 7°C (Western Regional Climate Center: WRCC - Alaska Climate Summaries: Kobuk 1971 to 2000). By early October, unpredictable weather limits safe helicopter travel to the Project. During winter months, the Project can be accessed by snow machine, track vehicle, or fixed wing aircraft. Winter temperatures are routinely below -25°C and can exceed -50°C. Annual precipitation in the region varies with elevation.

It is expected that any future mining activity will be conducted on a year-round basis. Exploration activities are generally confined to the period from late May to late September.

Kotzebue is a potential source of limited mining-related supplies and labourers, and is the nearest centre serviced by regularly scheduled, large commercial aircraft (via Nome or Anchorage). In addition, there are seven other villages in the region that will be a potential source of some of the workforce for the Project. Fairbanks (population 32,515; 2020 US Census) has a long mining history along with currently operating mines and can provide most mining-related supplies and support that cannot be sourced closer to the Project area.

Drilling and mapping programs are seasonal and have been supported out of the Bornite Camp and Dahl Creek Camp. The Bornite Camp facilities are located on Ruby Creek on the northern edge of the Cosmos Hills. The camp provides office space and accommodations for the geologists, drillers, pilots, and support staff. Power is supplied by two diesel generators – one 300 kW and one 225 kW. Water was supplied by the permitted artesian well located 250 m from camp; however, a water well was drilled in camp during the 2017 field season that was permitted by 2019 to provide all potable water for the Bornite Camp.

1.4 History

Prior to Trilogy's Project interest, work programs were conducted by Bear Creek Mining Company (BCMC), an exploration subsidiary of Kennecott Exploration (Kennecott) and Anaconda. Exploration activities included geological and reconnaissance mapping, geochemical sampling, airborne and ground geophysical surveys, drilling, metallurgical testwork, petrological and mineralogical studies, and resource estimates.

Trilogy obtained its project interest from NovaGold in 2011. NovaGold obtained its project interest in 2004, when the Alaska Gold Company, a wholly-owned subsidiary of NovaGold completed an Exploration and Option Agreement with Kennecott to earn an interest in the Ambler land holdings. In 2010, NovaGold acquired a 100% controlling interest by buying out Kennecott's interest, although Kennecott retained an NSR royalty. Work conducted by NovaGold and Trilogy Metals (formerly, NovaCopper) included geological mapping, soil and silt geochemical sampling, time-domain electromagnetic (TDEM) ground geophysical surveys, airborne DIGHEM geophysical surveys, down-hole geophysics, drilling programs, metallurgical testwork, Mineral Resource and Mineral Reserve estimates, mining studies, and baseline environmental studies.

1.5 Geological Setting, Mineralization, and Deposit

The Arctic deposit is hosted in the Ambler Sequence, in the upper part of the regional Anirak Schist, in the Ambler Mining District on the southern margin of the Brooks Range in Alaska. Ambler Sequence is a group of Middle Devonian to Early

Mississippian, metamorphosed, bimodal volcanic rocks with interbedded tuffaceous, graphitic, and calcareous volcaniclastic metasediments. The Arctic deposit has characteristics that are representative of a volcanogenic massive sulphide (VMS) deposit based on its geologic setting, associated host rocks, ore morphology, and ore mineralogy. VMS-style mineralization is found along the entire 110 km strike length of the Ambler Sequence.

Mineralization occurs as stratiform semi-massive sulphide (SMS) to massive sulphide (MS) beds within primarily graphitic chlorite schists and fine-grained quartz schists. The sulphide beds average 4 m in thickness but vary from less than 1 m up to as much as 18 m in thickness.

The bulk of the mineralization occurs within eight modelled SMS and MS zones lying along the upper and lower limbs of the interpreted Arctic isoclinal anticline. All the zones are within an area of roughly 1 km^2 with mineralization extending to a depth of approximately 250 m below the surface. Mineralization is predominately coarse-grained sulphides consisting mainly of chalcopyrite, sphalerite, galena, tetrahedrite-tennantite, pyrite, arsenopyrite, and pyrrhotite. Trace amounts of electrum are also present.

1.6 Exploration

Drilling at the Arctic deposit and within the Ambler Mining District has been ongoing since its discovery in 1966. Approximately 67,639 m of drilling was completed within the Ambler Mining District, including 55,038 m of drilling in 285 drill holes at the Arctic deposit or on potential extensions in 32 campaigns spanning 56 years. Drill programs were completed by Kennecott and its subsidiaries, Anaconda, NovaGold, Trilogy and Ambler Metals.

Drill collar and downhole survey measurement collected since 2004 have used industry-recognized instrumentation and methods. Many historical collar locations have been resurveyed using these current methods. Between 1998 and 2011, Specific Gravity (SG) measurements were collected from short whole core samples using water displacement or water immersion methods. Since 2011 SG measurements are collected from assay sample intervals of half or whole core using water immersion methods. Core recovery is good.

1.7 Sample Preparation, Analysis and Security

Analytical methods and laboratory accreditations used for historical samples are not known. Samples from the NovaGold/NovaCopper/Trilogy/Ambler Metals programs were submitted to ALS Minerals of Vancouver, British Columbia, Canada for multielement analysis by Inductively Coupled Plasma Mass Spectrometry (ICP MS) following a 4-acid digestion, and for gold analysis of a 30-gram sample by Fire Assay (FA) with an Atomic Absorption (AA) finish. Over limit ICP-MS samples were resubmitted for analysis by ICP-Atomic Emission Spectroscopy (AES) or AA following a 4-acid digestion. Over limit gold results were resubmitted for analysis of a 30-gram sample by FA with a Gravimetric finish.

Standard reference materials, blanks, duplicates, and check samples have been regularly submitted for all NovaGold/NovaCopper/Trilogy and Ambler Metals era sampling campaigns. No significant quality control issues are evident in samples analysed since 2004. Samples analyzed since 2004 are in the QP's opinion appropriate for the mineralization style observed at Arctic and provide adequate confidence in the reported assay values. Historical copper and lead values (pre-2004) that remain in the primary assay database appear to be biased high and low, respectively. The broad spatial distribution of these original historical samples and density of samples with more recent assay values surrounding these samples in the QP's opinion reduces the risk associated with these observed biases.

1.8 Data Verification

Kennecott entered the historical drill hole information into tables in 1995. In 2006, NovaGold geologists verified the geologic data from the original paper logs against the Kennecott electronic format, and then merged the data into a Microsoft SQL database. In 2013, NovaCopper retained GeoSpark Consulting to complete a 100% verification of the collar survey, downhole survey, and sample interval data. Geospark was also retained to generate QA/QC reports for the NovaGold-era 2004 to 2008 and NovaCopper/Trilogy-era 2011, 2015, 2016, 2017, and 2019 drill campaigns. All data for the Arctic resource area is stored in the GeoSpark Core Database System created and managed by GeoSpark Consulting.

Between 2004 and 2005 NovaGold completed a resampling program of historic drill holes. As a result, 85 % of the entire assay interval database now has well supported recent assay results. The resampling program included reassay of 289 previously assayed historic sample intervals. Analysis of the paired historic and reassay results indicates there is a 10 % high bias in the legacy Cu values and a 13 % low bias in the legacy Pb values. Legacy sample represent only 15% of the entire assay database and are generally evenly distributed spatially between samples with more recent assay reducing the risk associated with these observed biases.

It is the QP’s opinion the drill database and topographic information for the Arctic deposit are reliable and sufficient to support the current estimate of Mineral Resources.

1.9 Mineral Processing and Metallurgical Testing

Since 1970, metallurgical testwork has been conducted to evaluate the ability of the Arctic deposit to produce copper, lead and zinc concentrates. In-general, the samples tested produced similar metallurgical performances and the project has seen the development of a robust metal recovery process to support the current operational plans. Work conducted included mineralogy and flotation testing, locked cycle tests, comminution tests, copper/lead separation testwork, talc optimization testwork, and thickening and filtration testing.

Testwork can be broken into four key time periods:

1. Historical testwork completed prior to 2012, primarily by Kennecott Research Centre (KRC) in Utah, and Lakefield Research Ltd., Lakefield, Ontario;
2. Preliminary Trilogy Metals testwork conducted at SGS Mineral Services, Vancouver (SGS Vancouver), in 2012 to 2015;
3. Detailed Trilogy Metals testwork conducted at ALS Metallurgy in Kamloops, BC (ALS Metallurgy) in 2015 to 2019; and
4. Ambler Metals testwork conducted at ALS Metallurgy and SGS Mineral Services in 2021 to 2022.

In 2012, SGS Vancouver conducted a metallurgical test program to further study metallurgical responses of the samples produced from Zones 1, 2, 3, and 5 of the Arctic deposit. The flotation test procedures used talc pre-flotation, conventional copper-lead bulk flotation and zinc flotation, followed by copper and lead separation. In general, the 2012-2015 test results indicated that the samples responded well to the flowsheet tested. The average results of the locked cycle tests (without copper and lead separation) were as follows:

- The copper recoveries to the bulk copper-lead concentrates ranged from 89% to 93% excluding the Zone 1 & 2 composite which produced a copper recovery of approximately 84%; the copper grades of the bulk concentrates were 24% to 28%.

- Approximately 92% to 94% of the lead was recovered to the bulk copper–lead concentrates containing 9% to 13% lead.
- The zinc recovery was 84.2% from Composite Zone 1 & 2, 93.0% from Composite Zone 3 and 90.5% from Composite Zone 5. On average, the zinc grades of the concentrates produced were higher than 55%, excluding the concentrate generated from Composite Zone 1 & 2, which contained only 44.5% zinc.
- Gold and silver were predominantly recovered into the bulk copper–lead concentrates. Gold recoveries to this concentrate ranged from 65% to 80%, and silver recoveries ranged from 80% to 86%.

Using an open circuit procedure, the copper and lead separation tests on the bulk copper–lead concentrate produced from the locked cycle tests generated reasonable copper and lead separation. The copper concentrates produced contained approximately 28% to 31% copper, while the grades of the lead concentrates were in the range of 41% to 67% lead. In this testwork program, it appeared that most of the gold reported to the copper concentrate and on average the silver was equally recovered into the copper and lead concentrates. Subsequent testwork to better define the copper and lead separation process was conducted in 2017, including a more detailed evaluation of the precious metal deportment in the copper and lead separation process.

Grindability testing was completed during both the SGS Vancouver and ALS Metallurgy testwork programs to support the design and economics of efficient grinding of the Arctic materials. Semi-autogenous grind (SAG) mill test results included a single JKTech drop-weight test and 19 SAG media competency (SMC) tests using variability samples. Test results show the material is amenable to SAG milling and is relatively soft, with a reported breakage (axb) average value of 189.7. Bond ball mill work index (BWi) tests were completed on 44 samples and values ranged from 5.4 to 13.1 kWh/t with an average BWi of 8.82 kWh/t. Abrasion index (Ai) tests were completed on five samples and values fluctuated from 0.017 to 0.072 g for the measured samples. The data indicate that the samples are neither resistant nor abrasive to ball mill grinding. The materials are considered to be soft or very soft in terms of grinding requirements. The grinding testwork was used to support detailed grinding circuit design.

In 2017, ALS Metallurgy conducted detailed copper and lead separation flotation testwork using a bulk sample of copper–lead concentrate produced from the operation of a pilot plant. This testwork confirmed high lead recoveries in locked cycle testing of the copper–lead separation process and confirmed precious metal recoveries into the representative copper and lead concentrates. This testwork indicated a clear tendency of the gold values to follow the lead concentrate, giving it a significant gold grade and value. Detailed mineralogical analysis showed that a majority of gold values were occurring as liberated fine-grained gold particles.

The conclusions of testwork conducted both in 2012 and 2017 indicate that the Arctic materials are well-suited to the production of high-quality copper and zinc concentrates using flotation techniques which are industry standard. Copper and zinc recovery data were reported in the range of 88% to 92%, which reflected the high-grade nature of the deposit as well as the coarse-grained nature of these minerals. Grade variations within the deposit will be observed as indicated by the grade variations observed in variability samples, however, mill feed variability is expected to be limited and readily manageable with good plant operational practices. Lead concentrates have the potential to be of good quality and can also be impacted by zones of very high talc. Considerable care will be required to ensure maximum talc recovery to remove talc, which has the potential to dilute lead concentrate grades. The lead concentrate is also shown to be rich in precious metals, which has some advantages in terms of marketability of this material.

Ancillary testwork was completed by third party consultants on representative concentrate samples, to provide thickening and filtration data for the various concentrates. Settling and filtration rates were observed to be typical for sulphide concentrates and moisture contents in final filter cakes were observed to be lower than expected.

Metallurgical testwork was completed to provide representative tailings samples for use in detailed solids settling and compaction testwork to provide data for tailings design studies.

A detailed study of water treatment chemistry was undertaken to evaluate and confirm the option of destroying cyanide contained in solutions from the proposed copper–lead separation process. The use of an SO2/air process in a small-scale pilot plant demonstrated removal of 99% of the contained cyanide and supported the concept of maintaining low cyanide concentrations within the proposed tailings pond solutions.

In 2021, various metallurgical testwork programs were conducted at ALS Metallurgy, SGS, and MO Group. ALS Metallurgy completed several testwork programs, including flotation testing with the Preflotation circuit only to establish talc performance; further flowsheet development testwork to investigate the benefits of sequential flotation versus the original bulk flow sheet; and a variability testwork to support the development of improved metallurgical recovery models.

The objective of the ALS Metallurgy program was to investigate bulk and sequential flotation flowsheets with composites formed from two parent composites, and then select a flowsheet for a geo-metallurgical evaluation through testing with variability samples.

The mineralization was amenable to either a bulk flowsheet followed by copper-lead separation, or a sequential flowsheet, both following a pre-flotation stage to remove talc.

Table 1-1 shows average performance obtained for the Avg Talc Composite in the Flowsheet Development phase of the testing.

Table 1-1: Comparison of Bulk versus Sequential Locked-Cycle Test Results – ALS 2021

Composite	Assays						Distribution (%)				
	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Mg (%)	Cu	Pb	Zn	Ag	Au
Avg Talc Bulk											
Copper concentrate	28.0	0.86	4.27	181	4.17	0.46	87.3	8.3	9.1	36.0	60.9
Lead concentrate	7.90	39.0	6.30	1124	4.75	1.23	5.1	78.1	2.8	46.0	14.3
Zinc concentrate	0.87	0.38	55.9	41	0.35	0.04	1.9	2.6	83.3	5.7	3.5
Avg Talc – Sequential											
Copper concentrate	27.6	0.87	2.05	168	3.23	1.96	90.2	8.9	4.7	34.9	48.7
Lead concentrate	2.72	49.3	9.71	1360	5.31	1.40	1.2	69.9	3.1	39.4	11.2
Zinc concentrate	0.98	1.09	54.5	47	0.77	0.17	2.1	7.3	83.5	6.5	7.7

Copper recovery to the copper concentrate was slightly higher for the sequential flowsheet; however, gold recovery to the copper concentrate was substantially lower. The lead concentrate grade for the Avg Talc composite could likely be improved over that shown above with optimization of copper-lead separation conditions given the higher lead concentrate grade measured with other composites.

Zinc circuit performance was similar for the two flowsheets, although higher zinc recovery to the copper concentrate was recorded for the bulk circuit. Magnesium content in the copper concentrate was higher for the sequential circuit, but similar in the lead concentrate for both circuits.

Based on economic analysis comparing the bulk and sequential circuit, the bulk circuit flowsheet was selected for the Variability testing.

An overall metallurgical balance for the project is summarized in Table 1-2. The projected metallurgical recoveries are based on an expected average recovery over the life-of-mine (LOM), and results of metallurgical variability testwork conducted in 2021 and 2022.

Table 1-2: Summary of Overall Metal Recovery – Arctic Project

Process stream	Mass (%)	Concentrate Grade					Metal Recoveries				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	Zn (%)	Au (%)	Ag (%)
Process Feed	100	2.1	0.5	2.8	0.4	31.1	-	-	-	-	-
Copper Conc.	6.3	30.3	1.7	0.7	3.4	160.5	92.1	19.4	1.6	52.2	32.5
Lead Conc.	0.6	2.0	53.9	5.9	14.1	2425.8	0.6	61.3	1.3	21.6	48.6
Zinc Conc.	4.7	1.0	0.5	53.7	0.3	38.3	2.2	4.4	88.5	3.2	5.7
Tailings	88.4	0.1	0.1	0.3	0.1	4.6	5.1	14.8	8.7	23.2	13.1

SGS conducted SAG Power Index (SPI®) tests to investigate the effect of friable ores on the plant throughput.

MO Group conducted talc circuit modelling using the data obtained from the ALS Metallurgy Preflotation testwork program to investigate the benefits of talc circuit open and closed-circuit cleaning. The MO Group also conducted dewatering and filtration testwork on the talc concentrate and final tailings generated from the Preflotation testwork program.

Thickening and filtration testwork were completed by the MO Group on representative preflotation concentrate and tailings samples, to investigate opportunities to improve water recovery and reduce operating costs. The results were used to incorporate a tailings thickener in the process plant flow sheet.

1.10 Mineral Resource Estimates

Mineral resource estimates are performed from a 3D block model based on geostatistical applications using Leapfrog software. The resource estimate was generated using drill hole sample assay results and the interpretation of a geological model which relates to the spatial distribution of copper, lead, zinc, gold, and silver. The grade models have been validated using a combination of visual and statistical methods. The resources were classified according to their proximity to the sample data locations and are reported using the standards and definitions of S-K 1300. Model blocks estimated by three or more drill holes spaced at a maximum distance of 100 m are included in the Indicated category. Inferred blocks are within a maximum distance of 150 m from a drill hole.

The deposit is amenable to open pit extraction methods. The reasonable prospects for economic extraction was established by the constraining mineralization within a pit shell derived based on a series of technical and economic assumptions. Mineral Resources were established within the constraining pit shell using the copper equivalent (CuEq) grade of 0.5%.

Mineral Resources are reported in accordance with the standards and definitions of S-K 1300. The Mineral Resource estimate inclusive of Mineral Reserves is stated in Table 1-3. The Mineral Resource estimate exclusive of Mineral Reserves is stated in Table 1-4. All Indicated Mineral Resources have been converted to Mineral Reserves. Mineral Resources are reported in place (point of reference) and on a 100% basis; however, Trilogy's attributable interest is 50% of the tonnes and metal content.

Table 1-3: Mineral Resource Summary Table, Inclusive of Mineral Reserves

Confidence Category	Tonnage (Mt)	Average Grade					Contained Metal Content				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Pb (Mlb)	Zn (Mlb)	Au (koz)	Ag (Moz)
Indicated	35.7	2.98	0.79	4.09	0.59	45.2	2,347	621	3,216	675	52
Inferred	4.5	1.92	0.70	2.93	0.43	35.6	189	69	288	62	5

Notes:
1. Mineral Resources are current as of November 30, 2022 and were verified by a Wood QP.
2. Mineral Resources were prepared in accordance with the standards and definitions of S-K 1300.
3. Mineral Resources stated are contained within a conceptual pit shell developed using metal prices of $3.00/lb Cu, $0.90/lb Pb, $1.00/lb Zn, $1300/oz Au and $18/oz Ag and metallurgical recoveries of 92% Cu, 77% Pb, 88% Zn, 63% Au and 56% Ag and operating costs of $3/t mining and $35/t process and G&A. The assumed average pit slope angle is 43°.
4. The cut-off grade is 0.5% copper equivalent. CuEq = (Cu%x0.92) + (Zn%x0.290) + (Pb%x0.231) + (Au g/tx0.398) + (Ag g/tx0.005).
5. As a result of flattening the north end of the reserve pit to stabilize the pit wall due to the presence of talc, a portion of the reserve pit extended beyond the resource constraining pit shell. Approximately 568kt of 1.72% Cu, 0.77% Pb, 0.23 g/t Au and 21.3 g/t Ag in the Indicated category, and approximately 319 kt of 2.01% Cu, 0.87% Pb, 2.53% Zn, 0.50 g/t Au and 37.5 g/t Ag in the Inferred category were added to the Mineral Resource tabulation.
6. The Mineral Resource estimate is reported inclusive of those Mineral Resource that were converted to Mineral Reserves.
7. Trilogy's attributable interest is 50% of the tonnage and contained metal stated in the table.
8. Figures may not sum due to rounding.

Table 1-4: Mineral Resource Summary Table, Exclusive of Mineral Reserves

Confidence Category	Tonnage (Mt)	Average Grade					Contained Metal Content				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Pb (Mlb)	Zn (Mlb)	Au (koz)	Ag (Moz)
Inferred	4.5	1.92	0.70	2.93	0.43	35.6	189	69	288	62	5

Notes:
1. Mineral Resources are current as of November 30, 2022 and were verified by a Wood QP.
2. Mineral Resources were prepared in accordance with the standards and definitions of S-K 1300.
3. Mineral Resources stated are contained within a conceptual pit shell developed using metal prices of $3.00/lb Cu, $0.90/lb Pb, $1.00/lb Zn, $1,300/oz Au and $18/oz Ag and metallurgical recoveries of 92% Cu, 77% Pb, 88% Zn, 63% Au and 56% Ag and operating costs of $3/t mining and $35/t process and general and administrative costs. The assumed average pit slope angle is 43º.
4. As a result of flattening the north end of the reserve pit to stabilize the pit wall due to the presence of talc, a portion of the reserve pit extended beyond the resource constraining pit shell and approximately 319 kt of 2.01% Cu, 0.87% Pb, 2.53% Zn, 0.50 g/t Au and 37.5 g/t Ag in the Inferred category were added to the Mineral Resource tabulation.

5. The cut-off grade is 0.5% copper equivalent: CuEq = (Cu% x 0.92) + (Zn% x 0.290) + (Pb% x 0.231) + (Au g/t x 0.398) + (Ag g/t x 0.005).
6. The Mineral Resource estimate is reported exclusive of those Mineral Resources that were converted to Mineral Reserves.
7. Trilogy's attributable interest is 50% of the tonnage and contained metal stated in the table.
8. Figures may not sum due to rounding.

1.11 Mineral Reserve Estimates

Mineral Reserves were classified in accordance with the standards and definitions of S-K 1300. Modifying factors were applied to the Indicated Mineral Resources to convert them to Probable Mineral Reserves. Mineral Reserves for the Arctic deposit incorporate appropriate mining dilution and mining recovery estimations for the open pit mining method.

The pit shell that defines the ultimate pit limit was derived in Whittle using the Pseudoflow pit optimization algorithm. The optimization procedure uses the block value and pit slopes to determine a group of blocks representing pits of valid slopes that yield the maximum profit. The block value is calculated using information stored in the geological block model, commodity prices, mining and processing costs, process recovery, and the sales cost for the metals produced. The pit slopes are used as constraints for removal precedence of the blocks (Xiaoyu Bai, et al., 2017). Table 1-5 provides a summary of the primary optimization inputs.

Table 1-5: Optimization Inputs

Parameter	Unit	Value	Cu Conc.	Pb Conc.	Zn Conc.
Metal Prices					
Copper	$/lb	3.46			
Lead	$/lb	0.91			
Zinc	$/lb	1.12			
Gold	$/oz	1,615			
Silver	$/oz	21.17			
Discount Rate	%	8			
Dilution and Mine Losses	%	Estimated in a block-by-block basis, adding up 30% to 40%.			
Mining Cost					
Reference Bench Elevation	m	790			
Base Cost	$/t	2.52			
Incremental Mining Cost					
Uphill (below 790m)	$/t/5m	0.02			
Downhill (above 790m)	$/t/5m	0.012			
Process Costs					
Operating Cost	$/t milled	18.31			
G&A	$/t milled	5.83			

Parameter	Unit	Value	Cu Conc.	Pb Conc.	Zn Conc.
Sustaining Capital	$/t milled	2.37			
Road Toll Cost	$/t milled	8.04			
Closure	$/t milled	4.27			
Processing Rate	kt/d	10			
Process Recovery					
Copper	%		89.9	2.4	2.7
Lead	%		8.1	79	2.2
Zinc	%		3.4	0.4	90.6
Gold	%		10.9	62.1	5.4
Silver	%		26.4	63.1	3.4
Payable – Main Element	%		96.5	95	85
Treatment Cost	$/dmt		80	160	215
Refining Cost					
Copper	$/lb		0.08	-	-
Gold	$/oz		5	10	-
Silver	$/oz		0.5	1.25	-
Transport Cost	$/dmt	271			
Concentrate Losses	% weight	0.42			
Insurance Cost	%	0.15			
Representation/Marketing	$/wmt	2.5			
Slope Angles					
Geotechnical Sector 1 (2L-E)	degrees	Variable based on slope dip direction. IRA ranging from 26 to 56.			
Geotechnical Sector 2 (2L-W)	degrees	Variable based on slope dip direction IRA ranging from 38 to 56.			
Geotechnical Sector 3 (2U)	degrees	Variable based on slope dip direction IRA ranging from 29 to 56.			
Geotechnical Sector 4 (3)	degrees	Variable based on slope dip direction IRA ranging from 30 to 56.			
Geotechnical Sector 5 (4L)	degrees	Variable based on slope dip direction IRA ranging from 34 to 56.			
Geotechnical Sector 6 (4U)	degrees	Variable based on slope dip direction IRA ranging from 37 to 56.			
Royalties					
NANA Surface Use	%NSR	1			

Note: IRA = inter ramp angle

The Mineral Reserve estimates are shown in Table 1-6. Only Probable Mineral Reserves have been classified. The point of reference for the Mineral Reserves is defined at the point where the ore is delivered to the processing plant. Trilogy's attributable interest is 50% of the tonnes of the Mineral Reserves.

Table 1-6: Mineral Reserve Statement

Confidence Category	Tonnage	Average Grade				
	Mt	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Probable Mineral Reserves	46.7	2.11	0.56	2.90	0.42	31.8

Notes:
1. Mineral Reserves estimates are current as of November 30, 2022 and were prepared by a Wood QP.
2. Mineral Reserves were estimated assuming open pit mining methods and include a combination of internal and contact dilution. Total dilution is expected to be between 30% and 40%. Pit slopes vary by sector and range from 26° to 56°. A marginal NSR cut-off of $38.8 /t is used.
3. Mineral Reserves are based on prices of $3.46/lb Cu, $0.91/lb Pb, $1.12/lb Zn, $1,615/oz Au, and $21.17/oz Ag.
4. Variable process recoveries averaging 92.2% Cu in Cu concentrate, 62.2% Pb in Pb concentrate, 8887.6% Zn in Zn concentrate, 16.0% Pb in Cu concentrate, 1.9% Zn in Cu concentrate, 47.2% Au in Cu concentrate, 32.7% Ag in Cu concentrate, 0.8% Cu in Pb concentrate, 1.3% Zn in Pb concentrate, 26.1% Au in Pb concentrate, 48.7% Ag in Pb concentrate, 2.1% Cu in Zn concentrate, 4.5% Pb in Zn concentrate, 3.3% Au in Zn concentrate, 5.8% Ag in Zn concentrate.
5. Mineral Reserves are based on mining cost of $2.52/t incremented at $0.02/t/5m and $0.012/t/5m below and above 790 m elevation, respectively.
6. Costs applied to processed material following process operating cost of $18.31/t, G&A of $5.83/t, sustaining capital cost of $2.37/t, closure cost of $4.27/t, road toll cost of $8.04/t.
7. Strip ratio (waste: ore) is 7.3:1.
8. Selling terms following payables of 96.5% of Cu, 95% of Pb and 85% of Zn, treatment costs of $80/t Cu concentrate, $160/t Pb concentrate and $215/t Zn concentrate; refining costs of $0.08/lb Cu in Cu concentrate, $10/oz Au, $1.25/oz Ag in Pb concentrate; and transport cost $270.98/t concentrate.
9. Fixed royalty percentage of 1% NSR.
10. Trilogy's attributable interest is 50% of the tonnage stated in the table.

Specific risks to the Arctic Mineral Reserve estimate include:

- The uncertainty in the construction and timing of the Ambler Mining District Industrial Access Project (AMDIAP) road, also called the Ambler Access Project (AAP), and whether a favourable road toll agreement can be negotiated.
- The presence of talc layers in the rock that have not been included in the current geological model could affect metallurgical recoveries and slope stability.

Other risk factors that may affect the Mineral Reserve estimates include: metal price assumptions; changes to the assumptions used to generate the NSR cut-offs that constrains the estimate; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical and hydrological assumptions, changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; assumptions as to concentrate marketability, payability and penalty terms; assumptions as to the continued ability to access the site, retain mineral tenure and obtain surface rights titles, obtain environment and other regulatory permits, and maintain the social license to operate.

1.12 Mining Methods

The Arctic Project is designed as a conventional truck-shovel operation with 144 t trucks and 15 m^3 shovels. The pit design includes four nested phases to balance stripping requirements while satisfying the concentrator requirements.

The design parameters include a ramp width of 30 m, road grades of 10%, bench height of 5 m, targeted mining width between 70 m and 100 m, berm interval of 20 m, variable slope angles by sector and a minimum mining width of 30 m.

The smoothed final pit design contains approximately 46.7 Mt of ore and 340.2 Mt of waste for a resulting stripping ratio of 7.3:1. Within the 46.7 Mt of ore, the average grades are estimated to be 2.11% Cu, 2.90% Zn, 0.56% Pb, 0.42 g/t Au and 31.8 g/t Ag.

The scheduling constraints set the maximum mining capacity at 35 Mt/a, and the maximum process capacity at 10 kt/d. The production schedule based on the Probable Mineral Reserves shows a total LOM of 15 years, including 2 years of pre-production and 13 years of production.

1.13 Processing and Recovery Methods

The 10,000 t/d process plant design is conventional for the industry and will operate two 12- hour shifts per day, 365 d/a with an overall plant availability of 92%. The process plant will produce three concentrates: 1) copper concentrate, 2) zinc concentrate, and 3) lead concentrate. Gold and silver are expected to be payable at a smelter; both silver and gold are expected to be payable in the copper and lead concentrates.

While there are several deleterious elements reporting to the concentrates at levels that could incur penalties, special processing provisions have been included in the flowsheet to make a readily saleable concentrate. The presence of naturally hydrophobic talc minerals was consistently observed in the various testwork programs. There is little reason to expect concentrates will be impaired by talc contamination as talc can be effectively removed from the flotation process prior to base metal flotation. Talc and fluorine levels will be managed by optimization of the talc pre-float circuit, effectively removing talc and fluorine to ensure the quality of the lead concentrate.

The mill feed will be hauled from the open pit to a primary crushing facility where the material will be crushed by a jaw crusher to a particle size of 80% passing 80 mm.

The crushed material will be ground by two stages of grinding, consisting of one SAG mill and one ball mill in closed circuit with hydrocyclones (SAB circuit). The hydrocyclone overflow with a grind size of approximately 80% passing 70 µm will first undergo talc pre-flotation, and then be processed by conventional bulk flotation (to recover copper, lead, and associated gold and silver), followed by zinc flotation. The bulk rougher concentrate will be cleaned and followed by copper and lead separation to produce a lead concentrate and a copper concentrate. The final tailings from the zinc flotation circuit will be pumped to a tailings management facility (TMF). Copper, lead, and zinc concentrates will be thickened and pressure-filtered before being transported by truck to a port and shipped to smelters.

Based on the mine plan developed for the Report and metallurgical testwork results, the LOM average metal recoveries and concentrate grades are as presented in Table 1-7.

Table 1-7: LOM Average Recovery and Grade

Description	Unit	Cu con	Pb con	Zn con
LOM Production	t/a	234,132	23,300	174,202
Grade	%	30.3% Cu	53.9% Pb	53.7% Zn
Recovery	%	92.1% Cu 52.2% Au 32.5% Ag	61.3% Pb 21.6% Au 48.6% Ag	88.5% Zn

The recovery plan includes provision for reagents, and water and power requirements.

1.14 Infrastructure

1.14.1 Infrastructure Requirements

The Project site is a remote, greenfield site that is remote from existing infrastructure. Infrastructure that will be required for the mining and processing operations will include:

- Open pit mine
- Stockpiles and Waste Rock Facility (WRF)
- Truck workshop, truck wash, mine offices, mine dry facility and warehouse
- Administration building
- Mill dry facility
- Plant workshop and warehouse
- Primary crushing building
- Fine ore stockpile building
- Process plant and laboratory
- Concentrate loadout building
- Reagent storage and handling building
- Raw water supply building
- Explosives storage silos and magazines
- Avalanche mitigation structures
- TMF
- Surface water diversion and collection channels, culverts, and containment structures
- Waste rock collection pond (WRCP)
- Water treatment plant (WTP)

- Process pond

1.14.2 Access

The Project site will be accessed through a combination of State of Alaska-owned highways (existing), an Alaska Industrial Development and Export Authority (AIDEA)-owned private road (proposed) and Ambler Metals-owned access roads (proposed). The AAP road is proposed by AIDEA to connect the Ambler Mining District to the Dalton Highway. The AAP road will be permitted as a private road with restricted access for industrial use. To connect the Arctic Project site and the existing exploration camp to the proposed AAP road, a 30.7 km access road (the Arctic access road) will need to be built.

The AAP road will be permitted as a private road with restricted access for industrial use and received a Federal Record of Decision on July 23, 2020, by the Bureau of Land Management (BLM) and the National Park Service (Joint Record of Decision or JROD). Lawsuits were filed shortly thereafter by a coalition of national and Alaska environmental non-government organizations in response to the BLM's issuance of the JROD for the AAP.

On February 22, 2022, the United States Department of the Interior (DOI) filed a motion to remand the Final Environmental Impact Statement (EIS) and suspend the right-of-way permits issued to AIDEA for the AAP and in mid-March, the BLM and DOI suspended the right-of-way grant and the right-of-way permit over federal lands.

The lawsuits have been temporarily suspended pending the additional work to be performed by the BLM on the EIS.

The State of Alaska-owned, public Dahl Creek airport will require upgrades to support the planned regular transportation of crews to and from Fairbanks. The cost of these upgrades has been included in the capital cost estimate.

1.14.3 Power

Power generation will be by five diesel generators, producing a supply voltage of 13.8 kV. The total connected load will be 25.9 MW with a normal running load of 21.0 MW. Diesel will be supplied via existing fuel supply networks in the region and shipped along the AAP road.

1.14.4 Accommodation

The Project will require three self-contained camps, in two different locations, equipped with their own power and heat generation capabilities, potable WTP (PWTP), sewage treatment plant (STP), and garbage incinerator. The existing 90-person Bornite Camp currently used for exploration will be expanded and used to start the construction of the Arctic access road and the logistics yard and construction camp. This Bornite Camp will be expanded and available prior to surface access from the Dalton Highway via the AAP road is available. A 250-person construction camp (CC) will be constructed at location near the intersection of the AAP road and Arctic Mine Access road (AMA), across the road from the Logistics Yard. CC will be constructed when limited access via the AAP is available for the transport of camp modules. A 400-person Permanent Accommodations Facility (PAF) will be constructed in the same location as CC. The PAF will be constructed when the AAP is available to transportation of the modules by truck and will be operational by the peak accommodation requirements for the construction phase. CC will be adjacent to the PAF, and the accommodation sections will be integrated and operated with the kitchen, dining, and support facilities of the PAF. The PWTP, STP, and garbage incinerator will be transported to site and constructed with the CC and will be sized to support the future PAF and the capacities of both the CC and PAF (650-persons).

1.14.5 Waste Rock Facility

The WRF will be developed north of the Arctic pit in the upper part of the Subarctic Creek valley. The WRF is designed as part of the tailings dam structure to provide a buttress for tailings containment in the adjacent footprint. The total volume of waste rock is expected to be 162.6 Mm^3 (340 Mt); however, there is potential for expanded volume in the waste if placement density is <2.0 t/m^3. The WRF will have a final height of 340 m to an elevation of 990 masl and is planned to be constructed in lifts of either 5, 10 or 20 m height with catch benches every 20 m to achieve an overall slope angle of 2.5H:1V.

Most of the waste rock is anticipated to be potentially acid-generating (PAG) and there will be no separation of waste based on acid generation potential. Rather, seepage from the WRF will be collected and treated.

1.14.6 Overburden Stockpiles

There will also be three overburden stockpiles to store the stripped topsoil and overburden from the TMF and WRF footprint. The topsoil stockpile will be placed between the haul roads to store up to 325,000 m^3 while the overburden stockpile will be located south of the WRF to store up to 2,200,000 m^3.

1.14.7 Tailings Management Facility

The TMF will be located at the headwaters of Subarctic Creek, in the upper-most portion of the creek valley. The 59 ha footprint of the TMF will be fully lined with a geomembrane liner. Tailings containment will be provided by an engineered dam, buttressed by the WRF that will be constructed immediately downstream of the TMF, and the natural topography on the valley sides. A starter dam will be constructed to elevation 830 m. Three subsequent raises will bring the final dam crest elevation to 892 m, which is 98 m lower than the final elevation of the WRF. The TMF is designed to store approximately 37.4Mm^3 (41.2 Mt) of tailings produced over the 13-year mine life, 3.3 Mm^3 of additional pond water, as well as 1.5 times the probable maximum flood, with 2.5 m of freeboard.

1.14.8 Water Management

The proposed mine development is located in the valley of Subarctic Creek, a tributary to the Shungnak River. A surface water management system will be constructed to segregate contact and non-contact water. Non-contact water will be diverted around mine infrastructure to Subarctic Creek. A groundwater seepage monitoring and collection system will be located down gradient of the WRF and seepage collection pond. Contact water will be conveyed to treatment facilities prior to discharge to the receiving environment.

A WRCP will be located directly below the toe of the WRF and will be used to collect seepage from the WRF, runoff from the WRF and haul road corridor area, and water pumped from the open pit.

The Project water and load balance model was updated to include the current water management plan. The model indicates that during operations, excess water from the WRCP will need to be treated prior to discharge to the receiving environment. During closure, water from the dewatering of the TMF will also need to be treated prior to discharge to the receiving environment.

1.14.9 Water Treatment Plant

It was assumed the site will be assigned water quality-based effluent limits (WQBEL) matching the state's Water Quality Standards (WQS) for the nearby Subarctic Creek. Therefore, Water Treatment Plant (WTP) is designed to treat all parameters in the predicted site wastewater to the WQS of Subarctic Creek. This will eliminate the need for a mixing zone and allow treated water to be discharged year-round if needed.

A single WTP, built in stages, will be used. During Operations phase the WTP will treat effluent from the Waste Rock Collection Pond (WRCP), and during Closure phase effluent from the pit. The WTP will initially consist of chemical/physical treatment with reverse osmosis (RO) filtration. During operations, the RO reject will be sent to the TMF, and only RO permeate will be discharged to Subarctic Creek. When the TMF is closed at the end of the operations, a biological/chemical/physical plant will be added to treat the RO reject. The biologic plant discharge will be mixed with the RO prior to discharge.

1.15 Market Studies

Metal pricing was guided by 3-year trailing average prices and long-term price forecasts from analysts as published by CIBC in 2022.

The long-term consensus metal price assumptions selected for the economic analysis in the Report were:

- Copper: $3.65/lb
- Zinc: $1.15/lb
- Lead: $1.00/lb
- Gold: $1,650/oz
- Silver: $21.00/oz

Smelter terms were prepared in January 2023 by StoneHouse Consulting Inc. Smelter terms were applied for the delivery of copper, zinc and lead concentrate. It was assumed that delivery of all concentrates would be to a smelter in the Asia Pacific region at currently available freight rates. Total transport costs for the concentrate are estimated at $324.37/dmt.

1.16 Environmental Studies, Permitting, and Plans, Negotiations, or Agreements with Local Individuals or Groups

The Arctic Project area includes the Ambler lowlands and Subarctic Creek within the Shungnak River drainage. A significant amount of baseline environmental data collection has occurred in the area including surface and groundwater quality sampling, surface hydrology monitoring, wetlands mapping, aquatic life surveys, avian and mammal habitat surveys, cultural resource surveys, hydrogeology studies, meteorological monitoring, and metal leaching and acid rock drainage (ML/ARD) studies.

1.16.1 Permitting Considerations

Current mineral exploration activities are conducted at the Arctic deposit under State of Alaska and Northwest Arctic Borough (NWAB) permits. The State of Alaska Miscellaneous Land Use Permit (MLUP) and the NWAB Permit both expires at the end of 2022 and will be renewed.

Mine development permitting will be largely driven by the underlying land ownership with regulatory requirements varying depending on land ownership. The Arctic Project area includes patented mining claims owned by Ambler Metals (private land), NANA land (private land), and State of Alaska land.

The Arctic deposit is mainly on private lands owned by Ambler Metals and infrastructure for the Arctic Project is situated to a large extent on State land. It will be necessary to obtain a Plan of Operation Approval (which includes the Reclamation Plan and Closure Cost Estimate) from the Alaska Department of Natural Resources (ADNR). The Project will also require certificates to construct and operate dams (tailings and water storage) from the ADNR (Dam Safety Unit) as well as water use and discharge authorizations, an upland mining lease and a mill site lease, as well as several minor permits including those that authorize access to construction material sites from ADNR.

The Alaska Department of Environmental Conservation (ADEC) would authorize waste management under an Integrated Waste Management permit, air emissions during construction and operations under an air permit, and an Alaska Pollutant Discharge Elimination System (APDES) permit for any wastewater discharges, and a Multi-Sector General Permit for stormwater discharges. The ADEC would also be required to review the US Army Corps of Engineers (USACE) Section 404 permit to certify that it complies with Section 401 of the Clean Water Act (CWA).

The Alaska Department of Fish and Game (ADF&G) would have to authorize any culverts or bridges that are required to cross fish-bearing streams or other impacts to fish-bearing streams that result in the altering or affecting fish habitat.

US Army Corps of Engineers (USACE) would require a CWA Section 404 permit for dredging and filling activities in Waters of the United States including jurisdictional wetlands. The USACE Section 404 permitting action would require the USACE to comply with the Natural Environmental Policy Act (NEPA) and, for a project of this magnitude, the development of an Environmental Impact Statement (EIS) is anticipated. The USACE would likely be the lead federal agency for the NEPA process. As part of the Section 404 permitting process, the Arctic Project will have to meet USACE wetlands guidelines to avoid, minimize and mitigate impacts to waters of the US including wetlands.

The Arctic Project will also have to obtain approval for a Master Plan from the NWAB. In addition, actions will have to be taken to change the borough zoning for the Arctic Project area from Subsistence Conservation and General Conservation to Resource Development and Transportation.

The overall timeline required for permitting would be largely driven by the time required for the NEPA process, which is triggered by the submission of the Section 404 permit application to the USACE. The timeline includes the development and publication of a draft and final EIS and ends with a Record of Decision and Section 404-permit issuance. In Alaska, the EIS and other State and Federal permitting processes are generally coordinated so that permitting and environmental review occurs in parallel. The NEPA process could require about three years to complete and could potentially take longer.

1.16.2 Social and Community

The Arctic Project is located approximately 40 km northeast of the villages of Shungnak and Kobuk, and 65 km east-northeast of the community of Ambler. The population in these villages are 151 in Kobuk (2020 Census), 210 in Shungnak (2020 Census), and 275 in Ambler (2020 Census). Residents largely live a subsistence lifestyle with incomes supplemented by guiding, local development projects, employment through tribal and city councils, government aid, and employment both in and outside of their home villages.

The Arctic Project has the potential to significantly improve work opportunities for residents during the exploration phase, construction, and during full operation. Trilogy's joint venture, Ambler Metals works directly with the Upper Kobuk villages and communities throughout the region to employ residents as mechanics, geotechnicians, core cutters, administrative staff, camp services, heavy equipment operators, drill helpers, and environmental technicians.

Stakeholder outreach and community meetings in the region by the Project's owners over many years have provided the opportunity to engage with residents, provide updated information on the project and future plans for the UKMP, hear concerns, answer questions, and build relationships.

This engagement has also identified various hurdles residents have faced when applying for employment. Opportunities have been created for NANA shareholders to apply and receive educational scholarships, participate in job shadowing at Bornite, driver's license courses, and heavy equipment operator training sponsored by the Project's owners.

It is the company's goal to continue and grow these efforts throughout the permitting process and the life of the project – encouraging and supporting education, job training, employment, and economic growth.

1.16.3 Closure Planning

Mine reclamation and closure are largely driven by State of Alaska regulations that specify that a mine must be reclaimed concurrent with mining operations to the greatest extent possible and then closed in a way that leaves the site stable in terms of erosion and avoids degradation of water quality from acid rock drainage or metal leaching on the site. A detailed Reclamation Plan will be submitted to the State of Alaska agencies for review and approval in the future, during the formal mine permitting process.

Owing to the fact that the Arctic Project is likely to have facilities on a combination of private (patented mining claims and native land) and State land, the Reclamation Plan will be submitted and approved as part of the Plan of Operations, which is approved by the ADNR. However, since the Reclamation and Closure Plan must meet regulations of both ADNR and the ADEC, both agencies will review and approve the Reclamation Plan. In addition, private landowners must formally concur with the portion of the Reclamation Plan for their lands so that it is compatible with their intended post-mining land use.

1.17 Capital and Operating Cost Estimates

1.17.1 Capital Costs

The capital cost estimate is a AACE Class 3 estimate with an accuracy of +15%-10% and uses Q4-2022 US dollars as the base currency. The total estimated initial capital cost for the design, construction, installation, and commissioning of the Arctic Project is estimated to be $1,177 million including ≤15% contingency. A summary of the estimated initial capital cost, sustaining capital cost and closure costs is shown in Table 1-8.

Table 1-8: Capital Cost Summary

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
1000	Mining	277.4	17.5	294.9
2000	Crushing	42.5	0	42.5
3000	Process Plant	158.0	1.3	159.3
4000	Tailings	88.3	32.4	120.7
5000	On-site Infrastructure	172.8	35.1	207.9
6000	Off-site Infrastructure	75.8	0	75.8
Sub-total Direct Costs		**834.1**	**86.3**	**920.4**
7000	Indirects	177.4	15.1	192.5
8000	Contingency (~12% of total project cost)	138.5	13.0	151.5
9000	Owner's Costs	26.8	0	26.8
Sub-total Indirect Costs		**342.7**	**28.1**	**370.8**
Project Total		**1,176.8**	**114.4**	**1,291.2**
Project Total – Closure Costs				**170.8**

1.17.2 Operating Costs

The operating cost estimates use US dollars as the base currency and have an accuracy of ±15% and include contingency. An average operating cost was estimated for the Arctic Project based on the proposed mining schedule. These costs included mining, processing, G&A, surface services, and road toll & maintenance costs. The average LOM unit operating cost for the Arctic Project is estimated to be $59.83/t milled. The breakdown of costs in Table 1-9 is estimated based on the LOM average mill feed rate of 3,650,000 t/a.

All pre-production costs have been included in the capital cost estimate in Section 1.1 above.

Table 1-9: Overall Operating Cost Estimate

Description	LOM Average Unit Operating Cost ($/ t milled)	LOM Average Annual Cost ($ M/a)	Percentage of Total Annual Operating Costs
Mining*	22.49	82.1	37.6%
Processing	22.60	82.5	37.8%
G&A	5.85	21.3	9.8%
Road Toll and Maintenance	7.72	28.2	12.9%
Water Treatment	1.17	4.3	2.0%
Total Operating Cost	**59.83**	**218.4**	**100%**

* Excludes pre-production costs. Includes contingency which is less than 15%.

1.18 Economic Analysis

The results of this economic analysis represent forward looking information. The results depend on the inputs that are subject to several known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented in this section. Information that is forward looking includes mineral reserve estimates, commodity prices, the proposed mine production plan, construction schedule, projected recovery rates, proposed capital and operating cost estimates, closure cost estimates, toll road cost estimates, and assumptions on geotechnical, environmental, permitting, royalties, and hydrogeological information.

An economic analysis was undertaken on a 100% project ownership basis to determine the internal rate of return (IRR), net present value (NPV) and payback on initial investment of the Arctic Project. Trilogy holds 50% interest in the Arctic Project through its ownership in Ambler Metals. The Project consists of a three-year construction period, followed by 13 years of production.

Ausenco developed a pre-tax cash flow model for the Arctic Project and the NPV and IRR were calculated at the beginning of the construction period in Year -3.

The pre-tax financial model incorporated the production schedule and smelter term assumptions to produce annual recovered payable metal, or gross revenue, in each concentrate stream by year. Off-site costs, including the applicable refining and treatment costs, penalties, concentrate transportation charges, marketing and representation fees, and royalties were then deducted from gross revenue to determine the NSR. The operating cash flow was then produced by deducting annual mining, processing, G&A, surface services, and road toll & maintenance charges from the NSR. Initial and sustaining capital was deducted from the operating cash flow in the years they occur, to determine the net cash flow before taxes. Initial capital cost includes all estimated expenditures in the construction period, from Year -3 to Year -1 inclusive. First production occurs at the beginning of Year 1. Sustaining capital expenditure includes all capital expenditures purchased after first production, including mine closure and rehabilitation. The model includes an allocation of a 1% NSR attributable to NANA.

With total capital costs of $1,719 million over LOM ($1,177 million initial capital cost, $114 million sustaining capital cost and $428 million closure costs), the project demonstrates a pre-tax NPV of $1,500 million at an 8% discount rate, IRR of 25.8% and payback period of 2.9 years. Post-tax financials have an NPV of $1,108 million at an 8% discount rate, IRR of 22.8% and payback period of 3.1 years.

1.19 Interpretations and Conclusions

The Arctic deposit will be mined at an annual rate of 35 Mt of ore per year with an overall stripping ratio of 7.3. Ore will be processed by conventional methods to annually produce 234 kt of copper, 23 kt of lead, and 174 kt of zinc, all in concentrates for provision to third party refiners. Waste and tailings materials will be stored in surface facilities, which will be closed and reclaimed at the end of the mine; contact water will be treated and discharged to the environment throughout the LOM. Precious metals attendant with the concentrates will be largely payable. While there are deleterious elements reporting to the concentrates at levels that could incur penalties, special processing provisions have been included in the flowsheet to make a readily saleable concentrate.

In terms of project execution, the mine requires nominally two years of pre-strip operations, tailings pond starter dam development and water accumulation before actual production mining operations can commence.

For that pre-strip work to start, the Arctic access road from the AMDIAP intersection to the mine site will have to be constructed to at least a pioneer road condition that will allow the mine fleet and the support facilities to be delivered, built, and made operational.

Positive financial results support the declaration of Mineral Reserves.

1.20 Recommendations

It is the QPs' opinion that the identified technical and economic risks and uncertainly related to the property can be reduced and improved with the additional recommended work programs as outlined below, helping to continue developing the Project through engineering and de-risking, and into construction at a total cost of $14.4 million.

Table 1-10 and the following subsections summarize the key recommendations arising from a review of each major area of investigation completed as part of this study to improve the base case design.

Table 1-10: Summary of Recommended Work Packages

Description	Cost ($)
Mining	1,073,000
Geology and Resource Models	10,190,000
Open Pit Geotechnical Work	150,000
Hydrogeology	490,000
Tailings Management Facility	1,420,000
Closure	250,000
Water Treatment	170,000
Metallurgical Testing	175,000
Recovery Methods	350,000
Operational Readiness Plan	120,000
Total	**14,388,000**

1.20.1 Mining

- Perform a SMU study to define an optimal block size that can support the envisioned production rate while minimizing dilution
- Requote explosives as Ammonium nitrate prices are currently in record highs due to the Ukraine-Russia conflict
- Assess the use of alternate fuel sources for the mine mobile equipment
- Confirm the location, thickness, and continuity of the talc layers near to the northeast wall of the final pit.

1.20.2 Geology and Resource Models

- Bias in historical copper and lead assays: compare the current Mineral Resource estimate with an estimate prepared using a correction factor applied to the historic Cu and Pb assay values that remain in the assay database or an estimate that does not use these historical intervals
- Bias in predicted SG values: review methodology and dataset used to generate the predicted SG values to minimize or mitigate apparent bias
- Mineral Resource Classification confidence related to geological complexity: consider putting additional in-fill holes in drill grids to mitigate the risk of complex geology within the first 5 years of production. This is recommended to classify Measured Resources which can then be converted to Proven Mineral Reserves and provide high confidence in the production plan at project start and payback period.
- Update the geological and resource model using all available data, including that from the 2022 drilling program
  - Review and update variogram models for all metals and SG using all available data
  - Review and update the resource model estimation parameters for all metals and SG
  - Review the current compositing length of 1m that is more in agreement with the average sample length
  - Update the geological model wireframes with all available data.
- Address minor database issues: transcription error checks, removing survey measurements causing excessive deviation, reviewing detection limits, and including additional fields to indicate if chosen values are supported by QC

1.20.3 Open Pit Geotechnical work

- Update the 3D talc model to include all available drilling and laboratory testing data and complete a geotechnical review to update the slope designs
- Consider potential slope instability risks identified in the interim and final mining phases of the open pit design
- Re-evaluate seismic impact on the slope design
- Review the pore pressure monitoring system and mitigation plan as additional data becomes available, adopting mitigation efforts as appropriate.

1.20.4 Hydrogeology

- Update the hydrogeological conceptual model for the pit and valley areas as more data becomes available
- For the pit, continue water level monitoring, assess potential effects of seasonal or rapid and high amplitude recharge events (i.e., freshet), assess pit sump locations, assess drainage ditches around and set back from the pit perimeter, consider snow removal or clearing off high elevation portions of the pit, update pit Groundwater Management Plan as mine design advances, and install general pit perimeter groundwater monitoring on the north, south and west sides of the pit.
- For the valley bottom, conduct hydrogeological drilling at location of the groundwater seepage interception system (SIS), update conceptual model, update estimates of potential groundwater bypass of the WRCP and the groundwater SIS design, and initiate baseline monitoring of groundwater quality around groundwater SIS.

1.20.5 Tailings Management Facility

- Complete geotechnical and hydrogeological investigations in the foundations of the TMF, WRF, WRCP, and Process Pond including borehole drilling and laboratory testing
- Conduct Dam Breach Assessment, based on site specific hazard assessment for the TMF, the WRCP and Process pond
- Finite element consolidation and seepage modelling of the TMF
- Tailings deposition planning and water balance update
- Evaluate tailings consolidation and requirement for an underdrain system
- Update WRF, TMF, WRCP, and Process Pond stability analyses (including liquefaction assessment) based on additional field investigation results and lab testing
- Evaluate whether heat or gas generation during oxidation of waste rock may adversely affect the closure cover.

1.20.6 Closure

- Conduct revegetation and cover study, and conduct trials during operations to develop a revegetation program, including climatic variables
- Consult with the Alaska Plant Materials Center to develop a revegetation program utilizing native species to determine the optimal growth media depth, amendments, and soil cover to optimize plant survival.

1.20.7 Water Treatment

- Review and adjust the WTP design as the discharge permitting advances and the actual permitted discharge criteria become more defined
- Continue to track and address identified project risks and opportunities
- Implement bench-scale and pilot-scale treatment tests to confirm high treatment rates required to meet the WQS and to right-size treatment equipment
- Evaluate RO reject treatment train once real mine-impacted water is generated during the operation phase
- Refine the acids and bases used in the WTP to reduce the risk of TDS WQS compliance issues
- Continue to refine WTP influent water quality predictions.

1.20.8 Metallurgical testing

- Additional testwork to further refine metallurgical performance and recovery estimates for the flowsheet, including:
    - Variability testing on samples with characteristics that match the LOM feed material
    - Circuit flowsheet modifications to improve concentrate quality and reduce deleterious elements, where applicable
    - Comminution testing on new samples to optimize comminution power.

1.20.9 Recovery methods

- Capital cost optimization and a thorough review of equipment sizing/selection based on the geometallurgy outcomes.

1.20.10 Operational Readiness Plan

- Develop a robust Operational Readiness (OR) plan to address risks and challenges associated with operating at a remote site:
    - Review project and operating assumptions to take into consideration the impacts of labour availability and logistical challenges with availability of equipment
    - Create a start-up/operational risk register to highlight issues for decisions taken during design
    - Incorporate development and implementation considerations for the remote cold-weather project site with limited access.
    - Develop alternatives for supply, transportation and logistics for reagents, concentrate, and any other goods
    - Track environment and operations-related permits to avoid costly alternatives
    - Explore opportunities to develop synergies with existing businesses and build an “industrial base” required to sustain mining and processing operations
    - Identify vendors to support contracts, parts, warehousing, training and first fills
    - Consider strategic Human Resources (HR) staffing decisions such as innovative labour scheduling and local sourcing
    - Identify Health, Safety, Environment and Community (HSEC) policies and procedures for a remote site to leverage local resources and other tie-ins where possible
    - Early mobilization of key technical functions to support mining and operations.

2 INTRODUCTION

2.1 Introduction

Trilogy Metals Inc. (Trilogy or Trilogy Metals) is listed on the Toronto Stock Exchange (TSX) and the New York Stock Exchange (NYSE). As a result, Trilogy is a reporting issuer in Canada and must comply with National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) and is a registrant with the United States Securities and Exchange Commission (SEC) and must also comply with subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations of Regulation S-K (S-K 1300).

Trilogy commissioned Ausenco Engineering Canada Inc. (Ausenco) to manage the update of the 2020 Arctic Feasibility Study Technical Report (2020 FS) prepared in accordance with NI 43-101 into a prefeasibility-level study (the Arctic Project) and summarize into a S-K 1300 Technical Report Summary (the Report) on the Arctic deposit in the Ambler Mining District of northwest Alaska.

The Arctic Project is directly held by Ambler Metals, a 50/50 joint venture formed between South32 and Trilogy in February 2020. Upon the formation of the joint venture, Trilogy contributed all of its Alaskan assets, including the Arctic Project and Trilogy's agreement with NANA (see Section 3.4.2), to Ambler Metals in exchange for a 50% ownership interest and at the same time, South32 contributed $145 million in cash for their 50% ownership interest.

2.2 Terms of Reference

The firms and consultants who have provided updates and are co-authors of this Report are Ausenco, B&C, SRK and Wood.

The Report excludes the Bornite group of claims, which is a separate project and subject to a separate technical report summary.

All units of measurement in this Report are metric, unless otherwise stated.

The monetary units are in US dollars, unless otherwise stated.

Mineral Resources and Mineral Reserves are prepared in accordance with the standards and definitions of S-K 1300.

2.3 Qualified Persons

Table 2-1 provides a list of the firms and individuals that acted as third party Qualified Persons (QPs) in the preparation of this Report.

Table 2-1: Report Contributors

Qualified Person	Report Responsibilities	Report Sections
Ausenco	Plant and infrastructure design, metallurgy, recovery methods, consolidation of the capital costs and operating costs and the overall financial model	1.1, 1.3, 1.4, 1.9, 1.13, 1.14.1, 1.14.2, 1.14.3, 1.14.4, 1.15, 1.17, 1.18, 1.19, 1.20.8, 1.20.9, 1.20.10, 2, 3.1, 3.2, 3.10, 4, 5.3.8, 10, 14, 15.1, 15.2, 15.3, 15.4, 15.5, 15.6, 15.7, 15.10.8, 15.12, 15.13, 15.14, 15.15, 16, 18.1.1, 18.1.2, 18.1.3, 18.1.4, 18.1.5, 18.1.7, 18.1.9, 18.1.10, 18.1.11, 18.1.12, 18.1.13, 18.1.14, 18.2.1, 18.2.3, 18.2.4, 18.2.5, 18.2.6, 19, 21, 22.1, 22.5, 22.9, 22.10, 22.12, 22.13, 22.14, 22.15, 22.16, 22.17.8, 22.17.9, 23.1, 23.9, 23.10, 23.11, 24, 25.1, 25.3, 25.5
B&C	Water treatment	1.14.9, 1.20.7, 2.3, 15.9, 17.3.4, 17.3.6.2, 18.1.9.1, 18.2.6, 23.8, 25.2
SRK	Tailings, waste design, pit slope design, hydrology, water management, hydrogeology	1.14.6, 1.14.7, 1.14.8, 1.16.3, 1.20.3, 1.20.4, 1.20.5, 1.20.6, 2.3, 2.4, 3.7, 3.8, 3.9, 3.10, 5.3.7, 7.2.4, 8.1.2, 8.2.5, 13.9, 13.10.1, 13.11, 15.8, 15.10.1, 15.10.2, 15.10.3, 15.10.4, 15.10.5, 15.10.6, 15.10.7, 15.11, 17.3.1, 17.3.2, 17.3.3, 17.3.5, 17.6, 22.17.3, 22.17.4, 22.17.5, 22.17.6, 22.17.7, 23.4, 23.5, 23.6, 23.7, 25.2, 25.4
Wood	Geology, mine design, mineral resource estimate, mineral reserve estimates	1.2.1, 1.5, 1.6, 1.7, 1.8, 1.10, 1.11, 1.12, 1.14.5, 1.20.1, 1.20.2, 2.3, 2.4, 2.5, 2.6, 3.3, 3.4, 3.5, 3.6, 3.10, 6, 7.1, 7.2.1, 7.2.2, 7.2.3, 7.2.5, 7.2.6, 7.2.7, 7.2.8, 8.1.1, 8.1.3, 8.1.4, 8.1.5, 8.1.6, 8.2.1, 8.2.2, 8.2.3, 8.2.4, 8.3, 9, 11, 12, 13.1, 13.2, 13.3, 13.4, 13.5, 13.6, 13.7, 13.8, 13.10, 17.1, 17.2, 18.1.6, 18.2.2, 22.2, 22.3, 22.4, 22.6, 22.7, 22.8, 22.17.1, 22.17.2, 23.2, 23.3, 25.2, 25.4, 25.5

2.4 Site Visits

Ausenco's process QP relied upon another experienced Ausenco engineer's visit to the Arctic Project site on July 25, 2017, during which the engineer inspected the property access, viewed the surface topography in the area proposed for the process plant and supporting infrastructure.

SRK's tailings QP visited the Arctic Project site from July 24-25, 2017, and July 10-12, 2018. He inspected property access and surface topography where the waste rock facility and tailings management facilities are to be located, as well as available space for other mine facilities.

SRK's geotechnical QP visited the Arctic Project site August 27-29, 2019. During the visit he reviewed selected drill core, and inspected the Arctic deposit discovery outcrop, 2019 drill pads at Arctic, and a talc outcrop.

Wood's geology and resource QP visited the Arctic Project site during August 29 to September 8, 2022. During the site visit, he observed the diamond drill core logging processes that include geology, structure and geotechnical logging, sampling, and specific gravity (SG) measurement process. He also visited Arctic site area, measured historical collar locations with a handheld global positioning system (GPS), reviewed representative drill cores, observed active drilling process. He visited drill core and pulp storage sites at camp and in Fairbanks office/warehouse.

Wood's mining QP visited the site on August 30, 2022. During the visit, Wood inspected the property access and viewed the surface topography in the areas proposed for the locations of the open pit, mine infrastructure and waste rock facility are to be located; inspected lithologies in selected drill cores that would support the pit walls; and observed talc outcrop and main foliation that could affect pit slope stability.

2.5 Currency of Report

This Report is current as of November 30, 2022.

2.6 Information Sources and References

Reports and documents listed in Section 2.7 and Section 24 were used to support the preparation of the Report. Additional information was sought from Trilogy where required.

Trilogy contributed to Sections 1.16, 2.3, 5, 17.1, 17.2, 17.3, 17.4, 18.1, 22.11 and 25 of this Report.

This is the first S-K 1300 Technical Report Summary filed by Trilogy on the Arctic Project.

2.7 Previous Reports

Previous reports publicly filed by NovaGold and NovaCopper/Trilogy and available on SEDAR (www.sedar.com) include:

- Staples, P., Davis, B., MacDonald AJ., Austin, J., Sim, R., Boese, C., Murphy, B., and Sharp, T., Peralta Romero, A., 2020: Arctic Project, Northwest Alaska, USA, NI 43-101 Technical Report on Feasibility Study, report prepared by Ausenco Engineering Canada Inc. for Trilogy Metals Inc., effective date August 20, 2020.
- Staples, P., Hannon, J., Peralta Romero, A., Davis, B., DiMarchi, J., Austin, J., Sim, R., Boese, C., Murphy, B., and Sharp, T., 2018: Arctic Project, Northwest Alaska, USA, NI 43-101 Technical Report on Pre-Feasibility Study, report prepared by Ausenco Engineering Canada Inc. for Trilogy Metals Inc., effective date February 20, 2018.
- Davis, B., Sim, R., and Austin, J., 2017: NI 43-101 Technical Report on the Arctic Project, Northwest Alaska, USA: report prepared by BD Resource Consulting, Inc., SIM Geological Inc., and International Metallurgical & Environmental Inc. for Trilogy Metals Inc., effective date April 25, 2017.
- Wilkins, G., Stoyko, H.W., Ghaffari, H., DiMarchi, J., Huang, J., Silva, M., O'Brien, M.F., Chin, M., and Hafez, S.A., 2013: Preliminary Economic Assessment Report on the Arctic Project, Ambler Mining District, Northwest Alaska: report prepared by Tetra Tech for NovaCopper inc., effective date September 12, 2013.
- Rigby, N., White, R., Volk, J., Braun, T., and Olin, E.J., 2012: NI 43-101 Preliminary Economic Assessment Ambler Project Kobuk, AK: report prepared by SRK Consulting (US) Inc. for NovaCopper inc., effective date February 1, 2012.
- Rigby, N., and White, R., 2011: NI 43-101 Preliminary Economic Assessment Ambler Project Kobuk, AK: report prepared by SRK Consulting (US) Inc. for NovaGold Resources Inc., effective date May 9, 2011.
- Rigby, N., and White, R., 2008: NI 43-101 Technical Report on Resources Ambler Project Arctic Deposit, Alaska: report prepared by SRK Consulting (US) Inc. for NovaGold Resources Inc., effective date January 31, 2008.

2.8 Abbreviations and Acronyms

A list of unit abbreviations is provided in Table 2-2 and an acronyms and abbreviations in Table 2-3.

Table 2-2: Unit Abbreviations

Abbreviation	Definition	Abbreviation	Definition
%	Percent	km/h	Kilometres per hour
‘	Minute (plane angle)	km^2	Square kilometer
“	Second (plane angle)	kPa	Kilopascal
<	Less than	kt	Thousand tonnes
>	Greater than	kV	Kilovolt
°	Degree	kW	Kilowatt
°C	Degrees Celsius	kWh	Kilowatt hour
°F	Degrees Fahrenheit	kWh/a	Kilowatt hours per year (annum)
µm	Microns	kWh/t	Kilowatt hours per tonne (metric ton)
A	Ampere	L	Litres
a	Annum (year)	L/m	Litres per minute
ac	Acre	lb	Pounds
B	Billion	lb/ton	Pounds per ton
cfm	Cubic feet per minute	m	Metres
cm	Centimetre	M	Million
cm^2	Square centimetre	m^2	Square metre
cm^3	Cubic centimetre	m^3	Cubic metre
d	Day	masl	Metres above sea level
d/a	Days per year (annum)	mg	Milligram
d/wk	Days per week	mg/l	Milligrams per litre
ft	Feet	mi	Mile
ft^2	Square foot	min	Minute (time)
ft^3	Cubic foot	mL	Millilitre
ft^3/s	Cubic feet per second	mm	Millimetre
g	Gram	mo	Month
g/cm^3	Grams per cubic centimetre	Mt	Million tonnes
g/L	Grams per litre	MW	Megawatts
g/t	Grams per tonne	MWh	Megawatt hour
GPM	US Gallons per minute	oz	Ounce
h	Hour	ppb	Parts per billion
h/a	Hours per year (annum)	ppm	Parts per million
h/d	Hours per day	psi	Pounds per square inch
h/w	Hours per week	rpm	Revolutions per minute
ha	Hectare (10,000 m^2)	s	Second (time)
hp	Horsepower	t	Tonnes (metric - 1,000 kg)
in	Inch	ton	Tons (imperial – 2,000 lb)
in^2	Square inch	t/a	tonnes per year (annum)
in^3	Cubic inch	USG	US Gallons

Abbreviation	Definition	Abbreviation	Definition
k	Kilo (thousand)	V	Volt
kg	Kilogram	wk	Week
kg/h	Kilograms per hour	y	Year (annum)
kg/m²	Kilograms per square metre		
km	Kilometre		

Table 2-3: Acronyms and Abbreviations

Acronym/Abbreviation	Definition
AA	Atomic Absorption
AACE	Association for the Advancement of Cost Engineering
AAP	Ambler Access Project
AAS	Atomic Adsorption Spectroscopy
ABA	Acid Base Accounting
ADEC	Alaska Department of Environmental Conservation
ADNR	Alaska Department of Natural Resources
AES	Atomic Emission Spectroscopy
AFD	Approved For Design
AHEA	Annual Hardrock Exploration Activity
AIDEA	Alaska Industrial Development and Export Authority
AMA	Arctic Mine Access
AMDIAP	Ambler Mining District Industrial Access Project
AMJ	Arctic Mine Junction
AMR	Aphanitic Metarhyolite
AMLT	Alaska Mining License Tax
ANCSA	Alaska Native Claims Settlement Act
ANFO	Ammonium Nitrate Fuel Oil
ANILCA	Alaska National Interest Land Conservation Act
AP	Acid Potential
APDES	Alaska Pollutant Discharge Elimination System
ARD	Acid Rock Drainage
AST	Alaska State Income Tax
BCMC	Bear Creek Mining Company
BFA	Bench Face Angle
BMAL	Bulk Mineral Analysis with Liberation
BV	Bureau Veritas
BWi	Ball Mill Work Index

Acronym/Abbreviation	Definition
CC	Construction Camp
ChS	Chlorite Schist
ChTS	Chlorite Talc Schist
CSAMT	Controlled Source Audio Magneto Telluric
CuEq	Copper Equivalent
CWA	Clean water act.
DCF	Discounted Cash Flow
DDH	Diamond Drill Hole
DGPS	Differential GPS
DTY	Dalton Transfer Yard
DWi	Drop Weight Index
EGL	Effective Grinding Length
EIS	Environmental Impact Assessment
EM	Electromagnetic
EPCM	Engineering, Procurement, and Construction Management
FA	Fire Assay
FEL	Front End Loaders (Loading)
FOS	Factors of Safety
FS	Feasibility Study
GISTM	Global Industry Standard on Tailings Management
GOH	Gross Operating Hours
GPS	Global Positioning System
GS	Grey Schist
HA	Heavy ANFO
HCTs	Humidity Cell Tests
HDPE	High-density Polyethylene
HR	Human Resources
HSEC	Health, Safety, Environment and Community
ICP	Inductively Coupled Plasma
ICP-MS	Inductively Coupled Plasma Mass Spectroscopy
IP	Induced Polarization
IPR	Instantaneous Penetration Rates
IRA	Inter Ramp Angle
IRR	Internal Rate of Return
JV	Joint Venture
KRC	Kennecott Research Centre

Acronym/Abbreviation	Definition
LBMA	London Bullion Market Association
LiDAR	Light Detection and Ranging Survey
LME	London Metal Exchange
LOM	Life of Mine
MAR	Mean Annual Runoff
MIBC	Methyl Isobutyl Carbinol
ML	Metal Leaching
MLUP	Miscellaneous Land Use Permit
MMU	Mobile Mixing (Manufacturing) Unit
MO	Metso Outotec
MS	Massive Sulphide
NAD	North American Datum
NEPA	natural environmental policy act
NI	National Instrument
NN	Nearest Neighbour
NOH	Net Operating Hours
NP	Neutralization Potential
NPV	Net Present Value
NSR	Net Smelter Return
NWAB	Northwest Arctic Borough
NYSE	New York Stock Exchange
OR	Operational Readiness
PAF	Permanent Accommodations Facility
PAG	Potentially Acid Generating
PE	Professional Engineer
PEA	Preliminary Economic Assessment
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
PGA	Peak Ground Acceleration
PIMA	Portable Infrared Mineral Analyzer
PMF	Probable Maximum Flood
PoF	Potential of Failure
PP	Process Plant
PWTP	Potable Water Treatment Plant
QA	Quality Assurance
QC	Quality Control

Acronym/Abbreviation	Definition
QPs	Qualified Persons
RAA	Resource Associates of Alaska
RF	Revenue Factor
RFQ	Request for Quotation
RMR	Rock Mass Rating
RO	Reverse Osmosis
ROD	record of decision
ROM	Run of Mine
RQD	Rock Quality Designation
RTDs	Rubber Tire Dozers
S-K 1300	Subpart 229.1300 – Disclosure by Registrants Engaged in Mining Operations in Regulations
SAB	SAG/Ball
SAG	Semi Autogenous Grind
SG	Specific Gravity
SMC	SAG Mill Comminution (?)
SMS	Semi Massive Sulphide
SPI	Significant Potential Incidents
ST	Sodium Tungsten
STP	Sewage Treatment Plant
TAC	Transportation Association of Canada
TCJA	Tax Cuts & Jobs Act
TDEM	Time Domain Electromagnetic
THH	Top Head Hammer
THPVC	Thompson Howarth Precision Versus Concentration
TIC	Total Inorganic Carbon
TMF	Tailings Management Facilities
ToC	Time of Concentration
TS	Talc Schist
TSX	Toronto Stock Exchange
UCS	Unconfined Compressive Strength
UKMP	Upper Kobuk Mineral
USACE	US Army Corps of Engineers
USFW	US Fish and Wildlife
USGS	US Geological Survey
UTM	Universal Transverse Mercator
VAT	Value Added Tax

Acronym/Abbreviation	Definition
VHF	Very High Frequency
VMS	Volcanogenic Massive Sulphide
VWPs	Vibrating Wire Piezometers
WGM	Watts, Griffis and McOuat
WQS	Waste Rock Collection Pond
WQBEL	Water Quality-Based Effluent Limits
WRCP	Waste Rock Contact Pond
WRF	Waste Rock Facility
WTP	Water Treatment Plant

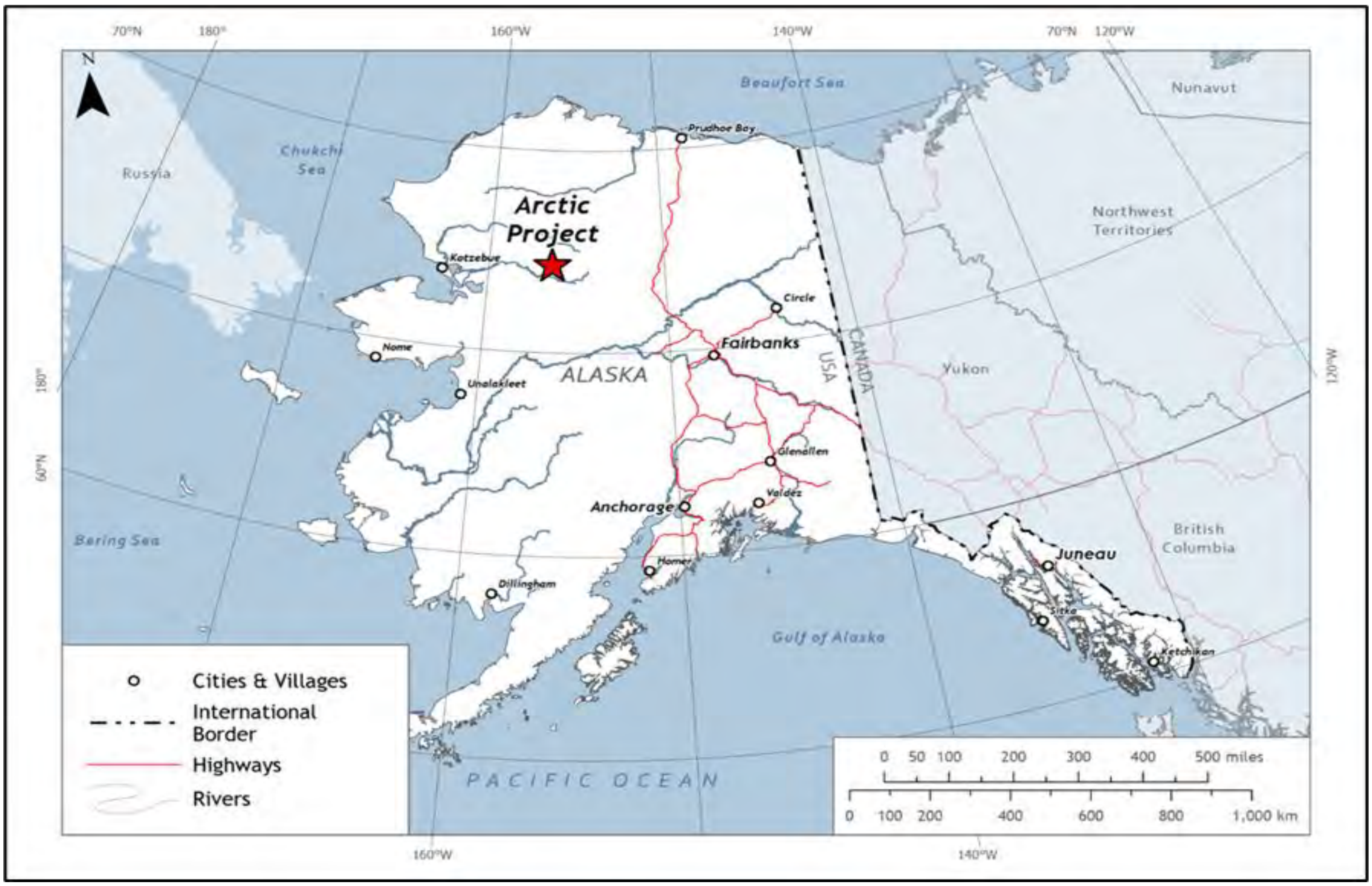
3 PROPERTY DESCRIPTION AND LOCATION

3.1 Introduction

The Property is situated in the Ambler mining district of the southern Brooks Range, in the Northwest Arctic Borough (NWAB) of Alaska (Figure 3-1). The Property is located in Ambler River A-2 quadrangle, Kateel River Meridian T 20N, R 11E, section 2 and T 21N, R 11E, sections 34 and 35.

The Property is about 270 km east of the town of Kotzebue, 37 km northeast of the village of Kobuk, and 260 km west of the Dalton Highway, an all-weather State maintained public road, at geographic coordinates N67.17° latitude and W156.39° longitude and Universal Transverse Mercator (UTM) North American Datum (NAD) 83, Zone 4 coordinates 7453080N, 613110E.

Figure 3-1: Arctic Project Location Map



Source: Ambler Metals, 2022.

3.2 Property Ownership

The Property is directly held by Ambler Metals, a 50/50 joint venture formed between South32 and Trilogy in February 2020. Upon the formation of the joint venture, Trilogy contributed all of its Alaskan assets, including the Property to Ambler Metals in exchange for a 50% ownership interest and at the same time, South32 contributed $145 million in cash for a 50% ownership interest.

Prior to the joint venture formation, the Property was held 100% by a wholly owned subsidiary of Trilogy. Trilogy acquired the Property from NovaGold in 2011. In 2011, NovaGold transferred all copper projects to NovaCopper and subsequently spun-out NovaCopper to its then existing shareholders by way of a Plan of Arrangement in 2012. NovaCopper Inc. subsequently underwent a name change to Trilogy Metals Inc. in 2016.

3.3 Mineral Tenure

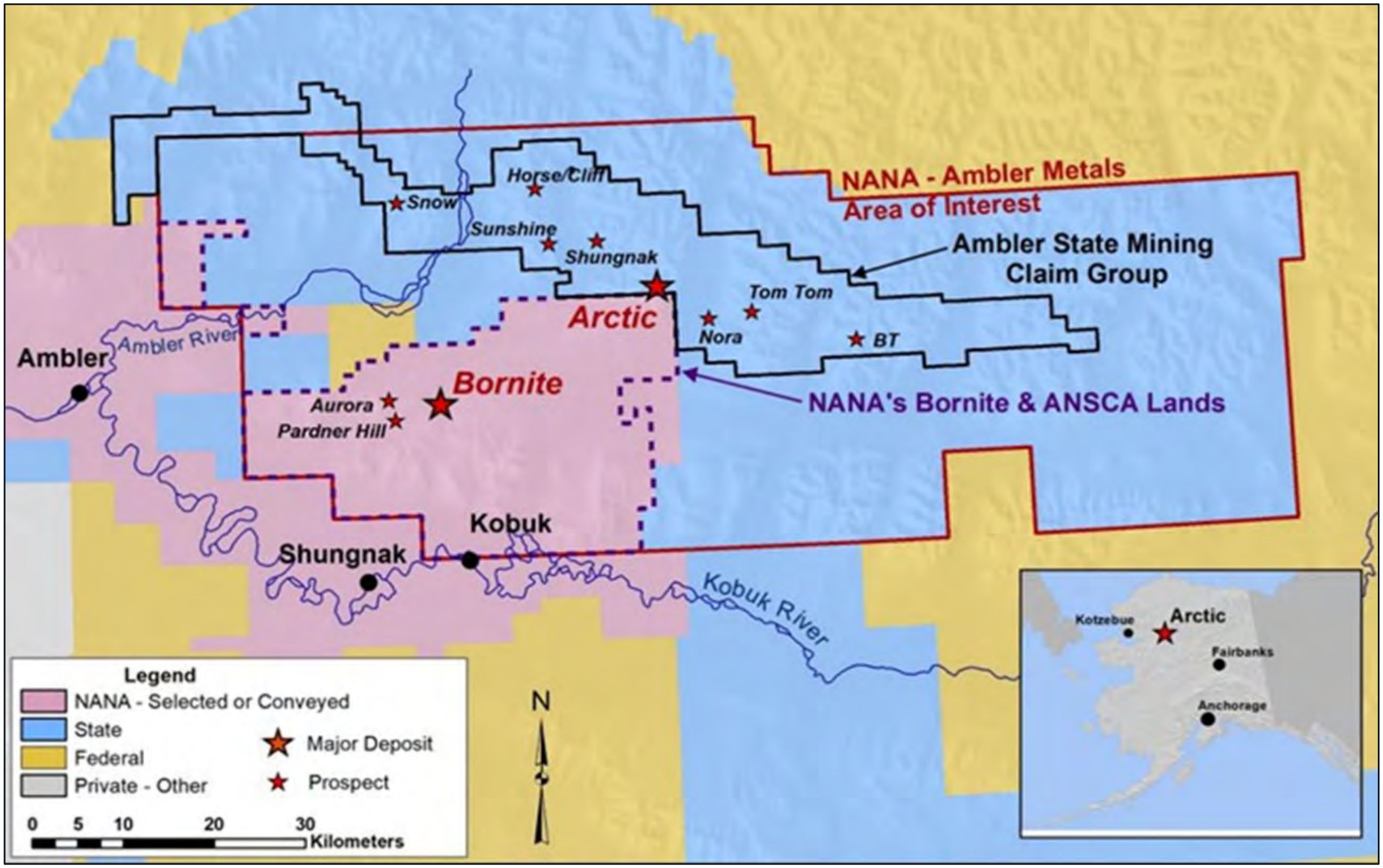
The UKMP consists of an approximately 448,217-acre land package containing state, patented and native lands within an area of interest. There are two discrete mineralized belts within the UKMP – the Devonian Ambler Schist Belt and the Devonian Bornite Carbonate Sequence. The Property is located within the Ambler Schist Belt which comprises approximately 231,008 acres (93,446 ha) of State of Alaska mining claims and US Federal patented mining claims in the Kotzebue Recording District. Exclusive of native lands, the UKMP land tenure consists of 2,136 contiguous State claims totalling 230,736 acres (93,336 ha), including 905 40-acre claims, 1,231 160-acre claims, and 18 Federal patented claims comprising 271.9 acres (110 ha) held in the name of Ambler Metals. Claim locations are shown in Figure 3-2 to Figure 3-4 and listed in Appendix A. The Arctic deposit is located near the southern edge of the centre of the claim block shown in Figure 3-5, primarily within the Federal patented claims, which do not expire and provides exclusive rights to the locatable minerals, and in most cases, the surface and all resources.

The Federal patented claim corners were located by the US Geological Survey (USGS). In Appendix A, a total of 18 Federal patented claims are reported using the completed mineral survey MS 2245 which includes Federal patent number 50-81-0127 covering16 Federal patented claims (Arctic 1, 2, 4, 9, 11, 13, 15, 17, 19, 23, 24, 25, 26, 27, 28, and 29), and Federal patent number 50-83-0174 covering 2 Federal patented claims (Arctic 10 and 495). Figure 3 4 included the locations of the Federal patented claims. There is no expiration date or labour requirement on the Federal patented claims.

The UKMP also consists of lands owned by NANA Regional Corporation, Inc. (NANA), who controls lands granted under the Alaska Native Claims Settlement Act (ANCSA) to the south of the Property boundary. Ambler Metals and NANA are parties to an agreement dated October 19, 2011 (the NANA Agreement) that consolidates the parties' land holdings into an approximately 190,929 ha land package and provides a framework for the exploration and development of the area, including non-exclusive rights for surface access across NANA lands. The NANA Agreement has a term of 20 years, with an option in favour of Ambler Metals to extend the term for an additional 10 years.

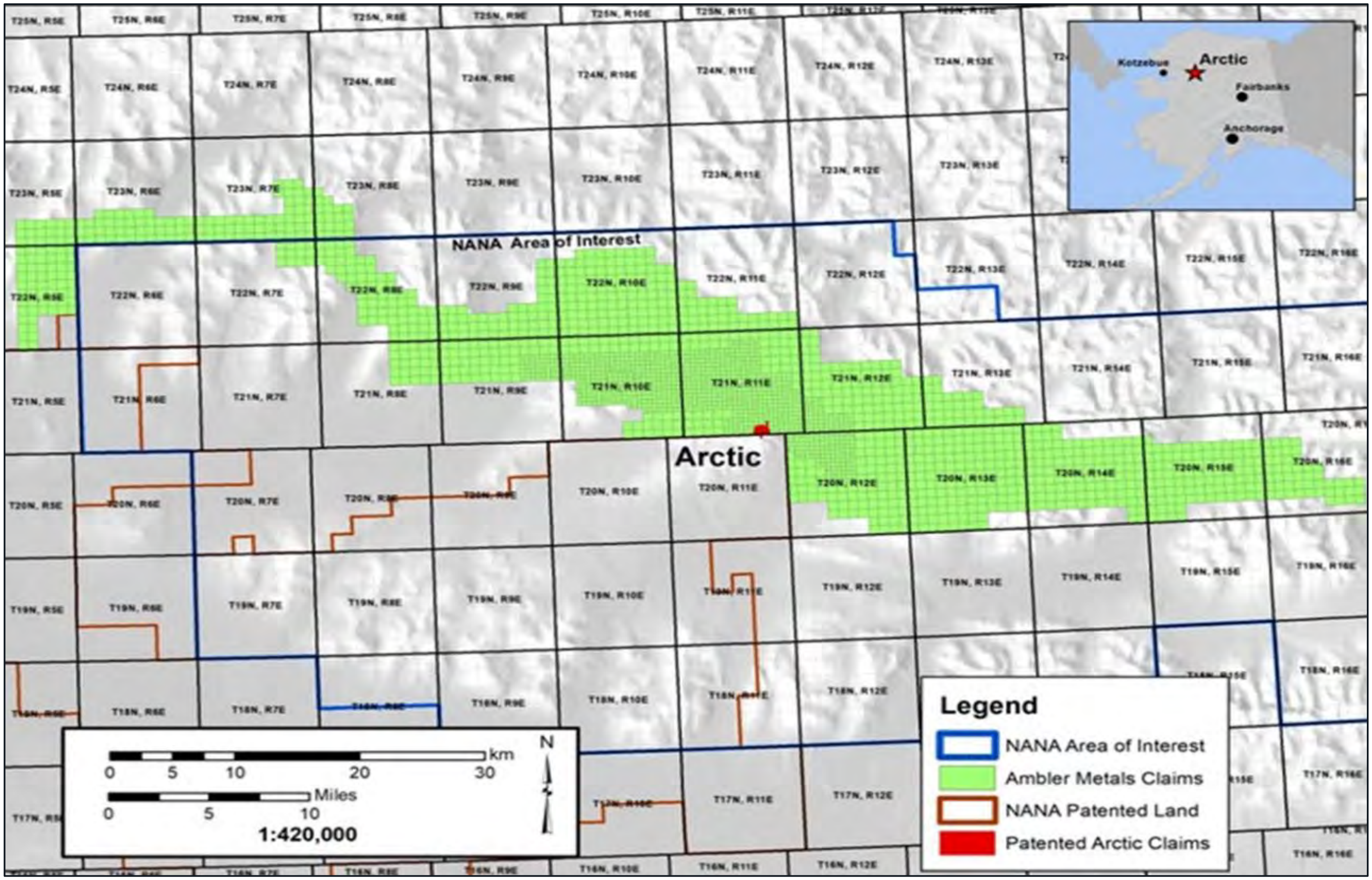
Rent for each State claim is up to date and is paid annually to the ADNR. An Annual Labour Statement must be submitted to maintain the State claims in good standing.

Figure 3-2: Upper Kobuk Mineral Projects Lands



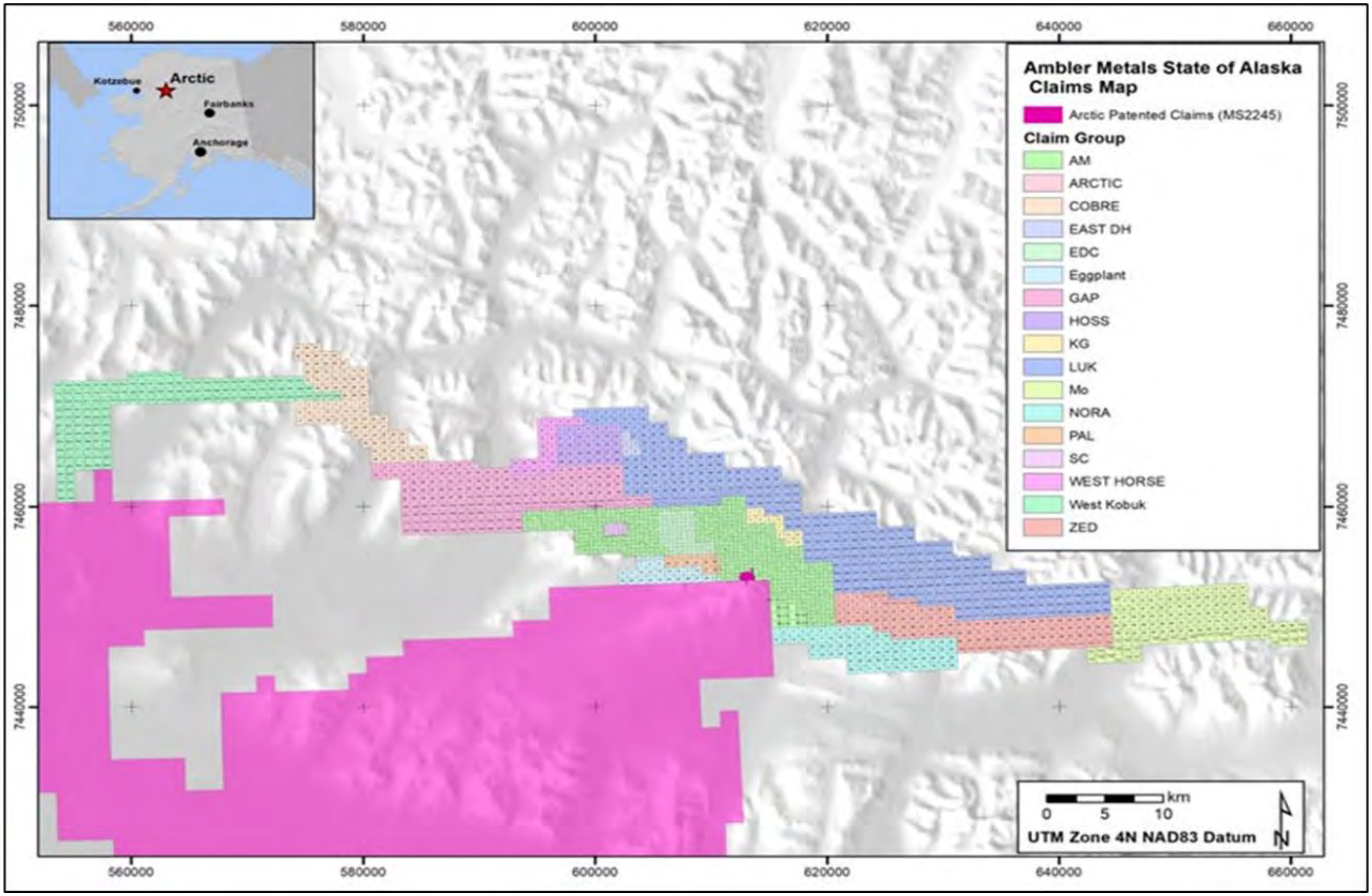
Source: Ambler Metals, 2022.
Note: NANA's Bornite and ANCSA Lands are not part of the Arctic Property

Figure 3-3: Arctic Project Mineral Tenure Plan



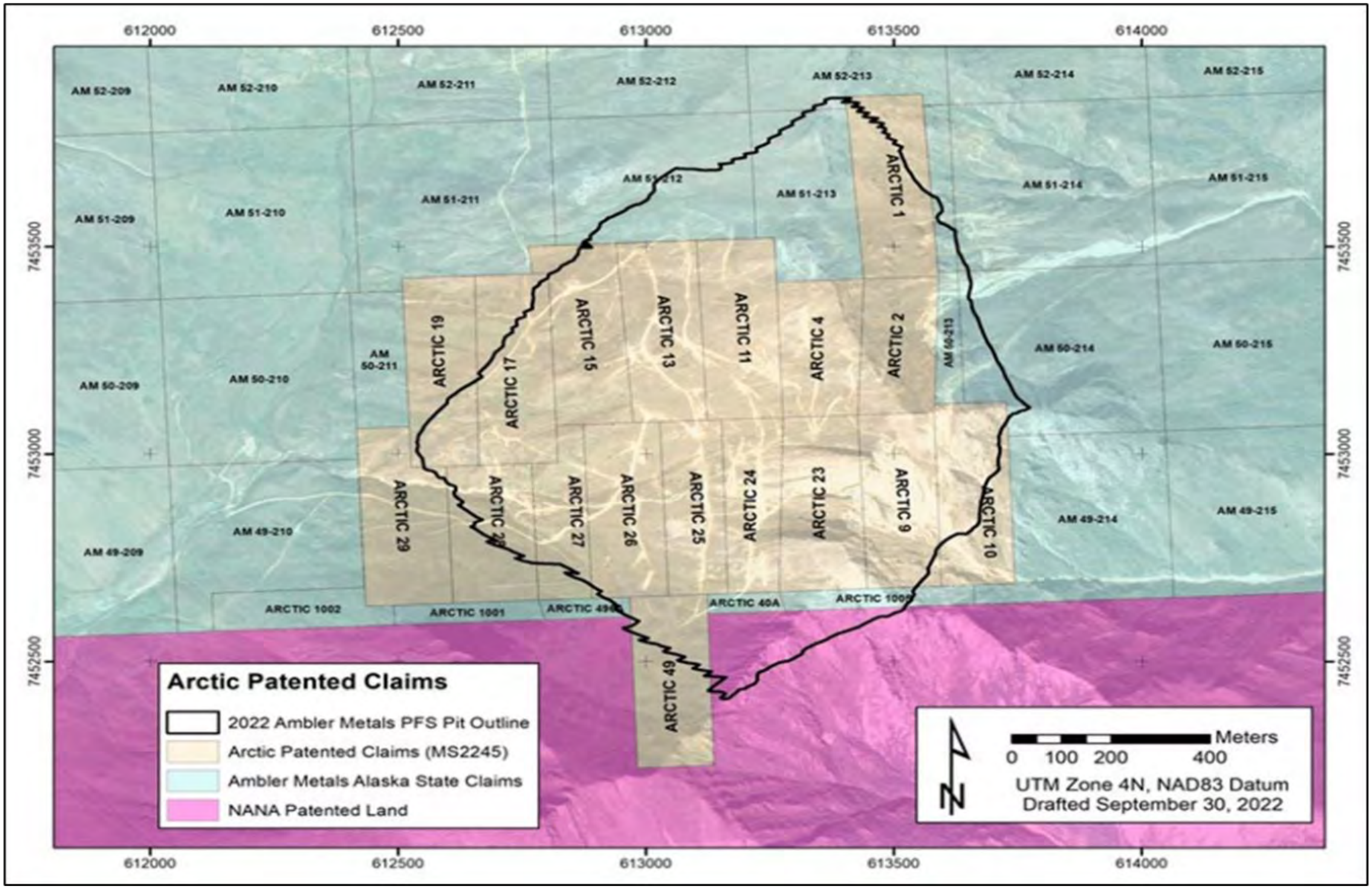
Source: Ambler Metals, 2022.

Figure 3-4: Mineral Tenure Layout Plan



Source: Ambler Metals, 2022.

Figure 3-5: Arctic Deposit Location



Source: Ambler Metals, 2022.

3.4 Royalties, Agreements and Encumbrances

3.4.1 Kennecott Agreements

The Kennecott Royalty Agreement dated effective January 7, 2010, was entered into by and among Kennecott, Alaska Gold Company and NovaGold. A copy of the Kennecott Agreement was recorded in the Kotzebue Recording District on January 8, 2010, as document no. 2010-000013-0.

The Kennecott Royalty Agreement documents a net smelter returns royalty that was reserved to Kennecott in a Purchase and Sale Agreement dated December 18, 2009, whereby Alaska Gold Company and NovaGold acquired mining properties from Kennecott. The mining properties referenced in the Kennecott Royalty Agreement consist of the Federal patented mining claims and many, but not all, of the State mining claims that are the subject of this Report.

Under the Kennecott Purchase and Termination Agreement, Kennecott retained a 1% NSR royalty that has been subsequently sold by Kennecott. The 1% NSR runs with the lands and is purchasable at any time from the royalty holder for a one-time payment of $10 million.

3.4.2 NANA Agreement

In 1971, the US Congress passed the ANCSA which settled land and financial claims made by the Alaska Natives and provided for the establishment of 13 regional corporations to administer those claims. These 13 corporations are known as the Alaska Native Regional Corporations (ANCSA Corporations). One of these 13 regional corporations is NANA. ANCSA Lands controlled by NANA bound the southern border of the Arctic Project claim block (refer to Figure 3-5).

On October 19, 2011, Trilogy and NANA entered into the NANA Agreement for the cooperative development of their respective resource interests in the Ambler mining district. The NANA Agreement consolidates Trilogy's and NANA's land holdings into an approximately 142,831 ha land package and provides a framework for the exploration and development of the area. The NANA Agreement provides that NANA will grant Trilogy the nonexclusive right to enter on, and the exclusive right to explore, the Bornite Lands and the ANCSA Lands (each as defined in the NANA Agreement) and in connection therewith, to construct and use temporary access roads, camps, airstrips, and other incidental works.

The NANA Agreement has been assigned by Trilogy to Ambler Metals upon the formation of the joint venture.

The NANA Agreement has a term of 20 years, with an option in favour of Ambler Metals to extend the term for an additional 10 years. The NANA Agreement may be terminated by mutual agreement of the parties or by NANA if Ambler Metals does not meet certain expenditure requirements on NANA's lands.

If, following receipt of a feasibility study and the release for public comment of a related draft EIS, Ambler Metals decides to proceed with construction of a mine on the lands subject to the NANA Agreement, Ambler Metals will notify NANA in writing and NANA will have 120 days to elect to either (a) exercise a non-transferrable back-in-right to acquire between 16% and 25% (as specified by NANA) of that specific project; or (b) not exercise its back-in-right, and instead receive a net proceeds royalty equal to 15% of the net proceeds realized by Ambler Metals from such project. The cost to exercise such back-in-right is equal to the percentage interest in the Arctic Project multiplied by the difference between (i) all costs incurred by Ambler Metals or its affiliates on the project, including historical costs incurred prior to the date of the NANA Agreement together with interest on the historical costs; and (ii) $40 million (subject to exceptions). This amount will be payable by NANA to Ambler Metals in cash at the time the parties enter into a joint venture agreement and in no event will the amount be less than zero.

In the event that NANA elects to exercise its back-in-right, the parties will, as soon as reasonably practicable, form a joint venture with NANA electing to participate between 16% to 25%, and Ambler Metals owning the balance of the interest in the joint venture. Upon formation of the joint venture, the joint venture will assume all of the obligations of Ambler Metals and be entitled to all the benefits of Ambler Metals under the NANA Agreement in connection with the mine to be developed and the related lands. A party's failure to pay its proportionate share of costs in connection with the joint venture will result in dilution of its interest. Each party will have a right of first refusal over any proposed transfer of the other party's interest in the joint venture other than to an affiliate or for the purposes of granting security. A transfer by either party of a net smelter royalty return on the project or any net proceeds royalty interest in a project other than for financing purposes will also be subject to a first right of refusal.

In connection with possible development on the Bornite Lands or ANCSA Lands, Ambler Metals and NANA will execute a mining lease to allow Ambler Metals or a joint venture vehicle to construct and operate a mine on the Bornite Lands or ANCSA Lands. The mining lease will provide NANA with a 2% NSR as to production from the Bornite Lands and a 2.5% NSR as to production from the ANCSA Lands.

If Ambler Metals decides to proceed with construction of a mine on its own lands subject to the NANA Agreement, NANA will enter into a surface use agreement with Ambler Metals which will afford Ambler Metals access to the Arctic Project along routes approved by NANA (the Surface Use Agreement). In consideration for the grant of such surface use rights, Ambler Metals will grant NANA a 1% NSR on production and an annual payment of $755 per acre (as adjusted for inflation each year beginning with the second anniversary of the effective date of the NANA Agreement) and for each of the first 400 acres and $100 for each additional acre, of the lands owned by NANA and used for access which are disturbed and not reclaimed.

Figure 3-5 showed the locations of the Bornite and ANCSA Lands that are included in the NANA Agreement. The Bornite Lands are not considered to be part of the Arctic Project, because the mineralization styles identified to date in the Bornite Lands are distinctly different to the mineralization styles in the Ambler claims, and it is expected that any mining operation in the Bornite Lands would be developed as a stand-alone operation using different infrastructure.

3.5 State Royalty

The owner of a State mining claim or lease will be obligated to pay a production royalty to the State of Alaska in the amount of 3% of net income received from minerals produced from the State mining claims.

This royalty does not apply to patented federal mining claims.

3.6 Surface Rights

Surface use of the private land held as Federal patented claims is limited only by reservations in the patents and by generally applicable environmental laws. These do not restrict access or the ability to do work.

Surface use of State claims allows the owner of the mining claim to make such use of the surface as is "necessary for prospecting for, extraction of, or basic processing of minerals."

3.7 Environmental Considerations

Environmental considerations are discussed in Section 17.

There may be some environmental liabilities associated with sites explored during the 1950s and 1960s. The exploration camp would require rehabilitation if the Arctic Project is closed.

3.8 Permits

Permitting considerations for the Arctic Project are discussed in Section 17. Permits necessary to advance to the next stage are also discussed in Section 17.

There have been no significant violations or fines on the Property.

3.9 Social Considerations

Social considerations for the Arctic Project are discussed in Section 17.

3.10 Comment on Property Description and Location

The QPs consider that there are no other significant factors or material risks that may affect access, mineral tenure, title or the right or ability to perform work on the Property other than what is described in this Report. All mineral tenure, mining leases and crown land title are in good standing. Surface and aerial access to the project site is permitted and well-established. Permits to authorize work program activities are in place and applied for sufficiently in advance of work requirements. It is a reasonable expectation that any additional surface rights that would be required to support Arctic Project development and operations can be obtained through appropriate negotiation.

4 ACESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Accessibility

4.1.1 Air

Primary access to the Property is by air, using both fixed wing aircraft and helicopters.

There are four well-maintained, approximately 1,500 m-long gravel airstrips located near the Arctic Project, capable of accommodating charter fixed wing aircraft. These airstrips are located 64 km west at Ambler, 46 km southwest at Shungnak, 37 km southwest at Kobuk, and 34 km southwest at Dahl Creek. There is daily commercial air service from Kotzebue to the village of Kobuk, the closest community to the Arctic Project. During the summer months, the Dahl Creek Camp airstrip is suitable for larger aircraft, such as a C-130 and DC-6.

In addition to the four 1,500 m airstrips, there is a 700 m airstrip located at the Bornite Camp. The airstrip at Bornite is suited to smaller aircraft, which support the Bornite Camp with personnel and supplies. There is also a 450 m airstrip (Arctic airstrip) located at the base of Arctic Ridge that can support smaller aircraft.

4.1.2 Road

A winter trail and a one-lane dirt track suitable for high-clearance vehicles or construction equipment links the Bornite Camp to the Dahl Creek airstrip southwest of the Arctic deposit. An unimproved gravel track connects the Arctic airstrip with the Arctic deposit.

4.1.3 Water

There is no direct water access to the Property. During spring runoff, river access is possible by barge from Kotzebue Sound to Ambler, Shungnak, and Kobuk via the Kobuk River.

4.2 Climate

The climate in the region is typical of a sub-arctic environment. Weather conditions on the Property can vary significantly from year to year and can change suddenly. During the summer exploration season, average maximum temperatures range from 10°C to 20°C, while average lows range from -2°C to 7°C (Western Regional Climate Center: WRCC – Alaska Climate Summaries: Kobuk 1971 to 2000). By early October, unpredictable weather limits safe helicopter travel to the Arctic Project. During winter months, the Arctic Project can be accessed by snow machine, track vehicle, or fixed wing aircraft. Winter temperatures are routinely below -25°C and can exceed -50°C. Annual total precipitation (rainfall and snowfall) in the region averages 1,219 mm with the most rainfall occurring from July through September, and the most snowfall occurring from October through February.

It is expected that any future mining activity will be conducted on a year-round basis. Past exploration activities are generally confined to the period from late May to late September.

4.3 Local Resources and Infrastructure

The Property is currently isolated from major public infrastructure. Infrastructure assumptions including sources of water and electricity, and the proposed infrastructure layout for the Arctic Project are discussed in Section 15 of the Report.

The Property is approximately 270 km east of the town of Kotzebue, on the edge of Kotzebue Sound, 37 km northeast of the village of Kobuk, 260 km west of the Dalton Highway, and 470 km northwest of Fairbanks. Kobuk (population 191; 2020 US Census) is the location of one of the airstrips near the Arctic Project. Several other villages are also near the Arctic Project, including Shungnak located 46 km to the southwest with a population of 272 (2020 US Census) and Ambler, 64 km to the west with a population of 274 (2020 US Census). Kotzebue has a population of 3,102 (2020 US Census) and is the largest population centre in the Northwest Arctic Borough. Kotzebue is a potential source of limited mining-related supplies and labourers, and is the nearest centre serviced by regularly scheduled, large commercial aircraft (via Nome or Anchorage). In addition, there are seven other villages in the region that will be a potential source of some of the workforce for the Arctic Project. Fairbanks (population 32,515; 2020 US Census) has a long mining history and can provide most mining-related supplies and support that cannot be sourced closer to the Property.

Drilling and mapping programs are seasonal and have been supported out of the Bornite Camp and Dahl Creek Camp. The Bornite Camp facilities are located on Ruby Creek on the northern edge of the Cosmos Hills. The camp provides office space and accommodations for the geologists, drillers, pilots, and support staff. Power is supplied by two diesel generators. Water was supplied by the permitted artesian well located 250 m from camp; however, a water well was drilled in camp during the 2017 field season that was permitted by 2019 to provide all potable water for the Bornite Camp.

There is sufficient area within the Property to host an open pit mining operation, including mine and plant infrastructure, and waste rock and tailings management facilities.

4.4 Physiography

The Property is located along the south slope of the Brooks Range, which separates the Arctic region from the interior of Alaska. Nearby surface water includes Subarctic Creek, the Shungnak and Kogoluktuk Rivers, the Kobuk River, and numerous small lakes. The Property is located at the eastern end of Subarctic Creek, a tributary of the Shungnak River to the west, along a ridge between Subarctic Creek and the Kogoluktuk River Valley. The Property area is marked by steep and rugged terrain with high topographic relief. Elevations range from 30 masl along the Kobuk River to 1,180 masl on a peak immediately north of the Arctic Project area. The divide between the Shungnak and Kogoluktuk Rivers in the Ambler Lowlands is approximately 220 masl.

The Kobuk Valley is located at the transition between boreal forest and Arctic tundra. Spruce, birch, and poplar are found in portions of the valley, with a ground cover of lichens (reindeer moss). Willow and alder thickets and isolated cottonwoods follow drainages, and alpine tundra is found at higher elevations. Tussock tundra and low, heath-type vegetation covers most of the valley floor. Intermittent permafrost exists on the Property.

Permafrost is a layer of soil at variable depths beneath the surface where the temperature has been below freezing continuously from a few to several thousands of years (Climate of Alaska 2007). Permafrost exists where summer heating fails to penetrate to the base of the layer of frozen ground and occurs in most of the northern third of Alaska as well as in discontinuous or isolated patches in the central portion of the State.

Wildlife in the Arctic Project area is typical of Arctic and subarctic fauna (Kobuk Valley National Park 2007). Larger animals include caribou, moose, Dall sheep, bears (grizzly and black), wolves, wolverines, coyotes, lynx and foxes. There are no anadromous fish species in the upper reaches of the Shungnak and Kogoluktuk Rivers due to natural fish barriers.

Other fish species such as trout, sculpin, and grayling are common. The caribou seen on the Property belong to the Western Arctic herd that migrate once a year heading south in late August through October from their summer range north of the Brooks Range. The caribou migrate north in March from their winter range along the Buckland River to the north slope of the Brooks Range, a more westerly route and do not cross the Project during that migration.

4.5 Comments on Accessibility, Climate, Local Resources, Infrastructure and Physiography

In the opinion of the QP:

- The planned infrastructure, availability of staff, power, water, and communications facilities, the design and budget for such facilities, and the methods whereby goods could be transported to the proposed mine, and any planned modifications or supporting studies are reasonably well-established, or the requirements to establish such, are reasonably well understood by Trilogy, and can support the declaration of Mineral Resources and Mineral Reserves.
- There is sufficient area within the Property to host an open pit mining operation, including mine and plant infrastructure, and waste rock and tailings management facilities.
- It is expected that any future mining operations will be able to be conducted year-round.

5 HISTORY

5.1 Regional History

Prospectors in search of gold, travelling up the Kobuk River in 1898-99 (Grinnell, 1901), found several small gold placer deposits in the southern Cosmos Hills, south of the Arctic deposit, which were worked intermittently over the ensuing decades. Around this time, copper mineralization at Ruby Creek and Pardner Hill in the northern Cosmos Hills was explored using small shafts and adits (Smith and Eakin, 1911). In 1947, Rhinehart “Rhiny” Berg staked claims over the Ruby Creek prospects, carried out extensive trenching and the first diamond drilling, and constructed an airstrip for access (alaskamininghalloffame.org 2012). BCMC, an exploration subsidiary of Kennecott, optioned the Ruby Creek property from Berg in 1957. The prospect became known as Bornite and Kennecott conducted extensive exploration over the next decade, culminating in the discovery of the high-grade No. 1 zone and the sinking of an exploration shaft to conduct underground drilling.

While exploring the Bornite deposit, BCMC carried out reconnaissance exploration throughout the western Brooks Range, including a large regional stream sediment survey in 1962. Initial follow up did not identify mineralization of interest however in 1965, Riz Bigelow (BCMC) and his team of geologists found boulders of massive sulphides at an anomaly (1400 ppm Cu) located 28 km northeast of Bornite that led to the discovery of outcropping mineralization the following year. The area was subsequently staked and, in 1967, nine core holes were drilled at the Arctic deposit, eight of which yielded massive sulphide intercepts over an almost 500-m strike length.

BCMC conducted intensive exploration on the property until 1977 and then intermittently through to 1998. No drilling or additional exploration was conducted on the Property between 1999 and 2003.

In addition to drilling and exploration at the Arctic deposit, BCMC also conducted exploration at numerous other prospects in the Ambler Mining District (most notably Dead Creek, Sunshine, Cliff, and Horse). The abundance of Volcanogenic Massive Sulphide (VMS) prospects in the district resulted in a series of competing companies in the area, including Sunshine Mining Company, Anaconda Company, Noranda Exploration Company, GCO Minerals Company, Cominco American Resource Inc. (Cominco), Teck Cominco, Resource Associates of Alaska (RAA), Watts, Griffis and McOuat Ltd. (WGM), and Houston Oil and Minerals Company, culminating into a claim staking war in the district in 1973. Falconbridge and Union Carbide also conducted work later in the district.

District exploration by Sunshine Mining Company and Anaconda resulted in two additional significant discoveries in the district; the Sun deposit located 60 km east of the Arctic deposit, and the Smucker deposit located 36 km west of the Arctic deposit. These two deposits are outside the current Property.

District exploration continued until the early 1980s on the four larger deposits in the district (Arctic, Bornite, Smucker and Sun) when the district fell into a hiatus due to depressed metal prices.

In 1987, Cominco acquired the claims covering the Sun and Smucker deposits from Anaconda. Teck Resources Limited, as Cominco’s successor company, continues to hold the Smucker deposit. In 2007, Andover Mining Corporation purchased a 100% interest in the Sun deposit for $13 million and explored the property through 2013. The Sun deposit and adjacent lands were acquired by Valhalla Metals Inc., a private company, staked over the Sun deposit in 2017 after the creditors for the bankrupt Andover Mining Corporation failed to pay the annual rent of the state claims and submit the Annual Labour Statement.

In 1981 and 1983, Kennecott received three US Mineral Survey patents (MS2245 totaling 240 acres over the Arctic deposit – later amended to include another 32 acres; and MS2233 and MS2234 for 25 claims totaling 516.5 acres at Bornite). The Bornite patented claims and surface development were subsequently sold to NANA Regional Corporation, Inc. in 1986.

No production has occurred at the Arctic deposit or at any of the other deposits within the Ambler Mining District.

5.2 Prior Ownership and Ownership Changes – Arctic Deposit and the Ambler Lands

BCMC initially staked federal mining claims covering the Arctic deposit area beginning in 1966. The 1960's drill programs defined a significant high-grade polymetallic resource at the Arctic deposit and, in the early 1970s, Kennecott began the patent process to obtain complete legal title to the Arctic deposit. In 1981, Kennecott received US Mineral Survey patent M2245 covering 16 mining claims totalling 240.018 acres. In 1983, US Mineral Survey patent M2245 was amended to include two additional claims totalling 31.91 acres.

With the passage of the Alaska National Interest Lands Conservation Act (ANILCA) in 1980, which expedited native land claims outlined in the ANSCA and State lands claims under the Alaska Statehood Act, both the State of Alaska and NANA selected significant areas of land within the Ambler Mining District. State selections covered much of the Ambler schist belt, host to the VMS deposits including the Arctic deposit, while NANA selected significant portions of the Ambler Lowlands to the immediate south of the Arctic deposit as well as much of the Cosmos Hills including the area immediately around Bornite.

In 1995, Kennecott renewed exploration in the Ambler schist belt containing the Arctic deposit patented claims by staking an additional 48 state claims at Nora and 15 state claims at Sunshine Creek. In the fall of 1997, Kennecott staked 2,035 state claims in the belt consolidating their entire land position and acquiring the majority of the remaining prospective terrain in the VMS belt. Five more claims were subsequently added in 1998. After a short period of exploration which focused on geophysics and geochemistry combined with limited drilling, exploration work on the Arctic Project again entered a hiatus.

On March 22, 2004, Alaska Gold Company, a wholly-owned subsidiary of NovaGold completed an Exploration and Option Agreement with Kennecott to earn an interest in the Ambler land holdings.

5.3 Previous Exploration and Development Results – Arctic Deposit

5.3.1 Introduction

Kennecott's ownership of the Property saw two periods of intensive work from 1965 to 1985 and from 1993 to 1998, before optioning the property to NovaGold in 2004.

Though reports, memos, and files exist in Kennecott's Salt Lake City office, only limited digital compilation of the data exists for the earliest generation of exploration at the Arctic deposit and within the VMS belt. Beginning in 1993, Kennecott initiated a re-evaluation of the Arctic deposit and assembled a computer database of previous work at the Arctic deposit and in the district. A computer-generated block model was constructed in 1995 and an updated resource estimate was preformed using from block model. Subsequently, Kennecott staked a total of 2,035 State of Alaska claims in 1997 and, in 1998 undertook the first field program since 1985.

Due to the number of companies and the patchwork exploration that occurred as a result of the 1973 staking war, much of the earliest exploration work on the Ambler Schist belt was lost during the post-1980 hiatus in district exploration. The following subsections outline the best documented data at the Arctic deposit as summarized in the 1998 Kennecott exploration report, including the assembled computer database; however, this outline is not considered to be either exhaustive or in-depth.

In 1982, geologists with Kennecott, Anaconda and the State of Alaska published the definitive geologic map of the Ambler schist belt (Hitzman et al. 1982).

Table 5-1 lists known exploration mapping, geochemical, and geophysical programs conducted for VMS targets in the Ambler Mining District.

Table 5-1: Known Mapping, Geochemical, and Geophysical Programs Targeting VMS Prospects in the Ambler Mining District

Area	Prospects	Company	Mineralization	Mapping	Soil Geochem	Geophysics	Reports
Arctic Center of the Universe (COU) Back Door	Arctic	BCMC-KEX	Two (or more) sulphide bands with thickness up to ~40 m with Zn, Cu, Pb, Ag, Au, ±Ba mineralization.	Proffett 1998; Lindberg and others 2004, 2005; NG personnel 2008 at 1:2,000 scale	Extensive 2006 NG program (>670 samples)	Numerous surveys including the 1998 Dighem EM and Mag aerial surveys, 1998 CSAMT survey, TEM downhole and surface surveys in 2005, TDEM ground survey in 2006	Numerous
	COU Back Door, 4th of July Creek	NG-Anaconda	No exposed or drilled mineralization, target is the projection of the Arctic horizon	NG 1:2,000 mapping in 2006	Extensive 2006 NG program	4 TDEM ground surveys in 2005 and 2006	2005 and 2006 NG Progress Reports; Lindberg's 2005 report
Sunshine Bud CS	Sunshine Creek	BCMC and BCMC-Noranda	Disseminated to semi-massive lens up to 18 m thick. Upper mineralized limb is Ba-rich	BCMC 1983; Paul Lindberg 2006; NG 2011	Numerous eras of soil sampling, most recent 1998 by Kennecott (Have data) and 2006 by NG	BCMC completed Recon IP survey and Crone vertical shoot back EM in 1977, 2 TDEM surveys to the NW	Various BCMC reports; Lindberg's 2006 Sunshine progress report; 2006 NG Progress report
	Bud-CS	SMC and TAC	Au-rich gossan and 3+ m intercept of 1.7% Cu, 0.4% Pb, 1.5% Zn, 2 oz/ton Ag, 0.017 oz/ton Au	Anaconda (TA C) and Sunshine (SMC)	SMC soil sampling	Anaconda completed downhole resistivity survey in 1981 on Bud 7	1981 through 1983 Anaconda Progress reports

Area	Prospects	Company	Mineralization	Mapping	Soil Geochem	Geophysics	Reports
Dead Creek Shungnak SK	Shungnak (Dead Cree k)	BCMC, Cominco	Thin (0.1 to 3 m) disseminated to semi-massive lenses of Cu, Zn, Pb, Ag mineralization	Bruce Otto and others 2006; Proffett 1998	NG in 2006 (355 samples); KEX in 1998 (~240 sampl es)	2 CEM surveys by BCMC at DH with no anomalous responses (do not have data)	2006 NG report; 1982 and 1983 Anaconda Ambler Progress reports
	SK	GCO and BCMC/GCO-HOMEX JV	Mineralized float up to 0.4% Cu, 4.8% Pb, 8.7% Zn, 5 oz/ton Ag	BCMC	BCMC 1982 soil grid	CEM and Max-min completed by BCMC (do not have data)	1982 Annual Progress Report, BCMC; Bruce Otto 2006 Memo
Horse Cliff DH	Horse-Cliff DH	Horse – BCMC, Cliff SMC, DH – BCMC and BCMC/GCO-HOMEX	Disseminate to semi-massive with local massive lens, thicknesses up to tens of feet.	KEX 1983 1:1000 prospect map	SMC soil surveys 1976-1978 and 1980	No known ground-based survey; occurrences within a large resistivity high	1985 Progress Report BCMC-GCO-Homex J; 1980 Summary of Ambler Field Investigations – Sunshine Mining, Horse Creek Memo – Robinson 1981; 1978 Ellis Geologic Evaluation and Assessment of the Northern Belt Claims
Snow Ambler RB Nani Frost	Snow	Cominco	Ag-Pb-Zn mineralization as massive and semi-massive bands hosted within thin bands	Noranda-Cominco scanned map with no 72eochem72e nced; Prospect scale	KEX Soil gird in 1997 or 1998	No known ground-based survey; Anaconda completed downhole resistivity survey	“Snow Prospect Miscellaneous Notes and Maps.pdf” is only known report

Area	Prospects	Company	Mineralization	Mapping	Soil Geochem	Geophysics	Reports
			of graphitic schist (GS).			in 1981 on Ambler-4	
	Ambler	Anaconda TAC	Massive, disseminated chalcopyrite and pyrite associated with chert	Numerous Anaconda geologists; no digitized maps	Only scattered soils in database	Max-min surveys, no data is available	1983 Ambler River Memo (Sunshine Progress Report); 1982 Anaconda Progress Report
	Nani-Frost	BCMC and BCMC-Noranda	Outcrops of 2-3 m of 0.8% Cu, 0.4% Pb, 1.2% Zn, 0.05 oz/ton Ag within felsic schist	BCMC (do not have data)	BCMC identified numerous weak soil anomalies (do not have data)	CEM, Max-min, and PEM completed by BCMC (do not have data)	1982 Annual Progress Report, BCMC
Red Nora	Nora	BCMC/GCO-HOMEX	Disseminated chalcopyrite within chlorite altered volcanics in two zones (Sulphide Gulch and Northern Horizon)	Generalized geologic map created by WGM for BCMC-GCO-HOMEX	No known data	Two PEM over the Sulphide Gulch horizon	1984 and 1985 Progress Report BCMC-GCO-Homex JV
	Red	BCMC	Thin discordant bands of sphalerite, chalcopyrite, galena, and pyrrhotite with calcite and fluorite cutting	None	KEX soil lines 1998	KEX identified EM anomalies 1998, follow-up gravity and Max-min EM; TDEM survey in 2006; DIGHEM helicopter EM and	Kennecott’s final 1998 field report; 2006 NG Progress Report

Area	Prospects	Company	Mineralization	Mapping	Soil Geochem	Geophysics	Reports
			'siltites' and metacarbonates			radiometric survey in 2006	
Other	BT, Jerri Creek	Anaconda, AMC	Massive sulphide bands up to 1.5 m thick extend nearly 2.3 km along an E-W strike	Hitzman and others	Historic soils at Jerri Creek	No known surveys.	Hitzman thesis and Anaconda (BT) and Bear Creek (Jerri) Assessment reports; 1982 and 1983 Anaconda Ambler Progress reports
	Kogo-White Creek	Bud – SMC or AMC	Discovered by hydrochemistry of high Cu ions in White Creek.	SMC?	Soil 74eochem surveys by SMC in 1978 and KEX in 1998	Recon IP survey in 1977; Max-Min Mag survey in 1980; Follow-up Max-Min and gravity by KEX in 1998; TDEM by NG in 2006.	1980 Summary of Ambler Field Investigations, SMC; Kennecott's Final 1998 Field Report
	Pipe	BCMC and SMC	Podiform zones of sulphide mineralization within calc-schists and QMS	Schmidt in 1978, SMC in 1982	Kennecott soil grid in 1997-1998	Not known	Schmidt's 1978 report (Part IV) for Anaconda's (?) annual report

Area	Prospects	Company	Mineralization	Mapping	Soil Geochem	Geophysics	Reports
	Tom	Anaconda and SMC	1982 'Discovery' trench by SMC uncovered massive sulphide boulders with up to 6 oz/ton Ag, 5.4% Pb, 6.3% Zn, only 0.2% Cu	Sunshine in 1982 (?)	SMC soils in 1982	Gamma mag survey by SMC in 1982; TDEM by NG in 2006.	1982 Sunshine Mining Company Memo by E.R. Modroo; Schmidt's 1978 report (Part IV) for Anaconda's (?) annual report
Sun	Sun-Picnic Creek	Anaconda – AMC-Cominco; Valhalla Metals is current owner	Three zones of sulphide mineralization varying from 1 to 10 m; Upper zone is Zn-Pb-Ag rich while the two lower zones are Cu rich	Various Anaconda geologists	Not known, but most likely extensive	Not known, but most likely extensive	1981 Anaconda progress report; Anaconda 1977 prefeasibility study (not in NG possession)
Smucker	Smucker-Charlie-Puzzle-4B-Patti	Anaconda, Cominco, and Bear Creek; now owned by Teck	A single mineralized Ag-Zn-Pb-Cu horizon varying from 1 to 8 m in thickness	Detailed mapping by Anaconda and BCMC geologists	Strong soil 75eochem anomalies in lowlands SE of Smucker horizon; Kennecott soil grid in 1997 or 1998	Not known	1985 Progress Report BCMC-GCO-Homex JV

Note: EM = electromagnetic; TDEM = time domain electromagnetic; CSAMT = Controlled Source Audio Magnetotelluric.

5.3.2 Geochemistry

Historical geochemistry for the district, compiled in the 1998 Kennecott database, includes 2,255 soil samples, 922 stream silt samples, 363 rock samples, and 37 panned concentrate samples. Data have been sourced from several companies including Kennecott, Sunshine Mining, RAA, and NANA. Sourcing of much of the data had been poorly documented in the database.

During 1998, Kennecott renewed its effort in the district, and, as a follow-up to the 1998 electromagnetic (EM) survey, undertook soil and rock chip sampling in and around EM anomalies generated in the geophysical targeting effort. During this period Kennecott collected 962 soils and 107 rocks and for the first time used extensive multi-element inductively coupled plasma (ICP) analysis.

5.3.3 Geophysics

Prior to 1998, Kennecott conducted a series of geophysical surveys which are poorly documented or are unavailable to Ambler Metals. With the renewed interest in the belt, Kennecott carried out a helicopter-supported airborne EM (five frequency DIGHEM system) and magnetic survey covering approximately a 70 km stretch of the Ambler belt in March 1998. A total 2,509-line kilometres was flown using 400 m line spacing except at the Arctic deposit where 200 m line spacing was used. The Arctic deposit presented a strong 900 Hz EM conductive signature. From the initial interpretation of the airborne EM data a two-year ground follow-up program to evaluate and drill test the best targets was recommended (Fueg, 1998).

During the summer of 1998, Kennecott carried out a well-planned and systematic ground follow-up program. Of the 46 EM anomalies that were identified 17 were ranked A and B and were mapped and evaluated in the field: 12 using soil geochemistry, 9 using a combination of ground EM (Maxmin 2 horizontal loop) and gravity (LaCoste and Romberg Model G gravimeter), and two anomalies were drilled. Eight of the 17 anomalies occurred in prospective geology and coincident anomalous geochemistry and two were drilled in the same summer (Kennecott, 1998).

At anomaly 98-3, later named East Dead Creek, located approximately 6 km northwest of the Arctic deposit and 2 km east-northeast of the Dead Creek prospect, Kennecott found sub-cropping gossan roughly coincident with the centre of their airborne and ground EM anomalies. A single vertical hole encountered semi-massive sulphides in three intervals in the upper 40 m of the hole, the best being 0.8% Zn, 0.35% Cu and 0.16% Pb over 15 feet (Kennecott, 1998).

Based on the results of the 1998 program, Kennecott made the following recommendations: anomaly 98-3 required further drilling, two other anomalies were drill ready, and five other anomalies, including anomaly 98-9, required additional exploration to define a drill target. Despite the encouraging regional exploration results, Kennecott conducted no further field exploration in the district after 1998 and subsequently optioned the property to NovaGold in 2004.

In addition to the regional program, Kennecott completed five lines of controlled source audio magneto telluric (CSAMT) data in Subarctic Valley. The Arctic deposit showed an equally strong conductive response in the CSAMT data as was seen in the EM data. As a result of the survey, Kennecott recommended additional CSAMT for the deposit area.

5.3.4 Drilling

Between 1967 and July 1985, Kennecott (BCMC) completed 86 holes (including 14 large diameter metallurgical test holes) totalling 16,080 m. In 1998, Kennecott drilled an additional six core holes totalling 1,492 m to test for:

- Extensions of the known Arctic mineralization.

- Grade and thickness continuity.
- EM anomaly 98-3 (East Dead Creek).

Drilling for all BCMC/Kennecott campaigns in the Arctic deposit area (1966 to 1998) totals 92 core holes for a combined 17,572 m (refer to Section 7 for additional information).

5.3.5 Specific Gravity

Prior to 1998, no SG measurements were available for the Arctic deposit rocks. A "factored" average bulk density was used to calculate a tonnage factor for resource estimations. A total of 38 samples from the 1998 drilling at the Arctic deposit were measured for SG determinations. Additional information on density determinations is provided in Section 8.

5.3.6 Petrology, Mineralogy, and Research Studies

There have been numerous internal studies done by Kennecott on the petrology and mineralogy of the Arctic deposit that exist as internal memos, file notes, and reports from as early as 1967. These data have been used in support of geological and mineralogical interpretations.

Portable infrared mineral analyzer (PIMA) alteration studies by Kennecott on a limited amount of core show that hydrothermal alteration (Na/Ba-rich micas) can still be spectrally recognized despite later regional metamorphism and continued use was recommended to assist in the structural interpretation of the known mineralization and in tracing the 'ore horizon'.

D. Schmandt completed an undergraduate thesis at Smith College in 2009 entitled "Mineralogy and origin of Zn-rich horizons within the Arctic Volcanogenic Massive Sulphide deposit, Ambler District, Alaska". Jeanine Schmidt completed a doctoral dissertation at Stanford University in 1983 entitled "The Geology and Geochemistry of the Arctic Prospect, Ambler District, Alaska"; and Bonnie Broman completed a master's thesis at University of Alaska, Fairbanks in 2014 entitled "Metamorphism and Element Redistribution: Investigations of Ag-bearing and associated minerals in the Arctic Volcanogenic Massive Sulphide deposit, SW Brooks Range, NW Alaska". These studies have provided additional information on geological and mineralogical settings in the Arctic Project area.

5.3.7 Geotechnical, Hydrogeological and Acid-Base Accounting Studies

A series of geotechnical, hydrological, hydrogeological, and acid-base accounting (ABA) studies were conducted prior to the 2020 FS. Rock geotechnical and hydrogeological studies completed after 1998 are listed in Table 5-2. ABA studies completed after 1998 are listed in Section 13.11.1.

5.3.7.1 Geotechnical and Hydrogeological Studies

In December 1998, URSA Engineering prepared a geotechnical study for Kennecott titled "Arctic Project – 1998 Rock Mass Characterization". Though general in scope, the report summarized some of the basic rock characteristics as follows:

- Compressive strengths average 6,500 psi for the quartz mica schists, 14,500 psi for the graphitic schists, and 4,000 psi for talc schists.
- Rock mass quality can be described as average to good quality, massive with continuous jointing except the talc schist, which was characterized as poor quality. The rock mass rating (RMR) averages 40 to 50 for most units except the talc schist which averages 30.

In 1998, Robertson Geoconsultants, Inc. (Robertson) of Vancouver prepared a report for Kennecott titled "Initial Assessment of Geochemical and Hydrological Conditions at Kennecott's Arctic Project". The report presented the results of the acid generation potential of mine waste and wall rock for the Arctic Project in the context of a hydrological assessment of the climate, hydrology, and water balance analyses at the Arctic deposit. Climatic studies at the time were limited to regional analyses as no climatic data had been collected at the Arctic Project site prior to the review. Regional data, most specifically a government installed gauging station about 32 km to the southwest at Dahl Creek, provided information in assessing the hydrology of the Arctic Project at the time. A total of nine regional gauges were utilized to evaluate the overall potential runoff in the area.

Table 5-2: Summary of Previous Geotechnical and Hydrogeological Work Completed After 1998

	Rock Geotechnical	Hydrogeological
Field Investigations	SRK 2020 Completed geotechnical and hydrogeological drill investigation program consisting of nine (9) drill holes. Detailed geotechnical core logging completed for four (4) drill holes, basic geotechnical logging completed for 5 drill holes. Laboratory testing conducted.	SRK 2020 Completed geotechnical and hydrogeological drill investigation program consisting of nine (9) drill holes. Hydraulic conductivity tests, piezometer installations, and extended duration injection tests were completed in five (5) drill holes.
	SRK 2017 Completed staged geotechnical field investigation programs in 2015 and 2016. Five HQ3 drill holes were completed for combined geotechnical and hydrogeological data acquisition purposes. Thirteen drill holes were surveyed using acoustic televiewer. Laboratory testing was conducted.	SRK 2017 Completed staged geotechnical field investigation programs in 2015 and 2016. Five HQ3 drill holes were completed for combined geotechnical and hydrogeological data acquisition purposes. Conducted downhole hydraulic conductivity tests and installed standpipe and vibrating wire piezometers.
	Tetra Tech 2013 No geotechnical field investigations were completed. Review of historical geotechnical studies suggests work completed to a high standard.	Tetra Tech 2013 No hydrogeological field investigations were completed.
	BGC 2012 Underground focus. Completed five HQ3 drill holes with lab testing, core logging and lithology descriptions (see Section 3.2). Completed structural geology mapping. Completed laboratory strength testing.	BGC 2012 Installed seven (7) vibe wires in five holes (see Section 3.2 for drill hole locations). Completed hydraulic conductivity testing for various geotechnical units. Installed one thermistor. Hydraulic head measurements at selected drill holes.
	URSA 1998 Mapping of major geological structures, faults, and joints. Completed structural lab strength testing and collected geotechnical data from resource drill holes	URSA 1998 No hydrogeological work completed

	Rock Geotechnical	Hydrogeological
Technical Findings	SRK 2020 Updated the geotechnical characterization and structural geology model. Sloped designs were updated based on updated geotechnical assessments. 3-D numerical stability models were utilized to review the influence of talc in the northeast wall.	SRK 2020 Hydrogeological assessments were completed for both the open pit and valley bottom water/waste management areas. Pit inflow and pore pressure conditions were assessed using numerical and analytical tools.
	SRK 2017 Established six geotechnical domains based on the rock mass characteristics and structural orientations. Kinematic and 2-D numerical stability analyses were conducted to provide the recommended slope configurations. The slope design for the east walls that are sub-parallel to the foliation consisted of 60 m high slopes that are stripped along the dip of the foliation fabric. The recommended IRA in the east walls were 26°-30°. Slope design of other areas of the proposed open pit were controlled by kinematic failure mechanisms.	SRK 2017 Two alternative conceptual hydrogeological models were established – a multiple water system model (regional shallow perched water table in northeast, deeper water table over the pit footprint), and compartmentalized water system model (water levels compartmentalized by faults, talc, and/or permafrost). A conservative pore pressure conditions were estimated for slope stability analysis.
	Tetra Tech 2013 Considers 7 lithogeochemical units; modeled as 3-D volumes (as supplied by Trilogy) PEA pit with OSA = 43°, triple benched with 5 m benches, 8 m catch berms (single domain/design sector)	Tetra Tech 2013 Assumes groundwater levels will follow topography. No continuous permafrost in area.
	BGC 2012 Two-dimensional geotechnical model using six (6) units combining lithological units of similar rock mass properties (RMR and other criteria defined for each unit). No 3-D structural model, plan view map only with two regional thrust faults (West Fault at 50° dipping south-southeast and WSFC at 40° dipping to the south). Mapped strong foliation sets dipping shallowly to the west and moderately to the southwest.	BGC 2012 Conceptual model of ground water flow. High flow rates in fault zones should be expected, groundwater flow follows topography. Not enough information to comment on recharge rate due to uncertainty in permafrost. Permeability data for main geotechnical units ranging from 3×10^{-9} to 6×10^{-7} m/s.
	URSA 1998 Developed six (6) geotechnical units based on drill holes and mapping, classified rocks as weak to moderately strong, talc mica schist is worst unit with RMR = 31. Mapped E-W trending faults with gouge of muscovite and talc and south dipping thrust fault. Two sets of joint/foliation present that intersect at 25°.	URSA 1998 No hydrogeological analyses completed. Suggests permafrost not present based on down hole observations

5.3.7.2 Acid-Base Accounting Studies

The 1998 Robertson study first documented ABA results, based on the selection of 60 representative core samples from the deposit. Since then, SRK has advanced the understanding of acid rock drainage potential with studies in 2010, 2013, and 2015-2017 based on an accumulated dataset from over 1550 ABA drill core samples from waste rock and ore, indicating that about 75% of samples were classified as potentially acid generating. Tailings from metallurgical testing in 2017 and 2019 were analysed for ABA and also shown to be potentially acid generating. Kinetic testing studies have subsequently been conducted on waste rock, ore and tailings, to inform rates of acid generation and metal leaching, and are documented in Section 13.11.2.

5.3.8 Metallurgical Studies

Kennecott undertook an extensive series of studies regarding the metallurgy and processing of the Arctic mineralization (refer to summary in Section 10.2 of this Report).

5.3.9 Development Studies

A number of mining and technical studies have been completed over the Arctic Project history, as summarized in Table 5-3.

Table 5-3: Mining and Technical Services

Company	Year	Consultant	Study
Kennecott	1974	internal	Ambler District Evaluation
	1976	internal	Arctic Deposit Order of Magnitude Evaluation
	1978	internal	Arctic Prospect Summary File Report Arctic Deposit
	1981	internal	Evaluation of the Arctic and Ruby Creek Deposits
	1984	internal	Evaluation Update
	1985	internal	Pre-AFD Report
	1990	internal	Re-Evaluation
	1997	internal	Arctic Project Mining Potential
	1999	internal	Interim Report Conceptual Level Economic Evaluations of the Arctic Resource
	1998	SRK	Preliminary Arctic Scoping Study
NovaGold	2011	SRK	Preliminary Economic Assessment
Trilogy (previously, NovaCopper)	2012	SRK	Preliminary Economic Assessment
	2013	Tetra Tech	Preliminary Economic Assessment
	2018	Ausenco	Pre-Feasibility Report
Trilogy	2020	Ausenco	Feasibility Report
Ambler Metals	2021	Ausenco	Tailings Evaluation

6 GEOLOGICAL SETTING, MINERALIZATION, AND DEPOSIT

6.1 Deposit Types

6.1.1 Deposit Model

The mineralization at the Arctic deposit and at several other known occurrences within the Ambler Sequence stratigraphy of the Ambler Mining District consists of Devonian age, polymetallic (zinc-copper-lead-silver-gold) VMS-like occurrences.

Observations and interpretations at the Arctic deposit such as: 1) the tectonic setting with Devonian volcanism in an evolving continental rift; 2) the geologic setting with bimodal volcanic rocks including pillow basalts and felsic volcanic tuffs; 3) an alteration assemblage with well-defined magnesium-rich footwall alteration and sodium-rich hanging wall alteration; and 4) typical polymetallic base-metal mineralization with massive and semi-massive sulphides, are indicative of a Volcanogenic Massive Sulphide (VMS) deposit that has undergone high strain and complex folding and faulting.

A variety of VMS types have been well documented in the literature (Franklin et al., 2005), with the Ambler Schist belt deposits most like deposits associated with bimodal felsic dominant volcanism related to incipient rifting. However, the abundance of volcaniclastic rocks with argillaceous sedimentary rocks and the tabular nature of mineralization are considered by Piercey (2022) to be similar to felsic silicilastic VMS environments.

Evidence exists for both exhalation and emplacement on the seafloor and replacement of rocks in the sub-seafloor, either via filling of void space or via dissolution of original rocks and replacement by new minerals (Piercey, 2022). For example, the presence of barite, attributed to the mixing of $BaCl_2$(aq) from hydrothermal fluids with seawater sulphate (SO_4(aq)) at the vent-seawater interface supports some of the mineralization at Arctic likely precipitated on the seafloor. In contrast, there is ample textural evidence of subseafloor replacement at Arctic, such as the presence of transitions from massive sulphides into selective replacement of interpreted permeable tuff beds in the hanging wall mudstones.

The tonnage, grades, and stratigraphic setting of the Arctic deposit, and its broader tectonostratigraphic setting, are similar to other felsic siliclastic VMS environments globally. The deposit has strong similarities to deposits found the Finlayson Lake VMS district, Yukon, Bathurst district, New Brunswick, and some parts of the Iberian Pyrite Belt, Spain-Portugal (Piercey, 2022).

A VMS model is considered applicable for use in exploration targeting in the project area and the interpretation of the geological model supporting the Mineral Resource estimate.

6.2 Regional Geology – Southern Brooks Range

The Ambler Mining District occurs along the southern margin of the Brooks Range within an east-west trending zone of Devonian to Jurassic age submarine volcanic and sedimentary rocks (Hitzman et al., 1986). The district covers both: 1) VMS-like deposits and prospects hosted in the Devonian age Ambler Sequence (or Ambler Schist belt or Schist belt), a group of metamorphosed bimodal volcanic rocks with interbedded tuffaceous, graphitic, and calcareous volcaniclastic metasediments; and 2) epigenetic carbonate-hosted copper deposits occurring in Silurian to Devonian age carbonate and phyllitic rocks of the Bornite Carbonate Sequence. The Ambler Sequence occurs in the upper part of the Anirak Schist, the thickest member of the Schist belt or Coldfoot subterrane (Moore et al., 1994). VMS-like stratabound mineralization can be found along the entire 110 km strike length of the district. Immediately south of the Schist belt in the Cosmos Hills, a time equivalent section of the Anirak Schist that includes the approximately 1 km thick Bornite Carbonate

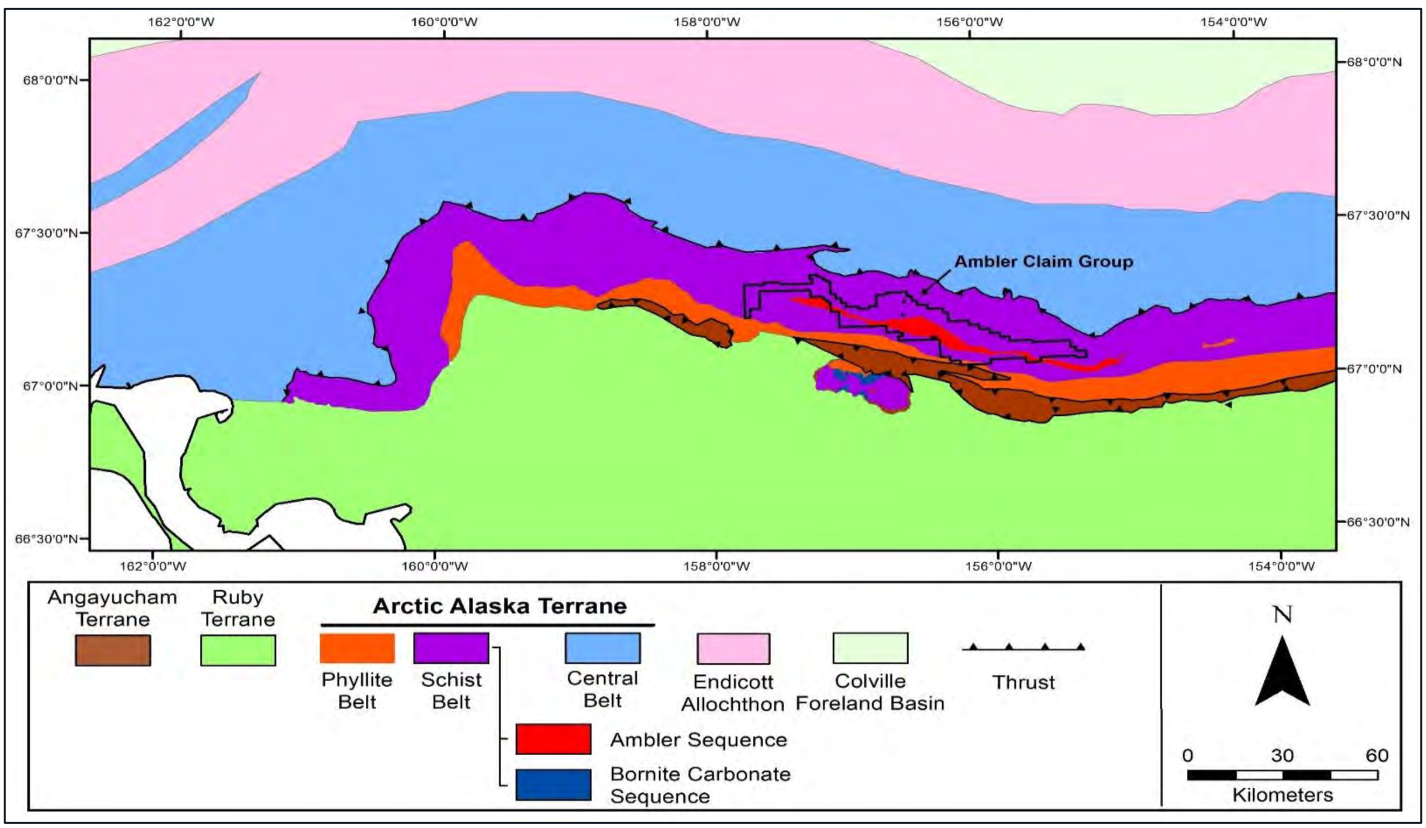
Sequence. Mineralization of both the VMS-like deposits of the Schist belt and the carbonate-hosted deposits of the Cosmos Hills has been dated at 375 to 387 Ma (Selby et al., 2009; McClelland et al., 2006).

The Ambler Mining District is characterized by increasing metamorphic grade to the north, perpendicular to the strike of the east-west trending units. The district shows isoclinal folding in the northern portion and thrust faulting to south (Schmidt, 1983). The Devonian to Late Jurassic age Angayucham basalt and the Triassic to Jurassic age mafic volcanic rocks are in low angle over thrust contact with various units of the Ambler Schist belt and Bornite Carbonate Sequence along the northern edge of the Ambler Lowlands.

6.2.1 Terrane Descriptions

The terminology used to describe the terranes in the southern Brooks Range evolved during the 1980s because of the region's complex juxtaposition of rocks of various compositions, ages and metamorphic grade. Hitzman et al. (1986) divided the Ambler Mining District into the Ambler and Angayucham terranes. Further work (Till et al., 1988; Silberling et al., 1992; Moore et al., 1994) includes the rocks of the previously defined Ambler terrane as part of the regionally extensive Schist belt or Coldfoot subterrane along the southern flank of the Arctic Alaska terrane as shown in Figure 6-1 (Moore et al., 1994). In general, the southern Brooks Range is composed of east-west trending structurally bound allochthons of variable metasedimentary and volcanogenic rocks of Paleozoic age.

Figure 6-1: Regional Geologic Terranes of the Southern Brooks Range



Source: Trilogy Metals, 2019.

The Angayucham terrane, which lies along the southern margin of the Brooks Range, is locally preserved as a klippen within the eastern Cosmos Hills and is composed of weakly metamorphosed to unmetamorphosed massive-to-pillowed basalt rocks with minor radiolarian cherts, marble lenses and isolated ultramafic rocks. This package of Devonian to Late Jurassic age (Plafker et al., 1977) mafic and ultramafic rocks is interpreted to represent portions of an obducted and structurally dismembered ophiolite that formed in an ocean basin south of the present-day Brooks Range (Hitzman et al., 1986; Gottschalk and Oldow, 1988). Locally, the Angayucham terrane overlies the Schist belt to the north along a poorly exposed south-dipping structure.

Gottschalk and Oldow (1988) describe the Schist belt as a composite of structurally bound packages composed of dominantly greenschist facies rocks, including pelitic to semi-pelitic quartz-mica schist with associated mafic schists, metagabbro and marbles. Locally, the Schist belt includes the Upper Silurian to middle Devonian age Bornite Carbonate Sequence, the lower Paleozoic age Anirak pelitic, variably siliceous and graphic schists, and the mineralized Devonian age Ambler sequence consisting of volcanogenic and siliciclastic rocks variably associated with marbles, calc-schists, metabasites and mafic schists (Hitzman et al., 1982; Hitzman et al., 1986). The lithologic assemblage of the Schist belt is consistent with an extensional, epicontinental tectonic origin.

Structurally overlying the Schist belt to the north is the Central belt. The Central belt is in unconformable contact with the Schist belt along a north-dipping low-angle structure (Till et al., 1988). The Central belt consists of lower Paleozoic age metaclastic and carbonate rocks, and Proterozoic age schists (Dillon et al., 1980). Both the Central belt and Schist belt are intruded by meta-to-peraluminous orthogneisses, which locally yield a slightly discordant U-Pb thermal ionization mass spectrometry zircon crystallization age of middle to late Devonian (Dillon et al., 1980; Dillon et al., 1987). This igneous protolith age is supported by Devonian orthogneiss ages obtained along the Dalton Highway, 161 km to the east of the Ambler Mining District (Aleinikoff et al., 1993).

Overlying the Schist belt to the south is the Phyllite belt, characterized in the Ambler mining district as phyllitic black carbonaceous schists of the Beaver Creek Phyllite which may underlie much of the Ambler Lowlands between the Brooks Range and the Arctic deposit to the north and the Cosmos Hills and the Bornite deposit to the south. The recessive weathering nature of the Beaver Creek Phyllite limits the exposure, but the unit is assumed to occur as a thrust sheet overlying the main Schist belt rocks.

6.2.2 Regional Tectonic Setting

Rocks exposed along the southern Brooks Range consist of structurally bound imbricate allochthons that have experienced an intense and complex history of deformation and metamorphism. Shortening in the fold and thrust belt has been estimated by some workers to exceed 500 km (Oldow et al., 1987) based on balanced cross sections across the central Brooks Range. In general, the metamorphic grade and tectonism in the Brooks Range increases to the south and is greatest in the Schist belt. The tectonic character and metamorphic grade decreases south of the Schist belt in the overlying Angayucham terrane.

During the late Jurassic to early Cretaceous, the Schist belt experienced penetrative thrust-related deformation accompanied by recrystallization under high-pressure and low-temperature metamorphic conditions (Till et al., 1988). The northward directed compressional tectonics were likely related to crustal thickening caused by obduction of the Angayucham ophiolitic section over a south-facing passive margin. Thermobarometry of schists from the structurally deepest section of the northern Schist belt yield relict metamorphic temperatures of 475°C, ±35°C, and pressures from 7.6 to 9.8 kb (Gottschalk and Oldow, 1988). Metamorphism grades from lower greenschist facies in the southern Cosmos Hills to upper greenschist facies, locally overprinting blueschist mineral assemblages in the Schist belt (Hitzman et al., 1986).

Compressional tectonics, which typically place older rocks on younger, does not adequately explain the relationship of young, low-metamorphic-grade over older and higher-grade metamorphic rocks observed in the southern Brooks Range hinterland. Mull (1982) interpreted the Schist belt as a late antiformal uplift of the basement to the fold and thrust belt. More recent models propose that the uplift of the structurally deep Schist belt occurred along duplexed, north-directed, thin-skinned thrust faults, followed by post-compressional south-dipping low angle normal faults along the south flank of the Schist belt, accommodating for an over-steepened imbricate thrust stack (Gottschalk and Oldow, 1988; Moore et al., 1994). Rapid cooling and exhumation of the Schist belt began at the end of the early Cretaceous age at 105 to 103 Ma, based on Ar40/Ar39 cooling ages of hornblende and white mica from near Mount Igikpak, and lasted only a few million years (Vogl et al., 2003). Additional post-extension compressive events during the Paleocene age further complicate the southern Brooks Range (Mull, 1985).

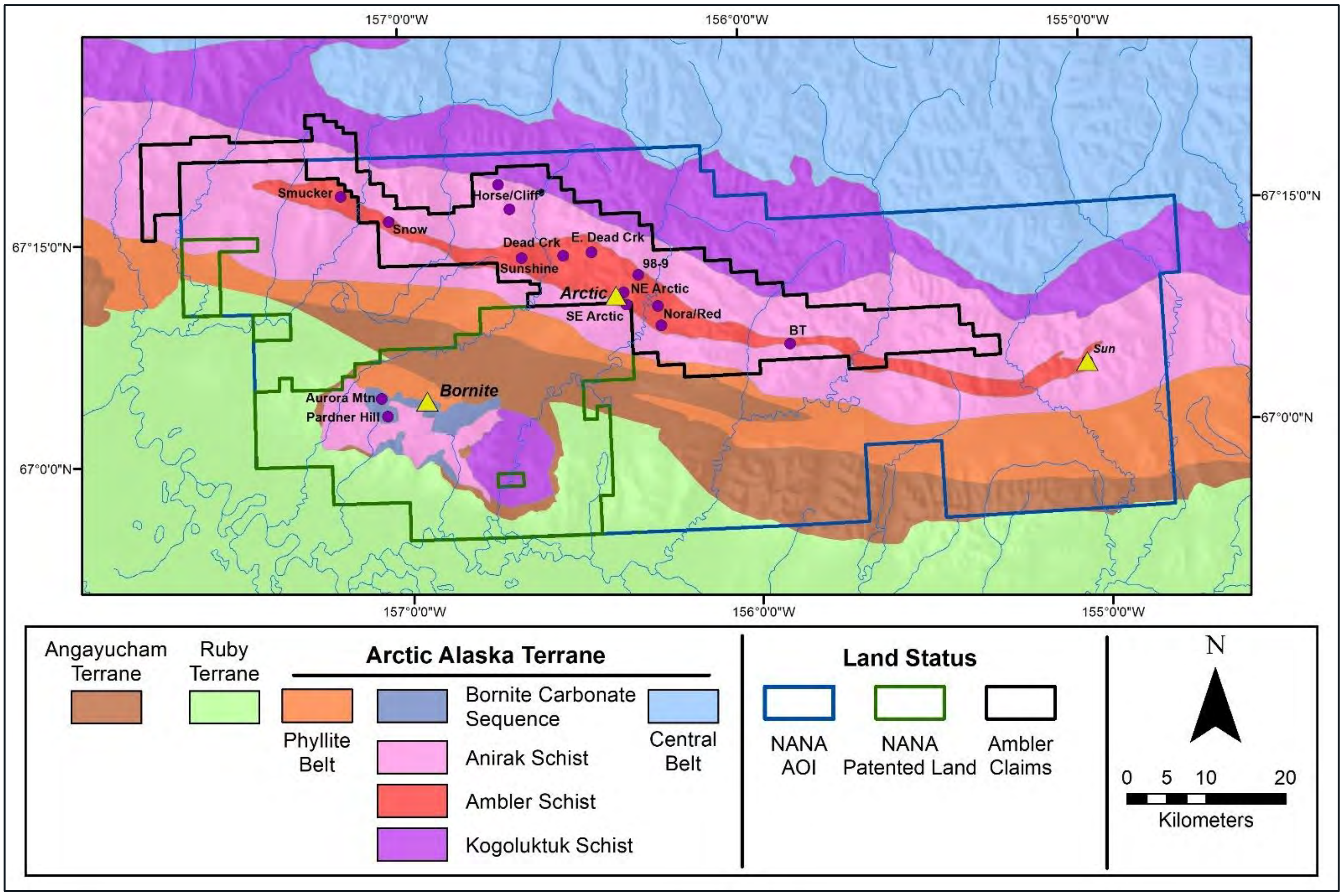
6.3 Local Geology

Rocks that form the Ambler Sequence consist of a lithologically diverse sequence of lower Devonian age carbonate and siliciclastic strata with interlayered mafic lava flows and sills. The clastic strata, derived from terrigenous continental and volcanic sources, were deposited primarily by mass-gravity flow into the sub-wavebase environment of an extending marginal basin.

The Ambler Sequence underwent two periods of intense, penetrative deformation. Sustained upper greenschist-facies metamorphism with coincident formation of a penetrative schistosity and isoclinal transposition of bedding marks the first deformation period. Pervasive similar-style folds on all scales deform the transposed bedding and schistosity, defining the subsequent event. At least two later non-penetrative compressional events deform these earlier fabrics. Observations of the structural and metamorphic history of the Ambler Mining District are consistent with current tectonic evolution models for the Schist belt, based on the work of others elsewhere in the southern Brooks Range (Gottschalk and Oldow, 1988; Till et al., 1988; Vogl et al., 2002).

Figure 6-2 shows the location and geology of the Ambler mining district and the Schist belt terrane including the Anirak Schist, the Kogoluktuk Schist and the Ambler Sequence, the contemporaneous Bornite Carbonate Sequence in the Cosmos Hills to the south, and the allochthonous overthrust Cretaceous sedimentary rocks and Devonian Angayucham Terrane volcanic rocks.

Figure 6-2: Local Geology of the Ambler Mining District



Source: Ambler Metals, 2022.

6.3.1 General Stratigraphy of the Ambler Sequence

Though the Ambler Sequence is exposed over 110 km of strike length, the following descriptions and comments refer to an area between the Kogoluktuk River on the east and the Shungnak River on the west where Trilogy has focused the majority of its exploration efforts over the last decade.

The local base of the Ambler Sequence consists of variably metamorphosed carbonates historically referred to as the Gnurgle Gneiss. Trilogy's joint venture, Ambler Metals interprets these strata as calc-turbidites, perhaps deposited in a sub-wavebase environment adjacent to a carbonate bank. Calcareous schists overlie the Gnurgle Gneiss and host sporadically distributed mafic sills and pillowed lavas. These fine-grained clastic strata indicate a progressively quieter depositional environment up section, and the presence of pillowed lavas indicates a rifting, basinal environment.

Overlying these basal carbonates and pillowed basalts is a section of predominantly fine-grained carbonaceous siliciclastic rocks that host most of the mineralization in the district including the Arctic deposit. This quiescent section indicates further isolation from a terrigenous source terrain.

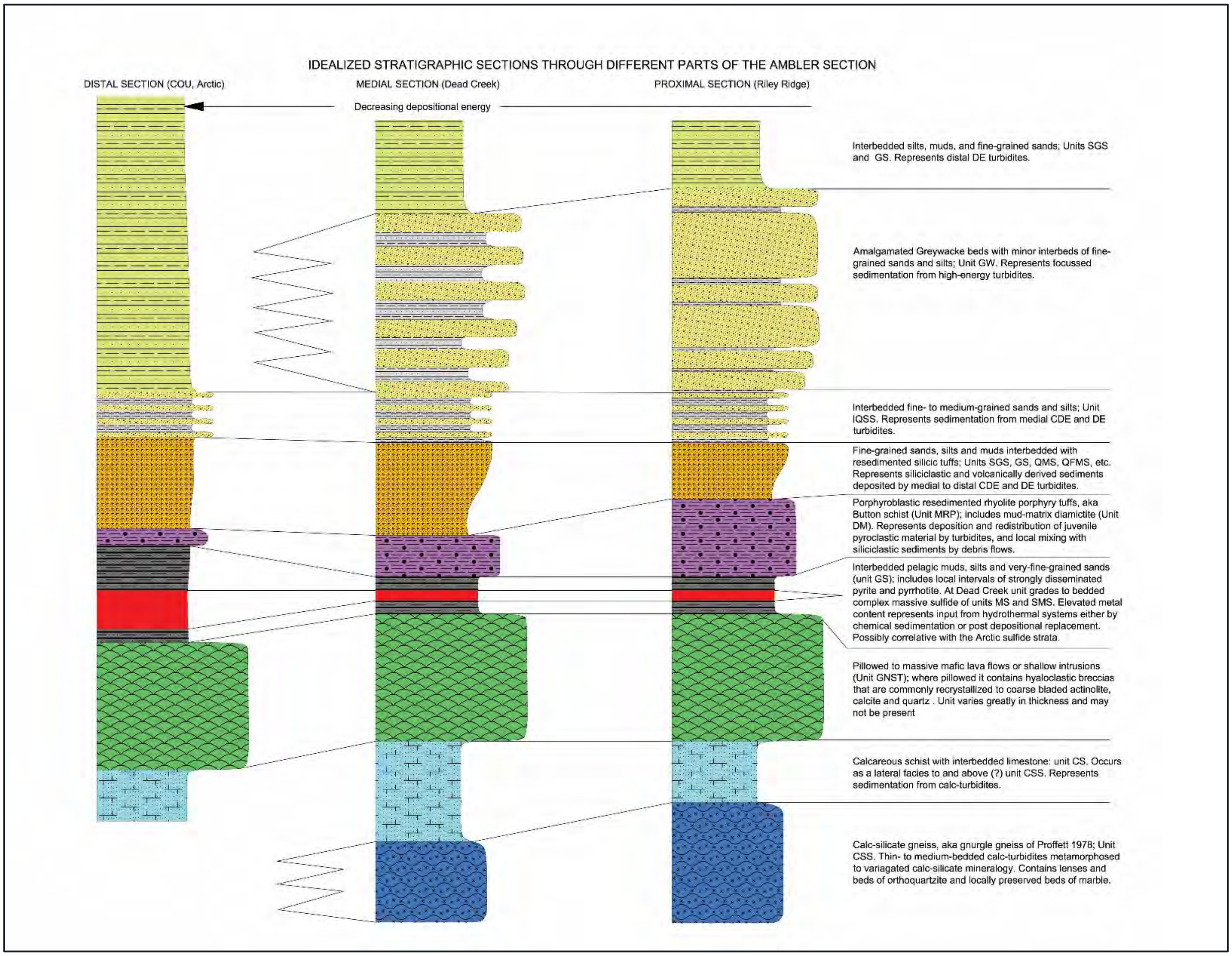
Above the stratigraphy hosting the Arctic deposit is a section with voluminous reworked silicic volcanic strata. At the base of this section is the Button Schist, a regionally continuous and distinctive K-feldspar porphyroblastic unit that serves as an excellent marker above the main mineralized stratigraphy. Figure 6-3 shows idealized sections for several different areas in and around the Arctic deposit.

Several rock units show substantial changes in thickness and distribution in the vicinity of the Arctic deposit that may have resulted from the basin architecture existing at the time of deposition. Between the Arctic Ridge, geographically above the Arctic deposit, and the Riley Ridge to the west, several significant differences have been documented including:

- The Gnurgle Gneiss is thickest in exposures along the northern extension of Arctic Ridge and appears to thin to the west.
- Mafic lavas and sills thicken from east to west. They occur as thick units in upper Subarctic Creek and to the west but are sparsely distributed to the east.
- The quartzite section within and above the Arctic sulphide horizon does not occur in abundance east of Arctic Ridge; it is thicker and occurs voluminously to the west.
- Button Schist thickens dramatically to the west from exposures on Arctic Ridge; exposures to the east are virtually non-existent.
- Greywacke sands do not exist east of Subarctic Creek but occur in abundance as massive, channelled accumulations to the west, centred on Riley Ridge.

These data are interpreted by Trilogy's joint venture, Ambler Metals to define a generally north-northwest-trending depocentre through the central Ambler Mining District. Volcanic debris flow occurrences as well as these formational changes suggest that the depocentre had a fault-controlled eastern margin. The basin deepened to the west; the Riley Ridge section was deposited along a high-energy axis, and the COU section lies to the west-southwest distally from a depositional energy point of view. This original basin architecture appears to have controlled mineralization of the sulphide systems at Arctic and Shungnak (Dead Creek), concentrating fluid flow along structures on the eastern basin margin.

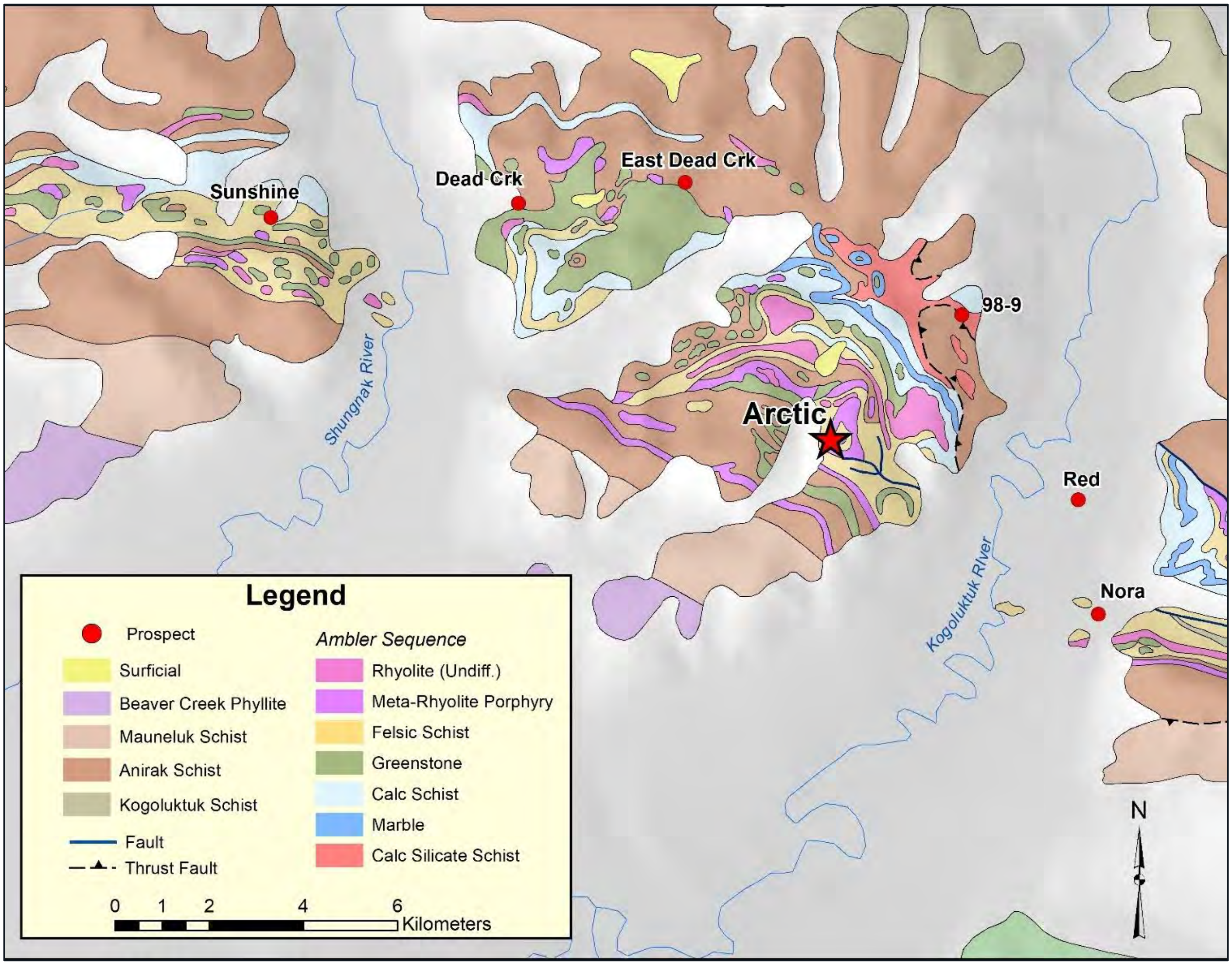
Figure 6-3: Ambler Sequence Stratigraphy in the Arctic Deposit Area



Source: Trilogy Metals, 2019.

Figure 6-4 is a simplified geologic map of the area between the Kogoluktuk and the Shungnak Rivers.

Figure 6-4: Generalized Geology of the Central Ambler Mining



Source: Trilogy Metals, 2022.

6.3.2 Structural Framework of the Ambler Mining District

In addition to the underlying pre-deformational structural framework of the district suggested by the stratigraphic thickening of various facies around the Arctic deposit, the Ambler Sequence is deformed by two penetrative deformational events that significantly complicate the distribution and spatial arrangement of the local stratigraphy.

6.3.2.1 F1 Deformation

The earliest penetrative deformation event is associated with greenschist metamorphism and the development of regional schistosity. True isoclinal folds are developed, and fold noses typically are thickened. The most notable F1 fold is the Arctic antiform that defines the upper and lower limbs of the Arctic deposit. The fold closes along a north-northeast-trending fold axis roughly mimicking the trace of Subarctic Creek and opening to the east. Importantly, the overturned lower limb implies that the permissive stratigraphy should be repeated on a lower synformal isocline beneath the currently explored limbs and would connect with the permissive mineralized stratigraphy to the northwest at Shungnak (Dead Creek). Figure 6-5 shows typical F1 folds developed in calcareous Gnurgle Gneiss.

Figure 6-5: Typical F1 Isoclinal Folds Developed in Calcareous Gnurgle Gneiss

Source: Trilogy Metals, 2019.

6.3.2.2 F2 Deformation

The earlier F1 schistosity is in turn deformed by an F2 deformational event that resulted in the local development of an axial planar cleavage. The deformational event is well defined throughout the Schist belt and results in a series of south-verging, open to moderately overturned folds that define a series of east-west trending folds of similar vergence across the entire Schist belt stratigraphy.

This event is likely temporally related to the emplacement of the Devonian Angayucham volcanics sequences, the obducted Jurassic ophiolites and Cretaceous sediments within the Schist belt stratigraphy.

In addition to the earlier penetrative deformation events, a series of poorly defined non-penetrative deformation events, likely due to Cretaceous extension, are seen as a series of warps or arches across the district.

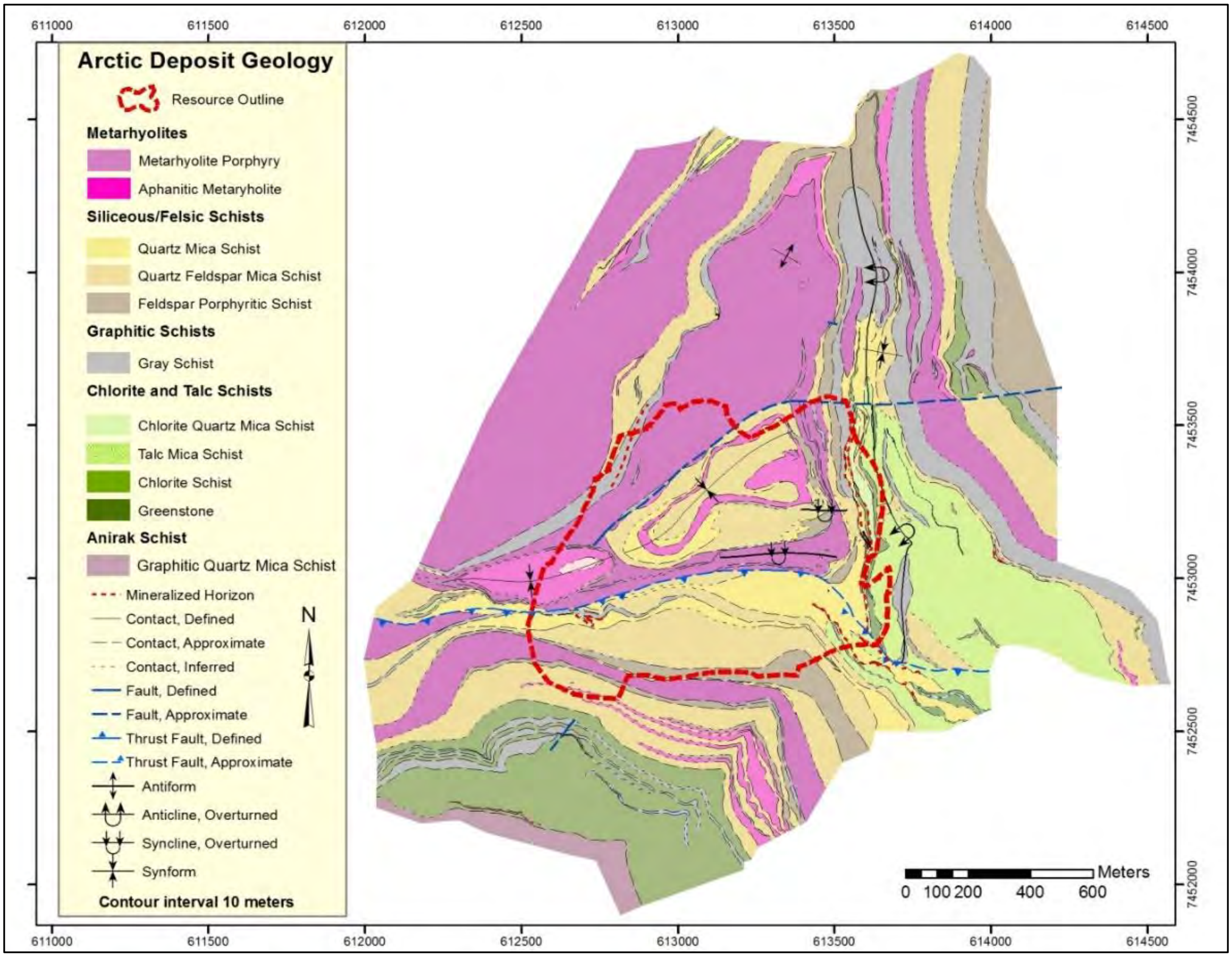
The interplay between the complex local stratigraphy, the isoclinal F1 event, the overturned south verging F2 event and the series of post-penetrative deformational events often makes district geological interpretation extremely difficult at a local scale.

6.4 Arctic Property Geology

Previous workers at the Arctic deposit (Russell 1995 and Schmidt 1983) describe three mineralized horizons: the Main Sulphide Horizon, the Upper South Horizon and the Warm Springs Horizon. The Main Sulphide Horizon was further subdivided into three zones: the southeast zone, the central zone, and the northwest zone. Previous deposit modelling was grade-based resulting in numerous individual mineralized zones representing relatively thin sulphide horizons.

Recent work by Ambler Metals defines the Arctic deposit as two or more discrete horizons of sulphide mineralization contained in a complexly deformed isoclinal fold with an upright upper limb and an overturned lower limb hosting the main mineralization. Nearby drilling suggests that a third upright lower limb, likely occurs beneath the currently explored stratigraphy. Figure 6-6 is a generalized geologic map of the immediate Arctic deposit area.

Figure 6-6: Generalized Geologic Map of the Arctic Deposit



Source: Trilogy Metals, 2019.

6.4.1 Lithologies and Lithologic Domain Descriptions

Historically, five lithologic groupings were used by Kennecott (URSA Engineering, 1998 and Russell, 1995) to describe the local stratigraphy of the deposit. These groupings of rock types and protoliths include: 1) metarhyolite (Button Schist) or porphyroblastic quartz feldspar porphyry and rhyolitic volcaniclastic and tuffaceous rocks; 2) quartz mica schists composed of tuffaceous and volcaniclastic sediments; 3) graphitic schists composed of carbonaceous sedimentary rocks; 4) base metal sulphide bearing schists; and 5) talc schists composed of talc altered volcanic and sedimentary rocks. Trilogy's joint venture, Ambler Metals has subsequently re-interpreted and modified the lithological groupings. In 2022, Piercey provided further interpretation of the stratigraphy, lithofacies, alteration and mineralization at Arctic based on a study of core from six holes drilled in 2022 and a field visit to Arctic.

The principal lithologic units captured in logging and mapping by Trilogy's joint venture, Ambler Metals are summarized and described in the following subsections, in broadly chronologically order from oldest to youngest. A summary of the interpretation of geology in cross section is shown in Figure 7-14, Figure 7-15, Figure 7-17 and Figure 7-18. These cross sections illustrate significant mineralized zones encountered on the Property including, a summary of the surrounding rock types, relevant geological controls, and the length, width, depth and continuity of the mineralization.

6.4.1.1 Greenstone (GNST)

Greenstones consists of massive dark-green amphibole and garnet-bearing rocks, differentiated by their low quartz content and dark green colour. Intervals of greenstone range up to 80 m in thickness and are identified as pillowed flows, sills, and dikes. Multiple ages of deposition are implied by the presence of both basal pillowed units as well as intrusive sill and dike-like bodies higher in the local stratigraphy. Textural and colour similarities along with similar garnet components and textures often cause confusion with some metasedimentary greywackes within the Ambler Sequence stratigraphy.

6.4.1.2 Chlorite Schist (ChS)

This unit is likely alteration-related but has been used for rocks where more than half of the sheet silicates are composed of chlorite. In the field, some samples of chlorite schist show a distinctive dark green to blue-green colour, but in drill core the chlorite schists commonly have a lighter green colour. Some intervals of chlorite schist are associated with talc-rich units. The footwall to the Arctic deposit is dominated by this lithofacies which Piercey (2022) describes as variably chlorite (and talc) altered tuffs and lapilli tuffs. Immediately beneath mineralization the rocks are tuffaceous and are strongly and pervasively chlorite and talc (- sericite) altered and, with depth, there are alternating layers of finer chlorite-talc bearing tuffs and quartz and feldspar and/or felsic fragment lapilli tuffs.

6.4.1.3 Talc Schist (TS)

Talc-bearing schists are often in contact with chlorite-rich units and reflect units which contain trace to as much as 30% talc, often occurring on partings. Like the chlorite schist this unit is likely alteration related.

6.4.1.4 Black to Grey Schist (GS)

Black or grey schists appear in many stratigraphic locations particularly higher in the stratigraphy but principally constitute the mineralized permissive stratigraphy of the Arctic deposit lying immediately below the Button Schist (MRP). The schist is typically composed of muscovite, quartz, feldspar, graphite, pyrite and/or pyrrhotite, and sometimes chlorite and/or biotite, interpreted to be an argillite with minor laminae of felsic ash and mm-sized crystals of feldspar and quartz (Piercey, 2022) The texture is phyllitic, variably crenulated, and suggests a pelitic protolith, likely deposited in a basin that was progressively filled with terrigenous fine sediment. This unit is host to the MS and SMS horizons that constitute the Arctic deposit.

Piercey (2022) recognized a distinct laminated argillite/mudstone that is more siliceous and 30 cm to several metres in thickness above massive sulphides. This argillite is interlayered with mm to cm scale tuffaceous beds that contain quartz and rhyolite ash and is variably pyritic with flecks and spots of pyrite.

6.4.1.5 Button Schist (MRP)

This rock type consists of quartz-muscovite-feldspar schists with abundant distinctive 1 to 3 cm K-feldspar porphyroblasts of metamorphic origin and occasional 0.5 to 2 cm blue quartz phenocrysts of likely igneous origin. The unit shows a commonly massive to weakly foliated texture, although locally the rocks have a well-developed foliation with elongate feldspars. Piercey (2022) observed the crystals are hosted within a grey to black argillaceous to tuffaceous matrix and interprets the Button Schist to be a crystal-rich lapilli tuff to tuff.

6.4.1.6 Quartz-Mica-(Feldspar) Schist (QMS/QFMS)

This schistose rock contains variable proportions of quartz, muscovite, and sometimes feldspar. The schist usually contains high amounts of interstitial silica, and sometimes have feldspar or quartz porphyroblasts. The texture of the unit shows significant variability and likely represents both altered and texturally distinct felsic tuffs and other volcaniclastic lithologies.

Piercey (2022) distinguishes four distinct units within the hanging wall quartz-mica (feldspar) schist: rhyolitic tuff with argillite laminae and wisps; rhyolitic tuffs to lapilli tuffs; clotted feldspar lapilli tuff, and a lapilli tuff with lenticular cm-sized fragments variably replaced by pyrite and sphalerite that is referred to as the lenticular unit. This unit sits directly atop mineralization or stratigraphically above the distinct laminated argillite/mudstone described above.

6.4.1.7 Debris Flow (DM)

This unit contains a range of unsorted, matrix supported polylithic clasts including clasts of the Button Schist, occurring in black to dark grey, very fine-grained graphitic schist. The unit occurs as lenses within other stratigraphic units, and likely represents locally derived debris flows or slumps.

6.4.1.8 Greywacke (GW)

This unit consists of massive green rocks with quartz, chlorite, probably amphibole, feldspar, muscovite, and accessory garnet, biotite, and calcite/carbonate. Voluminous accumulations of medium-grained greywacke occur within, but generally above, the quartz mica schist and are differentiated from texturally similar greenstones by the presence of detrital quartz, fine-grained interbeds, graded bedding and flute casts.

6.4.1.9 Lithogeochemistry of Immobile Trace Elements

In 2007, work by NovaGold suggested that many of the nondescript felsic metavolcanic lithologies were simply alteration and textural variants of the felsic rock units and logging was not adequately capturing true compositional lithological differences between units. Twelker (2008) demonstrated that the use of immobile trace elements, specifically $Al2O3:TiO2$ (aluminum oxide: titanium dioxide) ratios, could be used to effectively differentiate between different felsic volcanic and sedimentary suites of rocks at the Arctic deposit.

Lithogeochemistry shows three major felsic rock suites in the Arctic deposit area: a rhyolite suite; and intermediate volcanic suite and a volcaniclastic suite. These suites are partially in agreement with the logged lithology but in some instances the lithogeochemistry showed that alteration in texture and composition masked actual lithologic differences.

Results of the lithogeochemistry have led to a better understanding of the stratigraphic continuity of the various units and have been utilized to model the lithologic domains of the Arctic deposit more accurately.

6.4.1.10 Lithologic Domains

Though a variety of detailed lithologies are logged during data capture, Ambler Metals models the deposit area as two distinct structural plates, an Upper Plate and a Lower Plate separated by the Warm Springs Fault. The Upper and Lower

Plates contain similar lithologic domains that are primarily defined by lithogeochemical characteristics but are also consistent with their respective acid-generating capacities and spatial distribution around the fold axes. The domains include the following units: the Button Schist (a meta-rhyolite porphyry - MRP), aphanitic meta-rhyolite (AMR), a series of felsic quartz mica schists (QMS), and carbonaceous schists of the Grey Schist unit (GS). An alteration model was built to adequately characterize the chlorite and talc schists found within the deposit (ChS, ChTS, and TS). The mineralization is modelled as eight distinct zones (Zones 1–8) found both in the Upper and Lower Plates and range from MS to SMS.

6.4.2 Structure

Earlier studies (Russell, 1977, 1995; Schmidt, 1983) concluded mineralization at the Arctic deposit was part of a normal stratigraphic sequence striking northeast and dipping gently southwest. Subsequent reinterpretation by Kennecott in 1998 and 1999 suggested the entire Ambler Sequence could be overturned. Proffett (1999) reviewed the Arctic deposit geology and suggested that a folded model with mineralization as part of an isoclinal anticline opening east and closing west could account for the mapped and logged geology. His interpretation called for an F2 fold superimposed on a north-trending F1 fabric.

Lindberg (2004) supported a similar folded model though he considered that the main fold at Arctic to be northwest-closing and southeast-opening. Lindberg named this feature the Arctic antiform and interpreted it to be an F1 fold.

Lindberg believed most folding within the deposit occurred in the central part, within a southwest plunging “Cascade zone.” The increased thicknesses of mineralized intervals in this part of the property can in part be explained by the multiple folding of two main mineralized horizons as opposed to numerous individual mineralized beds as shown in the 1995 geologic model. The Cascade zone appears to be confined to the upper sulphide limbs of the Arctic antiform.

in 2008, closer spaced drilling across the Cascade zone confirmed the continuity of the two mineralized horizons but did not support the complexity proposed by Lindberg. Dodd et al. (2004) suggested that some of the complexity might be related to minor thrusting. Results of mapping in 2006 supported the interpretation of an F2 fold event that may fold the lower Button Schist back to the north under the deposit in this area (Otto, 2006). To test this concept, deep drilling on the north side of the deposit in 2007 encountered the appropriate upright stratigraphy at depth. Though the target horizon was not reached due to the drill rig limitations, the hole did encounter significant mineralization below the Button Schist immediately above the sulphide-bearing permissive stratigraphy.

6.4.3 Arctic Deposit Alteration

Schmidt (1988) defined three main zones of hydrothermal alteration occurring at the Arctic deposit:

- A main chloritic zone occurring within the footwall of the deposit consisting of phengite and magnesium-chlorite.
- A mixed alteration zone occurring below and lateral to sulphide mineralization consisting of phengite and phlogopite together with talc, calcite, dolomite, and quartz.
- A pyritic zone overlying the sulphide mineralization.

Field observations conducted by NovaGold in 2004 and 2005 supported by logging and short-wave infrared (SWIR) spectrometry only partially support Schmidt’s observations.

Talc and magnesium chlorite are the dominant alteration products associated with the sulphide-bearing horizons. Talc alteration grades downward and outward to mixed talc-magnesium chlorite with minor phlogopite, into zones of dominantly magnesium chlorite, then into mixed magnesium chlorite-phengite with outer phengite-albite alteration zones. Thickness of alteration zones vary with stratigraphic interpretation, but tens of metres for the outer zones is likely, as seen in phengite-albite exposures on the east side of Arctic Ridge. Stratigraphically above the sulphide-bearing horizons significant muscovite as paragonite is developed and results in a marked shift in sodium/magnesium ratios across the sulphide bearing horizons.

Piercey (2022) describes footwall alteration dominated by an assemblage of chlorite and talc with lesser sericite, quartz and carbonate with intense, pervasive talc-chlorite alteration extending between 10-50 m into the immediate footwall. Chlorite-talc alteration extends at least +200m into the footwall, decreasing in intensity with depth.

Hanging wall alteration extends approximately 50-100 metres into the hanging wall with intense talc-sericite alteration and sulphide replacement in the lenticular unit, decreasing in intensity upwards, to be patchier to semi-pervasive talc-sericite alteration in the upper parts of the stratigraphy (Piercey, 2022).Visual and quantitative determination of many of the alteration products is difficult at best due to their light colours and the well-developed micaceous habit of many of the alteration species. Logging in general has poorly captured the alteration products and the SWIR methodology though far more effective in capturing the presence or absence of various alteration minerals adds little in any quantitative assessment.

Of particular note are the barium species including barite, cymrite (a high-pressure barium phyllosilicate), and barium-bearing muscovite, phlogopite and biotite. These mineral species are associated with both alteration and mineralization and demonstrate local remobilization during metamorphism (Schmandt, 2009). Though little has been done to document their distribution to date, they do have a significant impact on bulk density measurements (refer to discussions in Section 10 and Section 11).

Talc is of particular importance at the Arctic deposit due to its potential negative impact on flotation characteristics during metallurgical processing, and on pit slope stability. The majority of the talc zones occur between the upper, stratigraphically up-right mineralized zones and the lower, overturned mineralized zones. Piercey (2022) noted that the strong talc alteration prevalent at Arctic is rather unique in this type of setting and may be due to the Mg-rich nature of the calc-silicates and carbonate-bearing rocks in the Anirak Schist underlying the felsic-dominated package.

6.4.4 Arctic Deposit Mineralization

Mineralization occurs as stratiform SMS to MS beds within primarily graphitic schists and fine-grained quartz mica schists. The sulphide beds average 4 m in thickness but vary from less than 1 m up to as much as 18 m in thickness. The sulphide mineralization occurs within eight modelled zones lying along the upper and lower limbs of the Arctic isoclinal anticline. The zones are all within an area of roughly 1 km^2 with mineralization extending to a depth of approximately 250 m below the surface. There are five zones of MS and SMS that occur at specific pseudo-stratigraphic levels which make up the bulk of the Mineral Resource estimate. The other three zones also occur at specific pseudo-stratigraphic levels but are too discontinuous.

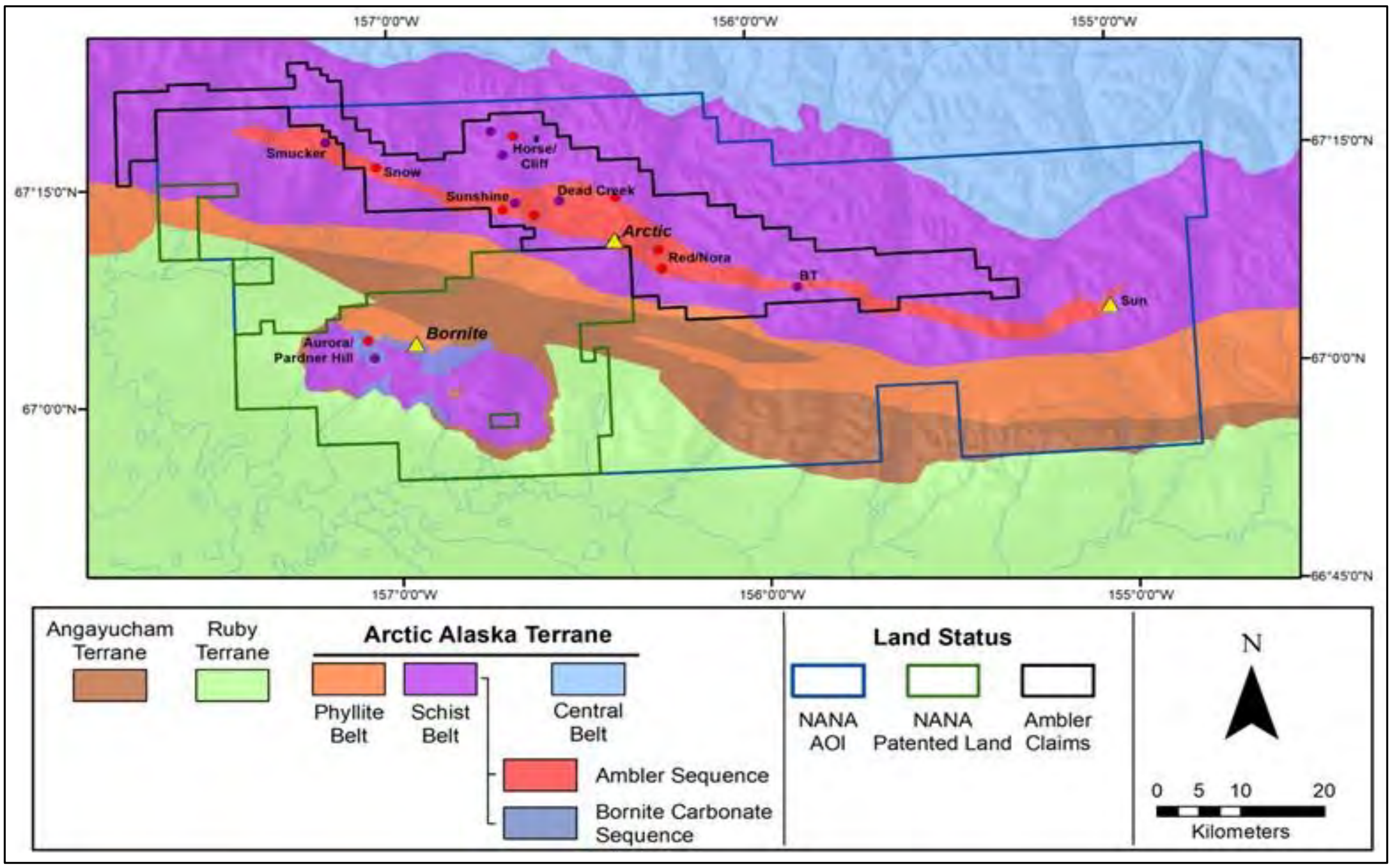
Unlike more typical VMS deposits, mineralization is not characterized by steep metal zonation or massive pyritic zones. Mineralization dominantly consists of sheet-like zones of base metal sulphides with variable pyrite and only minor zonation, usually on a small scale.

Mineralization is predominately coarse-grained sulphides comprising chalcopyrite, sphalerite, galena, tetrahedrite-tennantite, pyrite, arsenopyrite, and pyrrhotite. Sulphides occur as disseminated (<30%), semi-massive (30 to 50% sulphide) to massive (greater than 50% sulphide) layers. Trace amounts of electrum are also present. Gangue minerals associated with the mineralized horizons include quartz, barite, white mica, chlorite, stilpnomelane, talc, calcite, dolomite and cymrite.

6.5 Prospects

In addition to the Arctic deposit, numerous other VMS-like occurrences are present in the UKMP land package. The most notable of these occurrences are the Dead Creek (also known as Shungnak), Sunshine, Cliff, Horse, and the Snow prospects to the west of the Arctic deposit and the Red, Nora, Tom-Tom and BT prospects to the east. Four kilometres northwest of Arctic is target 98-9 where chalcopyrite and sphalerite associated with pyrrhotite and quartz carbonate veins are found in chlorite-biotite schists. Figure 6-7 shows the UKMP land package and the prospect locations. Figure 6-7 also shows: 1) the Smucker deposit on the far west end of the Ambler Sequence which is currently owned by Teck Alaska Inc.; 2) the Sun deposit at the eastern end of the Ambler Sequence and owned by Valhalla Metals Inc, and 3) carbonate-hosted deposits and prospects in the Bornite Carbonate Sequence controlled by Ambler Metals/NANA.

Figure 6-7: Major Prospects of the Ambler Mining District



Source: Ambler Metals, 2022.

7 EXPLORATION

7.1 Exploration

Table 7-1 summarizes the exploration work conducted by NovaGold, Trilogy (formerly, NovaCopper), and Ambler Metals from 2004 to the present. Field exploration was largely conducted during the period between 2004 to 2007 with associated engineering and characterization studies between 2008 and the present.

Table 7-1: Summary of NovaGold/Trilogy/Ambler Metals Exploration Activities Targeting VMS-style Mineralization in the Ambler Sequence Stratigraphy and the Arctic Deposit

Work Completed	Year	Details	Focus
Geological Mapping			
-	2004	-	Arctic deposit surface geology
-	2005	-	Ambler Sequence west of the Arctic deposit
-	2006	-	COU, Dead Creek, Sunshine, Red
-	2015, 2016	SRK	Geotechnical Structural Mapping
-	2016	-	Arctic deposit surface geology
-	2021	-	Snow, Ambler, Nani, DH, Cliff, Sunshine, Dead Creek, BT, 98-9/Pipe, COU, SE Arctic, Nora
-	2022	-	Snow, Ambler, Nani, DH, Bud, Sunshine, Dead Creek, BT, 98-9/Pipe, COU, East Arctic, Nora, South Cliff, SK, Cynbad, Z, Tom Tom, Kogo/White Creek
Geophysical Surveys			
SWIR Spectrometry	2004	2004 drill holes	Alteration characterization
TDEM	2005	2 loops	Follow-up of Kennecott DIGHEM EM survey
	2006	13 loops	District targets
	2007	6 loops	Arctic extensions
Downhole EM	2007	4 drill holes	Arctic deposit
VTEM Plus (Versatile Time Domain Electromagnetic) airborne helicopter geophysical	2019	400m line spacing with 200m infill with tie lines 4000m spacing	Ambler Mining District and Cosmos Hills with infill over Arctic, Sunshine and Horse-Cliff
ZTEM (Z-Axis Tipper Electromagnetic) airborne helicopter geophysical	2019	400m line spacing with tie lines 4000m spacing	Ambler Mining District and Cosmos Hills with infill over

Work Completed	Year	Details	Focus
			Arctic, Sunshine and Horse-Cliff
Geochemistry			
-	2005	-	Stream silts – core area prospects
-	2006	-	Soils – core area prospects
-		-	Stream silts – core area prospects
-	2007	-	Soils – Arctic deposit area
-	2021	-	Soils - VTEM 26-29, JA Creek, West Dead Creek, Dead Creek
-	2022	-	Soils - Sub Arctic Valley, South Cliff, VTEM 26-29, VTEM-41, VTEM-23 , East and West Sunshine, Tom Tom, Kogo/White Creek, SK, Cynbad, East Arctic, West Dead Creek, Dead Creek, 98-9/Pipe, Z, Nora, Ambler, Nani
-			Streams silts - Core area prospects
Survey			
Collar	2004 to 2011, 2018, 2019, 2021, 2022	DGPS	All 2004 to 2019 NovaCopper drill holes
	2004, 2008	Resurveys	Historical Kennecott drill holes
Photography/Topography	2010	-	Photography/topography
LiDAR Survey	2015, 2016	-	LiDAR over Arctic deposit
Technical Studies			
Geotechnical	2010	BGC	Preliminary geotechnical and hazards
ML/ARD	2011	SRK	Preliminary ML and ARD
Metallurgy	2012	SGS	Preliminary mineralogy and metallurgy
Geotechnical and Hydrology	2012	BGC	Preliminary rock mechanics and hydrology
Geotechnical and Hydrology	2015, 2016, 2018, 2019, 2021, 2022	SRK	Arctic PFS and FS slope design
ML/ARD	2015, 2016, 2017, 2018, 2019	SRK	Static kinetic tests and ABA update - ongoing

Work Completed	Year	Details	Focus
Metallurgy	2015, 2016, 2017, 2018, 2019, 2021	SGS, ALS	Cu-Pb Separation Testwork; Flotation and Variability Testwork; SAG Mill Comminution (SMC) Testwork, filtration Testwork, thickener Testwork, and tailings settling testing
Arctic Project Evaluation			
Resource Estimation	2008	SRK	Resource estimation
PEA	2011	SRK	PEA - Underground
	2012	Tetra Tech	PEA – Open Pit
PFS	2018	Ausenco	Pre-Feasibility Study
FS	2020	Ausenco	Feasibility Study

Note: SWIR = short wave infrared; LiDAR = light detection and ranging; ML = metal leaching; BGC = BGC Engineering Inc.; SGS = SGS Canada; ALS = ALS Metallurgy; PEA = preliminary economic assessment.

7.1.1 Grids and Surveys

Survey and data capture during the Kennecott's programs used the UTM coordinates system Zone 4, NAD27 datum. In 2010, NovaGold converted all historical geology and topographic data for the Arctic deposit into the NAD83 datum for consistency. At that time NovaGold contracted WH Pacific, Inc. (WHPacific) to re-establish project-wide survey control and benchmarks for the Arctic deposit. Current Mineral Resource estimate and geologic models use topography completed in 2010 by PhotoSat Inc. The resolution of the satellite imagery used was at 0.5 m and a 1 m contour map and digital elevation model were generated.

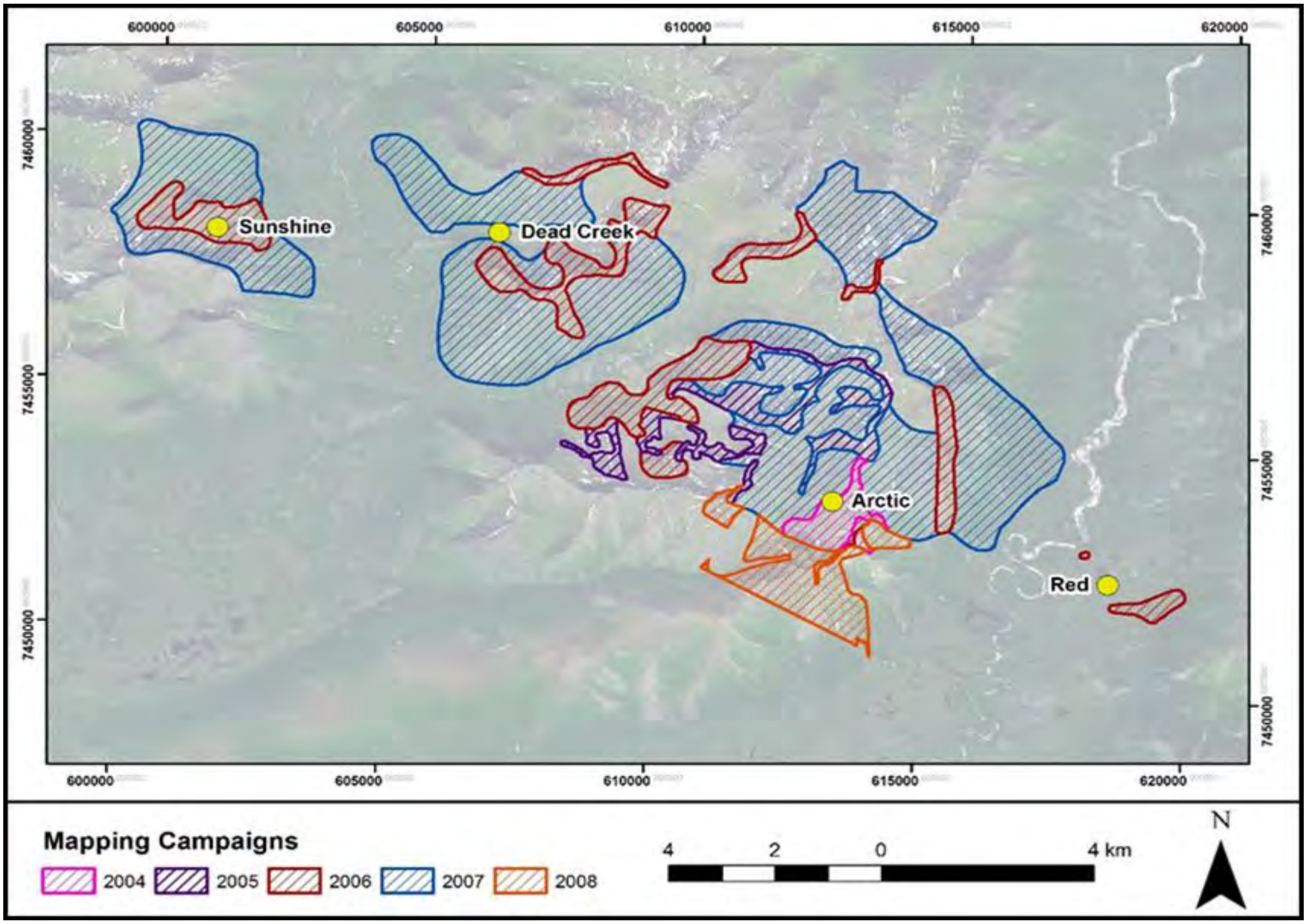
Trilogy retained WHPacific (and sub-consultant Quantum Spatial, Inc.) to conduct an aerial LiDAR survey over the Upper Kobuk area during 2015. Due to scheduling difficulties and poor weather conditions only 70% of the survey was completed in 2015. The remaining 30% of the aerial survey, as well as the final post-processing work, was completed between June and October 2016.

7.1.2 Geological Mapping

NovaGold focused its exploration mapping efforts on an area covering approximately 18 km of strike length of the permissive Ambler Sequence rocks of the Schist belt stratigraphy. This area is centred on the Arctic deposit and covers the thickest portion of the Ambler Sequence rocks. The area covers many of the most notable mineralized occurrences including the Red prospect east of the Kogoluktuk River, the Arctic deposit, and the nearby occurrences at the West Dead Creek and Dead Creek prospects, and the CS, Bud and Sunshine prospects west of the Shungnak River.

In 2004, mapping focused on the surface geology in and around the Arctic deposit while exploration in 2005 extended the Ambler Sequence stratigraphy to the west. In 2006 with expansion of the exploration focus to encompass the immediate district and to support a major TDEM geophysical program, mapping was extended to include the area between the Sunshine prospect on the west and the Red prospect on the east. Figure 7-1 shows areas mapped by successive campaigns, which resulted in the generalized geological interpretation shown in Figure 6-4.

Figure 7-1: Mapping Campaigns in and around the Arctic Deposit



Source: Trilogy Metals, 2019.

SRK was contracted in 2015 to create a structural geology model primarily based on brittle structures of the Arctic deposit for pit design and mine scheduling. The majority of the structural mapping took place along the north-south trending Arctic Ridge, and along the northwest trending ridge above a cirque to the south of the deposit, both of which provided the greatest exposure.

Geologic and structural mapping were completed by Trilogy geologists during the 2016 field season. The objectives of the mapping project were threefold; 1) to ground-truth the northeast and north-south trending fault structures identified by SRK in 2015 and to otherwise support SRK's 2016 geotechnical mapping efforts, 2) field check the outcrops mapped in 2006 and 2008 recorded in the current GIS database, and 3) determine the nature of the Warm Springs Fault by mapping in the immediate hanging wall of this apparent structural feature. The first objective was successfully accomplished and the pending SRK geotechnical structural model was considered to be robust. The two other objectives were partly met during the short field season.

Geological mapping was completed by Ambler Metals geologists during the 2021 and 2022 field seasons. The mapping and geochemical sampling occurred at numerous prospects throughout the Ambler Belt, including the Snow, Ambler, Nora, Sunshine, Centre of the Universe, Pipe, DH, Cliff, 98-9 and BT prospects. The purpose of geologic mapping and sampling was to increase mapped extents at known prospects, collect mineralized and non-mineralized surface samples for various analyses and evaluate VTEM anomalies identified from the 2019 survey.

7.1.3 Geochemistry

Soil and silt geochemical sampling were used to target many of the VMS prospects in the Ambler Sequence particularly in the core area around the Arctic deposit. Between 2005 and 2007, NovaGold collected 2,272 soils and 278 silt samples. Much of the reconnaissance soil sampling used gridding layouts of 200 m lines and 50 m sample intervals oriented perpendicular to stratigraphy. Results of the sampling were used to refine areas for geophysical surveying, and to define drill targets.

Ambler Metals completed soil sampling on numerous prospect areas within the Ambler VMS Belt during the 2021 and 2022 field seasons. In 2021, 731 soil samples were collected at Dead Creek, West Dead Creek, JA Creek, and over four VTEM (anomalies 26-29). Collection points spacing varied by prospect but were generally at 30 m spacing along parallel lines 100-200 m apart oriented to cross-cut stratigraphy. The 2021 program used the same analysis methods as the NovaGold programs.

In 2021, soil and silt samples were submitted directly to either ALS Minerals in Fairbanks (a division of ALS Global, formerly ALS Chemex) or Alaska Assay Labs in Fairbanks for sample preparation. The samples were dried and sieved to 80 mesh and forwarded to ALS Minerals for analysis. The samples were analysed using the ME-ICP61 method and a four acid near total digestion with 27 elements measured.

7.1.4 Geophysics

During NovaGold's tenure, the geophysical surveys were largely focused on ground and downhole EM methods to follow-up on the 1998 DIGHEM airborne EM survey conducted by Kennecott.

From 2005 to 2007, NovaGold conducted TDEM surveys and completed 21 different loops targeting the Arctic deposit, extensions to the Arctic deposit and a series of DIGHEM airborne anomalies in and around known prospects and permissive stratigraphy. Table 7-2 summarizes the TDEM loops and locations.

Frontier Geosciences of Vancouver, BC completed all of the geophysical programs using a Geonics PROTEM 37 transmitter, a TEM-57 receiver and either a single channel surface coil or a three component BH43-3D downhole probe.

Table 7-2: TDEM Loops and Locations

Area	2005	2006	2007
Arctic	1	-	6
COU	1	3	-
Dead Creek	-	4	-
Sunshine	-	2	-
Red	-	1	-
Tom	-	1	-
Kogo/Pipe	-	2	-
Total	**2**	**13**	**6**

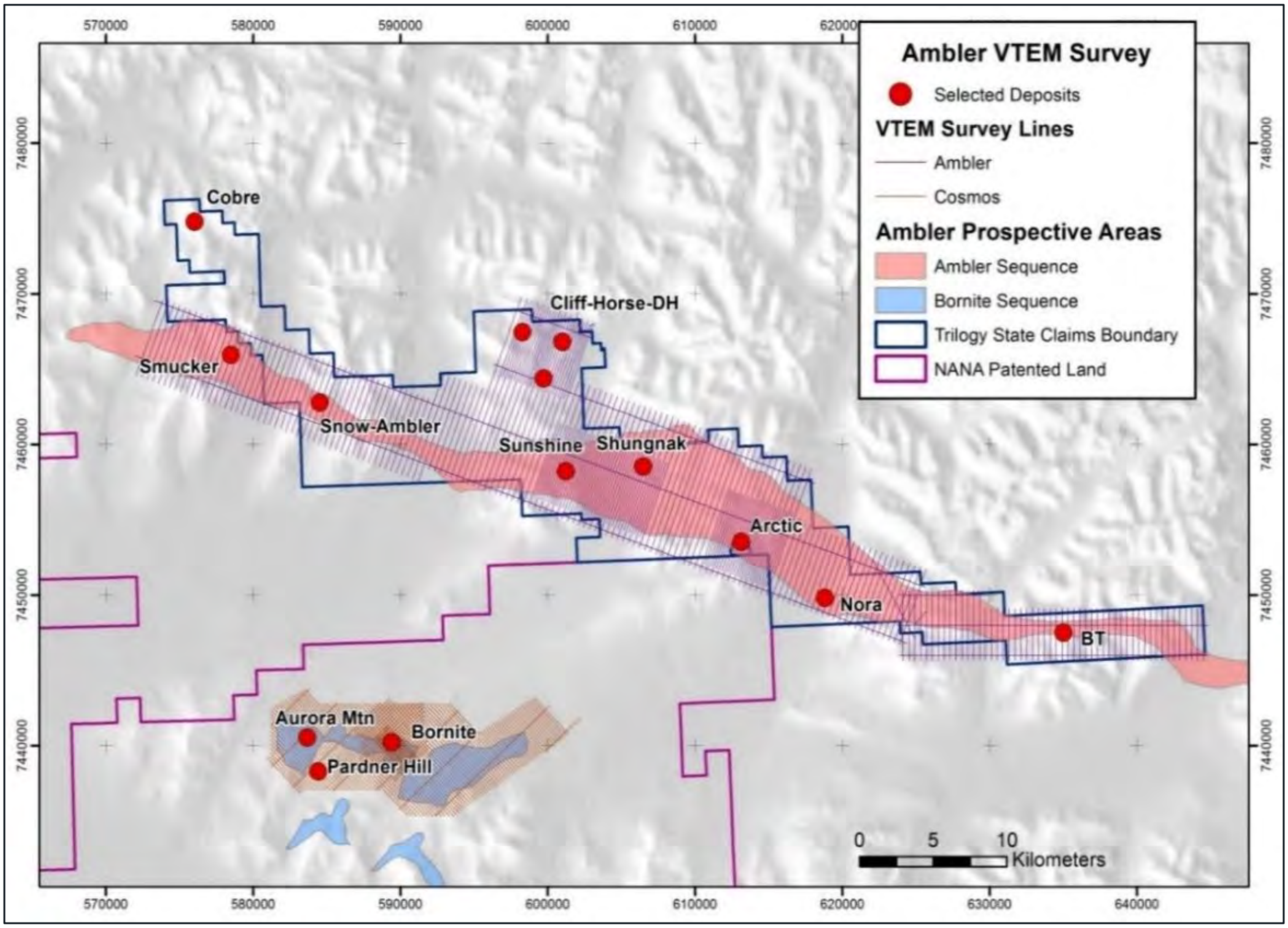
In addition to the TDEM surveys, Frontier Geosciences surveyed four drill holes (AR05-89, AR07-110, AR07-111, and AR07-112). All of the drill holes produced off-hole anomalies, notably AR07-111, which showed evidence of a strong EM conductor north of the hole. Follow-up of this conductor is warranted.

In 2019, Trilogy contracted GeoTech Ltd. of Aurora, Ontario to complete VTEM Plus (Versatile Time Domain Electromagnetic) and ZTEM (Z-Axis Tipper Electromagnetic) airborne helicopter geophysical surveys over the Cosmos Hills and the Ambler VMS belt. These survey methods are a significant upgrade over the previous DIGHEM survey flown by Kennecott in 1998 over the VMS belt and the DIGHEM survey flown by NovaGold over the Bornite Sequence in 2006 due to the greater resolution and deeper penetration ability. The magnetic field was also measured using a cesium vapor sensor, though radiometric data were not collected due to snow cover.

The program was designed, managed, and results interpreted by Resource Potential, a geophysical consulting company based in Perth, Australia.

The VTEM survey was flown at a 400 m line spacing over the Ambler VMS Belt along lines oriented at 20°-200° for the western portion of the belt and along north-south lines for the eastern portion (see Figure 7-2). In-fill lines at 200 m spacing were flown over the Arctic, Sunshine, and Horse-Cliff areas to provide greater resolution in those high priority areas. Tie lines at ~4 km spacing were flown perpendicular to the EM flight lines to provide control for the magnetic survey.

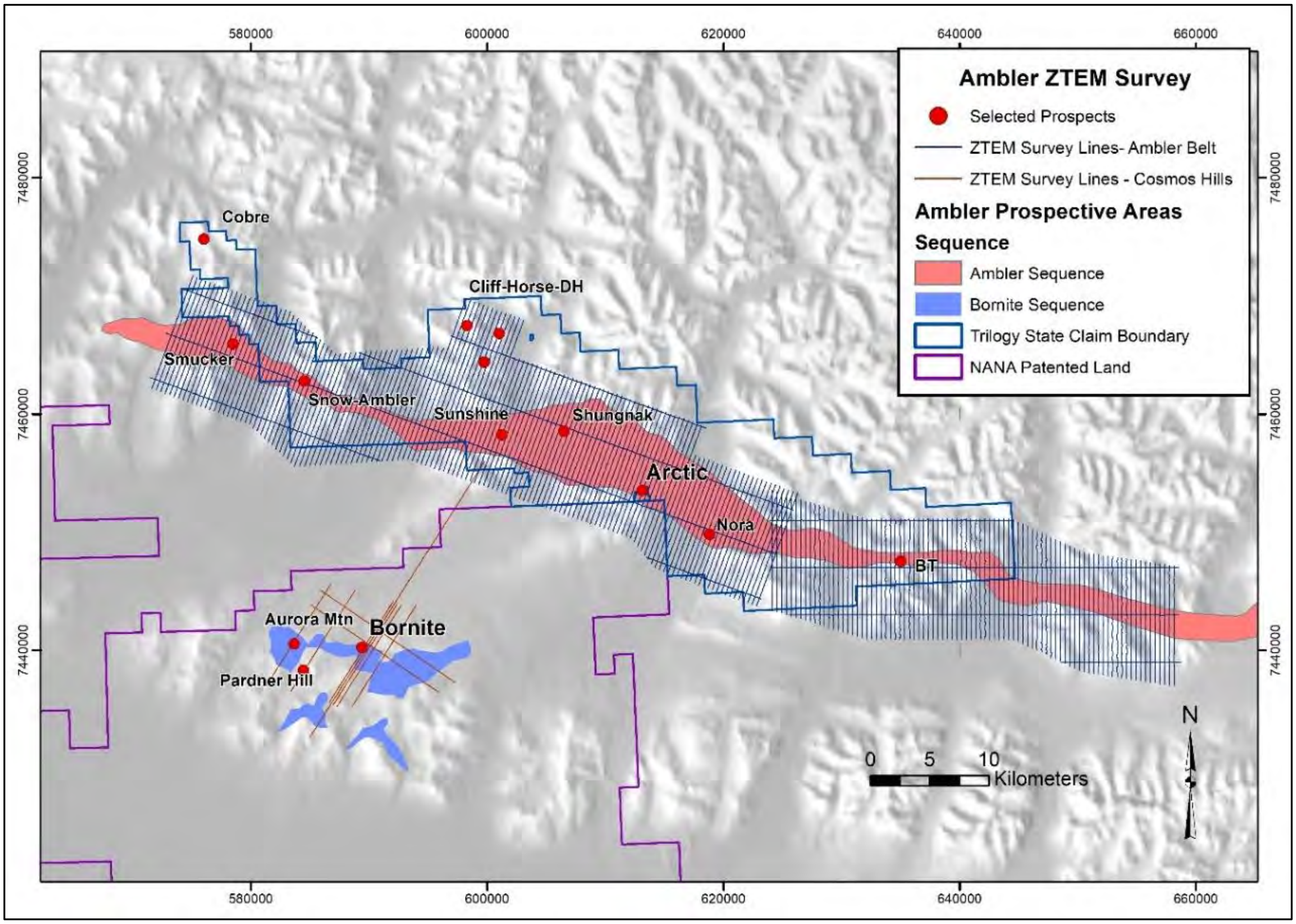
Figure 7-2: VTEM Flight Lines Over the Ambler Belt and Cosmos Hill Prospective Areas



Source: Ambler Metals, 2019.

The ZTEM survey was flown along lines with the same orientation as the VTEM survey at 400m line spacing, with tie lines at every 4 km. (Figure 7-3). Resource Potential re-processed the data from GeoTech and provided a 3D EM block model and both plan view depth slices and sectional EM images for the ZTEM survey. Numerous anomalies identified from both the VTEM and ZTEM surveys need further evaluation.

Figure 7-3: ZTEM Flight Lines Over the Ambler VMS Belt and the Bornite Deposit



Source: Trilogy Metals, 2019

7.1.5 Petrology, Minerology and Research Studies

Trilogy supported a series of academic studies of the Arctic deposit. In 2009, Danielle Schmandt completed an undergraduate thesis entitled "Mineralogy and Origin of Zn-rich Horizons within the Arctic Volcanogenic Massive Sulphide deposit, Ambler District, Alaska" for Smith College. The Schmandt thesis focused on a structural and depositional reconstruction of the Arctic deposit with the goal of locating the hydrothermal vents to aid in exploration.

Bonnie Broman, a Trilogy geologist, completed a Master of Science thesis in 2014 at the University of Alaska Fairbanks, focusing on the nature and distribution of the silver-bearing mineral species within the Arctic deposit. The thesis is titled "Metamorphism and Element Redistribution: Investigations of Ag-bearing and associated minerals in the Arctic Volcanogenic Massive Sulphide deposit, SW Brooks Range, NW Alaska."

7.2 Drilling

7.2.1 Overview

Drilling at the Arctic deposit and within the Ambler Mining District has been ongoing since the initial discovery of mineralization in 1966. Approximately 67,639 m of drilling has been completed within the Ambler Mining District, including 55,038 m of drilling in 285 drill holes sat the Arctic deposit or on potential extensions in 32 campaigns spanning 56 years. Drilling outside the Arctic deposit area is discussed in Section 7.2.2.

All of the drill campaigns at the Arctic deposit utilize diamond drill holes and have been run under the supervision of either: 1) Kennecott and its subsidiaries (BCMC), 2) Anaconda, or 3) Trilogy (formerly, NovaCopper) and its predecessor company, NovaGold and its successor joint venture Company, Ambler Metals. Table 7-3 summarizes operators, campaigns, holes, and meters drilled on the Arctic deposit. All drill holes listed in Table 7-3 - except 11 geotechnical holes in 2017, 24 geotechnical holes drilled in 2018, 8 holes from the 2021 program and 34 holes from the 2022 program, for which assay results were not available - were used to validate the geologic model.

Table 7-3: Companies, Campaigns, Drill Holes and Meters Drilled at the Arctic Deposit

Year	Company	No. of Holes	Metres
1967	BCMC	7	752
1968	BCMC	18	3,836
1969	BCMC	3	712
1970	BCMC	3	831
1971	BCMC	1	257
1972	BCMC	1	407
1973	BCMC	2	557
1974	BCMC	3	900
1975	BCMC	26	4,942
1976	BCMC, Anaconda	10	805
1977	BCMC, Anaconda	4	645
1979	BCMC, Anaconda	3	586
1980	Anaconda	1	183
1981	BCMC, Anaconda	2	632
1982	BCMC, Anaconda	5	677
1983	BCMC	1	153
1984	BCMC	2	253
1986	BCMC	1	184
1998	Kennecott	6	1,523
2004	NovaGold	11	2,996
2005	NovaGold	9	3,393
2007	NovaGold	4	2,606
2008	NovaGold	14	3,306
2011	NovaGold	5	1,193
2015	NovaCopper	14	3,055

Year	Company	No. of Holes	Metres
2016	NovaCopper	13	3,058
2017	Trilogy Metals	5	790
2018	Trilogy Metals	24	906
2019	Trilogy Metals	9	2,433
2021	Ambler Metals	18	4,131
2022	Ambler Metals	47	8,376
Total	**-**	**272**	**55,078**

Trilogy (formerly, NovaCopper) and its predecessor company, NovaGold and its successor joint venture Company, Ambler Metals, drilled 36,243 m in 173 holes targeting the Arctic deposit and several other prospects within the Ambler Schist belt. Table 7-4 summarizes all the NovaGold/Trilogy/Ambler Metals tenure drilling on the Project.

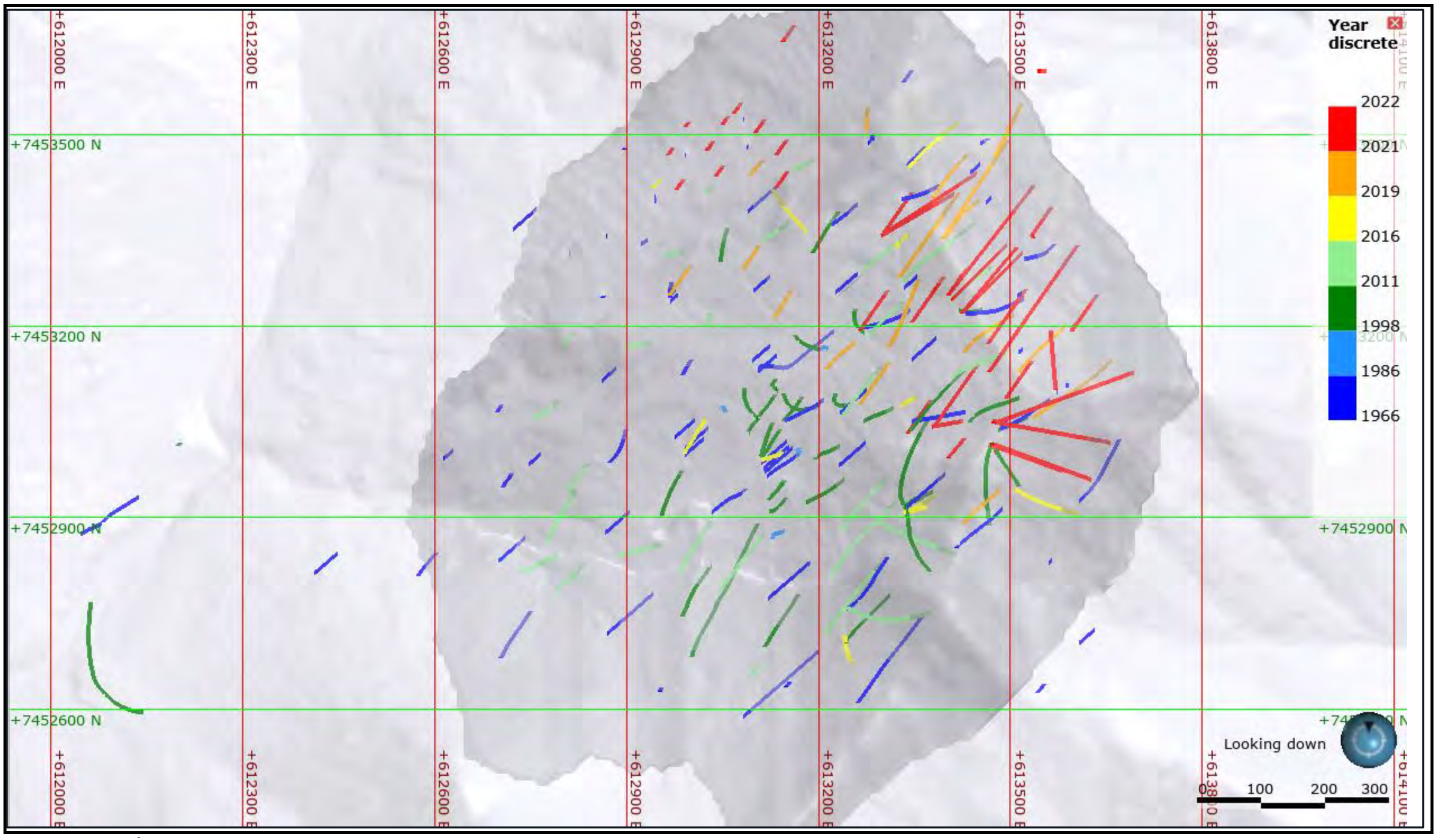
Table 7-4: Summary of NovaGold/Trilogy/Ambler Metals Arctic Deposit Drilling

Year	Metres	No. of Drillholes	Sequence	Purpose of Drilling
2004	2,996	11	AR04-78 to 88	Deposit scoping and verification
2005	3,393	9	AR05-89 to 97	Extensions to the Arctic deposit
2006*	3,010	12	AR06-98 to 109	Property-wide exploration drilling
2007	2,606	4	AR07-110 to 113	Deep extensions of the Arctic deposit
2008*	3,306	14	AR08-114 to 126	Grade continuity and metallurgy
2011	1,193	5	AR11-127 to 131	Geotechnical studies
2012*	1,752	4	SC12-014 to 017	Exploration drilling – Sunshine
2015	3,055	14	AR15-132 to 145	Geotechnical-hydrogeological studies, resource infill
2016	3,058	13	AR16-146 to 158	Geotechnical-hydrogeological studies, resource infill
2017**	790	5	AR17-159 to 163	Ore sorting studies
2018	906	24	GT18-AR-01 to 19	Geotechnical studies for site facilities
			MS18-AR-01 to 05	
2019*	2433	9	AR19-0164 to 172	Geotechnical and hydrogeological studies for 2020 FS
	1,357	6	SC19-018 to 023	Exploration drilling – Sunshine
2021**	4,131	18	AR21-0173 to 190	Geotechnical and hydrogeological studies for 2021 FS and In-fill
	2,414	6	Various holes	Property-wide exploration drilling
2022***	8,376	47	AR22-0191 to 237	Geotechnical and infill drilling
	1,644	5	Various holes	Property-wide exploration drilling
Total	**46,420**	**206**	-	-

Notes:
*Drilling in 2006, 2012, and some holes in 2019 and 2021 targeted exploration targets elsewhere in the VMS belt.
**Holes drilled in 2018 are not included in the current resource estimate as they were completed geotechnical site facilities studies.
***8 holes drilled in 2021 and 2022 are not included in the current resource database and were used to validate the resource model, as the results were not available at the time of updating the resource database.

Geotechnical holes drilled in 2017 and 2018 are not included in the current resource estimate as they were completed for geotechnical site facilities studies. Assays for 8 out of 18 holes drilled at Arctic in 2021 and 34 out of 34 holes drilled in 2022 were not available for use in grade estimation in the current mineral resource estimate because the assays were not available at the time of resource estimation. Figure 7-4 shows the locations of drill holes in the vicinity of the Arctic deposit.

Figure 7-4: Plan Map of Drill Holes coloured by Year in the Vicinity of the Arctic Deposit. Dark Grey is Limits of 2021 Conceptual Pit



Source: Wood, 2022.

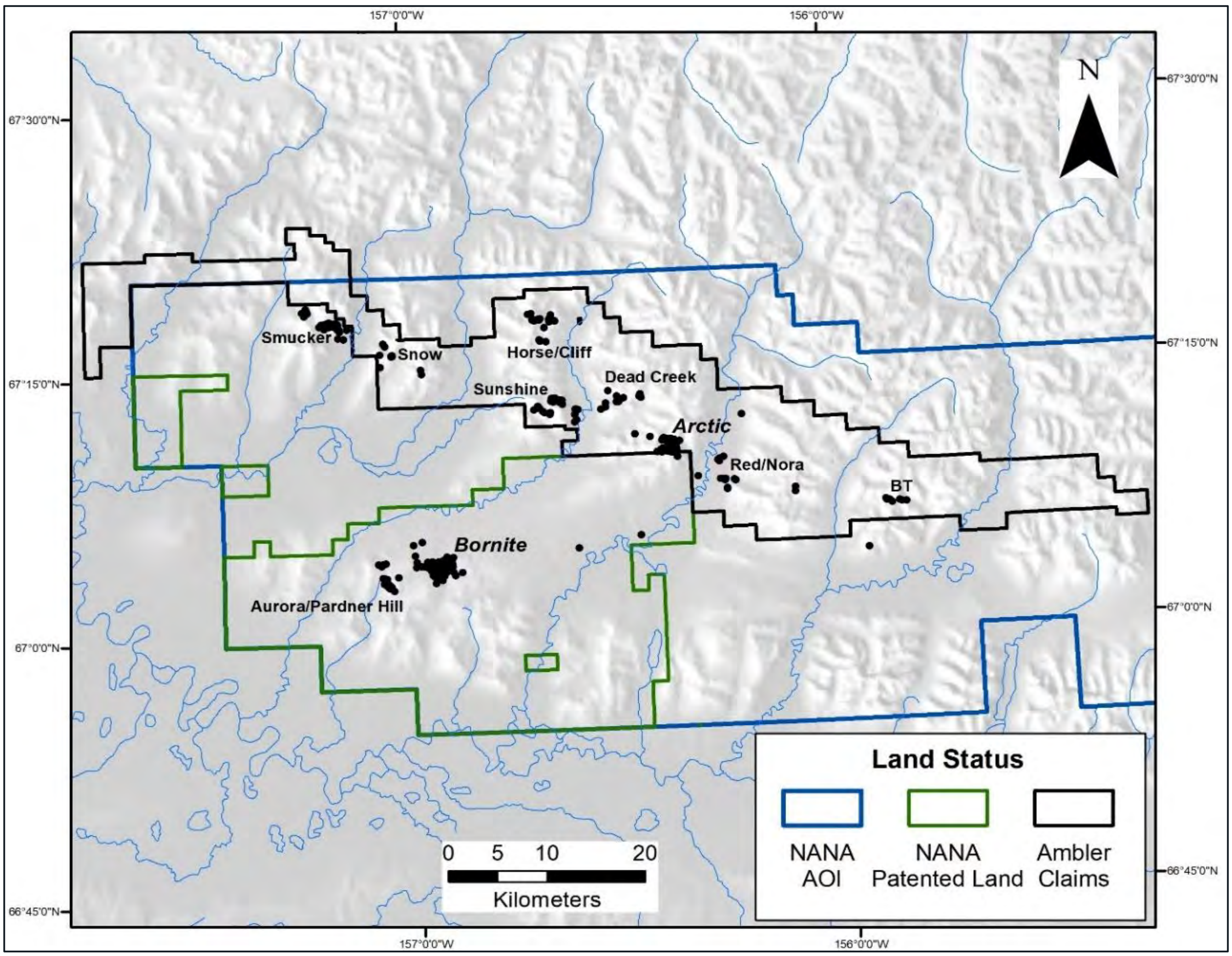
Significant exploration drilling has been carried out elsewhere within the UKMP targeting numerous occurrences along the Ambler Schist belt. Table 7-5 summarizes the drilling within the UKMP outside of the Arctic deposit.

Figure 7-5 shows the locations of known major prospects and drill collar locations for the Ambler Mining District. Note that none of these drill holes are located within the current Arctic Project and are therefore not included in the resource calculation.

Table 7-5: Drill, Meterage and Average Drill Depth for Ambler Sequence VMS Targets

Area	Drill Holes (number)	Metres	Average Depth (m)
Dead Creek/West Dead Creek	21	3,470	165
Sunshine/Bud	42	8,468	201
Snow/Trilogy	11	1,527	139
Horse/Cliff/DH	22	2,277	104
Red/Nora/BT	18	2,399	133
Total	**114**	**18,141**	**148**

Figure 7-5: Collar Locations and Principal Target Areas – Ambler Mining District



Source: Ambler Metals, 2022.

Exploration in 2006 investigated a series of geophysical anomalies in the central portion of the Ambler Schist belt near the Arctic deposit. In 2012, Trilogy drilled an additional four holes totalling 1,752 m to explore the down dip extension of the Sunshine prospect. Twelve holes totalling 3,100 m were drilled. In 2012, Trilogy drilled an additional four holes totalling 1,752 m to explore the down dip extension of the Sunshine prospect. In 2019, Trilogy drilled 10 holes totalling 3,823 m at four regional targets. Regional drilling is summarized in Table 7-6 and Table 7-7 and in Figure 7-6 to Figure 7-11.

Table 7-6: Ambler Metals Exploration Drilling – Ambler Schist Belt

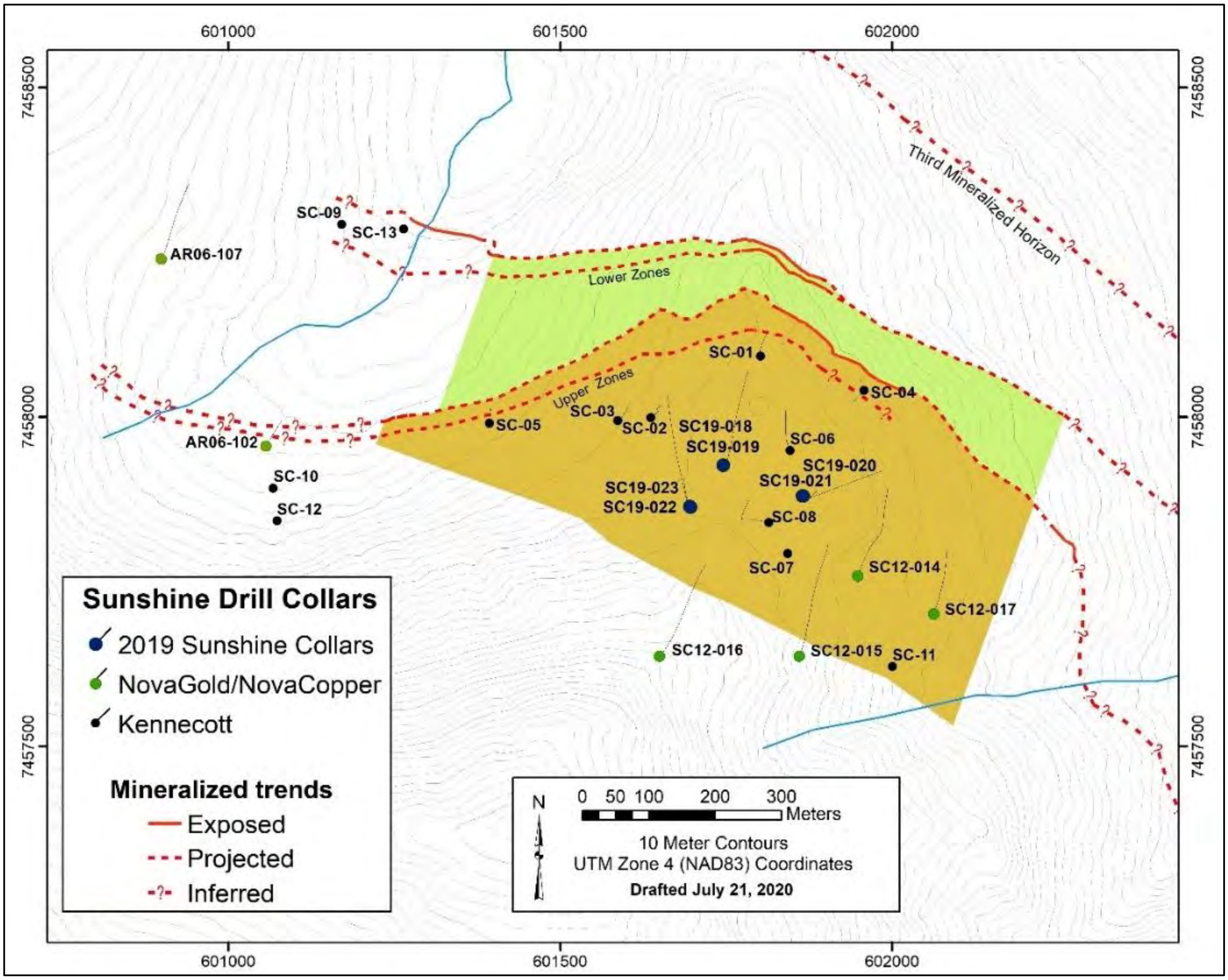
Hole ID	Area	Target	UTM East	UTM North	Azimuth (°)	Dip (°)	Depth (m)
AR06-98	COU	EM Anomaly	609490	7454374	0	-90	712.6
AR06-99	98-3	EM Anomaly	610111	7458248	0	-90	420.0
AR06-100	98-3	EM Anomaly	609989	7458633	0	-90	225.6
AR06-101	Red	EM Anomaly	618083	7451673	0	-90	141.7
AR06-102	Sunshine	West Extension	601176	7457834	30	-65	97.8
AR06-103	Red	EM Anomaly	618073	7451806	0	-90	209.7
AR06-104	Red	EM Anomaly	617926	7451693	0	-90	183.2
AR06-105	Red	EM Anomaly	618074	7451537	0	-90	136.6
AR06-106	Red	EM Anomaly	618083	7451677	310	-60	185.0
AR06-107	Sunshine	West Extension	601018	7458119	30	-60	294.4
AR06-108	Dead Creek	Downdip Extension	607618	7458406	0	-90	289.0
SC12-014	Sunshine	Sunshine Extension	601948	7457759	20	-57	537.8
SC12-015	Sunshine	Sunshine Extension	601860	7457637	20	-65	477.0
SC12-016	Sunshine	Sunshine Extension	601649	7457637	45	-77	386.2
SC12-017	Sunshine	Sunshine Extension	602063	7457701	20	-60	351.1
SC19-018	Sunshine	Sunshine Infill	601748	7457922	15	-52	296.3
SC19-019	Sunshine	Sunshine Infill	601748.2	7457923	0	-90	160.6
SC19-020	Sunshine	Sunshine Infill	601863.2	7457873	70	-48	230.4
SC19-021	Sunshine	Sunshine Infill	601862.2	7457872	70	-48	212.8
SC19-022	Sunshine	Sunshine Infill	601692.2	7457866	345	-80	203.6
SC19-023	Sunshine	Sunshine Infill	601691.6	7457868	345	-45	253.0
NEN22-001	98-9	Surface occurrence	615906	7456176	90	-55	223.72
NEN22-002	98-9	Surface occurrence	615907	7456178	90	-50	376.12
NEN22-003	98-9	Surface occurrence w/EM anomaly	615881	7456028	109	-45	451.1
NEN22-004	98-9	Surface occurrence w/EM anomaly	615880	7456029	140	-45	348.09
SNO21-001	Snow	Extension of Snow w/coincident EM Anomaly	584650	7462550	340	-80	529.74
SNO21-002	Snow	Extension of Snow w/coincident EM Anomaly	584650	7462551	11	-45	413
XAR21-001	NE Arctic	Extension of Arctic	613803	7453218	135	-45	448.67
XAR21-002	NE Arctic	Extension of Arctic	613805	7453217	135	-80	291.39
XAR21-003	SE Arctic	EM Anomaly	614750	7451885	340	-55	401.42
XAR21-004	SE Arctic	EM Anomaly	614894	7451990	32	-57	339.85
AR06-109	Dead Creek	Stratigraphy Hole	608187.7	7458314.5	0	-90	114.91

Table 7-7: 2019-2021 Regional Prospect Drilling Significant Intercepts

Sunshine Drill Results											
1.5% CuEq Cut-off											
	Hole	**From (m)**	**To (m)**	**Length_m**	**CuEq%**	**Cu_%**	**Zn_%**	**Pb_%**	**Au_g/t**	**Ag_g/t**	**Ba_pct**
Sunshine	SC19-018	139.52	144.76	5.24	3.93	2.08	3.13	0.63	0.154	41.643	-
	SC19-018	238.72	245.06	6.34	2.38	1.63	1.45	0.09	0.070	13.380	-
	SC19-018	247.86	255.06	7.20	1.69	0.72	2.18	0.21	0.030	6.640	-
	SC19-018	260.46	261.60	1.14	1.71	1.53	0.35	0.01	0.027	3.670	-
	SC19-019	57.00	66.14	9.14	3.95	3.02	1.42	0.27	0.137	24.651	-
	SC19-019	68.85	72.15	3.30	2.82	1.68	1.77	0.47	0.121	27.573	-
	SC19-019	98.81	102.54	3.73	5.51	4.74	0.97	0.13	0.153	28.957	-
	SC19-019	122.36	125.40	3.04	1.63	0.75	1.40	0.35	0.077	21.020	-
	SC19-019	138.17	146.05	7.88	5.23	2.23	5.62	1.10	0.180	46.947	-
	SC19-020	176.37	179.74	3.37	6.54	4.15	3.42	0.48	0.258	74.354	-
	SC19-020	188.55	190.10	1.55	3.77	1.43	1.65	0.40	0.061	23.300	-
	SC19-020	204.15	209.09	4.94	5.77	4.47	3.42	0.01	0.002	0.117	-
	SC19-020	219.30	221.98	2.68	3.87	3.70	0.44	0.00	0.002	0.396	-
	SC19-021	146.62	156.28	9.66	6.10	3.93	3.00	0.77	0.216	73.104	-
	SC19-022	114.12	115.47	1.35	5.90	2.89	4.87	1.41	0.172	68.300	-
	SC19-022	130.40	134.61	4.21	1.85	0.34	2.28	1.07	0.066	30.634	-
	SC19-022	143.73	159.01	15.28	3.08	1.35	2.91	0.78	0.158	32.584	-
	SC19-023	163.50	168.51	5.01	2.09	0.87	1.92	0.66	0.101	24.687	-
Northeast Arctic 2021	XAR21-001	421.00	422.52	1.52	0.20	0.07	0.04	0.26	0.014	1.000	-
	XAR21-001	436.12	436.57	0.45	2.08	0.11	0.02	0.05	0.003	216.000	-
	XAR21-002	235.65	236.11	0.46	2.14	0.55	1.04	2.77	0.025	23.800	-
	XAR21-002	239.02	240.10	1.08	0.25	0.09	0.01	0.39	0.009	0.710	-
	XAR21-003	358.75	359.34	0.59	0.48	0.29	0.10	0.35	0.003	3.710	-
Snow	SNO21-001	137.87	138.21	0.34	5.85	0.35	5.47	7.04	0.321	99.600	0.16
	SNO21-001	138.21	140.04	1.83	0.81	0.04	0.96	0.83	0.088	10.500	0.21
	SNO21-001	137.87	140.04	2.17	1.60	0.09	1.67	1.80	0.125	24.460	0.20
	SNO21-001	329.59	330.70	1.11	0.92	0.61	0.15	0.51	0.035	6.260	0.16
	SNO21-002	167.16	167.48	0.32	7.49	0.23	7.25	9.13	0.391	139.000	0.08
	SNO21-002	167.48	168.26	0.78	1.07	0.09	0.89	1.22	0.147	16.350	0.34
	SNO21-002	168.26	168.62	0.36	1.22	0.02	1.35	1.66	0.072	10.200	0.29
	SNO21-002	168.62	169.16	0.54	1.49	0.08	1.55	2.06	0.075	10.000	0.29

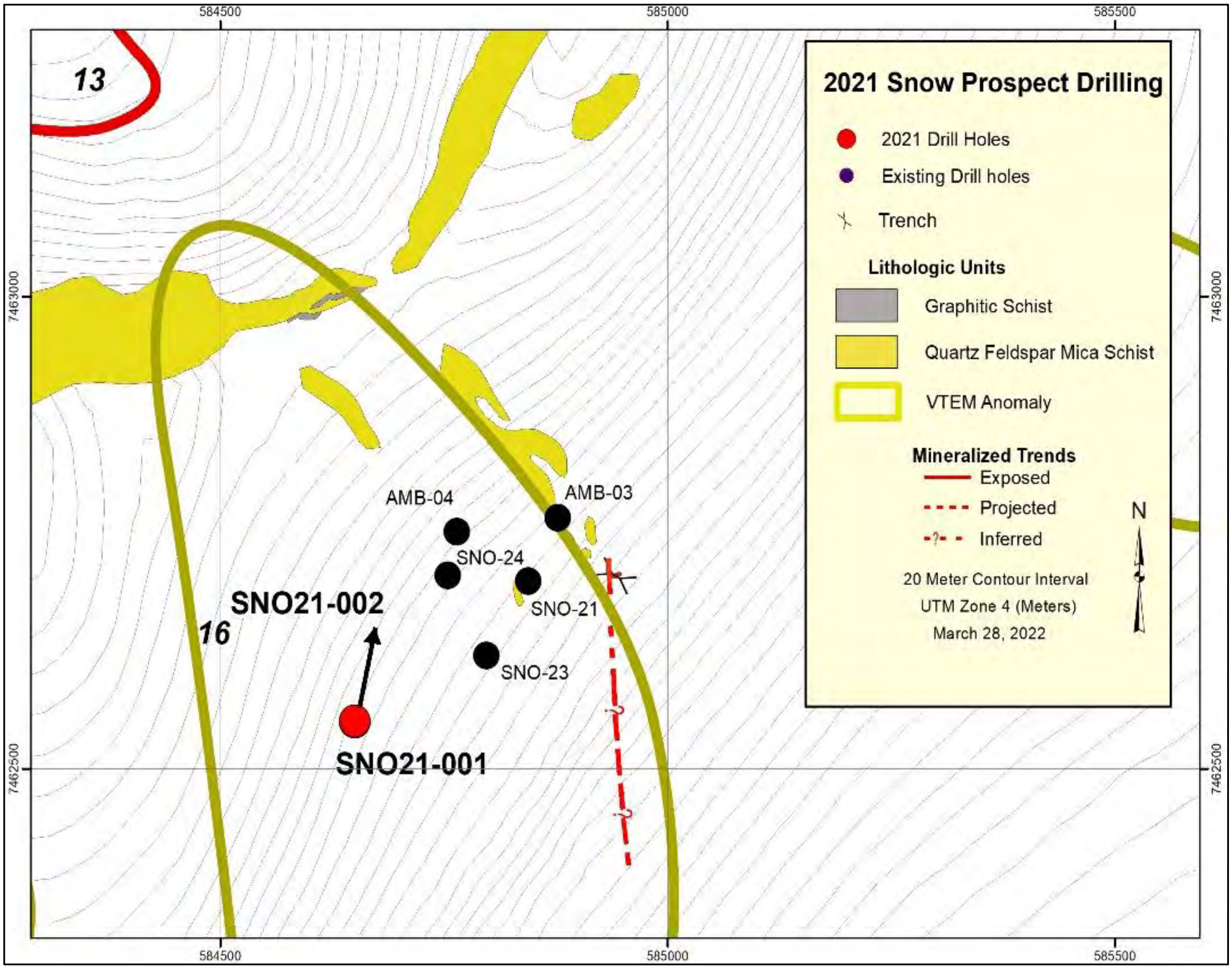
*Cu-Eq calculation uses the Trilogy's 2020 Arctic FS metal prices of $3.00 Cu, $1.10 Zn, $1.00 Pb, $18.00 Ag and $1,300 Au.

Figure 7-6: Sunshine Prospect and Drill Hole Locations



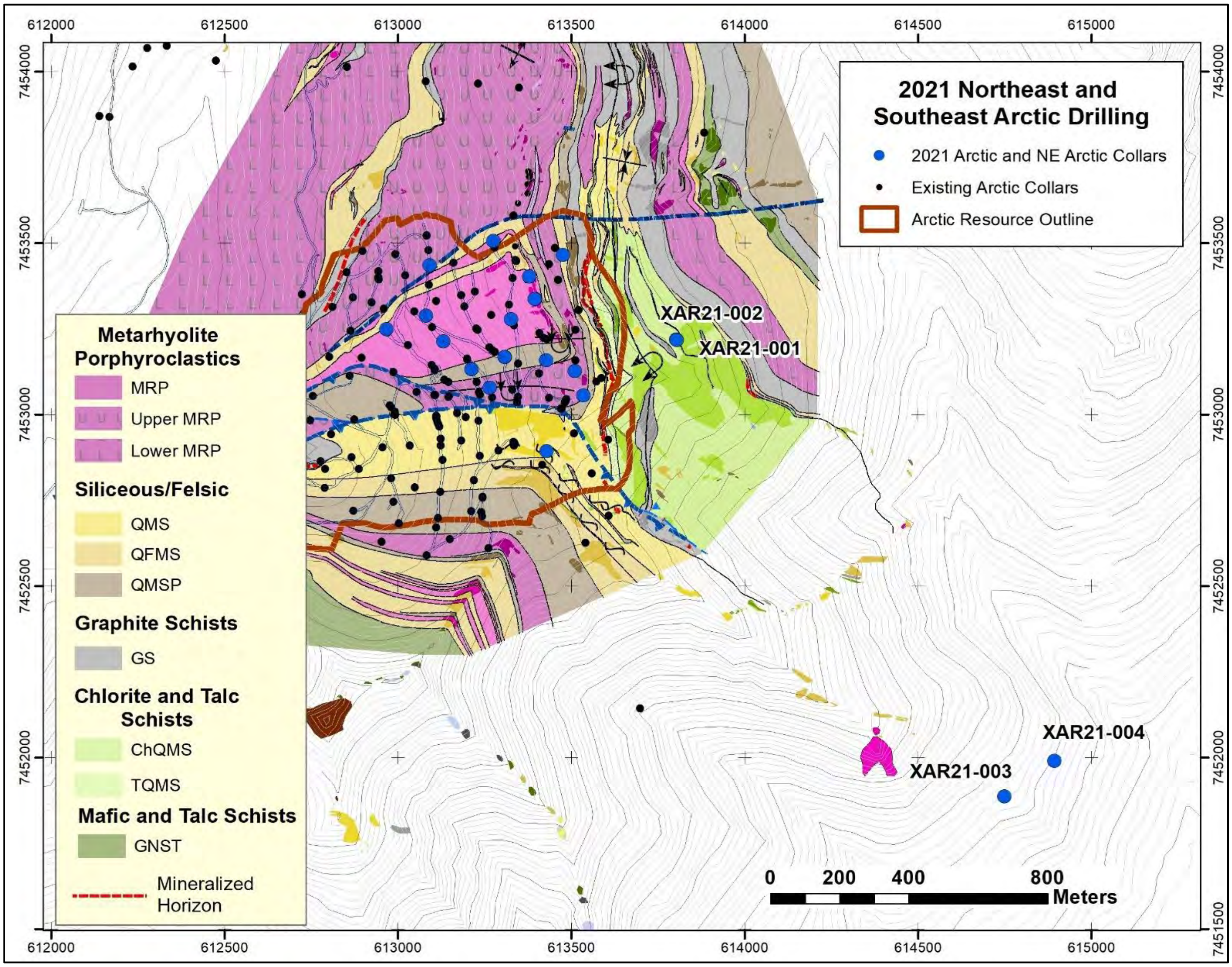
Source: Trilogy Metals, 2020.

Figure 7-7: Snow Project and Drill Hole Locations



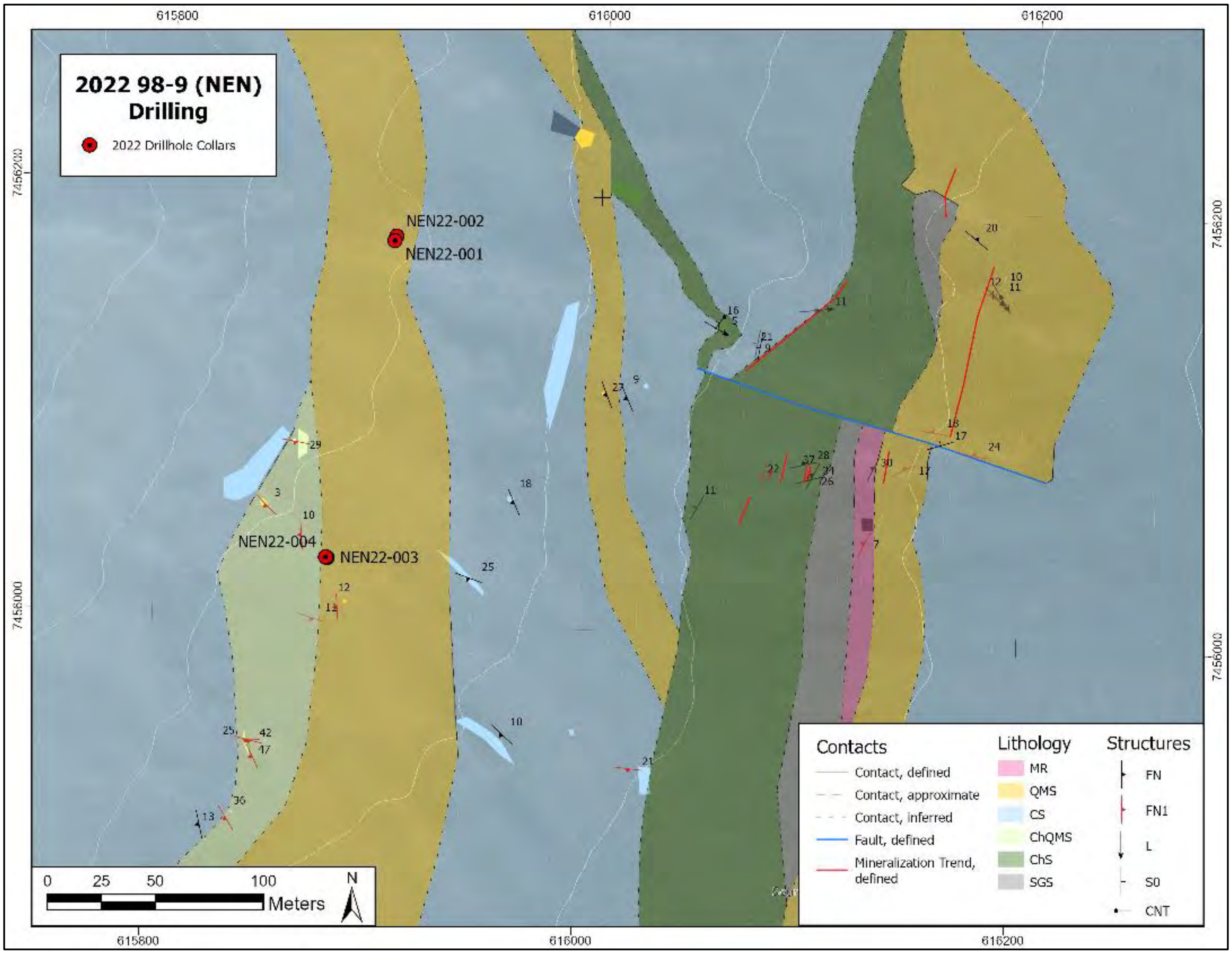
Source: Ambler Metals, 2021.

Figure 7-8: Northeast and Southeast Arctic and Drill Hole Locations



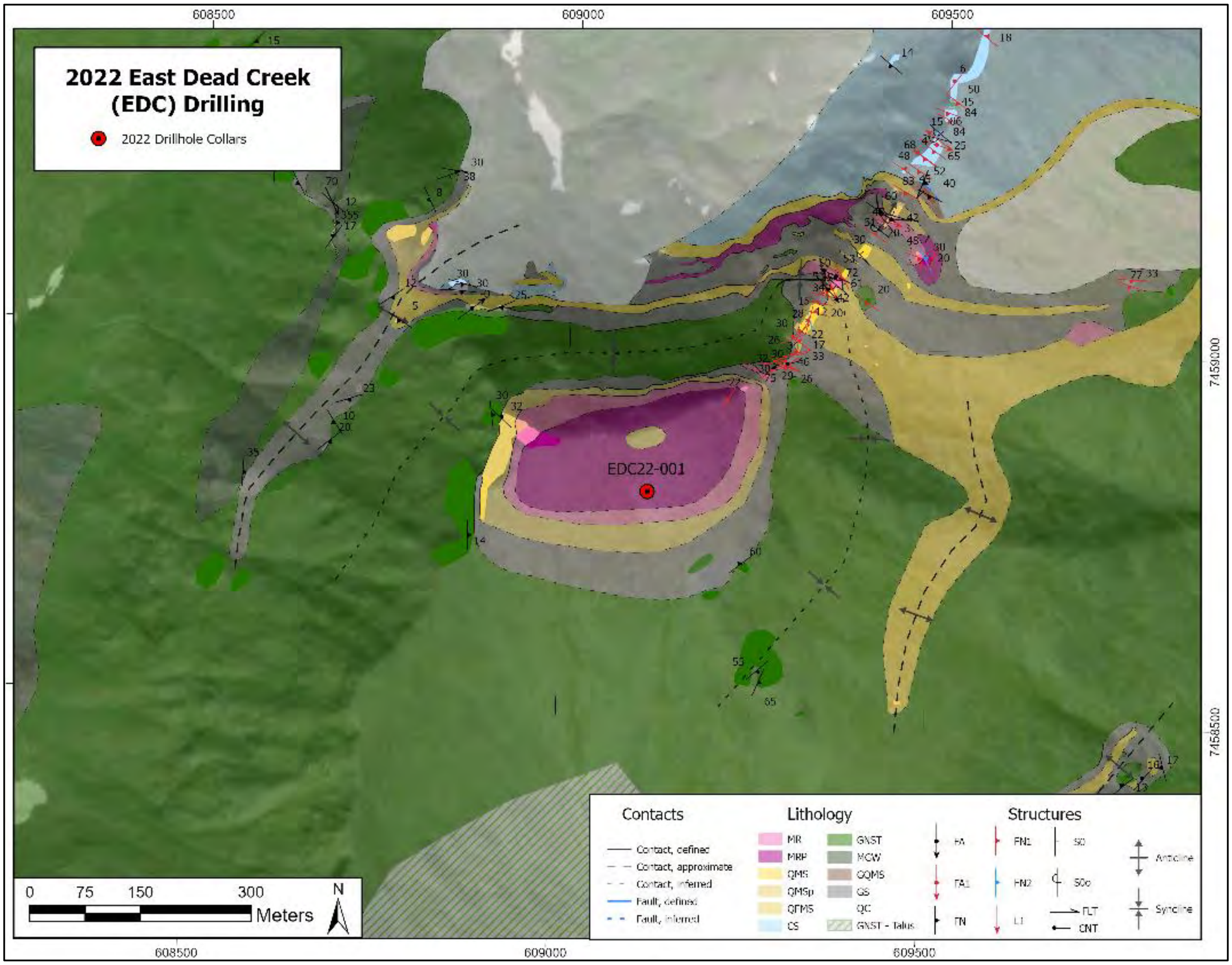
Source: Ambler Metals, 2021.

Figure 7-9: 98-9(NEN) Prospect and Drill Hole Locations



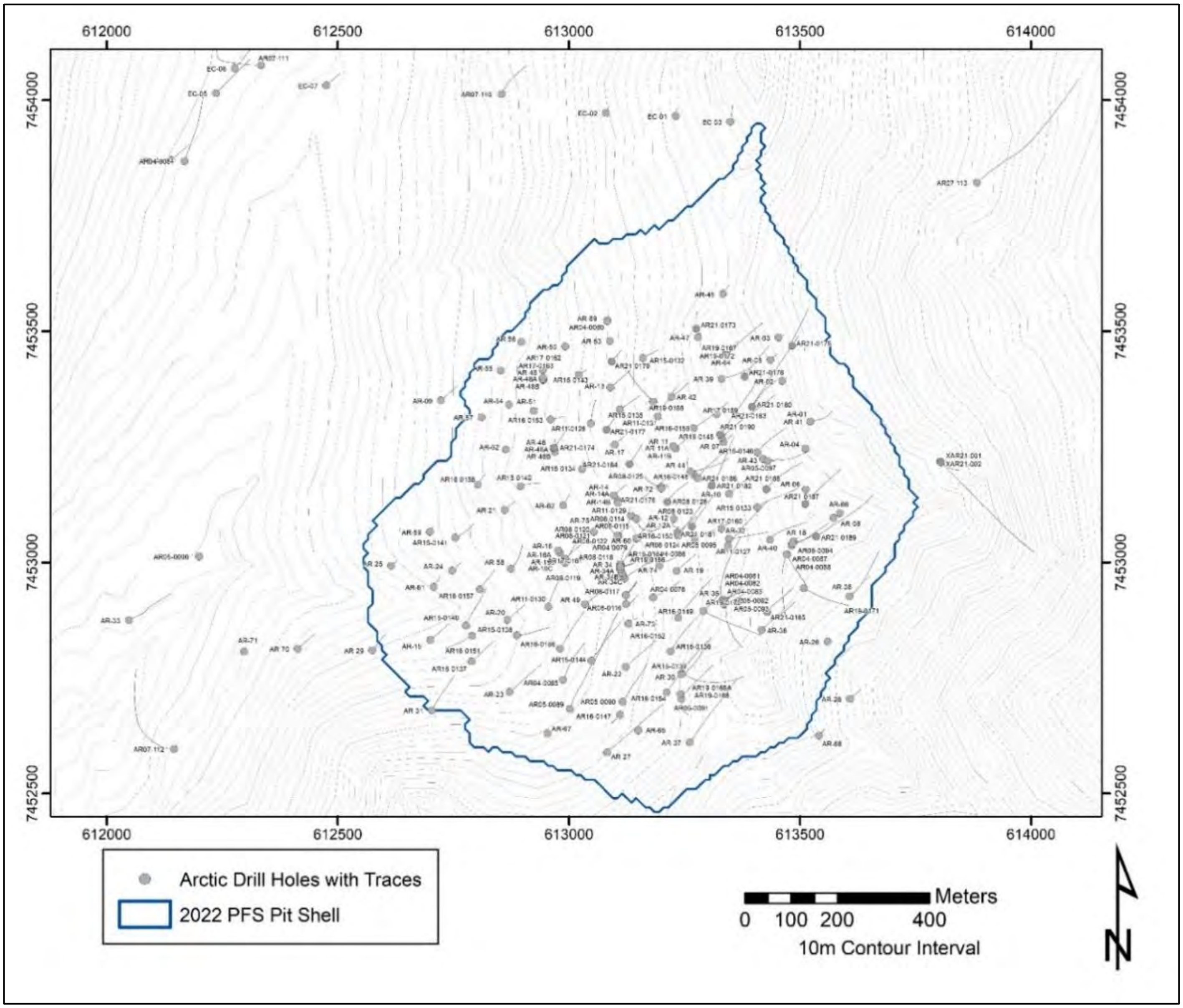
Source: Ambler Metals, 2021.

Figure 7-10: East Dead Creek Prospect and Drill Hole Locations



Source: Ambler Metals, 2021.

Figure 7-11: Plan Map of Drill Holes in the Vicinity of the Arctic Deposit



Source: Trilogy Metals, 2022.

7.2.2 Drill Companies

Over the Arctic Project's history, a relatively limited number of drill companies have been used by both Kennecott, NovaGold, Trilogy/NovaCopper and Ambler Metals at the Arctic deposit. During Kennecott's work programs, Sprague and Henwood, a Pennsylvania-based drilling company was the principal contractor. Tonto Drilling provided services to Kennecott during Kennecott's short return to the district in the late 1990s. NovaCopper and NovaGold used Boart Longyear as their only drill contractor. Trilogy has used Major Drilling and Tuug Drilling. Ambler Metals used Tuuq Drilling in 2021. Table 7-8 summarizes drill companies and core sizes used.

Table 7-8: Drill Contractors, Drill Holes, Meterage and Core Sizes by Drill Campaign at the Arctic Deposit

Year	Company	No. of Drill Holes	Metres	Core Size	Drill Contractor
1967	Bear Creek	7	752	BX	Sprague and Henwood
1968	Bear Creek	18	3,782	BX	Sprague and Henwood
1969	Bear Creek	3	712	BX	Sprague and Henwood
1970	Bear Creek	3	831	BX	Sprague and Henwood
1971	Bear Creek	2	257	BX	Sprague and Henwood
1972	Bear Creek	1	407	BX	Sprague and Henwood
1973	Bear Creek	2	557	BX	Sprague and Henwood
1974	Bear Creek	3	900	NX and BX	Sprague and Henwood
1975	Bear Creek	26	4,942	NX and BX	Sprague and Henwood
1976	Bear Creek	8	479	NXWL and BXWL	Sprague and Henwood
1977	Bear Creek	3	497	NXWL and BXWL	Sprague and Henwood
1979	Bear Creek	2	371	NXWL and BXWL	Sprague and Henwood
1981	Bear Creek	1	458	NXWL and BXWL	Sprague and Henwood
1982	Bear Creek	4	494	NXWL and BXWL	Sprague and Henwood
1983	Bear Creek	1	153	NXWL and BXWL	Sprague and Henwood
1984	Bear Creek	2	253	NXWL and BXWL	Sprague and Henwood
1986	Bear Creek	1	184	NXWL and BXWL	Sprague and Henwood
1998	Kennecott	6	1,523	HQ	Tonto
2004	NovaGold	11	2,996	NQ and HQ	Boart Longyear
2005	NovaGold	9	3,393	NQ and HQ	Boart Longyear
2007	NovaGold	4	2,606	NQ and HQ	Boart Longyear
2008	NovaGold	14	3,306	NQ and HQ	Boart Longyear
2011	NovaGold	5	1,193	NQ3 and HQ3	Boart Longyear
2015	NovaCopper	14	3,055	NQ and HQ	Boart Longyear
2016	NovaCopper	13	3,058	NQ and HQ	Boart Longyear
2017	Trilogy Metals	5	790	PQ	Major Drilling/Tuuq Drilling
2018	Trilogy Metals	24	906	PQ	Tuuq Drilling
2019	Trilogy Metals	9	2,433	PQ and HQ3	Tuuq Drilling
2021	Ambler Metals	18	4,131	PQ and HQ3	Tuuq Drilling
2022	Ambler Metals	47	8,376	HQ3	Major Drilling

Sprague and Henwood used company-manufactured drill rigs during their work programs on the Project. Many of their rigs remain at the Bornite deposit and constitute a historical inventory of 1950s and 1960s exploration artifacts. The 2004 to 2011 NovaGold drill programs used a single skid-mounted LF-70 core rig, drilling HQ (63.5 mm core diameter) or NQ (47.6 mm) core. The drill was transported by skid to the various drill pads using a D-8 bulldozer located on site. The D-8 was also used in road and site preparation. Fuel, supplies, and personnel were transported by helicopter. The 2015 and 2016 NovaCopper drill programs used two helicopter-portable LF-70 core rigs, drilling HQ or NQ core. The drill was transported by helicopter to various drill pads. The 2017 Trilogy metallurgical drill program used a helicopter-portable LF-90 core rig, drilling PQ (85 mm) core to be used in future metallurgical testwork. The drill was transported by helicopter to various drill pads. In 2018, Trilogy used a helicopter-portable Duralite 1400N, drilling PQ core to be used for geotechnical studies in plant site facilities. Drilling during 2019 consisted of two helicopter-portable Duralite 1400N core rigs, drilling PQ and HQ core used for geotechnical and hydrogeological studies to support Trilogy's 2020 FS. The 2021 Ambler Metals program was completed by Tuuq Drilling utilizing three helicopter-portable Duralite 1400N rigs for geotechnical, hydrogeological, resource conversion drilling, and to collect mineralized material for metallurgical testwork. The 2022 Ambler Metals program was completed by Major Drilling utilizing 3 helicopter-portable (2 Discovery EF-75 and 1 Longyear LF-90) rigs for geotechnical, hydrogeological, resource conversion drilling, and to collect mineralized material for metallurgical testwork.

7.2.3 Drill Core Procedures

7.2.3.1 BCMC/Kennecott

There is only partial knowledge of specific drill core handling procedures used by Kennecott during their drill programs at the Arctic deposit. The drill data collected during the Kennecott drilling programs (1965 to 1998) were logged on paper drill logs, copies of which are stored in the Kennecott office in Salt Lake City, Utah. Electronic scanned copies of the paper logs, in PDF format, are held by Trilogy. Drill core was hydraulically split or cut with half core submitted to various assay laboratories and the remainder stored in Kennecott's core storage facility at the Bornite Camp. Between 1965 and 1986 analyses were conducted primarily by Union Assay Office Inc. of Salt Lake City, Utah. At least six other labs were also used during that time period, but mostly as check labs or for special analytical work. ALS Minerals was used for analyses submitted by Kennecott in 1998.

7.2.3.2 NovaGold/Trilogy (formerly, NovaCopper)/Ambler Metals

Throughout Trilogy's work programs, the following standardized core handling procedures have been implemented. Core is slung by helicopter to either the Dahl Creek (2004 to 2008) or Bornite (2011 to 2019) Camp core-logging facilities. Upon receiving a basket of core, geologists and geotechnicians first mark the location of each drilling block on the core box, and then convert footages on the blocks into metres. All further data capture is then based on metric measurements. Geotechnicians or geologists measure the intervals (or "from/to") for each box of core using the drilling blocks and written measurements on the boxes.

Geotechnicians fill out metal tags with the hole ID, box number and "from/to", and staple them to each core box. Geotechnicians then measure the core to calculate percent recovery and rock quality designation (RQD).

Geologists then mark sample intervals to capture each lithology or other geologically appropriate intervals. Geologists staple sample tags on the core boxes at the start of each sample interval and mark the core itself with a wax pencil to designate sample intervals. Sample intervals used are well within the width of the average mineralized zones in the resource area. This sampling approach is considered appropriate for the style of mineralization and alteration.

Core is logged with lithology and visual alteration features captured on observed interval breaks. Geological and geotechnical parameters are recorded based on defined sample intervals and/or drill run intervals (defined by the placement of a wooden block at the end of a core run). Logged parameters are reviewed annually and slight modifications have been made between campaigns, but generally include rock type, mineral abundance, major structures, SG, point load testing, recovery and rock quality designation measurements, and magnetic susceptibility. Mineralization data, including total sulphide (recorded as percent), sulphide type (recorded as an absolute amount), gangue and vein mineralogy are collected for each sample interval with an average interval of approximately 2 m. Structural data are collected as point data. Geotechnical data (core recovery, RQD) are collected over drill run intervals.

Drill hole data are recorded in a digital format and, after a QA/QC review, are forwarded to the Database Manager who then imports them into the master database.

After logging, the core is digitally photographed and cut in half using diamond core saws. Specific attention to core orientation is maintained during core sawing to ensure the most representative sampling. Not all core is oriented; however, core that has been oriented is identified to samplers by a line drawn down the core stick. If core was not competent, it was split by using a spoon to transfer half of the core into the sample bag.

One-half of the core is returned to the core box for storage on site and the other half is bagged and labelled for sample processing and analysis. Select specific gravity measurements are also taken (refer to 8.1.3).

7.2.4 Geotechnical and Hydrogeological Drilling

Geotechnical and hydrogeological drilling is summarized in Table 7-9 and Table 7-10. The collar locations of the geotechnical drilling and hydrogeological drilling are presented Figure 7-12 and Figure 7-13. The number of holes reported for each year are the holes that were staffed by a geotechnician and/or hydrogeologist at the rig and primary purpose was to gather geotechnical and hydrogeological data. Geotechnical data (besides PLT's) and hydrogeological installs may have been completed on holes from other programs, such as in-fill or metallurgical holes. Triple tube core barrels were used for geotechnical drill holes in 2015, 2016, and 2019. The geotechnical and hydrogeological field and laboratory testing approach and the results are discussed in Section 13.9 and Section 17.1.5.

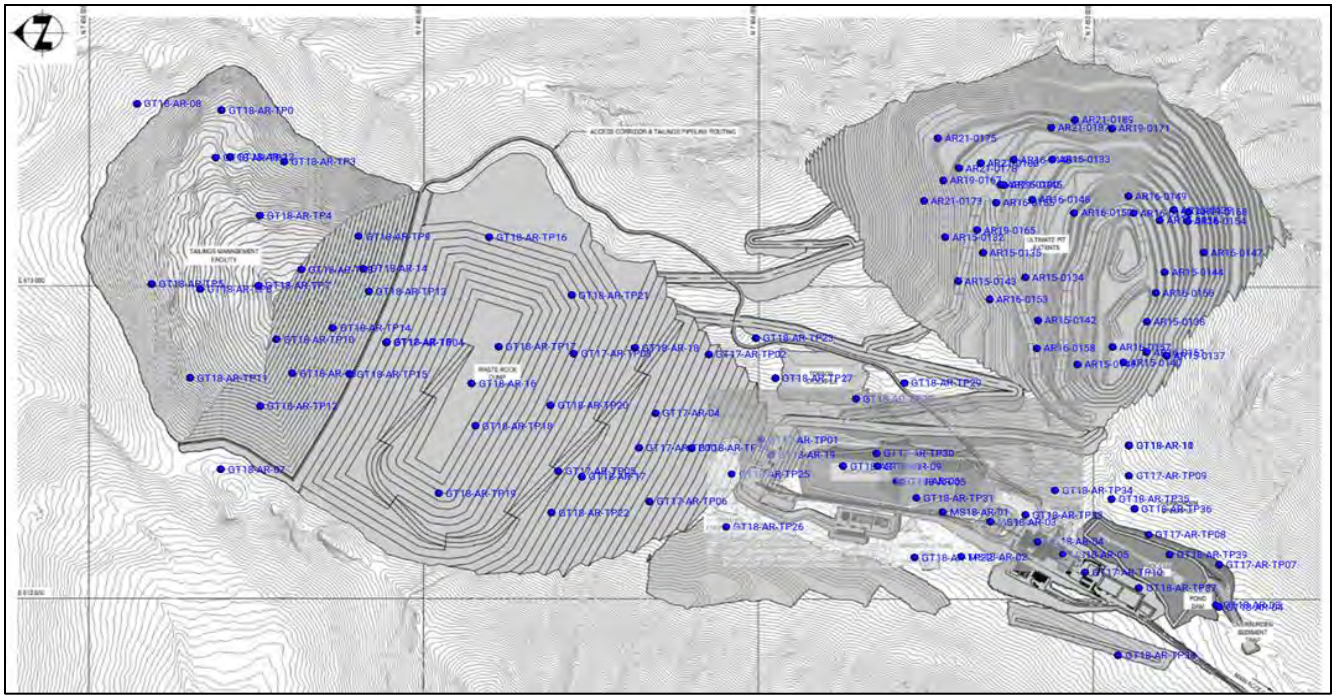
Table 7-9: Summary of Geotechnical Drilling

	2011	2015	2016	2017	2018	2019	2021	2022
Number of Holes	5	2	3	11	24	4	8	5
Oriented core	X	X	X			X	X	X
Water level monitoring	X	X	X	X	X	X	X	X
Falling head packer tests	X							
Point load tests	X	X	X			X	X	X
Uniaxial compressive strength		X	X			X		
Direct shear testing		X	X			X	X	X
Modulus testing		X	X			X		
Triaxial testing		X	X			X	X	X
Acoustic Televiewer			X					
Falling Head, Single or Straddle packer tests	X		X					
Airlift pump test			X					
Hydraulic conductivity testing (slug testing)				X				
Cohesive and residual shear strength tests on soils				X	X			
Compressive strength test on core and rock				X	X			
Extended duration injection tests						X		

Table 7-10: Summary of Geotechnical Drilling by Year and Purposes

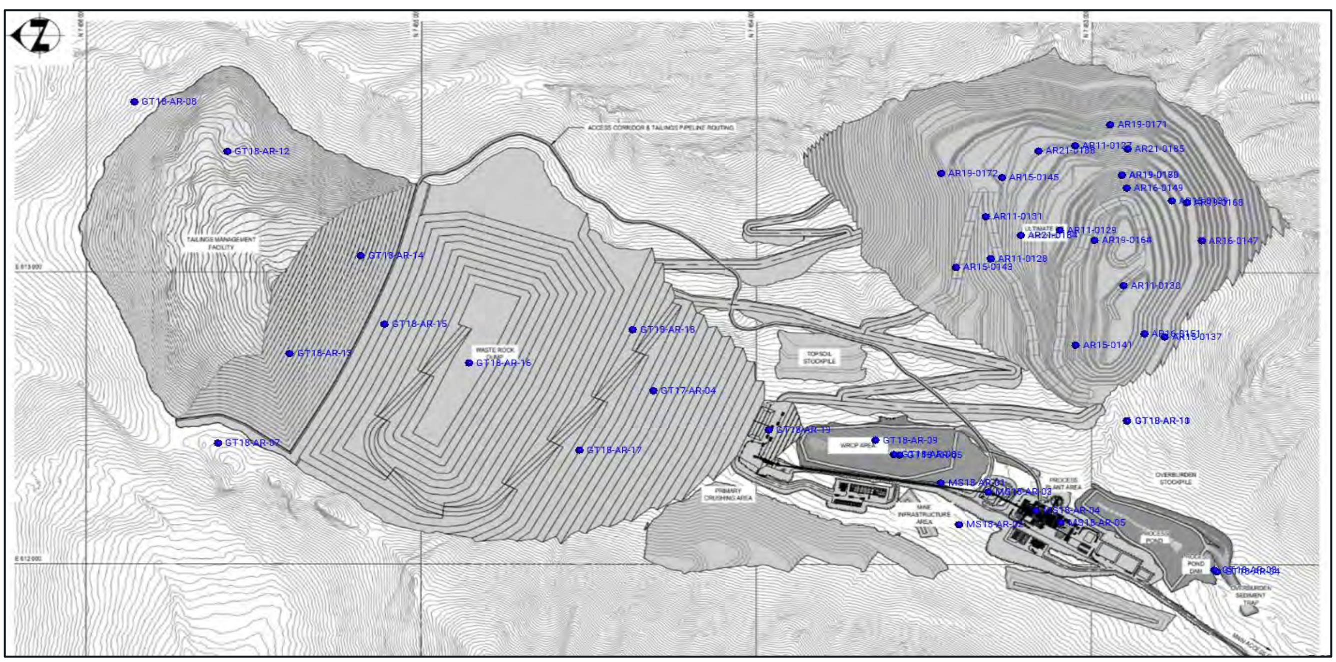
Year	Purpose
2011	Obtain geotechnical data in areas of the deposit that may host underground infrastructure or could pose issues with underground mining.
2015	Collect geotechnical and hydrogeological data to better understand the wall rock characteristics and hydrogeology within the open pit area.
2016	Complete the 3 drill holes that were deferred/not completed from the 2015 program.
2017	Collect geotechnical and hydrogeological data for tailings management and waste rock facilities within the entire Sub Arctic Creek valley.
2018	Collect geotechnical and hydrogeological data for WRF, TMF, and surface infrastructure in the Upper Sub Arctic Creek Valley.
2019	Provide additional geotechnical and hydrogeological data for pit design for the 2020 FS
2021	Define talc horizons on east side of pit for pit design.
2022	Define extent of lower talc horizons on northeast side of pit for pit design.

Figure 7-12: Locations of Geotechnical Drilling



Source: SRK, 2022

Figure 7-13: Locations of Hydrogeological Drilling



Source: SRK, 2022

7.2.5 Collar Surveys

7.2.5.1 Kennecott

Kennecott provided NovaGold with collar coordinates for all historical holes in UTM coordinates using the NAD27 datum. NovaGold re-surveyed collars of selected historical holes in 2004 and again in 2008. The re-surveys showed little variation compared to the historical surveys.

7.2.5.2 NovaGold/Trilogy (formerly, NovaCopper)/Ambler Metals

Collar location coordinates were determined for the 2004 to 2016 NovaGold/Trilogy drill campaigns with two Ashtech ProMark 2 GPS units using the Riley Vertical Angle Benchmark (VABM; 611120.442E, 7453467.486N) as the base station for all surveys. Raw GPS data were processed with Ashtech Solutions 2.60 software. All surveyed data were collected in the NAD27 datum and later converted to NAD83.

A 2010 survey by a Registered Land Surveyor from WHPa observed differences between the 2010 and historical coordinates used for the Riley VABM, which were of the same magnitude (0.5 m east, 0.1 m north and 1.0 m down) as other Arctic drill collars that were re-surveyed for the third time. A correction was applied to all Arctic drill holes based upon the newly established coordinates for the Riley VABM, together with converting from NAD27 to NAD83 datums. All post 2010 surveys are completed in NAD83.

During Tetra Tech's 2013 site visit, nine drill collars were located using a Garmin™ Etrex 20 GPS unit. The difference between reported and measured positions ranged between 3.4 and 7.8 m with an average discrepancy of 4.8 m. These differences are within the tolerances expected for GPS verification.

A Registered Land Surveyor, Eric Cousino of Windy Creek Surveys, completed collar coordinate survey locations of the 2019 drilling in August 2019. The Horizontal Datum used on this project for point position determinations was NAD_83(CORS96) (EPOCH 2003), and the Vertical Datum is NAVD88 computed using Geiod09. Data were output into NAD83 Zone 4N (meters) using JAVAD Justin post-processing software.

The 2021 collars were surveyed by Layne Lewis, the Ambler Metals drill coordinator, using a Leica GS18 differential GPS system (cm-scale accuracy). Kuna Engineering surveyed the 2022 collar locations using a Trimble R12 GPS/GNSS receiver.

7.2.6 Downhole Surveys

BCMC did not perform downhole surveys prior to 1971 (drill hole AR-32). In 1971, BCMC began to survey selected (mineralized) drill holes using a Sperry-Sun downhole survey camera usually at 30.5 m (100 ft) intervals. BCMC was able to re-enter and survey a few of the older drill holes. BCMC, and later Kennecott, applied a single azimuth (49°) and uniform dip deviation every 15.24 m (50 ft) that flattens with depth to all holes collared vertically that were not surveyed.

Downhole surveys from 2004 to 2017 were collected using either a Reflex EZ-shot camera or a Ranger single-shot tool with individual survey readings collected at the drill rig on approximately 30 to 60 m intervals. During the 2019 drill program, downhole surveys were collected using a continuous north seeking gyroscope with readings collected at the drill rig on roughly 30m intervals. Downhole surveys in 2021 were taken every 100 feet (30.5 meters) with a Reflex EZ-Trac multi-shot instrument. The magnetic declination correction from 2004 to 2021 were calculated using the NOAA Magnetic Field Calculator website and the longitude and latitude of the Arctic deposit.

The downhole survey data show a pronounced deviation of the drill holes toward an orientation more normal to the foliation.

7.2.7 Recovery

7.2.7.1 Kennecott

Incomplete Kennecott data exist with regards to overall core recovery but based on 917 intervals of 3.05 m or less in the historical database, the average recovery was 92%. Kennecott RQD measurements in the 1998 program averaged 87.0%. There has been no systematic evaluation of recovery by rock type.

7.2.7.2 NovaGold/Trilogy (formerly, NovaCopper)/Ambler Metals

Core recovery during NovaGold/NovaCopper/Trilogy and Ambler Metals drill programs were good to excellent, resulting in quality samples with little to no bias. There are no other known drilling and/or recovery factors that could materially impact accuracy of the samples during this period.

Table 7-11 shows recoveries and RQD for each of the NovaGold/NovaCopper/Trilogy and Ambler Metals campaigns exclusive of the geotechnical drill holes in 2011 and 2021.

Table 7-11: Recovery and RQD 2004 to 2008 Arctic Drill Campaigns

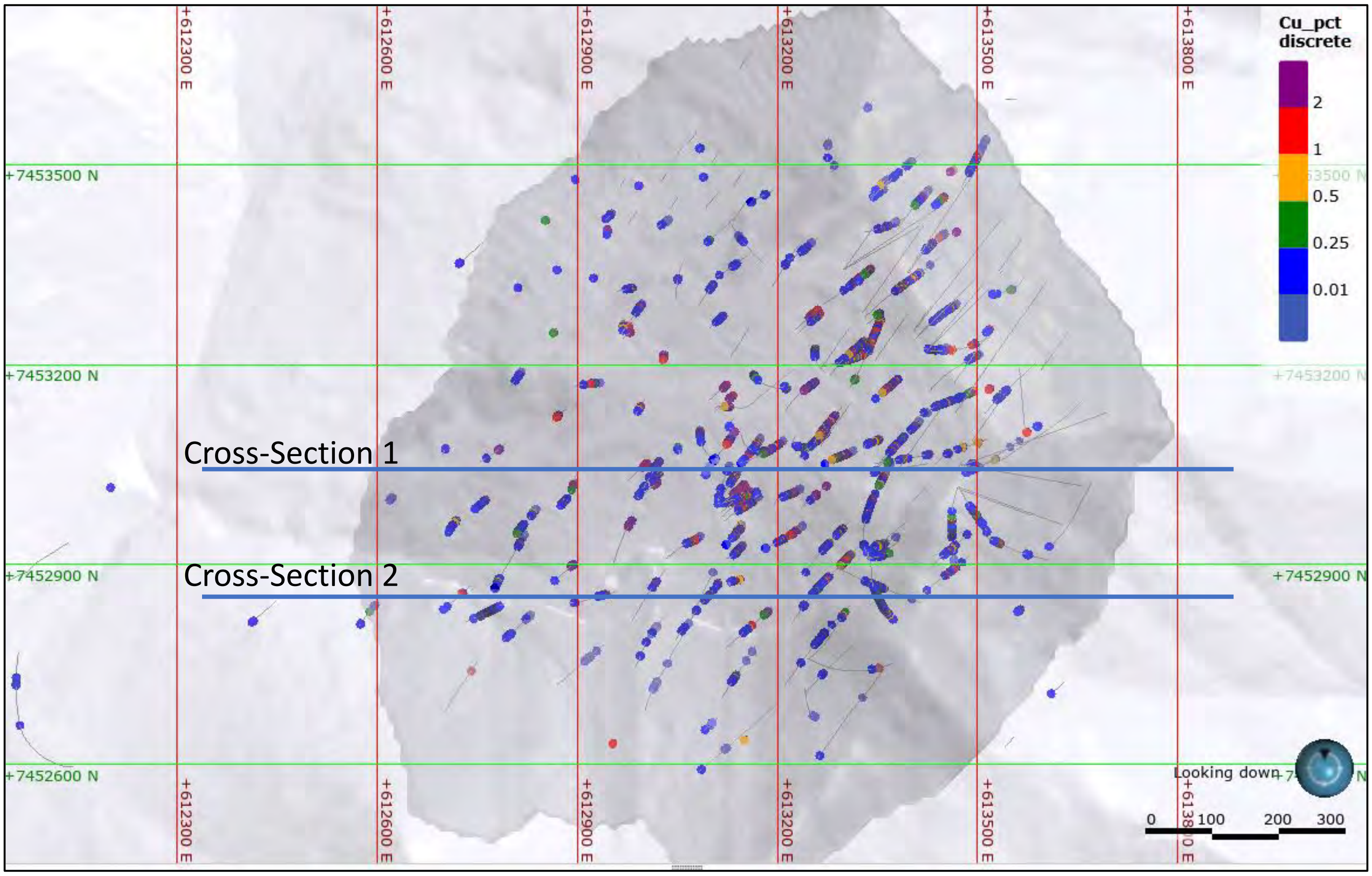
Year	Metres	Recovery (%)	RQD (%)
2004	2,996	98.0	73.4
2005	3,030	96.0	74.4
2007	2,606	95.7	73.1
2008	3,306	98.0	80.1
2011	1,193	96.0	68.8
2015	3,055	91.3	69.0
2016	3,058	91.5	69.7
2017	790	95.5	75.0
2019	2,433	96.3	77.1
2021	2,282	95.1	70.3

7.2.8 Drill Intercepts

All drill holes at the Arctic deposit are collared on surface and are generally vertically oriented, or steeply inclined in a northeast direction. The majority of drill holes are spaced at 75 m to 100 m intervals, but there are instances where drill holes are located within 10 m of one another. Drill holes typically intersect the generally shallow-dipping mineralized horizon at approximately right angles.

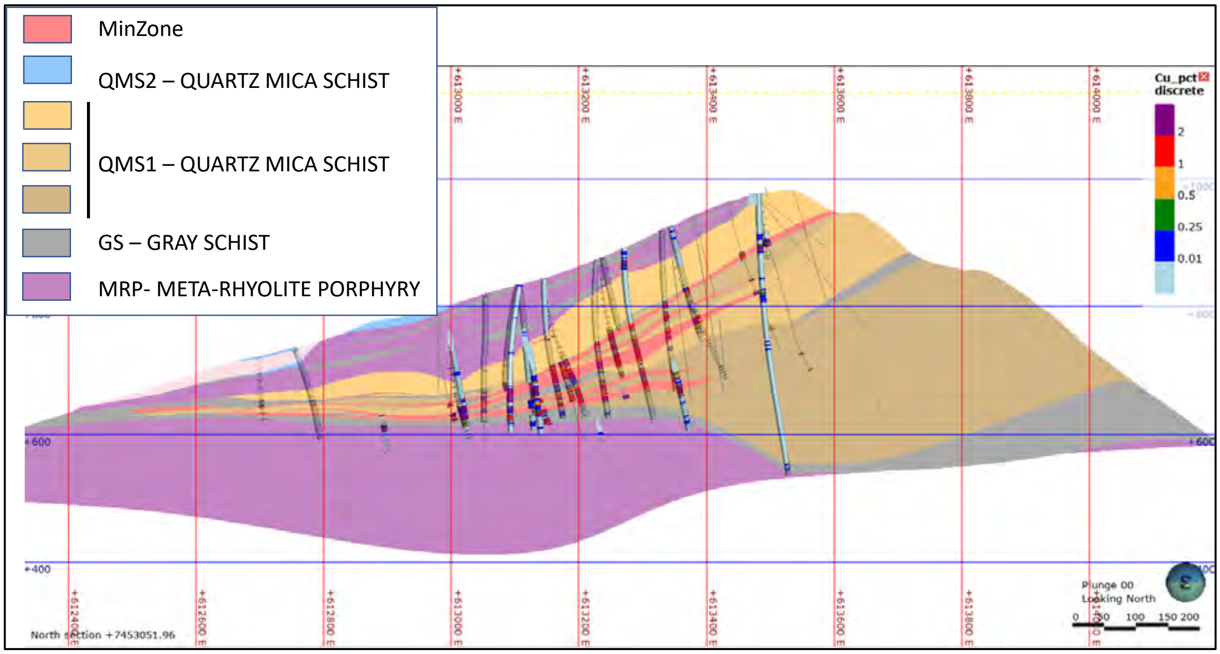
Drilling at the Arctic deposit covers an area measuring roughly 1,000 m east-west by 1,000 m north south with holes that approach 750 m below surface. Significant results of the distribution of copper in drilling is shown in plan and in vertical cross-sectional views in Figure 7-14 to Figure 7-16. Significant results of the distribution of zinc in drilling is shown in plan and in vertical cross-sectional views in Figure 7-17 to Figure 7-19. These results represent a summary of the QPs interpretation of the exploration information.

Figure 7-14: Drill Plan of Arctic Copper Results



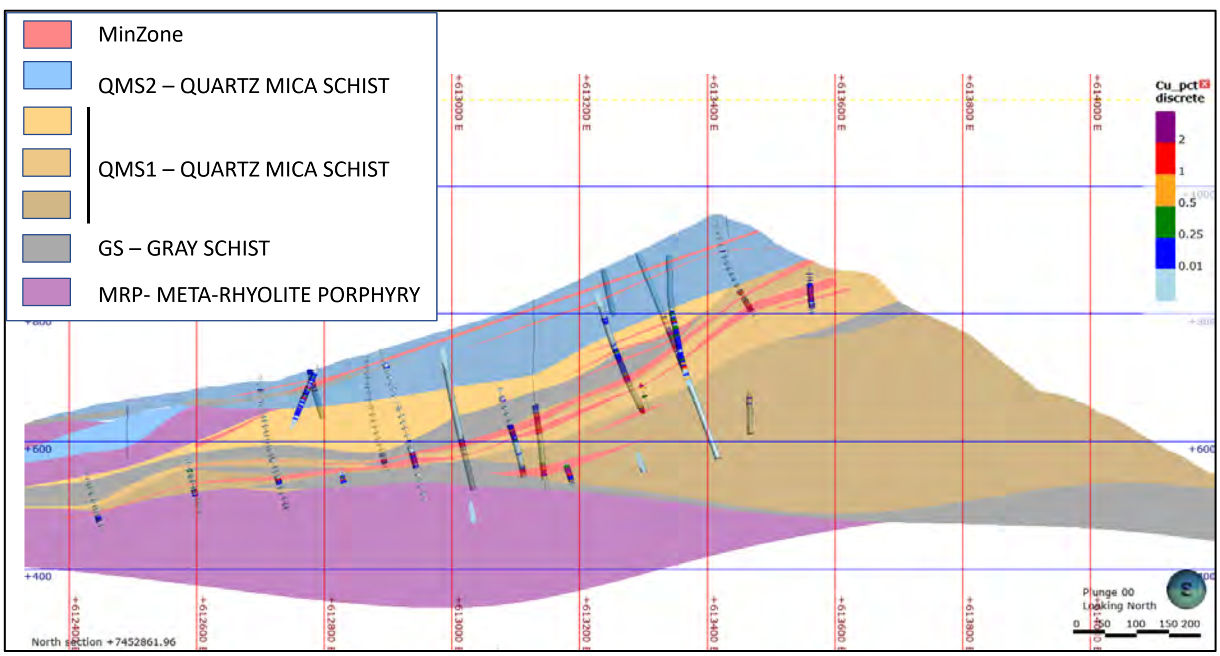
Source: Wood, 2022

Figure 7-15: Cross-Section 1 of Arctic Copper Results Looking South



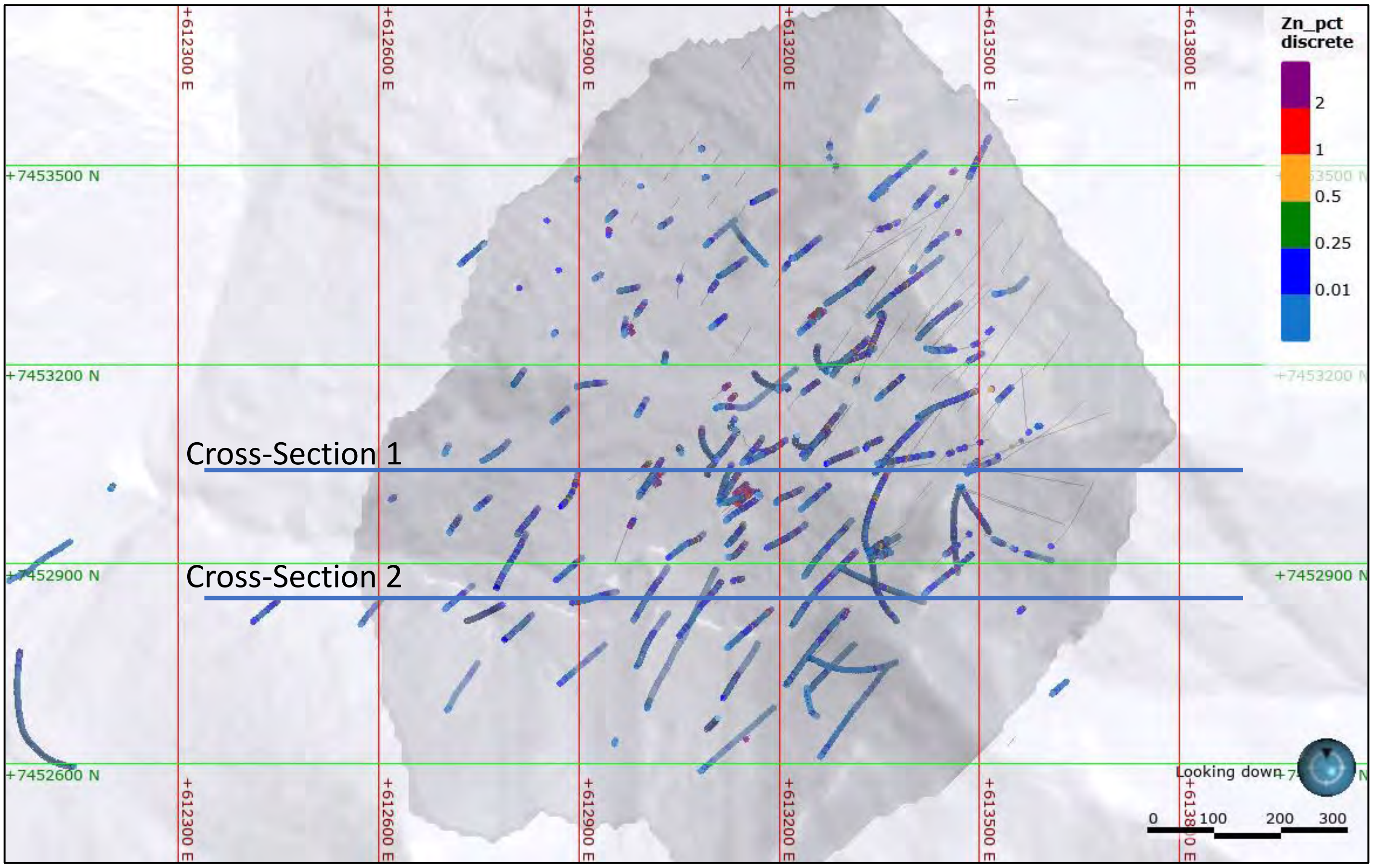
Source: Wood, 2022

Figure 7-16: Cross-Section 2 of Arctic Copper Results Looking South



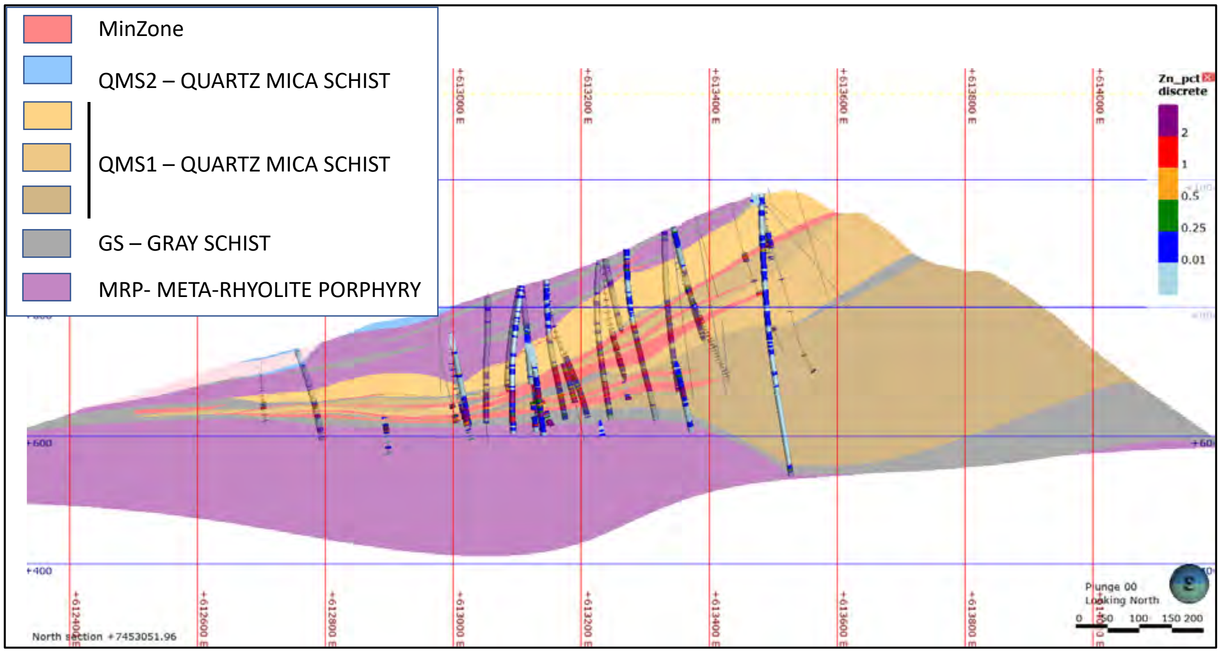
Source: Wood, 2022

Figure 7-17: Drill Plan of Arctic Zinc Results



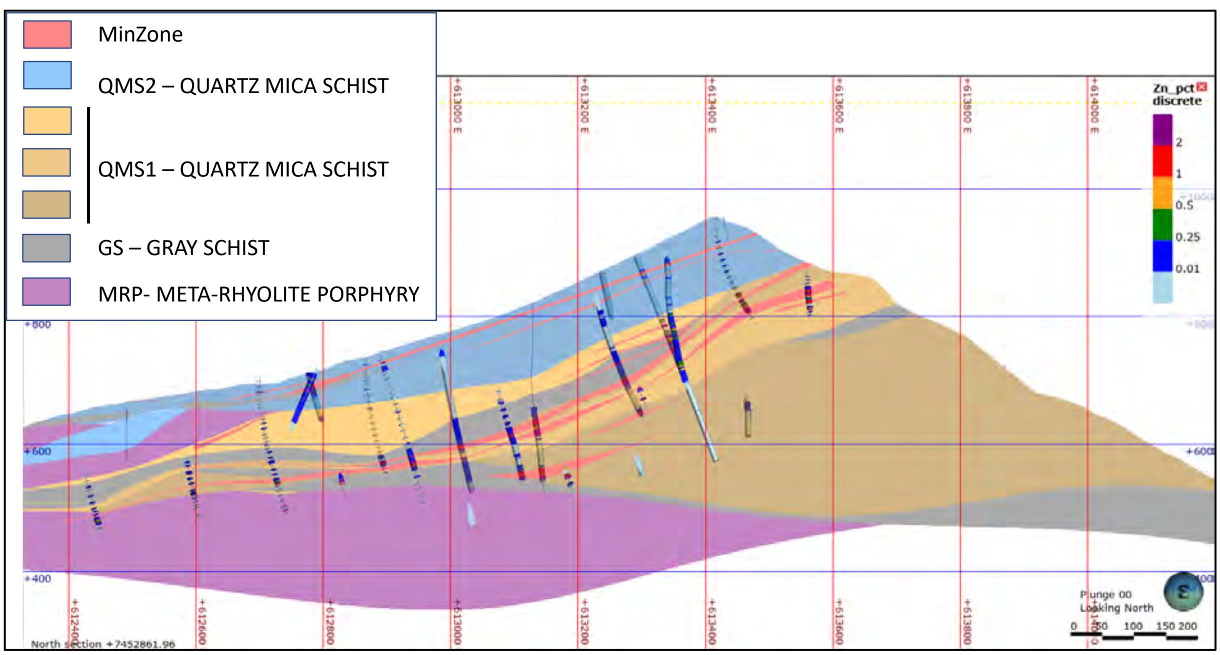
Source: Wood, 2022

Figure 7-18: Cross-Section 1 of Arctic Zinc Results Looking South



Source: Wood, 2022

Figure 7-19: Cross-Section 2 of Arctic Zinc Results Looking South



Source: Wood, 2022

8 SAMPLE PREPARATION, ANANLYSES AND SECURITY

8.1 Sample Preparation

8.1.1 Core

The data for the Arctic deposit were generated over three primary drilling campaigns: 1966 to 1986 when BCMC, a subsidiary of Kennecott was the primary operator, 1998 when Kennecott resumed work after a long hiatus, and 2004 to present under NovaGold. Trilogy (formerly, NovaCopper), and Ambler Metals.

8.1.1.1 Kennecott and BCMC

Sampling of drill core prior to 1998 focused primarily on the mineralized zones; numerous intervals of weak to moderate mineralization were not sampled during this period. During the 1998 campaign, Kennecott did sample some broad zones of alteration and weak mineralization, but much of the unaltered and unmineralized drill core was left unsampled. Little documentation on historic sampling procedures is available.

8.1.1.2 NovaGold/Trilogy (formerly, NovaCopper)

Between 2004 and 2005, NovaGold conducted a systematic drill core re-logging and re-sampling campaign of Kennecott and BCMC era drill holes AR-09 to AR-74. NovaGold either took 1 to 2 m samples every 10 m or sampled entire lengths of previously unsampled core within a minimum of 1 m and a maximum or 3 m intervals. The objectives of the sampling were to generate a full (ICP) geochemistry dataset for the Arctic deposit and ensure continuous sampling throughout the deposit.

From 2004 to 2019, sample intervals are determined by the geological relationships observed in the core and limited to a 2.5 m maximum length and 1 m minimum length. Sample intervals terminate at lithological and mineralization boundaries. Sampling is generally continuous from the top to the bottom of the drill hole unless otherwise directed by the Exploration Manager. Occasionally, if warranted by the need for better resolution of geology or mineralization, smaller sample intervals may be employed. When the hole is in unmineralized rock, the sample length is generally 2.5 m, whereas in mineralized units, the sample lengths are shortened to 1 to 2 m.

After logging, the core was cut in half using diamond core saws. If core was not competent, it was split by using a spoon to transfer half of the core into the sample bag. One-half of the core was returned to the core box for storage on site and the other half was bagged, labelled, and sent to ALS Minerals Laboratories in Vancouver for analysis, via the (ALS preparation facility in Fairbanks Alaska) and the other half was archived in the core storage facility at the Bornite Camp facilities or at the Ambler Metals warehouse in Fairbanks. For the 2021 metallurgical holes, ¼ core was sampled for analysis at ALS, ¼ retained, and ½ sent for metallurgical testing.

Standard and Blank control samples were inserted at site into the shipments at the approximate rate of one standard reference material (standard), one blank per 17 core samples. At the Exploration Manager direction, ALS also prepared one coarse duplicate per 17 core samples. A sample tag for these duplicates was inserted into an empty sample bag at site.

8.1.2 Acid-Base Accounting Sampling

In 2010, SRK collected 148 samples from drill core based on their position relative to the massive and semi-massive sulphide mineralization (SRK 2011). Samples were targeted within, immediately adjacent to, adjacent to, and between lenses of mineralization; the sampling program focused on characterization for a potential underground development scenario. Samples were shipped to SGS Canada Inc., Burnaby, BC, for sample preparation and analysis. Samples were analysed for acid base accounting (ABA) and metals. ABA tests were conducted using the Sobek method with sulphur speciation and total inorganic carbon (TIC) analysis. Metal concentrations were determined using aqua regia digestion followed by ICP-MS analysis. In addition, barium and fluorine were analysed by X-ray fluorescence (XRF) following a lithium metaborate fusion.

In 2015, Trilogy retained SRK to provide metal leaching (ML) and ARD characterization services for the Arctic deposit. Activities focused on three objectives: 1) construction of on-site barrel tests and parallel humidity cells, 2) expansion of the ABA database to support future evaluation for ARD potential management for open pit mining, and 3) evaluation of the use of proxies for ABA parameters in the exploration database with the purpose of being able to use the exploration database for block modelling of ML/ARD potential, if needed. Barrel test samples were collected during July and August 2015 and eight on-site barrel tests (including two QC tests) were constructed and initiated in late August 2015. Following the set-up of the on-site barrel tests, representative composite rock samples were shipped to Maxxam Analytics of Burnaby, British Columbia and parallel humidity cells were initiated in late October 2015. Trilogy and SRK selected 321 samples to be analysed for a conventional static ABA package with a trace element scan using the same method as the exploration database (four-acid digestion). Samples were analysed by Global ARD Testing Services of Burnaby, British Columbia.

In 2016, Trilogy evaluated the distribution of the existing samples to select additional samples in preparation for block modelling of ML/ARD potential. A drill program was designed, and infill samples were collected from holes drilled in 2015 and 2016. This program was completed with 1004 samples analysed for a conventional static ABA package with a trace element scan using the same method as the exploration database. Samples were analysed by Global ARD Testing Services of Burnaby, British Columbia. The resulting data were combined with the previous datasets.

In 2018 and 2021, the kinetic test program was expanded to further characterize ML/ARD potential. Samples were selected using the exploration geochemistry database, to target drill core with geochemical characteristics representing the range of compositions present in the database for key parameters such as sulphur and selenium. Kinetic testing was initiated in 2019 and 2021 at Bureau Veritas (previously Maxxam Analytics) of Burnaby, BC, and the kinetic samples were also analysed by static methods at Bureau Veritas for ABA and metals by the same methods as the 2015 and 2016 programs, in addition to metal concentrations determined using aqua regia digestion followed by ICP-MS analysis.

As described above, several laboratories were used for acid base accounting studies and associated kinetic testwork as summarized in Table 8 1. Accreditations, where known, are listed below. All the labs are independent of NovaGold, NovaCopper, Trilogy (formerly, NovaCopper), and Ambler Metals.

Table 8-1: Analytical Laboratories used for Acid Base Accounting and Kinetic Studies for the Arctic Project

Laboratory Name	Laboratory Location	Years Used	Accreditation	Comment
SGS Canada Inc.	Burnaby, BC	2010	ISO 90001 and ISO/IEC 17025.	ABA samples
Global ARD Testing Services	Burnaby, BC	2015, 2016	ISO/IEC 17025	ABA samples
ARS Aleut Analytical	Anchorage and Fairbanks, AK	2015, 2016, 2017, 2018	Accreditations are not known.	Barrel test leachate samples
Bureau Veritas (previously Maxxam Analytics)	Burnaby, BC	2015, 2016, 2017, 2018, 2019, 2020, 2021, 2022.	ISO/IEC 17025	ABA samples (2015 and 2019), HCTs (2015 to 2022), and barrel test leachate samples (2019-2022)

8.1.3 Density Determinations

Representative (SG) determinations conducted before 1998 for the Arctic deposit are lacking. Little information regarding sample size, sample distribution and SG analytical methodology are recorded for determinations during this period.

In 1998, Kennecott collected 38 core samples from that year's drill core, of which 22 were from mineralized zones and 16 from non-mineralized lithologies. Mineralized samples were defined as MS (more than 50% total sulphides), SMS (less than 50% total sulphides) or lithology samples (non-mineralized country rock containing up to 10% sulphides). SG determinations were conducted by ALS Minerals and Golder and Associates and were based on short (6 to 12 cm) whole core samples. SG was determined based on the water displacement method.

In 1999, Kennecott collected 231 samples from pre-1998 drill core for SG analysis. The samples were from NQ- and BQ-sized core and averaged 7.27 cm in length. The samples were shipped to Anchorage but were not forwarded to a laboratory.

In 2004, NovaGold forwarded the 231 samples from the pre-1998 drill campaigns stored in Kennecott's Anchorage warehouse, as well as 33 new samples from the 2004 drill program to ALS Minerals for SG determination.

Additionally, in 2004 NovaGold collected 127 usable field SG measurements. Samples were collected from HQ-sized core and averaged 9.05 cm in length. An Ohaus Triple Beam Balance was used to determine a weight-in-air value for dried core, followed by a weight-in-water value. The wet-value was determined by suspending the sample by a wire into a water-filled bucket. The SG value was then calculated using the following formula.

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

In 2011, NovaGold geologists stopped collecting short interval "point data" (as described above) within the mineralized zone, and instead collected "full-sample-width" determinations from existing 2008 split core and all of the sampled 2011 whole core. The samples averaged 1.69 m in length. Samples were collected continuously within mineralized zones and within a 2 to 3 m buffer adjacent to mineralized zones. A total of 193 measurements were collected. In 2011, 266 sample pulps were also submitted to ALS Minerals for SG determination by pycnometer analysis.

Between 2015 and 2019, Trilogy geologists collected SG data consistent with the 2011 full sample width water immersion campaign. A total of 2,406 specific gravity measurements were collected, with SG values ranging from 2.01 to 4.96. The samples averaged 1.49 m in length. Samples were collected continuously within mineralized zones and within a 2 to 3 m buffer adjacent to mineralized zones.

In 2021 Ambler Metals collected 830 SG measurements from half-core intervals ranging from 0.08 to 2.19 m with SG values ranging from 2.01 to 2.88. In 2022, Ambler Metals collected 2,545 SG measurements from half-core intervals ranging from 0.14 m to 2.81 m with SG values ranging from 2.79 to 5.06.

8.1.4 Sample Security

Security measures taken during historical Kennecott and BCMC programs are unknown to NovaGold, Trilogy, or Ambler Metals. Ambler Metals is not aware of any reason to suspect that any of these samples have been tampered with.

The 2004 to 2019 samples were always either in the custody of NovaGold personnel or at the assay laboratories, and the chain of custody of the samples is well documented.

The site has restricted access with only means of access via charter flight. Core is stored in a core storage yard and warehouse with the perimeter surrounded by a chain-link fence and secure locked gates. Shipment of core samples from site occurred on a drill hole by drill hole basis. Rice bags, containing two to four poly-bagged core samples each, were marked and labelled with the ALS Minerals address, project and hole number, bag number, and sample numbers enclosed. Rice bags were secured with a pre-numbered plastic security tie and a twist wire tie and then assembled into standard fish totes for transport by chartered flights on a commercial airline to Fairbanks, where they were met by a contracted expeditor for delivery directly to the ALS Minerals preparation facility in Fairbanks.

8.1.5 Assay Laboratories

At least six laboratories were used during the Kennecott/BCMC time period, but mostly as check laboratories or for special analytical work. Accreditations are not known. The laboratories were independent of Kennecott/BCMC. Bondar Clegg, now ALS Minerals, was used for analyses conducted by Kennecott. During the BCMC work, analyses were conducted primarily by Union Assay Office Inc. of Salt Lake City, Utah.

Since 2004 NovaCopper, NovaGold, Trilogy metals, and Ambler Metals have used ALS Minerals (Fairbanks Alaska for sample preparation and Vancouver British Columbia as the primary laboratory. ALS Minerals is accredited for a number of specific test procedures including fire assay of gold by AA, ICP, or gravimetric finish, multi-element ICP and AA assays for silver, copper, lead and zinc. ALS Minerals is independent of NovaGold, NovaCopper, Trilogy, and Ambler Metals.

The laboratories used during the various exploration, infill, and step-out drill analytical programs completed on the Arctic Project are summarized in Table 8 2.

Table 8-2: Analytical Laboratories Used by Operators of the Arctic Project

Laboratory Name	Laboratory Location	Years Used	Accreditation	Comment
Union Assay Office, Inc.	Salt Lake City, Utah	1968	Accreditations are not known.	Primary Assay laboratory
Rocky Mountain Geochemical Corp.	South Midvale, Utah	1973	Accreditations are not known.	Primary and secondary assays
Resource Associates of Alaska, Inc.	College, Alaska	1973, 1974	Accreditations are not known.	Primary and secondary assays
Georesearch Laboratories, Inc.	Salt Lake City, Utah	1975, 1976	Accreditations are not known.	Primary and secondary assays
Bondar-Clegg & Company Ltd.	North Vancouver, BC	1981, 1982	Accreditations are not known.	Primary and secondary assays
Acme Analytical Laboratories Ltd. (AcmeLabs)	Vancouver, BC	1998, 2011,	Accreditations are not known.	2012 and 2013 secondary check sample lab
SGS Canada Inc.	Burnaby, BC	2010	ISO 90001 and ISO/IEC 17025.	2015 to 2019 secondary check lab
ALS Analytical Lab	Fairbanks, Alaska (prep) and Vancouver, BC (analytical)	1998, 2004, 2005, 2006, 2007, 2008, 2011, 2012, 2013 2015, 2016, 2017, 2018, 2019	In 2004, ALS Minerals held ISO 9002 accreditations but updated to ISO 9001 accreditations in late 2004. ISO/International Electrotechnical Commission (IEC) 17025 accreditation was obtained in 2005.	2004 - 2019 primary assay laboratory

8.1.6 Sample Preparation and Analytical Methods

Samples from the NovaGold/Trilogy/Ambler Metals programs were logged into a tracking system on arrival at ALS Minerals and weighed. Samples were crushed to 70% passing 2 mm, dried, and a 250 g split pulverized to greater than 85% passing 75 µm.

Samples were submitted for multielement analysis of a 0.25 gram sample by ICP MS following a 4-acid digestion, and for gold analysis of a 30 gram sample by FA with an AA finish. Over limit ICP-MS Cu, Pb, and Zn samples were resubmitted for analysis of a 0.4 gram sample by ICP-Atomic Emission Spectroscopy (AES) or AA following a 4 acid digestion. The overlimit value for Cu, Pb, and Zn is 10,000 ppm. Over limit gold results were resubmitted for analysis of a 30 gram sample by FA with a Gravimetric finish. The overlimit value for Au is 10 ppm. The Lower detection limits for Cu, Pb, and Zn by ICP-MS are 0.2 ppm, 0.5 ppm, and 2 ppm respectively. The lower detection limit for Au by FAAA is 0.05 ppm.

8.2 Quality Assurance/Quality Control

8.2.1 Database Verification

In 1995, Kennecott entered the drill assay data information from the geologic core logs, and the downhole collar survey data into an electronic format. In 2006, NovaGold geologists verified the geologic data from the original paper logs against the Kennecott electronic format, and then merged the data into a Microsoft SQL database.

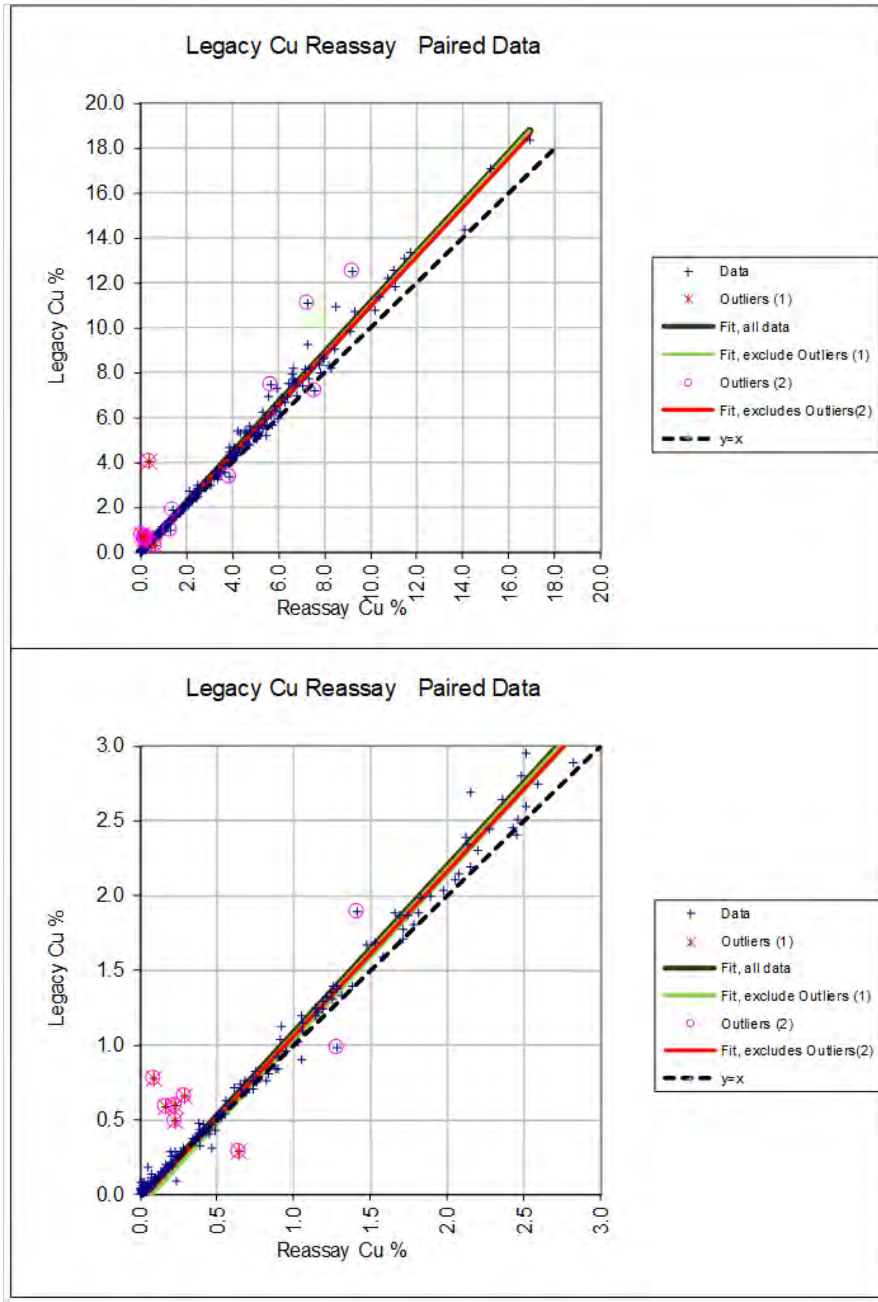
Between December 11th and January 9th, 2013, NovaCopper and GeoSpark Consulting completed a 100% verification of the collar survey, downhole survey, and sample interval data for the Arctic resource area. 20% of the pre-2004 assay data and ~26% of the 2004 to 2008 assay data were also verified at this time. Trilogy also retained GeoSpark to generate QA/QC reports for the NovaGold era 2004 to 2008 and NovaCopper/Trilogy era 2011, 2015, 2016, 2017, and 2019 drill campaigns. All data for the Arctic resource area is stored in the GeoSpark Core Database System created and managed by GeoSpark Consulting.

8.2.2 NovaGold QA/QC Review of Historical Analytical Results

The current assay database contains results for 11,429 sample intervals including 3,077 (27%) historical hole sample intervals. Between 2004 and 2005 NovaGold completed a resampling program of historical drill holes. As a result, 85% of the assay intervals now have recent assay results from ALS Minerals. The re-assay program includes 289 re-assays of previously assayed historical hole sample intervals, representing approximately 17% of the original historical sampling program. In the database reviewed by Wood, the original Cu, Pb, and Zn values from the KCC/Utah laboratory results are given priority over ALS laboratory results for the 289 re-assay intervals. The database reviewed does not distinguish between previously sampled and newly sampled historical intervals or original and new assay values. Wood used the presence and absence of multielement analysis values to establish resampling and re-assay frequency. To determine assay priority used, Wood compared reported values to original logs.

Reduced to Major Axis (RMA) charts prepared by Wood (Figure 8-1 and Figure 8-2) which evaluates the paired original historical and re-assay values, indicates there is a 10% high bias in the historical Cu values after exclusion of 15 outliers and a 13% low bias in the historical Pb values after exclusion of 17 outliers. An inflection in the trend of the paired Cu values starting at 1% Cu may indicate the bias is related to an upper detection limit for the original assay procedure and or a change to an overlimit method. The Pb RMA chart shows a cluster of paired data with high biased original results. The low bias indicated by the RMA slope does not change after exclusion of these pairs.

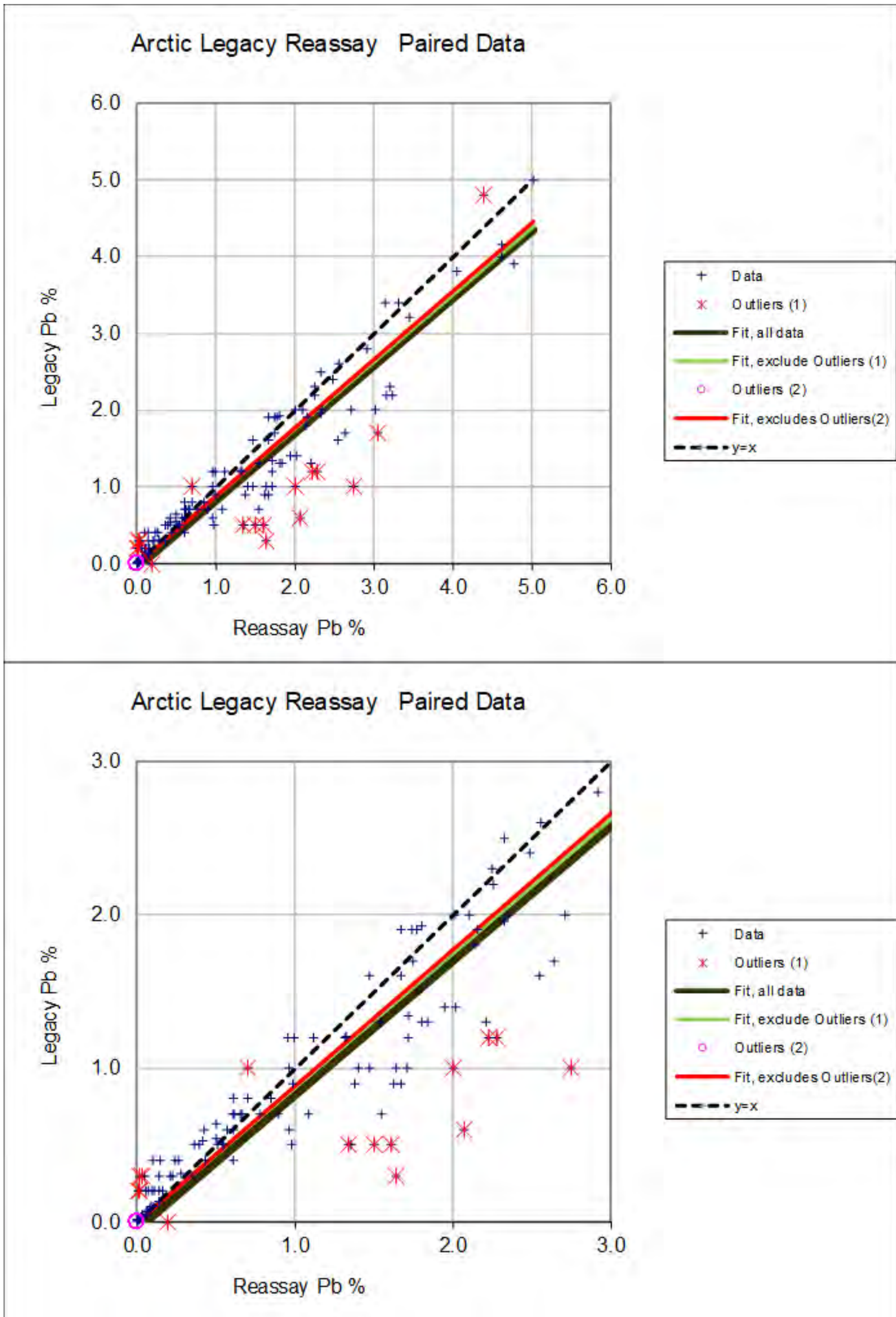
Figure 8-1: Historical Cu Re-Assay RMA Chart



	All Data	Excluding Outliers (1)	Excluding Outliers (2)
N	272	265	258
Percent Rejected	0.0%	2.6%	5.1%
R squared	0.9839	0.9881	0.9937
R	0.9919	0.9940	0.9968
slope	1.1130	1.1155	1.1019
y intercept	-0.0317	-0.0589	-0.0403
error in slope	0.0086	0.0075	0.0055
error in intercept	0.0353	0.0311	0.0224

Source: Wood, 2022

Figure 8-2: Historical Pb Re-Assay RMA Chart

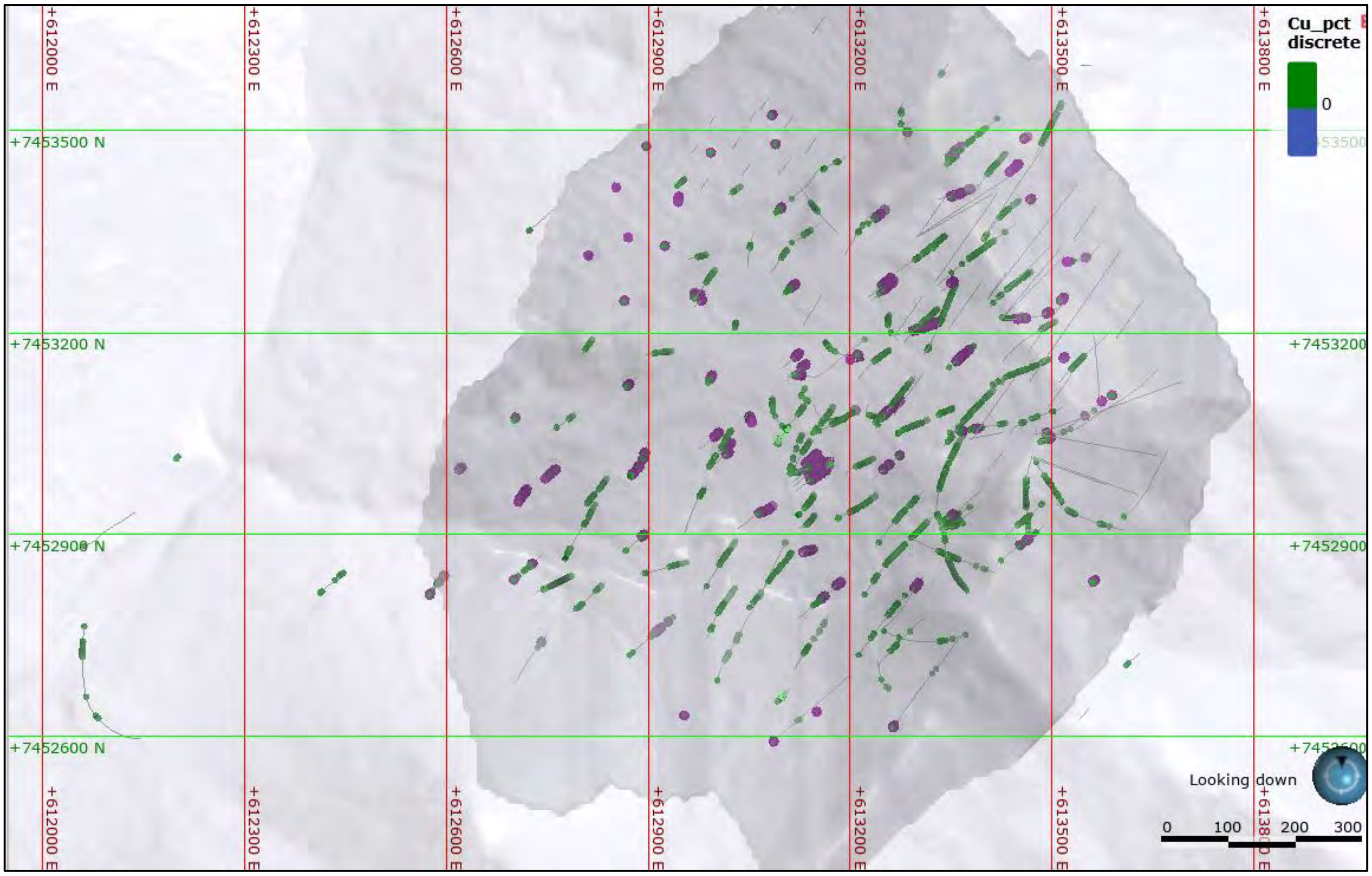


	All Data	Excluding Outliers (1)	Excluding Outliers (2)
N	134	117	--
Percent Rejected	0.0%	12.7%	--
R squared	0.869	0.927	--
R	0.932	0.963	--
slope m	0.87	0.87	--
intercept b	-0.050	0.006	--
error in slope	0.027	0.022	--
error in intercept	0.049	0.038	--

Source: Wood, 2022

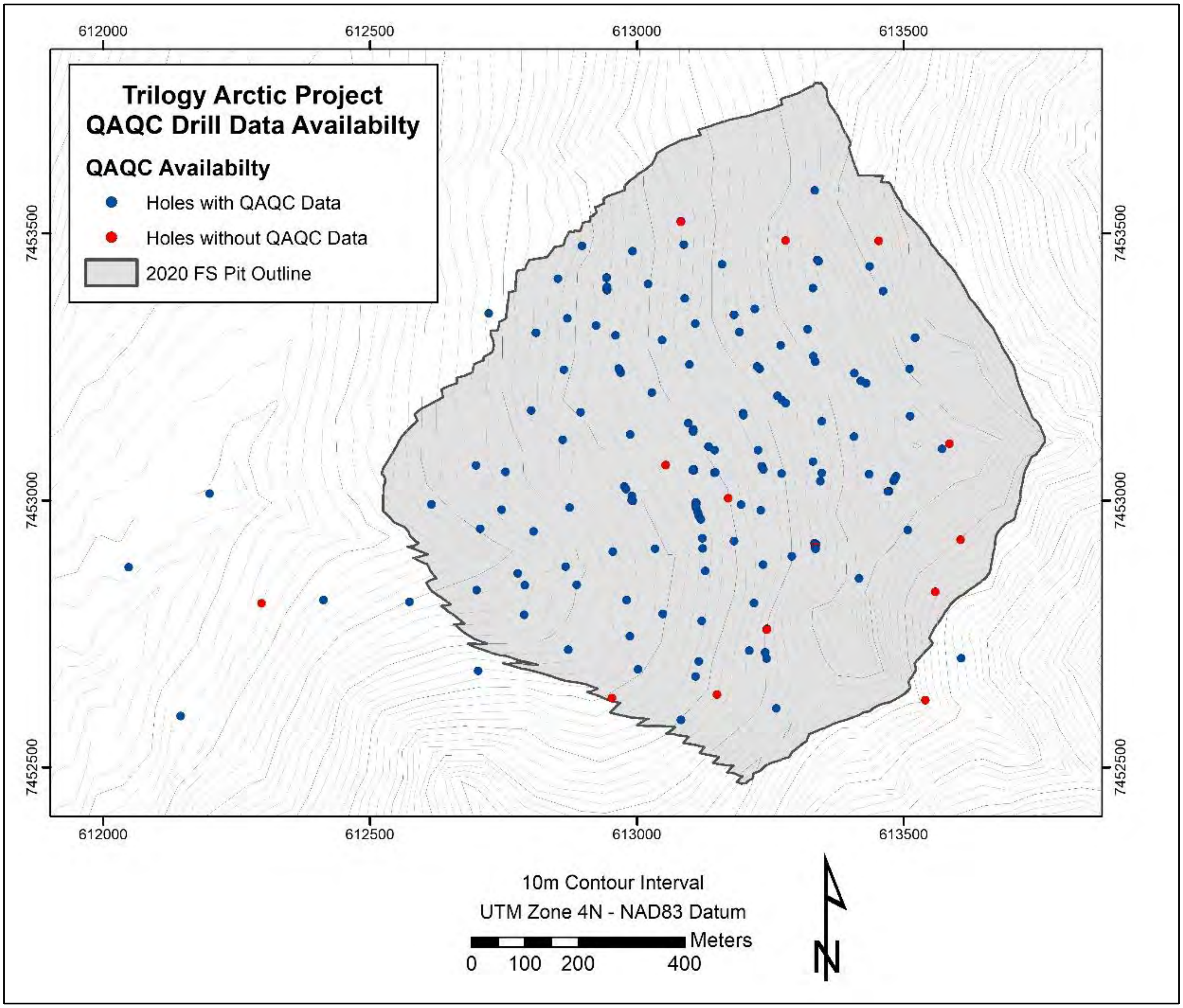
The spatial distribution of the historical samples that remain as primary samples in the database is shown in Figure 8-3. These remaining historical assays are evenly distributed through the deposit area and are surrounded by assay intervals analysed since 2004. Spatial availability of QA/QC data is shown in Figure 8-4.

Figure 8-3: Distribution of Historical Samples with Original Laboratory Results (Original Historic Assay Intervals are Indicated in Magenta, Recent Assays are Indicated in Green)



Source: Wood, 2022.

Figure 8-4: Spatial Availability of QA/QC Data



Source: Trilogy Metals, 2019.

8.2.3 NovaGold/Trilogy (2004 to 2019) QA/QC Results Review

All core and pulp reject samples submitted to the ALS Minerals laboratory since 2004 were accompanied by standard, blank and duplicate control samples. Secondary laboratory check samples were analysed at Acme in Vancouver or SGS Burnaby. The secondary laboratory check samples were selected to represent the data population using a random selection of 5% of the samples within percentile range groups.

GeoSpark has prepared several reports summarizing the control sample results received between 2004 and 2019. The reports include CRM and blank control charts, pulp duplicate absolute difference precision charts and check sample scatter and absolute difference charts. The standard and blank control sample results were defined as failing when results were in excess of plus and minus three standard deviations from the expected mean for the standard. No threshold for check samples was provided in the reports.

Failing blanks or standards were re-analysed along with the adjacent samples to address potential accuracy deficiencies and to maintain quality assays in the database. Initial review of the assay certificates as they were reported identified a few instances of failing standards; for any fails the adjacent samples were also rerun.

The following QA/QC sections are based on a review of those reports.

8.2.3.1 2004

The 2004 exploration program included drilling and sampling of 11 drill holes amounting to 989 primary samples assayed within 61 assay certificates reported by ALS Minerals. The review of pulp duplicates, blanks and standards, and check control samples results showed acceptable levels of precision, accuracy, and between-laboratory bias.

8.2.3.2 2005

The 2005 exploration program included drilling and sampling of nine drill holes, AR05-0089 through AR05-0097, amounting to 1,228 primary samples assayed within 36 assay certificates reported by ALS Minerals.

The review of pulp duplicates, blanks and standards, and check samples during allowed for inference of a reasonable level of precision, good accuracy, and insignificant levels of bias within the primary sample results reported by ALS Minerals related to the 2005 data.

This detailed QA/QC review on the analytical results reported during 2005 allowed for overall confidence in the analytical result quality.

8.2.3.3 2006

The 2006 exploration program included drilling and sampling of 12 drill holes, AR06-98 through AR06-109, amounting to 1,175 primary samples analysed at ALS Minerals.

The review of pulp duplicates, blanks and standards, and check samples for the 2006 program allowed for inference of a good level of precision, good accuracy, and insignificant levels of bias within the primary 2006 sample results reported by ALS Minerals.

8.2.3.4 2007

The 2007 exploration program included drilling and sampling related to four drill holes, AR07-110 through AR07-113, amounting to 950 primary samples analysed at ALS Minerals.

The review of pulp duplicates, blanks and standards, and check samples for the 2007 program allowed for inference of a good level of precision, good accuracy, and insignificant levels of bias within the primary sample results reported by ALS Minerals.

8.2.3.5 2008

The 2008 exploration program included drilling and sampling related to 14 drill holes, AR08-0114 through AR08-0126 and drill hole AR08-0117w, amounting to 1,406 primary samples assayed within 44 assay certificates reported by ALS Minerals.

The review of pulp duplicates, blanks and standards, and check samples for the 2008 program allowed for inference of a reasonable level of precision, good accuracy, and insignificant levels of bias within the primary sample results reported by ALS Minerals.

8.2.3.6 2011 (analysed in 2013)

Laboratory assay certificates FA13021131, FA13021132, FA13021133, FA13021134, and FA13021135 included results for six pulp duplicate pairs, six blank instances, and three standards. There were analysed by Geospark.

The duplicates for gold, silver, copper, lead, and zinc were found to correlate well with the primary sample results and it can be inferred that the primary results are of good precision.

Each of the blanks was analytical values within the control limits for the material. There are no issues with sample contamination and instrumentation difficulties. In addition, the accuracy can be inferred to be acceptable.

Each standard returned within the acceptable range for gold, silver, copper, lead, and zinc values; it is inferred that there is strong accuracy within the reported primary sample assay results.

A detailed review of secondary laboratory check sample results reported by ALS Minerals for the 2011 drill holes assayed in 2013 showed that the gold, silver, copper, lead, and zinc results indicated no material bias.

8.2.3.7 2015

The 2015 exploration program included drilling and sampling related to 14 drill holes, amounting to 1,0971 primary samples primary samples analysed at ALS Minerals. The review of pulp duplicates, blanks and standards, and check control samples results showed acceptable levels of precision, accuracy, and between-laboratory bias.

8.2.3.8 2016

The 2016 exploration program included drilling and sampling related to 13 drill holes, amounting to 1,181 primary samples primary samples analysed at ALS Minerals. The review of pulp duplicates, blanks and standards, and check control samples results showed acceptable levels of precision, accuracy, and between-laboratory bias.

8.2.3.9 2017

The 2017 exploration program included drilling and sampling related to 5 drill holes, amounting to 313 primary samples primary samples analysed at ALS Minerals. The review of pulp duplicates, blanks and standards, and check control samples results showed acceptable levels of precision, accuracy, and between-laboratory bias.

8.2.3.10 2019

The 2017 exploration program included drilling and sampling related to 9 drill holes, amounting to 586 primary samples primary samples analysed at ALS Minerals. The review of pulp duplicates, blanks and standards, and check control samples results showed acceptable levels of precision, accuracy, and between-laboratory bias.

8.2.4 Density Determination QA/QC

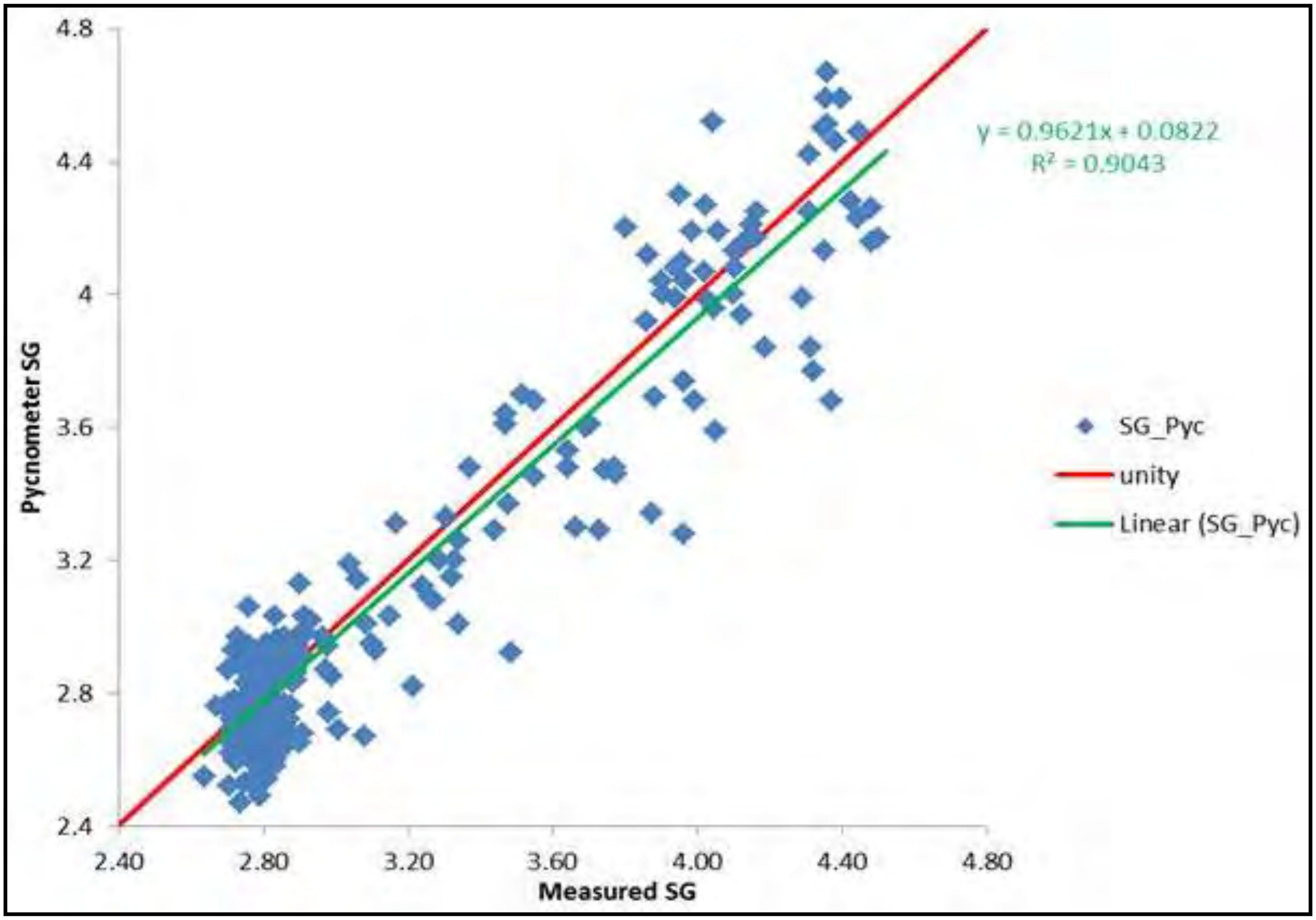
8.2.4.1 Laboratory vs. Fields Determinations

Paired laboratory and field determinations from for mineralized zone SG measurements from 1998 and the 2004 program show very low variation.

In 2010, NovaGold measured 50 unwaxed samples representing a full range of SG values for a variety of lithologies and then submitted the samples to ALS Minerals for wet-dry SG determinations after being sealed in wax. The mean difference between the NovaGold unwaxed and the ALS Minerals waxed SG determinations was 0.01.

In 2011, NovaGold submitted 266 pulps to ALS Minerals for determination of SG by pycnometer (ALS code OA-GRA08b). Figure 8-5 shows the paired pycnometer and whole core water immersion results compare well. The chart shows the pycnometer results display a slight low bias. Generally, intact samples are considered more acceptable for accurate SG determinations since they more closely resemble the in-situ rock mass.

Figure 8-5: Graph showing Good Agreement between Wet-Dry Measured Specific Gravity and Pycnometer Measured Specific Gravity

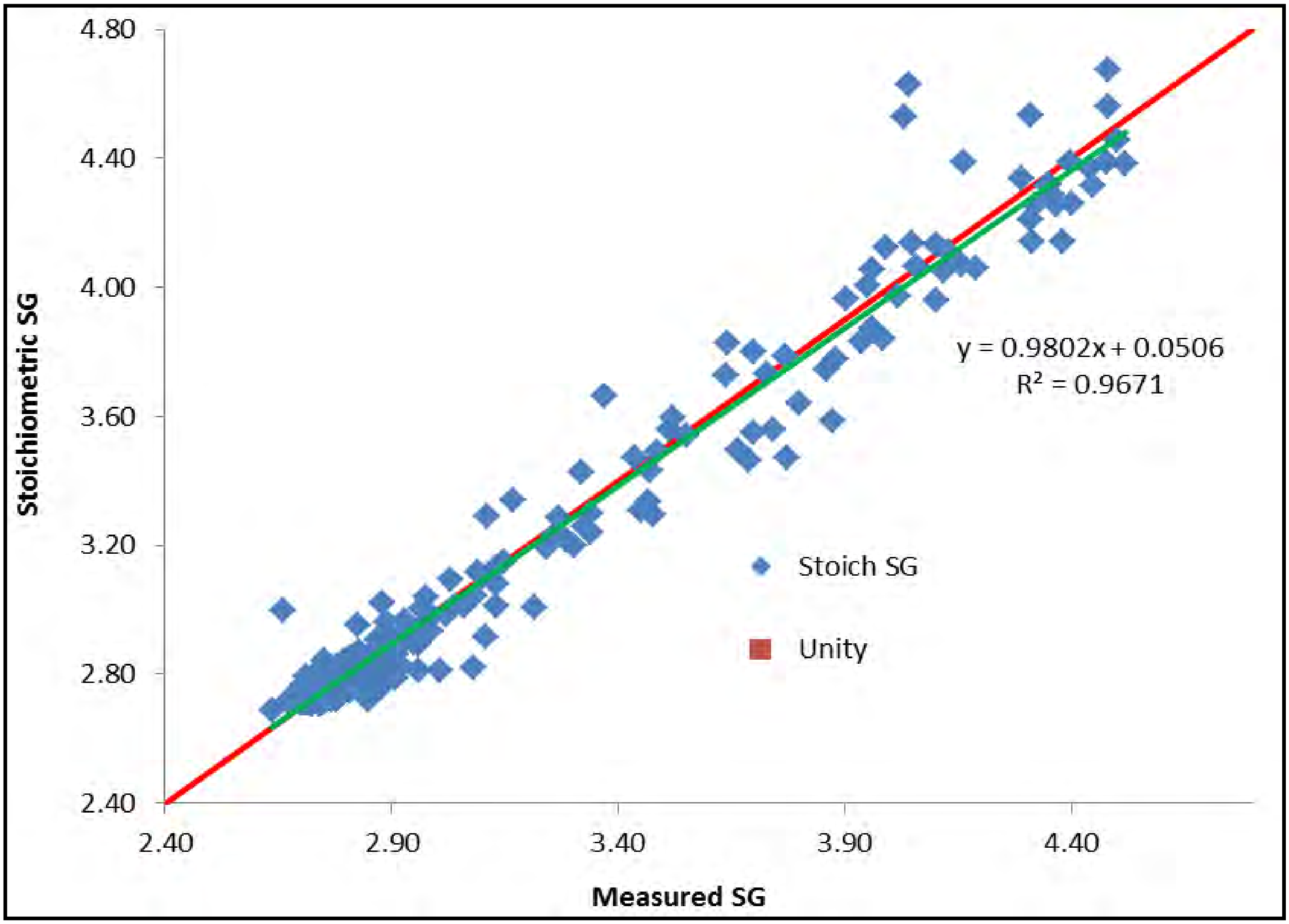


Source: West, A., 2014

8.2.4.2 Stoichiometric Method

Trilogy compared full sample length water immersion SG determinations with stoichiometric calculated SG values for 279 sample intervals that have copper, zinc, lead, iron, and XRF barium results, the major constituents of the sulphide and sulphate species for the Arctic deposit. Figure 8-6 shows That overall, there is good correlation between the two SG populations although the stoichiometric estimates are slightly lower with increasing SG.

Figure 8-6: Measured vs. Stoichiometric Specific Gravities



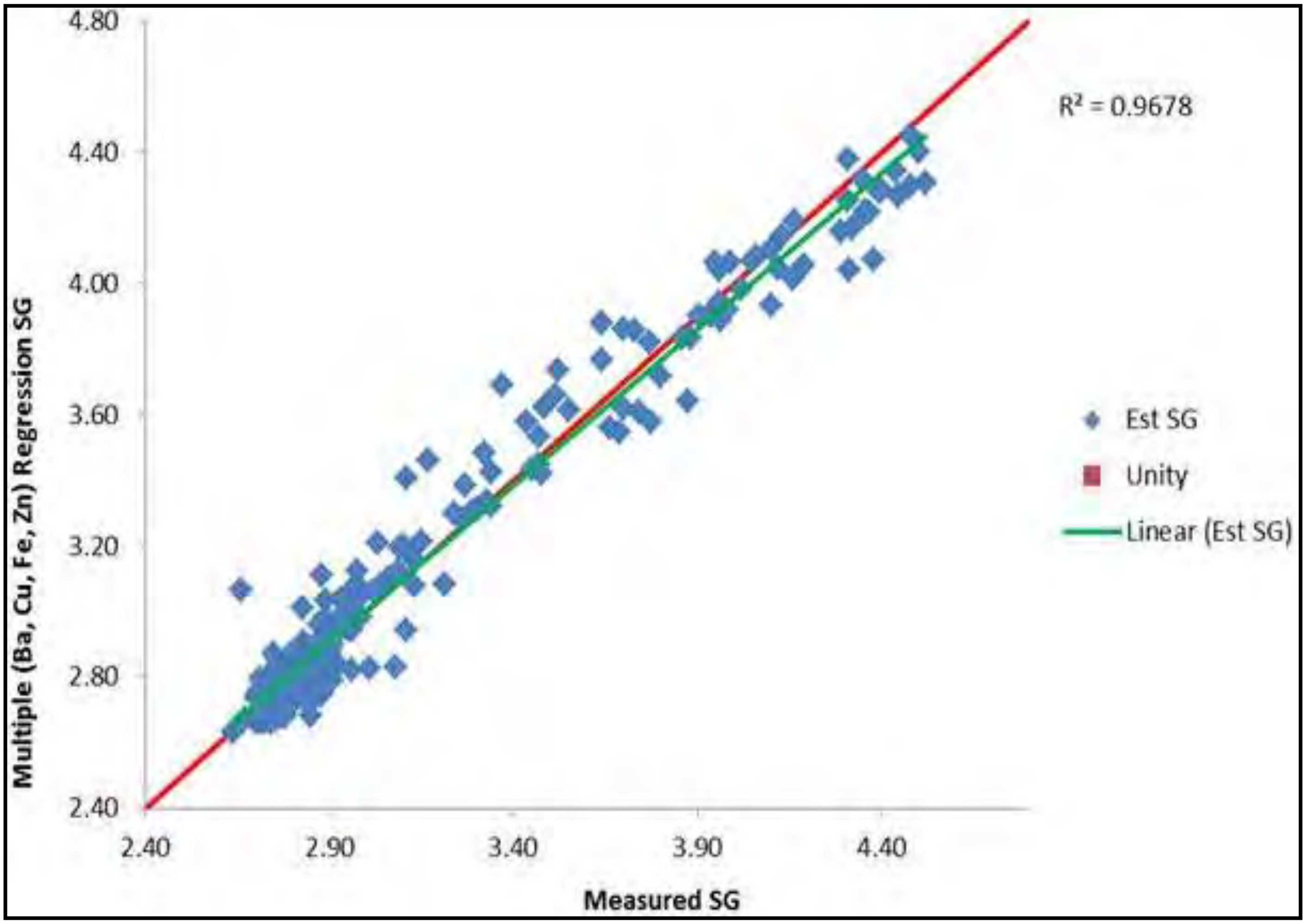
Source: West, A., 2014.

8.2.4.3 Multiple Regressions Method

The best fit to the data was achieved by using the multiple regression tool in Microsoft Excel on barium, iron, zinc, and copper for the entire dataset (Figure 8-7). The estimate correlates very well (R2=0.9678) with observed data and has a sinusoidal pattern that fits the low and moderately high SG very well and has high bias for moderate SG values and a low bias for very high SG values. The resultant SG formula is as follows:

$$SG_{(Regression)} = 2.567 + 0.0048*Cu_{(wt\%)} + 0.045*Fe_{(wt\%)} + 0.032*Ba_{(wt\%)} + 0.023\%*Zn_{(wt\%)}$$

Figure 8-7: Scatter Plot Showing the Measured Specific Gravity vs. Multiple (Copper, Iron, Zinc, Barium) Regression

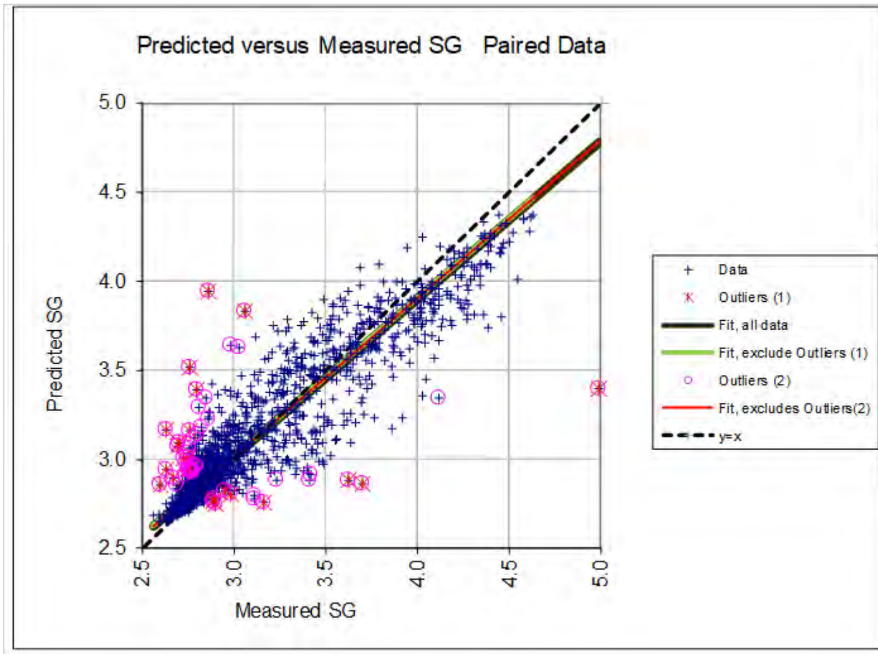


Source: West, A., 2014.

8.2.4.4 Random Forest Machine Learning Method

A random forest classification method was used to assist prediction of SG values for 12,039 sample intervals. Information regarding the inputs to the random forest method were not available for review. A comparison of 8,542 paired predicted and paired measured values shows a reasonable correlation between measured and random forest assisted prediction values but an obvious low bias low bias for predicted values (Figure 8-8). The bias is unexplained but may be a result of insufficient Ba analyses. The Random Forest predicted SG values were used in the current resource estimate and the impact of the bias is discussed in Section 11.5.1.

Figure 8-8: Reduced to Major Axis (RMA) Scatter Plot Showing the Predicted and Measured SG Values



	All Data	Excluding Outliers (1)	Excluding Outliers (2)
N	1849	1824	1799
Percent Rejected	0.0%	1.4%	2.7%
R squared	0.8645	0.8847	0.8943
R	0.9298	0.9406	0.9457
slope m	0.8852	0.8890	0.8903
intercept b	0.3619	0.3491	0.3438
error in slope	0.0076	0.0071	0.0068
error in intercept	0.0234	0.0218	0.0211

Source: Wood, 2022.

8.2.4.5 Comments on Density Determinations

Stoichiometric and multiple regression method results generally produce SG values that compare well with measured results. SG values predicted using random forest methods produce biased results. In the opinion of the QP, use of these predicted SG valued for estimation of SG in the block model may introduce a significant low bias for high SG areas.

8.2.5 Acid-Base Accounting Sampling QA/QC

SRK conducted a QA/QC review of the 2010 ABA dataset for the Arctic Project in July 2011 and concluded that data quality was acceptable.

QA/QC of the ABA dataset generated in 2015 and 2016 was conducted by SRK in November 2016 through January 2017 and data quality was concluded to be acceptable.

SRK conducted a QA/QC review of the ABA dataset from the various kinetic test samples in July 2019, June 2021, and November 2022, concluding that the data quality was acceptable.

SRK conducts monthly QA/QC review of kinetic test leachates for all operating kinetic tests. The kinetic test program also includes duplicate and blank tests. Data are reviewed for ion balance, potential contamination, reproducibility, changes in long-term trends, and anomalous spikes in the data. Where considered necessary by SRK, the laboratory is asked to rerun leachates from kinetic tests to investigate QC concerns.

8.3 Comments on Sample Preparation, Analyses and Security

The drill core sampling procedures at site, the primary laboratory sample preparation and analytical procedures, and the QAQC and security procedures applied by NovaGold and Trilogy for samples collected and analyzed since 2004 are in the QP's opinion appropriate for the mineralization style observed at Arctic and provide adequate confidence in the reported assay values. Historical copper and lead values (pre 2004) that remain in the primary assay database appear to be biased high and low, respectively. The broad spatial distribution of these original historical samples and density of samples with more recent assay values surrounding these samples in the QP's opinion reduces the risk associated with these observed biases and are suitable to be used in resource estimation.

9 DATA VERIFICATION

Wood reviewed database verification and laboratory QA/QC reports and made data entry error spot checks, inspected down hole survey results for anomalous kinks and excessive bends in the drill hole traces, reviewed reports summarizing the results of drill core sampling and assaying completed since 2004, reviewed the assay database for gaps and overlaps, and reviewed the historical re-assay program results. The following two significant issues were observed:

- A significant high bias in historic Cu and low bias in historic Pb assay results (see Section 8).
- Apparent low bias in Random Forest assisted specific gravity predictions (see Section 8).

Excessive between-survey deviation was observed in 30 historical drill holes and transcription errors related to priority of KCC/Utah laboratory assay results were observed in two randomly selected holes. Database adjustments by NovaGold and NovaCopper include resurvey and transformation of historical hole collar coordinates from UTM NAD27 feet to UTM NAD83 metres, historical down hole survey bearings from quadrant to azimuth and down hole survey depth from feet to metres, and gold and silver results from ounces per ton to grams per tonne. A spot comparison comparing original result documents and the database revealed transcription errors and inconsistencies in data selection priority. In the QP's opinion, these issues are not expected to have a material impact on the grade estimation but should be resolved in the next model.

In the current assay table historical sample interval assay results are given priority over the historic sample interval re-assay results. In the QP's opinion, this is not expected to have a material impact on the grade estimation but using the re-assay results would further mitigate the risk associated with the observed biases in the historical Cu and Pb values, as discussed in Section 8.

There are no QAQC analysis for the deleterious variables (As, Sb, Cd, Hg, Fe, S).

Overall, the database verification and management and the laboratory QAQC monitoring completed by NovaGold, Trilogy and Ambler Metals has resulted in a reasonably reliable drill hole database suitable for supporting the Mineral Resource estimated for the Arctic deposit. Some deficiencies exist that when rectified will make the drill hole database even more robust.

As referenced in Section 6, the interplay between the complex local stratigraphy, the isoclinal F1 event, the overturned south verging F2 event and the series of post-penetrative deformational events often makes district geological interpretation extremely difficult at a local scale. The current geological model captures the complexity reasonably well, but it is possible there is more variability at the local scale than is modelled. Tight spaced drilling in key areas of the deposit is warranted.

9.1 Comments on Data Verification

It is the QP's opinion that the drill database and topographic information for the Arctic deposit are reliable and sufficient to support the current estimate of Mineral Resources.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 Introduction

Metallurgical studies have spanned over 30 years with metallurgical testwork campaigns undertaken at the Kennecott Research Centre (KRC) in Salt Lake City, Utah; Lakefield Research Ltd., Lakefield Ontario (Lakefield); SGS Mineral Services, Burnaby, BC (SGS); and ALS Metallurgy, Kamloops, B.C. (ALS Metallurgy). Metallurgical testwork laboratories are typically not accredited. The KRC was not independent of Kennecott at the time the testwork was completed; all other laboratories were and are independent of NovaGold, Trilogy and Ambler Metals.

The current accreditation status of the KRC is unknown. Lakefield joined the SGS group in 2002. SGS and ALS Metallurgy conform to the requirements of ISO/IEC 17025 for specific tests as listed on their respective scope of accreditation documents, which can be found at www.scc.ca/en/search/laboratories.

The testwork conducted in 2012 through 2019 was under the technical direction of International Metallurgical and Environmental Inc (IME). Testwork prior to 2012 is considered historical in nature and has provided some guidance to the project development but is not used in any predictive manner or used in the generation of design criteria. The testwork was focused on a conventional process flowsheet employing crushing, grinding, bulk flotation of a copper and lead concentrate, flotation of a zinc concentrate and the subsequent separation of copper and lead values via flotation.

Various metallurgical testwork programs were conducted in 2021 through 2022 at ALS Metallurgy, SGS, and Metso-Outotec Group (Metso Group). ALS Metallurgy completed several testwork programs, including flotation testing with the pre-flotation circuit only to establish talc performance; further flowsheet development testwork to investigate the benefits of sequential flotation versus the original bulk flow sheet; and a variability testwork to support the development of improved metallurgical recovery models.

SGS conducted SAG Power Index (SPI®) tests to investigate the effect of friable ores on the plant throughput.

Metso Group conducted talc circuit modelling using the data obtained from the ALS Metallurgy pre-flotation testwork program to investigate the benefits of talc circuit open and closed-circuit cleaning. The Mesto Group also conducted dewatering and filtration testwork on the pre-flotation concentrate and final tailings generated from the pre-flotation testwork program.

The LOM average metallurgical performance forecasts, based on recent testwork completed and expected mine production grades is shown in Table 10-1. This overall project metallurgical accounting is based on locked cycle testwork, conducted on a distribution of samples from the deposit. Since 2012, testwork has been focused on optimizing the performance of the recommended flowsheet.

Table 10-1: Summary of Overall Forecast Metal Recovery – Arctic Deposit

Process Stream	Mass %	Concentrate Grade					Metal Recoveries				
		Cu %	Pb %	Zn %	Au g/t	Ag g/t	Cu %	Pb %	Zn %	Au %	Ag %
Process Feed	100.0	2.11	0.56	2.90	0.42	31.8	-	-	-	-	-
Copper Conc.	6.3	30.3	1.7	0.7	3.4	160.5	92.1	19.4	1.6	52.2	32.4
Lead Conc.	0.6	2.0	53.9	5.9	14.1	2425.8	0.6	61.3	1.3	21.4	48.8
Zinc Conc.	4.7	1.0	0.5	53.6	0.3	38.3	2.2	4.4	88.4	3.2	5.7
Tailings	88.4	0.1	0.1	0.3	0.1	4.6	5.1	14.8	8.7	23.2	13.1

A summary of the testwork programs completed for the Arctic Project, dates of testwork and testwork objectives is shown in Table 10-2.

Table 10-2: Summary of Testwork Chronology and Reporting

Laboratory	Project No.	Report Date	Testwork Objectives
Historical Testwork			
KRC	-	1970-1976	Preliminary mineralogy and flotation testing
Testwork from 2012 to 2019			
Lakefield	-	1999	Preliminary flotation testwork.
SGS Burnaby	50173-001	Oct. 4, 2012	Flotation scoping and locked cycle testing (LCT), Bond work index, (BWi), using 4 large composite samples
ALS Metallurgy	KM5000	Mar. 27, 2017	Flotation scoping and LC Testing, BWi, using a master composite and 14 variability samples, preliminary copper/lead separation testwork.
ALS Metallurgy	KM5372	July 11, 2017	Additional copper/lead separation testwork and detailed precious metal mineralogy
ALS Metallurgy	KM5567	Feb. 26, 2018	Talc optimization testwork and copper/lead separation testwork.
JKTech	18017/P14	July 2018	Drop Weight testing of Comp. sample
JKTech	19017/P16	Sept. 2019	SMC testing of Variability Samples
Inter. Metallurgy	-	April 22, 2019	Cyanide destruction testwork
Pocock Industrial	-	August 2019	Thickening and filtration testing
Testwork from 2021 to 2022			
Metso-Outotec Group	-	Nov. 3, 2021	Talc pre-flotation simulations
Metso-Outotec Group	93730- TQ1-TM001-R0	Oct 8, 2021	Dewatering testing for Talc concentrate and final tailings.
Metso-Outotec Group	93720- TQ1-TM001-R1	Oct 13, 2021	Filtration testing for Talc concentrate and final tailings.
ALS Metallurgy	KM6442	Dec. 15, 2021	Talc optimization testwork
SGS Burnaby	19157-01	June 3, 2022	SAG Power Index testing on 15 samples
ALS Metallurgy	KM6498	May 24, 2022	Flotation development testwork
ALS Metallurgy	KM6498	July 14, 2022	Flotation development and Variability testwork

Detailed testwork has concluded that the mineralization is well-suited to the production of separate copper, lead and zinc concentrates. There are no significant metallurgical impediments outside of the presence of talc minerals and fluorine levels in the lead concentrates as observed in the various testwork programs. The presence of naturally hydrophobic talc minerals was consistently observed, and talc can be effectively removed from the flotation process prior to base metal flotation. There is little reason to expect concentrates will be impaired by talc contamination as talc can be effectively removed from the base metal flotation process, mitigating the potential of talc diluting the base metal concentrates. Talc and fluorine levels will be managed by optimization of the talc pre-float circuit, effectively removing talc and fluorine to ensure the quality of the lead concentrate.

The flotation process uses industry standard flotation processes, with three major rougher flotation stages and re-grinding and flotation cleaning of copper/lead bulk concentrates and zinc concentrates. Copper and lead are separated from a bulk copper and lead concentrate. Testwork was broken into separate phases, with copper/lead separation being a key distinct phase of testing in the later stages of the program. The full-scale metal recovery and upgrading process can be well-managed with modern process control and ensuring that variability of feed grades, including talc content, can be accommodated.

Ancillary testwork including solid-liquid separation and cyanide detoxification testwork was completed.

10.2 Historical Testwork Review

10.2.1 Metallurgical Testing – Kennecott Research Centre (1968 to 1976)

Between 1970 and 1976, KRC conducted two initial mineralogical studies to evaluate and identify the potential beneficiation or metallurgical options. This early work was a cornerstone of planning later phases of testwork.

In the 1970 mineralogy investigation, KRC reported that the host rock of the mineralization is generally muscovite, chlorite, or talc schist. Principal economic minerals in the deposit were identified as chalcopyrite, sphalerite, and argentiferous galena.

The grain sizes of sulphide mineral particles ranged from sub-micron to a maximum of several centimetres; most of the sulphide particles were relatively large (coarser than 74 µm). KRC noted that the target sulphide minerals should be liberated from gangue at a primary grind size of 100% passing 100 mesh.

In 1976, KRC conducted preliminary comminution testwork using the standard BWi determination procedure (refer to discussion in Section 10.3.4).

Between 1968 and 1976, KRC carried out initial flotation testing. The focus was on selective flotation to provide separate copper, lead, and zinc concentrates for conventional smelting. In 1968, initial amenability testing was conducted on core composites from eight diamond drill holes (which is not available to review). Other tests were conducted in 1972 on four composites from three additional diamond core holes. The laboratory-scale tests conducted between 1968 and 1976 included the conventional selective flotation approach to produce separate lead, copper, and zinc concentrates.

The major problem encountered for the tests by KRC was the separation between lead and copper minerals, and the reduction of zinc deportment to the copper and lead concentrates. The copper concentrates produced from open circuit tests contained 30 to 32.4% Cu, 0.45 to 3.48% Zn and 0.15% to 1.31% Pb. The copper recoveries were < 80.7%. The lead concentrate grades were low, ranging from 17.1 to 36.5%.

Sphalerite flotation was generally efficient, producing zinc flotation concentrates grading approximately 55% zinc. Because of the low gold content of the test samples, no appraisal was made of gold recoveries.

From 1975 and 1976, large diameter cores from 14 drill holes were used for more detailed testing. Two composites labelled as Composite No. 1 (Eastern Zone) and Composite No. 2 (Western Zone), were prepared. The test program included bench-scale testing of various process parameters for sequential flotation, including locked cycle tests. A talc flotation step prior to sulphide flotation was considered to be necessary, as previously established. It was determined that chalcopyrite and sphalerite could be recovered into separate commercial grade copper and zinc concentrates. However, the production of a selective high-grade lead concentrate was not successful.

Using zinc sulphate and sodium bisulphate to suppress galena and sphalerite, 90% of the copper was recovered into a concentrate containing 26% Cu, 1.5% Pb, and 6% Zn. KRC indicated that because of close interlocking of chalcopyrite and sphalerite, the zinc content of the copper concentrate could not be reduced to below 6% without sacrifice of copper recovery.

Only low-grade silver-bearing lead concentrates were obtained. Under the best test conditions, approximately 65% of the silver reported to the low-grade lead concentrate. Some of the silver in the mineralization occurred as tetrahedrite, which was recovered to the copper concentrate. The KRC testwork did not focus on bulk copper and lead flotation and the attempt to focus on a sequential copper-lead-zinc flowsheet is considered a technical error. Subsequent testwork moved to a bulk copper-lead flotation process and subsequent separation of a bulk concentrate into copper and lead concentrates. Metallurgical results have improved with the change to a bulk copper-lead flowsheet in later testwork.

10.2.2 Metallurgical Testwork – Lakefield (1998 to 1999)

In 1998, Lakefield conducted a metallurgical test program to confirm and improve upon the results from the KRC testwork program. The Lakefield work was carried out on test composites prepared from three separate drill holes. The test composite from the upper portion of AR-72 was identified as being low in talc content; however, composites from the lower portion of AR-72 were high in talc content, as were AR-74 and AR-75. The head analyses for the respective resulting test composites are summarized in Table 10-3.

Table 10-3: Head Analysis, Lakefield Research 1999

Composite	Talc	Cu (%)	Zn (%)	Pb (%)	Fe (%)	Au (g/t)	Ag (g/t)	S^T (g/t)
Hole #72 – Upper	Low	5.28	7.16	1.86	15.6	1.14	72.3	23.4
Hole #72 – Lower	High	2.68	5.85	1.34	13.0	1.60	75.9	16.9
Hole #74	High	2.46	4.43	0.90	17.0	1.55	45.1	23.7
Hole #75	High	2.35	8.36	1.95	15.7	1.23	77.3	21.8

Note: ST = total sulphur.

Lakefield conducted a series of five flotation tests using a flowsheet similar to the one adopted in the 2012–2019 testwork and incorporated a bulk copper-lead flotation stage followed by copper and lead separation.

The bulk copper-lead rougher concentrate was reground and subjected to two stages of cleaner flotation and one stage of copper and lead separation, using zinc oxide and sodium cyanide to depress the copper while floating the lead. The resulting lead rougher concentrate was upgraded with two stages of cleaner flotation to produce the final lead concentrate. The lead rougher flotation tailings were the final copper concentrate.

The zinc rougher concentrate was reground and upgraded with two stages of cleaner flotation. The results of the best open circuit flotation test for the low talc composite are summarized in Table 10-4. The test results are indicative of the results obtained later in test programs and optimization of the process would improve these results. Of note, is the high recovery of precious metals to the lead concentrates, which was also confirmed in later testwork programs.

Table 10-4: Flotation Test on Ambler Low Talc Composite

Item	Weight (%)	Assays					Distribution (%)				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu	Pb	Zn	Au	Ag
Lead Concentrate	2.22	6.5	58.8	3.43	38.9	1,703	2.7	68.1	1.1	48.7	47.3
Copper Concentrate*	15.76	29.1	1.2	2.61	1.23	73.5	86.8	9.8	5.7	10.9	14.5
Zinc Concentrate	9.91	0.44	0.36	59.1	0.65	14.7	0.8	1.9	81.1	3.6	1.8
Zinc Tailings**	61.6	0.11	0.13	0.22	0.4	3.47	1.2	4.3	1.9	13.7	2.7
Head (Calculation)	100.0	5.28	1.92	7.21	1.78	80.1	100.0	100.0	100.0	100.0	100.0

Notes:
*Pb Rougher Tailings
**Does not include intermediate cleaner tailings.

Lakefield also conducted flotation tests on each of the high talc composites using a test procedure similar to the one used for the low talc composite, with the exception that carboxymethyl cellulose (CMC) was added as a depressant for talc. The results of these tests showed that the presence of talc had a significant negative impact on the copper and lead mineral recoveries. Lakefield also used talc pre-flotation prior to sulphide flotation in an effort to reduce talc effect on base metal flotation. It appears that the talc pre-flotation improved copper and lead metallurgical performances. However, the test results showed that elevated talc content had a significant effect in copper and lead flotation response.

In the test report, Lakefield also concluded that grind particle size as coarse as approximately 80% passing 74 µm provided good results.

10.3 Mineralogical and Metallurgical Testwork – 2012 to 2019

10.3.1 Introduction

Testwork conducted prior to 2012 is considered relevant to the Arctic Project, but predictive metallurgical performance is best estimated from testwork conducted on sample materials obtained from exploration work under the direction of Trilogy, conducted from 2012 to 2019 (refer to Table 10 2).

In 2012, SGS conducted a test program on the samples produced from mineralization zones 1, 2, 3, and 5. Drill core samples were composited from each of the zones into four different samples for the SGS testwork which included process mineralogical examination, grindability and flotation tests.

SGS used quantitative evaluation of materials by scanning electron microscopy QEMSCAN™, to develop grade limiting/recovery relationships for the composites.

Standard Bond grindability tests were also conducted on five selected samples to determine the BWi and abrasion index (Ai).

The flotation testwork investigated the effect of various process conditions on copper, lead and zinc recovery using copper-lead bulk flotation and zinc flotation followed by copper and lead separation. The testwork conducted in 2012 at SGS forms the basis for predicting metallurgical performance of the mineralized zone in terms of recovery of copper and lead to a bulk concentrate as well as predicting zinc recovery to a zinc concentrate.

In 2017, testwork moved to ALS Metallurgy and was focused on predicting the expected performance of the proposed copper and lead separation process, which required the use of larger test samples. A pilot plant was operated to generate approximately 50 kg of copper and lead concentrate, which became test sample material for use in locked cycle testing of the copper and lead separation process. This testwork allows for the accurate prediction of copper and lead deportment in the process as well as provided detailed analysis of the final copper and lead concentrates, expected from the process. Additional metallurgical testwork in the form of variability samples being subject to grindability and baseline flotation tests was also completed.

10.3.2 Test Samples

The 2012 test program used 90 individual drill core sample intervals totalling 1,100 kg. Individual samples were combined into four composites representing different zones and labelled as Composites Zone 1 & 2, Zone 3, Zone 5, and Zone 3 & 5. The sample materials used in the 2012 test program at SGS were specifically obtained for metallurgical test purposes. The drill cores were stored in a freezer to ensure sample degradation and oxidation of sulphide minerals did not occur.

The head grades of the composites from the 2012 SGS testwork program is shown in Table 10-5.

Table 10-5: SGS Burnaby Head Grades – Composite Samples – 2012

Sample	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	MgO (%)
Zone 1 & 2	2.63	0.95	3.43	7.73	8.36	0.79	57.6	5.78
Zone 3	3.56	1.73	8.58	17.5	25.8	0.67	80.4	1.94
Zone 3 & 5	4.41	1.60	7.76	16.7	23.5	0.97	82.0	3.96
Zone 5 B	2.56	1.33	5.68	15.8	21.2	1.16	63.0	0.90

The 2017 test program involved the collection of approximately 4,000 kg of drill core from five drill holes. The core was shipped in its entirety to ALS Metallurgy for use in grinding and flotation testwork. Fifteen separate composites samples were generated by crushing defined intercepts of mineralization. These samples were riffle split to generate 15 individual samples which were separately tested for grindability and flotation response, as well, a large portion of each sample was blended to make a single large composite sample for use in copper-lead separation testwork. The copper-lead separation testwork involved operating a pilot plant for the production of a single sample of copper/lead concentrate which was then used in bench-scale flotation testing, including open circuit flotation tests as well as locked cycle flotation tests.

The feed grades of samples used in the 2017 testwork program at ALS Metallurgy are shown in Table 10-6.

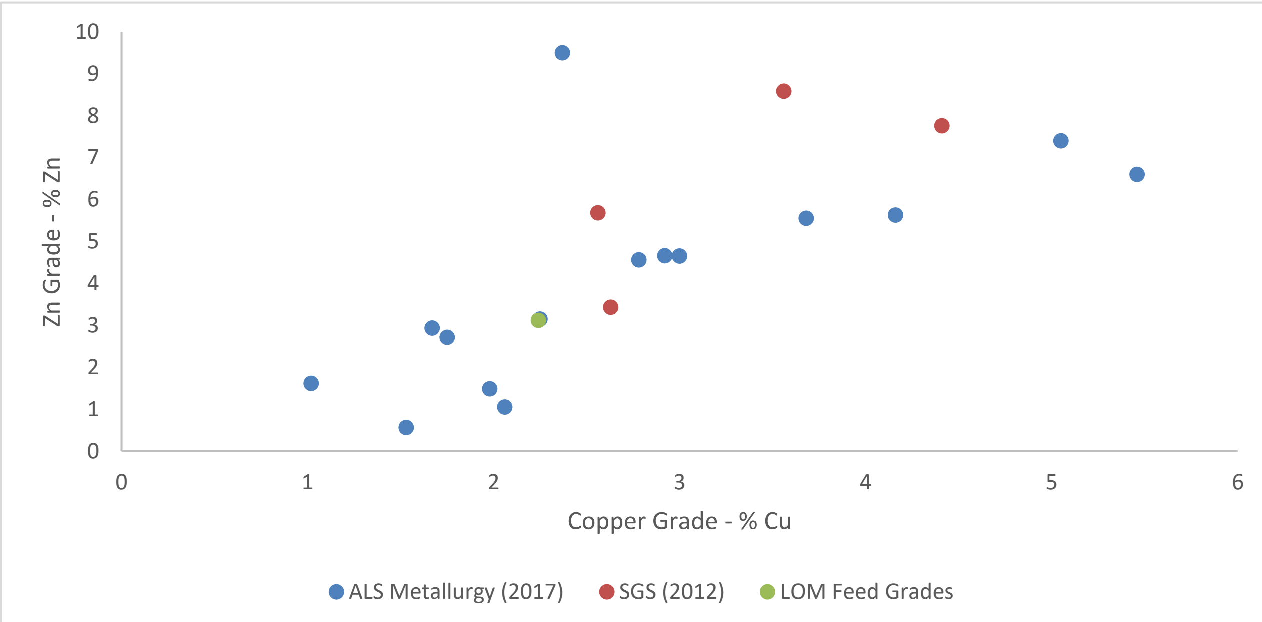
Table 10-6: ALS Metallurgy Head Grades – Composite Samples – 2017

Sample ID	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)	Mg (%)
Var. Comp 1	5.05	1.53	7.40	15.0	24.4	0.68	64	2.69
Var. Comp 2	2.06	0.25	1.05	4.6	3.68	0.52	34	11.2
Var. Comp 3	1.67	0.80	2.93	6.6	4.93	0.10	43	7.51
Var. Comp 4	2.25	0.24	3.15	13.1	16.2	0.20	18	6.26
Var. Comp 5	3.68	1.01	5.55	10.6	13.9	0.78	69	7.16
Var. Comp 6	1.02	0.36	1.61	8.0	9.45	0.45	24	1.92
Var. Comp 7	1.75	0.58	2.71	8.9	12.9	0.21	32	3.46
Var. Comp 8	3.00	0.68	4.65	10.0	13.3	0.75	56	9.18
Var. Comp 9	5.46	1.37	6.60	9.2	14.0	0.15	50	5.65
Var. Comp 10	4.16	1.24	5.63	13.4	22.1	0.06	34	3.58
Var. Comp 11	2.78	0.40	4.56	12.7	16.9	0.64	40	6.84
Var. Comp 12	1.53	0.07	0.56	4.4	3.15	0.59	16	10.6
Var. Comp 13	1.98	0.30	1.48	6.2	6.25	0.26	38	9.61
Var. Comp 14	2.37	2.43	9.50	15.1	23.7	0.84	62	0.81
PP Comp.	2.92	0.86	4.66	10.8	13.8	0.56	41	5.93

Shown in Figure 10 1 are the copper and zinc grades of the SGS and ALS Metallurgy testwork samples compared to the designed plant feed grades. The ALS Metallurgy samples are more representative of the expected mine production due to the lower range of copper and zinc grades that are available. There is a strong correlation between copper and zinc grades within the test samples. The metallurgical balance shown in Table 10-1 is based on LOM feed grades and is consistent with the LOM data point shown in Figure 10 1.

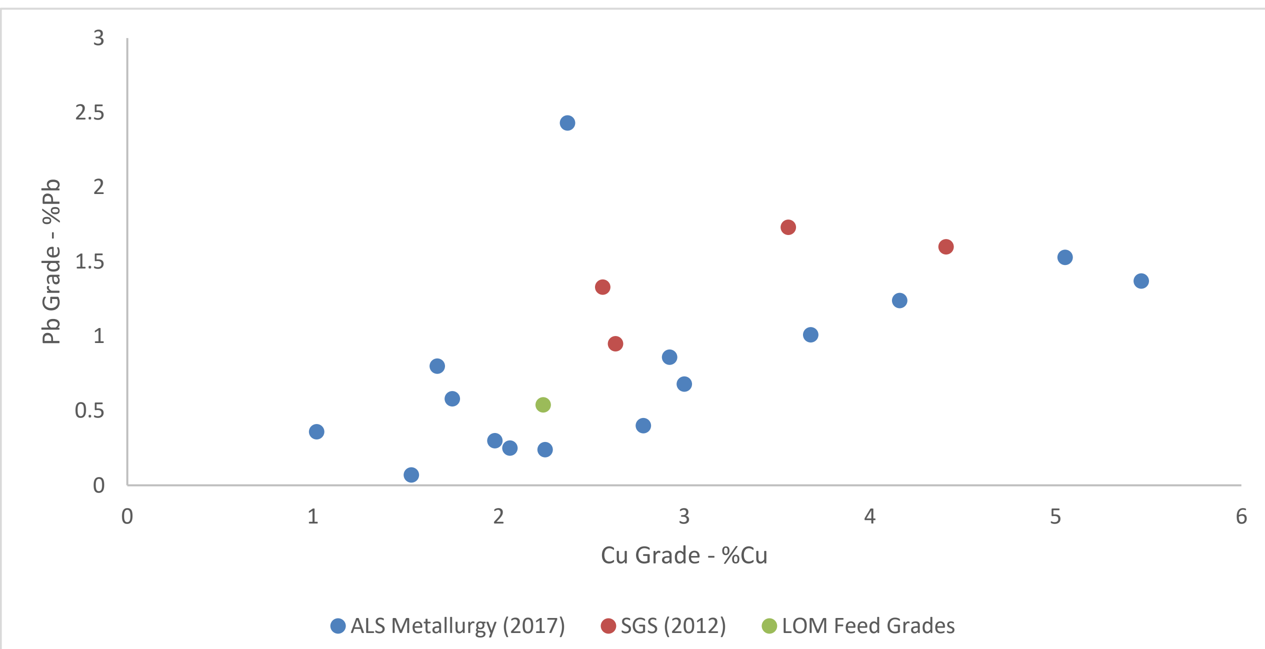
Shown in Figure 10 2 are the copper and lead grades of the various test samples used in the ALS Metallurgy and SGS test programs. There is a consistent copper to lead ratio of approximately 3.5–4.5 within the test samples and the LOM grades are shown to be within the distribution of test samples.

Figure 10-1: Cu and Zn Test Sample Grades for ALS Metallurgy/SGS Burnaby Programs



Source: Austin, 2020.

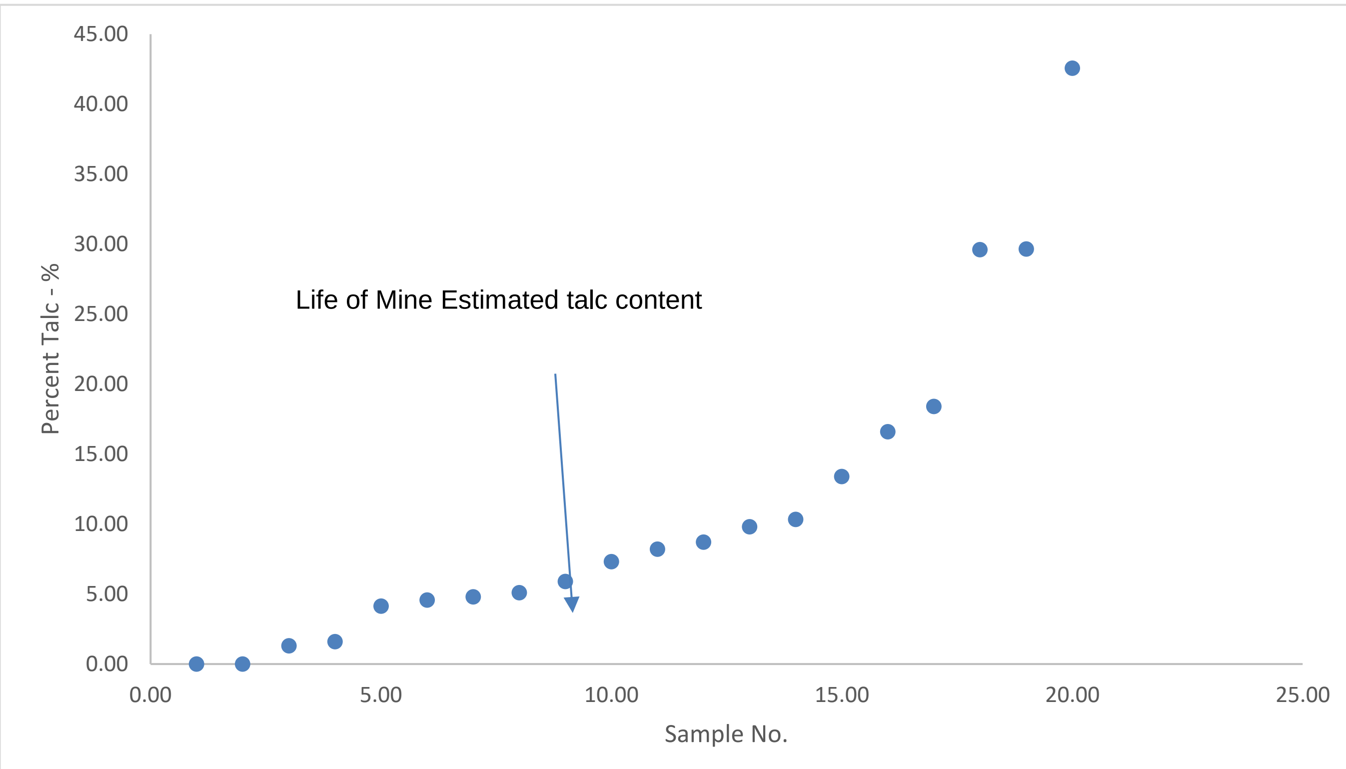
Figure 10-2: Cu and Pb Test Sample Grades for ALS Metallurgy/SGS Burnaby Programs



Source: Austin, 2020.

The volume of talc observed in the various test samples ranged from a low of zero contained talc to a high of 42% by weight floatable talc. Shown in Figure 10 3 is the distribution of talc within both the SGS and ALS Metallurgy samples. The estimated LOM talc content is estimated at 5.1%. Only one of the SGS test samples exceeded the LOM talc content, while 11 of the 14 ALS Metallurgy samples exceeded the LOM talc content. A composite of the ALS Metallurgy samples was approximately twice the talc content of the expected mine production. Talc, while a significant issue for the flotation process is likely overestimated in terms of its potential negative impact, owing to the large number of high talc samples seen in the ALS Metallurgy sample set. It has also been clearly demonstrated that even high volumes of talc can be effectively removed from the base metal flotation process and remove the impact of talc diluting base metal concentrates.

Figure 10-3: Distribution of Talc Content within the 2012 and 2017 Test Samples



Source: Austin, 2020.

10.3.3 Mineralogical Investigations

SGS used QEMSCAN™ to complete a detailed mineralogical study on each composite to identify mineral liberations and associations. The mineral modal abundance for the composites is shown in Table 10-7.

The mineralogical results obtained by SGS were typical for the balance of other samples observed in the ALS Metallurgy QEMSCAN analysis and no significant liberation or changes in mineral occurrences were observed in later QEMSCAN work. Metal and talc grades all were observed to be variable in the mineralogical evaluations but did not significantly impact textural relationships.

Table 10-7: Mineral Modal Abundance for Composite Samples – SGS Burnaby 2012

Mineral	Mass (%)			
	Zone 1 & 2	Zone 3	Zone 3 & 5	Zone 5
Chalcopyrite	9.2	9.4	12.2	6.4
Bornite	0.02	0.01	0.03	0.4
Tetrahedrite	0.1	0.4	0.2	0.2
Antimony	0.03	0.2	0.005	0.3
Galena	1.3	2.1	2.1	2.1
Sphalerite	7.2	14.6	14.3	11.3
Pyrite	6.7	30.4	23.8	27.8
Pyrrhotite	2.2	0.2	0.2	1.4
Arsenopyrite	0.5	0.1	0.6	2.2
Other Sulphides	0.1	0.1	0.2	0.1
Quartz	30.2	8.6	9.0	16.6
Feldspar	0.9	0.2	0.4	0.3
Magnesium-Chlorite	11.9	3.4	2.8	1.1
Talc	2.0	0.8	6.3	0.1
Micas	14.2	1.9	7.0	9.4
Cymrite	3.5	3.9	1.8	1.9
Clays	0.6	0.05	0.2	0.1
Iron Oxides	0.3	0.3	0.5	0.3
Carbonates	3.4	1.3	4.2	2.0
Barite	3.0	21.8	13.4	14.5
Fluorite	1.7	0.1	0.4	1.2
Other	1.1	0.3	0.4	0.4
Total	**100.0**	**100.0**	**100.0**	**100.0**

The mineralogical study showed that the mineralogy of all four composites was similar, but mineral volumes were highly variable between samples. Each composite was composed mainly of pyrite, quartz, and carbonates. However, Composite Zone 1 & 2 contain approximately 30% quartz, compared to 8.6% for Composite Zone 3, and 16.6% for Composite Zone 5. The study also showed that Composite Zone 1 & 2 had the lowest pyrite content (6.7%) while Composites Zone 3 and Zone 5 contained approximately 30.4% and 27.8% pyrite, respectively.

In all samples, the major floatable gangue minerals were talc and pyrite. Chalcopyrite was the main copper carrier. Combined bornite, tetrahedrite, and other sulphides accounted for less than 5% of the copper contained in the various samples. Galena was the main lead mineral and sphalerite was the main zinc mineral.

All the composites contained a significant amount of talc, which may have the potential to dilute final concentrates. Therefore, SGS recommended that talc removal using flotation be employed prior to base metal flotation and this recommendation was carried throughout all testwork.

At a grind size of approximately 80% passing 70 µm, chalcopyrite liberation ranged from approximately 80 to 87% (free and liberated combined) for all composites. The chalcopyrite is mostly free, with 7 to 10% associated with pyrite. For all composites, galena liberation ranged from 54 to 68% (free and liberated combined). Sphalerite liberation varied between 81 to 89%. Sphalerite is mostly free with about 7 to 10% associated with pyrite. Mineral liberation plays a significant role in removing talc during the pre-float stage and finer grinding benefits the removal of talc prior to sulphide flotation and subsequent re-grinding.

10.3.4 Comminution Testwork

KRC completed BWi tests on seven specific samples from the Arctic project in the 1976 testwork program. SGS conducted comminution tests on five selected samples during their testwork program in 2012. The SGS tests included the BWi tests and Ai tests. ALS Metallurgy also conducted BWi determinations on a number of samples during the 2017 to 2019 testwork program and these results from all three programs are summarized in Table 10-8. The BWi values range from 6.5 to 11 kWh/t for the materials sampled. The data indicates that the samples are relatively soft to ball mill grinding. The Ai ranged from 0.017 to 0.072 g, which indicates that the samples are not abrasive.

Grinding testwork for semi-autogenous grind (SAG) mill characterization was completed in conjunction with the ALS Metallurgy 2017 work. JKTech completed SMC and DWi testing of samples specifically selected for SAG amenability testing. Additional BWi determinations were also completed on the samples selected for SMC testwork; those BWI test results are included in Table 10-9 with the SMC results. Drill core data for the various SMC samples are contained in Table 10-8.

A total of 46 BWi tests were completed.

Table 10-8: Bond Ball Mill Work Index and Abrasion Index Test Results

Sample	Mesh of Grind Size	P_{80} (µm)	BWi (kWh/t)	Ai (g)
1976 KRC samples				
Hole – 11B	150	-	11.96	-
Hole – 34B	150	-	8.33	-
Hole – 34B	150	-	5.71	-
Hole – 34B	150	-	11.3	-
Hole – 34C	150	-	9.98	-
Hole – 48A	150	-	10.5	-
Hole – 48B	150	-	9.60	-
2012 SGS samples				
MET – 1105341	150	88	6.7	0.032
MET – 1106043	150	88	6.5	0.019
MET – 1105868	150	85	7.4	0.030
MET – 1106033	150	87	9.3	0.072
MET – 1105853	150	89	11.1	0.017
2017 ALS Metallurgy samples				
Composite 1	106	106	9.0	-
Composite 2	300	228	8.6	-
Composite 3	300	232	8.1	-
Composite 4	300	226	6.6	-
Composite 5	300	233	7.1	-
Composite 6	300	233	6.1	-

Sample	Mesh of Grind Size	P_{80} (µm)	BWi (kWh/t)	Ai (g)
Composite 7	300	223	6.2	-
Composite 8	300	234	9.0	-
Composite 9	300	236	6.4	-
Composite 10	300	237	5.3	-
Composite 11	300	225	7.2	-
Composite 12	300	234	10.3	-
Composite 13	300	229	10.1	-
Composite 14	300	231	6.4	-
PP Composite 1	300	231	7.2	-

ALS Metallurgy, in conjunction with the JKTech completed DWi testing and SMC tests on 19 individual samples.

SMC testwork was also completed by JKTech, including breakage parameters a x b, BWi, and autogenous grind/SAG mill specific energy (SCSE). The SMC data show the materials to be soft to very soft in terms of SAG milling characteristics. Test results are provided in Table 10-9.

Table 10-9: Summary of SMC Test Results and Additional BMI Data

Sample	a x b	SCSE (kWh/t)	BWi (kWh/t)
SMC - 1	148.9	6.13	10.9
SMC – 2	NA	NA	11.9
SMC – 3	239.9	5.31	7.1
SMC – 4	106.8	6.89	8.5
SMC – 5	203.5	5.56	6.8
SMC – 6	272.7	4.71	5.4
SMC – 7	629.3	3.98	12.5
SMC – 8	301.8	5.09	11.7
SMC – 9	220.5	5.11	8.2
SMC – 10	180.2	5.77	6.4
SMC – 11	143.8	6.25	10.8
SMC – 12	150.7	6.15	11.4
SMC – 13	98.8	7.07	7.6
SMC – 14	115.2	6.68	9.7
SMC – 15	182.5	5.77	9.0
SMC - 16	71.6	8.21	9.9
SMC – 17	86.2	7.27	10.5
SMC – 18	94.2	7.26	10.3
SMC – 19	169.3	5.83	8.4
Average Value	**189.8**	**6.05**	**-**

10.3.5 Flotation Testwork

The predictive metallurgical results for the Arctic Project are based on locked cycle flotation testwork which mirrors the performance of an operational plant and accounts for circulating loads and intermediate products. Each research program consisted of a large number of open circuit flotation tests that provided guidance to the selection of operating conditions to be used in locked cycle tests.

In 2012, SGS conducted bench-scale flotation testwork to investigate the recovery of copper, lead, zinc, and associated precious metals using bulk copper-lead flotation and zinc flotation, followed by copper and lead separation. Four wide-ranging composite samples were tested for rougher flotation kinetics, cleaner efficiency, and copper and lead separation flotation efficiency. All of the testwork used a phased approach to completing testwork, with a preliminary phase of flotation testwork generating a bulk copper-lead concentrate and a final zinc concentrate. The bulk copper-lead concentrate was typically used in a second phase of testing, which was focused on separation of copper and lead minerals into copper and lead concentrates. The second phase of testwork typically also involved open-circuit flotation tests and locked cycle tests.

All flotation testwork either at SGS or ALS Metallurgy had the same froth characteristics.

The froth product is typically light in density and can require extensive flotation time. Very high talc recoveries are required to protect the lead concentrate from contamination as the lead concentrate is the destination for mis-reporting talc. Talc levels in test samples ranged from 0.0 to 40% talc on a weight basis, the LOM average for talc is approximately 5.1%.

The ratio of copper to lead is approximately four in the feed samples tested and within the Mineral Reserve estimates. This copper and lead concentrate is readily upgraded to provide feed to a copper and lead separation circuit. This froth is heavy and flotation rates for copper and lead are considered fast. In all copper and lead flotation, Cytec reagent 3418A was used as a copper and lead collector.

Zinc flotation follows both talc flotation and copper/lead bulk flotation. Zinc is depressed through both stages of talc and copper/lead flotation with the use of zinc sulphate and cyanide. Zinc minerals are activated with the addition of copper sulphate and a xanthate-based collector.

Zinc flotation is relatively fast, and froths are generally heavily laden with mineral.

The separation of copper and lead is completed using a high-grade concentrate of copper and lead minerals. The basis of the copper and lead separation is the depression of copper minerals using cyanide to render the copper minerals hydrophilic.

Locked cycle test results form the basis of metallurgical predictions for the Arctic project and are reported within Table 10-13 for the SGS testwork program and within Table 10-13 for the ALS Metallurgy testwork program.

The SGS testwork produced similar metallurgical performances among the composite samples tested and results were consistent with expectations outlined in the various mineralogical examinations and preliminary testwork results. Further optimization testwork would likely have improved the flotation performance of the composite sample for Zone 1 & 2 (LCT6), sample availability was limited for this composite.

The SGS flotation testwork points to high recoveries of copper and lead to a bulk concentrate of copper and lead which would be subsequently separated. Copper recoveries were in the range of 90 to 92%. Zinc recoveries were typically in the range of 89 to 92%. The majority of misreporting of copper was to the zinc concentrate and the majority of mis-reporting zinc was to the copper-lead concentrate. Ongoing optimization of the flotation process will likely reduce the misreporting of metals, through changes in reagent additions, grind size optimization and concentrate mass recovery.

Flotation testwork conducted in 2017 at ALS Metallurgy, was focused on a detailed evaluation of the performance of a copper and lead separation process and included open circuit flotation tests and locked cycle flotation testing of the copper-lead separation process.

A master composite was made from 14 variability samples from the deposit. Locked cycle testing for this composite was completed to provide a comparison with the SGS test results.

A summary of the locked cycle test results for both the SGS and ALS Metallurgy test programs in contained in Table 10-10.

Table 10-10: Summary of Locked Cycle Recovery Data for Composite Sample Testing

Sample	Recovery to Bulk Cu/Pb Concentrate or Zinc Concentrate				
	Cu[1] (%)	Pb[1] (%)	Zn[2] (%)	Au[1] (g/t)	Ag[1] (g/t)
SGS Burnaby 2012					
Zone 3	92.5	92.6	93.0	77.6	85.9
Zone 5	91.3	92.0	89.3	70.9	84.2
Zone 3&5	91.7	92.3	91.6	75.8	85.0
Zone 1&2	84.2	94.0	85.7	79.7	84.2
ALS Metallurgy 2017					
Master Comp.	94.1	88.7	87.8	74.4	85.4

Notes:
1. Represents recovery to a bulk concentrate prior to Cu/Pb separation.
2. Represents recovery to a final zinc concentrate.

Locked cycle test results for the ALS Metallurgy master composite are contained in Table 10-10 and reports results for the production of a bulk copper-lead concentrate and a zinc concentrate. Table 10-11 provides the results for the SGS testwork program and Table 10-12 includes the results for the ALS Metallurgy testwork program.

Table 10-11: Locked Cycle Metallurgical Test Results – SGS Burnaby 2012

Test No.	Product	Regrind Size 80% Passing	Weight %	Assays						Distribution (%)					
				Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Cu	Pb	Zn	Au	Ag	S
Zone 3 LCT-2	Talc Concentrate	Cu/Pb Rougher Concentrate: 43 µm; Zn Rougher Concentrate: 41 µm	1.6	2.39	2.44	4.05	0.51	105.0	9.97	0.4	0.8	0.3	0.4	0.8	0.2
	Cu/Pb Cleaner 2 Conc.		12.9	24.7	12.4	3.61	4.73	506	30.5	92.5	92.6	5.5	77.6	85.9	15.4
	Zn Cleaner 2 Concentrate		12.9	1.02	0.38	61.4	0.40	41.7	32.9	3.8	2.8	93.0	6.5	7.1	16.5
	Zn Cleaner 1 Sc. Tailings		5.9	0.85	0.33	0.86	0.97	35.0	38.7	1.5	1.1	0.6	7.3	2.7	9.0
	Zn Rougher Tailings		66.7	0.10	0.07	0.09	0.10	4.01	22.5	1.9	2.7	0.7	8.3	3.5	58.9
	Feed		100.0	3.42	1.71	8.43	0.78	75.3	25.4	100.0	100.0	100.0	100.0	100.0	100.0
Zone 5 LCT-3	Talc Concentrate	Cu/Pb Rougher Concentrate: 36 µm; Zn Rougher Concentrate: 35 µm	1.3	7.15	3.71	2.46	1.22	187.0	13.7	1.2	1.2	0.2	0.3	1.4	0.3
	Cu/Pb Cleaner 2 Conc.		9.9	23.8	12.9	5.04	11.2	499	31.5	91.3	92.0	9.1	70.9	84.2	14.7
	Zn Cleaner 2 Concentrate		8.3	0.91	0.56	59.1	0.55	46.4	30.5	2.9	3.4	89.3	2.9	6.6	11.9
	Zn Cleaner 1 Sc. Tailings		7.1	0.80	0.28	0.56	4.55	30.0	32.4	2.2	1.4	0.7	20.5	3.6	10.7
	Zn Rougher Tailings		73.4	0.09	0.04	0.05	0.11	3.38	18.1	2.4	2.0	0.7	5.3	4.2	62.4
	Feed		100.0	2.56	1.37	5.47	1.55	58.2	21.1	100.0	100.0	100.0	100.0	100.0	100.0
Zone 3 & 5 LCT 4	Talc Concentrate	Cu/Pb Rougher Concentrate: 45 µm; Zn Rougher Concentrate: 23 µm	7.3	0.72	0.38	1.37	0.11	17.6	3.01	0.4	0.6	0.4	0.3	0.6	0.3
	Cu/Pb Cleaner 2 Conc.		16.0	25.3	9.25	3.13	4.28	408	29.4	91.7	92.3	6.4	73.8	85.0	21.3
	Zn Cleaner 2 Concentrate		11.8	1.78	0.39	60.9	0.48	50.7	32.5	4.8	2.9	91.6	6.1	7.8	17.4
	Zn Cleaner 1 Sc. Tailings		4.8	1.15	0.38	1.09	2.6	39.8	27.1	1.3	1.1	0.7	13.5	2.5	5.9
	Zn Rougher Tailings		60.2	0.14	0.08	0.13	0.1	5.19	20.2	1.9	3.2	1	6.3	4.1	55.1
	Feed		100.0	4.41	1.6	7.85	0.93	76.6	22.1	100	100	100	100	100	100
Zone 1 & 2 LCT-6	Talc Concentrate	Cu/Pb Rougher Concentrate: 62 µm; Zn Rougher Concentrate: 55 µm	4.8	0.67	0.34	0.90	0.40	13.9	1.88	0.4	0.6	0.4	0.8	0.4	0.3
	Cu/Pb Cleaner 2 Conc.		9.5	23.7	9.54	5.12	6.65	481	30.2	84.2	94.0	14.3	79.7	84.2	32.5
	Zn Cleaner 2 Concentrate		6.4	5.84	0.49	44.5	0.91	101.5	32.8	14.0	3.2	83.7	7.4	12.0	23.9
	Zn Cleaner 1 Sc. Tailings		7.4	0.22	0.06	0.17	0.91	12.3	19.6	0.6	0.5	0.4	8.4	1.7	16.4
	Zn Rougher Tailings		71.8	0.03	0.02	0.06	0.04	1.34	3.30	0.8	1.7	1.2	3.7	1.8	26.8
	Feed		100.0	2.69	0.97	3.42	0.80	54.6	8.8	100.0	100.0	100.0	100.0	100.0	100.0

Table 10-12: Locked Cycle Metallurgical Test Results – ALS Metallurgy 2017

Test No.	Product	Regrind Size 80% Passing	Weight %	Assays						Distribution (%)					
				Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Cu	Pb	Zn	Au	Ag	S
Zone 3 LCT-2	Talc Concentrate	Cu/Pb Rougher Concentrate: 43 µm; Zn Rougher Concentrate: 41 µm	1.6	2.39	2.44	4.05	0.51	105.0	9.97	0.4	0.8	0.3	0.4	0.8	0.2
	Cu/Pb Cleaner 2 Conc.		12.9	24.7	12.4	3.61	4.73	506	30.5	92.5	92.6	5.5	77.6	85.9	15.4
	Zn Cleaner 2 Concentrate		12.9	1.02	0.38	61.4	0.40	41.7	32.9	3.8	2.8	93.0	6.5	7.1	16.5
	Zn Cleaner 1 Sc. Tailings		5.9	0.85	0.33	0.86	0.97	35.0	38.7	1.5	1.1	0.6	7.3	2.7	9.0
	Zn Rougher Tailings		66.7	0.10	0.07	0.09	0.10	4.01	22.5	1.9	2.7	0.7	8.3	3.5	58.9
	Feed		100.0	3.42	1.71	8.43	0.78	75.3	25.4	100.0	100.0	100.0	100.0	100.0	100.0

10.3.6 Copper/Lead Separation Testwork

SGS performed preliminary open-circuit copper and lead separation tests on the bulk copper-lead concentrates produced from the locked cycle tests in open circuit flotation tests. Sodium cyanide was used to suppress copper minerals; 3418A was used as the lead collector and lime was added to adjust the pulp pH to 10. Table 10-13 summarizes the separation test results. These results were obtained using small concentrate samples of approximately 200 g following the production of bulk concentrates in locked cycle testwork. These results also indicated that the separation of copper and lead was feasible using depression of copper and the flotation of lead minerals. Lead concentrate grades as high as 60% lead were obtained in these preliminary open-circuit tests at SGS and rougher flotation recoveries for lead were observed in the range of 88 to 95%.

Additional testwork to provide detailed estimates of the performance of the copper and lead flotation process was conducted at ALS Metallurgy. Approximately 50 kg of bulk copper and lead concentrate was produced in a pilot plant program with the specific objective of completing detailed copper and lead separation testwork. The separation testwork performed very well, however, lower than expected lead concentrate grades were obtained due to talc contamination of the final lead concentrate. This contamination was due to non-optimal talc flotation conditions within the pilot plant.

Results for two locked cycle tests using the pilot plant bulk concentrate are shown in Table 10-14. Recovery of lead to a final concentrate was consistent at 88% of the lead contained in the bulk concentrate. Recovery of copper to a final concentrate was consistent at 97.4% of the copper contained in a bulk concentrate.

Table 10-13: SGS Burnaby Open Circuit Copper and Lead Separation Test Results

Test	Product	Weight %	Assays						Distribution (%)					
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)	Cu	Pb	Zn	Ag	Au	S
Zone 3 & 5 Cu/Pb Separation Feed from LCT-4 (Cycle 2)	Pb 2nd Cleaner Concentrate	8.2	5.99	41.0	2.02	2,330	18.9	13.1	1.9	37.0	6.0	44.7	35.9	3.4
	Pb 1st Cleaner Concentrate	22	6.87	37.5	4.34	1,665	13.6	20.6	5.9	90.8	34.8	85.7	69.5	14.3
	Pb Rougher Concentrate	37.7	16.4	23.0	3.43	1,033	9.17	26.2	24.1	95.5	47.4	91.3	80.3	31.4
	Pb Rougher Tailings (Cu Concentrate)	62.3	31.3	0.65	2.31	59	1.36	34.7	75.9	4.5	52.6	8.7	19.7	68.6
	Cu/Pb 2nd Cleaner Concentrate (Head)	-	25.7	9.07	2.73	4.27	4.31	31.5	-	-	-	-	-	-
Zone 3 Cu/Pb Separation from Open Circuit Test (Test F25)	Pb 2nd Cleaner Concentrate	2.1	2.22	58.8	5.58	1,622	0.3	20.8	1.4	74.9	1.4	44.2	1.0	1.8
	Pb 1st Cleaner Concentrate	2.9	4.51	48.3	6.94	1,369	0.5	24.1	3.8	83.8	2.4	50.9	2.0	2.8
	Pb Rougher Concentrate	4.3	12.4	33.6	6.54	1,026	1.05	26.9	15.3	86.0	3.3	56.3	6.6	4.6
	Pb Rougher Tailings (Cu Concentrate)	8.3	31.5	0.29	4.33	231	5.24	33.3	75.1	1.4	4.2	24.5	63.9	11.0
	Cu/Pb 2nd Cleaner Concentrate (Head)	12.6	25.0	11.6	5.08	502	3.81	31.1	90.4	87.4	7.5	80.8	70.5	15.5
Zone 5 Cu/Pb Separation Feed from LCT-5 (Cycle 2)	Pb 2nd Cleaner Concentrate	6.6	2.42	69.0	2.68	1,230	1.27	15.8	0.6	41.1	3	17.2	1.8	3.3
	Pb 1st Cleaner Concentrate	15.2	3.78	57.6	4.18	993	1.92	20.5	2.3	78.8	11.5	31.9	6.1	9.8
	Pb Rougher Concentrate	25.5	10.3	40.3	4.82	778	6.31	25.1	10.5	92.4	22.1	41.9	33.6	20.1
	Pb Rougher Tailings (Cu Concentrate)	74.5	30.0	1.13	5.79	369	4.26	34.1	89.5	7.58	77.9	58.1	66.4	79.9
	Cu/Pb 2nd Cleaner Concentrate (Head)	-	25.0	11.1	5.54	473	4.78	31.8	100.0	100.0	100.0	100.0	100.0	100.0
Zone 1 & 2 Cu/Pb Separation Feed from LCT-6 (Cycle 2)	Pb 2nd Cleaner Concentrate	7.59	2.4	57.3	5.59	0.54	1,313	15.1	0.76	47.1	8.1	0.7	20.1	3.78
	Pb 1st Cleaner Concentrate	16.4	4.38	45.3	7.96	0.77	1,038	19.9	2.98	80.5	24.9	2.2	34.4	10.8
	Pb Rougher Concentrate	23.6	9.6	34.3	7.19	1.13	849	22.9	9.4	87.7	32.3	4.6	40.4	17.8
	Pb Rougher Tailings (Cu Concentrate)	76.4	28.6	1.49	4.64	7.14	386	32.6	90.6	12.34	67.7	95.4	59.6	82.2
	Cu/Pb 2nd Cleaner Concentrate (Head)	-	24.1	9.23	5.24	5.72	495	30.3	100.0	100.0	100.0	100.0	100.0	100.0

Table 10-14: ALS Metallurgy Locked Cycle Testing of Copper-Lead Separation Process

Test	Product	Weight %	Assays					Distribution (%)				
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Cu	Pb	Zn	Ag	Au
Test 33 Bulk Conc. From Pilot Plant Operation	Pb 2nd Cleaner Concentrate (Pb Conc.)	22.1	2.68	24.2	1.35	960	6.9	2.6	87.3	11.4	70.5	85.1
	Pb 1st Cleaner Tail	11.3	25.8	1.64	2.08	138	0.39	12.9	3.0	9.0	5.2	2.5
	Pb Rougher Tail	66.6	28.8	0.89	3.11	110	0.33	84.5	9.7	79.5	24.3	12.5
	Combined Tailings (Cu Concentrate)	77.9	28.3	1.00	2.96	144	0.34	97.4	12.7	88.6	29.5	14.9
	Bulk Cleaner Concentrate (Feed)	-	22.7	6.13	2.61	301	1.79	100.0	100.0	100.0	100.0	100.0
Test 34 Bulk Conc. From Pilot Plant Operation	Pb 2nd Cleaner Concentrate (Pb Conc.)	21.9	2.75	23.3	1.29	906	7.33	2.6	86.0	10.8	69.5	85.3
	Pb 1st Cleaner Tail	11.2	26.8	1.76	2.02	132	0.44	12.9	3.3	8.7	5.2	2.6
	Pb Rougher Tail	66.9	29.4	0.95	3.14	106	0.34	84.5	10.7	80.5	25.3	12.1
	Combined Tailings (Cu Concentrate)	78.1	29.4	1.06	2.98	111	0.35	97.4	14.0	89.2	30.5	14.7
	Bulk Cleaner Concentrate (Feed)	-	23.3	5.93	2.61	285	1.88	100.0	100.0	100.0	100.0	100.0

In order to further evaluate the issue of talc contamination of the lead concentrate that was observed in some of the ALS Metallurgy testwork, additional lead concentrate production testwork was undertaken using a variety of samples with moderate to very high talc contents. This testwork was completed as open-circuit tests and involved the production of a bulk concentrate and the subsequent separation of the copper and lead minerals from the bulk concentrate. The objective of this testwork was to demonstrate that when talc recovery was optimized or maximized, the lead concentrate grade target could be readily achieved. These test results are summarized in Table 10-15. The open circuit tests included operating the talc flotation process until no visible talc was being recovered in the talc flotation stage. The results of the locked cycle separation testwork where lead concentrate grades were observed to be low are included in the table. Also shown is the value of the lead concentrate carried in the predictive metallurgical balance for the Arctic Project.

Table 10-15: Summary of Lead Concentrate Grades for Various Talc Grades in Feed

Sample	Talc Content	Lead Conc Grade	Copper Conc. Grade
	%	% Pb	% Cu
Pilot Plant Comp.	12.7	23.5	29.4
5567-04(test 4)	18.4	61.1	32.3
5567-14(test 5)	0	61.5	30.2
5567-1(test 6)	8.5	46.9	30.8
5567-5(test 7)	30.0	47.1	32.4
5567-7(test 8)	10.5	48.2	29.2
5567-9(test 9)	17.7	63.0	31.4
5567-14(test 10)	0	56.3	29.1
5567-5(test 11)	31.5	49.6	31.8
5567-9(test 12)	19.0	60.2	31.0
LOM Prediction	**5.1**	**55.0**	**30.3**

The lead concentrate grades observed in the testwork summarized in indicates the ability to manage lead concentrate grades by virtue of the efficiency of the talc removal process. This testwork also indicates that lead concentrate quality can be achieved when talc feed grades are several times higher than the LOM estimated talc grade.

Table 10-16 provides descriptions of the key metallurgical parameters for the operation of the proposed recovery and upgrading of base and precious metals. These have been estimated from numerous tests and will be modulated during the operation as feed grades and metal ratios dictate.

Table 10-16: Summary of Flotation Reagent and Grind Size Objectives

Process Stage	Parameter	Comment
Flotation Feed		
Primary Grind Size	P_{80}	70 µm
Zinc Depressant	$ZnSO_4$	200/100 g/t
Zinc Depressant	NaCN	100 g/t
Talc Flotation		
Frothers	MIBC	10 – 40 g/t
Bulk Cu/Pb Flotation		
Flotation Collector	3418A	25-40 g/t
Frother	MIBC	5 – 10 g/t
Re-grind Size	P_{80}	40 µm
Zinc Flotation		
Zinc Activator	$CuSO_4$	100-200 g/t
Zinc Collector	SIPX	10 - 50 g/t
Re-grind Size	P_{80}	40 µm
Cu/Pb Separation		
Copper Depressant	NaCN	60 - 120 g/t
Lead Collector	3418A	10 g/t

Note: Expected Concentrate Quality.

ICP assays were conducted on the copper and lead concentrates produced from the locked cycle tests at ALS Metallurgy and the zinc concentrate from the locked cycle tests at SGS. The samples are thought to represent the expected concentrate quality. The main impurity elements are shown in Table 10-17.

The results indicated that key penalty elements, as well as precious metals are typically concentrated into a lead concentrate, leaving the copper concentrate of higher-than-expected quality.

Table 10-17: Summary of Lead Concentrate Quality

Lead Conc.	Pb	Zn	Cu	Au	Ag	As	Sb	Bi	F
	%	%	%	g/t	g/t	g/t	g/t	g/t	g/t
Average Value	55.0*	1.8	6.9	37.3*	2805*	2128	1375	3876	3260
Sample Max	-	2.83	-	-	-	4400	5370	5140	5900
Sample Min	-	0.74	-	-	-	125	32	3270	160

* Reported to be consistent with predictive balance.

The issue of fluorine in the lead concentrate can be an issue for smelters. The level of fluorine in lead concentrates produced in laboratory tests, has been shown to be the result of small volumes of talc mineral that escape the talc pre-float circuit and will report with the lead concentrates. Talc levels and ultimately fluorine levels will be managed by optimization of the talc pre-float circuit, talc and fluorine will be effectively removed to ensure the quality of the lead concentrate. It is recommended that a value of 1,500 g/t F be used in marketing evaluations of the lead concentrate. Bismuth may also be an element that will be an issue with the lead concentrates.

Precious metal deportment into the lead concentrate is very high and should benefit the payable levels of precious metals at a smelter.

Table 10-18 provides the key features expected in the copper concentrate.

Table 10-18: Summary of Copper Concentrate Quality

Copper Conc.	Cu	Zn	Pb	Au	Ag	As	Sb	Bi	F
	%	%	%	g/t	g/t	g/t	g/t	g/t	g/t
Average Value	30.3*	1.6	0.70	0.8*	138*	1996	1163	175	246
Sample Max	-	2.98	1.52	-	-	3350	1675	324	330
Sample Min	-	0.87	0.53	-	-	102	264	115	180

* Reported to be consistent with predictive balance.

Copper concentrates are shown to be of high quality with arsenic levels somewhat elevated, but likely below penalty levels. Sulphur levels in copper concentrates are consistent with the mineralization being chalcopyrite at approximately 30–32% sulphur.

Table 10-19 shows the predicted key features of the zinc concentrate.

Table 10-19: Summary of Zinc Concentrate Quality

Zinc Conc.	Zn	Cu	Pb	Au	Ag	Fe	As	Sb	Cd	Bi	F
	%	%	%	g/t	g/t	%	g/t	g/t	g/t	g/t	g/t
Average Val.	59.2*	1.3	0.25	0.53*	24.5*	5.47	966	115	3514	60	100
Sample Max.	-	2.98	0.06	-	-	12.4	7000	570	4000	351	-
Sample Min.	-	0.87	0.22	-	-	1.93	100	16	2900	11	-

* Reported to be consistent with predictive balance.

Zinc concentrates are high grade but have elevated levels of cadmium that may incur economic penalties. Average iron content for the zinc concentrate is considered very good and further optimization of the rejection of pyrite may further reduce these reported iron levels.

10.4 Mineralogical and Metallurgical Testwork – 2021 to 2022

10.4.1 Introduction

In 2021, ALS Metallurgy conducted a test program on several composites constructed in previous Arctic test programs, followed by testing on composites constructed from the drill core samples generated from the 2021 drill program. The objectives of the test program were the characterization of talc in the composites, and determination of the talc distribution to the flotation flowsheet streams. Most of the flotation testing was completed with the preflotation circuit only to establish talc performance.

The generation of final tails for dewatering testing by the Metso Group was completed using a flowsheet which included pre-flotation, bulk, and zinc circuits. Copper-lead separation tests were also completed using the bulk concentrate produced from the tails generation test.

The Metso group prepared simulations for different talc pre-flotation circuit options using the ALS Metallurgy testwork results. The models were used to simulate different scenarios and to select the best circuit configuration.

In 2022, ALS Metallurgy conducted testwork to investigate bulk and sequential flotation flowsheets with composites formed from two parent composites, and then select a flowsheet for a geometallurgical evaluation through testing with variability samples.

The 2022 metallurgical samples were selected using Cancha geometallurgy software, which has unique geostatistical functions to ensure that samples are representative and combined with visualization of drill logs improved sample selection from the Arctic deposit. This software will be utilized along with a review of the material composition and key properties to select representative samples and analyze the metallurgical test results in future testwork programs.

The flowsheet development testwork investigated the effect of various process conditions on copper, lead and zinc recovery using copper-lead bulk flotation and zinc flotation followed by copper and lead separation. The testwork demonstrated that the mineralization was amenable to either a bulk flowsheet followed by copper-lead separation, or a sequential flowsheet, both following a pre-flotation stage to remove talc. In addition, Jameson cell cleaner flotation tests were conducted on a single parent composite to provide data to support the sizing and selection of the Jameson cell technology.

SGS conducted SAG Power Index (SPI®) testing on 15 samples to expand the comminution dataset and investigate the impact of friable ore types on the plant throughput.

Based on economic analysis comparing the bulk and sequential circuit, the bulk circuit flowsheet was selected for the variability testing. Many of the variability samples measured much higher copper, lead, and/or zinc feed grades than the composites tested in the flowsheet development phase. The variability sample selection were not intended to reflect sequences of the mine plan, but instead to represent discrete lithology types from which the results of the flotation testing would be used as inputs for recovery and grade models for the deposit.

Ore characterization and process mineralogical examination was conducted on all the flowsheet development and variability samples. The talc content was considered during sample selection to avoid mis-representing the talc content of future samples.

10.4.2 Test Samples

The pre-flotation flow sheet development test program was carried out on a number of composites constructed in previous Arctic test programs, followed by testing on composites constructed with drill core samples received under this test program.

Table 10-20 summarizes the assay results for the composites constructed in previous Arctic test programs KM5000 and KM5718, as well as the composites formed in this metallurgical test program.

Table 10-20: ALS Head Grades – Pre-Flotation Samples – 2021

Composite	Chemical Content							
	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Mg (%)
Composite 13 (KM5000)	1.98	0.30	1.48	6.20	6.25	38	0.26	9.14
PP Composite 1 (KM5000)	2.92	0.86	4.66	10.8	13.8	41	0.56	5.78
Zone 5 (KM5718)	2.45	1.35	5.59	15.9	22.7	67	1.24	0.61
Zone 3-5 (KM5718)	4.14	1.55	7.80	16.5	24.0	86	0.99	2.58
Master Composite 1B (KM5718)	3.16	1.12	5.52	14.0	16.6	48	0.58	5.22
Constructed under KM6442								
Master Composite 2	3.67	1.30	6.50	12.9	19.1	65	0.84	4.15
HT 1	4.40	0.80	4.79	10.2	14.0	42	0.12	8.54
HS 2	0.91	0.07	0.38	5.0	3.51	5	0.03	3.57
HT 2	0.25	0.04	0.39	1.8	0.67	3	0.01	9.11
FT 1	2.11	0.60	2.72	7.3	9.19	36	0.63	4.28

Initially, two parent composites were constructed for the flowsheet development testing: the Mineralized Composite and Talc Composite. The Mineralized Composite contained about 1.5% talc, while the Talc Composite contained about 55% talc. From these two composites, a range of blends were constructed to generate samples of various talc grades for testing.

Table 10-21 summarizes the flow sheet development blends and their calculated talc content.

Table 10-21: ALS Flowsheet Development Head Composites – 2021

Blend Composite	Proportion of Composite		Talc Content (%)
	Mineralized Composite	Talc Composite	
Very Low Talc	97	3	3.2
Low Talc	93	7	5.1
Average Talc	90	10	6.9
High Talc	87	13	8.7
Very High Talc	83	17	10.5
11% Talc	82	18	11.4
15% Talc	74	26	15.5

As would be expected, head assays measured for each of the blend composites were relative to the proportions of each of the two parent composites. A summary of key head assays for each blend is shown in Table 10-22.

The Talc Composite contained lower metal contents of Cu, Pb, Zn, Ag and Au, as compared to the mineralized (Pure Min) composite. As a result, samples with increased talc content were also of lower head grade, both of which would be expected to lead to poorer metallurgical response.

Some sulphate minerals, as well as organic carbon, were measured in the Pure Min Composite; both were almost negligible in the Talc Composite. The presence of sulphate minerals indicates that a portion of the total sulphur likely would not be recovered via sulphide flotation. Organic carbon, like talc, is usually naturally hydrophobic, so it would be expected to report to the pre-flotation concentrate with the talc.

Table 10-22: ALS Flowsheet Development Composites Head Assays – 2021

Composite	Chemical Content											
	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	S(s) (%)	$S(SO_4)$ (%)	Ag (g/t)	Au (g/t)	C (%)	TOC	Mg (%)
Pure min	2.31	0.84	3.57	8.0	10.9	9.82	1.03	39	0.47	0.59	0.14	3.12
V. Low Talc	2.12	0.78	3.20	7.7	10.2	-	-	34	0.47	-	-	3.61
Low Talc	2.17	0.78	3.21	7.6	10.2	-	-	35	0.42	-	-	4.05
Avg Talc	2.15	0.75	3.09	7.2	9.54	-	-	35	0.42	-	-	4.52
High Talc	2.11	0.76	3.07	7.1	9.44	-	-	36	0.44	-	-	4.91
V. High Talc	1.97	0.67	2.78	6.6	8.49	-	-	32	0.42	-	-	5.30
11% Talc	1.98	0.66	2.73	6.6	8.74	-	-	33	0.32	-	-	5.49
15% Talc	1.81	0.60	2.56	6.1	8.14	-	-	32	0.40	-	-	6.16
Pure Talc	0.93	0.09	0.55	2.0	1.51	1.49	0.03	14	0.25	0.43	0.04	15.7

The drill holes from the most recent 2021 Arctic exploration drilling program assessed to construct geometallurgical samples. The length of the geometallurgical sample was taken as short as possible to ensure that the mineralogy and lithology rock texture was consistent throughput the sample.

The 2022 variability test program used a total of 34 geometallurgical samples constructed from 11 drill holes and are summarized in Table 10-23. The drill cores were stored in a freezer to ensure sample degradation and oxidation of sulphide minerals did not occur.

Table 10-23: ALS Variability Sample Details – 2022

Sample ID	Drill Hole	From	To	Length (m)	Zone	Lithology
ARC-003	AR21-0174	86.9	92.1	5.2	Zone 5	MS
ARC-004	AR21-0174	92.9	95.1	2.2	Zone 5	MS
ARC-005	AR21-0176	128	129.4	1.4	Zone 4	SMS
ARC-006	AR21-0176	129.4	134.4	5	Zone 4	ChTS
ARC-007	AR21-0176	134.4	138.3	3.9	Zone 3	MS
ARC-008	AR21-0176	138.3	147.9	9.6	Zone 3	MS
ARC-010	AR21-0176	165.6	168.7	3.1	Zone 1	QMS
ARC-017	AR21-0177	94.6	99.8	5.2	Zone 5	GS
ARC-024	AR21-0182	126.6	127.7	1.1	Zone 5	MS
ARC-025	AR21-0182	127.7	130.9	3.2	Zone 5	ChTS
ARC-039	AR21-0182	158.6	162.4	3.8	Zone 3	MS
ARC-044	AR21-0182	176.3	178.6	2.3	Zone 3	SMS
ARC-050	AR21-0182	205.1	210.2	5.1	Zone 1	SMS
ARC-051	AR21-0185	123.5	125.4	1.9	Zone 5	MS
ARC-052	AR21-0187	85.2	89.7	4.5	Zone 5	MS
ARC-054	AR21-0181	119.7	121.8	2.1	Zone 5	SMS
ARC-055	AR21-0181	131.7	134.6	2.9	Zone 4	SMS
ARC-056	AR21-0181	134.6	136.9	2.3	Zone 4	MS
ARC-057	AR21-0181	136.9	140.4	3.5	Zone 4	TS
ARC-058	AR21-0181	140.4	144.8	4.4	Zone 4	MS
ARC-059	AR21-0181	147.8	149.4	1.6	Zone 4	SMS
ARC-062	AR21-0181	159.8	163.9	4.1	Zone 3	MS
ARC-063	AR21-0181	168.3	169.7	1.4	Zone 3	SMS
ARC-064	AR21-0181	169.7	173.1	3.4	Zone 3	TS
ARC-068	AR21-0181	177.1	180.4	3.3	Zone 3	SMS

Sample ID	Drill Hole	From	To	Length (m)	Zone	Lithology
ARC-069	AR21-0181	187.1	191.1	4	Zone 2	MS
ARC-072	AR21-0181	203.3	209.1	5.8	Zone 1	MS
ARC-080	AR21-0184	136.3	139.3	3	Zone 5	QMS
ARC-084	AR21-0188	222	224.4	2.4	Zone 2	MS
ARC-086	AR21-0190	141.9	145.1	3.2	Zone 4	SMS
ARC-087	AR21-0190	145.1	148.7	3.6	Zone 4	MS
ARC-092	AR21-0190	186.8	189.1	2.3	Zone 2.5	MS
ARC-098	AR21-0190	217.9	221.2	3.3	Zone 1	CHS
ARC-099	AR21-0189	50.4	61.54	11.14	Zone 5	MS

The head grades of the composites from the 2022 ALS variability testwork program are shown in Table 10-24.

Many of the variability samples measured much higher copper, lead, and/or zinc feed grades than the composites tested in the flowsheet development phase. The Variability sample selection were not intended to reflect sequences of the mine plan, but instead to represent discrete lithology types from which the results of the flotation testing would be used as inputs for recovery and grade models for the deposit.

Table 10-24: ALS Variability Samples Head Assays – 2022

Sample ID	Chemical Content									
	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Sb (g/t)	Barite (%)	Talc (%)
ARC-003	6.4	2.3	10.0	11.8	24.2	81	1.03	309	33.4	0.1
ARC-004	6.1	2.0	8.9	15.1	23.9	61	0.53	85	6.1	0.1
ARC-005	8.1	1.0	4.3	7.2	12.4	127	0.99	315	9.0	11.9
ARC-006	1.1	0.1	0.1	1.9	1.6	18	0.38	149	0.1	71.3
ARC-007	5.9	2.5	12.2	14.4	29.1	140	1.33	450	16.0	2.7
ARC-008	8.9	2.3	9.5	17.4	26.6	120	1.52	445	18.4	0.1
ARC-010	1.5	1.3	8.2	5.5	8.4	22	0.07	13	0.1	12.6
ARC-017	0.6	0.1	0.2	4.6	4.8	4	0.16	44	0.1	0.1
ARC-024	8.6	2.5	13.8	14.7	28.0	133	1.21	57	9.4	2.6
ARC-025	0.8	0.2	0.3	2.2	1.5	11	0.24	114	0.1	54.0
ARC-039	4.6	3.7	14.1	19.4	27.6	110	1.48	648	0.1	0.1
ARC-044	1.8	0.1	0.1	4.9	5.2	18	0.20	82	0.1	0.2
ARC-050	3.1	0.2	1.0	9.5	8.8	36	0.38	54	0.1	0.5
ARC-051	3.7	2.3	13.2	15.8	30.6	70	0.52	31	23.8	0.5

Sample ID	Chemical Content									
	Cu (%)	Pb (%)	Zn (%)	Fe (%)	S (%)	Ag (g/t)	Au (g/t)	Sb (g/t)	Barite (%)	Talc (%)
ARC-052	8.5	2.2	13.0	16.7	28.1	120	1.11	227	7.6	0.8
ARC-054	4.7	1.3	5.9	14.4	23.0	89	0.34	129	21.2	1.1
ARC-055	2.8	0.8	6.3	10.3	15.2	67	0.43	23	0.1	21.4
ARC-056	7.4	0.8	7.2	18.5	25.5	109	0.85	198	0.1	7.9
ARC-057	1.0	0.1	0.2	2.9	1.8	7	0.07	4	0.1	37.2
ARC-058	6.0	2.4	13.6	19.6	31.0	110	1.31	412	7.9	0.5
ARC-059	5.3	0.6	6.6	9.3	13.2	70	0.53	14	0.1	12.6
ARC-062	2.2	6.5	17.7	10.2	20.6	55	0.28	95	0.4	1.1
ARC-063	4.0	0.1	2.1	7.3	8.3	13	0.13	4	0.2	38.8
ARC-064	0.7	0.0	0.4	1.7	1.3	2	0.02	1	0.1	67.1
ARC-068	2.3	0.1	1.2	7.1	6.4	22	0.16	140	0.1	0.3
ARC-069	2.9	1.3	9.2	12.6	23.2	55	0.12	40	19.3	7.6
ARC-072	4.8	0.5	9.0	18.2	24.8	114	1.14	102	0.1	1.8
ARC-080	1.2	0.1	0.2	2.6	2.7	22	0.34	82	0.1	3.7
ARC-084	2.4	0.4	5.3	16.1	24.3	29	0.29	9	0.1	1.6
ARC-086	3.8	1.0	7.0	11.1	21.4	89	1.60	937	51.3	0.3
ARC-087	10.3	4.2	13.7	12.5	24.3	171	3.36	578	6.0	0.8
ARC-092	3.9	0.1	5.1	20.2	31.0	22	0.31	44	0.1	3.2
ARC-098	1.0	0.0	0.1	2.9	1.5	25	0.06	2	0.1	0.7
ARC-099	5.8	2.3	11.4	12.6	22.3	92	0.33	57	11.2	0.9

Head characterization was completed by ALS Geochemistry North Vancouver on all 34 variability samples including whole rock analysis by fusion and XRF, and multi-element determinations by ICP, ion chromatography, and Bulk Mineral Analysis with Liberation estimation (BMAL).

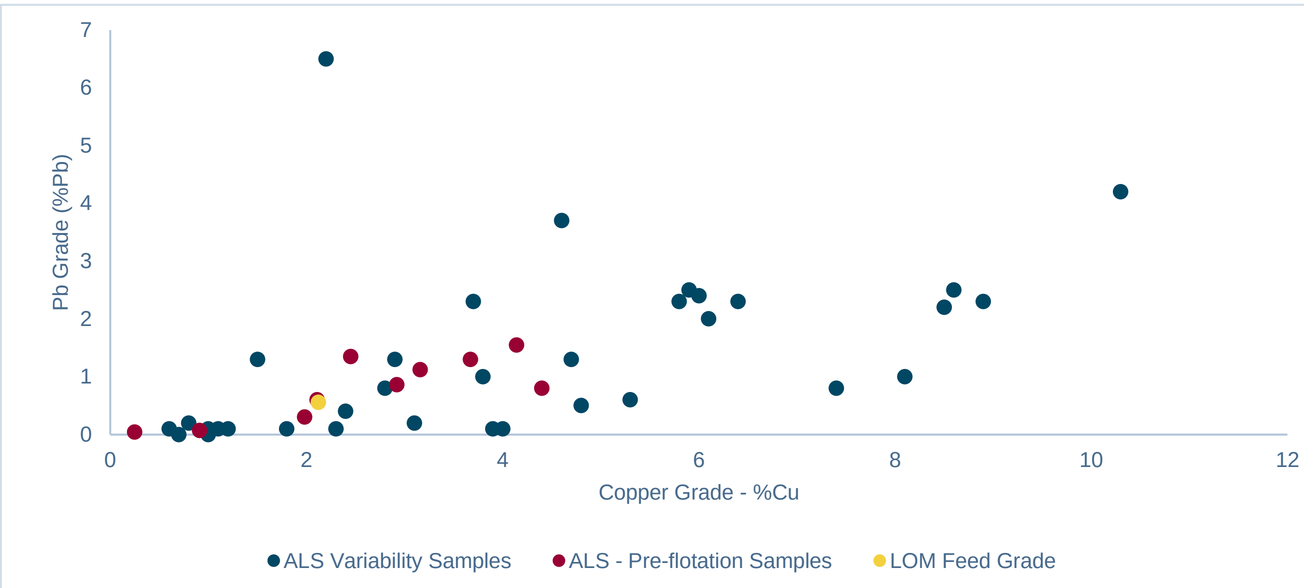
A total 29 samples were submitted for open circuit flotation tests and nine samples were used in locked cycle flotation tests. Eight samples produced sufficient bulk concentrate mass to undertake copper-lead separation testing.

Comminution testing was completed only on selected variability samples to extend the historical testwork dataset which included Bond ball mill work index tests and SPI tests. A Bond rod mill work index and Bond Abrasion test was completed with only three of the variability samples.

Shown in Figure 10 4 are the copper and lead grades of the ALS Metallurgy testwork samples compared to the designed plant feed grades. The pre-flotation samples are more representative of the expected mine production due to the lower range of copper and lead grades that are available. There is a strong correlation between copper and zinc grades within the pre-flotation samples. The metallurgical balance shown in Table 10-1 is based on LOM feed grades and is consistent with the LOM data point shown in Figure 10 4.

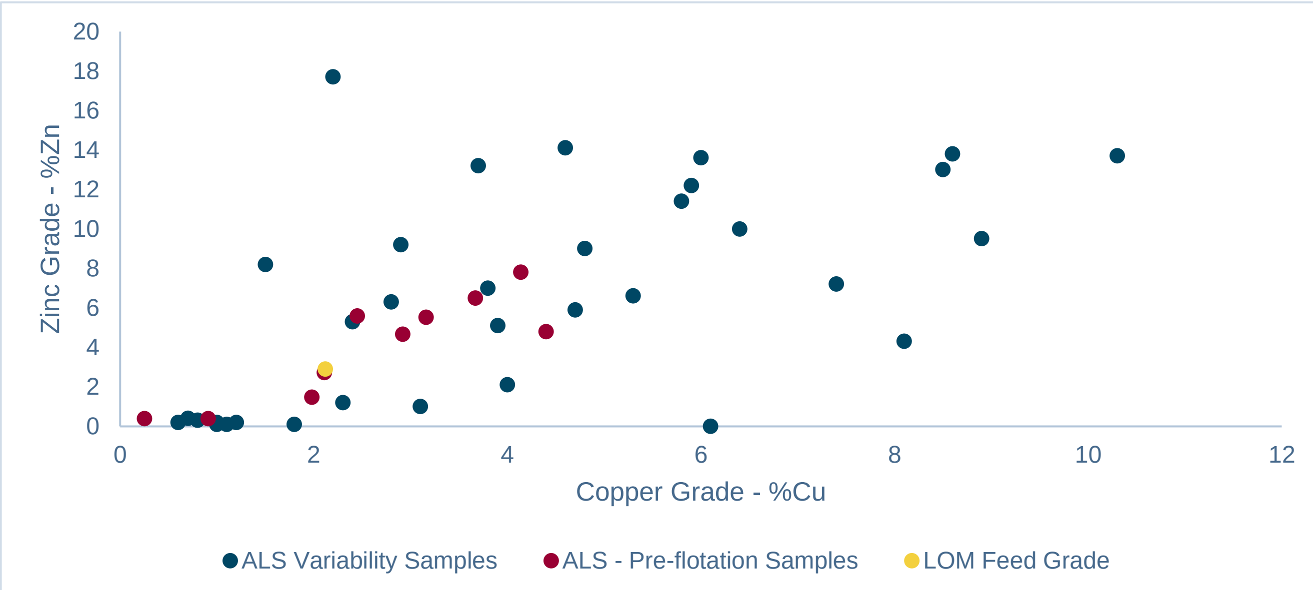
Shown in Figure 10 5 are the copper and zinc grades of the various test samples used in the ALS Metallurgy test program. There is a consistent copper to zinc ratio of approximately 3.5–4.5 within the test samples and the LOM grades are shown to be within the distribution of test samples.

Figure 10-4: Cu and Pb Test Sample Grades for ALS Metallurgy Test Program



Source: Ausenco, 2022

Figure 10-5: Cu and Zn Test Sample Grades for ALS Metallurgy Test Program



Source: Ausenco, 2022

10.4.3 Mineralogical Investigations

10.4.3.1 Talc Optimization Samples

The mineral content and elemental deportment of the composites tested in the metallurgical test program were measured by QEMSCAN using Bulk Mineral Analysis (BMA) protocols as well as by semi-quantitative X-Ray Diffraction (XRD). A summary of the mineral content mineral content is presented in Figure 10 6.

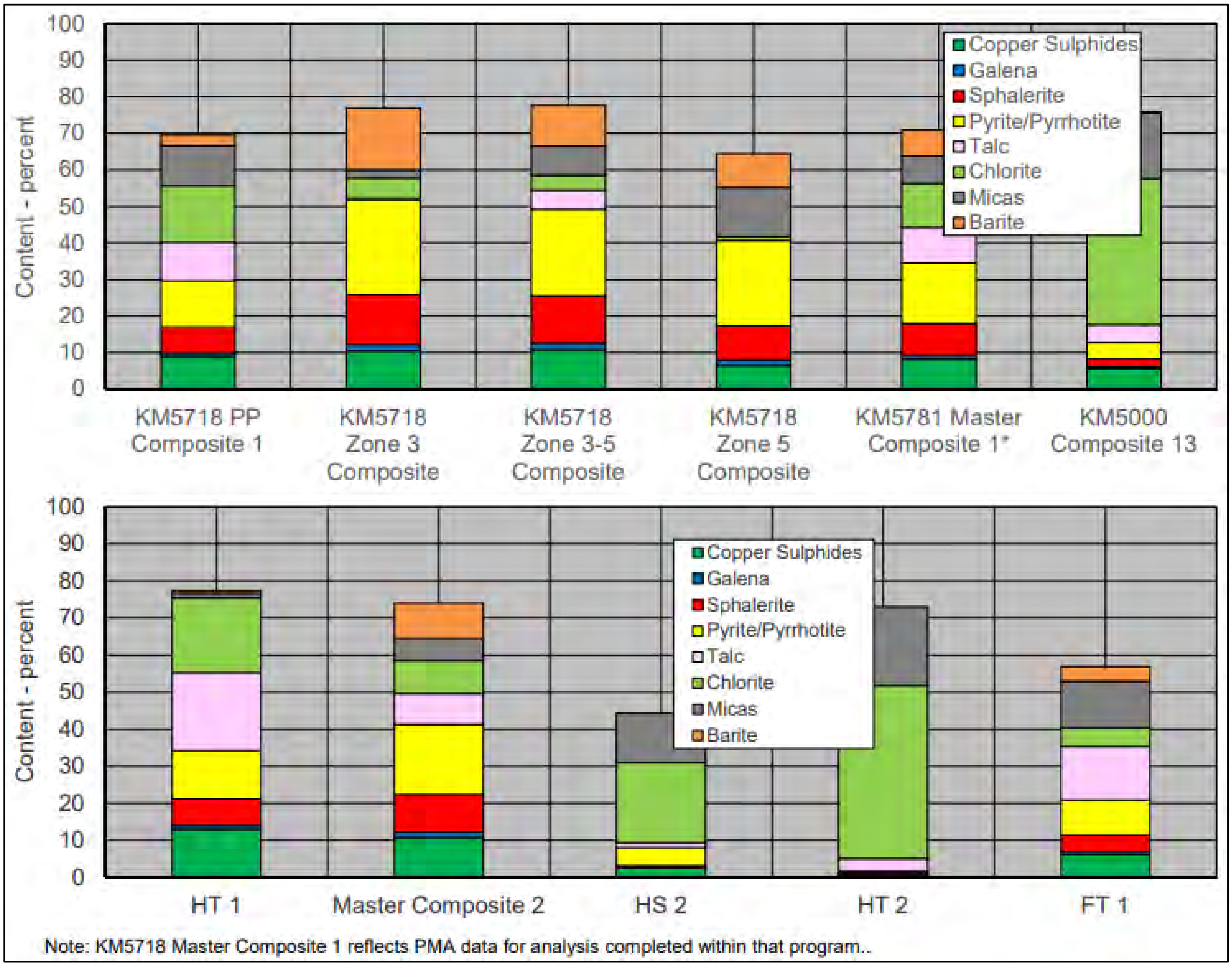
The talc content varied widely between the composites. Highest talc content at about 21% was measured for the HT1 (High Talc) sample. A second sample HT2, received with the intention of blending as a high talc sample, measured only about 3% as talc and 47% as chlorite by QEMSCAN and was not utilized for testing in this study. The Master Composite 2, which had been prepared using the KM5718 Master Composite 1B and Zone 3-5 Composite, measured about 8% talc, and the FT1 composite which was utilized to generate a talc pre-flotation concentrates and zinc circuit tails for dewatering testwork measured about 14% talc.

Magnesium was distributed amongst a number of minerals for the Arctic deposit which included primarily talc, micas, chlorite, and calcium carbonate minerals dolomite and ankerite. The distribution of magnesium to these minerals varied widely between the composites, so magnesium assays on flotation test products would not accurately reflect talc recoveries.

To improve the estimate of talc in the deposit a talc predictive algorithm has been developed by South32. The talc algorithm includes aluminium, calcium, potassium, and magnesium to improve the predictability of talc. The predicted talc assay form each of the individual variability composites were compared with the mineralogical analysis of the head sample. The predictive talc algorithm aligned well with the measured talc content. Understanding that the blending of composites reduces the predictability of the talc in the plant feed, the Arctic Project has considered incorporating the predicted talc and metallurgical performance on each of the mine blocks and then weighing the overall metallurgical performance. Future testwork will investigate blends to validate this approach.

Copper was contained primarily as chalcopyrite, although low percentages of bornite, tennantite/enargite, covellite, chalcocite, and bornite were also measured.

Figure 10-6: Talc Optimization Composites Mineral Content

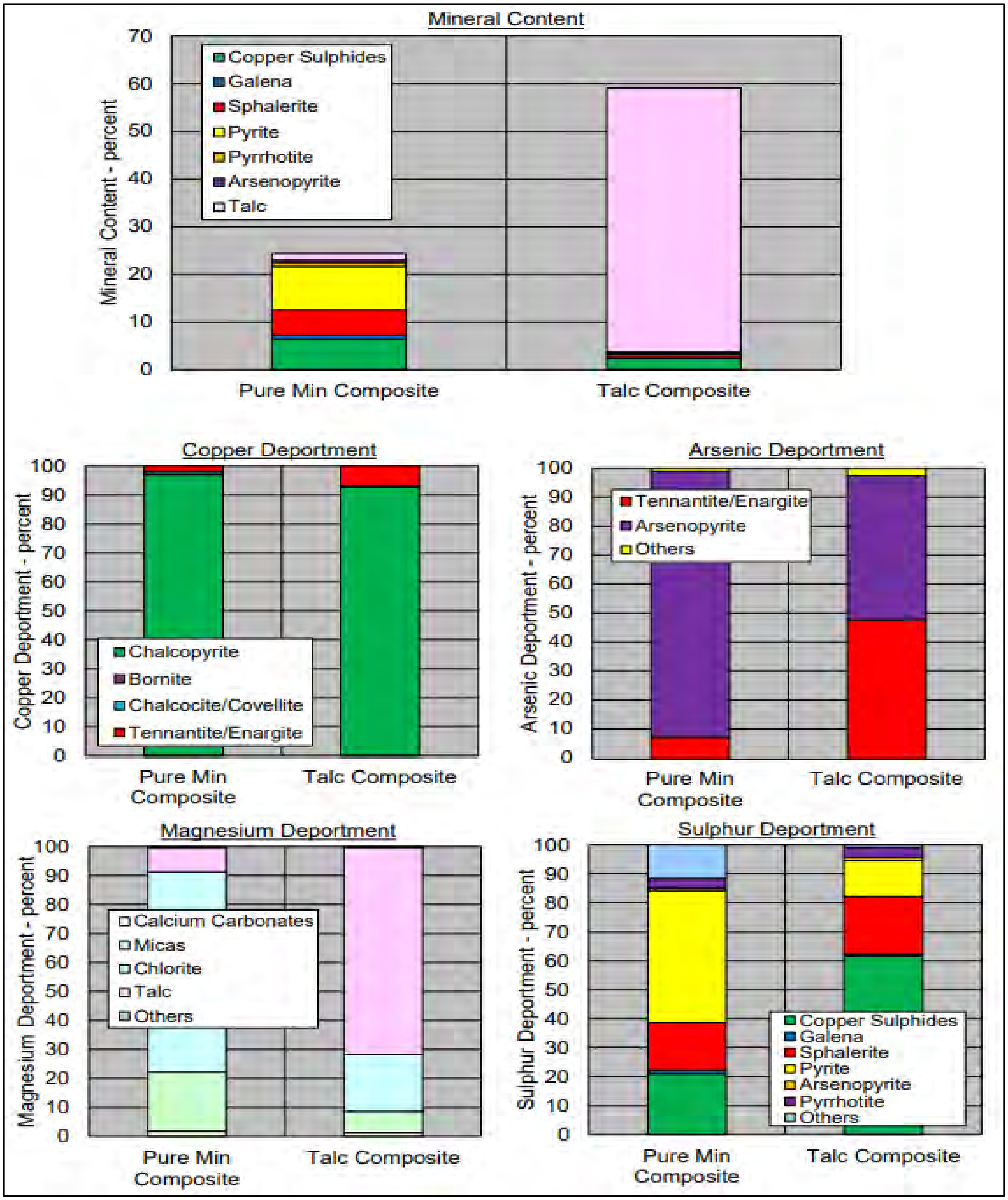


Source: ALS Metallurgy, 2021.

10.4.3.2 Flow Sheet Development Samples

Particle Mineral Analysis (PMA) and XRD were conducted on each the Pure Min and Talc Composites. The semi-qualitative XRD data confirmed non-sulphide gangue mineral contents reported in the Particle Mineral Analyses. A summary of pertinent mineral content data is shown in Figure 10 7.

Figure 10-7: Parent Composite Mineral Content



Source: ALS Metallurgy, 2021

As expected, based on the head assays, the sulphide mineral content for the Pure Min Composite was significantly higher than the Talc Composite. Conversely, the Talc Composite contained very high talc content, about 55%, while the Pure Min Composite only measured about 1.5% talc.

Copper was measured mostly in the form of chalcopyrite. However, between around 2 and 8% of the copper was in copper arsenic sulphosalts (enargite/tennantite /tetrahedrite group minerals). These copper arsenic sulphosalts are typically recovered by flotation in a similar manner to other copper sulphide minerals. On this basis, elevated arsenic and/or antimony levels in the copper concentrate would be expected.

Only a small portion of the magnesium in the Pure Min Composite was present in talc and most of the magnesium was measured as chlorite. In the Talc Composite, just over 70% of the magnesium was measured within talc, with chlorite being the main other magnesium bearing mineral. Mica minerals also contributed a lesser, but still significant component of the magnesium in both composites (7 to 21%). Given the presence of magnesium in minerals other than talc, magnesium is likely not a useful proxy for quantifying talc content across the deposit.

Sulphur was measured mostly as sulphide minerals in both samples. However, in the Pure Min Composite, around 11% was measured as sulphate minerals (i.e., barite, calcium sulphate, and jarosite).

Spectra collected on the sphalerite in the Pure Min and High Talc parent composites measured between 4 and 5 percent iron within the mineral matrix. The iron content within the zinc concentrates produced from the blend composites would be expected to therefore be at least 4%.

10.4.3.3 Variability Samples

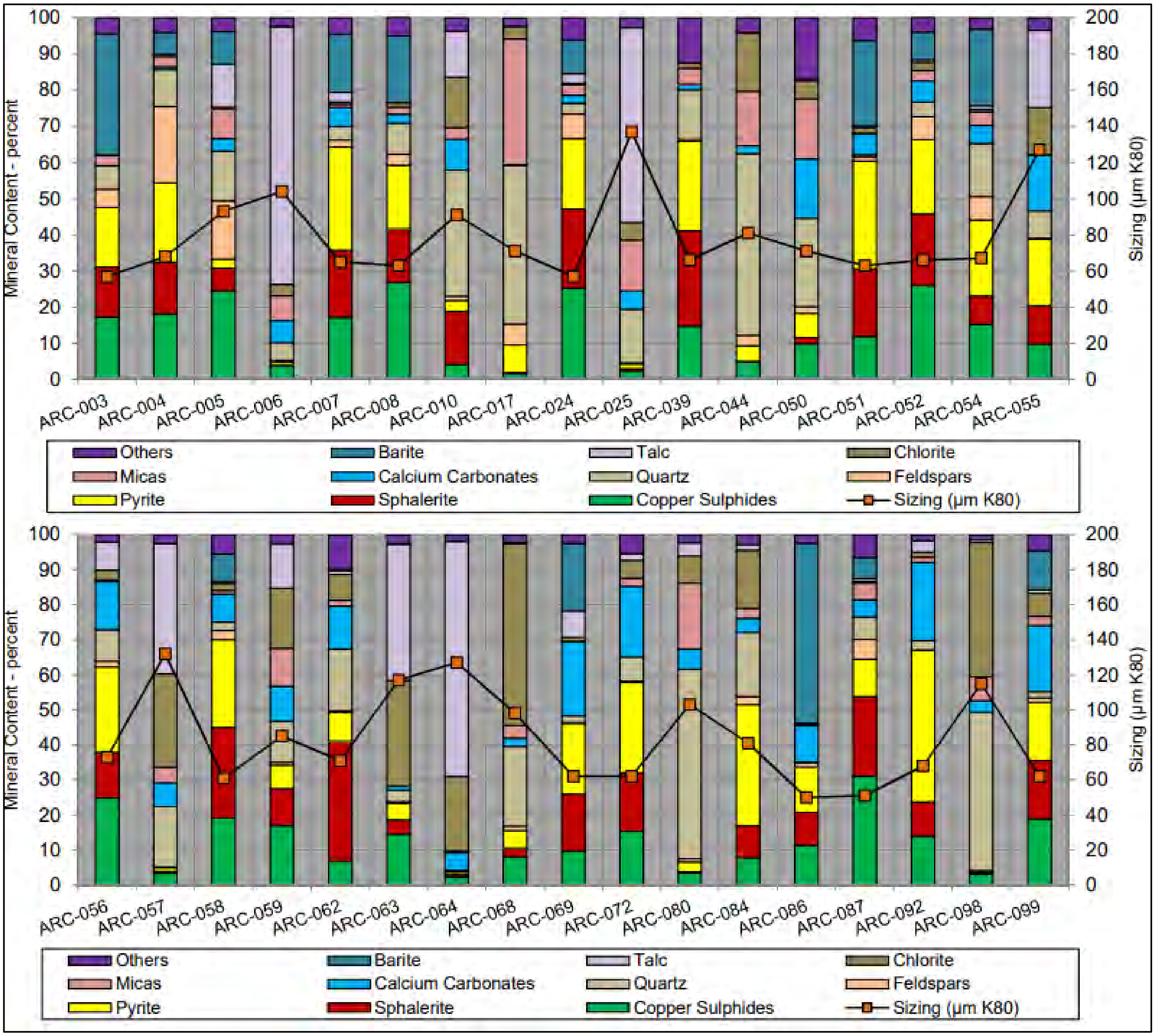
A Bulk Mineral Analysis with Liberation estimation (BMAL) by QEMSCAN was conducted at ALS Metallurgy on each of the variability samples to provide mineral composition and estimates of liberation for key sulphide minerals. A summary of the mineral content data is presented in Figure 10 8.

Talc content varied widely between the samples, ranging from less than detection limits to as high as about 71% for ARC-006. The average talc content for the variability samples was about 12% which is four times the LOM talc content of 3%. MgO contents, measured by XRF or whole rock analysis, could be used to make a preliminary estimate of talc content, despite magnesium being contained in minerals other than talc. Samples high in MgO but low in talc could be identified based on higher Al_2O_3 content greater than 10%. Since talc does not contain aluminium, higher Al_2O_3 content indicates the presence of other potentially magnesium bearing minerals in important concentrations. X-ray spectra collected from talc from the Pure Min and Talc composites did not identify fluorine within the mineral structure. However, based on a trend between fluorine and talc content, it is possible that there is fluorine within talc at low concentrations.

Copper was measured predominantly in the form of chalcopyrite. Copper as copper arsenic sulphosalts enargite/tennantite, and tetrahedrite averaged about 3% and measured as high as 14% for ARC-086. Elevated arsenic and/or antimony levels in the copper concentrate would be expected for feeds with higher percentages of copper in these forms.

Magnesium in the variability samples was measured primarily within talc, chlorite, and micas. For ARC-050, about one half of the magnesium was associated with amphibole/pyroxenes and magnesite/siderite. A summary of the magnesium deportment is shown in Figure 10-8. Sulphur was measured mostly as sulphide minerals chalcopyrite, sphalerite, pyrite, and galena. About 51% of the feed mass was measured as barite for ARC-086, and about 33% as barite for ARC-003. Barite would be unlikely to be recovered via sulphide flotation techniques.

Figure 10-8: Mineral Modal Abundance for Variability Samples



Source: ALS Metallurgy, 2022

Liberations were estimated by QEMSCAN BMAL protocols using sub-samples of grind calibrations for each of the variability samples. The analysis was completed over a wide range of primary grind sizes between 50 and 137µm K80 due to the manner in which the grind calibrations were completed. An initial grind calibration targeting 65µm K80 was completed at 18 minutes for variability samples which measured lower talc content, and at 23 minutes for the samples which measured high talc content. A second grind calibration at 30 or 35 minutes was included for a few samples where sizing's measured greater than 100µm K80.

Copper sulphide mineral liberations, estimated from the QEMCSAN BMAL, averaged about 68% for the variability samples which would be considered moderate to adequate for expected copper recovery in the bulk roughers. About 8% of the copper sulphide minerals on average were estimated to be in binary form with either sphalerite or pyrite, with about 10% of the copper sulphides in binary form with non-sulphide gangue.

An average galena liberation off 55% was estimated, but an additional 11% of the galena was estimated to be in binary form with copper sulphides which would also be recoverable in the bulk roughers, and as such the galena liberation would also be considered acceptable for good lead recovery in the bulk roughers. About 11% of the galena was estimated to be in binary form with sphalerite.

Average sphalerite liberation was estimated to be 72%. This would be expected to lead to high zinc recovery in the zinc roughers. About 9% of the sphalerite, on average, was estimated to be in binary form with copper sulphide minerals which could be potentially recovered in the bulk roughers, but a percentage of this sphalerite could be returned to the zinc roughers after bulk circuit regrinding.

10.4.4 Comminution Testwork

Comminution testing included Bond ball mill work index tests and SPI tests on several samples; a Bond rod mill work index and Bond Abrasion test was completed with only three of the variability samples.

ALS Metallurgy completed Bond ball mill work index tests (BWi) on one set of variability samples with a 150-mesh Tyler closing screen (106µm), and for a second set of samples with a 48-mesh Tyler screen (300µm). Several the samples selected for the Bond work index tests completed with the coarser closing screen measured high talc content (ARC-006, ARC-025, ARC-057, and ARC-064). All variability samples selected for Bond ball mill work index testing with the 150-mesh Tyler screen measured low talc content of 1% or less, except for the ARC-069 composite which measured about 8% talc.

The highest Bond ball mill work index recorded for those samples tested was about 21 kWh/t, at the coarser screen sizing, for ARC-006 which measured about 71% talc. The high Bond ball mill work index is likely a result of the resistance to breakage of the talc particles which has been observed in previous test programs with Arctic composites. A Bond ball mill work index of less than 12 kWh/t was measured for most of the variability samples tested which would indicate material that was soft in terms of ball milling.

Abrasion test results for the three variability samples submitted for testing ranged between 0.01 and 0.05, which would be considered mildly abrasive.

Table 10-25 summarizes the bond work comminution test results.

Table 10-25: Bond Ball Mill Work Index and Abrasion Index Test Results

Sample ID	Bond Ball Mill Work Index (BBMWi)		Bond Rod Mill Work Index (BBRWi)	Bond Abrasion
	@ 300µm kWh/t	@ 106µm kWh/t	@ 1180µm kWh/t	
ARC-006	20.8	-	-	-
ARC-008	-	8.2	-	-
ARC-017	9.8	-	11.3	0.049
ARC-025	11.4	-	-	-
ARC-039	-	8.6	-	-

Sample ID	Bond Ball Mill Work Index (BBMWi)		Bond Rod Mill Work Index (BBRWi)	Bond Abrasion
	@ 300µm kWh/t	@ 106µm kWh/t	@ 1180µm kWh/t	
ARC-044	-	17.7	-	-
ARC-050	-	11.8	-	-
ARC-052	-	9.0	-	-
ARC-057	10.3	-	-	-
ARC-058	-	8.1	-	-
ARC-062	-	13.1	-	-
ARC-064	14.7	-	-	-
ARC-069	-	8.2	-	-
ARC-072	-	9.1	6.4	0.034
ARC-099	5.9	-	4.8	0.014

ALS Metallurgy, in conjunction with SGS Lakefield (SGS) conducted SPI tests on 15 selected variability samples during the testwork program in 2022. The SPI test is considered more reliable when estimating specific energy of soft ores.

The SPI data show most materials to be very soft in terms of SAG milling characteristics. Test results are provided in Table 10-26.

Table 10-26: Summary of SPI Test Results

Sample	SPI (min)	Hardness Percentile	Category
ARC-017	35.2	15	Soft
ARC-025	12.8	3	Very Soft
ARC-044	25.3	8	Very Soft
ARC-050	19.5	5	Very Soft
ARC-056	14.0	3	Very Soft
ARC-057	13.8	3	Very Soft
ARC-058	10.4	2	Very Soft
ARC-062	12.5	3	Very Soft
ARC-063	13.5	3	Very Soft
ARC-064	5.5	1	Very Soft
ARC-069	8.2	1	Very Soft
ARC-072	12.2	3	Very Soft
ARC-086	5.9	1	Very Soft
ARC-087	10.7	2	Very Soft
ARC-099	9.8	2	Very Soft
Minimum Value	5.5	-	
Maximum Value	35.2		

Sample	SPI (min)	Hardness Percentile	Category
Average Value	**14.0**		

10.4.5 Flotation Testwork

10.4.5.1 Talc Optimization Testwork

Initial testing was completed on composites which had been retained in storage from previous Arctic test programs KM5000 and KM5718 with a flowsheet including pre-flotation rougher and cleaner stages. Table 10-27 summaries the baseline pre-flotation test results.

The results of these tests indicated similar pre-flotation circuit performance compared to results from the previous metallurgical test programs. One substantial difference noted was the extremely negative redox potentials measured in the pre-flotation rougher stage recorded for the tests in this program, although this did not appear to affect pre-flotation rougher performance in terms of mass recovery and metals losses to the pre-flotation concentrate.

Table 10-27: Summary of Baseline Pre-Flotation Tests

Test	Original Tests Distribution (%)					Current Tests Distribution (%)				
	Cu	Pb	Zn	Au	Ag	Cu	Pb	Zn	Au	Ag
Composite 13 (KM5000)	3.3	3.7	3.0	4.3	4.0	2.7	4.4	2.9	1.3	3.4
PP Composite 1 (KM5000)	1.0	1.1	0.3	2.4	0.3	1.3	5.1	1.2	1.3	3.3
Zone 5 (KM5718)	0.4	0.7	0.2	0.3	0.6	0.4	0.7	0.2	0.3	0.6
Zone 3-5 (KM5718)	0.3	0.9	0.3	1.4	0.8	0.4	0.8	0.4	0.6	0.8
Composite 1B (KM5718)	1.4	4.8	1.3	1.0	3.7	1.6	3.4	1.6	3.2	2.4

Notes: Values reflect recoveries to pre-flotation rougher for Composite 13, Zone 5, and PP Comp 1, and to pre-flotation cleaner concentrate for Master Composite 1B and Zone 3-5 composite

The Master Composite 2 was subjected to a series of eight pre-flotation was conducted which included two stages of cleaners was carried out using a variable screening test matrix. In these tests, whole rock and QEMSCAN mineralogical analyses were completed on the test products. High talc recoveries between 95 and 98% were recorded in all tests, but lower metal losses were recorded at a coarser primary grind sizing.

High grade talc concentrates ranging between 88 and 92% talc were measured for all variable screening tests after two cleaner stages. The percentage of the feed talc recovered to the pre-flotation cleaner concentrate varied widely between 61 and 92% and correlated with mass recovery. A total MIBC frother dosage between 55 and 77 g/ton appeared to be required to ensure a stable flotation froth and therefore high mass and talc recoveries. Copper, lead, zinc, and silver recoveries to the pre-flotation cleaner concentrate ranged between 0.5 and 1.0% indicating good rejection from the rougher concentrate by dilution cleaning with potential for recovery in the downstream sulphide flotation circuits Kinetic pre-flotation rougher and cleaner tests were completed with Master Composite 2 to provide flotation kinetic data for modelling by an external laboratory. A high-grade copper concentrate which measured between 22 and 24% copper was recovered to the final pre-flotation rougher concentrate following 8 minutes of initial rougher residence time; these tests were characterized by highly negative redox values in the initial pre-flotation rougher stages which increased to positive values by the final pre-flotation rougher stage.

Kinetic pre-flotation rougher and cleaner tests were completed with Master Composite 2 to provide flotation kinetic data for modelling by Metso Group. A high-grade copper concentrate which measured between 22 and 24% copper was recovered to the final pre-flotation rougher concentrate following 8 minutes of initial rougher residence time; these tests were characterized by highly negative redox values in the initial pre-flotation rougher stages which increased to positive values by the final pre-flotation rougher stage.

A 64-kilogram test was carried out in locked-cycle fashion using a bulk flotation flowsheet to generate pre-flotation concentrate and bulk final tails for dewatering and tailing consolidation tests by external laboratories. The bulk concentrate produced from the test was used for copper-lead separation tests using sodium cyanide to depress the copper sulphide minerals.

Test results indicated that on average, 93% of the copper was recovered to the lead rougher tails, and about 86% of the lead was recovered to the lead rougher concentrate. Lead losses were very high to the lead cleaner tails, and the flowsheet was more effective recovering talc to the lead cleaner concentrate. As a reverse flotation flowsheet with combined 3rd and 4th lead cleaner tails, about 56% of the lead and 30% of the silver was recovered to a lead concentrate with a calculated grade of about 58% lead and 0.2% silver.

10.4.5.2 Flowsheet Development Testwork

The objective of the program was to investigate bulk and sequential flotation flowsheets with composites formed from two parent composites, and then select a flowsheet for a geometallurgical evaluation through testing with variability samples.

The mineralization was amenable to either a bulk flowsheet followed by copper-lead separation, or a sequential flowsheet, both following a pre-flotation stage to remove talc. Table 10-28 shows average performance obtained for the Avg Talc Composite in the Flowsheet Development phase of the testing.

Table 10-28: Comparison of Bulk vs. Sequential Locked-Cycle Test Results – ALS 2021

Composite	Assays						Distribution (%)				
	Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	Mg (%)	Cu	Pb	Zn	Ag	Au
Avg Talc Bulk											
Copper concentrate	28.0	0.86	4.27	181	4.17	0.46	87.3	8.3	9.1	36.0	60.9
Lead concentrate	7.90	39.0	6.30	1124	4.75	1.23	5.1	78.1	2.8	46.0	14.3
Zinc concentrate	0.87	0.38	55.9	41	0.35	0.04	1.9	2.6	83.3	5.7	3.5
Avg Talc - Sequential											
Copper concentrate	27.6	0.87	2.05	168	3.23	1.96	90.2	8.9	4.7	34.9	48.7
Lead concentrate	2.72	49.3	9.71	1360	5.31	1.40	1.2	69.9	3.1	39.4	11.2
Zinc concentrate	0.98	1.09	54.5	47	0.77	0.17	2.1	7.3	83.5	6.5	7.7

Copper recovery to the copper concentrate was slightly higher for the sequential flowsheet; however, gold recovery to the copper concentrate was substantially lower. The lead concentrate grade for the Avg Talc composite could likely be improved over that shown in Table 10-28 with optimization of copper-lead separation conditions given the higher lead concentrate grade measured with other composites.

Zinc circuit performance was similar for the two flowsheets, although higher zinc recovery to the copper concentrate was recorded for the bulk circuit. Magnesium content in the copper concentrate was higher for the sequential circuit, but similar in the lead concentrate for both circuits.

10.4.5.3 Variability Testwork

Based on economic analysis comparing the bulk and sequential circuit, the bulk circuit flowsheet was selected for the Variability testing. A total of 28 variability samples were selected for flotation testing which included open circuit cleaner tests, and 9 of these variability samples were selected for locked-cycle testing. Table 10-29 shows performance obtained for Variability testwork phase of the testing.

Most of the variability samples responded well to the developed bulk circuit flowsheet, when adjustments to dosages were made to account for variations in head grades. In the cleaner tests with the bulk flowsheet, an average of 88% of the copper and 76% of the lead was recovered to the bulk concentrate. Average zinc recovery was about 78% to a zinc concentrate which averaged 57% zinc. High zinc rougher recoveries were recorded when the copper sulphate addition in the conditioner stage was ratioed to the zinc feed grade at 100 g/t to 1% feed grade. Further optimization of copper sulphate additions might be possible with additional testing.

The variability test phase culminated with locked-cycle testing of 9 variability samples to determine closed-circuit performance. On average, about 93% of the copper and 84% of the lead was recovered to the bulk concentrate which measured 25% copper and 10% lead, on average. Good zinc performance was recorded in these tests with an average zinc recovery of 90% to a zinc concentrate which averaged 55% zinc. Most of the gold and silver was recovered to the bulk concentrate, averaging 74% and 84%, respectively.

Table 10-29: Summary of Variability Open Circuit Flotation Test Results – ALS 2022

Test	Product	Regrind Size 80% Passing	Weight (%)	Assays						Distribution (%)					
				Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Cu	Pb	Zn	Au	Ag	S
ARC-004 Test 64	Talc Concentrate	Flotation Feed: 68 µm; Cu/Pb Rougher Concentrate: 38 µm; Zn Rougher Concentrate: 20 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		20.3	26.0	8.5	7.6	1.85	224	31.9	89.0	79.7	18.3	80.6	75.9	27.0
	Zn Cleaner 2 Concentrate		2.2	1.7	1.0	58.0	0.27	40	32.6	0.6	1.0	15.4	1.3	1.5	3.0
	Zn Cleaner 1 Sc. Tailings		0.9	5.0	2.4	14.6	0.51	106	20.4	0.7	1.0	1.5	1.0	1.5	0.7
	Zn Rougher Tailings		72.4	0.3	0.3	6.1	0.06	8	21.1	3.5	10.4	52.3	9.3	9.5	63.7
	Feed		100.0	5.9	2.2	8.4	0.47	60	24.0	100.0	100.0	100.0	100.0	100.0	100.0
ARC-007 Test 71	Talc Concentrate	Flotation Feed: 64 µm; Cu/Pb Rougher Concentrate: 47 µm; Zn Rougher Concentrate: 55 µm	4.6	0.3	0.4	0.5	0.07	16	1.2	0.3	0.6	0.2	0.2	0.5	0.2
	Cu/Pb Cleaner 2 Conc.		19.0	25.9	12.6	3.2	5.63	506	30.9	85.9	89.1	4.8	76.8	69.3	20.0
	Zn Cleaner 2 Concentrate		17.2	0.7	0.2	63.8	0.30	38	33.6	2.0	1.3	85.5	3.7	4.7	19.6
	Zn Cleaner 1 Sc. Tailings		14.6	1.4	0.3	1.4	0.66	91	48.9	3.4	1.7	1.6	6.9	9.6	24.2
	Zn Rougher Tailings		40.0	0.4	0.2	0.5	0.21	12	22.4	2.5	3.4	1.4	6.0	3.5	30.4
	Feed		100.0	5.7	2.7	12.8	1.39	139	29.5	100.0	100.0	100.0	100.0	100.0	100.0
ARC-008 Test 79	Talc Concentrate	Flotation Feed: 64 µm; Cu/Pb Rougher Concentrate: 41 µm; Zn Rougher Concentrate: 35 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		28.4	29.4	7.4	2.1	4.42	356	32.4	91.3	90.6	6.0	72.7	81.7	34.3
	Zn Cleaner 2 Concentrate		12.9	0.9	0.3	60.9	0.44	38	33.1	1.2	1.6	80.4	3.3	4.0	15.9
	Zn Cleaner 1 Sc. Tailings		3.0	2.9	0.5	5.6	4.55	121	34.6	1.0	0.6	1.7	7.9	2.9	3.9
	Zn Rougher Tailings		51.6	0.5	0.2	0.4	0.33	10	21.4	2.9	3.6	2.0	9.8	4.2	41.1
	Feed		100.0	9.2	2.3	9.8	1.73	124	26.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-024 Test 89	Talc Concentrate	Flotation Feed: 60 µm; Cu/Pb Rougher Concentrate: 44 µm; Zn Rougher Concentrate: 45 µm	3.3	0.5	0.5	0.6	2.81	24	1.7	0.2	0.5	0.1	7.1	0.6	0.2
	Cu/Pb Cleaner 2 Conc.		27.7	27.1	9.3	1.8	3.36	410	32.6	90.1	91.8	3.5	70.6	81.6	31.1
	Zn Cleaner 2 Concentrate		20.0	0.7	0.2	61.8	0.29	40	33.7	1.8	1.6	86.6	4.4	5.8	23.3
	Zn Cleaner 1 Sc. Tailings		4.5	2.1	0.4	3.3	1.10	49	45.0	1.1	0.7	1.0	3.8	1.6	7.1
	Zn Rougher Tailings		39.7	0.6	0.2	0.7	0.27	13	23.6	2.6	2.4	1.9	8.1	3.7	32.4
	Feed		100.0	8.3	2.8	14.3	1.32	139	28.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-039 Test 111	Talc Concentrate	Flotation Feed: 75 µm; Cu/Pb Rougher Concentrate: 45 µm; Zn Rougher Concentrate: 48 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		21.0	21.3	16.8	6.0	3.44	420	30.3	88.0	87.1	8.3	42.7	83.1	22.6
	Zn Cleaner 2 Concentrate		22.8	0.5	0.4	54.4	0.79	22	33.4	2.3	2.5	81.9	10.7	4.7	27.1
	Zn Cleaner 1 Sc. Tailings		10.3	0.5	0.4	1.0	2.16	15	49.0	1.1	1.0	0.7	13.2	1.5	18.0
	Zn Rougher Tailings		37.0	0.4	0.4	1.1	0.50	11	14.7	3.2	3.6	2.7	11.0	3.8	19.3
	Feed		100.0	5.1	4.0	15.1	1.69	106	28.1	100.0	100.0	100.0	100.0	100.0	100.0
ARC-044 Test 81	Talc Concentrate	Flotation Feed: 81 µm; Cu/Pb Rougher Concentrate: 57 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		6.0	31.1	0.2	0.3	3.57	260	33.8	96.0	23.9	23.9	79.2	91.4	38.9
	Zn Cleaner 2 Concentrate		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Cleaner 1 Sc. Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Rougher Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Feed		100.0	1.9	0.0	0.1	0.27	17	5.2	100.0	100.0	100.0	100.0	100.0	100.0
ARC-050 Test 77	Talc Concentrate	Flotation Feed: 71 µm; Cu/Pb Rougher Concentrate: 43 µm; Zn Rougher Concentrate: 46 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		10.1	31.1	1.0	1.1	3.06	294	33.0	93.3	67.2	9.8	71.5	79.8	37.8
	Zn Cleaner 2 Concentrate		1.8	2.6	0.3	47.1	0.85	92	29.8	1.4	3.7	75.4	3.5	4.4	6.1
	Zn Cleaner 1 Sc. Tailings		9.6	0.2	0.1	0.1	0.18	7	6.0	0.4	3.3	1.2	4.0	1.8	6.5
	Zn Rougher Tailings		73.9	0.1	0.0	0.0	0.08	3	5.3	1.5	20.6	2.6	13.7	6.0	44.2
	Feed		100.0	3.4	0.1	1.1	0.43	37	8.8	100.0	100.0	100.0	100.0	100.0	100.0
ARC-051 Test 110	Talc Concentrate	Flotation Feed: 64 µm; Cu/Pb Rougher Concentrate: 48 µm; Zn Rougher Concentrate: 50 µm	1.0	0.3	1.6	0.6	0.06	40	1.9	0.1	0.7	0.0	0.2	0.6	0.1
	Cu/Pb Cleaner 2 Conc.		14.4	24.4	13.0	3.4	1.67	368	31.7	84.2	84.2	3.8	62.4	82.1	15.1
	Zn Cleaner 2 Concentrate		19.5	0.8	0.3	59.0	0.20	10	33.5	3.5	2.7	88.6	10.1	3.0	21.5
	Zn Cleaner 1 Sc. Tailings		12.5	0.6	0.3	0.9	0.18	9	50.4	1.6	1.6	0.8	5.8	1.7	20.8
	Zn Rougher Tailings		47.3	0.3	0.2	0.6	0.08	8	22.9	3.7	4.9	2.2	9.8	5.8	35.6
	Feed		100.0	4.2	2.2	13.0	0.39	65	30.4	100.0	100.0	100.0	100.0	100.0	100.0
ARC-052 Test 110	Talc Concentrate	Flotation Feed: 69 µm; Cu/Pb Rougher Concentrate: 45 µm; Zn Rougher Concentrate: 43 µm	1.1	0.6	0.2	0.6	0.01	13	1.7	0.1	0.1	0.0	0.0	0.1	0.1
	Cu/Pb Cleaner 2 Conc.		26.5	29.7	7.2	2.0	3.67	370	32.2	88.1	88.8	3.9	78.8	77.6	29.9
	Zn Cleaner 2 Concentrate		16.2	0.6	0.3	63.8	0.22	24	33.6	1.0	2.0	73.7	2.9	3.1	19.0
	Zn Cleaner 1 Sc. Tailings		4.9	2.3	0.4	7.5	0.57	44	38.6	1.3	0.9	2.6	2.3	1.7	6.6
	Zn Rougher Tailings		43.8	0.5	0.1	0.4	0.08	9	23.3	2.2	2.6	1.1	2.8	3.1	35.8
	Feed		100.0	8.9	2.2	14.0	1.24	126	28.5	100.0	100.0	100.0	100.0	100.0	100.0
ARC-054 Test 60	Talc Concentrate	Flotation Feed: 67 µm; Cu/Pb Rougher Concentrate: 61 µm; Zn Rougher Concentrate: 38 µm	2.0	3.6	0.3	0.4	0.15	32	4.8	1.4	0.4	0.1	0.6	0.7	0.4
	Cu/Pb Cleaner 2 Conc.		16.8	28.3	6.4	1.8	1.69	460	33.0	90.3	89.9	5.2	56.8	83.4	24.6
	Zn Cleaner 2 Concentrate		7.3	0.8	0.3	60.2	0.64	38	32.1	1.1	1.9	76.2	9.3	3.0	10.3
	Zn Cleaner 1 Sc. Tailings		1.9	1.4	0.3	6.3	0.39	51	20.4	0.5	0.5	2.1	1.5	1.1	1.7
	Zn Rougher Tailings		68.4	0.2	0.1	0.2	0.09	6	18.9	2.3	2.9	2.0	12.3	4.1	57.3
	Feed		100.0	5.3	1.2	5.7	0.50	93	22.5	100.0	100.0	100.0	100.0	100.0	100.0
ARC-055 Test 92	Talc Concentrate	Flotation Feed: 72 µm; Cu/Pb Rougher Concentrate: 69 µm; Zn Rougher Concentrate: 54 µm	26.9	0.4	0.2	0.8	0.01	15	1.9	3.7	6.3	3.5	0.7	6.2	3.5
	Cu/Pb Cleaner 2 Conc.		9.9	26.3	6.4	1.5	3.08	508	29.9	84.9	78.3	2.4	81.5	77.9	20.0
	Zn Cleaner 2 Concentrate		8.9	0.7	0.2	60.0	0.15	18	32.0	2.0	2.1	84.9	3.6	2.5	19.2
	Zn Cleaner 1 Sc. Tailings		2.2	1.0	0.3	2.9	0.57	23	13.7	0.7	0.7	1.0	3.3	0.8	2.0
	Zn Rougher Tailings		48.4	0.2	0.1	0.3	0.01	8	15.5	2.7	7.7	2.4	1.3	6.0	50.4
	Feed		100.0	3.1	0.8	6.3	0.38	65	14.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-056 Test 82	Talc Concentrate	Flotation Feed: 73 µm; Cu/Pb Rougher Concentrate: 52 µm; Zn Rougher Concentrate: 29 µm	11.7	0.7	0.1	0.7	0.10	17	2.1	1.0	2.0	1.0	1.2	1.8	0.9
	Cu/Pb Cleaner 2 Conc.		23.5	30.6	3.0	1.3	3.03	390	33.3	90.5	84.8	3.9	71.1	82.3	30.2
	Zn Cleaner 2 Concentrate		10.7	0.9	0.2	60.4	0.26	20	32.4	1.2	1.9	85.1	2.8	1.9	13.4
	Zn Cleaner 1 Sc. Tailings		1.0	4.3	0.5	7.2	1.51	90	27.7	0.6	0.6	1.0	1.6	0.8	1.1
	Zn Rougher Tailings		49.7	0.5	0.1	0.4	0.28	14	26.2	2.9	6.5	2.7	13.9	6.2	50.1
	Feed		100.0	8.0	0.8	7.6	1.00	112	26.0	100.0	100.0	100.0	100.0	100.0	100.0
ARC-057 Test 83	Talc Concentrate	Flotation Feed:132µm; Cu/Pb Rougher Concentrate: 87 µm	44.1	0.2	0.0	0.1	0.03	2	0.4	9.1	20.4	14.7	12.3	9.5	10.0
	Cu/Pb Cleaner 2 Conc.		3.9	21.0	0.9	0.2	1.57	184	21.2	80.1	52.1	2.6	56.8	76.7	45.7
	Zn Cleaner 2 Concentrate		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Cleaner 1 Sc. Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Rougher Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Feed		100.0	1.0	0.1	0.2	0.11	9	1.8	100.0	100.0	100.0	100.0	100.0	100.0
ARC-058 Test 74	Talc Concentrate	Flotation Feed: 68 µm; Cu/Pb Rougher Concentrate: 50 µm; Zn Rougher Concentrate: 44 µm	0.7	1.6	1.0	0.9	0.42	47	4.1	0.2	0.3	0.0	0.2	0.3	0.1
	Cu/Pb Cleaner 2 Conc.		21.1	26.1	10.2	2.6	4.47	394	32.9	89.5	90.3	3.8	68.0	79.4	22.5
	Zn Cleaner 2 Concentrate		21.8	0.6	0.2	58.0	0.35	22	33.7	2.0	1.9	88.1	5.5	4.6	23.8
	Zn Cleaner 1 Sc. Tailings		4.7	1.3	0.4	2.6	1.67	27	42.8	1.0	0.7	0.8	5.6	1.2	6.4
	Zn Rougher Tailings		46.9	0.3	0.1	0.6	0.21	7	27.4	2.3	2.7	1.9	7.1	3.1	41.5

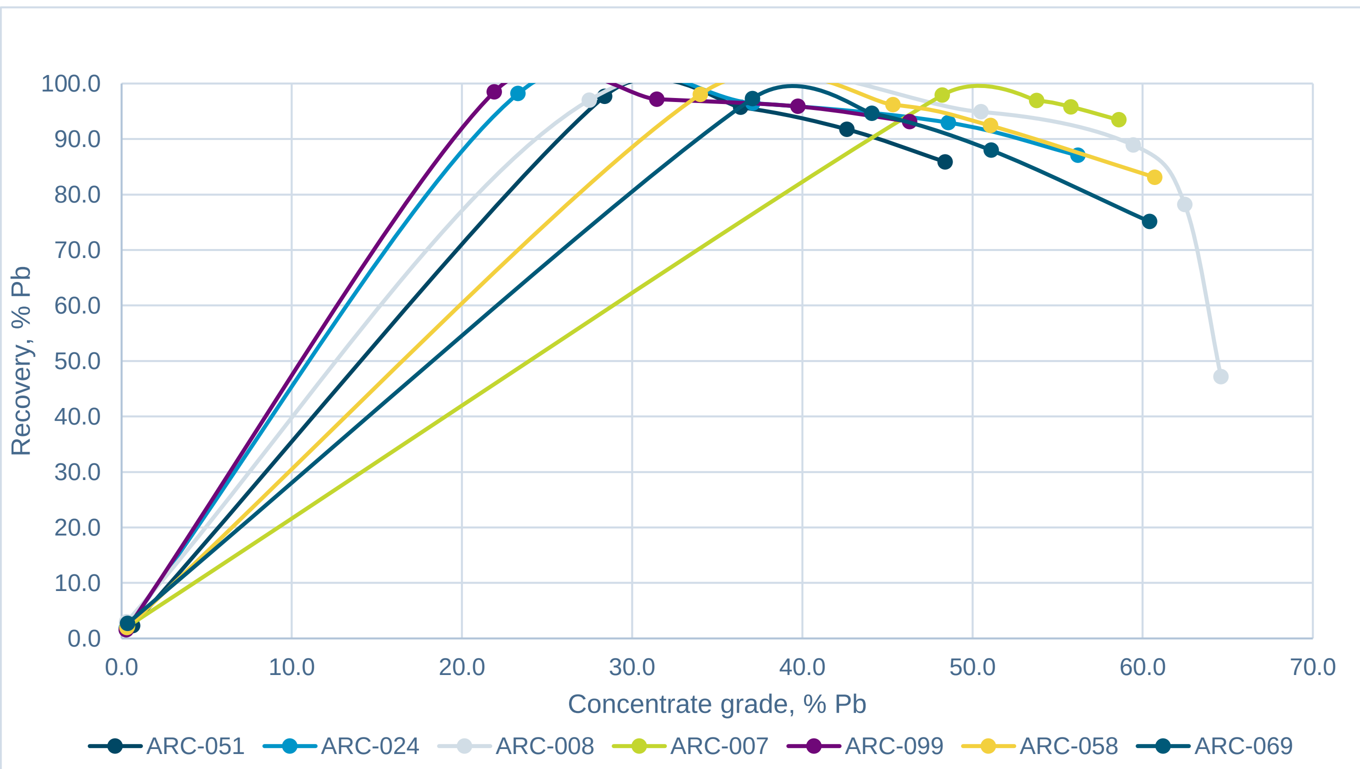
Test	Product	Regrind Size 80% Passing	Weight (%)	Assays						Distribution (%)					
				Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	S (%)	Cu	Pb	Zn	Au	Ag	S
	Feed		100.0	6.2	2.4	14.4	1.39	105	30.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-059 Test 84	Talc Concentrate	Flotation Feed: 85 µm; Cu/Pb Rougher Concentrate: 71 µm; Zn Rougher Concentrate: 69 µm	16.5	0.9	0.2	0.9	0.96	17	2.1	2.4	4.2	2.2	29.8	3.6	2.6
	Cu/Pb Cleaner 2 Conc.		18.1	29.9	2.9	1.4	1.70	366	32.2	91.0	87.8	4.1	57.9	85.3	45.0
	Zn Cleaner 2 Concentrate		9.9	0.9	0.1	55.0	0.14	22	32.0	1.5	2.2	85.9	2.6	2.8	24.4
	Zn Cleaner 1 Sc. Tailings		2.7	0.7	0.1	1.2	0.56	14	7.4	0.3	0.3	0.5	2.9	0.5	1.6
	Zn Rougher Tailings		48.3	0.1	0.0	0.1	0.01	5	5.5	0.8	2.5	1.1	0.9	3.1	20.6
	Feed		100.0	5.9	0.6	6.3	0.53	77	12.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-062 Test 75	Talc Concentrate	Flotation Feed: 71 µm; Cu/Pb Rougher Concentrate: 42 µm; Zn Rougher Concentrate: 43 µm	1.8	0.8	7.6	1.6	0.14	56	4.3	0.6	2.0	0.1	0.8	1.8	0.4
	Cu/Pb Cleaner 2 Conc.		14.6	13.6	41.0	3.6	1.36	326	21.7	88.2	91.8	2.7	63.2	87.8	16.1
	Zn Cleaner 2 Concentrate		30.7	0.3	0.4	58.6	0.12	6	32.3	4.2	1.7	92.0	11.7	3.4	50.3
	Zn Cleaner 1 Sc. Tailings		4.4	0.7	0.5	3.1	0.61	10	37.1	1.3	0.4	0.7	8.6	0.8	8.4
	Zn Rougher Tailings		44.9	0.1	0.2	0.4	0.04	3	8.2	2.0	1.0	1.0	5.7	2.5	18.7
	Feed		100.0	2.3	6.5	19.5	0.31	54	19.7	100.0	100.0	100.0	100.0	100.0	100.0
ARC-063 Test 85	Talc Concentrate	Flotation Feed:117µm; Cu/Pb Rougher Concentrate: 72 µm; Zn Rougher Concentrate: 69 µm	43.1	1.1	0.1	0.7	0.01	6	2.4	11.2	25.3	12.3	6.8	15.2	12.3
	Cu/Pb Cleaner 2 Conc.		11.4	29.3	0.4	0.6	0.36	102	30.4	78.6	56.3	3.0	64.7	68.7	41.5
	Zn Cleaner 2 Concentrate		3.1	1.7	0.1	55.6	0.17	14	32.7	1.2	2.2	73.0	8.4	2.6	12.2
	Zn Cleaner 1 Sc. Tailings		3.2	1.0	0.1	1.4	0.06	7	6.6	0.8	1.9	1.9	3.0	1.3	2.5
	Zn Rougher Tailings		35.3	0.2	0.0	0.2	0.01	2	5.9	1.4	8.3	2.5	5.5	4.2	24.9
	Feed		100.0	4.3	0.1	2.4	0.06	17	8.4	100.0	100.0	100.0	100.0	100.0	100.0
ARC-068 Test 97	Talc Concentrate	Flotation Feed: 68 µm; Cu/Pb Rougher Concentrate: 69 µm; Zn Rougher Concentrate: 59 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		8.1	31.4	0.8	0.7	1.36	186	33.7	95.6	71.0	4.4	75.5	72.4	42.9
	Zn Cleaner 2 Concentrate		2.2	0.9	0.1	50.5	0.52	34	32.0	0.7	2.8	84.9	7.9	3.6	11.1
	Zn Cleaner 1 Sc. Tailings		3.4	0.3	0.1	0.4	0.12	18	21.7	0.4	2.1	1.0	2.8	3.0	11.6
	Zn Rougher Tailings		83.3	0.0	0.0	0.1	0.01	4	1.9	0.6	17.4	3.8	5.7	16.0	25.3
	Feed		100.0	2.7	0.1	1.3	0.15	21	6.4	100.0	100.0	100.0	100.0	100.0	100.0
ARC-069 Test 106	Talc Concentrate	Flotation Feed: 62 µm; Cu/Pb Rougher Concentrate: 49 µm; Zn Rougher Concentrate: 56 µm	9.6	0.3	0.3	0.7	0.01	16	1.7	0.8	2.4	0.6	0.7	2.7	0.7
	Cu/Pb Cleaner 2 Conc.		10.8	25.7	9.6	2.7	0.58	396	32.8	86.4	83.3	3.0	44.8	74.7	15.0
	Zn Cleaner 2 Concentrate		15.3	0.6	0.2	58.4	0.20	24	33.9	3.0	2.7	91.2	22.0	6.4	22.1
	Zn Cleaner 1 Sc. Tailings		12.2	0.4	0.2	0.7	0.12	12	50.0	1.5	1.7	0.9	10.5	2.6	25.8
	Zn Rougher Tailings		48.5	0.2	0.1	0.3	0.04	9	14.8	3.2	3.9	1.6	13.9	7.6	30.5
	Feed		100.0	3.2	1.2	9.8	0.14	57	23.6	100.0	100.0	100.0	100.0	100.0	100.0
ARC-072 Test 78	Talc Concentrate	Flotation Feed: 69 µm; Cu/Pb Rougher Concentrate: 46 µm; Zn Rougher Concentrate: 36 µm	3.0	0.6	0.4	1.3	0.32	57	3.2	0.4	2.7	0.4	0.8	1.6	0.4
	Cu/Pb Cleaner 2 Conc.		16.3	29.3	1.8	3.5	5.38	476	33.6	91.5	70.2	6.1	71.3	71.6	21.9
	Zn Cleaner 2 Concentrate		14.2	0.7	0.2	56.2	0.69	48	33.3	1.8	5.0	85.5	8.0	6.3	18.9
	Zn Cleaner 1 Sc. Tailings		3.8	0.9	0.2	2.1	0.54	51	27.1	0.6	1.8	0.8	1.7	1.8	4.1
	Zn Rougher Tailings		58.8	0.2	0.1	0.2	0.24	19	21.2	2.0	13.9	1.2	11.5	10.3	49.9
	Feed		100.0	5.2	0.4	9.3	1.23	108	25.0	100.0	100.0	100.0	100.0	100.0	100.0
ARC-080 Test 113	Talc Concentrate	Flotation Feed: 68 µm; Cu/Pb Rougher Concentrate: 45 µm	6.4	0.2	0.0	0.0	0.15	10	0.4	0.9	4.7	1.7	1.6	2.7	1.0
	Cu/Pb Cleaner 2 Conc.		3.6	30.4	1.0	0.4	12.7	426	31.3	88.4	66.5	9.8	77.2	65.0	43.3
	Zn Cleaner 2 Concentrate		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Cleaner 1 Sc. Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Rougher Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Feed		100.0	1.3	0.1	0.2	0.60	24	2.6	100.0	100.0	100.0	100.0	100.0	100.0
ARC-084 Test 91	Talc Concentrate	Flotation Feed: 78 µm; Cu/Pb Rougher Concentrate: 46 µm; Zn Rougher Concentrate: 43 µm	2.1	0.5	0.3	0.9	0.04	15	3.9	0.4	1.5	0.3	0.5	1.1	0.3
	Cu/Pb Cleaner 2 Conc.		9.0	25.2	3.0	5.5	1.40	180	35.0	85.8	60.1	9.3	71.7	57.6	13.0
	Zn Cleaner 2 Concentrate		7.5	0.9	0.3	54.6	0.18	42	33.1	2.7	5.3	77.4	7.7	11.3	10.3
	Zn Cleaner 1 Sc. Tailings		2.5	1.3	0.4	3.1	0.18	29	30.2	1.3	2.4	1.5	2.6	2.6	3.2
	Zn Rougher Tailings		75.9	0.2	0.1	0.3	0.01	7	21.9	4.3	22.3	4.4	4.3	18.9	68.9
	Feed		100.0	2.6	0.4	5.3	0.18	28	24.1	100.0	100.0	100.0	100.0	100.0	100.0
ARC-086 Test 86	Talc Concentrate	Flotation Feed: 59 µm; Cu/Pb Rougher Concentrate: 45 µm; Zn Rougher Concentrate: 35 µm	0.4	0.3	0.1	0.2	0.81	14	1.7	0.0	0.1	0.0	0.3	0.1	0.0
	Cu/Pb Cleaner 2 Conc.		10.7	28.7	7.1	1.2	6.54	430	33.3	75.9	87.3	1.8	63.0	55.4	15.6
	Zn Cleaner 2 Concentrate		12.4	5.1	0.2	53.8	0.79	176	32.1	15.7	2.4	92.9	8.9	26.4	17.5
	Zn Cleaner 1 Sc. Tailings		3.0	1.8	0.3	2.0	1.53	70	28.2	1.3	0.9	0.8	4.1	2.5	3.7
	Zn Rougher Tailings		71.0	0.1	0.1	0.1	0.18	9	19.1	2.3	4.9	1.2	11.5	7.7	59.6
	Feed		100.0	4.0	0.9	7.2	1.11	83	22.7	100.0	100.0	100.0	100.0	100.0	100.0
ARC-087 Test 87	Talc Concentrate	Flotation Feed: 67 µm; Cu/Pb Rougher Concentrate: 36 µm; Zn Rougher Concentrate: 31 µm	1.4	0.9	0.5	0.7	0.21	24	2.0	0.1	0.2	0.1	0.1	0.2	0.1
	Cu/Pb Cleaner 2 Conc.		37.1	26.4	10.4	6.6	6.04	394	30.7	91.9	91.9	16.6	66.5	83.9	47.6
	Zn Cleaner 2 Concentrate		14.1	0.7	0.4	61.7	0.78	30	32.4	0.9	1.3	59.0	3.3	2.4	19.0
	Zn Cleaner 1 Sc. Tailings		2.9	3.3	0.9	13.3	12.2	96	27.1	0.9	0.6	2.6	10.5	1.6	3.3
	Zn Rougher Tailings		36.9	0.6	0.2	0.7	0.61	18	13.1	1.9	1.7	1.8	6.7	3.8	20.2
	Feed		100.0	10.7	4.2	14.7	3.37	174	23.9	100.0	100.0	100.0	100.0	100.0	100.0
ARC-092 Test 90	Talc Concentrate	Flotation Feed: 68 µm; Cu/Pb Rougher Concentrate: 46 µm; Zn Rougher Concentrate: 29 µm	5.1	0.5	0.0	0.2	0.03	5	2.6	0.5	1.6	0.2	0.6	1.2	0.4
	Cu/Pb Cleaner 2 Conc.		13.7	29.1	0.4	2.5	1.01	108	34.6	87.6	46.6	6.7	54.3	67.0	15.4
	Zn Cleaner 2 Concentrate		7.8	1.2	0.2	54.4	0.59	24	33.8	2.0	9.0	83.0	18.0	8.4	8.5
	Zn Cleaner 1 Sc. Tailings		1.7	2.6	0.2	3.3	0.06	26	27.0	0.9	2.6	1.1	0.4	1.9	1.5
	Zn Rougher Tailings		68.8	0.3	0.1	0.2	0.04	4	31.8	4.4	31.8	2.8	10.8	12.4	71.0
	Feed		100.0	4.6	0.1	5.1	0.26	22	30.8	100.0	100.0	100.0	100.0	100.0	100.0
ARC-098 Test 93	Talc Concentrate	Flotation Feed:103µm; Cu/Pb Rougher Concentrate: 89 µm	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu/Pb Cleaner 2 Conc.		4.0	25.6	0.7	0.8	1.32	578	29.5	97.8	69.9	34.5	75.8	92.6	82.1
	Zn Cleaner 2 Concentrate		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Cleaner 1 Sc. Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Zn Rougher Tailings		-	-	-	-	-	-	-	-	-	-	-	-	-
	Feed		100.0	1.1	0.0	0.1	0.07	25	1.5	100.0	100.0	100.0	100.0	100.0	100.0
ARC-099 Test 80	Talc Concentrate	Flotation Feed: 69 µm; Cu/Pb Rougher Concentrate: 48 µm; Zn Rougher Concentrate: 41 µm	2.0	0.9	0.5	0.6	0.04	22	1.8	0.3	0.4	0.1	0.2	0.5	0.2
	Cu/Pb Cleaner 2 Conc.		19.9	28.3	9.9	2.8	0.95	404	31.9	90.4	89.6	4.7	55.9	86.6	29.0
	Zn Cleaner 2 Concentrate		15.7	0.6	0.3	63.2	0.11	18	32.5	1.4	2.1	83.3	5.1	3.0	23.3
	Zn Cleaner 1 Sc. Tailings		4.1	1.8	0.4	7.5	0.71	30	29.1	1.2	0.7	2.6	8.7	1.3	5.5
	Zn Rougher Tailings		54.2	0.2	0.1	0.2	0.10	4	14.7	2.1	2.0	0.8	16.1	2.3	36.4
	Feed		100.0	6.2	2.2	11.9	0.34	93	21.9	100.0	100.0	100.0	100.0	100.0	100.0

10.4.5.4 Copper/Lead Separation Testwork

A copper-lead separation test was included for the bulk concentrate produced in selected cleaner tests and later cycles of 5 locked-cycle tests. Table 10-30 summarises the copper/lead separation testwork results. Since there are only 2 concentrate products in the separation test, lead recover increases with lead concentrate grade (copper is supressed) until lead concentrate grades are increased to the point of pushing lead into the copper concentrate.

Reasonable separations were achieved for all but ARC-003. For ARC-003, a low-grade lead concentrate which measured only about 16% lead was produced, even when the test was repeated with a much higher sodium cyanide dosage. It is unclear what caused the poor copper-lead separation for ARC-003. For the other 7 samples tested, lead recoveries from the bulk concentrate ranged between 75 and 95% to a lead concentrate with an average grade of about 56% lead in open circuit.

Figure 10-9: Variability Composite Lead Recovery vs. Concentrate Grade

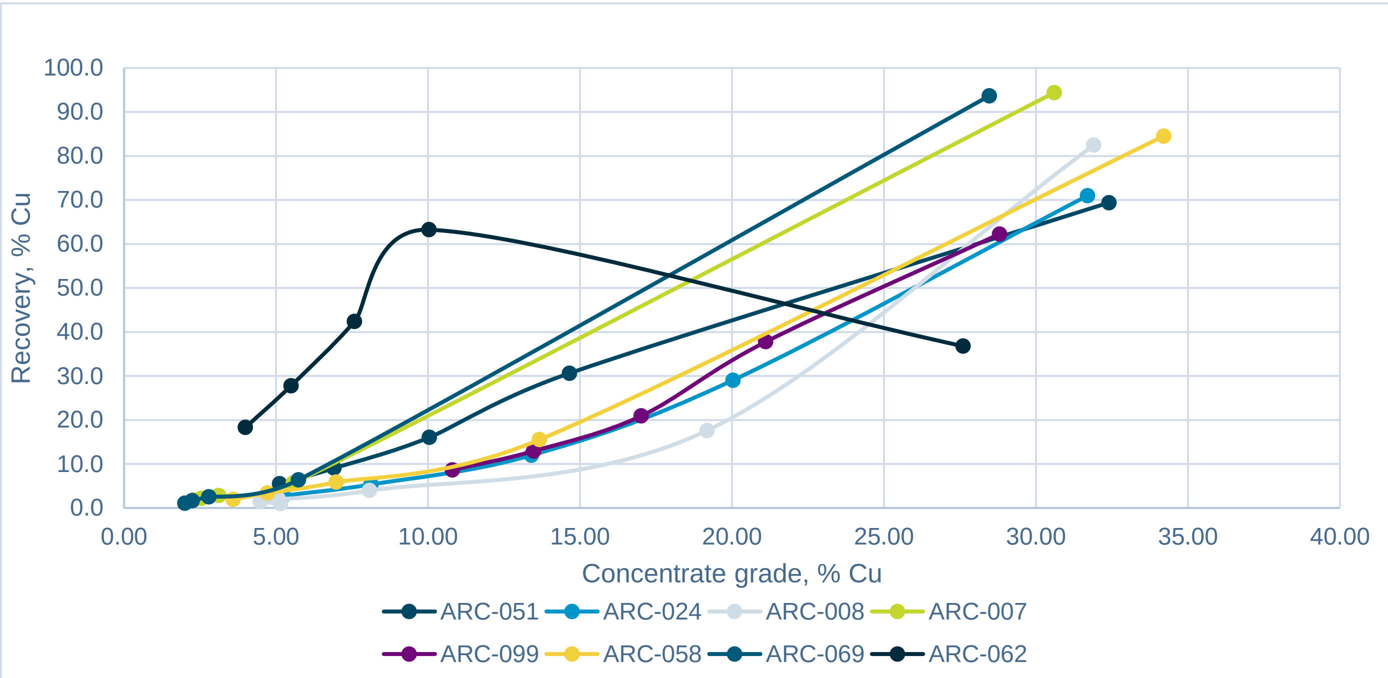


Source: Ausenco, 2022.

Copper-lead separation recoveries from tests on the locked cycle test concentrates were applied to bulk circuit recoveries to calculate overall recoveries. Silver recovery to the lead concentrate averaged about 75%, but gold recoveries in the copper separation tests varied widely and might indicate differences in gold deportment and associations between lithology types.

Figure 10-9 summarises the results showing the individual and combined concentrate grades and copper recovery. Figure 10-10 summarises the results showing the individual and combined concentrate grades and lead recovery.

Figure 10-10: Variability Composite Copper Recovery vs. Concentrate Grade



Source: Ausenco, 2022.

Table 10-30: ALS Lead Separation Test Results

Test	Product	Weight (%)	Assays						Distribution (%)					
			Cu (%)	Pb (%)	Zn (%)	Ag (g/t)	Au (g/t)	S (%)	Cu	Pb	Zn	Ag	Au	S
ARC-051 Cu/Pb Separation Feed from Open Circuit Test (Test 88)	Pb 3rd Cleaner Concentrate	25.5	5.1	48.4	9.3	1242	4.6	21.5	5.5	86.2	68.2	83.9	75.9	17.7
	Pb 2nd Cleaner Concentrate	30.9	6.9	43.8	9.11	1098	3.94	23.5	9.0	92.0	81.0	90.0	79.7	23.4
	Pb 1st Cleaner Concentrate	37.7	10.0	37.3	8.0	938	3.35	25.3	16	95.8	86.9	93.9	82.8	30.8
	Pb Rougher Concentrate	49.4	14.7	29.1	6.41	734	2.68	27.3	30.6	97.8	91.1	96.2	86.4	43.6
	Pb Rougher Tailings (Cu Concentrate)	50.6	32.4	0.7	0.6	28	0.41	34.5	69.4	2.2	8.9	3.8	13.6	56.4
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	23.6	14.7	3.47	377	1.53	31	100	100	100	100	100	100
ARC-024 Cu/Pb Separation Feed from Open Circuit Test (Test 89)	Pb 3rd Cleaner Concentrate	14.4	5.2	58.6	5.9	2230	17	18.6	2.8	87.5	47.3	78.5	72.9	8.2
	Pb 2nd Cleaner Concentrate	17.8	8.1	50.5	5.7	1929	14.4	21.0	5.3	93.2	56.4	83.9	76.2	11.5
	Pb 1st Cleaner Concentrate	24.2	13.4	38.5	4.8	1481	11.3	24.4	12.0	96.6	64.6	87.5	81.1	18.2
	Pb Rougher Concentrate	39.3	20.0	24.2	3.3	949	7.4	28.3	29.0	98.2	72.7	90.8	86.5	34.2
	Pb Rougher Tailings (Cu Concentrate)	60.7	31.7	0.3	0.8	62	0.75	35.3	81	1.8	27.3	9.2	13.5	65.8
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	27.1	9.7	1.8	410	3.4	32.6	100	100	100	100	100	100
ARC-007 Cu/Pb Separation Feed from LCT-96 (Cycle 5)	Pb 3rd Cleaner Concentrate	19.3	2.3	58.6	6.0	1710	0.4	19.9	1.8	93.4	29	57.3	1.7	12.1
	Pb 2nd Cleaner Concentrate	20.8	2.5	55.8	6.04	1629	0.5	21.4	2.2	95.8	31.4	58.8	2.2	14.0
	Pb 1st Cleaner Concentrate	21.9	3.1	53.8	6.04	1574	0.54	21.8	2.8	96.9	33.0	59.6	2.4	15.1
	Pb Rougher Concentrate	24.6	5.6	48.2	5.8	1438	1.05	22.9	5.6	97.9	35.4	61.3	5.4	17.8
	Pb Rougher Tailings (Cu Concentrate)	75.4	30.6	0.3	3.4	296	6.03	34.6	94.4	2.1	64.6	38.7	94.6	82.2
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	24.4	12.1	4	577	4.8	31.7	100	100	100	100	100	100
ARC-099 Cu/Pb Separation Feed from LCT-100 (Cycle 5)	Pb 3rd Cleaner Concentrate	20.3	10.7	46.5	5	1420	0.98	22.1	8.6	94.7	18.3	81.2	12.4	14.2
	Pb 2nd Cleaner Concentrate	23.9	13.1	40.2	4.8	1230	0.94	23.8	12.4	96.3	20.7	82.8	14.0	18.0
	Pb 1st Cleaner Concentrate	29.0	15.7	33.6	4.52	1033	0.88	25.5	18.0	97.4	23.6	84.2	15.9	23.4
	Pb Rougher Concentrate	37.6	18.9	26.1	4.3	815	0.96	27.4	28.1	98.2	29.2	86.1	22.4	32.5
	Pb Rougher Tailings (Cu Concentrate)	62.4	29.2	0.3	6.3	79	2.0	34.2	71.9	1.8	70.8	13.9	77.6	67.5
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	25.3	10.0	5.5	356	1.6	31.6	100	100	100	100	100	100
ARC-058 Cu/Pb Separation Feed from LCT-104 (Cycle 5)	Pb 3rd Cleaner Concentrate	14.9	3.6	60.7	5.2	2012	8.75	19.2	1.9	83.1	32.1	68.8	37.6	8.8
	Pb 2nd Cleaner Concentrate	19.8	4.7	51.1	5.9	17.1	7.5	22.8	3.4	92.4	47.8	76.9	42.6	13.9
	Pb 1st Cleaner Concentrate	23.2	7.0	45.3	5.6	1523	6.8	24.4	5.8	96.2	53.7	80.8	45.1	17.4
	Pb Rougher Concentrate	31.5	13.7	34.0	4.6	1166	6.1	27.0	15.5	98.1	59.8	84.0	54.9	26.2
	Pb Rougher Tailings (Cu Concentrate)	68.5	34.2	0.3	1.4	102	2.3	35.0	84.5	1.9	40.2	16.0	45.1	73.8
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	27.7	10.9	2.4	437	3.5	32.5	100	100	100	100	100	100
ARC-062 Cu/Pb Separation Feed from LCT-105 (Cycle 5)	Pb 3rd Cleaner Concentrate	60.0	4.0	63.4	7.0	482	1.5	17.8	18.3	92.1	74.8	85.6	65.5	45.3
	Pb 2nd Cleaner Concentrate	66.2	5.5	59.3	6.9	454	1.5	19.0	27.8	95.0	81.4	89.0	72.1	53.3
	Pb 1st Cleaner Concentrate	73.2	7.6	54.6	6.6	424	1.7	20.2	42.4	96.7	85.9	91.8	93.1	62.8
	Pb Rougher Concentrate	82.6	10.0	49.1	6.2	389	1.5	21.7	63.2	98.1	90.5	94.9	94.9	75.9
	Pb Rougher Tailings (Cu Concentrate)	17.4	27.6	4.6	3.1	98	0.4	32.6	36.8	1.9	9.5	5.1	5.1	24.1
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	13.1	41.3	5.6	338	1.3	23.6	100	100	100	100	100	100
ARC-069 Cu/Pb Separation Feed from LCT-114 (Cycle 5)	Pb 3rd Cleaner Concentrate	12.0	2.0	60.4	3.2	2168	0.2	16.5	1.1	75.1	11.1	64.0	4.3	6.2
	Pb 2nd Cleaner Concentrate	16.5	2.2	51.1	4.6	1848	0.23	20.9	1.6	88.0	2.1	75.6	6.6	10.8
	Pb 1st Cleaner Concentrate	20.6	2.8	44.1	5.1	1596	0.25	23.9	2.5	94.6	30.6	81.4	8.7	15.4
	Pb Rougher Concentrate	25.2	5.7	37.1	5.1	1355	0.3	25.3	6.4	97.3	37.7	84.5	12.4	19.9
	Pb Rougher Tailings (Cu Concentrate)	74.8	28.5	0.4	2.9	84	0.7	34.3	93.6	2.7	62.3	15.5	87.6	80.1
	Cu/Pb 2nd Cleaner Concentrate (Head)	100.0	22.7	9.6	3.4	405	0.6	32.0	100	100	100	100	100	100
ARC-008 Cu/Pb Separation Feed from LCT-116 (Cycle 5)	Pb 4th Cleaner Concentrate	5.4	5.1	64.6	3.1	2540	3.1	16.1	1.0	47.2	6.5	39.0	4.3	2.6
	Pb 3rd Cleaner Concentrate	9.3	4.5	62.5	4.0	2298	2.3	17.4	1.5	78.2	14.6	60.5	5.4	4.9
	Pb 2nd Cleaner Concentrate	11.1	5.1	59.5	4.5	2176	2.2	18.4	2.0	88.9	19.4	68.4	6.2	6.1
	Pb 1st Cleaner Concentrate	13.9	8.1	50.5	4.9	1858	2.5	21.0	3.9	94.9	26.8	73.4	8.9	8.8
	Pb Rougher Concentrate	26.2	19.2	27.5	3.5	1060	3.8	27.5	17.6	97.0	36.0	78.7	25.5	21.7
	Pb Rougher Tailings (Cu Concentrate)	73.8	31.9	0.3	2.2	102	4.0	35.2	82.4	3.0	64.0	21.3	47.5	78.3

10.5 Comment on Mineral Processing and Metallurgical Testwork

The materials tested in the metallurgical programs as described in this Section are representative of the LOM production.

The bulk flotation flowsheet developed based on the 2012 to 2022 testwork is feasible.

Recent grinding and concentrate dewatering testwork have supported the selected flow sheet. Additional concentrate characterization is also recommended for future metallurgical samples to bolster the confidence in the design of the proposed grinding plant.

The continuation of the geometallurgical program is recommended to further confirm the flowsheet and better understand the continuity of the Arctic deposit with respect to the metallurgical response. This testwork is recommended to take the form of locked cycle tests using a variety of metallurgical samples representing both lithology types and spatial zones within the deposit. A continuation of a phased approach to additional testwork is recommended to ensure that representative testwork is managed properly.

Historical testwork has shown the lead concentrate quality to be impacted by talc flotation efficiency. Recent testwork demonstrated a better understanding of the level of talc in an expected process feed to avoid mis-representing the talc content of future samples. There is little reason to expect concentrates will be impaired by talc contamination as talc can be effectively removed from the base metal flotation process, mitigating the potential of talc diluting the base metal concentrates. Talc and fluorine levels will be managed by optimization of the talc pre-float circuit, effectively removing talc and fluorine to ensure the quality of the lead concentrate.

There are no outstanding metallurgical issues related to the production of a copper or zinc concentrate from all of the materials tested.

In the QP’s opinion, based on the summarized testwork and predictions made from that testwork in terms of mineralogy, plant design considerations, recovery forecasts, and presence of deleterious elements, the predictions of proposed throughput and metallurgical performance are adequate for the purposes used in the Report.

11 MINERAL RESOURCE ESTIMATES

11.1 Introduction

In early 2022, Ambler Metals completed an update of the Arctic Mineral Resource model based on all available drill results through to the end of 2021 drilling campaign. Wood's QP reviewed the model and visited the Arctic Property and core storage sites.

The current Mineral Resource was prepared by Ambler Metals using the following:

- Leapfrog software;
- Updated geological models including the talc model using the additional drill hole data collected from 2019 and 2021;
- Updated resource estimation using the additional drill hole data collected from 2019 and 2021 (see Table 11-1, and Section 11.3.3).

The Wood QP reviewed and validated the Mineral Resource model.

Composites and 3D solid models were constructed using Leapfrog commercial modelling software. The block model is setup in NAD 83 datum and extends a total of 1,910 m in the east-west direction, 1,950 m north-south, and 705 m in the vertical direction. The block model has a parent block size of 10 x 10 x 5 m with a sub-block size of 2 x 2 x 1 m.

The Mineral Resource estimate is accepted by Wood's QP as being current as of November 30, 2022 and is in accordance with the with the standards and definitions required under S-K 1300.

Table 11-1: List of Drill Holes used in 2022 Ambler Metals Resource Model that were Not Included in the 2020 SRK Model

Hole ID	Geologic Model	Resource Model	Notes
AR19-0164	Yes	Yes	-
AR19-0165	Yes	Yes	-
AR19-0165a	Yes	Yes	-
AR19-0166	Yes	Yes	-
AR19-0167	Yes	Yes	-
AR19-0168	Yes	Yes	-
AR19-0168a	Yes	Yes	-
AR19-0169	Yes	Yes	-
AR19-0170	Yes	Yes	-
AR19-0171	Yes	Yes	-
AR19-0172	Yes	Yes	-
AR21-0173	Yes	Yes	-
AR21-0174	Yes	Yes	-
AR21-0175	Yes	Partially missing talc information	Talc samples are not interpreted to be in Minzones
AR21-0176	Yes	Yes	-

Hole ID	Geologic Model	Resource Model	Notes
AR21-0177	Yes	Yes	-
AR21-0178	Yes	Partially missing talc information	Talc intercepts within Minzone
AR21-0179	Yes	No	-
AR21-0180	Yes	Partially missing talc information	Talc intercepts within Minzone 3
AR21-0181	Yes	No	-
AR21-0182	Yes	Yes	-
AR21-0183	Yes	No	-
AR21-0184	Yes	No	-
AR21-0185	Yes	Yes	-
AR21-0186	Yes	Partially missing assays	No assay data for Minzone 4, 3, 2.5, 2 and 1
AR21-0187	Yes	Partially missing talc information	Talc not currently in Minzone, this may change when assays are back
AR21-0188	Yes	Partially missing assay information (only below 202.23 meters)	No assay data for Minzone 3 and 5
AR21-0189	Yes	No	-
AR21-0190	Yes	No	-

11.2 Drill Hole Database

Ambler Metals uses GeoSpark software for the drill hole logging data entries and management. Core logging data is entered directly to the GeoSpark software by core logging geologists, and the data is synchronized to the local server during the field season. After the field season is over, data validation is performed within the GeoSpark software, and the data is transferred to the master server.

For the current resource model, the cut-off date for the assay drill hole database is November 30, 2021. The assay database consists of collar, survey, assay, lithology, SG, geology, and geotechnical information. The assay database contains 171 drill holes with 3,224 assay samples, totalling 3,539.73 m.

At the assay cut-off date, out of 18 holes drilled in 2021, Ambler Metals had yet to receive assay results for six full holes and six partial holes due to heavy backlog at the assay laboratory.

Sample data for copper, lead, zinc, gold, and silver were extracted from this database for use in the generation of this resource estimate.

Individual sample intervals range from 0.08 m to 5.85 m in length and average 1.10 m. Drill hole sample interval protocol has varied throughout time at Arctic. Typically, shoulder samples were taken outside the mineralized zones up to 20 m.

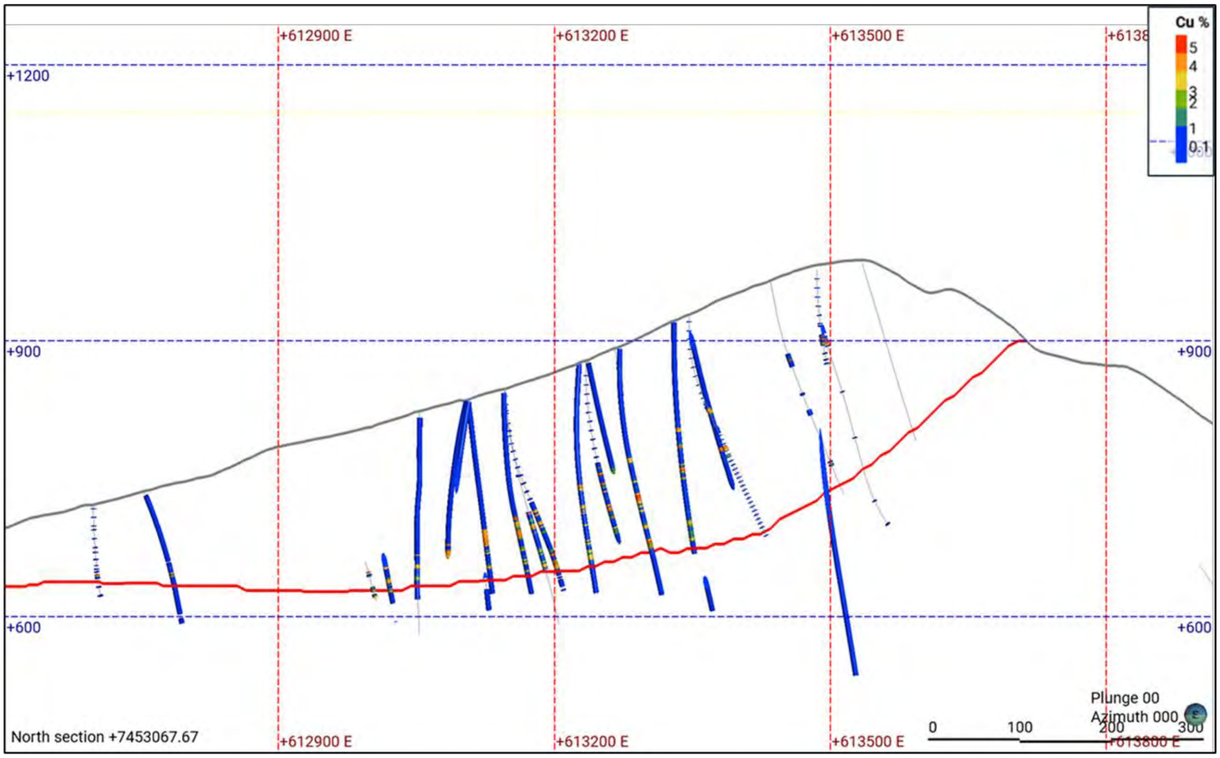
Table 11-2 shows a summary statistic table of the raw assay.

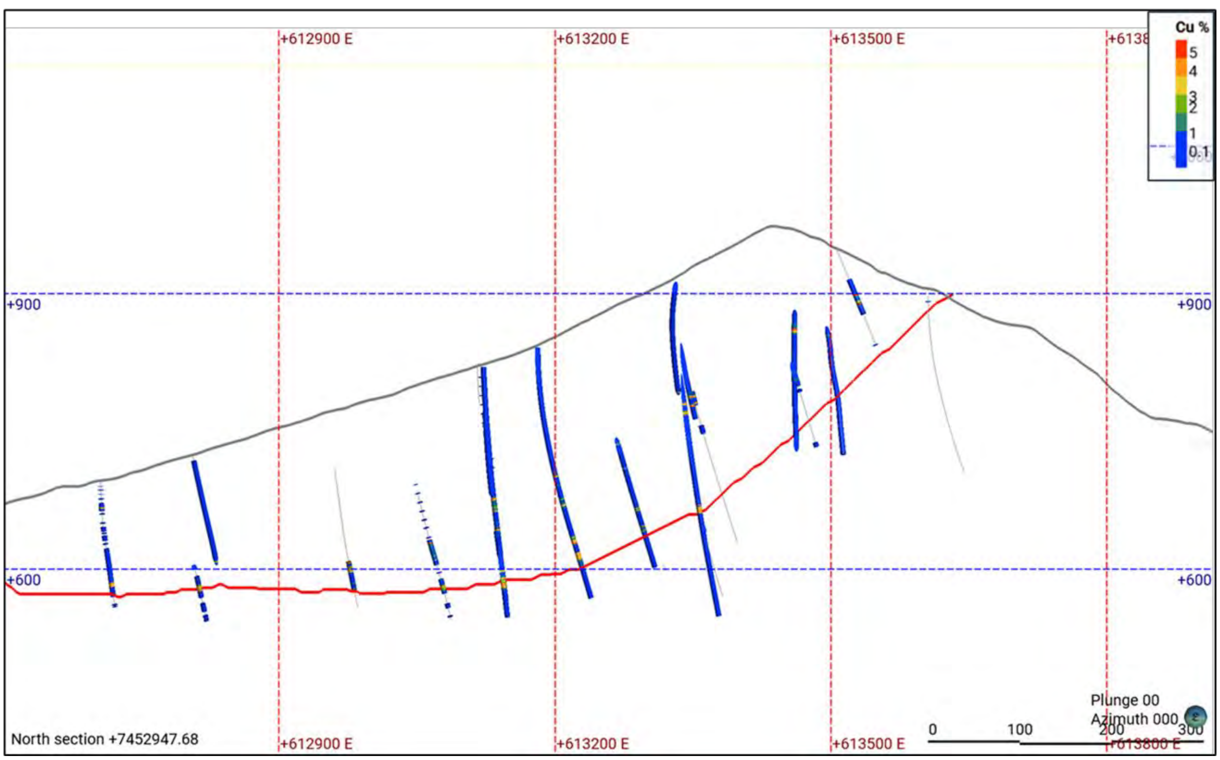
Table 11-2: Summary Statistics of Assay Database, Length Weighted

Element	Unit	Count	Total Length (m)	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation
Ag	g/t	2,983	3,298	0.0	1,155.0	46.1	59.77	1.30
As	ppm	1,891	2,131	0.6	10,000.0	1,028.1	1,837.60	1.79
Au	g/t	2,841	3,141	0.00	32.80	0.66	1.60	2.43
Cd	ppm	1,891	2,131	0.08	1,000.00	198.52	247.27	1.25
Cu	%	3,000	3,324	0.00	31.00	2.92	3.13	1.07
Fe	%	1,891	2,131	0.37	32.10	9.69	6.69	0.69
Hg	ppm	1,176	1,068	0.0	43.9	1.9	2.74	1.47
Pb	%	2,935	3,246	0.00	21.80	0.86	1.59	1.84
S	%	1,890	2,130	0.01	30.70	6.14	4.50	0.73
Sb	ppm	1,891	2,131	0.2	5,210.0	192.3	352.31	1.83
SG	-	3,184	3,540	2.01	4.99	3.29	0.36	0.11
Zn	%	2,974	3,294	0.00	30.00	3.95	4.96	1.26

The distribution of copper grades in drill holes proximal to the Arctic deposit is shown from two cross section maps in Figure 11-1.

Figure 11-1: East-West Cross Section for Copper Drill Hole Data, Looking North (Resource pit depicted in red)





Source: Ambler Metals, 2022.

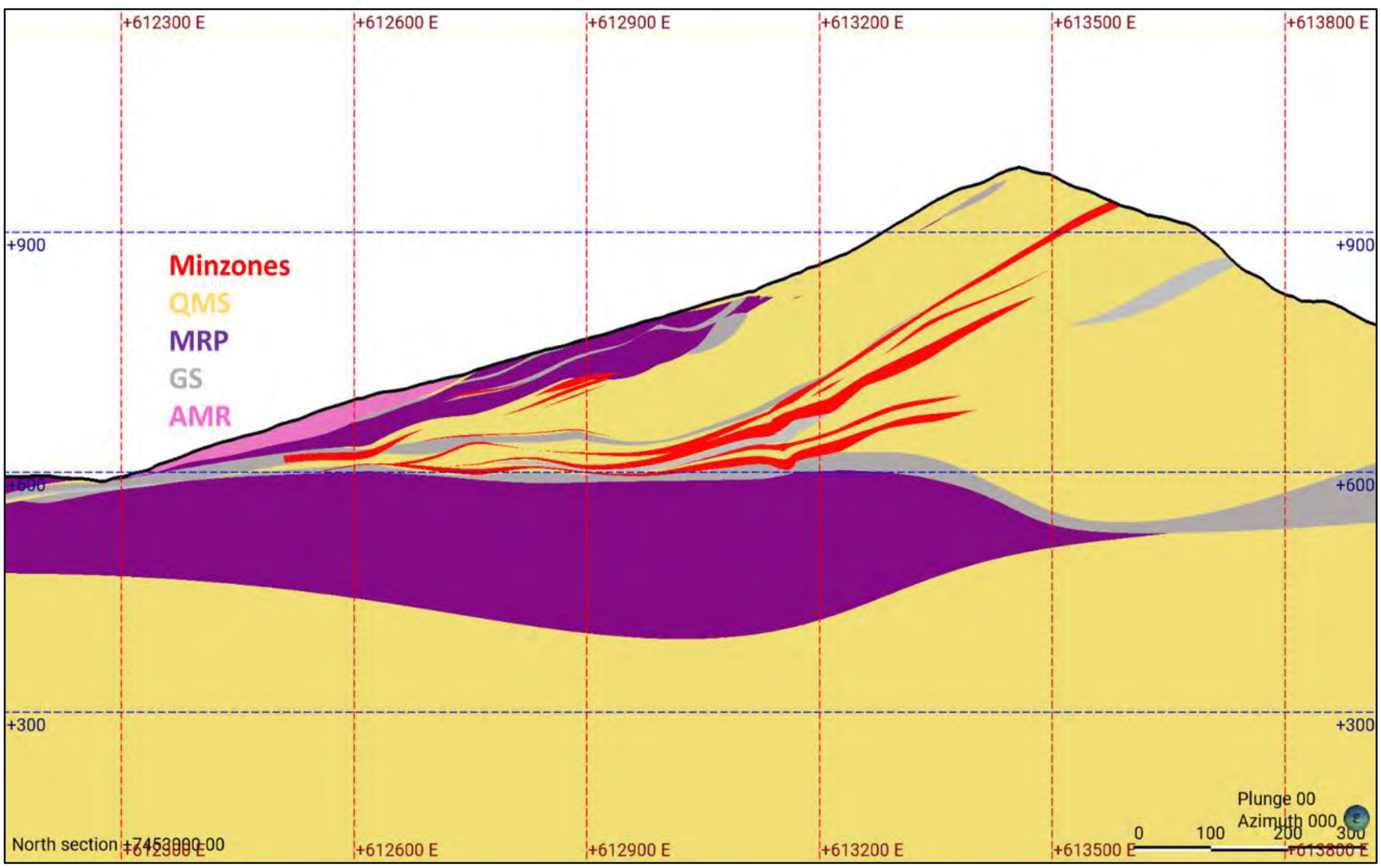
South32 developed and provided Ambler Metals with an algorithm to calculate talc proxy. The algorithm was applied in the database in 2019 and 2021. The algorithm uses the multi element assay results, mainly Mg, Al and K for the data from 1968 to 2017, and the four-acid digest-ICPMS data collected between 2004 to 2017 and generated calculated talc percent proxy values in the database. The algorithm was applied in the 2021 database to create talc model wireframes and estimate talc grades within the model.

11.3 Interpretations and Geological Modelling Procedures

11.3.1 Lithology Modelling

In 2019, SRK revised three-dimensional (3D) geological model wireframes generated by Trilogy in 2017. The geological model updates were completed using Leapfrog Geo software. In October of 2021, Ambler further updated the 2019 geological model wireframes using drilling information collected during 2019 and 2021 drilling campaign. Figure 11-2 shows East-West cross-section map of the modelled Arctic geological wireframes including mineralized zones (Minzones). The main lithological units in the Arctic deposit are shown in Table 11-3.

Figure 11-2: East-West Cross-Section of the Arctic Geological Model, Looking North



Source: Ambler Metals, 2022.

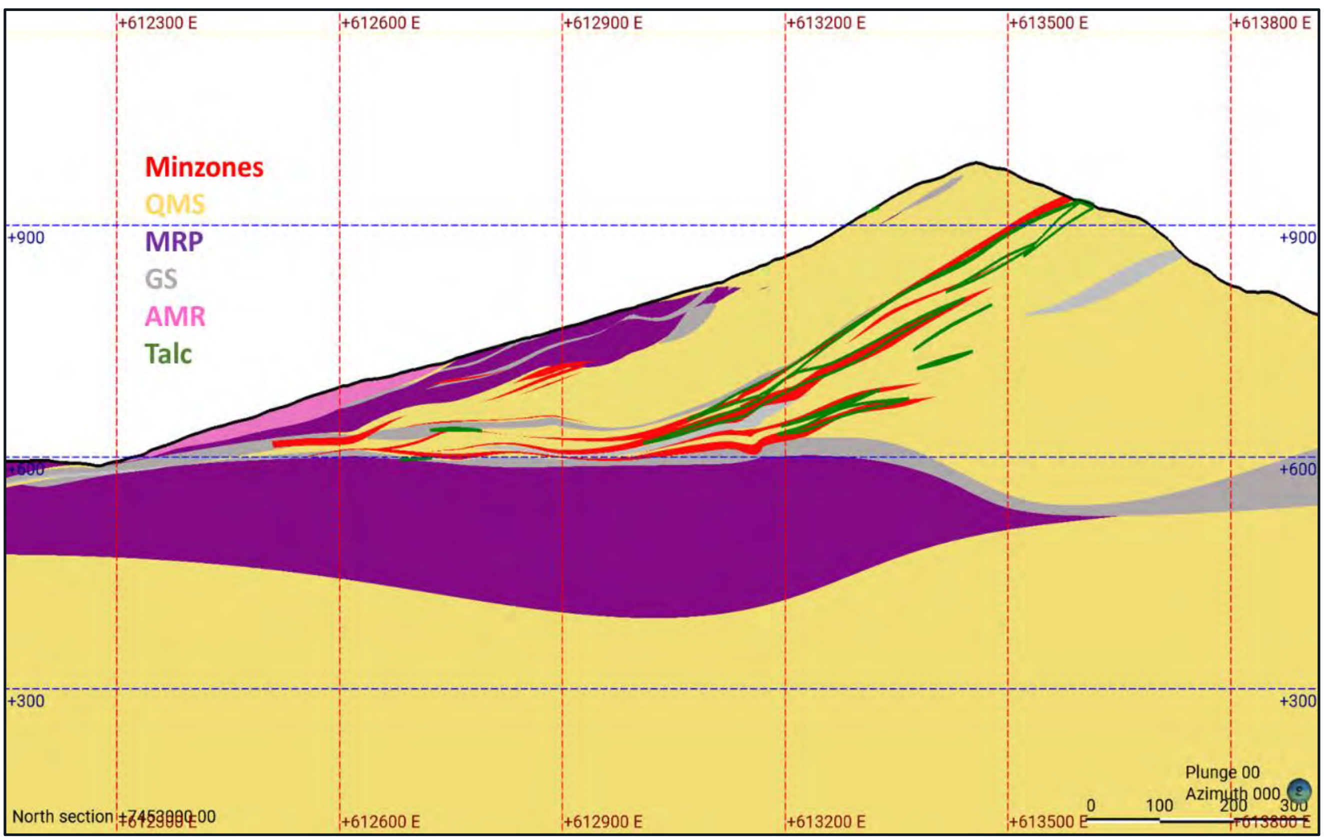
Table 11-3: Arctic Lithological Units

Lithological Unit Codes	Lithological Name
QMS	Series of felsic quartz mica schists
MRP	Meta-rhyolite porphyry
GS	Grey schist of the carbonaceous schists
AMR	Aphanitic meta-rhyolite

11.3.2 Talc Modelling

Three-dimensional talc model wireframes were generated using the updated talc proxy database using the algorithm provided by South32 by capturing high-grade and low-grade talc intervals. Samples that showed talc proxy percentages above 0% were included in the talc zones. Figure 11-3 shows an East-West cross-section map of the modelled Arctic talc wireframes (green).

Figure 11-3: East-West Cross-Section of Talc Model Wireframes, Looking North

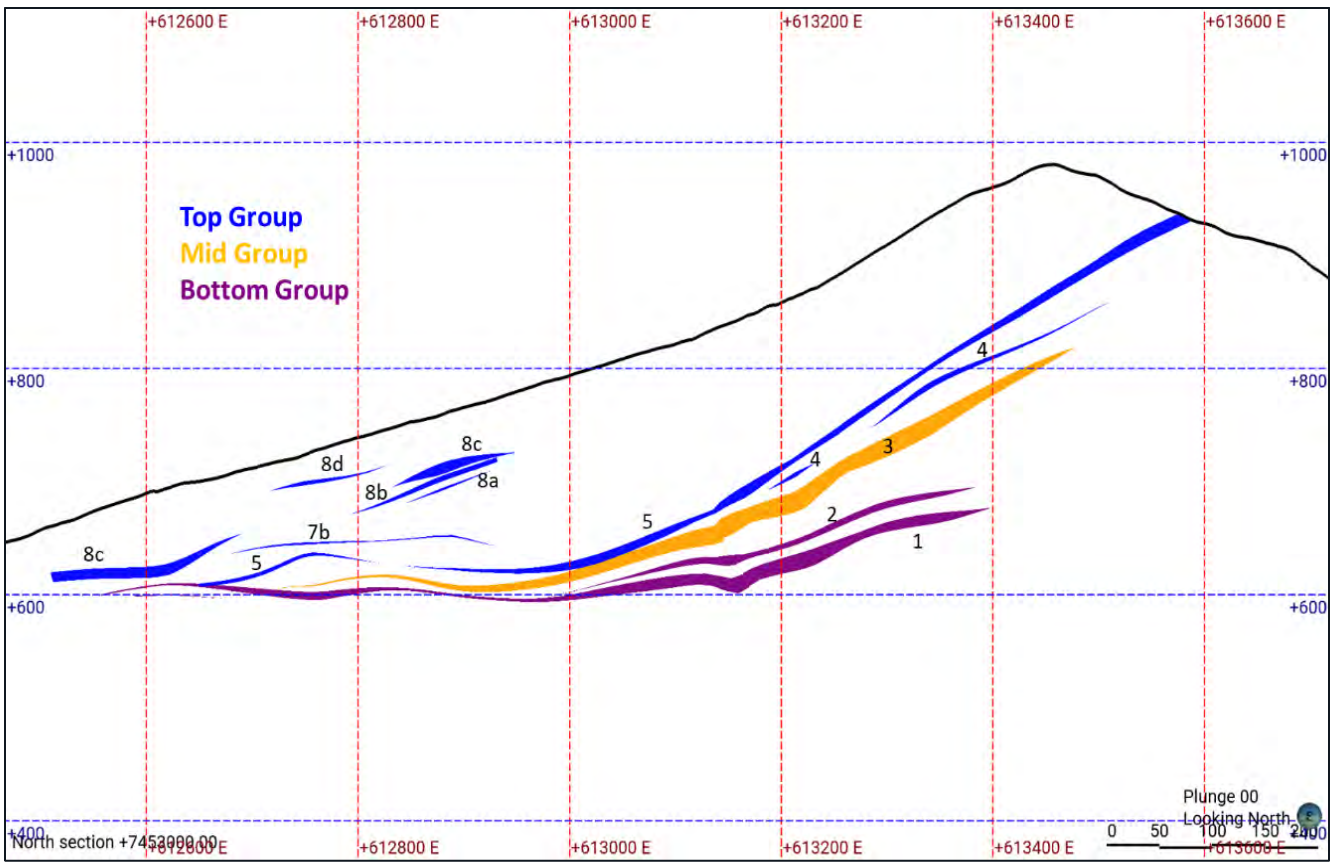


Source: Ambler Metals, 2022.

11.3.3 Mineralization Domains

The mineralized zone (Minzones) wireframes were updated using new assay data, typically using the 0.5% CuEq cut-off grade, as a guide where data is available. Drill holes that did not have assay results received by the assay database cut-off date were used to update the geological model but not used in the resource model. For these holes, the geological model was updated using the geology logging data to update the mineralized zone wireframes. Only minor revisions of the wireframes were made to the main mineralized zones (Zones 1, 1a, 2, 2.5, 3, 3_sub, 4 and 5) compared to the previous model. The main mineralized zones are Zones 1, 3, and 5. A cross section of mineralized zone wireframes, with estimation domain group and individual domain names, are shown in Figure 11-4. Table 11-4 shows mineralized zone domain codes used in the resource model.

Figure 11-4: East-West Cross-Section of the Arctic Deposit Showing Mineralized Zones



Source: Ambler Metals, 2022.

Table 11-4: Summary of Mineralized Zone Domains

Leapfrog Minzone	Domain	Estimation Domain
1	Zone 1	Bottom Group
1cz_sub	Zone 1 a	Bottom Group
2	Zone 2	Bottom Group
2.5	Zone 2.5	Bottom Group
3b	Zone 3	Mid Group
3by_sub	Zone 3a	Mid Group
4	Zone 4	Top Group
5cx	Zone 5	Top Group
7	Zone 7b	Top Group
7a	Zone 7c	Top Group
7a_hw	Zone 7a hw	Top Group
7b_hw	Zone 7b hw	Top Group
7hw	Zone 7c hw	Top Group
7_sub	Zone 7a	Top Group
8	Zone 8c	Top Group
8a	Zone 8d	Top Group
8_sub	Zone 8b	Top Group
8_subA	Zone 8a	Top Group

11.4 Exploratory Data Analysis

11.4.1 Specific Gravity

SG database includes a total of 12,088 SG values, with 3,497 measured SG values and the rest are predicted and assigned values, ranging from 2.01 to 4.99 with average sample length of 2.33 m.

Prior to 2019, SG measurement at Arctic was inconsistent and the SG database contained many missing SG intervals, especially within the mineralized zones. In 2019, SRK provided a solution to fill the missing SG intervals by using the Random Forest Regressor to predict SG values in the mineralized zones. Outside of the mineralized zones, within non mineralized lithological units where SG data was not available, Ambler assigned an SG value of 2.78.

Since 2019, Ambler Metals collected SG values at a regular interval from both inside and outside of the mineralized zones and it was no longer necessary to use the Random Forest Regressor.

From 2019, SG measurements were taken using half cut core and taking dry and water submerged weights to calculate SG value. From 2011 to 2017, SG measurements were taken on whole core prior to cutting. Ambler Metals updated the existing SG database created by SRK with predicted SG values in 2021 with SG measurements collected since 2019.

SG was estimated in all mineralized zones and lithologies.

Table 11-5 shows summary statistics of SG by Minzone.

Wood reviewed 1,850 measured SG values with Random Forest predicted values in the SG database to assess for bias. Table 11-6 shows that there is no significant difference between the means of the measured and predicted SG values, however, the standard deviation of the predicted SG value is slightly smaller. Correlation coefficient of the measured and predicted SG values is 0.930.

Table 11-7 shows average measured and predicted SG values within selected bins of measured SG values. A low bias in the predicted SG values is observed and only exceed 5% bias for SG values below 2.7 and above 3.7. Table 11-5 shows that the proportion of blocks with estimated SG below 2.7 and above 3.7 is less than 1%. Small number of samples and number of blocks affected by this is not expected to have any material impact. A capping strategy was used during estimation process of SG values to control the impact of the extreme SG values. No other corrections in the predicted SG were applied for the current resource estimate.

Table 11-5: Summary Statistics for SG by Mineralized Zone

Minzone	Count	Mean	Min	Max
Zone 1	646	3.36	2.70	4.96
Zone 1a	36	3.12	2.75	3.32
Zone 2	145	3.29	2.74	4.63
Zone 2.5	89	3.24	2.75	4.39
Zone 3	762	3.24	2.50	4.46
Zone 3a	4	3.32	3.31	3.32
Zone 4	222	3.24	2.70	4.43
Zone 5	849	3.36	2.01	4.99
Zone 7a	12	2.93	2.71	3.32
Zone 7b	32	3.12	2.80	4.00
Zone 7c	8	3.03	2.73	3.32
Zone 7a HW	30	2.99	2.70	3.32
Zone 7b HW	21	2.97	2.72	3.32
Zone 7c HW	33	3.06	2.73	3.32
Zone 8a	4	3.32	3.32	3.32
Zone 8b	5	3.31	3.31	3.32
Zone 8c	44	3.06	2.72	3.32
Zone 8 HW	7	3.32	3.31	3.32

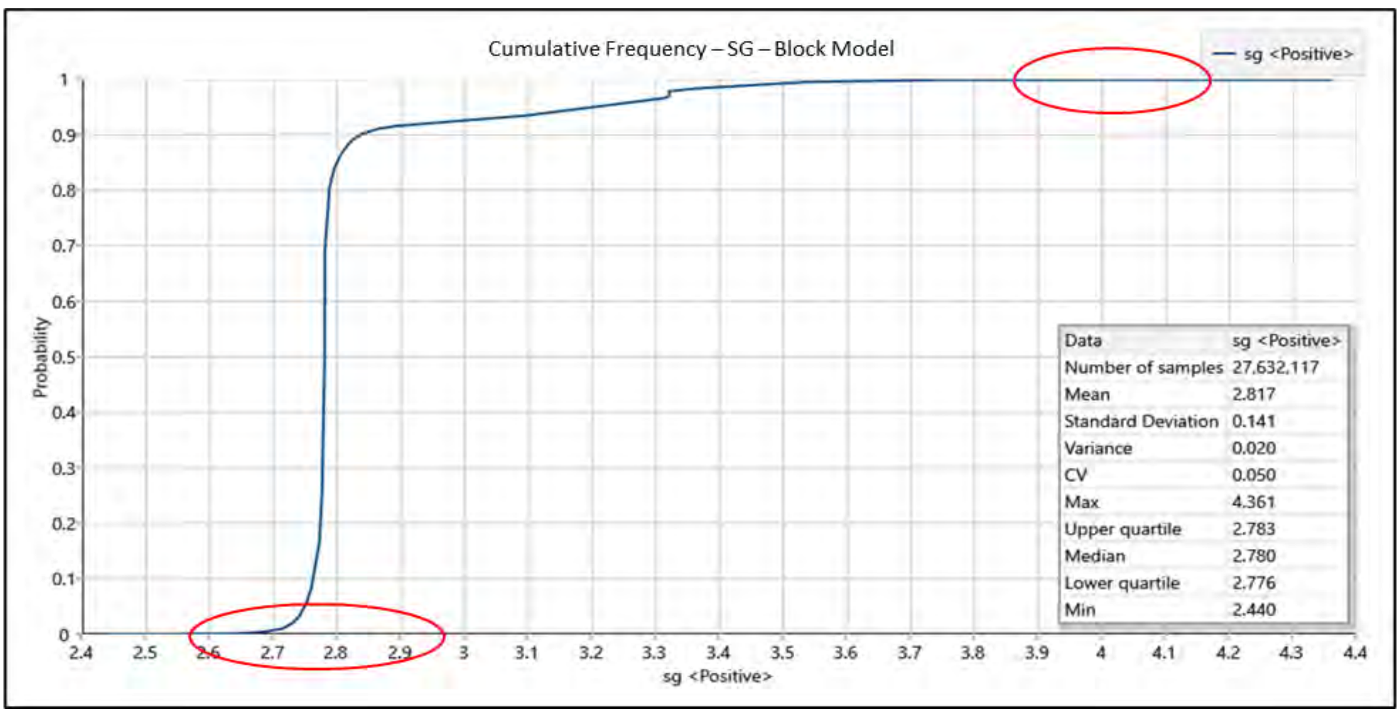
Table 11-6: Summary Statistics for SG Measured and SG Predicted

	Number of Data	Mean	Min	Max	Std. Dev
SG Measured	1,849	3.05	2.56	4.99	0.20
SG Predicted	1,849	3.06	2.65	4.38	0.16

Table 11-7: Mean SG Values for Measured and Predicted by Bins

Number of Data	Target SG Value	Mean		% Difference
		Measured SG	Predicted SG	
5	2.5	2.589	2.708	5%
754	2.7	2.746	2.788	2%
238	3	2.982	3.045	2%
118	3.2	3.2	3.249	2%
97	3.5	3.49	3.446	-1%
80	3.7	3.701	3.614	-2%
64	4	4.01	3.822	-5%
48	4.1	4.305	4.054	-6%
18	4.3	4.472	4.208	-6%
8	4.6	4.574	4.279	-6%

Figure 11-5: Cumulative Probability Plot for SG in the Resource Block Model



Source: Wood, 2022

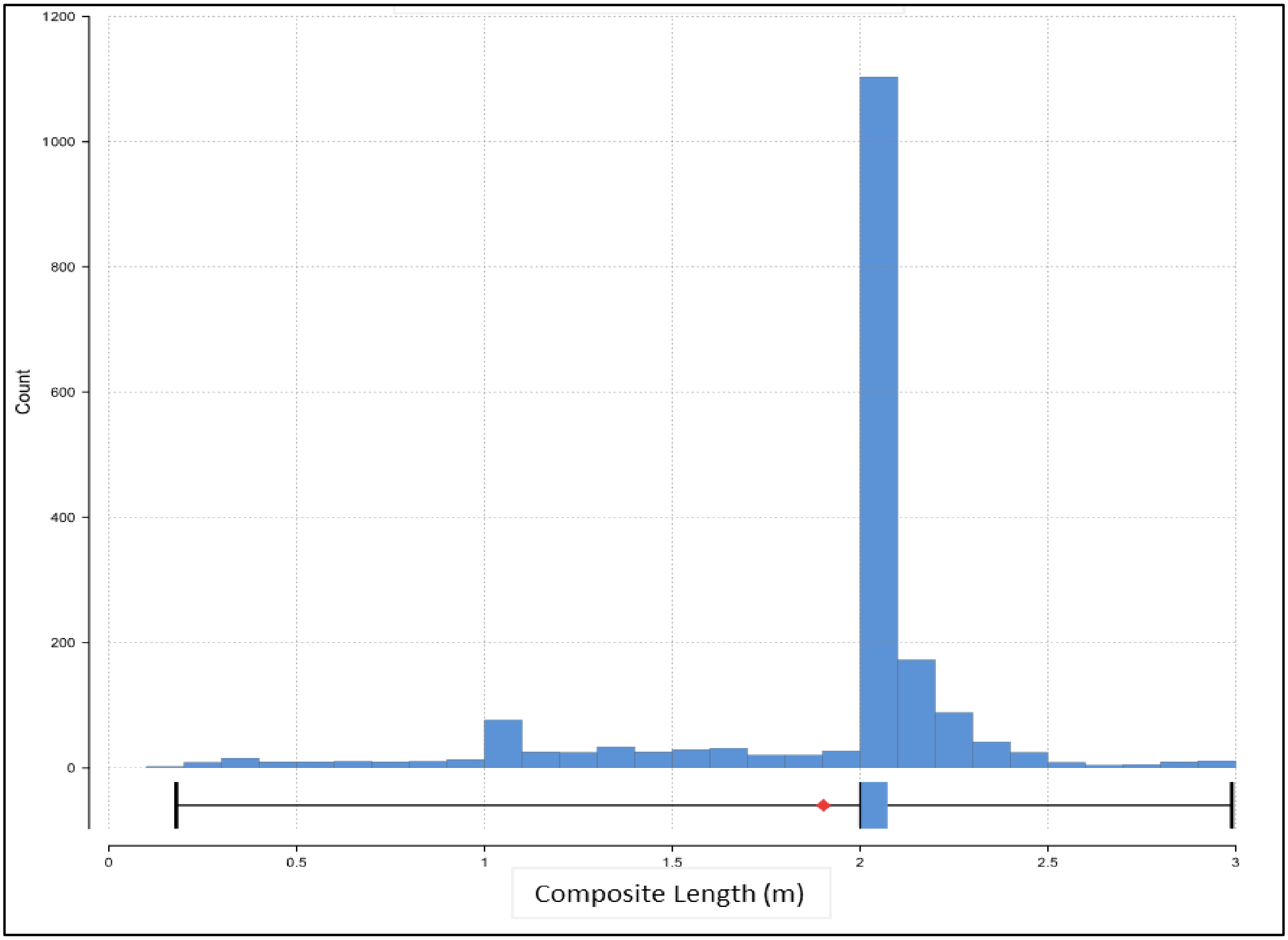
11.4.2 Compositing

The raw assay database was weighted using SG, meaning the weighted sample length equals sample length times SG times grade. Weighting composites by SG accounts for density having a positive correlation to grade in massive sulphide mineralization. The weighted assays were composited into 2-m intervals broken by lithology and mineralized zones. Residual composites that are less than 1 m were distributed equally into composites that are in the same domain.

Figure 11-6 shows a histogram of composite length, with 1.9 m as the average length of the composite.

Table 11-8 shows summary statistics of the composite database by mineralized zone.

Figure 11-6: Histogram of Composite Lengths



Source: Ambler, 2022.

Table 11-8: Summary Statistics of the Composite Database

Element	Unit	Count	Total Length (m)	Minimum	Maximum	Mean	Standard Deviation	Coefficient of Variation
Ag	g/t	1,747	3,328	0.0	578.9	46.5	47.71	1.03
As	ppm	1,123	2,152	2.1	10,000.0	1,032.7	1,614.66	1.56
Au	g/t	1,685	3,219	0.00	28.08	0.66	1.32	1.99
Cd	ppm	1,123	2,152	0.08	1,000.00	201.58	208.44	1.03
Cu	%	1,753	3,339	0.00	16.35	2.94	2.61	0.89
Fe	%	1,123	2,152	0.37	27.77	9.78	5.85	0.60
Hg	ppm	563	1,077	0.0	13.4	1.9	2.11	1.11
Pb	%	1,738	3,311	0.00	17.65	0.88	1.35	1.54
S	%	1,122	2,151	0.01	28.57	6.19	3.94	0.64
Sb	ppm	1,123	2,152	0.2	2,293.9	192.9	290.37	1.50
SG	-	1,860	3,540	2.51	4.60	3.30	0.34	0.10
Zn	%	1,749	3,333	0.00	24.10	3.97	4.14	1.04

11.4.3 Grade Capping

Probability plots of declustered data (by polygonal) and sensitivity curves were assessed in determining an appropriate capping value for all economic elements and specific gravity in each of the mineralized zones and lithological domains. The capping values used in the current model are the same as the values determined in the previous SRK model and are shown in Table 11-9. Summary statistics comparing mean values before and after the capping values are applied is shown in Table 11-10 for payable metals, and Table 11-11 for non-payable metals.

11.4.4 Spatial Analysis

Experimental and modelled variograms were generated using 2m capped composites to analyse the spatial continuity of mineralization within all mineralized zones. Mineralized zones define the resource estimation domains. Mineralized zones are combined into three groups (bottom, middle, and top) for the variogram analysis. Non-mineralized lithological units are combined into a single group, and the high-grade talc zones are also merged into a single group for the same purpose.

Table 11-12 summarizes the estimation domains. Figure 11-7 shows a cross-section of grouped estimation domains.

The composites for each combined mineralized zone were unfolded into 2D flattened space for experimental and modelling variograms and final variogram models are reported in GSLib format. Table 11-13 to Table 11-16 show modelled variogram model parameters used in the grade estimations.

Table 11-9: Capping Values

Minzone/Lithology	Capping Values											
	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	SG	As (ppm)	Sb (ppm)	Cd (ppm)	Hg (ppm)	Fe (%)	S (%)
Zone 1, 1a	8.0	3.00	14.00	2.5	150	4.5	7,000	1,000	800	7.0	25.0	10.0
Zone 2	6.0	2.80	11.00	2.5	110	4.5	4,000	1,000	600	4.0	20.0	10.0
Zone 2.5	6.0	2.50	8.00	0.6	90	4.0	1,000	700	400	5.0	20.0	10.0
Zone 3, SubZone 3	10.0	3.50	16.00	1.5	125	5.0	7,000	1,000	800	5.0	22.0	10.0
Zone 4	10.0	1.75	13.00	1.5	150	5.0	4,000	400	525	3.0	20.0	10.0
Zone 5	11.0	3.50	14.00	5.0	150	5.0	5,000	1,000	750	6.0	22.0	10.0
SubZone 7a, 7b, 7c, 7a HW, 7b HW, 7c HW	2.0	1.00	3.50	0.7	30	3.5	500	300	400	7.0	8.0	5.0
SubZone 8a, 8b, 8c, 8d, 8c HW	1.2	1.00	4.50	0.8	70	5.0	200	300	200	10.0	8.0	5.0
Aphanitic Meta-Rhyolite, Meta-Rhyolite Porphyry	0.1	0.10	0.20	0.2	9	3.1	200	30	15	0.3	7.5	9.5
Grey Schist	0.5	0.20	0.75	0.5	10	3.1	900	100	30	0.5	7.0	9.5
Quartz Mica Schist	0.8	0.50	0.75	0.5	15	3.1	1,000	100	75	2.0	10.0	9.5

Table 11-10: Summary Statistics of Uncapped and Capped Values for Payable Metal

Zone/ Lithology	Mean Cu (%)		Mean Pb (%)		Mean Zn (%)		Mean Au (ppm)		Mean Ag (ppm)	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Zone 1	2.80	2.76	0.83	0.74	4.12	4.03	0.76	0.73	53.3	51.1
Zone 1a	0.73	0.73	0.54	0.34	1.96	1.82	0.25	0.25	17.5	17.5
Zone 2	2.19	2.08	0.63	0.61	3.30	3.19	0.78	0.62	43.3	40.0
Zone 2.5	1.97	1.90	0.59	0.54	3.20	2.93	0.29	0.26	29.8	29.1
Zone 3	3.11	3.08	0.86	0.77	3.83	3.81	0.39	0.37	38.1	36.6
Zone 3a	1.38	1.38	1.06	1.06	6.46	6.46	0.00	0.00	24.7	24.7
zone 4	2.71	2.62	0.90	0.57	3.46	3.29	0.53	0.51	45.2	42.3
Zone 5	3.67	3.65	1.18	1.03	4.93	4.87	0.95	0.78	58.0	54.7
Zone 7a	0.22	0.22	0.07	0.07	0.20	0.20	0.22	0.20	5.2	5.2
Zone 7b	0.60	0.49	0.52	0.17	2.05	0.84	0.24	0.20	15.9	9.2
Zone 7c	0.62	0.62	0.04	0.04	0.44	0.44	0.20	0.20	5.6	5.6
Zone 7a HW	0.24	0.19	0.02	0.02	0.17	0.17	0.02	0.02	4.9	3.0
Zone 7b HW	0.09	0.09	0.02	0.02	0.21	0.21	0.03	0.03	1.4	1.4
Zone 7c HW	0.10	0.10	0.02	0.02	0.13	0.13	0.03	0.03	1.8	1.8
Zone 8a	0.58	0.58	0.21	0.21	2.90	2.90	0.60	0.60	14.2	14.2
Zone 8b	1.16	0.73	0.85	0.56	3.95	2.31	0.32	0.32	29.4	29.4
Zone 8c	0.36	0.23	0.22	0.20	1.00	0.83	0.26	0.19	7.9	7.9
Zone 8 HW	0.70	0.70	0.89	0.72	2.09	2.09	1.30	0.37	115.0	41.8

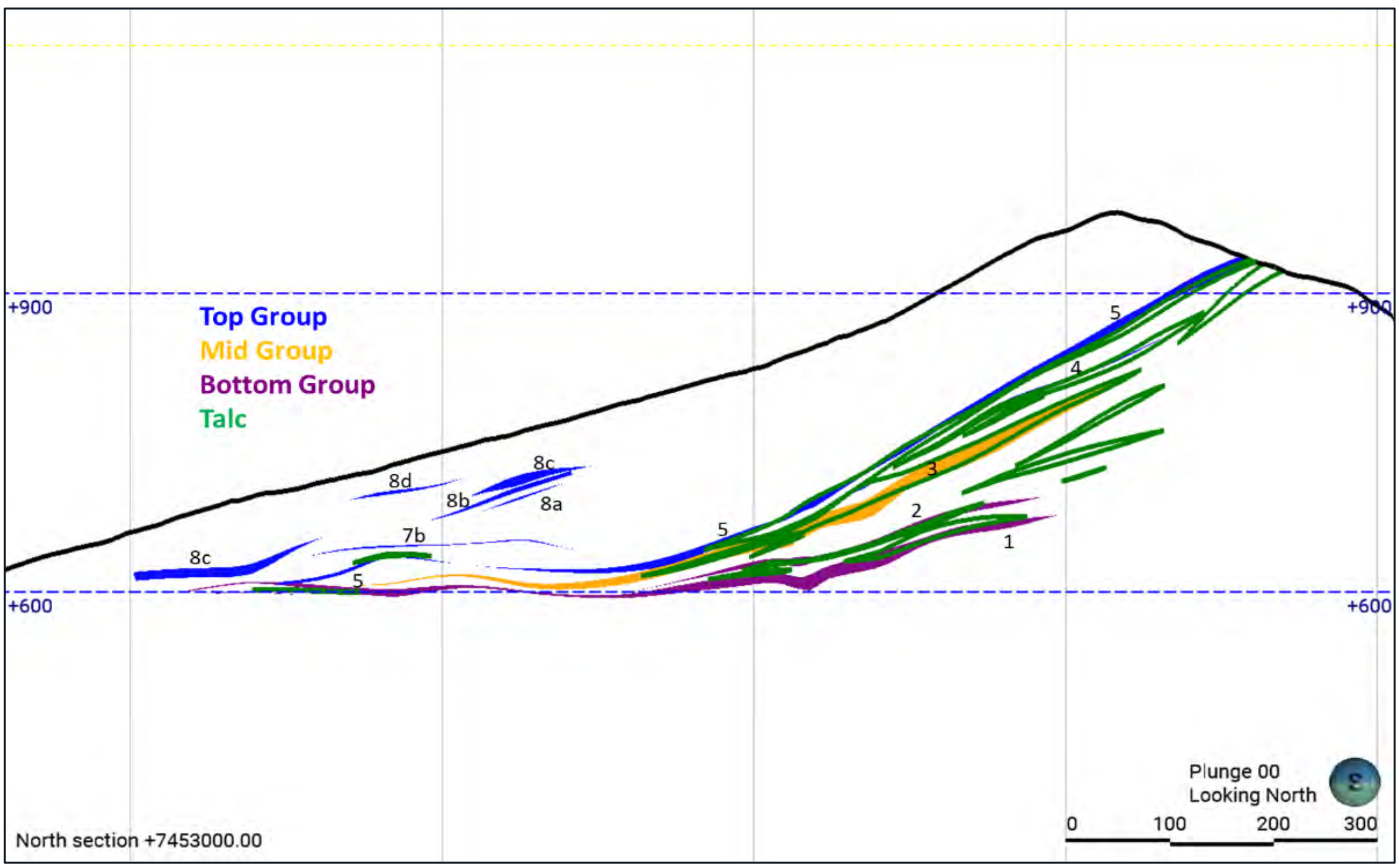
Table 11-11: Summary Statistics of Uncapped and Capped Values for Deleterious Elements

Zone/ Lithology	Mean As ppm		Mean Sb ppm		Mean Cd ppm		Mean Hg ppm		Mean Fe %		Mean S %	
	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped	Uncapped	Capped
Zone 1	1275.2	1251.7	276.8	264.7	192.91	191.03	2.3	2.2	11.29	11.25	6.58	6.50
Zone 1a	244.5	246.1	96.6	96.6	111.68	102.64	2.0	1.6	5.91	5.91	4.52	4.52
Zone 2	939.4	810.5	266.6	241.4	160.18	158.07	1.2	1.1	8.70	8.62	5.26	5.17
Zone 2.5	403.5	325.2	158.1	126.0	169.01	160.17	1.6	1.4	9.95	9.49	6.44	6.38
Zone 3	1074.6	1048.7	123.7	119.8	212.05	211.35	1.1	1.1	9.27	9.18	5.69	5.54
Zone 3a	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	0.00	0.00	0.00	0.00
zone 4	1169.2	912.9	103.4	88.1	179.17	173.39	1.4	1.2	8.19	8.16	4.96	4.68
Zone 5	1101.6	1061.3	222.4	213.6	261.36	258.50	2.5	2.5	10.93	10.82	7.53	6.84
Zone 7a	128.1	128.1	68.8	68.8	13.94	13.94	0.6	0.6	4.71	4.71	3.81	3.81
Zone 7b	214.2	196.1	105.0	70.8	117.17	68.43	3.8	2.6	6.06	5.04	4.54	3.93
Zone 7c	209.2	209.2	75.7	75.7	22.07	22.07	1.4	1.4	3.76	3.76	3.08	3.08
Zone 7a HW	26.6	26.6	11.0	11.0	7.40	7.40	0.9	0.9	2.80	2.77	0.80	0.68
Zone 7b HW	50.8	50.2	1.6	1.6	3.16	3.16	0.3	0.3	2.17	2.17	0.58	0.58
Zone 7c HW	64.9	64.9	20.5	20.5	7.23	7.23	0.8	0.7	2.67	2.67	0.74	0.70
Zone 8a	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	0.00	0.00	0.00	0.00
Zone 8b	42.3	42.3	19.2	19.2	40.00	40.00	2.2	2.2	3.58	3.58	1.22	1.22
Zone 8c	154.5	76.6	106.2	66.9	19.55	19.55	1.4	1.4	4.84	4.23	2.07	1.90
Zone 8 HW	0.0	0.0	0.0	0.0	0.00	0.00	0.0	0.0	0.00	0.00	0.00	0.00

Table 11-12: Estimation Domains

Estimation Domain	Minzones or Lithologies
Bottom Group	1, 1a, 2, and 2.5
Mid Group	3, and 3a
Top Group	4, 5, 7a, 7b, 8c, 8a, 8b, 8c, 8 hw, 7a hw, 7b hw, 7c hw
Talc	All high-grade talc zones
Lithology (Outside of Minzone)	All non-mineralized lithologies

Figure 11-7: Cross-Section of Estimation Domains and Talc



Source: Ambler Metals, 2022.

Table 11-13: Variogram Models for the Bottom Group

Bottom Group Variograms										
Variable	Variogram Model							GSLIB Angles		
	Nugget	Str. No.	Type	CC	X Range (m)	Y Range (m)	Z Range (m)	ANG1 (degree)	ANG2 (degree)	ANG3 (degree)
Cu	0.10	1	Exponential	0.50	30	70	7.5	30	0	0
		2	Spherical	0.40	20	200	8.0			
Pb	0.15	1	Exponential	0.50	30	50	7.5	30	0	0
		2	Spherical	0.35	220	180	10.0			
Zn	0.15	1	Exponential	0.55	40	60	9.0	30	0	0
		2	Spherical	0.30	240	200	9.0			
Ag	0.15	1	Exponential	0.45	30	75	6.0	30	0	0
		2	Spherical	0.43	240	180	12.0			
Au	0.15	1	Exponential	0.30	30	40	12.0	30	0	0
		2	Spherical	0.55	180	140	12.0			
SG	0.05	1	Exponential	0.60	30	60	9.0	30	0	0
		2	Spherical	0.35	250	20	12.0			
As	0.15	1	Exponential	0.40	40	60	9.0	30	0	0
		2	Spherical	0.45	26	200	12			
Cd	0.20	1	Exponential	0.65	55	55	9.0	30	0	0
		2	Spherical	0.15	200	200	9.0			
Fe	0.15	1	Exponential	0.50	40	55	10.5	30	0	0
		2	Spherical	0.35	250	200	10.5			
Hg	0.20	1	Exponential	0.65	75	75	9.0	30	0	0
		2	Spherical	0.15	200	200	12.0			
SG	0.10	1	Exponential	0.60	24	51	10.5	30	0	0
		2	Spherical	0.30	250	200	12.0			
Sb	0.15	1	Exponential	0.48	30	57	12.0	30	0	0
		2	Spherical	0.37	220	200	15.0			

Table 11-14: Variogram Models for the Middle Group

Middle Group Variograms										
Variable	Variogram Model							GSLIB Angles		
	Nugget	Str. No.	Type	CC	X Range (m)	Y Range (m)	Z Range (m)	ANG1 (degree)	ANG2 (degree)	ANG3 (degree)
Cu	0.15	1	Exponential	0.65	21	21	12.0	60	0	0
		2	Spherical	0.20	180	140	12.0			
Pb	0.15	1	Exponential	0.62	39	21	12.0	60	0	0
		2	Spherical	0.23	220	160	15.0			
Zn	0.15	1	Exponential	0.65	60	24	10.5	60	0	0
		2	Spherical	0.20	220	160	15.0			
Ag	0.15	1	Exponential	0.50	30	21	12.0	60	0	0
		2	Spherical	0.35	230	150	18.0			
Au	0.10	1	Exponential	0.45	30	21	15.0	60	0	0
		2	Spherical	0.45	240	160	20.0			
SG	0.05	1	Exponential	0.20	20	20	12.0	60	0	0
		2	Spherical	0.75	240	160	20.0			
As	0.20	1	Exponential	0.30	75	50	12.0	60	0	0
		2	Spherical	0.50	240	160	20.0			
Cd	0.15	1	Exponential	0.50	24	49	7.5	60	0	0
		2	Spherical	0.35	220	140	9.0			
Fe	0.10	1	Exponential	0.30	39	21	9.0	60	0	0
		2	Spherical	0.60	240	160	20.0			
Hg	0.15	1	Exponential	0.50	39	21	7.5	60	0	0
		2	Spherical	0.35	240	160	8.0			
SG	0.10	1	Exponential	0.15	39	21	6.0	60	0	0
		2	Spherical	0.75	22	160	12.0			
Sb	0.05	1	Exponential	0.20	39	21	12.0	60	0	0
		2	Spherical	0.75	240	180	18.0			

Table 11-15: Variogram Models for the Top Group

Top Group Variograms										
Variable	Variogram Model							GSLIB Angles		
	Nugget	Str. No.	Type	CC	X Range (m)	Y Range (m)	Z Range (m)	ANG1 (degree)	ANG2 (degree)	ANG3 (degree)
Cu	0.05	1	Exponential	0.60	45	33	6.0	60	0	0
		2	Spherical	0.35	250	160	9.0			
Pb	0.05	1	Exponential	0.55	45	30	4.5	60	0	0
		2	Spherical	0.40	240	180	9.0			
Zn	0.20	1	Exponential	0.60	45	33	6.0	60	0	0
		2	Spherical	0.20	250	160	9.0			
Ag	0.10	1	Exponential	0.50	45	33	6.0	60	0	0
		2	Spherical	0.40	250	180	9.0			
Au	0.30	1	Exponential	0.50	36	36	6.0	60	0	0
		2	Spherical	0.20	250	200	12.0			
SG	0.00	1	Exponential	0.50	45	45	6.0	60	0	0
		2	Spherical	0.50	220	160	13.0			
As	0.15	1	Exponential	0.24	45	33	6.0	60	0	0
		2	Spherical	0.61	250	160	12.0			
Cd	0.15	1	Exponential	0.30	45	33	4.5	60	0	0
		2	Spherical	0.55	250	160	12.0			
Fe	0.20	1	Exponential	0.15	36	30	6.0	60	0	0
		2	Spherical	0.65	240	160	12.0			
Hg	0.10	1	Exponential	0.70	105	105	6.0	60	0	0
		2	Spherical	0.20	250	250	9.0			
SG	0.15	1	Exponential	0.35	45	33	7.5	60	0	0
		2	Spherical	0.50	250	160	12.0			
Sb	0.05	1	Exponential	0.65	45	33	7.5	60	0	0
		2	Spherical	0.30	250	160	12.0			

Table 11-16: Variogram Model for Talc

Talc Variograms										
Variable	Variogram Model							GSLIB Angles		
	Nugget	Str. No.	Type	CC	X Range (m)	Y Range (m)	Z Range (m)	ANG1 (degree)	ANG2 (degree)	ANG3 (degree)
Talc	0.10	1	Exponentia	0.55	75	50	10	220	-23	0
		2	Spherical	0.35	150	80	30			

11.4.5 Block Model Setup

The non-rotated block model was set up in UTM coordinate system, Zone 4, NAD 83 datum. The block model has a parent block size of 10 x 10 x 5 m with a sub-block size of 2 x 2 x 1 m. Table 11-17 shows the Arctic block model parameters.

The original sub-block model was regularized to 5 x 5 x 5 m for the Reserve estimation.

Table 11-17: Arctic Block Model Parameters

Block Size (m)			Model Origin UTM	Boundary Size (m)	No. of Parent Blocks
Axis	Parent	Sub	Bottom Left Corner		
X	10	2	612,190	1,910	191
Y	10	2	7,452,095	1,950	195
Z	5	1	345	705	141
Rotation	Dip (degree)	0	-		
	Azi. (degree)	0			

11.5 Mineral Resource Estimation

Inside mineralized zones, payable metals (Cu, Zn, Pb, Ag and Au), deleterious elements, and SG were estimated using ordinary kriging (OK) in three passes with increasing search ellipsoid sizes. Talc was also estimated in the high-grade talc zones using OK with three passes. Outside of mineralized zones, payable metals, deleterious elements, and SG were estimated using inverse distance to the power of two (ID2) with three passes. Sub-blocks receive the estimated grade values of parent blocks. Each mineralized zone is treated as hard boundaries, meaning that the data from one mineralized zone was not used to estimate grades in other zones. Table 11-18 to Table 11-21 show search ellipsoid sizes, and minimum and maximum number of composites used for the block grade estimations.

Table 11-18: Search Ellipsoid Parameters Used for Payable Metals (Cu, Zn, Ag, and Au)

Search Parameters Payable Metals											
Minzone/ Lithology	Method	Search Distances				Composites			Octants		
		Pass	Y (m)	X (m)	Z (m)	Min	Max	Max Per DDH*	Min	Min Comps	Max Comps
Bottom Group	Ordinary Kriging with LVA**	1	125	100	6	3	12	2	2	1	2
		2	250	200	12	3	16	2	2	1	2
		3	750	600	36	1	21	2	-	-	-
Mid Group	Ordinary Kriging with LVA	1	130	80	9	3	12	2	2	1	2
		2	260	160	18	3	16	2	2	1	2
		3	780	480	54	1	21	2	-	-	-
Top Group	Ordinary Kriging with LVA	1	130	90	6	3	12	2	2	1	2
		2	260	180	12	3	16	2	2	1	2
		3	780	540	36	1	21	2	-	-	-
Lithologies	Inverse Distance squared with LVA	1	125	90	7	3	12	2	2	1	2
		2	250	180	14	2	16	2	2	1	2
		3	750	540	42	1	21	2	-	-	-
Talc	Ordinary Kriging with LVA	1	200	100	35	3	12	2	-	-	-
		2	400	200	70	3	16	2	-	-	-
		3	4,000	2,000	700	1	21	2	-	-	-

*DDH: Diamond Drill Hole
**LA: Locally Varying Anisotropy

Table 11-19: Search Ellipsoid Parameters Used for Deleterious Elements

Search Parameters Deleterious Elements												
Minzone/ Lithology	Method	Search Distances				Composites			Capped	Octants		
		Pass	Y (m)	X (m)	Z (m)	Min	Max	Max Per DDH		Min	Min Comp	Max Comp
Bottom Group	Ordinary Kriging with LVA	1	45	60	6	3	12	2	No	-	-	-
		2	90	120	12	2	16	2	No	-	-	-
		3	780	600	36	1	21	2	Yes	-	-	-
Mid Group	Ordinary Kriging with LVA	1	45	30	9	3	12	2	No	-	-	-
		2	90	60	18	2	16	2	No	-	-	-
		3	780	480	54	1	21	2	Yes	-	-	-
Top Group	Ordinary Kriging with LVA	1	60	45	6	3	12	2	No	-	-	-
		2	120	90	12	2	16	2	No	-	-	-
		3	780	510	36	1	21	2	Yes	-	-	-
Lithologies	Inverse Distance squared with LVA	1	60	45	6	3	12	2	No	-	-	-
		2	120	90	12	2	16	2	No	-	-	-
		3	780	510	36	1	21	2	Yes	-	-	-

Table 11-20: Search Ellipsoid Parameters Used for Talc

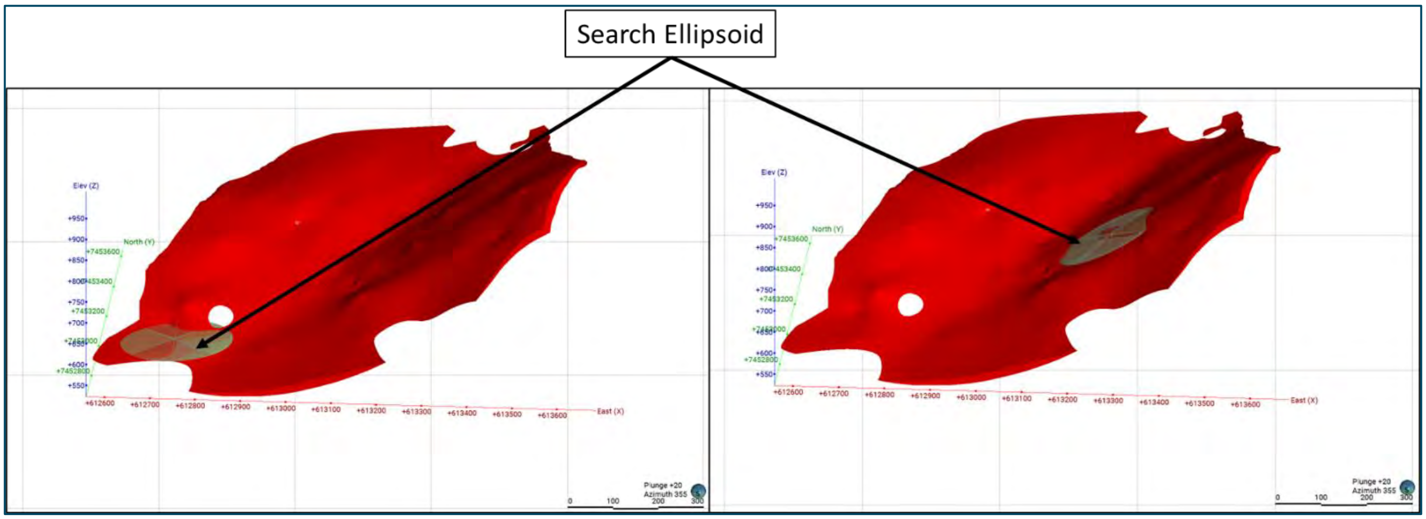
Search Parameters Talc											
Minzone/ Lithology	Method	Search Distances				Composites			Octants		
		Pass	Y (m)	X (m)	Z (m)	Min	Max	Max Per DDH	Min	Min Comp	Max Comp
Talc	Ordinary Kriging with LVA	1	200	100	35	3	12	2	-	-	-
		2	400	200	70	3	16	2	-	-	-
		3	4,000	2,000	700	1	21	2	-	-	-

Table 11-21: Search Ellipsoid Parameters Used for SG

Search Parameters Specific Gravity											
Minzone/ Lithology	Method	Search Distances				Composites			Octants		
		Pass	Y (m)	X (m)	Z (m)	Min	Max	Max Per DDH	Min	Min Comp	Max Comp
Bottom Group	Ordinary Kriging with LVA	1	130	100	6	2	12	-	-	-	-
		2	260	200	12	1	15	-	-	-	-
		3	780	600	36	1	15	-	-	-	-
Mid Group	Ordinary Kriging with LVA	1	130	80	9	2	12	-	-	-	-
		2	260	160	18	1	15	-	-	-	-
		3	780	480	54	1	15	-	-	-	-
Top Group	Ordinary Kriging with LVA	1	115	85	6	2	12	-	-	-	-
		2	230	170	12	1	15	-	-	-	-
		3	690	510	36	1	15	-	-	-	-
Lithologies	Inverse Distance squared with LVA	1	125	90	7	2	12	-	-	-	-
		2	250	180	14	1	15	-	-	-	-
		3	750	540	42	1	15	-	-	-	-

Locally Varying Anisotropy (LVA) was used to orient search ellipsoids at each block centroid location. The LVA parameters were calculated using each mineralized zone top and bottom surfaces. Table 11-10 shows search ellipsoids, at two different block locations, oriented by LVA parameters at each location following the orientation of the mineralized Zone 5 wireframes used for Cu estimation within the mineralized Zone 5.

Figure 11-8: LVA Search Ellipsoids Used in Minzone 5 (Red, Top Group) for Cu Estimation



Source: Ambler Metals, 2022.

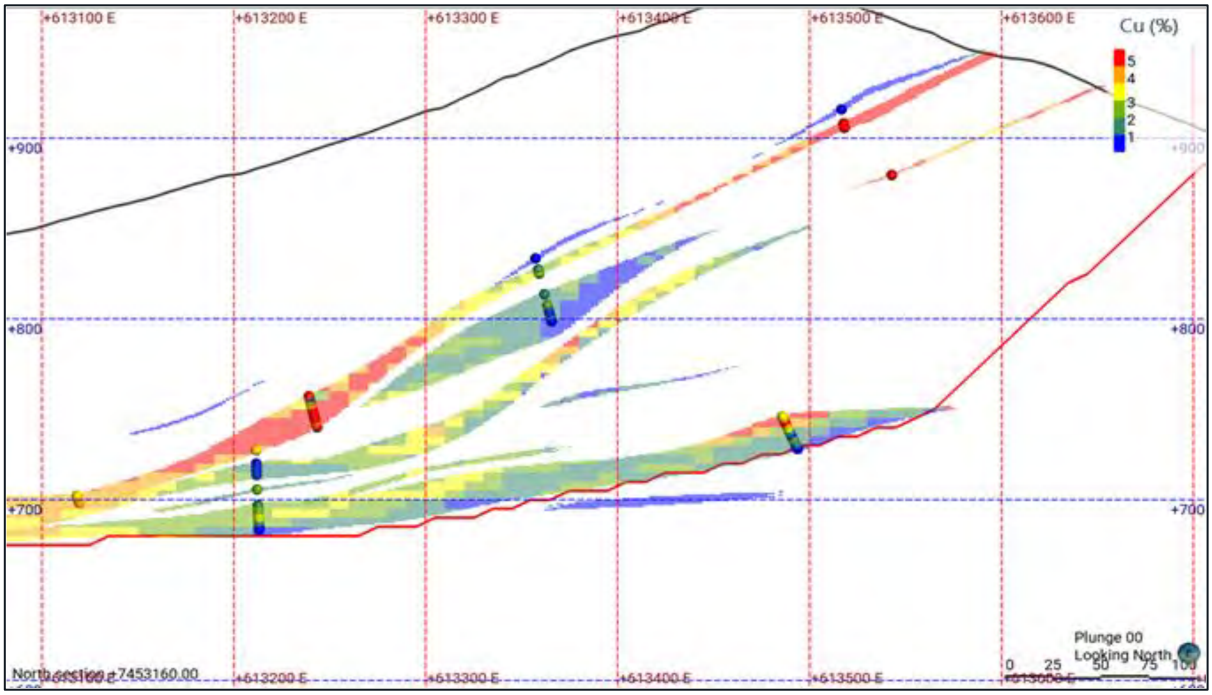
11.5.1 Validation

11.5.1.1 Visual Inspection

A detailed visual inspection of the block model was conducted in both section and plan to compare estimated grades against underlying sample data. This included confirmation of the proper coding of blocks within the respective domains.

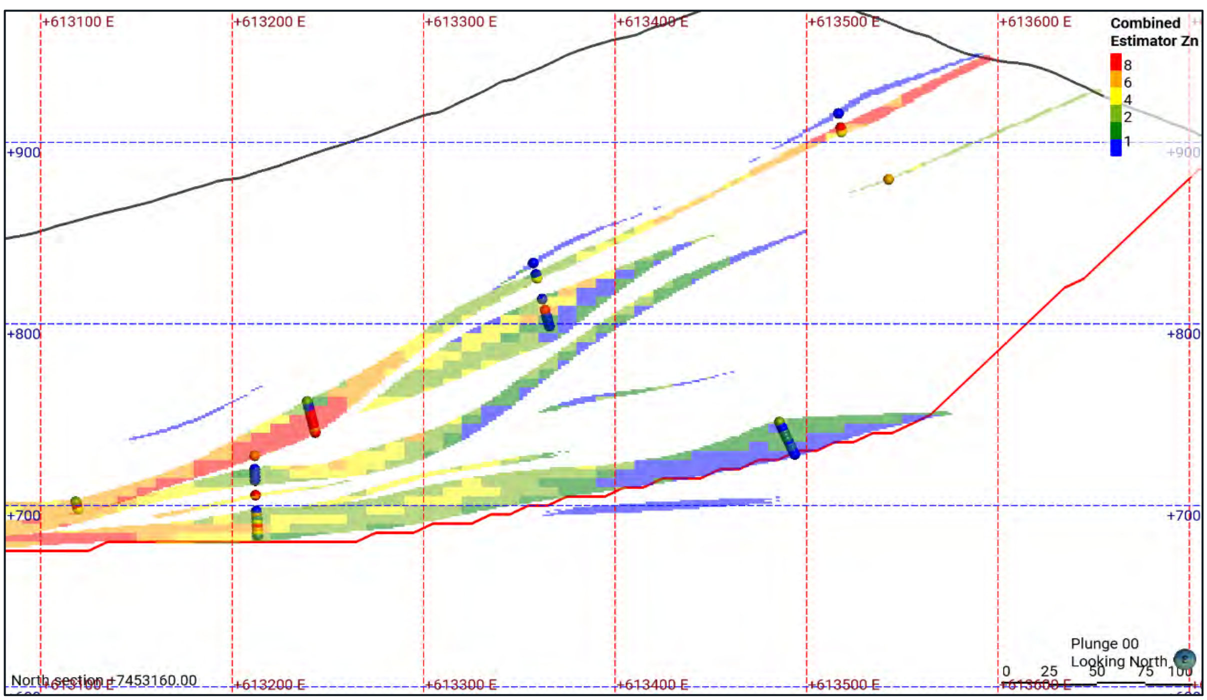
Figure 11-9 to Figure 11-13 show East-West cross-section maps for the payable metals block grade and composite database used for the estimation. In general, globally, block estimates compared well with composites.

Figure 11-9: East-West Cross-Section for Cu Block Model and Composite Data, Looking North (Resource pit depicted in red)



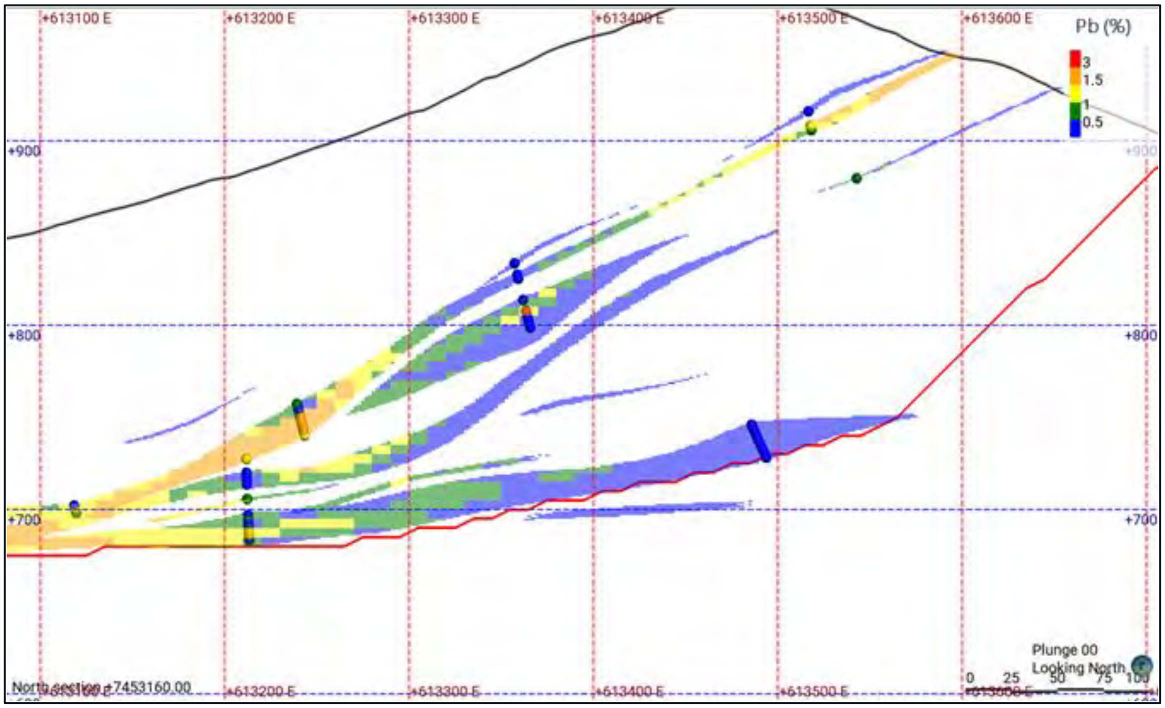
Source: Ambler Metals, 2022.

Figure 11-10: East-West Cross-Section for Zn Block Model and Composite Data, Looking North (Resource pit depicted in red)



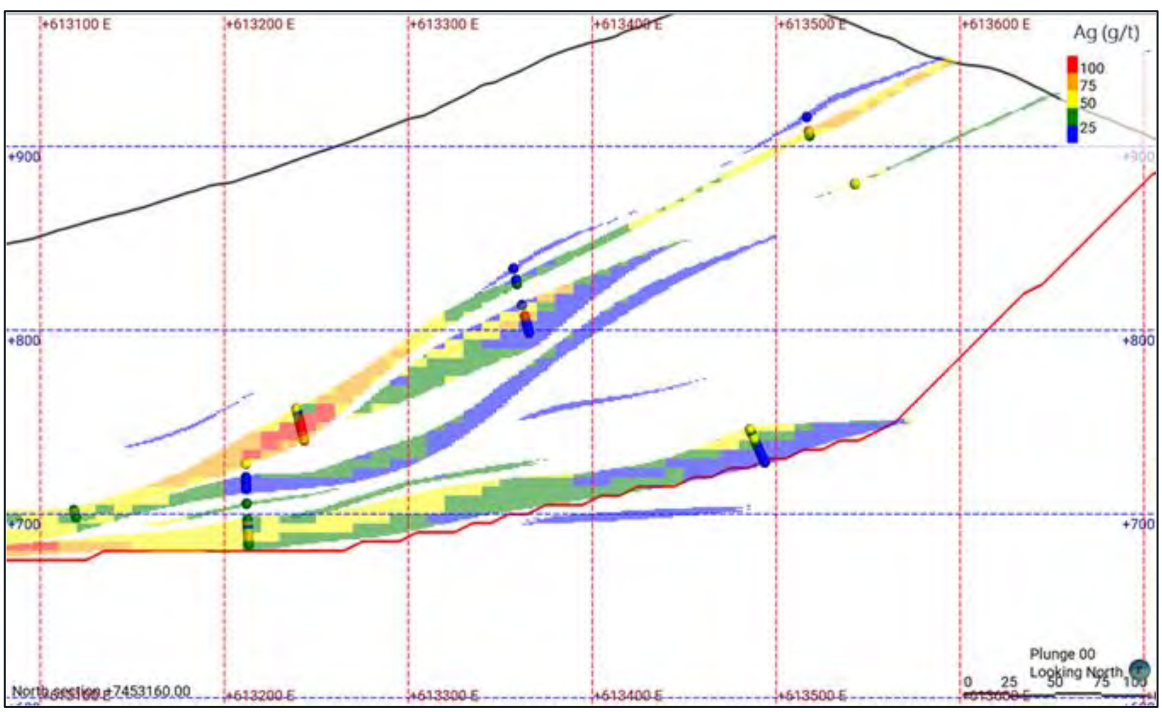
Source: Ambler Metals, 2022.

Figure 11-11: East-West Cross-Section for Pb Block Model and Composite Data, Looking North (Resource pit depicted in red)



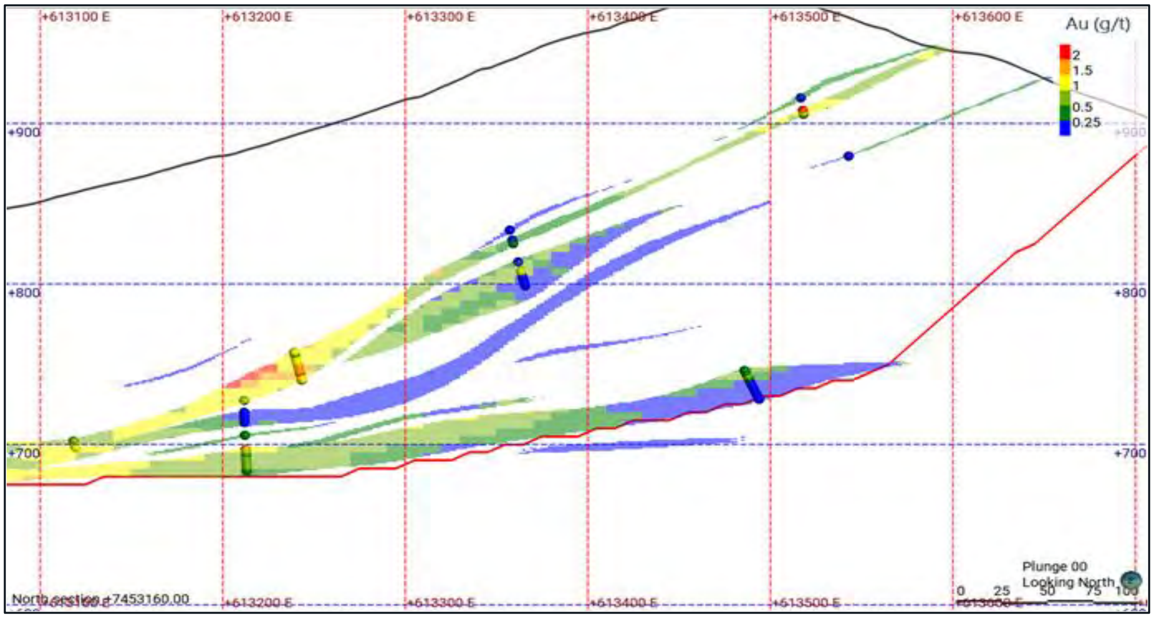
Source: Ambler Metals, 2022.

Figure 11-12: East-West Cross-Section for Ag Block Model and Composite Data, Looking North (Resource pit depicted in red)



Source: Ambler Metals, 2022.

Figure 11-13: East-West Cross-Section for Au Block Model and Composite Data, Looking North (Resource pit depicted in red)



Source: Ambler Metals, 2022.

11.5.1.2 Statistical Validation

Comparing the basic statistics of the OK block model with the Nearest Neighbour (NN) model help quantify whether the estimate is over or underestimated. In general, Wood considers differences of less than five percent between the two models mean grades is reasonable for validating global grade bias.

Summary statistics of payable metal mean grades for the main mineralized zones (Zones 1, 3, and 5) are shown in Table 11-22. Globally, the models validate well as OK estimated block model mean grades and declustered composites (the nearest neighbours model) means are close except for some smaller domains. These smaller domains were estimated with a small number of samples, which needs further investigation. In other minor mineralized domains, some domains show larger differences due to the lack of supporting drill hole data.

11.5.1.3 Swath Plots

Swath plots are used to detect any spatial-geographical bias in an estimated model. Plots compare average grades of the NN and estimated models over swaths that are typically one block wide in all directions.

The estimated models (ID2 and OK) and NN estimates swaths should follow the same trend with the NN plot typically experiencing higher highs and lower lows than the smoother estimated models. In areas where there are large discrepancies or deviations from each other, the models should be further investigated to determine the reason and modify as required.

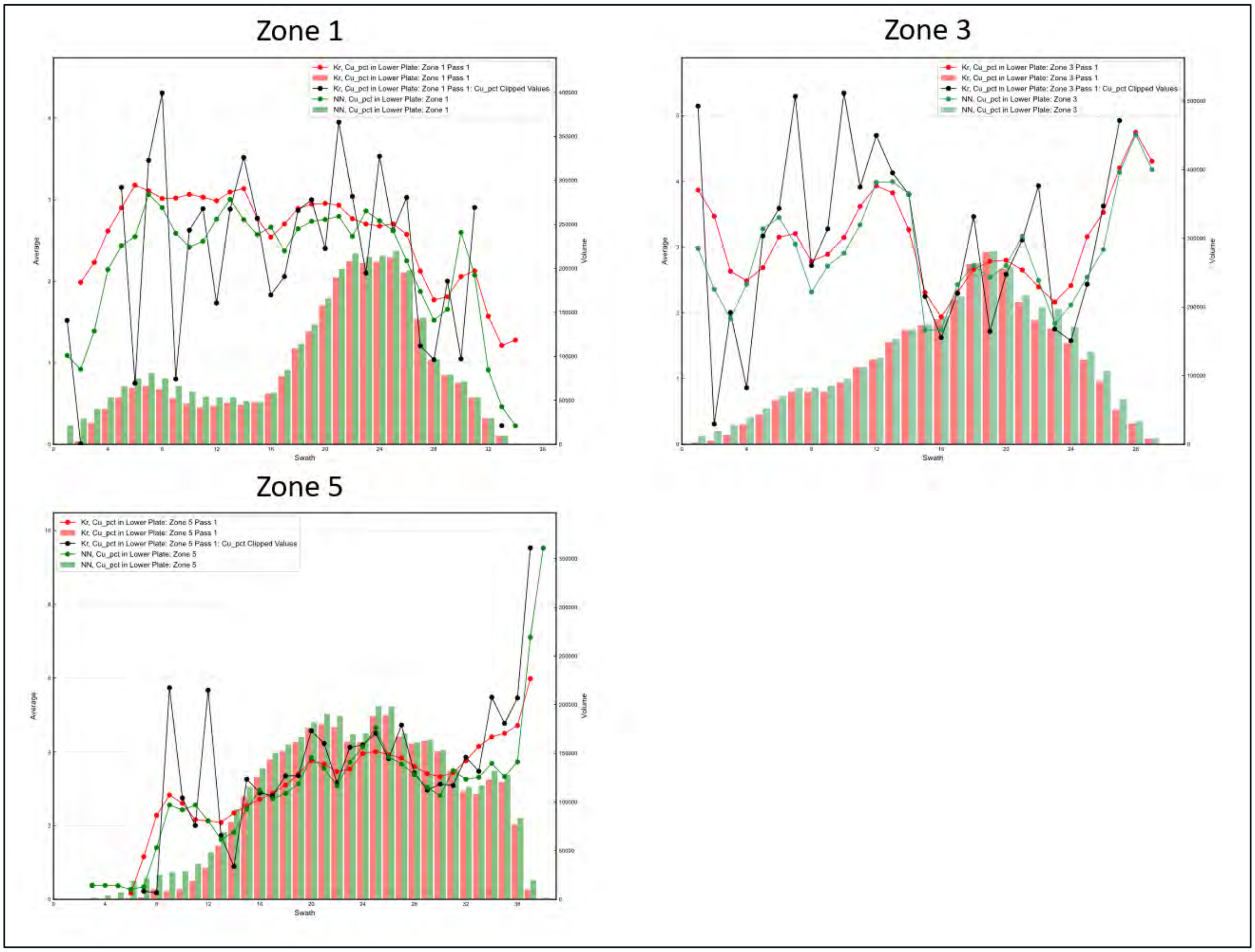
There is good correspondence between the models in most areas. The validation results indicate that the OK model is a reasonable reflection of the underlying sample data.

Figure 11-14 to Figure 11-18 show swath plots for the payable metals for the main mineralized zones (zone 1, 3 and 5).

Table 11-22: Statistics of OK and NN for Payable Metals for the Main Mineralized Zones

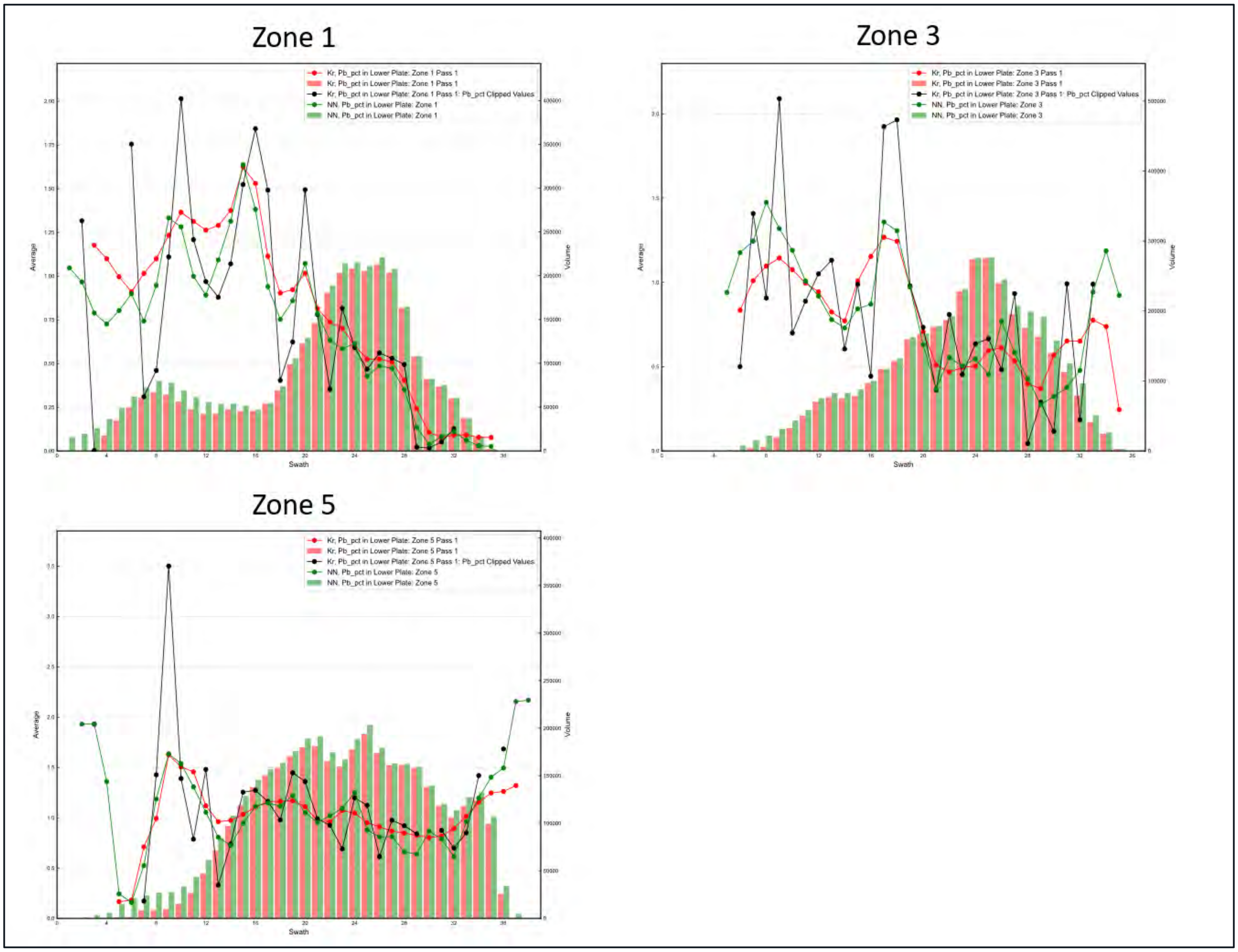
Minzone	No. of Blocks	Mean Cu (%)			Mean Pb (%)			Mean Zn (%)			Mean Au (g/t)			Mean Ag (g/t)		
		OK	NN	% Diff	OK	NN	% Diff	OK	NN	% Diff	OK	NN	% Diff	OK	NN	% Diff
Zone 1	693,631	2.587	2.448	-5	0.736	0.694	-6	3.860	3.636	-6	0.706	0.661	-6	50.022	47.206	-6
Zone 3	870,408	2.801	2.759	-2	0.684	0.657	-4	3.565	3.494	-2	0.314	0.323	3	33.120	33.283	0
Zone 5	930,886	3.451	3.287	-5	1.016	0.982	-3	4.808	4.394	-9	0.723	0.709	-2	51.867	50.029	-4

Figure 11-14: Cu Swath Plot, OK (Red), NN (Green) and Composite (Black), North-South Direction



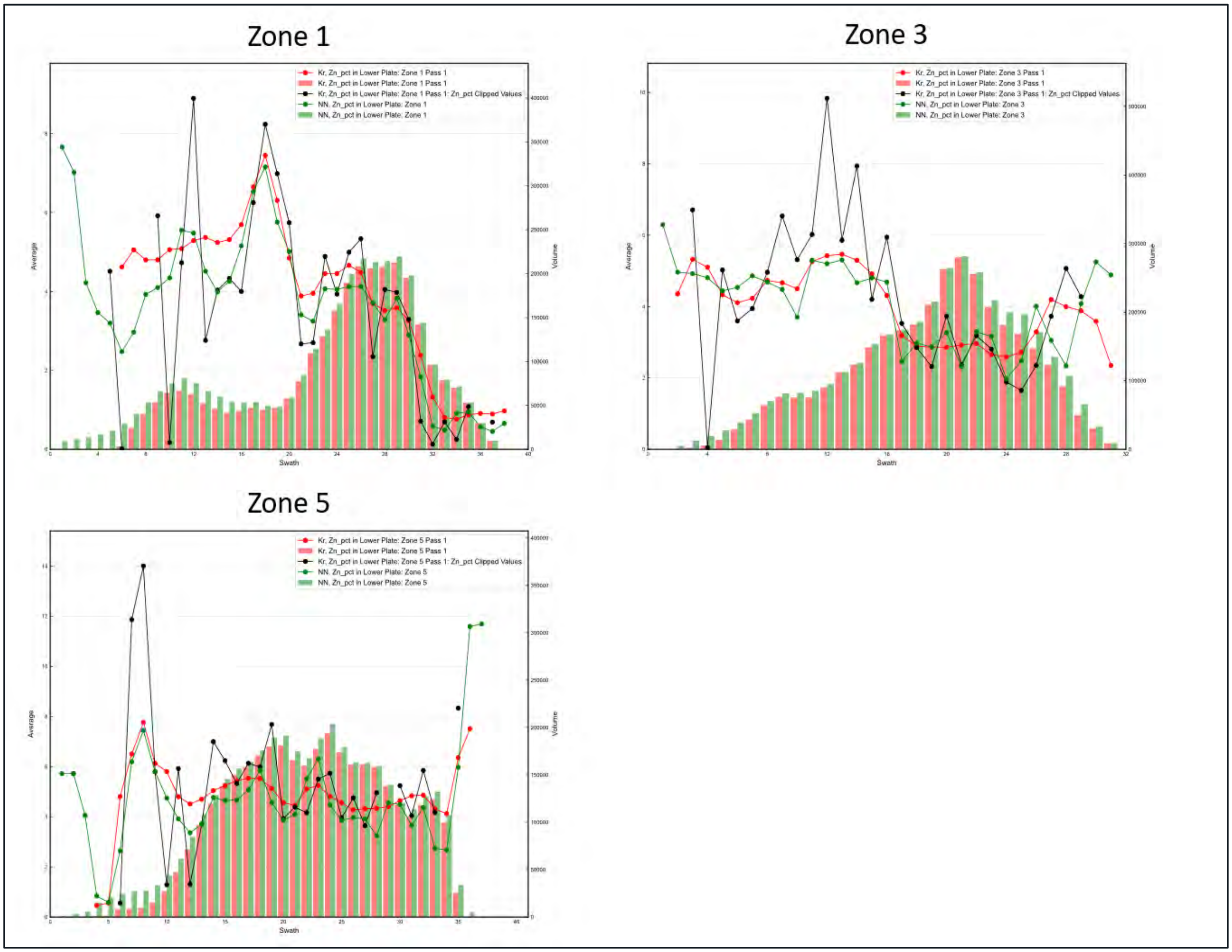
Source: Ambler Metals, 2022.

Figure 11-15: Pb Swath Plot, OK (Red), NN (Green) and Composite (Black), North-South Direction



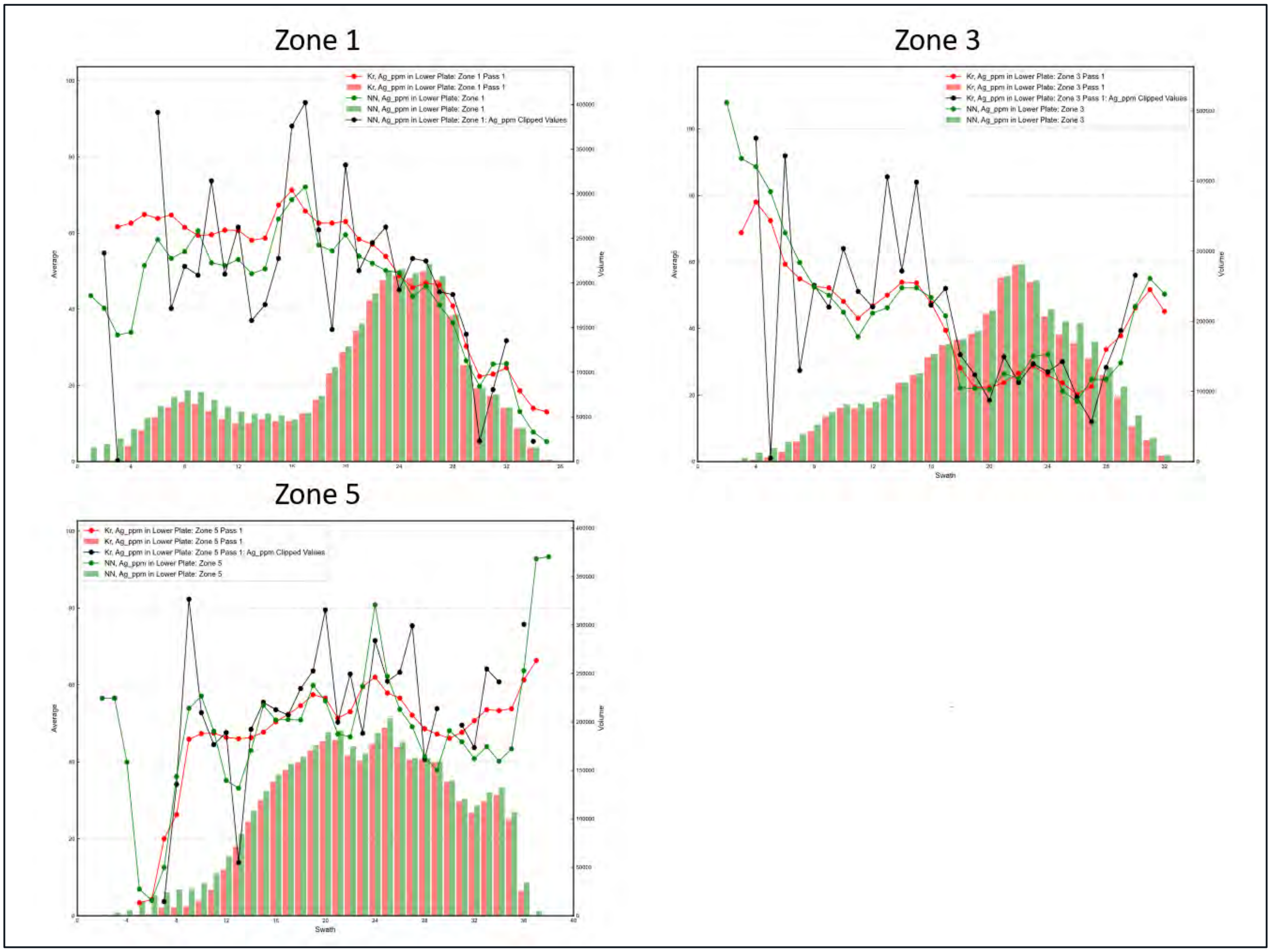
Source: Ambler Metals, 2022.

Figure 11-16: Zn Swath Plot, OK (Red), NN (Green) and Composite (Black), North-South Direction



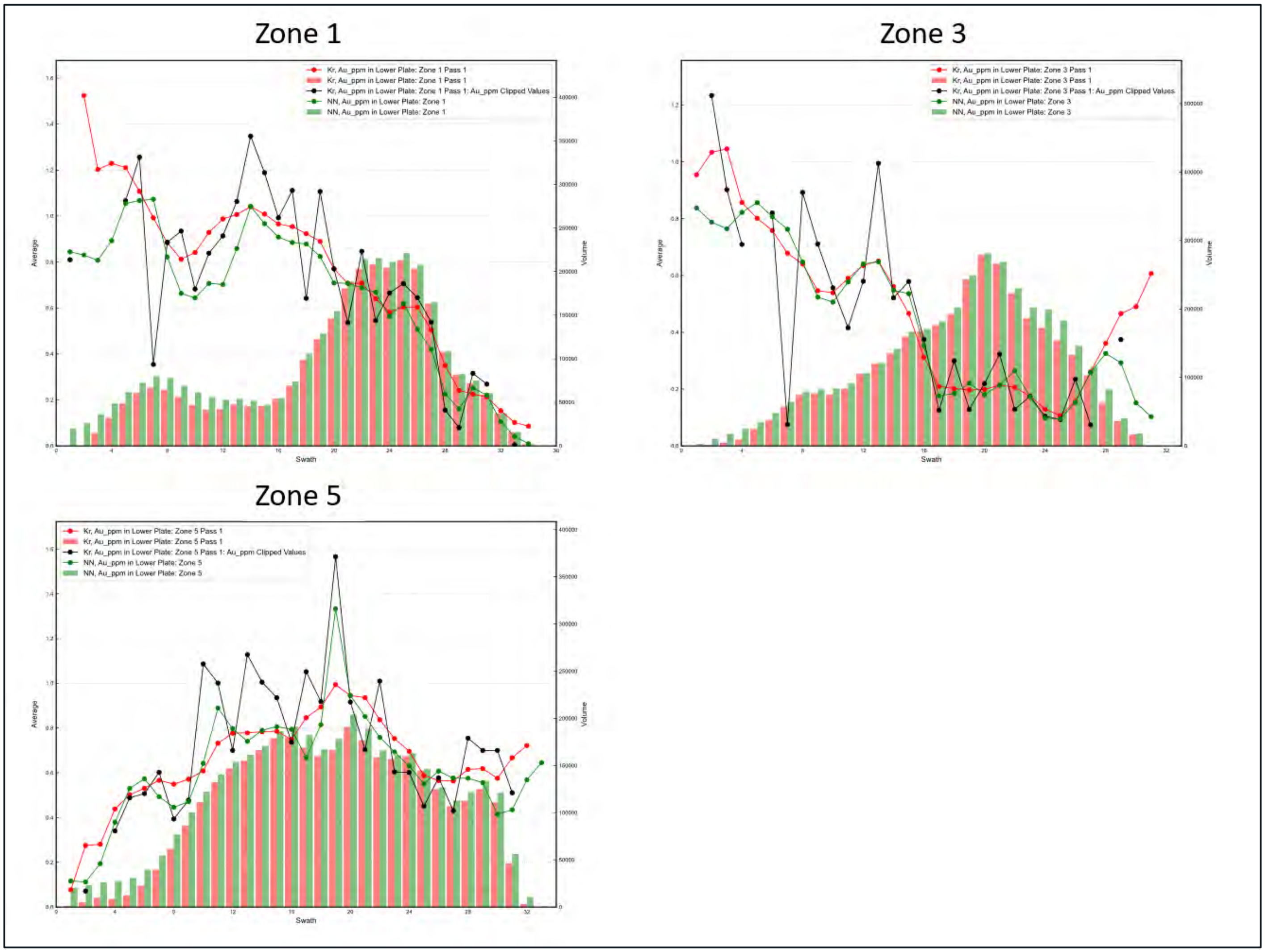
Source: Ambler Metals, 2022.

Figure 11-17: Ag Swath Plot, OK (Red), NN (Green) and Composite (Black), North-South Direction



Source: Ambler Metals, 2022.

Figure 11-18: Au Swath Plot, OK (Red), NN (Green) and Composite (Black), North-South Direction



Source: Ambler Metals, 2022.

11.5.1.4 Validation of Block Model with the New Drill Holes

Wood reviewed the new drilling data collected after the closing of the database for the current mineral resource model by displaying copper grades for both the drill holes and the current block model in East-West cross sections.

Copper grades in the new drill holes match well with the current block model copper grades visually and confirm mineralization zones defined.

11.5.2 Mineral Resource Classification

The Mineral Resources were classified in accordance with the standards and definitions of S-K 1300. The classification parameters are defined relative to the distance between sample data and are intended to encompass zones of reasonably continuous mineralization that exhibit the desired degree of confidence in the estimate.

Classification parameters are generally linked to the scale of a deposit: a relatively narrow, high-grade deposit would likely be mined using selective mining methods, at a much lower daily rate than a large, low-grade, bulk mineable deposit. The scale and selectivity of these two examples differs significantly and is reflected in the drill-hole spacing required to achieve the desired level of confidence to define a volume of material that represents, for example, a year of production. Based on engineering studies completed to date, the Arctic deposit is amenable to open pit extraction methods at a production rate of approximately 10,000 t/d. A drill hole spacing study, which tests the reliability of estimates for a given volume of material at varying drill hole spacing, suggests that drilling on a nominal 100 m grid pattern would provide annual estimates of sufficient confidence to classify them in the Indicated category. These results were combined with grade and indicator variograms and other visual observations of the nature of the deposit in defining the criteria for mineral resource classification as described below. At this stage of the Project, there is insufficient density of drilling information to support the definition of Mineral Resources in the Measured category. Manual “smoothing” of the Resource Classification was not performed.

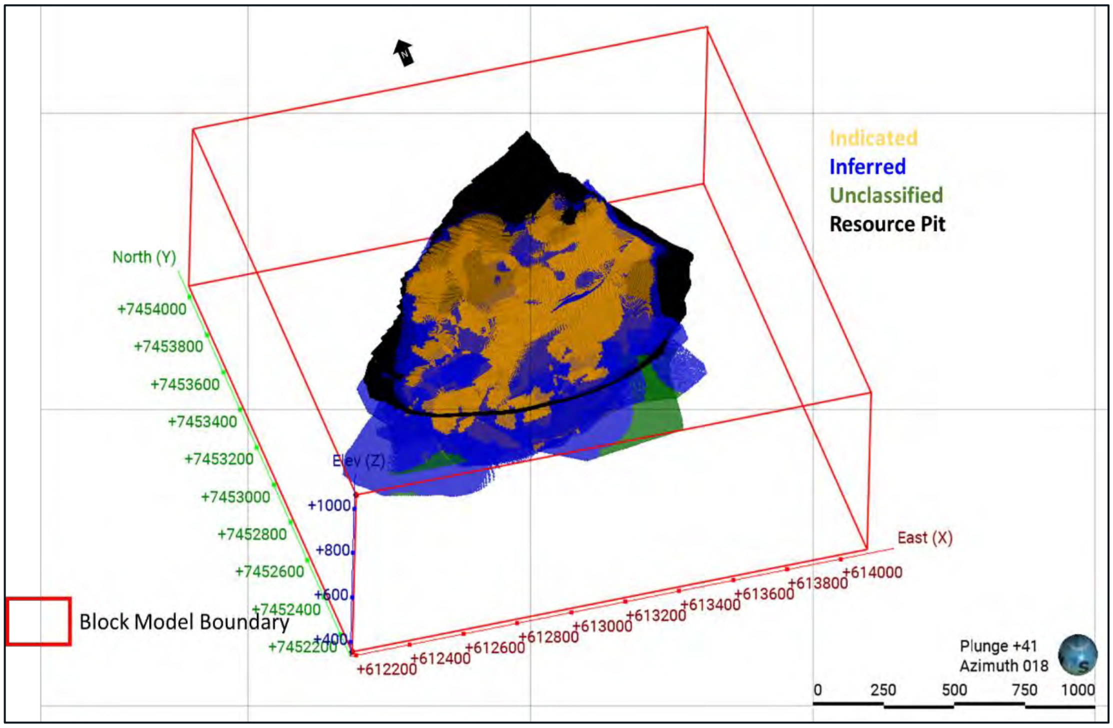
The Wood QP expects the majority of Inferred Mineral could be upgraded to Indicated Mineral Resources with additional exploration.

Table 11-23 shows the Mineral Resource classification criteria used in the resource model. Figure 11-19 and Figure 11-20 show plan view and cross-section map of the Mineral Resource Classification model.

Table 11-23: Mineral Resource Classification and Criteria

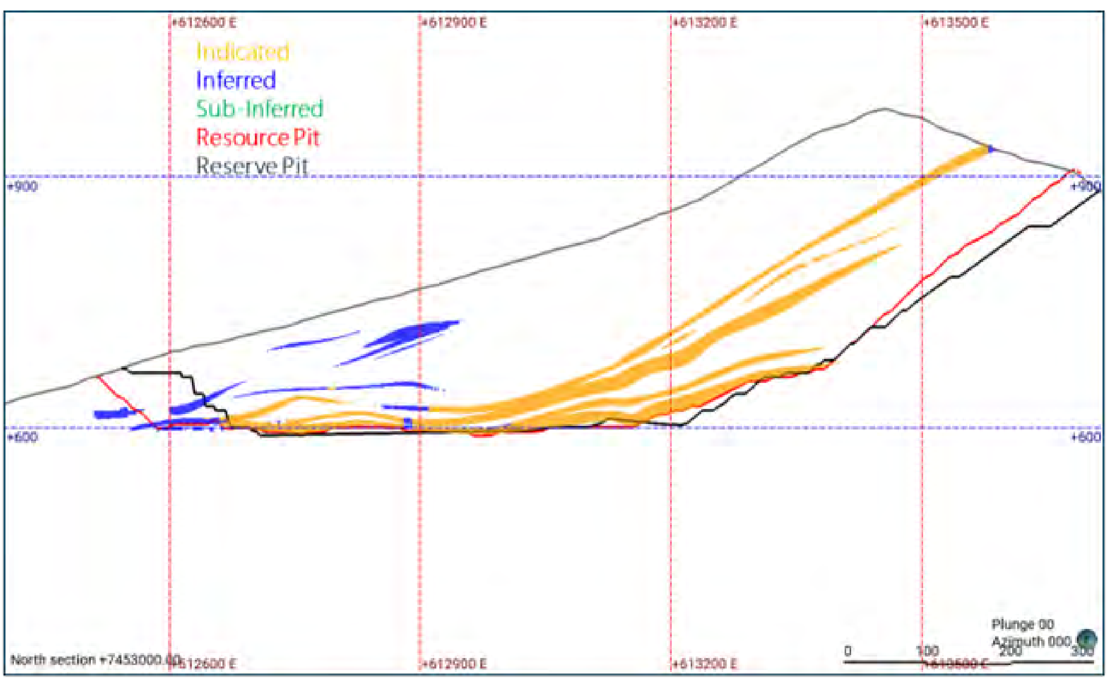
Resource Classification	Criteria
Measured	Not used
Indicated	Mineralized zones parent block centroid estimated with composites within 100 m of three drill holes.
Inferred	Mineralized zones parent block centroid estimated with composites within 150 m of one drill holes.

Figure 11-19: Oblique View of Mineral Resource Classification Model



Source: Ambler Metals, 2022.

Figure 11-20: Cross-Section of Mineral Resource Classification Model, Looking North



Source: Ambler, 2022.

11.5.3 Uncertainties Affecting Mineral Resource Confidence Classification

Factors that may affect the mineral resource estimate are listed below:

- Uncertainties in sampling and drilling methods, data processing and handling
- Metal price assumptions.
- Uncertainties in the assumptions used to generate the cut-off grade.
- Uncertainties in the geological and mineralization shapes, and geological and grade continuity assumptions.
- Uncertainties in the historically predicted (estimated) SG values determined by Random Forest Regressor.
- Uncertainties in the geotechnical, mining, and metallurgical recovery assumptions.
- Uncertainties represented by historical assay values for payable metals.
- Uncertainties in the resource estimation process including parameters such as capping values, search ellipsoids, variogram models, number of composites.
- Uncertainties to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates.
- Uncertainties in the assumptions made to the concentrate marketability, payability and penalty terms.
- Uncertainties in the assumptions regarding the continued ability to access the site, retain mineral and obtain surface rights titles, obtain environment and other regulatory permits, and maintain the social license to operate.

The above uncertainties were considered in Mineral Resource classification criteria and in the opinion of the QP all issues relating to all relevant technical and economic factors likely to influence the prospect of economic extraction can be resolved with further work.

11.5.4 Reasonable Prospects for Economic Extraction

The Arctic deposit comprises several zones of relatively continuous moderate to high-grade polymetallic mineralization that extends from surface to depths of over 250 m below surface. The deposit is potentially amenable to open pit extraction methods. The “reasonable prospects for economic extraction” was tested using a floating cone pit shell derived based on a series of technical and economic assumptions considered appropriate for a deposit of this type, scale, and location. These parameters are summarized in Table 11-24.

Using these parameters, copper equivalent grades that incorporate contributions of the five different metals present in the deposit were calculated using the following equation:

$$CuEq\% = (Cu\% \times 0.92) + (Zn\% \times 0.290) + (Pb\% \times 0.231) + (Au\ g/t \times 0.398) + (Ag\ g/t \times 0.005)$$

Using these parameters and the copper equivalent formula, an open pit marginal cut-off grade of 0.5% CuEq was determined.

Table 11-24: Parameters Used to Generate a Resource Constraining Pit Shell

Optimization Parameters	Unit	Value
Item		
Open Pit Mining Cost	$/t	3
Process Cost + G&A	$/t	35
Pit Slope	degree	43
Copper Price	$/lb	3.00
Lead Price	$/lb	0.90
Zinc Price	$/lb	1.00
Gold Price	$/oz	1,300
Silver Price	$/oz	18
Metallurgical Recovery: Copper	%	92
Lead	%	77
Zinc	%	88
Gold	%	63
Silver	%	56

11.5.4.1 Basis of Commodity Prices and Cost Assumptions

Copper, lead, zinc, gold and silver are exchange-traded contracts on all of the world's major commodity exchanges.

The input costs, recoveries and metal prices used to determine the metal cut-off for the Mineral Resources are based on the Arctic 2020 pre-feasibility study and are within the range of the long-term metal price forecast by mining analysts and investment banks at that time. These metal prices and cost inputs presented in Table 11-24 were fixed over the LOM of 13 years.

11.5.5 Mineral Resource Statement

The Mineral Resource estimate inclusive of Mineral Reserves is stated in Table 11-25. The Mineral Resource estimate exclusive of Mineral Reserves is stated in Table 11-26. All Indicated Mineral Resources have been converted to Mineral Reserves. Mineral Resources are reported in place (point of reference) and on a 100% basis; however, Trilogy's attributable interest is 50% of the tonnes and metal content.

Figure 11-21 shows the isometric view of the Arctic resource model.

Table 11-25: Mineral Resource Summary Table, Inclusive of Mineral Reserves

Confidence Category	Tonnage (Mt)	Average Grade					Contained Metal Content				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Pb (Mlb)	Zn (Mlb)	Au (koz)	Ag (Moz)
Indicated	35.7	2.98	0.79	4.09	0.59	45.2	2,347	621	3,216	675	52
Inferred	4.5	1.92	0.70	2.93	0.43	35.6	189	69	288	62	5

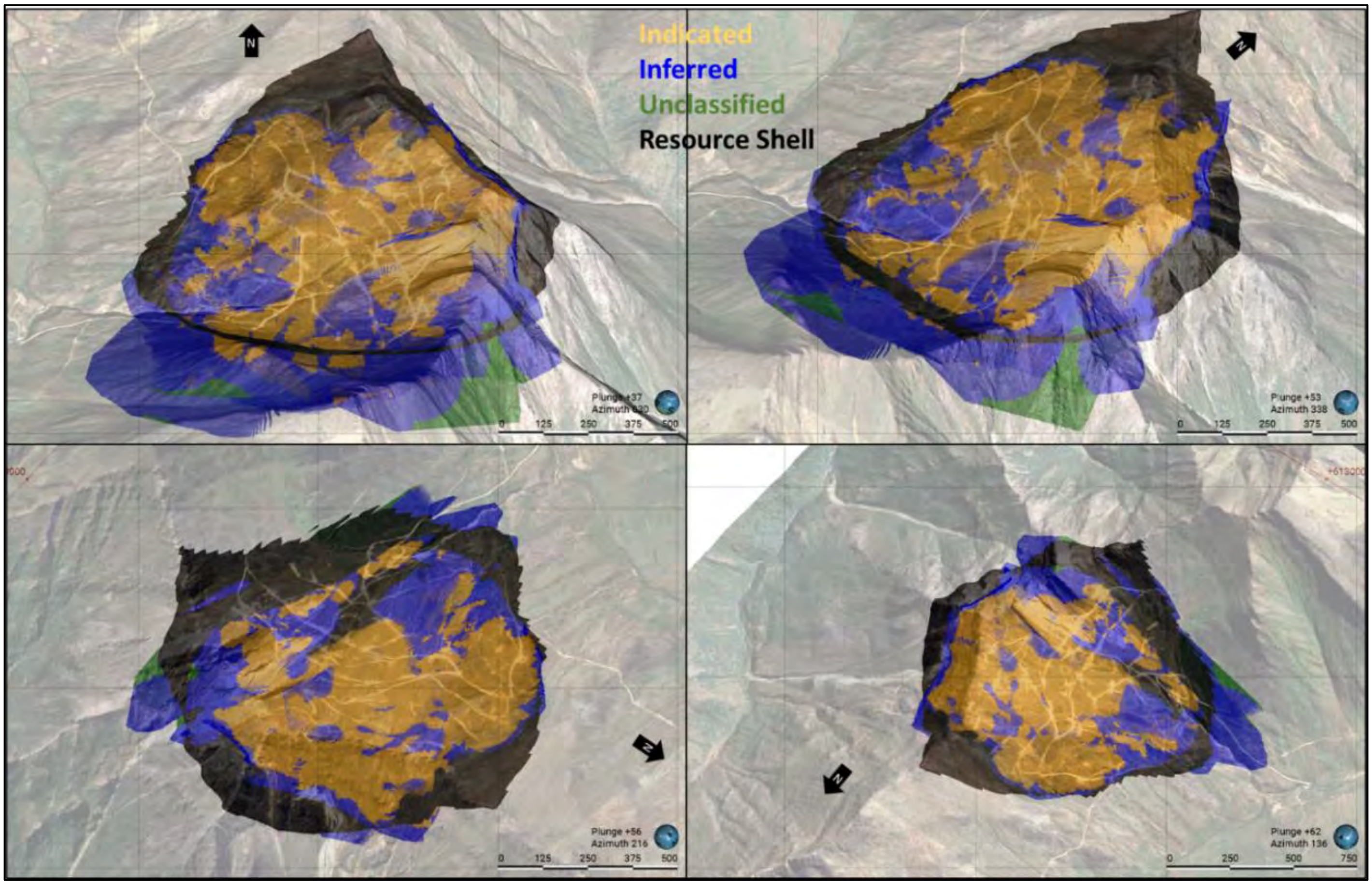
Notes:
1. Mineral Resources are current as of November 30, 2022 and were verified by a Wood QP.
2. Mineral Resources were prepared in accordance with the standards and definitions of S-K 1300.
3. Mineral Resources stated are contained within a conceptual pit shell developed using metal prices of $3.00/lb Cu, $0.90/lb Pb, $1.00/lb Zn, $1300/oz Au and $18/oz Ag and metallurgical recoveries of 92% Cu, 77% Pb, 88% Zn, 63% Au and 56% Ag and operating costs of $3/t mining and $35/t process and G&A. The assumed average pit slope angle is 43°.
4. The cut-off grade is 0.5% copper equivalent. CuEq = (Cu%x0.92) + (Zn%x0.290) + (Pb%x0.231) + (Au g/tx0.398) + (Ag g/tx0.005).
5. As a result of flattening the north end of the reserve pit to stabilize the pit wall due to the presence of talc, a portion of the reserve pit extended beyond the resource constraining pit shell. Approximately 568kt of 1.72% Cu, 0.77% Pb, 0.23 g/t Au and 21.3 g/t Ag in the Indicated category, and approximately 319 kt of 2.01% Cu, 0.87% Pb, 2.53% Zn, 0.50 g/t Au and 37.5 g/t Ag in the Inferred category were added to the Mineral Resource tabulation.
6. The Mineral Resource estimate is reported inclusive of those Mineral Resource that were converted to Mineral Reserves.
7. Trilogy's attributable interest is 50% of the tonnage and contained metal stated in the table.
8. Figures may not sum due to rounding.

Table 11-26: Mineral Resource Summary Table, Exclusive of Mineral Reserves

Confidence Category	Tonnage (Mt)	Average Grade					Contained Metal Content				
		Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)	Cu (Mlb)	Pb (Mlb)	Zn (Mlb)	Au (koz)	Ag (Moz)
Inferred	4.5	1.92	0.70	2.93	0.43	35.6	189	69	288	62	5

Notes:
1. Mineral Resources are current as of November 30, 2022 and were verified by a Wood QP.
2. Mineral Resources were prepared in accordance with the standards and definitions of S-K 1300.
3. Mineral Resources stated are contained within a conceptual pit shell developed using metal prices of $3.00/lb Cu, $0.90/lb Pb, $1.00/lb Zn, $1,300/oz Au and $18/oz Ag and metallurgical recoveries of 92% Cu, 77% Pb, 88% Zn, 63% Au and 56% Ag and operating costs of $3/t mining and $35/t process and G&A. The assumed average pit slope angle is 43°.
4. As a result of flattening the north end of the reserve pit to stabilize the pit wall due to the presence of talc, a portion of the reserve pit extended beyond the resource constraining pit shell and approximately 319 kt of 2.01% Cu, 0.87% Pb, 2.53% Zn, 0.50 g/t Au and 37.5 g/t Ag in the Inferred category were added to the Mineral Resource tabulation.
5. The cut-off grade is 0.5% copper equivalent. CuEq = (Cu%x0.92) + (Zn%x0.290) + (Pb%x0.231) + (Au g/tx0.398) + (Ag g/tx0.005).
6. The Mineral Resource estimate is reported exclusive of those Mineral Resources that were converted to Mineral Reserves.
7. Trilogy's attributable interest is 50% of the tonnage and contained metal stated in the table.
8. Figures may not sum due to rounding

Figure 11-21: Isometric View of Arctic Mineral Resource



Source: Ambler Metals, 2022.

12 MINERAL RESERVE ESTIMATES

12.1 Introduction

Mineral Reserves is classified in accordance with standards and definitions of S-K 1300. Modifying factors were applied to the Indicated Mineral Resources to convert them to Probable Mineral Reserves. Mineral Reserves for the Arctic deposit incorporate appropriate mining dilution and mining recovery estimations for the open pit mining method.

The Mineral Reserve estimate for the Arctic deposit is based on the resource block model provided by Ambler Metals, as well as other information provided by Ambler Metals, and information generated by Wood based on the preceding 2020 Feasibility Study.

The following sections outlines the procedures used to estimate the Mineral Reserves. The mine plan is based on the detailed mine design derived from the optimal pit shell produced by applying the Pseudoflow algorithm.

12.2 Basis of Commodity Prices and Cost Assumptions Used in Pit Optimization and Cut-Off Determination

To establish the long-term metal price forecast the Wood QP used a combination of information derived from 22 financial institutions, from pricing used in technical reports filed with Canadian regulatory authorities over the previous 12-month period, from pricing reported by major mining companies in public filings such as annual reports in the previous 12-month period, spot pricing, and three-year trailing average pricing. From this assessment the Wood QP considers industry consensus on a long-term price forecast on Mineral Reserves as stated in Table 12-1.

The costs stated in Table 12-1 are based on the 2020 Feasibility study and are fixed over the 13-year LOM production.

12.3 Pit Optimization

The pit shell that defines the ultimate pit limit, was derived in Whittle using the Pseudoflow pit optimization algorithm. The optimization procedure uses the block value and pit slopes to determine a group of blocks representing pits of valid slopes that yield the maximum profit. The block value is calculated using information stored in the geological block model, commodity prices, mining and processing costs, process recovery, and the sales cost for the metals produced. The pit slopes are used as constraints for removal precedence of the blocks (Xiaoyu Bai, et al., 2017). Table 12-1 provides a summary of the primary optimization inputs.

Table 12-1: Optimization Inputs

Parameter	Unit	Value	Cu Conc.	Pb Conc.	Zn Conc.
Metal Prices					
Copper	$/lb	3.46	-	-	-
Lead	$/lb	0.91	-	-	-
Zinc	$/lb	1.12	-	-	-
Gold	$/oz	1,615	-	-	-

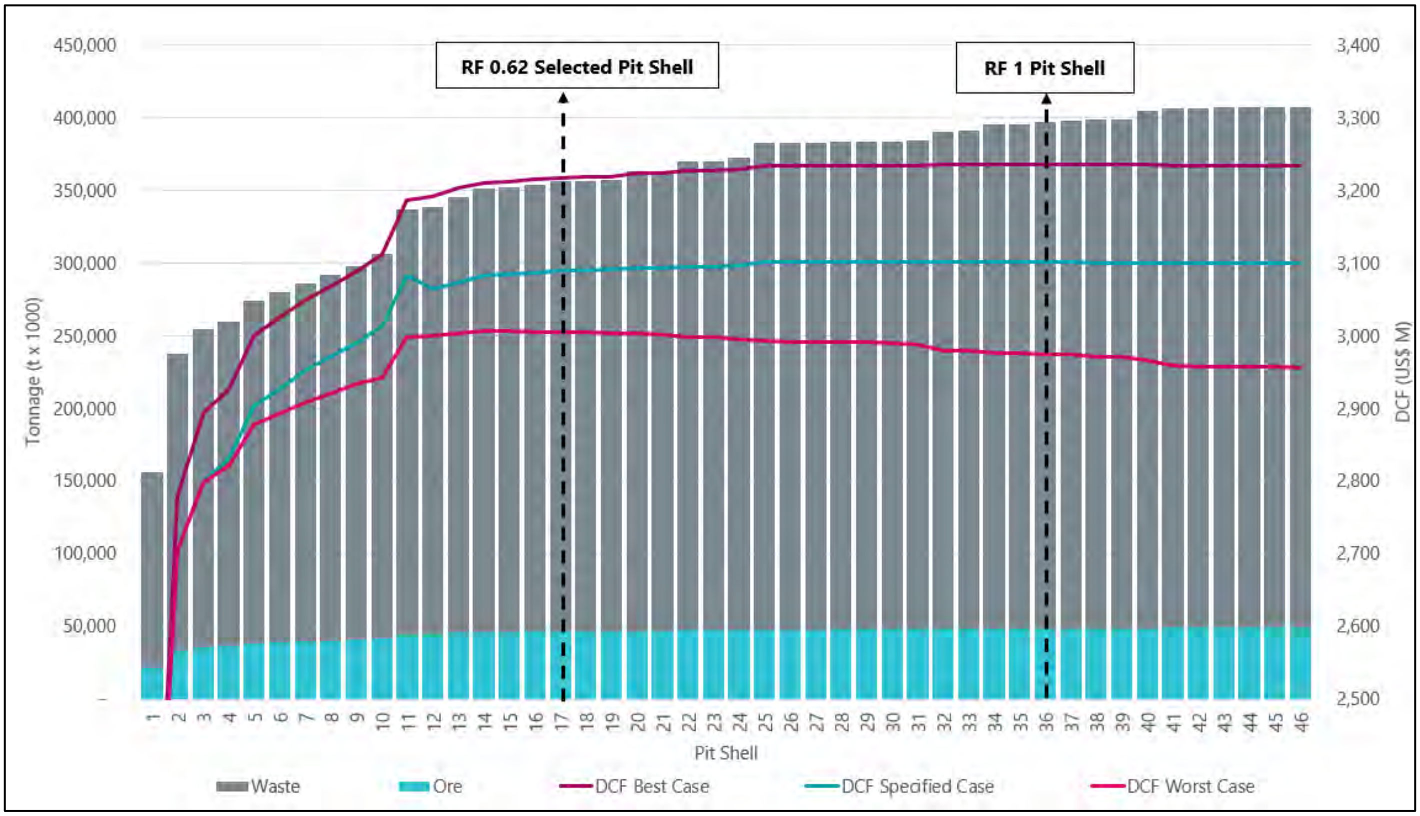
Parameter	Unit	Value	Cu Conc.	Pb Conc.	Zn Conc.
Silver	$/oz	21.17	-	-	-
Discount Rate	%	8	-	-	-
Dilution and Mine Losses	%	Estimated in a block-by-block basis, adding up 30% to 40%.			
Mining Cost					
Reference Bench Elevation	m	790	-	-	-
Base Cost	$/t	2.52	-	-	-
Incremental Mining Cost			-	-	-
Uphill (below 790m)	$/t/5m	0.02	-	-	-
Downhill (above 790m)	$/t/5m	0.012	-	-	-
Process Costs					
Operating Cost	$/t milled	18.31	-	-	-
G&A	$/t milled	5.83	-	-	-
Sustaining Capital	$/t milled	2.37	-	-	-
Road Toll Cost	$/t milled	8.04	-	-	-
Closure	$/t milled	4.27	-	-	-
Processing Rate	kt/d	10	-	-	-
Process Recovery					
Copper	%	-	89.9	2.4	2.7
Lead	%	-	8.1	79	2.2
Zinc	%	-	3.4	0.4	90.6
Gold	%	-	10.9	62.1	5.4
Silver	%	-	26.4	63.1	3.4
Payable – Main Element	%	-	96.5	95	85
Treatment Cost	$/dmt	-	80	160	215
Refining Cost					
Copper	$/lb	-	0.08	-	-
Gold	$/oz	-	5	10	-
Silver	$/oz	-	0.5	1.25	-
Transport Cost	$/dmt	271	-	-	-
Concentrate Losses	% weight	0.42	-	-	-
Insurance Cost	%	0.15	-	-	-
Representation/Marketing	$/wmt	2.5	-	-	-

Parameter	Unit	Value	Cu Conc.	Pb Conc.	Zn Conc.
Slope Angles					
Geotechnical Sector 1 (2L-E)	degrees	Variable based on slope dip direction. IRA ranging from 26 to 56.			
Geotechnical Sector 2 (2L-W)	degrees	Variable based on slope dip direction IRA ranging from 38 to 56.			
Geotechnical Sector 3 (2U)	degrees	Variable based on slope dip direction IRA ranging from 29 to 56.			
Geotechnical Sector 4 (3)	degrees	Variable based on slope dip direction IRA ranging from 30 to 56.			
Geotechnical Sector 5 (4L)	degrees	Variable based on slope dip direction IRA ranging from 34 to 56.			
Geotechnical Sector 6 (4U)	degrees	Variable based on slope dip direction IRA ranging from 37 to 56.			
Royalties					
NANA Surface Use	%NSR	1	-	-	-

Wood converted the resource model provided by Ambler Metals from a sub-blocked to a regularized percentage model. The model was imported into the optimization software containing grades, block percentages, material density, slope sectors, rock types, and net smelter return. The optimization run was carried out only using Indicated Mineral Resources to define the optimal mining limits.

The optimization run included 46 pit shells defined according to different revenue factors, ranging from a revenue factor of 0.3 to 1.2, where a revenue factor of 1 is the base case. To select the optimal pit shell that defines the ultimate pit limit, Wood conducted a pit-by-pit analysis to evaluate the contribution of each incremental shell to the discounted cash flow (DCF), assuming a processing plant capacity of 10 kt/d and a yearly discount rate of 8% (Figure 12-1). Following this analysis, pit shell 17 (revenue factor of 0.62) was selected. Although the DCF for the Selected pit shell is slightly lower in comparison to the Base Case pit shell, the selected pit shell eliminates mining 39.3 Mt of waste while only losing 1.9 Mt of ore. The DCF does not account for initial capital expenditures.

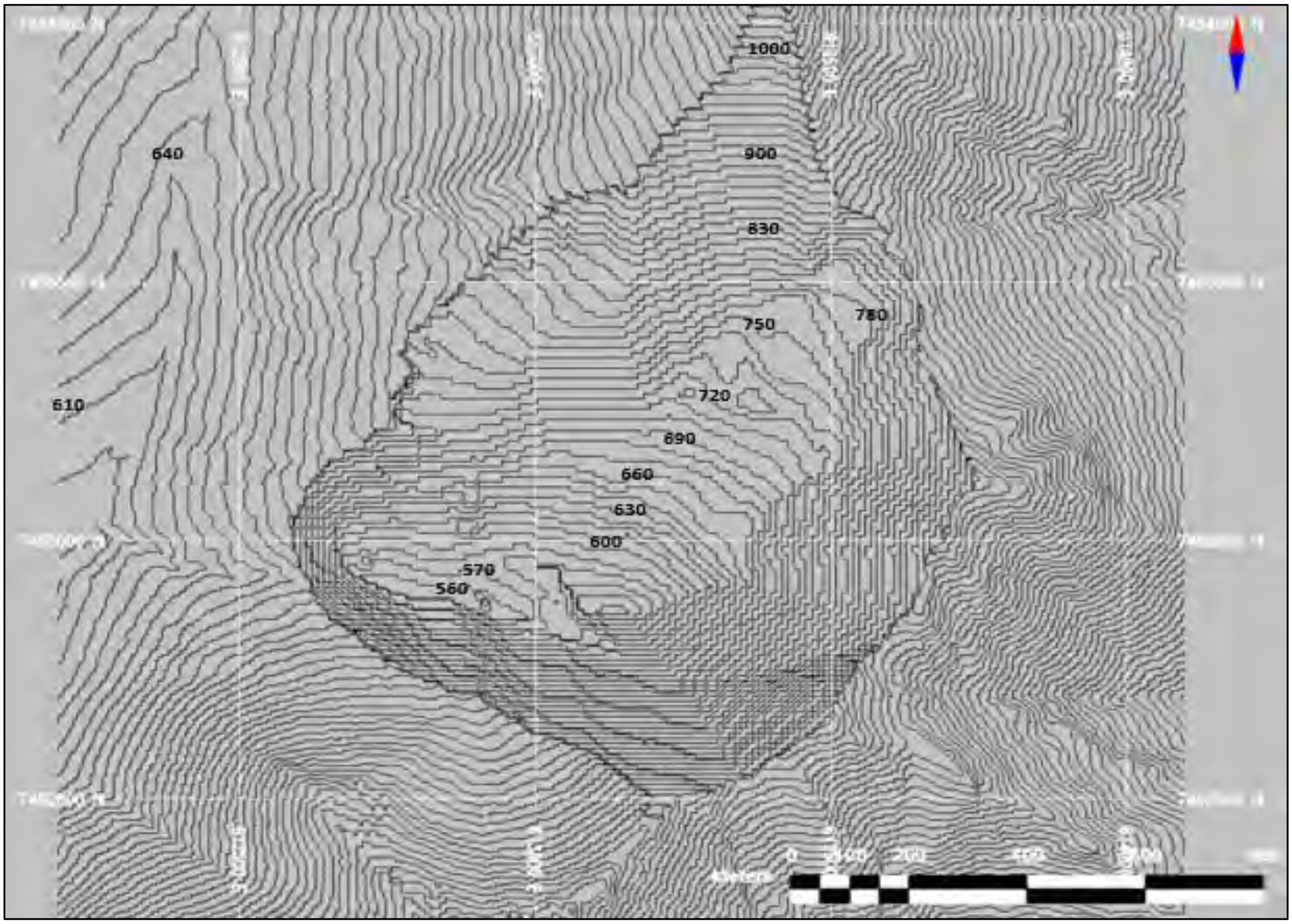
Figure 12-1: Pit-by-Pit Analysis



Source: Wood, 2022.

Following the pit-by-pit analysis, a secondary exercise was performed on the selected pit shell to incorporate a higher level of detail to the slope definition and to force the northeast wall to fully mine areas in the pit where talc zones are problematic for slope stability. As a result of this exercise, material contained within the selected pit shell increases by 13 Mt of waste and 0.8 Mt of ore. The final pit shell to be used as the basis for the ultimate pit and mine planning for the Arctic Project is shown in Figure 12-2.

Figure 12-2: Selected Pit Shell



Source: Wood, 2022.

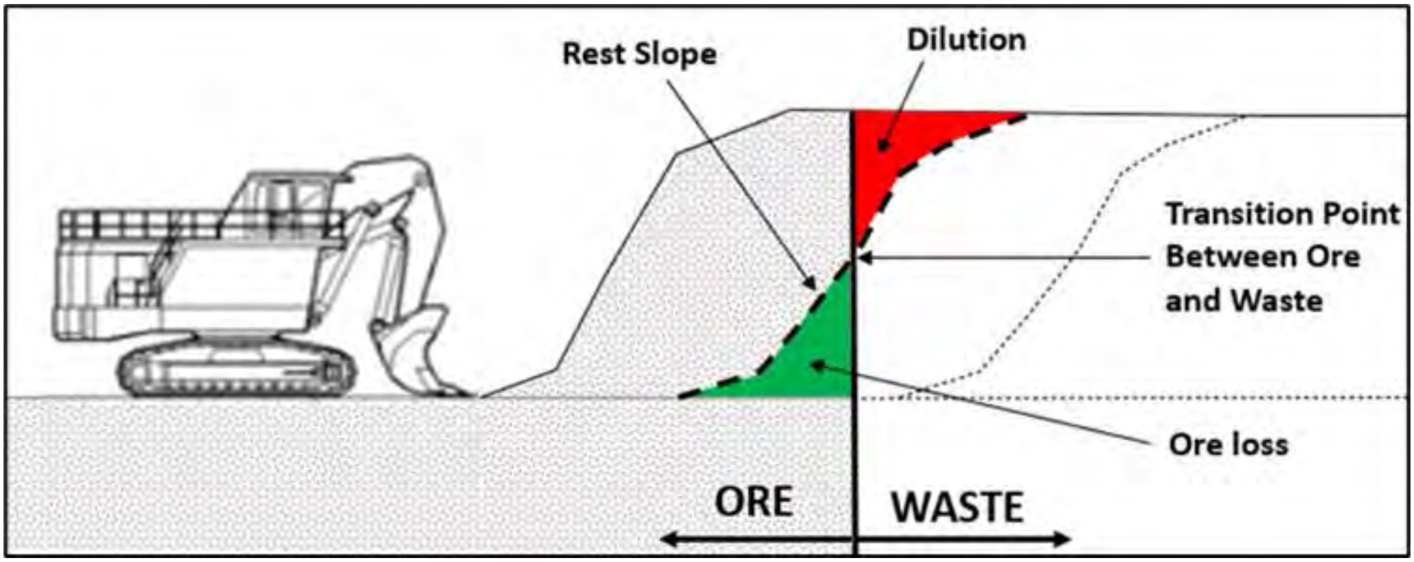
12.4 Dilution and Ore Losses

The Mineral Resources for the Arctic Project were reported undiluted. In order to determine Mineral Reserves, dilution was applied to the resource model in two steps: internal dilution and contact dilution.

Internal dilution accounts for the waste material entrenched within the deposit at the scale of the selective mining unit (SMU) block size. Contact dilution accounts for the dilution and ore losses caused by a block mixing with its surroundings during the mining process and includes ore losses.

For transition points between ore and waste, ore loss and dilution will take place as shown in Figure 12 3. Blocks with diluted grades that fell below the cut-off grade were treated as waste and removed from the Mineral Reserve as ore losses.

Figure 12-3: Contact Dilution Concept



Source: Wood, 2022.

Internal dilution was estimated as follows:

- The resource model was regularized to 5x5x5 m to better approximate mining selectivity. The regularized block model contained the volume percentage within the mineralized zone, and grades and densities for the mineralized zone and host rock.
- The grades were diluted per each block using the formula:

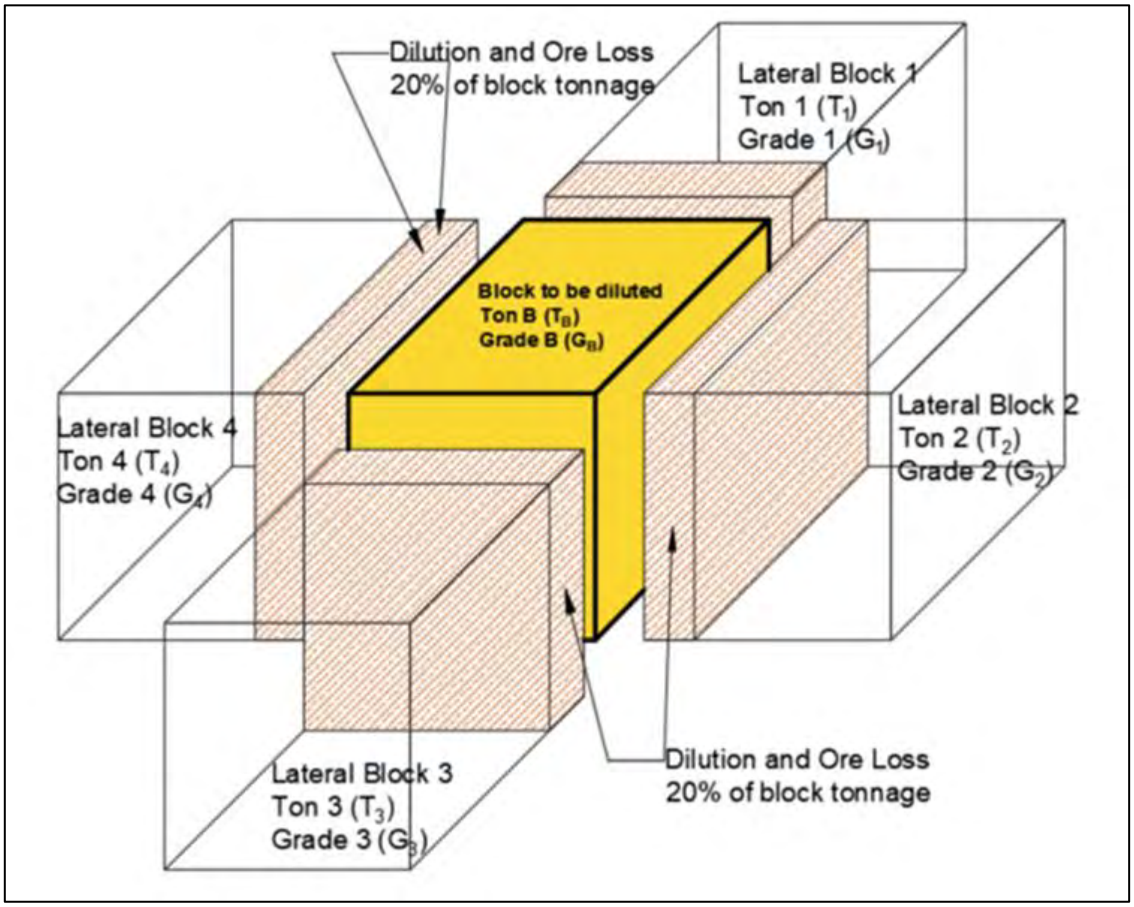
$$Diluted\ Grade = \frac{Ore\ Grade\ \times Ore\ tonnage + Waste\ Grade\ \times Waste\ Tonnage}{Total\ Tonnage}$$

Contact dilution was estimated as follows:

- To simulate the concept shown in Figure 12-3, the grade of a given block was diluted by blending it with 20% of the tonnage from each of its four lateral blocks. The tonnage of the diluted block remains unchanged in the block model.
- For lateral blocks classified as Inferred Mineral Resource, grade was set to zero for dilution purposes.

The procedure is illustrated in Figure 12-4.

Figure 12-4: Contact Dilution Estimation Procedure



Source: Wood, 2022.

12.5 Cut-Off Definition

The cut-off was based on the NSR value. The NSR has been calculated according to the following formulas:

$$Revenue = (Element\ Grade * Process\ Recovery) * (Element\ Price - Sales\ Cost)$$

$$NSR = \left(Revenue_{Copper\ Conc.} + Revenue_{Lead\ Conc.} + Revenue_{Zinc\ Conc.}\right) * (1 - Royalty)$$

Cost, metal pricing and sales costs are listed in Table 12-1. Variable process recoveries used in the NSR calculation are listed in Table 12-2.

The marginal cut-off pays for all the costs except the mining cost and was estimated to be \$38.8/t. The economic cut-off pays for all the costs including the mining cost and varies by depth with incremental mining cost between \$41.3/t to \$42.2/t. Ore above the marginal cut-off was included in the Mineral Reserves.

Table 12-2: Variable Process Recoveries Used in the NSR Calculation

Parameter	Cu Conc	Pb Conc	Zn Conc
Copper	92.2	0.8	2.1
Lead	16.0	62.2	4.5
Zinc	1.9	1.3	87.6
Gold	47.2	26.1	3.3
Silver	32.7	48.7	5.8

12.6 Mineral Reserves Statement

Individual blocks captured within the final pit design were identified as either ore or waste by applying the parameters shown in Figure 12 1 with the exception of the metallurgical recoveries which were applied variably on a block-by-block basis. Using the partial block percentages within the final pit design, the ore tonnage and average grades were calculated. The Mineral Reserves statement is shown in Table 12 2. The point of reference for the Mineral Reserves is defined at the point where the ore is delivered to the processing plant. Trilogy's attributable interest is 50% of the tonnes of the Mineral Reserves.

Table 12-3: Mineral Reserve Statement

Confidence Category	Tonnage	Average Grade				
	Mt	Cu (%)	Pb (%)	Zn (%)	Au (g/t)	Ag (g/t)
Probable Mineral Reserves	46.7	2.11	0.56	2.90	0.42	31.8

Notes:
1. Mineral Reserves estimates are current as of November 30, 2022 and were prepared by a Wood QP.
2. Mineral Reserves were estimated assuming open pit mining methods and include a combination of internal and contact dilution. Total dilution is expected to be between 30% and 40%. Pit slopes vary by sector and range from 26° to 56°. A marginal NSR cut-off of $38.8 /t is used.
3. Mineral Reserves are based on prices of $3.46/lb Cu, $0.91/lb Pb, $1.12/lb Zn, $1,615/oz Au, and $21.17/oz Ag.
4. Variable process recoveries averaging 92.2% Cu in Cu concentrate, 62.2% Pb in Pb concentrate, 87.6% Zn in Zn concentrate, 16.0% Pb in Cu concentrate, 1.9% Zn in Cu concentrate, 47.2% Au in Cu concentrate, 32.7% Ag in Cu concentrate, 0.8% Cu in Pb concentrate, 1.3% Zn in Pb concentrate, 26.1% Au in Pb concentrate, 48.7% Ag in Pb concentrate, 2.1% Cu in Zn concentrate, 4.5% Pb in Zn concentrate, 3.3% Au in Zn concentrate, 5.8% Ag in Zn concentrate.
5. Mineral Reserves are based on mining cost of $2.52/t incremented at $0.02/t/5m and $0.012/t/5m below and above 790 m elevation, respectively.
6. Costs applied to processed material following: process operating cost of $18.31/t, G&A of $5.83/t, sustaining capital cost of $2.37/t, closure cost of $4.27/t, road toll cost of $8.04/t.
7. Strip ratio (waste:ore) is 7.3:1.
8. Selling terms following: payables of 96.5% of Cu, 95% of Pb and 85% of Zn, treatment costs of $80/t Cu concentrate, $160/t Pb concentrate and $215/t Zn concentrate; refining costs of $0.08/lb Cu in Cu concentrate, and $10/oz Au, $1.25/oz Ag in Pb concentrate; and transport cost $270.98/t concentrate.
9. Fixed royalty percentage of 1% NSR.
10. Trilogy's attributable interest is 50% of the tonnage stated in the table.

12.7 Risk Factors that Could Materially Affect the Mineral Reserve Estimates

In the opinion of the QP, the Mineral Reserves are subject to the types of risks common to open pit polymetallic mining operations that exist in Alaska and may be materially affected by the following risk factors:

- Changes in metal prices from what was assumed;
- Changes to the assumptions used to generate the cut-offs;

- Changes in local interpretations of mineralization geometry and continuity of mineralized zones;
- Changes to geological and mineralization shapes, and geological and grade continuity assumptions;
- Changes to density and domain assignments from what was assumed;
- Changes to geotechnical, hydrogeological design assumptions;
- Changes to mining and metallurgical recovery assumptions;
- Change to the input and design parameter assumptions that pertain to the open pit constraining the estimates;
- Assumptions as to concentrate marketability, payability and penalty terms;
- Assumptions as to the continued ability to access the site, retain mineral tenure and obtain surface rights titles, obtain environment and other regulatory permits, and maintain the social license to operate.

More specifically the presence of certain talc layers in the rock have not been included in the current geological model and could affect the metallurgical recoveries and slope stability. Additionally, there is currently no developed surface access to the Arctic Project area and beyond. Access to the Arctic Project is proposed to be via AAP, a road approximately 340 km (211 miles) long, extending west from the Dalton Highway where it would connect with the proposed Arctic Project area. Construction costs of the road are not yet final. The working assumption is that AIDEA would arrange financing in the form of a public-private partnership to construct and arrange for the construction and maintenance of the access road. AIDEA would charge a toll to multiple mining and industrial users (including the Arctic Project) in order to pay back the costs of financing the AAP. The amount paid in tolls by any user would be affected by the cost of the road, its financing structure, and the number of mines and other users of the road which could also include commercial transportation of materials and consumer items that would use the AAP to ship concentrates to the Port of Anchorage in Alaska and possibly provide goods and commercial materials to villages in the region.

The Mineral Reserve estimation assumes toll payments of $5.52/t processed, plus a road maintenance fee of $2.52/t milled processed, resulting in a total road toll and maintenance LOM unit cost of $8.04/t processed. There is a risk that a negotiated road toll agreement may result in higher costs than what has been assumed.

13 MINING METHODS

13.1 Overview

The following section outlines the parameters and procedures used for the design of the mine as a conventional open pit, estimates of the Mineral Reserves within the open pit mine plan, and establishes a practical mining schedule for this Report. The mine plan is based on the Probable Mineral Reserves presented in Section 12.

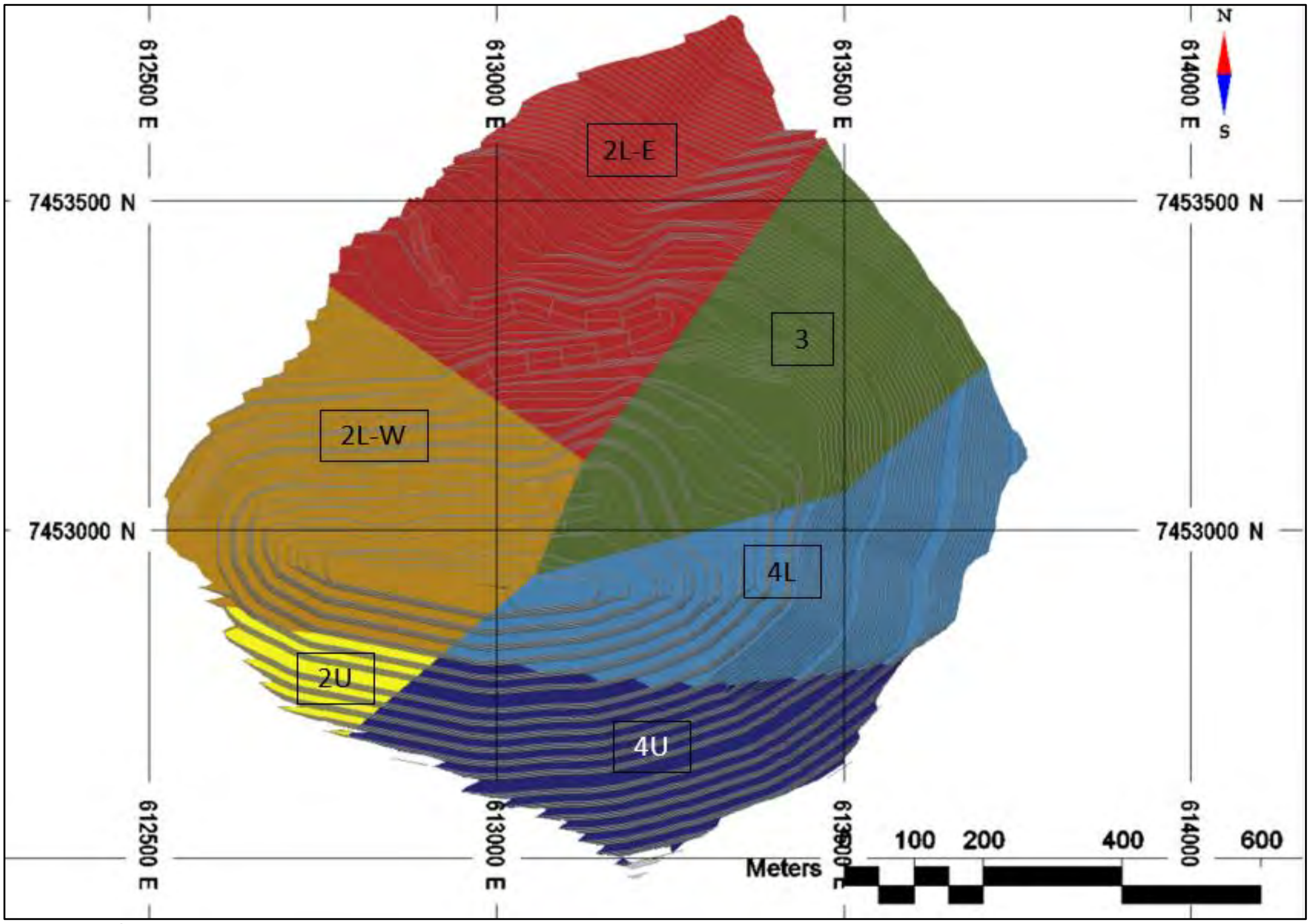
13.2 Mine Design

The Arctic Project is designed as a conventional truck-shovel operation with 144 t trucks and 15 m^3 shovels. The pit design includes four nested phases to balance stripping requirements while satisfying the concentrator requirements.

The design parameters include a ramp width of 30 m, road grades of 10%, bench height of 5 m, targeted mining width between 70 and 100 m, berm interval of 20 m, variable slope angles by sector and a minimum mining width of 30 m.

Wood performed a review of the pit geotechnical and hydrogeological assessment performed by SRK. Based on the review of the information provided by SRK, Wood agrees with their conclusions. Figure 13-1 shows the geotechnical slope design domains defined by SRK on the 2020 pit design. Table 13-1 shows a summary of the geotechnical design parameters recommended by SRK.

Figure 13-1: Geotechnical Slope Design Domains



Source: Wood, 2022.

Table 13-1: Geotechnical Design Parameters

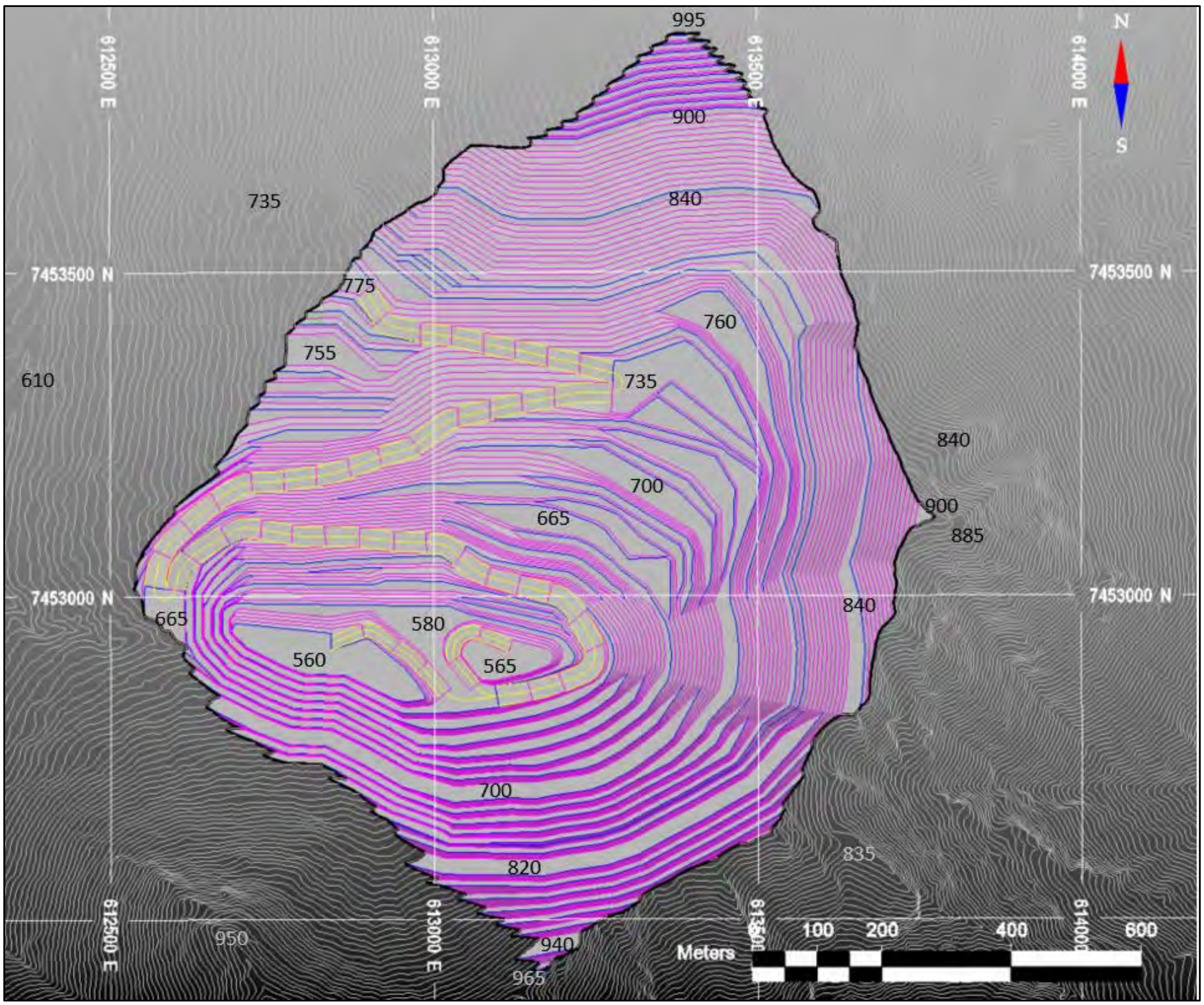
Slope Design Domain	Face Slope Dip Direction (*)	Bench Height (m)	IRA (°)	Design BFA* (°)	Design Bench Width (m)
2L-E	330 - 020	10	40	65	7.5
		20		65	14.5
	020 - 050	20	56	80	10.0
	050 - 100	20	45	65	10.5
	100 - 160	20	38	65	16.0
	160 - 250	60	26	27	8.0
	250 - 300	20	38	80	22.0
	300 - 330	10	50	80	7.5
		20	55	80	10.5
2L-W	330 - 020	10	40	65	7.5
		20		65	14.5
	020 - 140	20	56	80	10.0
	140 - 170	20	45	65	10.5
	170 - 250	60	38	41	8.0
	250 - 300	20	40	65	14.5
	300 - 330	10	50	80	7.5
		20	55	80	10.5
2U	330 - 020	20	45	65	10.5
	020 - 140	20	56	80	10.0
	140 - 170	20	45	65	10.5
	170 - 250	60	29	31	8.0
	250 - 300	20	40	65	14.5
	300 - 330	10	55	80	7.5
		20		80	10.5
3	310 - 020	10	38	65	8.0
		20		65	16.0
	020 - 050	20	56	80	10.0
	050 - 100	20	45	65	10.5
	100 - 140	20	56	80	10.0
	140 - 220	20	38	65	16.0
	220 - 310	60	30	32	8.0
4L	330 - 020	10	40	65	7.5
		20		65	14.5
	020 - 050	20	56	80	10.0
	050 - 100	20	45	65	10.5
	100 - 140	20	56	80	10.0
	140 - 180	20	38	65	16.0
	180 - 300	60	34	37	8.0
	300 - 330	10	38	65	8.0
		20		65	16.0
4U	330 - 020	20	45	65	10.5
	020 - 050	20	56	80	10.0
	050 - 100	20	45	65	10.5
	100 - 140	20	56	80	10.0
	140 - 180	20	45	65	10.5
	180 - 260	60	37	40	8.0
	260 - 300	20	40	65	14.5
	300 - 330	10	50	80	7.5
		20	55	80	10.5

*BFA: Bench Face Angle

As mentioned in Section 13.9.10 according to the SRK's geotechnical recommendations, a large talc zone located in the northeast pit slope must be completely removed. To achieve this, a pit optimization giving dummy values to the talc zone blocks was performed. This pit optimization was used as the basis for the mine design.

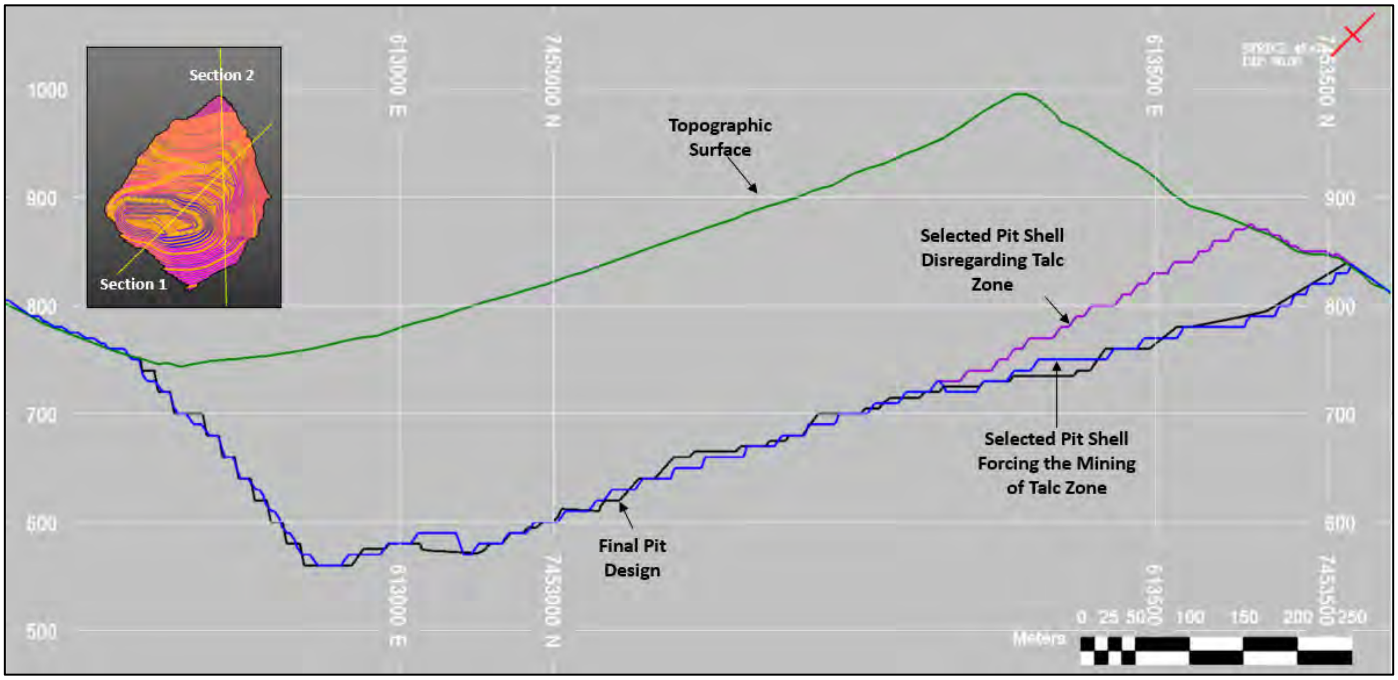
The smoothed final pit design contains approximately 46.7 Mt of ore and 340.2 Mt of waste for a resulting stripping ratio of 7.3:1. Within the 46.7 Mt of ore, the average grades are 2.11% Cu, 2.90% Zn, 0.56% Pb, 0.42 g/t Au and 31.8 g/t Ag. Figure 13-2 shows the ultimate pit design. Figure 13-3 and Figure 13-4 show pit sections comparing the mine design to the selected pit shells for each optimization.

Figure 13-2: Ultimate Pit Design



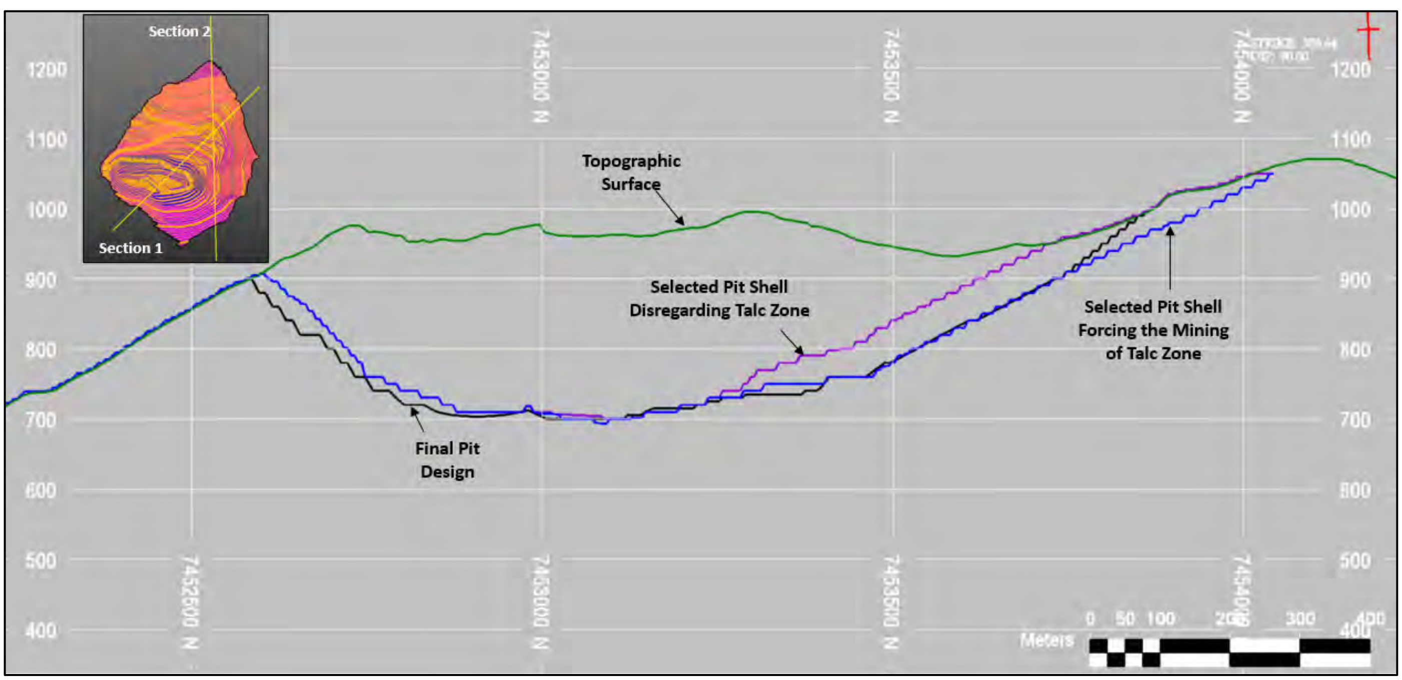
Source: Wood, 2022.

Figure 13-3: Section 1 Showing Mine Design and Selected Pit Shells (looking NW)



Source: Wood, 2022.

Figure 13-4: Section 2 Showing Mine Design and Selected Pit Shells (looking W)

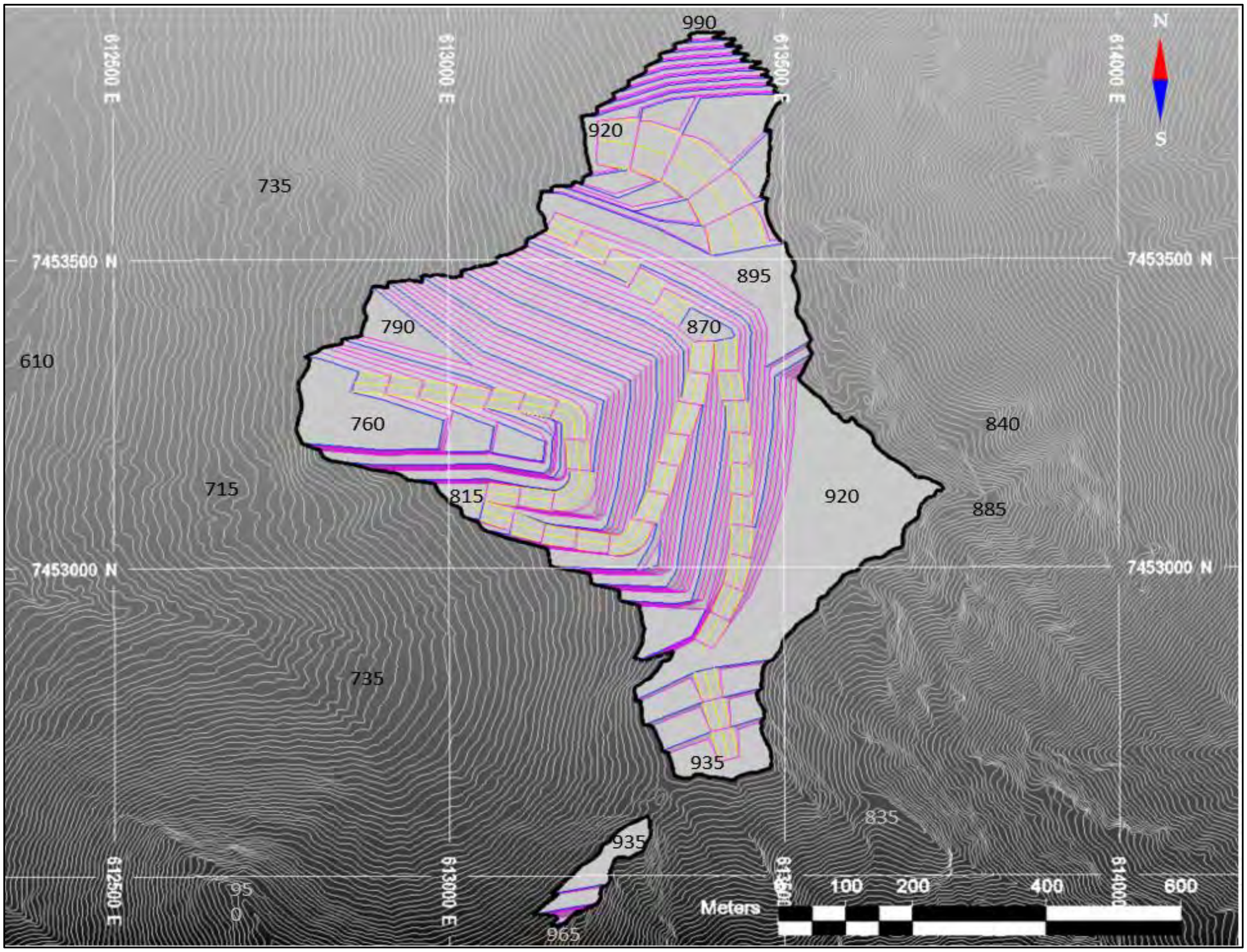


Source: Wood, 2022.

13.3 Interim Phase Design

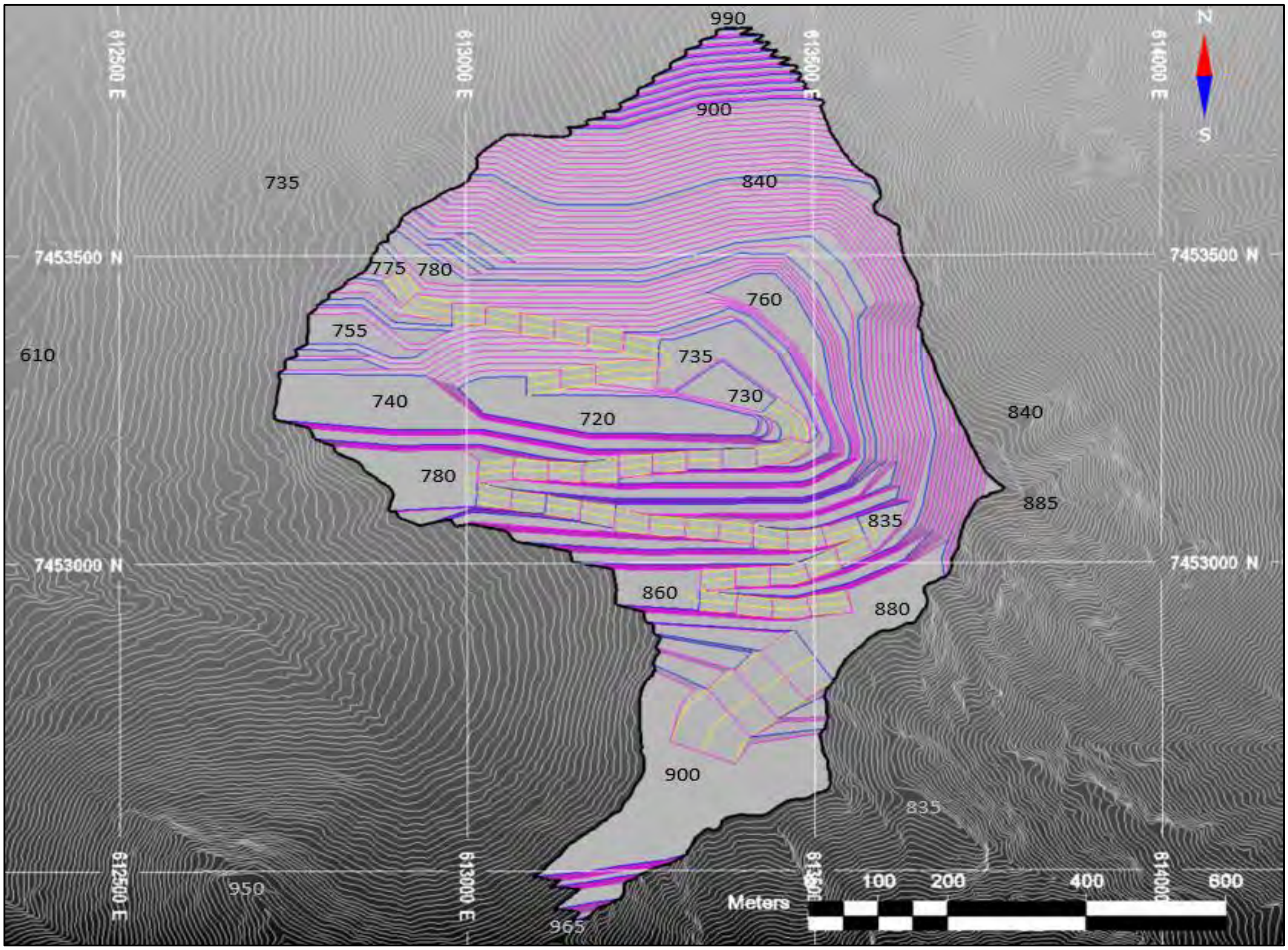
The deposit is mined in four nested phases, including the ultimate pit limit. During pre-production, the phasing strategy maximizes waste mining while minimizing the ore tonnage. This is primarily due to the lack of space to build ore stockpiles. In addition, the phasing strategy maintains enough exposed ore to guarantee the continuous operation of the process plant during the production. The phase designs are shown in Figure 13-5 to Figure 13-7.

Figure 13-5: Phase 1 Design



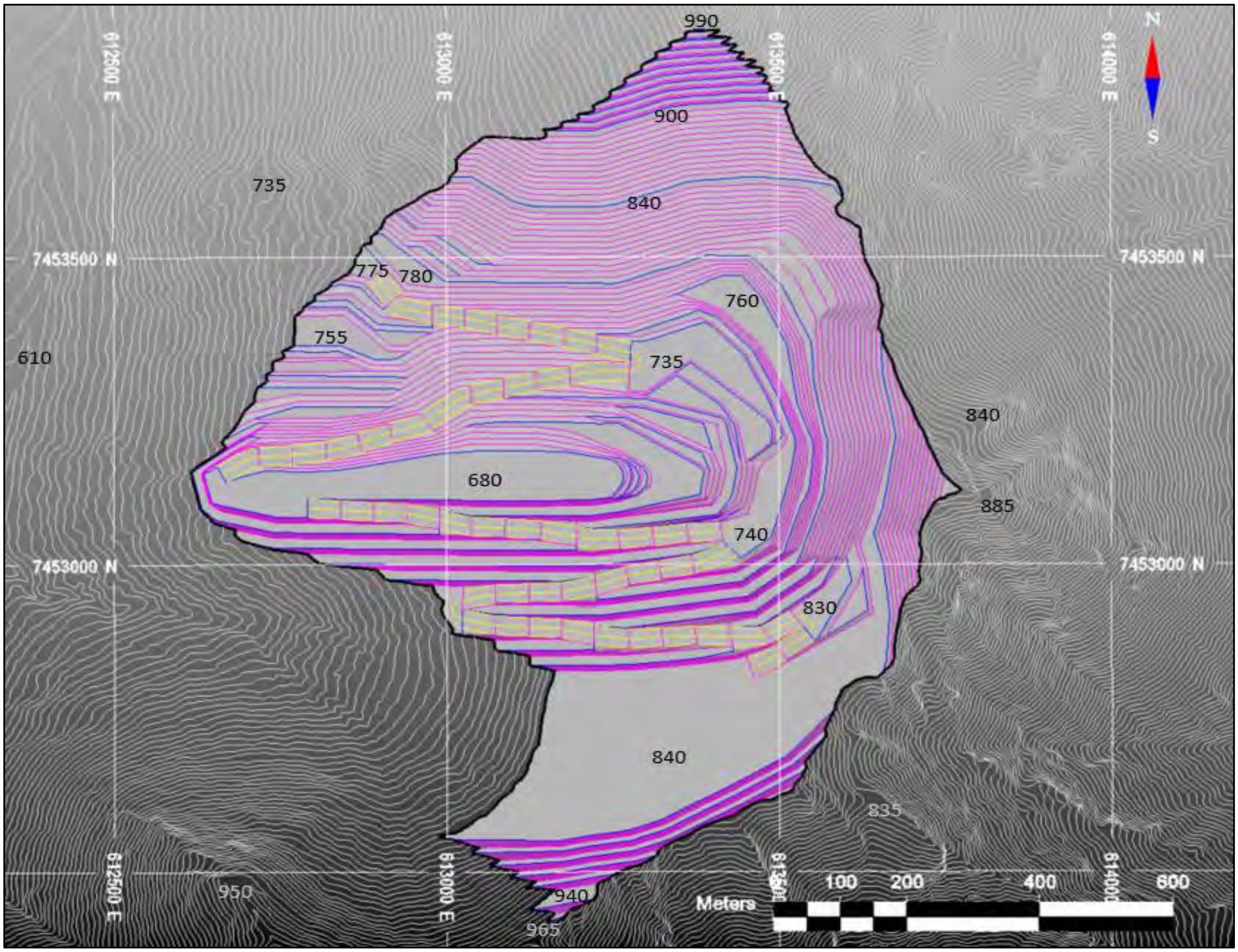
Source: Wood, 2022.

Figure 13-6: Phase 2 Design



Source: Wood, 2022.

Figure 13-7: Phase 3 Design



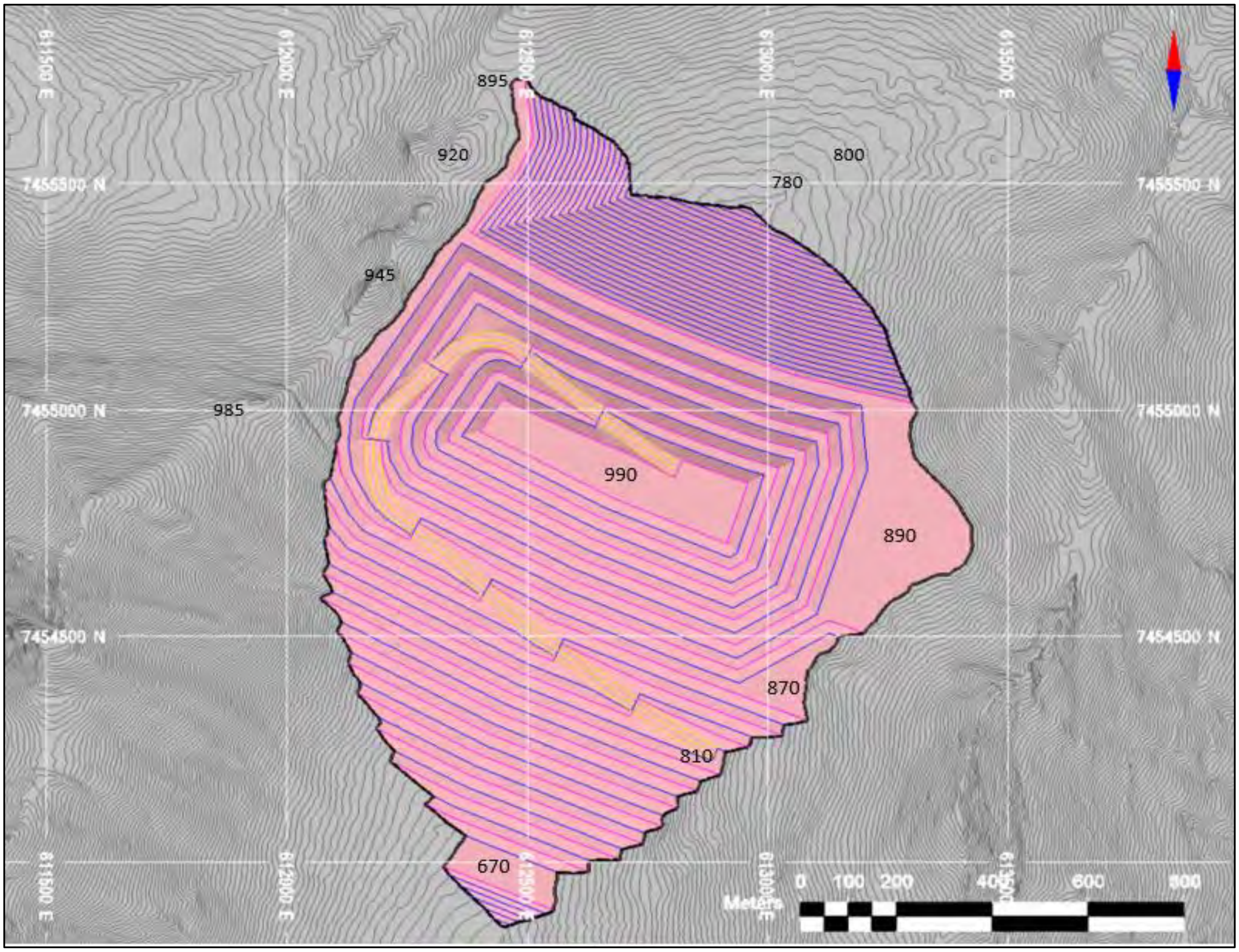
Source: Wood, 2022.

13.4 Waste Rock Facilities and Stockpile Designs

The design and construction of the WRF should ensure physical and chemical stability during and after mining activities. To achieve this, the waste areas and stockpiles are designed to account for benching, drainage, geotechnical stability, and concurrent reclamation.

The WRF design criteria include 23.5-m wide benches every 20 m in elevation with 2.5H:1V overall slopes, lifts from 5-m to 20-m high, and a 33% swell factor for estimating volumes. The overburden mined represents approximately 6% of the total waste and it is considered to be encapsulated within the waste rock. Figure 13-8 shows the WRF design.

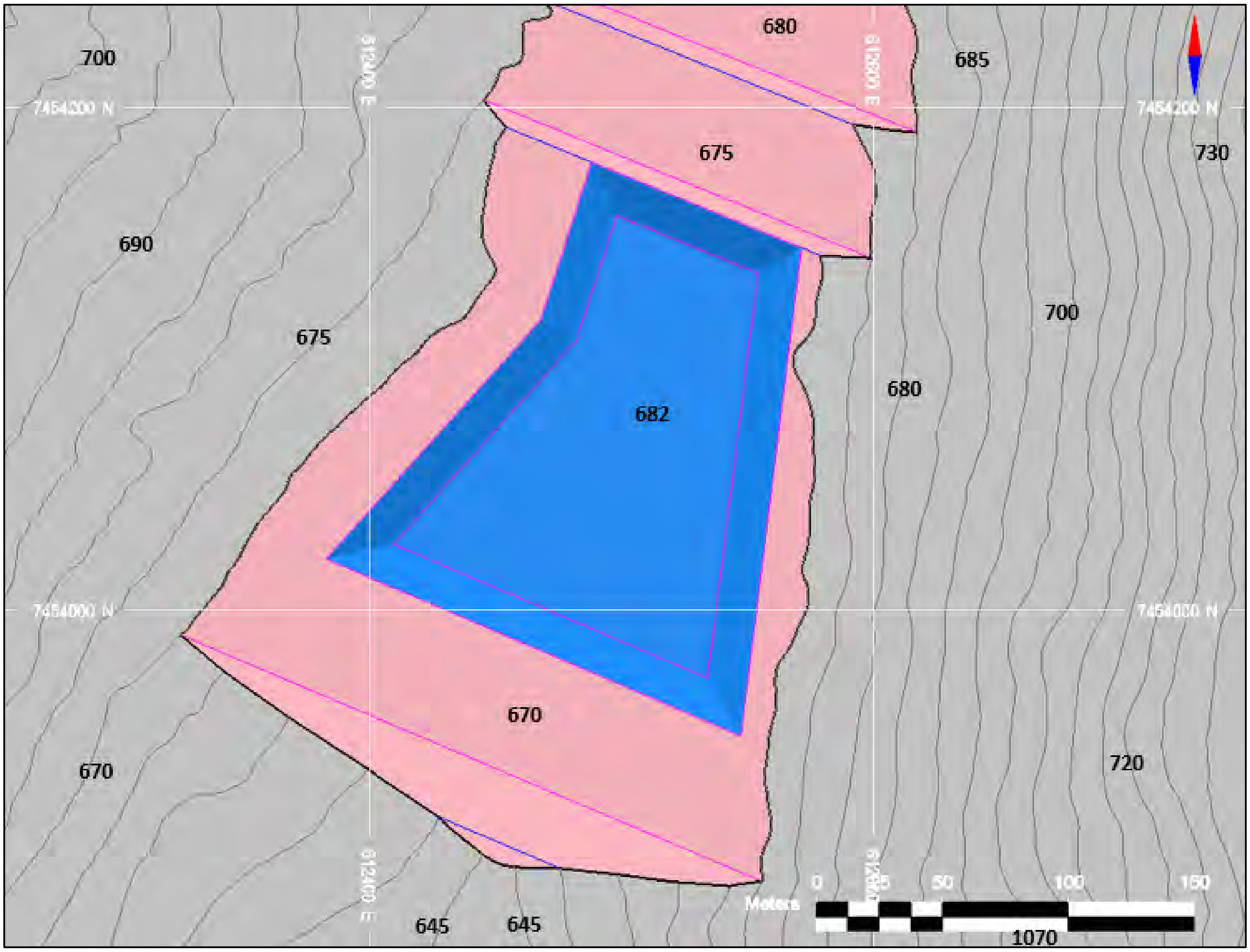
Figure 13-8: Waste Rock Facility



Source: Wood, 2022.

A stockpile is required to store the ore mined during the pre-production period. Because this stockpile is depleted at the beginning of the operation, it is located within the WRF footprint and has a total storage capacity of 223,000 m3. This volume is enough to satisfy the maximum stockpiling capacity of approximately 776 kt. Figure 13-9shows the stockpile design with respect to the WRF at the end of the pre-production period.

Figure 13-9: Ore Stockpile



Source: Wood, 2022.

13.5 Production Schedule

The production schedule includes the processing ramp up. The processing plant ramp-up considers the normal inefficiencies related to the start of operations, and includes the tonnage processed as well as the associated recoveries, which increases the design capacity during the second quarter of operation. The mine requires two years of pre-production before the start of operations in the processing plant.

The deposit is mined in four nested phases, including the ultimate pit limit. The schedule was developed in months for the pre-production period and the first two years of production, in quarters from Year 3 to Year 5 of production, and annually thereafter. The scheduling constraints set the maximum mining capacity at 35 Mt per year and the maximum number of benches mined per year at ten in each phase.

The production schedule based on the Probable Mineral Reserves shows a LOM of 15 years, including two years of pre-production. The amount of rehandled mill feed is 776 kt, which is the ore mined during the pre-production period. The average grades to the mill over the LOM are 2.11% Cu, 2.90% Zn, 0.56% Pb, 0.42 g/t Au, 31.83 g/t Ag and 2.92% Talc. The annual LOM summary schedule is shown in Table 13-2 and Figure 13-10. Figure 13-11 shows the annual tonnage mined by phase.

Table 13-2: Production Schedule

Period	Tonnage					Average Grade					
	To Mill			Mine to	Total	Cu	Zn	Pb	Au	Ag	Talc
	Direct (kt)	Stockpile (kt)	Total (kt)	Stockpile (kt)	Waste (kt)	(%)	(%)	(%)	(g/t)	(g/t)	(%)
Year -2	-	-	-	493	26,907	-	-	-	-	-	-
Year -1	-	-	-	283	34,761	-	-	-	-	-	-
Year 1	2,235	776	3,012	-	31,842	2.59	2.87	0.55	0.36	33.07	1.59
Year 2	3,650	-	3,650	-	30,540	2.06	2.48	0.50	0.38	29.03	3.35
Year 3	3,650	-	3,650	-	27,365	1.94	2.61	0.56	0.39	28.42	7.46
Year 4	3,650	-	3,650	-	26,350	1.88	2.82	0.58	0.42	28.03	4.13
Year 5	3,650	-	3,650	-	25,580	1.87	2.86	0.53	0.33	26.36	32.97
Year 6	3,650	-	3,650	-	24,913	1.99	2.51	0.49	0.34	28.66	3.31
Year 7	3,650	-	3,650	-	24,413	2.37	3.11	0.57	0.40	31.75	3.2
Year 8	3,651	-	3,651	-	19,850	2.00	3.107	0.61	0.44	32.44	1.32
Year 9	3,650	-	3,650	-	20,350	2.05	2.51	0.39	0.37	28.50	4.56
Year 10	3,650	-	3,650	-	17,922	2.33	3.09	0.53	0.50	36.80	2.76
Year 11	3,650	-	3,650	-	15,350	2.41	2.41	0.64	0.56	38.68	21.95
Year 12	3,650	-	3,650	-	11,629	1.98	3.15	0.66	0.47	34.08	0.59
Year 13	3,529	-	3,529	-	2,126	2.09	3.20	0.69	0.52	38.39	0.14
Total	**45,915**	**776**	**46,691**	**776**	**340,168**	**2.11**	**2.90**	**0.56**	**0.42**	**31.83**	**2.92**

Note: Totals may not sum due to rounding

Figure 13-10: Annual Production Schedule

Source: Wood, 2022

Figure 13-11: Total Tonnage Mined, Scheduled by Phase

Source: Wood, 2022

13.6 Waste Material Handling

Waste is hauled to the WRF using 144 t trucks. The construction sequence starts at the bottom of the dump by dumping the material in 5-m lifts, leaving a 23.5-m bench every four lifts. The resulting overall slope angle of the dump face is 2.5H:1V.

13.7 Operating Schedule

The Arctic mine is scheduled to operate 24 hours a day, seven days a week utilizing three rotating crews working 12-hour shifts. There are two 12-hour shifts scheduled, consisting of a day shift and a night shift. A number of duties only require work during the dayshift hours. For these duties, two crews rotate to provide seven day-a-week day-shift coverage.

For the rotating mine operations crews, approximately three hours are lost per day to standby time, inclusive of two hours for breaks, 20 minutes for fuelling 20 minutes for shift change, and 20 minutes for blast delay (Table 13-3).

Over a year, approximately 13 days are lost to poor weather conditions, predominantly in the wintertime. It is assumed that the equipment is manned but delayed during these weather events.

Table 13-3: Gross Operating Hours per Year

Time Definition	Hydraulic Shovel	Front End Loaders	Trucks	Drills
Calendar Time				
Calendar Days	365	365	365	365
Shifts per day	2	2	2	2
Shift length (h)	12	12	12	12
Calendar Time (h/a)	**8,760**	**8,760**	**8,760**	**8,760**
Available Time				
Availability	85%	85%	85%	85%
Down time (h/a)	1,314	1,314	1,314	1,314
Available Time (h/a)	**7,446**	**7,446**	**7,446**	**7,446**
Gross Operating Time				
Operating Standby	-	-	-	-
Lunch & breaks (min/day)	120	120	120	120
Fueling (min/day)	20	20	20	30
Shift change (min/day)	20	20	20	20
Meetings (min/day)	20	20	20	20
Blasting (min/day)	-	-	-	20
Weather (d/a)	13	13	13	13
Operating Standby (h/a)	**1,196**	**1,196**	**1,196**	**1,351**
Gross Operating Hours (h/a)	**6,250**	**6,250**	**6,250**	**6,095**

Accounting for standby time and weather delays, equipment accumulates on average 6,250 gross operating hours (GOH) per year in the example above. For productivity calculations, it is assumed that following preproduction, the trucks and shovels are in a productive cycle of approximately 50 minutes each hour, or 83% of the time. For drills, the productive utilization is lower and in the range of 65% based on benchmarking with similar mines.

As with mine operations, mine maintenance is scheduled to work a 24/7 schedule to allow for continuous maintenance coverage. However, the majority of planned maintenance work is done during the day shift with a skeleton crew scheduled for the night shift.

Blasting is only scheduled during the day shift. Two blasting crews rotate on a 12-hour day shift, for seven day-a-week coverage.

13.8 Mining Equipment

A conventional owner-operated truck fleet utilizing a combination of hydraulic shovels and front-end loaders (FELs) is planned. The truck fleet consists of large trucks for waste stripping and for mining the ore zones. The trucks are diesel powered with a combined capacity to mine a maximum of 40 Mt/a operating on a combination of 5 m and 10 m benches. The loading fleet is also diesel powered. Blasting is contract performed with explosives manufactured off-site and delivery to site by trucks.

Equipment requirements are estimated annually over the life of the mine. Equipment sizing and numbers are based on the mine plan, the operational factors shown in Table 13-4, and a twenty-four hour per day, seven day a week work schedule.

Table 13-4: Equipment Utilization and Efficiency

Description	Average Mechanical Availability (%)	Efficiency (%)
171 mm Production Drill	83%	65%
265 t/15 m^3 Hydraulic Face Shovel	85%	83%
124 t/12 m^3 Front End Loader	85%	83%
144 t Haul Truck	85%	83%
41t Articulated Truck	86%	83 %
26 t/5 m^3 Front End Loader	86%	83%
68 t/4 m^3 Hydraulic Excavator	86%	83%
35 t/1.4 m^3 Hydraulic Excavator	85%	83%
74 t/455 kW Track Dozer	85%	83%
50 t/419 kW Rubber Tired Dozer	85%	83%
40 t Articulated Sand Truck	85%	83%
33 t/217 kW Motor Grader	85%	83%
35,000-litre water truck	85 %	83%
41t Articulated Fuel/Lube Truck	85%	83%

13.8.1 Blasting

Blasting operations are contracted to a blasting explosives provider who is responsible for the blast design, loading, stemming, and initiation. Bulk emulsion is manufactured off-site. The explosive provider transports explosive material from Fairbanks, Alaska. The explosive is stored in four explosive silos with 80-ton capacity each: two silos for storing Ammonium Nitrate and the other two for storing emulsion produced on site.

In performing the explosive services, the blasting contractor is proposing to provide:

- Two five person blasting crews providing seven day a week coverage
- Transportation of the explosive from Fairbanks to site

Additionally, two (2) Mobile Manufacturing Unit (MMU) trucks and one stemming truck are provided by the mine. The MMU explosive trucks delivers a bulk emulsion product when required.

Two types of Heavy ANFO blend (HA) are used: 70% emulsion/30% Ammonium Nitrate Fuel Oil (ANFO) for wet material, and 30% emulsion/70% ANFO for dry material, with specific gravity of 1.25 g/cm^3 and 1.23 g/cm^3, respectively.

The wall control design patters are based on the 2022 Arctic Drill and Blast Study recommendations There are three main types of geotechnical zones within the pit: one group in the North and East walls, where the slope design is governed by the shallower foliation, with IRA lower than 38° and with geotechnical berms every 60 m, other group with IRA of between 38° and 47°, and other group with IRA greater than 47°. Pre-split blasting only used for the third group with IRA greater than 47°. For other zones, multiple stab trim holes are used for face outlining.

Table 13-5 shows the design parameters for production and wall control blasting.

Table 13-5: Blasting Design Input

Description	Units	Waste	Ore Dry	Waste Wet	Ore Wet	Trim 1 (1)	Trim 2 (2)	Trim 3 (3)	Pre-split (3)
Rock Density	t/m^3	2.76	3.22	2.76	3.22	2.76	2.76	2.76	2.76
Explosive		HA37	HA37	HA73	HA73	HA37	HA37	HA37	HA37
Explosive Density	kg/m^3	1,230	1,230	1,250	1,250	1,230	1,230	1,230	1,230
Bench Height	m	10	10	10	10	8.3	10	10	10
Hole Diameter	m	0.171	0.171	0.171	0.171	0.171	0.171	0.171	0.171
Burden	m	5.1	4.2	5.1	4.2	6.0	5.3	3.7	1.5
Spacing	m	5.9	4.8	5.9	4.8	5.5	4.8	3.9	2.5
Sub drill	m	1.0	1.0	1.0	1.0	0.0	0.1	0.0	0.0
Stemming	m	3.5	4.0	3.5	4.0	3.8	5.4	5.9	9.6
Drill length per hole	m	11	11	11	11	8.3	10	10	10
Rock Volume per hole	m^3	301	202	301	202	273	258	143	38
Rock Tonnage per hole	t	830	649	830	649	755	712	394	104
Rock Tonnage	t/m	75	59	75	59	91	70	39	10
Explosive Column	m	7.5	7.0	7.5	7.0	4.5	4.7	4.1	0.4
Weight of explosive	kg	212	198	215	201	128	133	116	13
Powder Factor (by rock volume)	kg/m^3	0.70	0.98	0.72	1.00	0.47	0.52	0.81	0.33
Powder Factor	kg/t	0.26	0.30	0.26	0.31	0.17	0.19	0.29	0.12

1. Geotechnical zones with IRA below 38 degrees and 60 m bench height. Domains 2L-E, 2L-W and 4L.
2. Geotechnical zones with IRA between 38 and 40 degrees and bench height shorter than 20 m. Domain 3 and 4L.
3. Geotechnical zones with IRA greater than 47 degrees and bench height shorter than 20 m. Domain 2U and 4U.

Although material is mined on a combination of 5 and 10 m benches, all material is drilled and blasted on 10 m benches.

The powder factor for ore is 0.3 kg/t and 0.26 kg/t for waste. For trim, variable powder factors were used depending on the characteristics of the geotechnical zone. Table 13-6 provides a summary of the material blasted on an annual basis.

Table 13-6: Material Blasted Quantities

Period	Powder Factor (kg/t)	Waste Blasted (kt)	Ore Blasted (kt)	Trim Blasted(1) (kt)	Total Blasted (kt)
Year -2	0.26	26,157	493	750	27,400
Year -1	0.25	30,262	283	4,499	35,044
Year 1	0.25	29,404	2,235	2,438	34,077
Year 2	0.25	25,754	3,650	4,786	34,190
Year 3	0.25	21,939	3,650	5,426	31,015
Year 4	0.25	22,156	3,650	4,194	30,000
Year 5	0.25	21,181	3,650	4,669	29,500
Year 6	0.25	19,687	3,650	5,226	28,563
Year 7	0.25	20,176	3,650	4,237	28,063
Year 8	0.25	15,976	3,651	3,873	23,500
Year 9	0.26	16,856	3,650	3,494	24,000
Year 10	0.26	15,809	3,650	2,113	21,572
Year 11	0.26	12,217	3,650	3,133	19,000
Year 12	0.25	7,949	3,650	3,680	15,280
Year 13	0.27	-	3,529	2,126	5,655
Total	**0.25**	**285,524**	**46,691**	**54,643**	**386,859**

1. Includes Pre-split.

Blasting products consumed on an annual basis are shown in Table 13-7.

Table 13-7: Explosives Quantities

Period	ANFO (t)	Emulsion Bulk (t)	Detonators Pyrotechnic (Units)	Boosters 0.45kg (Units)
Year -2	3,147	3,669	33,301	33,301
Year -1	3,980	4,416	42,917	42,917
Year 1	3,932	4,491	42,135	42,135
Year 2	3,958	4,369	43,090	43,090
Year 3	3,596	3,895	39,403	39,403
Year 4	3,493	3,851	38,027	38,027
Year 5	3,438	3,752	37,537	37,537
Year 6	3,331	3,587	36,557	36,557
Year 7	3,285	3,594	35,920	35,920
Year 8	2,767	3,004	30,543	30,543

Period	ANFO (t)	Emulsion Bulk (t)	Detonators Pyrotechnic (Units)	Boosters 0.45kg (Units)
Year 9	2,836	3,105	31,369	31,369
Year 10	2,568	2,878	28,298	28,298
Year 11	2,269	2,457	25,538	25,538
Year 12	1,847	1,917	21,310	21,310
Year 13	746	722	9,206	9,206
Total	**45,193**	**49,708**	**495,151**	**495,161**

13.8.2 Drilling

Throughout the project life, drilling is required for both ore control and production blasting. Rock fragmentation achieved through blasting is the overriding design criteria for the drill hole pattern design. The blast hole drilling design from Section 13.8.1 with the drill penetration rates described below are used to estimate drilling requirements.

Drill penetration is a function of bit size, bit load, drilling method, and rock strength properties. Wood relied on SRK's 2022 Pre-Feasibility Study Geotechnical and Hydrogeological Assessment report. SRK completed unconfined compressive testing (UCS) and point load testing on Arctic's primary rock types. The results of the uniaxial compressive tests for the primary ore hosting rocks (lithological units) are shown in Table 13-8. The weighted average UCS value for the ore hosting rocks is 83 MPa and 59 MPa for the waste areas.

Table 13-8: Rock Type Weight and UCS

Lithology	UCS (MPa)	Samples (Units)
Ore Hosting		
Black to Grey Schist (GS)	97	14
Massive Sulphide (MS)	56	6
Semi-massive Sulphide (SMS)	68	3
Average Ore	**83**	**23**
Waste		
Aphanitic Metarhyolite (AMR)	88	5
Chlorite Schist (CHS)	20	2
DIKE	62	2
Debris Flow (DM)	71	3
Button Schist (MRP)	73	9
Quartz-Felspar-Mica Schist (QFMS)	64	5
Quartz- Mica Schist QMS	50	5
QMSP	62	2
Talc Schist (TS)	1	2
CHTS	3	2
Average Waste	**59**	**37**

Instantaneous penetration rates (IPR) were calculated based on UCS data, rock factor and the Workman Szumanski (1996, 1997) formula. Using the Workman Szumanski (1996, 1997) formula Penetration Rate shown in Equation 13 1, and the inputs in Table 13-9, the instantaneous penetration rate (P) for waste and ore were estimated at 67 m/h and 45 m/h, respectively.

Equation 13 1: Workman, Szumanski (1996,1997) Formula

$$P = (RF - 28\ log_{10}Sc)\left(\frac{P}{D}\right) \times \frac{RPM}{17.6}$$

Table 13-9: Workman, Szumanski (1996,1997) Inputs

Item	Waste	Ore
RF - Rock factor	148	120
Sc – UCS (MPa)	59	83
P – pulldown (lb)	50,000	50,000
D - Bit Size (mm)	171	171
P/D - Pulldown (kg/m)	133,000	133,000
RPM – Revolutions per minute	90	90
P - Instantaneous Pen Rate (m/h)	**67**	**45**

The total drilling cycle time accounts for drilling time, tramming time between holes, setup time, and bit change time. Because the CAT MD6250main drill equipment can drill to a 13.6 m depth, no time is required for drill steel addition. The average penetration rate shown in Table 13-10 also assumes 65% productive utilization, or 39 minutes out of 60 minutes that the drill is in the drilling cycle.

Table 13-10: Drill Penetration Rates

Material Type	IPR	Drill Hole Length	Tram Time	Setup Time	Bit Change	Total Cycle	NOH Pen Rate	GOH Pen Rate[1]
	(m/h)	(m)	(min)	(min)	(min)	(min)	(m/h)	(m/h)
Waste	67	11	0.55	3.00	0.38	13.81	47.79	31.1
Ore	45	11	0.55	3.00	0.38	18.61	35.47	23.1

[1]Assumes 65% efficiency.

Table 13-11 shows the drill requirements, the meters drilled, the hours operated, and the average penetration rate by period. During Year -2, two production drills are required. From Year -1 to Year 7, mining requires three production drills. Following Year 7, drill requirements drop to two and after Year 12, drill requirements drop to one along with a drop in of total tonnes mined. Penetration rates average 29.8 m/h over the LOM.

Table 13-11: Drill Requirements and Performance

Period	Drills Required (Units	Meters Drilled (m)	Gross Operating Hours (GOH)	Average Penetration Rate (m/h)
Year -2	2	375,812	12,795	29.4
Year -1	3	470,586	15,783	29.8
Year 1	3	469,235	15,811	29.7
Year 2	3	472,384	16,299	29.0
Year 3	3	429,789	14,661	29.3
Year 4	3	418,968	14,089	29.7
Year 5	3	413,125	13,742	30.1
Year 6	3	400,873	13,206	30.4
Year 7	3	396,514	13,008	30.5
Year 8	2	335,237	11,659	28.8
Year 9	2	344,926	11,616	29.7
Year 10	2	313,880	11,616	27.0
Year 11	2	278,087	9,218	30.2
Year 12	2	228,627	7,708	29.7
Year 13	1	97,266	3,007	32.3
Total		**5,445,309**	**184,219**	**29.6**

The 2022 Arctic Drill and Blast study (Floyd, 2022) recommends 90° presplit for zones with inter-ramp angle greater than 47°, with the same drill hole diameter than the production drill holes. Therefore, only rotary drills are considered at this point. The study also suggests that in case the 90° presplit is not suitable according to field testing, smaller diameters, inclined holes, and a top head hammer (THH) drill with a 121 mm bit will be needed.

13.8.3 Loading

Wood completed an equipment trade-off study based in quantitative and qualitative characteristic of the equipment and vendor's quotations received. From the trade-off study, the primary loading units selected are two 15 m^3 hydraulic shovels. After Year 11 only one hydraulic shovel is required. To assist the hydraulic shovels, two 12 m^3 front end loaders (FEL) are scheduled until Year 7 of production, dropping to one until the end of the LOM. The FEL is also used for stockpile rehandling, all of which is scheduled in Year 1. Loading requirements are shown in Table 13-12.

Table 13-12: Loading Requirements and Performance

	15 m^3 shovel		12 m^3 FEL	
	Number	Productivity (t/GOH)	Number	Productivity (t/GOH)
Year -2	2	1,241	2	829
Year -1	2	1,698	2	995
Year 1	2	1,718	2	1,006
Year 2	2	1,673	2	1,003
Year 3	2	1,690	2	750

	15 m³ shovel		12 m³ FEL	
	Number	Productivity (t/GOH)	Number	Productivity (t/GOH)
Year 4	2	1,643	2	738
Year 5	2	1,630	2	729
Year 6	2	1,606	2	714
Year 7	2	1,576	2	698
Year 8	2	1,501	1	813
Year 9	2	1,531	1	834
Year 10	2	1,384	1	734
Year 11	2	1,224	1	615
Year 12	1	1,532	1	311
Year 13	1	675	1	233
Average		**1,488**		**733**

The 15 m³ shovel five-pass loads the 144 t truck in approximately three minutes. The LOM production rate averages 1,488 dry t/GOH, compared to the maximum productivity of 1,923 dry t/GOH shown in Table 13-13. The peak productivity scheduled for the shovel occurs in Years 1, when it is scheduled at 1,718 dry t/GOH.

Table 13-13: Hydraulic Shovel Productivity

Loader	Unit	15 m³ Shovel
Truck		**144 t Truck**
Bucket Capacity	m³	15
Bucket Capacity	t	30
Truck Capacity	m³	94
Truck Capacity	t	144
Insitu Bulk Density	t/m³	2.8
Bulk Factor		1.3
Loose Density	t/m³	2.15
Moisture	%	3%
Fill Factor		0.86
Effective Bucket Capacity	m³	12.97
Wet/Loose Density	t/m³	2.22
Tonnes/Pass	t	28.80
Theoretical Passes (Volume)		7.25
Theoretical Passes (Weight)		5.00
Actual Passes		5
Truck Load	m³	64.85
Truck Load	t	144.0
Truck Fill % (Volume)		69%

Loader	Unit	15 m³ Shovel
Truck		144 t Truck
Truck Fill % (Weight)		100%
Loader Cycle Time	seconds	35
Loader Spot Time	seconds	30
Load Time per Truck	seconds	170
Maximum Truck Loads per hour		21.18
Maximum Productivity	(wet t/adj. NOH)	3,049
Maximum Productivity	(wet t/NOH)	2,379
Maximum Productivity	(wet t/GOH)	1,982
Maximum Productivity	(dry t/GOH)	1,923Sh
Maximum Productivity	(wet t/d)	34,460
Maximum Productivity	(dry t/a)	12,200,469
Maximum Productivity	(dry t/d)	33,426

The FEL seven-pass loads the 144 t truck in approximately three and a half minutes. The LOM production rate averages 797 tdry/GOH, compared to the maximum productivity of 1,139 tdry/GOH shown in Table 13-14. The peak productivity scheduled for the FELs occurs in Year 1, when the FEL is scheduled at 1,008 tdry/GOH.

Table 13-14: Front End Loader Productivity

Loader	Unit	12 m³ FEL
Truck		144 t Truck
Bucket Capacity	m³	11.7
Bucket Capacity	t	21.0
Truck Capacity	m³	94.0
Truck Capacity	t	144.0
Insitu Bulk Density	t/m³	2.8
Bulk Factor		1.3
Loose Density	t/m³	2.15
Moisture	%	3.0%
Fill Factor		0.79
Effective Bucket Capacity	m³	9.26
Wet/Loose Density	t/m³	2.22
Tonnes/Pass	t	20.57
Theoretical Passes (Volume)		10.15
Theoretical Passes (Weight)		7.00
Actual Passes		7.00
Truck Load	m³	64.85
Truck Load	t	144.0
Truck Fill % (Volume)		69%

Loader	Unit	12 m³ FEL
Truck		144 t Truck
Truck Fill % (Weight)		100%
Loader Cycle Time	seconds	42
Loader Spot Time	seconds	35
Load Time per Truck	seconds	287
Maximum Truck Loads per hour		12.54
Maximum Productivity	(wet t/adj. NOH)	1,806
Maximum Productivity	(wet t/NOH)	1,409
Maximum Productivity	(wet t/GOH)	1,174
Maximum Productivity	(dry t/GOH)	1,139
Maximum Productivity	(wet t/d)	20,246
Maximum Productivity	(dry t/a)	7,168,141
Maximum Productivity	(dry t/d)	19,639

In addition to the primary loading units, a 5 m^3 FEL is paired with two 41 t articulated trucks throughout the mine life for bench development/pioneering work and winter snow removal. A 3.8 m^3 excavator is also used to maintain haul roads and scale the pit walls as needed.

13.8.4 Hauling

From the TCO analysis, the primary hauling unit selected for ore and waste mining is a mechanical drive truck with a payload capacity of 144 t wet, assuming a standard body with a full set of liners. The dry capacity is estimated at 140 t, assuming 3% moisture and carry back.

Wood estimated truck requirements on a period-by-period basis using travel distances from a road network developed within Minesight®. Haul segment distances were reported for each material type from their location on a mining bench to their final destination. Assuming 2% rolling resistance for haul roads, travel speeds were estimated from the manufacture's performance curves, and applied to each haul segment to estimate travel time. To better reflect actual speeds in an operating environment, the truck speeds were speed-limited according to Table 13-15.

Table 13-15: Truck Speed Limits

Segment	Loaded (kph)	Empty (kph)
Maximum Overall Speed	45	45
Maximum Speed within 100 m of dump face and shovel	25	25
Maximum Speed on a 10% downhill grade	30	30

Table 13-16 provides the adjusted speeds and fuel burns by haul grade used to estimate truck requirements and diesel fuel usage for the 144 t trucks.

Table 13-16: Truck Speed Limits

Actual Grade (%)	Haul Loaded (kph)	Return Empty (kph)	Fuel Loaded (l/h)	Return Empty (l/h)
-10%	30	30.0	19.1	19.1
10%	11.5	27	190.8	190.8
Flat Ex-pit	45	45	124.9	72.8
Flat In-pit	25	25	162.0	72.7
Idle	-	-	19.1	

Truck requirements by period are shown in Table 13-17 for 144 t trucks, together with the average one-way haul distance, average fuel consumption, and average truck productivity. Fourteen trucks are commissioned during pre-production. During Year 1 the fleet is ramped up to 15, reaching its peak. After Year 9, truck requirements drop to 14. Following Year 10, the 144 t truck requirements decline with declining mining rates.

Table 13-17: Truck Requirements and Performance

	Trucks Required	Average one-way Haul Distance	Average Fuel Burn	Average Truck Production
	#	**(m)**	**l/GOH**	**t/GOH**
Year -2	11	2,693	62	384
Year -1	14	2,558	63	387
Year 1	15	3,127	64	362
Year 2	15	2,953	68	356
Year 3	15	2,874	69	346
Year 4	15	3,034	73	318
Year 5	15	3,117	76	316
Year 6	15	2,962	77	310
Year 7	15	3,131	80	305
Year 8	15	3,353	82	275
Year 9	15	3,734	85	262
Year 10	14	3,847	88	252
Year 11	14	4,039	91	222
Year 12	12	4,206	92	209
Year 13	4	3,449	87	232
Average	-	**3,173**	**75**	**317**

13.8.5 Support

Support equipment includes excavators, track dozers, rubber-tired dozers (RTDs), sand trucks, graders, water trucks, fuel/lube trucks, and water trucks. The major tasks for the support equipment include:

- Bench and road maintenance

- Shovel support/clean-up
- Blasting support/clean-up
- WRF maintenance
- Stockpile construction/maintenance
- Road building/maintenance
- Pioneering and clearing work
- Field equipment servicing

A description of each support equipment fleet follows:

- 35 t Excavator – One excavator scheduled throughout the mine life. Its primary functions are to support dewatering, maintain pit drainage, break rocks utilizing the rock breaker attachment, and backup the 5 m^3 FEL.
- 72t/455 kW track dozers are estimated at 0.5 dozers per production blast hole drill and production loading unit. The dozer fleet peaks at 4 machines in pre-production. Their primary functions are to maintain pit floors, maintain dumps and stockpiles, build pit roads, and clean final pit walls. Due to limited mobility, the 72 t dozers are transported between working areas using a 90 t capacity transport trailer. The transport is also used to transport the 65 t drills and the 35 t excavator.
- 47t/419 kW rubber-tired dozer requirements are estimated at approximately one RTD per hydraulic shovel. Their primary function is to maintain shovel floors, provide drill pattern clean-up, clear rock spillage, and provide backup dump and stockpile maintenance. At peak, two-wheel dozers support two 15 m^3 shovels and associated mining areas.
- 41t Articulated Sand Truck – One articulated truck is fitted with a sander and used to support winter operations throughout the LOM.
- 227 kW motor graders are estimated at approximately one grader per 8 trucks. Their primary function is to maintain roads, dump areas, and pit areas. Two graders are scheduled along the life of the mine, dropping to 1 the last year, Y12.
- Two 35,000 litre water trucks are estimated to support the mine operation. During the winter season, from October to April, water trucks are lightly scheduled. They are primarily used for watering the drills and for fire patrol; nonetheless, even during the winter season roads become dusty. During May to September, when dust suppression requirements are at their highest, the water trucks are fully scheduled.
- Fuel/lube truck requirements are estimated at a ratio of one lube truck per 10 pieces of tracked field equipment. They are used to fuel and service shovels and other tracked field equipment. At peak, the mine operates one fuel/lube truck.
- One 1.4 m^3 backhoe is scheduled throughout the mine life. Its primary functions are to support dewatering, maintain pit drainage, and break rocks utilizing the rock breaker attachment.

Support equipment requirements are shown in Table 13-18.

Table 13-18: Support Equipment

Period	35 t Excavator	Bulldozer - 455 kW	Wheel dozer - 419 kW	Motor grader – 217 kW	Sand Truck - 41 t	Water Truck - 35000 l	Fuel / Lube Truck	Backhoe - 1.4 m³
Year -2	1	4	2	2	1	2	1	1
Year -1	1	4	2	2	1	2	1	1
Year 1	1	4	2	2	1	2	1	1
Year 2	1	4	2	2	1	2	1	1
Year 3	1	4	2	2	1	2	1	1
Year 4	1	4	2	2	1	2	1	1
Year 5	1	4	2	2	1	2	1	1
Year 6	1	4	2	2	1	2	1	1
Year 7	1	3	2	2	1	2	1	1
Year 8	1	3	2	2	1	2	1	1
Year 9	1	3	2	2	1	2	1	1
Year 10	1	3	2	2	1	2	1	1
Year 11	1	2	1	2	1	2	1	1
Year 12	1	2	1	2	1	2	1	1
Year 13	1	2	1	2	1	2	1	1

13.8.6 Auxiliary

To support mine maintenance and mine operation activities, a fleet of auxiliary equipment is required. The types and numbers of auxiliary equipment are listed in Table 13-19in five-year increments.

Table 13-19: Auxiliary Equipment

Equipment	Year 1	Year 5	Year 13
Mine Maintenance			
Truck Mounted 40 t Crane	1	1	1
100 t Rough Terrain	1	1	1
5t Rough Terrain Forklift	2	2	2
10t Forklift	2	2	2
Mechanic Service Truck	3	3	3
Small Fuel/Lube truck	1	1	1
55 kW Skid Steer	1	1	1
Flatbed Truck	2	2	2
45t Telehandler	1	1	1
Mine Operations			
Small backhoe/loader	1	1	1
Hydraulic hammer/impactor	1	1	1
90 t Lowboy	1	1	1
Compactor	1	1	1

Equipment	Year 1	Year 5	Year 13
Light Plant	11	8	5
Transport Tractor	1	1	1
Tire Handler Truck	1	1	1
3/4 ton Pickup	4	4	4
1 ton Pickup	5	4	2
Crew Bus	5	4	2
Fleet Management System	1	1	1
Mine & Geology Software	10	10	10
Heavy ANFO (blend) Truck	2	2	2
Stemming Truck	1	1	1
Laser Scan	2	2	2
Survey Drones	2	2	2
Total Station System	**1**	**1**	**1**

13.8.7 Labour

Wood developed the staffing plan based on the primary assumptions that the owner will perform the equipment maintenance and the blasting operations will be subcontracted.

At peak staffing levels in Year 1, the mine department includes 205 employees including 31 salaried and 174 hourly. The blasting contractor estimated that they will have 10 people on site on two crews including one supervisor, two blaster, two MMU operators per crew.

The salaried staff is estimated based on level of effort. The hourly staff is estimated on both level of effort and on mobile equipment coverage.

Mine maintenance hourly staffing is estimated by multiplying a ratio of mechanic time required by machine-operated hours. Table 13-20 provides a listing of the ratios by machine. Wood used its historical database to estimate ratios for the mine equipment.

Table 13-20: Ratio Maintenance Hours to Equipment Operated Hours

Equipment	Ratio Maintenance Hours to Equipment Operated Hours
171 mm Production Drill	0.90
265 t/15 m^3 Hydraulic Face Shovel	0.90
124 t/12 m^3 Front End Loader	0.90
144 t Haul Truck	0.52
41t Articulated Truck	0.52
26 t/5 m^3 Front End Loader	0.90

Equipment	Ratio Maintenance Hours to Equipment Operated Hours
68 t/4 m^3 Hydraulic Excavator	0.90
35 t/1.4 m^3 Hydraulic Excavator	0.50
74 t/455 kW Track Dozer	0.50
50 t/419 kW Rubber Tired Dozer	0.50
40 t Articulated Sand Truck	0.50
33 t/217 kW Motor Grader	0.50

13.9 Geotechnical and Hydrogeological Assessments

SRK reviewed the previous work and recommendations, identified potential risks and opportunities highlighted in the previous studies and designed the field program and laboratory testing to update the structural, geotechnical and hydrogeological models to support this pre-feasibility study. Detailed slope stability assessments were completed using the previously designed pit shells, dated Nov 2019, and updated slope design recommendations were provided with a focus on the slope stability risk related to potential sliding failure along talc zones in the northeast wall. Wood produced the open pit mine designs that include interim and final phases using these recommendations. SRK completed a geotechnical review of the updated mine designs, which is discussed in Section 13.10. The estimated annual average pit inflow, discussed in Section 13.9.7, has been updated based on the review of the new open pit designs.

13.9.1 Data Sources

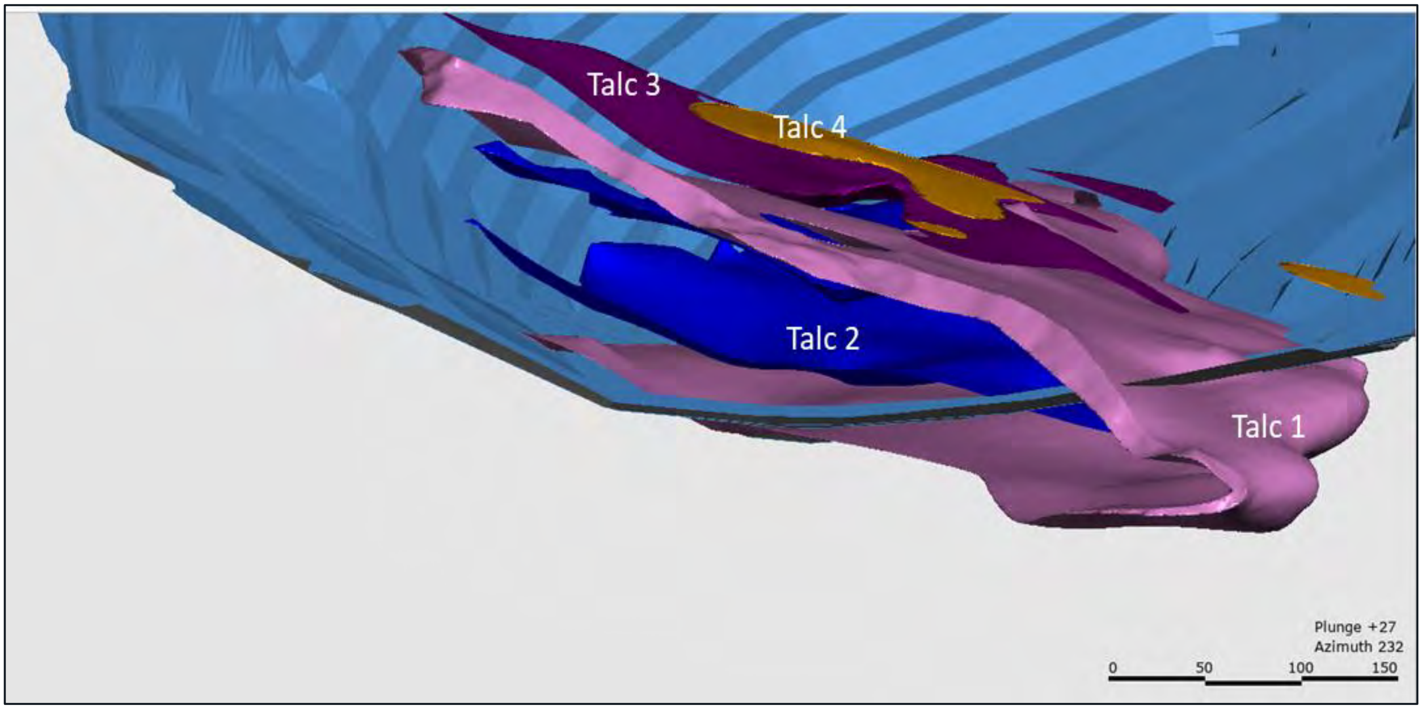
SRK has conducted geotechnical and hydrogeological drilling programs to collect information to support the Arctic Project. The locations of the drill holes are presented in Section 7.2.4.

13.9.2 Geology and Structure

The talc alteration occurs as continuous to semi-continuous massive bands that locally host economic mineralization many meters thick to much thinner (2 to 20 cm) bands parallel to the dominant foliation within more competent quartz-mica and quartz-chlorite mica schists. Both occurrences represent potential weak or slip foliation parallel surfaces. Trilogy developed talc wireframes delineating the spatial distribution and extent of the main talc-rich horizons.

These wireframes have been updated by SRK as the talc-rich horizons have been further delineated by additional drilling completed in 2021 (Figure 13-12). The focus of the update was to refine the geometry of talc layers that extend behind the northeast and east walls of the designed open pit. The entire talc model is planned to be updated in next phase of the engineering study with the results of the 2022 drill program. These talc wireframes were used in the geotechnical and hydrogeological assessments.

Figure 13-12: Isometric View (looking southeast) of Talc Zone Domain Modelled by SRK (2022) Using Leapfrog on the EoY12 design pit

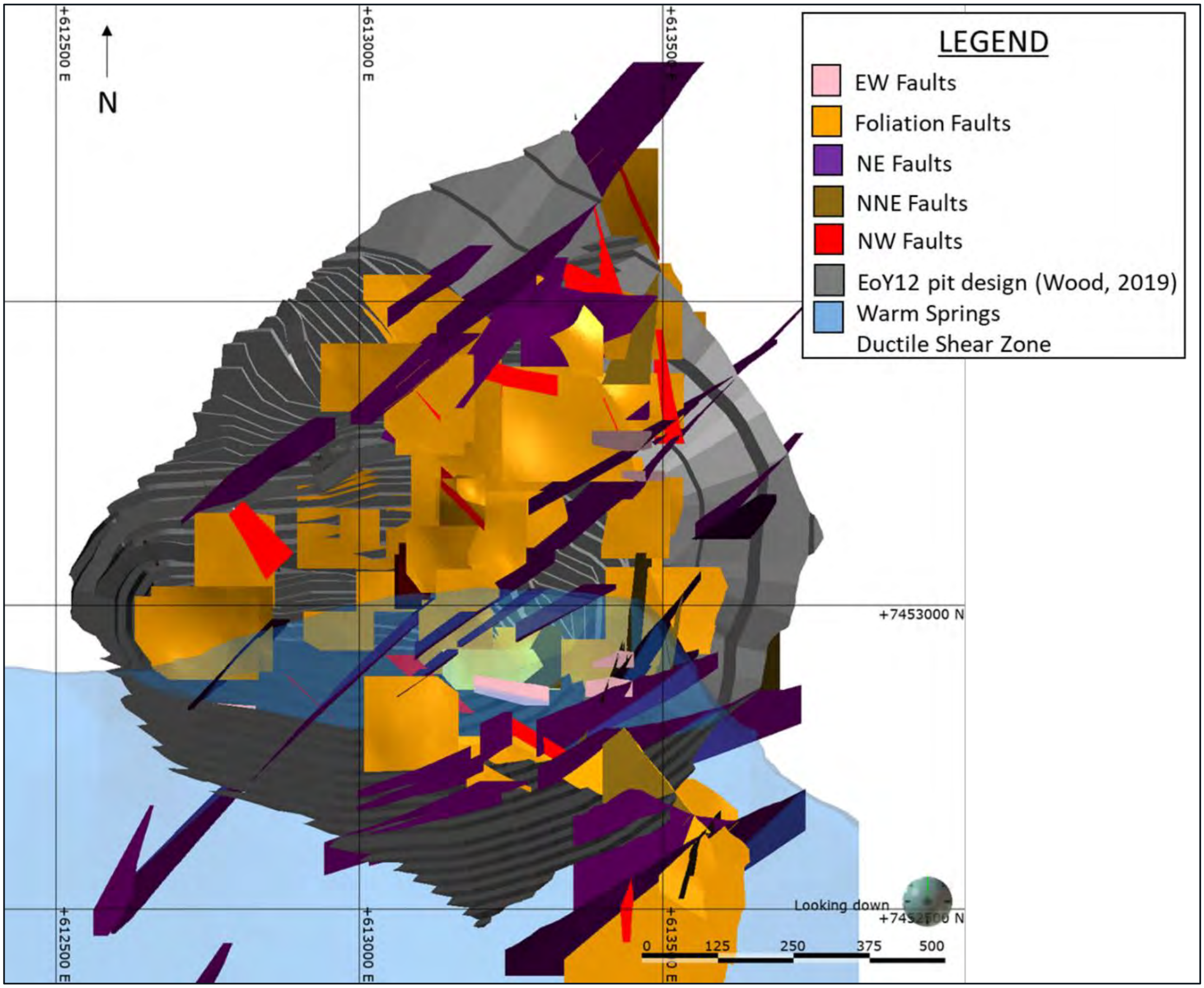


Note: Only talc layers that extend behind the final pit wall are shown. Source: Wood, 2019.

Pervasive weathering is present in the upper levels of the Arctic deposit. SRK has reviewed the geotechnical data and core photographs in order to define a base of weathering isosurface that represented the boundary of the upper, more pervasive weathering.

In 2019, SRK updated the 3-D structural model by integrating new structural drill hole data into the model, refining existing structures and modelling additional faults. The “structural matrix”, which provides information on the geometry, physical properties and confidence of major and minor structural features, was also updated. The final structural model is shown in Figure 13-13.

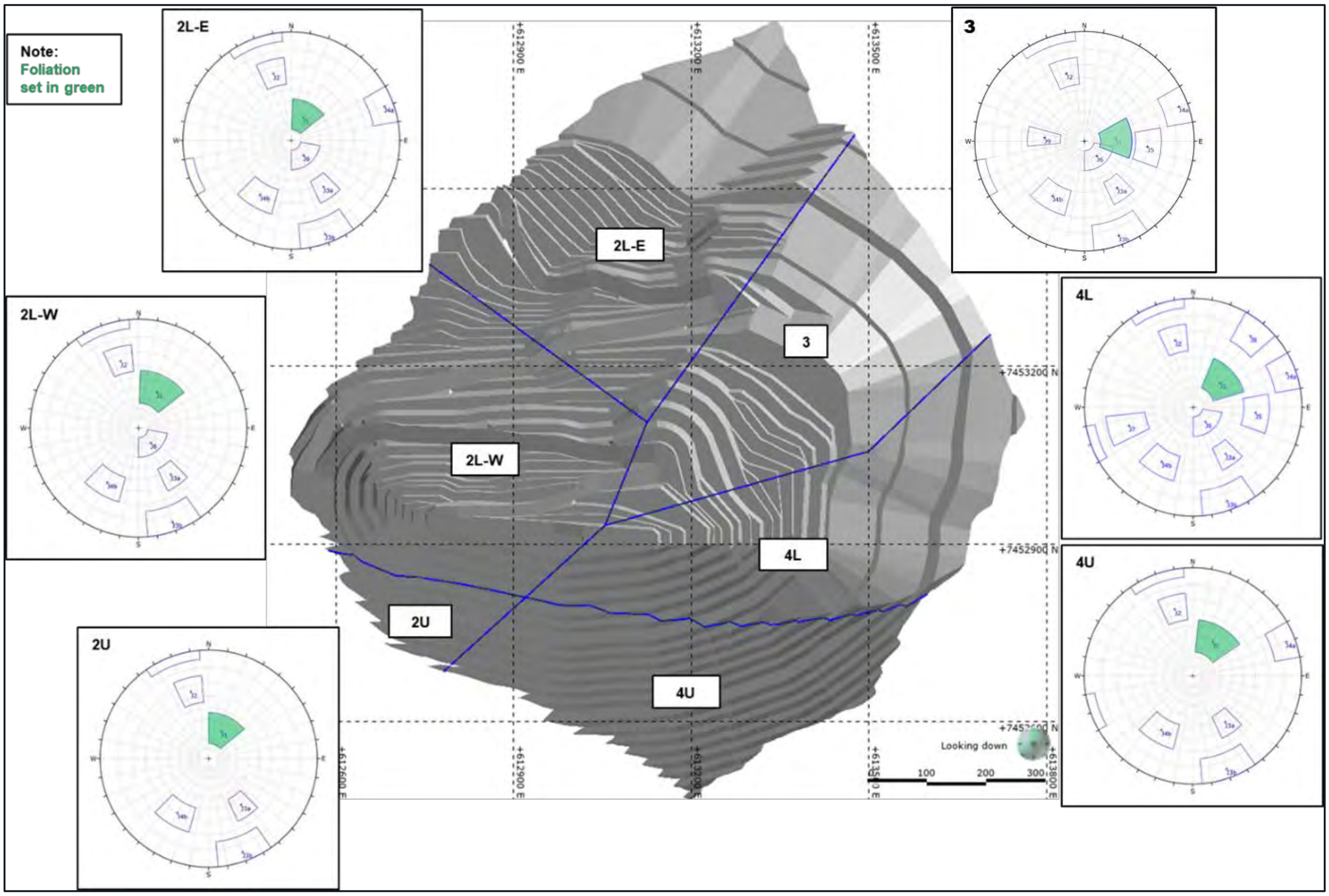
Figure 13-13: SRK Structural Model (SRK, 2019) used in the Slope Stability Analysis on the EoY12 Design Pit



Source: SRK, 2022.

Six structural and geomechanical domains were identified (Figure 13-14). These domains, each containing representative discontinuity sets and major structures, formed the basis of the kinematic assessment.

Figure 13-14: Location Plan, Structural and Geomechanical Domains



Source: SRK, 2022.

13.9.3 Geotechnical Laboratory Testing

The historical open pit geotechnical laboratory testing database includes uniaxial compressive strength (UCS) tests, modulus tests for Young's Modulus and Poisson's Ratio, triaxial compressive strength (TCS) tests, direct shear (DS) tests, and tilt table tests. The testing programs were directed by SRK (2015, 2016, 2019, 2021), BGC (2012), and URSA (1998) and completed at accredited laboratories in North America that follows industry best practices. SRK reviewed the testing database and utilized reliable and valid test results to derive representative compressive and shear strength values of the intact rock and discontinuities of the Arctic rock mass.

Strength anisotropy is known to exist as a result of foliation fabric through the various lithologies in the Arctic rock mass. UCS and TCS tests that have failed primarily along foliation fabric were rejected to minimize the influence of anisotropy to the intact rock strength. The mean and standard deviation values of UCS test results considered for geotechnical assessments are presented in Table 13-21.

The shear strength of discontinuities and the foliation rock fabric were assessed using DS and tilt table tests (Table 13-22). The test results suggest a variation in the friction angle between different lithologies and discontinuity orientations. After examining the laboratory test results, the peak friction angles of foliation fabric and other discontinuities, respectively, were characterized as 24° and 27°. These values represent the Arctic rock mass excluding the Talc Zone domain.

The Talc Zone domain consisting of talc schist (TS) and chlorite-talc schist (ChTS), is the weakest rock type (outside of fault zones) observed at the Arctic deposit. The domain is characterized by low intact rock strength, well developed fabric and low shear strength discontinuity surfaces. The shear strength, extent and persistence of the unit is considered a risk for pit slope stability and was the target of the 2021 and 2022 geotechnical drilling programs.

A geotechnical laboratory testing program dedicated to determining the shear strength of the weakest talc layers within the Talc Zone rock mass domain was completed in 2021. The program included both rock and soil TCS and DS tests. Available historical shear strength tests on talc were reviewed using before and after test photographs, and data points interpreted as invalid, or outliers, were removed from the analysis. The assessment indicates that the talc has an approximate peak friction angle of 11° and cohesion of 30 kPa (Figure 13-15). Additional laboratory tests utilizing core samples from the 2022 drilling program is in progress.

Table 13-21: Summary of Unconfined Compressive Strength by Lithological Units

Litho Code	Unconfined Compressive Strength						
	Valid Results Count	UCS (MPa)		Young's Modulus (GPa)		Poisson's Ratio	
		Average	Std Dev	Average	Std Dev	Average	Std Dev
AMR	5	88	61	22	9	0.19	0.06
CHS	2	20	-	4	-	0.23	-
DIKE	2	62	-	19	-	0.25	-
DM	3	71	-	22	-	0.25	-
GS	14	97	22	24	11	0.14	0.06
MRP	9	73	19	18	-	0.21	-
MS	6	56	20	24	11	0.25	0.04
QFMS	5	64	15	19	6	0.25	0.05
QMS	5	50	7	15	6	0.21	0.03
QMSp	2	62	-	20	-	0.23	-
SMS	3	68	-	24	-	0.21	-
TS	2	1*	-	0.9	-	-	-
ChTS	2	13*	-	3	-	-	-

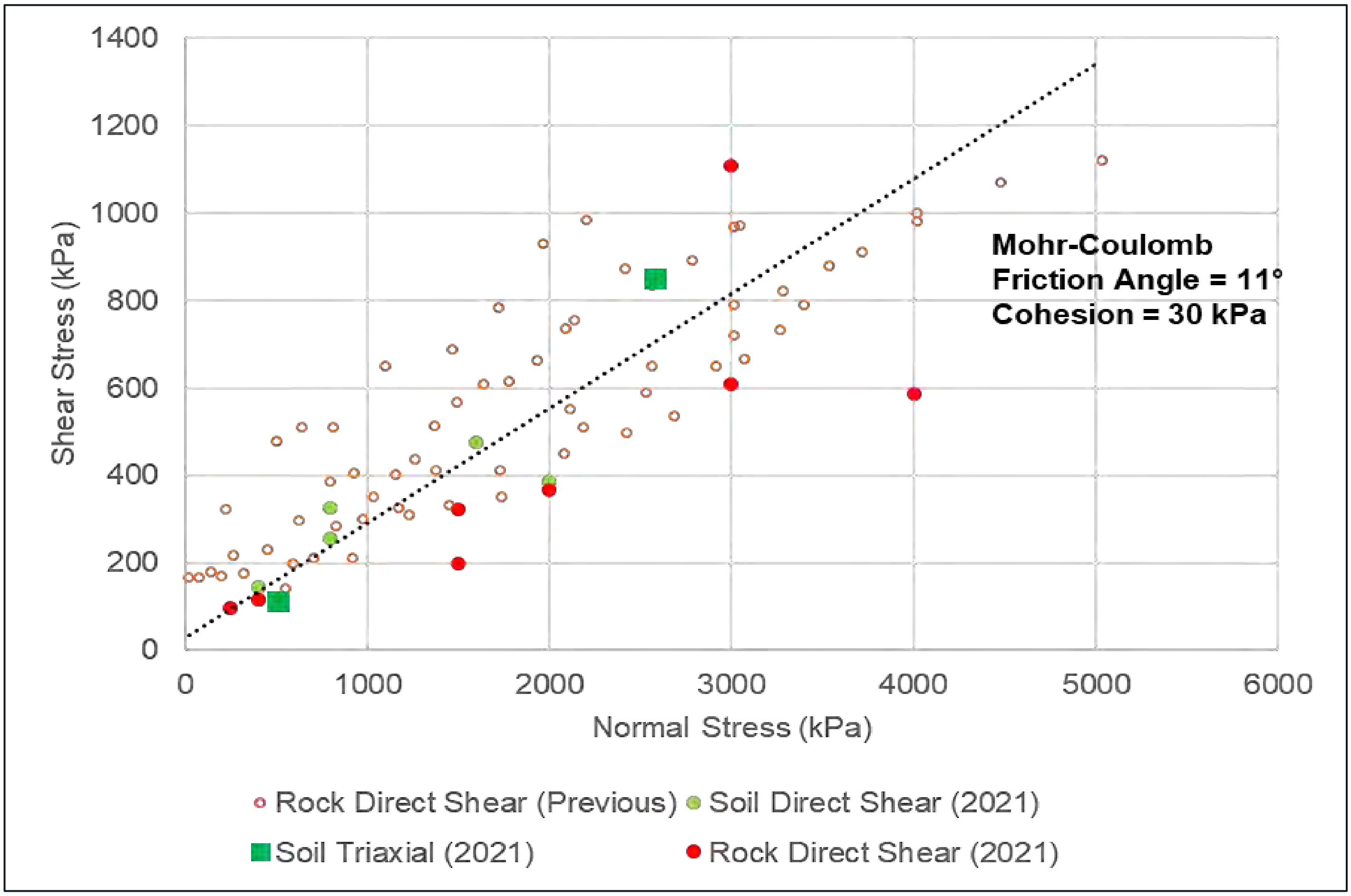
* TS and ChTS average UCS values were derived using triaxial confinement test data
Source: SRK, 2020

Table 13-22: Shear Strength Laboratory Test Results Excluding Talc Samples

Hole ID	From (m)	To (m)	Lithology	Test Type(1)	Peak Friction Angle (°) Natural Surface	Base Friction Angle (°) Saw Cut	Base Friction Angle (°) Tilt Table
AR15-0139	250.66	250.86	CHS	DS	23.7	-	-
AR15-0139	231.43	231.67	CHS	DS	17.2	-	-
AR19-0171	103.29	103.52	CHS	DS	7.7	-	-
AR11-0128	118.18	118.42	GS	DS	22.6	-	-
AR11-0129	177.12	177.52	GS	DS	32.5	-	-
AR11-0129	221.67	222.05	GS	DS	34.2	-	-
AR11-0130	161.04	161.44	GS	DS	23.5	-	-
AR15-0140	115.97	116.19	GS	SC	-	-	25.3
AR15-0142	47.20	47.4	GS	SC	-	-	24.1
AR19-0165A	159.36	159.64	MRP	DS	18	-	-
AR15-0142	82.46	82.68	MRP	SC	-	22.1	23.3
AR15-0140	134.39	134.64	MRP	SC	-	-	24.1
AR15-0143	36.72	36.97	MRP	SC	-	-	24.6
AR19-0166	286.16	286.39	MRP	SC	-	-	27.0
AR16-0149	156.80	157.01	MS	DS	24.5	-	-
AR16-0149	159.93	160.13	MS	DS	29.9	-	-
AR15-0139	75.30	75.51	QFMS	DS	22.1	-	-
AR11-0127	150.63	150.88	QFMS	DS	29.1	-	-
AR19-0168	198.68	198.95	QFMS	DS	25.6	-	-
AR19-0167	163.03	163.23	QFMS	DS	12	-	-
AR19-0164	138.78	138.96	QFMS	SC	-	-	29.4
AR19-0165A	216.69	216.97	QFMS	SC	-	-	25.8
AR19-0168	50.53	50.75	QFMS	SC	-	-	25.0
AR19-0168	139.94	140.17	QFMS	SC	-	-	23.0
AR19-0171	211.59	211.85	QMS	DS	21.8	-	-
AR15-0139	184.65	184.92	QMS	SC	-	24.1	23.3
AR15-0137	200.45	200.69	QMS	SC	-	-	26.3
AR15-0145	219.34	219.57	QMS	SC	-	-	24.0
AR19-0165A	105.13	105.38	QMS	SC	-	-	26.7
AR19-0171	129.77	130.02	QMS	SC	-	-	28.2
AR11-0130	141.10	141.65	QMSp	DS	21.2	-	-
AR11-0130	149.84	150.29	QMSp	DS	31.4	-	-
AR16-0149	154.18	154.48	SMS	DS	19.9	-	-

Source: SRK, 2020

Figure 13-15: Talc Geotechnical Laboratory Testing Results



Source: SRK, 2022

13.9.4 Rock Mass Assessment

Based on similar geotechnical conditions, most lithological units have been grouped together into broad domains represented by the Upper and Lower Plates that are separated by the Warm Springs Fault. The exceptions to these groupings are the weaker talc units, shallow weathered material, and fault zones. The following rock mass domains were defined:

- Upper Plate
- Lower Plate
- Weathered
- Talc Zone
- Fault zones

Mean rock mass parameter values and ranges were defined for each rock mass domain (e.g., fracture frequency, rock mass rating). Particular attention was paid to the assessment of intact rock strength within the defined rock mass domains. Laboratory strength testing, supported by point load testing and empirical field estimates, suggests that strength within the various lithological groups of the Upper and Lower Plates is reasonably homogeneous (Table 13-23).

Table 13-23: Summary of Derived Rock Mass Parameter Values per Rock Mass Domain

Domain	RQD (%)	FF/m	UCS MPa	RMR_{89} Joint Condition (0-30)	RMR_{89}	GSI	E_i (GPa)	m_i
Upper Plate	85-95	1-4	50-60	20	60-70	55-65	18	18
Lower Plate	75-85	2-5	50-60	19	55-65	50-60	18	18
Weathered	45-55	10-20	40-50	16	40-50	35-45	18(1)	13(1)
Talc Zone(2)	20-40	25-40	1-10	10	20-30	15-25	4	6
Faults	0-20	> 40	30-40	16	30-40	25-35	-(3)	-(3)

(1) Estimated parameter after BGC (2012)
(2) The rock mass parameters for Talc Zone are representative of the talc rock mass but not the talc layer itself
(3) No data available; estimated for stability assessments

13.9.5 Kinematic Stability Assessment

A complete assessment of bench and inter-ramp kinematic stability was undertaken. Full descriptions of toppling, planar, and wedge instability risks were developed per structural domain and design sector.

The most significant discontinuity sets, in terms of limiting slope angles, are related to shallow to intermediate dipping S1/S0 (foliation) fabric, which impacts the northeast, east and southeast slopes.

13.9.6 Hydrogeology

Hydrogeological investigations and assessments were completed for both the open pit and valley bottom water/waste management areas.

Hydrogeological data are available from 39 boreholes (Figure 7 13); 15 in the area of the open pit area and 24 in the valley bottom area, comprising:

- Hydrogeologic testing data from 50 packer-based hydraulic tests; 11 slug tests; two pumping tests in valley bottom, each with an observation well; three injection tests in pit area, one of six hours duration and two greater than 24 hours, each with monitoring at nearby vibrating wire piezometers (VWPs) (SRK 2017, SRK 2020, SRK 2022); 122 particle size distributions from test pits in valley floor (SRK 2020).
- Water level data from 25 VWPs in pit area; 3 standpipe piezometers (a fourth piezometer is planned to be installed in 2022) in the pit area;12 water level dataloggers in valley bottom (SRK 2017, SRK 2020, SRK 2022).
- Dedicated ground temperature cables at six locations in valley floor; temperature from all VWP sensors, including those in the open pit area (SRK 2020).
- 22 monitoring wells and two pumping wells in valley bottom; three standpipe piezometers in pit area (SRK 2020).

These data were collected following industry best practices. The data collected using the techniques described above have been used to develop conceptual models for the project area and hydrostratigraphic units:

Overburden is thickest in the valley bottom, typically ranging between 10 m to 20 m with localized areas of 25 m or more. It is comprised of colluvium and glacial till. On the valley sides, and in the area of the pit, overburden thickness is relatively minor, typically around 5 m or less. In the open pit area, the hydraulic conductivity (K) of the overburden ranges between 6x10-7 and 9x10 5 m/s, with a geomean of 4x10-6 m/s. In the valley bottom the K varies between 6x10-7 and 9x10-5 m/s, with a geomean of 5x10-6 m/s.

Weathered bedrock, defined as bedrock with enhanced permeability due to weathering or isostatic rebound post-glaciation, typically ranges between 0 m to 60 m thickness based on available drill hole data. Specific testing of weathered bedrock was completed in valley bottom drill holes; presence of weathered bedrock in the pit area was inferred from logging. The K in the weathered bedrock in both the open pit and valley bottom ranges between 6x10 8 m/s and 4x10-5 m/s, with a geomean of 5x10 7 m/s.

Competent bedrock encompasses all lithologic units, in both the valley bottom and pit areas, with sub-units that include upper and lower fractured bedrock (upper and lower are relative to talc position); talc unit; fault barriers; fault conduits. In the open pit area, the K varies between 2x10-9 m/s and 5x10-5 m/s with a geomean of 7x10 8 m/. In the valley bottom area, the K is interpreted to be primarily related to fractures and ranges between 2x10-9 m/s and 4x10-5 m/s, with a geomean of 7x10 8 m/s.

The talc unit is present in the pit area, with relatively low hydraulic conductivity and is considered an aquitard, but the aquitard is likely discontinuous based on talc distribution and thickness.

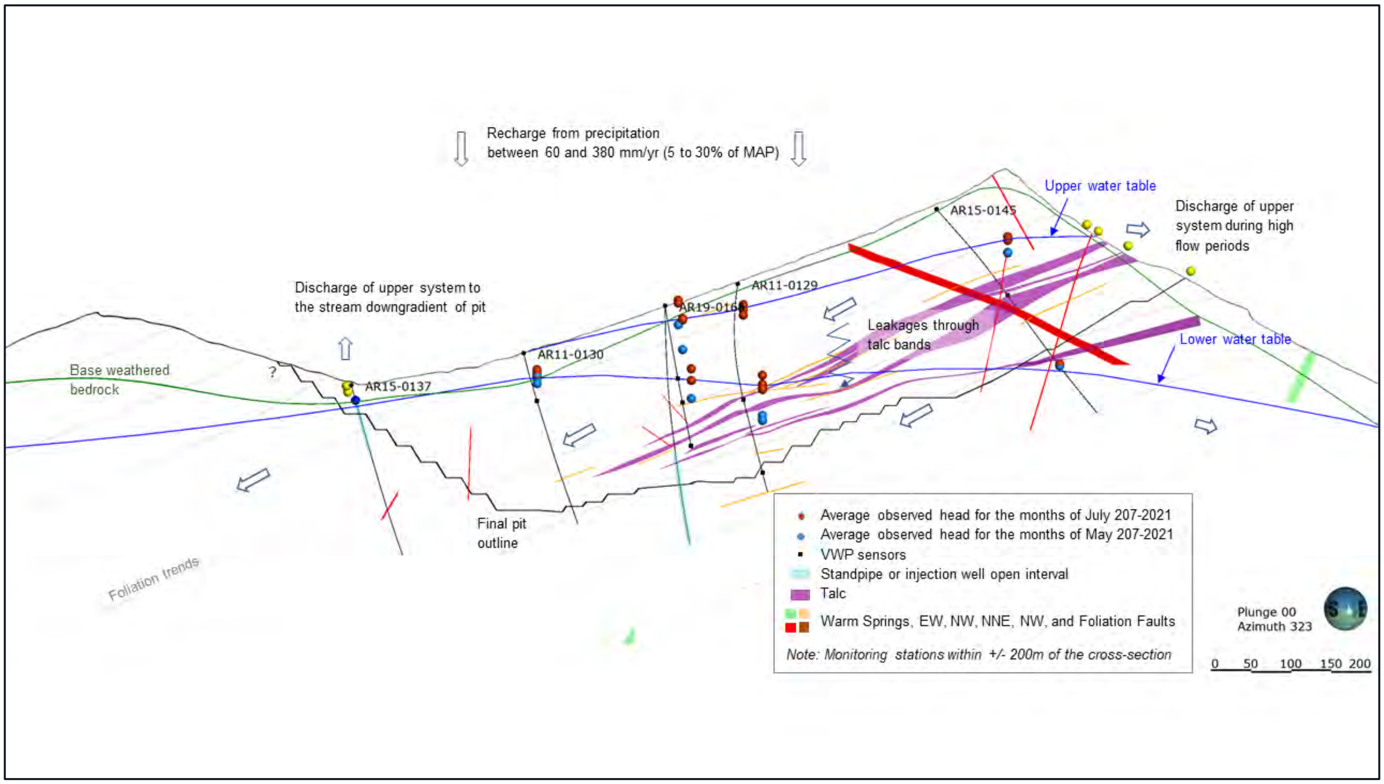
Testing of specific geological structures was completed in the pit area. Results range from similar to surrounding bedrock to higher or lower permeability (K). No continuous, relatively high or relatively low permeability structures were identified.

The hydrogeological conceptual model for valley bottoms is relatively simple. Overall water flow directions are similar to the topography, with the majority of water flow at relatively shallow depth in overburden or weathered bedrock flowing towards valley bottoms. In the valley bottom, overburden heterogeneity exists, with flow occurring within relatively coarser grained, higher hydraulic conductivity units and weathered bedrock. The majority of groundwater flow in the Subarctic Creek watershed is assumed to discharge to Subarctic Creek. Flow within competent bedrock is much lower in magnitude.

In the valley bottom, the outstanding data gap is hydrogeological characterization of overburden and fractured rock in the specific area of the proposed valley bottom groundwater seepage interception system. The characterization program should include completion of monitoring wells down gradient of this system to allow for development of baseline groundwater quality conditions.

Figure 13-16 presents the conceptual hydrogeological model for the pit area. Overall flow directions follow topography, with flow systems all within the competent bedrock hydrostratigraphic unit, with the talc, upper and lower fractured bedrock and structures sub-units present. Hydraulic gradients are downwards from the upper fractured bedrock sub-unit to the lower fractured bedrock unit, separated by the talc aquitard. The lower fractured bedrock unit is confined at lower elevations, with the potentiometric surface above the talc sub-unit. Water levels are assumed to vary seasonally, with highest levels occurring as a result of freshet snow melt and or spring/summer precipitation.

Figure 13-16: Summarized Hydrogeological Conceptual Model for Pit Area



Source: SRK, 2022

13.9.7 Pit Inflow

Pit inflow and pore pressure conditions for the 2019 mine designs were assessed using numerical and analytical tools. Comparison of the 2019 mine designs with 2022 mine design showed relatively minor differences, focused primarily in the area where the pit floor will be mined below the talc for slope stability purposes.

Considering the relatively minor differences, inflow estimates for the 2022 mine design were scaled from previous estimates using comparisons of the footprints, elevations, and bottom depths for open pit designs at different phases. Table 13-24 summarizes estimated pit inflow for three cases reflecting the range of hydraulic conductivity and groundwater recharge that could be expected based on data and modelling. The “Base Case” estimate reflects groundwater model runs with parameters calibrated to average stream baseflow conditions. “Low Case” and “High Case” estimates reflect model runs with parameters calibrated to minimum and maximum baseflow conditions, respectively. Estimates of pit wall runoff, reflecting direct precipitation on the pit, were obtained from the site wide water balance model, using a runoff coefficient of 0.9. The presented values are annual averages; shorter duration higher peak flows and periods of lower inflow can be expected to occur.

Pit inflows are expected to be low magnitude, relative to many other projects worldwide, and flows of this magnitude can be handled with in-pit sumps. It was assumed that sumps would be positioned at the lowest elevation points in the pit. Perimeter dewatering wells are not considered necessary for this scale of flow. A trend of increasing hydraulic conductivity with depth appears to occur at this site. As the pit advances into later years and gets deeper, in-pit dewatering wells may be beneficial.

Detailed design of dewatering capacity needs to consider:

- Normal duty pumping capacity based on estimated pit inflow rates.
- Event pumping capacity based on the highest of design hydrological event (e.g., 1 in 20, 24-hour precipitation), annual freshet flow, or 2x normal duty capacity.
- Pit sumps sized based on event pumping capacity/volume.

It is the QP's opinion that characterization of and assumptions for aquifer and aquitard parameters (i.e., permeability, recharge) used in groundwater models used to estimate inflows follow industry best practice, have an acceptable water balance and are appropriate for this level of study.

Table 13-24: Estimate Pit Inflow

Mining Year	Average Yearly Pit Groundwater Inflow (m^3/d)			Pit Wall Runoff (m^3/d)	Base Case Total Inflow (m^3/d)
	Low Case	Base Case	High Case		
02	130	300	650	1890	2190
05	150	350	770	2410	2760
07	160	370	830	2830	3200
09	160	400	900	3140	3540
11	170	420	950	3320	3740
13	180	440	990	3320	3760

13.9.8 Pore Pressure

Pore pressure influences on slope stability analyses were assessed through combinations of hydrogeological modelling and geotechnical stability modelling. Pore pressures were estimated with four 2-D cross-sectional models that represented the four geotechnical sections used to evaluate slope stability. Conservative model assumptions were used considering the risks posed by the low shear strength of the talc, and uncertainties with hydrogeological conditions (i.e., lateral extent of the hydrostratigraphic zones above the talc, vertical leakage rates between units above and below the talc, and presence of geological structures acting as barriers or conduits).

Sensitivity to model assumptions were tested including variations to hydraulic conductivity of the fractured rock, specific storage, specific yield, as well as to anisotropy within the fractured rock and presence of impermeable geological structures. Results suggested highest sensitivity (as measured via changes in water table distribution and hydraulic head) related to anisotropy and the addition of impermeable geological structures.

Based on slope stability assessments, it is the QP's opinion that pore pressures could pose stability risks under two conditions:

- Stability analyses indicate that slopes along geotechnical section E (SRK, 2022), in the northeast and east walls of the pit are sensitive to potential planar sliding on talc or foliation. Elevated pore pressures in these areas would exacerbate this condition. If pore pressure in these areas does not drain naturally as mining advances, active depressurization should be implemented.

- Compartmentalization by sub-vertical structures could impede natural drainage of the slopes, potentially leading to localized stability issues. Specific areas where this could occur have not been identified in the available data but are possible.

A conceptual pore pressure monitoring system and mitigation plan for the area of geotechnical section E was developed and is presented in Figure 13-17. Assumptions used for this plan are based on the hydrogeological conceptual model and results of hydrogeological testing. The system must be capable of maintaining pore pressure targets during freshet high flow conditions and operate under winter freezing conditions. The plan is sufficient to support the current level of study but should be reviewed as additional information becomes available, and mitigation efforts adapted as appropriate to achieve pore pressure reduction needs.

Figure 13-17: Pore Pressure Management Assumptions for East and Northeast Pit Walls

Assumptions:
Pore pressure system advanced or components replaced as needed at least through End of Year 6. At that time, the zone below talc will become accessible. Permanent horizontal drains can be installed.

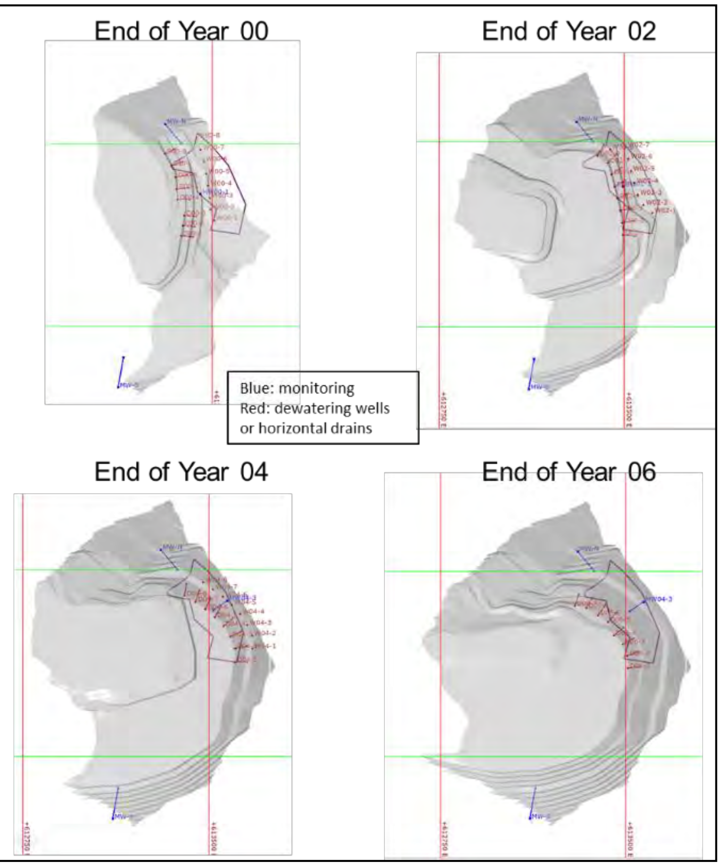
- 8 vertical sacrificial in-pit pumping wells per pit stage
- Completed at depth below talc
- Spacing 50m, depth 250 to 150m, decreasing as mining advances
- Rotary drilling diameter 6", well diameter 5"
- Design flow rate per well 100 m^3/d

- 8 horizontal drains per pit stage, at bottom benches
- Drilled to below talc
- Spacing 50m, length 50m
- 125mm diameter PVC drain installations

- Drain and well spacing based on connectivity observations during 2019 injection tests to distances of 50 to 100m

- 3 vertical in-pit monitoring wells per pit stage, from upper bench
- Completed to similar depths as in-pit pumping wells.
- Spaced along drainage system for performance monitoring
- VWPs
- HQ diameter

Overall Pit Monitoring:
- 2 multi-point monitoring systems north of pit perimeter, completed to depth of pit
- 2 multi-point monitoring systems south of pit perimeter, completed to depth of pit
- 2 multi-point monitoring systems west of pit, between pit and Subarctic Creek, completed to depth of pit. HQ drilling, VWPs



Source: SRK, 2022.

13.9.9 Stability Modelling

The minimum acceptable factor of safety (FoS) for the planned Arctic open pit varies depending on the pit slope component. Based upon the current plans, there is no major infrastructure set to be constructed proximal to any pit walls. If this were to change, it would be necessary to examine the selected acceptance criteria.

Table 13-25 summarizes the selected acceptance criteria for the Arctic pit slope design.

Table 13-25: Selected Slope Stability Acceptance Criteria

Slope Component	Acceptance Criteria (1)			
	Static		Dynamic	
	Minimum FoS	Maximum PoF Displayed as Probability of FoS ≤ 1 (%)	Minimum FoS	Maximum PoF Displayed as Probability of FoS ≤ 1 (%)
Bench	1.1	50%	NA	50%
Inter-ramp	1.3	10%	1.1	10%
Overall	1.3	5%	1.1	5%

(1) FoS = Factor of Safety and PoF = Probability of Failure

Two dimensional (2-D) RS2 modelling, carried out on four sections around the pit, validated the findings from the kinematic assessment and showed that existence of talc layers in the slopes are geotechnical risks. The models suggest that final pit wall slopes are sensitive to the pore pressure when the talc layers are exposed in the pit walls and relatively insensitive to pore pressure in other scenarios. A seismic hazard assessment was incorporated considering the annual exceedance probability of 10% chance of being exceeded in 50 years indicating a peak ground acceleration of 0.26 g for the region of the proposed Artic Pit. The peak ground acceleration used to evaluate the 2-D sections was a conservative 66% of the estimated regional peak ground acceleration for this region, or 0.17 g.

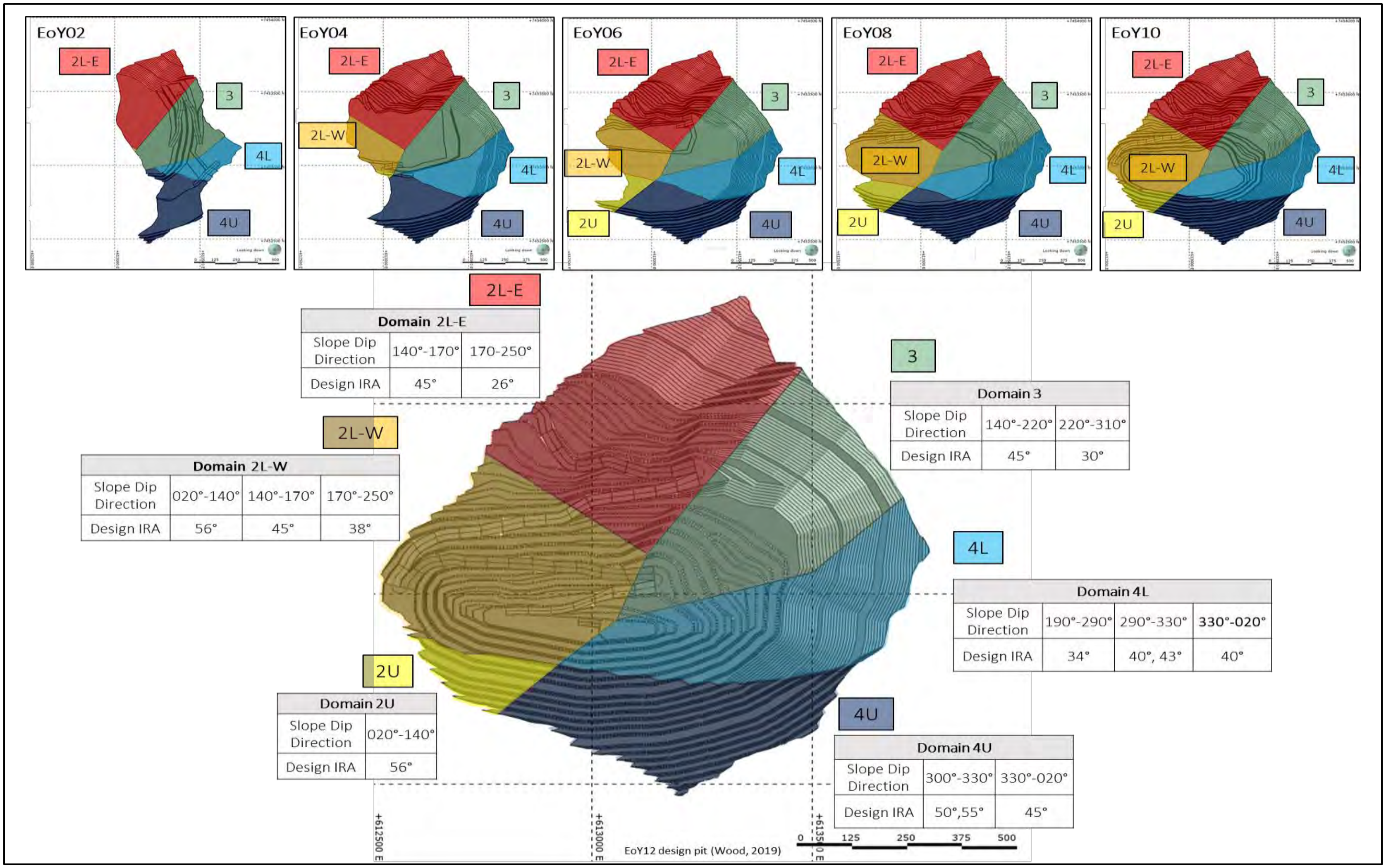
Three dimensional (3-D) RS3 modelling focused on analyzing the stability of the east and northeast walls that are designed to be mined to foliation fabric and contain talc layers. These slope areas did not meet the stability design criteria in 2-D but it was anticipated that confinement in 3D would improve slope performance. The effect of pore pressures was not analyzed in 3-D due to limitations with the software when modelling complex groundwater conditions. As with the results of the 2-D findings, the absence of pore pressure in the 3-D model was assumed to be representative if the pore pressure management plan successfully drains the groundwater above the talc zones.

The 3-D analysis result suggests that the area in the northeast wall above the Cz Talc Zone is, in terms of stability, the most sensitive slope to shear strength reduction. The resulting SRF of 1.3, although meeting the design criteria, represents an optimistic estimation of the factor of safety as the strengths of the talc zone and foliation could be lower.

13.9.10 Slope Design

Slope design recommendations were based on the findings of the kinematic evaluation, with additional adjustments from the numerical modelling analyses. Hydrogeological influences were considered in the 2-D numerical stability analyses using the predictions of phreatic surfaces in the interim and final pit phases. The models suggested that stability of the northeast and east wall was dependent on full excavation of the talc domains and successful management of the pore water pressures. Inter-Ramp slope design angles were determined for each slope design domain as seen in Figure 13-18.

Figure 13-18: Recommended Inter-Ramp Angles by Slope Design Domains Projected on EOY12 Design Pit



Source: SRK, 2019.

13.10 Geotechnical Review of Open Pit Design

Wood updated the open pit mine designs to support the Report. The updated design includes four nested phases including the ultimate pit. The notable geotechnical recommendations considered for the mine design update include:

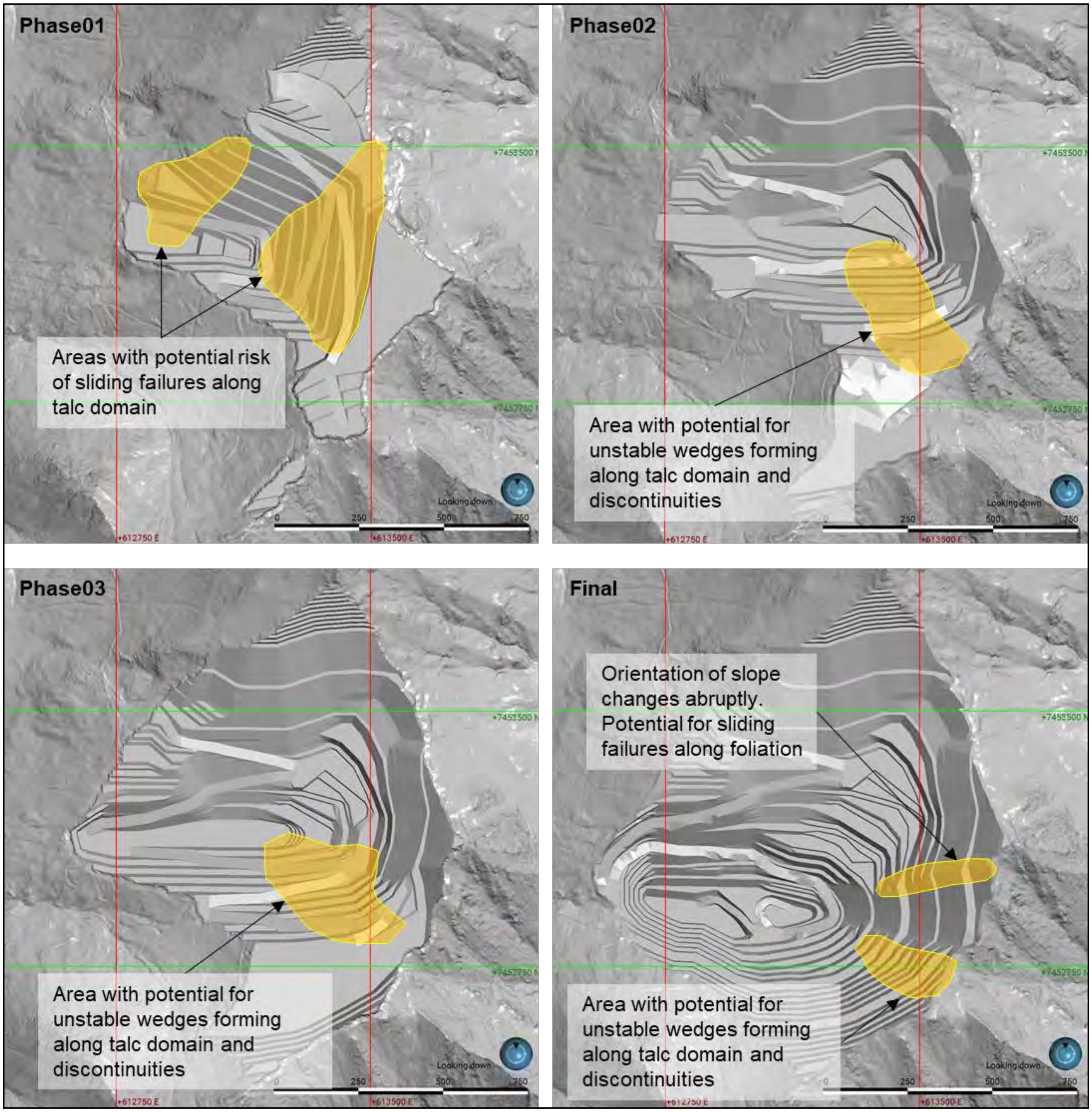
- Where slopes are oriented sub-parallel to foliation, conventional bench-berm designs have been adjusted to excavate along foliation design
- Slope angle of north wall, above 900 m elevation, has been optimized to account for more favourable orientation of foliation
- The east and northeast walls of the interim mining phases do not expose the talc domain
- Talc domains are fully excavated in the northeast and east walls of final mining phase

13.10.1 Slope Stability Risks

Based on a high-level geotechnical review, SRK believes that the updated open pit mine designs have honoured the geotechnical slope design recommendations. The review also identified potential slope stability risks for the various mining phases, shown in Figure 13-19, that were assessed using 3D finite element numerical models with simplified geometries and groundwater conditions. It is recommended that these potential risks are considered in the next stages of the Arctic Project.

- Potential sliding failure along talc domains and fault structures in the northwest and east walls of Phase01 pit. The 3D stability model results suggest that the east wall in the interim mining phase is stable in dry conditions but the factor of safety falls below the design acceptance criteria when the pore pressures become elevated above the talc domains which could occur seasonally during freshet. This highlights the importance of pit water management plan as well as controlling the amount of snow accumulating within the potential areas of risk. Flexibility in mine sequencing may be required to accelerate the excavations above the talc domains if pore pressures can't be controlled.
- Interaction of talc domains and discontinuities to form potentially unstable multi-bench wedges in the south walls of Phase02, Phase03 and Final pit designs. 3D numerical model results show that multi-bench scale failure is unlikely but elevated pore pressures could exacerbate the conditions. Localized blocks could fail along the bench crest and rockfall risk should be considered in these areas.
- Sliding failures along foliation are expected where the slope orientation changes abruptly in the final east wall. Slope design adjustment is recommended to remove any convex slope shapes in the elevated risk areas.

Figure 13-19: Potential Areas of Slope Instability Risks in the 2022 Open Pit Designs



Source: SRK, 2022

13.11 Geochemistry Assessment

13.11.1 Acid Base Accounting Studies

Five geochemical sampling campaigns in 1998, 2010, 2013, 2015 and 2016 resulted in accumulation of a dataset of 1,557 samples tested using various methods. In January 2017, following a large (1004 sample) infill sampling campaign in 2016, Trilogy consolidated the databases, which enabled SRK to use the geochemical database and mineralogical data to develop site-specific methods for acid potential (AP) and neutralization potential (NP) (SRK 2019).

About 75% of samples were classified as potentially acid generating (PAG), which is defined as total sulphur content >0.1% and NP/AP<2 where NP and AP are calculated using site-specific methods. Acid rock drainage (ARD) potential occurs in all rock types to varying degrees. Ore and the associated Gray Schist have the strongest ARD potential, followed by felsic schists. The rock types with weaker ARD potential are chlorite talc schist and metarhyolite porphyry.

In 2017 2019, and 2022 tailings generated by metallurgical testing were tested for acid-base accounting (ABA) methods and found to have potential for acid generation.

13.11.2 Geochemical Kinetic Studies

13.11.2.1 Waste Rock Program

Geochemical kinetic testing using six rock type composites was initiated in 2015. The composites were used to fill seven barrels (including one duplicate) at the site with 240 to 270 kg of waste rock in each test. Leachates from the barrels were collected and tested once in 2015 and three to five times during each summer from 2016 through 2019, and 2021 to 2022. Parallel laboratory humidity cells were also initiated in 2015 and are evaluating the weathering behaviour of the composites. Three of the humidity cells are still operating and had reached 370 weeks as of November 30, 2022, and three of the humidity cells were terminated in 2021/2022 as weathering rates were stable.

An additional 10 waste rock humidity cell tests (HCTs) were initiated in 2019, and a further two were initiated in 2021. While the tests initiated in 2015 generally represented typical compositions of the main waste rock types in terms of sulphur content, the tests initiated in 2019 and 2021 were designed to represent the range of compositions present for key parameters such as selenium and sulphur, including some tests that would represent "worst case" conditions. The humidity cells initiated in 2019 had been operating for 184 weeks as of November 14, 2022, with the exception of three of the tests which were terminated in 2021/2022 as weathering rates were stable. The tests initiated in 2021 had been operating for 92 weeks as of November 14, 2022.

All of the kinetic samples (from the 2015, 2019 and 2021 programs) have the potential to generate acid. To date, ten HCTs have generated acid (pH<5) including samples from all the main rock types. A further four tests had pHs that were most recently between pH 5.0 and 6.5 indicating that carbonate minerals have likely been exhausted and acid generation is buffered by silicate minerals. pH is expected to continue declining in the tests. The remaining four tests had recent pHs around 7.0 and were likely still buffered by carbonate minerals. The testwork has provided data on metal release rates under acidic and non-acidic conditions which have been used to estimate waste rock contact water composition.

13.11.2.2 Tailings Program

Kinetic testing of tailings from the 2017 metallurgical test program was completed late in 2020 (conducted for 193 weeks). Tailings from the 2019 and 2022 metallurgical test programs has been in operation for 172 weeks and 30 weeks respectively (as of November 14, 2022). Testing includes HCTs, and the 2019 and 2022 tailings are also undergoing subaqueous column testing. None of the tests have generated acid and recent pHs are above 7.0. The HCT testwork has provided data on metal release rates under non-acidic conditions which have been used to estimate contact water composition of tailings upon exposure to subaerial weathering. Subaqueous column results are used for predicting tailings water chemistry during operations and upon closure as water is drawn down in the TMF and porewater is released during tailings consolidation.

13.11.2.3 Ore Program

Kinetic testing of ore composites was initiated in 2019 and 2022. The 2019 HCT was terminated in 2021 after 112 weeks of operation. The test initiated in 2022 had been operating for 30 weeks as of November 14, 2022. The samples had not generated acid and pHs were above 7.0. The testwork has provided data on metal release rates under non-acidic conditions which have been used to estimate the ore contact water composition.

14 PROCESSING AND RECOVERY METHODS

14.1 Overview

The current process plant design is updated from the 2020 Arctic Feasibility Study Technical Report (2020 FS) by considering the verification testwork conducted by ALS in 2022, along with process optimization based on Ausenco's benchmark database. The process plant will operate two 12-hour shifts per day, 365 day a year, with an overall plant availability of 92%. The process plant will produce three concentrates: 1) copper concentrate, 2) zinc concentrate, and 3) lead concentrate. Gold and silver are expected to be payable at a smelter and are recovered in both the copper and lead concentrates.

The process is developed to treat the MS and SMS ores from the Arctic deposit hosted in the Ambler Sequence within a VMS belt. The deposit mineralisation occurs as stratiform SMS to MS beds within graphitic chlorite schists and quartz mica schists. The mined material will be comminuted via a primary crushing circuit and a grinding circuit configured in semi-autogenous ball (SAB) mode. Due to the significant levels of talc contained in the mined materials (2.92% in the plant feed), a talc flotation is required to remove the mineral. This is then followed by a bulk flotation of a copper and lead concentrate, and the subsequent separation of copper and lead concentrate. At this stage, flotation of a zinc concentrate process takes place to produce the three flotation concentrates. The talc flotation concentrate and the zinc tailings will be combined and thickened before discharged to the tailings storage facility.

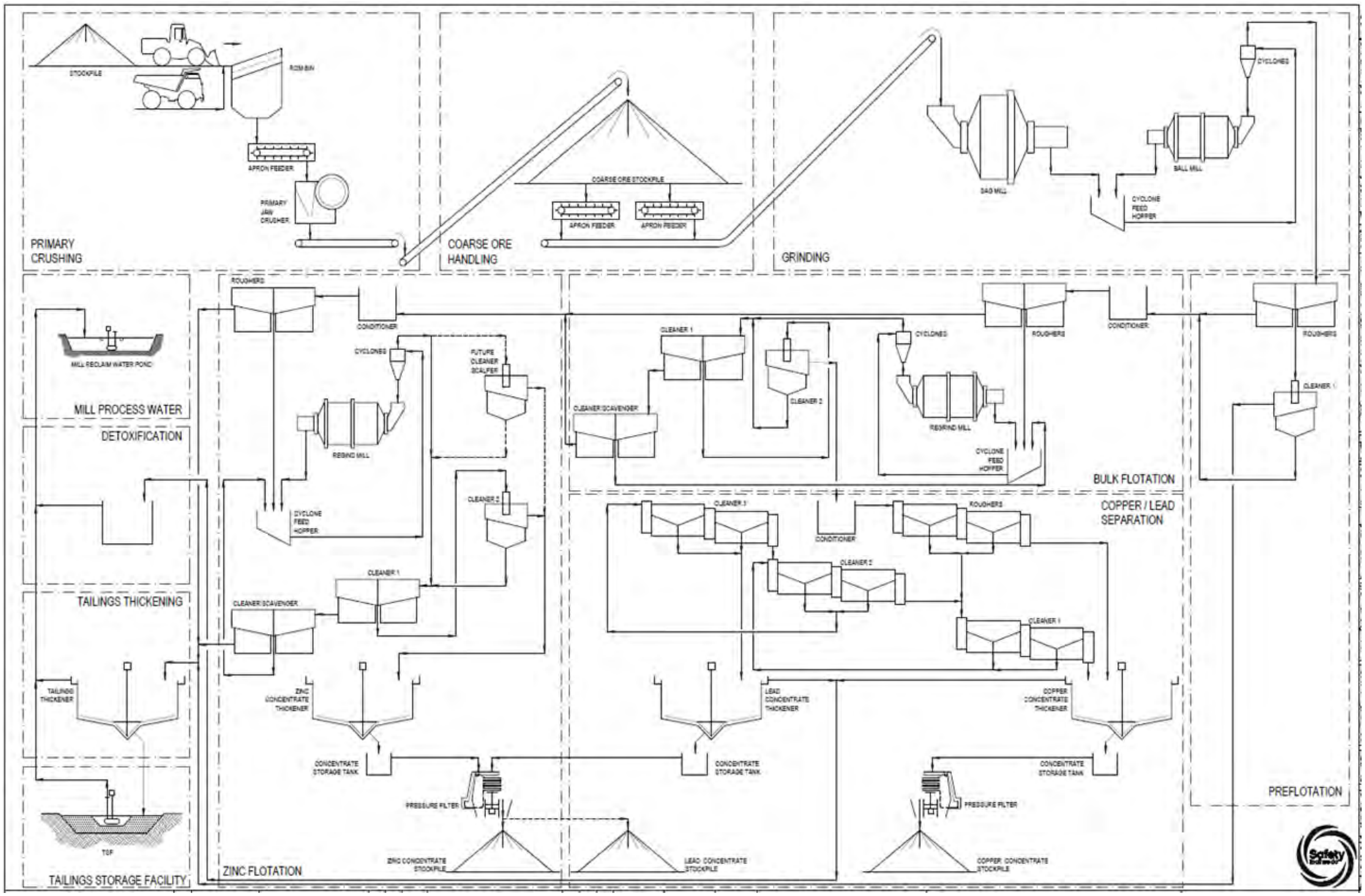
The overall circuit consisting of crushing, grinding and flotation to generate the respective metallic concentrates is based on the optimal process route to produce saleable concentrates from the ores in the Arctic deposit. This process flowsheet is shown in Figure 14-1. Figure 14-2 shows a plan of the overall plant area.

14.2 Process Flowsheet and Area Plan

The process plant will consist of the following unit operations:

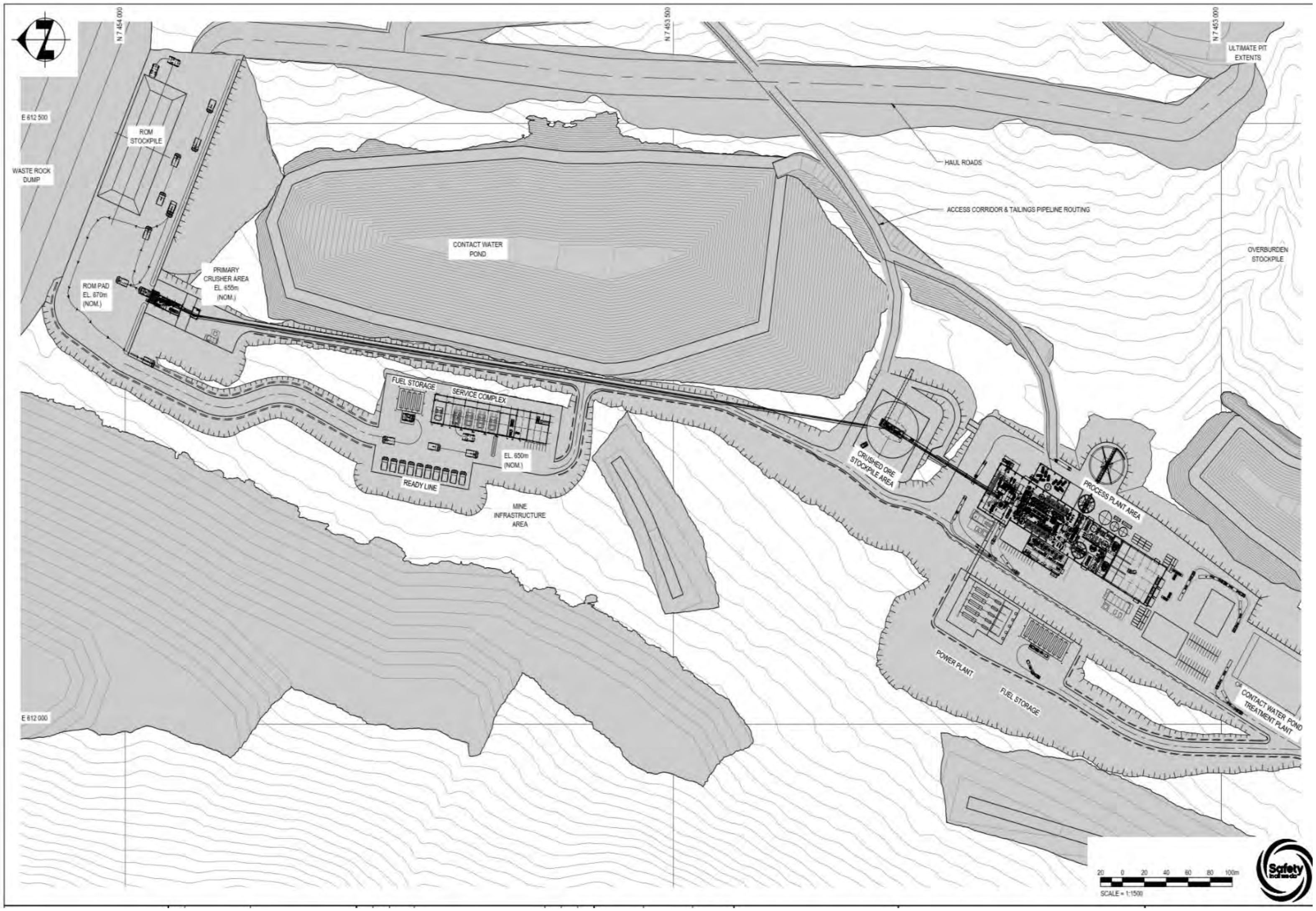
- Primary (jaw) crushing;
- Course ore stockpile;
- SAG and ball mill grinding:
    - o primary grinding using an open circuit semi-autogenous grinding (SAG) mill; and
    - o secondary grinding using a closed-circuit ball mill.
- Talc pre-flotation;
- Bulk copper and lead flotation;
- Copper and lead separation flotation;
- Zinc flotation;
- Thickening, filtration, and loading of copper, lead, and zinc concentrates;
- Cyanide destruction; and
- Tailings thickening and disposal.

Figure 14-1: Process Plant Flowsheet



Source: Ausenco, 2022.

Figure 14-2: Overall Plant Area Plan



Source: Ausenco, 2022.

14.3 Process Design Criteria

The process plant is designed for a throughput of 10,000 t/d, equivalent to 3,650,000 t/a. The key process design criteria considered for the design and selection of equipment for the processing facilities are listed in Table 14-1 and the comminution parameters are provided in Table 14-2. The process plant design is based on a robust metallurgical flowsheet developed for optimum recovery.

Table 14-1: Process Design Criteria – Overview

Criteria	Unit	Value
Plant Design Capacity	t/a	3,650,000
	t/d	10,000
Operating Availability - Crushing	%	65
- Grinding and Flotation	%	92
- Concentrate and Tailings Filtration	%	84
Process Plant capacity, nominal @ 92% availability	t/h	453
ROM Specific Gravity	-	3.25
Plant Feed Grades - Copper, Design	%	3.00
- Lead, Design	%	0.74
- Zinc, Design	%	4.20
- Gold, LOM Average	g/t	0.47
- Silver, LOM Average	g/t	35
- Talc, LOM Average	%	8

Table 14-2: Comminution Design Criteria

Criteria	Unit	Value
Crushing (Single Stage)		
Availability	%	65
Primary Crusher	Type	Jaw Crusher
Crushing Feed Size, 80% Passing	mm	504
Crushing Circuit Product Size, 80% Passing	mm	80
Grinding		
Availability	%	92
Circuit Type	-	SAG Mill, Ball Mill
Bond Ball Mill Work Index, Design	kWh/t	11.6
Bond Abrasion Index, Design	g	0.038
A x b, design	-	109
Feed Particle Size, F_{80}	mm	80
Product Particle Size, P_{80}	μm	70
Regrinding Circuit Product Size, 80% Passing - Bulk Rougher Concentrate	μm	40

Design criteria for the flotation plant were determined from the testwork conducted by ALS Metallurgy (described in Section 13) and is summarized in Table 14-3.

Table 14-3: Flotation Plant Design Criteria

Criteria	Unit	Value
Feed Rate	t/h	453
Pre-Flotation		
Cell Type	Type	Conventional Tank Cells
Stage Recovery to concentrate, mass	% flotation feed	21.5-25.7
Stage Recovery, talc	%	88.6
Copper-Lead Bulk Rougher Flotation		
Cell Type	Type	Jameson Cell
Stage Recovery to concentrate, mass	% flotation feed	12.7-15.0
Regrind Mill		
Type		Ball Mill
Feed Rate, design	t/h	68
Feed Particle Size, F_{80}	µm	63
Product Particle Size, P_{80}	µm	40
Specific Grinding Energy (SGE)	kWh/t	9.2
Copper-Lead Cleaners		3 stages
Cell Type	Type	Conventional Tank Cell – Stage 1 & 2 Jameson Cell – Stage 3
Stage Recovery to Concentrate, mass	% Flotation feed	16.1-18.6
Stage Recovery, Copper	%	92.1
Stage Recovery, Gold	%	52.0
Stage Recovery, Silver	%	32.5
Lead Rougher Flotation		
Cell Type		Conventional Tank Cells
Stage Recovery to concentrate, mass	% flotation feed	1.7-2.5
Lead Cleaners		3 Stages
Cell Type		Conventional Tank Cell
Stage Recovery to concentrate, mass		2.6-3.6
Stage Recovery, Lead	%	61.3
Stage Recovery, Gold	%	21.6
Stage Recovery, Silver	%	48.6
Zinc Rougher Flotation		
Cell Type		Conventional Tank Cell
Stage Recovery to Concentrate, mass	% flotation feed	7.4-8.8
Zinc Regrind Mill		

Criteria	Unit	Value
Type		Ball Mill
Feed Rate, design	t/h	40
Feed Particle Size, F_{80}	µm	63
Product Particle Size, P_{80}	µm	40
Specific Grinding Energy (SGE)	kWh/t	9.2
Zinc Cleaners		3 Stages
Cell Type		Conventional Tank Cell – Stage 1 & 2 Jameson Cell – Stage 3
Stage Recovery to concentrate, mass		11.
Stage Recovery, Zinc	%	88.5
Stage Recovery, Gold	%	3.3
Stage Recovery, Silver	%	5.7
Concentrate Handling		
Copper Concentrate		
Type		Hi-rate
Unit Area Thickening rate, design	$t/m^2.h$	0.25
Thickener Underflow Density	%w/w	65
Copper Concentrate Filter		
Type		Horizontal plate pressure filter
Filtration Rate	$kg/m^2/h$	575
Nominal Filter Cake Moisture	%w/w	6
Lead Concentrate		
Type		Hi-rate
Unit Area Thickening rate, design	$t/m^2.h$	0.2
Thickener Underflow Density	%w/w	65
Lead Concentrate Filter		
Type		Horizontal plate pressure filter
Filtration Rate	$kg/m^2/h$	800
Nominal Filter Cake Moisture	%w/w	6
Zinc Concentrate		
Type		Hi-rate
Unit Area Thickening rate, design	$t/m^2.h$	0.25
Thickener Underflow Density	%w/w	65
Zinc Concentrate Filter		
Type		Horizontal plate pressure filter
Filtration Rate	$kg/m^2/h$	555
Nominal Filter Cake Moisture	%w/w	6

14.4 Plant Description

14.4.1 Crushing Plant

Run-of-mine ore will be trucked to the crushing station and dumped into a 200-t receiving bin protected with a 1,000-mm-aperture stationary grizzly. Ore will be reclaimed from the bin with an apron feeder and scalped of fines with a 75-mm-aperture vibrating grizzly. The grizzly oversize will be passed to the primary jaw crusher. The selected conventional jaw crusher will operate with a closed side setting of 100 mm. The crushed material together with the grizzly undersize, with a P80 of 80 mm, will be discharged to the conveyor. A tramp metal magnet will be installed at the head end of the conveyor to remove tramp metal.

Crushed ore will be sent to a single covered conical stockpile via the stockpile feed conveyor.

The major equipment in the crushing circuit will include:

- One apron feeder, installed power 30 kW;
- One vibrating grizzly, installed power 30kW; and
- One single toggle jaw crusher, installed power 160 kW.

14.4.2 Coarse Ore Storage

The coarse ore stockpile will have a live capacity of 5,000 t, equivalent to approximately 12 h of mill feed at the nominal mill feed rate. The stockpile total capacity will be approximately 20,000 t. The coarse ore stockpile will be a covered facility to mitigate freezing of the stockpiled material, which will be also equipped with a dust collecting system. The coarse ore stockpile building will have sufficient space to allow for the operation of mobile equipment as required. Reclaim of ore from the stockpile will be accomplished using two 1,000-mm-wide by 4,800-mm-long apron feeders at a nominal rate of 226 t/h per feeder. Reclaimed material from the apron feeders will be discharged onto the SAG mill feed conveyor.

The major equipment of the coarse ore storage area will include:

- Two apron feeders, unit installed power 11 kW.

14.4.3 Grinding and Classification

14.4.3.1 Primary Grinding and Classification

The grinding circuit will consist of a SAG mill followed by a ball mill arranged in a closed circuit with a hydrocyclone cluster. The nominal feed throughput of the circuit will be approximately 453 t/h. The SAB circuit will reduce the crushed ore particle size from a P80 of 80 mm to 70 µm.

SAB circuit is selected by considering the low competence of the ore materials. The SAG mill will be a grate discharge type with 12 mm apertures and no pebble ports. As required, steel balls will be added to the SAG mill to maintain mill power. The SAG mill product will discharge onto a trommel screen. Trommel screen undersize will report to a hydrocyclone feed pumpbox and the oversize to a scats bunker.

The ball mill discharge will report to the hydrocyclone feed pumpbox, where it will be combined with the SAG mill trommel undersize prior to feeding the hydrocyclone cluster. Process water will be added to the SAG mill feed chute and hydrocyclone feed pumpbox to maintain a target slurry density. The hydrocyclone underflow will feed the ball mill by gravity, while the hydrocyclone overflow, with a solids content of 37%, will gravitate to the flotation circuit. The ball mill circulating load will be 350%.

The major equipment of the grinding and classification circuit will include:

- One SAG mill, 6.1 m in diameter (20 ft) by 4.9 m (16 ft) EGL, installed power 3,000 kW;
- One ball mill, ;6.1 m diameter (20 ft) by 9.1 m (30 ft) EGL, installed power 6,000 kW;
- Two 55 kW slurry pumps to pump hydrocyclone feed, with one pump in operation and one in standby; and
- One hydrocyclone cluster with fourteen 400 mm hydrocyclones, nine in operation, four in standby and one blank.

14.4.4 Flotation

The flotation plant will produce a talc concentrate for disposal, as well as three payable base metal concentrates for shipment to market and sale. The talc concentrate will be produced prior to base metal flotation to minimize dilution of the base metal concentrates. A standard bulk copper and lead concentrate followed by the flotation of a zinc concentrate will be used in the flotation circuit. The copper and lead in the bulk concentrate will be separated by floating the lead from the bulk concentrate to produce individual copper and lead concentrates.

14.4.4.1 Talc Pre-Flotation

The ball mill hydrocyclone overflow will feed the talc rougher circuit. The talc flotation will be composed of rougher and cleaner flotation stages to float the talc mineral and reject any entrained payable sulphide minerals. The rougher flotation will be performed in the conventional forced-air cells and the cleaner flotation will be performed in a Jameson cell. The talc cleaner concentrate will be pumped to the final tailings.

The major equipment in the talc pre-flotation circuit will include:

- Four 120 m^3 tank cells, unit installed power 93 kW
- One Jameson cell (B4500/12).

14.4.4.2 Copper – Lead Bulk Flotation and Regrinding

The talc pre-flotation tailings will be pumped to the copper/lead bulk flotation conditioning tanks where copper and lead mineral collectors will be added. The conditioned slurry will undergo rougher flotation in conventional tank flotation cells for recovery of a bulk copper and lead concentrate.

The bulk rougher concentrate will be reground in a 600 kW regrind ball mill configured in reverse, closed circuit with hydrocyclones. The bulk rougher concentrate will be reduced to a particle size of 80% passing 40 µm prior to being further upgraded by two stages of cleaner flotation.

The 1st bulk cleaner and cleaner scavenger flotation will be conducted in conventional tank cells. The 1st cleaner flotation tailings will gravitate for scavenging of residual copper and lead minerals. Concentrate from the cleaner-scavenger flotation will be pumped to the regrind hydrocyclone feed pumpbox for further regrinding, while the cleaner-scavenger tailings will join the rougher flotation tailings for pumped transfer to the zinc flotation conditioners. The 2nd bulk cleaner flotation will be performed in a Jameson cell to upgrade the concentrate obtained from the last stage. The 2nd cleaner tailings will be reprocessed in the 1st cleaner flotation stage.

Copper-lead bulk cleaner flotation dilution water and launder sprays will be supplied by a dedicated process-water system. The supply for this system will be a combination of copper/lead bulk thickener overflow and main plant process water.

The major equipment in the bulk flotation and regrinding circuit will include:

- Five 81 m^3 bulk rougher flotation tank cells, unit installed power 75 kW;
- One regrind ball mill, 3.0 m diameter by 5.3 m EGL, installed power 600 kW;
- Four 37 m^3 bulk 1st cleaner flotation tank cells, unit installed power 45 kW;
- Three 37 m^3 bulk cleaner scavenger flotation tank cells, unit installed power 45 kW; and
- One bulk 2nd cleaner flotation Jameson cell (B4500/12).

14.4.4.3 Copper and Lead Separation Flotation Circuit

The bulk 2nd cleaner concentrate will be pumped to the copper-lead separation flotation conditioning tank, where sodium cyanide and lime will be added to supress copper minerals. Lead flotation will be conducted at a pH ranging from 9.0 to 9.5. The conditioned slurry will flow to the conventional lead rougher flotation cells. Frother will be added in the rougher and cleaner flotation cells and lead promoter will be added in the lead rougher concentrate pumpbox to collect lead minerals.

The lead rougher concentrate will be further upgraded in three stages of cleaner flotation to produce the final lead concentrate that will report to the lead concentrate thickener. Tailings from the 2nd and 3rd stages of cleaner flotation will gravity flow to the proceeding flotation cell feed boxes. Tailings from the 1st cleaner flotation will be combined with the tailings from the lead rougher flotation to produce the final copper concentrate, which will report to the copper concentrate thickener.

The major equipment in the lead flotation circuit will include:

- Five 6 m^3 lead rougher flotation cells, unit installed power 11 kW;
- Five 4 m^3 1st cleaner flotation cells, unit installed power 7.5 kW;
- Two 4 m^3 2nd cleaner flotation cells, unit installed power 7.5 kW; and
- One 4 m^3 3rd cleaner flotation cell, unit installed power 7.5 kW.

14.4.4.4 Zinc Flotation and Regrind

The copper-lead bulk rougher tailings will be further floated to recover zinc in the zinc flotation circuit, along with the copper-lead bulk cleaner scavenger tailings. The circuit will consist of feed slurry conditioning, rougher/scavenger flotation, zinc rougher concentrate regrind, and two stages of cleaner flotation. A cleaner scalper stage will be installed when the plant feed zinc grade are expected to be higher.

Tailings from the copper-lead bulk flotation circuit will be conditioned with lime (to a pH above 10.5, to depress pyrite) and copper sulphate (to activate zinc minerals), and then sent to the head cell of a bank of zinc rougher flotation cells.

Zinc rougher concentrate will report to the regrinding circuit; zinc rougher tailings will be combined with the Talc concentrate and pumped to the tailings thickener. Thickened tails will be pumped to the Tailings Storage Facility (TSF).

The zinc rougher and cleaner scavenger concentrates will be reground in a 355 kW regrind ball mill configured in, closed circuit with hydrocyclones. The zinc concentrates will be reduced to a particle size of 80% passing 40 µm prior to being further processed by the zinc 1st cleaner flotation circuit.

The zinc cleaner circuit will operate at a pH of 11 or above to reject pyrite. The 1st cleaner concentrate will be further upgraded in the 2nd stage of cleaner flotation to produce the final zinc concentrate. The 1st cleaner flotation tailings will be further floated in the cleaner scavenger flotation cells. Concentrate from the cleaner scavenger flotation will return to the zinc regrind circuit. The cleaner scavenger tailings will also report to the final tailings pump box, joined by the zinc rougher tailings and talc flotation cleaner concentrate.

The main equipment used for the zinc flotation circuit will include:

- Five 81 m^3 zinc rougher flotation tank cells, unit installed power 75 kW;
- One regrind ball mill, 2.7 m diameter by 4.1 m EGL, installed power 355 kW;
- Four 23 m^3 1st cleaner flotation tank cells, unit installed power 37 kW;
- Two 23 m^3 cleaner scavenger flotation tank cells, unit installed power 37 kW;
- One Jameson flotation cell (E4232/10), Year 1 and Year 7 as 2nd cleaner flotation cell and repurposed to cleaner scalper flotation cell after Year 7; and
- One Jameson flotation cell (E2532/6), future installation on Year 7 as 2nd cleaner flotation cell.

14.4.5 Product Dewatering

The copper, lead, and zinc concentrates will report to separate higher-rate thickeners, where flocculant will be added to assist in the settling of the solids. The thickened slurries will be further dewatered by using pressure filters to a design moisture content of 6%. Two identical tower press filters are recommended for the concentrate filtration duties, including one common unit for lead and zinc concentrates dewatering; while another unit is dedicated to dewater the copper concentrate.

14.4.5.1 Copper Concentrate Dewatering

The copper concentrate will be pumped to a 15 m diameter high-rate thickener which is covered for outdoor application. The thickener overflow water will be reused in the lead flotation circuit, along with overflow from lead concentrate thickener. The excess overflow will be treated by a sulphur dioxide-air procedure to destroy residual cyanide Neutralized overflow will be pumped to the tailings thickener, where the thickener overflow will report to the process pond before recycling back to the plant process water system.

The copper concentrate thickener underflow at approximately 65% solids density by weight will be pumped to an agitated concentrate stock tank with 24 hours storage capacity prior to the filtration process. The final filter cake design moisture is 6% considering the site climate conditions. The copper concentrate will be discharged into a stockpile, from which front end loaders will load concentrate onto containers for shipment. Filtrate will return to the copper concentrate thickener. The stockpile will be capable of storing up to 4 days of copper concentrate production in case of potential truck haulage interruptions.

The main equipment in the copper concentrate dewatering circuit will include:

- One 15-m-diameter high-rate thickener, installed power 3.7 kW;
- One 84 m^2 copper concentrate tower filter press, installed power 105 kW; and
- Two 44 m^3 cyanide destruction tanks.

14.4.5.2 Lead Concentrate Dewatering

Lead concentrate will be blended with the flocculant solution and discharged to a 6 m diameter high-rate thickener. The thickener underflow containing approximately 65% solids will be pumped to an agitated concentrate stock tank prior to pressure filtration. The concentrate stock tank will have the capacity to contain lead concentrate production for 24 hours. The thickener overflow will be directed to the lead separation circuit for reuse.

Lead concentrate will be filtered in a pressure filter which will be designed to alternate between lead and zinc filtration. Use of a single filter, for 2 products, will require automation to ensure there is no lead-zinc cross contamination. The filter discharge will fall on a reversable conveyor which will send the product to the lead concentrate storage area. The filtrated lead concentrate cake with 6% design moisture will be loaded onto concentrate containers for shipment. The filtrate from the filter will return to the lead concentrate thickener feed well. The in-plant lead concentrate storage facility will be capable of storing at least 4 days of lead concentrate production.

The main equipment in the lead concentrate dewatering circuit will include:

- One 6-m-diameter high-rate thickener, installed power 1.5 kW; and
- One 84 m^2 tower filter press, shared with zinc concentrate filtration, installed power 105 kW.

14.4.5.3 Zinc Concentrate Dewatering

The final zinc concentrate will be pumped to a 14 m diameter high-rate thickener. Flocculant will be added to improve settling of the concentrate. The thickener underflow, with a solid density of 65%, will be stored in an agitated concentrate stock tank prior to pressure filtration. Thickener overflow will be delivered to the zinc rougher feed. The concentrate stock tank is designed to be capable of containing zinc concentrate production for 24 hours.

Zinc concentrate will be filtered in a pressure filter which will be designed to alternate between lead and zinc filtration. Use of a single filter, for 2 products, will require automation to ensure there is no lead-zinc cross contamination. The filter discharge will fall on a reversable conveyor which will send the product to the zinc concentrate storage area. The final filter cake design moisture is 6%. Filtrate from the filtration will return to the zinc concentrate thickener. The zinc concentrate will be loaded onto concentrate containers for shipment. The zinc concentrate storage facility will be capable of storing at least 4 days of zinc concentrate production in case of potential transport interruptions.

The main equipment in the zinc concentrate dewatering circuit will include:

- One 14-m-diameter high-rate thickener, installed power 5.5 kW; and
- One 84 m^2 tower filter press, shared with lead concentrate filtration, installed power 105 kW.

14.4.6 Tailings Disposal

The final flotation tailings consist of the talc flotation concentrate, zinc rougher flotation tailings and zinc cleaner scavenger flotation tailings. The final flotation tailings will be directed to a 32 m diameter high-rate thickener. Flocculant will be added to improve settling of the tailings. Thickener underflow with a solid density of 47% will be pumped towards the tailings storage facility. Thickener overflow will be delivered to the Process Pond, from where it will be pumped to the process water tank for distribution to the plant.

Water pump barge will be installed to in the tailings pond to reclaim water Tailings water will be pumped to the Process Pond. Water barge pumps will be installed in the Process Pond and will be able to send water to the process water tank or the WTP. Tailings management is discussed further in Section .

14.4.7 Reagent Handling and Storage

Various chemical reagents will be added to the grinding and flotation circuits to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the concentrate products. The reagents will be prepared and stored in a separate, self-contained area inside the process plant and delivered by individual metering pumps or centrifugal pumps to the addition points. All reagents will be prepared using WRCP water via a bulk reagent handling system, mixing and holding tanks.

Estimated annual reagent consumptions are provided in Table 14-4.

Table 14-4: Annual Reagent Consumption

Reagent	Unit	Value
Flocculant	t/a	104
SIPX	t/a	69
3418A	t/a	102
MIBC	t/a	301
Lime	t/a	3,424
Descalant	t/a	36
Cyanide	t/a	1,324
$ZnSO_4$	t/a	529
$CuSO_4$	t/a	690
SMBS	t/a	917

14.4.7.1 Collectors

Sodium isopropyl xanthate (SIPX) in pellets form will be shipped to the mine site in bulk-bags. The SIPX will be diluted to 20% w/w solution strength in a mixing tank and stored in a holding tank, before being dosed to the copper-lead bulk flotation circuit and the zinc flotation circuit via metering pumps.

Collector 3418A will be received as a liquid in Intermediate Bulk Containers (IBC) totes. It will be delivered to the copper-lead bulk flotation circuit via metering pumps without dilution.

14.4.7.2 Frother

MIBC frother will be received as a liquid in IBC totes. The reagent will be used at the supplied solution strength. Metering pumps will deliver the frother to the talc, copper, lead, and zinc flotation circuits.

14.4.7.3 Lime

Lime will be trucked to the site as unslaked pebbles and stored in a 100-t-capacity bulk lime silo. Lime will be conveyed to the lime slaker and slaked with water. The slaked lime will be stored in an agitated mixing tank and distributed to the addition points via a lime slurry loop, to the zinc flotation circuits and WTP. The slaker will be controlled automatically based on the lime slurry levels in the holding tank.

14.4.7.4 Flocculant

Solid flocculant will be prepared in a packaged preparation system, including a screw feeder, a flocculant educator, and mixing devices. Flocculant will be diluted to a 0.5% w/v solution strength and added via metering pumps to the copper, lead, zinc and final tailings thickeners' feed wells where the reagent will be further diluted.

14.4.7.5 Other Reagents

Sodium cyanide, sodium metabisulphite (SMBS), zinc sulphate, and copper sulphate will be supplied in powder/solid form and will be dissolved and diluted by WRCP water. The strength of the reagent solutions will be approximately 15–20%. Cyanide monitoring/alarm systems will be installed at the cyanide preparation areas. Emergency medical stations and emergency cyanide detoxification chemicals will be provided at the areas as well.

Anti-scaling chemicals may be required to minimize scale build-up in the reclaim or recycle water lines. These chemicals will be delivered in liquid form and metered directly into the intake of the reclaim water pumps or process water tank.

Storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection equipment and devices will be provided at reagent preparation areas.

14.4.8 Power Supply

Plant power will be generated on site, with no utility interconnection using diesel generator sets that will be installed to operate in parallel (four operating and one on standby). Each generator will be rated 13.8 kV, 5.5 MW, and the total power output capacity will be 21.6 MW excluding the redundant unit.

The total connected load for the plant will be 25.9 MW with a normal operating load of 21.0 MW (~167 GWh/a).

14.4.9 Water Supply

There will be three separate water supply systems: a fresh water system, a WRCP water supply system, and a process water supply system. The contact water will originate principally from the Waste-Rock Control Pond. The total water usage for the process plant is 42,240 m^3/d.

14.4.9.1 Fresh Water Supply System

Fresh water will be used for fire water and potable water applications that will be supplied from ground water wells. A 350 m^3 fresh water/fire water storage tank will hold well water, with a live capacity of 4 hours.

14.4.9.2 WRCP Water Supply System

WRCP water will be fed to the TMF during winter months and treated and discharged in the summer. Additional process water will be supplied from the WRCP to the TMF through the winter months to meet mill water demands and provide water stored in the TMF as a contingency for dry years. Contact water will also be supplied form the WRCP, which will feed the potable water and gland water tanks.

14.4.9.3 Process Water Supply System

Process water will mainly come from the Process Pond which is fed by tailings thickener overflow and reclaimed water from TMF. In addition, contact water is also another process water source as makeup water. Process water will be stored in a 14-m-diameter x 14-m-high tank, from where the water will be distributed to the process plant and other service locations. The process water consumption is 40,326 m^3/d.

14.4.10 Air Supply

High-pressure compressed air will be provided by three duty and one standby screw compressors and a duty plant air receiver. The instrument air will be dried and then stored in a dedicated air receiver. The plant air will be fed directly from the plant air receiver.

High-pressure air for the concentrate filters will come from a dedicated system of two duty and one standby screw compressors and a concentrate filtration area air receiver.

Low-pressure air for flotation-cell air requirements will be provided by four duty and one standby centrifugal blower.

Air service systems will supply air to various applications throughout the plant, as follows:

- A separate high-pressure air service system will supply air to the crushing plant by a dedicated air compressor. The air will be provided for dust suppression and equipment services.
- Flotation and Cyanide Destruction: low-pressure air will be provided for flotation tank cells and cyanide destruction tank by air blowers.
- Filtration Circuit: wet high-pressure air will be provided for filtration and drying by air compressors.
- Plant Air Service: wet high-pressure air will be provided for various services by dedicated air compressors.
- Instrumentation Air: dry high-pressure air will come from the plant air compressors; it will be dried and stored in a dedicated air receiver.

14.4.11 Assay/Metallurgical Laboratory and Quality Control

The final concentrate and intermediate streams will be monitored by one on-stream sample analyser with three multiplexers to analyse a total of 16 streams for metal contents. The measured data will be fed back to the central control room and used to optimize the process operation. Routine samples of head, intermediate products, tailings, and final products will be collected and assayed in the assay laboratory, where standard assays will be performed. The data obtained will be used for product quality control and routine process optimization.

The site assay laboratory will consist of a sample preparation facility, a metallurgical laboratory and a full set of assay instruments for base metal analysis as well as gold and silver assays. The metallurgical laboratory will perform metallurgical tests for quality control and process flowsheet optimization. The laboratory will be equipped with laboratory crushers, ball mills, particle size analysis devices, laboratory flotation cells, balances, and pH metres. The assay laboratory will be equipped with the following equipment, including:

- Fire assay equipment;
- An Atomic Adsorption Spectroscopy (AAS);
- An ICP for the routine assay laboratory;
- An ICP-MS for the environmental laboratory;
- A Leco furnace;
- Other determination instruments such as pH and redox potential meters and experimental balances.

14.5 Personnel

The number of process operations and maintenance personnel is provided in Section 18.2.3.4.

15 INFRASTRUCTURE

15.1 Introduction

The proposed Arctic mine is a greenfield site, remote from existing infrastructure. Infrastructure that will be required for the mining and processing operations will include:

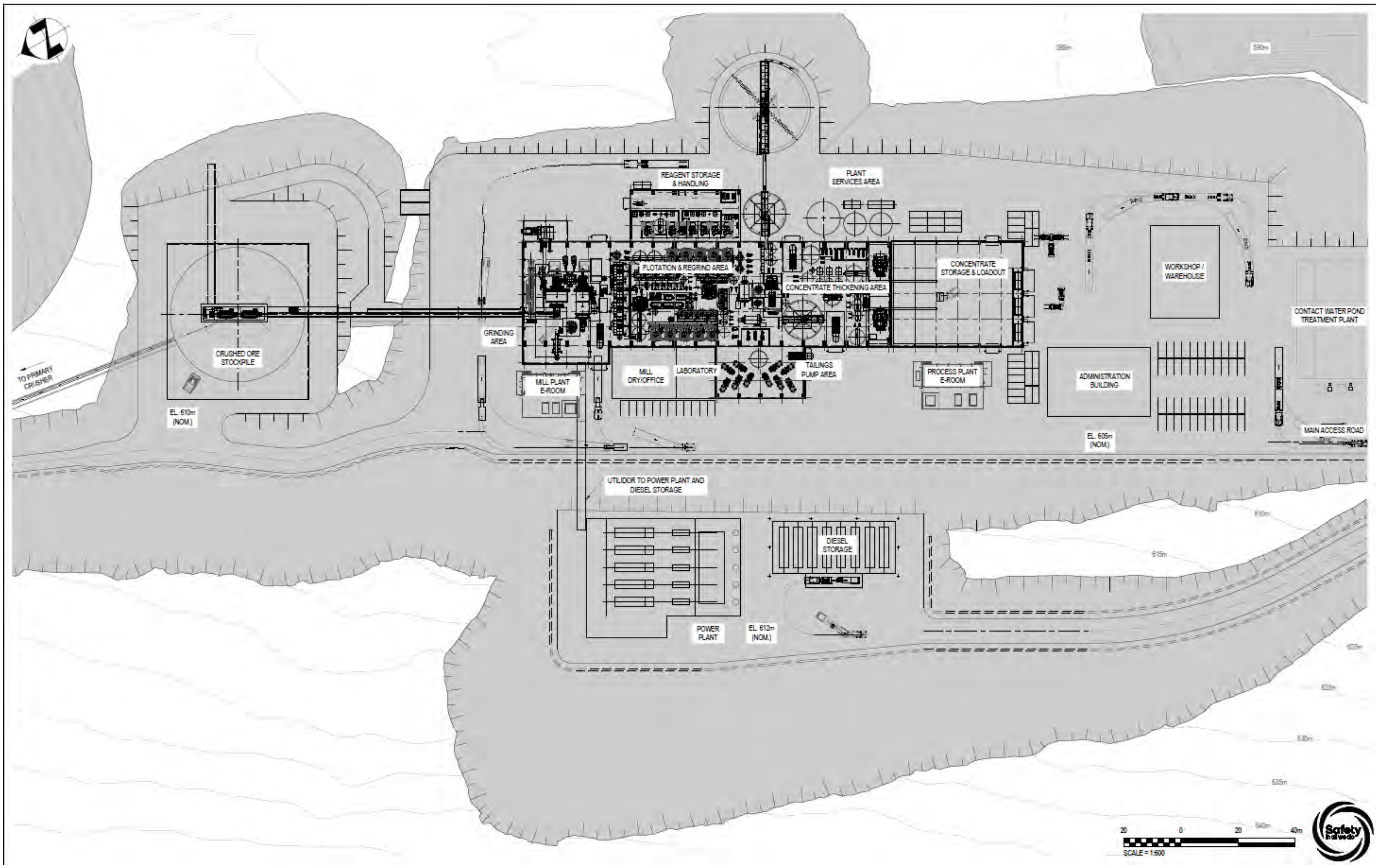
- Open pit mine
- Stockpiles and WRF
- Truck workshop, truck wash, mine offices, mine dry facility and warehouse
- Power house
- Administration building
- Mill dry facility
- Plant workshop and warehouse
- Primary crushing building
- Fine ore stockpile building
- Process plant and laboratory
- Concentrate loadout building
- Reagent storage and handling building
- TMF
- Diversion and collection channels, culverts, and containment structures
- WRCP
- Process water pond
- Water treatment plants (WTPs)
- Camp

Figure 15-1 shows the proposed site layout and Figure 15-2 shows the proposed locations of the processing plant, truck workshop and mine offices (collectively referred to as the mine infrastructure area) and administration buildings.

Figure 15-1: Proposed Site Layout

Source: Ausenco, 2022.

Figure 15-2: Proposed Location of the Processing Plant and Other Buildings



Source: Ausenco, 2022.

15.2 Access Roads

The Arctic Project site will be accessed through a combination of State of Alaska owned highways, a proposed AIDEA owned private AAP road, and proposed Trilogy-owned access roads.

15.2.1 Ambler Mining District Industrial Access Road

The Arctic Project assumes that the ADIEA will construct a road connecting the Arctic Project to the rest of Alaska's transportation infrastructure. The access road is referred to as the AMDIAP or the AAP.

There is currently no road access to the Arctic Project area. Access to the Arctic Project area is proposed to be via the AAP road, which is approximately 340 km (211 miles) long, extending west from the Dalton Highway where it would connect with the proposed Arctic Project area. The AAP road will be permitted as a private road with restricted access for industrial use and received a Federal Record of Decision on July 23, 2020, by the Bureau of Land Management (BLM) and the National Park Service (Joint Record of Decision or JROD). Lawsuits were filed shortly thereafter by a coalition of national and Alaska environmental non-government organizations in response to the BLM's issuance of the JROD for the AAP.

On January 6, 2021, BLM, the National Park Service and AIDEA signed Right-of-Way agreements giving AIDEA the ability to cross federally owned and managed lands along the route for the AAP approved in the JROD. During the second quarter of 2021, AIDEA signed a land access agreement with Doyon Limited to conduct feasibility and permitting activities to advance the AAP and in September 2021 AIDEA signed a land access agreement with NANA Regional Corporation, Inc. to conduct similar activities.

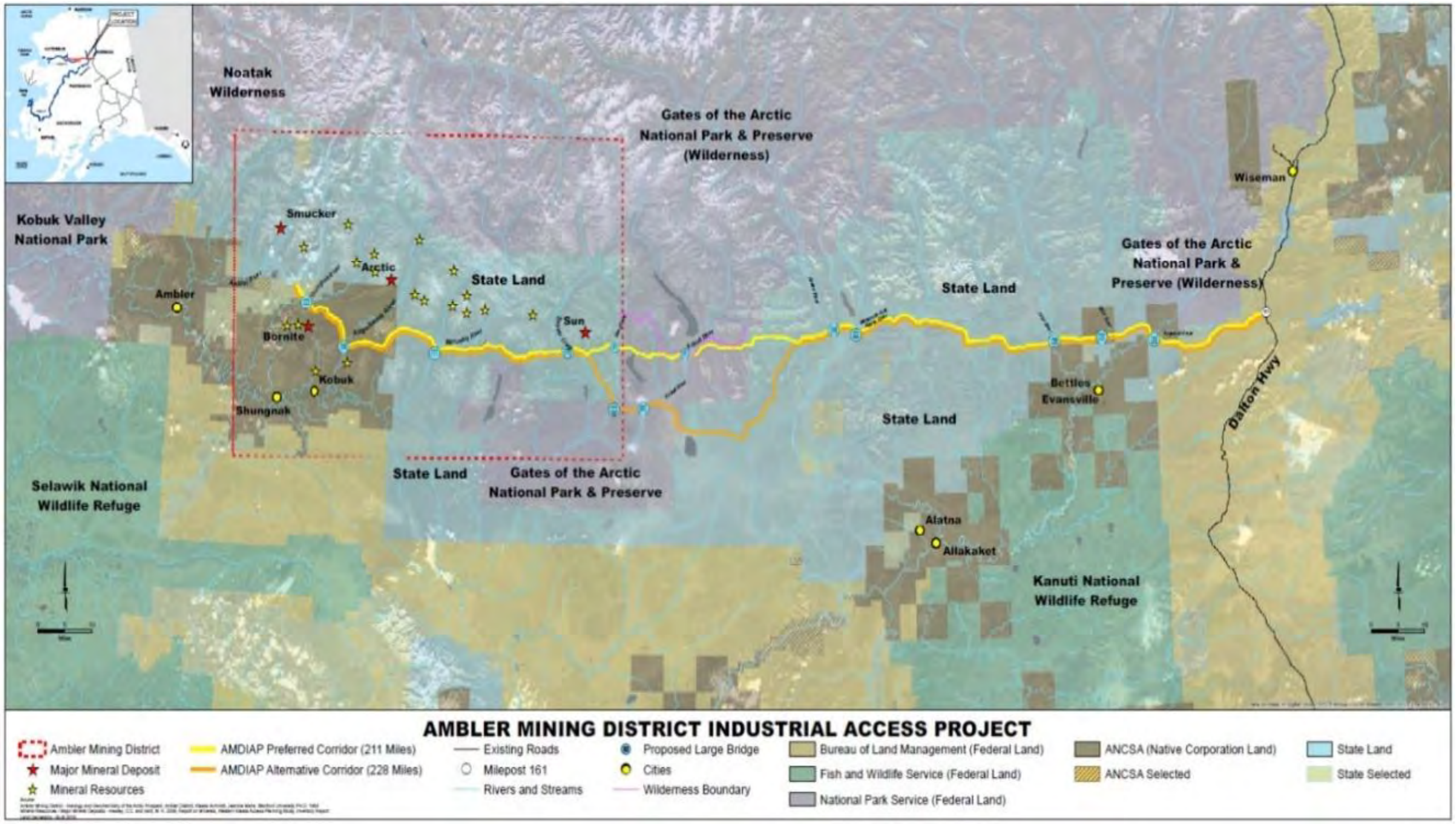
On February 22, 2022, the United States Department of the Interior (DOI) filed a motion to remand the Final Environmental Impact Statement (EIS) and suspend the right-of-way permits issued to AIDEA for the AAP and in mid-March, the BLM and DOI suspended the right-of-way grant and the right-of-way permit over federal lands.

On September 20, 2022, the BLM published in the Federal Register a Notice of Intent that it will prepare a Supplemental Environmental Impact Statement (SEIS) for the proposed Ambler Mining District Industrial Access Road. The BLM anticipates publishing a draft SEIS during the second quarter of calendar year 2023 and a final SEIS within the fourth quarter of calendar year 2023.

The lawsuits have been temporarily suspended pending the additional work to be performed by the BLM on the EIS.

Figure 15-3 shows the proposed route of the AAP road.

Figure 15-3: Proposed Route of AAP Road



Source: Ambler Access Website, 2018.

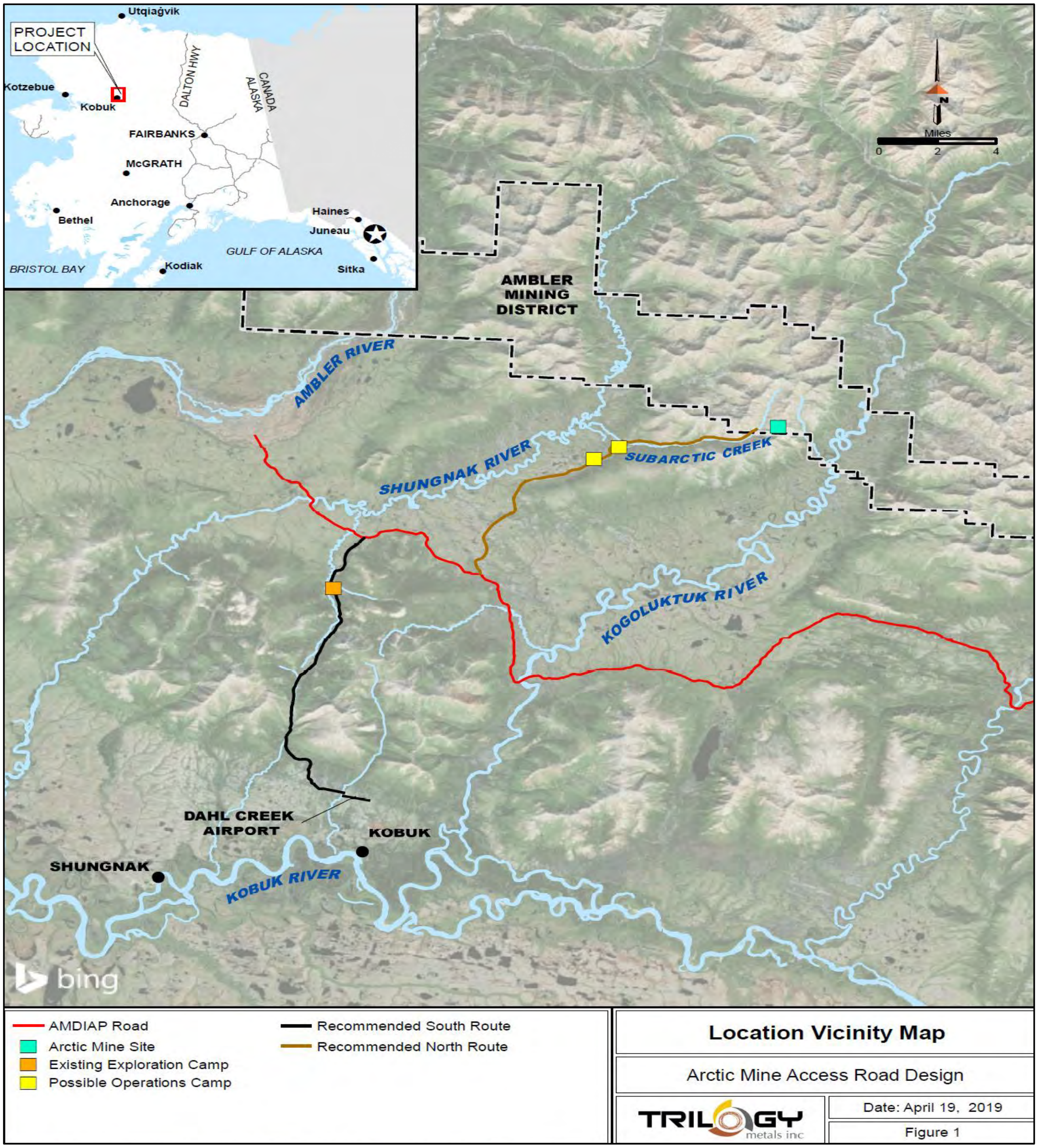
15.2.2 Arctic Access Road

The Arctic Project envisions the development of two access roads to support the proposed Arctic mine: a southern route that will connect the Dahl Creek airport to the AAP road, and a northern route that will connect the AAP road to the Arctic Mine, as seen in Figure 15-4. Development will include upgrading existing roads and trails and construction of new road segments where they currently do not exist. The roads will be designed to accommodate the types of vehicles expected to serve the intended mine and process plant operations.

The south route will be 21.4 km long and will be used to transport employees and air freight from the Dahl Creek airport to the Arctic mine. The first 17 km will generally follow the alignment of the existing road between the airport and the existing exploration camp. The remaining 4.5 km to the junction with the AAP road will require new construction.

The north route will be 22 km long and will support operations at the Arctic mine by transporting employees, mining equipment, supplies, and ore concentrate to and from the mine site. Approximately the first 8.8 km of the north route will be new construction across the Ambler lowlands. The remaining 13 km will upgrade an existing undeveloped summer/winter trail, including 7.7 km that extend up a narrow and steep valley to the Arctic mine site.

Figure 15-4: Arctic Access Road



Source: Trilogy Metals, 2019.

15.3 Airstrip

The Dahl Creek airport is situated approximately 32 km south of the Arctic deposit. The airport would need to be upgraded with a lighting system and an automated weather observation system to be functional for Project purposes. These upgrades would support the use of Dash 8 aircraft for transporting crew to and from the Fairbanks International airport. The Dahl Creek airport is owned by the State of Alaska, and it is assumed that the upgrades would be accepted if funded by the Arctic Project.

New infrastructure that would be associated with the airstrip would include:

- Pre-fabricated 90 m^2 passenger building to accommodate personnel waiting to depart the site.
- Fabric structure approximately 175 m^2 in area to accommodate snow-removal and aircraft service equipment.
- Precision approach pathway indicators on each end of the runway to aid pilots in acquiring and maintaining the correct approach.
- Local utilities such as generator power and water services.

15.4 Camps

The Arctic Project will require the use of three different accommodation camps, with one remaining for permanent operations. Each camp will generally be self-contained and have its own power generation and heating capabilities, potable WTP (PWTP), garbage incineration and sewage treatment plant (STP).

15.4.1 Bornite Exploration Camp

The existing Bornite Camp, which is currently used for exploration, is envisioned to be utilized initially for the Arctic Project. The Bornite Camp can currently hold approximately 90 persons. The Bornite Camp will be used to support the initial construction of the Arctic Access Road, and for the initial construction camp. It is anticipated that the camp will be temporarily expanded with seasonal accommodations to allow for about 30 additional persons during construction. Any improvements or expansion will be based on the same airlift transportation logistics as currently utilized to support the Bornite Camp.

15.4.2 Construction Camp

This initial Construction Camp (CC) will be constructed along the Arctic Mine Access Road approximately 8 km from the intersection of the AAP road and Arctic access road, across the road from the Arctic Mine Junction (AMJ) Logistics Yard. The camp will be constructed to provide room and board for 250 persons. The camp will be designed to operate 365 d/a, and of a modular type that can be transported to the site via the Dalton Highway on a semi-prepared AAP road route prior to full truck access being available to the site. The camp will be self-contained and will be integrated and operated as part of the Permanent Accommodations Facility (PAF). It is anticipated that the CC will be demobilized or shuttered upon the commencement of mill operations and the construction phase requirements completed. The transportation and construction of the CC will include the construction of the permanent PWTP, STP, and Garbage Incinerator that will be eventually used to support both the CC and PAF with a combined total occupancy of 650 persons.

15.4.3 Permanent Accommodations Facility

The PAF will be constructed on the same pad as CC. The PAF will be constructed when the AAP road is available for the transport of the modules, and the pad is upgraded for the permanent construction. The PAF will be located approximately 8 km from the intersection of the AAP road and Arctic access road, and 12 km from the process plant area. The PAF will be operating for approximately 2 years prior to the commissioning of the Arctic mill and production of concentrates. During this period, it will be used to accommodate both construction personnel and the initial Ambler Metals Mine Operations and support personnel required for pre-production mining.

An indicative breakdown of personnel onsite during operations is shown in Table 15-1.

Table 15-1: Personnel Onsite during Operations

Personnel Description	Number of Personnel Onsite
Owner Personnel	
General and Administration	30
Mill Operations	47
Mill Maintenance	101
Mine Operations and Maintenance	135
Contractor Personnel	
Concentrate Trucking Contractor	21
Camp Contractor	29
Other Contractor	8
Other Guests and Non-Employees	7
Total Personnel	**378**

15.5 Fuel Supply, Storage and Distribution

Diesel will be stored onsite in double-walled steel tanks for use in the power plant at the processing facility and for use in the mine fleet vehicles with a capacity of 30,000 US gallons. A total of 12 horizontal tanks will be used, four tanks within the mine infrastructure area for mine fleet fuel requirements, and eight tanks adjacent to the power plant for generator fuel.

The volume stored at the mine infrastructure area will represent nine days of storage and the volume stored at the power plant will represent seven days of storage.

15.6 Power Generation

The plant power supply will be generated on site, with no utility interconnection. A total of five diesel generator sets will be installed to operate in parallel with four operating and one on standby. Each generator will be rated 13.8 kV, 5.5 MW, and the total power output capacity will be 21.6 MW excluding the redundant unit. All five generators will be housed within a shared power plant building; the area adjacent to the building has been left free to allow for addition of a sixth generator in the future if required.

15.7 Electrical System

The primary power distribution will be at 13.8 kV and will run via above ground cable trays to the area electrical rooms. There will be a total of four major electrical rooms, one of which will be part of the power plant building. The remaining three will be prefabricated modularized type buildings that will be shipped with all equipment pre-installed; these will be placed in the grinding, process and crushing areas. The smaller electrical room for the reclaim water barge will be supplied integral with the barge.

The total connected load for the plant will be 25.9 MW with a normal operating load of 21.0 MW.

15.8 Surface Water Management

The proposed mine development is in the upper reach of Subarctic Creek valley, a tributary of the Shungnak River. The combination of the climate, terrain, soils and sparse vegetation result in high runoff potential, especially in May and June when soils are still frozen.

The catchment area of Subarctic Creek is approximately 25.8 km^2. All mine infrastructure and mine affected water is within the Subarctic Creek valley. A surface water management system will be constructed to segregate contact and non-contact water. Non-contact water will be diverted around mine infrastructure to Sub-Arctic Creek. Contact water will be collected and treated prior to discharge to Subarctic Creek.

The objectives of the water management system are to:

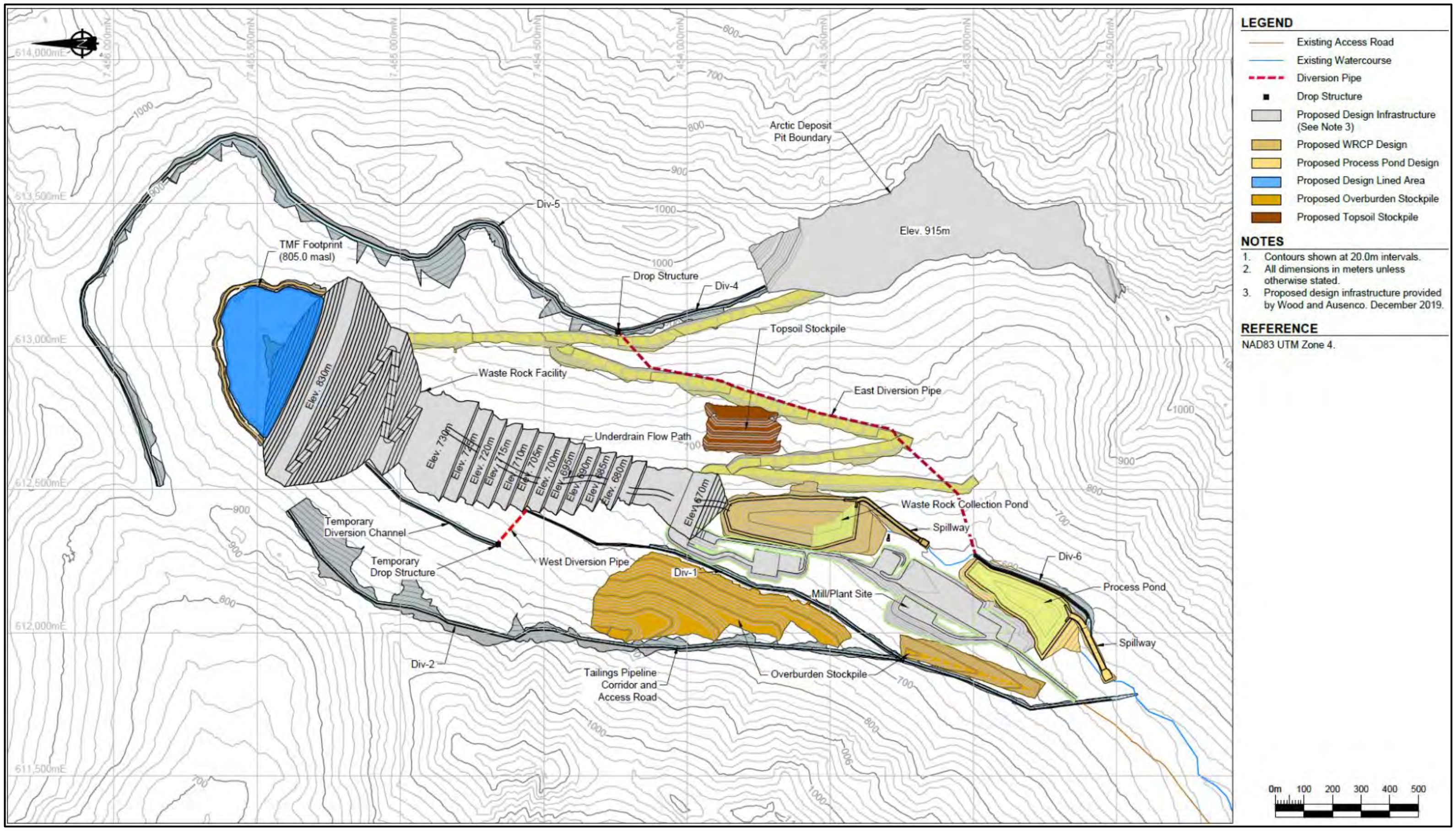
- Ensure sufficient water quantity is available to support processing.
- Manage contact and non-contact water separately to minimize volume of contact water collected on site.
- Collect and treat contact and high-sediment water that could otherwise impair the water quality of the receiving streams.
- Protect mine infrastructure from damage of unmanaged run-off.
- Reduce suspended solid loading in surface runoff prior to discharge.
- Store process water to provide for mill water needs

Water on site will be managed as one of the following:

- Non-contact water: non-contact water will be diverted away from mine infrastructure to the extent possible to reduce infiltration into the WRF, TMF inflows, and pit inflows. Non-contact water will be discharged to Subarctic Creek during operations and closure.
- Contact water: Contact water will be generated when precipitation or run-on comes in contact with mine waste rock, tailings or is collected in the pit. Contact water is treated prior to discharge to Subarctic Creek.
- High-sediment water: High-sediment water (i.e., surface flows from pads and topsoil stockpiles) is collected and conveyed to downstream ponds to treat prior to discharge in Subarctic Creek, during operations and closure.

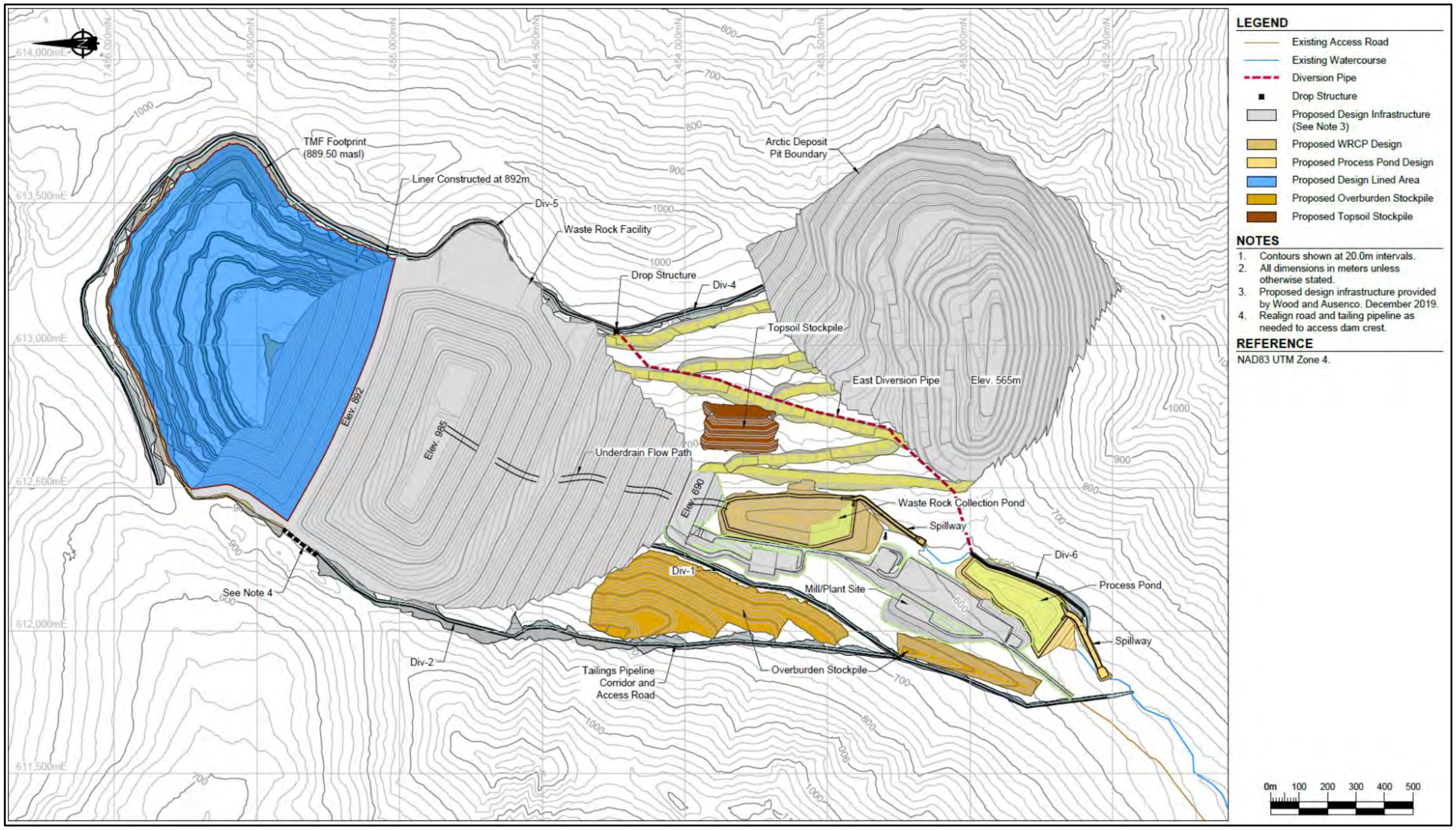
Figure 15-5 and Figure 15-6 illustrate the water management plan for the initial construction phase and the LOM footprint of the Arctic Project.

Figure 15-5: Surface Water Management Plan During Operations (Year 2)



Source: SRK, 2022.

Figure 15-6: Surface Water Management Plan at the End of LOM (Year 13)



Source: SRK, 2022.

15.8.1 Process Water Supply

Process water, gland water, and reagent make-up water will be supplied from the Process Pond and TMF reclaim. Additional process water will be supplied from the WRCP to the TMF through the winter months to meet mill water demands and provide water stored in the TMF as a contingency for dry years. Table 15-2 summarizes mill water supply throughout the year. The TMF dam, WRCP, and Process Pond will be constructed prior to the processing of ore. A water volume of 1.6 Mm^3 will be collected in the TMF prior to operations to supply the mill during start-up. Water that accumulates in the WRCP during construction may need to be treated for suspended solids by a temporary water treatment system.

Table 15-2: Process Water Supply Summary

Months	Supply Source	Comments
January to April	Process Pond, TMF reclaim water and WRCP	Water from the WRCP is pumped to the TMF from January to April. This water is accumulated from groundwater flows during the winter months.
May to December	Process Pond and TMF reclaim water	No WRCP water is pumped to the TMF. From May to December WRCP water is treated and discharged to Sub-Arctic Creek.

15.8.2 Water Management Infrastructure

Stormwater will be managed within the Arctic Project boundaries by capturing and conveying contact and high sediment water for treatment and diverting non-contact water away from developed areas. Water conveyance and storage facilities proposed include the following structure types:

- Diversion and collection channels.
- Drop structures, diversion pipes and culverts.
- Collection ponds.

Soils found within the Arctic Project limits consist of alluvial sands and gravels, with limited fines. Runoff from the overburden stockpile and mill site areas should be mitigated by temporarily seeding or using material to create durable less erodible surfaces to prevent or reduce erosion. In addition, water could be recirculated into the TMF or mill flow stream in extreme circumstances. Sediment that accumulates in the Process Pond will be removed periodically to maintain the design capacity.

15.8.2.1 Diversions and Collection Channels

The steep mountainous terrain and compact development footprint will require additional geotechnical and geochemical studies to confirm soil and bedrock depths along the proposed conveyance alignments and acid rock drainage/metal leaching potential. Bedrock is expected to be relatively shallow creating challenges for construction, however shallow bedrock may increase the collection efficiency of surface and shallow sub-surface water interception. In addition, construction of channels within bedrock may eliminate the need for riprap protection while reducing potential seepage through soils in the channel bottom.

Diversion channels will be constructed around the perimeter of the TMF and WRF, above the overburden stockpiles, and above the mill site infrastructure area. The channels will convey non-contact water runoff to Subarctic Creek. Runoff from haul roads is anticipated to be managed within the road footprint via roadside ditches with runoff reporting to the WRCP. The main purpose of the diversion channels is to reduce the amount runoff contacting mined materials. The diversion of non-contact water will reduce water treatment costs, and storage volume requirements. Channel construction will be phased during the construction period, and prior to commencement of mining to maximize non-contact water diversion.

Non-contact and contact water diversions are designed to convey the peak flow from a 100-year 24-hour rain-on-snow event. All channels will have a minimum depth and width of 1.0 m with lining either consisting of bedrock or riprap to provide erosion protection. Riprap will be sized to provide a stable non-erodible conveyance route, with riprap of reasonable size sourced locally on-site (max median particle size, D50 ≤ 300-450 mm). The riprap thickness is specified at two times the D50 and will be placed on geotextile fabric to limit erosion below the armour and reduce sediment loading downstream. Channel depths will include a minimum freeboard of 0.3 m above the design peak flow depth to accommodate potential ice or sediment accumulation, unforeseen climatic events, or other uncertainties.

A 4-m wide access road will be located adjacent to each of the channels to provide access during construction and then for maintenance and cleaning of ice and sediment accumulation during operations and closure.

Localized seeps are anticipated to be encountered along the alignments during construction. A thorough survey prior to development of engineered design is required to identify areas of potential seep locations and accommodate mitigation measures as needed. Seepage collection and management may require widening of channels in the immediate area to allow for ice build-up during the late fall as conditions change from flowing to frozen water, or to intercept and manage subsurface flowing water within talus slopes. Maintenance of these seeps during the winter could prove challenging during freeze-up and prior to freshet. Prior to freshet flows, channel maintenance will be critical to ensure that adequate freeboard is available, and run-off is contained within the channels.

15.8.2.2 Culverts

Culverts will be required to manage contact water intercepted on the haul roads, and to pass diverted non-contact water through access and haul road corridors. The haul road corridors will traverse the hillside above the WRCP, between the pit and the WRF. The switch-backing layout of the haul roads will present a challenge to manage contact water for each of the small catchment areas, which will change throughout the project life. However, the location of the WRCP immediately down slope of the haul road corridor will allow for contact water to be collected in roadside channels and discharge downslope towards the WRCP for collection. The location and sizing of the culverts along the haul road should be incorporated into the haul road design. A small allowance has been included in the capital expenditures for culvert construction.

A major pipe structure (the eastern diversion pipe) will be located above the haul road corridors on the eastern edge of the Arctic Project. This pipe will collect and convey non-contact water collected upslope of diversion channels DV-4 and DV-5 and discharge into Subarctic creek via DV-6 through the Process Pond spillway. Concrete inlet or drop structures will be constructed at the terminus of the channels connecting to the diversion pipe.

A temporary diversion channel and pipe (upper reaches of DV-1) will be constructed during the pre-operations period on the western boundary of the WRF. The temporary diversion channel will terminate in an excavated sump, pre-cast manhole or constructed concrete manhole. A pipe will be installed in the sump/manhole conveying water immediately downslope into a receiving channel. The temporary diversion channel pipe, and sump will be buried at the end of the pre-operations period. The temporary channel will be breached periodically at the boundary of the WRF to prevent contact water entering the non-contact water system downstream.

All pipe structures are designed to accommodate the 100 year, 24-hour rain-on-snow event. The culverts will be trenched in and backfilled with engineered fill. Diversion pipes located on hillsides will be stabilized by burial or mounding of soils sourced in the immediate area, trenching and backfill, or placed on a prepared bed and secured with concrete thrust block anchors if bedrock depths prevent conventional trenching or burial anchors. All pipes will discharge onto a stabilized rip-rap apron to reduce erosion potential.

15.8.2.3 Collections Ponds

Contact and high sediment water will be routed to the WRCP. Water collected in the WRCP will be pumped to the WTP or TMF for use as mill make-up water. Water collected in the Process Pond will be used as make-up water to the mill.

15.8.3 Waste Rock Collection Pond

The WRCP will be located directly below the toe of the WRF. The pond will collect seepage from the WRF, runoff from the WRF and haul road corridor area, runoff from the ore stockpile on the ROM pad, and water pumped from the pit. The pond is sized to store the 100-year, 24-hour rain-on-snow storm volume plus two days of average snow melt runoff and operational volume. The pond capacity will be approximately 700,000 m^3 to the spillway invert of 640.5 masl with a crest height of 642 masl.

Water from the WRCP will be pumped to the TMF for reuse as process water or to the treatment plant to be treated and discharged to Sub-Arctic Creek. The WTP operates from May to December during operations but may be operated year-round if needed during extreme water years.

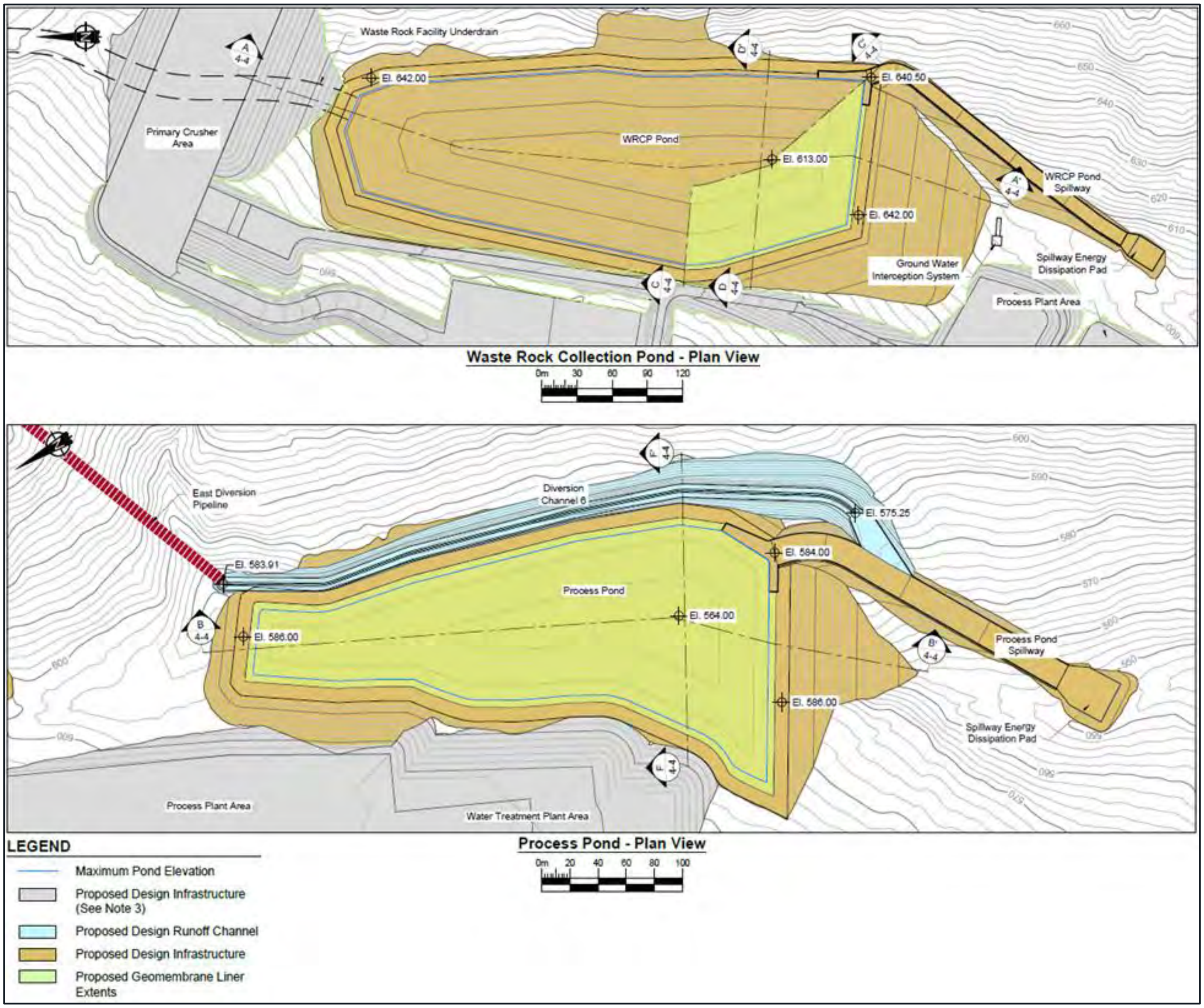
The dam will be constructed using overburden and bedrock material from the WRCP footprint or a suitable borrow source. Excess overburden will be stockpiled for use during closure activities.

Overburden material below the footprint of the dam will be removed and backfilled with well drained soils (as needed) to provide foundation stability. The excavation limit and dam foundation design requires additional geotechnical investigation. The design presented assumes all material upstream of the embankment centreline will be excavated to bedrock and material downstream excavated to suitable soils. The embankment will be constructed in 300 mm lifts, or as determined by the engineer. The downstream face of the dam will be constructed to 3H:1V while the upstream face will be 2.5H:1V. The upstream face of the pond will be lined with a geomembrane liner connected to a concrete plinth (curb) and grouted curtain wall to maximize interception of surface water and shallow groundwater.

A concrete-lined spillway will be constructed to manage flows up to the potential maximum flood (PMF) event. The spillway will include an intake weir and collection throat prior to discharging in the spillway. A stilling basin will be located at the outfall of the spillway, to reduce flows to non-erosive velocities. The spillway length could be reduced by using one of various methods of energy dissipation within the channel; however, energy reduction in the channel generally requires additional depth of flow, resulting in larger concrete requirements. The construction cost would likely be similar. A riprap-lined spillway was investigated; however, steep terrain, lack of geotechnical information, and the channel configuration would likely result in a large costly excavation that could encroach on the haul road located up-slope of the WRCP.

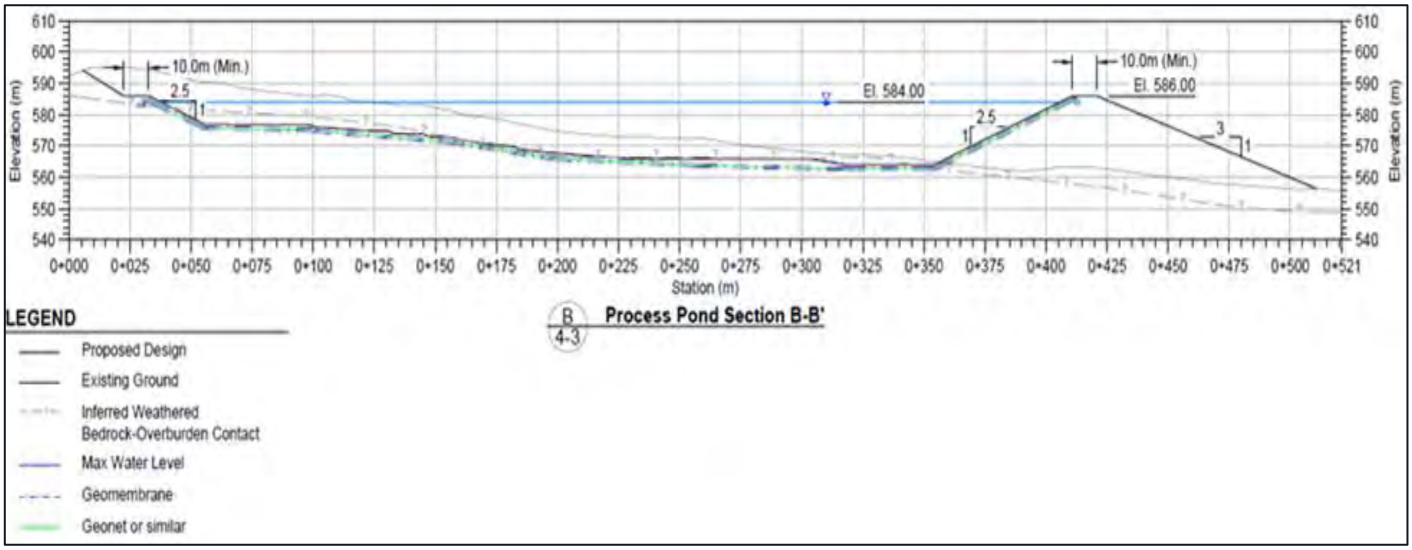
Figure 15-7, Figure 15-8, and Figure 15-9 illustrate the proposed design of the WRCP and process water pond.

Figure 15-7: WRCP and Process Pond Plan



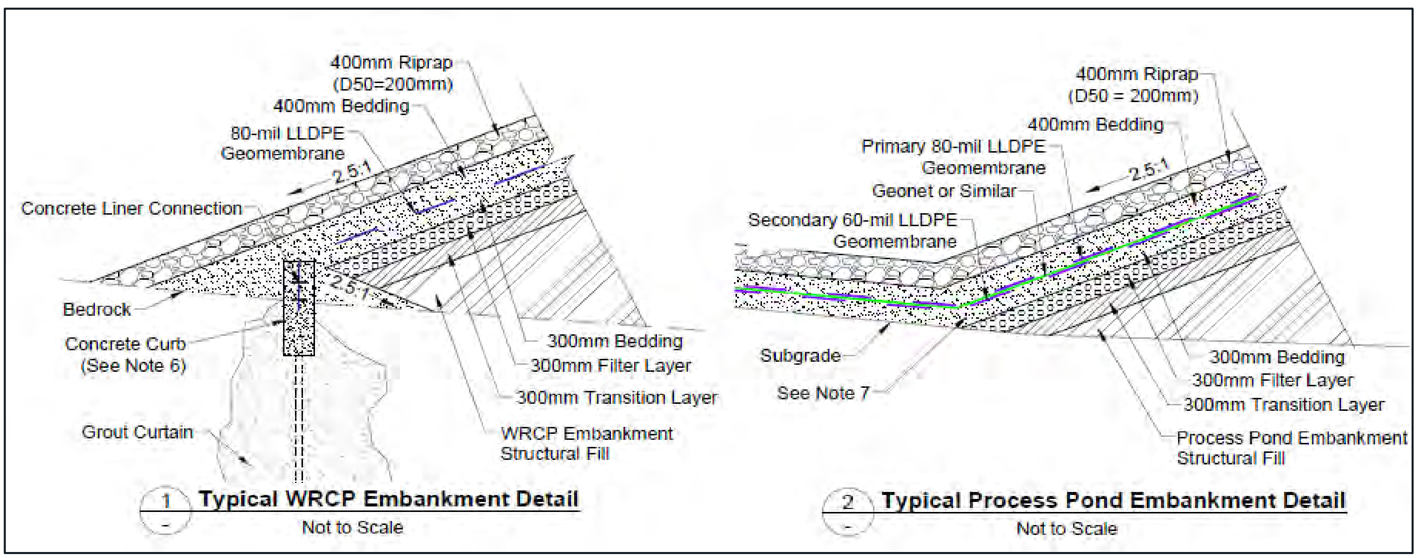
Source: SRK, 2022.

Figure 15-8: WRCP and Process Pond



Source: SRK, 2022.

Figure 15-9: WRCP and Process Pond



Source: SRK, 2022.

15.8.4 Process Pond

The Process Pond will be located directly below the process plant. The pond will retain water from the tailings thickener overflow for a minimum of 10 days. The pond capacity will be approximately 300,000 m³ to the spillway invert of 584 masl with a crest height of 586 masl.

The dam will be constructed using the overburden and bedrock material stripped from within the Process Pond footprint or a suitable borrow source. Excess overburden will be stockpiled for use during closure activities.

Overburden material below the footprint of the dam will be removed and backfilled with well drained soils (as needed) to provide foundation stability. The excavation limit and dam foundation design requires additional geotechnical investigation. The design presented assumes all material upstream of the embankment centerline will be excavated to bedrock and material downstream excavated to suitable soils. The embankment will be constructed in 300 mm lifts, or as determined by the engineer. The down-stream face of the dam will be constructed to 3H:1V while the upstream face will be 2.5H:1V. The pond will be lined to crest level with 80-mil primary and 60-mil secondary LLDPE geomembrane liners. A geonet or similar drainage layer will be installed above the secondary liner to convey any leak through the primary liner to a leak collection and recovery sump (LCRS). An underdrain will be constructed beneath the process pond to convey any shallow groundwater below the pond and prevent liner uplift.

A concrete-lined spillway will be constructed to manage flows up to the potential maximum flood (PMF) event. The spillway will include an intake weir and collection throat prior to discharging in the spillway. A stilling basin will be located at the outfall of the spillway, to reduce flows to non-erosive velocities. The spillway length could be reduced by using one of various methods of energy dissipation within the channel; however, energy reduction in the channel generally requires additional depth of flow, resulting in larger concrete requirements. The construction cost would likely be similar. A riprap-lined spillway was investigated; however, steep terrain, lack of geotechnical information, and the channel configuration would likely result in a large costly excavation that could encroach on the haul road located up-slope of the WRCP.

15.8.5 Groundwater Management System

Results from the preliminary water and load balance model suggest that small amounts of groundwater could bypass the WRCP, and a cut-off wall may need to be constructed to prevent degradation of water quality in Subarctic Creek. A groundwater interception system was designed downgradient of the WRCP. This system would include:

- Shallow collector trench across the Sub-Arctic valley bottom for overburden water.
- Small diameter collector pumping wells completed in bedrock for deep water bypass.
- Downgradient performance monitoring wells with completions in overburden, weathered bedrock and competent bedrock.

A second system located downgradient of all site water management infrastructure was included as a contingency for the closure period, when the open pit contains water and leakage from mine components could bypass the primary system. This system includes small diameter collector pumping wells completed in bedrock, and downgradient performance monitoring wells.

Groundwater management system operation would be guided under an adaptive management plan or trigger action response plan.

15.8.6 Site Water and Load Balance

The water balance model is based on the current water management plan and was used to predict water use, surpluses, and deficits for the site and mine water management infrastructure (TMF reclaim pond, Process Pond, and WRCP) over the 13-year mine life through to closure and post-closure. Mine facility footprints vary according to the mine plan over the LOM. The model evaluated the following hydrological conditions:

- Average hydrological conditions with mean annual precipitation of 1,219 mm/a over the LOM.
- Consecutive 1-in-25 year wet years with annual precipitation of 1,816 mm/a in years 12 and 13.
- Consecutive 1-in-25 year dry years with annual precipitation of 743 mm/a in years 12 and 13.

The load balance integrated source terms with the water balance. Source terms define the water chemistry of each water type. For potentially acid generating materials in the WRF and pit, two source terms were developed:

- Operations source terms prior to onset of acid generation.
- Closure source terms post-onset of acid generation.

PAG waste rock was assumed to generate acid at closure. This change in source terms increased the mass leaching from the WRF and pit during closure.

New and existing source terms will need to be updated to incorporate the revised water management plan and updated hydrology assessment. Loading from sources requiring updates are estimated to contribute less than 20% of the total load to the WTP for constituents of concern during operations and Phase 1 of closure but may account for up to 25% to 60% of the total load to the WTP in Phase 2 of closure. This indicates the model may underpredict WTP influent concentrations in Phase 2 of closure. Subsequent revisions of the model will incorporate updated source terms.

Constituent mass loads and flows were mixed to estimate concentrations at various locations on the site and in the receiving environment. Contact water will be treated to the Alaskan Water Quality Standards (WQS; 18 AAC 70), as described in Alaska Water Quality Criteria Manual for Toxic and Other Deleterious Organic and Inorganic Substances (ADEC 2008) prior to being discharged to Subarctic Creek.

The results from the water and load balance indicate that during operations, excess water from the WRCP will need to be treated prior to discharge to the receiving environment. During closure, the WRCP and water from dewatering of the TMF will be pumped to the pit and will also need to be treated for selenium and other constituents prior to discharge to the receiving environment (see description in Section 17.3).

15.9 Water Treatment Plant

The results from the water and load balance were used to develop a water treatment strategy. The treatment processes were defined based on the approximately 25 parameters predicted in the load balance to be present in WTP influent at peak concentrations above their WQS in the Subarctic Creek or the Shungnak River, including pollutants with a range of treatment mechanisms—cationic metals (such as cadmium and copper), anions (sulphate, fluoride), forms of nitrogen (ammonia, cyanide, and nitrate), selenium, other oxyanions (such as arsenic), and temperature.

Key aspects of the selected water management overall approach include:

- Assume the site will be assigned water quality-based effluent limits matching the state's WQS for the nearby Creek or River. Therefore, design the WTP to treat to all parameters to the WQS applicable to Subarctic Creek, so that site is not reliant on receiving a permit allowing a mixing zone for compliance.
- Provide a single WTP, built in stages.
    - The WTP will initially consist of a chemical/physical plant with reverse osmosis (RO). In the Operations phase, when the TMF is in use, the RO reject will be sent to the TMF, and only RO permeate will be discharged to the Creek. The plant will include:
        - Chemical treatment: Includes addition of caustic, soda ash, ferric chloride, and polymer into mixed reactor tanks. Solids formed are then settled out and removed in a clarifier. This step removes a significant portion of cationic metals such as cadmium and zinc. Liquid chemicals will be stored in tanks; soda ash will be stored in silos.
        - Ultrafilter: Protects the RO system by removing suspended solids carrying over from the chemical treatment system.
        - RO: Concentrates contaminants into the RO reject stream. Permeate water quality will be within WQS.
        - Reconstitution: RO permeate is reconstituted prior to discharge to Subarctic Creek. A calcite contactor tank dissolves carbonate into the discharge water to maintain alkalinity greater than 20 mg/L as calcium carbonate. Caustic and hydrochloric acid are provided as needed for pH adjustment.
    - When the TMF is closed at the end of the Operations phase, a biological/chemical/physical plant will be added to treat the RO reject. This phased approach provides multiple benefits, including allowing time to gain insight on the site's true wastewater flow and quality so that the RO reject treatment design can be improved upon. This treatment train will include:
        - Gypsum clarifier: Reduces sulphate concentration in the reject treatment stream train to the gypsum saturation limit by adding lime stored in silos. The system will also remove cationic metals such as cadmium and zinc.
        - Anaerobic biological treatment: Biochemically reduces nitrate to nitrogen gas and selenate and selenite to elemental selenium (a solid). Selenium and metal solids removed in the sludge are settled out and removed by the biological ballasted clarifier.
        - Aerobic biological treatment: Polishes residual organics added in the upstream anaerobic biological treatment. Solids are separated via a membrane filter with iron coprecipitation for polishing selenium.
    - During the Closure phase, the WTP will either be run year-round or for partial year based on water storage requirements. During the Operations phase, mill water demands will affect site water flows such that the WTP can be shut down and the discharge eliminated during roughly half of the year (winter).
    - During Operations, sludge from the WTP will be disposed of at the TMF. During Closure, sludge from the WTP will be disposed in filter bags on a lined area which will decant solution back to the pit.
- This WTP will operate in perpetuity.

15.10 Tailings Management Facility

15.10.1 General Description

The TMF will be located at the headwaters of Subarctic Creek, in the upper-most portion of the creek valley (refer to Figure 15-6). The maximum storage capacity of the facility will be about 40.7 Mm^3 (tailings and process water) at an elevation of 889.5 m plus an additional 2.5 m of freeboard. The 59 ha footprint of the TMF will be fully lined with a geomembrane liner.

Tailings containment will be provided by an engineered dam, buttressed by the WRF constructed immediately downstream of the TMF, and the natural topography on the valley sides. A starter dam will be constructed to elevation 805 m two years prior to mine start up and then increased to 830 m by the end of the construction period. Three subsequent raises will bring the final dam crest elevation to 892 m, which is 98 m lower than the final elevation of the WRF. The expected maximum tailings production rate is 8,900 t/d and the TMF is designed to store approximately 37.4 Mm^3 (41.2 Mt) of tailings over the 13-year mine life, 3.3 Mm^3 of additional pond water, as well as 1.5 times the probable maximum flood (PMF). Instruments installed in the tailings dam are anticipated to include piezometers, settlement monuments, and shape arrays.

Tailings will be deposited as thickened slurry from the dam crest by multiple spigots. In the winter, other deposition methods, such as subaqueous deposition and/or single point discharge, may be needed to minimize ice entrainment in the tailings. At the time of this report, detailed laboratory analysis of thickened tailings is underway. For this report, a dry density of 1.1 t/m^3 was used to estimate the storage capacity of the facility. The final consolidated tailings dry density is assumed to be 1.25 t/m^3 based on simple 2 D consolidation calculations (SRK 2020). To further reduce the duration of tailings consolidation, a drainage system should be evaluated, that would be installed above the TMF liner, convey seepage from the tailings to a sump at the upstream toe of the TMF dam, and then pump the seepage into the reclaim water system.

15.10.2 Design Criteria

The basis of the TMF design is provided in Table 15-3. Values were determined from project-specific information, judgment, and experience with other projects.

The TMF design was performed in accordance with the following guidelines, standards, and regulations:

- Guidelines for Cooperation with the Alaska Dam Safety Program, July 2017.
- Alaska Administrative Code Title 11, Chapter 93, Article 3 Dam Safety.
- State of Alaska Mining Laws and Regulations, Alaska Department of Natural Resources Division of Land and Water, 2014.
- Canadian Dam Association, 2019. Technical Bulletin: Application of Dam Safety Guidelines to Mining Dams.
- Global Industry Standard on Tailings Management (GISTM), 2020.

Table 15-3: TMF Design Parameters and Design Criteria

Design Item	Criteria	Reference
Operational life of TMF	13 years	Wood
Total tailings	41.2 Mt	Wood
Annual tailings production (solids)	3 Mm^3/a	Wood
Tailings percent solids (in the slurry pipeline)	47% (Ms/(Ms+Mw))	Ausenco
Tailings solids specific gravity	3.0	SRK
Tailings settled dry density	1.1 t/m^3	SRK
Expected tailings deposition angles*	1.0 %	SRK (2020)
Tailings deposition	Spigots from dam crest (summer)	SRK
Target tailings storage requirement	37.4 Mm^3	SRK
Dam hazard classification	Class I	ADSP (2017)
Minimum freeboard above tailings	2.5 m	SRK
Tailings dam crest width	30 m	SRK
Tailings dam max elevation (height)	892 masl (192 m)	SRK
Design earthquake return period and ground motion	1:10,000 year, PGA = 0.374	ADSP (2017), GISTM (2020)
Stability Factors of Safety (FOS) (minimum)	1.0 (pseudostatic) to 1.5 (static)	CDA (2019)
Maximum allowable seismic displacement	0.5 m	SRK
Dam construction materials	Waste rock compacted in 1 m lifts	SRK
Dam downstream and upstream slopes	2.5H:1V (maximum)	SRK
Starter dam storage capacity (elevation)	1.5 Mm^3, Year -1, (805 masl)	SRK
End of construction capacity (elevation)	6.5 Mm^3, Year 0, (830 masl)	SRK
Dam Raise 1 storage capacity (elevation)	13.9 Mm^3, Year 1, (850 masl)	SRK
Dam Raise 2 storage capacity (elevation)	24.8 Mm^3, Year 4, (870 masl)	SRK
Dam Raise 3 storage capacity (elevation)	40.7 Mm^3, Year 7, (892 masl)	SRK
Storm inflow design flood without overtopping	1.5 x Probable Maximum Flood (PMF)	SRK, ADSP (2017), GISTM (2020)

Notes: *Values from 2020 Trilogy Feasibility Study, based on whole slurry tailings.

The design criteria were based on site conditions as of August 2019, on assumptions interpreted from review of available information, and on pre-feasibility-level field investigations and associated reporting. Where data were not available, pre-feasibility-level assumptions were made.

15.10.3 Overburden Geotechnical Investigation

An overburden geotechnical investigation was carried out in 2017 (SRK, 2017) and 2018 (SRK, 2018) to provide overburden characterization in support of waste facility siting evaluation and geotechnical design of the WRF and TMF.

Surficial deposits mapped in the project area include glacial, aeolian, and fluvial deposits, with colluvium-covered slopes. Most morainal deposits at the site are from the late Pleistocene Walker Lake glaciation, and consist of drift deposits characterized by boulders, cobbles, and gravels in a fine-grained matrix, as well as outwash deposits of silty sand. The Subarctic Creek valley mainly consists of well-graded silty sand and gravel colluvium and alluvium, with some zones of large cobbles and boulders. Overburden thickness within the Subarctic Creek valley generally increases down-valley towards the Shungnak River valley.

In the proposed TMF and starter dam footprints, groundwater was encountered during drilling and test pitting at depths between 0.2 and 4.9 m. Groundwater was encountered at depths between 6 and 21 m below ground in the bedrock around the perimeter of the TMF. Based on groundwater elevation records in these areas, the planned facilities will be located in groundwater recharge zones.

In the planned WRF footprint, groundwater was encountered during drilling and test pitting at depths between 0.8 m and 13.5 m. Groundwater levels are generally near the surface on the western and central parts of the footprint and deeper on the eastern side of the footprint. Drill holes on both sides of the valley appear to be located in groundwater recharge zones. The center of the valley may correspond to both recharge and discharge zones.

The groundwater depth within the footprint of each planned infrastructure within the Subarctic Creek valley is relatively shallow. The average groundwater depth is 1.5 m and seems to correspond to the weathered bedrock–overburden contact. The water depth seems to be relatively constant despite varying surface slope angles.

15.10.4 Site Selection

The TMF site was selected in August 2017 during a workshop (Ausenco 2017) to evaluate locations for the TMF and WRF.

A weighting system was applied to four broad categories including environmental concerns, permitting, capital costs, and operating costs.

The site was reviewed again in 2021 during a tailings alternative review and found to be the preferred alternative. This work included a thorough review of the GISTM (2020) and the review found the proposed design to be compliant with GISTM.

15.10.5 Starter Dam

Overburden in the TMF starter dam area is characterized as thin well-graded silty sand with gravel or silty gravel with sand (SM or GM). The contact with the underlying fractured weathered bedrock occurs at depths varying between 0.2 and 3.1 m in the TMF and between 1.0 and 9.1 m in the starter dam area. The thicker overburden occurs in the center of the valley, south of the toe of the starter dam. The overburden is interpreted to be colluvial in origin. Bedrock crops out at the north end of the TMF.

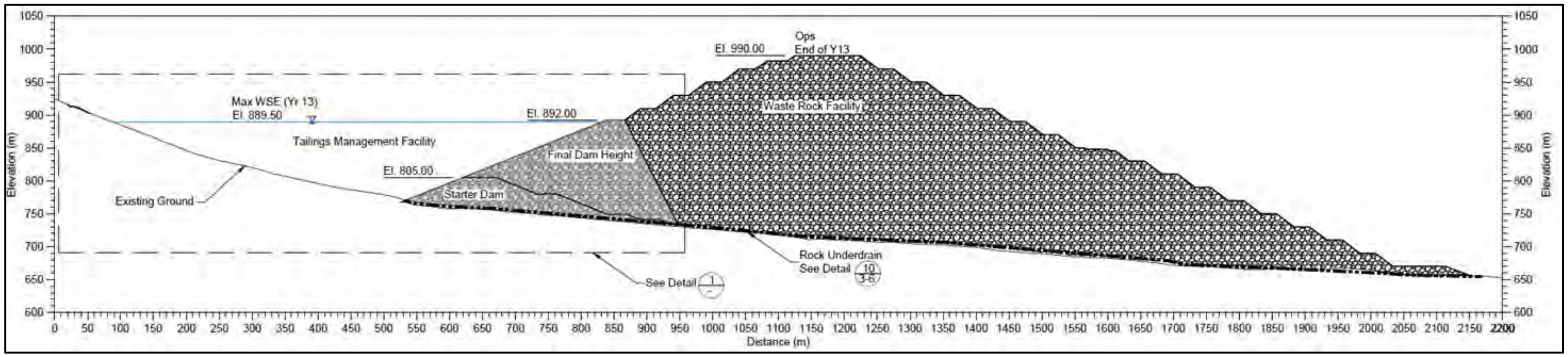
The overburden will be excavated underneath the footprint of the starter dam and removed to reduce potential settlement and deformation of the TMF dam; overburden removal will therefore reduce the potential for underperformance of the liner that will be installed on the dam's upstream face.

The topsoil and overburden material will be stockpiled below the pit and WRF, respectively, for use in future reclamation of the waste facilities.

The starter dam will be constructed in two phases to an ultimate lined elevation of 830 m, which will allow for storage of the pre-production water and approximately one year of storage (~1.6 Mm^3) to supply the mill during start-up. The upstream and downstream faces of the starter dam will be constructed at 2.5H:1V and constructed entirely of waste rock from the open pit. Waste rock will be placed in 1 m lifts and compacted by the mine fleet haul trucks. Figure 15-10 shows a cross section through the complete TMF and WRF, illustrating the starter dam in relation to the final dam and the downstream WRF.

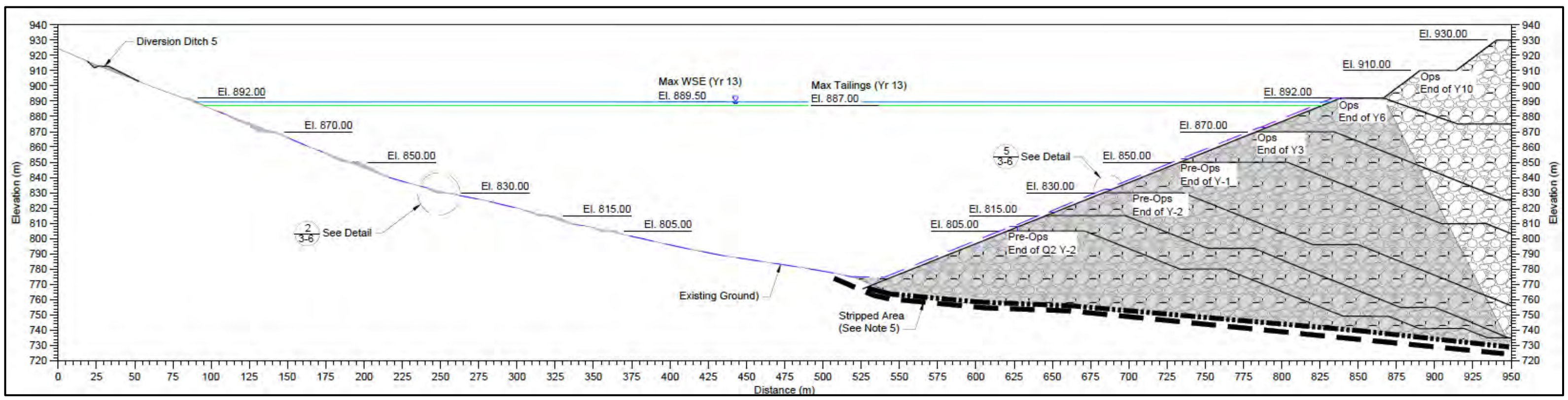
The TMF footprint and the upstream face of the dam will be lined with a textured geomembrane, placed over geotextile on prepared subgrade. To prepare for liner installation, the TMF area will be cleared, grubbed, and stripped of topsoil prior to grading and placement of the geotextile. The face of the dam will be covered with bedding material to protect the liner against puncture from potential sharp edges in the waste rock.

Figure 15-10: Cross Section of the TMF & WRF showing Starter Dam to Elevation 805 m



Source: SRK, 2022.

Figure 15-11: Cross Section of the TMF and Raises to Final Design Evaluation



Source: SRK, 2022.

15.10.6 Dam Raises and Final Dam

Three dam raises will be completed to reach the final dam height of 892 m. The construction for these raises will be completed in Years 1, 4, and 7 to elevations 850 m, 870 m and 892 m respectively. Construction will be completed using the downstream method and will connect the structural fill in the tailings dam with the uncompacted WRF. Tailings deposition will be required from the perimeter of the TMF during the construction sequence of the dam crest. The dam crest will be constructed to a width of 30 m at the end of each construction campaign.

An upstream slope of 2.5H:1V will be maintained throughout the construction of the dam to facilitate the placement of the bedding layer and the installation of the liner. The downstream portion of the dam will about the WRF at each stage of dam raising. Construction material for the dam will be waste rock compacted in 1 m lifts.

Additional liner will be placed within the expanded TMF footprint using the same procedures as for the initial starter dam. The TMF area will be cleared, grubbed, and stripped, and bedding material and geotextile placed prior to the geomembrane being laid down and seamed to the existing liner.

Figure 15-11 shows a cross section of the complete TMF and the raises constructed to reach the final design elevation.

15.10.7 TMF Water Pool and Water Return

Under normal situations, the pool will be maintained deeper towards the north portion of the TMF away from the deposition spigots. Recycle water will be obtained using a floating barge and pipeline system situated on the pool.

Tailings deposition in winter will be managed to limit ice formation in the tailings, for example, by subaqueous deposition and/or single point discharge of the tailings.

Thickened tailings may necessitate modification of pond and deposition management.

15.10.8 Tailings Delivery and Return Systems

The tailings delivery system will transport slurried tailings from the processing plant to the TMF. This will consist of one three-kilometre pipeline, with two kilometres of this being 500 mm (20 inch) diameter carbon steel rubber lined pipeline and the remaining one kilometre being 600 mm (24 inch) HDPE. This pipeline will transport up to 1,843 m^3/h of tailings to the TMF.

The return water delivery system for recycle water from the TMF has been sized on the basis of 1,308 m^3/h of water being pumped from the TMF to the process water pond. This system will consist of a barge pump and a 3 km-long pipeline, run adjacent to the tailings pipeline. The pipeline will consist of 2 kms of 450 mm (18 inch) diameter carbon steel pipeline, with the remaining 1 km being 500 mm (20 inch) HDPE.

Both pipelines will be heat-traced to prevent freezing.

15.11 Waste Rock Facility and Overburden Stockpiles

The WRF will be developed north of the planned Arctic pit in the upper part of the Subarctic valley. The waste rock placed in the northernmost portion of the WRF will be compacted to provide structural fill for the TMF.

There will also be three stockpiles to store the stripped topsoil and overburden from the TMF footprint.

15.11.1 Waste Rock Facility

The overburden in the WRF area is characterized as well-graded silty sand with gravel or silty gravel with sand (SM or GM) and angular cobbles interpreted to be colluvial in origin. Overburden is slightly thicker on the west side of the valley (± 10 m) than on the east side of the valley (± 6 m), and ranges between 3.0 and 5.6 m in the centre of the valley. Based on exposed bedrock, overburden is inferred to thin on the valley sides.

The total volume of waste rock is expected to be 162.6 m^3 (340 Mt); however, there is potential for expanded volume in the waste if placement density is < 2.0 t/m^3. Most of the waste rock is anticipated to be PAG and there will be no separation of waste based on acid generation potential. Rather, seepage from the WRF will be collected and treated. An underdrain at the base of the WRF will be built to collect and manage the seepage.

The WRF is planned to be constructed in 5 m lifts with benches set back at 23.5 m after every fourth lift, to achieve an overall slope of 2.5H:1V. The final design is expected to have variations in the slope aspect, to best manage overland flow, resulting in a transitioning convex to concave shape of the WRF face.

The maximum stacking height (above existing grade) of the WRF is planned at 340 m to an elevation of 990 masl. The effective normal stress for potential critical failure surfaces at the base could reach values up to 5,000 kPa, which is higher than the weathered bedrock's unconfined resistance (on the order of 3,000 kPa) and significantly lower that the competent bedrock resistance (± 32,000 kPa). These values were obtained from uniaxial compressive strength (UCS) tests. Considering the potentially mobilized zone under the stress (and potential strains) imposed by the stacked material, crushing of the waste rock should be expected. A sensitivity analysis was performed to assess the effect of finer-grained, granular waste rock on WRF stability.

A rockfill underdrain will be constructed under the WRF in the current Subarctic Creek channel. The underdrain will be excavated into overburden prior to WRF construction and will be capable of handling base flow through the Subarctic Creek valley. Water will be collected in the WRCP at the base of the WRF and held for treatment. The underdrain system will be used to maintain the phreatic water level as low as practicable within the WRF to increase the stability of the facility.

15.11.2 Overburden and Topsoil Stockpiles

Three stockpiles will be developed to store topsoil and overburden materials for use in final site reclamation. The topsoil stockpile will be placed in between the haul roads west of the planned pit and will store up to 325,000 m^3. The overburden stockpiles will be located south of the WRF and will store up to 2,200,000 m^3. The overburden stockpile geometries may be modified to act as or integrate with avalanche deflection berms.

15.12 Site Communications

External communications, including voice and data, will use a satellite connection with a line-of-sight to an orbiting geostationary satellite. Earth stations are expected to be located at the process plant area, camp area, contractor laydown yard proposed to be located at the road junction between the planned mine access road and the proposed AAP road, and the Dahl Creek airport. There will be separate systems for business operations and personal use.

Radio communications around the operation will be a very high frequency (VHF) Hi-Band system with radios in all mobile equipment and operations offices. There will be multiple frequencies with a frequency assigned to a specific operational group. The Dahl Creek airport will have a UHF radio to communicate with in-bound aircraft.

It is expected that there will be cell phone access for personal use in a limited area. This system will be installed and maintained by a communications service provider.

Communications for truck operators on the AAP road will be with VHF radios utilizing a system that will be installed as part of the AAP road.

15.13 Fire Protection

The firewater distribution network will be maintained under a constant pressure with a jockey pump and will be looped and sectionalized to minimize loss of fire protection during maintenance. Where run outside buildings, fire water piping will be above ground and be heat traced and insulated.

Yard hydrants will be limited to the fuel storage tank area. Wall hydrants will be used in lieu of yard hydrants, and these will be located on the outside walls of the buildings in heated cabinets.

Fire protection within buildings will include standpipe systems, sprinkler systems and portable fire extinguishers. Standpipe systems will be provided in structures that exceed 14m in height and additionally where required by regulations, local authorities or the insurance underwriter.

Camp modules will be purchased with fire detection; fire rated walls and will use separation as a means of fire protection. Handheld extinguishers will be located throughout the buildings.

Fire protection of the generators will be provided by a water mist system. Gas detection will be provided to detect dangerous levels of diesel gas within the generator building.

15.14 Plant Buildings

15.14.1 Truck Shop and Mine Offices

The mine truck shop and mine offices area will be 2,400 m^2 in area and will be positioned on an upper pad adjacent to the fuel storage area. This building will consist of the truck workshop, truck wash, mine offices, and mine dry. The truck workshop will have lifting and handling activities fulfilled by an overhead gantry crane. This building will be a pre-engineered steel frame and metal-clad building. The building will be heated using a system that will use waste heat produced from the diesel generators.

15.14.2 Laboratory

The laboratory will be 300 m^2 in area and will be situated adjacent to the process building. The building will house all laboratory equipment for the daily operational process control including the metallurgical and environmental requirements. Any mechanical items associated with the dust collection equipment will be located external to the building. The building will be constructed as a single-storey modular wood-frame building. This building will be heated using an air handler system that using waste heat produced from the diesel generators.

15.14.3 Administration Building

The administration building will consist of HR, accounting and senior site management. The building will have an approximate area of 850 m^2. The building will be constructed as a single-storey modular wood-frame building. This building will be heated using an air handler system that will use waste heat produced from the diesel generators.

15.14.4 Mill Dry Facility

The mill dry facility will consist of plant change rooms for the process plant area. The building will be approximately 400 m^2 in area. These facilities will have clean and dirty areas and will be complete with showers, basins, toilets, and lockers. The building will be constructed as a single-storey modular wood-frame building. This building will be heated using an air handler system that will use waste heat produced from the diesel generators.

15.14.5 Plant Workshop and Warehouse

The plant workshop will be used to perform maintenance on process equipment and equipment spares. The plant workshop will be approximately 780 m^2 in area. This building will be a pre-engineered steel frame and metal-clad building. This building will be heated using an air handler system that utilizes waste heat produced from the diesel generators.

15.14.6 Primary Crushing

The primary crushing area will feature a fabric structure approximately 70 m^2 in area that will be located above and adjacent to the crusher dump pocket to assist with dust collection measures.

15.14.7 Crushed Ore Stockpile

A geodesic dome will be used over the crushed ore stockpile to keep the ore dry during the winter. The building will be approximately 2,700 m^2 in area.

15.14.8 Process Plant

The process plant building will have an approximate area of 4,800 m^2, and will house all of the milling, flotation and concentrate thickening equipment. The building will be divided into two sections. The first section will contain the mill and will have dimensions of 30 x 24 m; the second section will contain the flotation, regrind and thickening equipment and will be 36 x 98 m in size. Both sections will be serviced by overhead cranes.

This building will be a pre-engineered steel frame and metal-clad building with internal insulation to reduce heat loss. This building will be heated using an air handler system that utilizes waste heat produced from the diesel generators.

15.14.9 Concentrate Loadout

The concentrate loadout building will house process equipment and provide a covered area for loading the concentrate on to the trucks. This building will be separated from the process plant building to minimize the building volume that requires process ventilation equipment. The building will be approximately 1,700 m^2 in area, will be a fabric structure and will not be heated to assist in drying out the concentrate prior to transport.

15.14.10 Reagent Storage and Handling

The reagent storage and handling building will be located outside the process plant building. The building will have an approximate area of 680 m^2. This building will be pre-engineered steel frame and metal-clad building. This building will be heated using an air handler system that will use waste heat produced from the diesel generators.

15.14.11 Water Supply

The water supply building will be located at the fresh water source and house the pumping equipment. The building will be 36 m^2 in area and will be a pre-engineered modular building. It will be heated using electric unit heaters.

15.15 Concentrate Transportation

Concentrate will be shipped from the Arctic mine site to the Port of Alaska in Anchorage in specialized 20ft intermodal bulk shipping containers for direct loading into bulk carrier vessels for ocean transport to the smelter or refinery. Containers will be trucked to Fairbanks, and then transferred to rail for delivery to the Anchorage port terminal.

A concentrate trucking contractor will be responsible for loading the containers in the concentrate storage building using a wheeled loader. Containers will be loaded with net 28.1 wmt of concentrate, resulting in 31.2 t gross weight per container. Based on a daily production of 1,289 wmt of concentrates (470,586 wmt per year), approximately 46 containers will be loaded per day and shipped from the Arctic site.

The base for the trucking operation will be at the junction where the road to the Arctic Mine would intersect the proposed AAP road, hereafter referred to as the Arctic Mine Junction (AMJ). This would be the primary laydown yard for concentrate containers. The trucking contractor would have a maintenance facility at AMJ, and mobile maintenance trucks along the AAP; however, most major equipment maintenance work is expected to be performed at the concentrate trucking contractor's Fairbanks operating base.

The trucking of the containers to Fairbanks is planned to be undertaken in three stages:

- Load trucking containers at the Arctic mine concentrate storage building and then transport the container to the AMJ.
- The containers would be trucked from AMJ using the AAP road to the Dalton Transfer Yard (DTY) facility. The DTY would be located near the intersection of the AAP road with the Dalton Highway and would be operated by the concentrate trucking contractor. A Super B-train configuration will be used on the AAP road, with each truck hauling two containers. At DTY, each truck will offload the two containers and return to the mine with two empty containers. This will require approximately 23 trips per day. Drivers would be based at either the Arctic site or AAP road and will complete one return trip per day from AMJ to DTY.
- The concentrate trucking contractor would use a Fairbanks-based fleet to move the containers using a single trailer configuration from the DTY to a depot in Fairbanks. When the trucks undertake the trip from Fairbanks to DTY, each truck would transport one empty container.

In Fairbanks, the containers would be loaded on railcars for transport to the Port of Alaska. At the port, the containers would be staged for direct loading into marine vessels using fixed shore cranes and a container rotator attachment. Concentrate would be shipped from Alaska in 10,000 dmt parcels for copper and zinc, and 5,000 dmt parcels for lead.

Table 15-4 provides details of the planned concentrate movement.

Table 15-4: Mode of Transport and Distances for Concentrate Shipping

Segment		Mode	Distance (km)	Trips/Day	Trips/Week
1	Arctic Site to AMJ	Truck – Single Trailer	16	46	322
2	AMJ to DTY	Truck – Double Trailer	324	23	161
3	DTY to Fairbanks	Truck – Single Trailer	391	46	322
4	Fairbanks to Port of Alaska	Rail	573	-	3
5	Port of Alaska to Asian Port	Marine Bulk Carrier	9,000	-	-

Concentrate shipping containers would be sourced from one of several suppliers and leased. It is expected that a fleet of approximately 1,770 containers would be required.

16 MARKET STUDIES

16.1 Introduction

The proposed operations will produce copper, zinc and lead concentrates from the Arctic deposit on site, which will then be transported to be sold in the Asia Pacific area. There are currently no contracts in place with any buyers for the concentrate.

16.2 Metal Prices

Metal prices are defined daily by several commodity markets via contract trading, some of which include the London Metal Exchange (LME), the Commodity Exchange (COMEX), the New York Mercantile Exchange (NYMEX), the Chicago Mercantile Exchange (CME), and the London Bullion Market Association (LBMA).

Prices are typically set each day, except weekends and holidays. Closing contract values, either based on spot pricing or based on AM/PM demand, are used to define future and expected commodity prices.

Metal price projections for use in the Report are guided by three-year trailing average commodity prices and long-term price forecasts from analysts as published by CIBC and approved by Trilogy Management as of November 30, 2022, as summarized in Table 16-1. The metal price assumptions used in the Technical Report Summary economic analysis are:

- Copper: $3.65/lb
- Zinc: $1.15/lb
- Lead: $1.00/lb
- Gold: $1,650/oz
- Silver: $21/oz

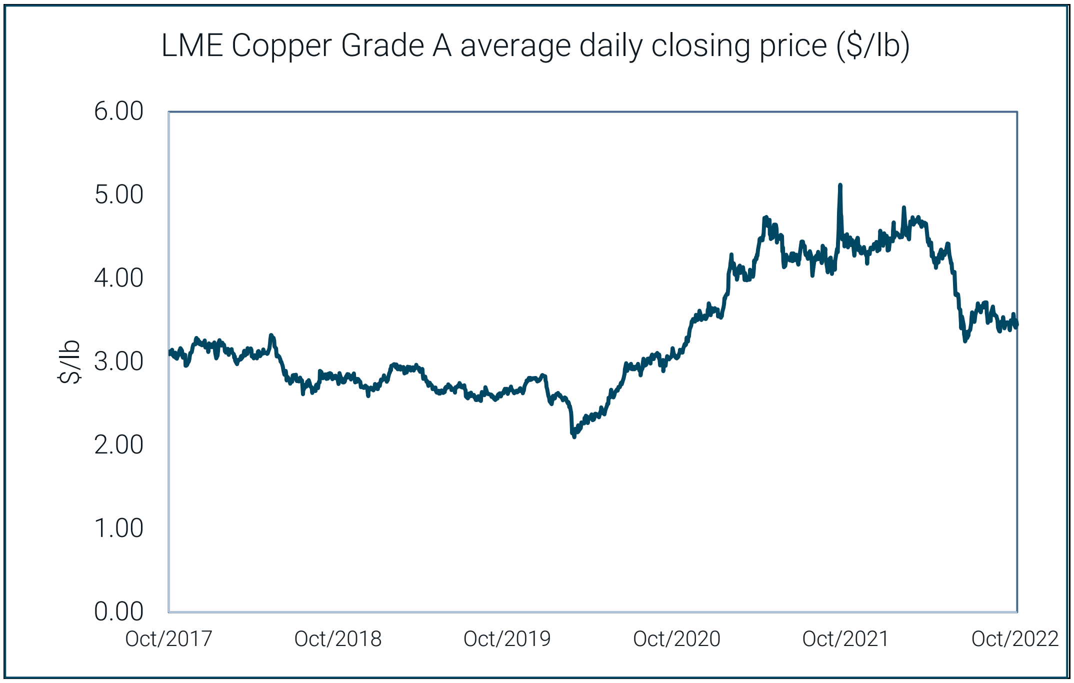
Metal price assumptions used for inputs to Mineral Resources are described in Section 11 and for Mineral Reserves in Section 12.

Table 16-1: Average Metal Prices (Data from S&P Market Intelligence and CIBC, 2022)

Average Price	Copper ($/lb)	Zinc ($/lb)	Lead ($/lb)	Gold ($/oz)	Silver ($/oz)
November 2022	3.45	1.24	0.91	1,628	18.92
1-year trailing	4.10	1.60	0.99	1,808	21.93
3-year trailing	3.62	1.31	0.93	1,775	22.19
5-year trailing	3.32	1.29	0.95	1,591	19.67
CIBC long-term consensus	3.63	1.14	0.91	1,641	21.35
Report economic analysis	**3.65**	**1.15**	**1.00**	**1,650**	**21.00**

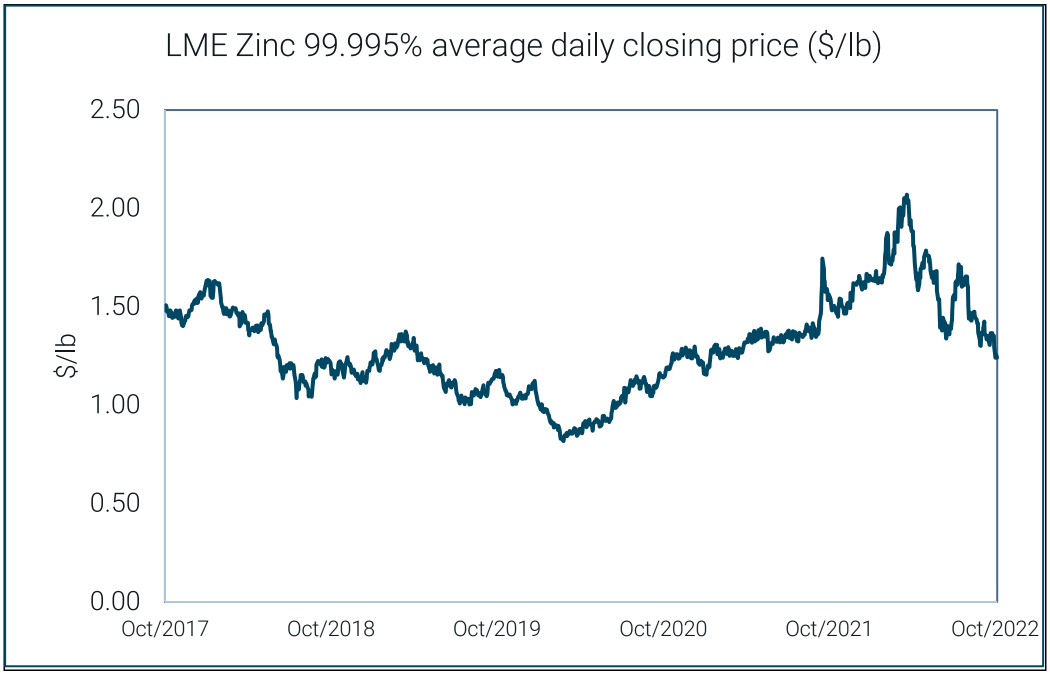
Figure 16-1 to Figure 16-5 illustrate historical pricing for the commodities above over the last 5 years, supporting the long-term metal price assumptions used in the Report.

Figure 16-1: 5-year Average Daily Closing Price for Copper



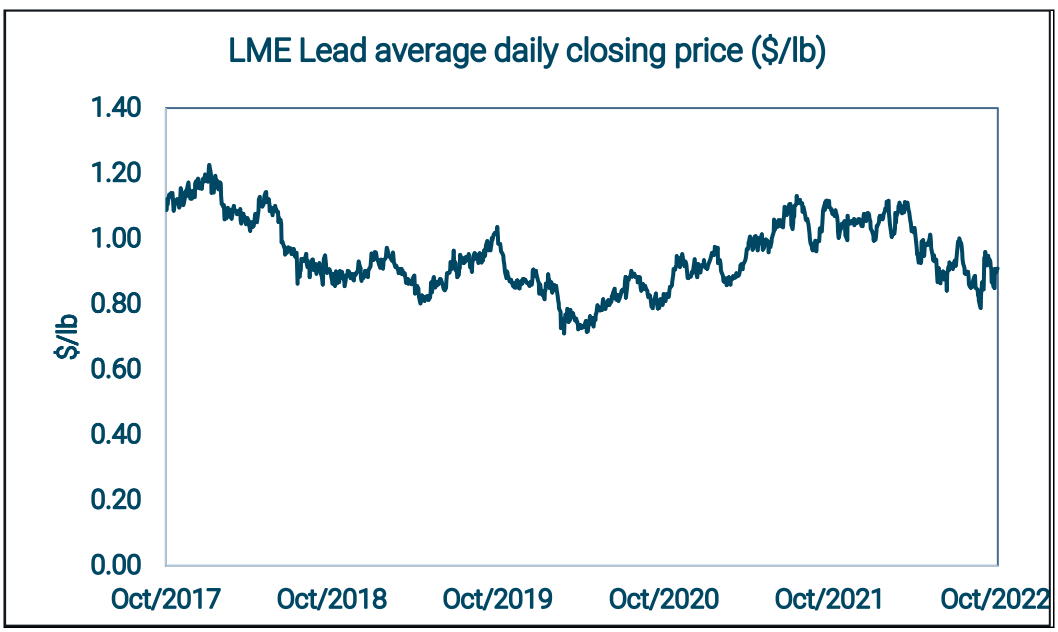
Source: Data from S&P Market Intelligence, 2022.

Figure 16-2: 5-year Average Daily Closing Price for Zinc



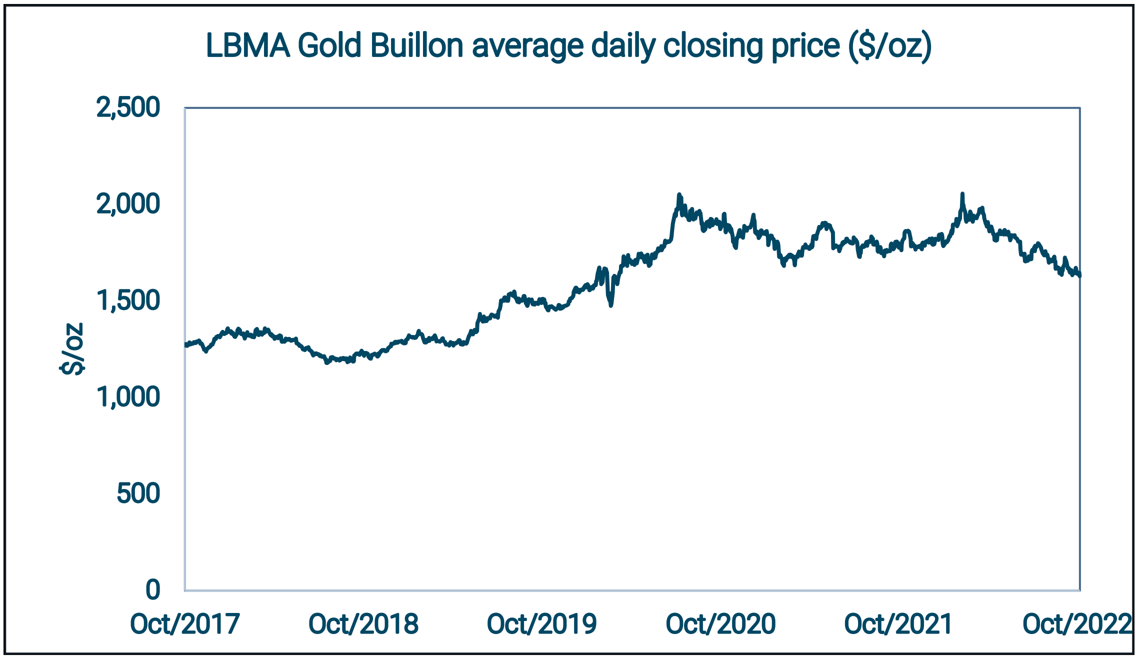
Source: Data from S&P Market Intelligence, 2022.

Figure 16-3: 5-year Average Daily Closing Price for Lead



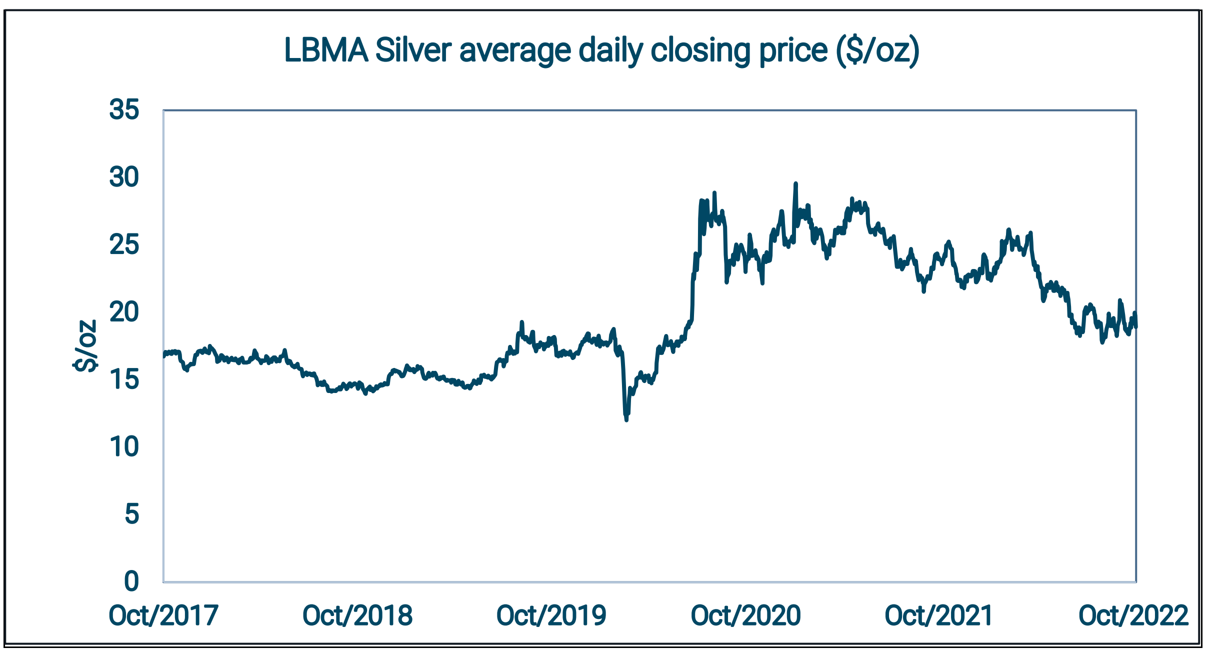
Source: Data from S&P Market Intelligence, 2022.

Figure 16-4: 5-year Average Daily Closing Price for Gold



Source: Data from S&P Market Intelligence, 2022.

Figure 16-5: 5-year Average Daily Closing Price for Silver



Source: Data from S&P Market Intelligence, 2022.

16.3 Markets and Contracts

A marketing review for the three Arctic concentrates, dated January 5, 2023, was conducted by StoneHouse Consulting based on the expected concentrate assays as provided by Trilogy.

Assumptions as to concentrate quality are based on the results of inductively coupled plasma (ICP) analysis of copper and lead concentrates produced from locked cycle tests at ALS Metallurgy and the zinc concentrate from locked cycle tests at SGS. The samples are thought to represent the expected concentrate quality. On the basis of this analysis, it has been assumed that the concentrates will be sent to Asian ports for smelting and refining.

No contracts have been entered into as of November 30, 2022 for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements. It is expected that the sale of concentrate will include a mixture of long-term and spot contracts.

- Most concentrate is traded on the basis of term contracts. Generally, a term contract is a frame agreement under which a specified tonnage of material is shipped from mine to smelter, with treatment and other charges negotiated at regular intervals (typically annually).
- Spot contracts are normally a one-off sale of a specific quantity of concentrate to a merchant or smelter. The material is paid for in much the same way as a concentrate shipped under a term contract. Merchant business is a mixture of one-off contracts with smelters and long-term contracts with both miners and smelters.

Often terms of sale for a term contract between miners and smelters are at “benchmark terms”, which is the consensus annual terms for the sale of concentrate and negotiated annually. Spot sales are made at spot terms, negotiated on a contract-by-contract basis.

16.3.1 Chinese Import Restrictions

The significance of the Chinese market for concentrate cannot be understated. China imports more than 1.5 Mt of zinc contained in concentrate per year, over 5 Mt of copper contained in concentrate, and over 700,000 t of lead contained in concentrates. China represents between 55- 75% of the traded spot concentrate business in the world. Being excluded from the Chinese market would be unfortunate for a new Project.

China is the only country that has national import quality standards for concentrates. Chinese import restrictions can dramatically affect the ability to sell concentrate in that country. For copper, zinc and lead concentrates the current import limits are indicated in Table 16-2 below.

Table 16-2: Chinese Import Restrictions for Copper, Zinc, and Lead Concentrates

Current Standard - Maximum Allowed (%)					
Concentrate	Pb	As	F	Cd	Hg
Copper	6.0	0.5	0.1	0.05	0.01
Zinc	-	0.6	-	0.3	0.06
Lead	-	0.7	-	-	0.05
Zinc/Lead Bulk	-	0.45	-	0.4	0.05

16.3.2 Copper Concentrate

Trilogy will have the option of selling some portion of the Arctic copper concentrate under long term contracts directly to smelters in China, Japan, and Korea, with the balance being sold under shorter term or spot contracts to the trade, with the same delivery points. The concentrate production at Arctic is too large to rely on the spot market for all its sales volume, although a portion sold under generally more favourable spot terms is recommended.

Arctic copper concentrate can be sold directly into China in the spot market and, as such, will be subject to better terms (lower TC) and lower penalty items. This material will not have to blended, but instead will be a quality that would be attractive for trading companies that need to blend down impurities in other concentrates.

US origin copper concentrates may be subject to Chinese import duties, which might limit the areas in where Arctic copper concentrate can be sold.

16.3.3 Zinc Concentrate

Because of the elevated level of cadmium, China will not be a market for the zinc concentrate. It is unlikely China will officially increase the Cd limit for imported concentrates, and Trilogy should not rely on exemptions at this point in the planning cycle to import Arctic zinc into China.

Even without the Chinese market Arctic zinc concentrate can still be sold to a number of smelters. Many zinc mines that have come onto the market over the past decade contain certain elements that are causing problems for smelters, principally iron, silica, manganese, selenium, and mercury. The most important aspects of Arctic zinc are the medium zinc, low silica, iron and mercury, and good copper levels. Smelters recover copper in zinc concentrates, and the 1.3% content in Arctic will be valued. The low silica, iron and mercury levels will be very attractive to many smelters in Asia and in Canada.

With respect to silver recovery, the two zinc plants in Korea, Onsan and Sukpo, several plants in Japan including Dowa's plant at Akita, Teck's plant in Trail, the Penoles plant in Torreon, Mexico and the Nyrstar plants in Europe all have good silver recovery and value silver content in zinc concentrates. Other plants have limited silver recovery, or specifically require high-grade, low-impurity zinc concentrate to limit residue production. These plants include Nyrstar facilities at Clarksville (US) and Budel (the Netherlands), and the Valleyfield smelter (Canada).

The balance of the smelters fall somewhere in between, with the desire for silver-bearing materials and the need for high-grade, low-impurity concentrates. With respect to Arctic zinc concentrate, even plants that have good silver recovery have impurity limits that could disqualify a high silver zinc concentrate, and these smelters would value some high zinc, low impurity concentrates like Arctic to blend the typically lower grade silver-bearing zinc concentrates. Smelters take a whole menu of concentrates, and they strive to create a feed blend that matches its technical capabilities. Arctic zinc concentrate would be a good fit in the blending portfolio of essentially every Western zinc plant.

However, given that most Japanese, Korean, and Canadian smelters are relatively well supplied in the current market, spot market opportunities for Arctic zinc will be limited. It is recommended that most Arctic zinc concentrate be sold under long-term contracts to Asian smelters, and perhaps to the Teck Trail smelter in Canada.

It may not be necessary for Arctic to consider shipments to Europe, although larger volume shipments to the Korea Zinc smelter in Australia might be cost effective.

16.3.4 Lead Concentrate

Over the past decade, smelters have become more sensitive to impurity levels for certain elements. For example, most lead plants have restrictions on arsenic, fluorine, selenium, chlorine, mercury, and thallium inputs. The increased vigilance on environmental issues is the primary reason for heightened scrutiny, although processing issues are also a concern with fluorine, chlorine, and selenium.

Selenium, at 0.55%, is 400% higher than the level of the next highest selenium in a lead concentrate (Peñasquito). Many smelters do not recover selenium and find selenium contamination a real problem. Some stand-alone lead smelters recover selenium, but most integrated (Zn/Pb) smelters cannot.

Korea Zinc, in particular, has tightened its internal controls on impurities in incoming feed. This is largely due to increased environmental enforcement in Korea, which has resulted in a forced temporary closure of a related zinc smelter. It is unlikely that Korea Zinc will be interested in this concentrate. Teck's Trail smelter will not purchase Peñasquito lead concentrate, which contains 0.07 - 0.15% selenium, so it will not be interested in Arctic lead concentrate. The Stolberg lead smelter in Germany, now owned by Trafigura, has a selenium limit that is much lower than the level in Arctic concentrates. These three smelters- Korea Zinc Onsan plant, Trail, and Stolberg- all have significant precious metals treatment capability but will be uninterested or unable to purchase this concentrate.

While high by market standards, fluorine, at 1,500 ppm, can generally be accounted for with a penalty. Fluorine at the high end of the specification, 3,260 ppm, is at a level that would likely require the concentrate to be blended with some low fluorine material to make a saleable concentrate. The cost of blending is borne by the more complex concentrate provider- in this case Arctic. The penalty for fluorine will be significant and will contain a higher per unit penalty above a certain threshold.

China remains the best market for this lead concentrate. China produces approximately 75% of the world's primary lead. Since the production volume of lead concentrate at Arctic will not be high, the concentrate may be a good fit at stand-alone Chinese lead smelters that can handle the selenium.

Assuming that Chinese smelters are interested in purchasing the lead concentrate for the precious metals content, there are other issues with selling the concentrate into China, depending on how it is characterized. As the concentrate does not exceed Chinese quality import limits, it could be sold directly into China as a lead concentrate. The silver content in an imported lead concentrate is charged a 13% VAT tax that is payable by the receiving smelter. The silver level in Arctic lead concentrate is high, but just below the 2,500 g/t threshold when it could be sold as a silver concentrate in China and not subject to this VAT. Ignoring the quality concerns with fluorine and selenium, smelters would be less interested in purchasing Arctic lead concentrate than some other materials with less silver content. This reluctance can generally overcome with a higher TC to compensate for the VAT disadvantage.

The challenge is to find an appropriate smelter that can value the silver and gold content but at the same time handle the fluorine and selenium impurities. As mentioned above, sales of this concentrate into China will have taxation implications based on the value of the silver content.

The copper content in the lead concentrate is good and adds to the value of the concentrate. As this concentrate will likely need to be blended due to the fluorine and selenium, the copper content, when blended, will be a value-added benefit to the smelter.

The current indication is that Arctic lead will also contain between 4 and 6% magnesium, presumably as MgO. As this is such a significant impurity the impact on a lead smelter should be evaluated. The QP's expectation is that the MgO will harmlessly slag off in the furnace, but each smelter may have different sensitivities, so the impact of the high Mg needs to be further evaluated.

Significant penalties have been assigned to fluorine, selenium and MgO, but penalty payments alone may not incentivize smelters to purchase this material.

At this time, it is felt that the best strategy is to sell the concentrate to trading companies who can blend with other lead concentrate containing low silver and selenium. In this way, the silver gets diluted, reducing the VAT impact, and the selenium gets reduced to an acceptable level.

Alternatively, the concentrate gets sold under a silver tolling licence and neither silver nor gold are subject to VAT. One issue with this strategy is that silver cannot fall under 2,500 g/t or else the concentrate will be improperly classified. For Arctic lead concentrate to be sold in this manner, silver needs to be the highest valued component in the concentrate. Depending on the prices of silver and gold, this might not always be the case.

It is recommended that a study be conducted to confirm marketability of this product. Smelters should be consulted so that Arctic can get some direction as to whether the elevated levels of fluorine and selenium can be managed.

16.4 Smelter Term Assumptions

Smelter terms were applied for the delivery of copper, zinc and lead concentrate. It was assumed that delivery of all concentrates would be to an East Asian smelter at currently available freight rates. These terms are forecasts and are considered to be in line with current market conditions. No contracts have been entered into at the Report effective date for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, and forward sales contracts or arrangements.

16.4.1 Copper

The contracts for the copper concentrate will generally include the following payment terms:

- Copper: pay 96.5% of the content, subject to a minimum deduction of 1 percentage unit, at the LME price for copper less a refining charge of $0.08 per payable pound. The minimum deduction applies to Cu grades less than 28.5% and is therefore not applicable to Arctic copper concentrate.
- Gold credit: If the gold grade is greater than 1 g/dmt and less than 5 g/dmt, then payment is 90% of the content less a refining charge of $5 per payable ounce at the LBMA gold price.
- Silver credit: If silver content is greater than 30 g/dmt, then the payment is 90% of the content less a refining charge of $0.50 per payable ounce.
- Treatment charge: $80 per dmt of concentrates
- Penalty charges:
    - $3.00 for each 1% that the zinc plus the lead grade exceeds 3%
    - $0.50 for each 100 ppm that the antimony grade is above 500 ppm
    - $2.00 for each 0.1% that the arsenic grade is above 0.1%
    - $2.00 for each 100 ppm that the selenium level exceeds 300 ppm

16.4.2 Zinc

The contracts for the zinc concentrate are assumed to generally include the following payment terms:

- Zinc: pay 85% of content, subject to minimum deduction of eight percentage units at the LME price for zinc. The minimum deduction applies to Zn grades less than 53.3% and is therefore not applicable to Arctic zinc concentrate.
- Gold credit: none.
- Silver credit: deduct 3 oz/dmt from the silver grade and pay for 70% of the balance at the LBMA price for silver.
- Treatment charge: $215/dmt of concentrate delivered; no price participation.
- Penalty charges:
  - $2 for each 0.1% that the cadmium content exceeds 0.3%

16.4.3 Lead

The contracts for the lead concentrate are assumed to include the following payment terms:

- Lead: pay 95% of the content subject to a minimum deduction of three percentage units at the LME price for lead. The minimum deduction applies to Pb grades less than 60%.
- Gold credit: pay 95% of content, subject to a minimum deduction of 1 g/dmt, less a refining charge of $20/payable oz at the LBMA price for gold.
- Silver credit: pay 95% of content, subject to a minimum deduction of 50 g/t less a refining charge of $1.25/payable oz. The minimum deduction applies to Ag grades less than 1,000 g/dmt and therefore does not apply to Arctic lead concentrate.
- Treatment charge: $160/dmt of concentrate delivered.
- Penalty charges:
  - $5.00 for each 1,000 ppm that the bismuth content exceeds 1,000 ppm
  - $4.00 for each 100 ppm that the selenium content exceeds 500 ppm
  - $2.00 for each 100 ppm that the fluorine content exceeds 500 ppm, plus $5.00 for each 100 ppm above 1,000 ppm, plus $10 for each 100 ppm above 2,000 ppm
  - $2.00 for each 0.1% that the MgO content exceeds 2%

16.5 Transportation and Logistics

Transportation cost assumptions for the concentrate are summarized in Table 16-3.

Table 16-3: Concentrate Transport Costs

Description	$/dmt
Container cost	7.80
Arctic Mine to Fairbanks (truck)	215.89
Fairbanks to Port of Anchorage (rail)	33.92
Port Terminal & Handling	24.21
Ocean Freight to Asian Port	42.55
Total Concentrate Transport Costs	**324.37**

16.6 Insurance

An assumed insurance rate of 0.15% was applied to the recovered value of the concentrates less refining, smelting, penalties, and treatment charges.

16.7 Representation and Marketing

An allowance of $2.50/wmt of concentrate was applied as an allowance for marketing and representation.

16.8 Comments on Market Studies and Contracts

The QP has reviewed the information on marketing, payable and penalty assumptions and metal price assumptions, and considers that they are acceptable for use to support the Mineral Reserve estimates in Section 12 and the economic analysis in Section 19.

17 ENVIRONMENTAL STUDIES, PERMITTING, AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

17.1 Environmental Studies

The Arctic Project area includes the Ambler Lowlands and Subarctic Creek within the Shungnak River drainage. A significant amount of baseline environmental data collection has occurred in the area including surface and ground water quality sampling, surface hydrology monitoring, hydrogeology studies, meteorological monitoring, wetlands mapping, aquatic life surveys, avian and mammal habitat surveys, cultural resource surveys, and metal leaching and acid rock drainage (ML/ARD) studies.

17.1.1 Hydrology

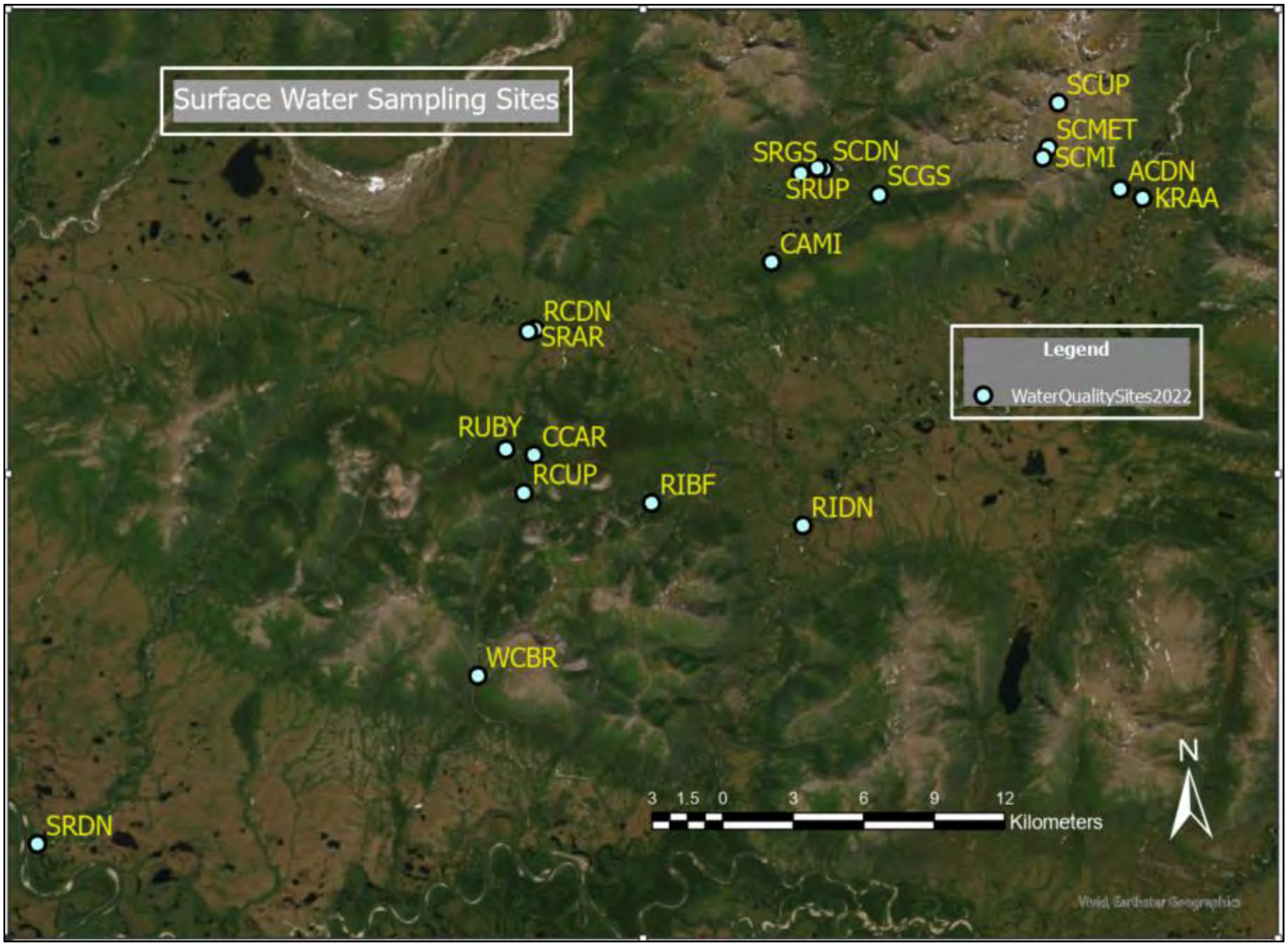
The Arctic Project hydrology studies are ongoing. Shaw Environmental collected water quality samples in 2007, 2008, and 2009 and measured stream flow at 13 stations on the Shungnak River, Subarctic Creek, Arctic Creek, and the Kogoluktuk River as seen in Figure 17 1 (Shaw, 2007, 2008, 2009).

Tetra Tech performed baseline studies in July 2010 to characterize stream flow and water quality that could be potentially impacted by the construction and operation of a proposed access road between the Bornite and Arctic airstrips, and the existing road between the Arctic airstrip and the Arctic deposit. Tetra Tech collected water quality and flow data at 14 sites (Tetra Tech, 2010a).

Two hydrologic gauging stations were installed on the Shungnak River (SRGS site) and Sub-Arctic Creek (SCGS site) respectively by DOWL HKM in July 2012. Each station is powered by dual solar panels and a battery, and continually measures and records water depth, temperature, pH, and conductivity. A third hydrologic gauging station was established at the Lower Ruby Creek Gauging Site (RCDN) by WHPacific in June 2013. The RCDN station was moved upstream in 2017 due to a beaver dam. An additional hydrologic gauge (SCMET) was installed near the proposed waste rock collection pond in July 2021. All four of these stations have remained in operation since their installation. Dave Brailey, a contracted hydrologist, has used the depth data to generate rating curves and daily discharge calculations for the SRGS, SCGS, and RCDN sites.

Ambler Metals staff have measured instantaneous stream flow and other standard field parameters (YSI 556 multi-parameter unit) during seasonal sampling events from 2013 to 2019, 2021, and 2022 to present in Sub-Arctic Creek, Ruby Creek, the Shungnak River, and select tributaries to these drainages. The baseline water quality and hydrology program were expanded in 2016 from seven to over 20 sampling sites, with additional locations in Cabin Creek, Riley Creek, Wesley Creek, and the Kogoluktuk River (Craig, 2016). Additional locations on Camp Creek and Sub-Arctic Creek were added in 2019.

Figure 17-1: Current Water Quality and Hydrology Stations' Location Map



Source: Craig, 2022.

A snow survey was conducted in March 2018 around the site area in the upper reaches of the Sub-Arctic Creek watershed. The survey consisted of 52 locations using a Federal Snow Sampling tube. The purpose of the snow survey was to determine the snow water equivalent of various representative locations in the Subarctic Valley (Craig, 2018).

Additional snow surveys were conducted in March 2021 and April 2022 using similar methodology (Stockert, 2021 & 2022).

SRK (2020) reviewed existing baseline hydrological data for the Arctic Project site and prepared a regional hydrology analysis. SRK conducted an update in 2022 to include the most recent precipitation and hydrologic data (SRK, 2022).

Project and publicly available data were used to estimate mean annual precipitation (MAP) and mean annual runoff (MAR) at the Arctic Project and extend the available record period. The 2020 analysis predicted a MAP of 1,294 mm and a MAR of 1,089 mm for the Sub-Arctic Valley. The increase in estimated MAP from the SRK 2018 investigation is mainly due to incorporating a precipitation undercatch correction developed based on the snow survey data collected in 2018. The 2022 update predicted a MAP of 1,219 mm and a MAR of 1,130 mm for the Sub-Arctic Valley. The Arctic Project's evaporation estimates were also updated in 2022 using climatic records and an empirical relationship. Evaporation estimates are stable and have not changed since SRK's 2018 climate and hydrology review (SRK, 2018).

Three precipitation gauges were installed in July 2021 by Boreal Services in the Sub-Arctic Valley. The precipitation gauges functioned properly through the winter of 2021/2022 and measured the precipitation from rain and snowfall. In addition, a snow monitoring gauge was installed to measure the weight of the snow at the Sub-Arctic Valley Meteorological Station. The additional sensors will help to improve the MAP and MAR estimates of the Sub-Arctic Valley drainage.

An evaporation pan was installed in July 2021 to measure the actual evaporation during the summer months. Winter evaporation is assumed to be negligible.

Unit measured flows at the nearby USGS Dahl Creek and the Kobuk River at Kiana stations were compared to unit measured flows at the SRGS and SCGS hydrologic gauging stations for a concurrent monitoring period Flows measured in Dahl Creek closely resemble unit flows at the Arctic Project (Brailey 2019 & SRK 2022).

17.1.1.1 Recommendations

The Arctic Project should add additional transducers to the Geonor precipitation gauges to guard against transducer failure or drift and to provide more certainty regarding the data quality.

Frequent checks and equipment calibration are necessary to maintain quality data collection and ensure correct instrumentation functioning. Annual reporting of instrumentation checks and data quality reviews should be completed.

17.1.2 Water Management

Water quality monitoring was conducted in the area over the past eleven years. The baseline monitoring data was supplemented with publicly available regional data to evaluate long-term trends.

In July 2010, Tetra Tech performed baseline studies to characterize flow and water quality in several streams that could be potentially impacted by construction and operation of a proposed access road between the Bornite airstrip and the Arctic airstrip, and the existing road between the Arctic airstrip and the Arctic deposit. Tetra Tech collected water quality and flow data at 14 sites. The results of the Tetra Tech sampling program indicate that, in general, the water quality for all sites meets applicable Alaska State water quality standards (WQS) for the parameters analyzed. Water quality sampling was conducted by Trilogy from 2012 to the present. Small sampling programs were performed from 2012-2015 during the summer field season. The water quality sampling program was expanded in 2016 to include more sample locations in the Shungnak River, Subarctic Creek, Ruby Creek, Riley Creek, Wesley Creek, and the Kogoluktuk River and to include sampling throughout the year. Several seeps in the Subarctic Creek drainage near the Arctic Project were sampled.

Samples were analyzed for both total and dissolved metals, mercury, cyanide, chloride, fluoride, nitrates, sulphate, acidity, alkalinity, total suspended solids, conductivity, pH, total dissolved solids, total organic carbon, and total phosphorus. The information will be used in the permitting and facilities design.

17.1.3 Wetlands Data

Tetra Tech performed a program of jurisdictional wetlands identification in a portion of the Arctic Project area in 2010, as part of a study to identify potential road alignment alternatives between the Bornite and Arctic airstrips. The work included data review, vegetation mapping, aerial photographic interpretation (segmentation), and field soil surveys. The work is summarized as follows.

The area between the Bornite and Arctic airstrips consists of a wide valley containing the Ambler lowlands and the Shungnak River. Wetlands are prevalent throughout much of the Ambler lowlands. Most of the wetlands within the area occur within tundra vegetation communities composed primarily of ericaceous shrubs, such as Alpine blueberry (Vaccinium uliginosum) and Northern-Mountain Cranberry (V. vitis-ideae) and graminoids, such as Tussock Cotton-Grass (Eriophorum vaginatum) and sedges such as Two-Color Sedge (Carex bigelowii) and Leafy Tussock Sedge (C. aquatilis). Forests of White Spruce (Picea glauca) and Black Spruce (P. mariana) and shrub birch communities of Swamp Birch (Betula nana) and Resin Birch (B. glandulosa) make up most of the upland communities.

In 2015, Trilogy engaged DOWL to perform additional wetlands mapping and generate two preliminary wetlands determinations for a 5,910-acre study area (DOWL, 2016). The study area included the entire Subarctic Creek drainage and the majority of the areas that could be directly impacted by the proposed Arctic open pit and mine facilities. The broad study area comprises 715 acres of potentially jurisdictional wetlands, 40 acres of Waters of the United States and 5,155 acres of non-jurisdictional uplands. According to DOWL (2016), the field work was performed in accordance with Part IV of the Corps of Engineers 1987 Wetlands Delineation Manual and the Regional Supplement to the Corps of Engineers Wetland Delineation Manual: Alaska Region (Version 2.0, 2007). Wetlands were classified and grouped according to the Class Level and system guidelines outlined in Classification of Wetlands and Deepwater Habitats of the United States (1979). The functional rating of potentially jurisdictional areas was determined using the criteria outlined in the 2009 Alaska Regulatory Guidance Letter, ID No. 09-10, the Cowardin Class, and observed hydrology. Ten ecological attributes were examined to subsequently rank wetland habitats as having low, moderate, or high functional ecological services. Riverine habitats (rivers and streams) perform vastly different functions compared to wetlands. Accordingly, riverine systems were evaluated based on the presence or absence of 17 functions according to the criteria outlined in the U.S. Department of the Interior Bureau of Land Management, Technical Report 1737-15, Riparian Area Management: A User Guide to Assessing Proper Functioning Condition and the Supporting Science for Lotic Areas.

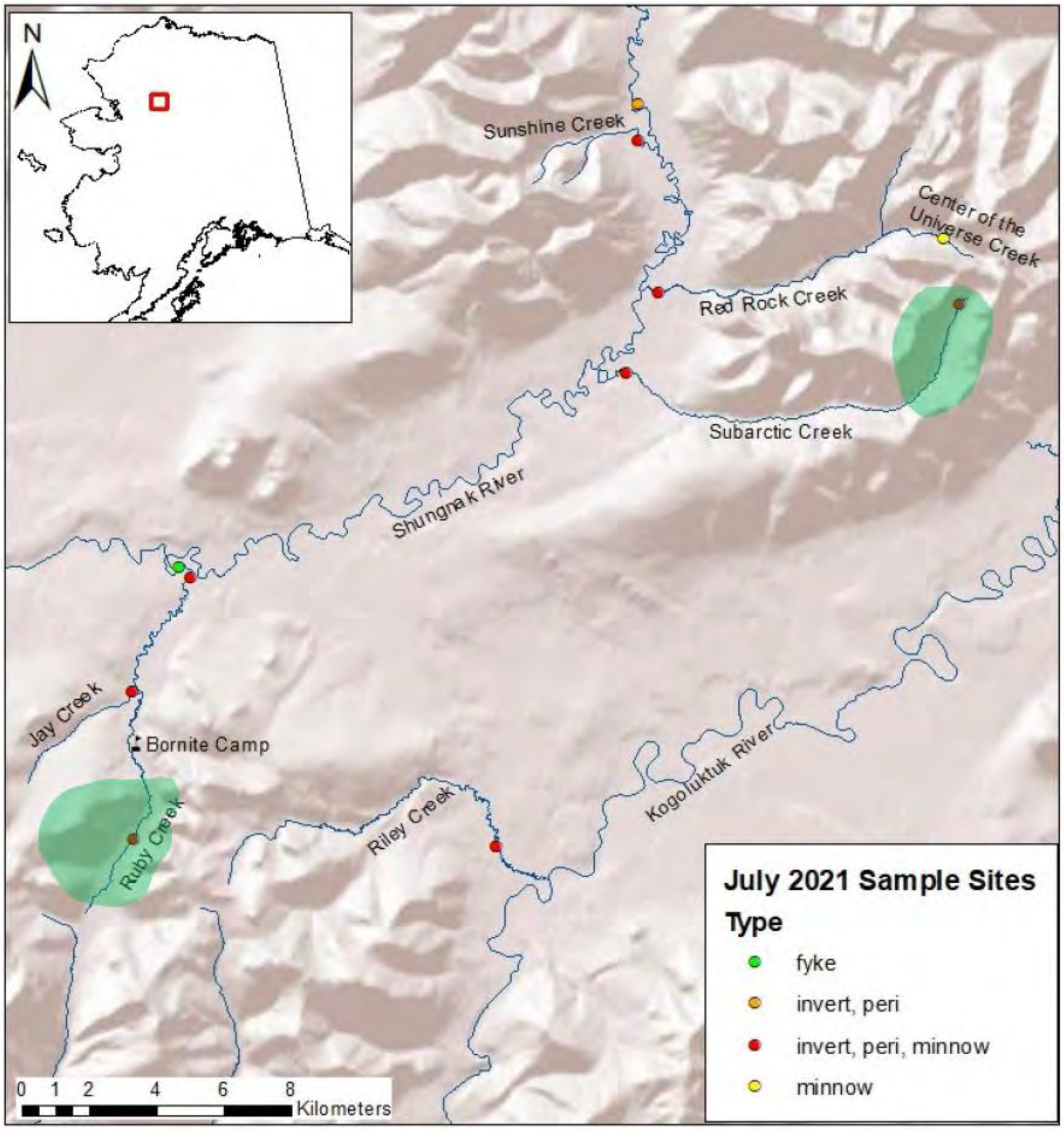
Additional wetlands delineation work was done by DOWL in 2016, 2018 and 2019 to provide wetlands delineation of the entire proposed Arctic Project footprint including access roads, camps, stockpiles, mining, and waste storage facilities.

17.1.4 Aquatic Life Data

A biological consultant (Tetra Tech) performed aquatic life studies in 2010 in the area between the Bornite and Arctic airstrips, and along the Arctic deposit road in Subarctic Creek. The purpose of this study was to characterize the aquatic life within the Shungnak River and select tributaries. Opportunistic observations were also collected in the Kogoluktuk River. Fish species documented included Arctic grayling (Thymallus arcticus), round whitefish (Prosopium cylindraceum), slimy sculpin (Cottus cognatus), and Dolly Varden (Salvenlinus malma). Fish and macroinvertebrate data were collected from July 8 to 14, 2010.

In 2016 Trilogy engaged Alaska Department of Fish and Game (ADF&G) to complete aquatic studies as an extension of the work done by consultants in prior years. ADF&G has performed aquatic biomonitoring surveys at several large hardrock mines in Alaska, and has performed aquatic surveys onsite from 2016-Present except in 2020 due to the COVID-19 pandemic. Each year, ADF&G sampled for periphyton, aquatic invertebrates, and fish. Fish were captured using minnow traps and fyke nets. A subset of fish were retained for whole body element analysis. Sample sites have been added and refined over the years, and in 2022 ten sites in the Shungnak River drainage were sampled, including two in Ruby Creek and two in Subarctic Creek. An additional site on Riley Creek in the Kogoluktuk River drainage was also sampled for a total of 11 sites (Figure 17 2).

Figure 17-2: 2021 Aquatics Sampling Sites



Source: ADF&G, 2021

Some of ADF&G's results show that Subarctic Creek, which drains the Arctic deposit area contained relatively low background concentrations of zinc and copper, as well as low total dissolved solids (TDS). Upper Subarctic Creek had the highest number of aquatic invertebrates but also had the lowest species richness with a total of 11 taxa identified.

Lower Subarctic Creek had a significantly lower average number of aquatic invertebrates but more diversity than upper Subarctic Creek.

Periphyton, or attached micro-algae, are sensitive to changes in water quality and are often used as evidence of in-situ productivity. The concentrations of chlorophyll-a are determined to estimate periphyton standing crop. Typically, Upper Ruby Creek has the highest concentrations of chlorophyll-a.

The aquatic invertebrate samples are sorted and identified to the lowest taxonomic level. Because invertebrates belonging to the orders Ephemeroptera (mayflies), Plecoptera (stoneflies), and Tricoptera (caddisflies) (EPT) are more sensitive to water quality, the proportion of EPT at each site was calculated and compared to other groups of aquatic invertebrates. Macroinvertebrate density for each sample was also calculated. In 2021, Upper Ruby Creek and Upper Subarctic Creek had the highest aquatic invertebrate densities. Lower Subarctic Creek and Lower Red Rock Creek had the highest proportion of EPT.

Despite being isolated from the Kobuk River by a large waterfall preventing migrations of anadromous fish, the Shungnak River drainage supports self-sustaining populations of Arctic grayling, Dolly Varden, round whitefish, longnose suckers, Alaska blackfish, and slimy sculpin. Minnow trap catches in Subarctic, Red Rock, and Center of the Universe creeks are dominated by Dolly Varden. Slimy sculpin is the majority of the catch in minnow traps in Ruby Creek (Clawson 2022a). The fyke nets at the mouth of Ruby Creek primarily capture Arctic grayling and round whitefish, although in 2017 large numbers of longnose suckers were also captured.

In 2021 and 2022, winter sampling for fish presence was conducted in March/April. Dolly Varden was captured at Upper Subarctic Creek, Center of the Universe Creek, and middle Subarctic Creek, demonstrating that at least some fish overwinter in these drainages instead of dropping down to the Shungnak River. The groundwater inputs into these streams are a steady temperature all winter and prevent the streams from freezing solid, enabling fish to survive over winter.

Over the six years of baseline data collection, different fish species have been sampled at different locations, therefore comparisons between sample results for fish whole body element concentrations must be evaluated accordingly. Generally, cadmium concentrations are highest in fish from Subarctic and Red Rock creeks and lowest in Ruby Creek fish. Whole body copper, selenium, and zinc concentrations are similar at all sample sites. Mercury concentrations are lowest in fish from the Subarctic and Red Rock creek drainages and highest in fish from Ruby Creek. Element concentrations fall within the range documented in fish from other drainages throughout the.

17.1.5 Hydrogeology Data

In total, from all programs over the last 10+ years, hydrogeological data is available from:

- monitoring wells including two pumping wells in the valley bottom area; 12 boreholes with single or multiple vibrating wire piezometers (i.e., 25 VWP sensors in total) and seven standpipe piezometers in proposed pit area;
- 39 boreholes; 15 in the area of the proposed open pit and 24 in the valley bottom area;
- Hydrogeologic testing data from 74 packer-based hydraulic tests; 19 slug tests; two pumping tests in valley bottom, each with an observation well; three extended injection tests in planned open pit area, one of six hours duration and two greater than 24 hours, each with monitoring at nearby vibrating wire piezometers (VWPs); 122 particle size distributions from test pits in valley floor;
- Groundwater level series data from 24 VWPs and two standpipe piezometers in pit area; 20 water level dataloggers in valley bottom, and manual observations from the monitoring wells;

- Dedicated ground temperature cables at six locations in valley floor; temperature from all VWP sensors, including those in the planned open pit area;
- Groundwater quality data from eleven monitoring wells, collected quarterly to once per year, depending on access and location. Most wells are within the Subarctic Creek valley bottom, upgradient, within and down gradient of the mine footprint.

Data were compiled into conceptual models for the planned valley bottom WRF and water management areas, as well as for the open pit area. Hydraulic head data indicate that groundwater flow is topographically driven, with recharge in the uplands and discharge in the valley bottom. At the large scale, hydrostratigraphic units include overburden, weathered bedrock and competent bedrock.

In the valley bottom, overburden, where coarse grained, and weathered bedrock are the primary aquifer units. Flow within competent bedrock is restricted to open fractures and generally has a lower hydraulic conductivity. Flow is in a down-valley direction, with groundwater divides generally aligned with the surrounding ridges.

In the planned pit area, the hydrostratigraphy is dominated by competent bedrock, with sub-units including upper and lower fractured rock, talc and geological structures. Talc may represent an aquitard located between the upper and lower fractured rock units. Water levels from the fractured rock units show a downwards gradient across the talc; the talc acts as a confining unit with the potentiometric surface for the lower fractured rock typically above the talc. Geological structures are present and can act as conduits or barriers to flow. There were no structures showing consistent barrier or conduit characteristics over the scale of the proposed pit. Compartmentalization is a possibility.

In the valley bottom, WRF and water management structures are designed to reduce release of contact water towards the down gradient receiving environment. Groundwater bypass of these systems could occur, though would be expected to be of low quantity as it would likely occur through the competent bedrock hydrostratigraphic unit. Groundwater moving down the valley can be assumed to largely discharge to Subarctic Creek before the creek enters the Shungnak River valley.

17.1.6 Cultural Resources Data

In 2016, Trilogy contracted WHPacific to perform a cultural resource assessment of the project area. WHPacific's assessment included a literature review, archival research, stakeholder meetings in the communities of Shungnak and Kobuk, and an archaeological field survey of the Arctic Project. WHPacific's literature and archival review included reviewing confidential Alaska Heritage Resource Survey Site information, which revealed that no archaeologic sites have been recorded in the project area (WHPacific, 2016). Stakeholder meetings with local community members indicated that the areas near the Arctic Project were not used by local residents in the past due to the lack of resources and passages to areas north, west, and east, where other resources and trading opportunities existed.

WHPacific's field survey was conducted by two archaeologists and included an initial aerial reconnaissance of all 2,327 acres of the survey area, executed through the use of helicopter overflights at low elevation for observation by the archaeologists. As a result of this flyover, WHPacific determined that the majority of the Project area had a low probability of containing cultural resources due to saturated and marshy terrain and dense vegetation. Areas of low to moderate probability included dryer areas of elevated terrain, lower valley slopes, ridges, flat areas overlooking valleys, and terraces along the waterways. Of the total Arctic Project acreage, WHPacific conducted pedestrian survey of 530 acres and excavated a total of 13 shovel probes on NANA-managed lands. No cultural resources were found in the survey area. WHPacific did note that late 20th century mining exploration is in evidence in the project area as seen by roads, abandoned equipment, and the airfield (outside of the direct survey area). These are of an age that is on the cusp of being considered historical period resources (50 years or older) according to the National Historic Preservation Act. However, these resources are not unique within northern Alaska or for 20th century mining materials.

WHPacific's (2016) recommendation based on their literature review, community interviews, and fieldwork was that no further cultural resources work was required, and that Project work associated with the proposed Arctic pit, facilities, tailings, and access road corridor project should proceed as planned.

Subsequent to WHPacific's investigation, Trilogy has made several minor additions to the Arctic Project, such as road corridors and material sites. To provide complete archaeological coverage for all areas of planned development, Trilogy contracted additional cultural resource fieldwork in 2018 by Kuna Engineering (2018) and in 2019 Walking Dog Archaeology (2020). These efforts included literature reviews and field investigations of the remaining areas potentially impacted by the project footprint including camps, access roads and material sites. Neither of these two investigations resulted in the identification of previously undocumented cultural resources.

17.1.7 Subsistence Data

Access to the Arctic Project area includes travel over private lands owned by NANA. Subsistence is important to local residents, and as a result, a Subsistence Committee comprised of locally appointed residents from five potentially-affected communities in the region has been formed to review and discuss subsistence issues related to activities within the UKMP, including the Arctic Project and to develop future compliance plans. Representatives from NANA and Ambler Metals facilitate the meetings and report a summary of the discussions and recommendations provided by the Subsistence Committee to the Oversight Committee, as defined by the NANA Agreement. The Subsistence Committee meets twice annually and discusses development plans and potential subsistence issues.

In 2012, Stephen R. Braund & Associates completed a subsistence data-gap memo under contract to the Alaska Department of Transportation and Public Facilities as part of the baseline studies associated with the proposed road to the Ambler Mining District. The purpose of this analysis was to identify what subsistence research had been conducted for the potentially affected communities, determine if subsistence uses and use areas overlap with or may be affected by the access road project, and identify what, if any, additional information (i.e., data gaps) needed to be collected to accurately assess potential effects to subsistence. Among other topics, the report outlined historic subsistence uses including maps and a literature review, and provided a synopsis, by village, including those villages closest to the Arctic Project, and suggested further study.

An ADF&G report titled, "Wild Food Harvests in 3 Upper Kobuk River Communities: Ambler, Shungnak, and Kobuk 2012-2013" (ADF&G 2015) provides a comprehensive analysis of subsistence food sources and their usage by the three Upper Kobuk villages. The report detailed the ethnographic history, contemporary usage, common species harvest methods, and abundance.

Previous sampling efforts established the presence of various salmon species, northern pike and sheefish in the lower Kogoluktuk River. Sampling efforts in the Shungnak River have established the presence of northern pike. The presences of fish are good indicators of the possibility of subsistence use of these rivers, but boat access is limited due to waterfalls and rapids. In comparison, the Kobuk River, a wide and easily navigable river on which the communities of the region exist, supports the bulk of subsistence fishing.

Determining the presence and distribution of caribou is complex because of seasonal and annual variability in migration patterns. The ADF&G and National Parks Service employ a radio collar monitoring program as well as aerial photography to estimate herd size and migration patterns. The ADF&G also estimates mortality rates for cows, calves, and bulls, as well as other biomonitors for health such as body fat and predator populations.

The Northwest Arctic Borough (NWAB), through its Title 9 Conditional Use permit, regulates the Arctic Project with respect to caribou interactions to assure the migration is minimally affected by mining and exploration activities. To this end, Ambler Metals has communicated with the ADF&G wildlife biologists, who monitor caribou herd movements in the spring and fall in proximity to the Arctic Project by using radio-collared caribou. Summary maps of those movements constructed from years of radio collar information indicate three main migration corridors to the west of the Arctic Project area for the Western Arctic caribou herd. The nearest herd is approximately 48 km west of the project area.

DOWL (2016) performed a large mammal habitat survey in the project area. Historic maps of caribou migration included in their report show that the project area is outside of main caribou corridor routes and calving areas, but that data from 1988-2007 suggested that the area may be used for wintering habitat.

17.1.8 Endangered Species, Migratory Birds, and Bald and Golden Eagle protection

In 2016, Trilogy engaged WHPacific, through subcontractor ABR, Inc., to perform aerial surveys of nesting raptors in the project area, including the Bornite area located some 24 km southwest of the project area. ABR (2017) identified a total of 26 nests, of which 18 were in the Bornite area and eight were in the project area. Fifteen of the totals were occupied in the initial occupancy survey; nine were occupied by rough-legged hawks, with three peregrine falcon nests and three raven nests. In the later productivity survey ABR observed that only one rough-legged hawk nest had a (single) nestling, one peregrine falcon nest had two young and an unhatched egg, and two raven nests had young (not counted).

In 2017, Trilogy engaged WHPacific to review requirements that would be necessary to comply with the Endangered Species Act, the Migratory Bird Treaty Act and the Bald and Golden Eagle Protection Act. WHPacific (2017) concluded that there are no endangered species or critical habitat in the project area. Further they reported that nine avian species of conservation concern are expected to occur or could potentially be affected by activities in the project area. They provided the timing guidelines for vegetation clearing that are meant to protect these species during nesting activities and advised that if impacts to migratory species are unavoidable for the Arctic Project that the US Fish and Wildlife Service (USFW) must be consulted. They also recommended that Trilogy request a project review from USFW when the Arctic Project is closer to initiation.

17.1.9 Metal Leaching / Acid Base Accounting Data

Drill core samples were used to characterize the acid generation potential of the mine waste for the Arctic Project. In 1998, Robertson collected 60 representative core samples from the deposit for their ABA characteristics; these samples provided a broad assessment of ARD potential at the Arctic deposit with a focus on characterization for surface development. In 2010, SRK collected 148 samples and prepared a preliminary analysis of the ML/ARD potential of waste rock at the Arctic deposit (SRK 2011). The SRK report focused on characterization for underground development rather than an open pit scenario; however, it did provide a more refined analysis of ARD potential based on advances that have been made in understanding the importance of sulphide mineralogy in assessing ARD potential. The criteria used for classifying ARD potential also differed slightly from the Robertson era work.

In 2015, Trilogy retained SRK to provide on-going ML/ARD characterization services for the Arctic Project. Activities in 2015 through 2018 focused on three objectives: 1) set up and on-going monitoring of on-site barrel tests (kinetics), 2) initiation and on-going monitoring of parallel laboratory humidity cell tests (kinetics), and 3) expansion of the ABA database (statics). Barrel test leachate samples were routinely collected during 2016 through 2018 and analyzed by ARS Aleut Analytical. Humidity cell tests were monitored on a weekly basis by Maxxam Analytics of Burnaby, British Columbia. Trilogy and SRK selected 1,119 samples to be analyzed for a conventional static ABA package with a multi element scan using the same methods as the exploration database. Samples were analyzed by Global ARD Testing Services of Burnaby, British Columbia.

In 2017, tailings humidity cell testing was initiated to investigate the ML potential of unsaturated tailings. Monitoring was as described above for the waste rock humidity cell tests.

Activities in 2019 focused on expanding the kinetic testing program to characterize the metal leaching potential of the range of geochemical compositions present in each of the main waste rock units, in addition to testing ore. New lab kinetic tests were initiated, and monitoring of the lab kinetic tests initiated in 2015 also continued, with all tests operating at Maxxam Analytics of Burnaby, British Columbia. Barrel test leachates were sampled routinely in 2019 and analyzed by Maxxam Analytics, to maintain consistency in detection limits between laboratory and field kinetic tests.

Additional kinetic tests were also started on tailings in 2019, including a further humidity cell, and a subaqueous column to investigate ML in subaqueous conditions. Tests were monitored on a weekly basis by Bureau Veritas Labs (BV; previously Maxxam Analytics) of Burnaby, British Columbia.

Static testing of overburden also occurred in 2019, including static leach tests, to investigate ARD/ML potential of freshly excavated overburden. Samples were collected from previous test pits and drill core used for geotechnical investigations.

In 2020, additional humidity cell tests were initiated on AMR at BV to improve the representivity of kinetic tests on this unit and to investigate the suitability of near-surface AMR for construction use. Monitoring of previous humidity cells and the tailings subaqueous column continued.

In 2021, monitoring of previous kinetic tests at BV labs continued except for three tests which were completed. Barrel test leachates were also sampled routinely in 2021 and analyzed at BV labs.

In 2022, additional humidity cell tests were initiated on ore and tailings and a subaqueous column was initiated on tailings, all at BV labs. Monitoring of previous kinetic tests at BV labs continued with six tests reaching completion in December 2022. Barrel test leachates were sampled routinely in 2022 and analyzed at BV labs.

17.2 Operational Site Monitoring

Requirements and plans for waste and tailings disposal during operations can be found in Sections 15.8.5 and 15.10.1. Information regarding water management during operations can be found in Section 15.8.

17.3 Mine Reclamation and Closure

17.3.1 Reclamation and Closure Plan

Mine reclamation and closure considerations are largely driven by State regulations (11 AAC 86.150, 11 AAC 97.100-910, and 18 AAC 70) and statutes (AS 27.19) that specify that a mine must be reclaimed concurrent with mining operations to the greatest extent possible and then closed in a way that leaves the site stable in terms of erosion and avoids degradation of water quality from acid rock drainage or metal leaching on the site. A detailed reclamation plan will be submitted to the State agencies for review and approval in the future, during the formal mine permitting process. For this report a preliminary reclamation and closure plan was developed by SRK. This is considered adequate for this stage of study as closure is delayed by the operations period and numerous studies will be developed during operations that will further inform the final reclamation and closure plan.

The approval process for the plan varies somewhat depending on the land status for any particular mine. Owing to the fact that the Arctic Project is likely to have facilities on a combination of private (patented mining claims and native land) and State land, it is likely that the reclamation plan will be submitted and approved as part of the plan of operations, which is approved by the ADNR. However, since the reclamation plan must meet regulations of both ADNR and the ADEC, both agencies will review and approve the Reclamation Plan. In addition, private landowners must formally concur with the portion of the reclamation plan for their lands so that it is compatible with their intended post-mining land use.

17.3.2 Closure Objectives and Closure Criteria

The overall closure objective is to establish stable chemical and physical conditions that protect the environment and human health. To the extent practicable, rehabilitation efforts will endeavour to return the site to a condition which generally conforms with the surrounding terrain. The site will be monitored and maintained post-closure in order to demonstrably meet these conditions.

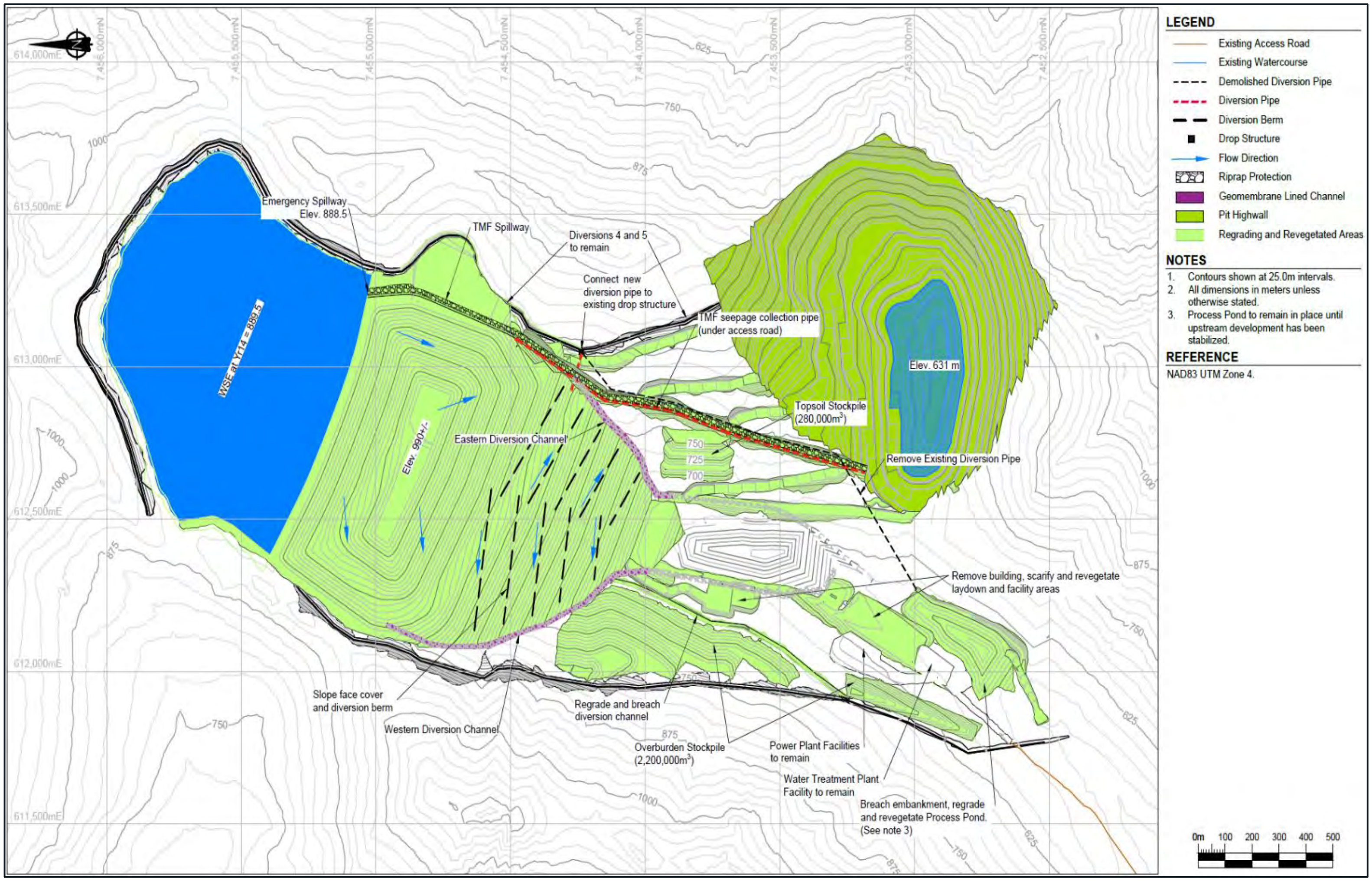
The following general closure objectives were considered:

- Demolish and remove all construction, camp and industrial facilities and reclamation of affected footprints.
- Achieve long-term slope stability of the pit, WRF, and TMF.
- Meet water quality criteria for all mine water and seeps prior to discharge to the environment.
- Prevent intrusion and migration of tailings porewater and water from the pit into the regional groundwater.
- Prevent and limit to the greatest extent practical, contact of humans and wildlife with the mine waste (waste rock and tailings).
- Establish adequate vegetation density to ensure erosion protection of the soil slopes.
- Re-establish vegetation or stabilized surface on areas returned to normal land use.

17.3.3 Closure Activities

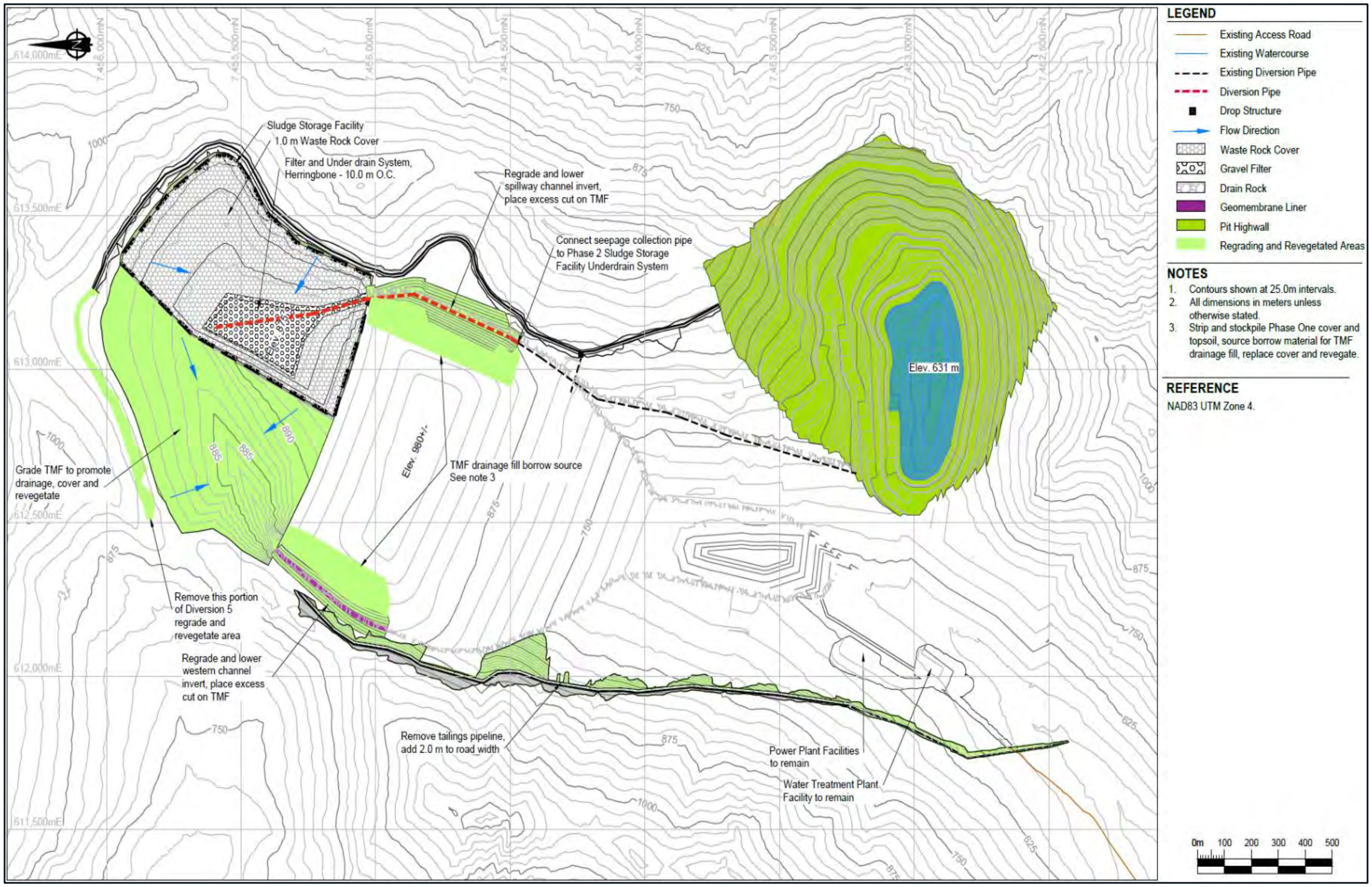
Closure activities will be undertaken at the end of mine life to bring the mine facilities in a state consistent with the stated closure objectives and compliant with the regulations for closure and reclamation. Major activities planned for the various mine components and facilities are detailed as follows. The closure plan will be conducted in two phases. Phase One (first 15 years) will include reclamation of the majority of the site and time to allow the majority of the tailings consolidation to occur, while Phase Two (16+ years) will include closure of the TMF and long term maintenance of the site. Both phases will include water treatment as discussed in Section 17.3.4. Figure 17 3 and Figure 17 4 illustrate the closure phases.

Figure 17-3: Phase One Closure



Source: SRK, 2022.

Figure 17-4: Phase Two Closure



Source: SRK, 2022.

17.3.3.1 Open Pit Workings

The pit wall is expected to be geotechnically stable in the long-term, and therefore no in-pit work is required at closure. The pit perimeter straddles a steep mountainous ridgeline: installation of typical preventative safety measures such as fencing, or warning berms are not practical to install. Clear signage will be installed to warn of existing dangers, such as highwall presence and falling rock in areas which could potentially be accessed by the public.

Once operations are completed, the water in the pit will be allowed to rise and the pit will be used as storage for the WTP. Seepage water from consolidation of the tailings, infiltration through the tailings cover, and seepage from the WRF will report to the open pit for seasonal treatment. Sludge generated from water treatment will be filtered through filter bags on a lined portion of the tailings area and filtrate solution will be returned to the pit to be treated through the WTP.

The pit lake will not be allowed to overflow, and the elevation will be carefully monitored to provide storage of the freshet volume and probable maximum precipitation event. Further hydrological and geotechnical studies are required to determine the maximum elevation to which water could be stored in the pit to prevent seepage through bedrock and into the environment. A conservative approach assumes that the maximum pit lake elevation will be maintained 40 m below the top of competent bedrock surface. An emergency spillway will be constructed.

17.3.3.2 Waste Rock Facility and Tailings Management Facility

It is anticipated that the WRF will be PAG after closure and into the foreseeable future. The face of the WRF will be regraded at closure for long-term stability at an overall slope of approximately 2.5H:1V. Efforts will be made to create a convex shape on the WRF face, which is the predominant shape of natural slopes in mature landscapes in the area. This shape together with installation of intermittent diversion berms on the face of the WRF slope will reduce rilling and erosion from uncontrolled runoff. The diversion berms on the WRF face will divert water to the perimeter. There, geosynthetic-lined and armoured perimeter channels will be installed at the interface of the existing ground and the WRF. The lined channels will limit infiltration of collected runoff both from the WRF surface and upstream run-on, limiting the volume of seepage reporting to the WRCP which requires treatment. Upon completion of grading, all surface areas of the WRF will be covered with geomembrane overlain by approximately 830 mm of overburden and 170 mm of growth media, for a full cover thickness of approximately one meter. Modelling of the geomembrane cover showed a significant reduction in the amount of water that would need to be treated.

In the later years of operation, water maintained in tailings will be kept at a minimum. The eastern half of tailings will be used as a WTP sludge storage area. The eastern side of the tailings dam will drain into the pit which, in turn, will be used to feed the WTP. An engineered drainage layer will be place on the eastern side of tailings so that equipment can be operate on top while consolidation solution can still drain to the pit. Equipment will be used to place pipes and filter bags. Sludge from the WTP will be pumped into the filter bags. Sludge will be retained in the bags and filtrate solution, along with tailings consolidation solution, will return to the pit. As bags fill, they will be taken off-line and allowed to dry. When the eastern half of tailings has been filled with one layer of bags, they will be covered with engineered drainage material and second layer of filter bags will be filled on top and the process will be repeated. As the level of filter bags and drainage material increases, the western side will be covered with geomembrane overlain by 830 mm of overburden and 170 mm of growth media to reduce the volume of water needing to be treated. Tailings consolidation is expected to continue for up to 100+ years but, the majority of the consolidation is expected to occur in the first 15–20 years post closure.

The western side of tailings will not be needed for sludge storage, so waste rock will be relocated the western half of the tailings surface to provide positive drainage towards the western diversion channel. Upon completion of grading and placement of waste rock, the western half of the TMF will be covered with 830 mm of overburden and 170 mm of growth media, for a full cover thickness of approximately one meter.

An engineered spillway will be installed between the east side of the TMF and the pit at the completion of mining. The spillway is designed to safely convey the probable maximum flood event from the TMF to the pit, both during Phase One and Phase Two closure.

Material for covering the WRF and TMF will be relocated from the overburden and topsoil stockpiles. Once all material has been relocated, the footprints will be reclaimed, and the Process Pond will be reclaimed and revegetated.

All disturbed areas will stabilized and seeded with native species of grasses and shrubs to re-establish vegetation. A revegetation and cover study must be conducted to verify plant survival at the elevations proposed of the waste rock dump and reclaimed tailings facility. Efforts to work with the Alaska Plants Material Center should be taken to maximize the potential for sustainable regrowth. The elevation, climate and latitude will likely limit the early growth success and trials should be conducted throughout the LOM to determine the management practices for revegetation and stabilization at this site. Although, revegetation is the preferred method of stabilization for many projects, other methods of stabilization could be implemented such as a non-reactive, durable rock cover.

17.3.3.3 Buildings and Equipment

The WTP and a modified power generation plant will be left in place, together with all appurtenant facilities and utilities. All other steel frame buildings including the mill, the truck shop, and the conveyors will be demolished selectively by removing the roof and siding and then dismantling the steel frames and trusses. Resulting debris will be disposed of in an approved landfill that will be located on the WRF. Concrete walls, pillars, and beams will be demolished to the ground and concrete foundations will be covered in place. Controlled blasting may be used to help with the demolition. All sumps and cavities will be backfilled to ground level. The rock fill pads underlying all buildings and equipment will be re-graded to prevent permanent ponding.

Non-hazardous and hazardous waste will be segregated. Hazardous waste will be placed in suitable containers and hauled to a licensed disposal facility, while non-hazardous waste will be placed in the landfill.

Any unwanted mobile or stationary equipment will be stripped of electronics and batteries, drained of all fluids (fuels, lubricants, coolants), decontaminated by power washing, and placed in the landfill for final disposal.

17.3.3.4 Mine Infrastructure

Mine support infrastructure will consist of internal access roads, haul roads, and rock fill pads underlying the site buildings and facilities.

All bridges and culverts associated with the haul roads and internal access roads will be removed and natural drainage will be restored. Swales will be created where needed, to allow continued use of the roads into post-closure water treatment, monitoring, and maintenance. Roads that are not needed in post-closure will be ripped and re-vegetated.

The surface of the rockfill pads underlying some of the buildings and facilities on site will be re-graded and/or crowned as necessary to prevent ponding of water. The pads will be scarified and re-vegetated. Additional overburden up to 300 mm may be applied, as needed, to promote revegetation.

The site access road will be maintained as long as water treatment is occurring on site.

17.3.3.5 Landfill

An unlined non-hazardous landfill will be located in the WRF. Demolition waste and other non-hazardous waste will be placed in the landfill and consolidated to minimize the occupied volume. The waste will then be covered with at least 1 m of waste rock. The final surface will then be graded to prevent permanent ponding and will be covered similarly to the rest of the WRF cover.

17.3.3.6 Water Management System

The water management system consisting of the pit lake, Process Pond, diversion channels, and various pipelines will be largely decommissioned. The Process Pond will be breached, regraded and re-vegetated. New diversion channels along the perimeter of the WRF will be created to manage surface runoff on and around the TMF and the WRF, as well as intake and discharge pipelines between the pit and the WRCP. Runoff from the WRF cover is expected to meet water quality standards and will be discharged to the environment. Seepage from the WRF will report to the WRCP and be pumped to the pit for treatment. As the pit fills naturally, water levels will be managed by treating and discharging treated water for the long-term.

The WRCP will collect seepage from the WRF in perpetuity. The expected seepage volume reporting to the WRCP in closure is approximately 800 m^3 per day. All contact water will be sent to the pit for storage prior to treatment. Water treatment is expected to occur year-round, beginning in May of the first year of closure.

17.3.4 Closure Water Treatment

When the TMF is closed at the end of the Operations phase, a biological/chemical/physical plant will be added to treat the RO reject. This phased approach provides multiple benefits, including allowing time to gain insight on the site’s true wastewater flow and quality so that the RO reject treatment design can be improved upon. This treatment train will include:

- Gypsum clarifier: Reduces sulphate concentration in the reject treatment stream train to the gypsum saturation limit by adding lime stored in silos. The system will also remove cationic metals such as cadmium and zinc.
- Anaerobic biological treatment: Biochemically reduces nitrate to nitrogen gas and selenate and selenite to elemental selenium (a solid). Selenium and metal solids removed in the sludge are settled out and removed by the biological ballasted clarifier.
- Aerobic biological treatment: Polishes residual organics added in the upstream anaerobic biological treatment. Solids are separated via a membrane filter with iron coprecipitation for polishing selenium.

During the Closure phase, the WTP will either be run year-round or for partial year based on water storage requirements. During the Operations phase, mill water demands will affect site water flows such that the WTP can be shut down and the discharge eliminated during roughly half of the year (winter).

During Operations, sludge from the WTP will be disposed of at the TMF. During Closure, sludge from the WTP will be disposed in filter bags on a lined area which will decant solution back to the pit.

17.3.5 Post-Closure

Continued presence at site will be required in the post-closure period for as long as water treatment is necessary. During the post-Closure phase, the WTP will either be run year-round or for part of the year based on water storage requirements. If operated for part of the year, during the winter months, the RO portion of the plant would be shut down and the bacteria in the biologic portion of the plant will be maintained with the feed of a concentrated brine solution from the RO plant. This will eliminate the need to bring large quantities of reagents to site during winter, eliminate winter operational issues, and reduce the cost of winter road maintenance. Generators and mobile equipment will also be required for the WTP operations as well as sampling and monitoring activities.

In the short-term (up to 10 years following closure) monitoring to confirm that the closure objectives are met will be based on the following requirements:

- The site should be visually inspected by a qualified Professional Engineer annually for three consecutive years and less frequently thereafter for up to 10 years to ensure that erosion-prone areas have stabilized.
- The cover systems over the WRF and the TMF should be regularly inspected by a qualified inspector to ensure the physical integrity of the cover is maintained. Inspection intervals should be by-annually for the first 10 years after construction.
- The site should be inspected by a vegetation specialist to confirm suitability of the revegetation efforts. Inspections should be completed at the following intervals, unless otherwise recommended by the vegetation expert: Year 1, Year 3, Year 7 and Year 10 post-closure.
- The WRCP and TMF dams will need a Periodic Safety Inspection by a qualified engineer every three years per ADSP (2017) for perpetuity.

In the long-term, continuous water quality monitoring will be implemented at sampling frequencies prescribed by future discharge permits.

Maintenance will be performed on areas that monitoring identifies as needing repairs. Water treatment in perpetuity will likely require a well-defined set of discharge quality criteria to be determined at the later stages of permitting and design

17.3.6 Closure Cost Estimate

17.3.6.1 General

The estimated closure costs are determined from unit rates based on projects located in Alaska. The indirect costs were included as percentages of the estimated direct costs. Indirect percentages are based on reclamation and closure guidelines for Alaska.

Long-term water treatment and maintenance of certain water management facilities were calculated separately, and an NPV value is provided for the first 100 years of closure, at a discount rate of 4.3%. The discount rate is the mutually agreed upon rate developed by the State of Alaska to be used for all financial assurance closure costs.

A summary of Trilogy's estimated closure and reclamation costs, for all items except water treatment, are provided in Table 17-1. Closure costs for the water treatment are captured separately in Section 17.3.6.2.

Table 17-1: Summary of Closure and Reclamation Costs

Cost Items	Subtotals Closure Costs ($M)	
DIRECT CLOSURE COSTS		
Waste Rock Facility	$	69.9
Yards and Laydown Areas	$	0.2
Buildings and Equipment	$	4.0
TMF Spillway (Phase 1)	$	4.9
Surface Water Management (Phase 1)	$	7.7
Ponds and Stockpiles	$	0.2
Roads and Diversion Channel Regrade	$	0.2
Hazardous Waste and Solid Waste Disposal	$	0.7
Camp and Turn-around Costs	$	2.2
Subtotal Direct Closure Cost	**$**	**90.1**
INDIRECT CLOSURE COSTS		
Contingency (25%)	$	22.5
Engineering Redesign (3%)	$	2.7
Contract Administration (7%)	$	6.3
Performance Bond (3%)	$	2.7
Liability Insurance (0.5%)	$	0.5
Subtotal Indirect Closure Cost	**$**	**34.6**
Total Closure Cost	**$**	**124.7**
POST-CLOSURE COSTS (100 years)		
Water Quality Sampling	$	8.3
Annual Reporting	$	2.7
Post-Closure Monitoring and Inspections	$	10.9
Vegetation and Cover Maintenance	$	5.7
Camp and Road Maint.	$	21.0
Mobile Equipment and Operating	$	13.9
TMF Reclamation & Diversions (Phase 2)	$	48.8
Sludge Repository Construction	$	5.5
Sludge Management Crew and Camp Support	$	4.6
Sludge Filtration	$	10.0
Subtotal Post-Closure Costs (undiscounted)	$	**131.4**
Post-Closure Costs NPV (4.3% Discount)	**$**	**46.1**
Discounted General Closure Cost	**$**	**170.8**

17.3.6.2 Closure Water Treatment Costs

Annual undiscounted costs associated with long-term operations of the WTP are estimated to be $11.7 million in Phase 1 closure (15-years post-closure) and $10.8 million in Phase 2 closure (85 years thereafter), amounting to $1,095 million over the 100-year closure period. These costs equate to $258 million when discounted at 4.3% p.a. to the first year of post-production.

17.3.7 Reclamation and Closure Financial Assurance

In the absence of activities on Federal Land in the mine area and excluding the access road to the district from the Dalton Highway, there would not be any financial assurance requirements from the Federal government for the mine.

There are three State of Alaska agencies that require financial assurance in conjunction with approval and issuance of large mine permits.

The ADNR, under authority of Alaska Statute 27.19, requires a reclamation plan be submitted prior to mine development and requires financial assurance, typically prior to construction, to assure reclamation of the site. The ADNR Dam Safety Unit also requires a financial assurance sufficient to cover the cost of decommissioning dams or the cost for long term maintenance and monitoring of dams that will remain in-place. The ADEC requires financial assurance both during and after operations, and to cover short and long-term water treatment, if necessary, as well as reclamation costs, monitoring, and maintenance needs. The State requires that the financial assurance amount also include the property holding costs for a one-year period.

The final financial assurance amount will be calculated through the process of reviewing and approving the Arctic Project reclamation plan during the formal permitting process. In general, the approach is to combine the reclamation costs, post-closure monitoring costs and the long-term annual water treatment costs into a financial amount that includes deriving the NPV of the long-term costs and combining that with the reclamation cost. The costs assume that a third-party contractor will perform and manage the Arctic Project. The estimate required for bonding is likely to be similar to the estimate provided above. Until actual Owner operating costs are known, the internal closure liability cannot readily be reduced.

Ambler Metals may satisfy the State financial assurance requirement by providing any of the following: (1) a surety bond, (2) a letter of credit, (3) a certificate of deposit, (4) a corporate guarantee that meets the financial tests set in regulation by the ADNR commissioner, (5) payments and deposits into the trust fund established in AS 37.14.800, or (6) for the dam- or ADEC-related obligation - any other form of financial assurance that meets the financial test or other conditions set in regulation by the ADNR or ADEC commissioners.

The adequacy of the reclamation plan, and the amount of the financial assurance, are reviewed by the State agencies at a minimum of every five years and must be updated whenever there is a significant change to the mine plan of operations, or other costs that could affect the reclamation plan costs.

17.4 Permitting Considerations

17.4.1 Exploration Permits

Current mineral exploration activities are conducted at the Arctic deposit under State of Alaska and Northwest Arctic Borough (NWAB) permits. The State of Alaska Miscellaneous Land Use Permit (MLUP) and the NWAB Permit both expires at the end of 2022 and will be renewed.

Cumulative surface disturbance for exploration activities on the Arctic Project remains less than 5 acres (excluding historic disturbance that includes roads and camp disturbances) and therefore there are currently no State requirements for reclamation bonding for the Arctic Project.

Ambler Metals also has NWAB Title 9 Conditional Use Permits authorizing exploration and bulk fuel storage, use of airstrips, operation of a landfill and gravel extraction expires on December 31, 2022. No bonding is required for the borough permits.

Ambler Metals obtained several other permits for camp-support operations. These permits include a drinking water permit, a domestic wastewater discharge permit, camp establishment permits, and construction and operation of a Class III Camp Municipal Landfill, all of which are issued by the Alaska Department of Environmental Conservation (ADEC). Temporary water-use authorizations were issued by the ADNR, a Title 16 Fish Habitat permit, and a wildlife hazing permit were issued by the ADF&G.

17.4.2 Major Mine Permits

The following discussion identifies the major permits and approvals that will likely be required for development of the Arctic deposit.

Permits would be required from Federal, State, and Regional agencies, including: the US Army Corps of Engineers (USACE), ADEC, ADF&G, ADNR, and the NWAB. The State of Alaska permit for exploration on the Arctic Project, the Annual Hardrock Exploration Activity (AHEA) Permit, is obtained and renewed every five years through the ADNR – Division of Mining, Land and Water.

The types of major mine permits required by the Arctic Project are largely driven by the underlying land ownership; regulatory authorities vary depending on land ownership. The Arctic Project area includes patented mining claims (private land under separate ownership by Ambler Metals and NANA), State of Alaska land, and NANA land (private land). The mine pit would situated mostly on patented land while the mill, TMF and WRF would be largely on State land. Other facilities, such as the camp, would be on NANA land. Federal land would likely underlie portions of an access road between the Dalton Highway and the project area. However, permits associated with such an access road are being investigated in a separate action by the State of Alaska and are not addressed in this report. A list of likely major mine permits is included in Table 17 2.

Because the infrastructure for the Arctic Project is situated to a large extent on State land, it will likely be necessary to obtain a Plan of Operation Approval (which includes the Reclamation Plan) from the ADNR. The Arctic Project will also require certificates to construct and then operate a dam(s) (tailings and water storage) from the ADNR (Dam Safety Unit) as well as water use authorizations, an upland mining lease and a mill site lease, as well as several minor permits including those that authorize access to construction material sites from ADNR.

The ADEC would authorize waste management under an integrated waste management permit, air emissions during construction and then operations under an air permit, and require an Alaska Pollution Discharge Elimination System (APDES) permit for any wastewater discharges to surface waters, and a Multi-Sector General Permit for stormwater discharges. The ADEC would also be required to review the USACE Section 404 permit to certify that it complies with Section 401 of the Clean Water Act (CWA).

The ADF&G would have to authorize any culverts or bridges that are required to cross fish-bearing streams or other impacts to fish-bearing streams that result in the loss of fish habitat.

The USACE would require a CWA Section 404 permit for dredging and filling activities in Waters of the United States, including jurisdictional wetlands. The USACE Section 404 permitting action would require the USACE to comply with National Environmental Policy Act (NEPA) and, for a project of this magnitude, the development of an Environmental Impact Statement (EIS) is anticipated. The USACE would likely be the lead federal agency for the NEPA process. The NEPA process will require an assessment of direct, indirect and cumulative impacts of the Arctic Project and the identification of project alternatives, and include consultation and coordination with additional federal agencies, such as the US Fish and Wildlife Service (if endangered or threatened species are present) and National Marine Fisheries Service (if essential fish habitat is present), and with the State Historic Preservation Office and Tribal Governments under Section 106 of the Historical and Cultural Resources Protection Act.

As part of the Section 404 permitting process, the Arctic Project will have to meet USACE wetlands guidelines to avoid, minimize and mitigate impacts to wetlands. The USACE may require Ambler Metals to develop a compensatory wetlands mitigation plan for mitigating unavoidable wetlands impacts.

The Arctic Project will also have to obtain approval for a Master Plan from the NWAB. In addition, actions will have to be taken to change the borough zoning for the Arctic Project area from Subsistence Conservation and General Conservation to Resource Development.

The overall timeline required for permitting would be largely driven by the time required for the NEPA process, which is triggered by the submission of the Sec. 404 permit application to the USACE. The timeline includes the development and publication of a draft and final EIS and ends with a Record of Decision (ROD), and 404-permit issuance. In Alaska, the EIS and other State and Federal permitting processes are generally coordinated so that permitting and environmental review occurs in parallel. The NEPA process could require between two to three years to complete and could potentially take longer.

Table 17-2: Major Mine Permits Required for the Arctic Project

Agency	Authorization
State of Alaska	
ADNR	Plan of Operations Approval (including Reclamation Plan)
	Upland Mining Lease
	Mill Site Lease
	Reclamation Bond
	Certificate of Approval to Construct a Dam
	Certificate of Approval to Operate a Dam
	Water Rights Permit to Appropriate Water
ADF&G	Title 16 Permits for Fish Passage (authorize stream crossings)
ADEC	APDES Water Discharge Permit
	Alaska Multi-Sector General Permit (MSGP) for Stormwater
	Stormwater Discharge Pollution Prevention Plan (part of MSGP)
	Section 401 Water Quality Certification of the CWA Section 404 Permit
	Integrated Waste Management Permit
	Air Quality Control – Construction Permit
	Air Quality Control – Title V Operating Permit
	Reclamation Bond
Federal Government	
EPA	Spill Prevention, Control, and Countermeasure (SPCC) Plan (fuel transport and storage)
USACE	CWA Section 404 Dredge and Fill Permit
NWAB	
NWAB	Master Plan Approval and rezoning lands from Subsistence Conservation to Resource Extraction

Note: "Major" permits generally define critical permitting path. Additional "minor" permits are also required.

17.5 Social or Community Considerations

The Arctic Project is located approximately 40 km northeast of the villages of Shungnak and Kobuk, and 65 km east-northeast of the community of Ambler. The population in these villages are 151 in Kobuk (2020 Census), 210 in Shungnak (2020 Census), and 275 in Ambler (2020 Census). Residents largely live a subsistence lifestyle with incomes supplemented by guiding, local development projects, employment through tribal and city councils, government aid, and employment both in and outside of their home villages.

The Arctic Project has the potential to significantly improve work opportunities for residents during the exploration phase, construction, and during full operation. Trilogy's joint venture, Ambler Metals works directly with the Upper Kobuk villages and communities throughout the region to employ residents as mechanics, geotechnicians, core cutters, administrative staff, camp services, heavy equipment operators, drill helpers, and environmental technicians.

Stakeholder outreach and community meetings in the region by the Arctic Project's owners over many years have provided the opportunity to engage with residents, provide updated information on the Arctic Project and future plans for the UKMP, hear concerns, answer questions, and build relationships.

This engagement has also identified various hurdles residents have faced when applying for employment. Opportunities have been created for NANA shareholders to apply and receive educational scholarships, participate in job shadowing at Bornite, driver's license courses, and heavy equipment operator training sponsored by the Arctic Project's owners.

It is the company's goal to continue and grow these efforts throughout the permitting process and the life of the Arctic Project – encouraging and supporting education, job training, employment, and economic growth.

17.6 Comment on Environmental Studies, Permitting, and Plans, Negotiations, or Agreements with Local Individuals or Groups

In the QP's opinion, the current plans are adequate to address any issues related to environmental compliance, permitting, and local individuals or groups.

18 CAPITAL AND OPERATING COSTS

18.1 Capital Costs

18.1.1 Introduction

The capital cost estimate for the Artic Project was developed at PFS-level study under S-K 1300 standards with an accuracy of +/-15% using the Association for the Advancement of Cost Engineering (AACE) Class 3 estimate standards with a contingency ≤15%. The estimate included the cost to complete the design, procurement, construction, and commissioning, of all the identified facilities.

The breakdown of the responsibilities for the capex by facility is as follows:

- Ausenco
  - Process Plant
  - Onsite Infrastructure
- Wood
  - Mining Fleet
  - Mine Development
  - Haul Roads
- Brown & Caldwell
  - Reverse Osmosis Water Treatment Plant
- Trilogy
  - Main Access Roads
  - Tailings Storage Facilities
  - WRF
  - Water Management
  - Reclamation & Closure
  - Owners Costs

This estimate collectively presents the entire costs for the Arctic Project.

18.1.2 Arctic Project Execution

The estimate was based on the traditional engineering, procurement, and construction management (EPCM) approach where the EPCM contractor will oversee the delivery of the completed project from detailed engineering and procurement to handover of working facility. The EPCM contractor would engage and coordinate several subcontractors to complete all work within the given scopes. Typical vertical and/or horizontal contract packages were identified and aligned with different pricing models such as, but not limited to:

- Schedule of rates (unit price)
  - This contract pricing model is based on estimated quantities of items included in the scope and their unit prices. The final contract price is dependent on the quantities needed to complete the work under the contract.
- Time and materials
  - Time and materials (T&M) fixes rates for labour and material expenditures, with the contractor paid on the basis of actual labour hours (time), usually at specified hourly rates, actual cost of materials and equipment usage, and an agreed upon fixed add-on to cover the contractor's overheads and profit.
- Design and construct
  - With this option one entity will provide design and construction services for an awarded scope of work. A higher degree of price certainty can be achieved when a lump sum arrangement is used; this method also provides a single point of accountability and an improved integration of the design with construction.

18.1.3 Work Breakdown Structure

The estimate was arranged by major area, area, major facility, and facility. Each sub-area was further broken down into disciplines such as earthworks, concrete etc. Each discipline line item was defined into resources such as labour, materials, equipment, etc., so that each line comprises all the elements required to complete each task.

The work breakdown structure (WBS) was developed in sufficient detail to provide the required level of confidence and accuracy and also to provide the basis for further development as the Arctic Project moves into execution phase.

18.1.4 Estimate Summary

The estimate is derived from a number of fundamental assumptions as shown on the PFDs, 3D model, mechanical equipment list, electrical equipment list, material take offs, scope definition and WBS, it includes all associated infrastructure as defined within the scope of works.

The Capital Cost Estimate has been summarized at the levels indicated with a base date of Q4-2022 with no provision for forward escalation. A summary of the capital costs estimate by Major Facilities and Major Disciplines are defined in Table 18-1and Table 18-2 respectively.

Table 18-1: Estimate Summary Level 1 Major Facility

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
1000	Mining	296.7	17.5	314.2
2000	Crushing	42.5	0	42.5
3000	Process Plant	158.0	1.3	159.3
4000	Tailings	88.3	32.4	120.7
5000	On-site Infrastructure	172.8	35.1	207.9
6000	Off-site Infrastructure	75.8	0	75.8
Sub-total Direct Costs		**834.1**	**86.3**	**920.4**
7000	Indirects	177.4	15.1	192.5
8000	Contingency (~12% of total project cost)	138.5	13.0	151.5
9000	Owner Costs	26.8	0	26.8
Sub-total Indirect Costs		**342.7**	**28.1**	**370.8**
Arctic Project Total		**1,176.8**	**114.4**	**1,291.2**

Table 18-2: Initial Estimate by Major Discipline

Discipline Code	Discipline Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
A	Architectural - Buildings	61.4	9.8	71.2
A	Architectural – Permanent Camp Facility	45.4	0	45.4
B	Earthworks	28.6	0.9	29.5
C	Concrete	42.1	4.6	46.7
S	Structural Steel	20.1	0	20.1
F	Platework	6.6	0	6.6
M	Mechanical Equipment	102.2	1.8	104.0
P	Piping & Fittings	51.3	0	51.3
L	Electrical Bulks	16.8	0	16.8
E	Electrical Equipment	42.6	0	42.6

Discipline Code	Discipline Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
I	Instrumentation	4.8	0	4.8
N	Mobile Equipment	7.6	0	7.6
R	Third Party	404.6	69.2	473.8
Sub-total Direct Costs		**834.1**	**86.3**	**920.4**
O	Owners Costs	26.8	0	26.8
T	Project Delivery	90.0	14.3	104.3
U	Filed Indirects	76.9	0.9	77.8
V	Spares/ First Fills / Vendor Reps	10.4	0	10.4
Y	Contingency (~12% of total project cost)	138.5	13.0	151.5
Sub-total Indirect Costs		**342.7**	**28.1**	**370.8**
Arctic Project Total		**1,176.8**	**114.4**	**1,291.2**

18.1.5 Definition

18.1.5.1 Definition of Costs

The estimate is broken out into direct and indirect capital costs.

Initial capital is the capital expenditure required to start up a business to a standard where it is ready for initial production.

Sustaining capital is the capital cost associated with the periodic addition of new plant, equipment or services that are required to maintain production and operations at their existing levels.

Direct costs are those costs that pertain to the permanent equipment, materials and labour associated with the physical construction of the process facility, infrastructure, utilities, buildings, etc. Contractor's indirect costs are contained within each discipline's all-in rates.

Indirect costs include all costs associated with implementation of the plant and incurred by the owner, engineer or consultants in the design, procurement, construction, and commissioning of the Arctic Project.

18.1.5.2 General Methodology

The estimate is developed based on a mix of detailed material take-offs and factored quantities and costs, detailed unit costs supported by contractor bids and budgetary quotations for major equipment supply as outlined in Ausenco's requirements.

The structure of the estimate is a build-up of the direct & indirect cost of the current quantities; this includes the installation/construction hours, unit labour rates and contractor distributable costs, bulk and miscellaneous material and equipment costs, any subcontractor costs, freight and growth.

The methodology applied and source data used to develop the estimate included:

- Define the scope of work.
- Quantify the work in accordance with standard commodities.
- Organize the estimate structure in accordance with an agreed WBS.
- Calculate "all in" labour rates for construction work.
- Determine the purchase cost of equipment and bulk materials.
- Determine the installation cost for equipment and bulks.
- Establish requirements for freight.
- Determine the costs to carry out detailed engineering design and project management.
- Determine foreign exchange content and exchange rates.
- Determine growth allowances for each estimate line item.
- Determine the estimate contingency value by probabilistic method (if required).
- Undertake internal peer review, finalize the estimate, estimate basis and obtain sign off by the Project Manager and Qualified Professional.

18.1.5.3 Basic Information

The following basic information pertains to the estimate:

- The estimate base date is Q4-2022.
- The estimate is expressed in United States Dollars ($).

18.1.5.4 Exchange Rates

The exchange rates in Table 18-3 were obtained from the XE.com website as of November 8, 2022 and confirmed for use the capital cost estimate by Trilogy.

Table 18-3: Estimate Exchange Rates

Exchange Rate	$
AUD 1.430	1.000
EUR 0.940	1.000
CAD 1.300	1.000

Notes: AUD = Australian Dollar, EUR = Euro, CAD = Canadian Dollar

18.1.5.5 Market Availability

The pricing and delivery information for quoted equipment, material and services was provided by suppliers based on the market conditions and expectations applicable at the time of developing the estimate.

The market conditions are susceptible to the impact of demand and availability at the time of purchase and could result in variations in the supply conditions. The estimate in this Report was based on information provided by suppliers and assumes there are no problems associated with the supply and availability of equipment and services during the execution phase.

18.1.6 Mining (WBS 1000)

The total mine capital cost estimated to develop the Arctic Project is $294.9 M (Table 18-4). The initial capital costs add a total of $277 M, including $150 M in capitalized preproduction mining operating costs for a two-year preproduction period, $90 M in initial capital expenditures for mobile equipment, $7 M in initial equipment spares, $1 M for the explosive storage and transfer site, and $29 M for haul roads construction. Following preproduction, a sustaining capital cost of $18 M is required primarily for additions to truck mining fleet, and to replace equipment beyond its manufacturer's recommended service life. An additional $40 M are estimated as a contingency for the initial capital cost. Equipment purchase payment has been estimated to be done in the period before it is required to account for the following payment schedule: 10% at order placement, 70% at shipping from ex-work, and 20% at site arrival.

Table 18-4: Mining Initial and Sustaining Capital Cost

Cost Area	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
Pre-Production Mining	149.9	0.0	149.9
Engineering and Management	4.0	1.1	5.1
Drilling	7.6	0.0	7.6
Blasting	2.0	0.0	2.0
Loading	16.3	0.6	16.9
Hauling	43.8	1.8	45.6
Support	18.9	9.3	28.2
Maintenance	6.0	4.7	10.7
Road Construction	28.8	0.0	28.8
Sub-total Mining Cost	**277.4**	**17.5**	**294.9**

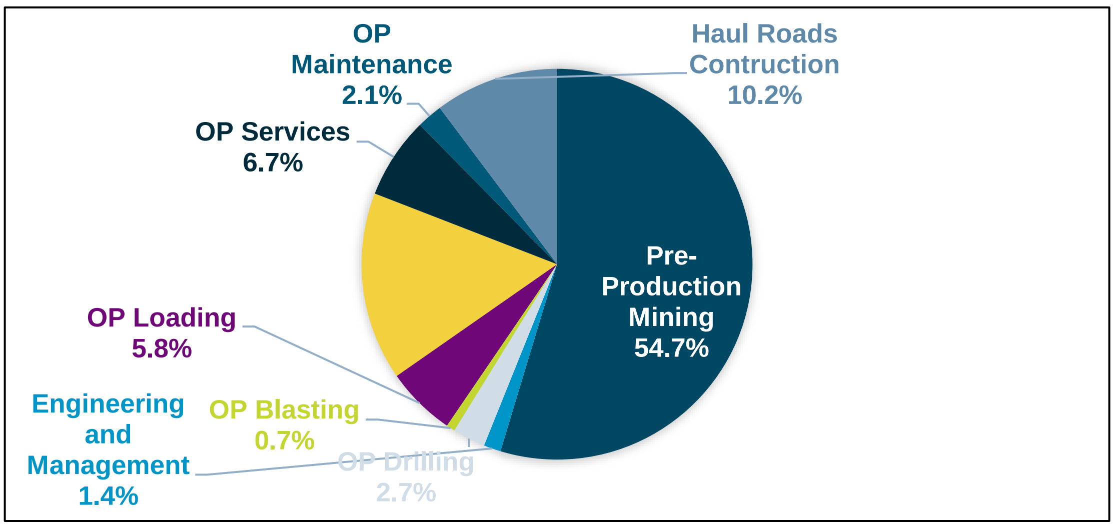
Table 18-5 provides an overview of the annual sustaining capital expenditures. Approximately $0.05/t mined is spent as sustaining capital. Capital expenditures are restricted during the last two years of mining.

Table 18-5: Sustaining Capital Cost Distribution

Period	$ M
Year 1	0.05
Year 2	-
Year 3	-
Year 4	1.46
Year 5	-
Year 6	0.75
Year 7	0.85
Year 8	0.20
Year 9	8.91
Year 10	1.93
Year 11	0.32
Year 12	3.05
Year 13	-
Total Sustaining Capital	**17.5**

Figure 18-1 provides a distribution of initial mining capital expenditures by cost area.

Figure 18-1: Initial Mining Capital Cost Distribution by Cost Centre



Source: Wood, 2022.

18.1.6.1 Mining Capital Cost Basis of Estimate

The basis of estimate for the mining equipment, ancillary equipment, capital spares, and other capital costs are provided in the following sub-sections.

18.1.6.2 Mining Equipment

The equipment capital costs were provided by multiple suppliers as part of a budgetary request for quotation process. The capital costs are inclusive of transport to Fairbanks, Alaska, assembly, training, tires, and options for cold weather operations. For example, Table 18-6 provides a capital cost buildup for the 141 t trucks.

Table 18-6: 141t Truck Capital Pricing

Capital Pricing 141t Truck	$ M
Base Price	2.1
Tires	0.1
Freight and Duty to Fairbanks, AK	0.1
Assembly	0.2
Total Purchase Price	**2.5**

In addition to capital pricing, equipment suppliers provided machine operational lives and mechanical availability, which were adjusted by Wood based on experience with similar type of equipment at other operating mines. Table 18-7 provides the basic inputs used to estimate mine equipment capital cost.

Table 18-7: Mine Equipment Capital Cost Inputs

Equipment	Capital Cost ($ M)	Machine Life (GOH)	Mechanical Availability (%)
171 mm Production Drill	2.36	100,000	83%
265 t/15 m^3 Hydraulic Face Shovel	4.40	100,000	85%
124 t/12 m^3 Front End Loader	2.08	80,000	85%
144 t Haul Truck	2.52	100,000	85%
41t Articulated Truck	0.88	42,000	86%
26 t/5 m^3 Front End Loader	0.55	60,000	86%
68 t/4 m^3 Hydraulic Excavator	1.08	60,000	86%
35 t/1.4 m^3 Hydraulic Excavator	0.59	48,000	85%
74 t/455 kW Track Dozer	1.38	60,000	85%
50 t/419 kW Rubber Tired Dozer	1.46	60,000	85%
40 t Articulated Sand Truck	0.88	42,000	85%
33 t/217 kW Motor Grader	1.39	60,000	85%
35,000-litre water truck	1.34	42,000	85%
41t Articulated Fuel/Lube Truck	1.05	42,000	85%

18.1.6.3 Ancillary Equipment

Ancillary equipment capital costs are estimated from a variety of sources including equipment supplier budgetary quotes, Wood's data base, and CostMine® costing service. Table 18-8 shows the ancillary equipment pricing and sourcing.

Table 18-8: Ancillary Equipment Capital Cost Inputs

Equipment	Source	Cost Each ($ M)
Truck Mounted 40 t Crane	Vendor	0.5
100t Rough Terrain	CostMine	1.4
5t Forklift	Vendor	0.1
10t Forklift	Vendor	0.4
Mechanic Service Truck	CostMine	0.2
Small Fuel/Lube truck	CostMine	0.2
55kW Skid Steer	Vendor	0.1
Flatbed Truck	CostMine	0.1
45t Telehandler	Vendor	0.8
Small backhoe/loader	Vendor	0.2
Hydraulic Hammer	Vendor	0.2
90t Lowboy	CostMine	0.3
Compactor	CostMine	0.2
Light Plant	Vendor	0.3
Transport Tractor	Wood	0.2
Tire Handler Truck	Vendor	0.4
3/4-ton Pickup	CostMine	0.5
1 ton Pickup	CostMine	0.6
Crew Bus	CostMine	0.6
Fleet Management System	Vendor	0.7
Mine & Geology Software	Vendor	0.5
MMU - Heavy ANFO (blend) truck	CostMine	0.3
Stemming Truck	CostMine	0.1
Laser Scan	Wood	0.02
Survey Drones	Wood	0.01
Total Station	Wood	0.03

18.1.6.4 Capital Spares

Capital spares are listed in Table 18-9. The capital spares are purchased during pre-production to minimize down time due to waiting for long lead time repair parts and wear items. Included in the capital spares are buckets, tires, and major and minor components. The capital spare costs for buckets and tires are supplier-provided. For minor and major components, an allowance of 5% of the initial purchase price was applied for each equipment fleet.

Table 18-9: Capital Spares

Description	$ M
Spare Bed - 144 t Haul Truck	0.2
Start Up Spares - 144 t Haul Truck	0.1
Spare Tires (2) - 144 t Haul Truck	0.04
FMS - 144 t Haul Truck	0.2
Spare Tires (2) - 41t Articulated Truck	0.01
Start Up Spares - 41t Articulated Truck	0.04
Spare Bucket- 265 t/15 m^3 Hydraulic Face Shovel	0.3
Startup Spares- 265 t/15 m^3 Hydraulic Face Shovel	0.2
FMS - Komatsu 265 t/15 m^3 Hydraulic Face Shovel	0.04
Spare Bucket- 68 t/4 m^3 Hydraulic Excavator	0.1
Startup Spares- 68 t/4 m^3 Hydraulic Excavator	0.05
FMS - 68 t/4 m3 Hydraulic Excavator	0.01
Spare Bucket- 124 t/12 m^3 Front End Loader	0.1
Startup Spares- 124 t/12 m^3 Front End Loader	0.1
FMS - 124 t/12 m3 Front End Loader	0.04
Spare Bucket- 26 t/5 m^3 Front End Loader	0.2
Startup Spares- 26 t/5 m^3 Front End Loader	0.03
FMS - 26 t/5 m3 Front End Loader	0.01
Startup Spares- 171 mm Production Drill	0.1
FMS - 171 mm Production Drill	0.07
Startup Spares- 35 t/1.4 m^3 Hydraulic Excavator	0.03
Startup Spares- 74 t/455 kW Track Dozer	0.07
Startup Spares- 50 t/419 kW Rubber Tired Dozer	0.07
Startup Spares- 41 t Articulated Sand Truck	0.04
Startup Spares- 33 t/217 kW Motor Grader	0.07
Startup Spares- 35,000-litre water truck	0.07
Startup Spares- 41t Articulated Fuel/Lube Truck	0.05
FMS - Service Equipment	0.03

18.1.6.5 Other Capital Costs

Other capital cost includes haul roads construction, equipment shipment from Fairbanks to site, and explosive storage facilities and transfer site.

Haul roads were estimated by Wood. The additional freight cost from Fairbanks, Alaska to site was estimated at $9,395/load during the preproduction period, $8,541/load during the production period, for loads less than 10’6” wide and 60,000 lbs. Heavier and wider loads, like trucks and shovels, were quoted separately. All freight costs were based on an in-state logistics quote. Explosive silos and transfer site cost was provided by the explosive supplier and an additional covered area of 10mx10m for blasting storage was estimated and costed using a unit rate of 3,500 $/m^2 provided by Ausenco.

Table 18-10 shows the summary of the capital cost mentioned above.

Table 18-10: Other Capital Costs

Description	$ M
Haul Roads Construction	28.8
Equipment Freight Fairbanks to Site	3.23
Explosive Silos and Transfer Site*	1.03
Blasting Explosive Storage Building	0.35
Equipment Rental for Assembly	0.44

*Note: Does not include earthworks, only infrastructure.

18.1.7 Crushing and Process Plant (WBS 2000 & 3000)

A summary of the crushing and process plant (WBS 2000 & 3000) capital cost estimate is presented in Table 18-11.

Table 18-11: Crushing and Process Plant Capital Costs

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
2100	Primary Crushing	13.3	0.0	13.3
2200	Ore Stockpile and Reclaim	24.6	0.0	24.6
2900	Crushing General	4.6	0.0	4.6
3100	Process Plant Building	31.2	0.0	31.2
3200	Grinding and Screening	29.0	0.0	29.0
3300	Flotation and Regrind	44.9	0.0	44.9
3400	Concentrate Thickening	9.4	0.0	9.4
3500	Concentrate Filtration, Storage, Loadout	13.7	0.0	13.7
3600	Reagents	7.8	0.0	7.8
3700	Process Services	9.4	0.0	9.4

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
3800	Process Control System	1.6	0.0	1.6
3900	Process Plant General	11.0	1.3	12.3
Sub-total Crushing and Process Plant Costs		**200.5**	**1.3**	**201.8**

The capital cost estimate for the process plant included provision for all mechanical and electrical equipment, mobile equipment, buildings as well as quantities for bulks such as earthworks, concrete, steel, piping, electrical and instrumentation.

All major processing equipment were sized based on the process design criteria. Once the mechanical equipment list was outlined, mechanical scopes of work were derived and sent to the market for firm and budgetary pricing by local and international equipment suppliers. Once the price quotations were reviewed and integrated, majority of the major mechanical and electrical equipment packages were sourced from firm and budgetary quotations, with the remainder of the minor process equipment pricing sourced from other recent reference projects and studies.

Additional indirect costs were included to cover field indirects, process capital spares, process vendor representatives, process commissioning support, process first fills, EPCM, construction camps, camp catering & maintenance, personnel transportation, fuel, heavy craneage and other miscellaneous construction services and temporary support costs.

18.1.8 Tailings Management Facility (WBS 4000)

The capital cost estimate for the TMF included provision for constructing the initial starter dam of the TMF to an elevation of 830 m, which is sufficient to store the first year of tailings production. The TMF would be constructed using waste rock material for the adjacent WRF and compacted in 1 m lifts. As the rock is already being delivered to the WRF, no allowance was provided for spreading and compacting the material as it is assumed that the dozers already on the WRF will handle that activity and the 1 m lifts would be compacted by haul truck traffic.

An allowance was made for excavating the overburden encountered beneath the starter dam footprint and portions of the WRF. This material will be stockpiled and used in reclamation activities at the end of mine life. Costs were estimated for the general foundation preparation within the footprint of the tailings impoundment in advance of the liner placement. Supply and installation of the geotextile and geomembrane was only considered in the capital cost estimate for placement up to the starter dam elevation of 830 m.

The storage capacity of the TMF will be increased through three additional raises of the dam in Years 1, 4 and 7 to an ultimate elevation of 892 m. Sustaining capital has been estimated for each of these raises to accommodate construction of access roads around the TMF, placement of underliner material, and installation of geosynthetics.

A summary of TMF capital costs is provided in Table 18-12.

Table 18-12: TMF Capital Costs

WBS Level	WBS Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
4110	Site Civil Infrastructure	13.3	0.0	13.3
4120	Water Systems	4.9	0.0	4.9
4130	Sewage, Waste and Water Systems	4.9	0.0	4.9
4140	Electrical Services	9.3	0.0	9.3
4910	IT & Communications	56.0	32.4	88.4
Sub-total TMF Costs		**75.8**	**88.3**	**32.4**

18.1.9 Onsite Infrastructure (WBS 5000)

18.1.9.1 Water Treatment Plant

Costs for major equipment items were based on vendor budgetary quotations solicited specifically for the Arctic Project, minor equipment costs sourced from similar recent projects.

A factor-based methodology was then used to estimate the total capital investment for the WTP. The factors used to convert the delivered equipment costs to total fixed capital investment are based on estimating norms established using experience from past projects. The cost estimate includes a total of 5,600 m^2 of heated buildings, built in phases as the elements of the WTP are built. A line item, to account for costs not itemized in other cost categories and contingency, of 30% was applied. B&C estimated the cost of the treatment system (equipment, piping, electrical systems, control systems, structural, yard improvements and freight); and Ausenco estimated the cost of the building, building foundation and civil works, and pipelines and utilities outside the building.

18.1.9.2 Other Onsite Infrastructure Costs

Other onsite infrastructure costs included provisions for civil infrastructure, electrical services, IT and communications, ancillary buildings, fuel storage and mobile equipment. A summary of onsite infrastructure costs is provided in Table 18-13.

Table 18-13: Onsite Infrastructure Costs

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
5100	Site Civil Infrastructure	12.8	0.0	12.8
5200	Water Systems	25.3	0.0	25.3

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
5300	Sewage, Waste and Water Systems	55.8	35.1	90.8
5400	Electrical Services	47.7	0.0	47.7
5500	IT & Communications	0.1	0.0	0.1
5600	Ancillary Buildings	12.4	0.0	12.4
5700	Diesel Storage and Distribution	2.1	0.0	2.1
5800	Plant Mobile Equipment	7.6	0.0	7.6
5900	Onsite Infrastructure General	9.1	0.0	9.1
Sub-total Onsite Infrastructure Costs		**75.8**	**172.8**	**35.1**

18.1.10 Offsite Infrastructure (WBS 6000)

18.1.10.1 Main Access Road

The total estimate for the main site access road is $20M. The access road includes approximately 8 km of road between the Bornite Camp and the Arctic intersection. The estimate included the access road design and construction package, stream crossings and drainage structures, road surfacing, project delivery costs, and contingency.

18.1.10.2 Other Offsite Infrastructure Costs

Other offsite infrastructure costs included provisions for an airstrip, logistics compound and PAF. A summary of offsite infrastructure costs is provided in Table 18-14.

Table 18-14: Offsite Infrastructure Costs

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
7100	Site Access Road	20.0	0.0	20.0
7200	Airstrip	4.6	0.0	4.6
7300	Logistics Compound	2.1	0.0	2.1
7400	PAF	48.6	0.0	48.6
7900	General Offsite Infrastructure	0.4	0.0	0.4
Sub-total Offsite Infrastructure Costs		**75.8**	**0.0**	**75.8**

18.1.11 Indirects (WBS 7000)

Indirect costs were estimated to include:

- Project preliminaries (field indirects) such as temporary construction facilities, temporary utilities, construction support, construction equipment, material storage and site office costs
- Construction camp and catering costs
- EPCM costs to cover such items as engineering and procurement services, construction management services, project office facilities, IT, staff transfer expenses, secondary consultants, field inspection and expediting, corporate overhead and fees
- Spares (operational, capital, commissioning) and first fills
- Costs for vendor representatives
- Contractor support for commissioning activities to make minor modifications or provide labour assistance during commissioning

Table 18-15 summarizes the indirect capital costs for the Arctic Project.

Table 18-15: Indirect Capital Costs

WBS Level 1	WBS Level 1 Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
7100	Project Preliminaries	18.7	0.0	18.7
7200	Temporary Facilities	4.3	0.0	4.3
7300	Temporary Services	11.8	0.8	12.6
7400	Camps	39.4	0.0	39.4
7500	Cranage	2.6	0.0	2.6
7600	EPCM Costs	89.8	14.3	104.1
7700	Commissioning Support	1.1	0.0	1.1
7800	First fills/spares	7.5	0.0	7.5
7900	Vendors	2.1	0.0	2.1
Sub-total Indirect Costs		**177.4**	**15.1**	**192.5**

18.1.12 Provisions / Contingency (WBS 8000)

Provisions are included in the cost estimate to account for growth allowances based on the level of definition of the study and estimate contingency. Estimate contingency is included to address anticipated variances between the specific items contained in the estimate and the final actual project cost.

The estimate contingency does not allow for the following:

- Abnormal weather conditions
- Changes to market conditions affecting the cost of labour or materials
- Changes of scope withing the general production and operating parameters
- Effects of industrial disputes

A summary of the estimate contingency is provided in Table 18-16, broken down by scope owner, totaling $138.5 M of initial capital costs, or ~12% of the total Arctic Project initial capital cost.

Table 18-16: Estimate Contingency

WBS Area	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
Crushing, Process Plant, Off-site Infrastructure	68.7	13.0	81.7
Mining	39.6	0.0	39.6
Tailings Management Facility	12.4	0.0	12.4
On-site Infrastructure	16.3	0.0	16.3
Owner's Cost	1.6	0.0	1.6
Sub-total Contingency	**138.5**	**13.0**	**151.5**

18.1.13 Owner's Costs (WBS 9000)

A breakdown of owner's costs is shown in Table 18-17.

Table 18-17: Owner's Costs

Description	Initial Capex ($ M)	Sustaining Capex ($ M)	Total Capex ($ M)
Owner's Project Management	11.3	0.0	11.3
Health & Safety	3.3	0.0	3.3
Start-up	12.2	0.0	12.2
Sub-total Owner's Costs	**26.8**	**0.0**	**26.8**

18.1.14 Sustaining Capital and Closure Costs

Sustaining capital costs include expenditure related to additions to the truck mining fleet, to replace equipment beyond its manufacturer's recommended service life, processing plant equipment, onsite infrastructure (largely comprising the cost of the Reverse Osmosis Reject WTP towards the end of the mine life) and miscellaneous indirect costs. In addition, sustaining costs are carried for the three TMF dam raises. However, costs associated with placement and compaction of the material for dam raises, being achieved through building up of the WRF, are captured as mining operating costs.

Table 18-18 summarizes the sustaining capital and closure costs. The total closure cost of $428 million includes $124.7 million in closure costs, $46.1 million in post-closure costs and $258 million in post-closure water treatment costs. Further details related to the closure cost total are provided in Section 17.3.6.

Table 18-18: Sustaining Capital and Closure Costs

WBS Level 1	WBS Description	Sustaining Capex ($ M)
1000	Mining	17.5
2000	Crushing	0
3000	Process Plant	1.3
4000	Tailings	32.4
5000	On-Site Infrastructure	35.1
6000	Off-Site Infrastructure	0
	Sub-Total Direct Costs	**86.3**
7000	Indirects	15.1
8000	Provisions (Contingency)	13.0
9000	Owners Costs	0
Sub-Total Indirect Costs		**28.1**
Arctic Project Total – Sustaining Capital		**114.4**
Arctic Project Total – Closure Costs		**428.4**

18.2 Operating Cost Estimate

The operating cost estimate is within the ± 25% cost accuracy for a PFS-level study under S-K 1300 standards.

18.2.1 Operating Cost Summary

An average operating cost was estimated for the Arctic Project based on the proposed mining schedule. These costs included, mining, processing, G&A, surface services, and road toll costs. The average LOM operating cost for the Arctic Project is estimated to be $59.83/t milled.

The processing plant throughput is designed to operate at approximately 10,000 t/d, or 3,650,000 t/a. The proposed mining schedule ramps up in Year 1 and ramps down in Year 13 resulting in a LOM average of approximately 3,592,000 t/a processed. Total throughput is estimated to be 46,691,000 t over the 13-year LOM. The breakdown of costs in Table 18-19 is based on the 3,650,000 t/a mill feed rate.

Table 18-19: Overall Operating Cost Estimate

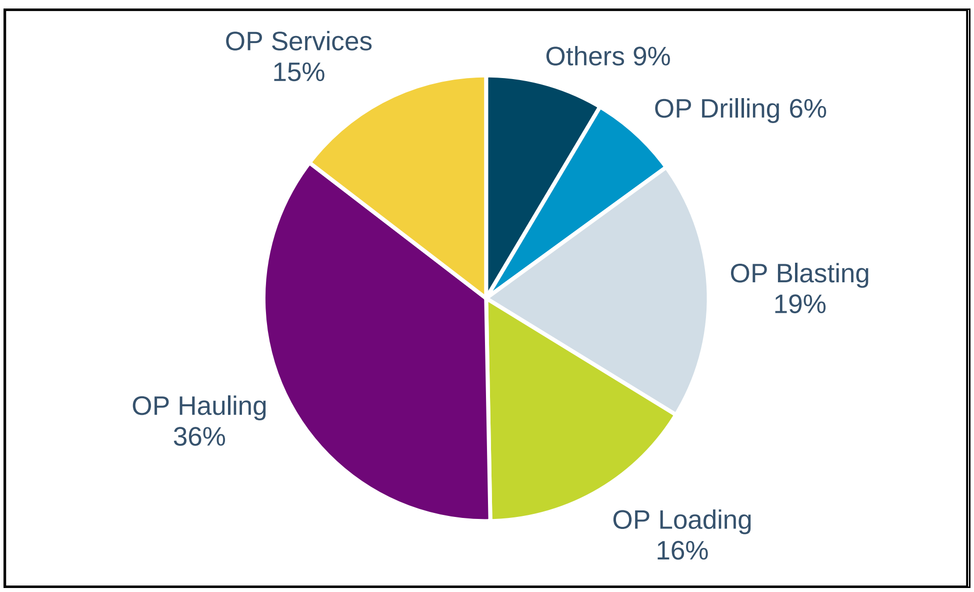
Description	LOM Average Unit Operating Cost ($/t milled)	LOM Average Annual Cost ($ M/a)	Percentage of Total Annual Operating Costs
Mining*	22.49	82.1	37.6%
Processing	22.60	82.5	37.8%
G&A	5.85	21.3	9.8%
Road Toll and Maintenance	7.72	28.2	12.9%
Water Treatment	1.17	4.3	2.0%
Total Operating Cost	**59.83**	**218.4**	**100%**

* Excludes pre-production costs.

18.2.2 Mining Operating Cost Estimate

The mining operating cost averages $2.40/t (moved) during preproduction and $3.24/t (moved) during production. The total tonnage moved includes 0.78 Mt of stockpile rehandle and 386.9 Mt of primary production. Figure 18-2 presents the operating cost distribution, where mine haulage accounts for 36% of the mine operating costs, open pit blasting accounts for 19% of the mine costs, followed by loading, services, and drilling each at 16%, 15%, and 6%, respectively. The Others category in the figure includes costs for engineering, geology, operations, and management overheads. The stockpile rehandling cost is minor due to the small amount of material rehandled along the life of the mine. Table 18-21 shows the operating cost break down by cost centre.

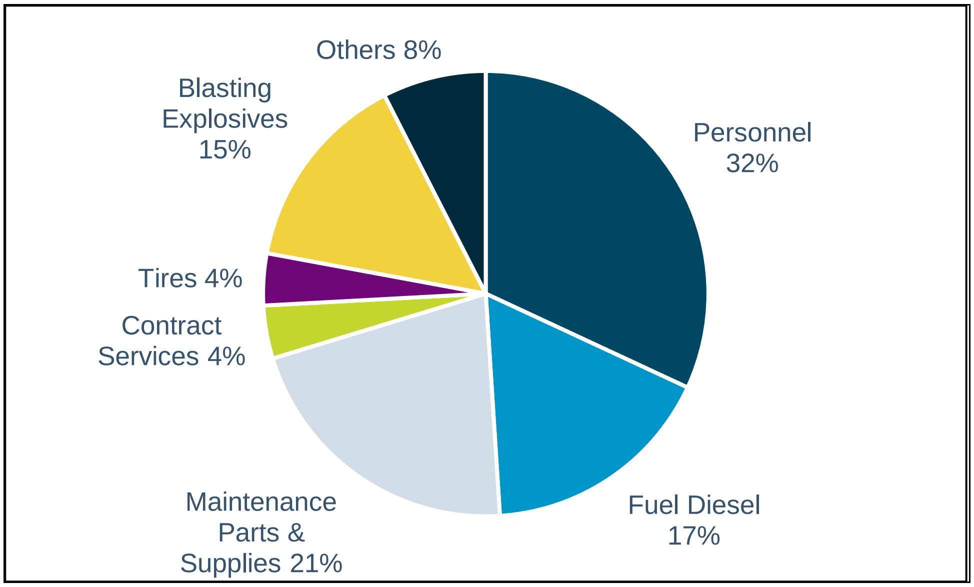
Figure 18-2: Costs by Cost Centre



Source: Wood, 2022.

Personnel costs are the largest cost items for the mine, followed by maintenance and repair supplies, fuel, and blasting explosives. The study diesel price is $1.04/l. The Others category shown in Figure 18-3 includes costs for: drilling supplies, auxiliary equipment, contract services, ground engaging tools, blasting accessories and lubricants.

Figure 18-3: Costs by Cost Item



Source: Wood, 2022.

Figure 18-4 provides an overview of the tonnage mined and the mine operating costs per tonne mined. During Year -2 and Year -1, the mining costs are at the lowest due to the short haul distances and high mining rates. From Year 1 to Year 4, the mining costs increase as the haul cycles increase with the deepening of the Phase 1 and Phase 2 lay backs and the mining of the upper zones of Phase 3. Mining costs then decrease in Year 5 as a new exit of the pit is used, decreasing the haulage distances. An increase in the following years can be noticed as Phase 2 and Phase 3 start deepening. After Year 9, only Phase 4 (final phase) is mined and the cost increases with the deepening of the pit and a reduction in the total tonnage mined.

Table 18-20 summarizes the average mining operating cost over the LOM.

Table 18-20: Mining Operating Cost Summary by Cost Centre

Cost Centre	LOM Average Annual Cost ($ M/a)	LOM Average Unit Operating Cost ($/t milled)
OP Management	0.4	0.12
OP Operations O/H	2.9	0.79
OP Engineering	2.0	0.55
OP Geology	1.5	0.41
OP Drilling	5.2	1.42
OP Blasting	14.9	4.08
OP Loading	13.0	3.56
OP Hauling	30.1	8.25
OP Stockpile Rehandle	0.1	0.02
OP Services	12.0	3.29
Total Operating Cost	**82.1**	**22.49**

Note: Figures may not add up due to rounding. OP – open pit

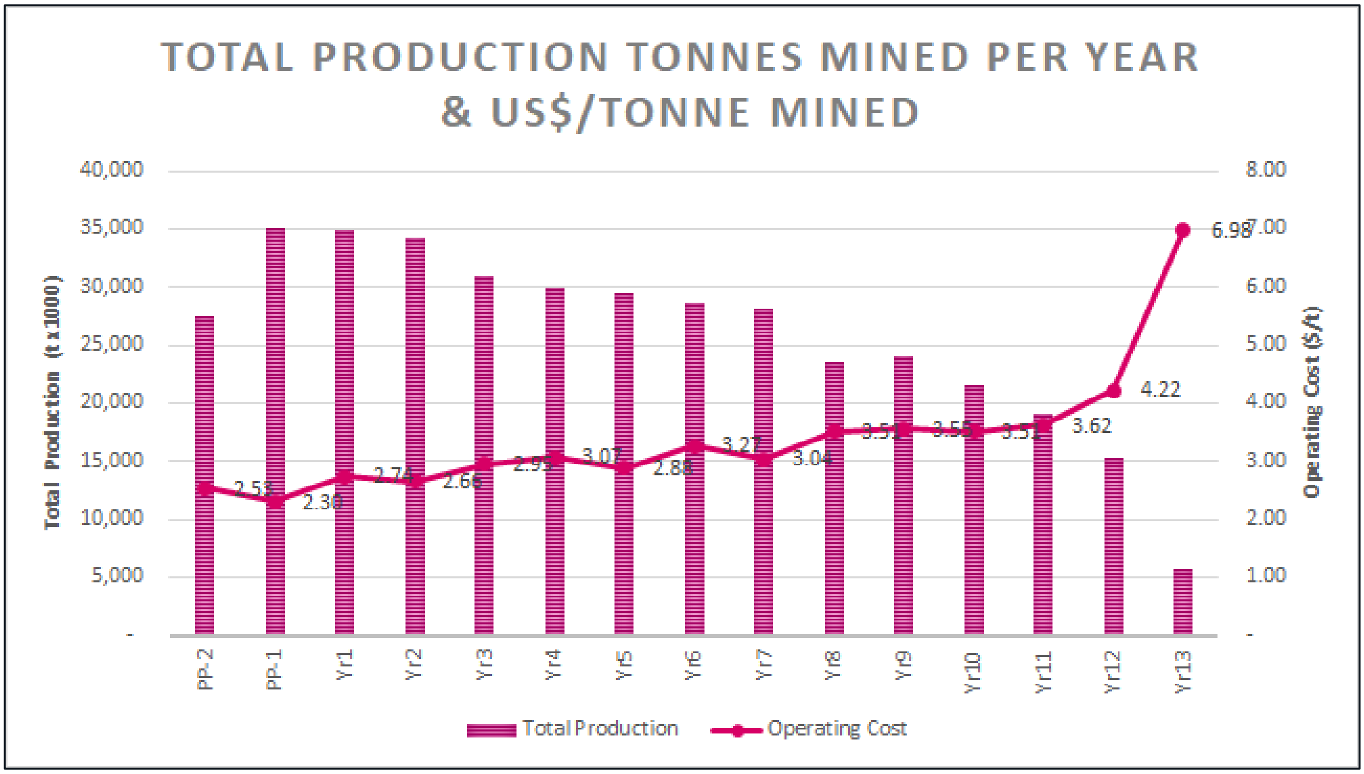
Average annual LOM mining costs are summarized in Table 18-21.

Table 18-21: Operating Cost Summary by Cost Centre

	Preproduction			Production			Total Mining Cost		
Cost Centre	**($M)**	**($/t mined)**	**($/t moved)**	**($M)**	**($/t mined)**	**($/t moved)**	**($M)**	**($/t mined)**	**($/t moved)**
OP Management	0.85	0.01	0.01	5.42	0.02	0.02	6.27	0.02	0.02
OP Operations O/H	5.77	0.09	0.09	37.04	0.11	0.11	42.81	0.11	0.11
OP Engineering	4.43	0.07	0.07	25.79	0.08	0.08	30.22	0.08	0.08
OP Geology	3.27	0.05	0.05	19.22	0.06	0.06	22.49	0.06	0.06
OP Drilling	11.15	0.18	0.18	66.21	0.20	0.20	77.36	0.20	0.20
OP Blasting	34.82	0.56	0.56	190.32	0.59	0.59	225.14	0.58	0.58
OP Loading	25.07	0.40	0.40	166.22	0.51	0.51	191.29	0.49	0.49
OP Hauling	43.14	0.69	0.69	385.42	1.19	1.19	428.56	1.11	1.11
OP Stockpile Rehandle	-	-	-	0.74	0.00	0.00	0.74	0.00	0.00
OP Services	21.40	0.34	0.34	153.62	0.47	0.47	175.02	0.45	0.45
Operating Cost	**149.9**	**2.40**	**2.40**	**1,050.0**	**3.24**	**3.23**	**1,199.90**	**3.10**	**3.10**
Tonnes Mined	62.4 Mt			324.4 Mt			386.9 Mt		
Tonnes Rehandled	-			0.8 Mt			0.8 Mt		
Tonnes Total	62.4 Mt			325.2 Mt			387.7 Mt		

Note: Figures may not sum due to rounding. OP – open pit.

Figure 18-4: Total Material Mined and Mine Operating Cost



Source: Wood, 2022.

Table 18-22 provides a summary of the annual mining costs and unit mining costs for both primary production and total material mined.

Table 18-22: Annual Mine Expenditure

Period	Mining Cost ($ M)	Primary Production		Total Material Moved*	
		(kt)	($/t)	(kt)	($/t)
Year -2	0.07	27,400	2.53	27,400	2.53
Year -1	0.08	35,044	2.30	35,044	2.30
Year 1	0.10	34,077	2.80	34,853	2.74
Year 2	0.09	34,190	2.66	34,190	2.66
Year 3	0.09	31,015	2.95	31,015	2.95
Year 4	0.09	30,000	3.07	30,000	3.07
Year 5	0.09	29,500	2.88	29,500	2.88
Year 6	0.09	28,563	3.27	28,563	3.27
Year 7	0.09	28,063	3.04	28,063	3.04

Period	Mining Cost ($ M)	Primary Production		Total Material Moved*	
		(kt)	($/t)	(kt)	($/t)
Year 8	0.08	23,500	3.51	23,500	3.51
Year 9	0.09	24,000	3.55	24,000	3.55
Year 10	0.07	21,572	3.51	21,572	3.51
Year 11	0.07	19,000	3.62	19,000	3.62
Year 12	0.06	15,280	4.22	15,280	4.22
Year 13	0.04	5,655	6.98	5,655	6.98
Total	**1.20**	**386,859**	**3.10**	**387,635**	**3.10**

Note: Total Material Moved is inclusive of rehandle

18.2.2.1 Mining Operating Cost Basis of Estimate

The basis for the study unit costs and the consumption estimates are detailed in the following sub-sections.

18.2.2.2 Personnel Costs

Salaries were based on Wood's database for similar projects, and benchmarking with publicly available salary ranges for mining job positions in Alaska. For hourly labour, Wood relied on local labour costs provided by Trilogy detailing expected hourly and salaried annual rates. Labour rates are inclusive of a 38% burden. Wood applied the labour rates to the staffing plan to estimate total labour costs.

18.2.2.3 Maintenance and Repair Cost

Equipment suppliers provided equipment maintenance and repair cost estimates in quotations obtained in 2022. Table 18-23 provides a listing of the average maintenance and repair costs for each major machine-type. Within the cost model a variable maintenance and repair cost was assigned every 6,000-hour increments.

The maintenance and repair costs provided by the equipment suppliers assume owner performed maintenance and include costs for:

- Planned maintenance, i.e., routine service and lubrication.
- Major repair and rebuilds including parts and rebuild bench labour.
- Routine minor repairs.

Table 18-23: Maintenance and Repair Cost

Equipment	Cost ($/h)
171 mm Production Drill	106
265 t/15 m^3 Hydraulic Face Shovel	125
124 t/12 m^3 Front End Loader	66
144 t Haul Truck	81
41t Articulated Truck	21
26 t/5 m^3 Front End Loader	36
68 t/4 m^3 Hydraulic Excavator	29
35 t/1.4 m^3 Hydraulic Excavator	21
74 t/455 kW Track Dozer	72
50 t/419 kW Rubber Tired Dozer	63
40 t Articulated Sand Truck	21
33 t/217 kW Motor Grader	31
35,000-litre water truck	21
40t Articulated Fuel/Lube Truck	21

18.2.2.4 Diesel and Fuel Consumption

Wood relied on a diesel fuel price of $1.04/L ($3.92/gal) provided by Ausenco - based on the 3-year trailing average weekly U.S. No 2 diesel ultra low sulphur retail price, adjusted for delivery to the project site. Diesel consumption for the haul trucks is estimated based on duty cycles. For all other machines, Wood used either the diesel consumption rates provided by equipment suppliers, or a consumption rate from the CAT Handbook Version 49 (Caterpillar, 2019). Consumption per operated hour for each machine is provided in Table 18-24.

Table 18-25 provides the annual diesel fuel requirements. Diesel fuel consumption peaks in Year 7 at 14.6 ML consumed.

Table 18-24: Equipment Fuel Burn Rates

Equipment	liters/op hour
171 mm Production Drill	83
265 t/15 m^3 Hydraulic Face Shovel	172
124 t/12 m^3 Front End Loader	90
144 t Haul Truck	76
41t Articulated Truck	22

Equipment	liters/op hour
26 t/5 m³ Front End Loader	21
68 t/4 m³ Hydraulic Excavator	40
35 t/1.4 m³ Hydraulic Excavator	18
74 t/455 kW Track Dozer	63
50 t/419 kW Rubber Tired Dozer	45
40 t Articulated Sand Truck	35
33 t/217 kW Motor Grader	30
35,000-litre water truck	35
40t Articulated Fuel/Lube Truck	35

Table 18-25: Annual Diesel Consumption

Period	Hauling (ML)	Loading (ML)	Support (ML)	Drilling (ML)	Blasting (ML)	Total (ML)
Year -2	4.6	3.7	2.3	1.1	0.2	11.9
Year -1	5.8	3.7	2.3	1.3	0.3	13.4
Year 1	6.3	3.6	2.3	1.3	0.3	13.8
Year 2	6.7	3.6	2.3	1.3	0.3	14.2
Year 3	6.4	3.6	2.3	1.2	0.3	13.7
Year 4	7.0	3.5	2.3	1.2	0.3	14.3
Year 5	7.3	3.5	2.3	1.1	0.3	14.5
Year 6	7.2	3.5	2.3	1.1	0.3	14.3
Year 7	7.5	3.5	2.3	1.1	0.2	14.6
Year 8	7.2	2.9	2.0	1.0	0.2	13.3
Year 9	7.9	2.9	2.0	1.0	0.2	14.1
Year 10	7.6	2.9	2.0	1.0	0.2	13.7
Year 11	7.9	2.9	2.0	0.8	0.2	13.8
Year 12	6.9	3.0	1.6	0.6	0.1	12.2
Year 13	2.3	1.9	1.6	0.2	0.1	6.0
Total	**98.5**	**48.7**	**31.8**	**15.2**	**3.4**	**197.6**

18.2.2.5 Explosives Cost

A multinational explosives service company provided explosives product costs as part of the RFQ process. Costs for explosive products and the monthly service cost is shown in Table 18-26. Operating fee includes $228k/month for personnel and $2.5k/month for forklift equipment costs rental. Other blasting equipment such as mobile mixing units (MMU) and stemming trucks were incorporated as part of the mine auxiliary equipment. Transfer site cost was included as part of the mine capital cost.

Table 18-26: Blasting Explosives and Accessories Cost and Service Fee

Item	Unit	Value
Blasting Product		
ANFO (Ammonium Nitrate – Fuel Oil)	$/t	1,282
Bulk Emulsion	$/t	2,042
Blasting Accessories		
Non-electric detonator	$/ea	14.00
Surface delays	$/ea	8.50
Booster 450 gram	$/ea	8.45
Booster 900 gram	$/ea	16.14
Monthly Service Fee		
Operating Fee	$/month	230,625

18.2.2.6 Tires Cost

Tire pricing was provided by equipment vendors as part of the RFQ process. Tire pricing is shown in Table 18-27along with tire lives used for estimating tire costs during operations.

Table 18-27: Tire Sizes, Pricing, and Life

Machine	Tires/machine	Size	Price each ($)	Life (operating h)
124 t/12 m^3 Front End Loader	4.00	45/65R45	43,750	5,000
144 t Haul Truck	6.00	33.00R51	20,214	5,000
41t Articulated Truck	6.00	29.5R25	5,560	5,000
26 t/5 m^3 Front End Loader	4.00	26.5R25	18,559	5,000
50 t/419 kW Rubber Tired Dozer	4.00	35/56-33	12,787	4,000
40 t Articulated Sand Truck	6.00	29.5R25	5,560	4,500
33 t/217 kW Motor Grader	6.00	23.5R25	3,495	4,000
35,000-litre water truck	6.00	29.5R25	5,560	4,500
40t Articulated Fuel/Lube Truck	6.00	29.5R25	5,560	4,500

During the preproduction period, the 144-t truck tire lives were derated to 3,000 hours in Year -2 and 4,000 hours in Year -1 to account for poor road, pit, and dump conditions anticipated in the early years of the operation.

18.2.2.7 Other Operating Costs

Other operating costs account for 7.57% of the total operating costs. The other cost items are listed in Table 18-28.

Table 18-28: Other Operating Costs

Cost Item	Percentage of Total (%)	LOM Cost ($ M)	$/t Mined
Drilling Supplies	1.14%	14.13	0.04
Auxiliary Equipment	2.65%	32.8	0.08
Ground Engaging Tools	1.26%	15.8	0.04
Equipment, Supplies, Materials	0.03%	0.39	0.00
Lubricants	2.33%	28.86	0.07
Other Miscellaneous Costs	0.16%	1.97	0.01
Total Cost	**7.57%**	**93.74**	**0.24**

18.2.3 Processing Operating Cost Estimate

The LOM average process operating cost is $22.60/t milled. This estimated unit cost was based on the designed 10,000 t/d throughput rate. Table 18-29summarizes the processing operating cost estimates. Labour costs based on a detailed staffing plan and local rates.

Operating costs include all regular, recurring costs of production, such as:

- Processing
- Power consumption
- Operating Consumables
- General Maintenance and
- General and administration (G&A)

The operating cost estimates were calculated based on the following assumptions:

- Plant capacity per day of 10,000 t/d over LOM
- Crushing and mill plant availabilities at 65% and 92%, respectively.
- Cost estimates were based on the third quarter (Q3) of 2022 pricing without allowances for inflation.
- Relevant exchange rates were used to convert to the base currency of US dollars ($)
- A diesel cost of $3.92/gal was based on the 3-year trailing average weekly U.S. No 2 diesel ultra low sulphur retail price, adjusted for delivery to the project site.

- The power consumption cost was derived from the diesel cost at $0.289/kWh.
- Labour will mainly be sourced from Alaska.
- Operating and maintenance consumables were provided by different vendors such as Univar, Molycop, Metso-Outotec, etc.
- Processing costs were determined based on labour, light vehicles and mobile equipment, operating and maintenance consumables and processing power requirements.
- Road maintenance and road toll costs were provided by Trilogy.

Table 18-29: Summary of Processing Operating Cost Estimates

Description	Average Annual LOM Operating Cost ($ M)	Average Annual LOM Operating Cost ($/t milled)
Plant Operations Labour	10.4	2.86
Plant Maintenance Labour	11.2	3.06
Power Supply (Mill and Tailings)	37.4	10.24
Processing Consumables	18.8	5.16
Maintenance Consumables	4.3	1.18
Light Vehicles & Mobile Equipment	0.4	0.10
Total (Processing)	**82.5**	**22.60**

18.2.3.1 Power

Power costs were calculated from an estimate of annual power consumption and using a unit cost of $0.289/kWh.

The processing power draw was based on the average power utilisation of each motor on the electrical load list for the process plant and services. Diesel generators along with the fuel will be transported to the site to service the facilities at the site.

Annual energy consumption is estimated at 129,546 MWh/a, costing an annual average of $25.2 M/a over LOM.

18.2.3.2 Consumables

Processing reagent and consumable costs were estimated based on the throughput. All costs include freight charges to site.

The operating consumables costs were developed with the following basis:

- Liner consumptions for the jaw crusher, SAG mill, Ball mill and regrind mill were determined based on the comminution and breakage data and Ausenco's calculations and in-house database.
- Grinding media consumption were based on the internal Ausenco calculations.
- Reagent consumption was estimated from metallurgical testwork.

- Filter cloth consumption were established by benchmarking different Ausenco projects.

All costs associated with operating consumables not stated above were determined using quotations and commercial proposals provided by vendors

18.2.3.3 Maintenance Consumables

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using a weighted average factor from 3% to 4%. The factor was applied to mechanical equipment, platework and piping. The total maintenance consumables operating costs is $1.18/t of feed.

18.2.3.4 Labour

Labour includes all processing and maintenance labour costs.

Processing production labour was provided by Trilogy and includes operation departments such as metallurgy, mill operations, maintenance, and the assay lab.

Each position was defined and classified as Salary and Wages. Costs included taxes and benefits. The annual cost is $10.3 M/a for process operations labour and $11.0 M/a for process maintenance labour. The estimated labour force for plant operations and plant maintenance was estimated at 69 and 74 people respectively. The estimate was based on providing a labour force to support continuous operations at 24 hr/d, 365 d/a.

Annual maintenance supplies costs were estimated as a percentage of major capital equipment costs plus an allowance for freight charges.

18.2.4 General and Administrative and Surface Services Cost Estimates

The LOM average G&A costs were estimated to be $5.85/t milled. These estimated costs were developed by Trilogy and Ausenco and include:

- Labour cost for the 57 administrative staff (27 hourly, and 30 salaried), with approximately 42 of these administrative staff onsite. These numbers included the 16 personnel (15 hourly and 1 salaried) surface operations crew.
- Service cost for safety, training, medical and first aid expenditure, computer supplies and software, human resources services, and entertainment/membership.
- Asset operations costs including operating vehicles, and warehouse costs.
- Contract services expenditures, including insurance, consulting, relocation expenses, recruitment, auditing and legal services.
- Camp costs and personnel transport.
- Operation and maintenance of the airport.
- Asset operations including heating, site services for general maintenance, general road maintenance, and ground transportation.
- Other expenses including road dust suppression.

- Other costs, including liaisons to local communities, sustainability costs, and an allowance for regional taxes and licenses.

Table 18-30 shows a summary of the G&A cost estimate.

Table 18-30: G&A Cost Estimates

Cost Description	Annual Costs ($ M)
Labour Costs – Salaried Staff	4.67
Labour Costs – Hourly Staff	3.92
Airport Operation	0.15
General Office Expense	0.11
Medical and First Aid	0.11
Environment	0.16
Travel	0.11
Training and Safety	0.16
Computer Supplies Including Software	0.11
Entertainment/Membership	0.06
Vehicles	0.31
Warehouse	0.22
Communications	0.44
Insurance	2.20
Consulting/External Assays	0.06
Relocation Expense	0.02
Recruitment	0.02
Audit	0.11
Legal Services	0.06
Rotational Travel and Camp	7.19
Surface Operations G&A	0.58
Surface Equipment	0.38
Liaison Committee/Sustainability	0.11
Other	0.11
Total (G&A)	**21.3**

18.2.5 Road Toll Cost Estimate

Trilogy will pay road toll costs to use the AAP that is proposed to be built by AIDEA. AIDEA anticipates that there will be multiple users of the access road for multiple purposes with significantly different levels of use. These variables make it difficult to accurately project, at this time, what the estimated road toll will be for the Arctic Project. Trilogy provided Ausenco with the company's view of the likely costs.

There is currently no road access to the project area. Access to the project area is proposed to be via the AAP road, which is approximately 340 km (211 miles) long, extending west from the Dalton Highway where it would connect with the proposed project area.

The working assumption for the 2020 FS report was that AIDEA would arrange financing in the form of a public-private partnership to arrange for the construction and maintenance of the access road. AIDEA would charge a toll to multiple mining and industrial users (including the Arctic Project) to pay back the costs of financing the AAP. This model is very similar to what AIDEA undertook when the DeLong Mountain Transportation System (also known as the Red Dog Mine Road and Port facilities) was constructed in the 1980s. The amount paid in tolls by any user would be affected by the cost of the road, its financing structure, and the number of mines and other users of the road which could also include commercial transportation of materials and consumer items that would use the AAP to ship concentrates to the Port of Anchorage in Alaska and possibly provide goods and commercial materials to villages in the region. This model has not changed in this Report.

For the purposes of this Report the $22M/a cost of the AAP toll is escalated from the 2020 FS, with an annual maintenance cost of an additional $9.1M/a. The final toll payments will be negotiated with AIDEA and the public-private partnership owners of the access road at some point in the future. A credit of $44M is applied towards road tolls payable in the first 2 years of operation to account for a pre-existing $35M investment, with an 8.0% annual interest rate, from Trilogy to AIDEA during the project development phase.

This equates to an average road toll and maintenance cost of $28.2 M annually over the LOM, or $7.72/t milled over the LOM.

18.2.6 Water Treatment Costs

Water treatment costs during the production period average $1.17/t milled over the LOM, or $4.2 M annually. These costs include Operation (labour, reagents, power) and maintenance costs for the RO WTP during plant operation. Reagents and chemical prices used are developed on a constant Q4 2022 basis, delivered to the project site. Operating costs for post-closure water treatment are captured under closure costs in Section 17.3.6.

19 ECONOMIC ANALYSIS

19.1 Forward-Looking Information Cautionary Statements

Trilogy is subject to the reporting requirements of the Exchange Act and this filing and other U.S. reporting requirements are governed by Subpart 1300 of Regulation S-K promulgated by the SEC. The results of the economic analyses discussed in this section represent forward-looking statements within the meaning of applicable securities laws relating to Trilogy Metals. These statements by their nature involve substantial risks and uncertainties. Statements involving the foregoing results of economic analysis are forward-looking statements. Without limiting the generality of the foregoing, words such as “may”, “anticipate”, “intend”, “could”, “estimate”, or “continue” or the negative or other comparable terminology are intended to identify forward-looking statements. Should one or more of these risks or uncertainties materialize or should the underlying assumptions prove incorrect, actual outcomes and results could differ materially from those indicated in the forward-looking statements. Information that is forward-looking includes the following:

- Probable Mineral Reserves that have been modified from Indicated Mineral Resource estimates.
- Assumed commodity prices and exchange rates.
- Proposed mine and process production plan.
- Projected mining and process recovery rates.
- Ability to market the three types of concentrate on favourable terms.
- Ability to control the levels of deleterious elements in some of the concentrate batches.
- Sustaining costs and proposed operating costs.
- Assumptions as to closure costs and closure requirements, including WTP requirements.
- Assumptions as to development of the AMDIAP, timeframe of such development and assumed toll charges.
- Assumptions as to ability to permit the project.
- Assumptions about environmental, permitting and social risks.

Additional risks to the forward-looking information include:

- Unrecognized environmental risks.
- Unanticipated reclamation expenses.
- Unexpected variations in quantity of mineralization, grade or recovery rates.
- Geotechnical or hydrogeological considerations during operations 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 generation 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.
- Changes to applicable laws.
- Receipt of all required permits.

19.2 Methodology

- An economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the Arctic Project based on an 8% discount rate.
- Years shown in the economic analysis are provided for conceptual purposes only. Permits still must be obtained in support of operations; and approval to proceed is still required from the Arctic Project's owners.

19.3 Financial Model Parameters

The Arctic Project will consist of a three-year pre-production construction period, followed by 13 years of production. Year -3 represents construction of a pioneer road and early purchases of equipment in preparation for site activities that occur in Year -2 and Year -1. The NPV and IRR were calculated at the beginning of the construction period in Year -3.

The cost and revenue estimates were assembled using real dollars, treating Year -3 as the base year. No escalation was applied to any of the estimates beyond this date.

The long-term consensus metal price assumptions were included in Section 16.2.

The LOM material tonnages and payable metal production used in the cash flow model are included in Table 19-1.

Table 19-1: Mine and Payable Metal Production for the Arctic Mine

Description	Units	Value
Total Tonnes Mined	kt	390,189
Mill Feed	kt	46,691
Concentrate		
Cu Concentrate	kt	2,995
Zn Concentrate	kt	2,228
Pb Concentrate	kt	298
Payable Metal		
Payable Cu	'000'lb	1,932,882
Payable Zn	'000'lb	2,243,771

Description	Units	Value
Payable Pb	'000'lb	334,785
Payable Au	'000'oz	423
Payable Ag	'000'oz	36,047

19.4 Economic Analysis

19.4.1 Pre-Tax Financial Analysis

19.4.1.1 Basis of Pre-Tax Financial Evaluation

The pre-tax financial model incorporated the production schedule and smelter term assumptions to produce annual recovered payable metal, or gross revenue, in each concentrate stream by year. Off-site costs, including the applicable refining and treatment costs, penalties, concentrate transportation charges, and marketing and representation fees, and royalties were then deducted from gross revenue to determine the NSR. Further details of the smelter terms used to calculate the recovered metal value and off-site operating costs can be found in Section 16. Royalties are discussed in Section 3.

The operating cash flow was produced by deducting annual mining, processing, G&A, surface services, and road toll & maintenance charges (Section 18.2) from the NSR.

Initial and sustaining capital (Section 18.1) was deducted from the operating cash flow in the years they occur, to determine the net cash flow before taxes.

Initial capital cost included all estimated expenditures in the construction period, from Year -3 to Year -1 inclusive. First production would occur at the beginning of Year 1. Sustaining capital expenditure includes all capital expenditures purchased after first production, including mine closure and rehabilitation.

Under the NANA Agreement, NANA has the right, following a construction decision, to elect to purchase a 16% to 25% direct interest in the Arctic Project or, alternatively, to receive a 15% Net Proceeds Royalty (refer to Section 3). This financial analysis was carried out on a 100% ownership basis and does not include any future potential impact on the Arctic Project interest if NANA elects to purchase an interest of between 16% and 25% in the Arctic Project under the NANA Agreement or, alternatively, the impact on the Arctic Project interest if the 15% net proceeds royalty becomes applicable. The financial analysis does include the 1.0% NSR to be granted to NANA under the NANA Agreement in exchange for a surface use agreement.

The financial analysis was carried out on a 100% Arctic Project ownership basis, of which Trilogy's share is currently 50%.

19.4.1.2 Pre-Tax Financial Results

A summary of the pre-tax financial results is provided in Table 19-2. The presentation is on a 100% project ownership basis of which Trilogy currently holds a 50% interest.

Table 19-2: Summary of Pre-Tax Financial Results

Description	Unit	LOM Value
Recovered Metal Value		
Copper	$ million	7,055.0
Lead	$ million	334.8
Zinc	$ million	2,580.3
Gold	$ million	697.8
Silver	$ million	757.0
Total Recovered Metal Value	$ million	11,424.9
Off-Site Operating Costs		
Royalties, Refining and Treatment Charges, Penalties, Insurance, Marketing and Representation & Concentrate Transportation	$ million	2,969.1
On-Site Operating Costs		
Mining	$/t milled	22.49
Processing	$/t milled	22.60
G&A	$/t milled	5.85
Water Treatment	$/t milled	1.17
Road Toll	$/t milled	7.72
Total Operating Cost	$/t milled	59.83
Total Operating Cost over LOM	$ million	2,793.6
Capital Expenditure		
Initial Capital	$ million	1,176.8
Sustaining Capital	$ million	114.4
Mine Closure & Reclamation	$ million	428.4
Total Capital Expenditure	$ million	1,719.6
Financial Summary		

Description	Unit	LOM Value
Pre-tax Undiscounted Cash Flow	$ million	3,942.6
Pre-Tax NPV at 8%	$ million	1,500.3
Cash Costs, Net of By-product Credits	$/lb Cu payable	0.72
All-in Cost*, Net of By-product Credits	$/lb Cu payable	1.61
Pre-Tax IRR	%	25.8
Pre-Tax Payback Period	years	2.9

Note: *All-in cost includes all operating and sustaining capital costs

19.4.2 Post-Tax Financial Analysis

The following tax regimes were incorporated in the post-tax analysis as provided by EY: US Federal Income Tax, Alaska State Income Tax (AST), and Alaska Mining License Tax (AMLT). Taxes were calculated based on currently enacted United States and State of Alaska tax laws and regulations, including the US Federal enactment of the Tax Cuts & Jobs Act (TCJA) on December 22, 2017, and the Coronavirus Aid, Relief and Economic Security Act (CARES Act) on March 27, 2020.

The Alaska Production Royalty tax of 3% is not applicable to the Arctic Project as the Arctic Project's claims are all federal mining patented claims.

19.4.2.1 US Federal Tax

For tax years beginning on or after January 01, 2018, the US Federal income tax corporate rate is 21% of taxable income, as opposed to a 35% rate which was applicable to prior tax years. Taxable income is calculated as revenues less allowable costs. In addition to other allowable costs, Alaska State Income Tax, AMLT, tax depreciation and the greater of the cost depletion or percentage depletion can be deducted. Cost depletion is the ratable recovery of cost basis as units are produced and sold, however, as noted, cost depletion is not calculated in the model because the initial cost basis of the mineral property has not been provided. IRC §613(a) governs percentage depletion and provides that the deduction for depletion shall be a statutorily prescribed percentage of the taxpayer's gross income from the mineral property during the taxable year. Such allowance shall not exceed 50% of the taxpayer's taxable income from the property that is mining related. Relevant statutorily prescribed percentages are 15% for gold, silver and copper, and 22% for lead and zinc. As a result of the TCJA, losses incurred for tax years beginning on or after January 01, 2018, are not eligible to be carried back to prior tax years but may be carried forward indefinitely. However, losses generated under the TCJA are only eligible to offset 80% of taxable income in future years.

For the purposes of this Report, as a stand-alone project, it was assumed that the initial adjusted cost base of the depletable and depreciable property was zero and that the initial loss carry-forwards were zero.

19.4.2.2 Alaska State Tax

Alaska State Taxes (AST) are determined on a basis similar to US federal tax. AST is calculated using a graduated rate table times taxable income with 9.4% being the highest applicable rate, where taxable income is calculated on the same basis as US federal tax (except that State tax is not deductible). The Alaskan Alternative Minimum Tax ("AMT") statutes are tied to the federal AMT statutes; therefore, the repeal of federal corporate AMT has effectively repealed Alaskan State AMT for tax years beginning on or after January 1, 2018.

19.4.2.3 Alaska Mining License Tax

The Alaska Mining Licence Tax (AMLT) is an income-based tax imposed on the mining income calculated for AST purposes, before any deduction of AMLT, except the percentage depletion is the lower of 15% of net metal revenues and 50% of net income before depletion. No loss carry-forwards or carry-backs are applied when calculating income subject to AMLT. No AMLT tax is charged for the first 3.5 years following commencement of production. In each year, AMLT can be reduced by up to 50% through the application of "Exploration Incentive Credits" (EICs); however, the credits may not exceed $20m in the aggregate for a mining operation and the credits must be utilized within 15 years. Note the EICs can be utilized against AST as well.

For the purposes of this Report, as a stand-alone project evaluated at the project level, it was assumed that the initial EIC balance is zero even though the Trilogy has a history of exploration at the Arctic Project. It was also assumed that no EICs would be earned over the life of the Arctic Project.

19.4.2.4 Post-Tax Financial Results

At the base case metal prices used for the Report, the total estimated taxes payable on the Arctic Project profits are $922.7 million over the 13-year mine life.

The post-tax financial results are summarized in Table 19-3.

Table 19-3: Summary of Post-Tax Financial Results

Description	Unit	LOM Value
Financial Summary		
Income Tax	$ million	922.7
Post-Tax Undiscounted Cash Flow	$ million	3,019.9
Post-tax NPV at 8%	$ million	1,108.1
Post-Tax IRR	%	22.8
Post-Tax Payback Period	years	3.1

19.5 Cash Flow

The annual production schedule and estimated cash flow forecast for the Arctic Project can be found in Table 19-4. The presentation is on a 100% Arctic Project ownership basis. Trilogy holds 50% interest in Ambler Metals.

Table 19-4: Pre and Post-Tax Arctic Project Production and Cashflow Forecast

Arctic Project	Units	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Mill Feed	**ktonnes**	**46,691**	**-**	**-**	**-**	**3,012**	**3,650**	**3,650**	**3,650**	**3,650**	**3,650**	**3,650**	**3,651**	**3,650**	**3,650**	**3,650**	**3,650**	**3,529**	**-**
Cu	%	2.11	-	-	-	2.59	2.05	1.94	1.88	1.87	1.99	2.37	2.00	2.05	2.33	2.41	1.98	2.09	-
Zn	%	2.90	-	-	-	2.87	2.48	2.61	2.82	2.86	2.51	3.11	3.07	2.51	3.09	3.41	3.15	3.20	-
Pb	%	0.56	-	-	-	0.55	0.50	0.56	0.58	0.53	0.49	0.57	0.61	0.39	0.53	0.64	0.66	0.69	-
Au	%	0.42	-	-	-	0.36	0.38	0.39	0.42	0.33	0.34	0.40	0.44	0.37	0.50	0.56	0.47	0.52	-
Ag	%	31.83	-	-	-	33.07	29.03	28.42	28.03	26.36	28.66	31.75	32.44	28.50	36.80	38.68	34.08	38.39	-
Concentrate																			
Cu Concentrate	**ktonnes**	**2,995**	**-**	**-**	**-**	**232.4**	**227.7**	**223.4**	**213.4**	**210.7**	**221.9**	**258.0**	**219.9**	**226.8**	**252.4**	**265.3**	**221.9**	**221.2**	**-**
Cu Recovery	%	92.1	-	-	-	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	92.1	-
Cu Concentrate Grade	%	30.3	-	-	-	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	30.3	-
Cu Payable	%	96.5	-	-	-	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	96.5	-
Ag Concentrate Grade	g/t	161	-	-	-	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	161.3	-
Ag Payable	%	90.0	-	-	-	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0	-
Zn Concentrate	**ktonnes**	**2,228**	**-**	**-**	**-**	**142.2**	**149.8**	**156.6**	**169.7**	**172.6**	**152.1**	**186.0**	**184.9**	**151.3**	**184.8**	**203.3**	**189.4**	**185.7**	**-**
Zn Recovery	%	88.5	-	-	-	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	88.5	-
Zn Concentrate Grade	%	53.7	-	-	-	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	53.7	-
Zn Payable in Concentrate	%	85.0	-	-	-	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	85.0	-
Pb Concentrate	**ktonnes**	**298**	**-**	**-**	**-**	**19.7**	**21.3**	**23.3**	**23.5**	**21.7**	**20.8**	**24.1**	**24.4**	**17.7**	**22.7**	**26.4**	**26.2**	**26.4**	**-**
Pb Recovery	%	61.3	-	-	-	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	61.3	-
Pb Concentrate Grade	%	53.9	-	-	-	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	53.9	-
Pb Payable	%	95.0	-	-	-	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	95.0	-
Ag Concentrate Grade	g/t	2,424	-	-	-	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	2,424	-
Ag Payable	%	95.0	-	-	-	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	-
Au Concentrate Grade	g/t	14	-	-	-	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	14.2	
Au Payable	%	95.0	-	-	-	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	95.00	-
Payable Metal																			
Payable Cu	('000'lb)	1,932,882	-	-	-	151,175	146,013	142,583	137,387	134,852	141,803	167,411	142,499	143,517	163,112	173,249	144,427	144,855	-
Payable Zn	('000'lb)	2,243,771	-	-	-	143,256	149,381	156,519	170,411	173,700	151,480	188,011	186,782	150,724	186,729	206,931	191,688	188,157	-
Payable Pb	('000'lb)	334,785	-	-	-	21,068	23,396	26,443	27,266	24,590	22,721	26,400	28,209	18,325	24,684	29,618	30,845	31,220	-
Payable Au	('000'oz)	423	-	-	-	22	29	31	34	26	26	30	34	28	38	45	38	40	-
Payable Ag	('000'oz)	36,047	-	-	-	2,362	2,530	2,593	2,541	2,355	2,512	2,755	2,877	2,446	3,179	3,449	3,112	3,336	-
Recovered Metal Value																			
Recovered Metal Value	$ M	11,424.9	-	-	-	824.2	828.9	833.0	833.8	809.4	810.3	961.8	880.4	813.4	964.6	1,046.7	906.2	912.2	-
Off-Site Charges		-																	
Royalties, Insurance, Marketing and Representation Fees, Refining, Treatment, Concentrate Transport, and Penalties	$ M	2,969.1	-	-	-	210.2	213.4	216.9	219.4	217.8	211.4	250.8	232.1	210.8	246.8	266.8	237.3	235.4	-
Capital Costs																			
Mining	$ M	314.2	97.0	111.5	88.2	0.1	-	-	1.5	-	0.8	0.8	0.2	8.9	1.9	0.3	3.0	-	-

Arctic Project	Units	LOM	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
Crushing	$ M	42.5	-	14.2	28.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing	$ M	159.3	-	79.5	78.5	-	-	0.2	0.2	0.2	0.2	0.2	0.2	0.2	-	-	-	-	-
Tailings	$ M	120.7	-	38.4	49.9	7.9	-	2.7	8.2	-	1.2	12.4	-	-	-	-	-	-	-
On-Site Infrastructure	$ M	207.9	19.0	61.7	92.1	-	-	-	-	-	-	-	-	-	17.5	17.5	-	-	-
Off-Site Infrastructure	$ M	75.8	20.1	31.1	24.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirects	$ M	192.5	60.9	62.9	53.5	-	-	-	-	-	-	-	-	-	7.6	7.6	-	-	-
Owner's Cost	$ M	26.8	7.9	9.4	9.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	$ M	428.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	428.4
Contingency	$ M	151.5	23.5	69.1	46.0	-	-	-	-	-	-	-	-	-	6.5	6.5	-	-	-
Total Capital Costs	$ M	1,719.6	234.4	473.2	469.2	8.0	-	2.8	9.9	0.2	2.1	13.4	0.4	9.1	33.5	31.9	3.0	-	428.4
Operating Costs																			
Mining	LOM - $/t mined	22.49	-	-	-	31.71	24.92	25.08	25.20	23.30	25.59	23.36	22.62	23.36	20.75	18.84	17.67	11.19	-
	$ M	1,050.0	-	-	-	95.5	91.0	91.5	92.0	85.1	93.4	85.2	82.6	85.3	75.7	68.8	64.5	39.5	-
Processing	LOM - $/t milled	22.60	-	-	-	23.91	22.33	22.50	22.50	22.50	22.50	22.50	22.50	22.50	22.50	22.50	22.50	22.77	-
	$ M	1,055.2	-	-	-	72.0	81.5	82.1	82.1	82.1	82.1	82.1	82.1	82.1	82.1	82.1	82.1	80.3	-
Water Treatment	LOM - $/t milled	1.17	-	-	-	1.40	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.15	1.19	-
	$ M	54.7	-	-	-	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	4.2	-
G&A	LOM - $/t milled	5.85	-	-	-	6.97	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.75	5.95	-
	$ M	273.0	-	-	-	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	21.0	-
Road Toll	LOM - $/t milled	7.72	-	-	-	3.03	2.50	8.53	8.53	8.53	8.53	8.53	8.53	8.53	8.53	8.53	8.53	8.82	-
	$ M	360.6	-	-	-	9.1	9.1	31.1	31.1	31.1	31.1	31.1	31.1	31.1	31.1	31.1	31.1	31.1	-
Total Operating Costs	LOM - $/t milled	59.83	-	-	-	67.02	56.65	63.02	63.14	61.24	63.52	61.29	60.56	61.30	58.69	56.78	55.61	49.92	-
	$ M	**2,793.6**	**-**	**-**	**-**	**201.8**	**206.8**	**230.0**	**230.5**	**223.5**	**231.9**	**223.7**	**221.1**	**223.7**	**214.2**	**207.3**	203.0	176.1	-
Undiscounted Pre-Tax Cash Flow	**$ M**	**3,942.6**	**(234.4)**	**(473.2)**	**(469.2)**	**404.3**	**408.7**	**383.3**	**374.1**	**367.9**	**364.9**	**473.8**	**426.9**	**369.7**	**470.1**	**540.7**	**462.9**	**500.6**	**(428.4)**
Discounted Pre-Tax Cash Flow at 8% Discount Rate	**$ M**	**1,500.3**	**(217.0)**	**(405.7)**	**(372.4)**	**297.1**	**278.1**	**241.5**	**218.3**	**198.8**	**182.5**	**219.5**	**183.1**	**146.8**	**172.8**	**184.1**	**145.9**	**146.1**	**(115.8)**
Income Tax	**$ M**	**-922.7**	**-**	**-**	**-**	**-10.4**	**-11.3**	**-30.0**	**-61.2**	**-64.6**	**-67.7**	**-106.4**	**-90.5**	**-76.6**	**-112.6**	**-132.3**	**-101.2**	**-57.9**	**-**
Undiscounted Post-Tax Cash Flow	**$ M**	**3,019.9**	**-234.4**	**-473.2**	**-469.2**	**393.8**	**397.3**	**353.3**	**312.9**	**303.3**	**297.2**	**367.5**	**336.5**	**293.0**	**357.5**	**408.4**	**361.7**	**442.7**	**-428.4**
Discounted Post-Tax Cash Flow at 8% Discount Rate	**$ M**	**1,108.1**	**-217.0**	**-405.7**	**-372.4**	**289.5**	**270.4**	**222.6**	**182.6**	**163.9**	**148.6**	**170.2**	**144.3**	**116.4**	**131.5**	**139.0**	**114.0**	**129.2**	**-115.8**

19.6 Sensitivity Analysis

Ausenco investigated the sensitivity of the Arctic Project's pre-tax NPV, and IRR to several project variables. The following variables were elected for this analysis:

- Copper price
- Zinc price
- Lead price
- Gold price
- Silver price
- Capital costs
- On-site operating costs
- Off-site operating costs (royalties, refining and treatment charges, penalties, insurance, marketing, and representation fees, and concentrate transportation).

Each variable was changed in increments of 10% between -20% to +20% while holding all other variables constant. Figure 19-1, Figure 19-2 and Table 19-5 show the results of the pre-tax sensitivity analysis.

The metal grade is not presented in these sensitivity graphs because the impacts of changes to metal grade mirror those of metal prices.

Table 19-5: Pre-Tax Sensitivity Summary

Pre-Tax Sensitivity Tables

Pre-Tax NPV Sensitivity To On-Site Opex

On-Site Opex	Commodity Price					
$1,500	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$693	$1,233	$1,773	$2,314	$2,854
(10.0%)	$556	$1,097	$1,637	$2,177	$2,717
--	$420	$960	$1,500	$2,041	$2,581
10.0%	$283	$824	$1,364	$1,904	$2,444
20.0%	$147	$687	$1,227	$1,767	$2,308

Pre-Tax IRR Sensitivity To On-Site Opex

On-Site Opex	Commodity Price				
26%	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	17.4%	23.2%	28.4%	33.1%	37.5%
(10.0%)	15.8%	21.8%	27.1%	31.9%	36.4%
--	14.1%	20.4%	25.8%	30.7%	35.2%
10.0%	12.2%	18.8%	24.4%	29.5%	34.1%
20.0%	10.3%	17.3%	23.1%	28.2%	32.9%

Pre-Tax NPV Sensitivity To Off-Site Opex

Off-Site Opex	Commodity Price				
$1,500	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$702	$1,243	$1,784	$2,326	$2,867
(10.0%)	$561	$1,102	$1,642	$2,183	$2,724
--	$420	$960	$1,500	$2,041	$2,581
10.0%	$279	$819	$1,358	$1,898	$2,437
20.0%	$138	$677	$1,216	$1,755	$2,294

Pre-Tax IRR Sensitivity To Off-Site Opex

Off-Site Opex	Commodity Price				
26%	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	17.5%	23.3%	28.4%	33.1%	37.5%
(10.0%)	15.8%	21.8%	27.1%	31.9%	36.4%
--	14.1%	20.4%	25.8%	30.7%	35.2%
10.0%	12.2%	18.8%	24.4%	29.4%	34.1%
20.0%	10.2%	17.2%	23.0%	28.2%	32.9%

Pre-Tax NPV Sensitivity To Initial Capex

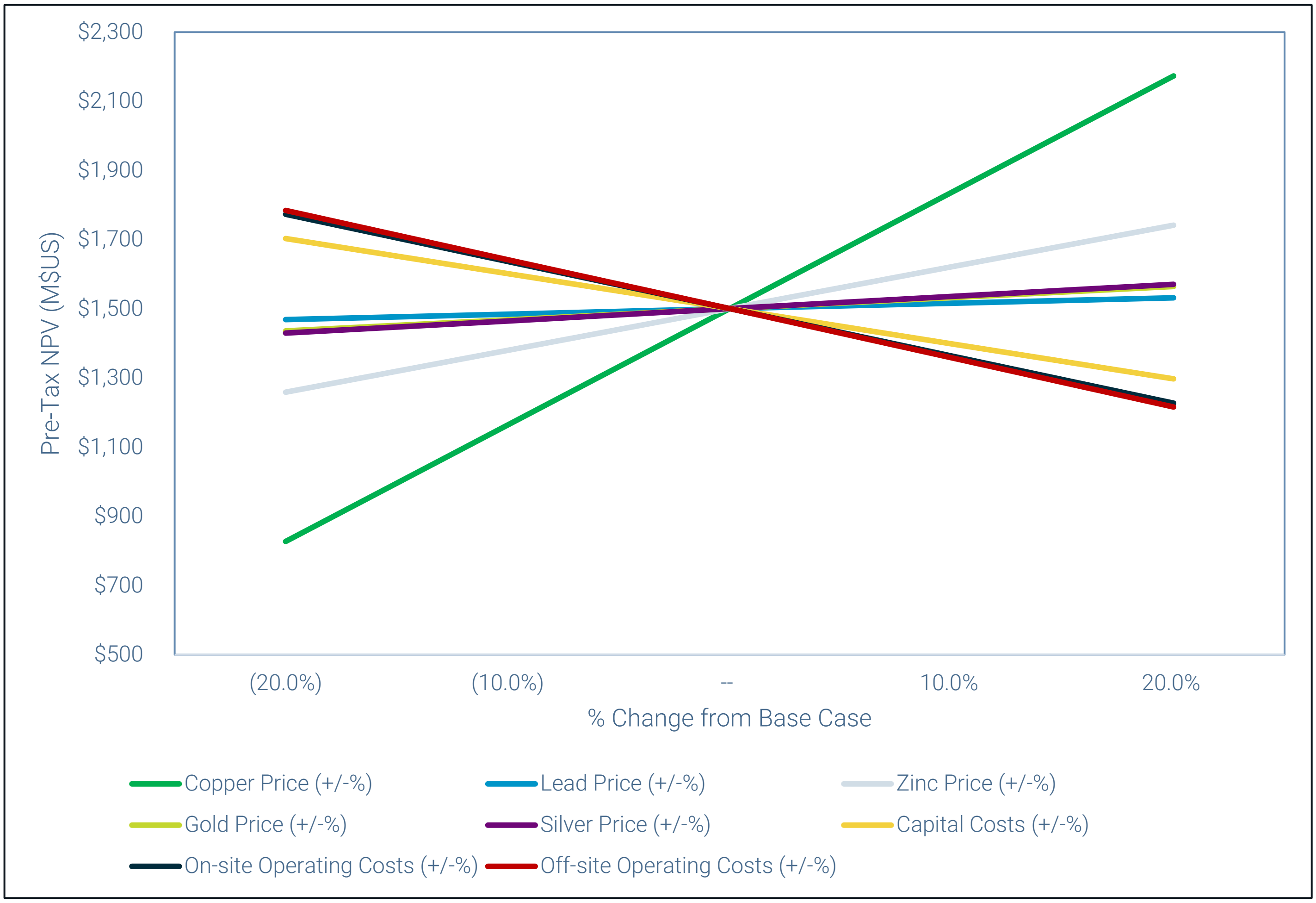
Initial Capex	Commodity Price				
$1,500	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	$623	$1,163	$1,703	$2,243	$2,784
(10.0%)	$521	$1,062	$1,602	$2,142	$2,682
--	$420	$960	$1,500	$2,041	$2,581
10.0%	$318	$859	$1,399	$1,939	$2,479
20.0%	$217	$757	$1,297	$1,838	$2,378

Pre-Tax IRR Sensitivity To Initial Capex

Initial Capex	Commodity Price				
26%	(20.0%)	(10.0%)	--	10.0%	20.0%
(20.0%)	18.5%	25.5%	31.6%	37.2%	42.3%
(10.0%)	16.1%	22.7%	28.5%	33.6%	38.5%
--	14.1%	20.4%	25.8%	30.7%	35.2%
10.0%	12.3%	18.3%	23.5%	28.2%	32.5%
20.0%	10.8%	16.6%	21.5%	26.0%	30.1%

Source: Ausenco, 2022.

Figure 19-1: Pre-Tax NPV Sensitivity Analysis



Source: Ausenco, 2022.

As shown in Figure 19-1, the Arctic Project's NPV at 8% discount rate is most sensitive to changes in copper price, followed by off-site operating costs, on-site operating costs, zinc price, capital costs, silver price, gold price and lead price. On-site operating costs follow closely the off-site operating costs.

Figure 19-2: Pre-Tax IRR Sensitivity Analysis

Source: Ausenco, 2022.

As shown in Figure 19-2 the Arctic Project's IRR is most sensitive to changes in copper price and capital cost, followed by off-site operating costs and on-site operating costs, and in then decreasing order, zinc price, silver price, gold price, and lead price. On-site operating costs follow closely the off-site operating costs.

Table 19-6 shows the results of the post-tax sensitivity analysis.

Table 19-6: Post-Tax Sensitivity Summary

Post-Tax Sensitivity Tables

Post-Tax NPV Sensitivity To On-Site Opex

On-Site Opex	Commodity Price					
	$1,108	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	$481	$897	$1,297	$1,690	$2,082
	(10.0%)	$376	$797	$1,204	$1,598	$1,990
	--	$269	$695	$1,108	$1,506	$1,899
	10.0%	$161	$591	$1,009	$1,413	$1,807
	20.0%	$51	$485	$907	$1,318	$1,714

Post-Tax IRR Sensitivity To On-Site Opex

On-Site Opex	Commodity Price					
	23%	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	15.3%	20.5%	24.9%	29.0%	32.7%
	(10.0%)	13.8%	19.3%	23.9%	28.0%	31.8%
	--	12.3%	18.0%	22.8%	27.1%	30.9%
	10.0%	10.6%	16.6%	21.7%	26.1%	30.0%
	20.0%	8.9%	15.2%	20.5%	25.1%	29.1%

Post-Tax NPV Sensitivity To Off-Site Opex

Off-Site Opex	Commodity Price					
	$1,108	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	$496	$916	$1,318	$1,713	$2,107
	(10.0%)	$383	$805	$1,214	$1,609	$2,003
	--	$269	$695	$1,108	$1,506	$1,899
	10.0%	$155	$583	$999	$1,402	$1,795
	20.0%	$40	$470	$889	$1,297	$1,691

Post-Tax IRR Sensitivity To Off-Site Opex

Off-Site Opex	Commodity Price					
	23%	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	15.4%	20.6%	25.1%	29.1%	32.9%
	(10.0%)	13.9%	19.3%	24.0%	28.1%	31.9%
	--	12.3%	18.0%	22.8%	27.1%	30.9%
	10.0%	10.5%	16.6%	21.6%	26.0%	29.9%
	20.0%	8.7%	15.1%	20.3%	24.9%	28.9%

Post-Tax NPV Sensitivity To Initial Capex

Initial Capex	Commodity Price					
	$1,108	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	$448	$869	$1,278	$1,672	$2,064
	(10.0%)	$359	$782	$1,193	$1,589	$1,981
	--	$269	$695	$1,108	$1,506	$1,899
	10.0%	$179	$607	$1,022	$1,422	$1,816
	20.0%	$88	$518	$936	$1,338	$1,733

Post-Tax IRR Sensitivity To Initial Capex

Initial Capex	Commodity Price					
	23%	(20.0%)	(10.0%)	--	10.0%	20.0%
	(20.0%)	16.4%	22.7%	28.1%	32.9%	37.3%
	(10.0%)	14.2%	20.1%	25.3%	29.7%	33.8%
	--	12.3%	18.0%	22.8%	27.1%	30.9%
	10.0%	10.6%	16.1%	20.8%	24.8%	28.5%
	20.0%	9.2%	14.5%	18.9%	22.8%	26.3%

Source: Ausenco, 2022.

20 ADJACENT PROPERTIES

This section is not relevant to this report.

21 OTHER RELEVANT DATA AND INFORMATION

21.1 Project Execution Plan

Key considerations for the execution of the Arctic Project are as noted below.

21.1.1 Constraints and Interfaces

The Arctic Project will be an integrated development with several consultants contributing to the overall design process. Specialist contractors will most likely be engaged for specific packages, such as the Arctic access road, and the construction camps, generally on a “design and construct” basis.

It is essential that these parties work together to ensure data being used is both current and meaningful. Data transfer between parties shall be strictly controlled and in accordance with Document Control protocols.

The early design interfaces for the Arctic Project will include at least:

- Mine development
- Waste Rock placement and Tails Dam
- Water management and treatment
- Arctic Access Road design and construction, in particular the pioneer road necessary to allow earliest possible access to the Mine pre-assembly construction site
- Bornite, Construction and Permanent Camps.

The Interface Management procedures will be developed to ensure services at the battery limits are clearly defined and understood by all parties affected.

21.1.2 Key Project Milestones

Key project milestones will be developed once the Arctic Project is committed to construction and the required permits are in hand.

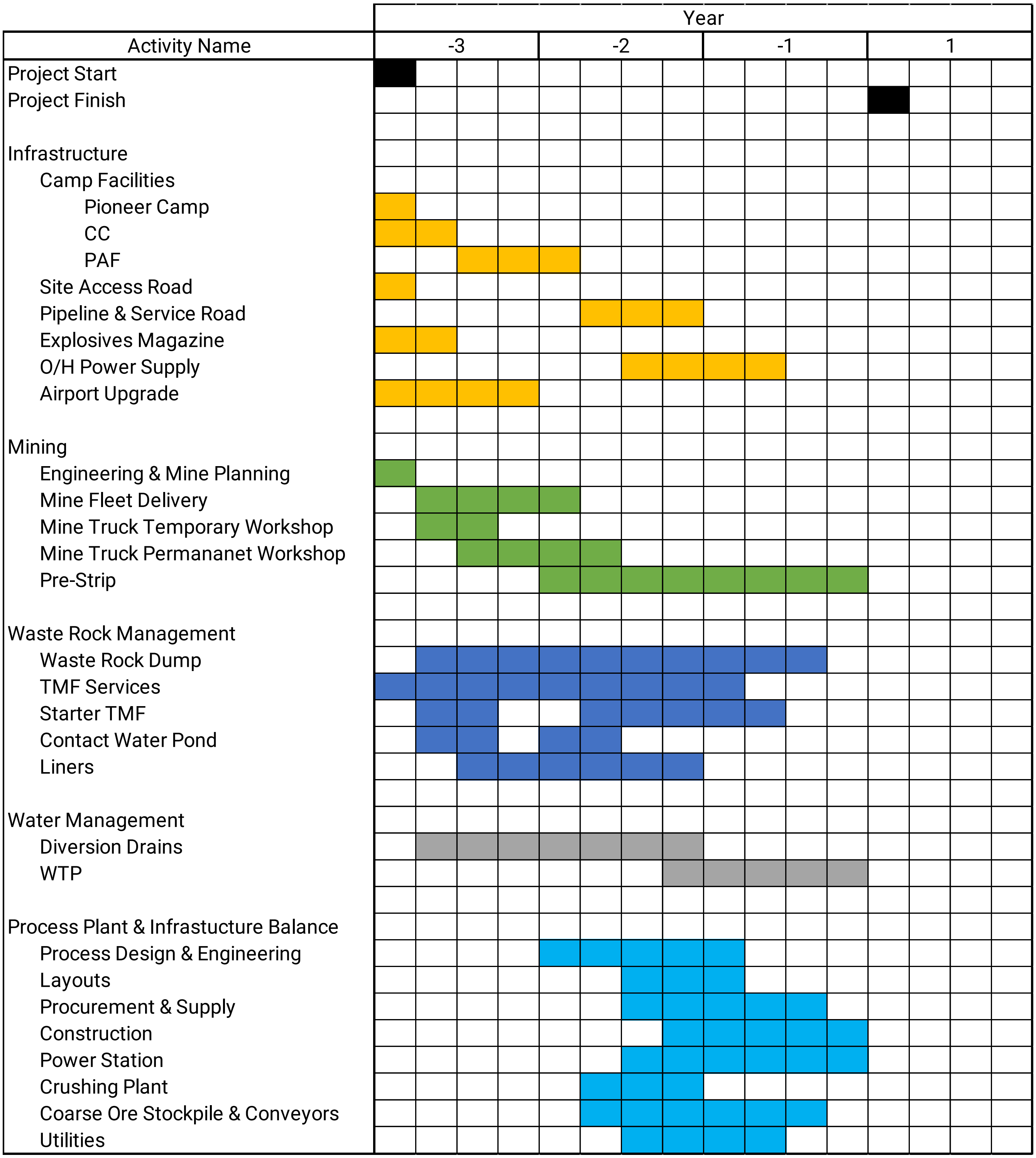
The mine requires nominally two years of pre-strip operations, tailings pond starter dam development and water accumulation before actual production mining operations can commence.

For that pre-strip work to start, the Arctic access road from the AAP intersection to the mine site will have to be constructed to at least a pioneer road condition that will allow the mine fleet and the support facilities to be delivered, built and made operational.

Tailings pond construction must be to a height to allow natural collection of water in quantities that will allow plant operations to commence.

Figure 21-1 shows a project execution schedule that covers key milestones and activities in the 3-year construction period.

Figure 21-1: Project Execution Schedule



Source: Ausenco, 2022.

21.1.3 Proven Technology

The Arctic Project will utilize proven technology and equipment that can be built, operated and maintained under adverse weather conditions.

The Design Criteria, Technical Specifications and Data sheets shall reflect the location, the environmental and initial logistics constraints that may affect the procurement and construction effort.

21.1.4 Engineering, Procurement and Construction Management Approach

Two EPCM strategies have been identified that are structured to account for the abnormally long pre-strip mining operation. The first option is the basis for the capital and operating cost estimate.

21.1.4.1 Early Engineering Only with 2-Stage Procurement

There is a need to establish the mine facilities and assemble the Mine Fleet in time to allow the pre-strip operation to start some two years before the Process Plant receives its first ore. This means that there will be a significant amount of detailed engineering requiring completion well in advance of the time required for conventional engineering, procurement, and construction of just the process plant and supporting infrastructure. This has been assessed as requiring detailed engineering to start some four years before the process plant starts production.

In particular, the pioneer access road design and contracts and civil design for the Mine Support facilities will be required early in the schedule. By default, the rest of the civil design would need to attach to that early works for simple plant layout and construction coordination purposes. For that to occur the plant layout will be required to be established at an early stage. That in turn is dependent on sizing and selection of the major process equipment items and the receipt of vendor data to complete the plant layout.

Effectively, the detailed design phase will need to follow the conventional approach and run its course but started at a time that meets the early works schedule requirements. Everything other than the mine support facilities will be designed some two years in advance of when it is needed.

With the early equipment order placement, the supply phase could become inordinately long, extending over three years in most cases, when in fact the equipment is not likely to be needed until the last eighteen months prior to plant start-up.

An unorthodox but proven option to this extended design, supply and construction schedule is to have the EPCM Contractor procure the major equipment in two steps:

- Step 1: Procure only the vendor certified engineering data to allow detailed engineering to continue to completion but hold the manufacturing functions until later in the overall schedule, effectively a delay of around twelve to fifteen months.
- Step 2: Based on agreed vendor manufacturing durations, apply a “late” release of the equipment for manufacture with deliveries effectively becoming a “Just-in Time” logistics operation.

This strategy provides the following advantages:

- Engineering can start and continue to completion using critical vendor data without the need for an extended “standby” involvement.

- Procurement functions can work in parallel with the engineering group with no disconnect between the two disciplines.
- The procurement team can generally disband early in the schedule with just key personnel retained to provide continuity of support.
- The expediting team can mobilise later in the schedule to drive manufacture and delivery in a concerted campaign.
- Equipment deliveries can be orchestrated to suit the conditions at the time with everything consolidated into a transit compound for coordinated shipping to site.
- Reduced cashflow demands.
- Potential issues to be mitigated with this approach are:
- The vendors need to be clearly briefed as to their manufacturing and delivery schedule.
- A payments formula needs to be in place to account for a delayed delivery strategy.
- Some vendors have difficulty in determining just what their actual engineering costs are.

21.1.4.2 Early EPCM Leading to Plant Care and Maintenance

Under this approach, the EPCM would work to conventional design and construction schedule, starting to suit the Mine access requirements but following on to completion without interruption. That would bring the total process plant and supporting infrastructure to a mechanical completion condition nominally twelve to fifteen months before it is able to start work.

The plant could not be commissioned through lack of ore and would have to be placed into care and maintenance mode until ore became available. This has an inherent advantage in that if the pre-strip operation was completed earlier than scheduled, and sufficient water is accumulated, the plant operations would be able to take advantage of the fact the plant was already mechanically complete. The care and maintenance requirements in that environment for that duration will require close assessment.

22 INTERPRETATIONS AND CONCLUSIONS

22.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report. The level of study was completed to a minimum of a PFS under the standards and definitions of S-K 1300.

22.2 Mineral Tenure, Surface Rights, Royalties and Agreements

Information from legal experts supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves.

Kennecott holds a 1% NSR royalty that is purchasable at any time for a one-time payment of $10 million. This royalty covers the patented federal mining claims and many, but not all, of the State mining claims.

The NANA Agreement between NovaCopper US and NANA, assigned to Ambler Metals in 2020, granted certain rights, for consideration, which include an exclusive right to explore for minerals and a nonexclusive right to enter upon and use certain NANA lands for various purposes including access to NovaCopper's Ambler mining properties. The NANA Agreement further provides that if NovaCopper completes a Feasibility Study and a Draft Environmental Impact Statement for a project to develop the Ambler properties, NANA will become entitled elect to purchase an interest in the Arctic Project by exercising a back-in right to acquire an undivided interest of between 16% and 25% (at NANA's option) of the mining project. If this back-in right were to be exercised by NANA the two parties would form a Joint Venture to proceed with the Arctic Project. The Joint Venture would lease the mining properties from NovaCopper US Inc. in exchange for a 1% NSR royalty and NANA would enter into a surface use agreement with the Joint Venture in exchange for a 1% NSR royalty. In the alternative, should NANA elect not to exercise this back-in right, the Joint Venture would not be formed, and NANA would instead become entitled to a 15% net proceeds royalty. As noted above, this agreement has been assigned to Ambler Metals.

The owner of a State mining claim or lease will be obligated to pay a production royalty to the State of Alaska in the amount of 3% of net income received from minerals produced from the State mining claims. The Arctic Project is not on State lands; thus, the State of Alaska production royalty is not applicable.

22.3 Geology and Mineralization

The Arctic deposit is considered to be an example of a VMS system.

Knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

22.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs conducted is sufficient to support Mineral Resource and Mineral Reserve estimation.

Analytical and density data are suitable to support Mineral Resource and Mineral Reserve estimation.

NovaGold, Trilogy and Ambler Metals sample security procedures met industry standards. Current sample storage procedures and storage areas are consistent with industry standards.

Data collected have been sufficiently verified that they can support Mineral Resource and Mineral Reserve estimation and be used for mine planning purposes.

22.5 Metallurgical Testwork

Metallurgical studies have spanned over 30 years.

Testwork conducted prior to 2012 is considered relevant to the Arctic Project, but predictive metallurgical results are considered to be best estimated from testwork conducted on sample materials obtained from exploration work under the direction of Trilogy conducted in 2012 and 2017 and Ambler Metals conducted in 2021.

To the extent known, the metallurgical samples are representative of the styles and types of mineralization and the mineral deposit as a whole.

Concentrate quality test results indicated that key penalty elements, as well as precious metals are typically concentrated into a lead concentrate, leaving the copper concentrate of higher-than-expected quality given the levels of impurities seen in the test samples. The lead concentrate may have penalties for the high arsenic and antimony concentrations seen in the results of this testwork. Precious metal deportment into a lead concentrate is very high and should benefit the payable levels of precious metals at a smelter. Silicon dioxide and fluoride assays should be conducted on the concentrates to determine whether or not they are higher than the penalty thresholds.

Talc will be managed through a pre-flotation step to minimize reporting to the metal concentrates.

In general, the flowsheet developed in the 2012 test program and further tested in the 2017 and 2021 testwork programs at ALS Metallurgy is feasible for the Arctic deposit mineralization. Further geometallurgical testwork is recommended to confirm and optimize the flowsheet. There are no outstanding metallurgical issues related to the production of a copper or zinc concentrate from all of the materials tested.

22.6 Mineral Resource Estimates

Mineral Resources have been prepared using industry-standard methods and software.

Mineral Resources have had reasonable prospects of economic extraction considerations applied and assume an open pit mining method.

Mineral Resources were prepared in accordance with the standards and definitions of S-K 1300.

Factors that may affect the Mineral Resource estimate include: metal price assumptions; changes to the assumptions used to generate the CuEq cut-off grade that constrains the estimate; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes and volumes; geological and grade continuity assumptions; geological and structural complexity assumptions; density and domain assignments; changes to the predicted SG values; changes to the Mineral Resource estimation parameters for the payable and deleterious metals; changes to geotechnical, mining, and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; assumptions as to concentrate marketability, payability and penalty terms; assumptions as to the continued ability to access the site, retain mineral tenure and obtain surface rights titles, obtain environment and other regulatory permits, and maintain the social license to operate; assumptions as to future site access.

Wood's QP recommends the followings tasks completed for the next model update:

- Review identified issues in the current database and update the database
- Review identified issues in the current geology model wireframes and update using the 2021 and 2022 drilling results.
- Review identified issues in the current talc model wireframes and update using the 2021 and 2022 drilling results.
- Investigate the predicted SG values in the current SG database that were generated by the Random Forest Regressor machine learning method. This is to mitigate the apparent bias identified in the predicted SG values. Use all available measured SG data collected to update the SG model.
- Investigate and update the historical assay values that shows apparent bias.
- Update variogram models for the payable and deleterious metals, and SG used in the current models by using all the available data including the 2021 and 2022 drilling results.
- Review and update the resource estimation parameters for the payable and deleterious metals, and SG.
- Plan for a tight drilling within the first 5-year pit production area to better understand the complexity of the deposit geology and provide higher confidence during the payback period.

22.7 Mineral Reserve Estimates

The 2022 mine plan is based on Probable Mineral Reserves resulting from modifying factors being applied to a subset of the Indicated Mineral Resource estimates.

Mineral Reserves were prepared in accordance with the standards and definitions of S-K 1300.

Risks that may affect the Mineral Reserve estimates include: metal price assumptions; changes to the assumptions used to generate the NSR cut-offs that constrains the estimate; changes in local interpretations of mineralization geometry and continuity of mineralized zones; changes to geological and mineralization shapes, and geological and grade continuity assumptions; density and domain assignments; changes to geotechnical and hydrological assumptions, changes to mining and metallurgical recovery assumptions; changes to the input and design parameter assumptions that pertain to the conceptual pit constraining the estimates; assumptions as to concentrate marketability, payability and penalty terms; assumptions as to the continued ability to access the site, retain mineral tenure and obtain surface rights titles, obtain environment and other regulatory permits, and maintain the social license to operate.

There is a risk to the estimate if the AMDIAP road is not constructed, or in the time frame envisaged, or that the toll charges assumed in this Report are not the final charges levied.

The northeast wall has been forced to fully mine the talc zones, which has pushed the wall, increased the footprint of the pit, and added waste material to mine plan. Any change to the talc shapes can modify the pit design, and therefore, the amount of material mined. Talc horizons that may not have been included in the geological model could affect slope stability and recoveries in the process plant and therefore could be a risk to the Mineral Reserves.

22.8 Mining Recovery

Wood selected conventional open-pit mining because of the deposit's geometry and proximity to surface. The deposit will be mined in four nested phases, including the ultimate pit limit.

An Owner-operated and maintained conventional truck–shovel operation was specified, with outside service providers supporting mine operations.

The planned open pit will operate for 13 years, and an additional two years of pre-production.

Proper management of groundwater will be important to maintaining pit slope stability; the east wall is highly sensitive to several geotechnical parameters, and talc horizons that may not have been included in the geological model might also affect its stability.

22.9 Recovery Methods

Recovery is planned using a conventional crushing, grinding and flotation circuit to produce metal concentrates. A 10,000 t/d throughput rate is envisaged, with an overall plant availability of 92%.

The process plant will produce three concentrates: 1) copper concentrate, 2) zinc concentrate, and 3) lead concentrate. Gold and silver are expected to be payable at a smelter; silver is expected to be payable in the copper and lead concentrates, with gold expected to be payable in the lead concentrate only.

While there are deleterious elements reporting to the concentrates at levels which would incur penalties, such as talc at 2.92% in the feed to the plant, special processing provisions have been included in the flowsheet to make a readily saleable concentrate.

22.10 Infrastructure

The planned mine will be a greenfield site and require construction of camp, mine and process-related infrastructure. Access roads in and around the project site will be required.

Site access assumes that the AAP road will be constructed.

Power generation will be provided by four diesel generators.

Process water will be supplied from the Process Pond, TMF reclaim, and WRCP. TMF dewatering must start towards the end of operations to prevent the TMF from exceeding the maximum storage capacity. Stormwater will be managed within the Arctic Project boundaries by capturing and conveying contact and impacted water for treatment and diverting non-contact water away from developed areas. Well water will be used for potable purposes.

A single WTP, built in stages, will be used. During the operations phase, the WTP will treat effluent from the WRCP, and during Closure phase effluent from the pit. The WTP will initially consist of chemical/physical treatment with RO. In the Operations phase, when the TMF is in use, the RO reject will be sent to the TMF, and only RO permeate will be discharged to the Creek. When the TMF is closed at the end of the Operations phase, a biological/chemical/physical plant will be added to treat the RO reject so that it can be mixed with the RO permeate and discharged.

The TMF will be a thickened tailings facility. The maximum storage capacity of the facility will be approximately 40.7 Mm^3 (tailings and process water) plus an additional 2.5 m of freeboard. The 59 ha footprint of the TMF will be fully lined with a geomembrane liner. Three dam raises will be completed.

22.11 Environmental, Permitting and Social

To date, a moderate amount of baseline environmental data collection has occurred in the area including surface water quality sampling, surface hydrology monitoring, wetlands mapping, stream flow monitoring, aquatic life surveys, avian and mammal habitat surveys, cultural resource surveys, hydrogeology studies, meteorological monitoring, and ML/ARD studies.

The Arctic Project will be subject to a mine permitting process typical for a mine of its size in Alaska. In order to support this process, Trilogy will have to broaden their existing baseline environmental program and complete a number of studies that will support the permit applications.

The Arctic Project is dependent on obtaining an APDES permit to discharge water that meets Alaska Water Quality Standards.

Ambler Metals has formally started engaging the Arctic Project stakeholders and recognizes the need to earn their trust and support by making the Arctic Project directly beneficial to them throughout the Arctic Project life and closure/post-closure periods.

Trilogy will be required to develop a Mine Reclamation and Closure Plan that is protective of the environment during mining operations as well as reclamation and closure.

Closure activities will be undertaken at the end of mine life to bring the mine facilities in a state consistent with the stated closure and post-closure objectives and compliant with the regulations for closure and abandonment.

22.12 Market Studies and Contracts

Metal pricing was provided by Trilogy, guided by long-term price forecasts from analysts as published by CIBC.

The long-term consensus metal price assumptions selected for the economic analysis in this Report were:

- Copper: $3.65/lb
- Zinc: $1.15/lb
- Lead: $1.00/lb
- Gold: $1,650/oz
- Silver: $21.00/oz

A marketing review for the three Arctic concentrates was conducted by StoneHouse Consulting in January 2023 based on the expected concentrate assays as provided by Trilogy.

The three Arctic concentrates – copper, zinc and lead – will need to be marketed differently but will likely be sold within the Asia Pacific area.

The copper concentrate is relatively high grade with low penalty impurities. The quality can be sold directly into China, Japan or Korea, and therefore will have the advantage of the most competitive terms. There is payable silver content, and potential penalties for zinc, lead and antimony will be low. The copper concentrate contains arsenic and selenium levels that are close to the threshold for allowable import into China. Overall, this is a good quality concentrate with few elements for which a penalty can be applied.

The zinc concentrate has a cadmium content that is above the current import limit set by the Chinese government so at this time cannot be sold into China. The obvious markets for this quality would be to smelters in Korea, Japan, Australia and Canada. The concentrate contains high grade zinc with relatively low iron. As the market is currently long in high silica and high manganese concentrates, this quality will be a welcome feed to blend down those impurities. Not all smelters will fully value a high zinc grade material as it is expensive per unit of zinc, so sales will need to be spread among several smelters.

The lead concentrate contains most of the payable gold and silver from the Arctic deposit. Unfortunately, the impurities, in particular fluorine and selenium, will limit the number of smelters interested in this concentrate, regardless of the smelter's capacity to recover precious metals. The best strategy is to sell the concentrate to trading companies who can blend with other lead concentrate containing low silver and selenium. In this way, the silver gets diluted and the selenium gets reduced to an acceptable level. It is recommended that a study be conducted to confirm marketability of this product. Smelters should be consulted so that Trilogy can get some direction as to whether the elevated levels of fluorine and selenium can be managed.

22.13 Capital Costs

Total capital costs are estimated at $1,291 million with initial capital of $1,177 million and sustaining capital costs of $114 million. Closure costs are estimated at an additional $428 million. The estimate accuracy is ±15% and developed at a Q4-2022 US dollars basis with a contingency less than 15%.

22.14 Operating Costs

The average LOM operating cost for the Arctic Project is estimated to be $59.83/t milled.

There is a risk to the capital and operating cost estimates if the toll road is not built in the time frame required for the Arctic Project, the design basis for the road cost estimate changes (for example from a single lane roadway as assumed in this Report to a dual lane), or if the annual toll charges that will be levied are significantly different from what was assumed.

22.15 Economic Analysis

Based on $1,177 million of initial capital costs, sustaining capital costs of $114 million, closure costs of $428 million, $2,969 million in LOM off-site operating costs and $2,794 million in LOM on-site operating costs, pre-tax financial results show an IRR of 25.8% and an NPV of $1,500 million at an 8% discount rate and a 2.9-year payback period. Post-tax financial results show an IRR of 22.8% and an NPV of $1,108 million at an 8% discount rate and a 3.1-year payback period.

22.16 Conclusions

The Arctic deposit will be mined at a maximum annual rate of 35 Mt/a with an overall stripping ratio of 7.3. Ore will be processed by conventional methods to annually produce 234 kt of copper, 23 kt of lead, and 174 kt of zinc, all in concentrates for provision to third party refiners. Waste and tailings materials will be stored in surface facilities, which will be closed and reclaimed at the end of the mine; contact water will be treated and discharged to the environment throughout the LOM. Precious metals attendant with the concentrates will be largely payable. While there are deleterious elements reporting to the concentrates at levels that could incur penalties, special processing provisions have been included in the flowsheet to make a readily saleable concentrate.

In terms of project execution, the mine requires nominally two years of pre-strip operations, tailings pond starter dam development and water accumulation before actual production mining operations can commence.

For that pre-strip work to start, the Arctic access road from the AMDIAP intersection to the mine site will have to be constructed to at least a pioneer road condition that will allow the mine fleet and the support facilities to be delivered, built, and made operational.

Positive financial results support the declaration of Mineral Reserves.

22.17 Risks and Opportunities

22.17.1 Permitting

Mine development permitting will be largely driven by the underlying land ownership; regulatory authorities vary depending on land ownership. The Arctic Project is situated to a large extent on State land, it will be necessary to obtain a Plan of Operation Approval (which includes the Reclamation Plan) from the ADNR. The overall timeline required for permitting would be largely driven by the time required for the NEPA process.

There is a risk to the capital and operating cost estimates, the financial analysis, and the Mineral Reserves if the NEPA process, Plan of Operation Approval and ancillary permits to support operations cannot be obtained in the timeframe envisioned for project start-up.

22.17.2 Mining

Mining through voids during open pit operations is a generally manageable risk where such voids are known to exist. However, unidentified voids may exist, and present a risk to mine and production plans if alternate schedules have to be derived, or new safety measures implemented.

Further data collection and interpretation tasks are recommended to fill in geotechnical data gaps to support future stages of design. A numerical groundwater flow model should be calibrated and developed under transient conditions to inform subsequent geotechnical evaluations and depressurization assumptions.

Grade control and mining near the ore contacts present a risk of potentially mining too much dilution material or losing high-grade material. Performance of grade control methods and mining techniques should be continually evaluated to manage this risk.

The support equipment fleet will be responsible for the usual road, pit and WRF maintenance requirements, but due to the climate conditions expected, will have a larger role in snow removal and water management. This is considered an important, but manageable operating risk to meet production targets.

22.17.3 Geotechnical

The Talc Zone domain represents the weakest geotechnical domain observed at the Arctic deposit. The extent and persistence of the unit is of concern to the pit slope stability. This extent has been further investigated during the 2022 investigation program. The 3D talc model should be updated to include all available drilling and laboratory testing data. Based on the updated geometries, a geotechnical review should be completed to update the slope designs.

Potential slope instability risks have been identified in the interim and final mining phases of the pre-feasibility open pit design. The risks include potential sliding failure along talc domains and fault structures in the east wall of Phase01 pit, localized wedge failures in the south wall of the Phase02, Phase03 and Final pits, and sliding failures along foliation in the final pit design. These risks, discussed in detail in Section 9.7.1.8, should be considered in the next stages of the Arctic Project.

The seismic impact on the slope design should be re-evaluated using the site-specific assessment of the potential seismicity.

Once mining begins, regular pit wall mapping is recommended as the new rock faces are exposed to enable a reconciliation against the geotechnical information used in the slope design. Foliation mapping will be critical for the short-term planning in the NE, E and SE slope sectors. The foliation model should be constantly updated as pit wall mapping is being undertaken and be utilized for making ongoing adjustments to the short-term slope design and mining plan.

The geotechnical evaluations identified that existence of talc layers in the slopes are significant geotechnical risks. Constant delineation of talc will be important to maintain stability of the pit slopes. As additional information, such as drilling and mapping, become available the talc model should be updated and incorporated into the mining plan.

The pore pressure monitoring system and mitigation plan would be critical in managing the slope stability risks during the interim mining phases. The open pit water management plan should include vertical in-pit monitoring piezometers, vertical in-pit pumping wells, and horizontal drain holes. This plan should be reviewed as additional data becomes available and mitigation efforts should be adopted as appropriate to achieve pore pressure reduction needs. This will be particularly important during the earlier phases of the open pit mining when the areas above the talc layers have not been excavated and the compartmentalized water system above the talc layers may exacerbate the geotechnical risks related to the foliation and talc.

Ongoing structural geology mapping should be undertaken so that these can be reconciled against the model used in the slope design to identify any potential risks or opportunities. It is the QP's opinion this structural model should be updated on a regular basis during operation, the risks assessed, and the designs and mining approaches adapted to manage these risks.

22.17.4 Hydrogeology

The hydrogeological conceptual model for the pit and valley areas should be updated over time as more data becomes available and designs are revised. For the pit, the potential effects of seasonal or rapid and high amplitude recharge events (i.e. freshet) on pore pressures and sump sizing need to be considered and opportunities for pit sump locations that could promote drainage from below talc layers should be identified. Where feasible, runoff into the pit should be reduced by drainage ditches around and set back from the pit perimeter, and internal ditching that directs water to pit sumps or out of the pit. Snow removal or clearing off of high elevation portions of the pit may need to be considered to reduce seasonal freshet increases in groundwater recharge and pore pressures.

For the valley bottom groundwater seepage interception system (SIS), data should be collected to characterize overburden and fractured rock properties in the specific area of the system and the conceptual model updated. Estimates of potential groundwater bypass of the WRCP should be updated and the groundwater SIS design updated as appropriate. Baseline monitoring of groundwater quality in this area should be initiated as soon as possible to provide a dataset for environmental permitting and to support performance monitoring of the groundwater SIS once in operation.

It is the QP's opinion the pit Groundwater Management Plan should be updated as mine design advances and implemented at the start of mining. Continued water level monitoring, updated interpretations of seasonal variation and development of baseline for comparison with slope monitoring are needed to assess drainage performance. Provisions should be included in the next level of study for groundwater monitoring and drainage works on the east/northeast slope and to install general pit perimeter groundwater monitoring on the north, south and west sides of the pit.

22.17.5 Tailings Management, Waste Rock and Water Management Facilities

The following are risks and opportunities to be evaluated:

- The increase in tailings volume decreases the volume available for water storage toward the end of operations.
- The current TMF capacity does not account for entrainment of ice in excess of tailings porosity – ice entrainment needs to be minimized through operational practices and/or accounted for in TMF capacity.
- To reduce the duration of tailings consolidation, a drainage system installed above the TMF liner should be evaluated.
- A TMF underdrain system may be needed to convey shallow groundwater below the TMF liner to the WRF rock drain.
- An underdrain system may be needed below the Process Pond to convey natural groundwater seepage into Subarctic Creek.
- A protective layer may be needed to prevent ice damage to the geomembrane liner in the tailings pool area.
- Waste rock placement in the WRF during operation should be optimized to minimize cuts for the western and eastern channels, which provide surface drainage from the TMF after closure.
- The capture and storage of non-acid-generating waste rock from the pit and overburden in the Subarctic Creek valley bottom in the early stages of construction may be optimized to maximize quantities of non-reactive material for closure and other construction needs. In addition, boulders that may need to be removed from liner subgrade and dam abutments for the TMF, WRCP, and Process Pond may be considered for use as rip-rap and/or sources for crushed aggregate.
- Geometries for the WRCP and Process Pond dams may be optimized to reduce earthwork costs. In particular, adjusting the WRCP dam crest so that the alignment is straight would reduce material quantities and complexity.
- Alternative channel linings or profiles may be evaluated to seek reduction in WRCP and Process Pond spillway costs.
- Continuing the East Diversion Pipe along the east bank of the Process Pond may reduce the cross-sectional area of and armouring requirements for the Div-6 diversion channel.

- The overburden stockpiles may be integrated with avalanche mitigation structures.

It is the QP's opinion that further geotechnical and hydrogeological investigations should be completed in the foundations of the TMF, WRF, WRCP, and Process Pond including borehole drilling and laboratory testing to support design objectives, including the following:

- Improve accuracy of overburden thickness and excavation volume estimates.
- Characterize construction materials, including underliner and pond embankment materials.
- Assess liquefaction susceptibility of overburden.
- Characterize the rock mass below the facilities to a depth equivalent to embankment height.
- Evaluate strength properties of pond cut slope materials.
- Assess suspected slope movements near waste or pond facilities.
- Characterize ground water regimes under the waste facilities, ponds, and seepage collection system.
- Characterize geotechnical properties of waste rock, including effect of talc content and physical/chemical degradation.

In addition, further engineering studies on the WRF and TMF design should include:

- Dam Breach Assessment.
- Finite element consolidation and seepage modelling of the TMF to better understand drainage, long-term settlement, and effects on capacity and closure.
- Tailings deposition planning and water balance update to account for additional tailings volume, thickened tailings, plan for pool management and winter deposition, and timing of tailings dewatering towards end of mine life.
- Update WRF, TMF, WRCP, and Process Pond stability analyses (including liquefaction assessment) based on additional field investigation results and lab testing.
- Evaluate whether heat or gas generation during oxidation of waste rock may adversely affect the closure cover.

22.17.6 Hydrology

- Hydrology parameters are based on the rating curve development and resulting discharge timeseries conducted by Brailey Hydrologic Consultants LLC (Brailey 2022). They are assumed to have undergone exhaustive review.
- The climate parameters presented by SRK are measured at discrete stations but are assumed to represent the general catchment area (SRK, 2022).
- Regional stations of assumed reasonable representativeness and gridded reanalysis were used to derive long-term records to calculate statistics such as extreme precipitation because of the limited site-specific record availability.
- SRK used gap-filling methods to fill missing records in the site-specific hydrometric records. Pairing methods are limited, assuming the frequency distribution developed is based on a concurrent record and consistent throughout the long-term record.

It is the QP's opinion that additional baseline studies and environmental permitting activities should be undertaken, including:

- Low flow measurements should be collected at the SRGS, SCGS, and RCDN hydrologic gauging stations to improve low flow and baseflow estimates.
- Snow course surveys should continue to be conducted annually to progress the understanding of freshet and peak flow timing.
- A monitoring gauging station was installed closer to headwaters of Sub-Arctic Creek. Once sufficient data has been collected, predicted Sub-Arctic Creek flows should be calibrated in the water and load balance.

22.17.7 Water and Load Balance

It is the QP's opinion that the water and load balance be updated as the project design is advanced in order to refine the understanding of the effluents to be treated during operations and at closure, and to refine the water treatment design and closure cost estimates.

Other items that require evaluation are:

- Surface run-off from cuts for roads is assumed to be non-contact water, however geochemical characterization of road cuts should be conducted to confirm this.
- Water quality predictions for the construction phase (Year -3 to -1) were not evaluated. Source terms for waste rock and pit wall run-off for the construction phase have not yet been incorporated in the water and load balance.
- There are 17 new source terms that have been developed (including those for the construction phase referenced in the previous point) but have not been incorporated into the model. New and existing source terms do not account for the most recent changes to the mine plan and water management plan, such as an additional year of operations, updated hydrology, and the inclusion of geomembrane covers.
- The new source terms (referenced in the above point) were developed to include annual values that account for differences in the geometry and volume of each facility over the mine life, include rock type specific mine scheduling information, and covering the full timeline over which all rock types will turn acidic. As above, these do not incorporate the most recent changes to the mine plan and have not been incorporated into the model.
- Based on kinetic data available to date, there is uncertainty with prediction of when waste rock and pit wall will transition from neutral pH to acidic conditions, and how long the transition will take.
- The impact of geomembranes on source terms for the WRF and TMF in closure have not been evaluated. As closure planning is refined, the water and load balance model will need to be updated based on revised waste management plans.
- Source terms should be updated to include the most recent changes to the mine plan and water management plan, refinement of source terms for road cuts and construction materials, and should incorporate all available data from on-going kinetic tests on waste rock, ore, and tailings.
- Baseline water quality source terms for Subarctic Creek and the Shungnak River should be updated to include data collected since 2019 and evaluated for seasonal variability.
- Inherent risk associated with assuming the groundwater interception system needs to be 100% efficient in order to prevent any seepage from impacting Subarctic Creek (and the Shungnak River).
- Uncertainty remains in predicted groundwater and seepage flows as these are a function of hydraulic conductivity and bedrock thickness which is only known at specific coordinates. It is the QP's opinion that these uncertainties can be reduced by the investigations described in this document (Section 22.17.3 and Section 23.5).

- Uncertainty in estimated tailings porewater release rate during closure phase of the Arctic Project. Further consolidation testwork should be carried out. Testwork should be done to confirm residence time required for the mill reagents to degrade. Water in the Process Pond has a residence time of 10 to 12 days. If this is insufficient, the pond design will need to be revised with additional capacity.
- The volume of water that accumulates prior to operations in the TMF for use during mill startup may exceed the TMF liner elevation as designed. If the liner raise cannot happen sooner than proposed, this surplus of water may be directed to the Process Pond. Any additional volume not required for mill startup will either be sent to the WRCP for treatment or directly to Subarctic Creek if it is non-contact runoff.
- The model was not evaluated for the hydrological condition where the site experiences a 1 in 25 dry year in the early years of operations. This scenario may result in the inability to meet mill water demands for part of the year if the dry year occurs prior to a sufficient reserve volume of water accumulates in the TMF. There is the potential to install groundwater wells to supplement mill water and mitigate this risk.
- For the model scenario involving two consecutive 1 in 25 wet years in Years 12 and 13 of operations, the WRCP alone does not have sufficient capacity in peak runoff months. The TMF and WTP operating year-round have sufficient capacity to manage this water surplus, but modelled pond volumes require further optimization. Cryo-concentration may impact WTP influent chemistry when the plant is operated year-round in operations.
- Climate change impacts should be incorporated into model and results should be assessed.

22.17.8 Metallurgical Testwork

Flowsheet development, locked cycle and variability testwork has shown the process flowsheet to be robust and stable at the study level under laboratory conditions. The ability to produce saleable metallic concentrates has been confirmed.

The process design assumed for this report has some risks identified that could impact delivery or economics and these need to be managed and mitigated by additional testwork and studies. The key aspects of the Arctic Project presenting most execution risk are:

- Performance of the flotation circuits at less than proven testwork results, which can be mitigated by further pilot level testing
- Metal recoveries not achieving proven testwork levels, which can be mitigated by further geometallurgical testwork, e.g: perform additional metallurgical testwork on the bulk Cu-Pb concentrate to determine if the Pb recovery can be increased and to better define the Au-Ag recovery
- Areas of low recovery in the pit, which can be mitigated by variability testing in several areas of the put to better define these zones based on updated recovery formulas

22.17.9 Road Access Rights

The Arctic Project assumes that the ADIEA will construct a road connecting the Arctic Project to the rest of Alaska’s transportation infrastructure (the AMDIAP or the AAP). Lawsuits were filed in 2020 by a coalition of national and Alaska environmental non-government organizations in response to the BLM's issuance of the JROD for the AAP. The lawsuits have currently been temporarily suspended pending additional work to be performed by the BLM on the final EIS.

23 RECOMMENDATIONS

23.1 Introduction

A work program is recommended to continue developing the Arctic Project through engineering and de-risking, and into construction at a total cost of $14.4 million.

Table 23-1 and the following sections summarize the key recommendations arising from a review of each major area of investigation completed as part of this study to improve the base case design. and described below.

Table 23-1: Summary of Recommended Work Packages

Description	Cost ($)
Mining	1,073,000
Geology and Resource Models	10,190,000
Open Pit Geotechnical Work	150,000
Hydrogeology	490,000
Tailings Management Facility	1,420,000
Closure	250,000
Water Treatment	170,000
Metallurgical Testing	175,000
Recovery Methods	350,000
Operational Readiness Plan	120,000
Total	**14,388,000**

23.2 Mining

- Perform a SMU study to define an optimal block size that can support the envisioned production rate while minimizing dilution. Estimated cost of $50,000.
- Requote explosives in the future. Ammonium nitrate prices are currently in record highs due to the conflict Ukraine-Russia. Since ammonium nitrate is the main raw material used to produce blasting explosives such as ANFO and emulsion, it has a significant impact on the blasting cost of the operation. Estimated cost of $5,000.
- Assess the use of alternate fuel sources for the mine mobile equipment including LNG. Estimated cost of $18,000.

- Confirm the location, thickness, and continuity of the talc layers near to the northeast wall of the final pit. The northeast wall has been forced to fully mine the talc zones, which has pushed the wall, increased the footprint of the pit, and added waste material to mine plan. Any change to the talc shapes can modify the pit design, and therefore, the amount of material mined. Estimated cost $1,000,000.

23.3 Geology and Resource Models

No issues were identified that are expected to have a material impact of the current Mineral Resource estimate. The following recommendations are made to confirm assumptions that non-material issues identified are not likely to materially impact the outcome of the resource estimate, but if addressed, they are expected to reduce potential risk associated with these minor issues and assist in more efficient and timely due diligence reviews in the future.

- Bias in historical copper and lead assays - Compare the current Mineral Resource estimate with an estimate prepared using a correction factor applied to the historic Cu and Pb assay values that remain in the assay database or an estimate that does not use these historical intervals. This work is expected to cost $5,000. If this work identifies a significant difference, then a continuation of the re-assay program may be warranted. The re-assay program is expected to cost $80,000.
- Bias in predicted SG values – A review of the methodology and dataset used to generate the predicted SG values is recommended and investigate to minimize or mitigate the apparent bias. Total estimated cost is $5,000.
- Mineral Resource Classification confidence related to geological complexity – To mitigate the risk of complex geology within the first 5 years of production area, consider putting additional in-fill holes around 25 m or less in drill grids. This is recommended to classify Measured Resources which can then be converted to Proven Mineral Reserves and provide high confidence in the production plan at project start and payback period. Estimated cost $10,000,000.
- Update the geological and resource model using all available data including the data collected during the 2022 drilling program. Total estimated cost is $80,000.
    - Review and update variogram models for all metals and SG, using all available data, especially for the payable metals.
    - Review and update the resource model estimation parameters for all metals and SG, especially for the minimum and maximum number of composites and number of composites used per drill hole.
    - Review the current compositing length of 1m that is more in agreement with the average sample length.
    - Update the geological model wireframes including talc wireframes with all available data.
- Address the following minor database issues. Total estimated cost is $25,000.
    - Complete and document a quantified transcription error check of collar, down hole survey, logging data and assays, especially for pre-2004 data.
    - Remove the few downhole survey measurements that are causing excessive deviation.
    - Prepare a document supporting applied detection limit decisions confirming high detection limits for legacy assays that remain in the database are not inadvertently adding economic value.
    - Include fields in the sample interval table indicating sample interval year, assay value year, and a flag to indicate if the value is supported by QC.

23.4 Open Pit Geotechnical work

It is the QP's recommendation that the following open pit geotechnical work be undertaken:

- The Talc Zone domain represents the weakest geotechnical domain observed at the Arctic deposit. The extent and persistence of the unit is of concern to the pit slope stability. The 3D talc model should be updated to include all available drilling and laboratory testing data. Based on the updated geometries, a geotechnical review should be completed to update the slope designs.
- Potential slope instability risks have been identified in the interim and final mining phases of the open pit design. The risks include potential sliding failure along talc domains and fault structures in the east wall of Phase01 pit, localized wedge failures in the south wall of the Phase02, Phase03 and Final pits, and sliding failures along foliation in the final pit design. These risks should be considered in the next stages of the Arctic Project.
- The seismic impact on the slope design should be re-evaluated using the site-specific assessment of the potential seismicity.
- The pore pressure monitoring system and mitigation plan would be critical in managing the slope stability risks during the interim mining phases. This plan should be reviewed as additional data becomes available and mitigation efforts should be adopted as appropriate to achieve pore pressure reduction needs.

Estimated total cost of $150,000.

23.5 Hydrogeology

It is the QP's recommendation that the following hydrogeology work be undertaken:

- In general, update the hydrogeological conceptual model for the pit and valley areas over time as more data becomes available and designs are revised.
- For the pit,
    - Continue water level monitoring
    - Assess the potential effects of seasonal or rapid and high amplitude recharge events (i.e., freshet) on pore pressure and on sump sizing needs.
    - Assess pit sump locations that could promote drainage from below talc layers.
    - Where feasible, assess drainage ditches around and set back from the pit perimeter to reduce runoff inflow, and internal ditching that directs water to pit sumps or out of the pit.
    - Consider snow removal or clearing off high elevation portions of the pit to reduce seasonal freshet increases in groundwater recharge and pore pressures.
    - Update the pit Groundwater Management Plan as mine design advances and plan to implement at the start of mining.
    - Estimated cost for points above of $40,000.
    - Install general pit perimeter groundwater monitoring on the north, south and west sides of the pit. Estimated cost $300,000

- For the valley bottom,
  - Conduct hydrogeological drilling at location of the groundwater seepage interception system (SIS) to characterize overburden and fractured rock properties in the specific area of the system
  - Update conceptual model
  - Update estimates of potential groundwater bypass of the WRCP and the groundwater SIS design as appropriate.
  - Initiate baseline monitoring of groundwater quality around groundwater SIS as soon as possible to provide a dataset for environmental permitting and to support performance monitoring of the groundwater SIS once in operation.
  - Estimated cost of $150,000.

23.6 Tailings Management Facility

It is the opinion of the QP, further engineering studies required for the TMF design include:

- Geotechnical and hydrogeological investigations to be completed in the foundations of the TMF, WRF, WRCP, and Process Pond including borehole drilling and laboratory testing to support design objectives, including the following. Estimated Cost $1,000,000.
- Dam Breach Assessment, based on site specific hazard assessment, for the TMF, the WRCP and Process pond. Estimated cost of $85,000.
- Finite element consolidation and seepage modelling of the TMF to better understand drainage, long-term settlement, and effects on capacity and closure. Estimated cost $75,000.
- Tailings deposition planning and water balance update to account for additional tailings volume, thickened tailings, plan for pool management and winter deposition, and timing of tailings dewatering towards end of mine life. Estimated cost $50,000.
- Evaluation of Tailings consolidation and requirement for an underdrain system. Estimated cost $25,000.
- Update WRF, TMF, WRCP, and Process Pond stability analyses (including liquefaction assessment) based on additional field investigation results and lab testing. Estimated Cost $150,000.
- Evaluate whether heat or gas generation during oxidation of waste rock may adversely affect the closure cover. Estimated cost $35,000.

23.7 Closure

It is the QP's opinion that a revegetation and cover study should be conducted to verify plant survival at the elevations proposed of the waste rock dump and reclaimed tailings facility. Trials should be conducted during operations to develop a revegetation program to meet 11 AAC 97.200 – Land Reclamation and Performance Standards. Alaska Statute AS 27.19.020 requires that the site be left in a stable condition. Generally, revegetation is the preferred method of stabilization, however other methods of stabilization could be implemented such as a non-reactive, durable rock cover.

Thriving plant communities at the proposed elevations are limited to localized areas with deeper growth media, and water availability. Much of the surrounding area is talus rock with minimal plant cover. The study should include climactic variables such as persistence of snow, limited rainfall, soil erosion or low temperature at altitude which may (or many not) limit plant survival. The proponent should consult with the Alaska Plant Materials Center to develop a revegetation program utilizing native species to determine the optimal growth media depth, amendments, and soil cover to optimize plant survival.

Estimated cost of $250,000.

23.8 Water Treatment

The following actions are recommended to further define the water treatment strategy. The cost of the first four items are already included in the estimated cost for the WTP:

- As the discharge permitting advances and the actual permitted discharge criteria become more defined, review and adjust the WTP design accordingly to target compliance with the criteria.
- Continue to track and address identified project risks and opportunities.
- Implement technology confirmation testing such as bench-scale and pilot-scale treatment tests to confirm the high treatment rates required to meet the WQS will be achieved and to right-size treatment equipment. Based on the outcome of influent water quality projections and these treatability studies, the system design can be refined. The RO reject treatment train can be evaluated once real mine-impacted water is generated during the operation phase.
- Refine the acids and bases used in the WTP to reduce the risk of TDS WQS compliance issues. This refinement can be done in later design phases.
- Continue to refine the WTP influent water quality predictions, which will include advancing the source term predictions. Estimated cost $170,000.

23.9 Metallurgical testing

Additional testwork is proposed to further refine metallurgical performance and recovery estimates for the selected flowsheet, including the following at an estimated cost of $175,000:

- Variability testing on samples with characteristics that match the LOM feed material to improve the recovery and concentrate grade models
- Investigate circuit flowsheet modifications to improve concentrate quality and reduce deleterious elements
- Comminution testing on new samples to increase the database of results to optimize comminution power

23.10 Recovery Methods

Further process plant activities to be considered for the next phase include capital cost optimization and a thorough review of equipment sizing/selection based on the geometallurgy outcomes. Estimated cost of $350,000.

23.11 Operational Readiness Plan

A robust OR plan would benefit the Arctic Project given the inherent risks and challenges associated with operating at a remote site. Estimated cost $120,000.

- Review project and operating assumptions to take into consideration the impacts of labour availability, logistical challenges with availability of equipment and associated goods/services and long lead times.
- Create a start-up/operational risk register to highlight issues for decisions taken during detailed design and for items related to the pre-operations/commissioning phase.
- Take into account specific development and implementation considerations in the OR plan for the remote cold-weather project site with limited access.
- Identify contingencies and challenge assumptions regarding supply, transportation and logistics for reagents, concentrate, and any other goods to develop potentially better alternatives.
- Identify strategies to track environment and operations-related permits to avoid other costly alternatives.
- Explore opportunities in the region to develop synergies with existing businesses and to understand what is available, as well as what needs to be developed to build the "industrial base" required to sustain mining and processing operations.
- Identify vendors and develop strategies to support contracts, parts, warehousing, training and first fills that will result in savings and positively impact HR planning.
- Consider strategic HR staffing decisions such as innovative labour scheduling and local sourcing.
- Identify HSEC policies and procedures for a remote site to leverage local resources and other tie-ins where possible.
- Mobilize key technical functions early, such as an assay lab and asset management functions to support mining and operations.

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25 RELIANCE ON INFORMATION PROVIDED BY THE REGISTRANT

25.1 Introduction

The QPs have relied on information provided by Trilogy Metals (the registrant) including expert reports, in preparing its findings and conclusions regarding the following modifying factors: macroeconomic information, marketing information, legal matters, environmental matters, accommodations the registrant commits or plans to provide to local individuals or groups in connection with its mine plans, and governmental factors.

The QPs consider it reasonable to rely on Trilogy Metals for this information since they have obtained opinion from appropriate experts.

25.2 Legal Matters

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Trilogy Metals and legal experts retained by Trilogy Metals for this information through the following documents:

- Kennecott Exploration Company, Kennecott Arctic Company, Alaska Gold Company and NovaGold Resources Inc., 2010: Net Smelter Returns Royalty Agreement dated effective January 7, 2010: 51 p.
- NovaCopper US Inc. and NANA Regional Corporation Inc., 2011: Exploration Agreement and Option to Lease, dated effective October 19, 2011: 144 p.
- NovaCopper US Inc. and NANA Regional Corporation Inc., 2012: Amending Agreement, dated effective May 10, 2012: 7 p.
- Reeves, J.N., 2018: Arctic Project: legal opinion prepared by Holmes Weddell & Barcott for Trilogy Metals Inc., 4 April 2018, 58 p.
- NovaCopper US Inc, Trilogy Metals Inc. and Ambler Metals LLC, 2020: Contribution Agreement, effective as of February 11, 2020: 39 p.

This information is used in Section 3 of the Report. The information is also used in support of the Mineral Resource estimate in Section 11, the Mineral Reserve estimate in Section 12 and the economic analysis in Section 19.

25.3 Taxation

The QPs have not independently reviewed the Arctic Project taxation position. The QPs have fully relied upon, and disclaim responsibility for, taxation information supplied by Ernst & Young LLP who was retained for this information. Arctic Project taxation information has been provided through:

- Ernst & Young LLP, 2023: Tax Module Build Arctic Project, January 20, 2023.

This information is used in the economic analysis in Section 19 of the Report.

25.4 Environmental Matters and Community Accommodations

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Trilogy Metals and experts retained by Trilogy Metals for information related to environmental (including tailings and water management) permitting, permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Alaska Department of Environment and Conservation. 2008. Alaska Water Quality Criteria Manual for Toxic and Other Deleterious Organic and Inorganic Substances.
- Craig, C., 2012, 2012 UKMP Water Quality Report, Internal Report by NovaCopper US Inc., January.
- DOWL, 2016, Ambler Metals Upper Kobuk Mining Project, Preliminary Wetlands Determination, Consultant Report for Ambler Metals Inc., November.
- Shaw Environmental Inc., 2007, Ambler Project, 2007 Environmental Baseline Sampling, Shaw Environmental, Inc.
- Shaw Environmental Inc., 2008, Water Quality Report July 2008 Event Final, Shaw Environmental, Inc.
- Shaw Environmental Inc., 2009, Hydraulics Data Report July 2009 Event Draft, Shaw Environmental, Inc.
- Shaw Environmental Inc., 2009, Water Quality Report July 2009 Event Final, Shaw Alaska, Inc.

This information is used in Section 17 of the Report. The information is also used in support of the Mineral Resource estimate in Section 11, the Mineral Reserve estimate in Section 12, capital and operating costs in Section 18 and the economic analysis in Section 19.

25.5 Marketing Information

The QPs have not independently reviewed the market studies, pricing or contract information. The QPs have fully relied upon, and disclaim responsibility for, information derived from Trilogy Metals and experts retained by Trilogy Metals for this information through the following documents:

- CIBC Global Mining Group, Analyst Consensus Commodity Price Forecasts, Consensus Forecast Summary, November 1, 2022.
- StoneHouse Consulting Inc.: Arctic Concentrate Marketing, prepared for Trilogy Metals Inc., dated January 5, 2023.

Metals price forecasting is a specialized business requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive global database that is outside of the purview of a QP. The QPs consider it reasonable to rely upon Trilogy Metals to provide metal price forecasts and marketing information on the base metal concentrates as they sought expert input for this information.

This information is used in Section 16 of the Report. The information is also used in support of the Mineral Resource estimate in Section 11, the Mineral Reserve estimate in Section 12, and economic analysis in Section 19.

Appendix A – Mineral Claims

To Affidavit of Annual Labor for Labor Year Ending September 1, 2022

Ambler Metals LLC

State of Alaska Mining Claims

Kateel River Meridian, Kotzebue Recording District

	ADL	Claim Name	Township	Range	Section	1/4 Section
1	540543	Arctic 40A	21N	11E	35	SW
2	540544	Arctic 496A	21N	11E	34	SE
3	540545	Arctic 1001	21N	11E	34	SE
4	540546	Arctic 1002	21N	11E	34	SE & SW
5	540549	Arctic 1005	21N	11E	35	SW
6	546144	SC 24	21N	10E	16	SW & SE
7	546145	SC 25	21N	10E	16	SW, SE, NW & NE
8	546146	SC 26	21N	10E	16	NW & NE
9	546147	SC 34	21N	10E	16	SE
10	546148	SC 35	21N	10E	16	SE & NE
11	546149	SC 36	21N	10E	16	NE
12	546150	SC 44	21N	10E	15; 16	SW; SE
13	546151	SC 45	21N	10E	15; 16	SW & NW; SE & NE
14	546152	SC 46	21N	10E	15; 16	NW; NE
15	546153	SC 54	21N	10E	15	SW
16	546154	SC 55	21N	10E	15	SW & NW
17	546155	SC 56	21N	10E	15	NW
18	546156	SC 64	21N	10E	15	SW & SE
19	546157	SC 65	21N	10E	15	SW, SE, NW & NE
20	546158	SC 66	21N	10E	15	NW & NE
21	590853	AM 63-165	21N	9E	14	NW
22	590854	AM 63-166	21N	9E	14	NW
23	590855	AM 63-167	21N	9E	14	NE
24	590856	AM 63-168	21N	9E	14	NE
25	590857	AM 63-169	21N	9E	13	NW
26	590858	AM 63-170	21N	9E	13	NW
27	590859	AM 63-171	21N	9E	13	NE
28	590860	AM 63-172	21N	9E	13	NE
29	590874	AM 64-165	21N	9E	14	NW
30	590875	AM 64-166	21N	9E	14	NW
31	590876	AM 64-167	21N	9E	14	NE
32	590877	AM 64-168	21N	9E	14	NE

33	590878	AM 64-169	21N	9E	13	NW
34	590879	AM 64-170	21N	9E	13	NW
35	590880	AM 64-171	21N	9E	13	NE
36	590881	AM 64-172	21N	9E	13	NE
37	590895	AM 65-165	21N	9E	11	SW
38	590896	AM 65-166	21N	9E	11	SW
39	590897	AM 65-167	21N	9E	11	SE
40	590898	AM 65-168	21N	9E	11	SE
41	590899	AM 65-169	21N	9E	12	SW
42	590900	AM 65-170	21N	9E	12	SW
43	590901	AM 65-171	21N	9E	12	SE
44	590902	AM 65-172	21N	9E	12	SE
45	590916	AM 66-165	21N	9E	11	SW
46	590917	AM 66-166	21N	9E	11	SW
47	590918	AM 66-167	21N	9E	11	SE
48	590919	AM 66-168	21N	9E	11	SE
49	590920	AM 66-169	21N	9E	12	SW
50	590921	AM 66-170	21N	9E	12	SW
51	590922	AM 66-171	21N	9E	12	SE
52	590923	AM 66-172	21N	9E	12	SE
53	590940	AM 67-165	21N	9E	11	NW
54	590941	AM 67-166	21N	9E	11	NW
55	590942	AM 67-167	21N	9E	11	NE
56	590943	AM 67-168	21N	9E	11	NE
57	590944	AM 67-169	21N	9E	12	NW
58	590945	AM 67-170	21N	9E	12	NW
59	590946	AM 67-171	21N	9E	12	NE
60	590947	AM 67-172	21N	9E	12	NE
61	590998	AM 56-186	21N	10E	27	NW
62	590999	AM 56-187	21N	10E	27	NE
63	591000	AM 56-188	21N	10E	27	NE
64	591001	AM 56-189	21N	10E	26	NW
65	591002	AM 56-190	21N	10E	26	NW
66	591003	AM 56-191	21N	10E	26	NE
67	591004	AM 56-192	21N	10E	26	NE
68	591005	AM 56-193	21N	10E	25	NW
69	591006	AM 56-194	21N	10E	25	NW
70	591007	AM 56-195	21N	10E	25	NE
71	591008	AM 57-176	21N	10E	19	SE
72	591009	AM 57-177	21N	10E	20	SW
73	591010	AM 57-178	21N	10E	20	SW
74	591011	AM 57-179	21N	10E	20	SE
75	591012	AM 57-180	21N	10E	20	SE
76	591013	AM 57-181	21N	10E	21	SW
77	591014	AM 57-182	21N	10E	21	SW
78	591015	AM 57-183	21N	10E	21	SE

79	591016	AM 57-184	21N	10E	21	SE
80	591017	AM 57-185	21N	10E	22	SW
81	591018	AM 57-186	21N	10E	22	SW
82	591019	AM 57-187	21N	10E	22	SE
83	591020	AM 57-188	21N	10E	22	SE
84	591021	AM 57-189	21N	10E	23	SW
85	591022	AM 57-190	21N	10E	23	SW
86	591023	AM 57-191	21N	10E	23	SE
87	591024	AM 57-192	21N	10E	23	SE
88	591025	AM 57-193	21N	10E	24	SW
89	591026	AM 57-194	21N	10E	24	SW
90	591027	AM 57-195	21N	10E	24	SE
91	591028	AM 58-176	21N	10E	19	SE
92	591029	AM 58-177	21N	10E	20	SW
93	591030	AM 58-178	21N	10E	20	SW
94	591031	AM 58-179	21N	10E	20	SE
95	591032	AM 58-180	21N	10E	20	SE
96	591033	AM 58-181	21N	10E	21	SW
97	591034	AM 58-182	21N	10E	21	SW
98	591035	AM 58-183	21N	10E	21	SE
99	591036	AM 58-184	21N	10E	21	SE
100	591037	AM 58-185	21N	10E	22	SW
101	591038	AM 58-186	21N	10E	22	SW
102	591039	AM 58-187	21N	10E	22	SE
103	591040	AM 58-188	21N	10E	22	SE
104	591041	AM 58-189	21N	10E	23	SW
105	591042	AM 58-190	21N	10E	23	SW
106	591043	AM 58-191	21N	10E	23	SE
107	591044	AM 58-192	21N	10E	23	SE
108	591045	AM 58-193	21N	10E	24	SW
109	591046	AM 58-194	21N	10E	24	SW
110	591047	AM 59-176	21N	10E	19	NE
111	591048	AM 59-177	21N	10E	20	NW
112	591049	AM 59-178	21N	10E	20	NW
113	591050	AM 59-179	21N	10E	20	NE
114	591051	AM 59-180	21N	10E	20	NE
115	591052	AM 59-181	21N	10E	21	NW
116	591053	AM 59-182	21N	10E	21	NW
117	591054	AM 59-183	21N	10E	21	NE
118	591055	AM 59-184	21N	10E	21	NE
119	591056	AM 59-185	21N	10E	22	NW
120	591057	AM 59-186	21N	10E	22	NW
121	591058	AM 59-187	21N	10E	22	NE
122	591059	AM 59-188	21N	10E	22	NE
123	591060	AM 59-189	21N	10E	23	NW
124	591061	AM 59-190	21N	10E	23	NW

125	591062	AM 59-191	21N	10E	23	NE
126	591063	AM 59-192	21N	10E	23	NE
127	591064	AM 59-193	21N	10E	24	NW
128	591065	AM 60-176	21N	10E	19	NE
129	591066	AM 60-177	21N	10E	20	NW
130	591067	AM 60-178	21N	10E	20	NW
131	591068	AM 60-179	21N	10E	20	NE
132	591069	AM 60-180	21N	10E	20	NE
133	591070	AM 60-181	21N	10E	21	NW
134	591071	AM 60-182	21N	10E	21	NW
135	591072	AM 60-183	21N	10E	21	NE
136	591073	AM 60-184	21N	10E	21	NE
137	591074	AM 60-185	21N	10E	22	NW
138	591075	AM 60-186	21N	10E	22	NW
139	591076	AM 60-187	21N	10E	22	NE
140	591077	AM 60-188	21N	10E	22	NE
141	591078	AM 60-189	21N	10E	23	NW
142	591079	AM 60-190	21N	10E	23	NW
143	591080	AM 60-191	21N	10E	23	NE
144	591081	AM 60-192	21N	10E	23	NE
145	591082	AM 60-193	21N	10E	24	NW
146	591083	AM 61-176	21N	10E	18	SE
147	591084	AM 61-177	21N	10E	17	SW
148	591085	AM 61-178	21N	10E	17	SW
149	591086	AM 61-179	21N	10E	17	SE
150	591087	AM 61-180	21N	10E	17	SE
151	591088	AM 61-181	21N	10E	16	SW
152	591089	AM 61-182	21N	10E	16	SW
153	591090	AM 61-183	21N	10E	16	SE
154	591091	AM 61-184	21N	10E	16	SE
155	591092	AM 61-185	21N	10E	15	SW
156	591093	AM 61-186	21N	10E	15	SW
157	591094	AM 61-187	21N	10E	15	SE
158	591095	AM 61-188	21N	10E	15	SE
159	591096	AM 61-189	21N	10E	14	SW
160	591097	AM 61-190	21N	10E	14	SW
161	591098	AM 61-191	21N	10E	14	SE
162	591099	AM 61-192	21N	10E	14	SE
163	591100	AM 61-193	21N	10E	13	SW
164	591101	AM 62-176	21N	10E	18	SE
165	591102	AM 62-177	21N	10E	17	SW
166	591103	AM 62-178	21N	10E	17	SW
167	591104	AM 62-179	21N	10E	17	SE
168	591105	AM 62-180	21N	10E	17	SE
169	591106	AM 62-181	21N	10E	16	SW
170	591107	AM 62-182	21N	10E	16	SW

171	591108	AM 62-187	21N	10E	15	SE
172	591109	AM 62-188	21N	10E	15	SE
173	591110	AM 62-189	21N	10E	14	SW
174	591111	AM 62-190	21N	10E	14	SW
175	591112	AM 62-191	21N	10E	14	SE
176	591113	AM 62-192	21N	10E	14	SE
177	591114	AM 62-193	21N	10E	13	SW
178	591115	AM 63-173	21N	10E	18	NW
179	591116	AM 63-174	21N	10E	18	NW
180	591117	AM 63-175	21N	10E	18	NE
181	591118	AM 63-176	21N	10E	18	NE
182	591119	AM 63-177	21N	10E	17	NW
183	591120	AM 63-178	21N	10E	17	NW
184	591121	AM 63-179	21N	10E	17	NE
185	591122	AM 63-180	21N	10E	17	NE
186	591123	AM 63-181	21N	10E	16	NW
187	591124	AM 63-182	21N	10E	16	NW
188	591125	AM 63-187	21N	10E	15	NE
189	591126	AM 63-188	21N	10E	15	NE
190	591127	AM 63-189	21N	10E	14	NW
191	591128	AM 63-190	21N	10E	14	NW
192	591129	AM 63-191	21N	10E	14	NE
193	591130	AM 63-192	21N	10E	14	NE
194	591131	AM 63-193	21N	10E	13	NW
195	591132	AM 64-173	21N	10E	18	NW
196	591133	AM 64-174	21N	10E	18	NW
197	591134	AM 64-175	21N	10E	18	NE
198	591135	AM 64-176	21N	10E	18	NE
199	591136	AM 64-177	21N	10E	17	NW
200	591137	AM 64-178	21N	10E	17	NW
201	591138	AM 64-179	21N	10E	17	NE
202	591139	AM 64-180	21N	10E	17	NE
203	591140	AM 64-181	21N	10E	16	NW
204	591141	AM 64-182	21N	10E	16	NW
205	591142	AM 64-183	21N	10E	16	NE
206	591143	AM 64-184	21N	10E	16	NE
207	591144	AM 64-185	21N	10E	15	NW
208	591145	AM 64-186	21N	10E	15	NW
209	591146	AM 64-187	21N	10E	15	NE
210	591147	AM 64-188	21N	10E	15	NE
211	591148	AM 64-189	21N	10E	14	NW
212	591149	AM 64-190	21N	10E	14	NW
213	591150	AM 64-191	21N	10E	14	NE
214	591151	AM 64-192	21N	10E	14	NE
215	591152	AM 64-193	21N	10E	13	NW
216	591153	AM 65-173	21N	10E	7	SW

217	591154	AM 65-174	21N	10E	7	SW
218	591155	AM 65-175	21N	10E	7	SE
219	591156	AM 65-176	21N	10E	7	SE
220	591157	AM 65-177	21N	10E	8	SW
221	591158	AM 65-178	21N	10E	8	SW
222	591159	AM 65-179	21N	10E	8	SE
223	591160	AM 65-180	21N	10E	8	SE
224	591161	AM 65-181	21N	10E	9	SW
225	591162	AM 65-182	21N	10E	9	SW
226	591163	AM 65-183	21N	10E	9	SE
227	591164	AM 65-184	21N	10E	9	SE
228	591165	AM 65-185	21N	10E	10	SW
229	591166	AM 65-186	21N	10E	10	SW
230	591167	AM 65-187	21N	10E	10	SE
231	591168	AM 65-188	21N	10E	10	SE
232	591169	AM 65-189	21N	10E	11	SW
233	591170	AM 65-190	21N	10E	11	SW
234	591171	AM 65-191	21N	10E	11	SE
235	591172	AM 65-192	21N	10E	11	SE
236	591173	AM 65-193	21N	10E	12	SW
237	591174	AM 66-173	21N	10E	7	SW
238	591175	AM 66-174	21N	10E	7	SW
239	591176	AM 66-175	21N	10E	7	SE
240	591177	AM 66-176	21N	10E	7	SE
241	591178	AM 66-177	21N	10E	8	SW
242	591179	AM 66-178	21N	10E	8	SW
243	591180	AM 66-179	21N	10E	8	SE
244	591181	AM 66-180	21N	10E	8	SE
245	591182	AM 66-181	21N	10E	9	SW
246	591183	AM 66-182	21N	10E	9	SW
247	591184	AM 66-183	21N	10E	9	SE
248	591185	AM 66-184	21N	10E	9	SE
249	591186	AM 66-185	21N	10E	10	SW
250	591187	AM 66-186	21N	10E	10	SW
251	591188	AM 66-187	21N	10E	10	SE
252	591189	AM 66-188	21N	10E	10	SE
253	591190	AM 66-189	21N	10E	11	SW
254	591191	AM 66-190	21N	10E	11	SW
255	591192	AM 66-191	21N	10E	11	SE
256	591193	AM 66-192	21N	10E	11	SE
257	591194	AM 66-193	21N	10E	12	SW
258	591195	AM 67-173	21N	10E	7	NW
259	591196	AM 67-174	21N	10E	7	NW
260	591197	AM 67-175	21N	10E	7	NE
261	591198	AM 67-176	21N	10E	7	NE
262	591199	AM 67-177	21N	10E	8	NW

263	591200	AM 67-178	21N	10E	8	NW
264	591201	AM 67-179	21N	10E	8	NE
265	591202	AM 67-180	21N	10E	8	NE
266	591203	AM 67-181	21N	10E	9	NW
267	591204	AM 67-182	21N	10E	9	NW
268	591205	AM 67-183	21N	10E	9	NE
269	591206	AM 67-184	21N	10E	9	NE
270	591207	AM 67-185	21N	10E	10	NW
271	591208	AM 67-186	21N	10E	10	NW
272	591209	AM 67-187	21N	10E	10	NE
273	591210	AM 67-188	21N	10E	10	NE
274	591211	AM 67-189	21N	10E	11	NW
275	591212	AM 67-190	21N	10E	11	NW
276	591213	AM 67-191	21N	10E	11	NE
277	591214	AM 67-192	21N	10E	11	NE
278	591215	AM 67-193	21N	10E	12	NW
279	591216	AM 67-194	21N	10E	12	NW
280	591217	AM 67-195	21N	10E	12	NE
281	591218	AM 67-196	21N	10E	12	NE
282	591219	AM 49-206	21N	11E	33	SW
283	591220	AM 49-207	21N	11E	33	SE
284	591221	AM 49-208	21N	11E	33	SE
285	591222	AM 49-209	21N	11E	34	SW
286	591223	AM 49-210	21N	11E	34	SW
287	591224	AM 49-214	21N	11E	35	SW
288	591225	AM 49-215	21N	11E	35	SE
289	591226	AM 49-216	21N	11E	35	SE
290	591227	AM 49-217	21N	11E	36	SW
291	591228	AM 49-218	21N	11E	36	SW
292	591229	AM 49-219	21N	11E	36	SE
293	591230	AM 49-220	21N	11E	36	SE
294	591231	AM 50-206	21N	11E	33	SW
295	591232	AM 50-207	21N	11E	33	SE
296	591233	AM 50-208	21N	11E	33	SE
297	591234	AM 50-209	21N	11E	34	SW
298	591235	AM 50-210	21N	11E	34	SW
299	591236	AM 50-211	21N	11E	34	SE
300	591237	AM 50-213	21N	11E	35	SW
301	591238	AM 50-214	21N	11E	35	SW
302	591239	AM 50-215	21N	11E	35	SE
303	591240	AM 50-216	21N	11E	35	SE
304	591241	AM 50-217	21N	11E	36	SW
305	591242	AM 50-218	21N	11E	36	SW
306	591243	AM 50-219	21N	11E	36	SE
307	591244	AM 50-220	21N	11E	36	SE
308	591245	AM 51-206	21N	11E	33	NW

309	591246	AM 51-207	21N	11E	33	NE
310	591247	AM 51-208	21N	11E	33	NE
311	591248	AM 51-209	21N	11E	34	NW
312	591249	AM 51-210	21N	11E	34	NW
313	591250	AM 51-211	21N	11E	34	NE
314	591251	AM 51-212	21N	11E	34	NE
315	591252	AM 51-213	21N	11E	35	NW
316	591253	AM 51-214	21N	11E	35	NW
317	591254	AM 51-215	21N	11E	35	NE
318	591255	AM 51-216	21N	11E	35	NE
319	591256	AM 51-217	21N	11E	36	NW
320	591257	AM 51-218	21N	11E	36	NW
321	591258	AM 51-219	21N	11E	36	NE
322	591259	AM 51-220	21N	11E	36	NE
323	591260	AM 52-206	21N	11E	33	NW
324	591261	AM 52-207	21N	11E	33	NE
325	591262	AM 52-208	21N	11E	33	NE
326	591263	AM 52-209	21N	11E	34	NW
327	591264	AM 52-210	21N	11E	34	NW
328	591265	AM 52-211	21N	11E	34	NE
329	591266	AM 52-212	21N	11E	34	NE
330	591267	AM 52-213	21N	11E	35	NW
331	591268	AM 52-214	21N	11E	35	NW
332	591269	AM 52-215	21N	11E	35	NE
333	591270	AM 52-216	21N	11E	35	NE
334	591271	AM 52-217	21N	11E	36	NW
335	591272	AM 52-218	21N	11E	36	NW
336	591273	AM 52-219	21N	11E	36	NE
337	591274	AM 52-220	21N	11E	36	NE
338	591275	AM 53-206	21N	11E	28	SW
339	591276	AM 53-207	21N	11E	28	SE
340	591277	AM 53-208	21N	11E	28	SE
341	591278	AM 53-209	21N	11E	27	SW
342	591279	AM 53-210	21N	11E	27	SW
343	591280	AM 53-211	21N	11E	27	SE
344	591281	AM 53-212	21N	11E	27	SE
345	591282	AM 53-213	21N	11E	26	SW
346	591283	AM 53-214	21N	11E	26	SW
347	591284	AM 53-215	21N	11E	26	SE
348	591285	AM 53-216	21N	11E	26	SE
349	591286	AM 53-217	21N	11E	25	SW
350	591287	AM 53-218	21N	11E	25	SW
351	591288	AM 53-219	21N	11E	25	SE
352	591289	AM 53-220	21N	11E	25	SE
353	591290	AM 54-206	21N	11E	28	SW
354	591291	AM 54-207	21N	11E	28	SE

355	591292	AM 54-208	21N	11E	28	SE
356	591293	AM 54-209	21N	11E	27	SW
357	591294	AM 54-210	21N	11E	27	SW
358	591295	AM 54-211	21N	11E	27	SE
359	591296	AM 54-212	21N	11E	27	SE
360	591297	AM 54-213	21N	11E	26	SW
361	591298	AM 54-214	21N	11E	26	SW
362	591299	AM 54-215	21N	11E	26	SE
363	591300	AM 54-216	21N	11E	26	SE
364	591301	AM 54-217	21N	11E	25	SW
365	591302	AM 54-218	21N	11E	25	SW
366	591303	AM 54-219	21N	11E	25	SE
367	591304	AM 54-220	21N	11E	25	SE
368	591305	AM 55-206	21N	11E	28	NW
369	591306	AM 55-207	21N	11E	28	NE
370	591307	AM 55-208	21N	11E	28	NE
371	591308	AM 55-209	21N	11E	27	NW
372	591309	AM 55-210	21N	11E	27	NW
373	591310	AM 55-211	21N	11E	27	NE
374	591311	AM 55-212	21N	11E	27	NE
375	591312	AM 55-213	21N	11E	26	NW
376	591313	AM 55-214	21N	11E	26	NW
377	591314	AM 55-215	21N	11E	26	NE
378	591315	AM 55-216	21N	11E	26	NE
379	591316	AM 55-217	21N	11E	25	NW
380	591317	AM 55-218	21N	11E	25	NW
381	591318	AM 55-219	21N	11E	25	NE
382	591319	AM 55-220	21N	11E	25	NE
383	591320	AM 56-206	21N	11E	28	NW
384	591321	AM 56-207	21N	11E	28	NE
385	591322	AM 56-208	21N	11E	28	NE
386	591323	AM 56-209	21N	11E	27	NW
387	591324	AM 56-210	21N	11E	27	NW
388	591325	AM 56-211	21N	11E	27	NE
389	591326	AM 56-212	21N	11E	27	NE
390	591327	AM 56-213	21N	11E	26	NW
391	591328	AM 56-214	21N	11E	26	NW
392	591329	AM 56-215	21N	11E	26	NE
393	591330	AM 56-216	21N	11E	26	NE
394	591331	AM 56-217	21N	11E	25	NW
395	591332	AM 56-218	21N	11E	25	NW
396	591333	AM 56-219	21N	11E	25	NE
397	591334	AM 56-220	21N	11E	25	NE
398	591335	AM 57-206	21N	11E	21	SW
399	591336	AM 57-207	21N	11E	21	SE
400	591337	AM 57-208	21N	11E	21	SE

401	591338	AM 57-209	21N	11E	22	SW
402	591339	AM 57-210	21N	11E	22	SW
403	591340	AM 57-211	21N	11E	22	SE
404	591341	AM 57-212	21N	11E	22	SE
405	591342	AM 57-213	21N	11E	23	SW
406	591343	AM 57-214	21N	11E	23	SW
407	591344	AM 57-215	21N	11E	23	SE
408	591345	AM 57-216	21N	11E	23	SE
409	591346	AM 57-217	21N	11E	24	SW
410	591347	AM 57-218	21N	11E	24	SW
411	591348	AM 57-219	21N	11E	24	SE
412	591349	AM 57-220	21N	11E	24	SE
413	591350	AM 58-206	21N	11E	21	SW
414	591351	AM 58-207	21N	11E	21	SE
415	591352	AM 58-208	21N	11E	21	SE
416	591353	AM 58-209	21N	11E	22	SW
417	591354	AM 58-210	21N	11E	22	SW
418	591355	AM 58-211	21N	11E	22	SE
419	591356	AM 58-212	21N	11E	22	SE
420	591357	AM 58-213	21N	11E	23	SW
421	591358	AM 58-214	21N	11E	23	SW
422	591359	AM 58-215	21N	11E	23	SE
423	591360	AM 58-216	21N	11E	23	SE
424	591361	AM 58-217	21N	11E	24	SW
425	591362	AM 58-218	21N	11E	24	SW
426	591363	AM 58-219	21N	11E	24	SE
427	591364	AM 58-220	21N	11E	24	SE
428	591365	AM 59-202	21N	11E	20	NW
429	591366	AM 59-203	21N	11E	20	NE
430	591367	AM 59-204	21N	11E	20	NE
431	591368	AM 59-205	21N	11E	21	NW
432	591369	AM 59-206	21N	11E	21	NW
433	591370	AM 59-207	21N	11E	21	NE
434	591371	AM 59-208	21N	11E	21	NE
435	591372	AM 59-209	21N	11E	22	NW
436	591373	AM 59-210	21N	11E	22	NW
437	591374	AM 59-211	21N	11E	22	NE
438	591375	AM 59-212	21N	11E	22	NE
439	591376	AM 59-213	21N	11E	23	NW
440	591377	AM 59-214	21N	11E	23	NW
441	591378	AM 59-215	21N	11E	23	NE
442	591379	AM 59-216	21N	11E	23	NE
443	591380	AM 59-217	21N	11E	24	NW
444	591381	AM 59-218	21N	11E	24	NW
445	591382	AM 60-202	21N	11E	20	NW
446	591383	AM 60-203	21N	11E	20	NE

447	591384	AM 60-204	21N	11E	20	NE
448	591385	AM 60-205	21N	11E	21	NW
449	591386	AM 60-206	21N	11E	21	NW
450	591387	AM 60-207	21N	11E	21	NE
451	591388	AM 60-208	21N	11E	21	NE
452	591389	AM 60-209	21N	11E	22	NW
453	591390	AM 60-210	21N	11E	22	NW
454	591391	AM 60-211	21N	11E	22	NE
455	591392	AM 60-212	21N	11E	22	NE
456	591393	AM 60-213	21N	11E	23	NW
457	591394	AM 60-214	21N	11E	23	NW
458	591395	AM 60-215	21N	11E	23	NE
459	591396	AM 60-216	21N	11E	23	NE
460	591397	AM 60-217	21N	11E	24	NW
461	591398	AM 60-218	21N	11E	24	NW
462	591399	AM 61-202	21N	11E	17	SW
463	591400	AM 61-203	21N	11E	17	SE
464	591401	AM 61-204	21N	11E	17	SE
465	591402	AM 61-205	21N	11E	16	SW
466	591403	AM 61-206	21N	11E	16	SW
467	591404	AM 61-207	21N	11E	16	SE
468	591405	AM 61-208	21N	11E	16	SE
469	591406	AM 61-209	21N	11E	15	SW
470	591407	AM 61-210	21N	11E	15	SW
471	591408	AM 61-211	21N	11E	15	SE
472	591409	AM 61-212	21N	11E	15	SE
473	591410	AM 61-213	21N	11E	14	SW
474	591411	AM 61-214	21N	11E	14	SW
475	591412	AM 61-215	21N	11E	14	SE
476	591413	AM 61-216	21N	11E	14	SE
477	591414	AM 61-217	21N	11E	13	SW
478	591415	AM 61-218	21N	11E	13	SW
479	591416	AM 62-202	21N	11E	17	SW
480	591417	AM 62-203	21N	11E	17	SE
481	591418	AM 62-204	21N	11E	17	SE
482	591419	AM 62-205	21N	11E	16	SW
483	591420	AM 62-206	21N	11E	16	SW
484	591421	AM 62-207	21N	11E	16	SE
485	591422	AM 62-208	21N	11E	16	SE
486	591423	AM 62-209	21N	11E	15	SW
487	591424	AM 62-210	21N	11E	15	SW
488	591425	AM 62-211	21N	11E	15	SE
489	591426	AM 62-212	21N	11E	15	SE
490	591427	AM 62-213	21N	11E	14	SW
491	591428	AM 62-214	21N	11E	14	SW
492	591429	AM 62-215	21N	11E	14	SE

493	591430	AM 62-216	21N	11E	14	SE
494	591431	AM 62-217	21N	11E	13	SW
495	591432	AM 62-218	21N	11E	13	SW
496	591433	AM 63-202	21N	11E	17	NW
497	591434	AM 63-203	21N	11E	17	NE
498	591435	AM 63-204	21N	11E	17	NE
499	591436	AM 63-205	21N	11E	16	NW
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502	591439	AM 63-208	21N	11E	16	NE
503	591440	AM 63-209	21N	11E	15	NW
504	591441	AM 63-210	21N	11E	15	NW
505	591442	AM 63-211	21N	11E	15	NE
506	591443	AM 63-212	21N	11E	15	NE
507	591444	AM 64-202	21N	11E	17	NW
508	591445	AM 64-203	21N	11E	17	NE
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510	591447	AM 64-205	21N	11E	16	NW
511	591448	AM 64-206	21N	11E	16	NW
512	591449	AM 64-207	21N	11E	16	NE
513	591450	AM 64-208	21N	11E	16	NE
514	591451	AM 64-209	21N	11E	15	NW
515	591452	AM 64-210	21N	11E	15	NW
516	591453	AM 64-211	21N	11E	15	NE
517	591454	AM 64-212	21N	11E	15	NE
518	591455	AM 65-202	21N	11E	8	SW
519	591456	AM 65-203	21N	11E	8	SE
520	591457	AM 65-204	21N	11E	8	SE
521	591458	AM 65-205	21N	11E	9	SW
522	591459	AM 65-206	21N	11E	9	SW
523	591460	AM 65-207	21N	11E	9	SE
524	591461	AM 65-208	21N	11E	9	SE
525	591462	AM 65-209	21N	11E	10	SW
526	591463	AM 65-210	21N	11E	10	SW
527	591464	AM 65-211	21N	11E	10	SE
528	591465	AM 65-212	21N	11E	10	SE
529	591466	AM 66-202	21N	11E	8	SW
530	591467	AM 66-203	21N	11E	8	SE
531	591468	AM 66-204	21N	11E	8	SE
532	591469	AM 66-205	21N	11E	9	SW
533	591470	AM 66-206	21N	11E	9	SW
534	591471	AM 66-207	21N	11E	9	SE
535	591472	AM 66-208	21N	11E	9	SE
536	591473	AM 66-209	21N	11E	10	SW
537	591474	AM 66-210	21N	11E	10	SW
538	591475	AM 66-211	21N	11E	10	SE

539	591476	AM 66-212	21N	11E	10	SE
540	591477	AM 67-197	21N	11E	7	NW
541	591478	AM 67-198	21N	11E	7	NW
542	591479	AM 67-199	21N	11E	7	NE
543	591480	AM 67-200	21N	11E	7	NE
544	591481	AM 67-201	21N	11E	8	NW
545	591482	AM 67-202	21N	11E	8	NW
546	591483	AM 67-203	21N	11E	8	NE
547	591484	AM 67-204	21N	11E	8	NE
548	591485	AM 67-205	21N	11E	9	NW
549	591486	AM 67-206	21N	11E	9	NW
550	591487	AM 67-207	21N	11E	9	NE
551	591488	AM 67-208	21N	11E	9	NE
552	591489	AM 67-209	21N	11E	10	NW
553	591490	AM 67-210	21N	11E	10	NW
554	591491	AM 67-211	21N	11E	10	NE
555	591492	AM 67-212	21N	11E	10	NE
556	591493	AM 68-208	21N	11E	9	NE
557	591494	AM 68-209	21N	11E	10	NW
558	591495	AM 68-210	21N	11E	10	NW
559	591496	AM 68-211	21N	11E	10	NE
560	591497	AM 68-212	21N	11E	10	NE
561	591498	AM 69-208	21N	11E	4	SE
562	591499	AM 69-209	21N	11E	3	SW
563	591500	AM 69-210	21N	11E	3	SW
564	591501	AM 69-211	21N	11E	3	SE
565	591502	AM 69-212	21N	11E	3	SE
566	591503	AM 49-221	21N	12E	31	SW
567	591504	AM 49-222	21N	12E	31	SW
568	591505	AM 49-223	21N	12E	31	SE
569	591506	AM 49-224	21N	12E	31	SE
570	591507	AM 49-225	21N	12E	32	SW
571	591508	AM 49-226	21N	12E	32	SW
572	591509	AM 49-227	21N	12E	32	SE
573	591510	AM 49-228	21N	12E	32	SE
574	591511	AM 49-229	21N	12E	33	SW
575	591512	AM 49-230	21N	12E	33	SW
576	591513	AM 50-221	21N	12E	31	SW
577	591514	AM 50-222	21N	12E	31	SW
578	591515	AM 50-223	21N	12E	31	SE
579	591516	AM 50-224	21N	12E	31	SE
580	591517	AM 50-225	21N	12E	32	SW
581	591518	AM 50-226	21N	12E	32	SW
582	591519	AM 50-227	21N	12E	32	SE
583	591520	AM 50-228	21N	12E	32	SE
584	591521	AM 50-229	21N	12E	33	SW

585	591522	AM 50-230	21N	12E	33	SW
586	591523	AM 51-221	21N	12E	31	NW
587	591524	AM 51-222	21N	12E	31	NW
588	591525	AM 51-223	21N	12E	31	NE
589	591526	AM 51-224	21N	12E	31	NE
590	591527	AM 51-225	21N	12E	32	NW
591	591528	AM 51-226	21N	12E	32	NW
592	591529	AM 51-227	21N	12E	32	NE
593	591530	AM 51-228	21N	12E	32	NE
594	591531	AM 51-229	21N	12E	33	NW
595	591532	AM 51-230	21N	12E	33	NW
596	591533	AM 52-221	21N	12E	31	NW
597	591534	AM 52-222	21N	12E	31	NW
598	591535	AM 52-223	21N	12E	31	NE
599	591536	AM 52-224	21N	12E	31	NE
600	591537	AM 52-225	21N	12E	32	NW
601	591538	AM 52-226	21N	12E	32	NW
602	591539	AM 52-227	21N	12E	32	NE
603	591540	AM 52-228	21N	12E	32	NE
604	591541	AM 52-229	21N	12E	33	NW
605	591542	AM 52-230	21N	12E	33	NW
606	591543	AM 53-221	21N	12E	30	SW
607	591544	AM 53-222	21N	12E	30	SW
608	591545	AM 53-223	21N	12E	30	SE
609	591546	AM 53-224	21N	12E	30	SE
610	591547	AM 54-221	21N	12E	30	SW
611	591548	AM 54-222	21N	12E	30	SW
612	591549	AM 54-223	21N	12E	30	SE
613	591550	AM 54-224	21N	12E	30	SE
614	591551	AM 55-221	21N	12E	30	NW
615	591552	AM 55-222	21N	12E	30	NW
616	591553	AM 55-223	21N	12E	30	NE
617	591554	AM 55-224	21N	12E	30	NE
618	591555	AM 56-221	21N	12E	30	NW
619	591556	AM 56-222	21N	12E	30	NW
620	591557	AM 56-223	21N	12E	30	NE
621	591558	AM 56-224	21N	12E	30	NE
622	591575	AM 37-226	20N	12E	16	SW
623	591576	AM 37-227	20N	12E	16	SE
624	591577	AM 37-228	20N	12E	16	SE
625	591578	AM 37-229	20N	12E	15	SW
626	591579	AM 37-230	20N	12E	15	SW
627	591590	AM 38-226	20N	12E	16	SW
628	591591	AM 38-227	20N	12E	16	SE
629	591592	AM 38-228	20N	12E	16	SE
630	591593	AM 38-229	20N	12E	15	SW

631	591594	AM 38-230	20N	12E	15	SW
632	591605	AM 39-226	20N	12E	16	NW
633	591606	AM 39-227	20N	12E	16	NE
634	591607	AM 39-228	20N	12E	16	NE
635	591608	AM 39-229	20N	12E	15	NW
636	591609	AM 39-230	20N	12E	15	NW
637	591620	AM 40-226	20N	12E	16	NW
638	591621	AM 40-227	20N	12E	16	NE
639	591622	AM 40-228	20N	12E	16	NE
640	591623	AM 40-229	20N	12E	15	NW
641	591624	AM 40-230	20N	12E	15	NW
642	591635	AM 41-225	20N	12E	9	SW
643	591636	AM 41-226	20N	12E	9	SW
644	591637	AM 41-227	20N	12E	9	SE
645	591638	AM 41-228	20N	12E	9	SE
646	591639	AM 41-229	20N	12E	10	SW
647	591640	AM 41-230	20N	12E	10	SW
648	591648	AM 42-223	20N	12E	8	SE
649	591649	AM 42-224	20N	12E	8	SE
650	591650	AM 42-225	20N	12E	9	SW
651	591651	AM 42-226	20N	12E	9	SW
652	591652	AM 42-227	20N	12E	9	SE
653	591653	AM 42-228	20N	12E	9	SE
654	591654	AM 42-229	20N	12E	10	SW
655	591655	AM 42-230	20N	12E	10	SW
656	591661	AM 43-221	20N	12E	8	NW
657	591662	AM 43-222	20N	12E	8	NW
658	591663	AM 43-223	20N	12E	8	NE
659	591664	AM 43-224	20N	12E	8	NE
660	591665	AM 43-225	20N	12E	9	NW
661	591666	AM 43-226	20N	12E	9	NW
662	591667	AM 43-227	20N	12E	9	NE
663	591668	AM 43-228	20N	12E	9	NE
664	591669	AM 43-229	20N	12E	10	NW
665	591670	AM 43-230	20N	12E	10	NW
666	591676	AM 44-219	20N	12E	7	NE
667	591677	AM 44-220	20N	12E	7	NE
668	591678	AM 44-221	20N	12E	8	NW
669	591679	AM 44-222	20N	12E	8	NW
670	591680	AM 44-223	20N	12E	8	NE
671	591681	AM 44-224	20N	12E	8	NE
672	591682	AM 44-225	20N	12E	9	NW
673	591683	AM 44-226	20N	12E	9	NW
674	591684	AM 44-227	20N	12E	9	NE
675	591685	AM 44-228	20N	12E	9	NE
676	591686	AM 44-229	20N	12E	10	NW

677	591687	AM 44-230	20N	12E	10	NW
678	591693	AM 45-217	20N	12E	6	SW
679	591694	AM 45-218	20N	12E	6	SW
680	591695	AM 45-219	20N	12E	6	SE
681	591696	AM 45-220	20N	12E	6	SE
682	591697	AM 45-221	20N	12E	5	SW
683	591698	AM 45-222	20N	12E	5	SW
684	591699	AM 45-223	20N	12E	5	SE
685	591700	AM 45-224	20N	12E	5	SE
686	591701	AM 45-225	20N	12E	4	SW
687	591702	AM 45-226	20N	12E	4	SW
688	591703	AM 45-227	20N	12E	4	SE
689	591704	AM 45-228	20N	12E	4	SE
690	591705	AM 45-229	20N	12E	3	SW
691	591706	AM 45-230	20N	12E	3	SW
692	591712	AM 46-217	20N	12E	6	SW
693	591713	AM 46-218	20N	12E	6	SW
694	591714	AM 46-219	20N	12E	6	SE
695	591715	AM 46-220	20N	12E	6	SE
696	591716	AM 46-221	20N	12E	5	SW
697	591717	AM 46-222	20N	12E	5	SW
698	591718	AM 46-223	20N	12E	5	SE
699	591719	AM 46-224	20N	12E	5	SE
700	591720	AM 46-225	20N	12E	4	SW
701	591721	AM 46-226	20N	12E	4	SW
702	591722	AM 46-227	20N	12E	4	SE
703	591723	AM 46-228	20N	12E	4	SE
704	591724	AM 46-229	20N	12E	3	SW
705	591725	AM 46-230	20N	12E	3	SW
706	591731	AM 47-217	20N	12E	6	NW
707	591732	AM 47-218	20N	12E	6	NW
708	591733	AM 47-219	20N	12E	6	NE
709	591734	AM 47-220	20N	12E	6	NE
710	591735	AM 47-221	20N	12E	5	NW
711	591736	AM 47-222	20N	12E	5	NW
712	591737	AM 47-223	20N	12E	5	NE
713	591738	AM 47-224	20N	12E	5	NE
714	591739	AM 47-225	20N	12E	4	NW
715	591740	AM 47-226	20N	12E	4	NW
716	591741	AM 47-227	20N	12E	4	NE
717	591742	AM 47-228	20N	12E	4	NE
718	591743	AM 47-229	20N	12E	3	NW
719	591744	AM 47-230	20N	12E	3	NW
720	591745	AM 48-217	20N	12E	6	NW
721	591746	AM 48-218	20N	12E	6	NW
722	591747	AM 48-219	20N	12E	6	NE

723	591748	AM 48-220	20N	12E	6	NE
724	591749	AM 48-221	20N	12E	5	NW
725	591750	AM 48-222	20N	12E	5	NW
726	591751	AM 48-223	20N	12E	5	NE
727	591752	AM 48-224	20N	12E	5	NE
728	591753	AM 48-225	20N	12E	4	NW
729	591754	AM 48-226	20N	12E	4	NW
730	591755	AM 48-227	20N	12E	4	NE
731	591756	AM 48-228	20N	12E	4	NE
732	591757	AM 48-229	20N	12E	3	NW
733	591758	AM 48-230	20N	12E	3	NW
734	622359	Eggplant 1	21N	10E	26	NW
735	622360	Eggplant 2	21N	10E	26	NW
736	622361	Eggplant 3	21N	10E	26	NE
737	622362	Eggplant 4	21N	10E	26	NE
738	622363	Eggplant 5	21N	10E	25	NW
739	622364	Eggplant 6	21N	10E	25	NW
740	622365	Eggplant 7	21N	10E	26	SW
741	622366	Eggplant 8	21N	10E	26	SE
742	622367	Eggplant 9	21N	10E	25	SW
743	622368	Eggplant 10	21N	10E	34	NW
744	622369	Eggplant 11	21N	10E	34	NE
745	622370	Eggplant 12	21N	10E	35	NW
746	622371	Eggplant 13	21N	10E	35	NE
747	622372	Eggplant 14	21N	10E	36	NW
748	622373	Eggplant 15	21N	10E	36	NE
749	622374	Eggplant 16	21N	11E	31	NW
750	622375	Eggplant 17	21N	11E	31	NE
751	622376	Eggplant 18	21N	11E	32	NW
752	622377	Eggplant 19	21N	10E	34	SW
753	622378	Eggplant 20	21N	10E	34	SE
754	622379	Eggplant 21	21N	10E	35	SW
755	622380	Eggplant 22	21N	10E	35	SE
756	622381	Eggplant 23	21N	10E	36	SW
757	622382	Eggplant 24	21N	10E	36	SE
758	622383	Eggplant 25	21N	11E	31	SW
759	622384	Eggplant 26	21N	11E	31	SE
760	622385	Eggplant 27	21N	11E	32	SW
761	622386	Eggplant 28	21N	11E	32	SE
762	622387	Eggplant 29	21N	11E	33	SW
763	622388	Eggplant 30	21N	11E	33	SW
764	626222	LUK 1	22N	10E	8	NW
765	626223	LUK 2	22N	10E	8	NE
766	626224	LUK 3	22N	10E	9	NW
767	626225	LUK 4	22N	10E	9	NE
768	626226	LUK 5	22N	10E	10	NW

769	626227	LUK 6	22N	10E	10	NE
770	626228	LUK 7	22N	10E	11	NW
771	626229	LUK 8	22N	10E	11	NE
772	626230	LUK 9	22N	10E	8	SE
773	626231	LUK 10	22N	10E	9	SW
774	626232	LUK 11	22N	10E	9	SE
775	626233	LUK 12	22N	10E	10	SW
776	626234	LUK 13	22N	10E	10	SE
777	626235	LUK 14	22N	10E	11	SW
778	626236	LUK 15	22N	10E	11	SE
779	626237	LUK 16	22N	10E	15	NE
780	626238	LUK 17	22N	10E	14	NW
781	626239	LUK 18	22N	10E	14	NE
782	626240	LUK 19	22N	10E	13	NW
783	626241	LUK 20	22N	10E	13	NE
784	626242	LUK 21	22N	10E	14	SW
785	626243	LUK 22	22N	10E	14	SE
786	626244	LUK 23	22N	10E	13	SW
787	626245	LUK 24	22N	10E	13	SE
788	626246	LUK 25	22N	10E	23	NE
789	626247	LUK 26	22N	10E	24	NW
790	626248	LUK 27	22N	10E	24	NE
791	626249	LUK 28	22N	11E	19	NW
792	626250	LUK 29	22N	11E	19	NE
793	626251	LUK 30	22N	10E	23	SE
794	626252	LUK 31	22N	10E	24	SW
795	626253	LUK 32	22N	10E	24	SE
796	626254	LUK 33	22N	11E	19	SW
797	626255	LUK 34	22N	11E	19	SE
798	626256	LUK 35	22N	10E	27	NE
799	626257	LUK 36	22N	10E	26	NW
800	626258	LUK 37	22N	10E	26	NE
801	626259	LUK 38	22N	10E	25	NW
802	626260	LUK 39	22N	10E	25	NE
803	626261	LUK 40	22N	11E	30	NW
804	626262	LUK 41	22N	11E	30	NE
805	626263	LUK 42	22N	11E	29	NW
806	626264	LUK 43	22N	11E	29	NE
807	626265	LUK 44	22N	11E	28	NW
808	626266	LUK 45	22N	11E	28	NE
809	626267	LUK 46	22N	10E	27	SE
810	626268	LUK 47	22N	10E	26	SW
811	626269	LUK 48	22N	10E	26	SE
812	626270	LUK 49	22N	10E	25	SW
813	626271	LUK 50	22N	10E	25	SE
814	626272	LUK 51	22N	11E	30	SW

815	626273	LUK 52	22N	11E	30	SE
816	626274	LUK 53	22N	11E	29	SW
817	626275	LUK 54	22N	11E	29	SE
818	626276	LUK 55	22N	11E	28	SW
819	626277	LUK 56	22N	11E	28	SE
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822	626280	LUK 59	22N	10E	35	NE
823	626281	LUK 60	22N	10E	36	NW
824	626282	LUK 61	22N	10E	36	NE
825	626283	LUK 62	22N	11E	31	NW
826	626284	LUK 63	22N	11E	31	NE
827	626285	LUK 64	22N	11E	32	NW
828	626286	LUK 65	22N	11E	32	NE
829	626287	LUK 66	22N	11E	33	NW
830	626288	LUK 67	22N	11E	33	NE
831	626289	LUK 68	22N	11E	34	NW
832	626290	LUK 69	22N	11E	34	NE
833	626291	LUK 70	22N	11E	35	NW
834	626292	LUK 71	22N	11E	35	NE
835	626293	LUK 72	22N	11E	36	NW
836	626294	LUK 73	22N	11E	36	NE
837	626295	LUK 74	22N	10E	34	SE
838	626296	LUK 75	22N	10E	35	SW
839	626297	LUK 76	22N	10E	35	SE
840	626298	LUK 77	22N	10E	36	SW
841	626299	LUK 78	22N	10E	36	SE
842	626300	LUK 79	22N	11E	31	SW
843	626301	LUK 80	22N	11E	31	SE
844	626302	LUK 81	22N	11E	32	SW
845	626303	LUK 82	22N	11E	32	SE
846	626304	LUK 83	22N	11E	33	SW
847	626305	LUK 84	22N	11E	33	SE
848	626306	LUK 85	22N	11E	34	SW
849	626307	LUK 86	22N	11E	34	SE
850	626308	LUK 87	22N	11E	35	SW
851	626309	LUK 88	22N	11E	35	SE
852	626310	LUK 89	22N	11E	36	SW
853	626311	LUK 90	22N	11E	36	SE
854	626312	LUK 91	21N	10E	3	NE
855	626313	LUK 92	21N	10E	2	NW
856	626314	LUK 93	21N	10E	2	NE
857	626315	LUK 94	21N	10E	1	NW
858	626316	LUK 95	21N	10E	1	NE
859	626317	LUK 96	21N	11E	6	NW
860	626318	LUK 97	21N	11E	6	NE

861	626319	LUK 98	21N	11E	5	NW
862	626320	LUK 99	21N	11E	5	NE
863	626321	LUK 100	21N	11E	4	NW
864	626322	LUK 101	21N	11E	4	NE
865	626323	LUK 102	21N	11E	3	NW
866	626324	LUK 103	21N	11E	3	NE
867	626325	LUK 104	21N	11E	2	NW
868	626326	LUK 105	21N	11E	2	NE
869	626327	LUK 106	21N	11E	1	NW
870	626328	LUK 107	21N	11E	1	NE
871	626329	LUK 108	21N	12E	6	NW
872	626330	LUK 109	21N	12E	6	NE
873	626331	LUK 110	21N	10E	1	SW
874	626332	LUK 111	21N	10E	1	SE
875	626333	LUK 112	21N	11E	6	SW
876	626334	LUK 113	21N	11E	6	SE
877	626335	LUK 114	21N	11E	5	SW
878	626336	LUK 115	21N	11E	5	SE
879	626337	LUK 116	21N	11E	4	SW
880	626338	LUK 117	21N	11E	4	NW of SE
881	626339	LUK 118	21N	11E	4	SW of SE
882	626340	LUK 119	21N	11E	4	NE of SE
883	626341	LUK 120	21N	11E	3	NW of SW
884	626342	LUK 121	21N	11E	3	NE of SW
885	626343	LUK 122	21N	11E	3	NW of SE
886	626344	LUK 123	21N	11E	3	NE of SE
887	626345	LUK 124	21N	11E	2	SW
888	626346	LUK 125	21N	11E	2	SE
889	626347	LUK 126	21N	11E	1	SW
890	626348	LUK 127	21N	11E	1	SE
891	626349	LUK 128	21N	12E	6	SW
892	626350	LUK 129	21N	12E	6	SE
893	626351	LUK 130	21N	10E	12	NW of NW
894	626352	LUK 131	21N	10E	12	NE of NW
895	626353	LUK 132	21N	10E	12	NW of NE
896	626354	LUK 133	21N	10E	12	NE of NE
897	626355	LUK 134	21N	11E	7	NW of NW
898	626356	LUK 135	21N	11E	7	NE of NW
899	626357	LUK 136	21N	11E	7	NW of NE
900	626358	LUK 137	21N	11E	7	NE of NE
901	626359	LUK 138	21N	11E	8	NW of NW
902	626360	LUK 139	21N	11E	8	NE of NW
903	626361	LUK 140	21N	11E	8	NW of NE
904	626362	LUK 141	21N	11E	8	NE of NE
905	626363	LUK 142	21N	11E	9	NW of NW
906	626364	LUK 143	21N	11E	9	NE of NW

907	626365	LUK 144	21N	11E	9	NW of NE
908	626366	LUK 145	21N	11E	11	NW
909	626367	LUK 146	21N	11E	11	NE
910	626368	LUK 147	21N	11E	12	NW
911	626369	LUK 148	21N	11E	12	NE
912	626370	LUK 149	21N	12E	7	NW
913	626371	LUK 150	21N	12E	7	NE
914	626372	LUK 151	21N	11E	12	SW
915	626373	LUK 152	21N	11E	12	SE
916	626374	LUK 153	21N	12E	7	SW
917	626375	LUK 154	21N	12E	7	SE
918	626376	LUK 155	21N	12E	18	NW
919	626377	LUK 156	21N	12E	18	NE
920	626378	LUK 157	21N	12E	17	NW
921	626379	LUK 158	21N	12E	17	NE
922	626380	LUK 159	21N	12E	16	NW
923	626381	LUK 160	21N	12E	16	NE
924	626382	LUK 161	21N	12E	15	NW
925	626383	LUK 162	21N	12E	15	NE
926	626384	LUK 163	21N	12E	14	NW
927	626385	LUK 164	21N	12E	14	NE
928	626386	LUK 165	21N	12E	18	SW
929	626387	LUK 166	21N	12E	18	SE
930	626388	LUK 167	21N	12E	17	SW
931	626389	LUK 168	21N	12E	17	SE
932	626390	LUK 169	21N	12E	16	SW
933	626391	LUK 170	21N	12E	16	SE
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936	626394	LUK 173	21N	12E	14	SW
937	626395	LUK 174	21N	12E	14	SE
938	626396	LUK 175	21N	12E	20	NW
939	626397	LUK 176	21N	12E	20	NE
940	626398	LUK 177	21N	12E	21	NW
941	626399	LUK 178	21N	12E	21	NE
942	626400	LUK 179	21N	12E	22	NW
943	626401	LUK 180	21N	12E	22	NE
944	626402	LUK 181	21N	12E	23	NW
945	626403	LUK 182	21N	12E	23	NE
946	626404	LUK 183	21N	12E	24	NW
947	626405	LUK 184	21N	12E	24	NE
948	626406	LUK 185	21N	13E	19	NW
949	626407	LUK 186	21N	13E	19	NE
950	626408	LUK 187	21N	12E	20	SW
951	626409	LUK 188	21N	12E	20	SE
952	626410	LUK 189	21N	12E	21	SW

953	626411	LUK 190	21N	12E	21	SE
954	626412	LUK 191	21N	12E	22	SW
955	626413	LUK 192	21N	12E	22	SE
956	626414	LUK 193	21N	12E	23	SW
957	626415	LUK 194	21N	12E	23	SE
958	626416	LUK 195	21N	12E	24	SW
959	626417	LUK 196	21N	12E	24	SE
960	626418	LUK 197	21N	13E	19	SW
961	626419	LUK 198	21N	13E	19	SE
962	626420	LUK 199	21N	12E	29	NW
963	626421	LUK 200	21N	12E	29	NE
964	626422	LUK 201	21N	12E	28	NW
965	626423	LUK 202	21N	12E	28	NE
966	626424	LUK 203	21N	12E	27	NW
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969	626427	LUK 206	21N	12E	26	NE
970	626428	LUK 207	21N	12E	25	NW
971	626429	LUK 208	21N	12E	25	NE
972	626430	LUK 209	21N	13E	30	NW
973	626431	LUK 210	21N	13E	30	NE
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975	626433	LUK 212	21N	13E	29	NE
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983	626441	LUK 220	21N	12E	27	SE
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986	626444	LUK 223	21N	12E	25	SW
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995	626453	LUK 232	21N	12E	34	NW
996	626454	LUK 233	21N	12E	34	NE
997	626455	LUK 234	21N	12E	35	NW
998	626456	LUK 235	21N	12E	35	NE

999	626457	LUK 236	21N	12E	36	NW
1000	626458	LUK 237	21N	12E	36	NE
1001	626459	LUK 238	21N	13E	31	NW
1002	626460	LUK 239	21N	13E	31	NE
1003	626461	LUK 240	21N	13E	32	NW
1004	626462	LUK 241	21N	13E	32	NE
1005	626463	LUK 242	21N	13E	33	NW
1006	626464	LUK 243	21N	13E	33	NE
1007	626465	LUK 244	21N	13E	34	NW
1008	626466	LUK 245	21N	13E	34	NE
1009	626467	LUK 246	21N	13E	35	NW
1010	626468	LUK 247	21N	13E	35	NE
1011	626469	LUK 248	21N	12E	33	SE
1012	626470	LUK 249	21N	12E	34	SW
1013	626471	LUK 250	21N	12E	34	SE
1014	626472	LUK 251	21N	12E	35	SW
1015	626473	LUK 252	21N	12E	35	SE
1016	626474	LUK 253	21N	12E	36	SW
1017	626475	LUK 254	21N	12E	36	SE
1018	626476	LUK 255	21N	13E	31	SW
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1020	626478	LUK 257	21N	13E	32	SW
1021	626479	LUK 258	21N	13E	32	SE
1022	626480	LUK 259	21N	13E	33	SW
1023	626481	LUK 260	21N	13E	33	SE
1024	626482	LUK 261	21N	13E	34	SW
1025	626483	LUK 262	21N	13E	34	SE
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1041	626499	LUK 278	20N	13E	2	NW
1042	626500	LUK 279	20N	13E	2	NE
1043	626501	LUK 280	20N	13E	1	NW
1044	626502	LUK 281	20N	13E	1	NE

1045	626503	LUK 282	20N	14E	6	NW
1046	626504	LUK 283	20N	14E	6	NE
1047	626505	LUK 284	20N	14E	5	NW
1048	626506	LUK 285	20N	14E	5	NE
1049	626507	LUK 286	20N	12E	3	SE
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1051	626509	LUK 288	20N	12E	2	SE
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1054	626512	LUK 291	20N	13E	6	SW
1055	626513	LUK 292	20N	13E	6	SE
1056	626514	LUK 293	20N	13E	5	SW
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1058	626516	LUK 295	20N	13E	4	SW
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1061	626519	LUK 298	20N	13E	3	SE
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1063	626521	LUK 300	20N	13E	2	SE
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1067	626525	LUK 304	20N	14E	6	SE
1068	626526	LUK 305	20N	14E	5	SW
1069	626527	LUK 306	20N	14E	5	SE
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1071	626529	LUK 308	20N	13E	8	NW
1072	626530	LUK 309	20N	13E	8	NE
1073	626531	LUK 310	20N	13E	9	NW
1074	626532	LUK 311	20N	13E	9	NE
1075	626533	LUK 312	20N	13E	10	NW
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1078	626536	LUK 315	20N	13E	11	NE
1079	626537	LUK 316	20N	13E	12	NW
1080	626538	LUK 317	20N	13E	12	NE
1081	626539	LUK 318	20N	14E	7	NW
1082	626540	LUK 319	20N	14E	7	NE
1083	626541	LUK 320	20N	14E	8	NW
1084	626542	LUK 321	20N	14E	8	NE
1085	626543	LUK 322	20N	14E	9	NW
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1088	626546	LUK 325	20N	14E	10	NE
1089	626547	LUK 326	20N	14E	11	NW
1090	626548	LUK 327	20N	14E	11	NE

1091	626549	LUK 328	20N	14E	12	NW
1092	626550	LUK 329	20N	14E	12	NE
1093	626551	LUK 330	20N	15E	7	NW
1094	626552	LUK 331	20N	13E	9	SW
1095	626553	LUK 332	20N	13E	9	SE
1096	626554	LUK 333	20N	13E	10	SW
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1484	629307	WK265	22N	05E	14	NW
1485	629308	WK266	22N	05E	14	NE
1486	629309	WK267	22N	05E	13	NW
1487	629310	WK268	22N	05E	13	NE
1488	629311	WK269	22N	05E	15	SW
1489	629312	WK270	22N	05E	15	SE
1490	629313	WK271	22N	05E	14	SW
1491	629314	WK272	22N	05E	14	SE
1492	629315	WK273	22N	05E	13	SW
1493	629316	WK274	22N	05E	13	SE
1494	629317	WK275	22N	05E	22	NW
1495	629318	WK276	22N	05E	22	NE
1496	629319	WK277	22N	05E	23	NW
1497	629320	WK278	22N	05E	23	NE
1498	629321	WK279	22N	05E	24	NW
1499	629322	WK280	22N	05E	24	NE
1500	629323	WK281	22N	05E	22	SW
1501	629324	WK282	22N	05E	22	SE
1502	629325	WK283	22N	05E	23	SW
1503	629326	WK284	22N	05E	23	SE
1504	629327	WK285	22N	05E	24	SW

1505	629328	WK286	22N	05E	24	SE
1506	629329	WK287	22N	05E	27	NW
1507	629330	WK288	22N	05E	27	NE
1508	629331	WK289	22N	05E	27	SW
1509	629332	WK290	22N	05E	27	SE
1510	629333	WK291	22N	05E	34	NW
1511	629334	WK292	22N	05E	34	NE
1512	629335	WK293	22N	05E	34	SW
1513	629336	WK294	22N	05E	34	SE
1514	634110	EDC 1	21N	10E	12	SW
1515	634111	EDC 2	21N	10E	12	SE
1516	634112	EDC 3	21N	10E	12	SE
1517	634113	EDC 4	21N	11E	7	SW
1518	634114	EDC 5	21N	11E	7	SW
1519	634115	EDC 6	21N	11E	7	SE
1520	634116	EDC 7	21N	11E	7	SE
1521	634117	EDC 8	21N	11E	8	SW
1522	634118	EDC 9	21N	10E	12	SW
1523	634119	EDC 10	21N	10E	12	SE
1524	634120	EDC 11	21N	10E	12	SE
1525	634121	EDC 12	21N	11E	7	SW
1526	634122	EDC 13	21N	11E	7	SW
1527	634123	EDC 14	21N	11E	7	SE
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1547	634143	EDC 34	21N	10E	13	SE
1548	634144	EDC 35	21N	10E	13	SE
1549	634145	EDC 36	21N	11E	18	SW
1550	634146	EDC 37	21N	11E	18	SW

1551	634147	EDC 38	21N	11E	18	SE
1552	634148	EDC 39	21N	11E	18	SE
1553	634149	EDC 40	21N	11E	17	SW
1554	634150	EDC 41	21N	10E	13	SW
1555	634151	EDC 42	21N	10E	13	SE
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1593	634189	EDC 80	21N	11E	19	SE
1594	634190	EDC 81	21N	11E	20	SW
1595	634191	EDC 82	21N	11E	20	SW
1596	634192	EDC 83	21N	11E	20	SE

1597	634193	EDC 84	21N	11E	20	SE
1598	634194	EDC 85	21N	11E	21	SW
1599	634195	EDC 86	21N	10E	25	NE
1600	634196	EDC 87	21N	11E	30	NW
1601	634197	EDC 88	21N	11E	30	NW
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1607	634203	EDC 94	21N	11E	29	NE
1608	634204	EDC 95	21N	11E	28	NW
1609	650291	HOSS 01	22N	10E	18	NW
1610	650292	HOSS 02	22N	10E	18	NE
1611	650293	HOSS 03	22N	10E	17	NW
1612	650294	HOSS 04	22N	10E	17	NE
1613	650295	HOSS 05	22N	10E	16	NW
1614	650296	HOSS 06	22N	10E	16	NE
1615	650297	HOSS 07	22N	10E	15	NW
1616	650298	HOSS 08	22N	10E	18	SW
1617	650299	HOSS 09	22N	10E	18	SE
1618	650300	HOSS 10	22N	10E	17	SW
1619	650301	HOSS 11	22N	10E	17	SE
1620	650302	HOSS 12	22N	10E	16	SW
1621	650303	HOSS 13	22N	10E	16	SE
1622	650304	HOSS 14	22N	10E	15	SW
1623	650305	HOSS 15	22N	10E	19	NW
1624	650306	HOSS 16	22N	10E	19	NE
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1629	650311	HOSS 21	22N	10E	22	NW
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1639	650321	HOSS 31	22N	10E	29	NW
1640	650322	HOSS 32	22N	10E	29	NE
1641	650323	HOSS 33	22N	10E	28	NW
1642	650324	HOSS 34	22N	10E	28	NE

1643	650325	HOSS 35	22N	10E	27	NW
1644	651152	ZED 1	20N	12E	10	NE
1645	651153	ZED 2	20N	12E	11	NW
1646	651154	ZED 3	20N	12E	11	NE
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1653	651161	ZED 10	20N	12E	14	NW
1654	651162	ZED 11	20N	12E	14	NE
1655	651163	ZED 12	20N	12E	13	NW
1656	651164	ZED 13	20N	12E	13	NE
1657	651165	ZED 14	20N	13E	18	NW
1658	651166	ZED 15	20N	13E	18	NE
1659	651167	ZED 16	20N	13E	17	NW
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1661	651169	ZED 18	20N	13E	16	NW
1662	651170	ZED 19	20N	13E	16	NE
1663	651171	ZED 20	20N	13E	15	NW
1664	651172	ZED 21	20N	13E	15	NE
1665	651173	ZED 22	20N	12E	15	SE
1666	651174	ZED 23	20N	12E	14	SW
1667	651175	ZED 24	20N	12E	14	SE
1668	651176	ZED 25	20N	12E	13	SW
1669	651177	ZED 26	20N	12E	13	SE
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1671	651179	ZED 28	20N	13E	18	SE
1672	651180	ZED 29	20N	13E	17	SW
1673	651181	ZED 30	20N	13E	17	SE
1674	651182	ZED 31	20N	13E	16	SW
1675	651183	ZED 32	20N	13E	16	SE
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1680	651188	ZED 37	20N	13E	19	NE
1681	651189	ZED 38	20N	13E	20	NW
1682	651190	ZED 39	20N	13E	20	NE
1683	651191	ZED 40	20N	13E	21	NW
1684	651192	ZED 41	20N	13E	21	NE
1685	651193	ZED 42	20N	13E	22	NW
1686	651194	ZED 43	20N	13E	22	NE
1687	651195	ZED 44	20N	13E	19	SE
1688	651196	ZED 45	20N	13E	20	SW

1689	651197	ZED 46	20N	13E	20	SE
1690	651198	ZED 47	20N	13E	21	SW
1691	651199	ZED 48	20N	13E	21	SE
1692	651200	ZED 49	20N	13E	22	SW
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1694	651202	ZED 51	20N	13E	23	NW
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1699	651207	ZED 56	20N	14E	19	NE
1700	651208	ZED 57	20N	14E	20	NW
1701	651209	ZED 58	20N	14E	20	NE
1702	651210	ZED 59	20N	14E	21	NW
1703	651211	ZED 60	20N	14E	21	NE
1704	651212	ZED 61	20N	14E	22	NW
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1732	651240	ZED 89	20N	14E	30	NW
1733	651241	ZED 90	20N	14E	30	NE
1734	651242	ZED 91	20N	14E	29	NW

1735	651243	ZED 92	20N	14E	29	NE
1736	651244	ZED 93	20N	14E	28	NW
1737	651245	ZED 94	20N	14E	28	NE
1738	651246	ZED 95	20N	14E	27	NW
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1741	651249	ZED 98	20N	14E	26	NE
1742	651250	ZED 99	20N	14E	25	NW
1743	651251	ZED 100	20N	14E	25	NE
1744	651252	ZED 101	20N	15E	30	NW
1745	651253	ZED 102	20N	13E	26	SW
1746	651254	ZED 103	20N	13E	26	SE
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1761	651269	ZED 118	20N	15E	30	SW
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1763	651271	PAL 2	21N	10E	25	NE
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1772	651280	PAL 11	21N	11E	28	NW
1773	651289	PAL 20	21N	10E	25	SE
1774	651290	PAL 21	21N	11E	30	SW
1775	651291	PAL 22	21N	11E	30	SE
1776	651292	PAL 23	21N	11E	29	SW
1777	651293	PAL 24	21N	11E	29	SE
1778	651294	PAL 25	21N	11E	28	SW
1779	651296	PAL 27	21N	11E	32	NE
1780	651297	PAL 28	21N	11E	33	NW

1781	651299	GAP 1	22N	8E	28	NW
1782	651300	GAP 2	22N	8E	28	NE
1783	651301	GAP 3	22N	8E	27	NW
1784	651302	GAP 4	22N	8E	27	NE
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1786	651304	GAP 6	22N	8E	26	NE
1787	651305	GAP 7	22N	8E	25	NW
1788	651306	GAP 8	22N	8E	25	NE
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1791	651309	GAP 11	22N	9E	29	NW
1792	651310	GAP 12	22N	8E	28	SW
1793	651311	GAP 13	22N	8E	28	SE
1794	651312	GAP 14	22N	8E	27	SW
1795	651313	GAP 15	22N	8E	27	SE
1796	651314	GAP 16	22N	8E	26	SW
1797	651315	GAP 17	22N	8E	26	SE
1798	651316	GAP 18	22N	8E	25	SW
1799	651317	GAP 19	22N	8E	25	SE
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1813	651331	GAP 33	22N	9E	31	NE
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1819	651337	GAP 39	22N	9E	34	NE
1820	651338	GAP 40	22N	9E	35	NW
1821	651339	GAP 41	22N	9E	35	NE
1822	651340	GAP 42	22N	9E	36	NW
1823	651341	GAP 43	22N	9E	36	NE
1824	651342	GAP 44	22N	10E	31	NW
1825	651343	GAP 45	22N	10E	31	NE
1826	651344	GAP 46	22N	10E	32	NW

1827	651345	GAP 47	22N	10E	32	NE
1828	651346	GAP 48	22N	10E	33	NW
1829	651347	GAP 49	22N	10E	33	NE
1830	651348	GAP 50	22N	10E	34	NW
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1846	651364	GAP 66	22N	9E	32	SE
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1849	651367	GAP 69	22N	9E	34	SW
1850	651368	GAP 70	22N	9E	34	SE
1851	651369	GAP 71	22N	9E	35	SW
1852	651370	GAP 72	22N	9E	35	SE
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1856	651374	GAP 76	22N	10E	31	SE
1857	651375	GAP 77	22N	10E	32	SW
1858	651376	GAP 78	22N	10E	32	SE
1859	651377	GAP 79	22N	10E	33	SW
1860	651378	GAP 80	22N	10E	33	SE
1861	651379	GAP 81	22N	10E	34	SW
1862	651380	GAP 82	21N	8E	3	NE
1863	651381	GAP 83	21N	8E	2	NW
1864	651382	GAP 84	21N	8E	2	NE
1865	651383	GAP 85	21N	8E	1	NW
1866	651384	GAP 86	21N	8E	1	NE
1867	651385	GAP 87	21N	9E	6	NW
1868	651386	GAP 88	21N	9E	6	NE
1869	651387	GAP 89	21N	9E	2	NW
1870	651388	GAP 90	21N	9E	2	NE
1871	651389	GAP 91	21N	9E	1	NW
1872	651390	GAP 92	21N	9E	1	NE

1873	651391	GAP 93	21N	10E	6	NW
1874	651392	GAP 94	21N	10E	6	NE
1875	651393	GAP 95	21N	10E	5	NW
1876	651394	GAP 96	21N	10E	5	NE
1877	651395	GAP 97	21N	10E	4	NW
1878	651396	GAP 98	21N	10E	4	NE
1879	651397	GAP 99	21N	10E	3	NW
1880	651398	GAP 100	21N	9E	2	SW
1881	651399	GAP 101	21N	9E	2	SE
1882	651400	GAP 102	21N	9E	1	SW
1883	651401	GAP 103	21N	9E	1	SE
1884	651402	GAP 104	21N	10E	6	SW
1885	651403	GAP 105	21N	10E	6	SE
1886	651404	GAP 106	21N	10E	5	SW
1887	651405	GAP 107	21N	10E	5	SE
1888	651406	GAP 108	21N	10E	4	SW
1889	651407	GAP 109	21N	10E	4	SE
1890	651408	GAP 110	21N	10E	3	SW
1891	651409	GAP 111	21N	10E	3	SE
1892	651410	GAP 112	21N	10E	2	SW
1893	651411	GAP 113	21N	10E	2	SE
1894	651412	GAP 114	21N	9E	11	NW
1895	651413	GAP 115	21N	9E	11	NW
1896	651414	GAP 116	21N	9E	11	NE
1897	651415	GAP 117	21N	9E	11	NE
1898	651416	GAP 118	21N	9E	12	NW
1899	651417	GAP 119	21N	9E	12	NW
1900	651418	GAP 120	21N	9E	12	NE
1901	651419	GAP 121	21N	9E	12	NE
1902	651420	GAP 122	21N	10E	7	NW
1903	651421	GAP 123	21N	10E	7	NW
1904	651422	GAP 124	21N	10E	7	NE
1905	651423	GAP 125	21N	10E	7	NE
1906	651424	GAP 126	21N	10E	8	NW
1907	651425	GAP 127	21N	10E	8	NW
1908	651426	GAP 128	21N	10E	8	NE
1909	651427	GAP 129	21N	10E	8	NE
1910	651428	GAP 130	21N	10E	9	NW
1911	651429	GAP 131	21N	10E	9	NW
1912	651430	GAP 132	21N	10E	9	NE
1913	651431	GAP 133	21N	10E	9	NE
1914	651432	GAP 134	21N	10E	10	NW
1915	651433	GAP 135	21N	10E	10	NW
1916	651434	GAP 136	21N	10E	10	NE
1917	651435	GAP 137	21N	10E	10	NE
1918	651436	GAP 138	21N	10E	11	NW

1919	651437	GAP 139	21N	10E	11	NW
1920	651438	GAP 140	21N	10E	11	NE
1921	651439	GAP 141	21N	10E	11	NE
1922	651440	GAP 142	21N	9E	5	NW
1923	651441	GAP 143	21N	9E	5	NE
1924	651442	GAP 144	21N	9E	4	NW
1925	651443	GAP 145	21N	9E	4	NE
1926	651444	GAP 146	21N	9E	3	NW
1927	651445	GAP 147	21N	9E	3	NE
1928	651446	GAP 148	21N	8E	3	SE
1929	651447	GAP 149	21N	8E	2	SW
1930	651448	GAP 150	21N	8E	2	SE
1931	651449	GAP 151	21N	8E	1	SW
1932	651450	GAP 152	21N	8E	1	SE
1933	651451	GAP 153	21N	9E	6	SW
1934	651452	GAP 154	21N	9E	6	SE
1935	651453	GAP 155	21N	9E	5	SW
1936	651454	GAP 156	21N	9E	5	SE
1937	651455	GAP 157	21N	9E	4	SW
1938	651456	GAP 158	21N	9E	4	SE
1939	651457	GAP 159	21N	9E	3	SW
1940	651458	GAP 160	21N	9E	3	SE
1941	651459	GAP 161	21N	8E	10	NE
1942	651460	GAP 162	21N	8E	11	NW
1943	651461	GAP 163	21N	8E	11	NE
1944	651462	GAP 164	21N	8E	12	NW
1945	651463	GAP 165	21N	8E	12	NE
1946	651464	GAP 166	21N	9E	7	NW
1947	651465	GAP 167	21N	9E	7	NE
1948	651466	GAP 168	21N	9E	8	NW
1949	651467	GAP 169	21N	9E	8	NE
1950	651468	GAP 170	21N	9E	9	NW
1951	651469	GAP 171	21N	9E	9	NE
1952	651470	GAP 172	21N	9E	10	NW
1953	651471	GAP 173	21N	9E	10	NE
1954	651472	GAP 174	21N	8E	10	SE
1955	651473	GAP 175	21N	8E	11	SW
1956	651474	GAP 176	21N	8E	11	SE
1957	651475	GAP 177	21N	8E	12	SW
1958	651476	GAP 178	21N	8E	12	SE
1959	651477	GAP 179	21N	9E	7	SW
1960	651478	GAP 180	21N	9E	7	SE
1961	651479	GAP 181	21N	9E	8	SW
1962	651480	GAP 182	21N	9E	8	SE
1963	651481	GAP 183	21N	9E	9	SW
1964	651482	GAP 184	21N	9E	9	SE

1965	651483	GAP 185	21N	9E	10	SW
1966	651484	GAP 186	21N	9E	10	SE
1967	651485	GAP 187	21N	8E	15	NE
1968	651486	GAP 188	21N	8E	14	NW
1969	651487	GAP 189	21N	8E	14	NE
1970	651488	GAP 190	21N	8E	13	NW
1971	651489	GAP 191	21N	8E	13	NE
1972	651490	GAP 192	21N	9E	18	NW
1973	651491	GAP 193	21N	9E	18	NE
1974	651492	GAP 194	21N	9E	17	NW
1975	651493	GAP 195	21N	9E	17	NE
1976	651494	GAP 196	21N	9E	16	NW
1977	651495	GAP 197	21N	9E	16	NE
1978	651496	GAP 198	21N	9E	15	NW
1979	651497	GAP 199	21N	9E	15	NE
1980	655537	ZED 119	20N	12E	12	NW
1981	655538	ZED 120	20N	12E	12	NE
1982	655539	ZED 121	20N	13E	7	NW
1983	655540	ZED 122	20N	13E	7	SW
1984	655541	ZED 123	20N	13E	7	SE
1985	655542	ZED 124	20N	13E	8	SW
1986	655543	ZED 125	20N	13E	8	SE
1987	655648	KG 1	21N	11E	11	SW
1988	655649	KG 2	21N	11E	11	SE
1989	655650	KG 3	21N	11E	14	NW
1990	655651	KG 4	21N	11E	14	NE
1991	655652	KG 5	21N	11E	13	NW
1992	655653	KG 6	21N	11E	13	NE
1993	655654	KG 7	21N	11E	13	SE
1994	655655	KG 8	21N	11E	24	NE
1995	714584	East DH 1	22N	10E	15	SE
1996	714585	East DH 2	22N	10E	22	NE
1997	714586	East DH 3	22N	10E	23	NW
1998	714587	East DH 4	22N	10E	22	SE
1999	714588	East DH 5	22N	10E	23	SW
2000	714589	COBRE 1	22N	7E	2	NW
2001	714590	COBRE 2	22N	7E	2	NE
2002	714591	COBRE 3	22N	7E	1	NW
2003	714592	COBRE 4	22N	7E	1	NE
2004	714593	COBRE 5	22N	8E	6	NW
2005	714594	COBRE 6	22N	8E	6	NE
2006	714595	COBRE 7	22N	8E	5	NW
2007	714596	COBRE 8	22N	8E	5	NE
2008	714597	COBRE 9	22N	7E	2	SW
2009	714598	COBRE 10	22N	7E	2	SE
2010	714599	COBRE 11	22N	7E	1	SW

2011	714600	COBRE 12	22N	7E	1	SE
2012	714601	COBRE 13	22N	8E	6	SW
2013	714602	COBRE 14	22N	8E	6	SE
2014	714603	COBRE 15	22N	8E	5	SW
2015	714604	COBRE 16	22N	8E	5	SE
2016	714605	COBRE 17	22N	7E	11	NW
2017	714606	COBRE 18	22N	7E	11	NE
2018	714607	COBRE 19	22N	7E	12	NW
2019	714608	COBRE 20	22N	7E	12	NE
2020	714609	COBRE 21	22N	8E	7	NW
2021	714610	COBRE 22	22N	8E	7	NE
2022	714611	COBRE 23	22N	8E	8	NW
2023	714612	COBRE 24	22N	8E	8	NE
2024	714613	COBRE 25	22N	8E	9	NW
2025	714614	COBRE 26	22N	8E	9	NE
2026	714615	COBRE 27	22N	8E	7	SE
2027	714616	COBRE 28	22N	8E	8	SW
2028	714617	COBRE 29	22N	8E	8	SE
2029	714618	COBRE 30	22N	8E	9	SW
2030	714619	COBRE 31	22N	8E	9	SE
2031	714620	COBRE 32	22N	8E	17	NW
2032	714621	COBRE 33	22N	8E	17	NE
2033	714622	COBRE 34	22N	8E	16	NW
2034	714623	COBRE 35	22N	8E	16	NE
2035	714624	COBRE 36	22N	8E	15	NW
2036	714625	COBRE 37	22N	8E	15	NE
2037	714626	COBRE 38	22N	8E	17	SE
2038	714627	COBRE 39	22N	8E	16	SW
2039	714628	COBRE 40	22N	8E	16	SE
2040	714629	COBRE 41	22N	8E	15	SW
2041	714630	COBRE 42	22N	8E	15	SE
2042	714631	COBRE 43	22N	8E	21	NW
2043	714632	COBRE 44	22N	8E	21	NE
2044	714633	COBRE 45	22N	8E	22	NW
2045	714634	COBRE 46	22N	8E	22	NE
2046	714635	COBRE 47	22N	8E	23	NW
2047	714636	COBRE 48	22N	8E	23	NE
2048	714637	COBRE 49	22N	8E	21	SW
2049	714638	COBRE 50	22N	8E	21	SE
2050	714639	COBRE 51	22N	8E	22	SW
2051	714640	COBRE 52	22N	8E	22	SE
2052	714641	COBRE 53	22N	8E	23	SW
2053	714642	COBRE 54	22N	8E	23	SE
2054	714643	West Horse 1	22N	9E	12	SW
2055	714644	West Horse 2	22N	9E	12	SE
2056	714645	West Horse 3	22N	10E	7	SW

2057	714646	West Horse 4	22N	10E	7	SE
2058	714647	West Horse 5	22N	10E	8	SW
2059	714648	West Horse 6	22N	9E	13	NW
2060	714649	West Horse 7	22N	9E	13	NE
2061	714650	West Horse 8	22N	9E	13	SW
2062	714651	West Horse 9	22N	9E	13	SE
2063	714652	West Horse 10	22N	9E	24	NW
2064	714653	West Horse 11	22N	9E	24	NE
2065	714654	West Horse 12	22N	9E	24	SW
2066	714655	West Horse 13	22N	9E	24	SE
2067	714656	West Horse 14	22N	9E	27	NE
2068	714657	West Horse 15	22N	9E	26	NW
2069	714658	West Horse 16	22N	9E	26	NE
2070	714659	West Horse 17	22N	9E	25	NW
2071	714660	West Horse 18	22N	9E	25	NE
2072	714661	West Horse 19	22N	9E	27	SE
2073	714662	West Horse 20	22N	9E	26	SW
2074	714663	West Horse 21	22N	9E	26	SE
2075	714664	West Horse 22	22N	9E	25	SW
2076	714665	West Horse 23	22N	9E	25	SE
2077	714748	AM 37-225	20N	12E	16	SW
2078	714749	AM 38-225	20N	12E	16	SW
2079	714750	AM 39-225	20N	12E	16	NW
2080	714751	AM 40-225	20N	12E	16	NW
2081	714752	AM 41-223	20N	12E	8	SE
2082	714753	AM 41-224	20N	12E	8	SE
2083	714754	AM 43-219	20N	12E	7	NE
2084	714755	AM 43-220	20N	12E	7	NE
2085	714756	AM 38-217	20N	12E	18	SW
2086	714757	AM 38-219	20N	12E	18	SE
2087	714758	AM 38-221	20N	12E	17	SW
2088	714759	AM 38-223	20N	12E	17	SE
2089	714760	AM 42-219	20N	12E	7	SE
2090	714761	AM 42-221	20N	12E	8	SW
2091	714762	AM 44-217	20N	12E	7	NW

2092	714763	AM 40-217	20N	12E	18	NW
2093	714764	AM 40-219	20N	12E	18	NE
2094	714765	AM 40-221	20N	12E	17	NW
2095	714766	AM 40-223	20N	12E	17	NE
2096	714767	AM 42-217	20N	12E	7	SW
2097	714768	KG 9	21N	12E	19	NW
2098	714769	KG 10	21N	12E	19	NE
2099	714770	KG 11	21N	12E	19	SW
2100	714771	KG 12	21N	12E	19	SE
2101	715147	Cobre 55	23N	7E	14	SW
2102	715148	Cobre 56	23N	7E	14	SE
2103	715149	Cobre 57	23N	7E	13	SW
2104	715150	Cobre 58	23N	7E	23	NW
2105	715151	Cobre 59	23N	7E	23	NE
2106	715152	Cobre 60	23N	7E	24	NW
2107	715153	Cobre 61	23N	7E	24	NE
2108	715154	Cobre 62	23N	8E	19	NW
2109	715155	Cobre 63	23N	7E	23	SE
2110	715156	Cobre 64	23N	7E	24	SW
2111	715157	Cobre 65	23N	7E	24	SE
2112	715158	Cobre 66	23N	8E	19	SW
2113	715159	Cobre 67	23N	8E	19	SE
2114	715160	Cobre 68	23N	7E	26	NE
2115	715161	Cobre 69	23N	7E	25	NW
2116	715162	Cobre 70	23N	7E	25	NE
2117	715163	Cobre 71	23N	8E	30	NW
2118	715164	Cobre 72	23N	8E	30	NE
2119	715165	Cobre 73	23N	8E	29	NW
2120	715166	Cobre 74	23N	8E	29	NE
2121	715167	Cobre 75	23N	7E	26	SE
2122	715168	Cobre 76	23N	7E	25	SW
2123	715169	Cobre 77	23N	7E	25	SE
2124	715170	Cobre 78	23N	8E	30	SW
2125	715171	Cobre 79	23N	8E	30	SE
2126	715172	Cobre 80	23N	8E	29	SW
2127	715173	Cobre 81	23N	8E	29	SE
2128	715174	Cobre 82	23N	7E	36	NW
2129	715175	Cobre 83	23N	7E	36	NE
2130	715176	Cobre 84	23N	8E	31	NW
2131	715177	Cobre 85	23N	8E	31	NE
2132	715178	Cobre 86	23N	8E	32	NW
2133	715179	Cobre 87	23N	8E	32	NE
2134	715180	Cobre 88	23N	8E	31	SE
2135	715181	Cobre 89	23N	8E	32	SW
2136	715182	Cobre 90	23N	8E	32	SE